Summary of Select Public Information on Environmental Geochemistry of Duluth Complex Rock

Prepared in Consideration of the Proposed Mineral Withdrawal Area, Superior National Forest, Minnesota

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EXHIBIT 4

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1 Executive Summary

The environmental geochemistry of rock from the Duluth Complex is well understood, due, in part, to a complementary set of research programs undertaken by researchers at the Minnesota Department of Natural Resources and others, including members of industry and academic community. These programs include:

- investigation of full-scale waste rock stockpiles at an operating mine,
- field leach test piles with a program duration of over 38 years,
- numerous laboratory dissolution tests on samples analogous to the field-based programs, and
- extensive solids characterization on samples from test programs.

The United States Forest Service has requested that the Department of Interior, Bureau of Land Management, withdraw 234,328 acres of the Superior National Forest located within the Rainy River watershed from disposition under mineral leasing laws for a period of 20 years. Before acting on USFS's withdrawal application, the federal agencies have indicated they would prepare an environmental impact statement in which they would analyze the impact of this action. Due to the extensive existing information on environmental geochemistry of Duluth Complex rock, there is already a robust understanding of chemical process that occur during weathering of this rock; the chemical constituents have the greatest potential to be released during weathering, and at what rate; the factors that lead to acid drainage; the factors that influence metal mobility in the system; and approaches that can be applied as mitigation.

Major outcomes of the previous characterization programs include the following findings for Duluth Complex rock:

1. Oxidation of sulfide minerals may release acidity and metals into mine contact water. Neutralization potential is provided by silicate mineral dissolution. Initial sulfur content of the rock is correlated with sulfate concentration in the leachate and anticorrelated with pH.
2. Tests on samples with less than or equal to 0.22% sulfate concentration have not produced acidic drainage. Dissolved metals in leachate associated with this rock may, without treatment or mitigation, be elevated relative to surface water quality standards.
3. Both laboratory testing and field leach piles indicate that rocks with higher sulfur concentration (starting at 0.4% or greater) do have, in the absence of appropriate environmental controls, the potential to produce acidic drainage under certain circumstances. These conditions may take significant time to develop.

4. Sulfate release rates decrease with time (over 90% decrease in 38 years).
5. At sufficiently low sulfur concentrations, rock drainage has been observed to meet surface water quality standards for discharge with no mitigation or water treatment necessary.
6. Constructed wetlands have been shown to be effective as a passive, or semi-passive, treatment option for removing metals from drainage.

In addition to these general outcomes, test work provides information on sulfate and metal release rates, factors for scaling laboratory-derived rates to operational-scale features, upper-end dissolved metals concentrations for neutral pH mine contact water, particle size distribution of potential future waste rock, and stockpile hydrology. Considered collectively, the available data, analyses, and studies indicate that the geochemical characteristics of Duluth Complex rock are predictable and attributable to established geochemical mechanisms. Therefore, this rock is amenable to industry mitigation measures, as overseen by federal and state permitting, that can prevent or minimize impacts associated with acid drainage or metals.[1] This available data and analyses provides the federal agencies with an abundance of existing information with which to assess potential impacts from mining activities in the proposed mineral withdrawal area.

2 Introduction

Environmental geochemistry of non-ore grade rock ("waste rock") from the Duluth Complex has been exceptionally well-studied by government researchers and mining project proponents. Waste characterization and research programs focused on environmental geochemistry of Duluth Complex rock are remarkable with regard to their long durations (multi-decadal, for some studies); the large number of samples tested (100s); the breadth of test scales represented (ranging from small-scale laboratory reactors to a full-scale operating mine); and the thorough solid phase analysis that commonly accompanies the experimental work. The result of this sustained effort is a robust technical basis that can be used in planning for and assessing water quality associated with potential future mining operations.

The United States Forest Service (USFS) has requested that the Department of Interior withdraw 234,328 acres of the Superior National Forest (SNF) located within the Rainy River watershed ("proposed withdrawal area") from disposition under mineral and geothermal leasing laws for a period of 20 years. USFS has also indicated it intends to amend the Land and Management Plan for the SNF with respect to minerals exploration and development in the Rainy River watershed. In connection with these proposed actions, the federal agencies intend to prepare an

[1] This report does not opine on the efficacy of such measures or the specific application of such measures to any proposed mine operations, but notes that a review of available literature, analysis, and data indicates that various measures are available to manage the Duluth Complex rock.

environmental impact statement (EIS), which is expected to include studies of the potential impacts of mining federal minerals in the Rainy River watershed (the "no action" alternative).

Any required environmental review would likely include some form of evaluation of water quality associated with mining in the proposed mineral withdrawal area. In general, water quality is dependent on: 1) the specific plan for waste rock and water management[2]; 2) climate; 3) mining and processing methods; 4) performance of engineered systems; and 5) intrinsic geochemical properties of the ore and waste rock (for example, potential for producing acidic drainage, release rates for chemical constituents, and the factors that control those release rates). With regard to the fifth item, since the recognized mineral deposits located within the proposed mineral withdrawal area are hosted by Duluth Complex rock, there is already a large existing body of information on geochemical behavior of potential waste rock. This report documents the extensive scope of this existing information and highlights some of the key outcomes of these geochemistry programs.

3 Background: Geologic Context of the Proposed Mineral Withdrawal Area

The proposed withdrawal area is located in the Rainy River Watershed, south of the Boundary Waters Canoe Area Wilderness (BWCAW), in northeastern Minnesota. A map showing the proposed withdrawal boundary, along with, bedrock geology in the region and recognized mineral deposits located within the proposed application boundary, is portrayed in Figure 3-1. The known mineral deposits that fall within the application boundary are hosted by the South Kawishiwi Intrusion of the Duluth Complex.

[2]Developing a waste rock and water management plan that is appropriate for the site-specific materials and water quality objectives is a key component to modern mine development. During mine development, a company would identify environmental objectives for the project, and, then, work backwards to design waste rock and water management strategies that would meet those objectives.

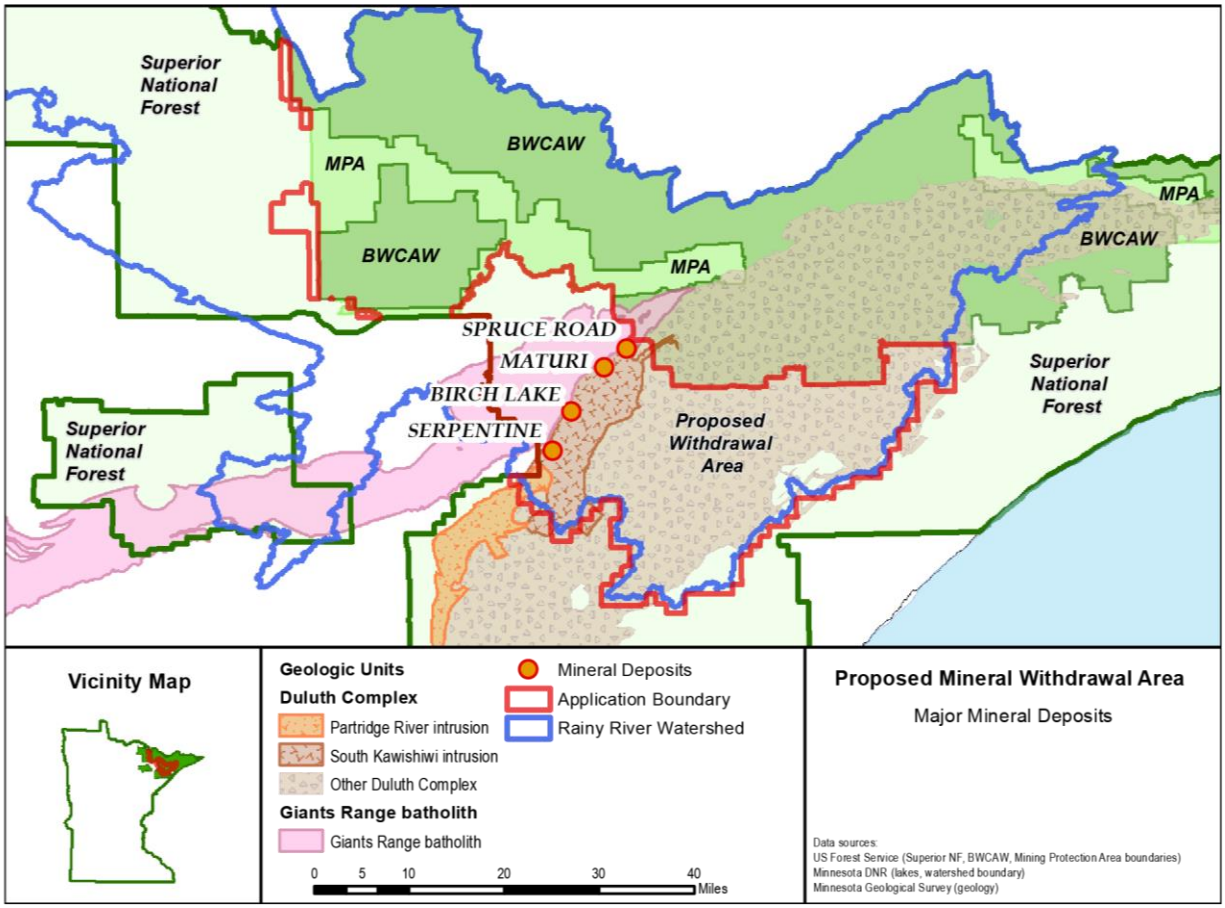

Figure 3-1. Map showing bedrock geology of the proposed mineral withdrawal area.

4 Geochemical Characterization of Duluth Complex Rock

Study of the environmental geochemistry of Duluth Complex rocks has been ongoing in Minnesota for approximately 40 years. A summary of test programs that have produced publicly available waste characterization data is shown in Large Table 1. The Minnesota Department of Natural Resources (MN DNR) produced a summary of research in 2012 (Lapakko and Antonson 2012), which is included as Attachment 1. Brief discussions on five of these programs with particular historical significance and/or are most directly relevant to mining in the proposed withdrawal area are provided below. These summaries are based on a literature review of publicly-available information, relevant portions of which are referenced in this report.

4.1 Regional Copper-Nickel Study

Exploration activities for Cu-Ni mineralization in the Duluth Complex began in the early 1950s[3]. For over twenty following years, rock from the Duluth Complex was disturbed during

[3]Mineralized rocks with copper staining where discovered near Spruce Road by F.S. Childers in 1948. Inco and Bear Creek Mining began exploration programs in 1952 (Miller et al. 2002).

exploration activities; and blasted, excavated, and stockpiled (at Dunka Mine, see below) with no significant work done to characterize its environmental geochemistry. In 1974, the International Nickel Company (INCO) requested approval to establish an open pit mine near Spruce Road, in area which is now part of a larger deposit, more generally known as the Maturi deposit), and AMAX Inc. (AMAX) announced plans to sink an exploration shaft into, what is now called, the Mesaba deposit. Due to the intense public interest in potential environmental impacts from these proposed projects, the Minnesota Legislature instructed the predecessor to the Minnesota Environmental Quality Board to conduct a comprehensive study of potential environmental impacts from this type of mining, referred to as the "Regional Copper-Nickel Study". Participants in the study included representatives of the MN DNR, other state agencies, and the University of Minnesota.

The Regional Copper-Nickel Study was conducted between 1976-1979 and had a scope that encompassed social, economic, ecological, air quality, and water quality potential impacts. It included research into environmental geochemistry of rock from the Duluth Complex, consisting of a complementary suite of laboratory tests (Hoffmann and Eisenreich 1979), field leach test piles (Eger et al. 1979), and operational-scale monitoring (Thingvold et al. 1979; Eger and Lapakko 1980). Some of this evaluation work is still ongoing. Reports with data on these respective efforts are provided as Attachments 2-4.

4.2 Dunka Mine Stockpiles

The Dunka Mine was an open pit taconite mine that operated from 1962 to 1995. In the course of accessing the taconite ore, approximately 50 million tons of rock from the overlying South Kawishiwi Intrusion of the Duluth Complex were removed and stockpiled alongside the open pit. This rock was segregated into separate stockpiles according to copper and nickel concentration, which resulted in stockpiles that were also, in effect, segregated by sulfur concentration. The low copper-nickel, but high sulfur, hornsfels from the Virginia Formation, however, was placed with the low copper-nickel-sulfur rock from the Duluth Complex, which increased the overall sulfur content of these stockpiles.

Monitoring of stockpile flow and drainage chemistry, as observed at stockpile seeps, began in 1979, with the Regional Copper-Nickel Study, and continued until at least 1993 (MN DNR 1996). Monitoring of outfalls continues at the site, and has been used for both research purposes and for compliance with permitting requirements. Data collected from 1976-1993 is included as Attachment 5.

Major outcomes of research at Dunka Mine

Outcomes of the work conducted at the Dunka Mine are:

- Measurements on seepage flow and chemistry have been used to calculate constituent loading rates resulting from weathering of the stockpiled Duluth Complex rock (MN DNR 1996).

- A comparison of sulfate loading between these operational-scale stockpiles and laboratory tests has been used to develop a factor for scaling laboratory-based rates to full-scale system (Lapakko and Olson 2015), provided as Attachment 6.

- The seepage from these stockpiles has remained predominantly at neutral pH with isolated values observed as low as 4.2 (MN DNR 1996). However, copper, nickel, cobalt, and zinc have been observed to be elevated in seepage.

An additional major focus of the research conducted at this site was in support of design, testing, and monitoring of mitigation and treatment strategies for impacted mine water. Surface water was diverted away from Duluth Complex rock. The stockpiles were capped, and seepage was directed towards constructed wetland systems that were designed for non-mechanical treatment of the metalliferous water. These activities have been found to significantly reduce water quality impacts for the site (Eger et al. 2000; ITRC 2010). Reports describing these activities are provided as Attachments 7 and 8.

Significance for Proposed Mineral Withdrawal Area

The Dunka Mine stockpiles contain Duluth Complex rock from the same intrusion as is present in the proposed mineral withdrawal area. Thus, information on constituent loading rates and stockpile hydrology may be useful in evaluating potential mitigation measures for mining or as part of an environmental review process. The stockpiles at Dunka Mine, however, are unlikely to be entirely analogous to those at a future potential mine. Stockpiling began on the site in 1965, prior to establishment of modern environmental regulatory frameworks, such as those represented by Minnesota Minelands Reclamation Act (1969) or the Federal Clean Water Act (1972). Consistent with the time period of operations, waste-rock management strategies at Dunka Mine were initiated before modern water quality standards were developed and "planning for closure" became industry best practice; detailed waste characterization was not conducted in advance of waste-rock excavation.

4.3 South Kawishiwi Intrusion Quarry Sites

Site investigations were conducted around two abandoned dimension stone quarries that had been mining Duluth Complex rock from the South Kawishiwi Intrusion. Both sites contained small pit lakes and waste rock stockpiles. The MN DNR collected samples of both the waste rock stockpiles and pit lake water.

Major outcomes from site investigations at quarry sites

The major outcome of the quarry study is the observation that pit lake water associated with low sulfur (less than 0.05%) rock had both neutral pH and sufficiently low dissolved metals as to meet surface water quality standards, even without any formal reclamation activities having been conducted at the sites (Lapakko and Oberhelman 1993). The report from this investigation is included as Attachment 9.

Significance for Proposed Mineral Withdrawal Area

The rock that is the subject of the investigation is from the same intrusion of the Duluth Complex as is primarily located within the proposed mineral withdrawal area. While the rock is not closely associated with mineralization, and accordingly, contains only a small amount of total sulfur, the collected data may be a useful source of information for low sulfur waste rock that might be excavated and managed on a future mining operation. An especially notable outcome of this study is the finding that for sufficiently low sulfur Duluth Complex rock, no mitigation, engineering controls, or water treatment are needed to maintain contact water that meets water quality standards.

4.4 AMAX Field Leach Test Piles

In 1976, AMAX completed a 1,700-ft exploratory shaft into the "Local Boy" ore zone of the Mesaba Deposit[4] (Severson and Hauck 2008). Development rock, equivalent to lean ore, from that effort was placed into six field leach test piles, each approximately 820 to 1300 metric tons, with sulfur contents of approximately 0.6-1.4%. The piles were underlain by an impervious liner. Drainage from the piles was collected and analyzed for chemistry. The leach piles were monitored from 1978-93, with data reported in Lapakko (1993), included as Attachment 10.

Three of the piles were covered in 1978, and the effect of different cover designs was gauged through continued drainage monitoring and comparison between the six piles. Campaigns were conducted in 1982 and 1994 to collect and characterize the solids within portions of the piles for chemistry, mineralogy, and particle size distribution (Lapakko et al. 2004). The report on these efforts is included as Attachment 11. The leach piles were mostly reclaimed in 1994. However, portions of several of the piles were relocated to the MN DNR facility in Hibbing, MN. Water has continued to be collected and monitored for flow and pH analysis, but only intermittently for chemistry testing.

[4]Referred to, at the time, as the "Babbit Deposit". The exploratory shaft at this site is sometimes referred to as the "Minnamax shaft".

Major outcomes of AMAX test pile program

Major outcomes of this test program include:

- All six piles, which had sulfur content exceeding 0.6%, started by producing neutral pH drainage, which decreased in pH over time, eventually becoming acidic. For the piles with sulfur content of approximately 0.6%, drainage pH decreased gradually from between 7 and 8 to the range of 4.8 to 5.3 (Lapakko 1993).

- Sulfate release rates and neutralization potential were calculated for all piles.

- Release rate data were compared laboratory data to draw conclusions about scaling effects (Lapakko 1994), Attachment 12.

- Recent work extended the evaluation of drainage chemistry through to 2015, showing approximately 90% decrease in sulfate release rate from the start to end of the 38 years of data (Wenz 2016), Attachment 13.

Significance for Proposed Mineral Withdrawal Area

The Duluth Complex rock used in these test piles, while from the Partridge River Intrusion, is very similar in terms of mineralogy and lithology to rock in much of the proposed mineral withdrawal area. The dataset that is derived from this experimental program is uniquely valuable to assessing potential future water quality because of the exceptionally long record of data (approximately 38 years) that demonstrates an eventual decrease in sulfate concentration in the drainage as sulfide mineral source is depleted over time. In addition, because of the relatively large scale of test piles, and the completeness of the analytical suite, the data can also be used to assess solubility-controlled metal concentration in drainage, especially during early years, when drainage from the 0.6% sulfur piles was circumneutral pH. Data from these programs have provided hydrologic information that could be applied to a full-scale system.

4.5 South Kawishiwi Intrusion Blast Hole Tests

The MN DNR initiated testing of samples collected from blast holes at Dunka Mine in 1989, following a method that was developed by the agency for this purpose. A 75-gram crushed sample that had been sieved to isolate the -11/+270 mesh fraction was placed in a custom reactor that allowed for the sample to be exposed to atmosphere while sitting on a filter with a chamber underneath. Once a week, the sample was flushed with 200 ml of distilled-deionized water. It remains in contact with the crushed rock for between four and seven minutes; then, the water is vacuum-pumped to the lower chamber, where it is collected and analyzed for chemical composition. Up to 24 years of data from these test were published in Kellogg et al. (2014), which is included as Attachment 14.

Major outcomes of the study

Over the entire 24 years of testing, samples with sulfur contents of 0.22% or less maintained circumneutral pH. In a number of samples, there was a significant time lag before acidic conditions developed. Minimum pH and maximum sulfate release rate were observed after 6 to 8 years of dissolution. The initial sulfur concentration of the sample was directly correlated with the sulfate concentration in the drainage and anticorrelated with pH. On the basis of drainage pH and chemistry, MN DNR proposed waste rock management criteria that would divide waste rock into three groups:

- Group I (S%≤0.22%): non-potentially acid generating; disposal would "probably require no rigorous reclamation"[5]
- Group II (0.40≤%S≤0.70): minimum pH observed in testing was 3.8; management requirements would be intermediate to those for Group I and Group III
- Group III (0.70<%S≤1.65): minimum pH observed in testing was 3; "would require the most rigorous control measures".

Significance for Proposed Mineral Withdrawal Area

The rock samples used in these tests are Duluth Complex rock from the same intrusion as is predominant in the proposed mineral withdrawal area. Proposed categories for mine waste management and release rates (after applying a scale factor, for example, from Lapakko and Olson (2015)) could potentially be applied to a mining operation in the area for purposes of determining appropriate environmental controls and mitigation measures whether during permitting or even scoping of an environmental review process.

5 Key Conclusions from Test Programs

The aforementioned test programs have provided a robust scientific basis to assess environmental geochemistry of Duluth Complex rock. Key conclusions from these test programs include:

1. Oxidation of sulfide minerals, in the absence of any environmental controls or mitigation measures, releases acidity and metals into mine contact water. Neutralization potential is provided by silicate mineral dissolution. Initial sulfur content of the rock is correlated with sulfate concentration in the leachate and anticorrelated with pH.

[5] The discontinuity in sulfur concentrations is due to lack of samples at those intermediate sulfur concentrations.

2. Test on samples with less than or equal to 0.22% have not produced acidic drainage. Dissolved metals in leachate associated with this rock may, without treatment or mitigation, be elevated relative to surface water quality standards.
3. Both laboratory testing and field leach piles indicate that rocks with higher sulfur concentration (starting at 0.4% or greater) do have potential to produce acidic drainage. Acidic conditions may take several years to develop.
4. Sulfate release rates decrease with time (over 90% decrease in 38 years).
5. At sufficiently low sulfur concentrations, drainage has been observed to meet surface water quality standards for discharge with no mitigation or water treatment necessary.
6. Constructed wetlands have been shown to be effective as a passive, or semi-passive, treatment option for removing metals from drainage.

In addition to these general outcomes, previous testwork provides information on sulfate and metal release rates, factors for scaling laboratory-derived rates to operational-scale features, upper end dissolved metals concentrations for neutral pH mine contact water, particle size distribution of potential future waste rock, and stockpile hydrology. Information provided from these studies indicates that the geochemical characteristics of Duluth Complex rock are predictable and attributable to understood geochemical mechanisms. As such, this rock would be amenable to industry mitigation measures, as overseen by federal and state permitting, that can prevent or minimize impacts associated with acid drainage or metals.

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Large Table 1. Summary of characterization programs potentially relevant to proposed mineral withdrawal area

	Test type	Conducted By	Test Quantity	Sample Provenance	Test Duration	Select References
Duluth Complex: South Kawishiwi Intrusion	Stockpile Monitoring	Erie Mining Company; LTV Steel Mining Company; Cliffs Natural Resources; MN DNR	50 million tons of waste rock, over 300 acres	Dunka Mine Site	Stockpile drainage chemistry reported from 1976-1993; monitoring ongoing	Eger et al 2000; MN DNR 1996
	Site Investigation	MN DNR	2 abandoned dimension stone quarries	South Kawishiwi Intrusion, Ely-Isabella area	One-time site investigation	Lapakko and Oberhelman, 1993
	Small MN DNR Reactors	MN DNR	15 75-gram samples (run in duplicate, 30 tests total)	Dunka Mine Site (collected from blast holes)	Up to 1252 weeks of data reported in 2014 with some test ongoing	Kellogg et al, 2014
	Small MN DNR Reactors	MN DNR	2 tests of 75-gram samples	Dunka Mine Site (collected from blast holes)	809 weeks of data reported in 2014 with tests ongoing	Kellogg et al, 2014
Duluth Complex: Partridge River Intrusion	Field Test Pile	AMAX; MN DNR	Six test piles from 820-1300 metric tons	Mesaba Deposit	Leachate chemistry available from 1978-1993	Eger et al, 1979; Lapakko, 1994; Lapakko 1993; Lapakko et al. 2004;
	Small MN DNR reactors	MN DNR	14 75-gram samples	Mesaba Deposit (13 samples were from AMAX drill core, 1 sample was from a AMAX field leach pile)	Test duration from 30-49 weeks	Lapakko et al. 2013 (Attachment 15)
	Small MN DNR Reactors	MN DNR	4 75-gram samples	Mesaba Deposit (one sample from bulk sample collected in 1996; three samples from AMAX exploration drill cuttings that had been stored for 20 yrs)	Up to 809 weeks of data reported in 2014 with 3 tests ongoing	Lapakko et al. 2013
	ASTM D5744 HCT	US Bureau of mines	8 1-kg samples (run in duplicate)	Mesaba Deposit (collected from surface of AMAX field leach piles)	Data reported up to 125 weeks (experiments terminated after 168 weeks)	Lapakko and White 2013
	Small MN DNR Reactors	MN DNR	6 75-g samples	Mesaba Deposit (from the bulk sample collected in 2001)	400 weeks of data reported in 2013 with tests ongoing	Lapakko et al. 2013

Large Table 1. Summary of characterization programs potentially relevant to proposed mineral withdrawal area (cont.)

	Test type	Conducted By	Test Quantity	Sample Provenance	Test Duration	Select References
Duluth Complex: Partridge River Intrusion (cont.)	Small MN DNR Reactors	MN DNR	1 75-g sample	NorthMet Deposit (selected from drill core)	400 weeks of data reported in 2013 with tests ongoing	Lapakko et al. 2013
	ASTM D5744 HCT	MN DNR	7 1-kg sample	Mesaba Deposit (from the bulk sample collected in 2001)	400 weeks of data reported in 2013 with tests ongoing	Lapakko et al. 2013
	ASTM D5744 HCT	MN DNR	3 1-kg samples	NorthMet Deposit (selected from drill core)	400 weeks of data reported in 2013 with tests ongoing	Lapakko et al. 2013
	Size fractionated	MN DNR	6 size fractions of one sample; three largest size fragments also ran in a parallel covered cell set up	Mesaba Deposit (collected from AMAX field leach piles)	943 weeks data reported in 2013 with tests ongoing	Wenz et al. 2013 (Attachment 16)
	Field Test Pile	MN DNR	1 field plot of tailings approximately 20 ft x 30 ft x 2 ft deep	Mesaba Deposit (ore from Minnamax site)	approximately 3 years of leachate chemistry (1979-1981)	MN DNR 2004
	Small MN DNR Reactors	MN DNR	6 75-g split of tailings sample (uncovered and covered tests run in triplicate)	Mesaba Deposit (from bulk sample collected in 2001)	520 weeks of data reported in 2013 with tests ongoing	Lapakko et al. 2013b (Attachment 17)
	ASTM D5744 HCT	MN DNR	1 1-kg sample, run in duplicate	Mesaba Deposit (from bulk sample collected in 2001)	520 weeks of data reported in 2013 with tests ongoing	Lapakko et al. 2013b

Attachment 1

Lapakko K, Antonson D (2012) Duluth Complex Rock Dissolution and Mitigation Techniques : A summary of 35 years of DNR research. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN.

Duluth Complex Rock Dissolution and Mitigation Techniques: A summary of 35 years of DNR research

February 2012

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Duluth Complex Rock Dissolution and Mitigation Techniques: A summary of 35 years of DNR research

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1. INTRODUCTION

Minnesota has an extensive taconite mining industry. The mines, tailings, and waste rock produced during this mining cover more than 135 square miles. Despite the vast amount of these wastes, drainage quality problems have been largely limited to recent concerns regarding sulfate release. Studies have been conducted to quantify the sulfate released with drainage from a tailings basin (Lapakko and Jagunich 1991) and to evaluate ground water quality impacts of disposing taconite tailings in existing open pits (e.g. Berndt et al. 1999, Lapakko and Jakel 1999).

Minnesota has potential for mining of nonferrous minerals, particularly within the Duluth Complex, a layered mafic intrusion in the northeastern part of the state. It is a large gabbroic body that contains low-grade copper and nickel sulfides, titanium oxides, and platinum group elements, and it has also been developed as a source of dimension stone. The Complex contains one of the largest known copper and nickel resources in the U.S. (Minnesota Environmental Quality Board 1979; Kingston et al. 1970).

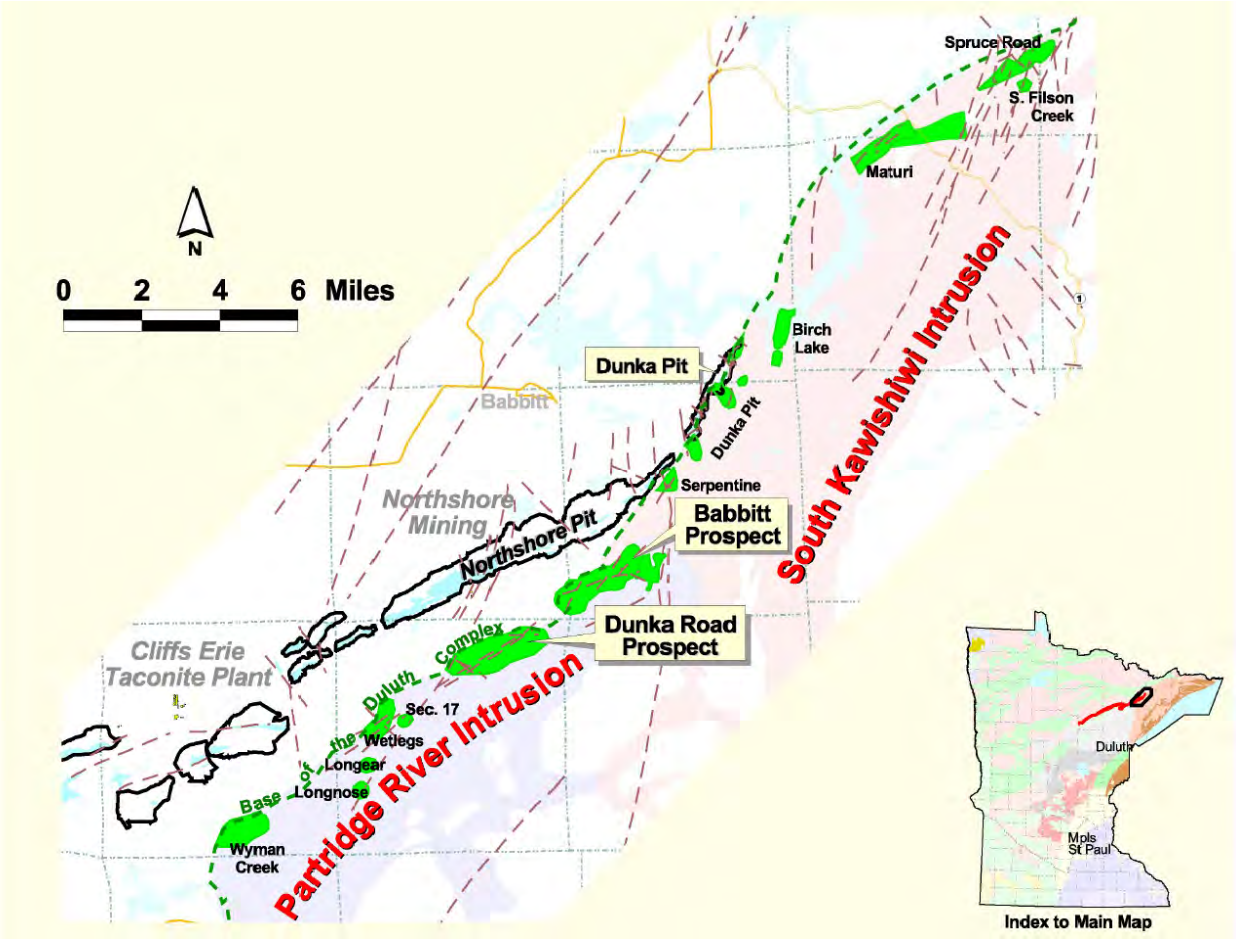
Miller et al. (2002) reported on the mineral occurrences and geology of the Duluth Complex, from which Ruhanen (2006) summarized the following information on the Partridge River and South Kawishiwi intrusions. Dissolution of rocks from these two intrusions is the focus of the studies discussed in the present report.

The Partridge River intrusion hosts at least four copper-nickel prospects (Babbitt, Dunka Road, Wetlegs, Wyman Creek) and at least seven potential iron-titanium prospects (Miller et. al., 2002) (Section 17, Longear, Longnose, Wyman Creek, Section 22, Skibo, Water Hen) (Figure 1). The Section 22, Skibo, and Water Hen prospects all lie south, and within 12 miles, of the Wyman Creek prospect and are not depicted in Figure 1.

The Partridge River intrusion consists mainly of troctolitic rocks that form an arc-shaped exposure (Figure 1). It is estimated to be 2.5 km thick, and is underlain by the Virginia Formation slates and greywackes and the Biwabik Iron Formation. The basal 900 meters of the intrusion has been subdivided into seven stratigraphic units (Severson and Hauck 1990, 1997; Geerts 1991; Severson 1991, 1994) that can be correlated over a strike length of 24 km. These units dip 10 to 20 degrees to the southeast and have been described in detail due to the presence of copper-nickel (Cu-Ni) mineralization. Rocks overlying the Partridge River intrusion include other Duluth Complex rocks and metamorphosed basalts (probably North Shore Volcanic Group). These rocks are also present in the interior of the intrusion as large inclusions.

The contact zone between the Partridge River and South Kawishiwi intrusions is poorly exposed and sparsely drilled. The seven basal units become unrecognizable in the contact zone. This contact is complicated by the presence of a fault (Grano fault of Severson 1994) that was repeatedly activated during and after the emplacement of the two intrusions.

Figure 1. Map of Duluth Complex mineral prospects.



The South Kawishiwi intrusion hosts at least five copper-nickel prospects (Spruce Road, South Filson Creek, Maturi, Dunka Pit, Serpentine) and a potential platinum group element copper-nickel prospect (Birch Lake) (Figure 1). The intrusion is composed of troctolitic rocks exposed in an 8-by-32 kilometer arc-shaped band (Figure 1). The Giant's Range batholith is the dominant footwall rock, although Virginia Formation and Biwabik Iron Formation underlie the intrusion in some areas. Five major map units in the South Kawishiwi intrusion have been identified, although some geologists have further subdivided these units (e.g. Severson 1994). The units are not continuous throughout the intrusion. Major sulfide mineralization occurs at the base of the intrusion. Virginia Formation and Biwabik Iron Formation inclusions occur throughout the intrusion. Inclusions of basaltic hornfels (probably from the North Shore Volcanic Group) and quartz sandstone hornfels (probably from the Puckwunge and Nopeming sandstones) occur near some of the copper-nickel prospects. The lower ultramafic unit contains the majority of the high platinum group element values.

The presence of sulfide minerals in Duluth Complex rock also represents an environmental concern. When exposed to oxygen and water, iron sulfides such as pyrrhotite will oxidize and produce acid. At relatively low iron sulfide concentrations, the rate of acid generation will be low and dissolution of other minerals present in the rock can neutralize the acid produced. At higher iron sulfide concentrations, the rate of acid production will exceed the rate of neutralization, resulting in acidic drainage. Sulfide minerals containing copper, nickel, cobalt, or zinc will also oxidize to release the associated trace metal. Drainage that is acidic and/or containing elevated concentrations of trace metals is an environmental concern.

Mine wastes that produce problematic drainage must be managed such that the quality of waters of the state is not adversely impacted. Mine waste management strategies directed at this objective include preoperational mine waste characterization and drainage quality prediction and prevention, control, and treatment of problematic drainage. The Reclamation Section within the Division of Lands and Minerals of the Minnesota Department of Natural Resources (MN DNR) has conducted numerous studies on mine waste characterization, drainage quality prediction, and mitigation of mine waste drainage, as well as field studies on the release and aqueous transport of mining related contaminants.

Mine waste drainage quality and the mitigation of problematic drainage have been studied in Minnesota since the mid-1970s. Studies were initiated in 1976 under the auspices of the Regional Copper-Nickel Study and, in 1978, were continued by the MN DNR. Many of the studies were conducted in cooperation with industry or other governmental agencies (Lapakko and Brice 1998).

A process for characterizing potential mine wastes prior to nonferrous mine development was developed. The characterization process includes determining the mass and composition of wastes to be generated and conducting dissolution tests on representative samples of mine waste to predict the drainage quality from mine wastes. Studies have been conducted to examine the design of dissolution tests and interpretation of their results. Drainage from operational scale stockpiles of Duluth Complex has been monitored, and field studies on Duluth Complex test

piles were conducted. Laboratory studies have been conducted to describe the quality of drainage from Duluth Complex waste rock. Laboratory and field studies have been conducted on tailings generated from copper-nickel deposits in the Duluth Complex.

The strategies using alkaline solids and subaqueous disposal have been examined for their potential in the environmental management of mine wastes. Laboratory and field experiments are in progress to examine the mitigative potential of mixing alkaline solids with acid-producing mine waste. Laboratory and field tests were conducted to determine the effectiveness of limestone beds for treating mine waste drainage. Operational scale mitigation measures implemented at the Dunka mine have been summarized (Eger et al. 1999). Finally, disposal of sulfidic rock in a subaqueous setting, with and without various amendments, is under investigation in both the laboratory and field.

The following provides a brief overview of monitoring and research activities regarding the environmentally sound management of Duluth Complex mine wastes. In particular, this report summarizes research on prediction of drainage quality from Duluth Complex mining wastes and mitigation of Duluth Complex mine waste drainage using alkaline solids and subaqueous disposal. Many of these projects remain in progress. More detailed information is available in the publications cited. A brief discussion of mine waste dissolution chemistry is presented in Appendix 1. Information on other MN DNR research on environmentally sound management of mine wastes from other lithologies is presented in previous research summaries (Lapakko et al. 1998a, 2001).

The MN DNR has also conducted research on capping mine wastes to reduce precipitation input, joint disposal of waste rock and tailings, and treatment of mine waste using wetlands and sulfate reduction. These studies are not described in the present report. The Mineral Resources Research Center at the University of Minnesota, Minneapolis conducted mineral processing studies and, to a far lesser degree, environmental research on Duluth Complex rock. Extensive research on the geology of the Duluth Complex has also been conducted by the Natural Resources Research Institute, University of Minnesota, Duluth, and the Minnesota Geological Survey. This research is also excluded from the present report.

2. MINE WASTE CHARACTERIZATION

2.1. Conceptual Approach

The quality of mine waste drainage is variable, largely due to the variability in mine waste composition. By characterizing mine wastes and predicting mine waste drainage quality prior to mineral resource development, waste disposal facilities can be designed, sited, constructed, and reclaimed in a manner that will most effectively reduce contaminant release and the associated drainage treatment cost. Mitigation can be tailored to the predicted drainage quality and directed toward meeting water quality objectives. Wastes with mitigative capacity (e.g. limestone) can be identified during the characterization process. Finally, the cost of reclamation can be considered along with other mining costs in the assessment of resource development economics. Mine development can then progress with consideration of both metal recovery and the environmentally sound management of mine wastes.

A mine waste characterization process was designed for application on a site-specific basis (Lapakko 1990b, 1988b). The approach and many of the analytical techniques used are similar to those employed for evaluation of potential for resource recovery. The extent of rock units potentially contributing to mine waste is delineated and pertinent compositional aspects of these units are quantified. The quality of drainage from the potential wastes is then projected. Existing data on a mine waste of similar composition, generated by similar mining methods, and exposed to similar environmental conditions for an extended time provide the best indicator of drainage quality. A directory of contacts was compiled to facilitate collection of information on mine waste composition and drainage quality across the United States (Wessels et al. 1991).

Since data on the drainage quality from a mine waste of specific composition are often not available, it is commonly necessary to conduct laboratory dissolution tests to predict drainage quality. These tests could be conducted on a variety of representative samples from individual rock units to determine the relationship between mine waste composition and drainage quality. Additional sampling, analysis, and dissolution testing may be required based on the information compiled. The relationship between solid composition and chemical behavior is then extrapolated to the waste as a whole. A mine waste management program would be designed based on the projected drainage quality and quantity.

The mine waste composition and predicted drainage quality would be subject to verification during operation if mine development proceeded. Small-scale field tests on well-characterized waste allow more accurate extrapolation to operational scale drainage quality and drainage volume, and also allow assessment of mitigation design. Initiating such verification tests as soon as resource development begins maximizes the benefit of these tests for drainage quality prediction and mine waste management. These concepts were integrated into Minnesota's Nonferrous Mining Rules (Chapter 6132).

2.2. Assessment of Tests for Predicting Mine Waste Drainage Quality

Several static and kinetic tests have been developed for predicting mine waste drainage quality. Static tests base predictions on solid phase analyses, while kinetic tests use data from dissolution tests on mine waste samples. Static tests attempt to quantify the capacity of a mine waste to produce acid (Acid Production Potential or AP) and to neutralize acid (Neutralization Potential or NP). The difference between these values or their ratio is used to estimate the potential of the mine waste to produce acidic drainage. They do not account for the availability of the acid-producing or acid-neutralizing minerals or the rate at which they dissolve. They are typically used as a tool for initial screening of mine wastes for their potential to produce acid.

AP is based on the quantities of sulfur present as acid-producing sulfide and sulfate minerals. Since techniques for determining these quantities have been developed and applied for geological assessment over many years, they are fairly accurate as long as the acid-producing phases present are known. NP is typically determined by subjecting the sample to partial digestion in an acidic solution, and then titrating the solution to determine the amount of acid neutralized during digestion.

Kinetic tests involve dissolution of mine waste samples and analysis of the resultant water quality. Tests such as these permit identification of some acid-producing mine wastes based on the laboratory drainage pH. However, if the mine waste contains a moderate amount of neutralization potential, it may not be depleted during the course of the test and drainage will remain neutral. Based on such data it might be concluded that the solids were not acid producing. However, over a longer duration the NP may be depleted while iron sulfides remain and continue to oxidize. In such a case the drainage would acidify upon depletion of the neutralizing minerals. Thus, to determine the degree to which laboratory tests model long-term field behavior, it is necessary to determine the extent to which acid producing and acid neutralizing minerals have been depleted in kinetic tests.

To evaluate several static and kinetic tests on a compositional range of mine wastes, the U.S. Environmental Protection Agency provided funding to the MN DNR for a project that was administered by the Western Governors' Association (WGA). A literature review was produced and followed by empirical evaluation of the ability of tests to predict mine waste drainage pH (Lapakko 1991b, 1992a). Insight into the accuracy of NP determinations has been gained as part of MN DNR testing of Duluth Complex rock and hydrothermal quartz-carbonate gold tailings. The influence of static test protocol variables on NP was also examined as part of the latter research, which was funded by the Minnesota Minerals Coordinating Committee and MN DNR Division of Minerals (Lapakko 1991a, 1992b). Much of the information from these and other studies is presented in White et al. (1997, 1999).

The EPA funded a continuation of the kinetic testing phase of the WGA project. In this continuation two dissolution tests were continued for a total duration of 132 weeks and the effect of particle size on mine waste drainage quality was examined (Lapakko et al. 1995). The U.S. Army Corps of Engineers provided funding for the final stage of this project, in which as much as 618 weeks of dissolution testing data were reported (Lapakko et al. 2004a).

2.3. Static Testing of Duluth Complex Rock

As mentioned above, the objective of static tests is to determine the potentials of a mine waste to produce and neutralize acid. To accurately quantify the potential for acid production, the content of acid-producing minerals must be determined. In Duluth Complex mine wastes, iron sulfide phases are the dominant acid producers, and the predominant mineral phases are pyrrhotite and the iron sulfide fractions of chalcopyrite, cubanite, and pentlandite (respectively $Fe_{1-x}S$, $CuFeS_2$, $CuFe_2S_3$, $(Fe, Ni)_9S_8$). Because sulfide is commonly associated with copper and nickel (and to a lesser degree with cobalt and zinc), quantification of acid production by determination of total sulfur or sulfide-sulfur tend to overestimate the potential for acid production. That is, oxidation of the trace metal sulfide fraction of these minerals will not lead to substantial acid production unless the trace metals released precipitate as hydroxides or oxides.

Standard ABA NP values overestimate the capacity for Duluth Complex rock to neutralize acid while maintaining an environmentally acceptable drainage pH. Although carbonate minerals are generally present at very low concentrations in Duluth Complex rock (Stevenson et al. 1979, Lapakko 1988a), Standard Acid Base Accounting (Standard ABA) NP values in the typical range of 10 to 20 kg $CaCO_3$ eq t^{-1} (inferring 1-2% $CaCO_3$; Lapakko 1993a, White et al. 1999) and as high as 33 kg $CaCO_3$ eq t^{-1} (Lapakko 1993d, 1994b) have been reported. Laboratory empirical NP values ranged from 0 to 2.0 kg $CaCO_3$ eq t^{-1} (Lapakko 1990a, 1993a, Lapakko and Antonson 1991, White et al. 1997), confirming that the Standard ABA NP values overestimated by (5 to 21 kg $CaCO_3$ eq t^{-1}) the capacity of this rock to neutralize acid while maintaining drainage pH $\geq$ 6.0.

Furthermore, even after the pH of effluent from the rock samples had decreased below pH 6, Standard ABA tests conducted on the leached solids indicated a residual NP, typically in the range of 8 to 14 kg t^{-1} $CaCO_3$ (White et al. 1999). Based on data from field test piles of Duluth Complex rock, the Empirical Neutralization Potential (ENP_6, the neutralization occurring prior to drainage pH decreasing below 6) observed in the field was determined to range from 0.04 to 1.3 kg $CaCO_3$ eq t^{-1} and, as was the case for laboratory values, tended to decrease as sulfur content increased (Lapakko 1994b).

Because there are limitations in the accuracy of ABA analyses for Duluth Complex waste, other analytical tools might be more appropriate for determination of the potentials for acid generation and acid neutralization. Some of these tools are identified and their applications are described in two publications (Lapakko 2001b, Lapakko 2003a).

2.4. Kinetic Testing of Duluth Complex Rock

To assess the accuracy of kinetic test results they must be compared, perhaps after modeling, to long-term field results for the same mine waste. While laboratory drainage pH from Duluth Complex rock agreed fairly well with field data (Lapakko 1994a, see also section 3.3.2.4), data for such comparisons are rarely available. Consequently, it is rarely possible to assess kinetic test accuracy based on field verification, and the extent to which the laboratory tests model field behavior must be addressed conceptually. Results of Duluth Complex dissolution in laboratory kinetic tests and in the field are presented in the following section.

3. DRAINAGE QUALITY FROM DULUTH COMPLEX MINE WASTES

3.1. Introduction

In 1976, The Regional Copper-Nickel Study was formed in response to public concerns regarding the environmental impact of copper-nickel mining. The final summary report was completed in 1979 (Minnesota Environmental Quality Board 1979). The investigations conducted included operational scale field studies at the Dunka Mine (an active iron ore operation), pilot scale field studies at the AMAX exploration site, monitoring of a bulk sample site, as well as laboratory experiments.

The earliest reports on the Duluth Complex focused on the impact of potential copper-nickel mining (Thingvold et al. 1979), and addressed the hydrology (Hewett 1980) and chemistry of stockpiles. The early laboratory studies (Lapakko 1980; Eisenreich et al. 1977a, 1977b) investigated the kinetics and mechanisms of gabbro dissolution as well as trace metal equilibria. The findings of the early field monitoring, the laboratory experiments, and the chemical equilibrium computer models were compiled in a comprehensive report (Eger and Lapakko 1980c), and summarized in shorter papers presenting the chemical mechanisms of leaching (Lapakko and Eger 1980a, 1980b).

3.2. Field Data

3.2.1. Operational Scale South Kawishiwi Intrusion Waste Rock

Field water quality data were collected at an operational taconite mine where Duluth Complex rock had been stripped and stockpiled, at six test piles, and at dimension stone quarries. Operational scale Duluth Complex rock stockpiles are present at the Dunka Mine near Babbitt, a full-scale open pit taconite operation now closed. The Dunka open pit intersects the geological contact between the South Kawishiwi intrusion of Duluth Complex and the Biwabik Iron Formation, and removal of Duluth Complex rock was required to mine the underlying taconite. More than 50 million tons of the sulfide-mineral-bearing rock is stored at the site, in stockpiles that cover 320 acres.

Since 1976, data have been collected on the drainage quality, quantity, and chemical mass release associated with stockpile drainages at the site. The Regional Copper-Nickel Study began data collection (Thingvold et al. 1979), and after 1980 monitoring was conducted by Erie Mining Company and LTV Steel Mining Company. Cleveland Cliffs has conducted monitoring since 2001. The most recent report (MN DNR 1996, unpublished) presented the results of monitoring from 1976 to 1993, and earlier reports are available (MN DNR 1987, 1994; Eger et al. 1981a; Eger and Lapakko 1980a; Eger et al. 1977). Other studies at the Dunka site have focused on the transport of trace metals leached from the stockpiles through a white cedar swamp (Eger et al. 1980a, 1981a; Eger and Lapakko 1988, 1989), Unnamed Creek (Eger and Lapakko 1980b), a second small drainage at the site (Strudell 1986), and Bob Bay (Lapakko and Eger 1981b; Strudell et al. 1984; MN DNR unpublished data 1990). Information on the kinetics, mechanisms, and capacities of metal sequestration by peat has been compiled in the form of

literature reviews (Lapakko et al. 1986b; Otterson 1978) and laboratory experiments on metal removal by peat have been conducted (Lapakko et al. 1986a).

The most recent report indicates total flow from the watershed is about one billion gallons per year, of which drainage associated with stockpiles has contributed roughly 5 to 20 percent (MN DNR 1996, table 5). During the period from 1976 through 1993, the majority of leachate samples had pH values between 6.0 and 8.5, but values as low as 4.2 were reported (MN DNR 1996, Figure 4). Particularly notable was the Seep 1 pH decrease from roughly 7.3 to near 5 during the period 1976-1991. There also were a number of pH values below 6 at Seep 3 in 1989. Sulfate concentrations from these two seeps were also elevated above the typical range of 1000 to 3000 mg L^{-1} for the seeps.

Concentrations of trace metals (Cu, Ni, Co, Zn) exceeded ambient levels by 10 to 10,000 times. Nickel contributed more than 90 percent of the trace metal load from the stockpiles, and its subsequent removal by natural processes in the Unnamed Creek system was less than 40 percent. Chemical mass release of trace metals from the stockpiles reached a maximum in 1988, when nickel concentrations were as high as 80 mg L^{-1} at Seep 3. Nickel concentrations at Seep 3 were notably elevated in the periods 1976-78 and 1988-90, with medians typically in the range of 20 to 50 mg L^{-1}, and decreased to roughly 10 to 20 mg L^{-1} by 1993. Nickel concentrations at Seep 1 were also elevated relative to other seeps, with median values often in the range of 15 to 20 mg L^{-1}. When nickel concentrations were elevated, concentrations of copper, cobalt, and zinc were often elevated as well.

A summary of the monitoring requirements for the current Dunka NPDES permit are presented in Attachment 1. Monitoring requirements are likely to change upon reissuance of the permit in 2012.

3.2.2. Duluth Complex Quarry Sites in the South Kawishiwi Intrusion

In late 1991, rock and water quality samples were taken from two abandoned dimension stone quarry sites in the Ely-Isabella area of northeastern Minnesota. Rock samples were also collected from three potential quarry sites. The abandoned sites and two of the potential quarry sites are located in Duluth Complex rock. The third potential site is in the Vermilion Granitic Complex.

Total sulfur content of the rocks was less than 0.05 percent and trace metal levels were low. Consequently, the pH of water in the abandoned pits ranged from 6.5 to 6.8, and sulfate concentrations were low. Sulfate concentrations in the three of the four water samples collected were below detection (2 mg L^{-1}) and the fourth sample had a sulfate concentration of 3.2 mg L^{-1}. Copper (0.002-0.004 mg L^{-1}) and zinc (0.008-0.017 mg L^{-1}) concentrations were higher than background levels in the area, but were well within water quality standards (Lapakko and Oberhelman 1993). It should be noted that the fraction of fine particles associated with rock at these sites may have been considerably less than that at a metal mine. That is, the surface area per unit mass at the quarry sites was probably less than that for metal mine waste rock. Consequently, the quarry rock would tend to react less rapidly than a similar mass of waste rock from a metal mine.

3.2.3. Test Piles of Rock from the Babbitt Deposit, Partridge River Intrusion

3.2.3.1. Introduction

Six small test piles containing 820 to 1300 metric tons of low-grade copper-nickel Duluth Complex rock, taken from an exploration shaft, were constructed next to the MINNAMAX test shaft near Babbitt, Minnesota. For runoff collection, a Hypalon liner was placed at the base of each pile. Both drainage flow and quality were determined. In 1982, approximately 40 percent of test pile FL4 was removed for use in another study of stockpile reclamation techniques. In 1994, the piles were dismantled and the rock was incorporated into one concrete/rock monolith as a final reclamation measure for site closure (Barr Engineering 1995).

Drainage quality, quantity, and effects of soil covers are addressed in a series of reports (Eger and Lapakko 1985; Eger et al. 1981b, 1980b, 1979; Lapakko 1993b) and symposium publications (Eger and Lapakko 1981; Lapakko 1994a). During dismantling of three of the piles (FL1, FL5, FL6), rock samples were collected for physical, chemical, and mineralogical characterization. The most recent report on this project focused on the characterization of leached rock samples (Lapakko et al. 2004b).

3.2.3.2. Drainage Quality

Sulfur contents of the test piles were 0.6 (four piles), 0.8 and 1.4 percent, copper contents were 0.35, 0.34, and 0.31 percent, and nickel contents were 0.08, 0.09, and 0.09 percent. The copper contents of the rock were relatively high, perhaps more representative of ore than waste rock. Consequently, a fairly large fraction of the sulfur was associated with copper rather than iron, and this would tend to reduce the potential for acid production at a specific sulfur content. The sulfur contents associated with iron (including that in chalcopyrite, cubanite, pentlandite, etc.) in the 0.6, 0.8, and 1.4 percent sulfur piles were 0.35, 0.51, and 1.19 percent (corresponding to FeS contents of roughly 0.6, 0.9, and 2.1 percent), and the corresponding sulfur contents associated with pyrrhotite were 0.13, 0.28, and 0.98 percent.

The minimum annual median drainage pH values produced by rock 0.6, 0.8, and 1.4 percent sulfur piles were approximately 4.8, 4.0 and 3.4, respectively. Drainage pH from the 0.6% S piles was still decreasing at the end of the tests. During the last year of operation, typical concentration ranges for copper, nickel, cobalt and zinc in drainage from pile FL1 (0.6% S) were 20-30, 50-70, 2-3, and 1-2 mg L^{-1}. Drainage pH values during this period typically ranged from 5.0-5.2. Maximum nickel concentrations for the 0.6% S piles ranged from roughly 70 to 200 mg L^{-1}, and those for the 0.8 and 1.4% S piles were roughly 220 and 700 mg L^{-1}, respectively.

The following general conclusions on Duluth Complex rock drainage quality from 1978 through 1993 were drawn based on this small-scale field program.

1. Trace metal concentrations in drainage from test piles containing low-grade sulfide mineralization exceeded concentrations in undisturbed streams of the area by as much as five orders of magnitude.

2. Drainage pH decreased while trace metal concentrations and mass release rates increased with the iron sulfide content of the stockpile.

3. For the 16-year period of record, a critical iron sulfide content appears to occur at roughly 0.6 weight percent FeS (0.6 % total S) in the bulk rock. Rock of similar mineralogy and particle size distribution containing more than this critical amount would generate acidic leachate.

4. The early drainage quality was not an accurate indicator of longer-term drainage quality. For the test piles that contained 0.6% S (FL1, FL2, and FL3), acid producing conditions did not occur for 5 to 6 years. Even after the 16 years of this study, less than 12% of the total sulfur was released from the piles as sulfate. Predictions based on data from the first years of this study would have underestimated trace metal release rates and acid production.

3.2.3.3. Geochemical Modeling

Lapakko et al. (2004b) provided a brief description of the mineral dissolution reactions influencing drainage quality. Acid release was the balance of acid producing and acid neutralizing reactions. Oxidation of pyrrhotite was the dominant source of acid production. Acid neutralization was dominated by dissolution of calcium-bearing minerals above roughly pH 5.0 and by dissolution of magnesium-bearing minerals below this value. The dominant calcium minerals were plagioclase and augite, and the dominant magnesium minerals were augite and olivine.

Copper release resulted from oxidation of chalcopyrite and cubanite, and pentlandite oxidation provided most of the nickel release. These metals were mobile in solution under most conditions observed in the drainages. The rates of trace metal sulfide oxidation, therefore, controlled trace metal release and were apparently accelerated under acidic conditions.

Geochemical modeling was used to predict the mineralogic and hydrologic controls on test pile FL1. Nine samples from test pile FL1 were subjected to a sequential extraction procedure that indicated that trace metals released during the oxidation of sulfides were adsorbed to amorphous ferric oxyhydroxides (Kelsey et al. 1996). Predictions, using the geochemical modeling programs NETPATH and MINTEQAK, generally agreed with the sequential extraction results, but predicted a lesser degree of nickel adsorption on amorphous ferric oxyhydroxides (Kelsey et al. 1996).

3.2.3.4. Post-Mortem Solid-Phase Analyses

About 40 percent of one of the 0.6% S piles (FL4) was partially removed in 1982 and sample were collected (Eger and Lapakko 1985, Lapakko et al. 1986a). Particle size distribution analysis indicated that 19 percent of the mass was finer than 6.4 mm (0.25 inches). Seven fractions finer than 2 mm were analyzed. Paste pH and sulfur, sulfide, copper, nickel, cobalt, and

zinc concentrations were determined. As particle size decreased, paste pH decreased and concentrations of the aforementioned components typically increased by factors of two to five.
In 1994, the piles were dismantled, and bulk and spatially distributed samples were collected from FL1 (0.63% S), FL6 (0.8% S), and FL5 (1.4% S) (Lapakko et al. 2004a). Samples were analyzed for particle size distribution, chemistry (including acid content), and semi-quantitative mineralogy. The major objective of this characterization was to identify physical, chemical, mineralogical, and hydrologic controls on the transport of acid and trace metals from the piles. Visual inspection of the piles found elemental sulfur, elemental copper, and a blue green precipitate (possibly a trace metal solid phase), in addition to the expected iron oxyhydroxides and gypsum. Substantial cementation was also observed in the higher sulfur piles, with agglomerates varying in size (1cm to 1m) and cohesiveness.

Particles finer than 0.5 mm constituted roughly five to six of the rock pile mass but were responsible for an estimated 25 to more than 50 percent of the pyrrhotite oxidation. An estimated 70 percent of the rock finer than 0.5 mm contributed to pyrrhotite oxidation. Rock at the surface of the piles oxidized to a greater degree than that at depth. Oxidation at depth may have been limited by different conditions that may have been present in different areas. For example, sulfide mineral oxidation may have been limited due to water-saturated conditions or cementation that limited oxygen transport, lack of contact with water, or limited oxygen transport in specific areas. Cementation was observed during the dismantling of the rock piles.

3.3. Laboratory Dissolution Studies

3.3.1. Test Methods

3.3.1.1. MN DNR Reactors

Laboratory studies on the dissolution of Duluth Complex rock were initially conducted at the University of Minnesota in batch reactor and column tests (Lapakko 1980). Subsequently, the MN DNR developed an experimental method based on the principle that sulfide minerals oxidize in the presence of atmospheric oxygen and water (Gottschalk and Buehler 1912; Caruccio et al. 1980). Samples were quite fine, ranging from 0.053 to 0.149 mm in diameter (-100/+270 mesh). Experiments were conducted on 75-g samples in units designed to permit the rinsing of solids and filtration of the rinse solution in one step (U.S. Patent No. 4,783,318). The solids (and reactors) were stored in a controlled temperature and humidity environment, and rinsed weekly with 200-mL volumes of distilled deionized water. This method was used for initial testing at the MN DNR and several subsequent experiments.

3.3.1.2. ASTM 5744

In 1992 US Bureau of Mines (USBM) initiated assessment of a laboratory test for weathering mine wastes, and in 1996 the American Society of Testing and Materials (ASTM) designated this test as ASTM D 5744-96 (ASTM 1996). This method uses a 1000 g sample of waste rock finer than 0.25 inches and recommends a cell with an internal diameter of 10.2 cm and height of 20.3 cm. The weekly rinse volume is either 500 or 1000 mL and can be added by either of two

methods ("drip" or "flood"). The day after the sample is rinsed a three-day cycle begins during which dry air is passed through the cell. This cycle is followed by a three-day cycle during which humidified air flows through the cell. On the seventh day the cell is rinsed, and the drainage is collected for analysis. The cells are weighed to determine the water retained after the rinse, on the fourth day of the cycle (concurrent with the switch of ASTM cells from wet to dry air cycle), and before the rinse. The method is described in detail in ASTM (2000), and assessments of the method are provided by White and Lapakko (2000), Lapakko et al. 2002a, and Lapakko (2003b). In general, the intralaboratory and interlaboratory replication were good. The major variable affecting results appeared to be temperature, with elevated temperatures yielding lower pH values and higher rates of sulfate release. (Variations in temperature were reported for the commercial laboratory.)

A kinetic test method similar to ASTM Method D 5744-96, described by Lapakko and White (2000) was designated as the MN DNR method. The MN DNR protocol uses the same humidity-cell diameter, waste-rock charge, and rinse volume and application method as described in ASTM D 5744-96. The MN DNR cell was about three centimeters shorter than the ASTM cell. The latter had a thicker base plate; a larger space, to accommodate a fitting for introducing air flow, between the base plate and perforated plastic support; and a thicker cover. Three 0.5-L rinses were used at week-0 rinse rather than the single 0.5-L volume described in the standard method. Furthermore, instead of subjecting the humidity cell apparatus to the humid or dry air flow into the cell, the cells were simply stored in a controlled temperature and humidity room between weekly rinses. Experiments were conducted to assess replication of test options and compare their results (Lapakko 1998b, 1999a; Lapakko and White 2000; White and Lapakko 2000; Lapakko and Antonson 2000a; Lapakko et al. 2002a). Because there was little difference between the two methods, the MN DNR approach was incorporated into the ASTM method as Option B in 2007. The most recent developments in the method are presented by Bucknam et al. (2009). Additional modifications are in progress as of December 2011.

3.3.2. Waste Rock

3.3.2.1. South Kawishiwi Intrusion (SKI)

In February 1989 an experiment using the small MN DNR reactors was initiated that ultimately examined the longer-term drainage quality from 17 South Kawishiwi Intrusion (SKI) Duluth Complex samples of variable sulfur content. The samples were collected from blast holes at the Dunka site in the SKI of the Duluth Complex. Dissolution testing was initiated on 15 of the samples in 1989 and 1990, and on two additional samples in August 1997. (Four Virginia formation hornfels samples also were tested from February 1989 to August 1990.) The following discussion pertains to only the initial 15 Duluth Complex samples. Data from the early stage of the experiment were presented by Lapakko and Antonson (1993) and Lapakko (1993a).

Presently in progress is a report that analyzes as much as 866 weeks of drainage quality data. These data are used to update classifications of rock reactivity based on sulfur content. The drainage quality data are also used, in conjunction with solid-phase characterization (particle

size, bulk chemistry, mineral content, mineral chemistry, surficial features of leached solids) to estimate mineral dissolution rates.

The particle size of the rock samples tested in the small MN DNR reactors was in the range of 0.053 to 0.149 mm and the sulfur contents ranged from 0.18 to 3.12 percent. Detailed analyses were conducted on these samples to determine chemistry, mineral content, and mineral chemistry. 75-g samples were rinsed weekly with 200 mL of deionized water, and the drainage was regularly analyzed for pH, specific conductance, sulfate, calcium and magnesium. Concentrations of other solutes were determined on a less frequent basis. Between rinses the samples were stored in a room in which temperature and humidity were controlled. Analyses were also conducted on rock samples for which dissolution testing was terminated.

Rates of sulfate release tended to increase and drainage pH tended to decrease as solid-phase sulfur content increased. Two drainage pH values are of particular interest with regard to managing the associated mine wastes in an environmentally sound manner. The first is pH 6.0, a common regulatory requirement for drainages. The second critical pH value is that below which microbial catalysis of sulfide mineral oxidation accelerates the rate of sulfide mineral oxidation and, consequently, acid production and trace metal release. Provisionally a pH value of 4.0 will be used for this threshold. Drainage with pH values in the range of 4.0 to 6.0 will have trace metal concentrations exceeding regulatory standards. Since this type of drainage is not highly acidic, it may be possible to mitigate the acid release by passive treatment (e.g. limestone beds, sulfate reduction). Rock that generates, drainage pH values below 4.0 will also require mitigation of both acid and trace metal release. In contrast to the previous case, the levels of acidity and metals in this low-pH drainage may be excessively high for practical application of passive treatment. Thus, rock that generates drainage of this nature would require the most rigorous management.

Samples with sulfur contents of 0.18 and 0.22 percent consistently generated drainage pH values above 6 and, therefore met standards for pH. Trace metal concentrations in drainage from this rock might still exceed regulatory standards. Thus, mine waste management plans for this rock might require mitigation of trace metal release but not acid neutralization.

The remaining samples produced drainage pH values below 6.0. Drainage pH from samples of higher sulfur ($S \geq 0.40\%$) contents generally decreased for five to seven years, plateaued for one to two years, and then increased. Samples with sulfur contents of 0.40 to 0.58 percent produced minimum drainage pH values near 4.0. Samples with sulfur contents of 0.71 to 1.64 percent generated minimum pH values in the approximate range of 3.0 to 3.5. Dissolution testing of samples with sulfur contents of 2.06 and 3.12 percent was terminated after 78 weeks, and it is likely that these samples would have generated minimum drainage pH values of 3.0 or lower had testing continued. Mine waste management plans for waste rock generating drainage pH values below 6.0 must mitigate release of both acid and trace metals. The mitigation of the highest sulfur group would be most rigorous.

3.3.2.2. Partridge River Intrusion (PRI)

3.3.2.2.1. Babbitt Prospect

Several laboratory testing studies were conducted on Babbitt prospect rock from AMAX drill core, AMAX test piles, and a 1996 bulk sample. **AMAX drill core and a test pile** sample were subjected to dissolution experiments using the small MN DNR reactors described above. The experiment was the first laboratory dissolution testing of Duluth Complex rock at the MN DNR and was initiated in March 1985. The intent was to investigate the effect of solid phase sulfur content on the quality of drainage from 14 rock samples ($0.47 \leq \%S \leq 2.57$) from the Babbitt prospect of the PRI. Thirteen of the samples were from AMAX drill core and one sample was collected from AMAX test pile FL4. The samples were subjected to dissolution testing for periods of 30 to 49 weeks, and rinse intervals varied from one to five weeks (Lapakko 1993e, 1988a). It should be noted that the experiment was somewhat exploratory in nature, given the short duration and changing rinse intervals.

The use of sulfur content alone as the controlling variable for drainage pH assumes that acid neutralization by host rock mineral dissolution is relatively constant in Duluth Complex rocks of similar particle size. The solids examined were divided into three groups based on sulfur content, maximum rate of sulfate release, and minimum pH of the drainages (Lapakko 1993e). Group 1 samples contained 0.47 to 0.80 percent sulfur and produced maximum sulfate release rates of 5.8×10^{-6} to 16×10^{-6} mol week^{-1}. The sum of calcium and magnesium release rates exceeded the sulfate release rates (that is, the rates of acid neutralization exceeded the rates of acid production) and, consequently, the drainage pH of Group 1 solids remained higher than 6.0.

Group 2 (0.92 - 1.26% S) and Group 3 (1.35 - 2.57% S) solids had higher sulfur contents, exhibited higher sulfate release rates (20×10^{-6} to 190×10^{-6} mol week^{-1}), and produced minimum drainage pH values in the range of 3.77 to 4.50. For all solids examined, less than nine percent of the sulfur in the samples was oxidized (Lapakko 1993e), suggesting that extrapolating the data for long-term predictions was tenuous. The drainage pH values were higher than those generated by the SKI samples from the Dunka Mine (section 3.3.2.1). The higher pH values were partly the result of the short duration of these tests.

The composition of the Duluth Complex rock examined in the aforementioned experiment was subsequently characterized using more sophisticated methods (Pignolet-Brandom and Lapakko 1990). The QEM*SEM automated scanning electron microscope image analysis system was used to quantify the volumetric abundances of iron sulfides, copper-iron sulfides, nickel-iron sulfides, and calcite, as well as the general categories of silicate, oxide and phosphate minerals. The sulfur, nickel, and copper contents based on volumetric abundances were typically within 20 percent of the values determined by standard chemical analyses. QEM*SEM can quantify other parameters affecting mine waste dissolution, including mineral grain and rock particle size, particle surface mineralogy, extent of sulfide liberation, as well as total and liberated mineral surface area per unit volume.

Four PRI Babbitt prospect samples were subjected to dissolution testing along with SKI samples in the small MN DNR reactors beginning in August 1997 (see also section 3.3.2.1). One sample

was from a **bulk sample** collected in 1996 and three samples were from **AMAX exploration drill cuttings** generated in the mid-1970s. Respective sulfur contents of these samples were 0.67, 0.72, 0.92, and 1.71 percent. The 1.71% S drill core sample was terminated after 143 weeks, and testing of the remaining samples continues. The drill cutting tests resulted in markedly lower sulfate release and higher pH than samples of similar sulfur content from the SKI samples with which they were tested (see section 3.3.2.1) or subsequently tested PRI samples (see section 3.3.2.2.1.).

The **United States Bureau of Mines** Salt Lake City Research Center (USBM) initiated dissolution tests on larger Duluth Complex particles (d < 6.25 cm) using a different method (ASTM 5744-96 method) in July 1992. Tests were conducted on eight Duluth Complex samples that were collected randomly in 1992 from the surface of AMAX test piles FL1 and FL6 (Babbitt prospect of the Partridge River intrusion). Samples with sulfur contents ranging from 0.56 to 1.39 percent (White 1998), virtually all of which was present as sulfide, were run in duplicate and the duplicates were terminated after 60 weeks. The remaining eight samples were continued for 168 weeks. Drainage pH and rates of sulfate, calcium, and magnesium release from the eight pairs of samples during the initial 59 weeks replicated well (White and Lapakko 2000). Data from the first 125 weeks were used for comparison with those from two other laboratories (Lapakko and Antonson 2000a; Lapakko and White 2000; White and Lapakko 2000; Lapakko et al. 2002a), but data from week 125 through 168 have not been reported.

Minimum pH values ranged from approximately 4 to 5, and were in general agreement with values observed at MN DNR for Dunka blast hole samples of similar sulfur content. However, the USBM drainage pH values did not decrease strictly (nor did sulfate release rates increase strictly) with increasing sulfide content, and the lowest pH values were produced by a sample containing 0.97 percent sulfide. The rates of sulfate, calcium, and magnesium release from the USBM samples were typically 25 to 75 percent of those observed in MN DNR experiments on Dunka blast hole samples of similar sulfur contents. The difference in rates was most likely due to the particle size in the USBM experiments, which was large relative to the 0.053-0.149 mm diameter particles used in the MN DNR experiments. A similar effect of particle size on release rates was observed in an experiment conducted as part of an EPA project (see section 3.3.2.3). However, testing of particle size fractions of Duluth Complex rock indicated drainage pH decreased with decreasing particle size. In contrast, the drainage pH values from the USBM study were roughly equal to or, in the case of the 0.97 percent sulfide sample, lower than the associated MN DNR samples.

Beginning in 1996 the **MN DNR conducted additional testing on the 1.39% S** sample tested by the USBM using method ASTM 5744-96 and three modifications of this method (Lapakko 1998b, 1999a; Lapakko and White 2000; White and Lapakko 2000; Lapakko and Antonson 2000a; Lapakko et al. 2002a). (As discussed in section 3.3.1.2, the results were compared to those from the USBM and a commercial laboratory.)

After 265 weeks of testing using the MN DNR drip and flood alternatives to the ASTM method (Lapakko et al. 2002a), the minimum drainage pH values observed were in the range of 4.4 to 4.6. These values were considerably higher than the minimum annual median pH values observed for the 0.8 and 1.4 percent sulfur test piles (see section 3.2.3), which were 4.0 and 3.4,

respectively. That is, the drainage pH values from these laboratory tests did not reach the minimums observed in the field for rock of similar or lower sulfur content. The copper and nickel contents of the laboratory sample were 0.35 and 0.06 percent, respectively, and these values were similar to those for the field test piles. This suggests that the sulfur speciation for the laboratory and field samples was similar. Therefore, the differences in drainage pH values do not appear to be the result of differences in sulfide mineral contents.

The reason for the differences cannot be conclusively identified at present. Possible contributing factors are 1) differences in particle size distribution of laboratory and field rock (for example, a more influential fine size fraction in the field), 2) reduced sulfide mineral reactivity of the laboratory rock because it had been weathered in the field for 15 years, 3) influence of a wider range of sulfur contents for the rock in the field piles, 4) excessive neutralization in laboratory tests resulting from excessive water retention (see Lapakko and White 2000), or 5) development of microbial communities in the field that accelerated sulfide mineral oxidation. It should be noted that drainage pH values reported for another laboratory conducting testing on this sample were as low as 3.7. The low pH was attributed to higher temperatures in this laboratory during the summer months and, therefore, the closer agreement with field conditions might be somewhat fortuitous.

3.3.2.2.2. Babbitt and Dunka Road Prospects

In August 2003 tests were initiated on the dissolution of Duluth Complex rock samples from the **Babbitt and Dunka Road prospects** of the **Partridge River intrusion (PRI)**. Tests were conducted on 75-g samples of –100/+270 mesh (0.053 <d < 0.149 mm) rock and on 1000-g samples of rock finer than 0.25 inches (6.35 mm) in diameter. Seven fine samples were tested, consisting of six samples from the Babbitt prospect and one from the Dunka Road prospect. Sulfur and carbon dioxide content ranges were 0.07 to 0.94 percent and 0.07 to 0.21 percent, respectively. Ten larger samples were tested, seven from the Babbitt prospect and three from the Dunka Road prospect. Sulfur and carbon dioxide content ranges were 0.13 to 1.36 percent and 0.05 to 0.21 percent, respectively. These experiments remain in progress and the following comments assess data generated through week 126 (Lapakko and Antonson 2006).

For both experimental methods drainage pH tended to decrease and SO_4, Ni, Cu, Co, Zn, Ca, Mg, Na, K release rates tended to increase with increasing sulfur content. The -100/+270 mesh rock produced lower drainage pH and higher rates of sulfate and metal release per unit mass than the –0.25-inch rock.

Comparison of the Babbitt prospect and Dunka Road prospect rocks is tenuous because of the limited number of Dunka Road samples, thus the following comments must be viewed as tentative. The variations of drainage pH with solid-phase sulfur content for Babbitt prospect and Dunka Road samples were not substantially different, although sulfate release from the Babbitt prospect samples was greater than or equal to that from the Dunka Road samples. Babbitt prospect samples tended to have a lower ratio of calcium to magnesium release than the Dunka Road samples. Trace metal release from the Babbitt prospect rock tended to be higher than that from the Dunka Road rock. Additional solid-phase analyses will provide insight on the behavior of these samples.

The dissolution behavior of PRI rock (both Babbitt and Dunka Road prospects) was compared to that of the South Kawishiwi rock (samples from the Dunka Mine) using data from laboratory experiments on -100/+270 mesh samples. Data from seven PRI samples (0.07 ≤ percent sulfur ≤ 0.94) and nine South Kawishiwi intrusion samples (0.18 ≤ percent sulfur ≤ 1.16) were used. The PRI samples tended to generate slightly lower pH and higher sulfate release rates than the SKI samples. For a specified sulfur content, drainage pH from the PRI samples was generally 0.1-0.2 units lower than that from the SKI samples. Sulfate release from the PRI samples tended to be about 50 percent higher than that for the SKI samples of the same sulfur content.

Comparison of trace metal release from the Partridge River and South Kawishiwi intrusions is limited to data prior to week 72. Trace metal concentrations from the SKI samples were determined only sporadically after this time. Prior to week 72, nickel release from the PRI samples was roughly 70% higher than that from the Dunka Mine samples. Copper and cobalt release from the PRI samples was not obviously different from that from the Dunka mine samples. Zinc release from the PRI samples was roughly 70% of that from the Dunka Mine samples.

The detailed chemistry, mineral content, and mineral chemistry of the PRI samples have not been determined. These analyses may shed light on the reasons for the observed drainage quality differences relative to the Dunka Mine samples.

3.3.2.3. Effects of Rock Particle Size

To determine the effects of particle size on drainage quality, Duluth Complex rock was separated into six different size fractions and subjected to dissolution testing in December 1993 (Lapakko et al. 1995, 1998, 2004a, 2006). The sample was from AMAX test pile FL6 (0.8% S, Babbitt prospect of the Partridge River intrusion), and its mineral content indicated it was a norite (Lapakko et al. 2006). The particle size diameter ranges for the six size fractions were <0.053, 0.053-0.149, 0.149-0.5, 0.5-2.0, 2.0-6.35, and 6.35-19 mm. The drainage quality results through week 465, reported by Lapakko et al. (2004a, 2006), indicated that as particle size decreased, lag times to pH decreasing below 6.0, minimum drainage pH values, and the ratio of neutralizing mineral dissolution rates to acid producing mineral oxidation rates tended to decrease. These three drainage quality descriptors correlated well with the degree of sulfide mineral exposure. This indicates that particle size reduction increased the acid-producing mineral surface area to a greater extent than the acid-neutralizing mineral surface area.

All size fractions produced drainage pH values below 6.0, and the dissolution time prior to pH decreasing below 6.0 varied from 16 weeks for finer size fractions to 403 weeks for the largest size fraction. The three finest fractions produced minimum pH values ranging from 4.0 to 4.2 after roughly 200 weeks of dissolution, and pH followed a generally increasing trend after 200 to 400 weeks of dissolution. Examination of unpublished data through week 652 revealed ultimate drainage pH values of approximately 6, 6.3, and 5.2 for the <0.053, 0.053-0.149, 0.149-0.5 mm fractions, respectively. Drainage pH values from the three larger fractions tended to decrease slowly and then plateau after 300 to 600 weeks. The minimum values for the 0.5-2.0 mm and 2.0-6.35 mm fractions ranged from 4.8 to 4.9, and the largest fraction produced a minimum value

of 5.8. After 652 weeks the drainage pH values were approximately 4.8, 5.0, and 5.8 for the 0.5-2.0, 2.0-6.35, and 6.35-19 mm fractions, respectively. Average sulfate release rates ranged from 0.014 to 0.80 mmol $(kg \bullet wk)^{-1}$ for the first 465 weeks and increased as particle size decreased.

3.3.2.4. Comparison of Field and Laboratory Data

A similar trend in drainage pH with respect to time and sulfur content was observed in both the laboratory after 78 and 150 weeks, and field test piles experiments over 17 years (Lapakko 1994a). The ultimate drainage pH values for the 0.63% sulfur test piles were similar to values observed in the laboratory. However, minimum pH values for higher sulfur content test piles were about one unit lower than the corresponding laboratory values for these sulfur contents. The rates of sulfate, calcium, and magnesium release in the laboratory were 3 to 10 times higher than those for the test piles.

The lower field rates may be due to several factors. First, field rates of sulfide mineral oxidation may be slower due to differences in the sulfide mineral surface area available for reaction. Since the iron sulfide mineral oxidation produces the acid that drives silicate mineral dissolution, lower release of calcium and magnesium in the field would be expected due to the lower sulfate release. Second, cooler temperatures in the field, and possibly lower oxygen concentrations (although oxygen transport is not likely limiting in the small piles), would retard oxidation. Third, lower field release may be due to limited transport of reaction products resulting from factors related to the magnitude of water addition in the laboratory rinse phase. The amount of water added in laboratory tests is designed to enhance reaction product transport. In comparison, reaction product transport in the field might be limited owing to 1) limited water available to transport reaction products from the piles or 2) increased chemical precipitation in the field due to higher solute concentrations. (That is, solute concentrations in the laboratory tend to be lower due to the relatively large amount of water added during the rinse.)

The observation of field pH values lower than laboratory values at the higher sulfur contents was puzzling. Considering the decrease of pH with decreasing particle size observed in the laboratory (Lapakko et al. 1995, 1998, 2006), drainage pH from the larger field particles would be expected to exceed that from the small laboratory particles. Continued laboratory testing shed light on this apparent contradiction, revealing that the pH of drainage from samples with sulfur contents exceeding 0.40 percent continued to decrease (see also the laboratory section on blast holes at the Dunka site). After 404 weeks of dissolution in the laboratory, the higher sulfur content samples generally approximated field pH values (MN DNR 1997, unpublished data). In contrast, the pH values for the lower sulfur content samples in the laboratory were typically a unit lower than those in the field. This is consistent with the particle size dependence of drainage pH observed in laboratory experiments.

However, given the laboratory-field agreement for drainage pH from the higher sulfur solids, it may also suggest that pH of drainage from the 0.63 percent sulfur field piles would have continued to decrease had the piles not been dismantled. A visually estimated relationship between pH and time was graphed for the low sulfur field piles, which suggested that drainage pH values asymptotically approach a minimum value in the range of 4.6 to 4.8. The 404-week

dissolution pH values observed in the laboratory are in better agreement with the estimated minimum drainage pH values for the low sulfur field piles.

Drainage pH values in the circumneutral range associated with the low-sulfur gabbro at dimension stone quarries (pH 6.5 to 6.8 for sulfur contents less than 0.05%; Lapakko and Oberhelman 1993) are also semiquantitatively consistent with the observed dependence of laboratory drainage pH on sulfur content. Correlation of field data from the Dunka site to laboratory data was tenuous due to uncertainties in field pile composition. The lowest pH at the site, 4.2, was associated with a stockpile reported to contain only 0.24 percent sulfur. Higher sulfur content rock at the site produced higher drainage pH values. It is assumed that the lack of correlation between sulfur content and drainage pH at the site is the result of inadequate data on stockpile composition and/or the presence of relatively small amounts of highly reactive mineralized Virginia formation hornfels in some piles. The presence of this rock could depress drainage pH even when present in small amounts.

3.3.3. Tailings

3.3.3.1. Field Tests

In November 1978, a tailings plot approximately 20 ft x 30 ft x 2 ft deep, denoted FL7, was constructed (Borovsky et al. 1983). A 30-mil Hypalon synthetic liner was placed at the base of the plot to provide an impermeable base for a drainage collection system. The tailings used in the plot were generated from Duluth Complex ore from the Minnamax site, which was processed by the Twin Cities Research Center of the U.S. Bureau of Mines. The tailings were placed on top of the liner, which was sloped to a six-inch PVC perforated pipe that collected leachate. The MN DNR monitored water quality from the plot from 1979 to 1981 (MN DNR 2004).

Based on analyses for a subsequent study, the total sulfur concentration in the tailings was 0.38%, and the copper, nickel, cobalt and zinc contents were 420, 350, 110, 180 parts per million. The pH of drainage from the tailings remained above 7 for the course of the study. Based on laboratory testing with Duluth Complex waste rock, acid drainage could occur at total sulfur contents of 0.2 to 0.4%. Some of these waste rock samples produced circumneutral drainage for 14 years then acidified. Consequently, the comparatively short three-year field study is inadequate to determine if the tailings would generate acidic drainage. Even though the tailings did not produce acid, low levels of trace metals, particularly nickel were released to solution. The maximum concentrations observed for copper, nickel, cobalt, and zinc in the drainage were 0.2, 0.4, 0.09, and 0.03 mg L^{-1}.

Drainage from this plot also contained elevated chloride and sulfate concentrations. Sulfate is generally produced from the oxidation of sulfide minerals present in the tailings, but the source of chloride is not known. Chloride concentrations in the Duluth complex are very low, and the chloride may have been added during the processing or there may have been some contamination of the tailings during the storage at the U S Bureau of Mines.

3.3.3.2.Laboratory Tests

A bulk sample of Duluth Complex rock was collected from the Babbitt prospect of the Partridge River intrusion in April 2001. The sulfide flotation used produced tailings with a 0.2% sulfur content. In February 2002, dissolution tests were begun on 75-g samples in uncovered and covered reactors (both in triplicate) and in duplicate 1000-g humidity cells (Lapakko et al. 2003, Lapakko and Berndt 2009).

Drainage pH from the small, uncovered reactors decreased from near 8 at week 0 to approximately 7 at week 100, and oscillated in the typical range of 6.8 to 7.1 to week 220. Sulfate concentrations decreased from a typical range of 10-20 mg L^{-1} during the first 20 weeks, to about 2 mg L^{-1} at week 100, and subsequently oscillated in the approximate range of 1 - 2 mg L^{-1}. Drainage pH from the small, covered reactors was slightly lower, decreasing from near 8 at week 0 to 6.5 at week 70, with subsequent values in the typical range of 6.5-6.8. Concentrations remained above 10 mg L^{-1} for roughly 60 weeks, and declined to a typical range of 2-4 mg L^{-1} after week 100. For cell 5, a subtle peak in sulfate concentrations occurred between weeks 40 and 66, concurrent with the decline in pH. Drainage pH values from the 1000-g humidity cells decreased slowly and steadily from near 8 initially to 7 at week 220. Sulfate concentrations decreased from 20-30 mg L^{-1} initially to 10 mg L^{-1} at week 100, and subsequently typically ranged from 8-10 mg L^{-1}. These results suggest that the potential for acid production in the first four years of dissolution is low.

If the tailings do not generate acid, trace metal release would potentially be the major environmental concern. For the small, uncovered reactors, nickel and zinc concentrations during the first three weeks typically ranged from 0.004 to 0.008 mg L^{-1} and decreased to a typical range of <0.002 (detection limit) to 0.004 mg L^{-1} subsequently. Copper and cobalt were at or near this same detection limit throughout the period of record. Trace metal concentrations in drainage from the covered reactors were considerably higher. Nickel concentrations were generally near the detection limit during the first 40 weeks and peaked roughly 50 to 80 weeks into the experiment, reaching values near 0.1 mg L^{-1}. The elevated nickel concentrations were concurrent with the decline of drainage pH below about 6.8. Concentrations subsequently decreased and generally ranged from 0.002-0.005 mg L^{-1} after week 150. During the period of maximum nickel concentrations, zinc concentrations peaked near 0.01 mg L^{-1} and cobalt concentrations reached 0.005-0.008 mg L^{-1}. Copper concentrations remained below or near the detection limit.

For the 1000-g humidity cells, nickel concentrations in the first rinse were near 0.05 mg L^{-1}, decreased immediately to near 0.01 mg L^{-1}, and were typically below or near the detection limit of 0.002 mg L^{-1} after week 50. However, concentrations from one of the duplicate cells (cell 7) were consistently reported as 0.03 mg L^{-1} after week 170. Copper concentrations in the intial two rinses were near 0.006 mg L^{-1} and decreased to values typically below detection (0.002 mg L^{-1}) within the first 10 weeks. Cobalt and zinc concentrations were almost always reported as less than detection (0.002 mg L^{-1}). The dependence of heavy metal release on test design, drainage pH, and water retention is discussed in more detail based on a roughly 200-week period of record by Lapakko and Berndt (2009).

Two tailings samples generated from bench scale tests on titanium ore from the Duluth Complex were also subjected to solid phase characterization and dissolution testing described above (Lapakko 1991a, 1993c). The test was very preliminary, and since the two tailings were generated from the same ore, the results do not represent the potential range of ore and tailings composition. The tailings were relatively coarse and contained very few sulfide or carbonate minerals. However, they did contain elevated concentrations of chromium, copper, and nickel. The neutralization potentials determined by static tests greatly overestimated the total calcium carbonate and magnesium carbonate content of these two samples.

These tailings generated drainage of neutral pH during the 52-week dissolution test, and presented virtually no potential for acid production since the sulfide content of both samples was reported as less than 0.01 percent. Copper, nickel, and zinc concentrations (to a lesser extent) were elevated in these tailings and in the initial drainage. Concentrations subsequently decreased, suggesting that only a small fraction of the metals present in the solid phase were readily leachable under the conditions of this experiment (Lapakko 1991a, 1993c).

3.4. Mineral Dissolution Literature Summaries

Extensive research has been conducted elsewhere that benefit interpretation of laboratory and field data on the dissolution of Duluth Complex rock. Thus, a series of annotated bibliographies focusing on rates of mineral dissolution was generated to identify and help summarize those studies most relevant to reactions taking place or expected to be taking place in Minnesota mine wastes. These include acid-producing reactions involving oxidative dissolution of pyrite (Leopold and Lapakko 2001a) and pyrrhotite (Leopold and Lapakko 2001b) and acid-neutralizing reactions involving carbonates (Doctor et al. 2001) and silicates (Phillips et al. 2001).

4. MITIGATION TECHNIQUES

4.1. Introduction

The degree of mitigation required for mining wastes depends directly upon the potential for problematic mine waste drainage quality at a site. Appropriate methods of mitigation can be designed and optimized based, in part, upon the predicted drainage quality. The MN DNR Division of Lands and Minerals has studied both passive and active mitigation techniques in the laboratory and on an operational scale. Results from these studies are presented below.

4.2. Literature Reviews.

Extensive literature exists on the subject of mitigation techniques. The MN DNR has compiled and summarized the current state of environmental mine waste technology, specifically as it relates to present and possible future ore deposits located in Minnesota. Fifteen environmental mine waste strategies were identified, which could be subdivided into three separate categories: those focused on prevention, control and treatment of problematic mine waste drainages (Jakel and Lapakko 1999; Leopold and Lapakko 2000, 2001c).

4.3. Operational Scale Mitigation

In the early 1980s, the U.S. Bureau of Mines funded a two-phase program examining the feasibility of removing trace metals (Cu, Ni, Co, Zn) from stockpile drainage using readily available materials (peat, till, wood chips, Cu-Ni tailings, zeolite) in low-cost, low maintenance systems. Following the initial literature review and laboratory experimental phase that used batch and column tests to evaluate metal sequestration by the materials, a field study phase was conducted at the AMAX/Kennecott site. The results of the entire program are presented in two reports (Lapakko et al. 1986a, 1986b), and synopses of various research segments are presented in several symposium proceedings (Eger et al. 1984; Lapakko and Eger 1988, 1983, 1981a; Lapakko et al. 1983).

Passive and active mitigation techniques have been applied to stockpiles and stockpile seepages at the Dunka Mine to prevent, control, and treat known and potential water quality problems (LTV Steel Mining Company 1996). Prevention methods include stockpile capping, diversion ditches, and moving rock off site (STS Consultants Ltd. 1993, 1994a, 1994b). Stockpile capping techniques were selected based on the stockpile drainage quality and chemical composition. Stockpiles with no known water quality problems and relatively low sulfur contents were capped using standard mineland reclamation (two feet of overburden). Stockpiles with known water quality problems or elevated sulfur contents were capped with either enhanced mineland reclamation (two feet of minus 6-inch uncompacted material with at least 10% in the -200-mesh fraction: stockpiles 8011, 8031, and 8012); enhanced mineland reclamation with linear low density polyethylene (LLDPE) membrane lined ditches (stockpiles 8013 and 8014); or covered with a LLDPE membrane liner (8018 and the hornfels pile on the 8014) (STS Consultants Ltd. 1994a).

Limestone was also added to the hornfels pile on the 8014 stockpile (LTV Steel Mining Company 1996). This pile was constructed of random hornfels rock identified on top of the 8014 and 8031 stockpiles (LTV Steel Mining Company 1996). The hornfels presence was the result of blast contamination "at the interface between the gabbro and/or waste rock" (LTV Steel Mining Company 1996). LTVSMC estimated that such contamination was on the order of "a few thousand tons on a 5 to 10 million ton stockpile."

Diversion ditches to reduce surface and ground water input to stockpiles were constructed on the west side of the 8014 draining to the pit; the west side of the 8013 draining to the pit; a bedrock cut in Unnamed Creek east of Seep 1 to lower the water table at the toe of the 8013 stockpile; from the Environmental Sump to the pit; on the south side to the 8011 draining into the Environmental Sump and subsequently into the pit; and to redefine the east 8011 drainage ditch.

A mass of 262,652 long tons of mineralized Virginia Formation hornfels rock was moved off site to LTV's tailings basin. The rock, which had a sulfur content of 3.2 percent, was mixed with 31,468 tons of dolomitic limestone (to neutralize acid) and buried in the tailings to inhibit oxygen transport to the rock and consequent oxidation of the sulfide minerals present. The limestone addition yielded a value of 1.2:1 for the ratio of NP added to acid production potential

present as sulfur. The tailings also contained ankerite for additional neutralization potential and iron oxyhydroxide minerals as potential trace metal adsorbing surfaces.

Treatment methods at Dunka include passive wetland treatment systems (Eger 1992; Eger et al. 1996, 1997, 1998) and an active treatment plant (STS Consultants Ltd. 1994b). Constructed wetland treatment systems include limestone/peat systems at seeps W2D/W3D (061) and Seep X (044), a limestone pretreatment followed by a limestone/peat system at W1D (051), and a limestone pretreatment followed by a peat system at Seep 1 (043). A limestone/peat system was constructed in 1998 for EM8 (041). The active treatment plant is a lime precipitation system designed to treat flow from EM8, Seep1, Seep X, and W1D treatment systems.

4.4. Blending Alkaline Solids with Mine Waste

4.4.1. Introduction

The addition of alkaline solids to acid-producing mine waste may provide 1) short-term mitigation by neutralizing acid produced by iron sulfide oxidation and 2) long-term mitigation by facilitating the passivation of iron sulfide mineral surfaces. Laboratory experiments blending Duluth Complex with rotary kiln fines and with limestone have been in progress since 1988. Field tests blending Archean greenstone with limestone were initiated on 24 October 2000.

4.4.2. Laboratory experiment with Duluth Complex rock

In a laboratory experiment which began in May 1988, rotary kiln fines (RK fines; a waste product generated by the conversion of limestone to lime), ˜10 mesh ($d < 2$ mm) limestone, and +10 mesh/˜0.25 inch ($0.2 \text{ mm} \leq d < 6.35$ mm) limestone were each mixed with finely-crushed Duluth Complex rock ($0.053 < d$ ˜ 0.149 mm) to examine their effectiveness in reducing the release of acid and trace metals in drainage from the rock. The finely crushed Duluth Complex rock, simulating tailings or fine waste rock, contained 2.1 percent sulfur and the predominant iron sulfide mineral was pyrrhotite. The combined calcium and magnesium carbonate (96.7% and 2.1%, respectively) content of the limestone was 99.1 percent as $CaCO_3$. The calcium content of the RK fines indicated a combined calcium carbonate/oxide content of 63 percent and 96.7 percent of the mass was finer than 0.075 mm.

In addition to duplicate control reactors, five loadings (0.5, 1.0, 2.0, 3.0, and 5.0 g alkaline solid (g sulfur in rock)$^{-1}$) of each solid were mixed with 75 g Duluth Complex rock. Duplicate reactors were used for the 1.0 and 3.0 g loadings. This produced neutralization potential to acid production potential quotients (NP/AP) of 0.11 to 1.1 for the RK fines and 0.16 to 1.6 for the limestone. The mixtures were subjected to a “wet-dry cycle” laboratory test, which is yet in progress. The following section summarizes the results of the experiment for the first 585 weeks. Additional detail on this study is available in three status reports (Lapakko and Antonson 1989a, 1990a; Lapakko et al. 1998a) and three papers (Lapakko and Antonson 1991; Lapakko et al. 1997, 2000).

The pH of drainage from the rock alone decreased below 6.0, a common minimum water quality standard, after 8 weeks and reached a minimum of 3.3 in 117 weeks, at which time the controls

were terminated. Maximum copper and nickel concentrations ranged from 1 to 2 ppm. The sulfate release did not decrease over time. The minus 0.25-inch/+10 mesh limestone treatment produced drainage similar to that of the controls and was discontinued after 40 weeks. In contrast, the RK fines and -10 mesh limestone maintained drainage pH above 6.0 for 75 to 585 (still in progress) weeks. While pH was in this range, sulfate concentrations decreased and trace metal concentrations were typically three to ten percent of those observed in drainage from the controls.

The RK fines elevated pH and inhibited sulfate release for time periods that increased with the mass of RK fines present. The pH of drainage from the 0.11 quotient (NP/AP) loading decreased below pH 6.0 after 75 weeks of dissolution and reached a minimum of 3.6 after 117 weeks, at which time the reactor was terminated. Drainage from the duplicate 0.22 quotient loadings of RK fines decreased below 6.0 after 95 and 170 weeks. The former was terminated after 117 weeks, and the latter (still in progress) reached a minimum pH of 2.8 after 512 weeks. Drainage pH from the 0.66 quotient decreased below 6.0 after 581 weeks. For the drainages that acidified, the cumulative calcium release indicated that 92 to 103 percent of the calcium carbonate/oxide initially present in the RK fines was dissolved at the time pH decreased below 6.0. Concentrations of nickel, cobalt, and zinc reached maximums after pH declined below 6.0, and copper concentrations tended to increase steadily as pH decreased. The pH of drainage from the 1.1 quotient, which remains operative, typically remained above 8.0 throughout the 585 week period of record.

Sulfate concentrations in the drainage generally declined until pH decreased below 6, and the highest values were observed after drainage acidification. With the 0.65 and 1.1 quotient loadings, drainage pH values were typically near 9.0, although after week 340 the pH of drainage from the 0.65 loadings began to oscillate occasionally into the sevens. Such oscillations preceded acidification of drainages from the 0.22 quotient loadings. Sulfate concentrations in the drainage from the 0.65 quotient loadings have decreased throughout the period of record. For each mixture, regression analysis for the equation $[SO_4] = ae^{bt}$ was used to describe the decrease of sulfate concentrations (mg L^{-1}) with time (t, weeks) over the period during which pH remained above 6. The variables "a" and "b" are constants, and the values of b for the six cases ranged from -0.016 to -0.0031 (Lapakko et al. 1998a).

The - ˜10 mesh limestone additions elevated drainage pH and net alkalinity while inhibiting sulfate release. The pH of drainage from the 0.16 quotient (NP/AP) loading was initially neutral, decreased below 6.0 after 109 weeks, reached a minimum of 5.0 at week 116, and subsequently increased to a typical range of 5.3 to 6.3 through week 709. This is in contrast to the pH from the lower loadings of RK fines, for which pH continued to decline after falling below 6.0. The lack of more extreme acidification was due to a decrease in the rate of sulfide oxidation by more than 80 percent. It is of interest to note that the sulfide oxidation rate did not increase after pH decreased below 6.0. Drainage pH from the higher limestone loadings remained above 6.0 and sulfate concentrations decreased with time.

As with the RK fines, regression analysis was used to describe the decrease of sulfate concentrations with time (see above), and the constant "b" for the seven cases ranged from -0.011 to -0.0066. The correlations for the fine limestone were considerably higher than those

for the RK fines. All loadings reduced the sulfide mineral oxidation rates, and the loadings of 1:1 and greater reduced them to levels at which host rock silicate mineral dissolution was adequate to neutralize the resultant acid production. Furthermore, the amount of silicate minerals present was large enough that their dissolution would neutralize the acid produced if the pyrrhotite present continued to oxidize slowly. The decline in sulfide oxidation may have been due to creation of a leached layer at the pyrrhotite surface, pyrrhotite coating by products of precipitation reactions enhanced by the presence of the limestone, or cementation of pyrrhotite grains due to reactions within the bed. Fresh and leached solids were examined using scanning electron microscopy (SEM). These examinations identified varying degrees of oxidation features on leached sulfide mineral grains but did not provide a clear indication of the mechanism producing the superior mitigation by the limestone (Nicholas et al. 2006, 2007).

This mitigative technique has excellent potential for neutralizing acid produced over a period of several years as an interim measure and promising potential for long term mitigation. However, certain precautions must be taken with the conclusions. First, the results are specific to the Duluth Complex mineralogy. For example, if sulfide minerals were coated by precipitates the coating composition, as well as mechanisms and rates of coating, may be specific to this rock formation. In addition, the dissolution of host rock minerals of other rock types is likely to provide different rates and capacities of acid neutralization than those of the Duluth Complex.

Second, these results were determined under laboratory conditions. The extent of limestone/mine waste mixing in the field is likely to be less complete than in the laboratory, and this extent is likely to influence the effectiveness of acid neutralization and sulfide mineral coating. In particular, it is difficult to obtain intimate contact when mixing relatively fine limestone with large waste rock particles. Environmental variables such as large fluctuations in temperature may also produce results deviant from those observed in the closely controlled laboratory environment. Of particular importance is determination of the longevity of the acid production inhibition.

4.4.3. Field Experiment with Archean Greenstone Rock

In October 2000 a field experiment was initiated to examine the effect of mixing limestone with Archean greenstone rock that was expected to produce acidic drainage (Lapakko et al. 2001, 2003). The tests are being conducted in six polyethylene plastic tanks (d = 48 in., h = 42 in.), consisting of two control tanks (tanks 1 and 6), two tanks with limestone mixed at a 1:1 ratio (tanks 2 and 5), and two tanks with limestone mixed at a 3:1 ratio (tanks 3 and 4). The 1:1 ratio represented a limestone loading that would neutralize all acid produced by oxidation of the iron sulfides present in the rock, assuming complete reaction of both limestone and iron sulfides. The 3:1 ratio represented a safety factor of three for the limestone addition. The tanks were fitted with 2-inch slotted PVC outlet pipe on the bottom, which drains into a 22-gallon polyethylene plastic sample collection sump. The tanks were filled on 24 October 2000 and input to the tanks is limited to precipitation. The tanks are housed in a 20' x 20' lined bin that serves as a double containment.

The 0.67% sulfur rock used in test pile 4 of the Archean greenstone prediction study was selected for use in the alkaline mixing experiment. The rock was screened to pass a 1.5-inch

screen at Casper Construction Inc. in Grand Rapids, MN. A bobcat loader was used to fill the tanks with the exception of tank 5 which could not be reached with the loader and was filled using five gallon buckets.

For the tanks that had the rock mixed with limestone, the loader scooped a bucket of rock and then the limestone (1/3 of the total addition), which had been weighed, was added to the bucket. A sample of rock was taken from each loader bucket, prior to adding the limestone for the alkaline mixing tanks, for a total of three samples per tank. A total of 55 pounds of limestone was added to the 1:1 ratio tanks and 165 pounds to the 3:1 ratio tanks.

The drainage pH from all tanks was initially slightly alkaline, ranging from the upper sevens to middle eights. Drainage pH from the controls decreased below 6.0 in September 2004. As of September 2011 pH values are in the low fours and acidity is in the range of 40 to 50 mg L^{-1}. Drainage pH values from the tanks with limestone added have been consistently above 7.3. The 1:1 limestone addition tanks presently have pH values in the middle to upper sevens and alkalinity ranging from 35 to 50 mg L^{-1} as $CaCO_3$. The 3:1 limestone addition tanks presently have pH values near eight and alkalinities in the range of 40 to 70 mg L^{-1} as $CaCO_3$. These tests remain in progress with the objective of determining the effectiveness of the limestone addition over time.

4.5. Drainage Treatment with Limestone Beds

4.5.1. Laboratory tests

Column laboratory experiments to examine the ability of limestone beds to neutralize acidic stockpile drainage began in April 1988. Triplicate columns containing 780 grams of minus 0.25-inch/+10-mesh high calcium limestone were used to treat each of three different drainages. Flow rates were adjusted to maintain an effluent in which the alkalinity exceeded the acidity. Although the columns were successful in treating all three drainages, problems with plugging typically led to termination of columns. These problems occurred at different times and, consequently, periods of record for the columns varied. The termination dates for individual columns ranged from September 1990 to December 1998. Excluding periods when columns were nonfunctional, the periods of operation ranged from 856 to 3682 days.

For Seep 1, input flow was maintained at an average of 4.9 bv d^{-1} (bed volumes per day) for most of the experiment, with total flow volumes ranging from 4200 to 5700 bed volumes. The mean input values for pH and net alkalinity were 5.6 and -22 mg L^{-1} as $CaCO_3$. Limestone dissolution elevated these values to 7.5 and 20 mg L^{-1} as $CaCO_3$ in the column effluent.

For FL3, input flows averaged 0.15 bv d^{-1}, with total flow volumes ranging from 160 to 610 bv. The mean pH and net alkalinity were 4.9 and -210 mg L^{-1} as $CaCO_3$. Dissolution of limestone elevated drainage pH and net alkalinity to approximate respective values of 7.9 and 75 mg L^{-1} as $CaCO_3$ in the column effluent.

For FL6, input flows averaged 0.51 bv d^{-1}, with total flow volumes of 550 to 990 bv. The mean input pH and net alkalinity were 4.1 and 580 mg L^{-1} as $CaCO_3$. Dissolution of limestone

elevated these values to respective means of 7.6 and 120 mg L^{-1} as $CaCO_3$ in the column effluent.

Copper concentrations were reduced by 85 to 98%. Nickel, cobalt and zinc concentrations were reduced to a lesser extent (Ni: 11-24%, Co: 7-22%, Zn: 35-65%). These results indicate that limestone beds are capable of neutralizing these stockpile drainages and substantially reducing copper concentrations if adequate detention time is allowed. Problems with maintaining flow, due to cementation of the bed, were encountered. Use of larger particles would reduce potential for such problems in the field but decrease the reactivity per mass limestone. Additional detail is available in the final report (Lapakko et al. 2002b), status reports (Lapakko and Antonson 1989b, 1990b), and a shorter technical publication (Lapakko and Antonson 1990d).

4.5.2. Field test

A field scale limestone treatment bed (1.4 m^3 bed volume) began treating the Seep 1 drainage at the Dunka site on 26 September 1988 (Lapakko and Antonson 1989c, 1990c; Lapakko et al. 1998). The objectives of this project were to:

1) elevate the pH and alkalinity while reducing the acidity and trace metal concentrations in the Seep 1 drainage;

2) describe the variation of treatment efficiency in terms of pH elevation and trace metal removal with the volume of drainage treated; and,

3) describe the variation of treatment efficiency as a function of detention time, or equivalently, flow rate.

The bed contained 2020 kg of high-calcium limestone (minus 6.4 mm), and was 1.3 m in diameter and 1 m deep. It received Seep 1 flow from 26 September to 28 October 1988, from 26 April until 31 October 1989, and from May 5 to July 3. Operation was terminated in 1988 and 1989 when freezing conditions were impending. The bed clogged persistently throughout the study and, consequently, the limestone was replaced with a coarser-sized limestone (6.4 to 38.1 mm) July 12, 1990 (Lapakko et al. 1998).

The total flow volumes through the fine particle-size limestone bed during 1988, 1989, and 1990 were 580, 6600, and 3600 m^3, respectively, yielding an average flow of 13, 25, and 45 bed volumes per day, respectively. The bed was 100% efficient in neutralizing acid in the seepage, raising the pH from approximately 5.1 to 7.0, and the net alkalinity from approximately -46 mg $CaCO_3$ L^{-1} to 28 mg $CaCO_3$ L^{-1}. Copper removal by the bed increased each year, reaching 68% in 1990. Zinc removal also increased from about 12% in 1988 and 1989 to 42% in 1990. Nickel and cobalt concentrations were reduced by approximately 10% throughout the study. The flow volume treated or the range of detention times observed in this study did not influence treatment efficiencies for acid neutralization and trace metal removal.

The average rates of acid neutralization by the fine particle-size limestone bed were 14, 32, and 72 mg $CaCO_3$ s^{-1} in 1988, 1989, and 1990 respectively, implying that 0.84 T of limestone was dissolved during the operation of this bed. The release rate was independent of the volume of

treatment, indicating that the treatment capacity of the bed was not taxed. The release rate did increase with flow, indicating that for the range of flows observed, the initial reaction of the Seep 1 drainage with the limestone was relatively rapid. Although the bed provided acceptable elevation of pH and net alkalinity, as well as some reduction in trace metal concentrations, the observed flow impedance was a problem that had to be addressed.

Larger-sized limestone particles were tested from July 12 to August 6, 1990, and although no overflow occurred, treatment efficiency decreased below acceptable levels. Total flow through this bed during the 24-day period of operation was 790 m^3, or 23 bv day^{-1}. However, the bed was only 25% efficient in neutralizing acidity in the seepage. The mean effluent pH only reached 4.9, and the effluent net alkalinity averaged -64 mg $CaCO_3$ L^{-1}. Furthermore, no more than 10% of copper, zinc, nickel, or cobalt was removed from the drainage. The mean rate of acid neutralization was a nominal 10 mg s^{-1}, indicating that only 0.02 T of limestone was dissolved.

The flow rates were too high to allow an adequate detention time for the larger limestone particles to effectively treat Seep 1 drainage quality. The rate of calcite dissolution is proportional to surface area. Since the surface area of the fine particles was approximately seven times that of the coarse particles, their dissolution rate would be approximately seven times faster than those of the coarse particles. It would therefore be expected that a limestone bed using the larger particles would require a detention time seven times that of the fine particles for adequate acid neutralization to occur. To achieve this detention time, a minimum bed volume of 3024 L would be required. However, only 25% of the Seep 1 drainage acidity was neutralized by the large particles in the limestone bed. This indicates that the bed volume should be increased by at least a factor of four to achieve adequate neutralization for the flow rates encountered.. Additional investigation will be necessary to determine the actual bed volume required to neutralize Seep 1 drainage.

4.6. Subaqueous Disposal

4.6.1. Introduction

Five laboratory experiments and one field experiment were conducted to examine the effectiveness of subaqueous disposal and subaqueous disposal with various barrier layers for controlling the oxidation of sulfide minerals present in mine wastes. 1) Unmodified subaqueous disposal tests were initially conducted in 2-liter Erlenmeyer flasks. 2) Subsequent tests were conducted in 2-inch-diameter columns to describe the release of sulfate as a function of sulfidic mine waste depth for two different mine waste particle sizes. 3) In a second phase of the Erlenmeyer flask experiment, composted yard waste was added to some of these flasks to determine if organic amendments would further inhibit sulfide oxidation. 4) Composted yard waste and fresh rock, as well as controls for compost and rock, were subjected to subaqueous dissolution in the laboratory. 5) A column experiment was conducted to determine the effectiveness of taconite tailings, yard waste compost/tailings mixtures, and limestone layers above tailings as potential oxygen diffusion barriers above sulfidic mine waste. 6) Barriers of tailings, a yard waste compost/tailings mixture, and limestone are presently under examination in small-scale field tests.

4.6.2. Unmodified subaqueous disposal

4.6.2.1. Laboratory flask experiments

The first 120 weeks of the flask experiment begun in September 1990 were summarized by Lapakko (1994c). The experimental objective was to determine the effect of three different subaqueous disposal techniques on the oxidation of sulfide minerals present in Virginia Formation hornfels rock containing about 5.5 percent sulfur and about 14 wt% pyrrhotite ($Fe_{0.9}S$). The subaqueous techniques were unmodified subaqueous disposal, subaqueous disposal with alkaline addition, and pretreatment (rinsing and neutralization) of rock prior to subaqueous disposal and subsequent alkaline addition. The pH of drainage from the two unsaturated controls, representing on-land disposal, ranged from 3.7 to 4.2 during the initial 100 weeks, then decreased rapidly to 3.2 at 120 weeks. During the first 100 weeks the sulfate release rates averaged roughly 2.0×10^{-12} mol (g rock)$^{-1}$ s^{-1}; concurrent with the pH decrease, rates for controls increased to 5.6×10^{-12} and 13×10^{-12} mol (g rock)$^{-1}$ s^{-1}. For unmodified subaqueous disposal, pH declined steadily from 4.5 initially to 3.5 after 120 weeks. Pyrrhotite oxidation rates for the three subaqueous disposal techniques were 1.5×10^{-12}, 1.7×10^{-12}, and 2.2×10^{-12} mol (g rock)$^{-1}$ s^{-1}, respectively. With the exception of some initial rapid sulfate release from rock that was not pretreated, these rates were relatively constant over the course of the experiment. The rates observed in the laboratory were used, along with oxygen transport theory, to determine acid production rates for disposal of similar mine waste in a flooded open pit.

4.6.2.2. Laboratory column experiments

Assuming oxygen transport was limited by diffusion through a quiescent layer of water, it was estimated that the depth of solids in the flasks (see previous section) was near that at which oxygen transport became limiting. Due to the uncertainty of values for variables necessary for this calculation, an experiment was initiated in March 1997 to 1) determine the depths of fine ($0.053 < d \tilde{} 0.149$ mm, 1.22 percent S) and coarse ($0.635 < d \tilde{} 1.905$ mm, 0.6 to 1.0 percent S) sulfidic Duluth Complex rock at which oxygen transport limited the rate of sulfide mineral oxidation and 2) better estimate values for oxygen transport variables for the materials used. Bed depths of 1 to 20 cm were used for the fine particles, and two standard Wet-Dry Cycle reactors were added after the columns were underway. Coarse particle bed depths of 5 to 150 cm were used. A 25-cm depth of water was maintained above the submerged beds, and unsaturated columns were run at three depths of fine and coarse particles for comparison with oxidation rates under saturated conditions.

The depth of solids at which oxygen transport limited the rate of sulfide mineral oxidation in the fine and coarse tailings was on the order of 20 and 100 cm, respectively. The respective maximum sulfate release rates observed were 2.5×10^{-8} and 1.5×10^{-8} mole m^{-2}s (m^2 represents the horizontal cross sectional area of the bed). These values are about an order of magnitude higher than those preliminarily predicted based on oxygen diffusion alone, perhaps due to advective oxygen transport due to thermal currents.

Although higher than indicated by preliminary theoretical calculations, sulfate release from the subaqueous bed was lower than that from the unsaturated beds. Sulfate release from the deeper fine unsaturated columns was apparently limited by formation of a saturated layer at the top of the bed and, consequently, duplicate 75-g masses were subjected to unsaturated oxidation in standard MN DNR wet-dry cycle reactors to obtain data for an unsaturated condition. Weekly sulfate release from these reactors (3.0×10^{-8} mole $m^{-2}s^{-1}$ = 0.50×10^{-6} mole (g rock•wk)$^{-1}$) exceeded the observed range of 1.2×10^{-8} to 2.5×10^{-8} mole $m^{-2}s^{-1}$ from all masses of saturated tailings (38 to 555 g). The sulfate release rates (per unit cross-sectional area) for the unsaturated coarse particle bed depths of 5, 50, and 150 cm were 1.5 to 3 times those for the corresponding saturated beds.

The sulfate release from the unsaturated fine and coarse rock was in good agreement with rates previously reported for unsaturated Duluth Complex rock. Release per unit mass unsaturated fine rock (0.50×10^{-6} mol (g rock•wk)$^{-1}$) was in good agreement with average values previously reported by Lapakko (1993a) for similarly sized samples with sulfur contents of 1.12 and 1.16 percent (0.59×10^{-6} and 0.31×10^{-6} mol (g rock•wk)$^{-1}$, respectively). Sulfate release from the unsaturated beds averaged 0.017×10^{-6} mole (g rock•wk)$^{-1}$, which agreed well with the 0.02×10^{-6} mole (g rock•wk)$^{-1}$ rate observed for the 0.25-0.75 inch fraction of 0.9-percent-sulfur gabbro (Lapakko et al. 1995).

The two experiments discussed above revealed that separating sulfidic mine waste from the atmosphere with a layer of water would reduce the rate of oxidation of sulfide minerals present. However, oxidation would continue and possibly produce acidic waters. Consequently, various modifications to subaqueous disposal are presently being examined.

4.6.3. Subaqueous Disposal with Modifications

4.6.3.1. Organic Amendment Flask and Beaker Experiments

The potential benefit of adding an oxygen consuming layer between sulfidic rock and the overlying water in the flasks experiment is under examination (MN DNR Hibbing laboratory, unpublished data). After 238 weeks the addition of alkalinity to the flasks was terminated (see above), and 75 grams of yard waste compost was added to one of each pair of duplicate reactors. The quality of water was monitored for an additional year, at which time the pH of water in the three reactors receiving no compost was in the range of 3.4 to 4.3. For reactors receiving compost, the pH remained near seven or, in the case of the initial unmodified subaqueous treatment, increased to this level shortly after the addition. After 26 weeks of reaction, the pH in one of the compost amendment reactors (initial treatment 3) decreased below pH 6.0. At the end of the period of record, the pH in the two remaining organic amendment reactors was in the upper sixes to the lower sevens. Rates of sulfide oxidation were calculated for the different disposal techniques based on the rate of sulfate appearance in solution. The aqueous concentrations of phosphorous, ammonia-ammonium nitrogen, total Kjeldahl nitrogen, nitrate, and nitrite from the organic substrate were also determined. These results have not been formally analyzed and compiled.

The solids in the previous experiments had been weathering in a subaqueous environment for roughly 4.5 years prior to the addition of compost. A second experiment was begun using six 4-liter beakers: two composted yard waste controls (75 g/3.6 L), two unleached mineralized Virginia Formation hornfels controls (4.47% S, 200 g/3.6 L), and two with the compost (75 g/3.6 L) and hornfels rock (200 g/3.6 L). The composted yard waste controls were run for 20 weeks and pH was in the typical range of 7.5 to 7.8. The rock controls initially had pH values in the upper fours, and then gradually decreased to near 3 at week 91. The pH for the combined rock and compost was initially in the mid-sevens, dropped below 6.0 at week 113, and decreased to the mid-fives at week 125. In contrast, the solids subjected to the initial subaqueous oxidation (in the aforementioned flask tests) maintained pH above 6 for 179 weeks. It is possible that during the 224-238 weeks of subaqueous oxidation in the previous test, a) some of the more reactive sulfides were removed or b) the sulfide mineral surface had become less reactive. Such a reduction in reactivity could be the result of formation of a leached layer on the pyrrhotite. The difference in effectiveness may also have been due to differences in the initial compositions of the rock and/or compost in the two experiments. Additional analysis and formal reporting of these results is in progress.

4.6.3.2. Subaqueous Disposal plus Barrier Layers Column Experiment

A second laboratory experiment, conducted in columns, was initiated in March 1997 to examine the effect of placing various barriers above sulfidic wastes (-4/+14 mesh rock) disposed in a subaqueous setting. The intent of these barriers is to further inhibit oxygen transport to the mine waste and, therefore, decrease the rate of sulfide mineral oxidation. The barriers under examination are tailings (1, 2, 3, 5, 10 cm depths), tailings/yard waste compost mixtures (0.1 percent compost with tailings at depths of 1, 3, 10 cm; 0.5 and 1.0 percent compost with tailings at 3 cm depth), and a layer of limestone above tailings (1 cm limestone over tailings depths of 0, 1, 3, 10 cm). Controls were included for unamended -4/+14 mesh rock and unscreened -3/4-inch rock in a subaqueous setting, and -4/+14 mesh rock in an unsaturated condition. Initial calculations indicated that sulfate release rates from the thicker covers were 15 to 50 percent of those from the subaqueous controls. Rates also tended to decrease as the amount of compost added increased. Rates from some of the thinner covers were similar to those for the subaqueous control, likely due to visible physical flaws in these covers. The initial results suggest that these modifications warrant additional examination. Additional analysis and formal reporting of these results is in progress.

4.6.3.3. Field Tests on Subaqueous Disposal plus Barrier Layers

A field test examining the effectiveness of limestone, tailings, and a tailings compost mixture above sulfidic rock in a subaqueous environment began in December 1997 at the MN DNR Research Site in Hibbing. The subaqueous tests are being conducted in large cylindrical tanks (d = 46 in., h = 120 in.), fitted with a 30 mil PVC liner. Inside the PVC liner was a filter fabric liner that extended to a height of 48 inches. Wells (i.d. = 1 in.) were installed to sample at distances of 3 and 39 inches from the bottom of the tanks.

The tanks were filled to a depth of 48 inches with -3/4-inch Duluth Complex rock, originally collected from a 1996 bulk sample and subsequently crushed at the NRRI Coleraine Research

Center. The sulfur contents of composites from each of the four tanks averaged 0.66% (0.63 to 0.69%). These values compare favorably with values determined for the same rock in the column experiments (0.63%), a composite sample taken from the barrels in which the samples were shipped (0.71%), and two grab samples taken by the NRRI from a larger pile of this material at Coleraine (0.76%).

The rock was placed into the tanks 5 November 1996, and water from the Hibbing Taconite Scranton Mine pit was added to the tanks from above (to a level one foot from the top of the tank) on 10 December 1996. The bottom port and surface water were sampled eight times and the middle port six times prior to the addition of barrier layer amendments. In August 1997, water was drained from the tanks and well water was added from below to the surface of the rock. Water addition to the control tank was continued to a level one foot from the top of the tank.

To the remaining three tanks, the amendments were added prior to filling the tanks to this level. 53 liters (14 gallons) of each amendment were added, representing an amendment depth of approximately 4.4 cm (1.75 inches). The mass additions of limestone, tailings, and tailings plus compost were 87 kg, 81 kg, and 81 kg, respectively (191, 179, 179 lbs). The limestone and tailings were added on 26 August and the tailings/compost mixture on 27 August. The tailings/compost mixture contained 0.406 kg dry compost, or five percent of the mixture. Over time, water quality samples will be collected from the two well depths and the surface water to determine the rate of sulfate release and, by implication, the rate of sulfide mineral oxidation in the various settings. The first samples were collected on 28 August 1997 (week 0).

Two unsaturated tanks (d = 48 in., h = 42 in.) were also established to generate comparative water quality data for an on-land disposal scenario. These tanks were fitted with 1-inch slotted PVC outlet pipe on the bottom and were filled to a height of 38 inches with the same Duluth Complex rock. Subsequently a bulk density of 106.5 lb ft^{-3} (also 37% porosity, 2.7 specific gravity) was determined, indicating the rock mass was approximately 1930 kg. On 28 August 1997, 65 gallons of tap water was added to each tank to check for plumbing leaks and to attain field capacity. A grab sample (week 0) was collected on the same day from the initial drainage. Subsequent input to the tank will be limited to precipitation.

Through week 222, pH of water in the four subaqueous treatments ranged from 7.0 to 8.9, and values did not vary markedly among the four tanks. Values for all subaqueous tanks tended to be higher in the surface water. In contrast, pH values for the unsaturated controls at week 222 ranged from 4.2 to 4.5. Additional analysis and formal reporting of these results is in progress.

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APPENDIX 1

MINE WASTE DISSOLUTION CHEMISTRY

1. Acid Production

1.1. Iron Sulfide Mineral Oxidation.

When metal sulfide minerals are exposed to the oxidizing conditions present in waste rock stockpiles, mine walls, and tailings basins, sulfide is oxidized and the associated metals (Fe, Cu, Ni, Co, Zn) are released (Nelson 1978; Garrels and Christ 1965; Sato 1960a, 1960b). Acid is produced as a result of the oxidation of iron sulfide minerals present in mine waste, as indicated by reaction 1 (Nelson 1978) and reaction 2 (Stumm and Morgan 1981). Two moles of acid are produced for each mole of sulfur oxidized.

$$FeS(s) + (3/2)H_2O + (9/4)O_2(g) = FeOOH(s) + 2H^+(aq) + SO_4^{2-}(aq) \quad (1)$$

$$FeS_2(s) + (5/2)H_2O + (15/4)O_2(g) = FeOOH(s) + 4H^+(aq) + 2SO_4^{2-}(aq) \quad (2)$$

The rate of iron sulfide oxidation by oxygen is proportional to the available sulfide surface area (Nelson 1978; Sato and Mooney 1960; Sato 1960a, 1960b), and dissolved oxygen concentration (Nelson 1978; Dobrokhotov and Maiorova 1962; McKay and Halpern 1958), with only a slight dependence on pH (Nelson 1978; Majima and Peters 1966). These oxidation reactions can elevate aqueous concentrations of iron, sulfate, and acid. These components can 1) accelerate the oxidation of other sulfide minerals, 2) precipitate hydrated iron-sulfates or other minerals, 3) contact host rock minerals which react to neutralize some or all of the acid, and/or 4) exit the mine waste as acidic flow, referred to as acid mine drainage (AMD) or acid rock drainage (ARD).

Ferric iron oxidizes iron sulfides more rapidly than oxygen (Singer and Stumm 1970). Research on pyrite oxidation indicates that as "pH decreases to 4.5, ferric iron becomes more soluble and begins to act as an oxidizing agent" (Nordstrom 1982). As pH further decreases, bacterial oxidation of ferrous iron becomes the rate-limiting step in the oxidation of pyrite by ferric iron (Singer and Stumm 1970), which is the only significant oxidizing agent in this pH range (Nordstrom 1982; Singer and Stumm 1970; Kleinmann et al. 1981). Under these conditions the reaction is independent of sulfide-mineral surface area (Singer and Stumm 1970).

1.2. Sulfate Mineral Dissolution.

Hydrated iron sulfate minerals may precipitate due to evaporation of acidic, iron- and sulfate-rich water within mine wastes. This essentially stores acid generated by iron sulfide mineral oxidation for potential subsequent release. The more common of these minerals that occur as efflorescent salts on the surfaces of weathering pyrite include melanterite, rozenite, szomolnokite, romerite, and copiapite ($FeSO_4 \cdot 7H_2O$, $FeSO_4 \cdot 4H_2O$, $FeSO_4 \cdot H_2O$, $Fe^{2+}Fe^{3+}{}_2(SO_4)_4 \cdot 14H_2O$, and $Fe^{2+}Fe^{3+}{}_4(SO_4)_6(OH)_2 \cdot 20H_2O$ respectively; Alpers et al. 1994). These salts are highly soluble and provide an instantaneous source of acidity upon dissolution

(Nordstrom 1982, Cravotta 1994). For example, dissolution of one mole of melanterite contributes two moles of acid (reaction 3). Similarly, Cravotta (1994) reported that dissolution of one mole of romerite produces six moles of acid. Reactions such as this may be largely responsible for increased acidity loadings during rainstorm events.

The formation of hydrated iron sulfates is an important intermediate step that precedes the precipitation of slightly soluble iron minerals such as jarosite (Nordstrom 1982). Jarosite is slightly soluble (Alpers et al. 1994) and yields 1.5 moles of H^+ per mole dissolved (reaction 4). In studies conducted at the U.S. Bureau of Mines, Salt Lake City Research Center (USBM SLRC), the pH of deionized water dropped from 6 to 3 or 4 after contact with synthetic and natural jarosites (White et al. 1997a). Because of jarosite's relatively low solubility, the acid contributed by its dissolution is small relative to that by dissolution of more soluble hydrated iron sulfates, although hydronium jarosite can contribute acid at a more rapid rate (Lapakko and Berndt 2003). It should be noted that sulfate minerals such as anhydrite ($CaSO_4$, reaction 5) or barite ($BaSO_4$) will not produce acid.

$$FeSO_4 \bullet 7H_2O + (1/4)O_2(g) = FeOOH(s) + 2H^+(aq) + SO_4^{2-}(aq) + (11/2)H_2O \quad (3)$$

$$KFe_3(SO_4)_2(OH)_6(s) = 3FeOOH(s) + 3H^+(aq) + 2SO_4^{2-}(aq) + K^+(aq) \quad (4)$$

$$CaSO_4(s) = Ca^{2+}(aq) + SO_4^{2-}(aq) \quad (5)$$

2. Acid Neutralization

2.1. Calcium and Magnesium Carbonate Mineral Dissolution

The most effective minerals for neutralizing acid are calcium carbonate and magnesium carbonate (reactions 6-9). Reactions 6 and 8 are dominant above approximately pH 6.3, while reactions 7 and 9 are dominant below this pH.

$$CaCO_3(s) + H^+(aq) = Ca^{2+}(aq) + HCO_3^-(aq) \quad (6)$$

$$CaCO_3(s) + 2H^+(aq) = Ca^{2+}(aq) + H_2CO_3(aq) \quad (7)$$

$$MgCO_3(s) + H^+(aq) = Mg^{2+}(aq) + HCO_3^-(aq) \quad (8)$$

$$MgCO_3(s) + 2H^+(aq) = Mg^{2+}(aq) + H_2CO_3(aq) \quad (9)$$

Iron carbonates will provide no net neutralization of acid. The initial dissolution of one mole of iron carbonate will consume one or two moles of acid (reactions 10, 11). However, under environmental conditions the one mole of ferrous iron released will oxidize to ferric iron (reaction 12), which will precipitate as ferric oxyhydroxide (reaction 13). The oxidation of ferrous iron is slower than the subsequent ferric oxyhydroxide precipitation, and is reported to be second-order with respect to hydroxide ion concentration (Sung and Morgan 1980; Eary and

Schramke 1990). The oxidation and precipitation reactions will yield two moles of acid (reaction 14). Thus, iron carbonate will not contribute to acid neutralization.

$$FeCO_3(s) + H^+(aq) = Fe^{2+}(aq) + HCO_3^-(aq) \quad (10)$$

$$FeCO_3(s) + 2H^+(aq) = Fe^{2+}(aq) + H_2CO_3(aq) \quad (11)$$

$$Fe^{2+}(aq) + (1/4)O_2(g) + H^+(aq) = Fe^{3+}(aq) + (1/2)H_2O \quad (12)$$

$$Fe^{3+}(aq) + 2H_2O = FeOOH(s) + 3H^+(aq) \quad (13)$$

$$Fe^{2+}(aq) + (1/4)O_2(g) + (3/2)H_2O = FeOOH(s) + 2H^+(aq) \quad (14)$$

2.2. Silicate Mineral Dissolution

Other minerals, such as anorthite and forsterite, can also consume acid but their dissolution rate (and associated rate of acid neutralization) is typically slow in the neutral pH range (reactions 15, Holdren and Berner 1979; and reaction 16, Hem 1970). These minerals dissolve more rapidly as pH decreases, and therefore, provide more buffering under acidic conditions.

$$CaAl_2Si_2O_8(s) + 2H^+(aq) + H_2O = Ca^{2+}(aq) + Al_2Si_2O_5(OH)_4(s) \quad (15)$$

$$Mg_2SiO_4(s) + 4H^+(aq) = 2Mg^{2+}(aq) + H_4SiO_4(aq) \quad (16)$$

3. Trace Metal Sulfide Oxidation

The oxidation of trace metal sulfide minerals releases trace metals and sulfate but does not necessarily contribute acid (reaction 17). Reactions subsequent to sulfide mineral oxidation affect

$$CuS(s) + 2O_2(g) = Cu^{2+}(aq) + SO_4^{2-}(aq) \quad (17)$$

the net trace metal release to the environment. The transport of a given component in the environment is the net result of release to solution by dissolution and removal from solution by precipitation, coprecipitation, exchange reactions, and adsorption. The degree of trace metal transport is dependent upon drainage composition (particularly pH), the chemistry of the released component, and the chemical character and surface area of solid surfaces present. Concentrations of trace metals tend to increase exponentially as solution pH decreases, therefore acidic drainages often contain elevated trace metal concentrations. However, circumneutral drainages can contain elevated concentrations of trace metals such as nickel (Eger and Lapakko 1985) and molybdenum (Brown 1989) that, compared to other trace metals, are relatively soluble in this pH range.

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Attachment 1. Dunka NPDES permit monitoring requirements.

Seeps WS001, WS003, WS004, WS005 monitored twice monthly, prior to treatment, for:

Total Co
Total Cu
Total Ni
Total Zn
pH
Total Hg (twice annually and at WS001 only)
Flow

Discharges SD005, SD006, SD007, SD008, SD009 monitored twice monthly (or monthly as noted), after treatment, for:

Total Cu
Total Ni
Total Zn
Hardness
pH
Toxicity (Additive Acute Toxicity for Cu, Ni and Zn)
Total Hg (twice annually and at SD005, SD007, SD009 only)
Sulfate (monthly)
TSS (monthly)
Dissolved Fe (monthly)
Flow

Key to Outfalls/Seeps

WS001, SD007 = Seep EM-8
WS003, SD008 = Seep 1
WS004, SD009 = Seep X
WS005, SD005 = Seep W-1d
SD006 = Seep W-2/3d

Attachment 2

Hoffmann M, Eisenreich S (1979) Kinetics and Mechanisms of the Oxidative-Dissolution of Metal Sulfide Minerals Found in Duluth-Gabbro Ore. Report for Regional Copper-Nickel Study.

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KINETICS AND MECHANISMS OF THE OXIDATIVE-DISSOLUTION OF METAL SULFIDE MINERALS FOUND IN DULUTH - GABBRO ORE

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March 1979

footnotes ?

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Chapter 1

Introduction

1.1. Composition of the Duluth-Gabbro Ore

The Duluth Gabbro Complex in northeastern Minnesota is a large body of mafic and anorthositic plutonic rocks of Late Precambian age. Primary mineral components in the feldspar-rich gabbro are plagioclase, olivine and pyroxene. Secondary minerals such as chalcopyrite, cubanite, pentlandite and pyrrhotite have potential economic and environmental significance. The chemical and mineral phase analyses of representative samples of operationally defined unmineralized and mineralized gabbro material, which were used in this study, are presented in Tables 1.1 and 1.1. Plagioclase ($NaAlSi_3O_8/CaAl_2Si_2O_8$) is the major feldspar mineral, chalcopyrite ($CuFeS_2$) the major copper mineral and pentlandite, the major nickel mineral. In general, gabbro ore deposits contain approximately 0.5 to 1.5 weight percent of combined copper and nickel with an average Cu to Ni ratio of 3 to 1.

1.2. Environmental Significance

Field studies (1) near an open-pit iron-ore mine covered by Duluth Gabbro overburden show that heavy metals and sulfate are released in significant concentrations from various seepage sites to a small stream which flows into a larger aquatic ecosystem. Potentially toxic trace metals such as Cu, Ni, Co, Cd and Zn are apparently released in aqueous solution from exposed Duluth-Gabbro ore even though the seepage effluent is buffered naturally to approximately pH 7. Leaching of mineral mining solids such as lean ore, waste rock, tailings and displaced overburden can occur through a combination of chemical, physical and biological processes. Historically, environmental concern with leaching of metal sulfides has been focused on acid mine drainage which results directly from the oxidation of iron pyrite. In general, the oxidative-dissolution of metal

sulfides depends on the availability of water, oxygen, acidity, exposed surface area, bacteria, iron content and the acid neutralizing capacity of the host mineral matrix. Release of the above metals to aquatic systems may be detrimental to the aquatic life and indirectly through the process of bioaccumulation to human health. Leachate from Duluth Gabbro mining solids is atypical because the pH and the alkalinity are unusually high. With high alkalinity and pH, in the absence of strong complexing agents, the concentrations of trace metals should be controlled at relatively low levels by their respective hydroxide and carbonate solids. However, due to the solubilizing impact of humic/fulvic material (2) present in the waters of northeastern Minnesota these metals are found primarily in the soluble phase.

1.3. Research Objectives

The primary objective of this research project was to study the detailed kinetics of the oxidative dissolution of metal sulfides and silicate minerals present in mining solids derived from the Duluth-Gabbro Complex. From a study of rate of leaching of various metals and sulfur compounds as a function of a variety of environmental parameters such a pH, T, E_h, D_p (particle size) and catalyst concentrations and empirical rate law and a mechanism consistent with this rate law can be derived. From a knowledge of the rates of oxidative-dissolution and the primary kinetic factors affecting these rates, the relative impact of the exposure of Duluth Gabbro to the aquatic environment can be predicted. An understanding of the detailed mechanism of oxidative-dissolution and subsequent trace metal release should lead to the design of optimal control procedures to eliminate, inhibit or minimize environmental degradation.

1.4. Relevant Research

Little direct information on the oxidative dissolution of sulfidic minerals is available in the primary chemical literature. More information is available

on the microbial leaching of metal sulfides as a beneficiation technique (3). However, the majority of relevant information is available in the hydrometallurgical literature (4,5,6). For an extensive review of the literature on metal sulfide leaching the reader is referred to an earlier paper by the principal authors (7).

Because the environmental conditions found in the mine drainage from Cu-Ni deposits, the role of microbial catalysis in the leaching of sulfidic minerals appears to be minimal. The two principal strains of sulfur metabolizing bacteria, Thiobacillus ferroxidans and Thiobaccillus thiooxidans, which are critical for microbial leaching, are acidophillic, chemolithotropic bacteria with pH optima near 2. Above pH 4, the primary mode of leaching is due to chemical oxidation by Fe^{+3} and O_2 (8).

Under typical environmental conditions of E_h (electrochemical potential) and pH the oxidation of metal sulfides to sulfate and elemental sulfur with the concomitant release of the principal metal is favored thermodynamically. Eisenreich et al., (7) have generated an equilibrium model for leaching in wasterock piles of Duluth Gabbro ore and subsequent speciation of trace metals released to aquatic systems. The model qualitatively predicts that metal sulfides will by oxidized by O_2 to sulfate in an aqueous environment. Subsequent mobility of the trace metals is predicted to depend highly on the nature and concentration of organic and inorganic liqands and on the nature and availability of adsorbing surfaces. The acidity which is generated upon sulfide oxidation appears to be insufficient to overcome the natural buffering capacity of silicate minerals, such as plagioclase, which undergo a unique series of weathering reactions. Examples of these reactions are given in Table 1.3.

Peters (9) has developed an extensive E_h-pH diagram for chalcopyrite, $CuFeS_2$, which is the predominant copper sulfide mineral in the Duluth-Gabbro complex. In a slightly acidic, oxidizing environment $CuFeS_2$ will be unstable with respect

to bornite, pyrite and H_2S. Further increases in oxidation potential result in the decomposition of bornite to chalcocite. Kinetically, $CuFeS_2$ is preferentially transformed to CuS, covellite, and FeS, troilite. These sulfides are oxidized in turn by Fe^{+3} or O_2 to Cu^{+2}, Fe^{+3}, SO_4^{-2} and S^o.

Metal sulfides such as ferrous sulfide (machinawite and pyrrhotite) and nickel sulfides (millerite and pentlandite) readily dissolve under oxic or anoxic conditions. Locker and deBruyn (10) have studied the anoxic dissolution of synthetic ZnS and CdS in acidic solution which dissolve according to the following stoichiometric equation:

$$CdS + 2H^+ \underset{k_{-1}}{\overset{k_1}{\rightleftarrows}} Cd^{+2} + H_2S(g). \qquad (1.1)$$

They showed that in a well-agitated system that the reaction rate is controlled by a heterogeneous surface reaction rather than mass transport in solution; and that in non-oxidative dissolution that the reaction products do not form a passive film that inhibits the dissolution process. The empirical rate expression obtained by these investigators is as follows:

$$(1/A)\ (d\,N_{H_2S}/\,dt) = k_1\,[H_2SO_4] - k_{-1}\,[Cd^{+2}]^{0.5}\ {}_{H_2S}^{\ 0.5}. \qquad (1.2)$$

where the dissolution rate (moles/m^2-sec) is a function of the surface area (A in M^2) of the solid/liquid interface and the concentration of acid in the system. They developed a model and provide experimental evidence to show that a slow adsorption of a proton on the solid surface is the rate determining step in the reaction. Their results clearly show that the rate of non-oxidative dissolution depends on both the properties of the solid and the solution.

Pankow (11) has studied the non-oxidative dissolution of FeS (mackinawite) over the pH range 3-7. The flux, F_s (moles/cm^2 m) from the surface of pressed FeS pellets obeyed the following empirically determined rate law:

$$F_s = \frac{V}{A}\frac{dS}{dt}t = k_1 \gamma\,[H^+] + k_2 \qquad (1.3)$$

where k_1 and k_2 are rate constants with values of 0.22

and 1.9 moles/cm^2-min, respectively, at 25°C. The k_2 term would predominate in the pH range of natural waters. Mechanistically, the k_1 term arises from direct adsorption of H^+ on the FeS crystal surface and the k_2 term arises from thermal vibrations and solvation effects which result in the dissolution of lattice constituents.

The role of oxygen in the oxidative-dissolution of FeS is apparent in the empirical rate law determined by Nelson (12).

$$-d\,[FeS]/dt = k\,A_s\,P_{O_2}\,[H^+]^{0.25}. \qquad (1.4)$$

The rate of dissolution of FeS has a first-order dependence on dissolved oxygen concentration and the total available surface area; however, unlike the anoxic dissolution, the oxidative dissolution exhibits a fractional dependence on the hydrogen ion concentration. The reaction is remarkably sensitive to catalysis by Ni(II) in concentrations in excess of 10^{-5} M. At pH 7, the rate of dissolution is accelerated ten-fold when the Ni concentration is 10^{-4} M. Other trace metals such as Cu(II), Ag(I) and Cd(II) exhibited no catalytic or inhibitory activity. Below pH 9 the major sulfur oxidation product was elemental sulfur with significant contributions from thiosulfate and sulfate. The exclusive iron oxidation product was γ-FeOOH (lepidocrocite). In the presence of high concentrations of Cl^- elemental sulfur was the exclusive sulfide oxidation product.

Under the conditions of pH7, $[O_2]$ = 4 mg/l, $[FeS]_0$ = 800 mg/l and rapid stirring the observed oxidative dissolution rate was 10^{-4} M^{-1}Min^{-1}. This is a significantly greater rate of oxidative-dissolution than would be predicted if the rate of either the homogeneous oxidation of Fe(II) (13) or HS^- (14) at saturation values was the overall rate-determining processes. Using the rate law of Stumm and Lee (13) under the above conditions Fe(II) would be oxidized at a rate of 10^{-6} Mmin^{-1}. Similarly, the rate law of Chen and Morris (14) predicts that HS^- would be oxidized at a rate of 10^{-10} M min^{-1}. Both of these calculated rates are below 10^{-4} M min^{-1}. Therefore, it was concluded that the

oxidative dissolution involves direct attack of O_2 at the FeS surface. Furthermore, considering the results of Pankow (11) and Nelson (12) jointly, it can be concluded that kinetically FeS will preferentially dissolve via an oxidative pathway although both mechanisms should be operative simultaneously under environmental conditions.

Oxidation of the three major sulfidic minerals of economic significance in the Duluth-Gabbro complex, chlacopyrite ($CuFeS_2$).

$$CuFeS_2 + 4Fe^{+3} \rightarrow Cu^{+2} + 5Fe^{+2} + 2S^0 \quad (1.5)$$

$$CuFeS_2 \rightarrow CuS + FeS \quad (1.6)$$

$$4FeS + 9O_2 + 4H^+ \rightarrow 4Fe^{+3} + 4SO_4^{-2} + 2H_2O \quad (1.7)$$

$$CuS + 2Fe^{+3} \rightarrow Cu^{+2} + S^0 + 2Fe^{+2} \quad (1.8)$$

pentlandite {$(Fe, Ni)_9S_8$},

$$18\ Fe^{+3} + (Fe, Ni)_9S_8 \rightarrow 22.5Fe^{+2} + 4.5Ni^{+2} + 8S^0 \quad (1.9)$$

$$(Fe, Ni)_9S_8 + 17.63O_2 + 6.5H^+ \rightarrow 4.5Fe^{+3} + 4.5Ni^{+2} + 8SO_4^{=} + 3.25\ H_2O \quad (1.10)$$

and cubanite ($CuFe_2S_3$)

$$CuFe_2S_3 + 6Fe^{+3} \rightarrow Cu^{+2} + 8Fe^{+2} + 3S^0 \quad (1.11)$$

$$CuFe_2S_3 + 6.5O_2 + 2H^+ \rightarrow Cu^{+2} + 2Fe^{+3}\ 3\ SO_4^{=} + H_2O \quad (1.12)$$

has been studied with the potential for mineral extraction as a primary goal; therefore, detailed kinetic and mechanistic information is not available, although some of the experimental results and general observations are pertinent to this

study. Kinetically, $CuFeS_2$ is the copper sulfide most resistant to oxidative dissolution (4). Madsen et al., (15) studied the large-scale (7 ton), long-term (500 days) leaching by Fe^{+3} of low-grade copper sulfide ores which contained predominately chalcopyrite imbeded in feldspars (monzonite and quartz monzonite). These investigators observed parabolic release kinetics in which the extent of leaching was a direct function of total exposed surface area. They used a mixed kinetic model, in which dissolution is controlled partially by diffusion through a reacted outershell of the host rock and partially by a chemical reaction at the mineral surface, to mathematically predict the observed leaching rates. Release of Cu is initially controlled by a surface reaction, however, as the reaction proceeds a layer of reaction products builds up on the surface and diffusion of the oxidant through this reacted zone becomes rate controlling.

Wadsworth (4) reports that the oxidation of $CuFeS_2$ by O_2 in acidic solution followed a mixed kinetic rate law involving the sum of a parabolic and linear term

$$\frac{\Delta n^2}{k_p} + \frac{\Delta n}{k_1} = t \quad . \qquad (1.13)$$

where Δn represents the amount of copper dissolved in solution in time t, and k_1 and k_p are respectively the parabolic and linear rate constants. The parabolic term describes the diffusion of O_2 through a deposited film of product sulfur and the linear term describes a surface reaction which doesn't involve charge transfer.

Dutrizac, MacDonald and Ingraham (16) followed the rate of dissolution of synthetic cubanite by Fe^{+3} oxidation at temperatures greater than 45°C. The dissolution rate was linear with an activation energy of 12 k cal/mole. At constant temperature, $k_1 = \gamma\ [Fe]^{0.6}$ and $[H^+]^{-0.1}$. From this evidence, it was concluded that the rate of dissolution was controlled by a chemical reation at the

sulfide surface. Chloride ion appeared to have a catalytic effect on the rate of dissolution.

Pentlandite {$(Ni, Fe)_9S_8$} ores are readily leached in column reactors by trickling solutions of ferric sulfate (17). Linear release rates were observed over a significant portion of the reaction where $k_1 = \gamma [Fe^{+3}]^{0.2} [H_2SO_4]^{-0.1}$. At low flow rates the dissolution of pentlandite appears to be diffusion controlled whereas at higher flow rates the reaction appears to be chemically controlled. Formation of elemental suflfur and jarosite ($HFe_3(SO_4)_2(OH)_6$) where observed during the course of the reaction.

Other studies relevant to the potential leaching of Cu and Ni from Duluth Gabbro ore pertain to firstly, the relationship between the solubility product and the rate of metal sulfide oxidation, secondly, the role of electrochemical interactions between different metal sulfides in influencing the rate of metal leaching and thirdly, the role of organic chelating agents. Torma and Sakaguchi (37) have shown that the rates of biooxidation of a series of synthetic metal sulfides is directly proportional to their solubility products for a given surface area. The sulfide with the highest solubility product (i.e. NiS) is oxidized most rapidily in the following order: NiS>CoS>ZnS>CdS>CuS.

Sato (18) has observed that one metal may enhance the dissolution rate of another sulfide through electrochemical interaction. Specifically, for two sulfides of the same metal, the one with the higher formal oxidation state is more stable than one with the lower formal oxidation state. This appears to be consistent with the relatively high resistance of chalcopyrite to chemically induced dissolution in mixed mineral systems. Bryner and Anderson (19) reported that the presence of FeS_2 accelerated the rate of Cu leaching from $CuFeS_2$. FeS_2, which is significantly more electropositive than chalcopyrite, should induce dissolution of $CuFeS_2$ by galvanic interactions. However, Ichikuni (20)

reports that no galvanic effect was observed on the dissolution rate of $CuFeS_2$ in the presence of Fe^{+3}.

Sato (18) has proposed that the rate determining step in the dissolution of metal sulfides involves an election transfer with the formation of elemental sulfur.

$$MeS \rightleftarrows Me^{+2} + S^{o} + 2\bar{e} \qquad (1.14)$$

The experimental potential generated by divalent metal-monosulfides measured against a reference electrode were best predicted by the half-cell reaction in equation 1.13. When Fe^{+3} is the election acceptor equations 1.13 and 1.14

$$2\bar{e} + 2Fe^{+3} \rightleftarrows 2Fe^{+2} \qquad (1.15)$$

can be combined to give the overall equation for oxidation.

$$MeS + 2Fe^{+3} \rightleftarrows Me^{+2} + 2Fe^{+2} + S^{o} \qquad (1.16)$$

Nelson (12) reports that leaching experiments conducted on ZnS, PbS, CuS and FeS_2 separately and in various combinations demonstrated that electrolytic action does substantially increase the extent and rate of oxidation of the sulfide with the lower emf, while offering considerable cathodic protection to the sulfide with the higher half-cell potential.

Chelation by naturally occurring organic macromolecules has been implicated or suggested by a number of investigators as an important pathway of weathering or solubilization of a wide variety of minerals (21, 22, 23, 24). Soil humic acids, which are heteropoly condensates with structurally indeterminate arrays of functional groups such as carboxyl, carbonyl, phenolic, amino and hydroxyl groups on the periphery of the molecule, have been shown to be effective reagents for the dissolution of a variety of metal sulfides (21). Humic acids isolated from soil exhibited a dramatic effect on the leaching of copper sulfides. Susceptibility to dissolution by humic acids was correlated with the relative bond strengths of the sulfide minerals. These minerals were susceptible to humic acid

dissolution in the following order: $Cu_2>S>CuS>PbS>Cu_5>FeS_4>CuFeS_2>ZnS$. Leaching was also accelerated by a decrease in average particle size.

Schalscha et al., (22) studied the role of the model organic acids such as salicylic, citrate, tartaric, and acetic acid on the extent of dissolution of a variety of ferrungnious oxides and silicates. The extent of leaching appears to be a function of the nature of the organic acid in terms of its complexing strength and the particle size of the oxide or silicate mineral. Gruner (25) demonstrated that water drawn through a peat bed from northeastern Minnesota attacked silicates, iron oxides and siderite but had little effect on pyrite.

From these studies, it can be concluded the concentration and nature of dissolved organic carbon in the waters of the Duluth Gabbro complex will be critical factors in the rate and extent of leaching. Field studies (1) show that dissolved organic carbon levels in groundwater near the Dunka pit range from 20 to 40 mg/l. Humic material released from peat beds surrounding the mining region may be responsible to a large extent for the observed rates of leaching in the field.

1.5. Weathering/Dissolution of Aluminum Silicates

Trace elements of interest in this study such as Cu, Ni, Co and Zn are present primarily as sulfides in Duluth Gabbro. The metal sulfides are disseminated as small particles in the interstices between silicate minerals (plagioclase, olivine, pyroxene) or intimately intergrown with the silicates. As a result, silicate mineral dissolution may play an important role in the release to solution of trace metals bound as sulfides. Tables 1.2 and A.8 list a detailed mineralogical analysis of the mineralized and unmineralized gabbro rock which was leached in this study. The primary minerals present in gabbro rock leached are in order of decreasing abundance (weighted mean %): plagioclase (47-64%) >clinopyroxene (7.6-26%) >olivine (11-14%) >biotite (3.3-5%). Plagioclase minerals have the general stoichiometry of (Na, Ca)

$AlSi_3O_8$ and represent a continuous solid solution feldspar series with end members albite ($NaAlSi_3O_8$) to anorthite ($CaAlSi_3O_8$). Clinoproxene and olivine have the general stoichiometry of $Ca(Fe, Mg)Si_2O_6$ and $(Fe, Mg)_2 SiO_4$, respectively. Since greater than 90% of the gabbro mass consists of silicate minerals, and the majority of silicate mineral comprised of plagioclase (a feldspar series), a discussion of the weathering/dissolution of feldspars is included.

The incongruent dissolution of aluminum silicates (weathering) can be represented by (26).

$$\text{cation - Al-silicate} + H_2CO_3^* + H_2O \rightleftarrows HCO_3 + H_4SiO_4 + \text{cation} + \text{Al-silicate(s)} \quad (1.17)$$

In this process, a primary silicate mineral is degraded into a secondary mineral which is often ill-defined structurally or X-ray amorphous. Aluminum is generally conserved in the reaction as it forms insoluble silicate minerals such as kaolinite ($Al_4[Si_4O_{10}](OH)_8$ or halloysite ($Al_4[Si_4O_{10}](OH)_8 2H_2O$), amorphous aluminum oxides ($AlC(OH)_3 nH_2O$) or microcrystalline Al oxides such as gibbsite ($Al_2O_3 \cdot 3H_2O$). The weathering of Al silicates imparts alkalinity (HCO_3^-) to the water, and therefore the resulting degraded Al silicate is more acidic than the primary mineral. Silicic acid (H_4SiO_4) and mineral cations are released to solution, with aqueous control mechanisms being different for both. The appearance of cations (Ca^{+2},Mg^{+2},Na^{+1},K^{+1}) and/or silicic acid to solution is offered as evidence of silicate mineral weathering, and is usually monitored in laboratory kinetic studies.

An example of the incongruent dissolution of a typical plagioclase (andesine) to give the degraded silicate kaolinite, can be represented by (26):

$$4\ Na_{0.5}Ca_{0.5}Al_{1.5}Si_{2.5}O_8 + 6\ H_2CO_3^* + 11\ H_2O \rightleftarrows 2Na^+ + 2Ca^{+2} + 4\ H_4SiO_4 + 6\ HCO_3^- + 3\ Al_2Si_2O_5(OH)_4(S) \quad (1.18)$$

In this reaction the rate of release of the cations Na^+ and Ca^{+2} and silicic acid can be used to determine the kinetics and possibly the mechanism of plagioclase dissolution.

In addition to the weathering of Al silicate minerals, incongruent dissolution can occur through the slower chemical process of hydrolysis. For example, the hydrolysis of anorthite, an end member of the plagioclase series, can be compared to the weathering of anorthite as represented below.

Hydrolysis

$$CaAl_2Si_2O_6(s) + 3H_2O \rightleftharpoons Ca^{+2} + 2OH^- + Al_2Si_2O_5(OH)_4(s) \qquad (1.19)$$

Weathering

$$CaAl_2Si_2O_6(s) + 2H_2CO_3 + H_2O \rightleftharpoons Ca^{+2} + 2HCO_3^- + Al_2S_2O_5(OH)_4 \qquad (1.20)$$

The hydrolysis of anorthite leads to the release of Ca^{+2} and OH^-, and formation of the secondary mineral, kaolinite, but no silicic acid is released. In contrast, the weathering of anorthite releases, in addition to the above, HCO_3^- and silicic acid. The rates of incongruent dissolution are slow for both reactions, but weathering processes exert a greater ultimate influence on the chemical composition of the aqueous phase.

The mechanism of aluminum silicate dissolution and the process exerting control on aqueous phase solubility on the reaction products have been studied under environmental conditions by Garrels and Howard (27), Wollast (28), Helgesen (29, 30), Luce et. al., (31), Paces (32), Petrovic (33), Busenberg and Clemency (34) Dayal (35) and Busenberg (36). Garrels and Howard (27) found that the initial interaction of feldspar with the aqueous phase consisted of a reversible ion-exchange reaction between H^+ from the aqueous phase and surface cations resulting in the release of a stoichiometric quantity of cation to solution, and an increase in solution pH. This was followed by the incongruent dissolution

of the feldspar surface (27, 34) leaving behind a slightly soluble product layer. Wollast (28), Helgesen (29) and others have modeled the diffusional transport of silicic acid and cations through the building product layer.

Busenberg and Clemency (34) studied the dissolution of kinetics of eight different feldspars at 25°C and 1 atm of CO_2 partial pressure for time periods of 400-1200 hours. The incongruent dissolution of the silicate minerals could be adequately described by a four-stage process. The first stage involved an ion-exchange reaction whereby H^+ were exchanged with surface cations resulting in an initial rise in pH. Negligible amounts of H_4SiO_4 or Al were released in this stage lasting ~ 1 minute. The second stage lasting to ~ 50 hours for cations and 1.5-100 hours for silicic acid could be described by an equation of the form $C=ket^n$ where C=concentration, ke=rate constant, t=time, and n=dimensionless factor. They attributed this complex release behavior to two or more processes occurring simultaneously. The second stage was typified by a rapid release of cations and silicic acid in solution. The diffusion controlled parabolic third stage lasted up to ~ 19-21 days and could be described by an expression of the type

$$C= kpt^{1/2} + A \qquad (1.21)$$

where kp=rate constant and A=integration constant. Parabolic release kinetics were attributed to the rate limiting process of diffusional transport of cations across a thickening product layer. Wollast (28), Busenberg and Clemency (34) and Busenberg (36) suggest that Al diffuses out of the Al silicate matrix in the first and second stages where it hydrolyzes to form an amorphous Al $(OH)_3$. Silicic acid diffusing through the secondary product layer reacts with the $Al(OH)_3$ to form an Al silicate which covers the surface. Diffusion of cations and silicic acid through the new mineral phase explain the parabolic kinetics observed in the third stage. The fourth stage lasting beyond 21 days exhibited

linear kinetics, which can be expressed as

$$C = k_1 t + B \tag{1.22}$$

where k_1 = rate constant, and B = integration constant. Linear kinetics may result when the rate of formation of the secondary product layer equals the rate at which the layer is destroyed by dissolution or fragmentation (32).

Busenberg (36) has studied the products of feldspar dissolution under conditions of 1 atm partial pressure of CO_2 and 25°C. He determined that the activity of Al in the aqueous phase within the initial 4 hours was controlled by pH and the solubility of microcrystalline gibbsite ($Al_2O_3\ 2H_2O$). After 100 hours, the activity of Al and H_4SiO_4 in solution was controlled by microcrystalline halloysite ($Al_y[Si_4O_{10}](OH)_8\ 2H_2O$) a hydrolyzed form of kaolinite ($Al_4[Si_4O_{10}]-(OH)_8$. The dissolution of the secondary product layer can be represented by the equation:

$$Al_2Si_2O_5(OH)_4 \cdot 2H_2O + 6H^+ \rightleftarrows 2Al^{+3} + 3H_2O + 2H_4SiO_4 \tag{1.23}$$

Busenberg and Clemency (34) reported that the parabolic stage rate constants ranged from 3.4×10^{-14} to 9.3×10^{-13} mole/cm^2/sec$^{1/2}$, the linear rate constants were $(.03 - 2.5) \times 10^{-15}$ mole/cm^2/sec, and the apparent diffusion coefficients (parabolic stage) were $(0.1 - 5) \times 10^{-21}$ cm^2/sec. Helgeson (29) determined that the incongruent dissolution of feldspars at 25-200°C exhibited parabolic constants from 10^{-10} to $>10^{-8}$ moles/cm^2 day$^{1/2}$

1.6. Interrelationship of Alumino-Silicate and Metal Sulfide Dissolution

The dissolution of alumino-silicate minerals (plagioclase) can effect the dissolution of base metal sulfides such as chalcopyite and pentlandite by 1) exposing additional metal sulfide surface to solution, 2) controlling solution pH through buffering reactions, 3) contributing anions (HCO_3^-), cations (Ca^{+2}, Mg^{+2}, Na^+, K^+) and silicic acid (H_4SiO_4) to solution which might

accelerate or inhibit metal sulfide dissolution or oxidation, 4) increasing ionic strength which increases the solubility of metal sulfides, and 5) sorbing released metal from metal sulfides onto primary (feldspar) or secondary product layers (amorphous $Al(OH)_3$; microcrystalline gibbsite, kaolinite, halloysite), and 6) forming insoluble metal precipitates such as metal silicates. The dissolution of silicate minerals, in general, increases with increasing acidity, increasing Pe and increasing tendency for complex formation (26).

The dissolution of silicate minerals increases with increasing surface area available to solution. Garrels and Howard (27) noted that K^+ release from a K^+ feldspar increased with increasing solution loading of -200/inch particles. Increases in solid surface area available for dissolution may be achieved by increasing loading of similar size particles or by adding equal masses of different size particles. For particles less an 1 μm in diameter or of specific surface areas greater than~3 m^2/g, particle surface energy may become large enough to alter surface properties. As a result, decreasing particle size may increase the solubility of particles through a thermodynamic influence of surface properties on the solubility product constant.

Solution acidity is a dominant variable in the dissolution of silicate minerals. A dilute aqueous suspension of feldspar results in an increase in pH (28) with the solution buffered at pH 7-9. Busenberg and Clemency (34) observed an increase in the pH of an acidic feldspar solution from 3.92 to 4.51.

Table 1.3 lists a series of equilibrium reactions describing the dissolution of a typical feldspar, anorthite. Reaction i represents the initial ion exchange reaction of a feldspar surface with solution where H^+ replace cations in surface exchangeable sites. Wollast (28) has shown that the initial reaction rate (15 mins. - 2 hrs.) depends on pH according to the expression

$$\text{rate} = k_1(H^+)^{1/3} \qquad (1.24)$$

Under natural conditions, the rate and extent of silicate dissolution increases from pH 8 to pH 4, and can be described by H_4SiO_4 diffusional transport through a residual product layer made up of slightly soluble $Al(OH)_3$ followed by formation of Al-silicates (equation A, Table 1.3). At pH <5, no protective product layer is formed due to continuous dissolution of amorphous $Al(OH)_3$. Therefore, the rate of appearance of silic acid and cations in solution at low pH should not follow parabolic kinetics, and should not be controlled by diffusion. Wollast (28) extrapolated the results of his experiments to the field in that natural conditions of good drainage, and rains of high intensity and frequency, but of short duration, should yield maximum dissolution rates. Reaction rates increase above pH 8 due to dissolution of the slightly soluble $Al(OH)_3$ formed on the feldspar surface. There is not agreement, however, that feldspar dissolution is controlled by diffusional transport from pH 3-10 (33).

Dissolved oxygen exerts an indirect influence on the dissolution of silicate minerals. As the dissolved oxygen (DO) content of the solution increases (i.e., high Pe) there is a greater tendency for oxidation of the metal sulfide minerals such as pyrite, pyrrohotite, chalcopyite and pentlandite. The release of H^+ through the oxidation of ferrous ion and reduced sulfur species will decrease solution pH at the surface water boundary, thereby increasing silicate dissolution. In aqueous suspensions constituted of 95% plagioclase and 5% mineral sulfide by weight, bulk solution pH will be only slightly affected by redox processes.

Organic and inorganic liqands may enhance the dissolution of silicate minerals. Reaction B (Table 1.3) demonstrates the effect of organic complexation with ligand (Y^-) undergoing a ligand exchange reaction with surface bound Al, thus degrading the silicate structure. Available ligand may also bind the released cations maintaining them in a disolved state. Organic chelation has been observed as a factor in enhancing silicate weathering (22). Salicylate,

8-hydroxyquinoline and EDTA increased the extent of biotite dissolution by binding structural components. Plagioclase in intimate contact with ground or surface water containing naturally occurring organic ligands may enhance the dissolution of silicate and metal sulfide minerals. In addition, organic complexation of released metals may prevent their removal by sorption or precipitation.

Reaction x. (Table 1.3) demonstrates the adsorption of trace metals on the silicate surface. The mineral surface is available as the native feldspar or as secondary reaction products such as amorphous $Al(OH)_3$, microcrystalline $Al_2O_3 \cdot 3H_2O$, kaolinite or halloysite. Thus trace metals released to solution through oxidative dissolution of the metal sulfide may be removed by adsorption on the silicate mineral surface. The relative extent and importance of adsorption as a metal removal mechanism depends on pH and total metal concentration.

1.7. Research Direction - Parametric Studies

From the above review it is apparent that a number of factors can control the rate and extent of metal leaching. Some of these factors include the availability of oxygen or other oxidants such as ferric ion, temperature, degree of surface saturation by water, pH, particle size, chemical or microbial catalysis, ionic strength, chelating agents, and galvanic interactions. In the following sections, laboratory research experiments will be described, that were designed to elucidate the relative importance of the above factors in controlling the rate and extent of leaching of heavy metals from Duluth Gabbro mining solids. Parametric studies were undertaken in an attempt to determine an empirical or operational rate law which can be used for predictive purposes and to elucidate a plausible mechanism for leaching consistent with observed kinetics. Kinetic and mechanistic models will be developed in subsequent chapters. Results of this research should prove to be useful in predicting the relative role of chemical leaching of Duluth Gabbro solids (mineralized, unmineralized and tailings) as a source of potential toxic metals to aquatic ecosystems in northeastern

Minnesota. A knowledge of mechanistic pathways should suggest reasonable control procedures to prevent or treat any mine drainage that results from the mining or beneficiation of Duluth Gabbro Ore.

Chemical Composition of Gabbro: Table 1.1
Values in Weight Percent

Element	Unmineralized	Mineralized
S[b]	1.1	3.92
Ni	0.095	0.36
Cu	0.24	1.4
Co[a]	0.082	0.039
Zn[a]	0.0009	0.012
Fe	10.4	17.2
Ca	2.6	2.3
Mg	2.0	1.9

a Neutron Activation Analysis (University of Wisconsin)
b Wet Chemical Analysis
c Atomic Absorption Sepectrophotometry (all other elements)

Mineral Analysis of Gabbro: Table 1.2

Mineral	Formula	Weighted Mean Volume Percent Unmineralized	Mineralized
Pentlandite	$(Fe,Ni)_9S_8$	0.032	0.34
Chalcopyrite-Cubanite	$CuFeS_2$-$CuFe_2S_3$	0.13	1.3
Pyrrhotite	Fe_7S_8-FeS	0.403	3.1
Ilmenite	$FeTiO_3$	4.0	3.1
Magnetite	Fe_3O_4		
Olivine	$(FeMg)_2SiO_4$	14	11
Clinopyroxene	$Ca(Fe,Mg)Si_2O_6$	7.6	26
Orthopyroxene	$(Fe,Mg)_2Si_2O_6$	2.8	2.3
Plagioclase	$NaAlSi_3O_8$-$CaAl_2Si_2O_8$	64	47
		92.97	94.14

Table 1.3
Silicate Dissolution, Anorthite

Initial Ion Exchange

i. $CaAl_2Si_2O_8(s) + 2H^+ = Ca^{2+} + H_2Al_2Si_2O_8(s)$

Dissolution Phase for Anorthite

Inorganic Reaction

ii. $CaAl_2Si_2O_8(s) + 2H^+ + H_2O = Al_2Si_2O_5(OH)_4(s) + Ca^{2+}$; log k = $^+14.4^a$ (Kaolinit

iii. $Al_2Si_2O_5(OH)_4(s) + SH_2O = 2H_4SiO_4 + Al_2O_3 \cdot 3G_2O(s)$; log k = -9.4^a (Gibbsite)

A. $CaAl_2Si_2O_8(s) + 2H^+ + 6H_2O = 2H_4SiO_4 + Al_2O_3 \cdot 3G_2L(s) + Ca^{2+}$; log k = 5.0

Organic Ligand Y^-, effect on kaolinite dissolution

iv. $Al_2Si_2O_5(OH)_4 + 2Y^- = 2AlY(aq) + 2H_4SiO_4 + H_2O^a$

B. $CaAl_2Si_2O_8(s) + 8H^+ + 2Y^-_+ = Ca^{2+} + 2AlY(aq) + 2H_4SiO_4$

Additional Reactions

v. $Al_2O_3 \cdot 3H_2O(s) = 2Al^{3+} + 6OH$ log k = 68.0^a

vi. $Al_2O_3 \cdot 3H_2O(s) + OH^- = Al(OH)_4^-$ log k = -1.0^a

vii. SiO_2 (amorph) $+ H_2O = H_4SiO_4$ log k = -2.7^a

viii. $H_4SiO_4 = SiO(OH)_3^- + H^+$ pk = 9.46^a

ix. $(-SIOH) = (-SiO^-) + H^+$ pk int = $6.8 \pm 0.2^{a,b}$

x. $Me^{ni} + N(-SiOH) = Me(OSi-) + NH^{+c}$
(similar to reaction of Me^{n+} with (OH^-))

a. Reference 26; 25°C, 1atm.
b. 0.1 M $NaClO_4$, Schindler & Kamber (1968) in Stumm & Morgan (26)
c. Dugger (1964) in Stumm & Morgan (26)

Chapter 2

Experimental Procedure

2.1. Sample Preparation

Duluth Gabbro rock was obtained as sample composites from mineralized and unmineralized areas of the Dunka Pit mine. The two samples were ground at the Mineral Resources Research Center at the University of Minnesota. The grinding process consisted of subjecting the rock sequentially to jaw crusheers of 2.54 cm and 1.27 cm capacity and gyro crusher. The sample was then split into ~ 1.8kg portions using a basic splitter and submitted to Bond Ball Mill grinding dry at 70 rpm. The resulting sample was sieved through a Tyler screen shaker for 5 minutes of grinding and 15 minutes of shaking. The minus 200 mesh portion was retained for use in laboratory leaching experiments, and stored in polyethylene containers.

In initial experiments, representative portions of the bulk Gabbro sample were selected by the coning and quartering method. In later experiments, samples to be leached were obtained by removing the desired quantity from the well-mixed bulk material.

2.2 Apparatus

The experimental apparatus used in the batch leaching studies consisted of six, 4-l Pyrex beakers in which the lips had been removed, and the assembly covered by a fitted Plexiglass top. Six air diffusers were connected to a manifold by Tygen tubing with a diffuser inserted into each beaker through the protective cover, and sealed with an aluminum-foil covered rubber stopper. The gabbro suspension in the reaction flask was mixed continuously with a laboratory stirrer consisting of a teflon shaft and blade protruding through the top of the Plexiglass cover. Samples were withdrawn by pipette through a stoppered sampling port. All reactors were maintained in a constant temperature room at $20.0 \pm 1^{o}C$.

The beakers, mixing shafts and gas diffusers were prepared for each experiment by washing with 3 portions of distilled water, 3 portions of 0.1 N HNO_3 and 3 additional portions of distilled water. Distilled water (3500 ml) was measured into each reaction flask and the air bubbled the solution for 24 hours prior to each experiment, except when disolved oxygen was being varied.

2.3 Procedure

Gabbro samples (-200 mesh) were weighed on a Sartorius Analytical Balance and poured into the reaction flasks, initiating the experiment. Samples were withdrawn at periods exceeding 1000 hours at the time intervals of 0.5, 6, 24 and 48 hours and 1, 2, 3, 4 and 5 weeks. The 150 ml sample was withdrawn through the sampling port from a point in the reaction flask two thirds depth of the liquid. With the exception of the particle size experiment, no water was replaced to the reaction flask because representative amounts of solid and liquid were taken for analysis. The samples were placed into washed, pre-weighed 250 ml polyethylene bottles, re-weighed and the weight of the sample recorded. The samples were filtered immediately through acid-washed, 0.4 μ Nuclepore filters placed in a Millipore Filtration Apparatus. The filtered sample was acidified to pH1 with concentrated HNO_3.

In some experiments, the quantity of metal precipitated or adsorbed to the leached particle surface was determined. The untreated Gabbro suspensions obtained from the reaction flask were filtered through 0.4μm Nuclepore membrane filters and the particulate-laden filter was re-suspended in 100 mls of 0.01N HNO_3 for 60 seconds. The sample was re-filtered and the filtrate saved for chemical analysis.

The metals studied in the leaching experiments were Cu, Ni, Fe, Ca and Mg, and were determined by Atomic Absorption Spectrophotometry (AAS) in the flame

(Varian Model 175 AA) or flameless mode (PE Model 360 AAS with D_2 background corrector and HGA 2100 graphite furnace). Analyses were performed according to the general instructions of the manufacturer with minor modifications to account for the high salt matrix encountered. Quantification was achieved by comparison to standards or by standard addition. Reproducibility measurements indicated a variation of <10%. Sulfate was determined by the recommended procedure in Standard Methods (37) with modification to smaller sample volumes. Soluble reactive silica determined in some experiments was analyzed for by the method of Strickland and Parsons (38). Spectrophotometric measurements were performed with a Beckman Model 26 UV-VIS Spectrophotometer.

Dissolved oxygen measurements were made with a YSI Model 54 D.O. Meter, with a Model 5739 D.O. probe. Dissolved oxygen of reaction solutions was adjusted using various combinations of O_2, N_2 and compressed air. Solution pH was determined with a Corning Model 10 pH meter and a microelectrode, or an Orion Digital pH meter (Model 701A). pH was maintained in controlled experiments by daily or hourly additions of HCL or NaOH.

Chapter 3

Experimental Results - Metal Sulfide Dissolution Kinetics

3.1 Background

Previous investigations, which were reviewed in Chapter one, on the dissolution kinetics of the slightly soluble metal sulfides have shown that the critical factors are available reactive surface area, pH, dissolved oxygen concentration and the presence of active catalysts. In order to formulate a reasonable kinetic expression which adequately describes the process of oxidative-dissolution of metal sulfides, each important parameter must be varied independently of the other parameters in order to ascertain the relative impact of that particular variable on the dissolution of the metal sulfides.

The study of reaction kinetics in heterogeneous systems is a difficult problem even for simple systems which involve only two phases (i.e. one liquid and one solid phase). In simple homogeneous systems the rate of reaction is most often controlled by formation of an "activated complex" to use the terminology of transition state theory (39). However in hetergeneous systems, mass transport or diffusion limitations to reaction rate become important. In a hetereogeneous system as complex as the Duluth Gabbro ore, which according to Table 1.2 involves at least a minimum of nine solid phases, one liquid phase and one gaseous phase, the problem of resolving the detailed kinetics and mechanism of dissolution is immense if not impossible. In order to simplify the presentation of experimental results the problem of metal sulfide dissolution will be separated from the problem of silicate mineral dissolution, although these two processes are not mutually exclusive. In the discussion of results presented in Chapter 5 the two problems will be interfaced in a unifying manner.

The primary metal sulfides identified by x-ray diffraction analysis were in their relative order of abundance pyrrhotite (FeS), chalcopyrite $(CuFeS_2)$, pentlandite $(Fe,Ni)_9S_8$ and cubanite $(CuFe_2S_3)$. From an environmental and

economic standpoint the rate of appearance of Cu and Ni in solution is of primary importance. X-ray analysis also shows that ore samples which are ground to a finely divided state (i.e., <74μ in particle diameter, Dp) contain distinct fractions of metal sulfide independent of the original silicate matrix. Assuming that the metal sulfides are found in discrete inclusions and are liberated totally from the surrounding silicate (plagioclase) matrix by grinding to a Dp <0.74μ , the problem of metal sulfide dissolution can be separated from the problem of silicate dissolution (weathering) as a fist approximation. Silicate dissolution kinetics are presented separately in Chapter four.

3.2. Preliminary Experiments and Kinetic Analysis

Well-mixed batch reactors described in section 2.2 were used for all kinetic runs. Preliminary experiments were run with powdered Gabbro material with a Dp <74μ suspended in doubly-distilled water by mechanical stirring. The goal of these initial experiments was to determine optimal conditions for following the rate of release of Cu, Ni and Fe into aqueous solution. During each kinetic run aliquots of the solid-liquid slurry were extracted while the system was well-mixed. Ideally the solid to liquid ratio was maintained constant during each experiment. A liquots were taken at pre-established time intervals during the course of each experiment. Total time of reaction was generally around 1000 hours.

Initial experiments in unbuffered, pH uncontrolled systems were designed to determine the impact of increasing solids to volume ratio in a search for optimal experimental conditions for subsequent studies. The effect of increasing total specific surface area or loading on the relase of Cu, Ni and Fe was studied at mass loadings of 2, 10, 50 and 100 grams of Dp <74μ of unmineralized Gabbro material. Increasing the Gabbro loading at constant volume increases the total specific surface area of sulfide minerals for reaction with oxygen,

on one hand, but on the other hand, an increase in total specific surface area of the silicate minerals, which have charged surface silanol groups, increases the possibility of adsorptive interactions with charged aquated and hydrolyzed metals in solution.

The overall impact of increased loading on total metal release is illustrated in Figure 3.1. As the loading increases the pH of the slurry rises in response to the incongruent dissolution of plagioclase which releases HCO_3^- to solution according to equation 3.1

$$4\ Na_{0.5}Ca_{0.5}Al_{1.5}Si_{2.5}O_8 + 6H_2CO_3^* + 11H_2O = 2Na^+ + 2Ca^{+2} + 4H_4SiO_4 + 6HCO_3^- + 3Al_2SiO_5(OH)_4(s) \quad (3.1)$$

This effect is shown more clearly in Figure 3.2. The release of bicarbonate to solution accounts for the unique buffering capacity exhibited by the Gabbro material in equilibrium with water. After 2,800 hours the Gabbro H_2O air system can be assumed to have reached equilibrium with respect to pH. This assumption is verified by the results presented in Figure 3.3 which show that the pH stablizes at a constant value after approximately 500 hours for different Gabbro loadings. The buffering capacity of the Gabbro ore appears to be a direct function of the total specific surface area and the total bicarbonate released to the system. For example, for a reactor system dominated by bicarbonate buffer exposed to the atmosphere equations 3.2 and 3.3 can be combined to show the approximate dependence of the sytem pH on the bicarbonate concentration if all other equilibrium are ignored.

$$H_2CO_3^* = K_H PCO_2 = 10^{-5} M \quad (3.2)$$

$$[H^+][HCO_3] = K_1\ [H_2CO_3^*] = 10^{-6.3} \times 10^{-5} \quad (3.3)$$

$$pH = 11.3 + \log [HCO_3^-] \quad (3.4)$$

If the $[HCO_3^-]$ in solution at equilibrium is 30 mg/l (log HCO_3^- = -3.3) the pH would be 8.0. From the results shown in Figure 3.2, it can be seen that the

equilibrium pH approaches 8.0 given sufficient time and high Gabbro loadings.

Referring to Figure 3.1 again, the greatest total release, which is the sum of the solution phase metal and the adsorbed phase metal, for Ni and Fe occurs at relatively low loading rates and consequently relatively low pH. Copper, on the other hand, shows a greater total release at higher pH although the majority of released metal was found in the adsorbed phase. In general as the pH rises the released metals are found predominately in the adsorbed phase. This is consistent with the known surface characteristics of α-quartz and other silicate minerals in aqueous solution.

In general, at lower pH FeS and $(Ni,Fe)_9S_8$ would be expected to dissolve more readily independent of any surface oxidation. However, $CuFeS_2$ dissolution is relatively insensitive to increasing acidity while surface oxidation plays a more important role. It is well-known that oxidation of HS^- increases with increasing pH (14, 40) and by analogy the oxidation of $CuFeS_2$ may also increase with increasing pH. A summary of the effect of loading on metal release is present in Table 3.1.

Using raw data presented in Table A. 1 (Appendix A) concentration versus time profiles on Cu and Ni can be constructed as shown in Figure 3.4. At mass loadings ≤ 10 g l^{-1}, the concentration of Ni as a function of time appears to be linear. This linear or apparent zero-order rate of dissolution of Ni was observed consistently over a wide variety of conditions. The rate of appearance of Cu with time was more complex and showed a greater sensitivity to pH. In order to observe Cu release rates, it was necessary to keep to solution pH below 6; whereas Ni release rates were less sensitive to pH and maintained apparent linearity. Initial release of Cu at lower pH could be described as parabolic with a later linear stage of release. The fate of Cu in solution is more sensitive to pH because of the formation of $Cu(OH)_2$ and $CuCO_3(s)$ at pH 6 and electrostatic and chemical interactions with surface $-SiOH^-$ groups. For this reason, it

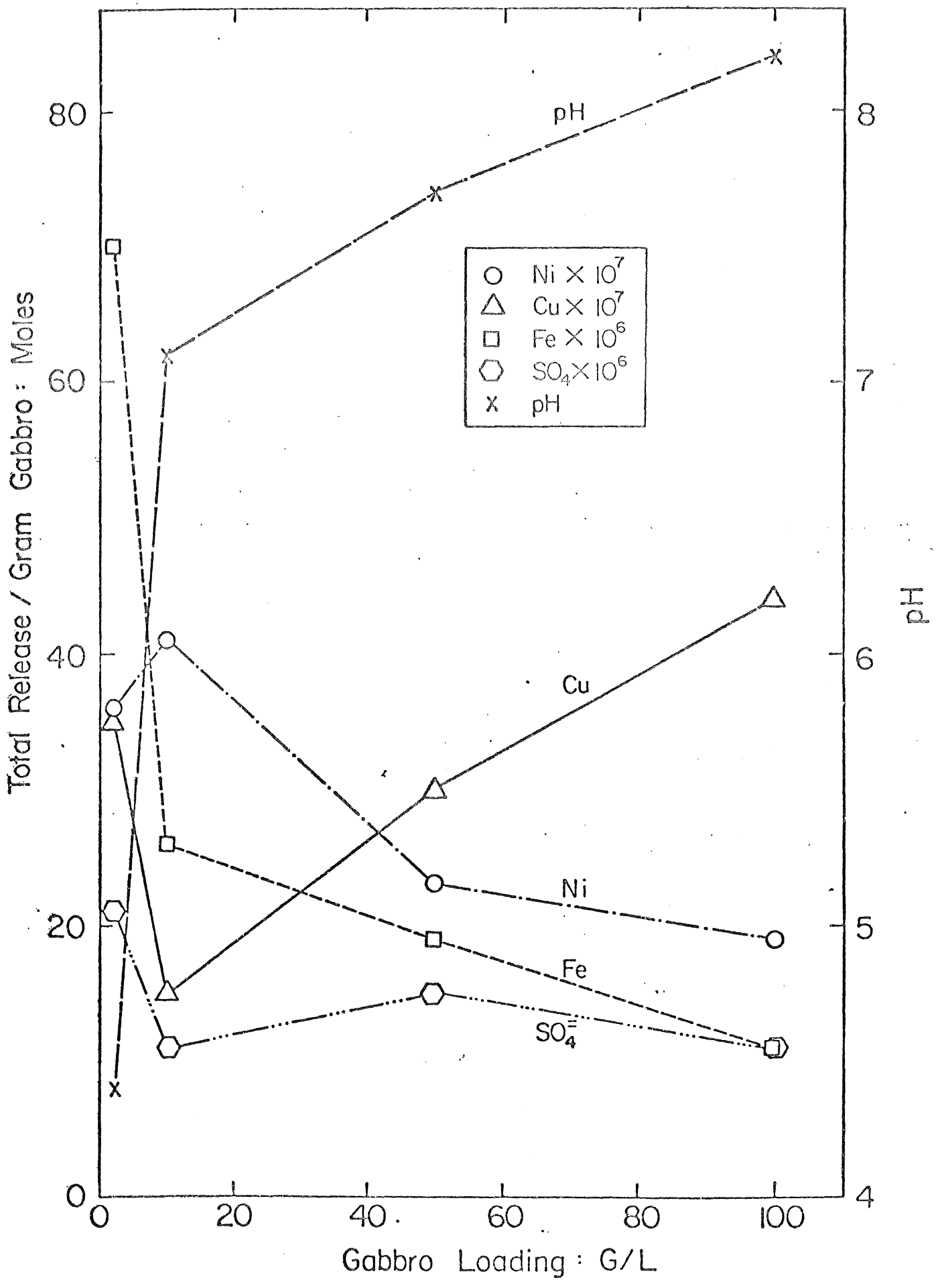

Total Release / Gram Gabbro : Moles
pH
Gabbro Loading : G/L
0
20
40
60
80
100
4
5
6
7
8
pH
Ni × 10^7
Cu × 10^7
Fe × 10^6
SO4 × 10^6
pH
Cu
Ni
Fe
SO4=

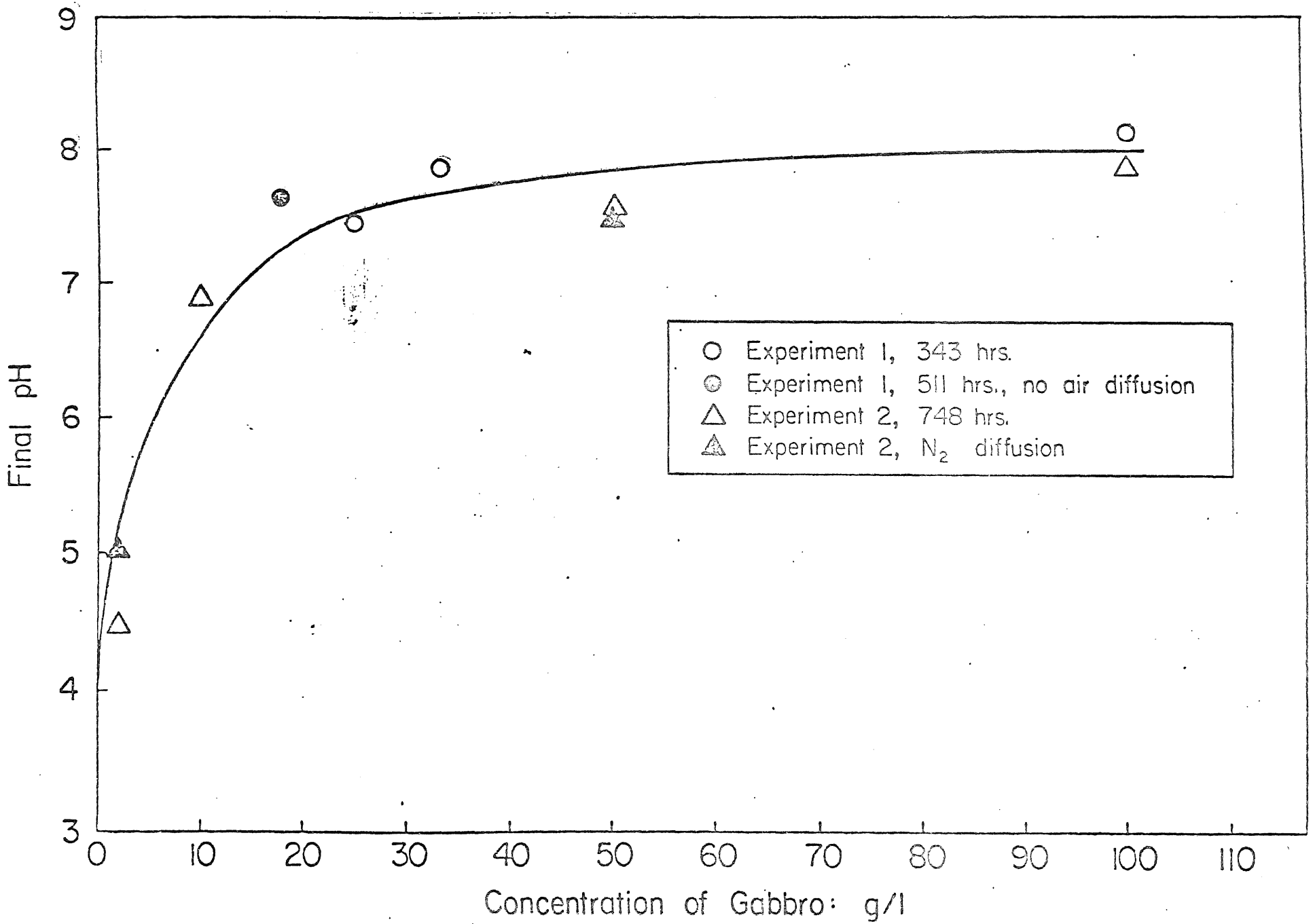
Table 3.1

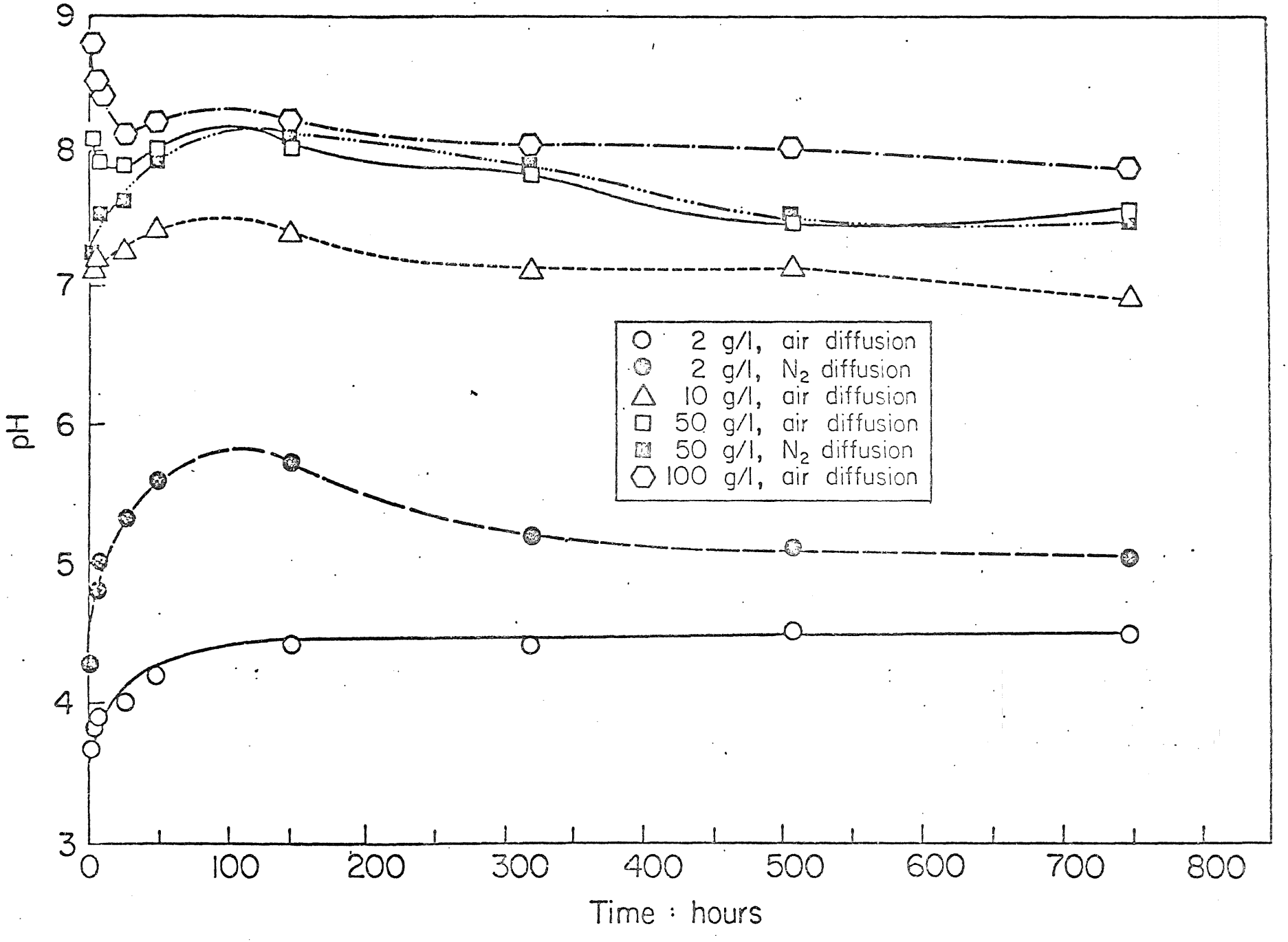
Total Release: Effect of loading on unmineralized gabbro
Ad: Adsorbed & Precipitated, Sol: In solution, Tot: Total Release
All values in moles * 10^6 per gram gabbro

Time: 750 Hrs

g Gabbro / Liter	pH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{Sol}	Mg_{Tot}	SO_{Tot}
2	4.4	0.083	3.6	3.6	0.24	0.24	3.5	7.0	63	7.0	6.6	32	.38	2.6	19	19	21
10	7.1	3.7	0.44	4.1	1.5	<0.01	1.5	26	0.027	26	N.D.	13	N.D.	N.D.	6.1	N.D.	11
50	7.7	2.2	0.035	2.3	3.0	0.01	3.0	19	0.01	19	58	7.6	65	13	3.3	16	15
100	8.2	1.8	0.02	1.9	4.4	0.01	4.4	11	0.01	11	N.D.	5.6	N.D.	N.D.	2.7	N.D.	11
2*	5.2	0.12	2.2	2.3	0.22	0.28	0.50	9.9	59	69	5.5	24	29	3.6	16	20	18
50*	7.7	1.8	0.01	1.8	2.2	0.01	2.2	20	0.01	20	56	6.0	62	16	3.5	19	4.4

* Nitrogen Diffusion
ND: Not Determined


Final pH
9
8
7
6
5
4
3
0
10
20
30
40
50
60
70
80
90
100
110
Concentration of Gabbro: g/l
Experiment 1, 343 hrs.
Experiment 1, 511 hrs., no air diffusion
Experiment 2, 748 hrs.
Experiment 2, N_2 diffusion


pH
9
8
7
6
5
4
3
0
100
200
300
400
500
600
700
800
Time : hours
2 g/l, air diffusion
2 g/l, N_2 diffusion
10 g/l, air diffusion
50 g/l, air diffusion
50 g/l, N_2 diffusion
100 g/l, air diffusion

was decided to follow the appearance of Ni with time as a measure of the intrinsic rate of oxidative dissolution of metal sulfides.

The rate of oxidative-dissolution can be expressed mathematically in the following way:

$$-d\,[(Fe,Ni)_9S_8]/dt = kA_s^{l}[O_2]^m[H^+]^n \qquad (3.5)$$

assuming that the kinetics of dissolution have a similar dependence on active surface area as oxygen concentration and hydrogen ion concentration. Superscripts ℓ, m, η are the reaction orders for A_s, $[O_2]$ and $[H^+]$, respectively. Under constant conditions any catalytic factors would be accounted for by k, the overall rate constant. From the stoichiometry for the oxidative-dissolution of pentlandite by oxygen

$$(Fe,Ni)_9S_8 + 17.63O_2 + 6.5H^+ \rightarrow 4.5Fe^{+3} + 4.5Ni^{+2} + 8SO_4^{=} + 3.25H_2O \qquad (3.6)$$

The rate of disappearance of pentlandite will be related to the rate of appearance of Ni^{+2} in solution by equation 3.7.

$$-d[(Fe,Ni)_9S_8]\,/dt = (1/4.5)d[Ni^{+2}]/dt \qquad (3.7)$$

and substituting into equation 3.5 gives

$$(1/4.5)d[Ni^{+2}]/dt = kA_s^{\,l}[O_2]^m[H^+]^n \qquad (3.8)$$

Under conditions of constant O_2 , pH and A_s equation 3.8 can be rewritten as a pseudo zero-order reaction

$$d[Ni^{+2}]/dt = k^{l} \qquad (3.9)$$

where

$$k^{l} = 4.5kA_s^{\,l}[O_2]^m[H^+]^n \qquad (3.10)$$

Equation 3.10 predicts that a plot of $[Ni^{+2}]$ versus time should be linear with a slope equal to k^{l} as is actually observed in all kinetic runs when all other variables are held constant. The pseudo zero-order rate constant, k^{l}, will have units of ug l^{-1} hr^{-1}.

The rate of dissolution of $(Fe,Ni)_9S_8$ can be also expressed as a flux when the total reaction volume V and active surface area, A_s, are considered. In general, for a heterogeneous reaction of a solid surface with a surface area A_s (cm^2) dissolving in a solvent system of volume V (Cm^3) the flux (moles/cm^2sec) of material escaping from the surface into solution can be written in terms of the rate of the bulk concentration value, Ni^{+2} (moles/cm^3) as follows:

$$\frac{d\,[Ni^{+2}]}{dt} = \frac{AsF}{V} \tag{3.11}$$

or

$$F\left(\frac{moles}{cm^2\text{-}sec}\right) = \frac{V}{As}\,(d[Ni^{+2}]/dt) \tag{3.12}$$

Since As was not determined experimentally but is a function of the actual surface area, A(cm^2/g), the mass of Gabbro, m, can be substituted for As assuming that

$$As = mA \tag{3.13}$$

Substituting this expression into equation 2.12 allows an "operational" surface flux to be calculated according to equation 3.14.

$$AF = F^1\left(\frac{moles}{g\text{-}sec}\right) = \frac{V}{m}\,\frac{d\,[Ni^{+2}]}{dt} \tag{3.14}$$

In subsequent sections results of a parametric study on the factors controlling the absolute value of k^1 or F^1 will be presented sequentially starting with particle size, followed by pH, dissolved oxygen, ionic strength and chelating agents.

3.3. Effect of Particle Size, Dp.

In general, finely divided solids have a greater solubility than large crystals. This phenomenon is commonly known as the Kelvin effect (26). The free energy change, ΔG, involved in subdividing a coarse solid suspended in aqueous solution into a finely divided one of molar surface area, S, is given by

$$\Delta G = 2/3\bar{\gamma}S \tag{3.15}$$

where $\bar{\gamma}$ is the surface tension of the solid-liquid interface. This change in free energy can be considered in the overall free energy change for dissolution.

$$\Delta G^{1} = \Delta G_{o} + \Delta G_{s} \tag{3.16}$$

which yields

$$\log k_{so(S)} = \log k_{so(S=0)} + \frac{2/3\bar{\gamma}S}{2.3RT} \tag{3.17}$$

Equation 3.17 shows that the thermodynamic behavior of a solid is a function of the surface area. If the rate of oxidative-dissolution were dependent on a pre-equilibrium dissociation of the sulfide mineral followed by a rate determining oxidation of $Fe^{+\Delta}$ or HS^{-} in solution, then a shift in the equilibrium constant due to a change in solid surface area would become significant. At 25°C with $\bar{\gamma}$ = 400 ergs/cm^2 and S = 10^4 m^2/mole the effect on log k_{so} is approximately 0.5.

Alternatively, if the rate of oxidative-dissolution is a function of the number of reactive sites on the surface, then increasing the total available surface area should increase the reaction rate. To test this hypothesis, the effect of four discrete particle size fractions at a fixed mass loading on the reaction rate was studied. The four size fractions examined were

$Dp \leq 74\mu$, $74\mu < Dp \leq 420\ \mu$, $420\mu < Dp \leq 840\mu$, and $840\mu < Dp \leq 2380$.

The approximate surface area of the $Dp \leq 74$ fraction was determined using a sub-sieve sizer. The areas of the other fractions were estimated using this approximate value, finer sieving analysis and mathematical extrapolation. The areas as determined by the above methods were 1700, 190, 62 and 25 cm^2/g. Elemental analyses for Ni, Cu, Fe, Ca and Mg found in each discrete particle size function are presented in Table 3.2

As particle size was reduced the rate of dissolution increased. This effect is shown clearly in Figure 3.5. Using kinetic data presented in Table A.3 it is possible to determine the sensitivity of k to changes in As where

Table 3.2a

Effect of Particle Size

COMPOSITION: All Values in mg (g gabbro)$^{-1}$

Diameter (mm)	Ni	Cu	Fe	Ca	Mg
$d \leq 0.074$	3.55	14.1	172.	22.7	19.2
$0.074\ d \leq 0.42$	1.82	8.25	155.	17.6	18.2
$0.42\ d \leq 0.84$	4.52	12.8	259.	29.3	37.9
$0.84\ d \leq 2.38$	4.08	23.5	460.	47.3	80.2

Table 3.2b

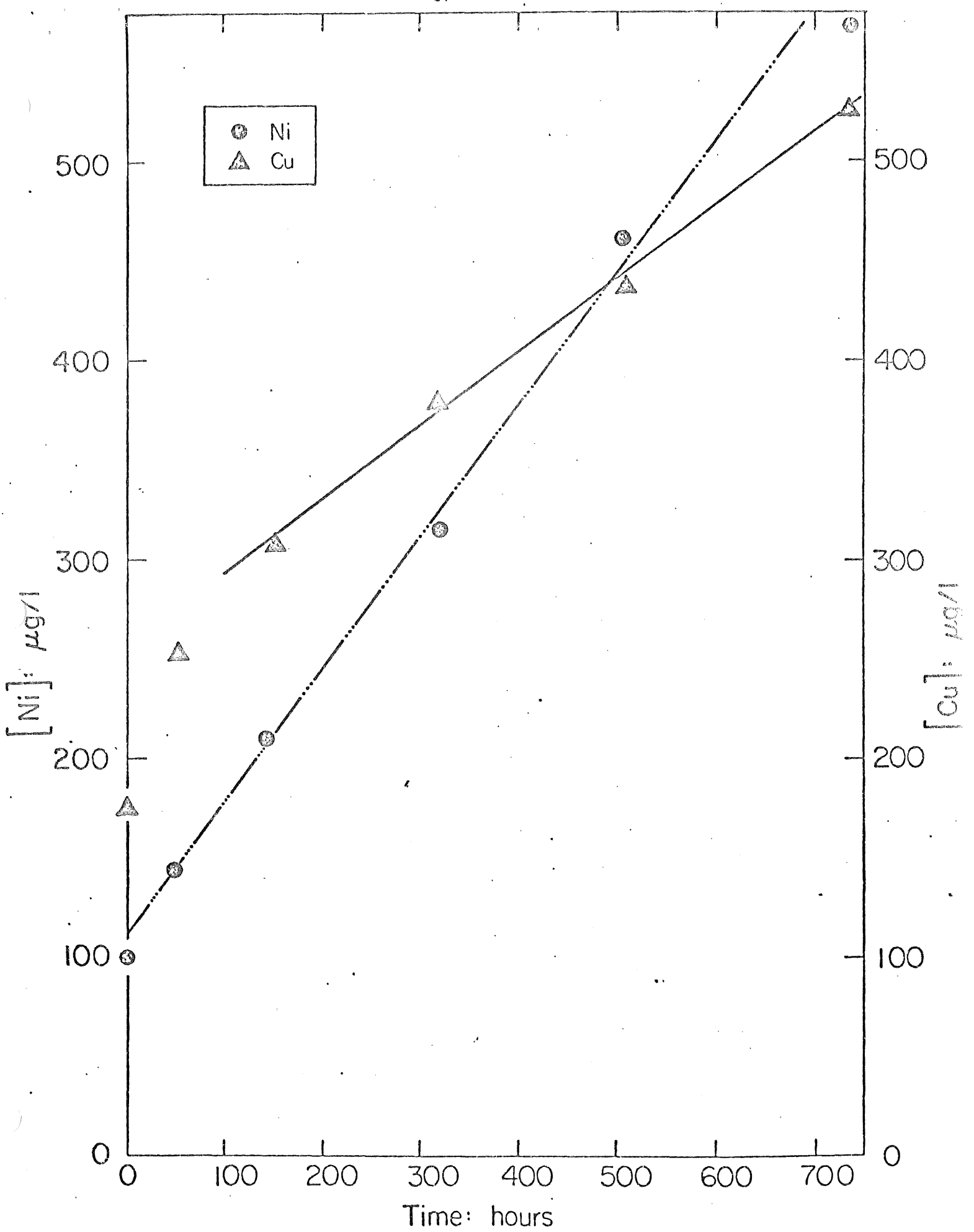
Total Release: Effect of particle size: Mineralized Gabbro
Ad: Adsorbed & Precipitated, Sol: In solution, Tot: Total Release
All values in moles * 10^6 per gram gabbro

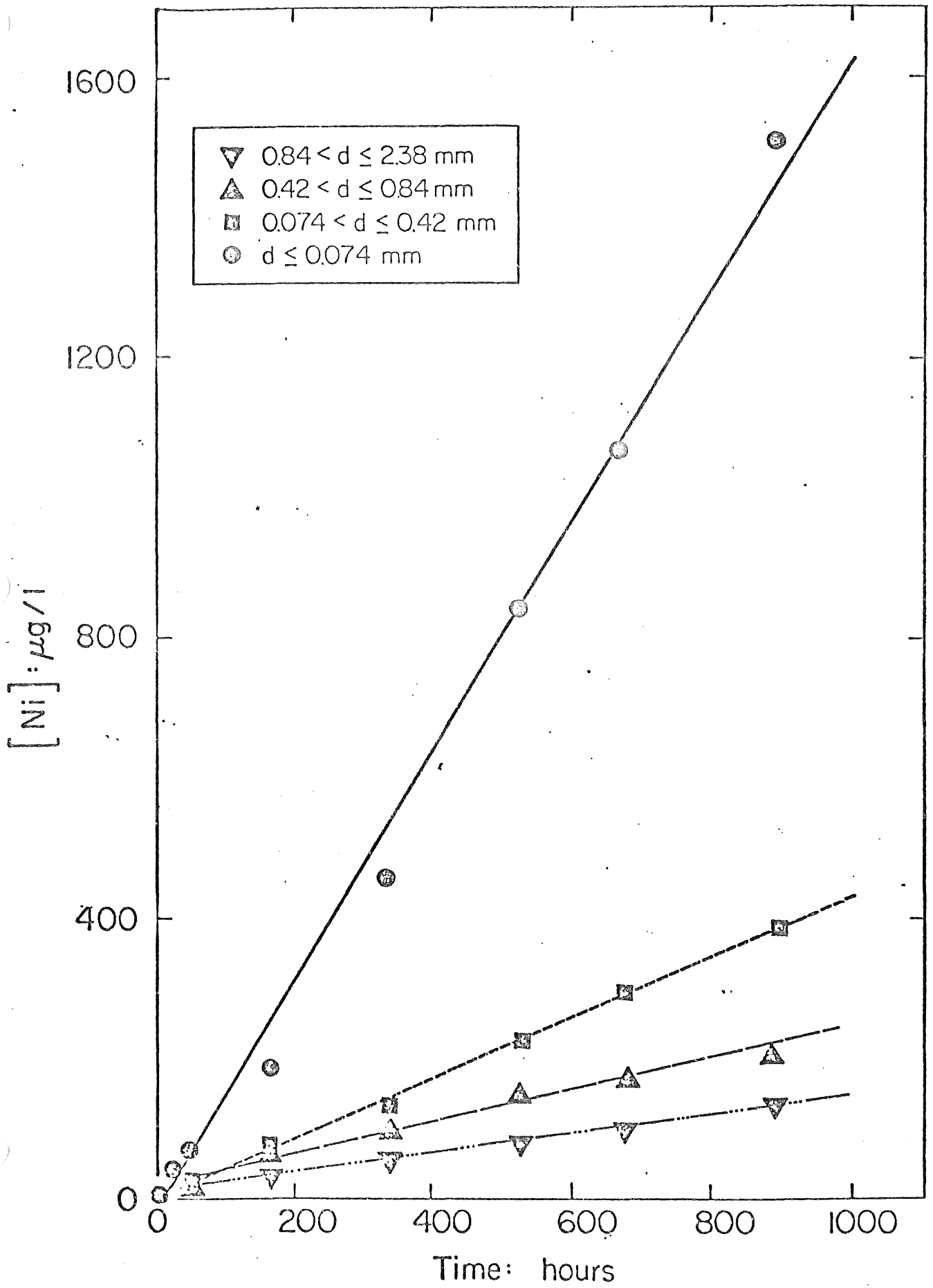
Time: 890 Hrs.

S.A. $cm^2 l^{-1}$	pH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{Sol}	Mg_{Tot}	SO_{Tot}
4900	6.0	0.09	6.3	6.4	0.08	0.33	0.41	0.47	0.27	0.74	2.2	9.1	11	0.57	6.1	6.7	30
540	6.4	0.08*	2.2	2.3	0.05*	0.03	0.3*	0.3*	0.02	0.32	2*	6.3	8.3	0.4*	2.8	3.2	19
180	6.5	0.05	1.2	1.2	0.02	0	0.17	0.17	0.03	0.19	1.4	4.7	6.1	0.08	2.2	2.2	17
72	6.2	0.04	0.76	0.80	0.02	<0.01	0.36	0.37	0.06	0.42	1.9	4.0	5.8	0.42	1.2	1.6	15

Table 3.3
Total Release: Effect of pH on unmineralized Gabbro
Ad: Adsorbed & Precipitated, sol: In solution, Tot: Total Release
All values in moles * 10^6 per gram gabbro

pH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{Sol}	Mg_{Tot}	SO_{Tot}
6.9	0.33	2.6	3.0	0.17	0.02	0.19	1.9	0.03	1.9	10	18	29	0.88	8.6	9.5	14
4.7	0	4.7	4.7	0.03	1.2	1.2	1.8	35	37	1.1	23	24	0.08	15	15	20
4.3	<0.01	3.2	3.2	0.05	1.5	1.5	2.7	71	74	1.5	46	48	0.49	23	23	19
4.0	<0.01	2.7	2.7	0.03	1.1	1.2	1.9	100	100	0.91	64	65	0.35	37	37	18


Ni
Cu
500
400
300
200
100
0
[Ni]: μg/l
[Cu]: μg/l
0
100
200
300
400
500
600
700
Time: hours


0.84 < d ≤ 2.38 mm
0.42 < d ≤ 0.84 mm
0.074 < d ≤ 0.42 mm
d ≤ 0.074 mm
1600
1200
800
400
0
[Ni] : μg/l
0
200
400
600
800
1000
Time: hours

$$k^1 = k\ A_s^{\ 1} \tag{3.18}$$

Taking the logarithm of both sides of this equation yields

$$\log k^1 = \log k + 1 \log As \tag{3.19}$$

which is a linear equation with a slope equal to the reaction order, 1, for the surface area. Equation 3.19 is plotted in Figure 3.6 with data taken from Table 3.10. From this linear relationship the reaction order in A_s was determined to be 0.4. From data presented in Table 3.3 it can be seen that total metal release for all metals studied increases with a decrease in particle size although this effect is most promounced for the metals which are thought to originate from sulfidic minerals. Copper and Ni appear predominately in solution whereas the majority of iron is found in the adsorbed state, most likely as FeOOH.

As total specific surface area is increased within the same particle size fraction the apparent dissolution rate remains unaffected as shown in Figure 3.7. This result indicates that metal solubility as defined by the solubility product relationship may control the rate of appearance of Ni in solution. Decreasing the particle size would have the net effect of increasing the solid solubility as predicted by equation 3.17.

If the rate of dissolution were controlled by a chemical reaction on the sulfide surface then increasing the total reactive surface area should result in a demonstrable increase in reaction rate.

Further evidence for a solution phase rate limiting step in the dissolution of pentlandite is presented in Figures 3.8 and 3.9 which show the time dependent changes in pH and sulfate, respectively. pH remained relatively constant in the larger particle size fractions; however, in the lowest size fraction the pH was progressively lowered as the reaction proceeded. This indicates that significant oxidation of HS^- and Fe^{+2} is occurring with a concomitant release of H^+ to solution.

Table 3.10

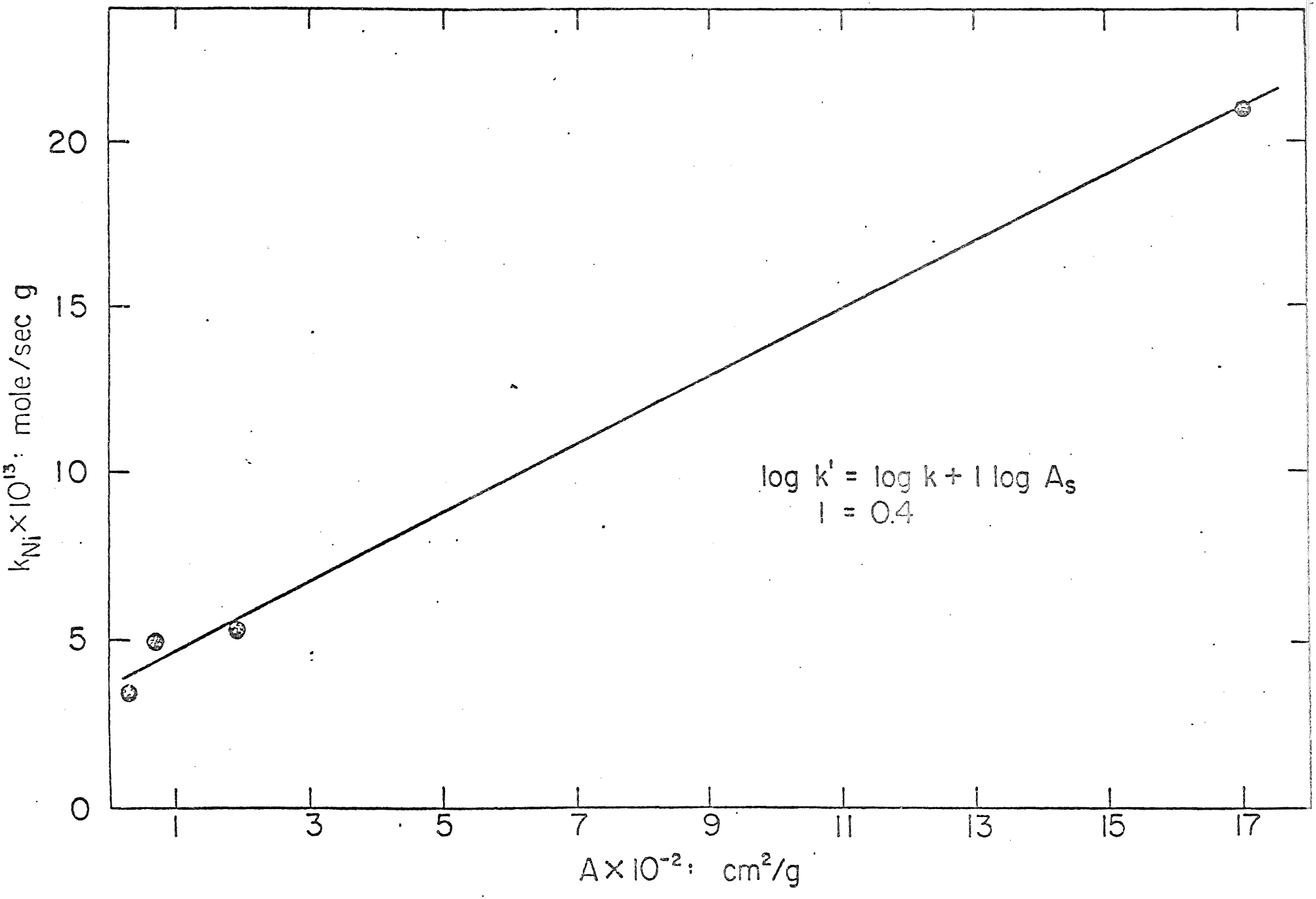
Nickel Kinetics - Summary of Data

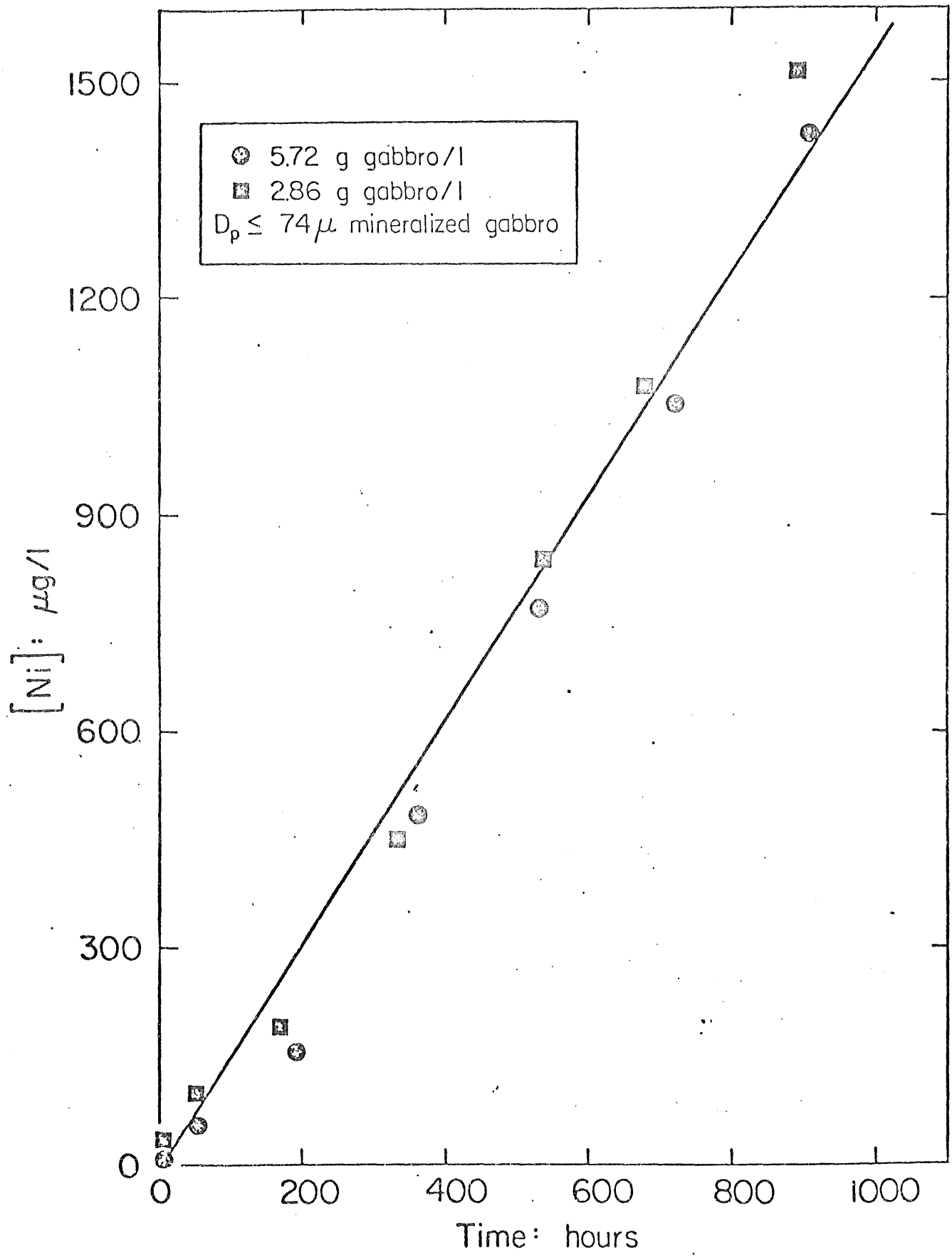
[Ni] = kt+b
: in mole sec^{-1} (g gabbro)$^{-1}$
) in mole (g gabbro)-1
: in seconds

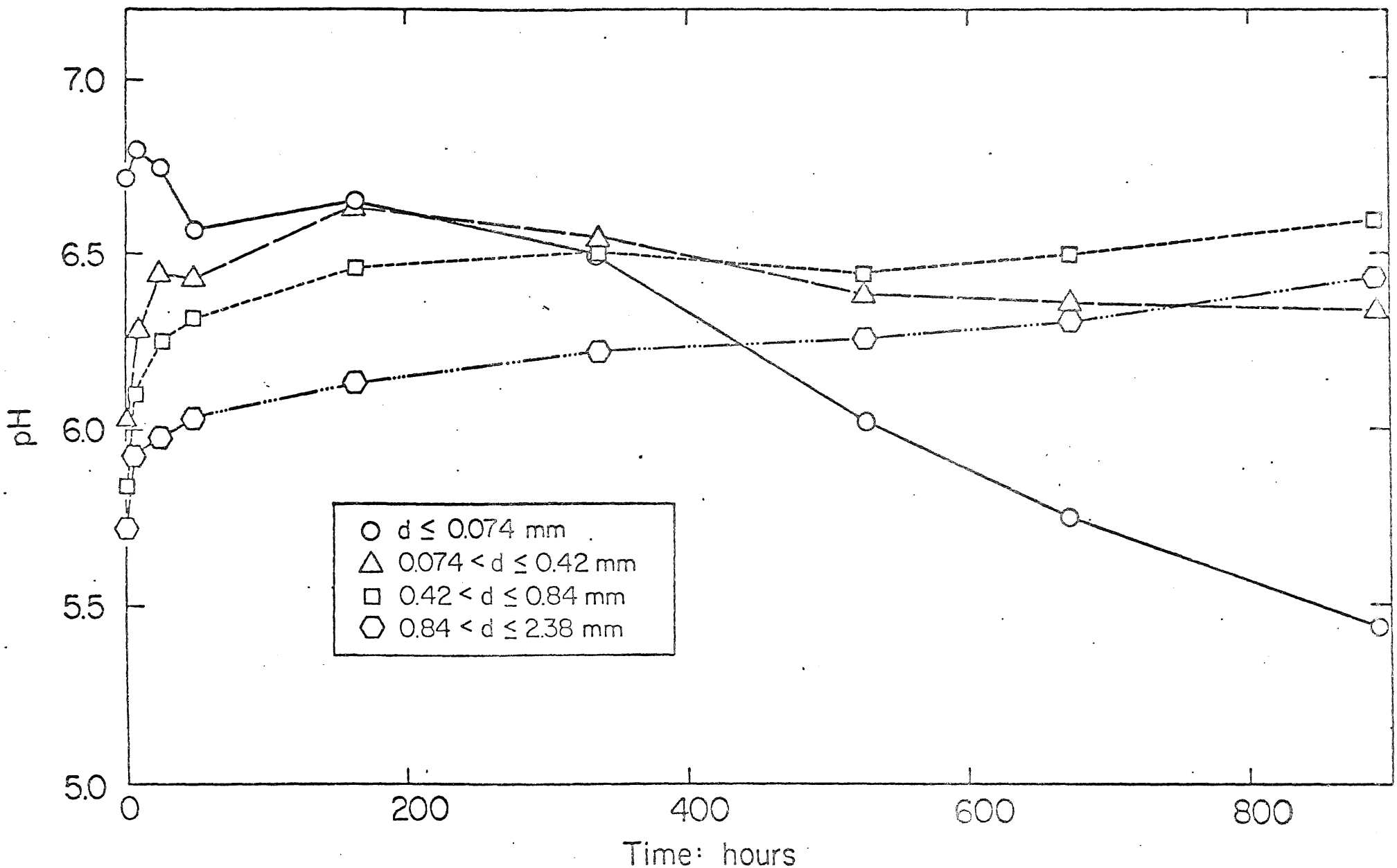
Reactor	r^2	$b*10^9$	$k*10^{13}$	Description
2-1	0.8814	890	24	2 gl^{-1} unmineralized gabbro, air diffusion
2-2	0.9914	17	16	10 gl^{-1} unmineralized gabbro, air diffusion
2-3	0.9687	-1.4	3.2	50 gl^{-1} unmineralized gabbro, air diffusion
2-4	0.8531	0.081	0.79	100 gl^{-1} unmineralized gabbro, air diffusion
2-5	0.8911	450	17	2 gl^{-1} unmineralized gabbro, N_2 diffusion
2-6	0.4274	2.5	-0.15	50 gl^{-1} unmineralized gabbro, N_2 diffusion
3-1	0.9938	3.1	30	2.86 gl^{-1} unmineralized gabbro, PH 6.9
3-2	0.9853	270	37	2.86 gl^{-1} unmineralized gabbro, PH 4.6
3-3	0.9424	360	27	2.86 gl^{-1} unmineralized gabbro, PH 3.9
3-4	0.8723	400	18	2.86 gl^{-1} unmineralized gabbro, PH 3.5
4-1	0.9972	53	21	2.86 gl^{-1} -200 mesh mineralized gabbro
4-2	0.9759	25	4.9	2.86 gl^{-1} +200 - -40 mesh mineralized gabbro
4-3	0.9710	15	5.1	2.86 gl^{-1} +40 - -20 mesh mineralized gabbro
4-4	0.8935	23	3.5	2.86 gl^{-1} +20 - -8 mesh mineralized gabbro
5-1	0.8903	-1.0	4.8	2.86 gl^{-1} mineralized gabbro, D.O.=1 mgl^{-1}
5-2	0.9879	58	11	2.86 gl^{-1} mineralized gabbro, D.O.=5 mgl^{-1}
5-3	0.9779	180	22	2.86 gl^{-1} mineralized gabbro, D.O.=9.3 mgl^{-1}
5-4	0.8929	170	21	2.86 gl^{-1} mineralized gabbro, D.O.=44 mgl^{-1}
6-1	0.9858	39	12	5.71 gl^{-1} mineralized gabbro, PH 5
6-2	0.9963	21	9.4	5.71 gl^{-1} mineralized gabbro, PH 6
6-3	0.9814	20	5.8	5.71 gl^{-1} mineralized gabbro, PH 7
6-4	0.9964	10	5.1	5.71 gl^{-1} mineralized gabbro, PH 8
7-3	0.9475	1.3	11	5.71 gl^{-1} mineralized gabbro, μ=0.005
7-4	0.9830	4.4	12	5.71 gl^{-1} mineralized gabbro, μ=0.05
8-1a	0.9230	20	1.2	
8-1b	0.6944	80	3.1	
8-2a	0.9780	2.4	0.31	
8-2b	0.9192	5.7	0.42	
8-3a	0.5324	0.43	0.0098	
8-3b	0.6554	3.4	-0.015	

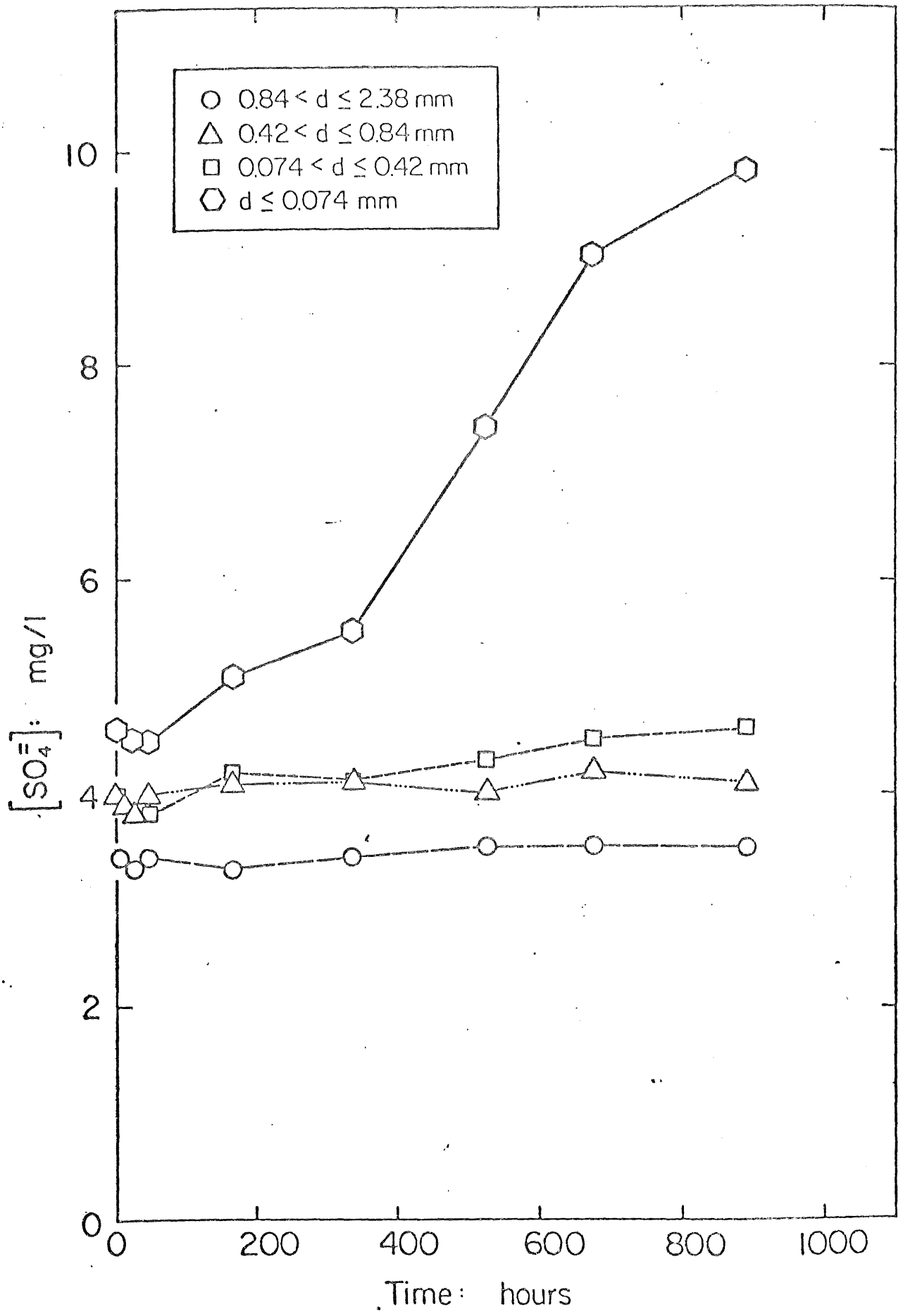
Table 3.10 Continued

Reactor	r^2	$b*10^9$	$k*10^{13}$	Description
3-5	0.9970	77	40	2.86 gl^{-1} unmineralized gabbro, $1.29x10^{-4}$ citrate
3-6	0.9977	55	91	2.86 gl^{-1} unmineralized gabbro, $1.29x10^{-4}$ citrate
4-5	0.8706	250	39	2.86 gl^{-1} unmineralized gabbro, $1.29x10^{-3}$ citrate
4-6	0.9989	160	110	2.86 gl^{-1} unmineralized gabbro, $1.29x10^{-4}$ citrate
5-5	0.9478	200	50	2.86 gl^{-1} unmineralized gabbro, $1.29x10^{-3}$ citrate, pH7
5-6	0.9868	240	56	2.86 gl^{-1} unmineralized gabbro, $1.29x10^{-}$ citrate, pH7
7-1	0.8120	0.50	1.4	5.71 gl^{-1} mineralized gabbro, bog water
7-2	0.9544	0.72	2.0	5.71 gl^{-1} mineralized gabbro, ground water
7-5	0.9862	0.94	2.6	5.71 gl^{-1} mineralized gabbro, $1.29x10^{-4}$ phthallic, pH7
7-6	0.8382	0.24	0.66	5.71 gl^{-} mineralized gabbro, $1.29x10^{-3}$ phthallic, pH7
8-4a	0.7509	2.0	0.025	
8-4b	0.9893	2.3	0.03k	
8-5a	0.0028	1.4	-0.00021	
8-5b	0.9881	1.5	0.043	


$k_{Ni} \times 10^{13}$: mole/sec g
0
5
10
15
20
1
3
5
7
9
11
13
15
17
$A \times 10^{-2}$: cm^2/g
$\log k' = \log k + l \log A_s$
$l = 0.4$


5.72 g gabbro/l
2.86 g gabbro/l
$D_p \leq 74\mu$ mineralized gabbro
1500
1200
900
600
300
0
[Ni]: μg/l
0
200
400
600
800
1000
Time: hours


7.0
6.5
6.0
5.5
5.0
pH
0
200
400
600
800
Time: hours
d ≤ 0.074 mm
0.074 < d ≤ 0.42 mm
0.42 < d ≤ 0.84 mm
0.84 < d ≤ 2.38 mm


0.84 < d ≤ 2.38 mm
0.42 < d ≤ 0.84 mm
0.074 < d ≤ 0.42 mm
d ≤ 0.074 mm
[SO4=]: mg/l
Time: hours
0
2
4
6
8
10
0
200
400
600
800
1000

This idea is corroborated by the $SO_4^{=}$ vs. time profile in Figure 3.9. Significant increases of sulfate with time occurred only in the lowest particle size fraction indicating that a pre-equilibrium dissolution of pentlandite may be rate controlling. Oxidation of Fe^{+2} to Fe^{+3} and subsequent hydrolysis to $Fe(OH)_3$ or formation of jarosite, $KFe_3(SO_4)_2(OH)_6$ will result in the release of either 3 or 6 moles of H+ released per mole of Fe^{+2} oxidized.

$$Fe^{+2} + 1/4O_2 + H^+ \rightarrow Fe^{+3} + 1/2H_2O \qquad (3.18)$$

$$Fe^{+3} + 3H_2O \rightarrow Fe(OH)_3 + 3H^+ \qquad (3.19)$$

$$K^+ + 2Fe^{+3} + 2SO_4^{=} + 6H_2O \rightarrow KFe_2(SO_4)_2(OH)_6 + 6H^+ \qquad (3.20)$$

Regardless of the mechanistic implications, the kinetic results show that empirically equation 3.9 can be rewritten to take surface area into account in the following fashion

$$-d\,[(Fe_1Ni)_9S_8] = d\,[Ni(II)] = 4.5k_sA_s^{0.4} \qquad (3.21)$$

where

$$k_s = k[O_2]^m[H^+]^n \qquad (3.22)$$

3.4. Effect of Dissolved Oxygen Concentration

The oxidation of FeS, $CuFeS_2$ and $(Fe, Ni)_9S_8$ by oxygen to a variety of products is favored thermodynamically under experimental conditions used in this study. However, the stoichiometries presented in equations 1.4 through 1.11 are incomplete. Oxidized sulfur is found in a variety of metastable oxidation states. These states include elemental sulfur, S^o, polysulfide, S_n^{-2}, dithionite, $S_2O_4^{-2}$, thiosulfate, $S_2O_3^{-2}$, trithionate, $S_3O_6^{-2}$, tetrathionate, $S_4O_6^{-2}$, sulfite, $S_2O_6^{-2}$, and dithronate, $S_2O_6^{-2}$; although under the conditions of these experiments sulfate, $SO_4^{=}$, is the predicted end-product of oxidation. For example, in the oxidation of pyrrhotite

$$FeS + MO_2 \rightarrow \text{products} \qquad (3.23)$$

the stoichiometric coefficient, M, can vary from 0.75 when sulfur is the exclusive oxidation product to 2.25 when sulfide is oxidized to sulfate. For a given set of experimental conditions where the reaction variables are held constant, the value of M should remain constant in the absence of any autocatalytic influences which may arise during the course of the reaction. Experimentally, the reaction order of oxygen, M, can be determined by studying the variation of k^1 with $[O_2]$ when all other variables are held constant.

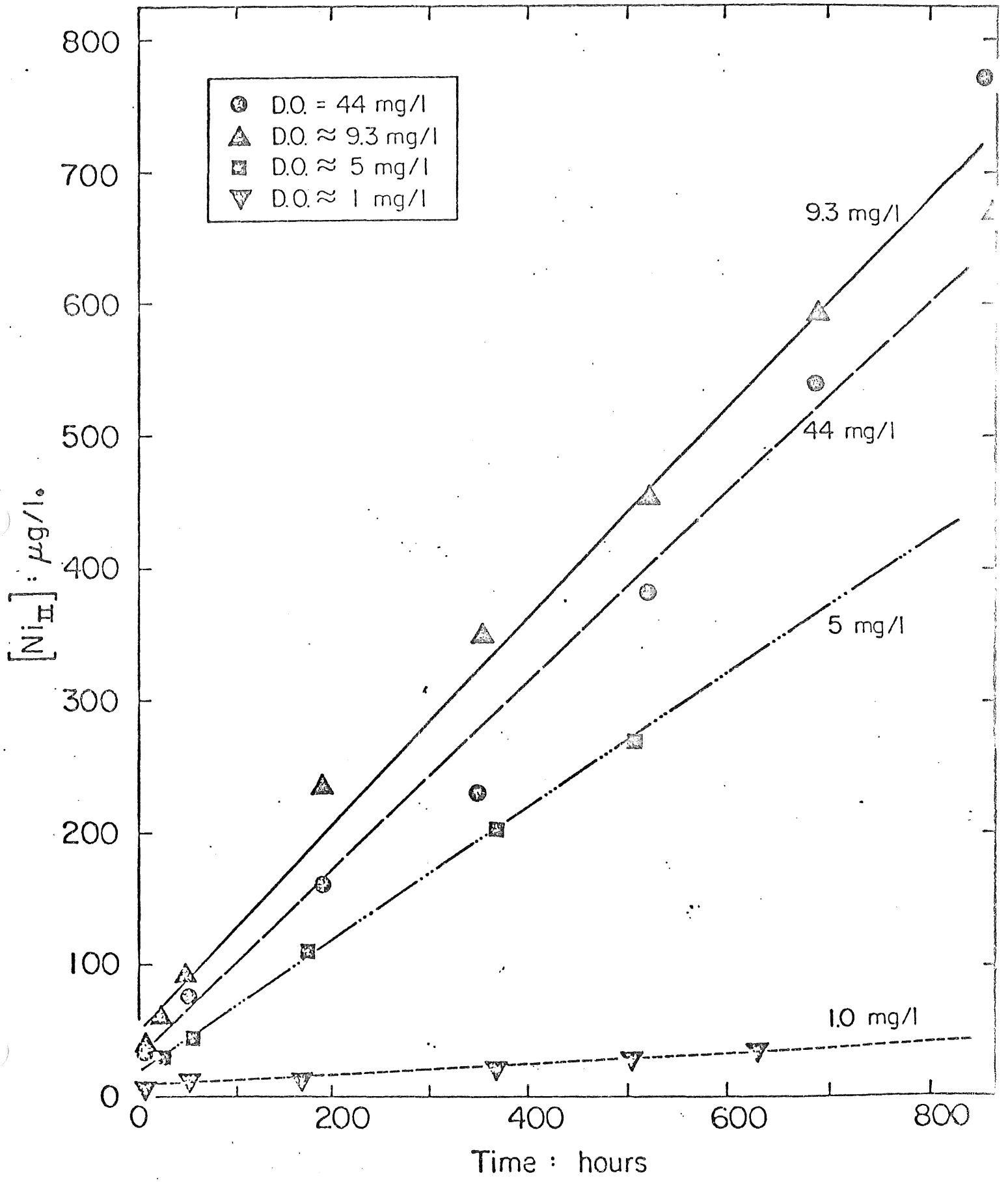
Variable $[O_2]$ concentrations were achieved by differential mixing of N_2 and O_2 gas through four different reactors. Concentrations of oxygen of 1, 5, 9.3 and 44 mg/l were obtained and held constant throughout the reaction by gas diffusion through a fritted glass dispersion tube with continuous mechanical stirring.

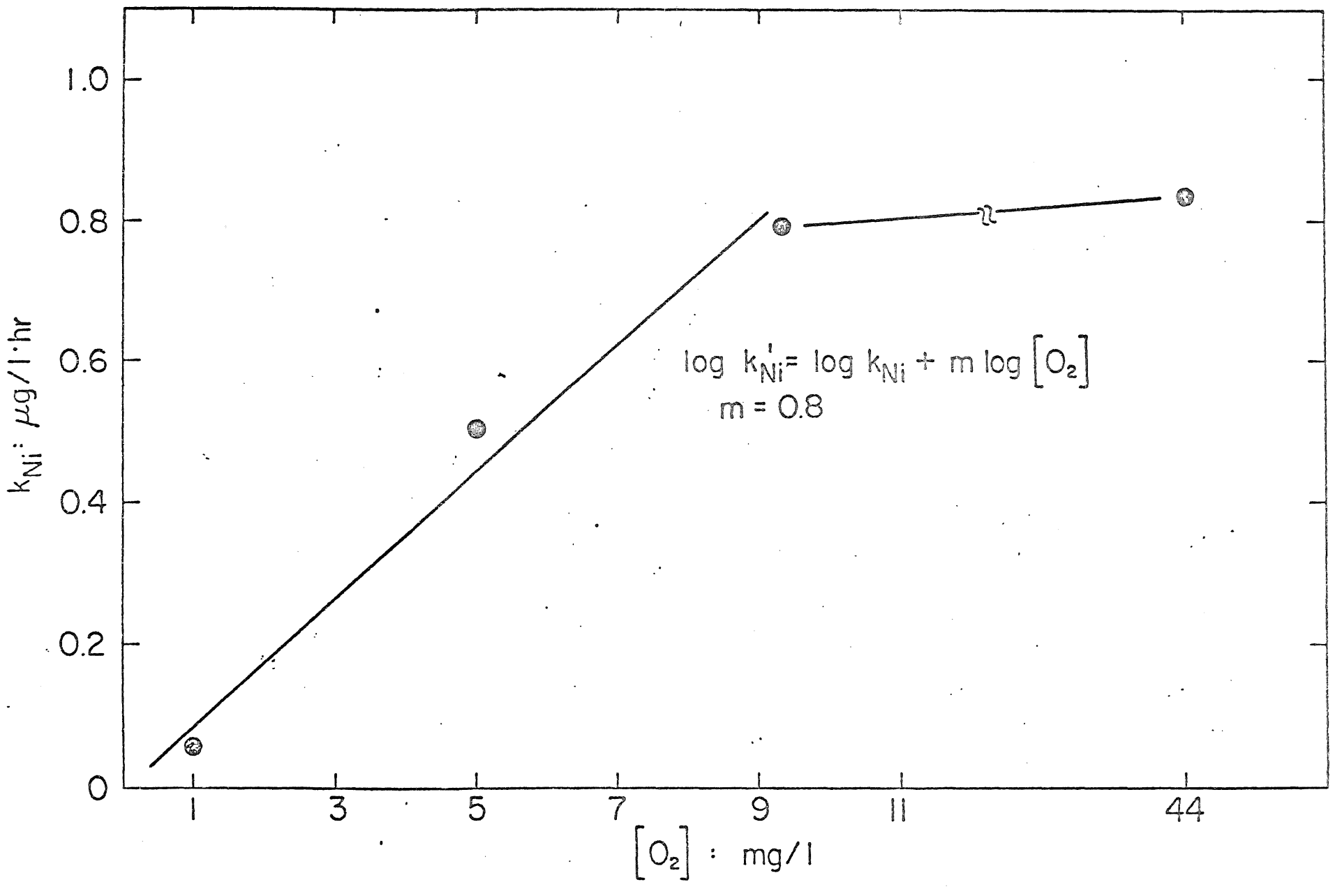
The apparent Ni release rates increased as a direct function of increasing $[O_2]$ until 9.3 mg/l as shown in Figure 3.10. Above 10 mg/l in $[O_2]$ the reaction rate shows no further increase. This result suggests that the rate may be limited by the number of reactive surface available for direct attack by oxygen. Once these reactive sites were fully saturated there would be no further rate increases with increasing $[O_2]$ in solution. The dependence of k^1 on $[O_2]$ can be ascertained from the following linear relationship

$$k^1 = k\,[O_2]^m \tag{3.24}$$

$$\log k^1 = \log k + m\log[O_2] \tag{3.25}$$

which is plotted for the kinetic data presented in Table A.4 in Figure 3.11. From this graphical relationship it can be seen that the reaction order in $[O_2]$ is 0.8 which is fairly close to one. This suggests that at low $[O_2]$ the reaction order is one and at high concentration it should be zero. This saturation effect is seen in Figure 3.11.


D.O. = 44 mg/l
D.O. ≈ 9.3 mg/l
D.O. ≈ 5 mg/l
D.O. ≈ 1 mg/l
9.3 mg/l
44 mg/l
5 mg/l
1.0 mg/l
[Ni II] : μg/l.
Time : hours
0
100
200
300
400
500
600
700
800
0
200
400
600
800


1.0
0.8
0.6
0.4
0.2
0
k_{Ni}: μg/l·hr
1
3
5
7
9
11
44
$[O_2]$: mg/l
$\log k'_{Ni} = \log k_{Ni} + m \log [O_2]$
m = 0.8

Combining this information with that contained in equation 3.21 allows the empirical rate law to be written as

$$\frac{d\,[Ni(II)]}{dt} = k^{11} As^{0.4} [O_2]^{0.8} \tag{3.26}$$

where

$$k^{11} = 4.5k[H^+]^n \tag{3.27}$$

Significant release of Cu with time during the O_2 variation experiments did not occur until the oxygen concentration exceeded 5.0 mg/l. Unfortunately during the course of these experiments the pH was not maintained absolutely constant. Initially, the pH at each $[O_2]$ was stable but as the reactions proceeded, especially when $[O_2]$ >9.0 mg/l the pH dropped as the buffering capacity of the Gabbro was exceeded. The change in pH during the later portion of each experiment may explain the Cu release rates observed in Figure 3.12. Even if oxidation proceeds at an appreciable rate, Cu oxidation products would be expected to accumulate as $Cu_2CO_3(OH)_2$ and $Cu(OH)_2$ in a reacted zone on the surface of $CuFeS_2$. When pH drops below 6, release of Cu to the aqueous solution will occur.

These interrelationships are illustrated more clearly in the data presented in Table 3.4. Nonetheless, the pronounced dependence of the dissolution rate and extent of metal release on the dissolved oxygen concentration implies that sulfide oxygen plays an important role in the leaching process. Saturation of the rate of release of Ni at higher $[O_2]$ suggests a surface controlled reaction which is limited by the number of available reaction sites.

Although the reaction rate is not significantly increased when the $[O_2]$ exceeds 10 mg/l, the extent of and rate of $SO_4^=$ formation (Figure 3.31) is increased at higher concentrations. This would be expected to occur if the initial sulfide oxidation product was a metastable intermediate such as S^o, $S_2O_3^=$ or $S_4O_6^=$. Given sufficient time these metastable sulfur products will be oxidized completely to sulfate.

Table 3.4

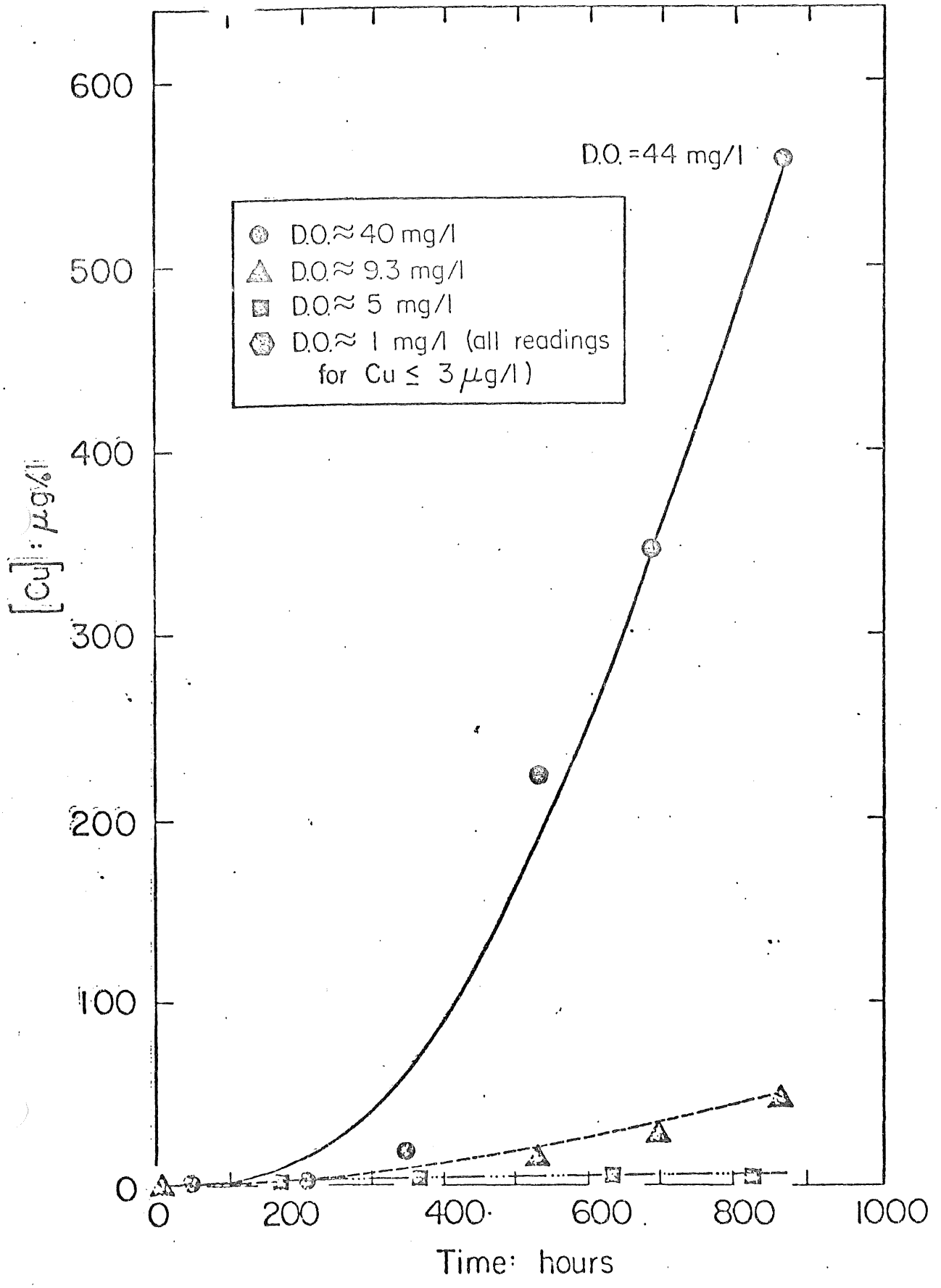
Total Release: Effect of pH; Mineralized Gabbro
Ad: Adsorbed & Precipitated, Sol: In solution, Tot: Total Release
All Values in Moles * 10^6 per gram gabbro

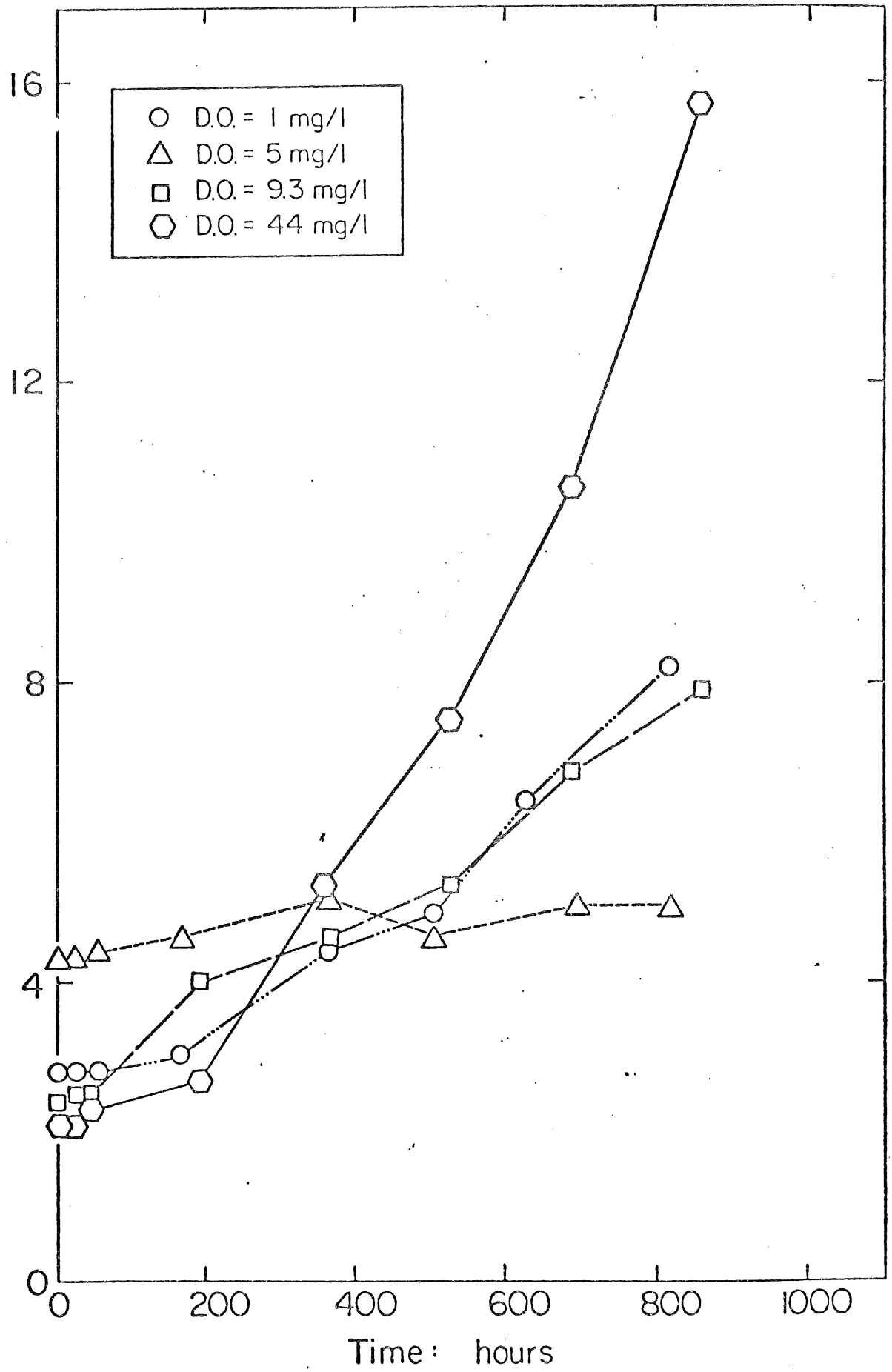
PH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{Sol}	Mg_{Tot}	SO_{Tot}
5.0	0.04	2.3	2.3	0.10	0.55	0.65	0.65	1.8	2.4	0.57	10	11	0.24	8.3	8.5	31
6.0	0.16	2.8	2.9	0.20	0.13	0.34	0.66	0.10	0.77	2.9	10	13	0.41	7.4	7.8	18
7.0	0.19	1.0	1.2	0.56	0.06	0.61	1.1	0.39	1.5	6.5	7.8	14	0.71	5.7	6.5	16
8.0	0.21	0.09	0.30	0.67	0.01	0.68	1.3	0.07	1.4	8.4	6.9	15	1.1	4.7	5.8	15

Table 3.5

Total Release: Effect of Dissolved Oxygen on Mineralized Gabbro
Ad: Adsorbed & Precipitated, Sol: In solution, Tot: Total Release
All values in moles * 10^6 per gram gabbro

D.O. MG/L	PH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{Sol}	Mg_{Tot}	SO_{Tot}
1	7.9	0.21	0.11	0.32	0.32	<0.01	0.32	1.9	0.07	2.0	15	10	25	4.6	6.0	11	22
5	7.0	0.19	0.27	0.47	0.35	0.02	0.36	1.4	0.04	1.5	9.1	10	19	1.7	6.9	8.5	16
9.3	6.0	0.30	2.9	3.2	0.29	0.17	0.45	0.90	0.05	0.95	10	14	24	1.1	5.6	6.7	22
44	5.3	0.06	3.0	3.1	0.11	1.9	2.0	0.57	3.4	3.9	5.5	22	27	0.70	7.7	8.4	39


600
500
400
300
200
100
0
[Cu] : μg/l
D.O. =44 mg/l
D.O.≈ 40 mg/l
D.O.≈ 9.3 mg/l
D.O.≈ 5 mg/l
D.O.≈ 1 mg/l (all readings for Cu ≤ 3 μg/l)
0
200
400
600
800
1000
Time: hours


D.O. = 1 mg/l
D.O. = 5 mg/l
D.O. = 9.3 mg/l
D.O. = 44 mg/l
16
12
8
4
0
0
200
400
600
800
1000
Time : hours

3.5. Effect of pH

From the results of previous studies (12, 16), on the oxidative-dissolution of metal sulfides, the role of pH is relatively unclear. At most there seems to be a reciprocal dependence of dissolution rate on the hydrogen ion concentration.

In order to determine the impact of pH on the observed rate constant for Ni release, four different reactors were run with pH maintained constant by the addition of acid or base. Mass loadings and $[O_2]$ were maintained constant in each reactor at 5.71 gl^{-1} of $\leq 74\mu$ mineralized Gabbro and 9.3 mg/l, respectively. Results of this controlled pH variation are presented graphically in Figure 3.14 and in tabular form in Table A.5. From the trends observed in Figure 3.14 it is apparent that the Ni release rate increases as the pH drops from 8 to 5 although the most significant change occurs between pH 8 and 6.

The observed rate constant, k^1, depends on the $[H^+]$ with all other variables held constant in the following way

$$k^1 = k_H [H^+]^n \tag{3.28}$$

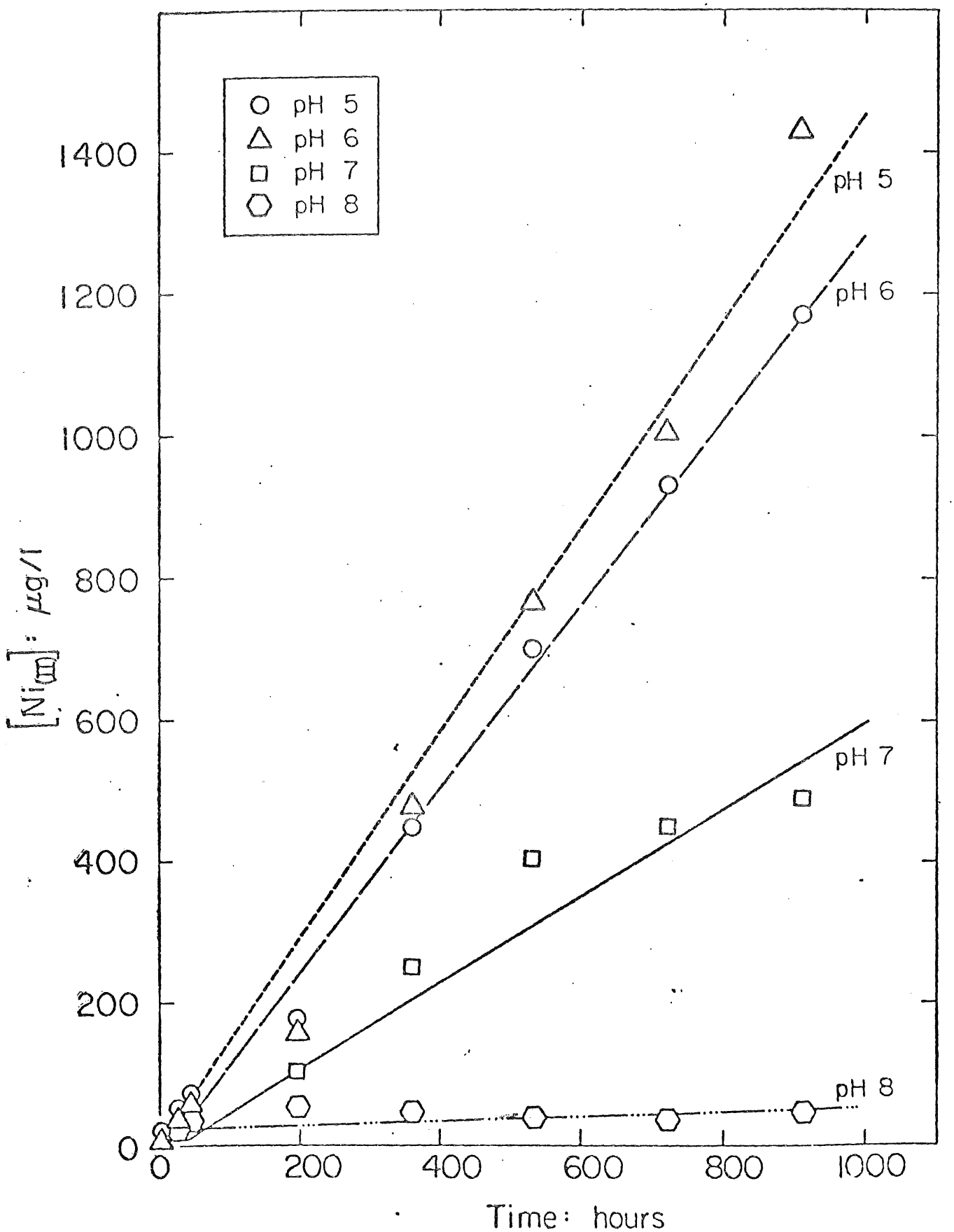
$$\log k^1 = \log k_H + n \log[H^+] \tag{3.29}$$

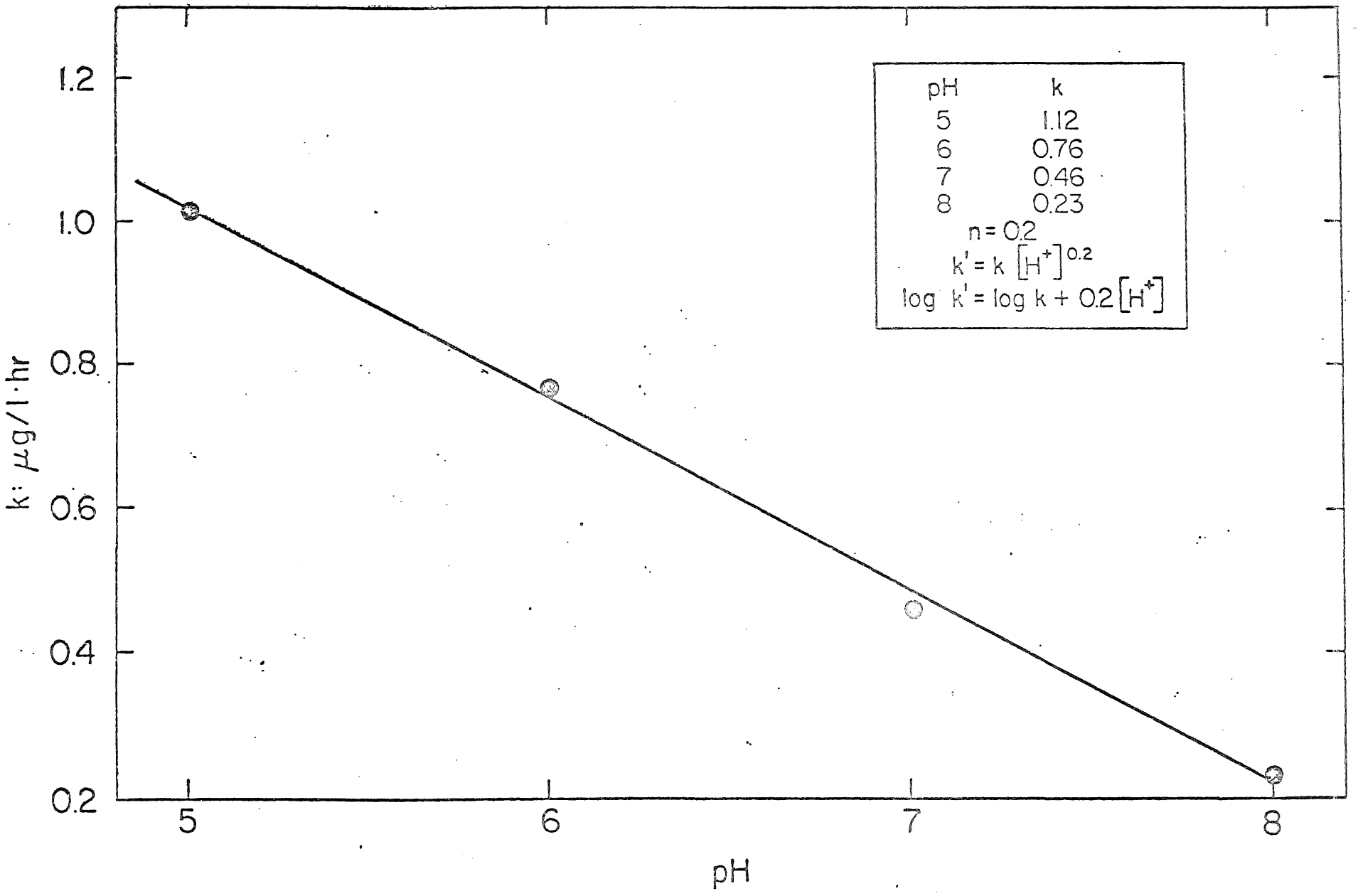
where

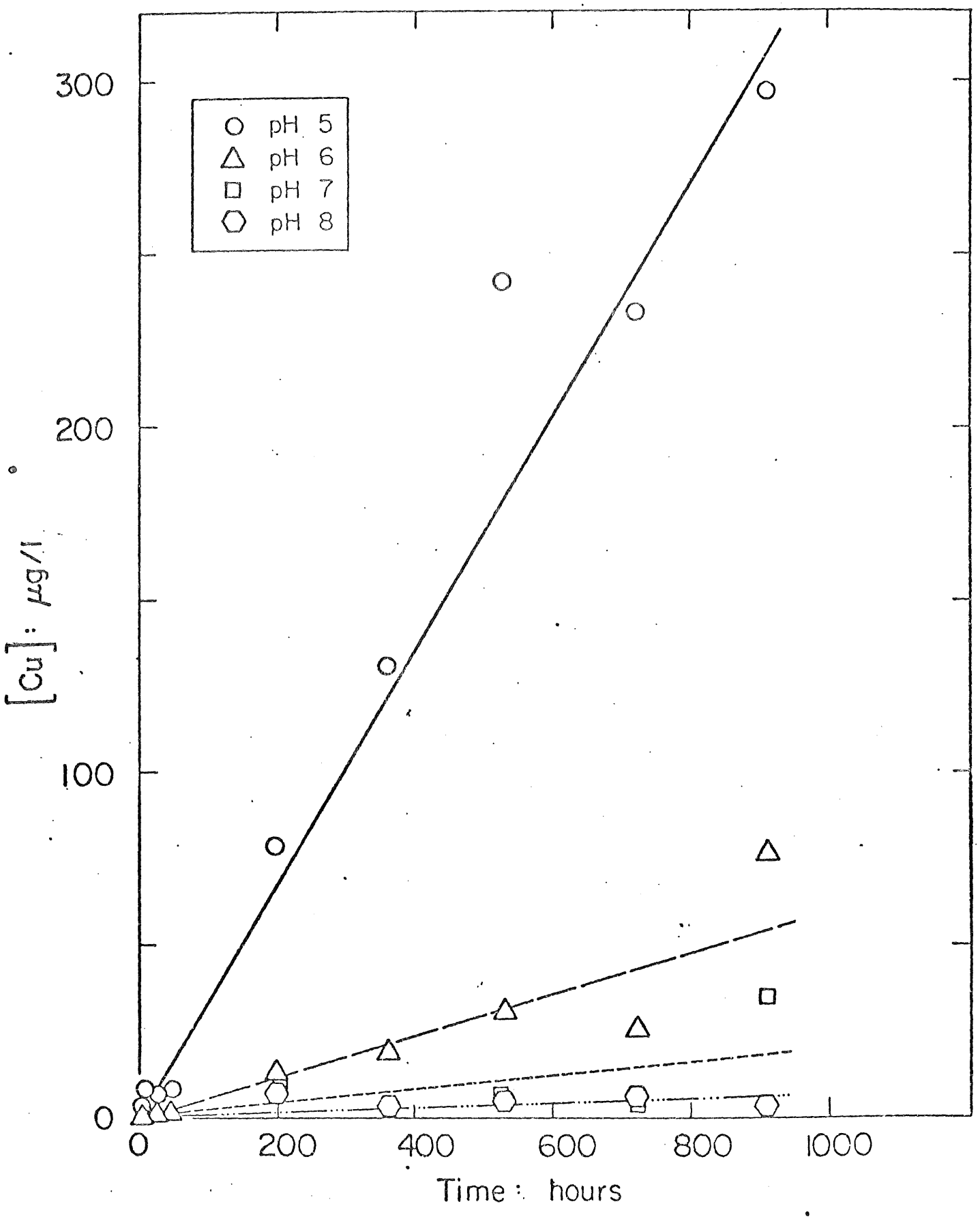
$$k_H = k A_s^{0.4} [O_2]^{0.8} \tag{3.30}$$

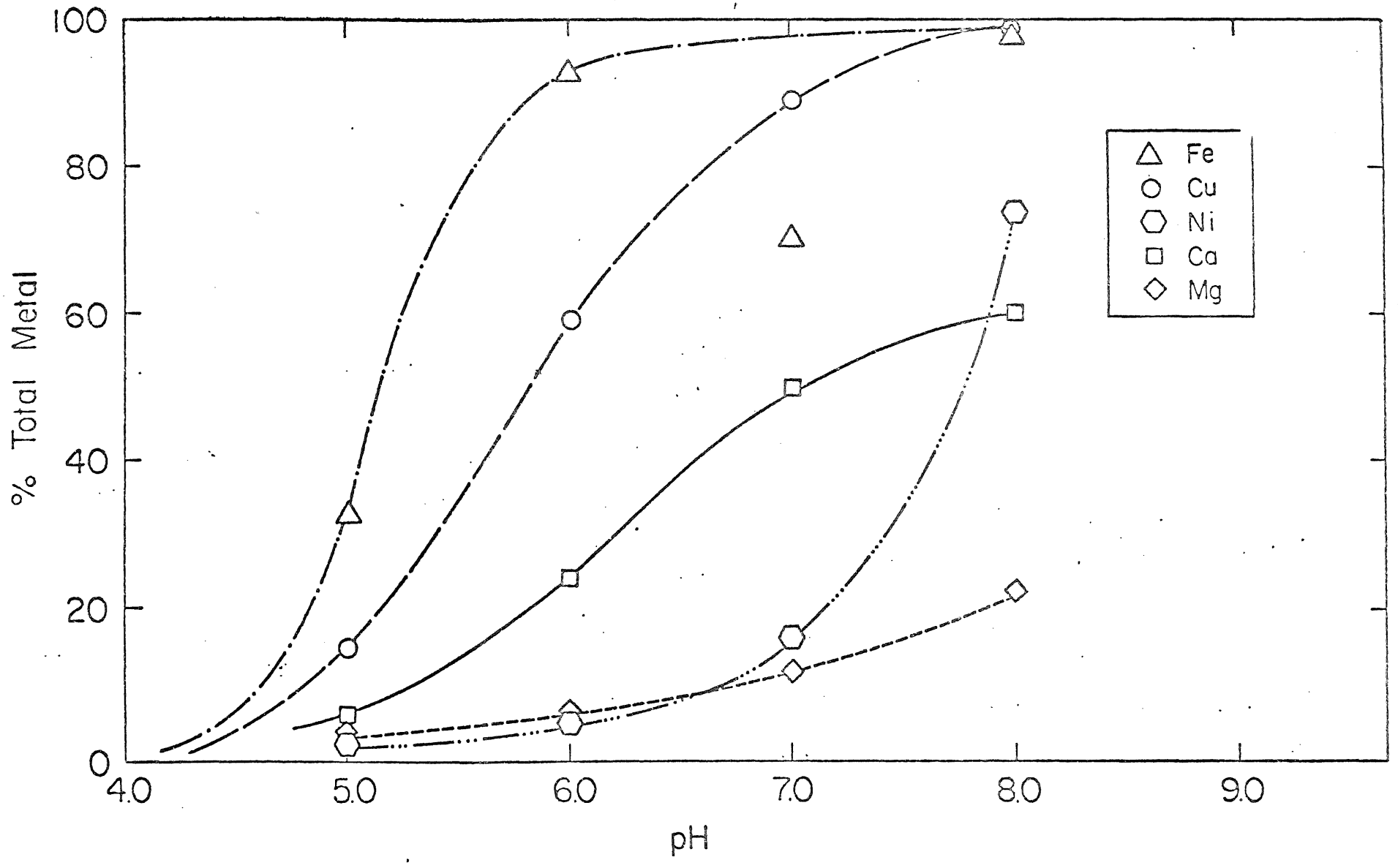
The reaction order, n, was determined to be 0.2 from a least-squares fit of the linear relationship presented in equation 3.29. The pH dependency and reaction order is illustrated in Figure 3.15. A similar pH dependency on Cu release kinetics is illustrated in Figure 3.16. Copper release rates are accelerated as pH drops from 8 to 5. The net impact of pH is summarized in Table 3.5.

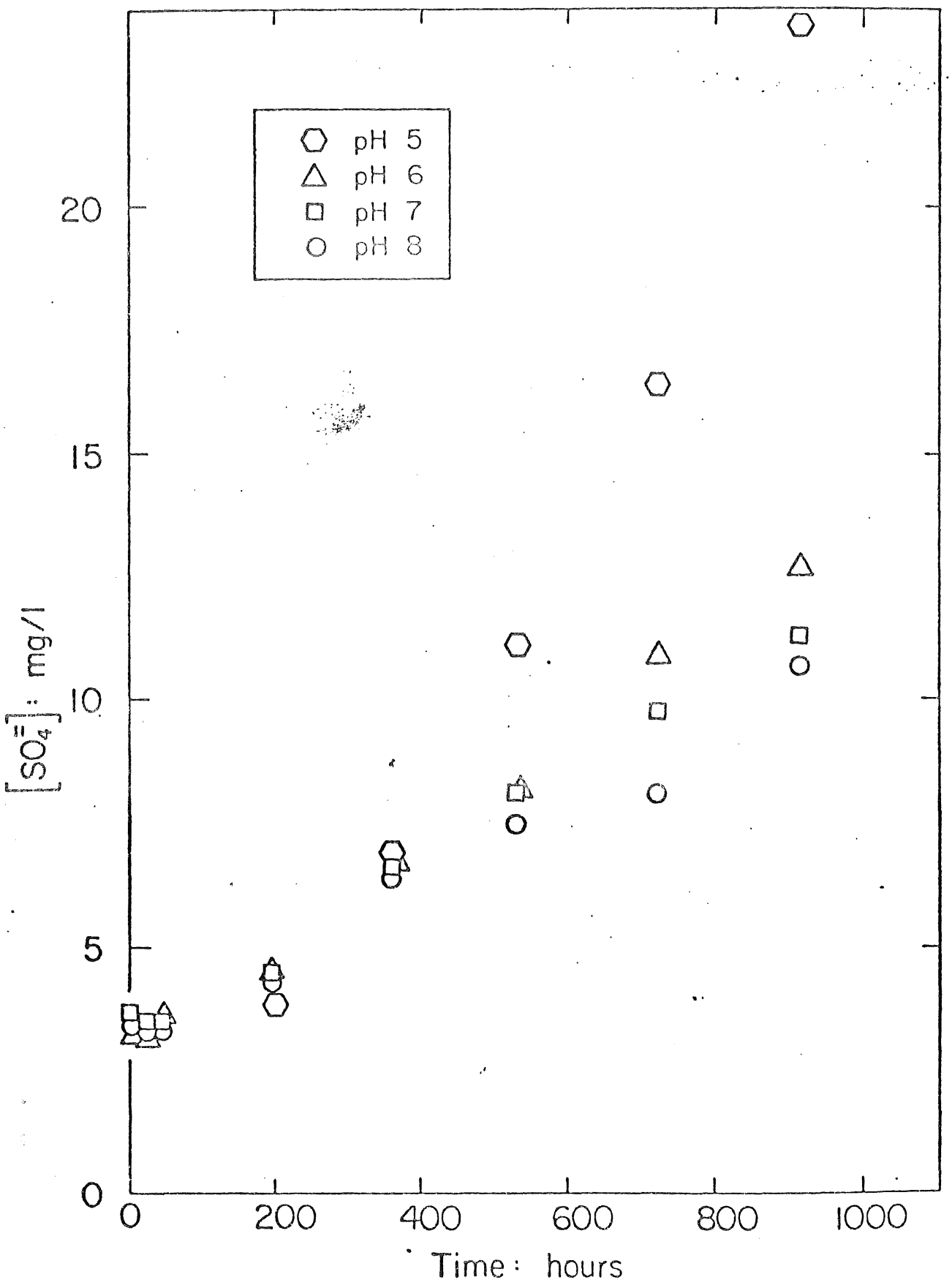
The increase in dissolution rate with a decrease in pH suggests that a surface controlled reaction which is dependent on protonation before the rate determining step. This could be visualized as a step involving protonation of


pH 5
pH 6
pH 7
pH 8
1400
1200
1000
800
600
400
200
0
[Ni(III)] : μg/l
0
200
400
600
800
1000
Time : hours
pH 5
pH 6
pH 7
pH 8


pH k
5 1.12
6 0.76
7 0.46
8 0.23
n = 0.2
k' = k [H+]^0.2
log k' = log k + 0.2 [H+]
k: μg/l·hr
1.2
1.0
0.8
0.6
0.4
0.2
5
6
7
8
pH


pH 5
pH 6
pH 7
pH 8
300
200
100
0
[Cu] : µg/l
0
200
400
600
800
1000
Time : hours


100
80
60
40
20
0
% Total Metal
4.0
5.0
6.0
7.0
8.0
9.0
pH
Fe
Cu
Ni
Ca
Mg


pH 5
pH 6
pH 7
pH 8
20
15
10
5
0
$[SO_4^=]$: mg/l
0
200
400
600
800
1000
Time: hours

surface sulfide groups from +S to +SH or $+SH_2$. Pankow (11) has shown that the rate of anoxic dissolution of FeS depends directly on the $[H^+]$ to the first-order. Without further experimental information, it would be difficult to conclude that reaction rate is controlled exclusively by a surface reaction that involves a proton in the transition state. The observed pH dependency would also be consistent with a solution phase reaction which depends on a pre-equilibrium dissolutionof the metal sulfide. However, the rate of oxidation of HS^- has been shown to decrease with a decrease in pH (14, 40).

This is also true for the oxidation rate of Fe^{+2} by O_2 which depends on $[OH^-]^3$ (13). Although unlikely in this pH range, Fe^{+3} could serve as the principal oxidant.

In addition to increasing the rate of oxidative dissolution with a decrease in pH, the solution phase mobility of Cu and Fe, in particular, is enhanced and to a lesser extent the mobility of Ni is also enhanced. As the pH drops from 8 to 5 the dominant solid and adsorbed phases become less important. This effect can be seen in Figure 3.17 which was calculated with the aid of REDEQL2 (41) using the maximum observed concentrations at the lowest pH. This effect is also illustrated by the data in Table 3.5.

Additional evidence for increased surface reactivity with an increase in $[H^+]$ is given in Figure 3.18 which shows that the sulfate concentration increases more rapidly at low pH. This trend is opposite the known solution phase kinetics of sulfide oxidation (14, 40) and suggests that increased protonation of reactive surface sites accelerates electron transfer on the surface.

3.6. Effect of Ionic Strength

Changes in ionic strength are expected to affect the rate of homogeneous reactions when charged species are reacting. In heterogeneous system such as

metal sulfide -H_2O-O_2 system ionic strength would be expected to affect the solubility product. For example considering the dissolution of FeS

$$FeS \underset{\leftarrow}{\overset{K_{so}}{\rightarrow}} Fe^{+2} + S^{-2} \tag{3.31}$$

the solubility product can be written as

$$K_{so} = [Fe^{+2}]\,[HS^{-}]\,f_{Fe}\,f_{s} \tag{3.32}$$

By using the extended Debye-Huckel law, equation 3.32 can be rewritten as

$$\log K_{so} = \log^{c}K_{so} - (Z_{Fe}^{2} + Z_{s}^{2})\left(\frac{0.5\ I}{1 + I}\right) \tag{3.33}$$

Equation 3.33 illustrates that the solubility increases with increasing ionic strength and that the effect is especially pronounced for solids containing multivalent ions.

To experimentally determine the effect of ionic strength, three reactors loaded with 5.71 g/l of Gabbro each were run under normal conditions except that one reactor contained 0.005 M NaCl antoher reactor contained 0.05 M NaCl and the third reactor was used as a control. The pH in each reactor was maintained constant at 7.0 with the controlled addition of acid or base. The effect of ionic strength on the rate of release of Ni is shown in Figure 3.19. A slight enhancement in the rate of release of Ni is observed as the ionic strenght is increased to 0.05 M NaCl. The pseudo zero-order rate constant for nickel release, k_{Ni}, is plotted as a function of ionic strenght, μ, in Figure 3.20. As predicted by equation 3.33, the solubility of pentlandite increases with an increase in μ. A summary of experimental results is presented in Table 3.6. In general total metal release for all metals is enhanced by an increase in ionic strength although the most pronounced effect was observed for the metal sulfides. Surprisingly, the sulfate concentrations remained relatively unaltered with changes in ionic strenth.

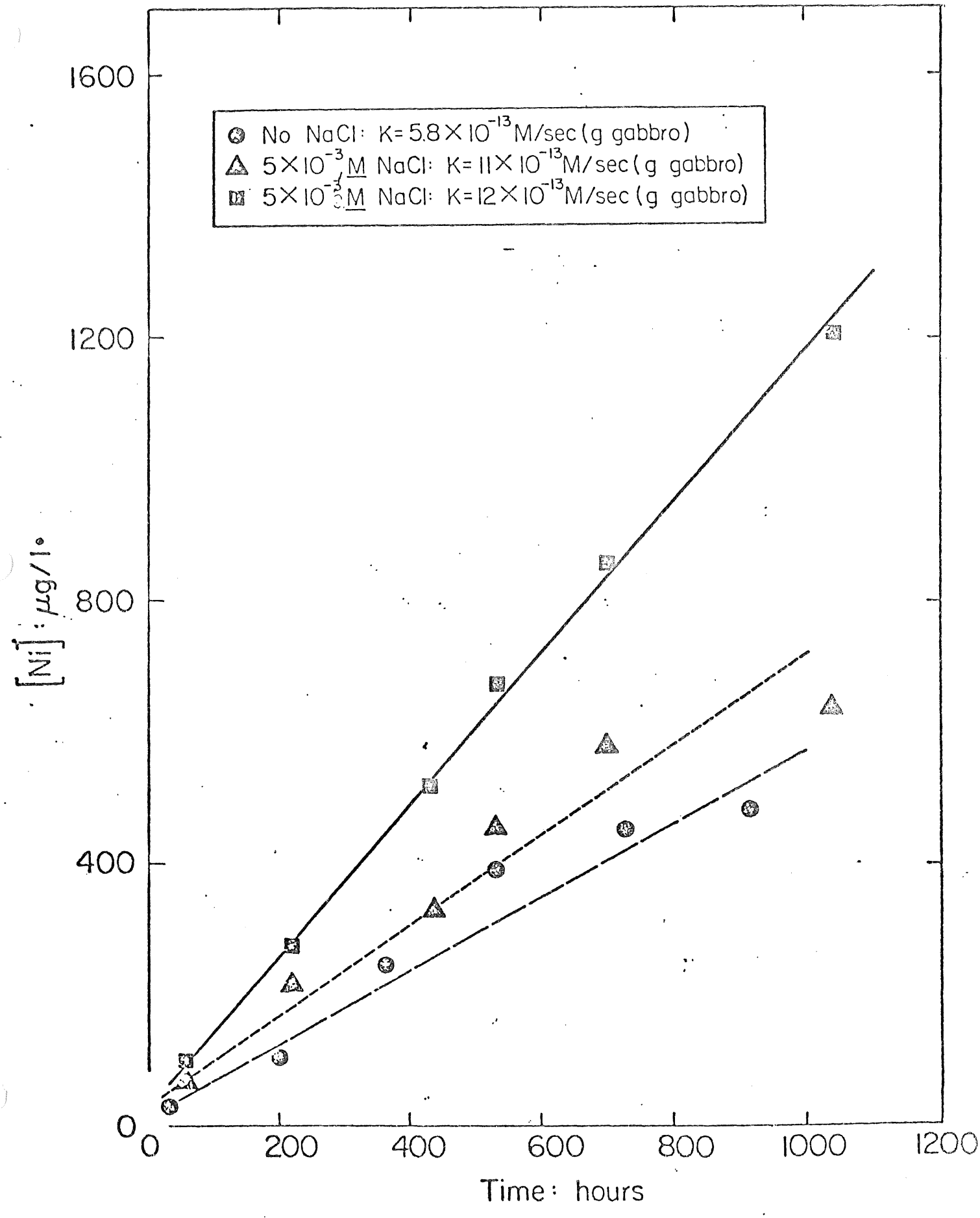

No NaCl: K=5.8×10^-13 M/sec (g gabbro)
5×10^-3 M NaCl: K=11×10^-13 M/sec (g gabbro)
5×10^-2 M NaCl: K=12×10^-13 M/sec (g gabbro)
[Ni] : μg/l
Time : hours
0
400
800
1200
1600
0
200
400
600
800
1000
1200

Table 3.6

Total Release: Effect of Ionic Strength
Ad: Adsorbed & Precipitated, Sol: In Solution, Tot: Total Release
All Values in Moles * 10^6 per gram gabbro

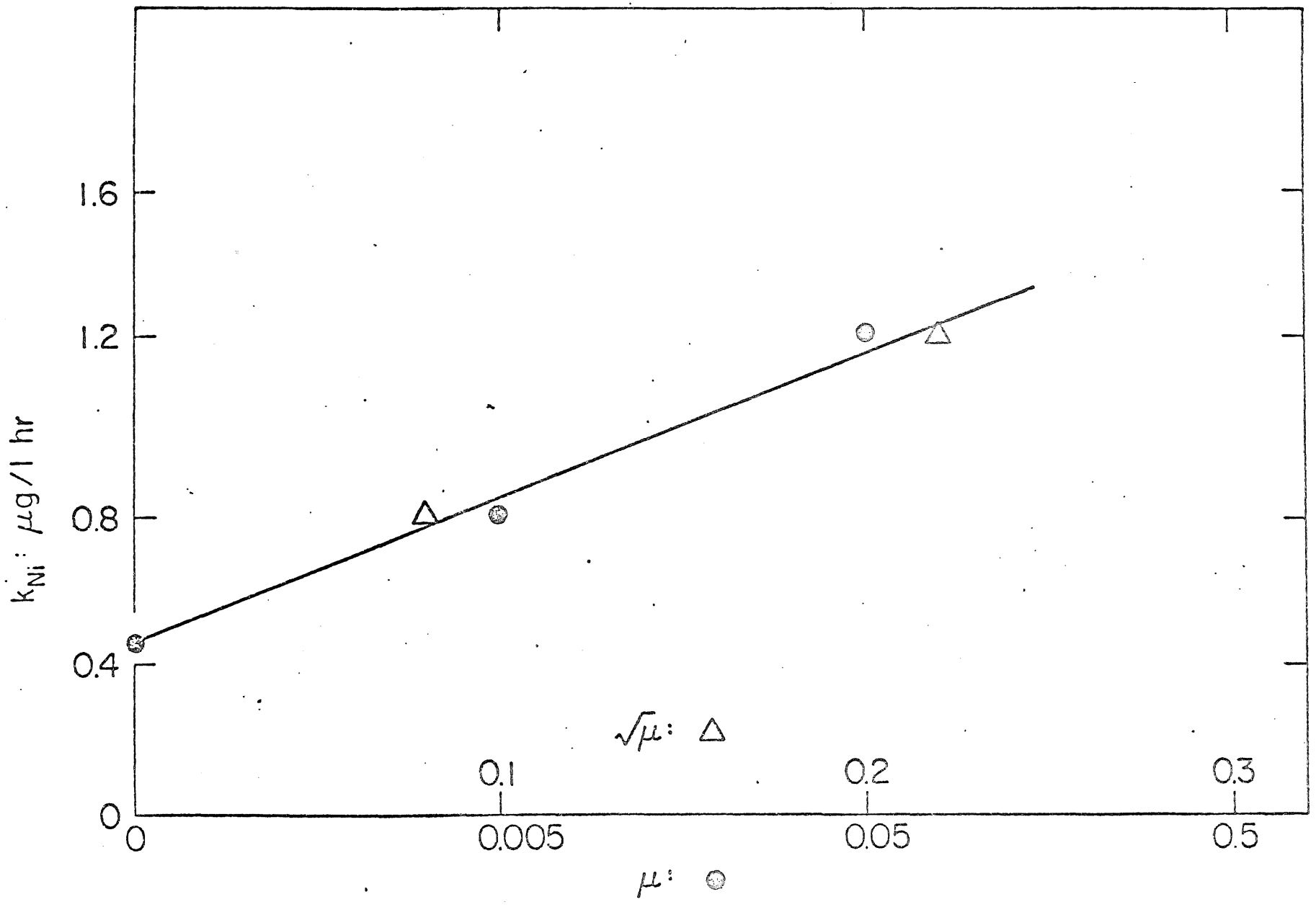
M	PH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{Sol}	Mg_{Tot}	SO_{Tot}
)[a]	7.0	0.19	1.0	1.2	0.56	0.06	0.61	1.1	0.39	1.5	6.5	7.8	14	0.71	5.7	6.5	16
).005[b]	7.0	0.67	1.4	2.1	0.50	0.01	0.51	2.7	0.04	2.7	9.8	9.0	19	1.2	7.0	8.2	19
).05[b]	7.0	0.42	2.6	3.0	0.94	0.01	0.95	4.1	0.05	4.2	6.7	11	18	0.97	7.0	8.0	21

a:t = 910 hrs.
b:t = 1030 hrs.

Table 3.7

Total Release: Effect of Citrate on Unmineralized Gabbro
Ad: Adsorbed & Precipitated, Sol: In Solution, Tot: Total Release
All Values in Moles * 10^6 per gram gabbro

[CIT]:M	PH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{Sol}	Mg_{Tot}	SO_{Tot}
1.3E-4	7.0	0.64	1.6	2.2	0.28	0.20	0.48	2.1	3.5	5.6	15	20	35	1.9	11	13	14
1.3E-3	7.0	0.09	5.3	5.4	0.16	0.95	1.1	4.8	130	140	18	19	37	5.4	38	44	43
1.3E-4	7.5	0.50	0.55	1.1	0.23	0.03	0.26	2.3	1.9	4.1	13	33	46	0.66	8.4	9.1	13
1.3E-3	8.2	0.28	4.8	5.1	0.12	0.86	0.98	4.4	23.0	39.0	13	29	41	1.9	12	14	11
1.3E-4	7.7	0.18	0.33	0.52	0.22	0.03	0.25	0.12	2.2	2.3	4.7	7.1	12	0.47	8.0	8.5	15
1.3E-3	8.3	0.22	0.02	0.24	0.36	0.12	0.47	2.5	9.0	12	6.5	6.9	13	1.9	9.9	12	11
)	6.9	0.33	2.6	3.1	0.17	0.02	0.19	1.9	0.03	1.9	10	18	29	0.88	8.6	9.5	14


1.6
1.2
0.8
0.4
0
k_{Ni}: μg/l hr
$\sqrt{\mu}$: △
0.1
0.2
0.3
0
0.005
0.05
0.5
μ: ○

3.7. Effect or Organic Ligands and Natural Waters

High concentrations of dissolved organic carbon have been observed in the natural waters surrounding the Erie Mining Company's Dunka Pit (1). Complexation capacity measurements show that a strong affinity exists between the dissolved components in the natural waters and added metals such as Cu, Pb, Cd and Zn. Most likely, organometallic complexes (2) are formed between the naturally-occurring organic material and the trace transition metals in the aqueous phase. Complexation should enhance the solubility and transport of metals such as Cu, Ni and Fe. Concentrations of these metals are normally controlled by their sulfide, hydroxide and carbonate solids.

The effect of synthetic and natural organic ligands on the weathering of silicate and sulfide minerals is well-documented (21-24). In general, the rate and extent of metal dissolution is enhanced in the presence of moderately strong chelating agents. This rate enhancement may arise from either a shifting of the equilibrium dissolution or from a surface catalytic influence. For example, if the rate of dissolution is limited by diffusion through a layer of reaction products on the surface of the sulfide mineral, chelation of a particular metal may accelerate the normally slow diffusion through the reacted zone. The other possibility is that an organometallic complex acts as a catalyst for the activation of molecular oxygen to a more reactive state and as a consequence accelerate the surface oxidation (40).

If the rate of dissolution is controlled by a solution phase oxidation of either Fe^{+2} or HS^{-} then shifting of the pre-equilibrium dissolution step will increase the apparent rate of oxidation. Using FeS as an example the following sequence of reactions can be hypothesized.

$$H^+ + FeS \underset{\leftarrow}{\overset{K_{so}}{\rightarrow}} Fe^{+2} + HS^- \tag{3.32}$$

$$Fe^{+2} + Y^{-2} \underset{\leftarrow}{\overset{B}{\rightarrow}} FeY \tag{3.33}$$

$$HS^- + O_2 \overset{k\ (slow)}{\rightarrow} HSO_2^- \tag{3.34}$$

$$HSO_2^- + O_2 \overset{fast}{\rightarrow} HSO_4^- \tag{3.35}$$

Since equation 3.34 represents the slow rate determining step in the reaction sequence, the rate of oxidative dissolution can be written as

$$\frac{-d\,[FeS]}{dt} = k\,[HS^-][O_2] \tag{3.36}$$

From the equilibrium expressions written in equations 3.32 and 3.33

$$[HS^-] = [H^+]\,K_{so}/[Fe^{+2}] \tag{3.37}$$

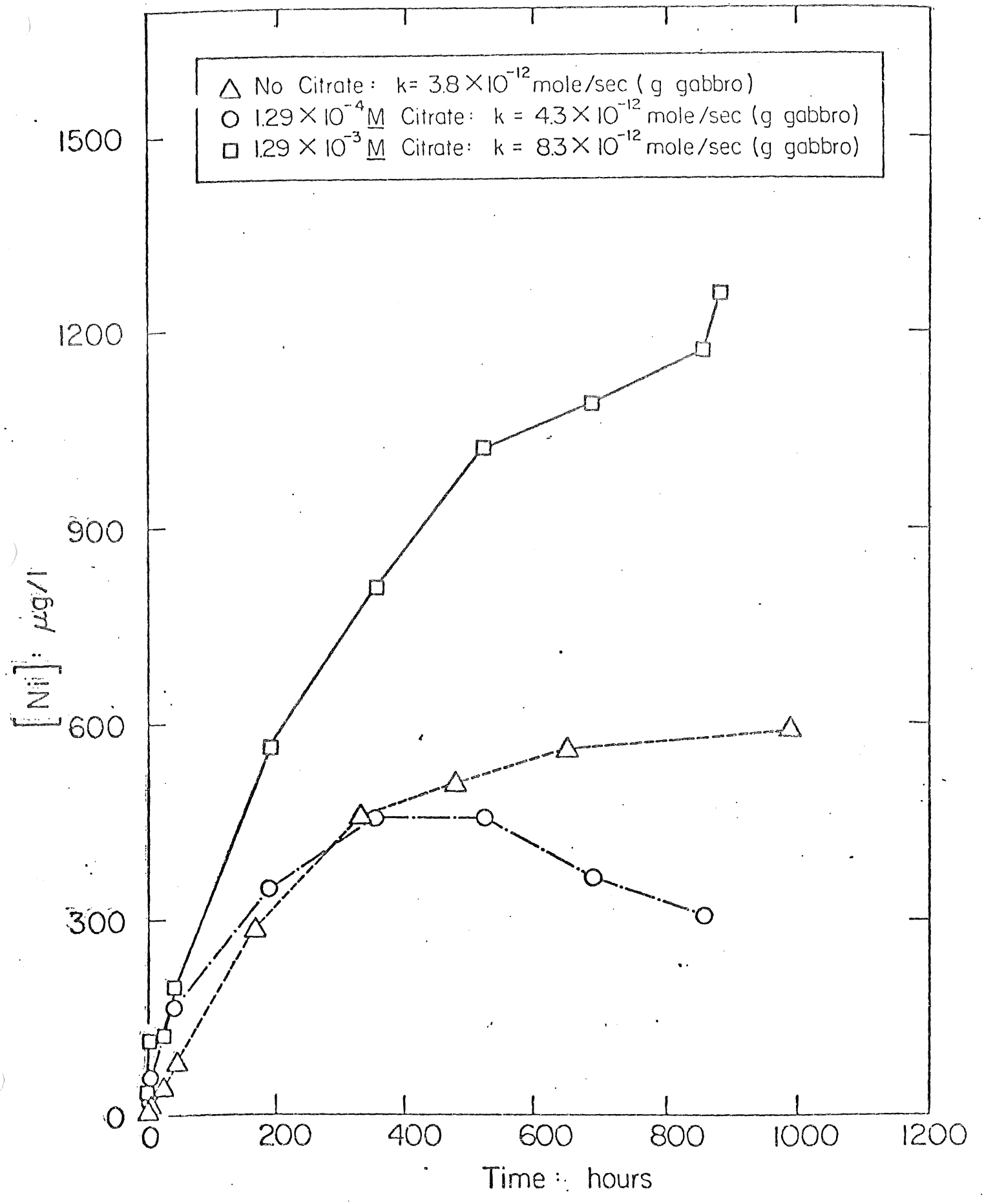
$$[Fe^{+2}] = [FeY]/B[Y^{-2}] \tag{3.38}$$

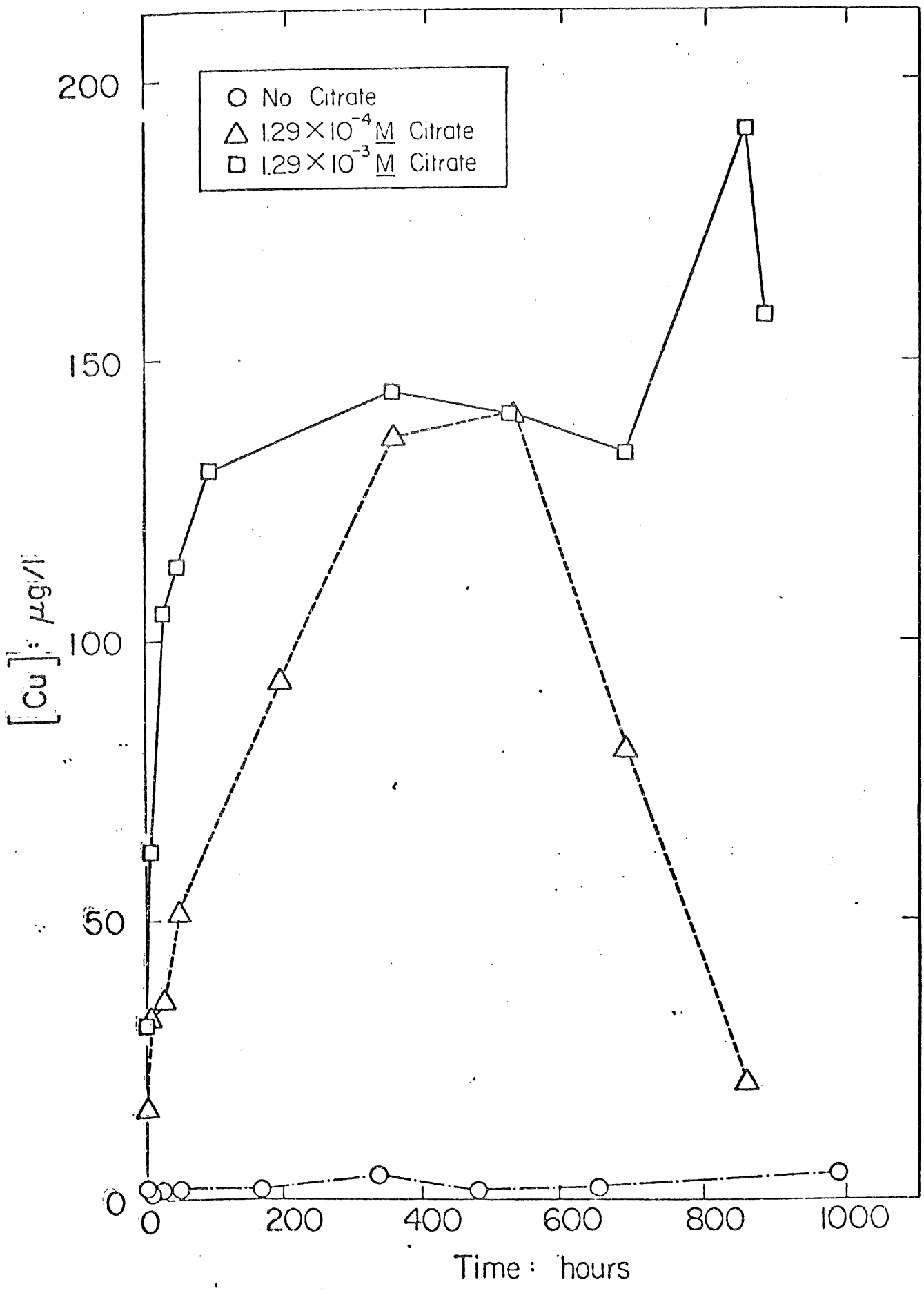
Substitution of these expressions into equation 3.34 yeilds

$$\frac{-d[FeS]}{dt} = \frac{kK_{so}B[Y^{-2}][O_2][H^+]}{[FeY]} \tag{3.39}$$

Equation 3.37 predicts that the rate of oxidative dissolution will be directly proportional to the ligand concentration, $[Y^{-2}]$. A similar rate expression should result if the complex FeY were an active catalyst for the surface oxidation of FeS.

To experimentally determine the impact of organic ligands on the rates of oxidative dissolution, two synthetic chelates, citrate and phthalate, and natural water plus peat bog organics were used in conventional leaching experiments according to procedures previously described. Experiments with citric acid, which is a highly soluble tri-carboxylic acid $\{HO_2CCH_2C(OH)(CO_2H)CH_2CO_2H\}$ with pk's of 3.13, 4.76 and 6.40 for the successive dissociation of the carboxylate protons,


△ No Citrate: k= 3.8 × 10^-12 mole/sec (g gabbro)
○ 1.29 × 10^-4 M Citrate: k = 4.3 × 10^-12 mole/sec (g gabbro)
□ 1.29 × 10^-3 M Citrate: k = 8.3 × 10^-12 mole/sec (g gabbro)
[Ni] : μg/l
0
300
600
900
1200
1500
0
200
400
600
800
1000
1200
Time : hours


No Citrate
1.29×10⁻⁴ M Citrate
1.29×10⁻³ M Citrate
[Cu] : μg/l
200
150
100
50
0
0
200
400
600
800
1000
Time : hours

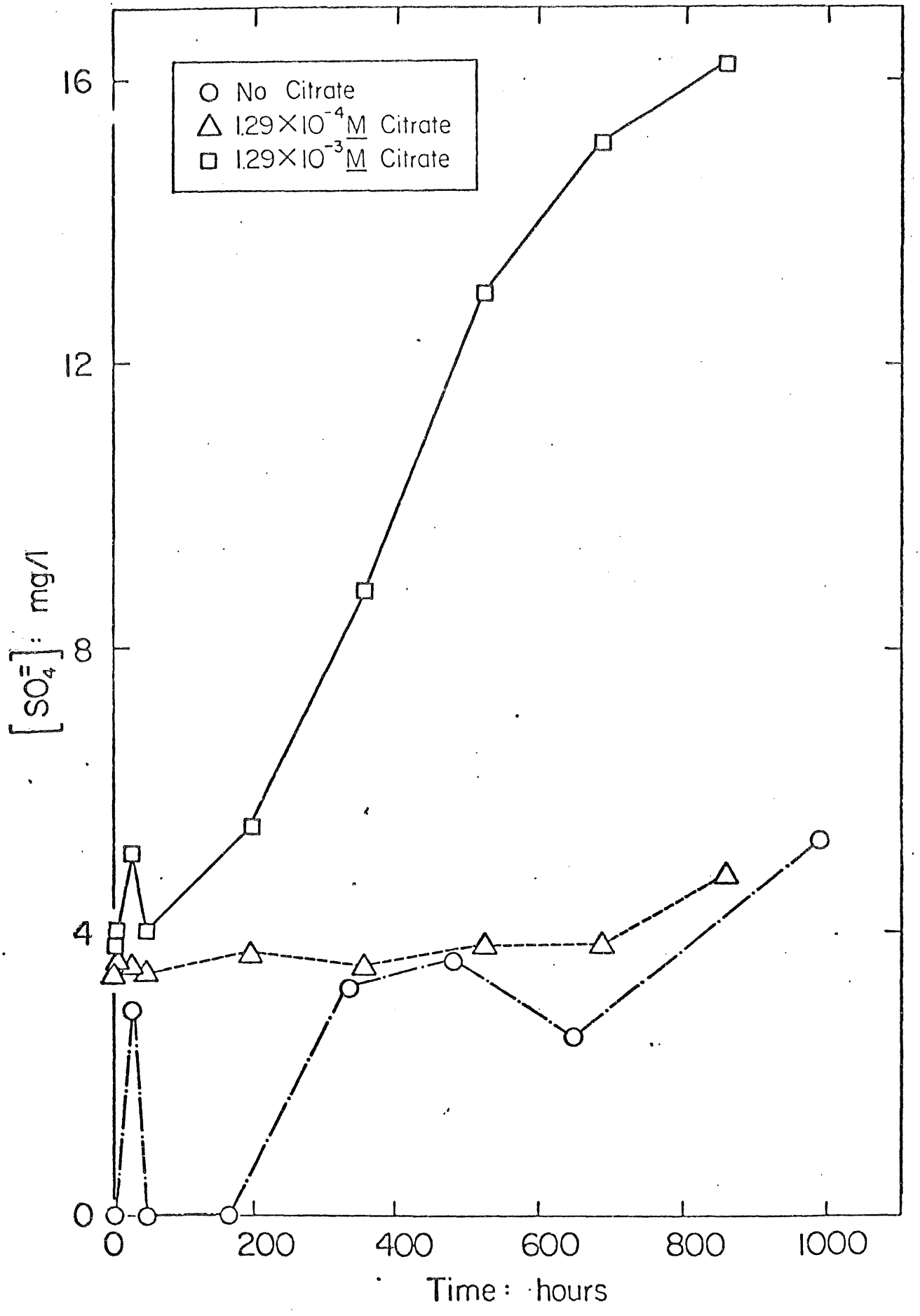

No Citrate
1.29×10⁻⁴ M Citrate
1.29×10⁻³ M Citrate
16
12
8
4
0
[SO₄⁻⁻]: mg/l
0
200
400
600
800
1000
Time: hours

Table 3.8

Total Release: Effect of Phthallic Acid
Ad: Adsorbed & Precipitated, Sol: In Solution, Tot: Total Release
All Values in Moles * 10^6 per gram gabbro

thallic id	PH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{So}	Mg_{Tot}	SO_{Tot}
	7.0	0.19	1.0	1.2	0.56	0.06	0.61	1.1	0.39	1.5	6.5	7.8	14	0.71	5.7	6.5	16
3E-4[b]	7.0	0.36	0.92	1.3	0.04	o.01	0.05	1.4	0.23	1.6	7.2	8.5	16	0.62	7.0	7.6	16
3E-3[b]	7.0	0.60	0.49	1.1	0.39	0.03	0.42	2.3	0.61	2.9	9.0	13.5	22	2.7	6.2	8.9	11

a. 912 hrs.
b. 1034 hrs.

Table 3.9

Total Release: Effect of Natural Waters
Ad: Adsorbed & Precipitated, Sol: In Solution, Tot: Total Release
All Values in Moles * 10^6 per gram gabbro

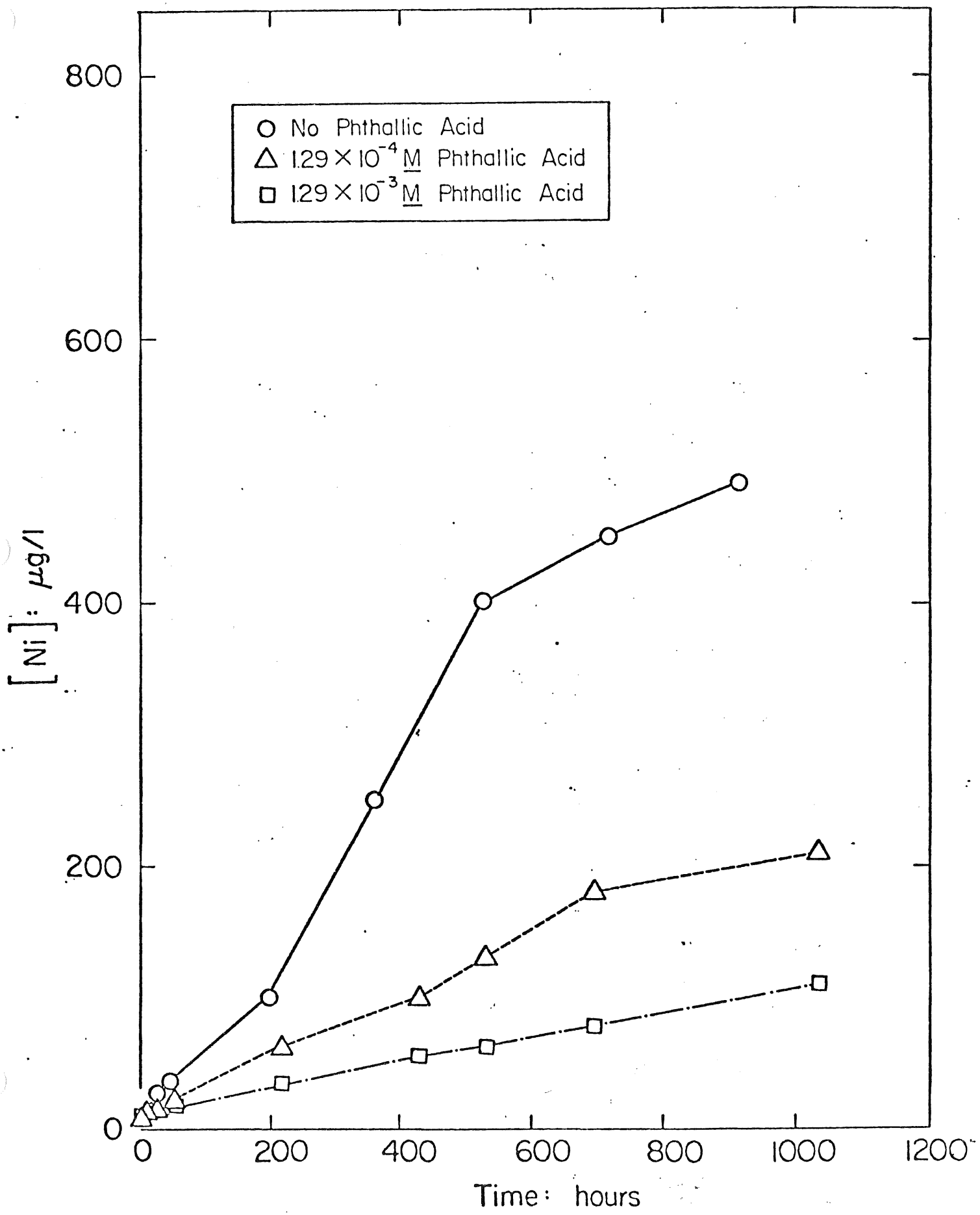
Water	PH	Ni_{Ad}	Ni_{Sol}	Ni_{Tot}	Cu_{Ad}	Cu_{Sol}	Cu_{Tot}	Fe_{Ad}	Fe_{Sol}	Fe_{Tot}	Ca_{Ad}	Ca_{Sol}	Ca_{Tot}	Mg_{Ad}	Mg_{Sol}	Mg_{Tot}	SO_{Tot}
Megapure	8.0	0.21	0.09	0.30	0.67	0.01	0.68	1.3	0.07	1.4	8.4	6.9	15	1.1	4.7	5.8	15
BOG		0.21	0.09	0.30	0.62	0.05	0.67	5.0	0.24	5.2	18	-1.6	16	5.3	-8.5	-3.2	4.3
Ground		0.37	0.07	0.44	1.3	0	1.3	9.9	0.01	9.9	34	-10	24	8.5	-3.5	5	6.6

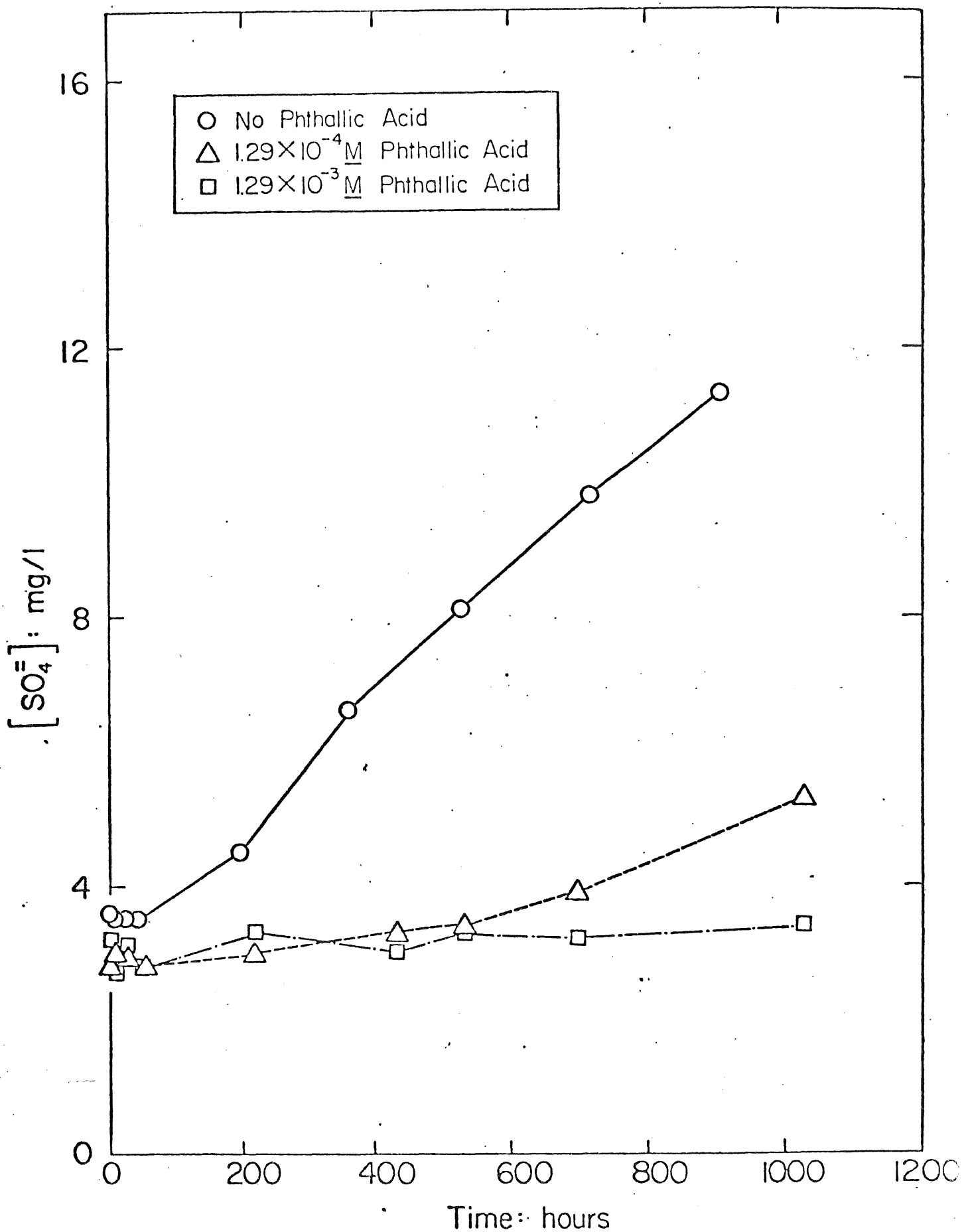
a:t = 912 hrs.
b:t = 1034 hrs.

were run at concentrations of 1.3×10^{-4} M and 1.3×10^{-3} M at a mass loading of 2.86 g/l of mineralized Gabbro adjusted to pH 7. At pH 7, which was maintained constant during the course of the reaction with the addition of acid or base, the citric acid added to the system is essentially non-protonated and would be expected to behave as a moderately strong chelate. Phthalic acid, an aryldicarboxylic acid, with successive pk's of 2.95 and 5.4, is a relatively weak bidentate ligand, which is commonly thought to be representative of the functionality found in aquatic humates (32). Reactions with phthalic acid added at concentrations of 1.3×10^{-4} M and 1.3×10^{-3} M were run with mineralized Gabbro loadings of 5.71 g/l at pH 8. At this pH the phthalic acid will be totally deprotonated and should exhibit weak to moderate chelating ability. Natural organic material present in bog water and ground water was tested with Gabbro loadings of 5.71 g/l adjusted to pH 8. All other conditions were identical to those used in previous experiments.

Results of these experiments are summarized in Tables 3.7-3.9, A.2, A.3, A.7 and A.8 and graphically depicted in Figures 3.21-3.27. The presence of citrate had a noticeable impact on rate and extent of mineral dissolution in addition to enhancing the solution phase mobility of the metals. However during the later stages of the reactions a sudden drop in the concentration of metals in solutions occurred. This occurrence was also observed by Schalscha et. al., (22) in their studies of silicate dissolution by organic chelates. Rapid dissolution of the silicate matrix may result in the solution being supersaturated in dissolved silica as SiO_2 or H_4SiO_4. At some point the soluble silica precipitates as amorphous, opaline silica is a discontinuous fasnion resulting in a sweep-floc removal of dissolved metals as coprecipitates or adsorbed species.

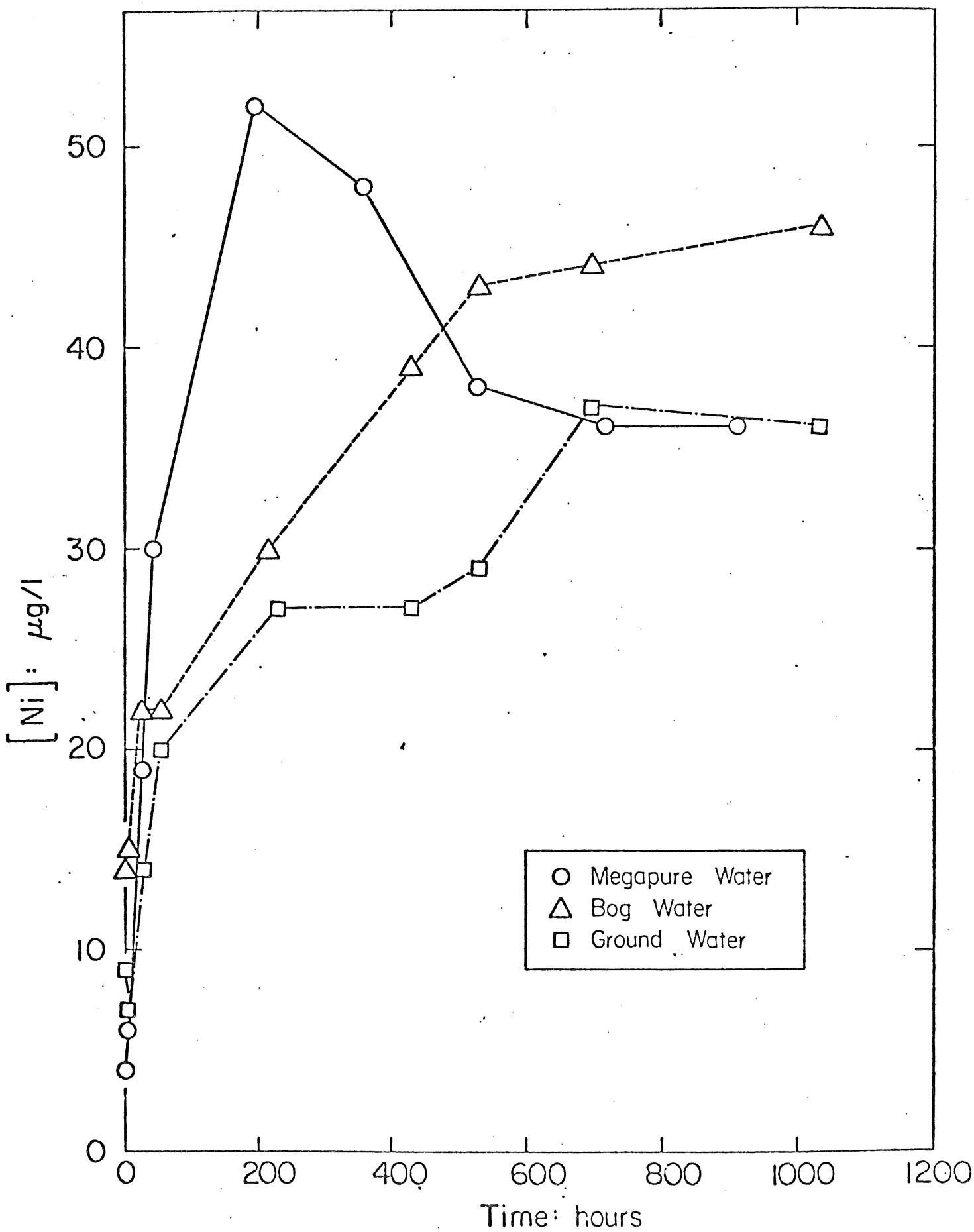
The effect of increasing citrate concentrations on the rate of appearance of Ni is illustrated in Figure 3.21. In these experiments linear dissolution

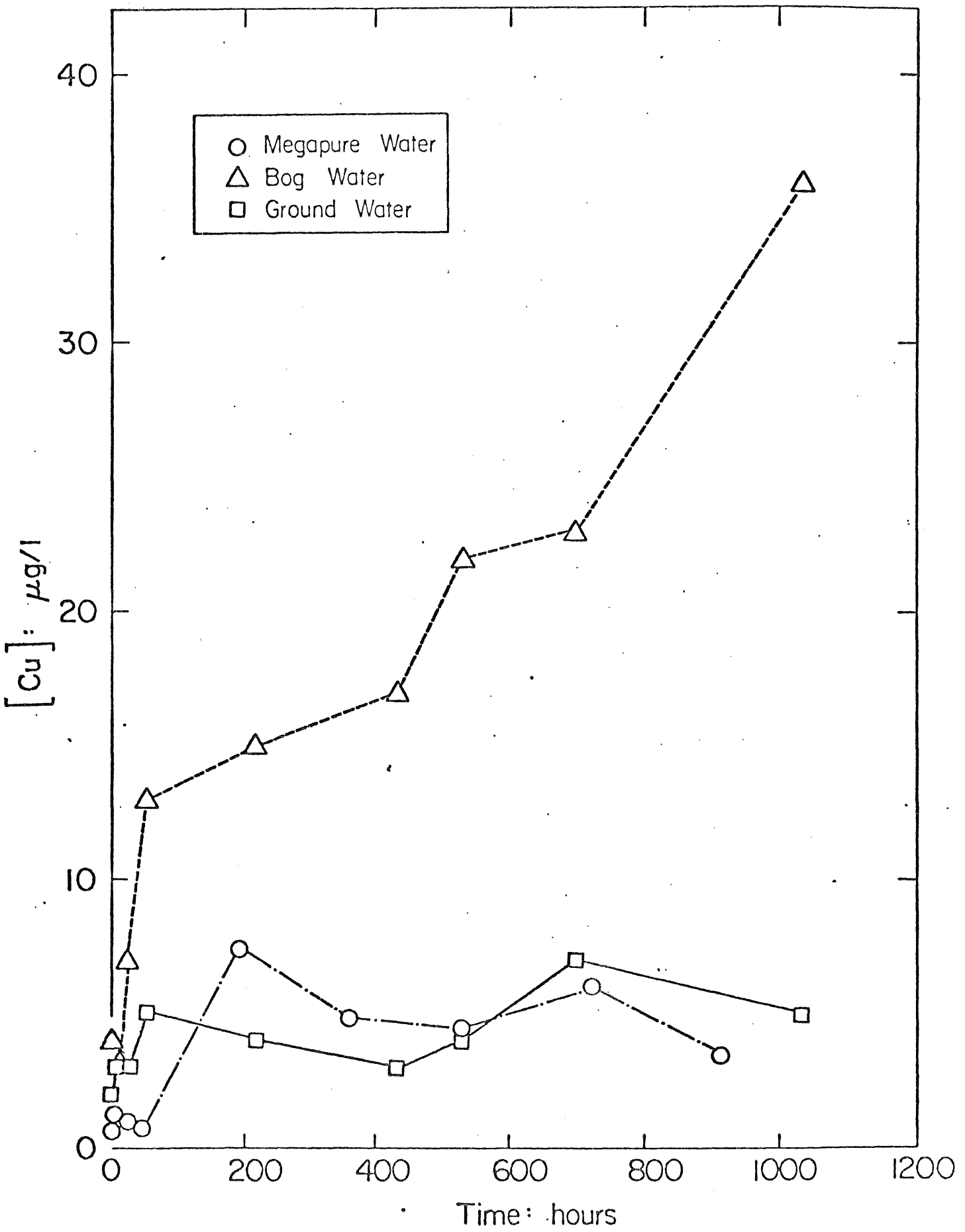

800
600
400
200
0
[Ni]: μg/l
O No Phthallic Acid
△ 1.29 × 10⁻⁴ M Phthallic Acid
□ 1.29 × 10⁻³ M Phthallic Acid
0
200
400
600
800
1000
1200
Time: hours


No Phthallic Acid
1.29×10⁻⁴ M Phthallic Acid
1.29×10⁻³ M Phthallic Acid
[SO₄⁼]: mg/l
Time: hours
0
4
8
12
16
0
200
400
600
800
1000
1200

kinetics were only observed for the first 100-200 hours. During later stages release appeared to be parabolic. Comparing the linear stages of Ni release shows that as the concentration of citrate increases tenfold the pseudo zero-order rate constant, k^1, approximately doubles. Similar trends are observed when these experiments are run in duplicate. From these results $k^1 \sim k_o$ [citrate]$^{0.3}$. Similar rate enhancements are shown for Cu release in Figure 3.22, although in later stages a sudden drop in concentration occurs due to the precipitation of silica. Support for the mechanism postulated in equations 3.32 to 3.35 is obtained from the results presented in Figure 3.23 which shows a dramatic increase in the rate of appearance of $SO_4^=$ as compared to the system with no citrate. This result would be predicted if equation 3.34 were rate limiting and equation 3.37 representative of the rate of dissolution.

The effect of phthallic acid on the apparent dissolution rates was dramatically different than citrate. This effect is predicted to some extent by equation 3.37 since B (phthalate) < B (citrate) with all other factors held constant. The effect of phthalate can be seen in Figures 3.24 and 3.25. Increasing concentrations of phthalate have an inhibitory effect on the rate of release of Ni and the rate of appearance of sulfate. Chen and Morris (33) have shown that the rate of autoxidation of sulfide in aqueous solutions is inhibited by a number of organic reagents. A similar inhibition in Ni release rate is observed when bog and ground water from the Dunka Pit region is used, (see Figures 3.26 and 3.27) although bog water does enhance the rate and extent of Cu release. This suggests a specificity of the complexing agents present in bog water for Cu. Preferentially binding of Cu over Ni for the same ligand is predicted thermodynamically.


50
40
30
20
10
0
[Ni]: μg/l
0
200
400
600
800
1000
1200
Time: hours
Megapure Water
Bog Water
Ground Water


Megapure Water
Bog Water
Ground Water
[Cu]: μg/l
Time: hours
0
10
20
30
40
0
200
400
600
800
1000
1200

Chapter 4

Results: Silicate Dissolution

4.1

The dissolution of the alumino-silicate minerals (primarily plagioclase) in continuous mix batch reactors was investigated as a function of particle surface area, solution pH, dissolved oxygen concentration, organic ligand concentration, and ionic strength. Results are presented in terms of a time-dependent chemical release as affected by the above dependent variables. Experiments were performed under natural conditions of constant temperature (20°C) and atmospheric pressure. Therefore, the dissolution rates and constants may be extrapolated to environmental conditions.

4.2

Kinetic Treatment

The incongruent dissolution of alumino-silicate minerals in aqueous solution under natural conditions has been described by a sequential four-stage process in which H^+ initially replaces surface cations followed by stages which were represented by the expressions (Busenberg and Clemency, 1976):

$$C = k_e t^n \qquad \text{2nd stage; competing processes} \qquad (4.1)$$

$$C = k_p t^{1/2} + A \qquad \text{3rd stage; parabolic kinetics} \qquad (4.2)$$

$$C = k_l t + B \qquad \text{4th stage; linear kinetics} \qquad (4.3)$$

The above expressions are derived from the general equation $C = kt^n$ where

$$C = \text{concentration} \qquad (4.4)$$

k = overall rate constant

t = time

n - integer between p and 1.

A single power law of the above type was used to describe the release (surface

flux) of Ca^{+2} and Mg^{+2} in continuous-mix batch reactors from aluminosilicates in Duluth Gabbro. The limited number of data points makes the four-stage delineation impractical. A power law equation of the type ($C = kt^n$) was used as an estimation of the release kinetics over the time span of the experiment to ∿ 1000 hours.

4.3

Effect of Particle Surface Area

The effect of particle surface area on Ca^{+2} and Mg^{+2} release with time was investigated in two separate experiments. Initially 2, 10, 50 and 100 g/l suspensions of unmineralized gabbro (-200 mesh) were subjected to the mineral batch leaching procedure for periods up to 750 hours. The observed concentrations of Ca^{+2} and Mg^{+2} increased with time at each mass loading, and increased in separate reactors with increased loading (Figure 4.1). The results are not unambiguous as the pH of the solution also increased with increased gabbro loading and may have resulted in changes in the dissolution mechanism.

The rates of Ca^{+2} and Mg^{+2} release decreased non-linearly with increased loading as shown below:

Loading	$k_{Ca^{+2}}$	$k_{mg^{+2}}$
2 $g.l^{-1}$	2.4	2.9
10 "	.51	.009
50 "	.06	.06
100 "	.009	.03

$k_{Ca^{+2}}$ (mole . sec^{-1} . g^{-1}) x 10^5;

$k_{Mg^{+2}}$ (mole . sec^{-1} . g^{-1}) x 10^6

The rate of Ca^{+2} release varied from (.009-2.4) x 10^{-5} mole . sec^{-1} . g^{-1} while Mg^{+2} rates varied from (.009-2.9) x 10^{-6} moles . sec^{-1} . g^{-1}.

Interpretation of the release behavior was complicated by the concurrent pH variations observed at increased loading.

To determine the effect of particle size on Ca^{+2} and Mg^{+2} release, four discrete size fractions at a fixed mass loading (2.86 $g.l^{-1}$) were leached for ∿ 890 hours. The four size fractions examined were $D_p \leq 74$ μm, 74 μm $< D_p \leq 420$ μm, 420 $< D_p \leq 840$ μm, and 840 μm $< D_p \leq 2380$ μm. Particle surface areas were estimated to be 1700, 190, 62 and 25 cm^2/g (Section 3). Elemental analyses (Table 3.2) show that Ca and Mg are enriched in the 840 μm $< D_p \leq 2380$ m particle size fraction, and decrease by factors of ∿ 2 and 4, respectively, in the smallest size fraction.

The concentrations of Ca^{+2} and Mg^{+2} and the dissolution rate of the silicate matrix increase with decreasing particle size (Figure 4.2). The cation release behavior is characterized by a rapid initial release, presumably due to the replacement of surface cations by H^+ and a subsequent buildup of the secondary product layer. After ∿ 50 hours, the concentrations of Ca^{+2} (Figure 4.2) and Mg^{+2} tend to plateau to a constant value except for the < 74 μm particles, which continue to release Ca^{+2} at a linear rate. In these experiments, final pH varied from 6.0 to 6.5, and was not a major factor in the concentrations observed.

To determine the sensititivy of k_{As} to changes in the particle surface area, an equation of the following type was developed (Section 3):

$$\log k' = \log k + \ell \log A_s \qquad 4.5$$

where k = rate constant

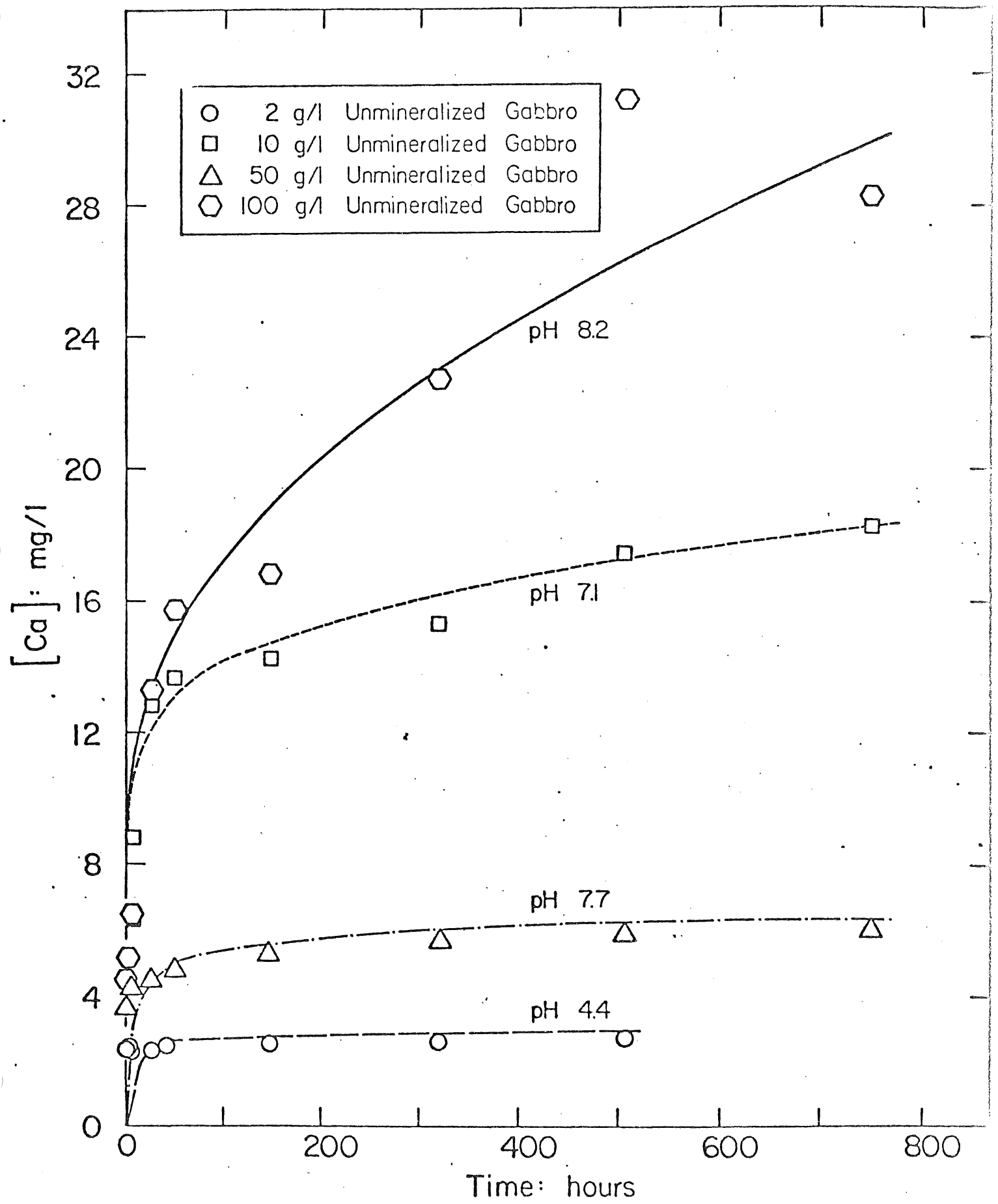
A_s = surface area

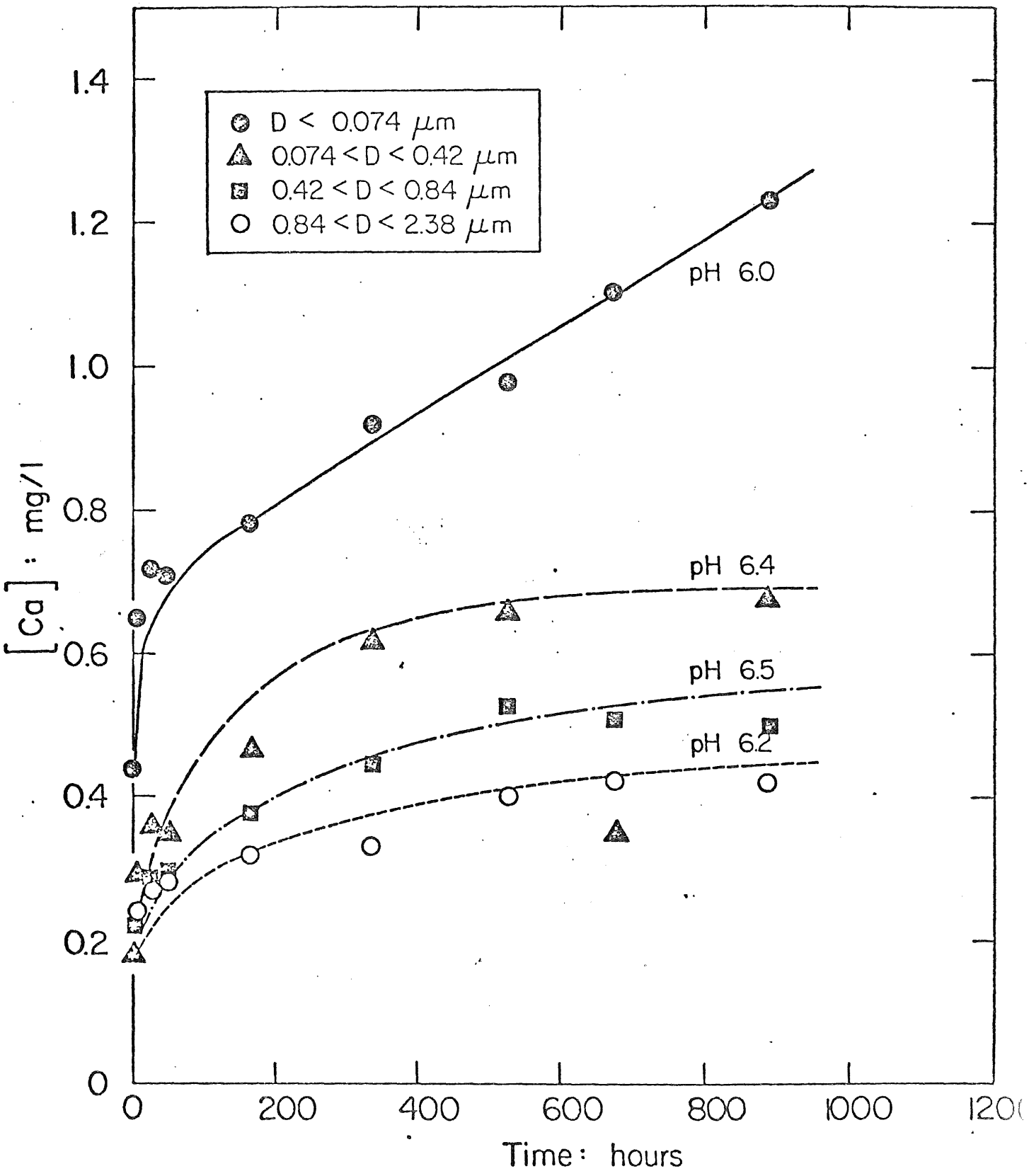
ℓ = reaction order

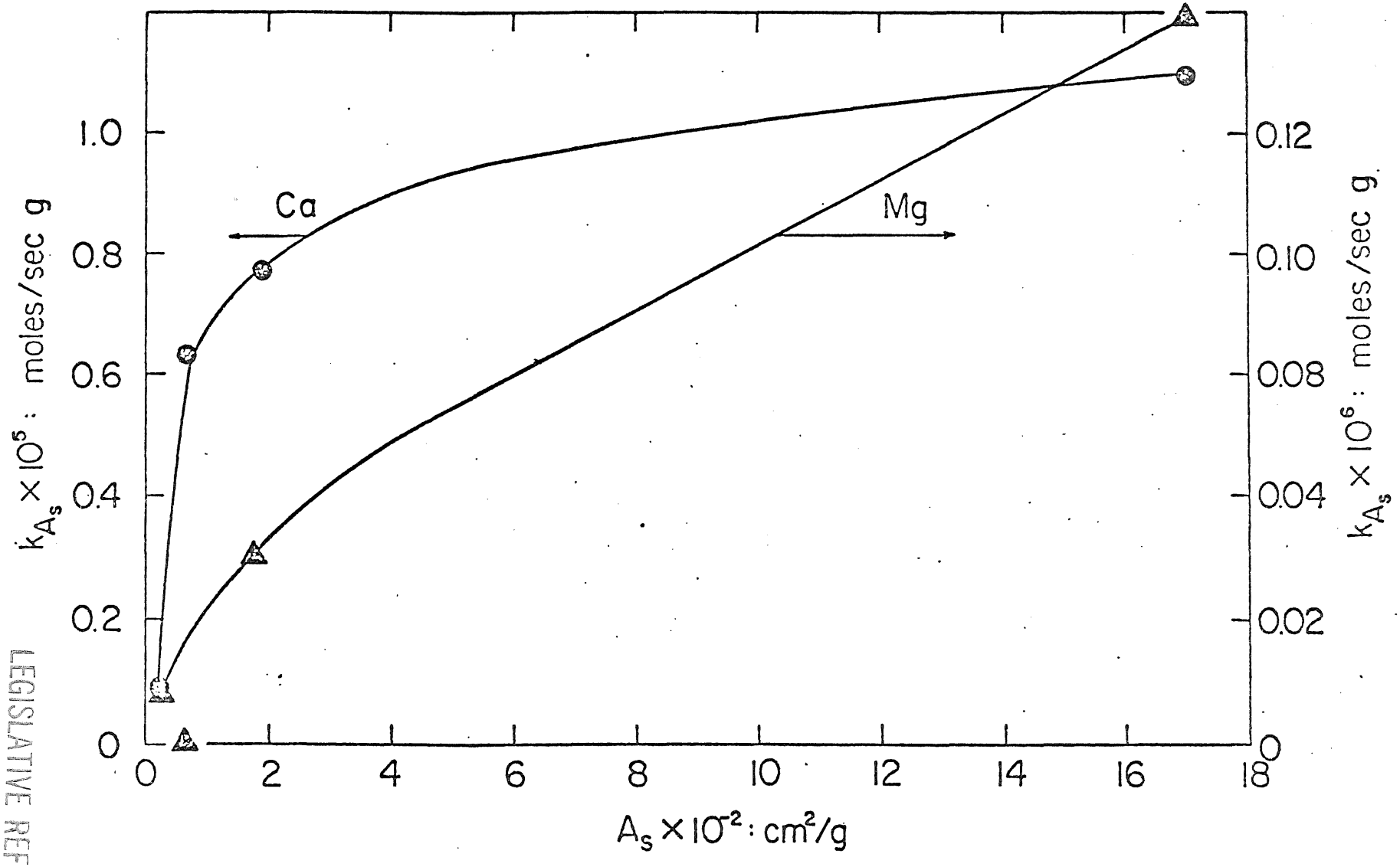
Equation 4.2 is a linear function, and if log k' is plotted versus log A_s,

the reaction order, ℓ, can be determined for the dependence of A_s on dissolution rate. The k_{As} values are plotted in Figure 4.3 while equation 4.2 is plotted in Figure 4.4 with k values obtained from Table . The reaction order, ℓ, for the effect of particle surface area, determined as the slope of the line in Figure 4.3 was $\sim$ 2.0 for the release of Ca^{+2} and 0.5 for the release of Mg^{+2} for $A_s \leq 200\ cm^2 . g^{-1}$. A discontinuity in the log k_{As}-log A_s plots was observed for both Ca^{+2} and Mg^{+2} for particles < 740 μm in particle size. In the case of Ca^{+2}, release rates became smaller than expected, while Mg^{+2} release rates increased. The increase in Mg^{+2} release rate may result from its enhanced concentration in the smallest size particle. The surface area loading for which particle size effects on release rates were greatest appears to be $\sim$ 200 $cm^2.g^{-1}$ gabbro. The differences in reaction rates and orders describing Ca^{+2} and Mg^{+2} release may be dependent on surface concentration, bulk solid concentration, diffusional transport through the secondary product layer and aqueous chemical removal processes. The data in Table 1.2 describing the mineralogical composition of the gabbro suggests that Ca^{+2} is largely derived from plagioclase and clinopyroxene while Mg^{+2} has olivine and clinopyroxine as its origin. The surface exchangeable Ca^{+2} is likely in higher concentration than Mg^{+2}.

At the conclusion of the experiment, from 60-90% of the Ca^{+2} released and > 90% of the Mg^{+2} released appeared in solution as soluble cation (Table 3.3). However, as the A_s available for dissolution decreased, the percentage of surface-associated Ca^{+2} (sorbed/precipitated) increased. The discontinuities observed in the $k_{Ca^{+2}}$ and $k_{Mg^{+2}}$ may be related to the tendency for formation of secondary precipitates.


2 g/l Unmineralized Gabbro
10 g/l Unmineralized Gabbro
50 g/l Unmineralized Gabbro
100 g/l Unmineralized Gabbro
pH 8.2
pH 7.1
pH 7.7
pH 4.4
[Ca]: mg/l
Time: hours
0
4
8
12
16
20
24
28
32
0
200
400
600
800


D < 0.074 μm
0.074 < D < 0.42 μm
0.42 < D < 0.84 μm
0.84 < D < 2.38 μm
pH 6.0
pH 6.4
pH 6.5
pH 6.2
[Ca] : mg/l
Time : hours
0
0.2
0.4
0.6
0.8
1.0
1.2
1.4
200
400
600
800
1000



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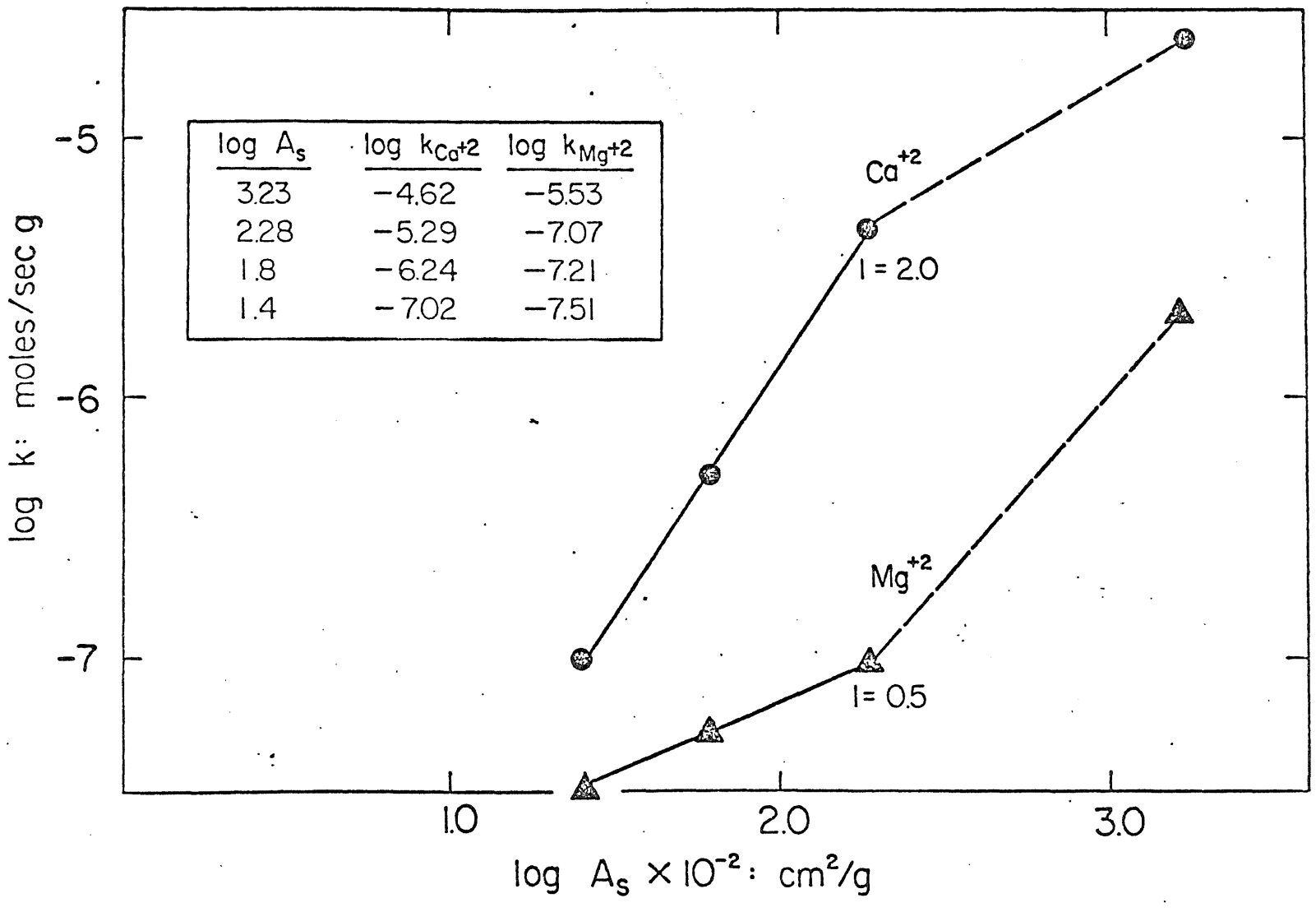


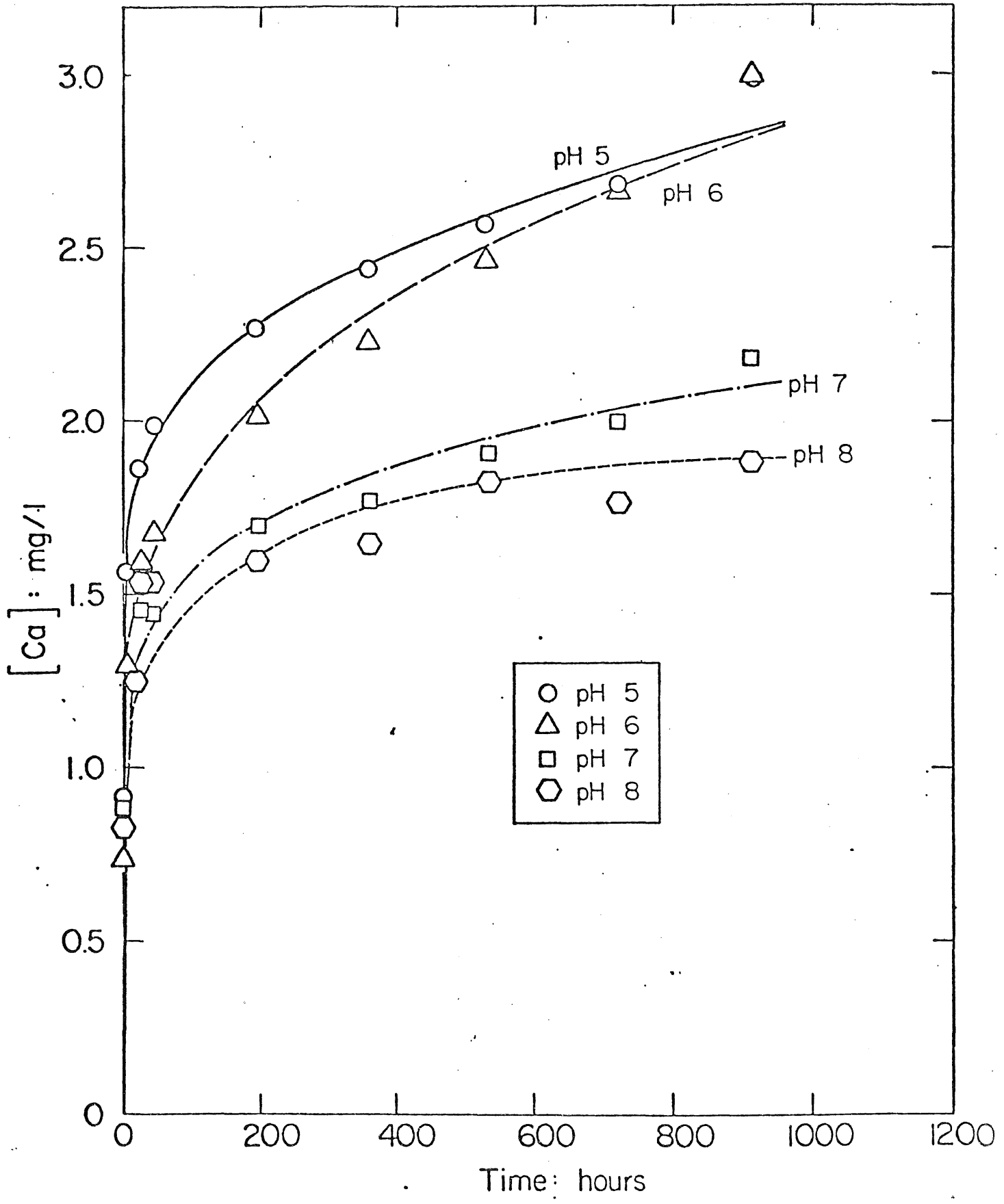
log A_s	log $k_{Ca^{+2}}$	log $k_{Mg^{+2}}$
3.23	−4.62	−5.53
2.28	−5.29	−7.07
1.8	−6.24	−7.21
1.4	−7.02	−7.51

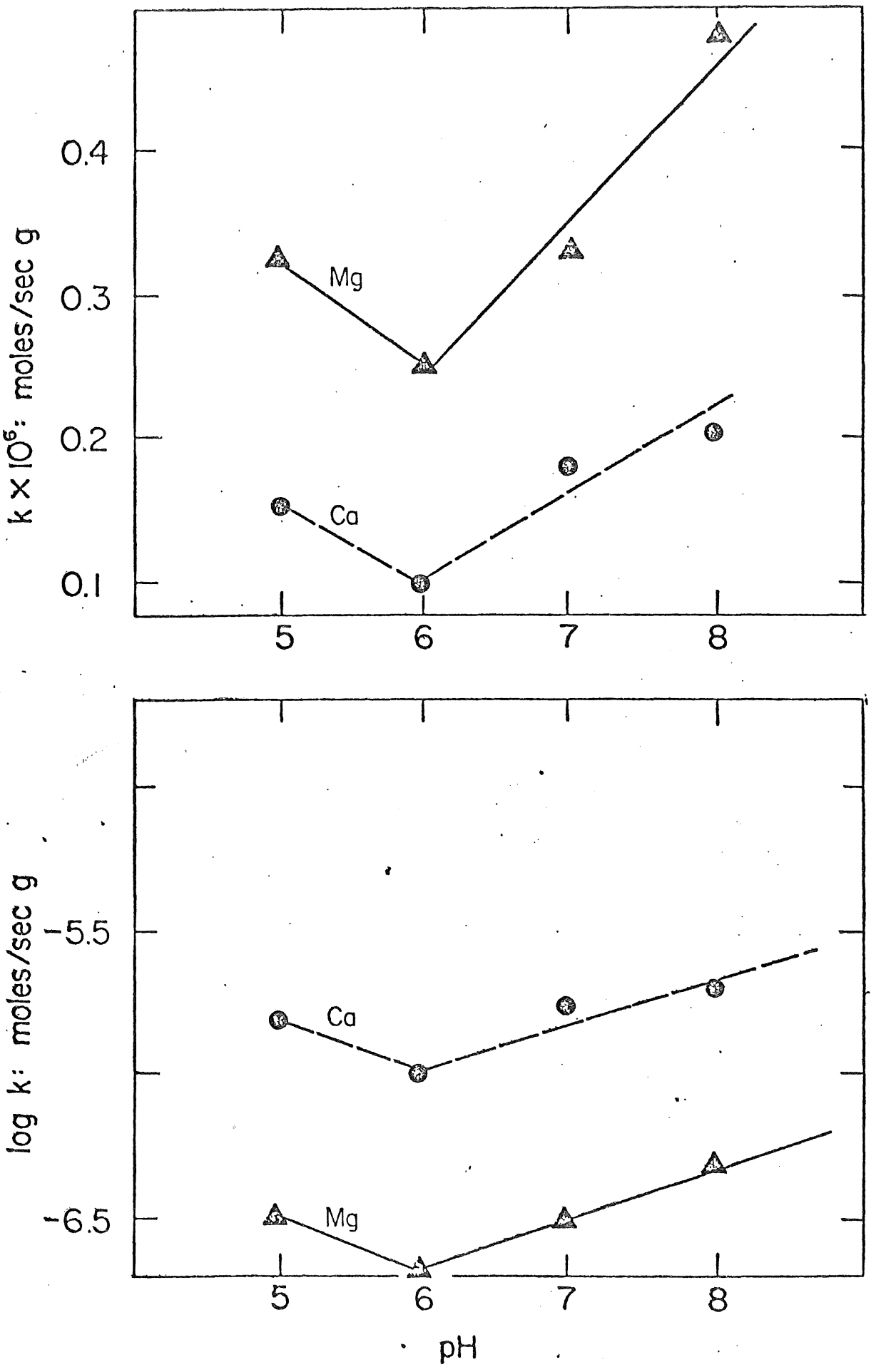
4.4

Effect of pH

The results of previous investigations (Section 1.6) suggest that the net effect of decreasing pH is to increase the extent of alumino-silicate dissolution. In order to determine the effect of solution acidity on silicate dissolution in our studies (Ca^{+2}, Mg^{+2} release), 5.71 $g.\ell^{-1}$ of ≤ 74 μm mineralized gabbro was leached in each of four reactors in which the solution was maintained at pH 5, 6, 7 and 8, respectively, by addition of strong acid or base. Dissolved oxygen was maintained at ∿ 9 mg/l and the temperature was 17°C. The results of the pH study are plotted in Figure 4.5 for Ca^{+2} release; similar results were obtained for Mg^{+2} release. The release curves are characterized by an initial rapid release of Ca^{+2} which tends to plateau at Ca^{+2} concentrations between 1.8 and 2.8 $mg.\ell^{-1}$. The quantity of Ca^{+2} and Mg^{+2} appearing in solution (Figure 4.5; Table 3.5) with time increases with decreasing pH. At low pH, less than 10% of the Ca^{+2} released is associated with the particle surface, but as the pH increases to 7 and 8, up to 56% of the total released Ca is sorbed to the particle surface or precipitated in a new solid phase. In contrast, less than 20% of the Mg^{+2} is associated with the solid phase at pH 8. A possible explanation of this behavior is that Ca^{+2} diffusing through the secondary product layer rapidly precipitates out as $Ca(OH)_2$ or $CaCO_3$ at high pH, but remains in solution at low pH. The total quantity of Ca^{+2} released increased with increasing pH while Mg^{+2} exhibited the reverse trend (Table 3.5). Aqueous chemical removal processes apparently control Ca^{+2} observed in solution, while solid dissolution exerts a greater control on Mg^{+2} concentrations.

The rate of Ca^{+2} and Mg^{+2} release to solution increases with increasing pH from pH 6 to 8, and increases below pH 6 (Figure 4.6). Preliminary experi-


3.0
2.5
2.0
1.5
1.0
0.5
0
[Ca] : mg/l
0
200
400
600
800
1000
1200
Time: hours
pH 5
pH 6
pH 7
pH 8
pH 5
pH 6
pH 7
pH 8


0.4
0.3
0.2
0.1
k × 10⁶: moles/sec g
Mg
Ca
5
6
7
8
-5.5
-6.5
log k: moles/sec g
Ca
Mg
5
6
7
8
pH

ments performed to study the pH effect (Table A) showed that Ca^{+2} release rates reached 2.9×10^{-5} moles . sec^{-1} . g^{-1} gabbro at pH 4.15, while the $k_{Ca^{+2}}$ achieved at pH 8 was only 0.2×10^{-5} moles . sec^{-1} . g^{-1}. The increased rate at low pH is likely due to the acid-assisted dissolution of the silicate surface preventing the buildup of a protective layer; therefore, no diffusion limited transport. Also, low pH inhibits formation of solid carbonates and hydroxides.

At solution values of pH 5-8, the $k_{Ca^{+2}}$ ranged from $(0.1\text{-}0.2) \times 10^{-5}$ moles . sec^{-1} . g^{-1} gabbro and the $k_{Mg^{+2}}$ varied from $0.3\text{-}0.5 \times 10^{-6}$ moles . sec^{-1} . g^{-1} gabbro. The rate of release of Ca^{+2} and Mg^{+2} is relatively independent of pH in the range pH 5-8 in contrast to the $k_{Ni^{+2}}$ values observed (Section 3.5). To determine the sensititivy of $k_{Mg^{+2}}$ and $k_{Ca^{+2}}$ on pH, a kinetic expression of the type below was developed (Section 3.5):

$$k' = k_H[H^+]^n \qquad 4.6$$

$$\log k' = \log k_H + n \log [H^+] \qquad 4.7$$

$$\text{where } k_H = kA_s^{\ell}[O_2]^m$$

The reaction order, n, was determined to be 2.9 for Ca^{+2} release, and 0.14 for Mg^{+2} release at pH 6-8, using a least-squares fit. Since Ca^{+2} and Mg^{+2} release rates increase with increasing pH (6 to 8), and Ni^{+2} release rates show the opposite behavior (Section 3.5), it is reasonable to assume that the dissolution kinetics of the silicate minerals have no major effect on metal sulfide dissolution/oxidation.

4.5

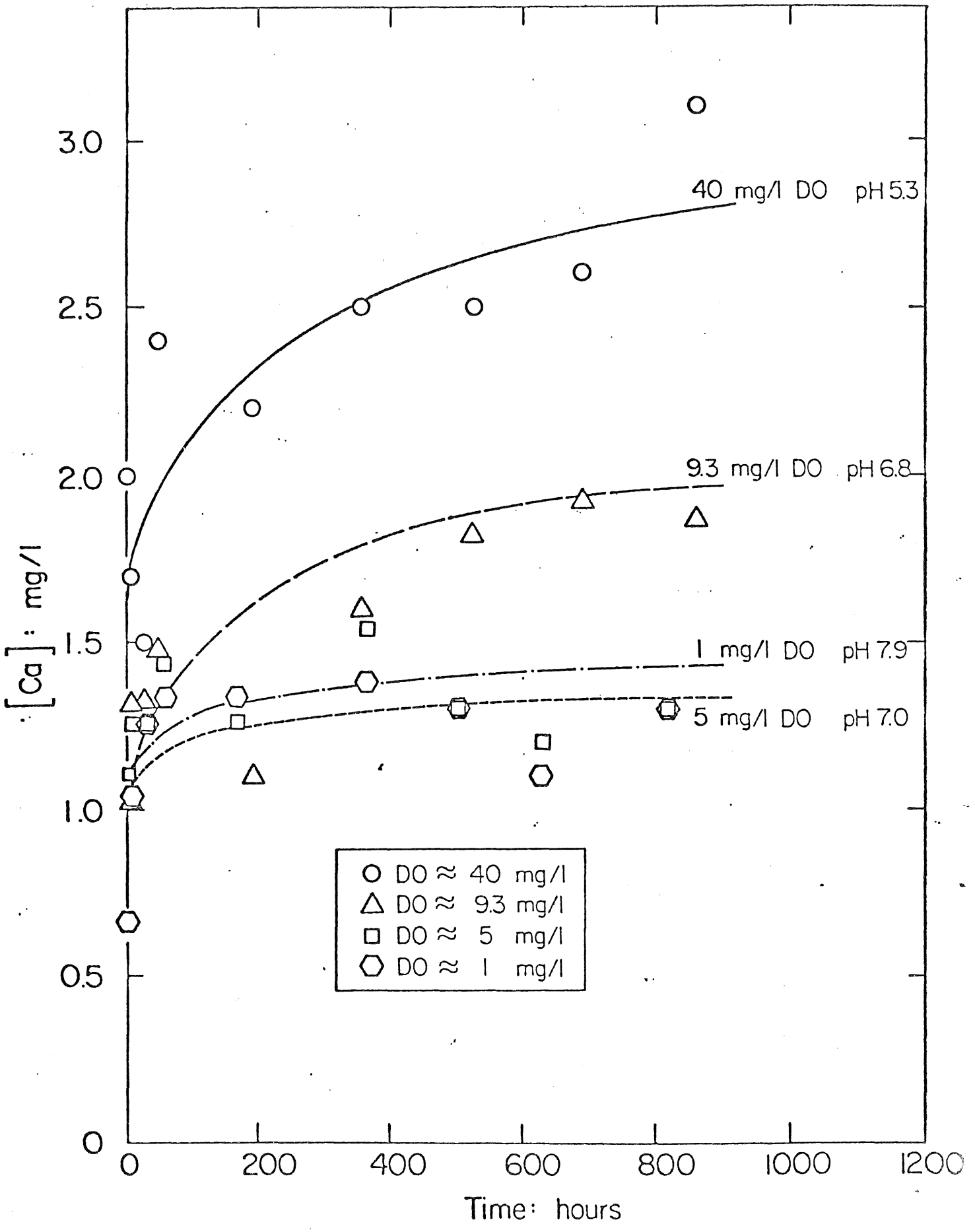
Effect of Dissolved Oxygen

The effect of dissolved oxygen on the dissolution of alumino-silicate minerals must of necessity be indirect if observed at all. In Section 4.4,

the solution pH was found to be relatively unimportant in the dissolution of silicate minerals in the pH range of natural waters. However, the data indicate that depression of the pH below 6 increases the reaction rate by up to a factor of 10 for Ca^{+2} from natural conditions. Increases in solution acidity may result from the increased oxidative dissolution of metal sulfides (Section 3.4). Acidity from this source may increase the release of Ca^{+2} and Mg^{+2} to solution.

To determine the effect of dissolved oxygen on alumino-silicate dissolution in the presence of finely-divided metal sulfide particles, 2.86 $g.\ell^{-1}$ of -200 mesh gabbro were leached in mixed reactors at 17°C under conditions of variable dissolved oxygen concentrations 1-1, 5, 9.3 and 40 $mg.\ell^{-1}$. The dissolved oxygen concentrations achieved by variable mixing of pure O_2, compressed air and N_2 in the batch reactors. Solution pH was monitored but not controlled so as to prevent sample disturbance.

The release of Ca^{+2} from mineralized gabbro as a function of dissolved oxygen concentration is shown in graphical form in Figure 4.7, and in tabular form in Table 3.4 (and Table A-). The results show that the concentration of Ca^{+2} in solution generally increases with increasing dissolved oxygen. However, the Ca^{+2} increase occurred concommitantly with a decrease in solution pH resulting from the oxidation of reduced S species and Fe^{+2}. Total Ca^{+2} and Mg^{+2} release (Table 3.4) either remain constant or decrease slightly with increasing dissolved oxygen. As expected, the dissolved oxygen content of the leaching solution has little or no effect on alumino-silicate dissolution, unles solution pH is altered by metal sulfide oxidation.


40 mg/l DO pH 5.3
9.3 mg/l DO pH 6.8
1 mg/l DO pH 7.9
5 mg/l DO pH 7.0
DO ≈ 40 mg/l
DO ≈ 9.3 mg/l
DO ≈ 5 mg/l
DO ≈ 1 mg/l
[Ca]: mg/l
Time: hours
0
0.5
1.0
1.5
2.0
2.5
3.0
200
400
600
800
1000
1200

4.6

Effect of Ionic Strength

The general effect of increasing salt content in heterogeneous reactions is to decrease the solubility product of the dissolving solid, thereby increasing its solubility. Equation 3.33 (Section 3.6) illustrates that increasing ionic strength increases solubility, and that the effect of ionic strength is especially important for solids containing multivalent cations. According to the extended Debye-Huckel equation (3.33), the decrease in the solubility product should be proportional to $\sqrt{\mu}$.

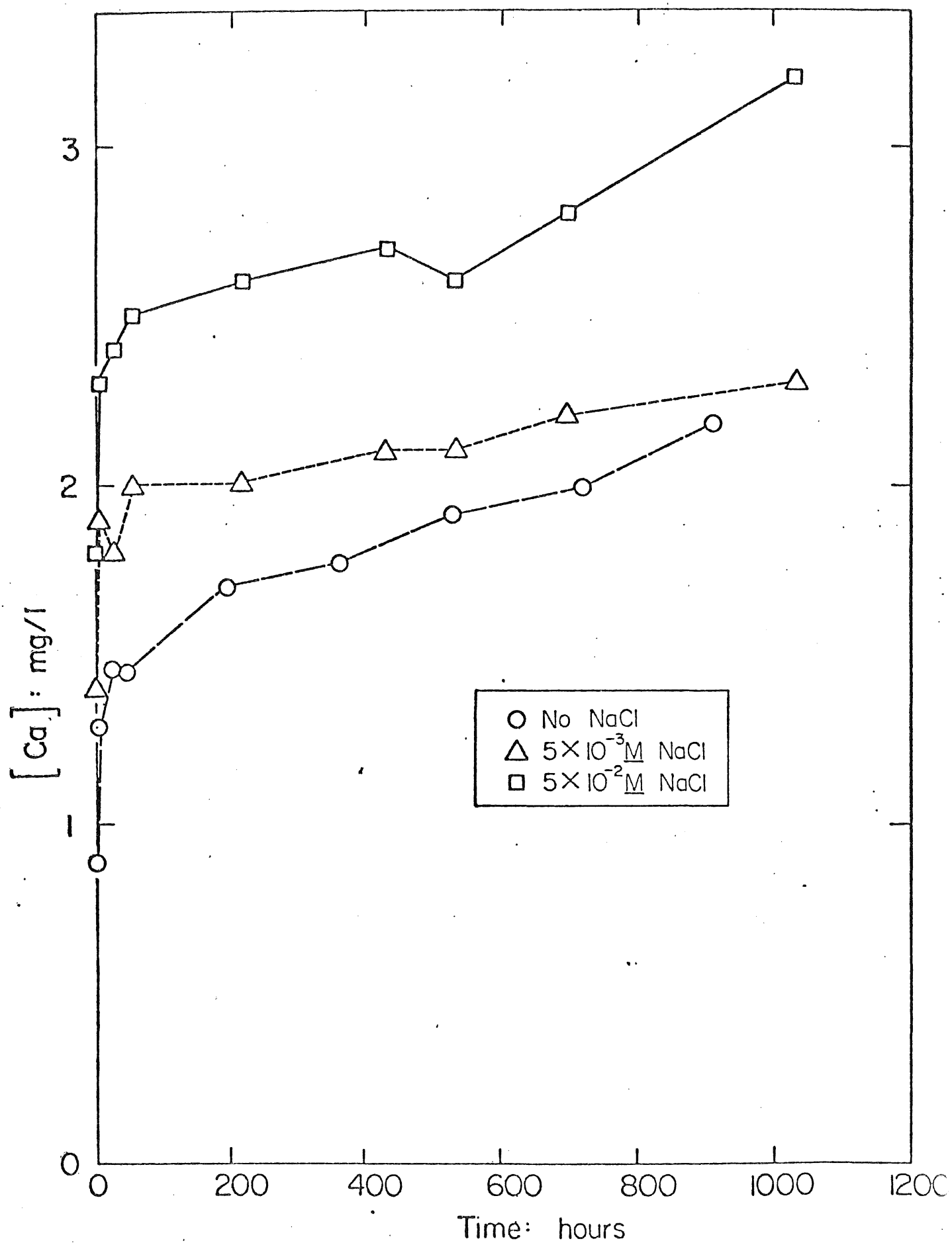
To determine the sensitivity of k_μ to ionic strength, batch reactors containing 5.71 g. ℓ^{-1} gabbro were adjusted to 0, 5 x 10^{-3}M and 5 x 10^{-2}M NaCl. The reactor solution was maintained at pH 7, 17°C and 9.3 mg.ℓ^{-1} D.O.

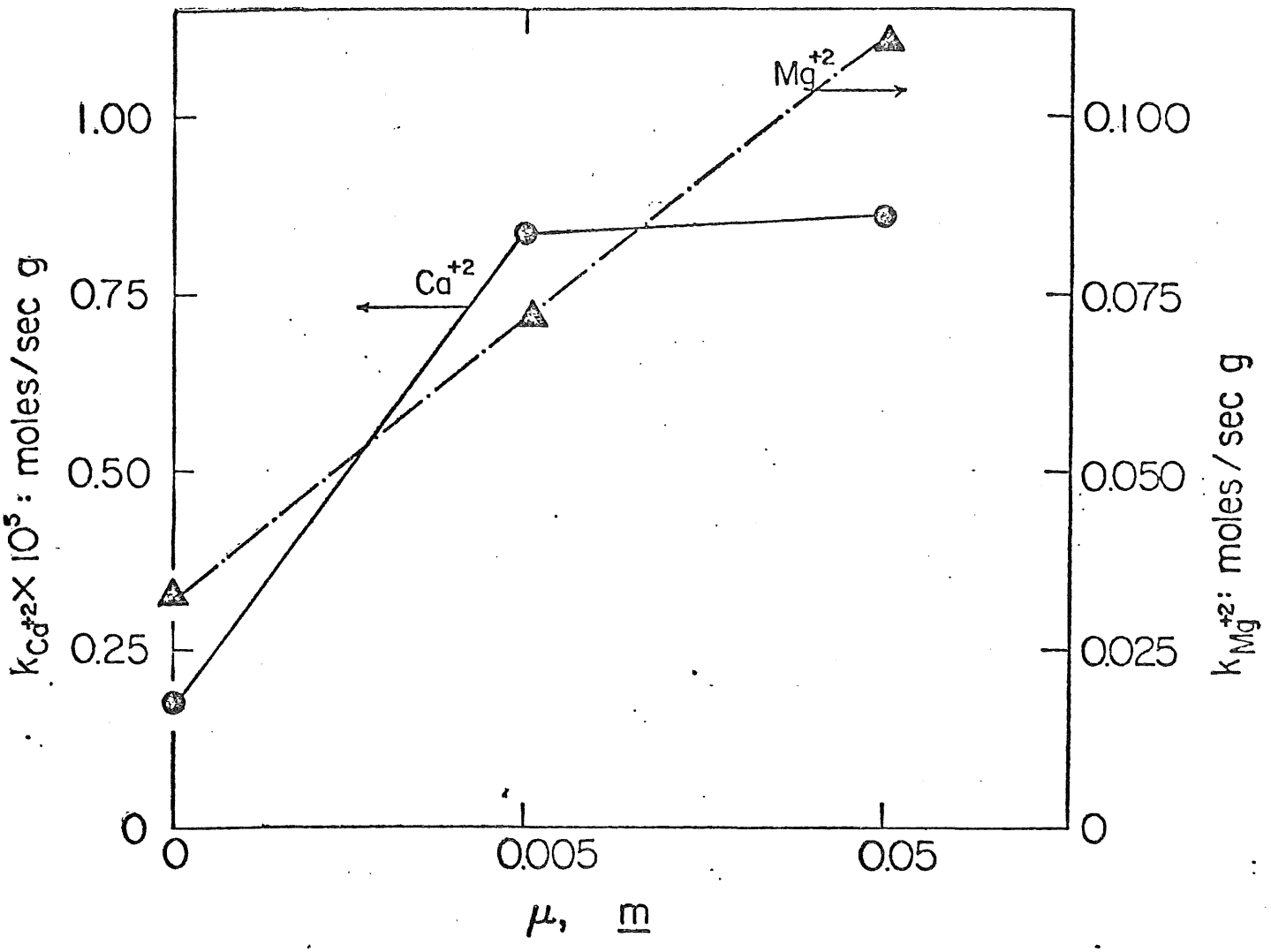
The results of the ionic strength experiment are plotted as a function of time in Figure 4.8. Silicate dissolution increased slightly with increasing μ as indicated by the release of Ca^{+2} (Figure 4.8) and Mg^{+2} (Table 3.6) to solution at pH 7. The greatest increase in Ca^{+2} and Mg^{+2} concentration in solution was observed in the reactor having the highest ionic strength. Increased salt content was expected to decrease the initial ion exchange reaction between surface cations and solution H^+, but Na^+ would probably also be effective in displacing surface Ca^{+2} and Mg^{+2}.

The rates of Ca^{+2} ($k_{Ca^{+2}}$) and Mg^{+2} ($k_{Mg^{+2}}$) release increased with increase in ionic strength as shown in Figure 4.9. The following rate constants were observed:

μ	$k_{Ca^{+2}}$	$k_{Mg^{+2}}$
0	.18	.033
.005M	.84	.072
.05M	.87	.11

(moles.sec^{-1}.g^{-1} gabbro) x 10^{-5}


3
2
1
0
[Ca]: mg/l
0
200
400
600
800
1000
1200
Time: hours
No NaCl
5×10⁻³M NaCl
5×10⁻²M NaCl


1.00
0.75
0.50
0.25
0
0.100
0.075
0.050
0.025
0
0
0.005
0.05
Ca^{+2}
Mg^{+2}
$k_{Ca^{+2}} \times 10^5$: moles/sec g
$k_{Mg^{+2}}$: moles/sec g
μ, m

The results indicate that the enhancement in Ca^{+2} release reaches a maximum at 0.005 M NaCl and plateaus, whereas Mg^{+2} release increases linearly at least up 59 0.05 M NaCl. The increase in $k_{Ca}+2$ at low μ may be responding to surface-associated cations, while the increase in $k_{Mg}+2$ with increasing μ may result from increased dissolution of the bulk solid. This interpretation is consistent with that made based on pH variations (Section 4.5).

4.7

Effect of Organic Ligands and Natural Waters

The effect of synthetic and naturally-occurring organic ligands on the weathering of silicate and mineral sulfides has been reported (21-24). In general, the extent and rate of silicate dissolution is enhanced relative to control conditions. The effect of organic ligands is to form weak to moderately-strong complexes or chelates with structural or surface-associated cations by ligand exchange reactions of the type (Table 1.3):

$$Al_2Si_2O_5(OH)_4\ (s) + 2Y^- \rightleftarrows 2\ AlY^-\ (aq) + 2H_4SiO_4 + H_2O \qquad (4.8)$$

In this example, kaolinite reacts with the organic ligand, Y^-, to form a soluble complex with structural Al atoms, thereby degrading the alumino-silicate. Similar reactions can be written for plagioclase minerals whereby Al and surface-associated cations (Ca^{+2}, Mg^{+2}) are complexed, resulting in an increased rate and extent of mineral dissolution. In experiments performed in this study, the appearance of Ca^{+2} and Mg^{+2} in the reactor solution at a faster rate than observed in the control was used to demonstrate the influence of organic ligands. A further discussion of organic ligand effects on minerals dissolution may be found in Section 3.7.

The effect of organic ligands on silicate dissolution was demonstrated by leaching gabbro under natural conditions by citrate, phthalate and natural

waters consisting of bog and ground water. Citrate and phthalate were chosen to be representative of functionalities thought to exist in natural waters. Citric acid is an aliphatic tri-carboxylic acid ($HO_2CH_2CCOHCO_2H\ CH_2CO_2H$) having pKa's equal to 3.13, 4.76 and 6.40 at 25°C and zero ionic strength for the successive acid dissociation constants. Under solution conditions used in this experiment (pH 7), greater than 90% of the citric acid is expected to be fully ionized. The reactions pertinent to the binding of Ca^{+2} and Mg^{+2} by citrate are:

$$Ca^{+2} + Y^{\equiv} \rightleftarrows CaY^{-} \qquad pK = 3.50 \qquad (4.9)$$

$$Ca^{+2} + HY^{=} \rightleftarrows CaHY^{\circ} \qquad pK = 2.10 \qquad (4.10)$$

$$Mg^{+2} + Y^{\equiv} \rightleftarrows MgY^{-} \qquad pK = 3.37 \qquad (4.11)$$

$$Mg^{+2} + HY^{=} \rightleftarrows MgHY^{\circ} \qquad pK = 1.92 \qquad (4.12)$$

Citric acid is a moderately strong ligand relative to both Ca^{+2} and Mg^{+2}, and binds each with about equal strength.

Phthalic acid is an aromatic di-carboxylic acid (benzene, 1-2, dicarboxylic acid) having successive acid dissociation constants of $pK_1 = 2.95$, $pK_2 = 5.10$.

Therefore, phthalic acid is fully de-protonated under the solution conditions used in these experiments. The reaction pertinent to the binding of Ca^{+2} by phthalate is:

$$Ca^{+2} + Y^{=} \rightleftarrows CaY^{\circ} \qquad pK = 2.42 \qquad (4.13)$$

The stability constant for the formation of MgY° is not available.

To determine the influence of organic ligands and natural waters on the dissolution rate of silicate minerals, citrate or phthalate was added at concentrations of 0, 1.3×10^{-4} M to reactors containing 2.86 $g.\ell^{-1}$. The solution was adjusted to pH 7 and maintained throughout the experiment. The control reactor (pH 6.9) contained no ligand, and the pH was not adjusted due

to its proximity to pH 7. In preliminary experiments, amorphous precipitates were observed which removed trace metals but had lesser effects on major cations.

Results of these experiments are summarized on Tables 3.7-3.9, A.2, A.3, A.7 and A.8, and represented graphically in Figures 4.10 and 4.11. Citrate tended to increase the initial rate and extent of Ca^{+2} release (Figure 4.9), but concentrations decreased in later stages of the reaction (after 400 hours). Schalscha, et al. (22) observed this behavior in their experiments and suggested that it would be explained by a breakdown of the organic complex, adsorption onto mineral reaction products or other complicated factors. In this study, precipitation of a new mineral phase (amorphous metal silicate) may be responsible. Up to 400 hours, the rate of appearance of Ca^{+2} in solution was enhanced over the control. The effect of the 1.3×10^{-4}M ligand concentration was much less than the effect of the 1.3×10^{-3}M solution, and did not differ markedly from the control after 400 hours.

Figure 4.11 shows the release of Ca^{+2} with time in the presence of increasing concentrations of phthalate. In this case, Ca^{+2} concentrations were only slightly enhanced over the control up to $\sim$ 100 hours. After 100 hours, increasing concentration of ligand inhibited Ca^{+2} release to solution. The appearance of a brown precipitate in acidified, filtered samples withdrawn from the reactors may have caused the apparent concentration decrease. In any case, phthalate does not apparently enhance silicate dissolution.

The effect of increasing citrate and phthalate concentrations on the rate of Ca^{+2} and Mg^{+2} release is shown in Figure 4.12. Assuming Ca^{+2} release

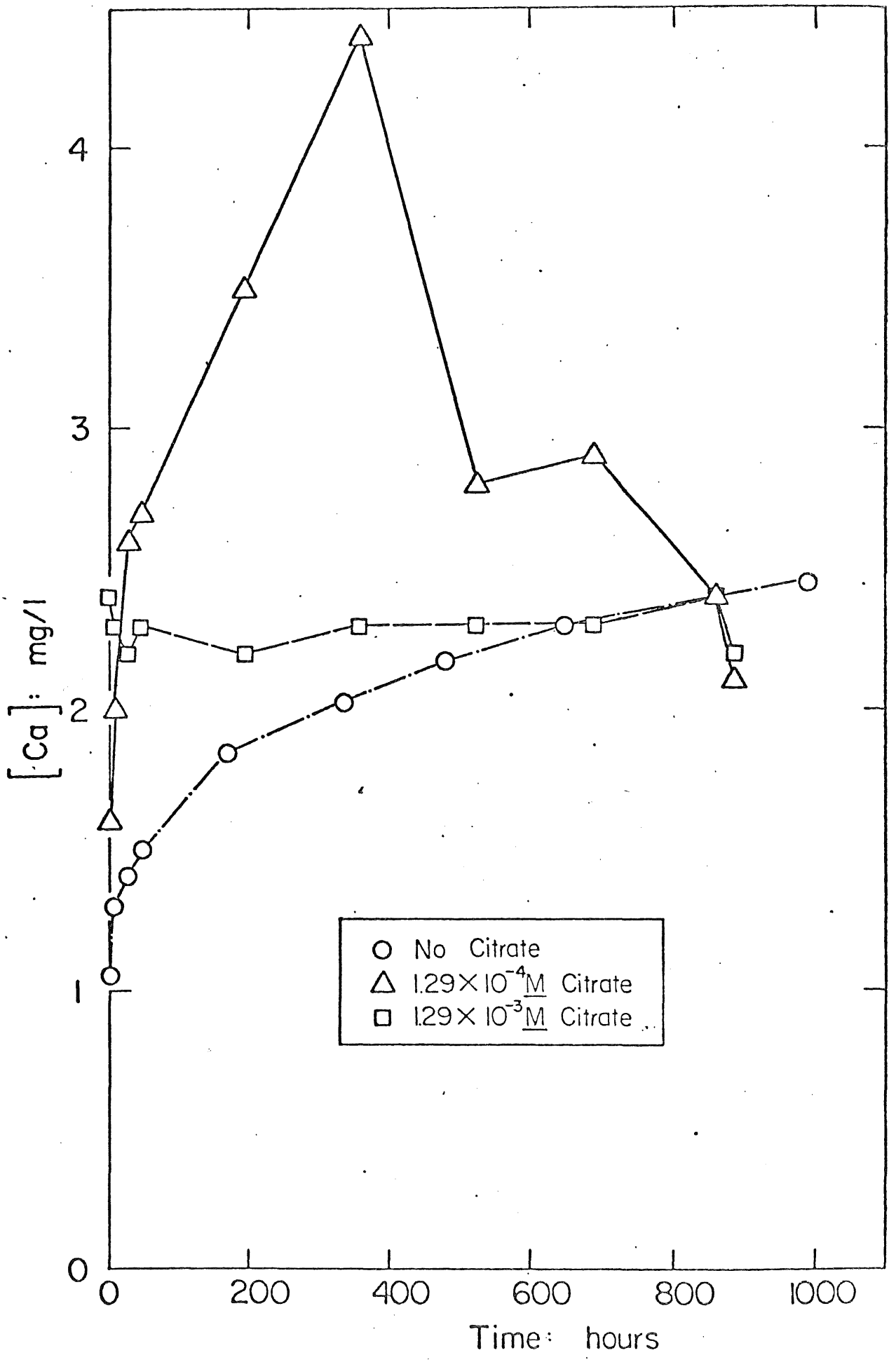
as influenced by citrate concentrations can be represented by the equatic.. :.' = $k_0[CiT]^n$, then log k' = log k + n log [CiT]. As indicated in the lower half of Figure 4.11, a plot of log k vs log [Cit] yields a line of slope equal to ∿ 0.3-0.4. This corresponds to an approximate doubling of the observed k as the [CiT] increased by ten times. The Mg^{+2} release rate remained essentially unchanged with increasing [CiT].

The rates of Ca^{+2} release in the presence of citrate ranged from .77-2.1 x 10^{-5} moles . sec^{-1} . g^{-1} while Mg^{+2} rates ranged only from .33-.58 x 10^{-6} moles . sec^{-1} . g^{-1}.

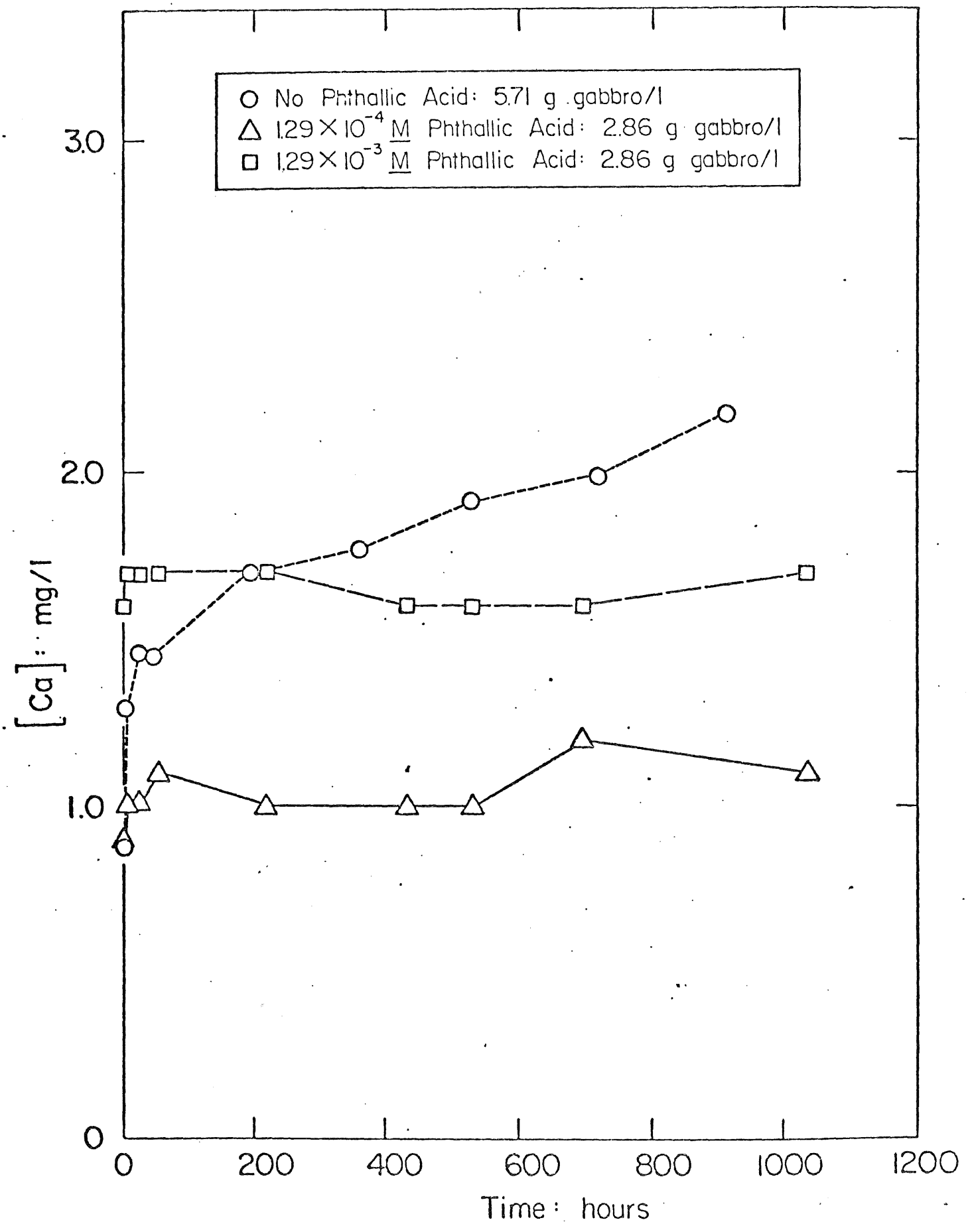
To determine the effect of natural waters on the dissolution of silicate minerals, bog and groundwater obtained in the vicinity of the Dunka Pit were used to leach 5.71 g . ℓ^{-1} of mineralized gabbro at pH 8. Natural concentrations of Ca^{+2} in bog and groundwater were 7.8 and 18.9 mg .ℓ^{-1}, respectively. Results of the dissolution experiment are shown in Figure 4.13. Calcium concentrations increased by 3.0 mg . ℓ^{-1} in the bog water case over ∿ 1000 hours, while the Ca^{+2} levels increased by 1.20 mg . ℓ^{-1} in the groundwater reactor control experiments showed an∿ 2.0 mg . ℓ^{-1} increase over the same time period. With this comparison, bog water enhanced the dissolution rate of gabbro by about 50%, while groundwater interactions may have actually removed Ca^{+2}. The Mg^{+2} concentration increased by 1.0 mg . ℓ^{-1} and 2.90 mg .ℓ^{-1}, respectively for the groundwater and bog water leach.

The initial reaction on mixing of surface waters with alumino-silicate minerals would be a decrease in the solution concentration due to precipitation/sorption followed by a slow increase as a result of dissolution

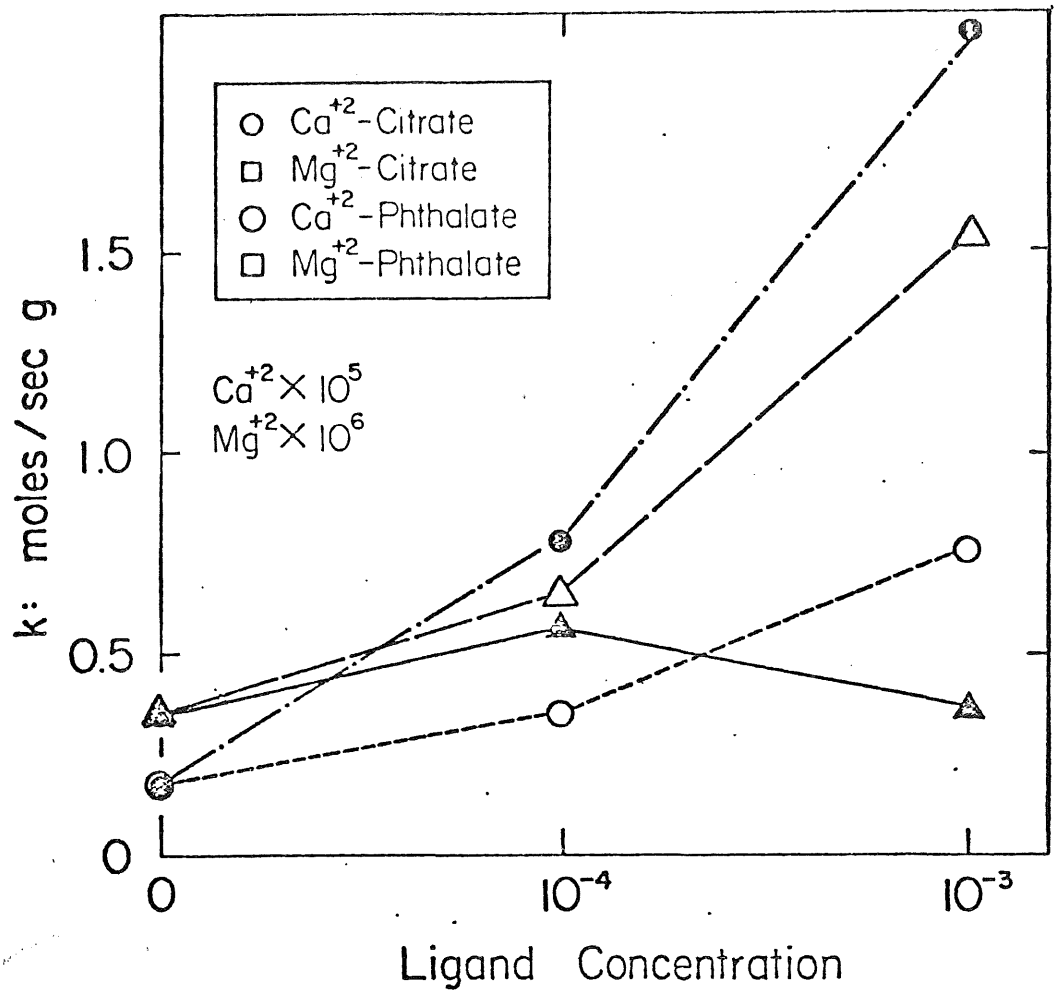
by binding structural elements, and maintain higher solution concentrations through complexation. The effect of silicate dissolution on mineral sulfide oxidation may simply be to increase A_s available for leaching.

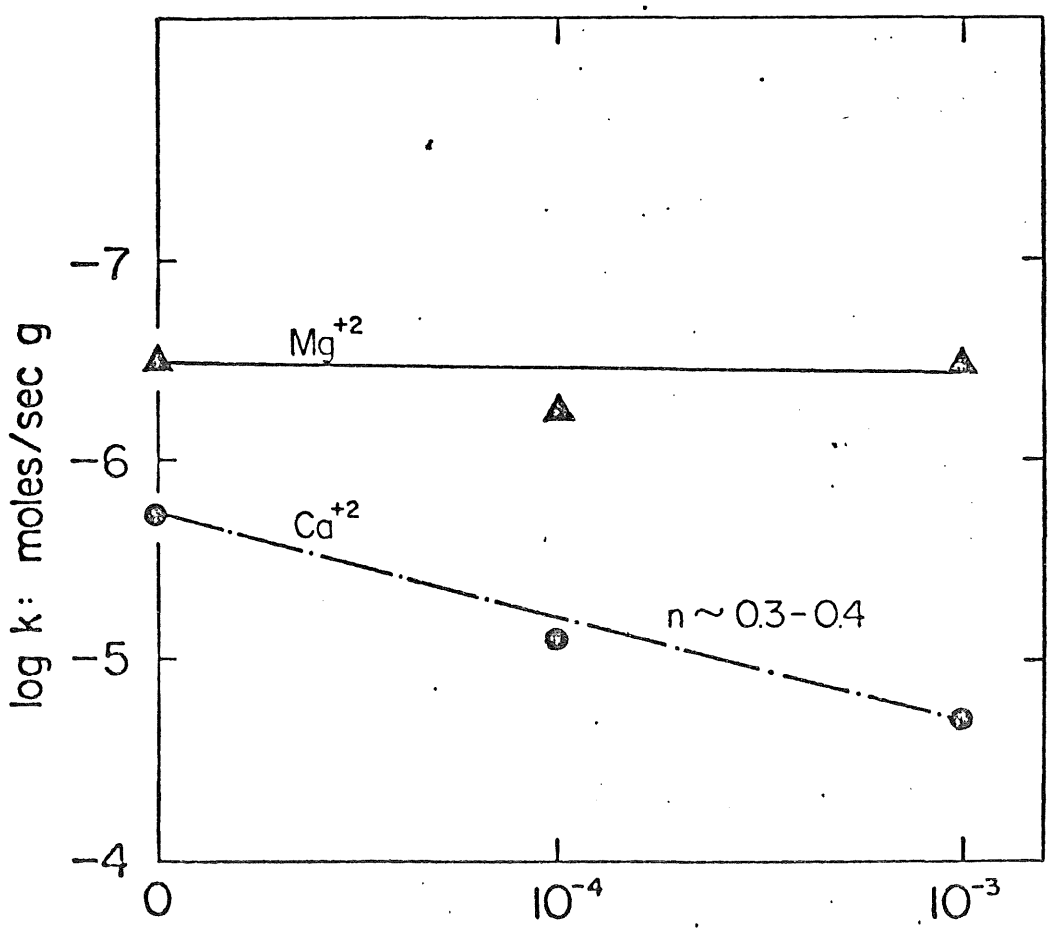

4
3
2
1
0
[Ca]: mg/l
0
200
400
600
800
1000
Time: hours
No Citrate
1.29×10⁻⁴M Citrate
1.29×10⁻³M Citrate

4-11

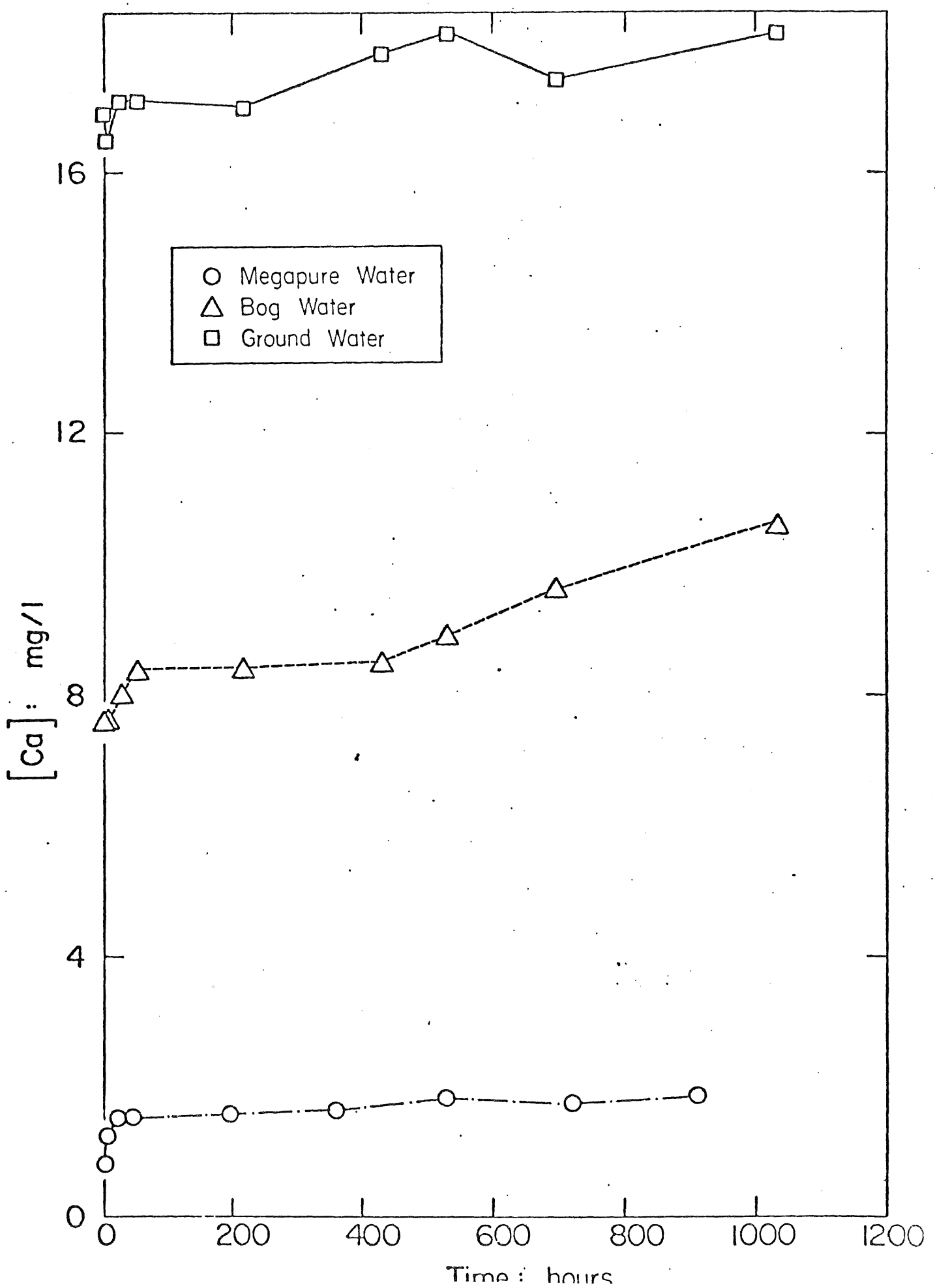

No Phthallic Acid: 5.71 g gabbro/l
1.29 × 10^-4 M Phthallic Acid: 2.86 g gabbro/l
1.29 × 10^-3 M Phthallic Acid: 2.86 g gabbro/l
3.0
2.0
1.0
0
[Ca]: mg/l
0
200
400
600
800
1000
1200
Time: hours

4-12


Ca+2-Citrate
Mg+2-Citrate
Ca+2-Phthalate
Mg+2-Phthalate
Ca+2 × 10^5
Mg+2 × 10^6
k: moles / sec g
1.5
1.0
0.5
0
0
10^-4
10^-3
Ligand Concentration


log k: moles/sec g
-7
-6
-5
-4
Mg+2
Ca+2
n ~ 0.3-0.4
0
10^-4
10^-3

4-13


Megapure Water
Bog Water
Ground Water
[Ca]: mg/l
0
4
8
12
16
0
200
400
600
800
1000
1200
Time : hours

Chapter 5

Discussion of Kinetic Results and Mechanisms

5.1. Oxidative Dissolution of Metal Sulfides

The kinetic results for the abiotic oxidative dissolution of pentlandite present in the Duluth Gabbro ore can be summarized by the following empirical rate expression:

$$\frac{-d[(Fe,Ni)_9S_8]}{dt} = \frac{d[Ni(II)]}{dt} = 2.5kA_s^{0.4}[O_2]^{0.8}[H^+]^{0.2}[CIT]^{0.3} \qquad (5.1)$$

When surface area, As, dissolved oxygen, $[O_2]$, pH and organic chelate, [CIT], remains constant, the rate of appearance of Ni is pseudo zero-order (i.e., concentration increases linearly with time).

$$\frac{d[Ni(II)]}{dt} = k^1 \qquad (5.2)$$

$$k^1 = 4.5kAs^{0.4}[O_2]^{0.8}[H^+]^{0.2}[CIT]^{0.3} \qquad (5.3)$$

Equation 5.3 predicts that magnitude of the observed pseudo zero-order rate constant will increase with an increase in surface area (or a decrease in particle size), with an increase in the dissolved oxygen concentration (over the range 1-10 mg/l which is typical of environmental conditions), with a decrease in pH (an increase in $[H^+]$) and with an increase in the concentration of organic chelates that exhibit specificity for the complexation of Ni(II) or Fe(II).

Similar kinetic dependencies were observed for the rate of release of Cu to solution from the dissolution of chalcopyrite although the reduced solution phase mobility of Cu over a broad pH range prevented the same detailed analysis of Cu dissolution rates. The oxidative dissolution of cubanite and chalcopyrite was significantly enhanced with a decrease in pH and an increase in organic chelate concentration. In fact, the impact of chelation on the release rate of Cu should be predicted to be greater than that of Ni based on fundamental considerations of ligand field stabilization energies (44).

The stability constnat, B, for the formation of analogous complexes of divalent ions follow the general trend $MN^{+2}<Ca^{+2}<Ni^{+2}<Cu^{+2}<Zn^{+2}$ which is called the Irving-Williams order (44). From the concentration time profiles for Cu, it appears that release of copper to solution may be diffusion controlled at higher pH in the absence of strong chelates. This idea is consistent with the observations of other investigators (4, 15, 16) who have observed mixed linear and parabolic release rates for Cu from $CuFeS_2$. Copper has a pronounced tendency to form $Cu_2CO_3(OH)_2$ (malachite) and $Cu(OH)_2$ at the pH and $[HCO_3^-]$ observed during these studies. Formation and deposition of these products on the $CuFeS_2$ surface may provide a barrier to the rapid diffusion of Cu(II) away from the surface and result in a reaction rate which is controlled by mass transport of Cu(II) through the reacted zone. In addition to these non-sulfide products of the oxidative dissolution of $CuFeS_2$, metastable products such as CuS and Cu_3FeS_4 may form on the surface.

Dissolution of solid reactants such as $CuFeS_2$ are thought to involve three distinct steps which are firstly; the detachment of ions or molecules which constitute the solid from the crystal lattice, secondly, the transport of the released solutes away from the dissolving crystal face and thirdly, in the case of a chemical reaction at the surface, the transport of this reactant to the crystal surface. For many readily soluble solids such as alkali and alkaline earth halides it has been observed (45) that step 1 is relatively fast and the solution near the surface is saturated. Dissolution of these compounds appears to be diffusion controlled (step 2). Generally, the rate law observed for such systems is

$$\frac{dc}{dt} = \frac{AD}{VS}(C_s - C) \qquad (5.4)$$

Where A is the surface area, V is the volume of the solvent, D is the solution phase diffusion coefficient and S, is the "Nernst diffusion layer thickness. (46)". A similar rate law would be predicted for dissusion of a reactant such a O_2

through a porous reacted layer on the surface of the dissolving crystal. Diffusion controlled reactions show a unique dependence on temperature and mixing. Increasing the rate of stirring will result in a decrease in S and an increase in the rate of dissolution. The activation energy obtained from the Arrhenius relationship (equation 5.5)

$$d \ln k/dt = Ea/RT^2 \quad (5.5)$$

$$k = Ze^{-Ea/RT} \quad (5.6)$$

for a diffusion-controlled reaction is typically <6 kcal/mole whereas a chemically controlled reaction will have activation energies which are generally >8 Kcal/mole (38).

In the future, the rates of dissolution of the sulfide minerals present in the Gabbro ore should be studied as a function of temperature and mixing rate to ascertain the relative importance of mass transport in the rate controlling process.

Even though the rate of oxidative dissolution is predicted to be a function of both the dissolved oxygen concentration and surface area for either a diffusion-controlled or chemically-controlled reaction; the bulk of experimental evidence suggests that the rate of dissolution of pentlandite is controlled by a chemical reaction either on the surface or in solution in close proximity to the surface. A simple hypothetical model for the dissolution of FeS has been presented in equations 3.32 through 3.37. This model assumes that the rate of dissolution is controlled by the solution phase oxidation of HS^- (alternatively, oxidation of Fe^{+2} could be rate controlling). The kinetic expression which results from this model predicts that the rate of dissolution should be a function of $[O_2]$, $[H^+]$, $[S_T]$ and $[Y^{-2}]$, the organic chelate concentration. Results predicted by this mechanistic model are consistent with the experimental observations of this study and of other investigations (10, 11, 12, 21, 37) although the range of experimental conditions was significantly different.

A similar model, in which the rate of dissolution is controlled by a solution phase reaction, can be presented for $(Fe, Ni)_9S_8$.

$$8H^+ + (Fe,Ni)_9S_8 \rightleftarrows 4.5\ Ni^{+2} + 4.5\ Fe^{+2} + 8HS^- \tag{5.7}$$

$$K_{so} = [Ni^{+2}]^{4.5}[Fe^{+2}]^{4.5}[HS^-]^8/[H^+]^8 \tag{5.8}$$

To simplify equation 5.8, it is assumed that $[Fe^{+2}] = [Ni^{+2}]$ and that $[Ni^{+2}]^{1.12}$ $[Ni^{-2}]$. Using these assumptions equation 5.8 reduces to

$$K^1_{so} = [Ni^{+2}][HS^-]/[H^+] \tag{5.9}$$

After an equilibrium dissociation of pentlandite the following reactions are postulated to occur in sequence:

$$Ni^{+2} = Y^{-2} \underset{\text{rapid}}{\overset{B}{\rightleftarrows}} NiY \tag{5.10}$$

$$HS^- + O_2 \underset{\text{slow}}{\overset{k}{\rightarrow}} HSO_2^- \tag{5.11}$$

$$HSO^-_2 + O_2 \underset{\text{fast}}{\rightarrow} HSO_4^- \tag{5.12}$$

Since equation 5.11 is the slow step in the reaction mechanism, the rate of dissolution will be controlled by this step and can be expressed as follows:

$$\frac{-d[(Fe,Ni)_9S_8]}{dt} = k\ [HS^-][O_2] = \frac{1}{4.5}\frac{d\ [Ni(II)]}{dt} \tag{5.13}$$

Substitution of the equilibrium expressions written in equation 5.9 and 5.10 into equation 5.13 yields equation 5.14 after rearrangement.

$$\frac{d[Ni(II)]}{dt} = \frac{4.5\ k\ K_{so}B[Y^{-2}][O_2][H^+]}{[NiY]} \tag{5.14}$$

Equation 5.14 predicts the observed kinetic dependencies within reason. The dependence of the rate on the total concentration of the chelating agent, Y^{-2}, will be fractional. The observed kinetic orders for O_2 and H^+ and 0.8 and 0.2 respectively. 0.8 is close enough to one to suggest a first-order dependence on oxygen, however,

the small fractional dependence is not adequately predicted by this model.

An alternative model that could account for the fractional dependence of both oxygen and hydrogen ion assumes that the rate of the reaction is controlled by a surface reaction between bound HS^- groups and oxygen. Based on the ideas presented by Nelson (12) the model would involve the following sequence of steps: Oxygen must first diffuse to the solid surface and then attach itself to the surface. After attachment of oxygen to the surface electron transfer from sulfur to oxygen must occur. This step would be followed by a molecular rearrangement and bond breaking. It is most likely that a proton assists the bond breaking at the surface. Additional secondary reactions may occur on the solid interface. The final step would involve the diffusion of reaction products into the bulk solution. Assuming the reaction is neither limited by the diffusion involved in the first or last step, the following model for the reaciton of oxygen alone would result. This model considers that a limited number of reactive sites are available on the solid surface.

$$=/\text{-}SH + O_2 \underset{k_2}{\overset{k_1}{\rightleftharpoons}} =/\text{-}SH^{\cdot}O_2^{*} \qquad (5.15)$$

$$=/\text{-}SH^{\cdot}O_2^{*} \xrightarrow{k_3} \text{products} \qquad (5.16)$$

Equation 5.15 represents a rapid pre-rate-determining equilbrium step and equation 5.16 is the rate-determining slow step in the reation sequence. The kinetic expression for the rate of formation of Products, P, is

$$\frac{dP}{dt} = k_3\,[=/\text{-}SH^{\cdot}O_2^{*}] \qquad (5.17)$$

Assuming steady-state conditions for the intermediate $=/\text{-}SH^{\cdot}O_2^{*}$ the concentration of this intermediate is given by

$$\frac{d[=/\text{-}SH^{\cdot}O_2^{*}]}{dt} = 0 = k_1\,[=/\text{-}SH][O_2] - k_2[=/\text{-}SH^{\cdot}O_2^{*}] - k_3[=/\text{-}SH^{\cdot}O_2^{*}] \qquad (5.18)$$

or

$$[=/\text{-}SH^{\cdot}O_2^{*}] = \frac{k_1}{k_2 + k_3}[=/\text{-}SH][O_2] \tag{5.19}$$

Substitution of equation 5.20 into equation 5.17 gives

$$= \frac{dP}{st} = \frac{k_3 k_1}{k_2 + k_3}[=/\text{-}SH][O_2] \tag{5.20}$$

However since the concentration of unoccupied reactive sites at any time is unknown, the [=/-SH] term must be written in terms of the total number of sites available before the reaction begins.

$$[=/\text{-}SH]_t = [=/\text{-}SH] + [=/\text{-}SH^{\cdot}O_2^{*}] \tag{5.21}$$

Using equation 5.15, equation 5.20 can be rewritten as

$$[=/\text{-}SH]_t = [=/\text{-}SH]_t/(1 + k_f[O_2]) \tag{5.23}$$

where $kf = \{k_1/(k_2 + k_3)\}$. Substitution of 5.23 into equation 5.20 yields

$$= \frac{k_3 K_f [=/\text{-}SH]_t[O_2]}{(1 + k_f[O_2])} \tag{5.24}$$

The concentration of $[=/\text{-}SH]_t$ can be written as

$$[=/\text{-}SH]_t = \frac{As^{\cdot}N}{V} \tag{5.25}$$

where Ni is equal to the number of surface sites per unit area, V is the volume of the reactor and As is the total surface area of the sulfide. Substitution of this expression gives

$$= \frac{k_3 K_f [O_2] As}{1 + K_f[O_2]} \frac{N}{V} \tag{5.26}$$

Equation 5.26 can be examined under the boundary condition of $K_f[O_2] >> 1$, which gives a reaction rate apparently independent of $[O_2]$; and that under the boundary condition of $K_f[O_2] << 1$ the reaction will be first-order in $[O_2]$.

The role of pH can be taken into account in the reaction model proposed in equation 5.15 and 5.16 by considering a pre-equilibrium protonation of the metal sulfide surface as follows:

$$=/\text{-}SH + H^+ \underset{\leftarrow}{\overset{K_a}{\rightarrow}} =/\text{-}SH_2^+ \tag{5.27}$$

$$=/\text{-}SH_2^+ + O_2 \underset{\leftarrow}{\overset{k_4}{\rightarrow}} =/\text{-}SH_2{\cdot}O_2^+ \tag{5.28}$$

$$=/\text{-}SH{\cdot}O_2^+ \overset{k_6}{\rightarrow} \text{products} \tag{5.29}$$

Using a similar mathematical development as used in the derivation of equation 5.26 in addition to a mass balance expression for the total reaction sites including protonated and non-protonated sites a new expression can be obtained. This approach is identical to that used by Nelson (12) and yields equation 5.30.

$$\frac{V}{A} = \frac{(k_3K_1 + k_6K_2[H^+]\ K_a)\ [O_2]A_s}{1 + K_1[O_2] + K_a[H^+] + K_aK_2[H^+][O_2]} \tag{5.30}$$

where $K_1 = k_1/k_2$ and $K_2 = k_4/k_6$. Equations 5.26 and 5.30 are similar mathematically and conceptually to monosubstrate and bisubstrate Michaelis-Mention enzyme kinetics, respectively. Equation 5.30 predicts fractional orders in both the hydrogen concentration and dissolved oxygen concentration.

In all likelihood the rate of dissolution is a function of all three mechanistic pathways. Thatis: diffusion, solution phase and surface phase reactions may occur simultaneously to give the overall rate of oxidation. Additional experimental work is needed to determine the primary mechanistic pathways and specific rate constants involved.

5.2. Dissolution of Alumino-Silicate Minerals

The incongruent dissolution of alumino-silicate minerals containing up to 4.0% sulfur and 1.5% combined Cu, Ni as sulfides within interstitial spaces was investigated in well-mixed batch reactors at 20°C and atmospheric pressure. The kinetic results for the abiotic dissolution of plagioclase (PL) present in Duluth Gabbro can be summarized by the following empirical rate expressions:

$$\frac{-d[PL]}{dt} = \frac{d[Ca^{+2}]}{dt} = k_1 A_s^{2.0} [H^+]^{2.9} [CIT]^{0.3} \qquad (5.31)$$

and

$$\frac{-d[PL]}{dt} = \frac{d[Mg^{+2}]}{dt} = k_2 A_s^{0.5} [H^+]^{.14} \qquad (5.32)$$

where A_s = surface area, CIT = citrate and H^+ = hydrogen ion.

Unlike the case of pentlandite dissolution described by equations (5.1-5.3), silicate mineral dissolution is not dependent on the dissolved oxygen (O_2) concentration in aqueous solution. However, pH solution may change as a result of oxidative dissolution of metal sulfides which will alter the rate and extent of silicate dissolution. Several qualitative differences were observed in the release characteristics of Ca^{+2} and Mg^{+2} to solution indicating that they are bound differently in the gabbro rock. Firstly, the reaction orders observed for Ca^{+2} release as a function of A_s (2.0) and H^+ (2.9) were significantly greater than the corresponding orders for Mg^{+2} release (As, 0.5; H^+, .14). Secondly, the release of Mg^{+2} to solution was not dependent on citrate concentration whereas Ca^{+2} release was fractionally dependent. This behavior was observed even though stability constants for the Ca^{+2}-citrate and Mg^{+2}-citrate complexes are similar. Thirdly, the rate of Ca^{+2} release under all experimental conditions exceeded Mg^{+2} release rates by up to 10^2.

Given this information, it is apparent that Ca^{+2} and Mg^{+2} are bound in different mineral forms. The principal Ca^{+2}-bearing mineral is plagioclase which represents 47 to 64% of the rock by weight. Principal Mg^{+2} minerals are olivine, clinopyroxene and biotite. In each of the above named cases, Ca^{+2} is the cation associated with the silicate mineral maintaining neutrality, while Mg^{+2} is a structural component of the mineral lattice. Therefore, the respective associations of Ca^{+2} and Mg^{+2} in Duluth Gabbro are borne out by the observed release behavior to solution.

A summary of the Ca^{+2} and Mg^{+2} kinetic constants are given below for parametric variations of As, pH, u, O_2, CIT, PHTA:

	$k_{Ca^{+2}}$ ($\times 10^5$ moles.sec^{-1}.g^{-1})	$k_{Mg^{+2}}$ ($\times 10^6$ moles.sec^{-1}.g^{-1})
As	.009-1.1	.008-0.9
pH	.1 - .2	.26-.58
u	.43-.52	.72-1.1
O_2	.4 - .9	.35- .8
CITRATE	.8 -2.1	.6 - 1.3
PHTHALATE	.3 - .7	.6 - 1.6

Under natural environmental conditions, the Ca^{+2} release rate is in the range of 0.1-1 mole.g^{-1}.sec^{-1}, while the Mg^{+2} release rate is $\sim$ 0.21-1.5 mole .sec^{-1}.g^{-1}. The slower release rates observed for Mg^{+2} release as compared to Ca^{+2} are a result of smaller concentrations in the bulk solid and occurrence in different mineral phases. The principal controls on the buildup of Ca^{+2} and Mg^{+2} in solution are pH and binding ligands. The occurrence of natural organic ligands in water contacting plagioclase minerals will increase the rate and extent of weathering (dissolution). The organic ligand may prevent the buildup of the secondary product layer, thereby eliminating solute diffusion through the layer as a rate-limiting step. Decreasing solution pH to less than 5 may cause similar effects.

The shapes of the Ca^{+2} and Mg^{+2} release curves with time are consistent

with the mechanism of aluminosilicate minerals proposed by Busenberg and Clemency (34). However, the complexity of the Gabbro rock leached in batch reactors precludes the proposed mechanism from being fully substantiated. According to Busenberg and Clemency (34), the dissolution of aluminosilicate minerals can be explained by a four-stage kinetic process in which the initial step is the exchange of solution protons for surface cations followed by the concurrent release of structural cations and silicic acid. In the second stage lasting $\sim$ 50 hours, silicic acid reacts with newly-formed amorphous $Al(OH)_3$ at the solid surface to form a slightly-soluble product layer consisting of microcrystalline gibbsite or halloysite (hydrated form of kaolinite). The third stage is parabolic, lasts up to 19 days and represents a diffusion-limited transport step. The fourth stage, lasting beyond 21 days, exhibits linear kinetics in which the rate of formation of the secondary product layer equals the rate at which the layer is destroyed by dissolution or fragmentation (32). Cation release curves were similar to those previously observed (34) and likely represent similar processes. Although parabolic kinetics are suspected after $\sim$ 50 hours in our studies, insufficient data points were obtained to verify this hypothesis.

The results of parametric studies of the dissolution of aluminosilicate minerals indicate that weathering increases with increasing particle surface area, acidity, pe, solution ionic strength and presence of organic ligands.

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Table A.1

Experiment 2

REACTOR 1: 2 g/l unmineralized-200 mesh gabbro, air diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)	Si (mg/l)
0.75	3.66	2.8	100.4	178.2	4240	2.36	0.35	6.43
3.25	3.82	3.0	111.2	156.2	5550	2.46	0.53	1.10
6.00	3.89	2.9	115.3	174.9	5910	2.21	0.57	1.10
26.75	4.00	2.6	130.8	232.1	6780	2.34	0.66	1.29
49.50	4.20	3.6	144.8	254.1	7140	2.45	0.80	1.81
146.0	4.41	3.6	210.0	307.3	7580	2.52	0.98	1.83
319.3	4.42	3.6	326.6	378.0	7520	2.60	1.0	1.84
507	4.51	3.4	460.0	436.5	7980	2.74	1.01	1.92
748	4.48	4.8	564.0	526.6	8000	2.88	1.13	2.22
1-AD-2		0	5.5	17.2	440	0.30	0.07	0.98

REACTOR 2: 10 g/l unmineralized-200 mesh gabbro, air diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)	Si (mg/l)
0.75	7.10	2.6	16.6	2.2	15.7	3.71	0.34	0.86
3.25	7.19	2.5	16.0	ND	16.9	3.9	0.7	1.00
6.0	7.18	2.5	31.4	0.3	13.3	4.23	1.02	24.56
26.75	7.21	2.5	102.2	ND	8.5	4.56	1.45	1.53
49.50	7.41	3.6	154.4	ND	22.2	4.83	1.10	1.89
146.0	7.39	3.3	298.8	0.6	16.6	5.26	1.21	2.54
319.3	7.11	9.9	373.0	1.0	16.8	5.73	1.15	3.36
507	7.12	12.9	410.8	2.0	17.8	5.86	1.40	4.15
748	6.90	13.7	317.4	ND	16.5	6.14	1.83	3.71
2-AD-2		3.8	242.5	105.2	1650			1.32

REACTOR 3: 50 g/l unmineralized-200 mesh gabbro, air diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)	Si (mg/l)
0.75	8.19	3.0	7.5	0.8	12.4	4.53	0.71	1.62
3.25	7.88	3.7	4.6	0.1	6.9	6.42	1.00	0.83
6.0	7.92	4.4	8.1	ND	6.2	8.75	1.31	1.22
26.75	7.89	5.2	89.5	3.7	45.6	12.87	2.20	2.97
49.50	8.01	7.0	95.7	0.1	9.5	13.69	2.40	2.91
146.0	8.02	10.5	107.1	0.7	11.4	14.23	3.10	3.77
319.3	7.81	20.4	110.8	0.6	22.3	15.33	3.85	3.76
507.0	7.49	53.3	145.6	ND	19.7	17.45	4.60	4.23
748	7.55	110	127.0	1.9	14.5	18.23	5.10	4.71
3-AD-2		3.8	707.7	1009	5660	12.45	1.66	0.45

Table A.1

Experiment 2

REACTOR 4: 100 g/l unmineralized-200 mesh gabbro, air diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)	Si (mg/l)
0.75	8.80	6.4	12.3	6.1	11.6	4.45	1.05	1.27
3.25	8.50	6.0	1.4	ND	44.2	5.17	1.23	1.52
6.00	8.41	7.6	2.3	ND	6.4	6.50	1.60	1.47
26.75	8.12	10.3	47.1	0.8	8.3	13.35	4.00	2.42
49.50	8.22	11.8	66.2	1.2	11.0	15.63	4.65	2.72
146.00	8.21	16.0	77.8	1.2	11.0	16.79	5.00	3.87
319.30	8.04	59.5	113.5	1.3	10.3	22.75	6.40	4.44
507.00	8.02	128.0	196.1	ND	9.6	31.13	7.50	4.28
748.00	7.87	148.0	145.5	ND	14.3	28.38	8.40	4.75
4-AD-2		3.7	2253.8	5850.0	13,272.0			14.66

REACTOR 5: 2 g/l unmineralized-200 mesh gabbro, N_2 diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)	Si (mg/l)
0.75	4.27	2.9	56.5	2.0	5010		0.47	2.38
3.25	4.79	3.0	51.1	7.9	6580		0.49	0.96
6.00	5.00	3.0	56.8	9.7	6920		0.78	0.96
26.75	5.32	2.5	71.8	10.8	7330	2.05	0.78	1.22
49.50	5.69	3.6	69.8	2.1	7250		0.74	1.33
146.00	5.72	3.6	101.8	1.7	7240		0.77	1.89
319.30	5.18	3.7	230.6	41.8	6880		0.80	1.67
507.00	5.12	3.5	330.4	46.1	6910	2.05	0.83	1.38
748.00	5.04	3.8	342.4	49.1	7210		0.92	2.30
5-AD-2		0	9.8	19.2	760	0.30	0.12	0.44

REACTOR 6: 50 g/l unmineralized-200 mesh gabbro, N_2 diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)	Si (mg/l)
0.75	7.21	3.6	10.1	ND	22.4	9.97	2.00	1.21
3.25	7.51	2.7	4.5	ND	37.3	11.20	2.02	1.36
6.00	7.51	4.6	5.8	ND	42.9	11.93	2.55	1.24
26.75	7.61	4.9	3.6	ND	41.4	11.63	3.05	1.23
49.50	7.93	6.0	3.7	ND	45.7	11.10	3.40	2.61
146.00	8.12	6.7	3.6	ND	48.2	11.73	4.05	3.09
319.30	7.85	10.1	17.1	1.6	39.1	13.03	4.80	3.34
507.00	7.51	25.1	15.2	ND	31.1	13.75	5.30	3.02
748.00	7.48	29.1	11.5	ND	33.4	13.60	5.05	3.13
6-AD-2		3.5	501.8	635.0	5150.0	10.40	1.76	1.90

Table A.2

Experiment 3

REACTOR 1: 2.86 g/l unmineralized-200 mesh gabbro, air diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Mg (mg/l)
0.5	6.89	0	2.6	1.7	0.022	1.05	0.140
6.0	6.86	0	10.3	1.1	0.009	1.30	0.260
24.5	6.80	2.9	39.7	1.5	<0.005	1.43	0.341
48.0	6.94	0	88.9	1.7	0.020	1.50	0.386
168.5	6.86	0	285.0	1.5	<0.005	1.85	0.465
335.0	6.80	3.2	460.0	4.1	0.026	2.05	0.574
480.0	6.90	3.6	590.0	1.2	<0.005	2.17	0.614
651.0	6.86	2.5	562.0	1.8	<0.006	2.30	0.652
987.5	6.90	5.3	586.0	4.6	<0.005	2.46	0.732
1-AD-3			117.0	66.1	0.640	2.50	0.128

REACTOR 2: 2.86 g/l unmineralized-200 mesh gabbro; air diffusion; add 1.6 ml 1N $HClO_4$

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Mg (mg/l)
0.5	4.09	3.2	42.9	74.6	3.96	2.28	0.605
6.0	4.52	3.3	66.0	92.7	6.06	2.34	0.708
24.5	4.72	3.2	94.6	101.0	6.60	2.46	0.805
48.0	4.80	3.1	156.0	109.0	6.63	2.28	0.793
168.5	4.76	3.1	392.0	153.0	6.41	2.39	0.968
335.0	4.73	4.0	548.0	182.0	6.39	2.61	1.020
480.0	4.71	3.6	720.0	205.0	6.25	2.57	1.040
651.0	4.67	3.5	824.0	240.0	6.71	2.81	1.150
987.5	4.59	7.0	1040.0	270.0	5.92	2.94	1.220
2-AD-3		0	ND	10.4	0.51	0.24	0.011
B-AD-3		0					

REACTOR 3: 2.86 g/l unmineralized-200 mesh gabbro; air diffusion; add 3.2 ml 1 N HClO

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Mg (mg/l)
0.5	3.57	3.2	50.9	87.2	7.02	4.14	0.713
6.0	3.76	3.3	80.6	149.0	9.38	4.30	0.899
24.5	3.92	3.3	99.5	187.0	10.70	4.46	1.170
48.0	4.06	4.1	136.0	205.0	11.70	4.38	1.350
168.5	4.30	3.2	268.0	227.0	12.50	4.98	1.600
335.0	4.40	3.9	440.0	265.0	12.50	5.25	1.660
480.0	4.44	4.0	562.0	289.0	13.00	5.30	1.700
651.0	4.45	4.7	679.0	317.0	13.00	5.33	1.780
987.5	4.43	6.2	715.0	310.0	12.20	5.95	1.820
3-AD-3		0	0.4	11.6	0.51	0.20	0.038

Table A.2

Experiment 3

REACTOR 4: 2.86 g/l unmineralized-200 mesh gabbro; air diffusion; add 4.8 ml 1 N $HClO_4$

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Ma (mg/l)
0.5	3.25	3.3	57.2	61.4	7.67	5.63	0.803
6.0	3.41	3.4	82.3	121.0	12.00	6.21	1.120
24.5	3.53	3.5	101.0	161.0	13.90	6.84	1.470
48.0	3.63	4.1	116.0	174.0	15.90	7.17	1.760
168.5	3.87	4.1	222.0	198.0	17.70	7.56	2.380
335.0	3.99	4.1	380.0	226.0	18.30	8.12	2.780
480.0	4.05	3.8	494.0	258.0	19.40	8.36	2.910
651.0	4.10	5.2	662.0	291.0	20.50	8.70	3.240
987.5	4.15	7.2	708.0	274.0	21.20	9.42	3.580
4-AD-3		0	1.9	12.2	0.74	0.26	0.060

REACTOR 5: 2.86 g/l unmineralized-200 mesh gabbro; air diffusion; citrate addition; $1.29 * 10^{-4}$ M

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Ma (mg/l)
0.5	7.24	0	13.2	9.8	9.064	2.61	0.258
6.0	7.01	0	30.6	16.3	1.580	2.68	0.356
24.5	7.23	2.7	68.6	32.4	1.980	3.08	0.480
48.0	7.30	3.6	130.0	45.5	2.770	3.15	0.540
168.5	7.64	2.9	99.5	1.8	0.143	3.84	0.627
335.0	7.66	2.5	110.0	2.4	0.041	4.06	0.644
480.0	7.63	2.7	110.0	1.8	0.034	4.21	0.704
651.0	7.58	2.9	122.0	2.0	0.028	4.43	0.692
987.5	7.55	5.3	114.0	1.7	0.024	4.72	0.720
5-AD-3		0	219.0	107.5	0.940	3.88	0.121

REACTOR 6: 2.86 g/l unmineralized-200 mesh gabbro; air diffusion; citrate addition; $1.29 * 10^{-3}$ M

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Ma (mg/l)
0.5	7.95	3.0	17.5	19.7	0.95	2.94	0.258
6.0	7.61	3.0	40.0	44.6	3.79	3.08	0.430
24.5	7.63	3.7	137.0	88.5	6.51	3.26	0.711
48.0	7.74	4.3	277.0	118.0	7.46	3.44	0.892
168.5	8.25	3.8	737.0	162.0	8.28	3.52	1.060
335.0	8.33	4.2	1010.0	175.0	8.13	3.59	1.080
480.0	8.38	3.8	1070.0	181.0	8.00	3.75	1.050
651.0	8.38	4.0	1050.0	213.0	6.80	3.95	1.070
987.5	8.49	3.4	1250.0	226.0	6.66	4.17	1.140
6-AD-3		0	117.0	54.0	1.71	3.63	0.322

Table A.3

Experiment 4

REACTOR 1: 2.86 g/l -200 mesh mineralized gabbro

TIME (Hrs)	pH	SO_4 (PPM)	Ni (PPB)	Cu (PPB)	Fe (PPB)	Ca (PPM)	Mg (PPM)
0.5	6.72	4.6	7.8	0	6.45	0.44	0.080
5.5	6.80	4.6	17.2	0.3	0.74	0.65	0.132
25.0	6.75	4.5	42.0	0.1	<5.00	0.72	0.178
48.0	6.57	4.5	69.5	0.2	0.62	0.71	0.188
163.5	6.65	5.1	194.0	0.6	9.02	0.78	0.258
336.0	6.50	5.5	460.0	1.4	8.09	0.92	0.326
526.0	6.02	7.4	839.0	8.8	<5.00	0.98	0.450
672.0	5.75	9.0	1070.0	34.8	10.90	1.10	0.443
891.0	5.44	9.8	1510.0	92.5	65.00	1.23	0.532
1-AD-4		3.9	19.9	18.6	95.60	0.32	0.046

REACTOR 2: 2.86 g/l $^{+}$200- $^{-}$40 mesh mineralized gabbro

TIME (Hrs)	pH	SO_4 (PPM)	Ni (PPB)	Cu (PPB)	Fe (PPB)	Ca (PPM)	Mg (PPM)
0.5	6.03	4.0	3.4	0.3	2.5	0.18	0.016
5.5	6.28	3.9	7.2	0.6	17.7	0.29	0.064
25.0	6.44	3.9	11.1	1.0	12.5	0.36	0.070
48.0	6.43	3.8	18.7	0.1	2.7	0.35	0.092
163.5	6.64	4.2	62.4	1.5	22.8	0.47	0.120
336.0	6.55	4.1	122.0	0.1	0.8	0.62	0.123
526.0	6.38	4.3	221.0	0.6	1.7	0.66	0.174
672.0	6.35	4.5	293.0	2.6	<5.0	0.35	0.162
891.0	6.33	4.6	384.0	4.8	1.6	0.68	0.190
2-AD-4							

REACTOR 3: 2.86 g/l $^{+}$40- $^{-}$20 mesh mineralized gabbro

TIME (Hrs)	pH	SO_4 (PPM)	Ni (PPB)	Cu (PPB)	Fe (PPB)	Ca (PPM)	Mg (PPM)
0.5	5.84	4.0	1.6	0.4	5.2	0.23	0.004
5.5	6.10	3.9	4.7	0.4	8.0	0.22	0.012
25.0	6.25	3.8	11.6	0.3	10.8	0.29	0.054
48.0	6.31	4.0	16.5	0.3	11.8	0.30	0.072
163.5	6.46	4.1	45.7	0	1.8	0.38	0.082
336.0	6.50	4.1	92.7	0	9.5	0.45	0.098
526.0	6.44	4.0	153.0	1.0	32.1	0.53	0.130
672.0	6.49	4.2	175.0	0.8	<5.0	0.51	0.152
891.0	6.59	4.1	203.0	0.4	<5.0	0.50	0.145
3-AD-4		4.0	13.4	6.8	45.4	0.27	0.009

Table A.3

Experiment 4

REACTOR 4: $^{+}20$- $^{-}8$ mesh mineralized gabbro

TIME (Hrs)	pH	SO_4 (PPM)	Ni (PPB)	Cu (PPB)	Fe (PPB)	Ca (PPM)	Mg (PPM)
0.5	5.72	4.0	2.1	0.2	4.1	0.22	0.025
5.5	5.93	3.4	6.6	1.2	15.2	0.24	0.026
25.0	5.98	3.3	10.1	0.7	27.5	0.27	0.035
48.0	6.03	3.4	13.3	0.5	25.2	0.28	0.041
163.5	6.13	3.3	33.2	0	15.0	0.32	0.047
336.0	6.22	3.4	60.0	0.6	2.4	0.33	0.055
526.0	6.25	3.5	79.6	0.7	38.0	0.40	0.093
672.0	6.30	3.5	86.3	0.4	4.1	0.42	0.075
891.0	6.52	3.5	130	0.7	5.5	0.42	0.082
4-AD-4		3.4	7.0	3.2	59.4	0.22	0.026

REACTOR 5: 2.86 g/l $^{-}200$ mesh unmineralized gabbro, [CIT]=1.29×10^{-4} M

TIME (Hrs)	pH	SO_4 (PPM)	Ni (PPB)	Cu (PPB)	Fe (PPB)	Ca (PPM)	Mg (PPM)
0.5	7.24	5.0	34.2	16.6	602	0.66	0.240
6.5	7.30	4.0	48.3	21.2	2150	0.84	0.358
24.0	7.28	3.6	127	35.7	3050	0.86	0.493
49.5	7.42	3.6	144	28.9	2200	0.80	0.542
142.0	7.58	3.6	64.8	1.0	169	0.72	0.543
358.0	7.66	4.2	68.4	0.9	11.3	0.86	0.598
531.0	7.72	4.5	54.1	0.4	5.5	0.92	0.601
721.0	7.69	4.9	48.5	0.7	<5	1.01	0.656
867.0	7.77	5.1	56.5	1.3	<5	0.98	0.704
1088	7.86	3.6					
5-AD-4			28.5	36.8	18.1	0.50	0.030

REACTOR 6: 2.86 g/l $^{-}200$ mesh unmineralized gabbro [CIT]=1.29×10^{-3} M

TIME (Hrs)	pH	SO_4 (PPM)	Ni (PPB)	Cu (PPB)	Fe (PPB)	Ca (PPM)	Mg (PPM)
0.5	7.04	0.8	30.6	31.7	2300	1.04	0.408
6.5	7.39	0.8	65.8	52.5	5250	1.04	0.594
24.0	7.47	1.0	193	90.3	8200	1.14	0.832
49.5	7.77	1.0	353	106	9550	1.10	1.01
142.0	8.40	2.2	484	76.2	7250	0.90	0.882
358.0	8.60	3.4	6.5	7.1	181.3	0.82	0.766
531.0	8.61	2.9	5.3	9.6	32.5	0.84	0.810
721.0	8.63	2.8	5.2	9.9	9.5	0.84	0.821
867.0	8.68	4.4	5.1	9.3	55.4	0.90	0.826
1088	8.71						
6-AD-4		1.4	42.7	72.8	451.0	0.84	0.15

Table A.4

Experiment 5

REACTOR 1: 2.86 g/l mineralized-200 mesh gabbro, nitrogen diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75	7.98	2.8	0	0.3	20.5	0.66	0.15
5.5	7.33	2.8	3.4	1.1	51.2	1.04	0.28
25.5	7.56	2.8	3.8	0.8	12.5	1.26	0.34
53	7.87	2.8	16.5	0.4	1.8	1.32	0.40
168	8.40	3.0	7.9	0.7	6.8	1.32	0.45
363	7.05	4.4	24.6	2.0	8.8	1.38	0.51
502	7.91	4.9	30	2.5	15.4	1.3	0.5
628	7.79	6.4	32	3.0	13.3	1.1	0.5
817	7.56	8.2	22	1.6	9.7	1.3	0.5
1-AD			33	53.7	280	1.6	0.3

REACTOR 2: 2.86 g/l, mineralized-200 mesh gabbro, N_2 & air diffusion

TIME (hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75	7.25	4.3	8.1	4.3	34.1	1.10	0.18
5.5	7.06	4.3	15.2	0.8	6.8	1.26	0.26
25.5	7.10	4.3	27.0	2.1	8.4	1.26	0.34
53	7.18	4.4	43.7	0	0.3	1.43	0.37
168	7.08	4.6	110.9	3.6	2.1	1.26	0.45
363	6.48	5.1	199.6	4.1	0.4	1.54	0.51
502	6.88	4.6	268	18.2	20.6	1.3	0.5
628	6.95	5.0	50	5.1	10.8	1.2	0.5
817	6.94	5.1	26	2.4	3.7	1.3	0.6
2-AD			28	54.4	200	0.9	0.1

REACTOR 3: 2.86g/l mineralized-200 mesh gabbro, air diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75	6.58	3.4	27.0	1.6	12.4	1.04	0.16
5	6.86	3.3	42.7	3.1	11.7	1.32	0.22
24	6.80	3.5	61.3	2.2	2.4	1.32	0.24
46	6.76	3.5	92.4	2.1	1.4	1.48	0.25
192	6.45	4.0	237.4	2.1	0.4	1.1	0.32
356	6.33	4.6	351	4.2	2.0	1.60	0.37
523	5.95	5.3	454	13.2	6.4	1.82	0.42
690	5.97	6.8	595	27.6	8.7	1.92	0.46
860	5.54	7.9	721	48.5	10.1	1.87	0.50
886			724	50.8	12.2	2.09	0.57
3-AD-5	-	-	53.4	54.8	151.2	1.21	0.08

Table A.4

Experiment 5

REACTOR 4: 2.86 g/l mineralized -200 mesh gabbro, O_2 diffusion

TIME (Hrs)	pH	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75	6.82	3.1	23.7	2.4	19.7	2.0	0.18
5	7.51	3.1	34.2	0.9	3.9	1.7	0.19
24	7.25	3.1	71.8	1.2	4.1	1.5	0.24
46	7.41	3.3	80.0	3.4	5.6	2.4	0.25
192	6.94	3.7	157	1.6	2.8	2.2	0.31
356	6.04	5.3	226	19.1	4.4	2.5	0.40
523	5.20	7.5	381	226	870	2.5	0.70
690	5.00	10.6	540	347	240	2.6	0.64
860	4.71	15.7	774	558	850	3.1	0.72
886			785	596	1110	3.4	0.75
4-AD-5			21.3	41.6	187.2	1.3	0.10

REACTOR 5: 2.86 g/l unmineralized -200 mesh gabbro, $1.29*10^{-4}$ M citrate, adjust to pH 7

TIME (Hrs)	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75	3.4	20	16.2	440	1.6	0.28
5	3.6	56	33.0	1120	2.0	0.51
24	3.5	122	36.3	2980	2.6	0.63
46	3.4	163	52.8	3350	2.7	0.70
192	3.7	348	92.6	2550	3.5	0.87
356	3.5	455	136	1030	4.4	0.90
523	3.8	456	140	470	2.8	0.95
690	3.8	362	79.8	107	2.9	0.95
860	4.8	307	20.7	64	2.4	0.96
886		276	5.6	85	2.1	1.04
5-AD-5		234	111.3	730	3.8	0.29

REACTOR 6: 2.86 g/l unmineralized -200 mesh gabbro, $1.29^{*}\ 10^{-3}$ M citrate adjust to pH 7

TIME (Hrs)	SO_4 (mg/l)	Ni (μg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75	3.8	34.0	31	1980	2.4	0.46
5	4.0	68.0	62	5000	2.3	0.72
24	5.1	123	105	8320	2.2	1.12
46	4.0	195	113	10,080	2.3	1.42
192	5.5	563	130	15,410	2.2	2.44
356	8.8	810	144	19,210	2.3	2.98
523	13.0	1020	140	24,100	2.3	3.09
690	15.1	1090	133	25,800	2.3	3.11
860	16.2	1170	191	28,800	2.4	3.51
886		1260	158	28,800	2.2	3.37
6-AD-5		16.9	32.9	870	2.4	0.43

Table A.5

Experiment 6

REACTOR 1: 5.7 g ℓ^{-1} -200 mesh mineralized gabbro, pH 5

TIME (Hrs)	pH	Ni (μg/l)	SO_4^{2} (mg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Mg (mg/l)
0.5		13	3.4	2.0	350	.91	.27
5.5		19	3.3	7.9	820	1.57	.43
24		52	3.3	6.6	340	1.87	.58
44		73	3.3	8.0	450	1.99	.69
196		180	3.8	78.7	1020	2.27	1.03
360		450	6.9	131	530	2.44	1.16
530		700	11.2	242	860	2.57	1.26
720		930	16.4	233	700	2.68	1.36
912		1170	23.7	297	620	2.99	1.50
3-AD-6		27		66.9	380	0.24	.06

REACTOR 2: 5.71 g ℓ^{-1} -200 mesh mineralized gabbro, pH 6

TIME (Hrs)	pH	Ni (μg/l)	SO_4^{2} (mg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Mg (mg/l)
0.5		6	3.2	0.2	140	.74	.22
5.5		15	3.3	0.6	230	1.30	.37
24		35	3.2	0.5	20	1.60	.53
44		57	3.5	0.2	12	1.68	.60
196		160	4.4	12.9	19	2.05	.87
360		480	6.8	19.1	13	2.23	.99
530		770	8.1	30.9	18	2.47	1.10
720		1050	10.9	25.6	9	2.66	1.23
912		1430	12.7	77.5	22	3.00	1.36
4-AD-6		950		129	370	1.18	0.10

REACTOR 3: 5.71 g $^{-1}$ -200 mesh mineralized gabbro, pH 7

TIME (Hrs)	pH	Ni (μg/l)	SO_4^{2} (mg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Mg (mg/l)
0.5		7	3.6	0.4	43	.88	.23
5.5		9	3.5	1.1	20	1.29	.32
24		26	3.5	0.3	5	1.46	.42
44		36	3.5	0.1	5	1.45	.50
196		100	4.5	6.6	13	1.70	.71
360		250	6.6	3.1	9	1.77	.77
530		400	8.1	5.3	8	1.91	.85
720		450	9.8	4.8	8	1.99	.93
912		490	11.3	34.5	210	2.18	1.04
1-AD-6		130		410	690	3.04	.20

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Table A.5

Experiment 6

REACTOR 4: 5.7 $g\ell^{-1}$ -200 mesh mineralized gabbro, pH 8

TIME (Hrs)	pH	Ni (μg/l)	So_4^{-2} (mg/l)	Cu (μg/l)	Fe (mg/l)	Ca (mg/l)	Mg (mg/l)
0.5		4	3.5	0.6	27	.83	.23
5.5		6	3.6	1.3	11	1.25	.34
24		19	3.3	1.0	21	1.52	.48
44		30	3.3	0.7	14	1.52	.51
196		52	4.3	7.2	35	1.60	.61
360		48	6.4	4.8	51	1.65	.68
530		38	7.5	4.5	103	1.83	.79
720		36	8.1	6.0	69	1.77	.77
912		36	10.7	3.5	14	1.88	.80
2-AD-6		87		306	520	2.40	.19

Table A.6

Experiment 7

REACTOR 1: 5.71 $g\ell^{-1}$ -200 mesh mineralized gabbro, bog water

TIME (Hrs)	pH	So^{-2} (μg/l)	Ni (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75	8.08	4.5	14	4	620	7.6	8.4
5	8.09	5.1	15	3	440	7.6	8.4
27	8.11	4.7	22	7	570	8.0	8.7
52	8.02	4.6	22	13	570	8.4	8.3
218	7.99	5.3	30	15	460	8.4	8.8
431	8.08	6.4	39	17	490	8.5	9.3
531	8.10	6.7	43	22	310	8.9	9.6
698	8.14	7.2	44	23	320	9.6	9.9
1034	8.12	9.9	46	36	250	10.6	11.3
1-AD-7			69	220	1550	3.8	0.73
T=0		3.9	0	3	160	7.8	8.9

REACTOR 2: 5.71 $g\ell^{-1}$ -200 mesh mineralized gabbro, ground water

TIME (Hrs)	pH	So^{-2} (g/l)	Ni (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75	8.03	6.7	9	2	17	16.9	9.8
5	8.11	7.1	7	3	27	16.5	9.9
27	8.13	7.7	14	3	33	17.1	9.1
52	8.04	7.9	20	5	42	17.1	9.9
218	8.01	7.9	27	4	26	17.0	9.8
431	8.11	8.9	27	3	26	17.8	10.2
531	8.14	9.5	29	4	21	18.1	10.3
698	8.16	12.0	37	7	32	17.4	10.0
1034	8.03	11.6	36	5	38	18.1	10.8
2-AD-7			30	110	680	1.9	0.29
T=0		6.4	4	4	37	18.9	10.2

REACTOR 3: 5.71 $g\ell^{-1}$ -200 mesh mineralized gabbro, $=5*10^{-3}$

TIME (Hrs)	pH	So^{-2} (μg/l)	Ni (mg/l)	Cu (μg/l)	Fe (g/l)	Ca (mg/l)	Mg (mg/l)
0.75		2.9	2	0.9	92	1.4	0.36
5		2.7	19	2	18	1.9	0.46
27		2.9	51	0.8	12	1.8	0.56
52		3.3	71	3	27	2.0	0.60
218		5.0	220	2	19	2.0	0.75
431		7.1	330	3	6	2.1	0.90
531		7.5	460	4	6	2.1	0.87
698		9.6	590	5	15	2.2	1.01
1034		13.3	640	7	12	2.3	1.13
3-AD-7			120	96	450	1.2	0.093

Table A.6

Experiment 7

REACTOR 4: 5.71 $g\ell^{-1}$ -200 mesh mineralized gabbro, $\mu=5*10^{-2}$

TIME (Hrs)	pH	So^{-2} (μg/l)	Ni (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75		3.1	22	6	250	1.8	0.36
5		2.8	32	6	130	2.3	0.76
27		3.1	70	2	8	2.4	0.71
52		3.4	97	2	7	2.5	0.80
218		4.6	275	0.3	9	2.6	0.85
431		7.7	520	2	4	2.7	0.98
531		9.1	670	2	23	2.6	1.05
698		10.8	860	2	13	2.8	1.03
1034		16.1	1200	4	4	3.2	1.20
4-AD-7			74	180	650	0.8	0.066

REACTOR 5: 5.71 $g\ell^{-1}$ -200 mesh mineralized gabbro, $1.29*10^{-4}$ phthallic acid

TIME (Hrs)	ph	So^{-2} (μg/l)	Ni (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75		2.8	6	0.6	45	0.9	0.21
5		3.0	9	1.3	41	1.0	0.30
27		3.0	14	0.1	13	1.0	0.30
52		2.8	23	0.6	16	1.1	0130
218		3.0	63	0.7	45	1.0	0.38
431		3.3	100	0	44	1.0	0.38
531		3.4	130	0.9	51	1.0	0.36
698		3.9	180	3	48	1.2	0.50
1034		5.3	210	3	44	1.1	0.6
5-AD-7			58	7	210	0.8	0.045

REACTOR 6: 5.71 $g\ell^{-1}$ -200 mesh mineralized gabbro, $1.29*10^{-3}$ phthallic acid

TIME (Hrs)	pH	So^{-2} (μg/l)	Ni (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
0.75		3.2	11	0	84	1.6	0.38
5		2.7	13	0	41	1.7	0.34
27		3.1	13	o	74	1.7	0.32
52		2.8	16	0.4	31	1.7	0.38
218		3.3	34	0.6	74	1.7	0.40
431		3.0	55	0	52	1.6	0.46
531		3.5	64	0	55	1.6	0.46
698		3.2	78	2	62	1.6	0.48
1034		3.4	110	7	120	1.7	0.52
6-AD-7		3.9	49	35	180	0.5	0.09
BOg H_2O		6.4	0	3	164	7.8	8.9
GRND H_2O			4	4	37	18.9	10.2

Table A.7

Experiment 8

REACTOR 1: 50 $g\ell^{-1}$ tailings, pH 4

TIME (Hrs)	PH=4	Ni (μg/l)	So^{-2} (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
1		22	2.7	0	78	13.9	0.9
27		77	2.7	5.6	2,450	23.4	2.8
218		440	4.1	170	16,300	23.5	7.1
432		550	6.4	320	33,200	25.6	12.4
532*		650	8.1	260	38,100	26.8	14.2
0.5		23	3.0	24	1,870	0.8	0.4
22		550	3.1	40	3,770	1.6	1.4
190		840	3.8	44	9,130	2.9	2.9

REACTOR 2: 50 $g\ell^{-1}$ tailings, pH 6

TIME (Hrs)	PH=4	Ni (μg/l)	So^{-2} (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
1		5	2.9	0	97	11.2	0.7
27		11	2.7	0	28	17.8	1.0
218		92	3.2	0.6	93	18.1	2.3
432		140	3.5	0	75	18.8	3.1
532*		160	3.7	0.4	66	19.3	3.4
95		5	2.8	0.1	34	0.8	0.1
22		41	3.0	4.1	60	2.6	0.4
190		100	2.8	3.7	48	3.1	0.7

REACTOR 3: 50 $g\ell^{-1}$ tailings, natural pH

TIME (Hrs)	PH	Ni (μg/l)	So^{-2} (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
1	9.10	3	3.2	1	76	7.7	0.3
27	8.02	1	2.9	2	25	13.9	0.6
218	8.08	1	3.1	1	25	14.0	0.9
432	8.01	7	3.7	2	63	14.2	1.1
532*	8.12	2	4.0	2	22	14.7	1.2
0.5	6.88	9	3.1	4	76	1.3	0.1
22	7.50	11	3.1	5	260	5.1	1.2
190	7.94	7	3.7	2	49	4.4	2.3

*Solids removed for 5 day drying period

Table A.7

Experiment 8

REACTOR 4: 50 $g\ell^{-1}$ tailings, bog water

TIME (Hrs)	PH	Ni (μg/l)	So^{-2} (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
1	8.16	5	3.8	9	180	9.3	6.5
27	8.17	5	3.8	9	180	9.4	5.7
218	8.26	16	4.2	10	130	11.0	6.4
432	8.16	15	5.3	14	92	11.5	7.3
532*	8.34	15	7.7	17	78	14.9	10.1
95	8.15	7	3.6	11	350	7.8	7.5
22	8.24	7	3.9	10	240	8.8	7.8
190	8.37	13	4.4	9	82	9.1	8.9
T=0		0	3.9	3	160	7.8	8.9

REACTOR 5: 50 $g\ell^{-1}$ tailings, ground water

TIME (Hrs)	PH	Ni (μg/l)	So^{-2} (mg/l)	Cu (μg/l)	Fe (μg/l)	Ca (mg/l)	Mg (mg/l)
1	8.15	5	7.8	6	91	17.5	8.1
27	8.20	3	7.2	5	30	17.0	7.7
218	8.24	4	7.4	29	80	17.3	8.5
432	8.18	4	9.7	32	29	18.2	9.0
532*	8.26	4	9.4	19	14	18.1	9.9
0.5	8.17	4	6.0	14	88	16.3	10.2
22	8.25	6	7.0	14	37	17.6	10.2
190	8.52	13	21.5	12	520	23.8	25.9
T=0		4	6.4	4	37	18.9	10.2

*Solids removed for 5 day drying period

Table A.8

Calcium and Magnesium Kinetics: Summary of Data

$[C] = kt^n$

k = moles, sec^{-1} (g gabbro)$^{-1}$

t = time

n = dimensionless

	Ca			Mg		
Reactor	$k \times 10^5$	r^2	n	$k \times 10^6$	r^2	n
2-1	2.39	.555	.021	2.94	.959	.140
2-2	.51	.987	.073	.009	.701	.531
2-3	.058	.931	.191	.061	.995	.286
2-4	.0095	.968	.300	.031	.957	.322
2-5	-	-	-	5.5	.727	.083
2-6	.38	.766	.038	.44	.971	.154
3-1	.36	.966	.114	.44	.995	.210
3-2	1.54	.619	.028	4.16	.957	.091
3-3	2.43	.834	.045	3.78	.983	.131
3-4	2.90	.966	.065	2.26	.986	.204
3-5	1.09	.916	.084	1.35	.980	.139
3-6	1.79	.903	.043	.94	.932	.198
4-1	1.11	.931	.119	.16	.965	.247
4-2	.77	.974	.180	.033	.936	.299
4-3	.63	.885	.129	.0019	.945	.481
4-4	.0089	.901	.091	.0075	.865	.178
4-5	.43	.586	.042	1.40	.947	.132
4-6	1.32	.533	-.037	3.88	.583	.082
5-1	.39	.618	.076	.71	.936	.163
5-2	.88	.248	.026	.77	.986	.159
5-3	.47	.931	.085	.59	.927	.162
5-4	.76	.559	.077	.35	.832	.216
5-5	.77	.536	.088	1.32	.953	.163
5-6	2.05	.019	-.002	.58	.992	.300
6-1	.16	.965	.139	.33	.996	.230
6-2	.099	.980	.167	.26	.997	.239
6-3	.18	.970	.106	.33	.986	.203
6-4	.20	.917	.095	.49	.991	.166
7-1	2.37	.700	.036	44.7	.575	.031
7-2	6.66	.614	.010	63.3	.292	.010
7-3	.43	.842	.054	.72	.964	.154
7-4	.52	.870	.059	1.13	.855	.131
7-5	.34	.432	.022	.64	.792	.111
7-6	.73	.008	-.001	1.56	.589	.051
8-1a	.34	.900	.096	.019	.981	.434
8-1b	.008	.987	.213	.027	1.000	.333
8-2a	.31	.882	.081	.060	.920	.258
8-2b	.007	.940	.238	.007	.993	.332
8-3a	.19	.841	.096	.042	.996	.215
8-3b	.014	.800	.223	.002	.972	.543
8-4a	.27	.599	.056	3.35	.286	.045
8-4b	.32	.975	.027	4.97	.803	.027
8-5a	.82	.255	.056	5.14	.489	.028
8-5b	.72	.205	.019	2.57	.604	.139

Note: Consult Table 3.10 for Reactor and Experiment conditions

Table A.8

Mineral Analysis of Gabbro

Mineral	Formula	Weighted Mean Percent Unmineralized	Weighted Mean Percent Mineralized
Plagioclase	$NaAlSi_3O_8$-$CaAl_2Si_2O_8$	64	47
Sericite	$K_2Al_4Si_6Al_2O_{22}(OH,F)_4$	1.6	0.069
Olivine	$(Fe,Mg)_2SiO_4$	14	11
Clinopyroxene	$Ca(Fe,Mg)Si_2O_6$	7.6	26
Monocrystalline Amphibole	$(Na,K)_{0-1}(Ca,Mg,Fe^{+2},Fe^{+3},Al)_7 \cdot (Si_{6-8}Al_{2-0})O_{22}(OH,F,Cl)_2$ non-acicular morphology	0.054	-
Fibrous Amphibole	Same as Above; fibrous or acicular morphology	0.005	-
Chlorite	$(mg,Fe,Al)_{12}(Si,Al)_8O_{20}(OH)_{16}$	1.7	0.40
Serpentine	$Mg_3Si_2O_5(OH)_4$	0.016	0.014
Iodingsite	Serpentine + iron oxide phase	0.017	0.053
Talc	$Mg_6Si_8O_{20}(OH)_4$	-	-
Biotite	$K_2(Mg,Fe^{2+})_{6-4}(Fe^{3+},Al,Ti)_{0-2} \cdot (Si_{6-5}Al_{2-3})O_{20}(OH,F)_4$	3.3	5.0
Smectite	$(1/2\ Ca,Na)_{0.7}(Al,Mg,Fe)_4(Si,Al)_8 \cdot O_{20}(OH)_4 \cdot NH_2O$	-	-
Celadonite	$KMgFe^{3+}Si_4O_{10}(OH)_2$	0.014	-
Orthopyroxene	$(Fe,Mg)_2Si_2O_6$	2.8	2.3

Table A.8 (co)

Mineral Analysis of Gabbro

Mineral	Formula	Weighted Mean Percent Unmineralized	Weighted Mean Percent Mineralized
Opaque Minerals (Following Six)			
Chalcopyrite-Cubanite	$CuFeS_2$-$CuFe_2S_3$	0.13	7.9
Pentlandite	$(Fe,Ni)_9S_8$	0.032	1.3
Pyrrhotite	Fe_7S_8-FeS	0.0403	0.34
Ilmenite	$FeTiO_3$	4.0	3.1
Magnetite	Fe_3O_4		3.1
Graphite	C	-	0.025
Spinel	$MgAl_2O_4$	-	-
Myrmekitic Intergrowths	Generally between plagioclase 7 olivine	-	-
Apatite	$Ca_5(PO_4)_3(OH,F,Cl)$	0.18	0.074
Epidote	$CaFe^{3+}Al_2Si_3O_{12}(OH)$	0.17	-
Allanite	$(Ca,Ce)_2(Fe^{2+},Fe^{3+})\cdot Al_2Si_3O_{12}OH$	-	-
Calcite	$CaCO_3$	0.026	0.006
Quartz	SiO_2	-	-
Cordierite	$Al_3(Mg,Fe^{+2})_2Si_5AlO_{18}$	-	-
	Σ=	104.6	107.7

Table A.9

Elemental Analyses: All Values in Weight Percent

	Mineralized			Unmineralized		
Element	NAA	Acid Digestion	Acid[1] Digestion	NAA	Acid Digestion	Acid Digestion
Ag	0.00046			<0.00014		
Al	4.55			8.04		
As	0.00073			0.00008		
Au	0.000003			0.000004		
Ba	0.036			0.034		
Ca	4.69	2.27		7.08	2.55	
Cd	<0.00065			<0.00029	0	
Co	0.039			0.082	0.01	
Cr	0.011			0.014		
Cu	1.18	1.41	0.85	9,156	0.245	0.28
Fe	20.18	17.2	15.17	13.14	10.4	11.13
Hg	0.0009			<0.00001		
Mg	3.12	1.92		2.000	2.05	
Mn	0.134			0.138	0.265	
Na	1.76			2.050		
Ni	0.624	0.355	0.23	0.138	0.0954	0.10
S			3.92			1.1
Sb	<0.00007			<0.00005		
Sc				<0.00017		
Se	0.00062					
Sn	<0.048			<0.050		
Sr	<0.057			0.062		
Ti	1.32					
V	0.012			0.018		
W	0.00014			<0.00016		
Zn	0.012			<0.00088		

[1] Mineral Resource Research Center, University of Minnesota

Attachment 3

Eger P, Johnson B, Hohenstein G (1979) 1978 DNR/AMAX Field Leaching and Reclamation Program: Progress Report on the Leaching Study. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN. 18 p.

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1978 DNR/AMAX FIELD LEACHING AND RECLAMATION PROGRAM

Progress Report on the Leaching Study

January 29, 1979

Minnesota Department of Natural Resources
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LAPAKKO

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Prepared by
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Bruce Johnson
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FOREWORD

This progress report describes the leaching portion of the joint DNR/Amax leaching and reclamation study. It contains the raw water quality data collected during 1978. Some analysis of the data is presented, but more is needed and planned. Additional literature work is also planned. Supporting references will be cited in future reports and the discussion section will be expanded. Analysis plans are discussed in this report and additional results will be presented as they are completed.

This document is preliminary but it does serve as a starting point for continued discussion of the leaching program. Comments would be appreciated.

ABSTRACT

Runoff from lean ore and waste rock Duluth gabbro stockpiles may pose significant environmental problems. Water quality samples collected from gabbro stockpiles have contained elevated concentrations of trace metals (copper, nickel, cobalt and zinc.) Observed metal concentrations in leachate samples range from 10 to 10,000 times the natural background concentrations of streams in the area.

In 1978, a joint DNR/Amax program was established to study the leaching and reclamation of stockpiled gabbro. Six test piles (FL 1-6) were constructed in 1977. In 1978, two were covered with glacial till (piles 3,5) and one with topsoil (pile 2). These were then fertilized and planted.

Intensive sampling was conducted on FL-4 during two rain events, to determine the fluctuation in runoff concentration over the storm. Copper concentration remained essentially constant while nickel and sulfate concentration varied considerably. Mass release followed the runoff hydrograph, with the maximum release occuring at peak discharge.

Preliminary results indicate that there is a difference in the amount of runoff from each pile. Piles 5 and 6 yield about 65% runoff; piles 1,3 and 4, 50-55%; and pile 2 35%. It appears that the glacial till treatment did little to control runoff, while the topsoil treatment reduced flow. Vegetation was sparse on the till pile but good growth was observed on the topsoil pile.

Both flow and concentration data indicate that pile 5 and 6 are different from piles 1 - 4. Pile 5 has produced acid leachate and the pH in pile 6 began to decline at the end of the field season. Metal concentrations in samples from piles 5 and 6 are one to two orders of magnitude higher than in samples from piles 1 - 4; the maximum nickel concentration observed was at pile 5, 83 mg/l. The median concentration values for pH, nickel sulfate are comparable for piles 1 - 4. Copper concentrations in samples collected from piles 2 and 3 increased in 1978. This increase appears to

be the result of increased copper mobility, rather than increased sulfide dissolution. Organic compounds removed from the overlying soil cover could produce a increase in copper mobility.

OVERVIEW

In 1974, Amax Exploration Inc. reached an agreement with Bear Creek Mining Company, and assumed Bear Creek's leases for properties located in T 60N R 12W, about 4 miles S.E. of Babbitt, Minnesota. As a significant proportion of the leases involved state land and state minerals, Amax requested permission from the State to construct an exploratory shaft.

In 1974, very little information existed on the potential environmental problems that might be associated with developing Minnesota's copper-nickel resources. The state and Amax agreed that the Amax project could serve as a pilot study. A comprehensive monitoring network was developed and an agreement was reached to conduct leaching tests on stockpiled materials.

In 1977, Amax constructed six (6) test piles containing lean ore material. The mass, grade, and completion date of each pile is given in Table I. Each pile was placed on an impervious liner (30 mil reinforced Hypalon) and all runoff is collected via a six (6) inch perforated drain line. The details of pile construction and pile layout are shown in Figures 1 and 2.

In 1977, Water quality data was collected periodically from each pile and has been present in a report "Field Leaching Summary Report 1977", prepared by Amax Environmental Services, Inc. May, 1978.

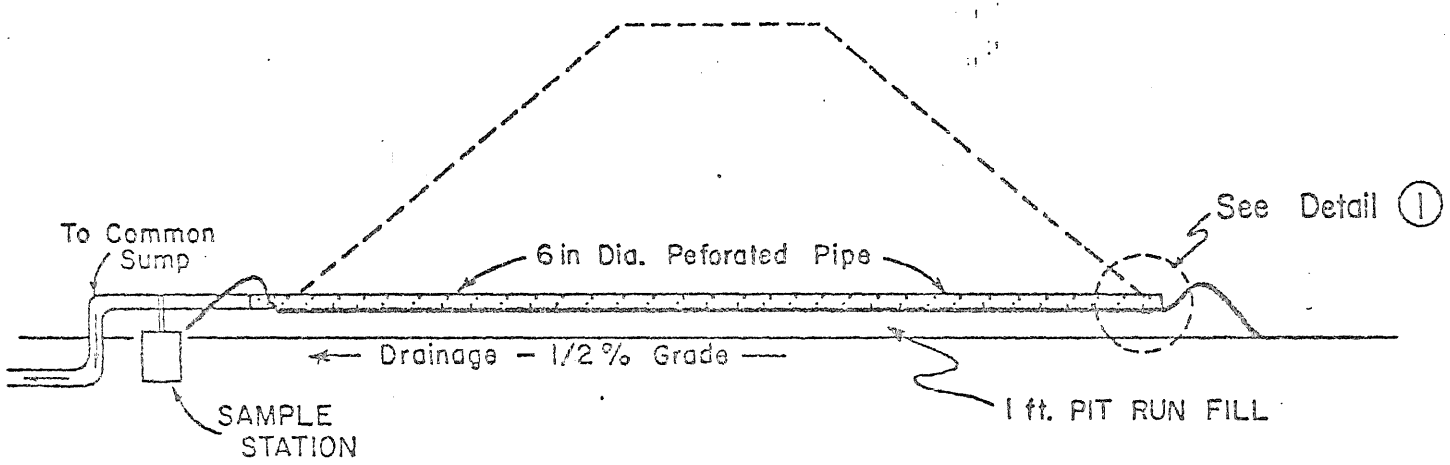
In 1978, a joint DNR/Amax program was initiated to study the leaching and revegetation of the lean ore test piles. This is a unique project not only from a scientific viewpoint but also from the stand point of industry-government cooperation. A concerted effort is being made by both the state and the company to identify significant environmental problems and to study methods of mitigation. In the joint program, Amax is responsible for the revegetation program and the DNR is responsible for the leaching aspects of the study. This is a preliminary progress

TABLE 1. COMPONENTS OF FIELD LEACHING PILES

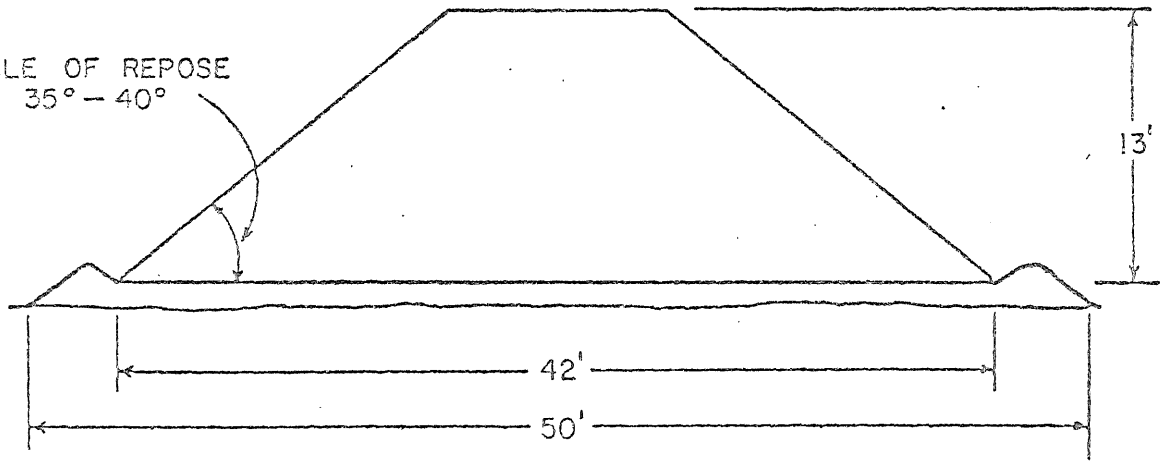
	Pile No.	Size (Tons)	Completion Date	Average Mineral Content % Cu	% Ni	% S	
	1	1766	April 20, 1977	.33-.35	.075	0.6	NONE
	2	1766	April 20, 1977	.33-.35	.075	0.6	TILLAGE & [illegible]
	3	1766	April 20, 1977	.33-.35	.075	0.6	TILLAGE
	4	1765	April 20, 1977	.33-.35	.075	0.6	NONE
	5	1951	Sept. 10, 1977	0.29	0.09	1.45	TILL & ELVIS
	6	1672	Sept. 30, 1977	0.38	0.09	0.83	NONE
otal	Piles 1-6	10,686		0.33	0.08	0.79	

(ton is short ton = 2000 lbs)

Figure 1 - Pit Construction

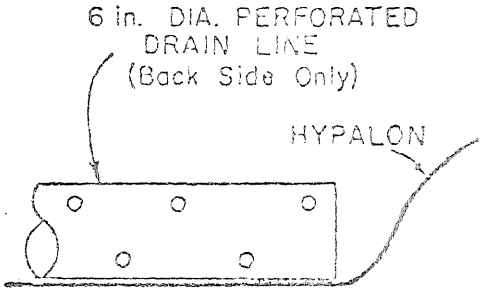


SECTION B-B
1" = 10'



SECTION C-C
1" = 10'

DETAIL 1
(NOT TO SCALE)



PIT RUN FILL

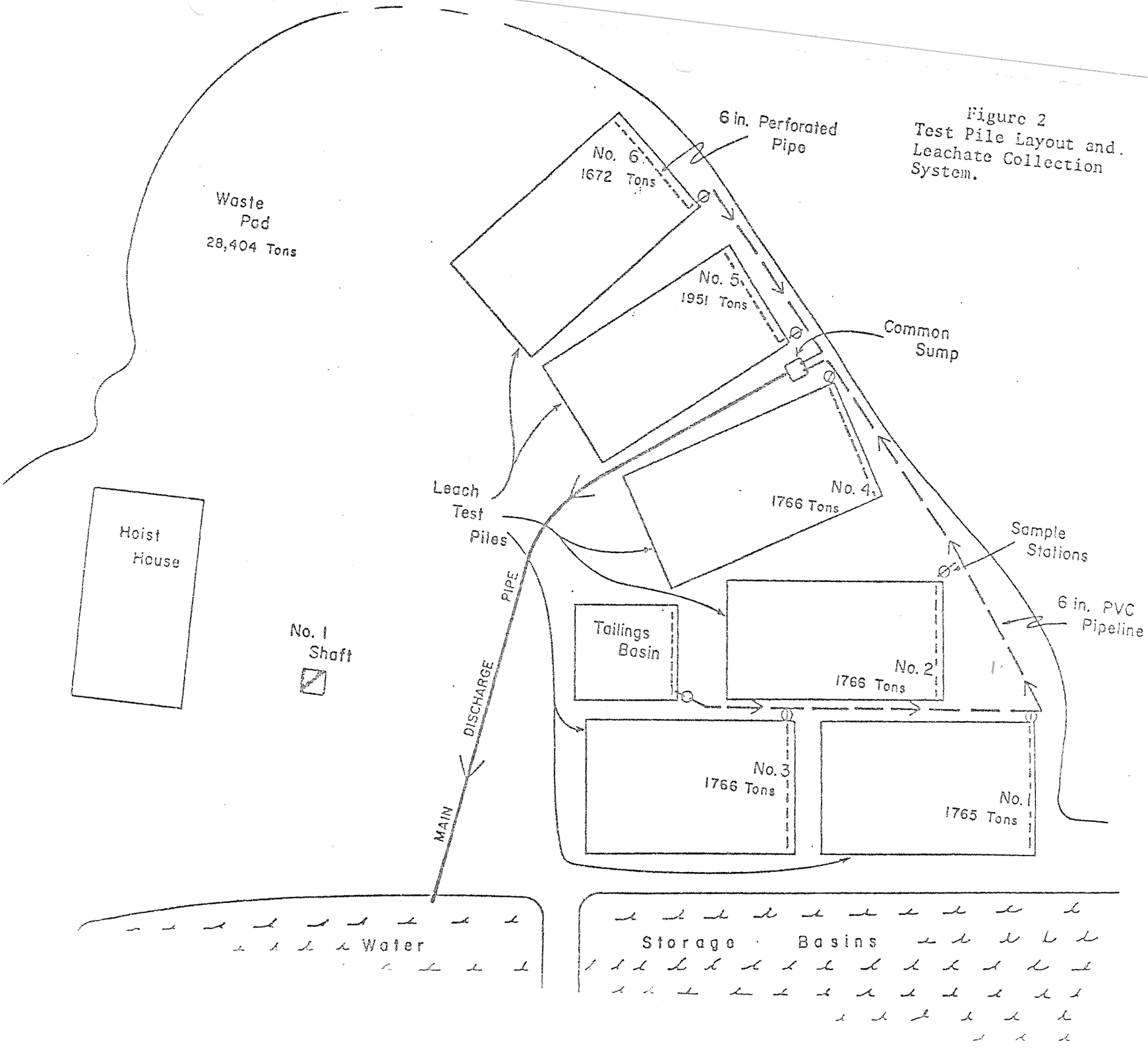


Figure 2
Test Pile Layout and
Leachate Collection
System.

report on the leaching program. Water quality data, flow data and rainfall data that have been collected are presented. Some preliminary discussion of the results is provided. Data analysis is continuing and additional results will be included in future reports.

I)ODUCTION

Historically one of the major problems associated with the mining of metal sulfide deposits has been the release of toxic trace metals through the leaching of mining wastes (Hawley, 1972, Eisenreich et al, 1976). In most documented cases of leaching problems, the leachate is acidic, often with pH values as low as 2 to 3.

Most of the field studies reported in the literature involved materials that contained a high concentration of sulfide minerals. Hawley reports that acid leachate has been associated with materials that contain 2.5 to 70 percent sulfide. The Duluth gabbro is a low grade disseminated deposit. "Model" lean ore and waste rock concentrations developed by the Regional Copper Nickel Study (R. Stevenson, personal communication) range from .655 percent sulfur for lean e to .207 for waste rock. Even though the gabbro is in general a low grade deposit, early water quality samples (1974) collected near stockpiled gabbro at Erie Mining Company's Dunka Pit, near Babbit, MN, indicated elevated levels of trace metals. Studies conducted by the Regional Copper Nickel Study (Level II leaching report, soon to be released) have confirmed the findings of the early samples. Metal concentrations in leachate samples are 10 to 10,000 times the natural background concentrations.

Lean ore and waste rock stockpiles, unless controlled, may pose significant environmental problems. Not only is there evidence that stockpile runoff will contain elevated metal concentrations, but the mass of material that will be stockpiled is extremely large. For its open pit prospect, Amax has projected a stockpile area of 3400 acres, 500-600 feet high.

The Duluth gabbro complex contains a large percentage of silicate minerals. As part of the Regional Copper-Nickel Study, samples of the gabbro were collected

across the contact from Hoyt Lakes to Ely (more detail will be given in the Regional Copper Nickel report on geology and mineralogy). The mineralogy of these samples was determined and a complete chemical analysis was made on each sample. Five samples from the Amax area were analyzed, one of which was selected from leach pile FL-1. The mineralogical composition is shown in Table II and the chemical composition in Table III. A description of the mineral forms is given in Table IV. Over ninety percent of the rock is composed of silicate minerals, the three major groups being plagioclase, olivine, and pyroxene. The sulfides make up a very small percentage of the rock, approximately 0.655 percent.

Generally, most of the sulfides found in mining wastes are unwanted iron sulfides, either in the form of pyrite or pyrrhotite. The percent of iron sulfide in the test pile material is quite low (Table II) and this is generally true for the disseminated material across the region (R. Stevenson, personal communication). Only in pockets of massive or semi-massive ore are high concentrations of iron sulfides found. Tables V and VI include the analysis of a semi-massive sample collected from the Amax test shaft. This sample contained 10.5 percent sulfur, most of which was associated with the pyrrhotite which comprised 12.8% of the sample.

Table II MINERALOGICAL COMPOSITION OF FL-1 (volume %)

Plagioclase	59.112
Sericite	2.176
Olivine	10.510
Clinopyroxene	11.185
Orthopyroxene	3.716
Monocrystalline amphibole	3.567
Fibrous amphibole	0.288
Chlorite	1.136
Serpentine	0.257
Iddingsite	0.075
Talc	-
Biotite	1.738
Smectite	0.021
Celadonite	-
Opaques*	5.098
Chalcopyrite-cubanite	0.769
Pentlandite	0.037
Pyrrhotite	0.844
Ilmenite-magnetite	3.447
Graphite	-
Spinel	-
Myrmekite	-
Apatite	0.085
Epidote	0.953
Allanite	-
Calcite	0.056
Quartz	-
Cordierite	0.027

* The value shown for opaques is the sum of the five following values

Table III CHEMICAL COMPOSITION OF FL-1

Element			Units
Si (SiO_2)	22.80	(48.81)	Pct
Al (Al_2O_3)	8.98	(16.96)	Pct
Fe (Fe(O))	9.22	(11.86)	Pct
Mg (MgO)	4.31	(7.15)	Pct
Ca (CaO)	5.55	(7.76)	Pct
Na (Na_2O)	2.00	(2.70)	Pct
K (K_2O)	.32	(.39)	Pct
Ti (TiO_2)	1.41	(2.35)	Pct
P (P_2O_5)	.02	(.05)	Pct
Mn (MnO)	.12	(.16)	Pct
Cr (Cr_2O_3)	.03	(.05)	Pct
B	662.00		PPM
Ba	1173.00		PPM
Be	1.00		PPM
Sr	279.00		PPM
V	276.50		PPM
Th	6.40		PPM
Zr	80.00		PPM
S	.655		Pct
Cu	.306		Pct
Ni	.073		Pct
Fe (S)	.696		Pct
Co	.009		Pct
Zn	157.00		PPM
Pb	0		PPM
Ag	1.35		PPM
As	0		PPM
Hg	<1.00		PPB

Table III (cont'd) CHEMICAL COMPOSITION OF FL-1

Element		Units
Mo	.50	PPM
Cd	0	PPM

Pct = Weight percent
PPM = parts per million by weight
PPB = parts per billion by weight

Table IV. Mineral Names, Abbreviations and Formulas

LAG	Plagioclase	$Na\ Al\ Si_3\ O_8$ - $Ca\ Al_2\ Si_2\ O_8$	a feldspar
ERCT	Sericite	$K_2\ Al_4\ Si_6\ Al_2\ O_{22}\ (OH, F)_4$	a mica, alters from feldspar
OL	Olivine	$(Fe, Mg)\ SiO_4$	Fe-Mg silicate
CPX	Clinopyroxene	$Ca\ (Fe, Mg)\ Si_2\ O_6$	a pyroxene, Fe-Mg silicate
OP	Orthopyroxene	$(Fe, Mg)_2\ Si_2\ O_6$	a pyroxene, Fe-Mg silicate
AMP	Monocrystalline amphibole	$(Na,K)_{0-1}\ (Ca, Mg, Fe^{+2}, Fe^{+3}, Al)_7$ $(Si_{6-8}\ Al_{2-0})\ O_{22}\ (OH, F, Cl)_2$	amphibole with non-acicular morphology
BAMP	Fibrous Amphibole	$(Na,K)_{0-1}\ (Ca, Mg, Fe^{+2}, Fe^{+3}, Al)_7$ $(Si_{6-8}\ Al_{2-0})O_{22}\ (OH, F, Cl)_2$	amphibole with fibrous or acicular morphology
HLRT	Chlorite	$(Mg, Fe, Al)_{12}\ (Si, Al)_8\ O_{22}\ (OH)_{16}$	alters from Fe-Mg silicates
SRPN	Serpentine	$Mg_3\ Si_2\ O_5\ (OH)_4$	alters from Fe-Mg silicates
IDNGT	Iddingsite	Serpentine + iron oxide phase	alters from olivine
TALC	Talc	$Mg_6\ Si_8\ O_{20}\ (OH)_4$	alters from Fe-Mg silicates
BIOT	Biotite	$K_2\ (Mg, Fe^{+2})_{6-4}\ (Fe^{+3}, Al, Ti)_{0-2}$ $(Si_{6-5}\ Al_{-3})\ O_{20}(OH, F)_4$	a mica
SMCTT	Smectite	$(\frac{1}{2}\ Ca, Na)_{0.7}\ (Al, Mg, Fe)_4\ (Si, Al)_8$ $O_{20}\ (OH)_4 \cdot nH_2O$	a clay, alters from feldspar
GLDNT	Celadonite	$K\ Mg\ Fe^{+3}\ Si_4\ O_{10}\ (OH)_2$	a mica, similar to glauconite
OPQ	Opaque minerals, sum of following seven minerals*		
CP-CB	*Chalcopyrite *Cubanite	$Cu\ Fe\ S$ $Cu\ Fe_2\ S_3$	undifferentiated
PN	*Pentlandite	$(Fe, Ni)_9\ S_8$	
PO	*Pyrrhotite	$Fe_7\ S_8$ - $Fe\ S$	
ILMETIO	*Ilmenite	$Fe\ Ti\ O_3$	

	*Magnetite	$Fe_3 O_4$	contains appreciable titanium (20 wt%)
.RPHT	*Graphite	C	present as flakes
?NL	Spinel	$Mg Al_2 O_4$	
MYRM	Myrmekitic intergrowths, generally between plagioclase and olivine		
APAT	Apatite	$Ca_5 (PO_4)_3 (OH, F, Cl)$	
.?I	Epidote	$Ca Fe^{+3} Al_2 Si_3 O_{12} (OH)$	(includes zeolites and prehnite ($Ca_2 Al Al Si_3 O_{10} (OH)$)
ALNT	Allanite	$(Ca, Ce)_2 (Fe^{+2} Fe^{+3} Al_2 Si_3 O_{12} :OH$	borders radiating epidote
CAL	Calcite	$Ca CO_3$	
TZ	Quartz	$Si O_2$	
CORD	Cordierite	$Al_3 (Mg, Fe^{+2})_2 Si_5 Al O_{18}$	associated with hornfels

rom R. Stevenson, Regional Copper-Nickel Study

Table V MINERALOGICAL COMPOSITION OF SEMI-MASSIVE, MINERALIZED ROCK SAMPLE (volume %)

Plagioclase	47.885
Sericite	0.091
Olivine	1.513
Clinopyroxene	2.656
Orthopyroxene	18.472
Monocrystalline amphipole	0.025
Fibrous amphibole	0.024
Chlorite	0.145
Serpentine	-
Iddingsite	-
Talc	-
Biotite	4.475
Smectite	-
Celadonite	-
Opaques*	19.239
Chalcopyrite-cubanite	3.231
Pentlandite	0.161
Pyrrhotite	12.816
Ilemnite-magnetite	2.564
Graphite	0.467
Spinel	-
Myrmekite	-
Apatite	0.118
Epidote	-
Allanite	-
Calcite	-
Quartz	0.037
Cordierite	5.350

* The value shown for opaques is the sum of the five following values

Table VI CHEMICAL COMPOSITION OF SEMI-MASSIVE, MINERALIZED ROCK SAMPLE

Element			Units
Si (SiO_2)	18.15	(38.85)	Pct
Al (Al_2O_3)	6.43	(12.15)	Pct
Fe (Fe(O))	7.71	(9.92)	Pct
Mg (MgO)	2.69	(4.46)	Pct
Ca (CaO)	3.44	(4.81)	Pct
Na (Na_2O)	1.18	(1.59)	Pct
K (K_2O)	.73	(.88)	Pct
Ti (TiO_2)	.65	(1.08)	Pct
P (P_2O_5)	.03	(.07)	Pct
Mn (MnO)	.11	(.14)	Pct
Cr (Cr_2O_3)	.04	(.06)	Pct
B	428.00		PPM
Ba	310.00		PPM
Be	1.20		PPM
Sr	151.50		PPM
V	213.00		PPM
Th	0		PPM
Zr	59.50		PPM
S	10.500		Pct
Cu	1.155		Pct
Ni	.418		Pct
Fe (S)	13.885		Pct
Co	.036		Pct
Zn	185.00		PPM
Pb	0		PPM
Ag	0		PPM
As	0		PPM
Hg	<1.00		PPB

Table VI (cont'd) CHEMICAL COMPOSITION OF SEMI-MASSIVE, MINERALIZED ROCK SAMPLE

Element		Units
Mo	5.50	PPM
Cd	0	PPM

Pct = Weight percent
PPM = parts per million by weight
PPB = parts per billion by weight

1978 FIELD PROGRAM

The overall goals of the joint leaching/reclamation program are measurement of the release of trace metals, sulfate, nutrients and chloride from the lean ore test piles and evaluation of the effect of various reclamation procedures on that release. To accomplish these goals the field program was separated into two phases.

The first phase was a characterization phase. Data was collected on the rock characteristics, top dressing characteristics (soil and fertilizer) and rainfall. In addition, the variability of leachate composition during a storm event was determined.

Phase II was a monitoring phase, in which data were collected to determine the total mass released from each pile. The total mass release will be used to evaluate the effectiveness of different reclamation procedures.

METHODS

As this was the first year of the program, a large effort was required to initiate the program, develop methodology, and provide instrumentation for the leach piles. The details of the problems encountered and the attempts to overcome them are given in Appendix I.

The critical parameters that had to be measured were 1-rainfall; the amount, duration, time of the event, and pH, 2-the discharge from individual piles and 3-a concentration representative of the runoff.

Rainfall

Rainfall is recorded by a standard rain gauge (Science Associates, Inc., NWS Spec. 450.230) and a recording rain gauge (Science Associates, Inc., NWS Spec. 450.220). The standard gauge is more accurate (± 0.01 inches) and is used to

determine the amount of rainfall while the recording gauge gives the duration and time of the rain event.

To collect samples for rainfall pH, a polyethylene funnel was attached to the outside of the weather instrument shelter. This funnel was connected by tygon tubing to a teflon bottle inside the shelter. The pH was recorded as soon as possible after rainfall ceased.

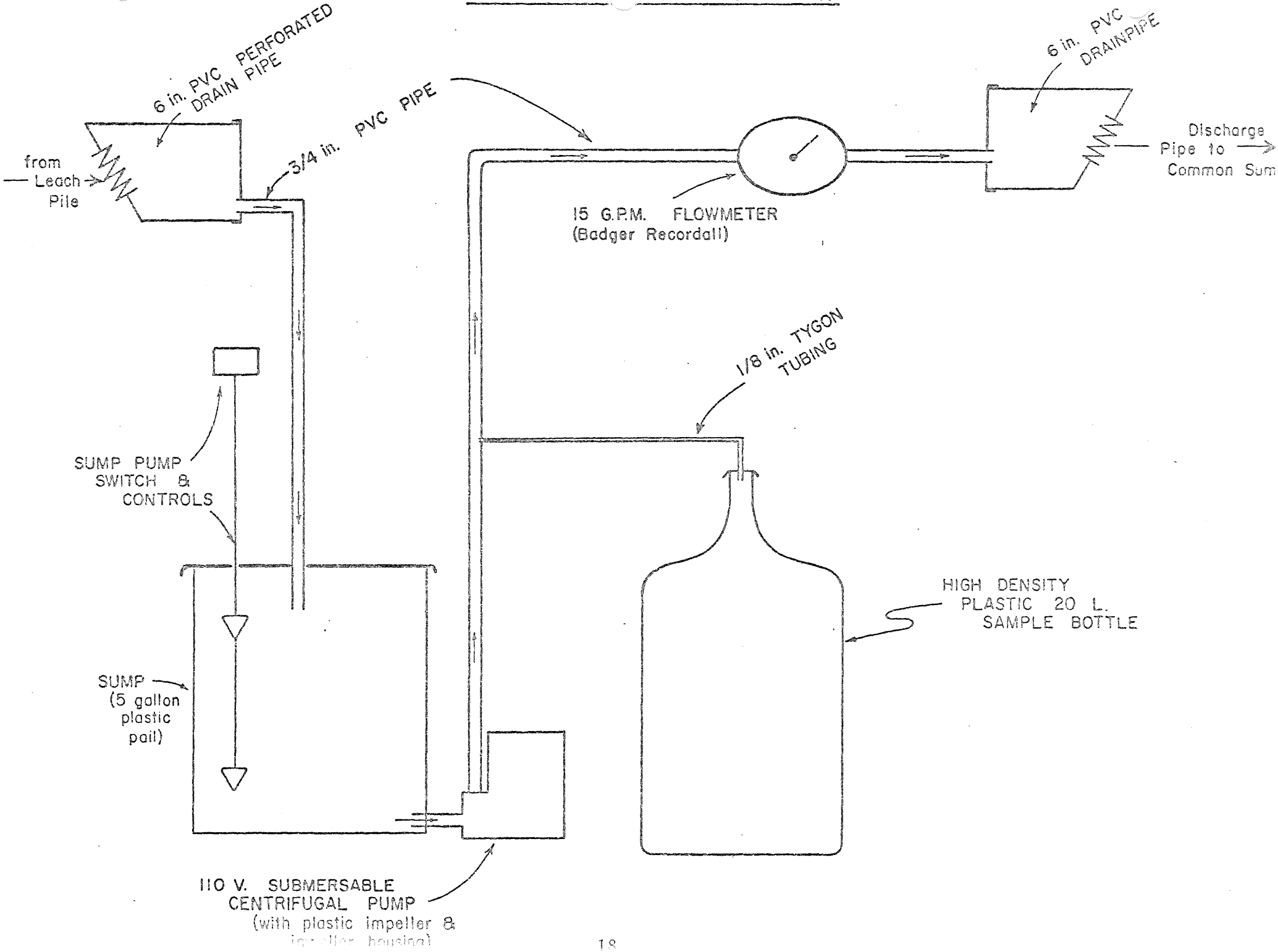
Flow

Daily volumetric flow measurements were made on each pile. Each pile is equipped with a flow meter (Badger Recordall Model 15). In July, fine mesh screens were installed upstream of the meters; and the slope was adjusted to increase the pressure drop through the meter. After these changes were made the meters worked satisfactorily at high flow, but failed to operate at low flow. Daily discharge from each pile was estimated using a combination of meter readings and volumetric measurements. Details and sample calculations are shown in Appendix II.

Water Quality Samples

The development of a satisfactory sampling device required a substantial amount of trial and error. As a result, the samples collected in 1978 are a mixture of composite and grab samples. A flow proportional sampler (Figure 3) was developed and tested on pile FL 4 and worked satisfactorily. This system provides a sample over an event that is weighted by flow. Concentrations have been shown to vary over a storm event (see Results-Phase I). In order to get an accurate measure of the total mass released from a pile, either a series of samples must be collected during each event or one flow weighted sample must be collected. The basic concept is that if 50% of the flow occurs in a 4 hour period then 50% of the volume of the sample should be collected in this period.

FLOW PROPORTIONAL SAMPLER



In our system, each time the pump discharges an aliquot of constant volume is placed in the sampling container. The higher the flow, the more often the pump discharges and the greater the volume of sample collected. This system has now been installed on all piles. (For more details refer to Appendix I).

RESULTS - PHASE I

Characterization

The objective of the characterization work is to provide information on all the test materials. Characterization data for the stockpiles are given in Table VII. Particle size distribution and surface area are estimates based on work done by the Regional Copper Nickel Study (Details of the calculation will be presented in the Level II Leaching Report). Piles 5 and 6 contain rock from the drifts which contain a higher degree of mineralization than the rock in the other four piles.

Additional mineralogical, chemical, and surface area information may be needed to clarify the differences between piles 1,2,3,4, and 5,6.

Top Dressing Characteristics

On May 11, test piles FL 3 & FL 5 were covered with glacial till and FL 2 with top soil. The volume and depth of soil cover for each pile is given in Table VIII. Composite samples of each soil type were collected. Each sample will be thoroughly mixed and split to obtain a representative sample. Particle size and total and extractable metal analysis will be performed by the Department of Natural Resources Laboratory in Hibbing. Soil classification, pH, and macro and micro nutrient analyses will be performed by the University of Minnesota Research Analytical Laboratory in St. Paul.

Fertilizer

Fertilizer was applied to test piles 2,3, and 5 on June 22. Sixteen kilograms (35.2 lbs) of a fertilizer with a guaranteed analysis of 10% NH_4NO_3, 20% P_2O_5, and 20% K_2O were applied to each pile. A composite sample of fertilizer was collected and will be analyzed for trace metal content.

TABLE VII. CHARACTERIZATION DATA FOR STOCKPILES

Pile	1	2	3	4	5	6
omposition						
%S	.6	.6	.6	.6	1.45	.83
%Cu	.33-.35	.33-.35	.33-.35	.33-.35	.29	..38
%Ni	.075	.075	.075	.075	.09	.09
ass (tons)	1766	1766	1766	1765	1951	1672
ollecting Ar (ft.^{2})	3173	3678	3178	3524	3648	3444
Surface rea stimate) m^2/ton	90-820	90-820	90-820	90-820	90-820	90-820
ate onstructed	(- - April 20 1977 - - - - - - - - - - - - -)				September 10 1977	September 3 1977
ock source	shaft	shaft	shaft	shaft	drift	drift
reatment	none	covered with topsoil and revegetated	covered with glacial till & revegetated	none	covered with glacial till & revegetated	none

* estimate based on an assumed particle size distribution (Detail of the calculation will be found in the Regional Copper Nickel Study Level II leaching report. This will be available soon.)

TABLE VIII. SOIL DEPTH AND FERTILIZER ADDITION

	Mean depth of soil cover (inches)	Mean width of pile (ft)	Length of pile (ft)	Volume of soil cover (ft^3)	(yds^3)	(m^3)	NH_4 added (g)	NO_3 added (g)	P added (g)	K added (g)
2	7.14	56	92	3065	114	87	360	1239	1397	1582
3	11.3	51	88	4219	156	119	360	1239	1397	1582
5	10.9	56	90	4578	170	130	360	1239	1397	1582

Rainfall

During 1977-78 monthly bulk precipatation samples were collected by the Regional Copper Nickel Study. Bulk samples collect both wet and dry fall, so concentrations may tend to be slightly higher than if only rainfall was sampled. Table IX gives the regional average which was based on data collected at four sites across the copper-nickel study area. More information will be available in the Regional Copper Nickel Study Technical Report on air/water interactions. Concentrations of all constituents are generally quite low.

In 1978, rainfall samples were collected at Amax and pH was analyzed from fourteen separate events. The data is given in Table X and summarized in Figure 4. In general, the rain tends to be acidic. Recent studies conducted in the area have indicated that there is already a trend for increasing rainfall acidity (D. Thingvold, personal communication). The exact effect this increasing acidity will have on the release from stockpiled materials is not known, but if the pH in the pile is lowered more metals may be released.

Variability of Runoff Concentration Over a Storm Event

Intensive sampling was conducted on FL4 for two storm events (0.35 inches on June 23 and 1.53 inches on July 7). The complete set of data for these events is in Appendix III. Figure 5 and 6 illustrate the variation in concentrations of copper, nickel, and sulfate during the runoff. On June 23, there was no flow prior to the storm. Nickel and sulfate concentrations rose rapidly, and then leveled off, while copper concentrations remained essentially constant throughout the runoff period. On July 7, there was a small base flow prior to the rain event. Nickel and sulfate concentrations were high in the base flow, decreased rapidly with the initial rise in flow, increased slightly during the peak runoff, decreased on falling limb of the hydrograph, and then rose gradually as flow diminished.

TABLE IX. REGIONAL RAINFALL AVERAGES

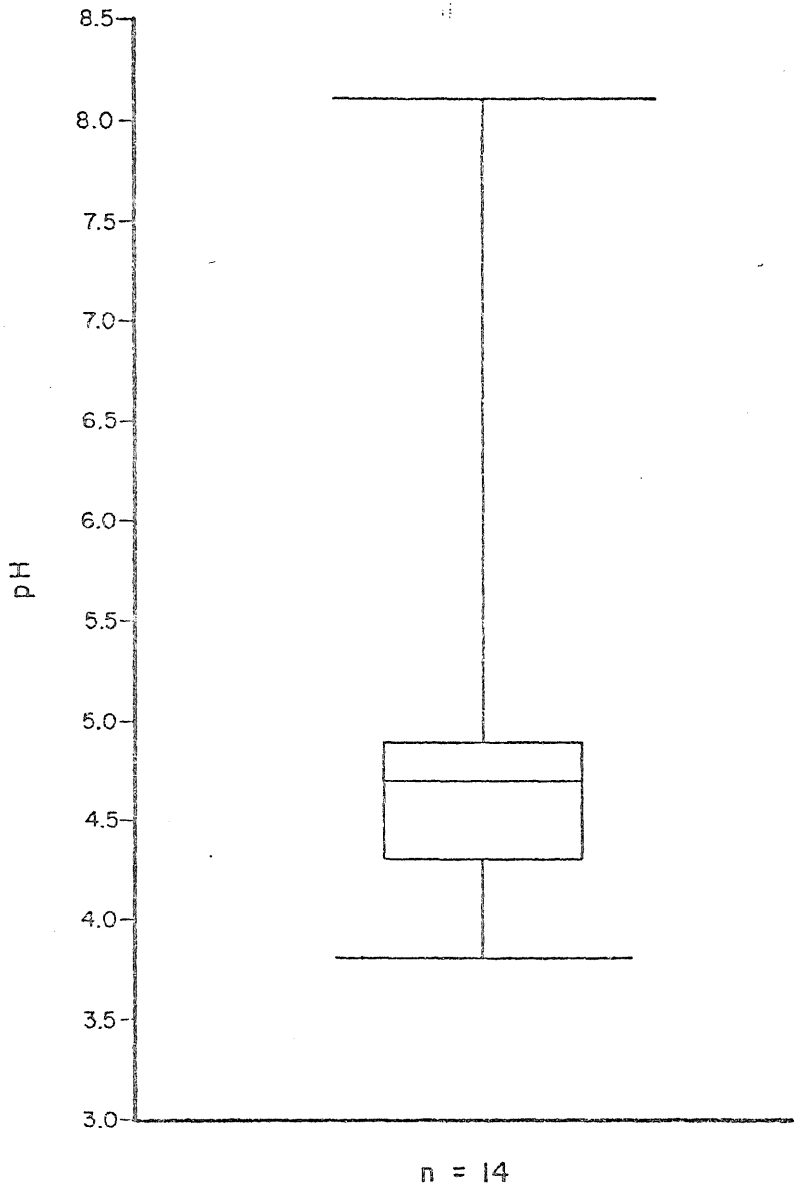
Parameter	
Specific Conductance, 25°C (umhos)	27
Suspended solids	12
Total dissolved solids, 180°C	10
NO_2,NO_3	.27
Total Phosphorus	.04
Total organic Carbon	4.1
Alkalinity, $CaCO_3$	$<$ 10
Cl	$<$1.9
Ca	1.6
Mg	$<$1.6
Na	$<$.5
K (mg/l)	$<$.5
F (mg/l)	$<$.1
Ni (ug/l)	2
Cd (ug/l)	.45
Zn (ug/l)	8.1
Pb (ug/l)	10.9
Al (ug/l)	120
Fe (ug/l)	($\sim$154)
As (ug/l)	$<$1.6
Cu (ug/l)	1.5

All concentrations in mg/l except as noted.

TABLE X
Rain pH

Date 1978	pH	Specific Conductance (umhos)	Remarks	$[H^+] \times 10^6$
6/07	5.2			6.31
6/23	6.3			0.501
7/02	4.9		pH taken 7/5, 1030	12.6
7/07	4.7			20.0
7/18	4.7			20.0
7/22	4.6	20	pH taken 7/24, 1100	25.1
7/24	4.9	15		12.6
7/28	8.1	18		7.94×10^{-3}
8/15	4.3			50.1
8/16	4.2	1.5		63.1
8/18	3.8	14		158
9/11	4.3			50.1
9/12	4.0			100
9/14	4.8		pH taken 9/15, 1610	15.8
n=14				
$\bar{x}$	4.4			38.2

Figure 4
RAINFALL pH, AMAX 1978



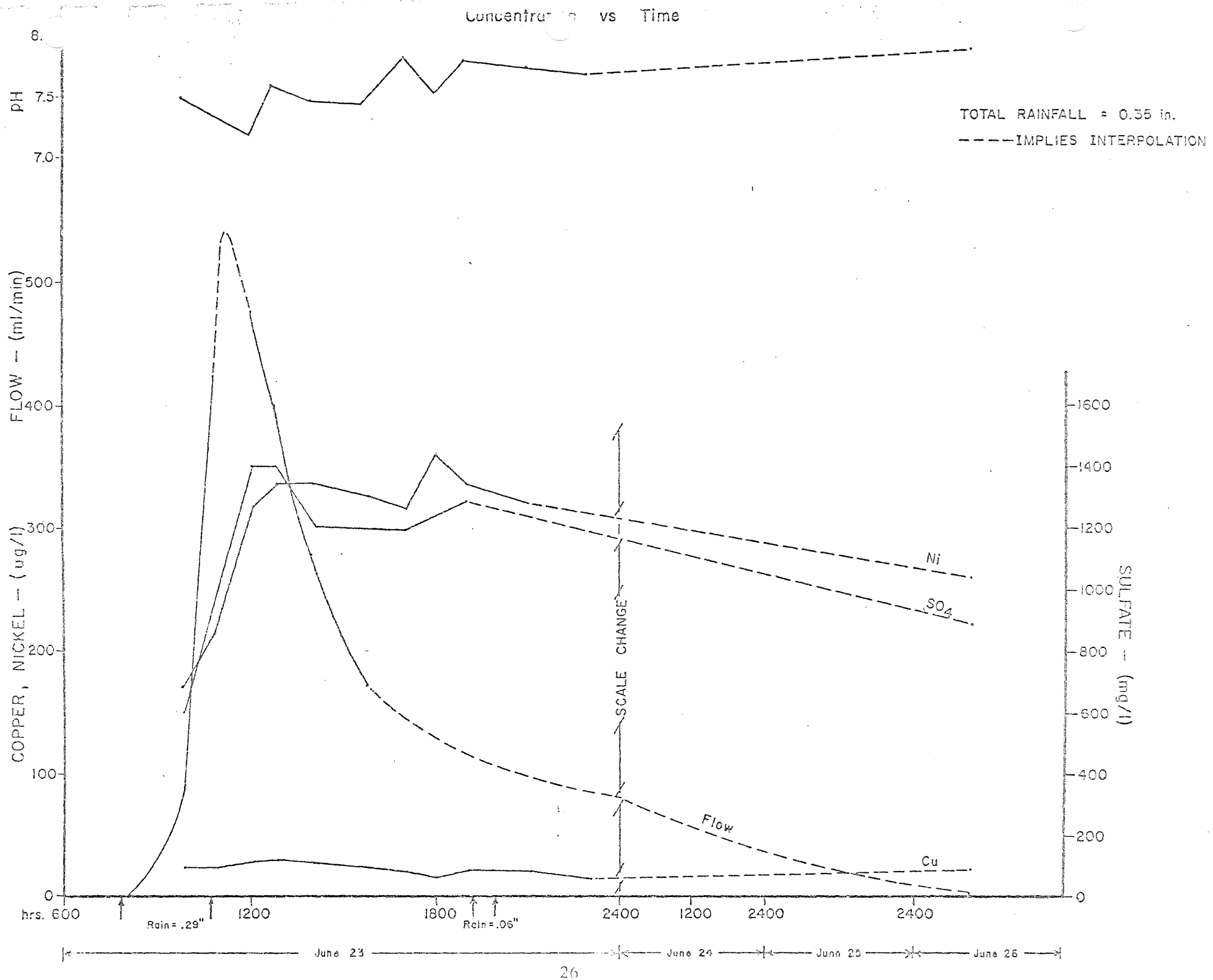
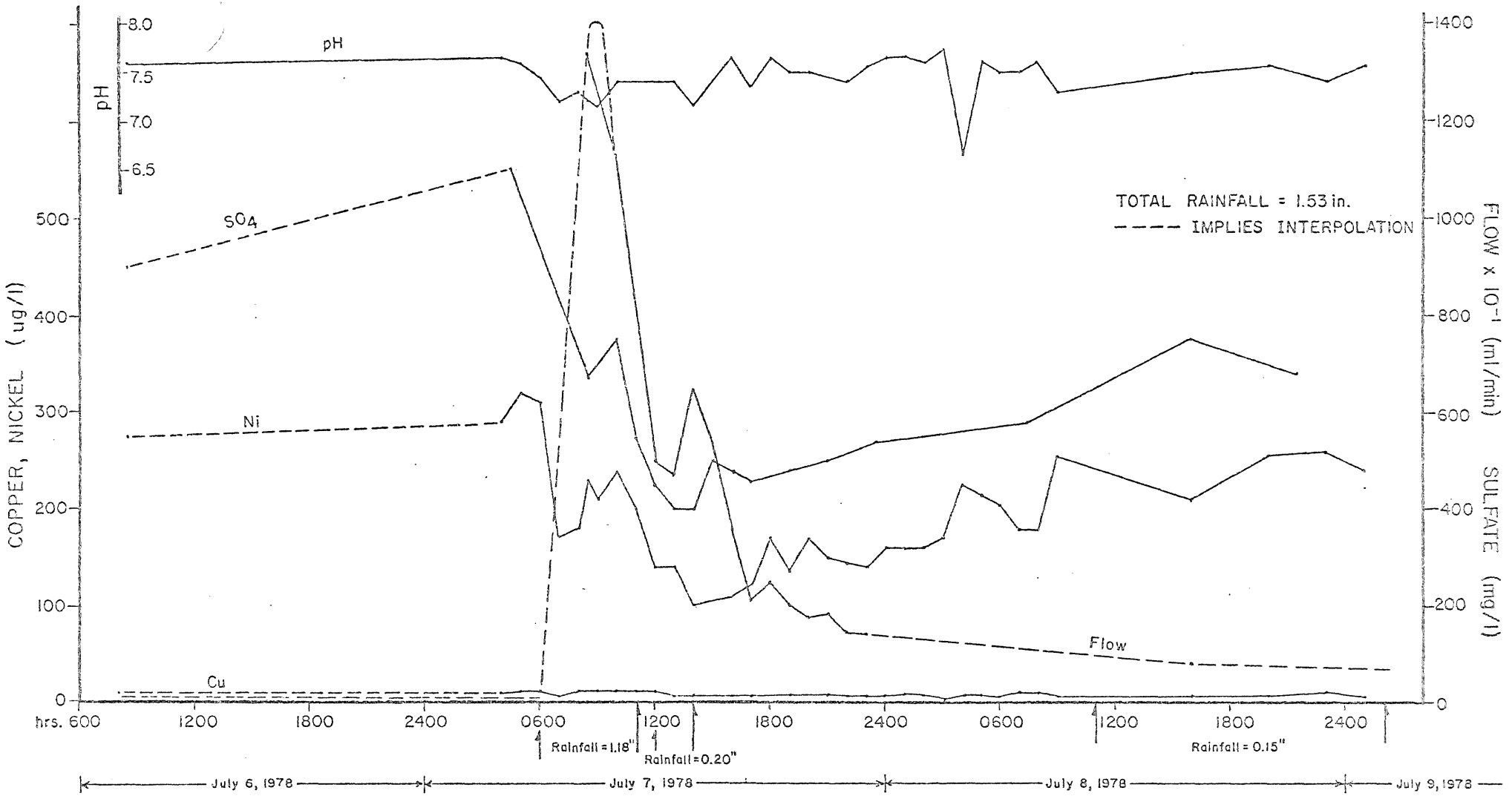

LEACH PILE 4
Concentration vs Time
TOTAL RAINFALL = 0.35 in.
----IMPLIES INTERPOLATION
PH
8.
7.5
7.0
FLOW - (ml/min)
500
400
COPPER, NICKEL - (ug/l)
300
200
100
0
SULFATE - (mg/l)
1600
1400
1200
1000
800
600
400
200
0
SCALE CHANGE
Ni
SO4
Flow
Cu
hrs. 600
Rain = .29"
1200
1800
Rain = .06"
2400
1200
2400
2400
June 23
June 24
June 25
June 26

Figure 6

INTENSIVE SAMPLING OF RAIN EVENT OF JULY 7 and JULY 8, 1978

LEACH PILE 4

Concentration vs Time



Copper concentrations essentially remained constant throughout the event.

DISCUSSION - PHASE I

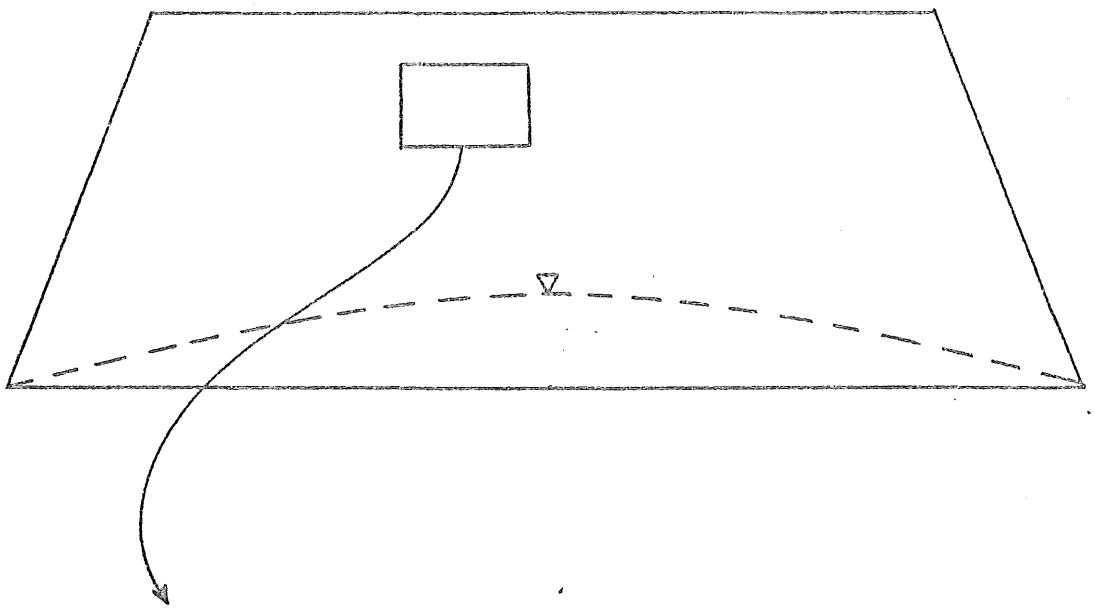
Leaching, as the term is generally used, is the net transfer of material from the solid phase to an environmentally observed aqueous phase. The net transfer is the complex result of a series of chemical and physical reactions that involve mineral dissolution and the transport of the dissolution products out of the stockpile. A schematic of the processes involved in the net transfer of material from the stockpile to an observation point is shown in Figure 7. A detailed discussion of each type of process will not be presented here, but will be provided in later reports as well as in the Regional Copper Nickel level II leaching report, and the technical report on the laboratory leaching experiments conducted at the University of Minnesota. Preliminary discussion can be found in earlier progress reports, (Eisenreich et al, 1977).

The two major considerations in leaching are 1) the reactivity of the specific mineral phase and 2) the mobility of the resulting chemical species; e.g. copper, nickel or sulfate. The reactivity of the mineral phase is a measure of how easily the mineral dissolves. Mobility is a measure of the tendency of an ion to remain in solution. It is the result of many competing reactions, and it is a function of the aqueous environment. Under acid pH conditions, mobility of most trace metals (e.g. copper) will be very high, while at neutral or basic pH, mobility may be low.

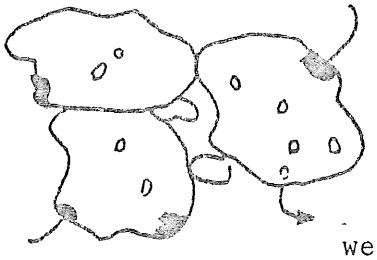
Although sulfate does undergo reactions in solution, under oxidizing conditions it is generally treated as a mobile ion. Therefore, it can be used as an indication of the dissolution of the metal sulfides in the pile. If it is assumed that the only source of the trace metals is the sulfide minerals and that the sulfate release is a measure of the total rate of sulfide mineral dissolution, then the percent of each metal that is transported out of the pile can be calculated. This

Figure 7

Schematic of Stockpile Leaching Process



Step 1: Reaction:



Step 2: Transport: A. reaction products are removed by percolating water

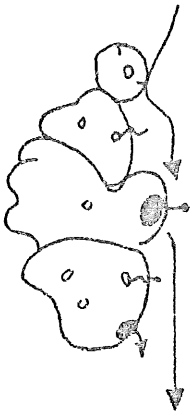
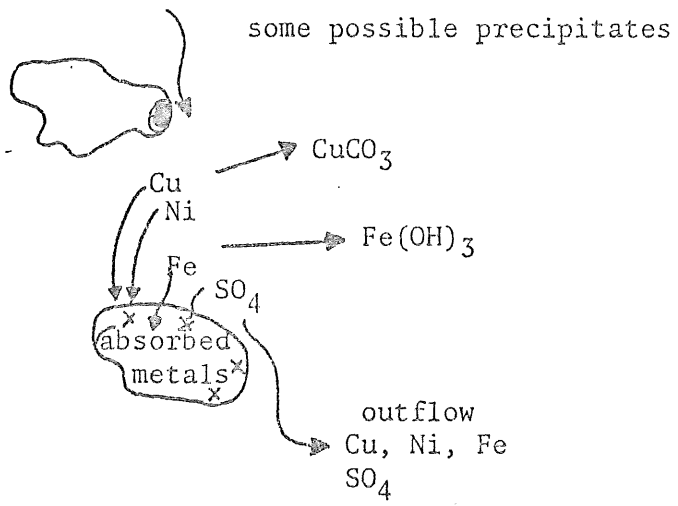


Figure 7 (cont'd.)

B. Some of the metals are readsorbed onto the silicate mineral surface or form precipitates.



(In saturated zone the processes are essentially the same, although water is always available to transport the reaction products.)

The outflow from the stockpile is composed of the weathering products of 1) the sulfide minerals (the prime source of the trace metals; copper, nickel, zinc, cobalt; and sulfate) 2) the silicate minerals (major contributors of calcium, magnesium) and the residual chemicals that were on the rocks when they were stockpiled (primary source of nitrate).

is illustrated in Table XI using the 1977 data for leach pile FL 1. The details of the calculation are given in Appendix IV. The metals exhibit very low mobility. Under present environmental conditions, only a very small percentage of each metal that is released from the mineral phase is transported from the stockpile.

The mobility factor is important in discussing the different behavior of the release of copper and nickel from the test piles. The pH values of the samples collected during the rain events of 6-23 and 7-7 are shown in Figures 5 and 6. The mean pH was 7.56 on 6-23 and 7.44 on 7-7. At this pH, in the absence of complexing ligands, copper is not very mobile. It can form carbonate and hydroxide precipitates and be adsorbed onto the silicate surface of the gabbro or onto iron hydroxide precipitates. The result is that very little copper is transported from the test pile. The constant concentration over the rain event supports the concept that the copper release is controlled by its mobility.

In general, nickel is a more mobile ion than copper, particularly in the neutral pH range. As can be seen from Figures 5 and 6, the nickel concentration is more than an order of magnitude higher than the copper concentration. The nickel concentrations fluctuate over a factor of two during the storm of 7-7. The final concentration in solution is influenced not only by reaction equilibrium but also by the amount and rate of water moving through the pile. Further analysis of the fluctuations in concentration over the events is being conducted.

Figures 8,9,10, show the rate of mass release (mg/min) for copper, nickel and sulfate for the storms of 6-23 and 7-7. The general trend is that mass release follows the hydrograph with the maximum mass release occuring during the peak run-off. Table XII summarizes the total release from the pile for each storm.

Table XI. PERCENT OF OXIDIZED METAL THAT IS RETAINED IN STOCKPILE

	Concentration (mg/l) predicted from chemical composition[1]	Concentration (mg/l) observed (median)	Percent of metal retained in pile
SO_4	610[1] basis	610[1]	0
Ni	22.7	.288	98.7
Cu	95.2	.03	99.97
Zn	5.0	.013	99.7
Co	2.8	.006	99.8
Fe	217	$<$ .05	$>$ 99.98

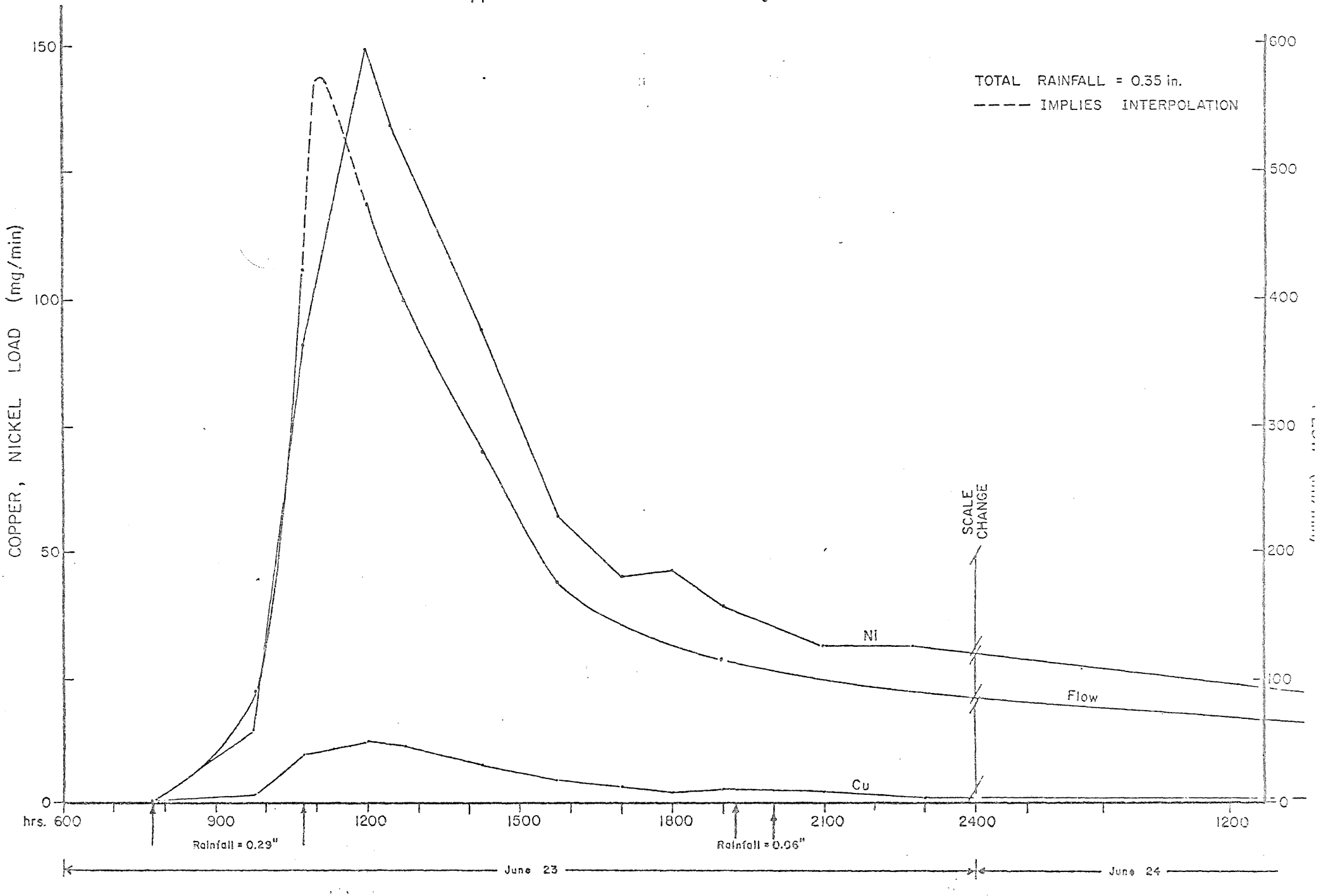
[1]SO_4 is the basis of the calculation, it is assumed that SO_4 indicates total sulfide dissolution and that it is 100% removed from pile

Fig 8

INTENSIVE SAMPLING OF RAIN EVENT OF JUNE 23, 1978

LEACH PILE 4

Copper and Nickel Mass Loading vs Time



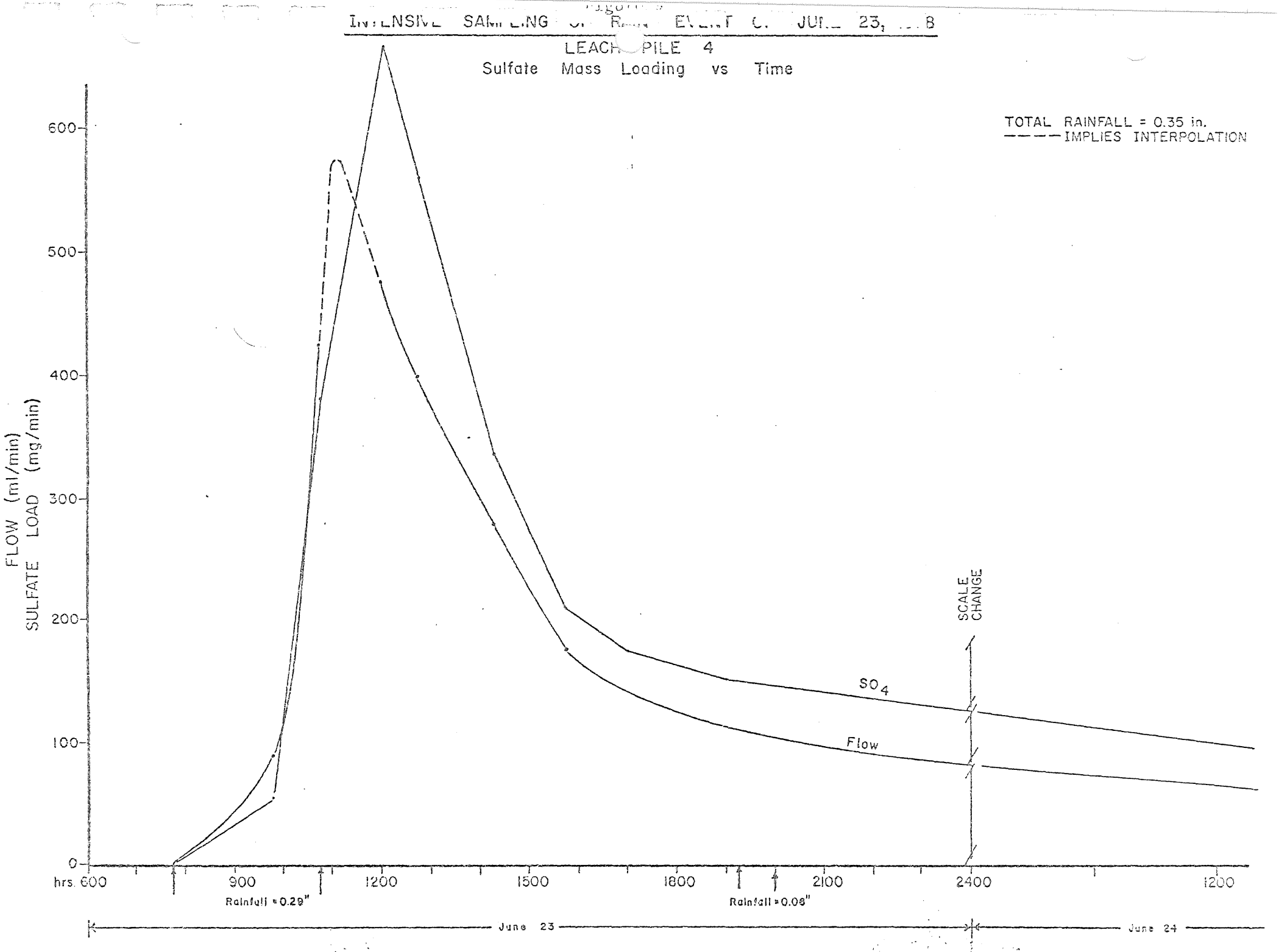

INTENSIVE SAMPLING OF RAIN EVENT OF JUNE 23, 1978
LEACH PILE 4
Sulfate Mass Loading vs Time
TOTAL RAINFALL = 0.35 in.
IMPLIES INTERPOLATION
FLOW (ml/min)
SULFATE LOAD (mg/min)
600
500
400
300
200
100
0
hrs. 600
900
1200
1500
1800
2100
2400
1200
Rainfall = 0.29"
Rainfall = 0.06"
SCALE CHANGE
SO4
Flow
June 23
June 24

Fig. 10

INTENSIVE SAMPLING OF RAIN EVENT OF JULY 7 and JULY 8, 1978

LEACH PILE 4

Mass Loading vs Time

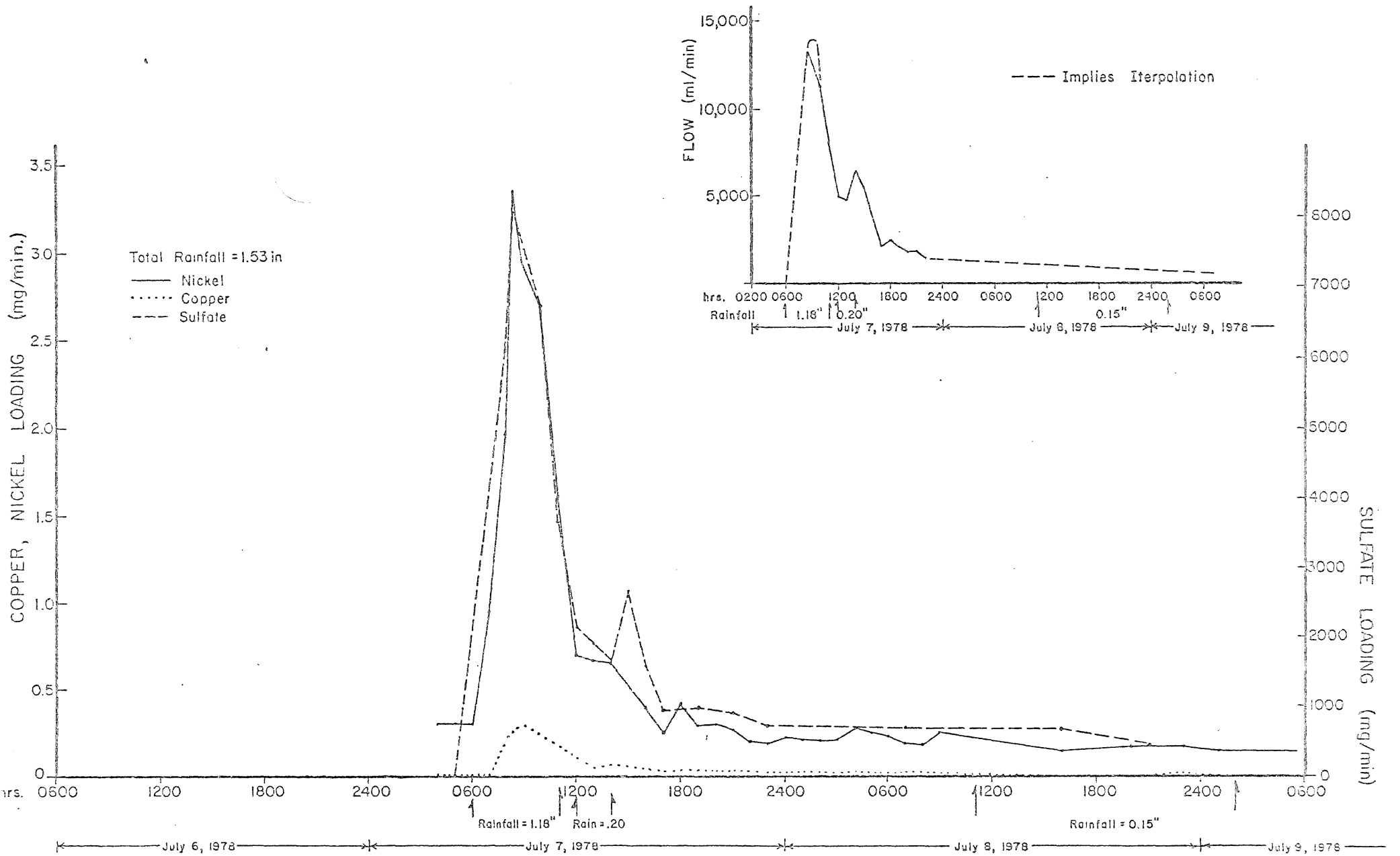


TABLE XII

MASS RELEASE SUMMARY FOR INTENSIVE EVENTS - FL-4

Parameter	6-23-78	7-7-78
Cu released (mg)	6	107
Ni released (mg)	111	1441
SO_4 released (g)	478	3738
Total Volume (l)	341	7599
Time Interval, Δt (min)	4415	4620
Total Rainfall (inches)	.35	1.53

how can rainfall $\frac{1.53}{.35}$ 4.5

& volume $\frac{7599}{341}$ ≈ 22 ?

SUMMARY AND CONCLUSIONS

Copper concentration remains constant during runoff events, while concentrations of sulfate and nickel vary by more than a factor of two. Mass release for all parameters parallel the hydrograph, with the maximum release occurring at peak flow.

The possibilities of large variation in concentrations during storm events makes the collection of a composite sample a necessity. Grab samples will not adequately represent the concentration of all parameters during runoff. For some parameters, the mass calculations made on the basis of grab samples could be in error by more than a factor of two.

RESULTS - PHASE II

It will be recalled that the objective of Phase II is monitoring of the mass release and flow from each pile. The ultimate goal is the use of total mass release to measure the success of different reclamation efforts.

Rainfall

To calculate the total water input to each pile, the total rainfall and the collecting area of each pile must be known. Rainfall data was collected using a standard and a recording rain gauge. Complete rainfall data is given in Appendix V. Table XIII provides a monthly summary of the rainfall. The collecting area of each pile was measured with two sets of measurements taken for each pile and the results averaged. The results are summarized in Table XIV.

Flow Data

The problems encountered with obtaining the flow data have been discussed in Appendix I. Estimates of daily discharge from each pile were made using a combination of flow meter readings and volumetric discharge measurements. (Appendix II.)

The overall comparison between the total of the estimated flow for each pile and the sump flow is given in Table XV. The overall agreement is reasonably good.

The overall runoff relation for each pile is given in Table XVI. Preliminary data suggest that the percent runoff varies as $5 \approx 6 > 4 \approx 3 \times 1 > 2$. Monthly runoff coefficients were also computed for each pile. The results of these calculations indicated that even though the overall agreement between the estimated flows and the sump flow was good, the individual pile data needed to be re-examined. In the early part of the field season, the flow meters would periodically jam and produce backwater conditions in the pile. Some of the volumetric measurements were influenced by this backwater, and as a result the daily flows were overestimated.

TABLE XIII
Rainfall Events, April through October, 1978

Month	Total # of Events	# of Events ≥ 0.3 in.	Total Rain	Rain provided by events ≥ 0.3 inches in.	%
April	7	2	1.28	0.78	60.9
May	8	2	3.37	2.30	68.2
June	9	3	2.21	1.58	71.5
July	14	4	6.38	4.90	76.8
August	10	5	5.53	4.58	82.8
September	9	3	3.28	2.84	86.6
October	7	2	1.05	0.72	68.6
TOTAL	64	21	23.10	17.70	76.62

Note: Rainfall greater than or equal to 0.01 inches is an event.

Table XIV.
Test Pile Area

Pile	Total Collecting Area (ft^2)
1	3173
2	3678
3	3183
4	3524
5	3648
6	3444

Table XV. FLOW SUMMARY

Month	Rainfall	6 Pile Total	Sump Discharge	% Difference
April				
May	3.37	87,000	79,000	+9.4
June	2.49	82,000	81,000	+1.6
July	6.38	156,000	164,000	-4.6
August	5.53	129,000	123,000	+4.4
September	3.28	89,000	81,000	+9.4
October		12,000	12,000	
May - October		555,000	540,000	+2.8

$$+ \% \text{ difference} = \frac{\text{6 pile total} - \text{sump}}{\text{sump}} \times 100$$

Table XVI. OVERALL RUNOFF CALCULATION

Pile	Total Collecting Area (ft^2)	Area of Exposed Hypalon (ft^2)	% of Area That Is Exposed Hypalon	Input liters/ in Rain	TOTALS Input (l)	TOTALS outflow (l)	TOTALS runoff (%)
1	3173	96	3.0	7487	166,137	83,184	50
2	3678	243	6.6	8679	192,587	67,421	35 *
3	3183	36	1.1	7511	166,669	95,265	57 *
4	3524	528	15.0	8315	184,510	92,231	50
5	3648	160	4.4	8608	191,012	120,522	63 *
6	3444	371	10.8	8127	180,338	117,209	65

* Pile has been covered and seeded.

This was a particular problem with FL 3.

All flows will be recomputed. An attempt will be made to correlate flow data between periods of good flow records and then, based on these correlations, make necessary flow estimations. A preliminary examination of the problem time period indicates that the net result may be a reduction in the total flow from FL 3 and possibly FL 2. This would not change the overall relationship in percent runoff that was discussed previously.

Water Quality Data

All the 1978 water quality data are in Appendix III. The sampling procedures are discussed in Appendix I. The data are preliminary. Some of the concentration values appear to be anomalous and will have to be reanalyzed.

Since the major purpose of this report is to serve as a progress report, there is still a large amount of data analysis that remains to be done. The intent of this discussion is illustration of some of the general findings. More detailed analyses, which would include calculation of mass release, will be available in future reports.

Variation In Concentration With Time

To compute the mass release,the total flow for each pile must be computed and representative concentrations must be assigned. These calculations have not been completed. However, general comparisons have been made among the different test piles and between 1977 and 1978.

Figures 11 through 24 present the variation of copper, nickel, sulfate and pH with time for 1977 and 1978. These parameters were selected since they are the key components and products of the leaching reactions. The other parameters will be discussed in future reports. The data presented in these figures are the concentrations in the sample, no attempt has been made to produce a complete set

of flow weighted or "representative" concentrations. The concentrations are a mixture of values; some from composite samples and some from grab samples which were collected both at high and low flow.

From the results of Phase I it is clear that concentrations vary during storm runoff. Therefore, the yearly concentration versus time graphs are useful for identifying long term general trends, but may not give an accurate representation of short time fluctuations. The variation between two samples may be the result of the differing flow conditions at which the samples were collected. Comparison of individual concentrations must be done with caution.

The box plot, or schematic diagram, (Figure 25) is a useful tool for summarizing data. In order to produce a meaningful summary of the concentration data, a uniform data set was selected. The objective was to choose a series of sampling dates on which a sample was collected from every pile. (Due to the start-up nature of the program, not all piles were sampled at the same frequency.) The set of samples that was chosen is listed in Table XVII. The concentrations obtained from this set are not necessarily representative of the entire year of of the flow distribution but they do provide a basis for comparing the different test piles.

Since a smaller number of samples was collected in 1977, all the samples were used to provide a reasonable number of data points. There were not enough data to construct a box plot for piles FL 5 and 6 because they were not constructed until September, 1977. The resulting box plots are shown in Figures 26 through 29.

DISCUSSION

The results indicate that the piles tend to separate into two general groups: piles 1 through 4 and piles 5 and 6. Piles 1-4 were constructed from material from the shaft while the material in 5 and 6 came from the drifts. Piles 5 and 6 contain a higher percentage of sulfur (Table VII) and may have a finer particle size distribution (Amax, personal communication). These factors should tend

enhance the reaction rate in these piles. (As mentioned previously, concentration alone is not a good indication of total release. Concentrations must be converted to mass before the total release can be calculated).

Piles 5 and 6 have higher median concentrations of copper, nickel, and sulfate than the other piles. Preliminary analysis of the flow data (Table XVI) indicates that these two piles also yield more water than the rest. It appears that the total mass release is higher in these piles. One of the major factors that affects the release is the pH. Figure 21 indicates the dramatic decline in pH observed for FL 5 and a corresponding increase in metal concentration. The exact cause of the pH decline is still being studied. Figure 30 shows additional pH data for pile 6. The pH dropped considerably in the last two months before freeze up. From Figure 30 it appears that the decline in pH began around the end of August. If this pH trend continues, acidic leachate may be produced. Nickel concentrations tended to increase throughout 1978. Concentration geban to increase sharply about one month prior to the initiation of the decline in pH. This phenomemon can be seen in FL 5.

The rise in copper concentrations tend to correspond more closely to the decline in pH. Copper is not very mobile at pH values above 7, but its mobility increases as the pH declines. Preliminary examination of the sulfate data indicate no clear trend in concentration. The concentration is variable and there is no dramatic increase in 1978, as there is for the metals.

Since the release of sulfate can be used as a measure of the dissolution rate of the metal sulfides, the preliminary indication is that the overall rate of dissolution has not increased dramatically. The increase in metal concentration results from the increased mobility of the metals at the lower pH. From laboratory studies conducted by Lapakko, Eisenreich, and Hoffman (Eisenreich, et al 1977), an increased rate of sulfide dissolution was observed as pH decreased. It is also well

Figure 1

FL - 1, 1977

CONCENTRATION VS TIME

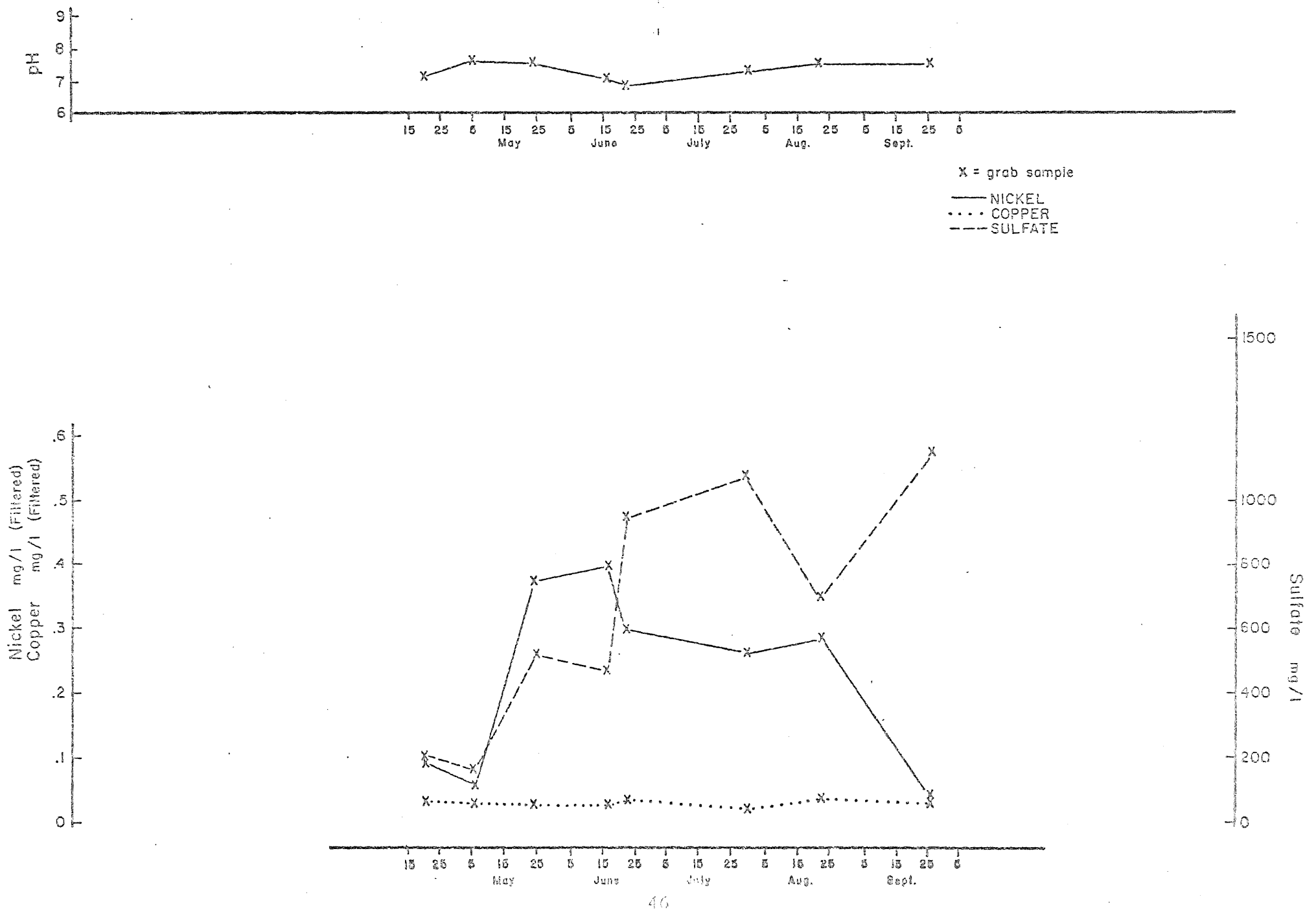
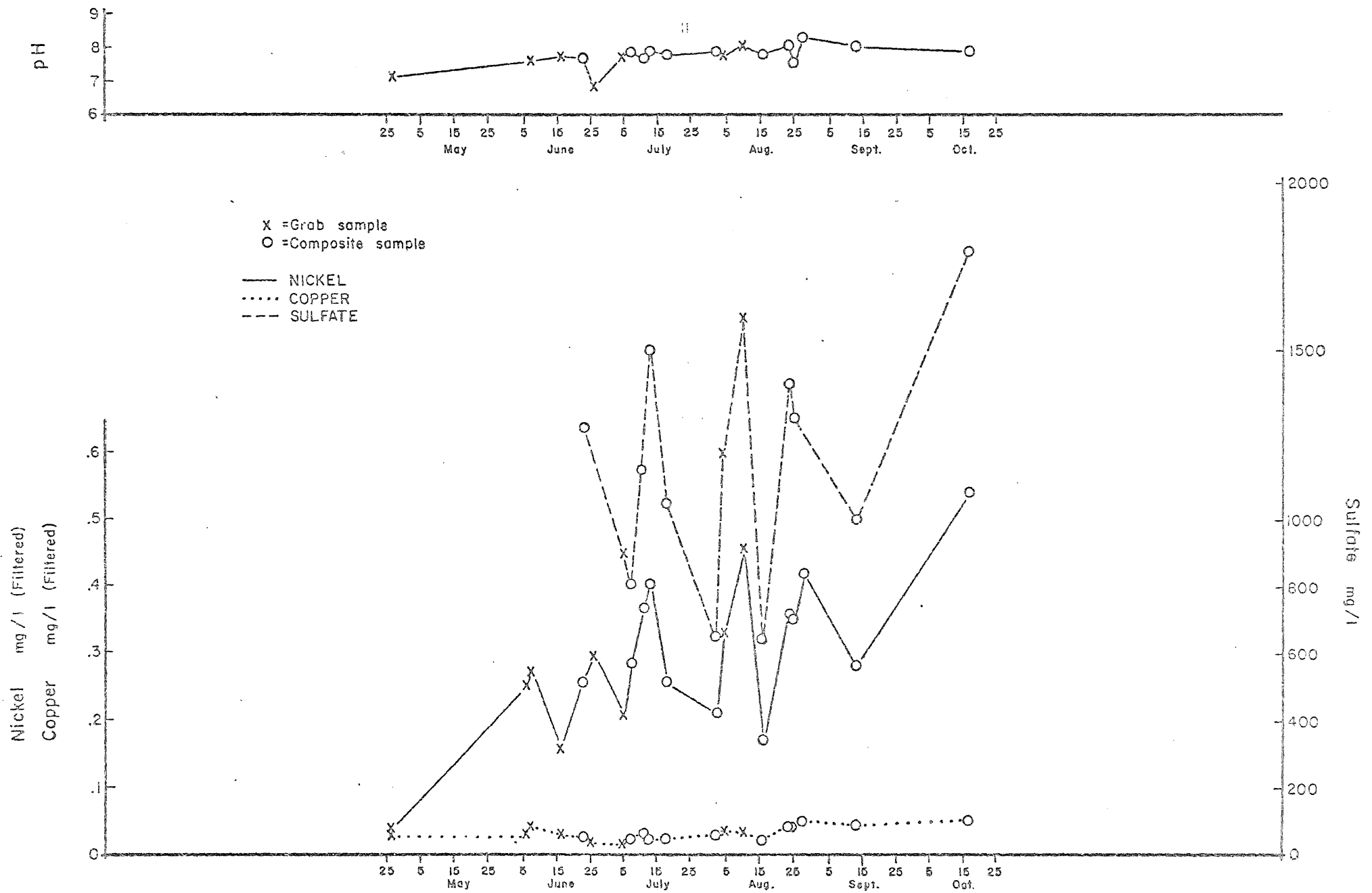


Figure 1.
FL - 1, 1978

CONCENTRATION VS TIME



CONCENTRATION VS TIME

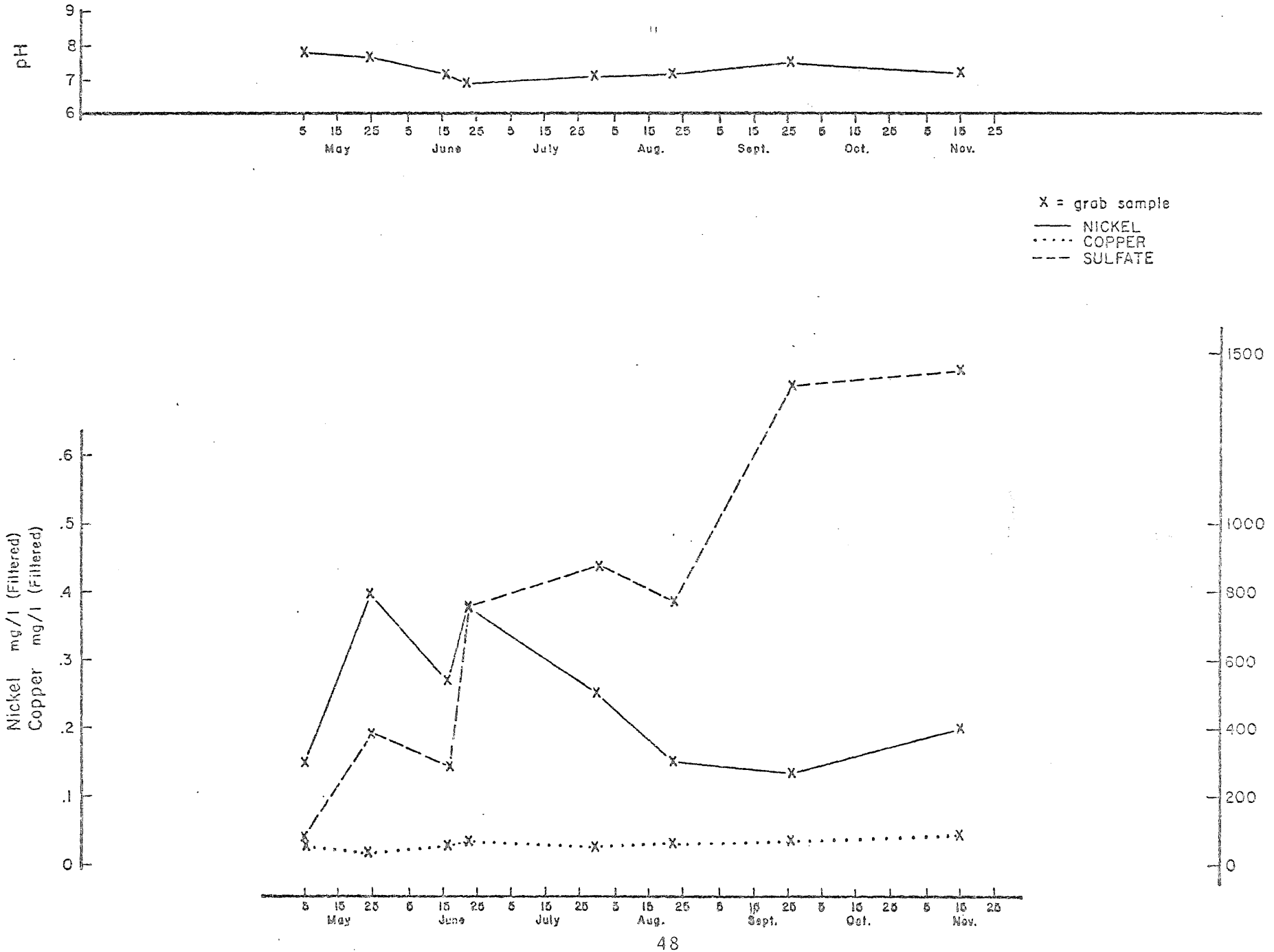
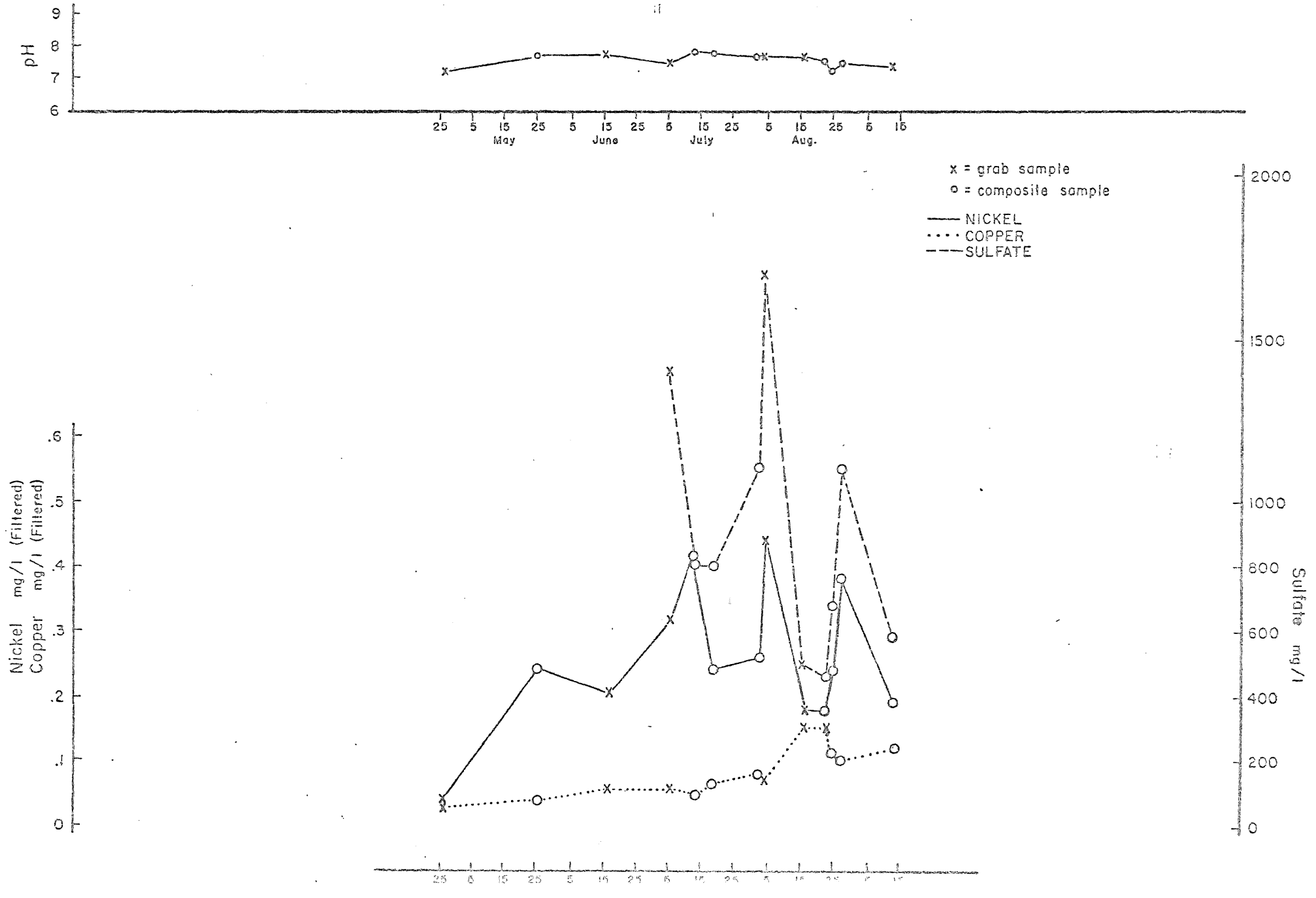


Figure 14

FL-2, 1978

CONCENTRATION VS TIME



Figur 15

FL – 3, 1977

CONCENTRATION VS TIME

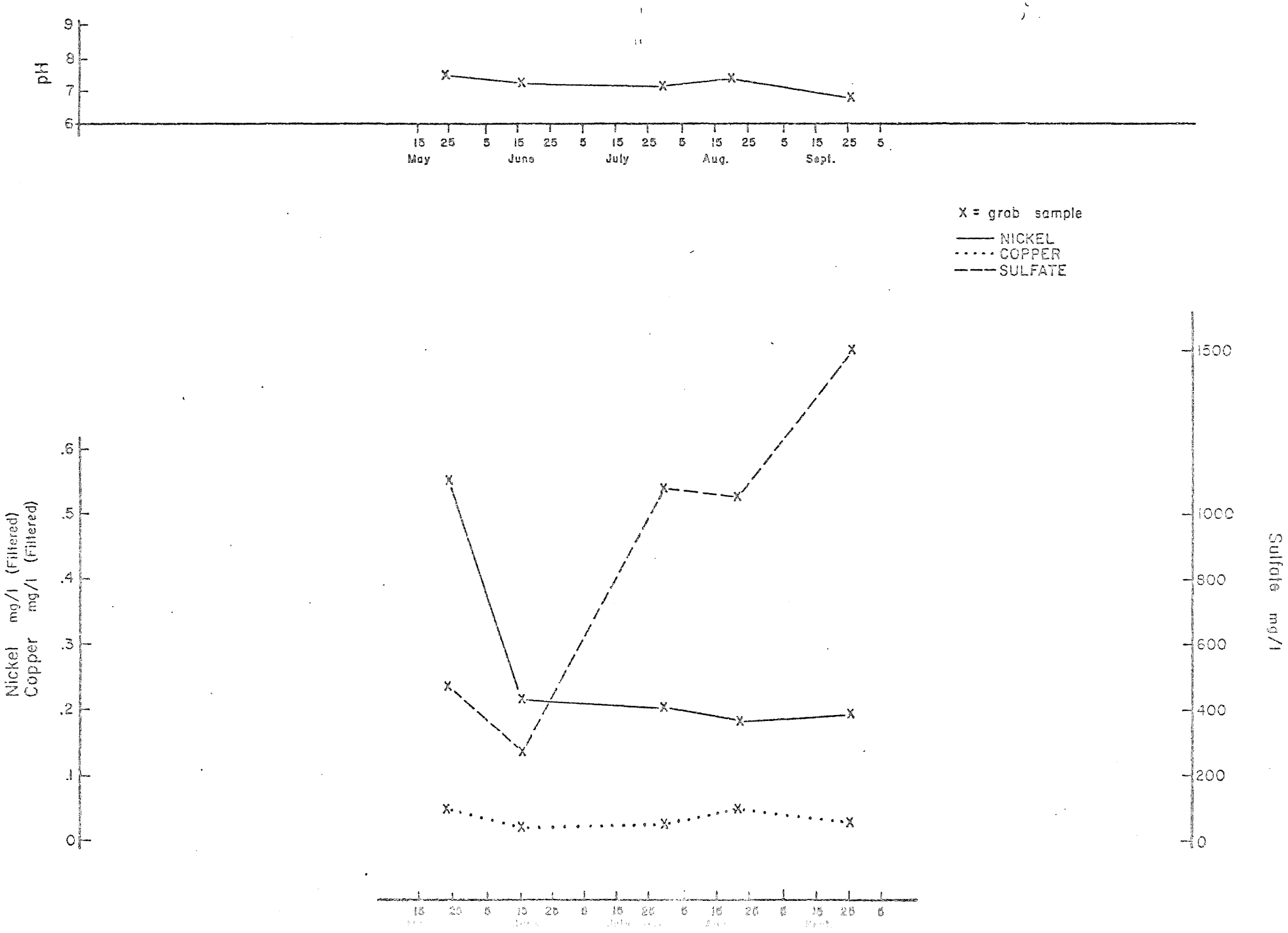
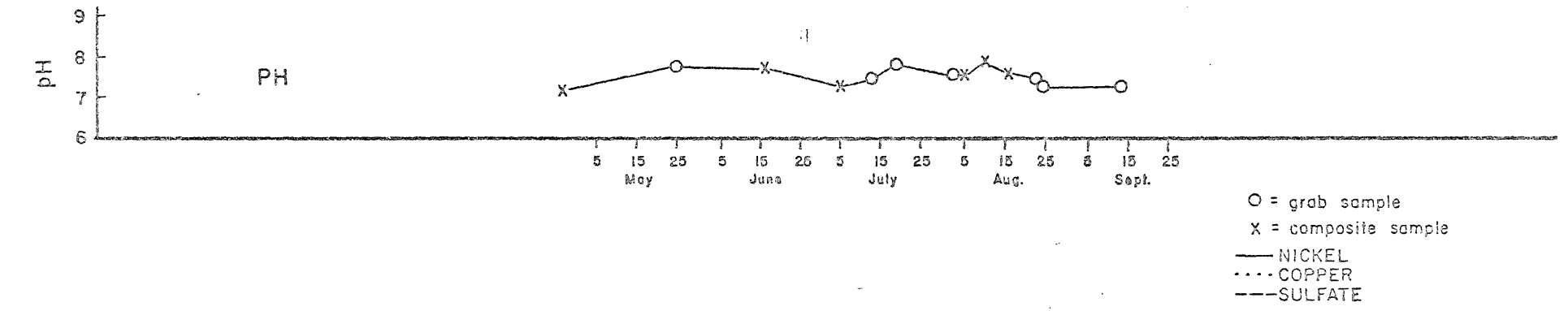


Figure 16

FL-3, 1978

CONCENTRATION VS TIME



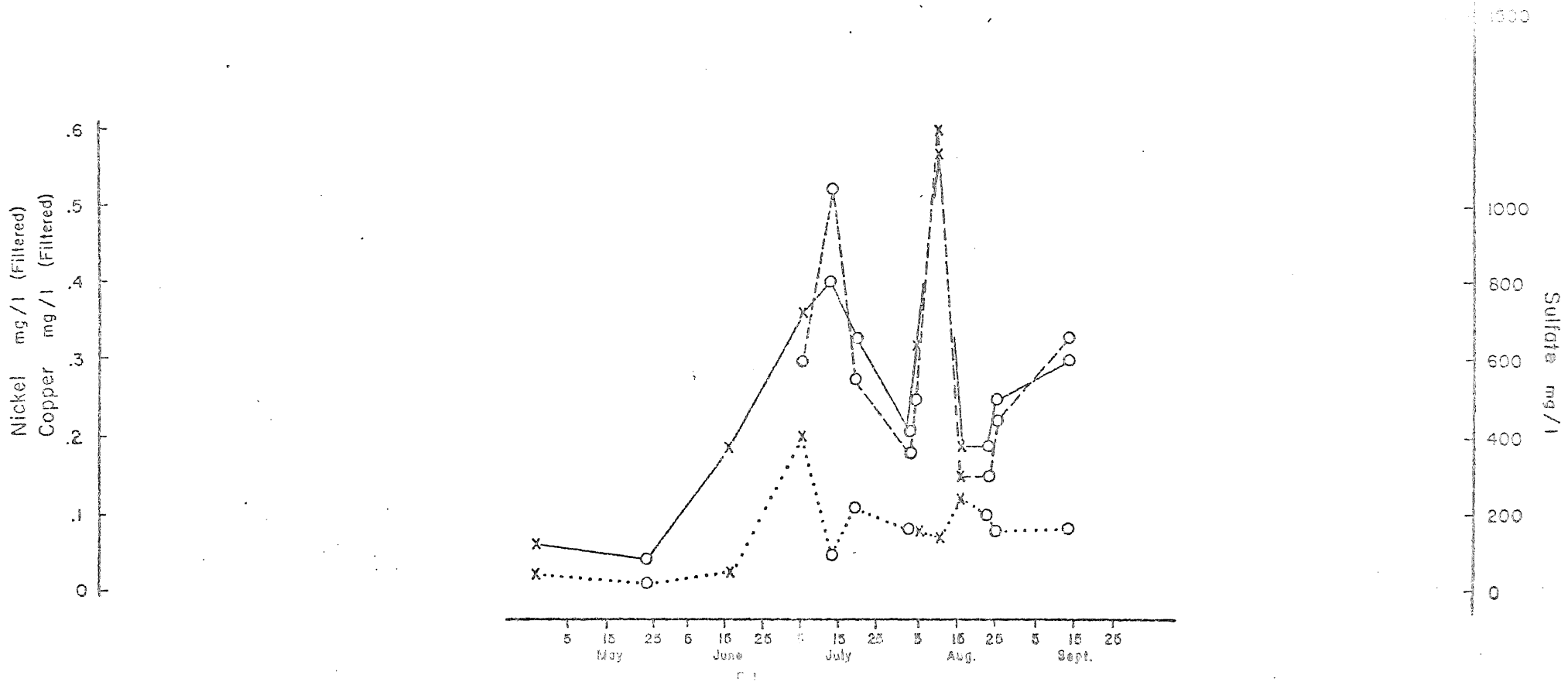
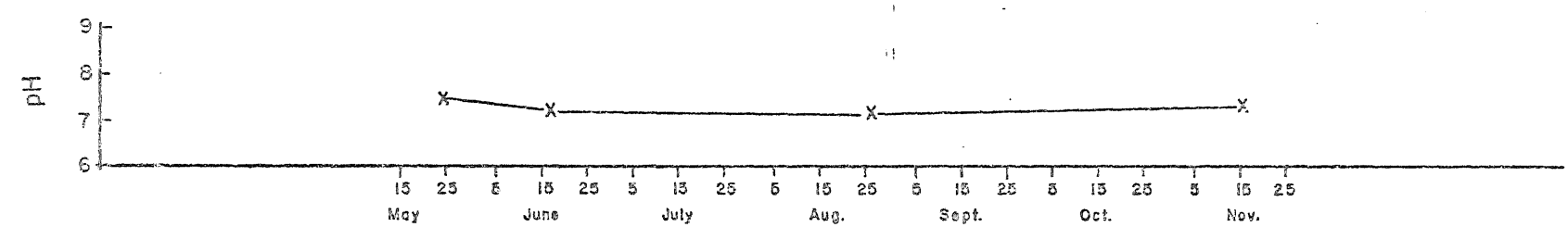


Figure
FL – 4, 1977

CONCENTRATION VS TIME



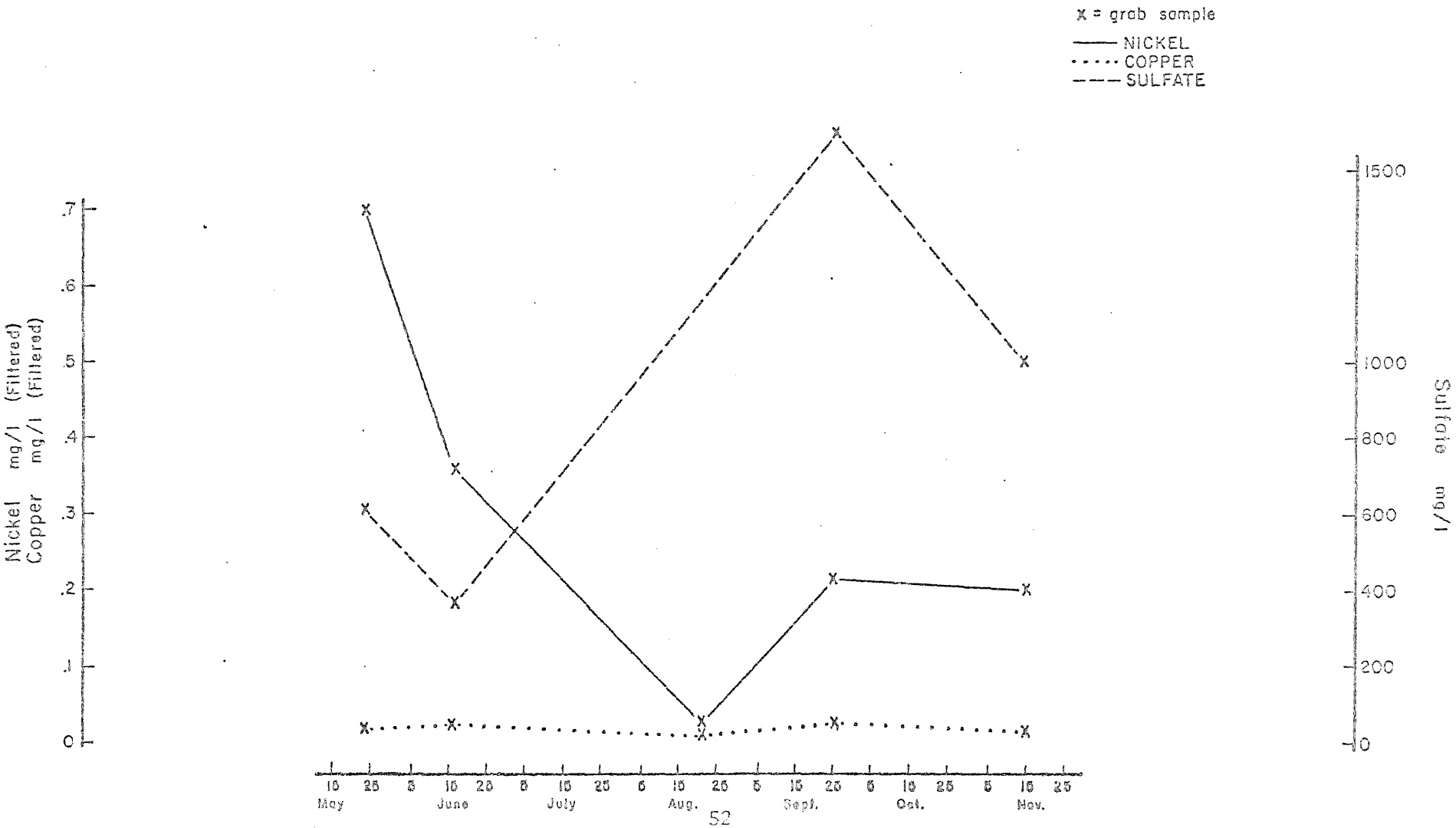
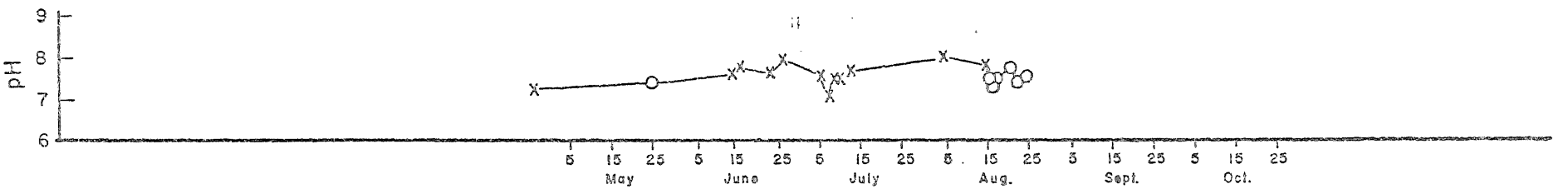


Figure 18

FL-4, 1978

CONCENTRATION VS TIME



X = grab sample

O = composite sample

—— NICKEL

···· COPPER

– – – SULFATE

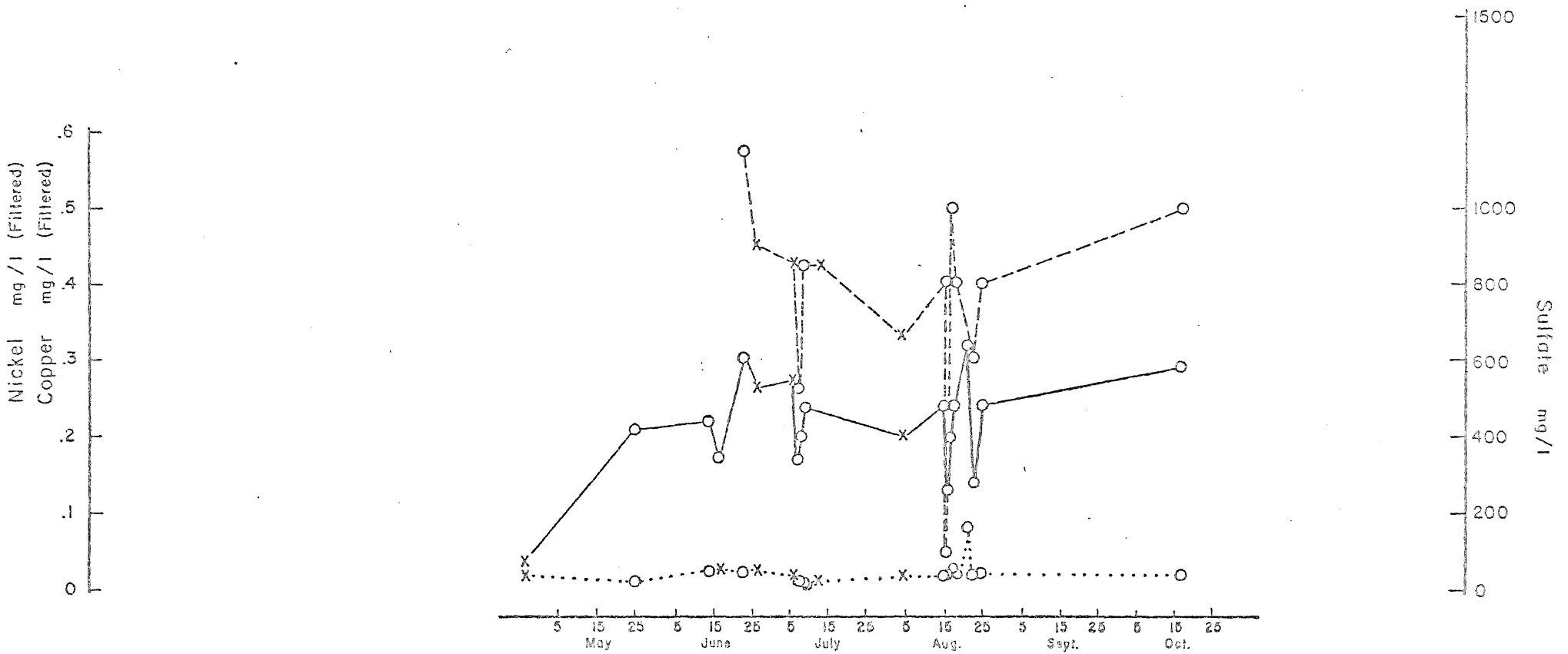


Figure 19

FL - 5, 1977

CONCENTRATION VS TIME

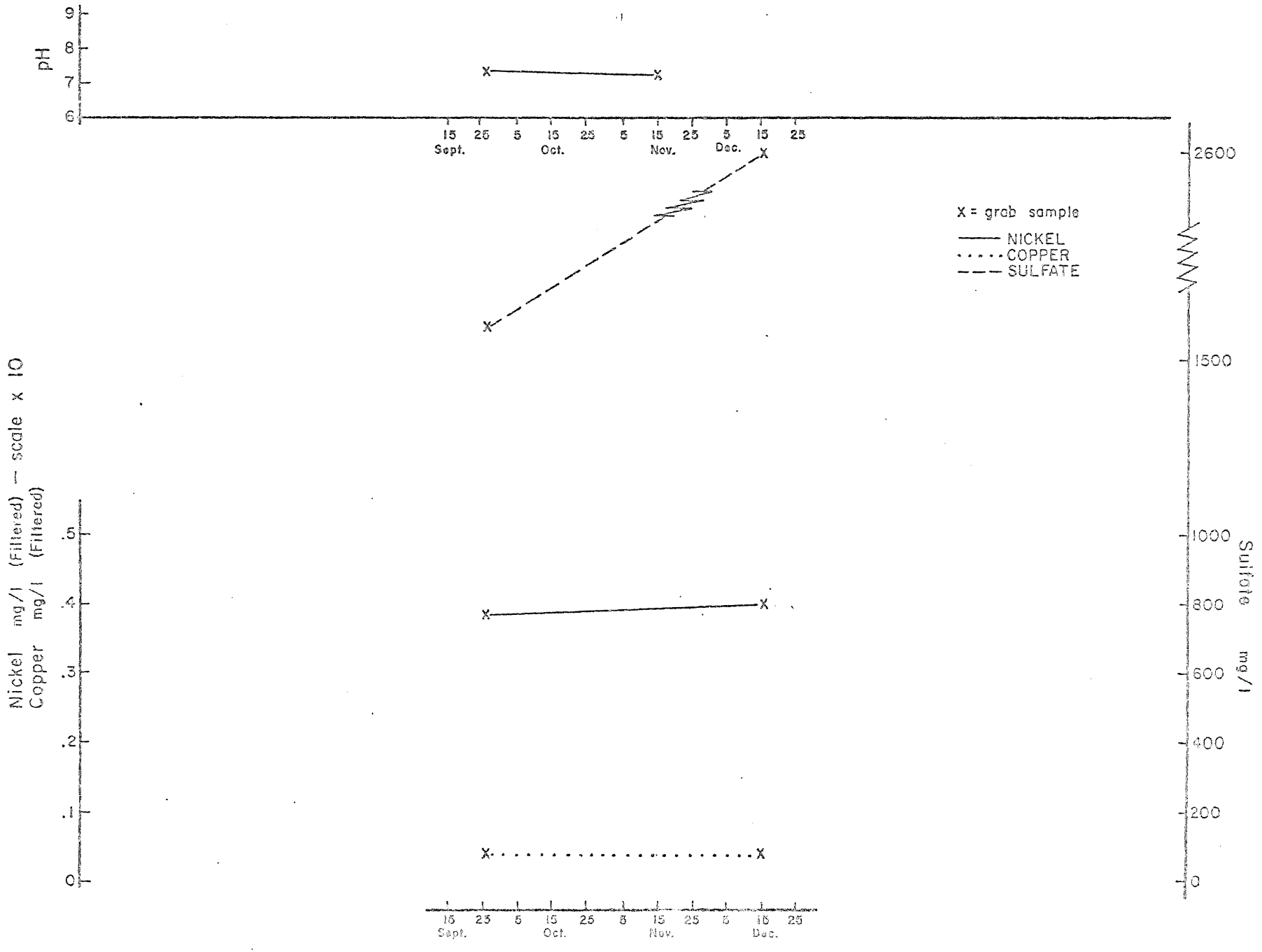


Figure 20

FL - 5, 1978

SULFATE CONCENTRATION VS TIME

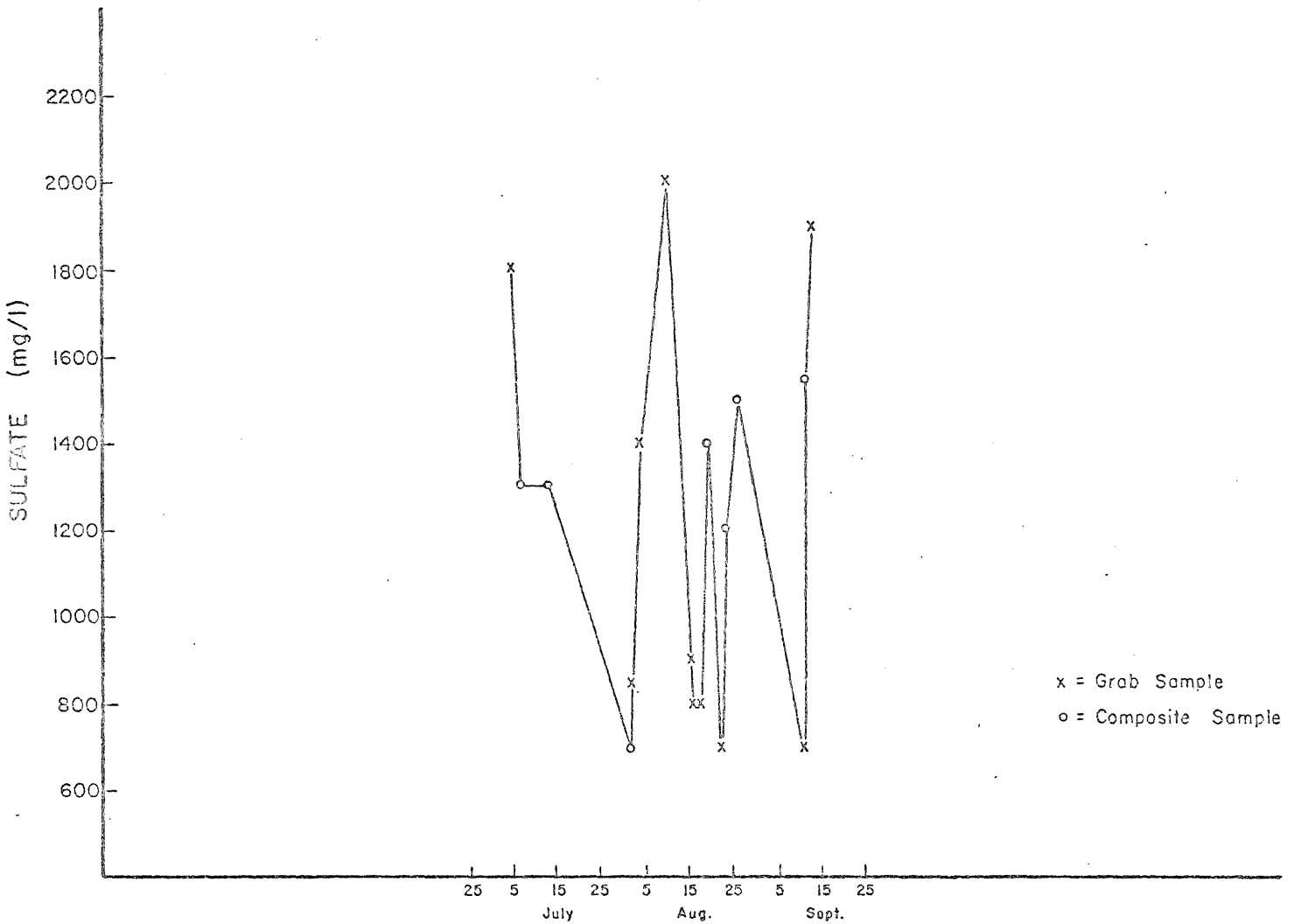


Figure 2

FL- 1978

CONCENTRATION VS TIME, NICKEL, pH

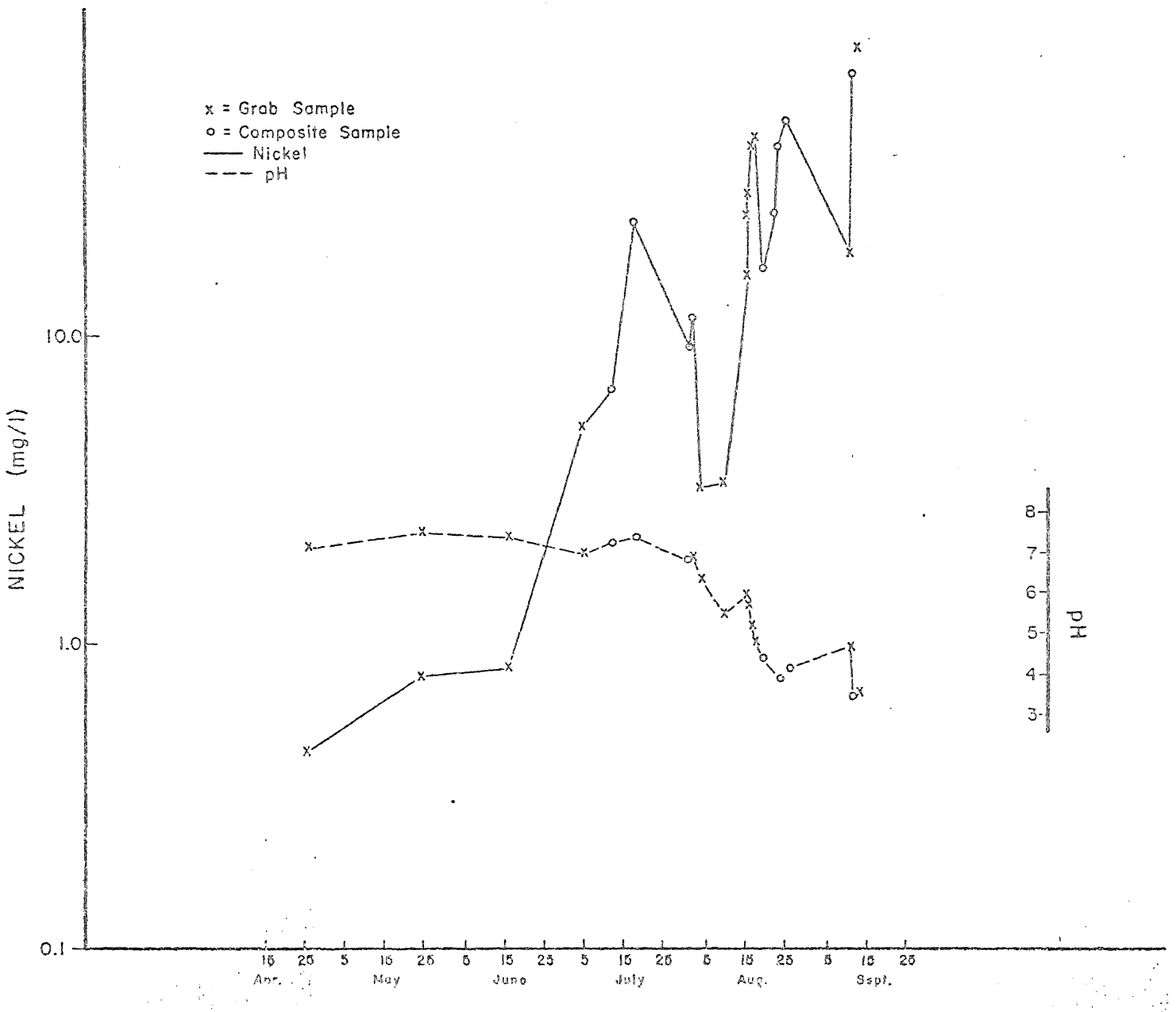


Figure 22

FL 6, 1978

CONCENTRATION VS TIME, COPPER, pH

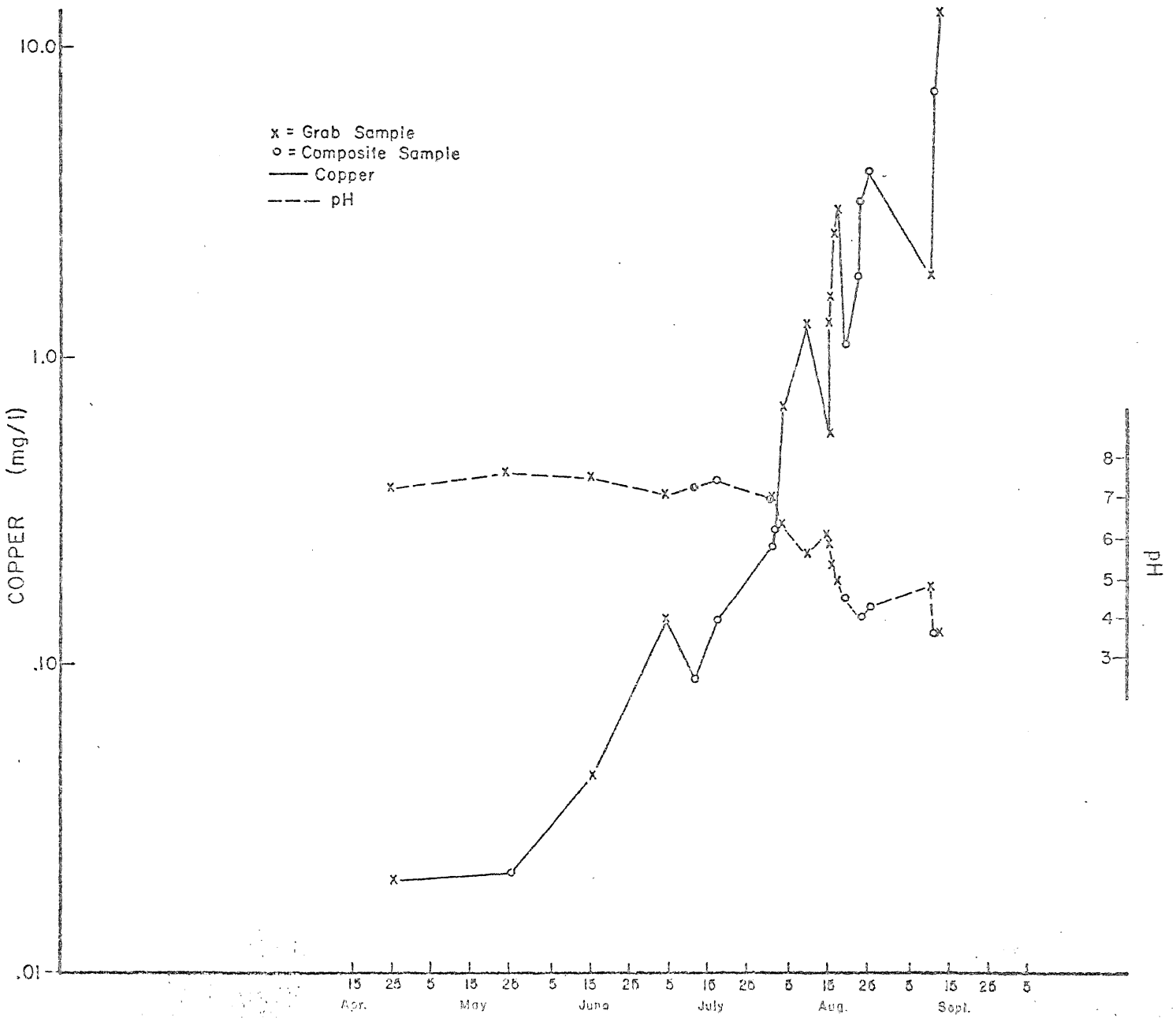


Figure 23

FL-[illegible], 1978

SULFATE CONCENTRATION VS TIME

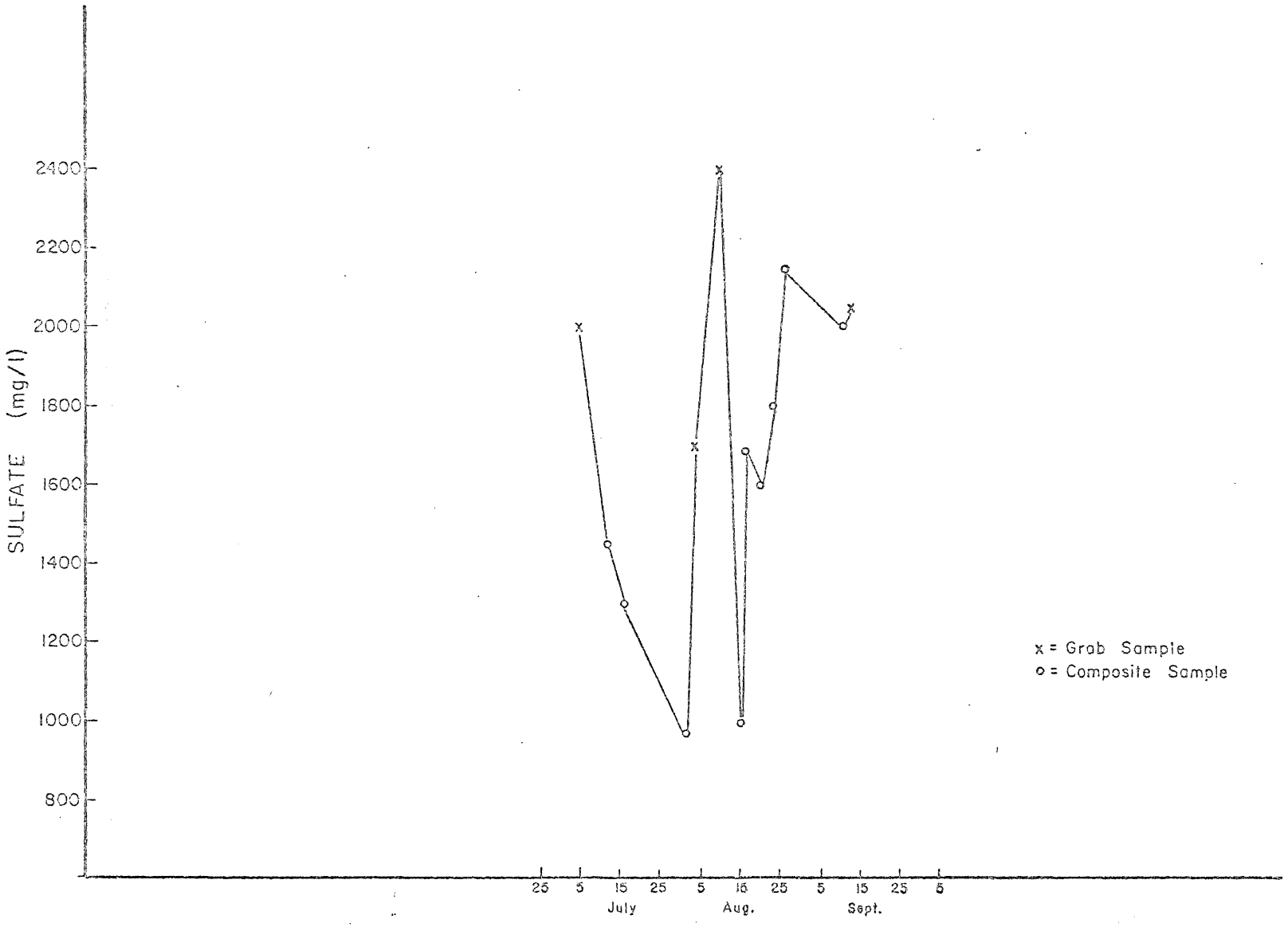
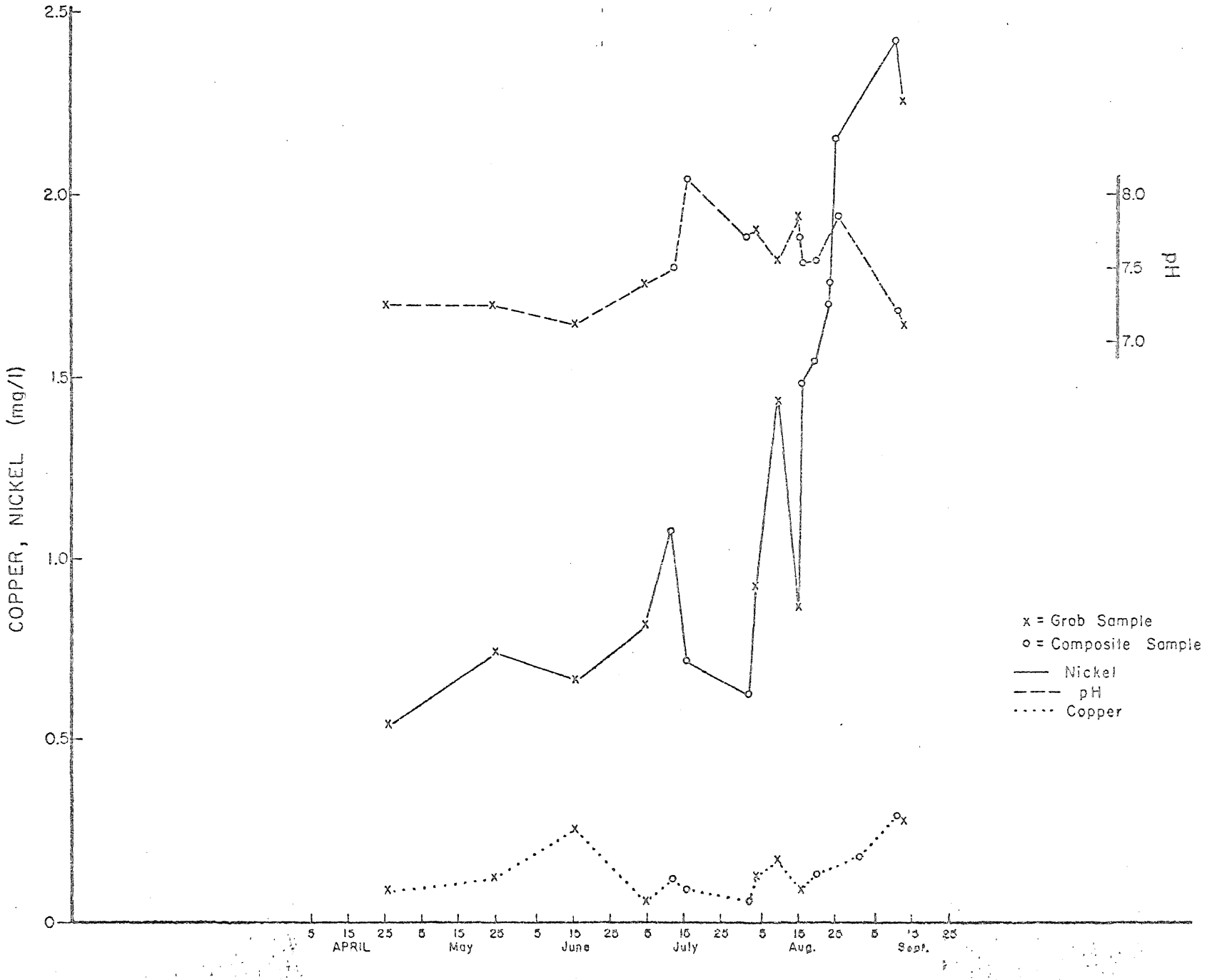


Fig. 24
FL - 1978

CONCENTRATION VS TIME, COPPER, NICKEL, pH



BOX PLOT

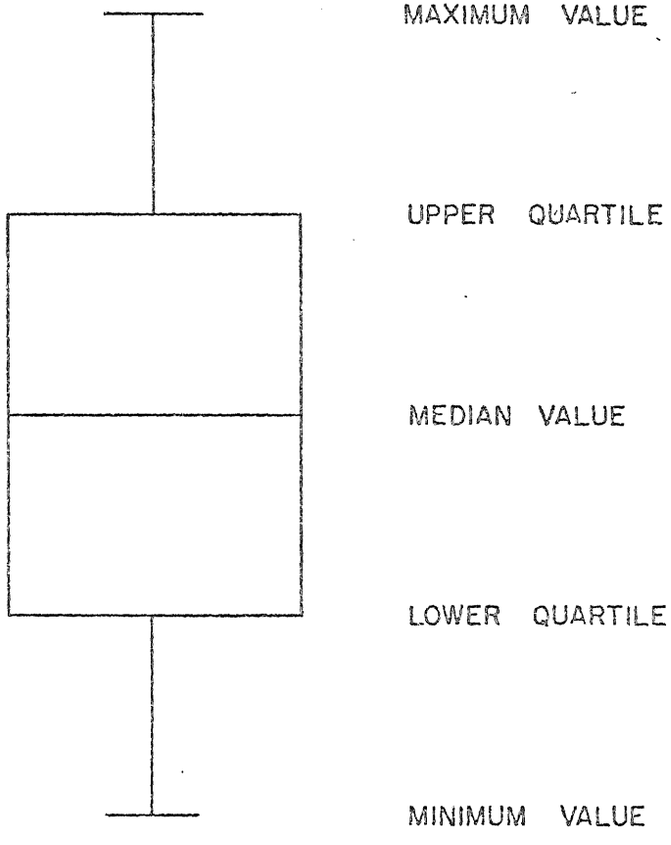

MAXIMUM VALUE
UPPER QUARTILE
MEDIAN VALUE
LOWER QUARTILE
MINIMUM VALUE

Table XVII. SAMPLES USED FOR BOX PLOT

DATE	TYPE OF SAMPLE	CONDITION
4-26	grab	low flow
6-16	grab	low flow
7-5	grab	tail of rain
7-12 to 7-13	composite	comp over rain for all, grab for FL4
7-14 to 7-19	composite	all piles except FL4
8-1 to 8-2	composite	over rain
8-4	grab	low flow
8-16	grab & composite	during & over rain event
8-23	composite	before and over rain
9-13	grab	tail of rain

more detailed information is given in Appendix V

documented that the oxidation of the metal sulfides results in acid production (Hawley, 1972; Eisenreich et al, 1976). Normally, the buffering capacity of the gabbro is sufficient to maintain the pH in the neutral to slightly basic range. If the oxidation rate increases or the buffering capacity decreases, the pH may drop. As pH decreases, the rate of metal transport from the stockpile increases. The drop in pH in FL 5, resulted in a 10 to 100 fold increase in metal transport.

With the exception of copper, the box plots indicate that piles 1-4 generally have comparable median concentrations. Median copper concentrations for piles 2 and 3 appear to have increased between 1977 and 1978, and the 1978 values are higher than those for piles 1 and 4. In 1977, the median copper concentration was comparable for all four piles. Median concentrations for piles 1 and 4 are essentially the same for 1977 and 1978. The fact that the median nickel concentrations are similiar for all piles, both in 1977 and 1978, suggests that the overall release rate has not changed. A selective increase in copper release has occured, due to increased copper mobility. Mobility could be increased by the introduction of organic compounds which have been washed out of the soil cover. Additional analysis is needed to determine the significance and the cause of this increase.

Median pH values for piles 1-4 increased slightly form 1977 to 1978. Comparison of the median sulfate data did not reveal any consistent pattern between the two years. Median sulfate concentrations decreased in FL 3 and 4, increased in FL 1 and essentially remained the same for FL 2.

Figure 26

BOX PLOT, SULFATE

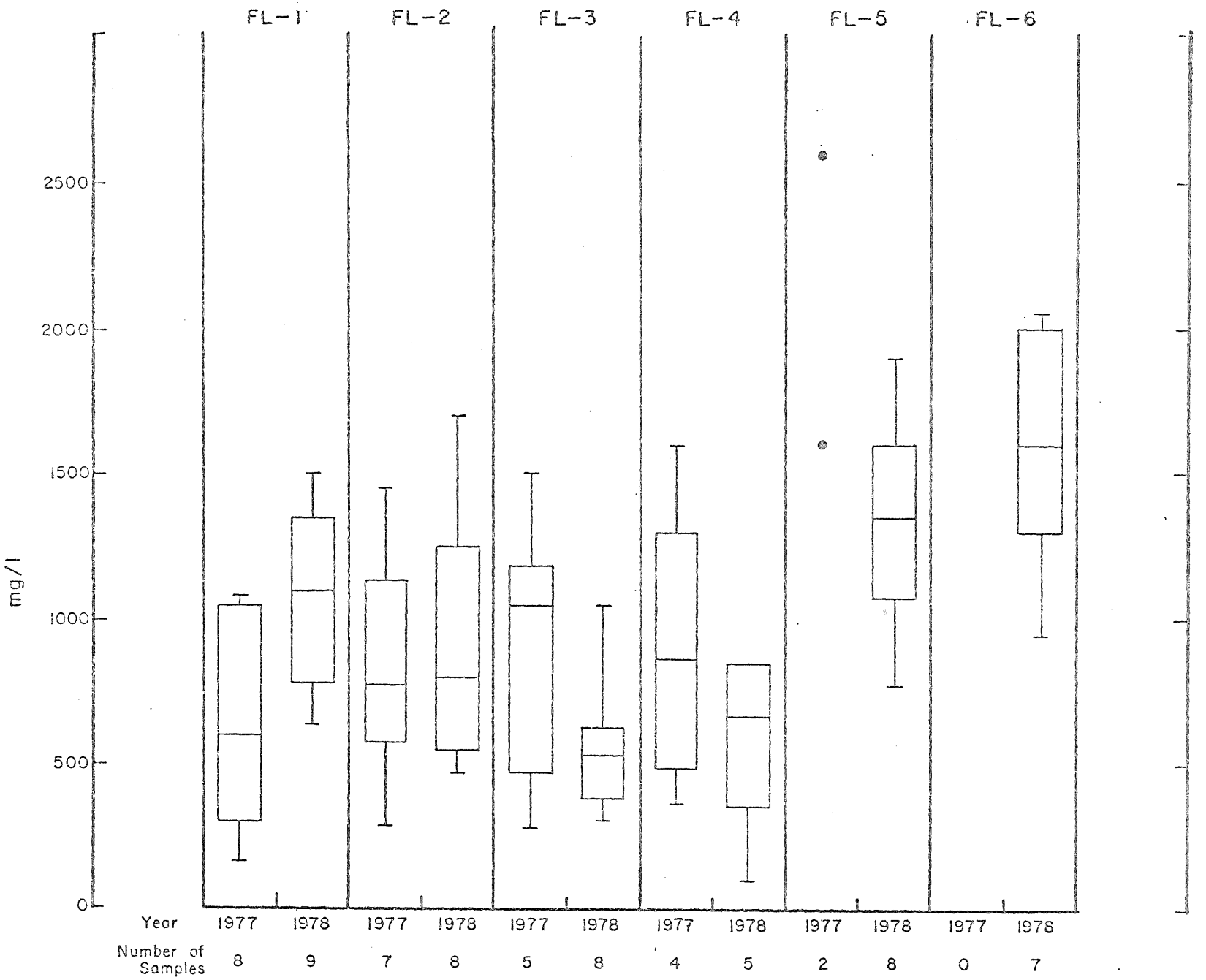


Figure 27

BOX PLOT, COPPER

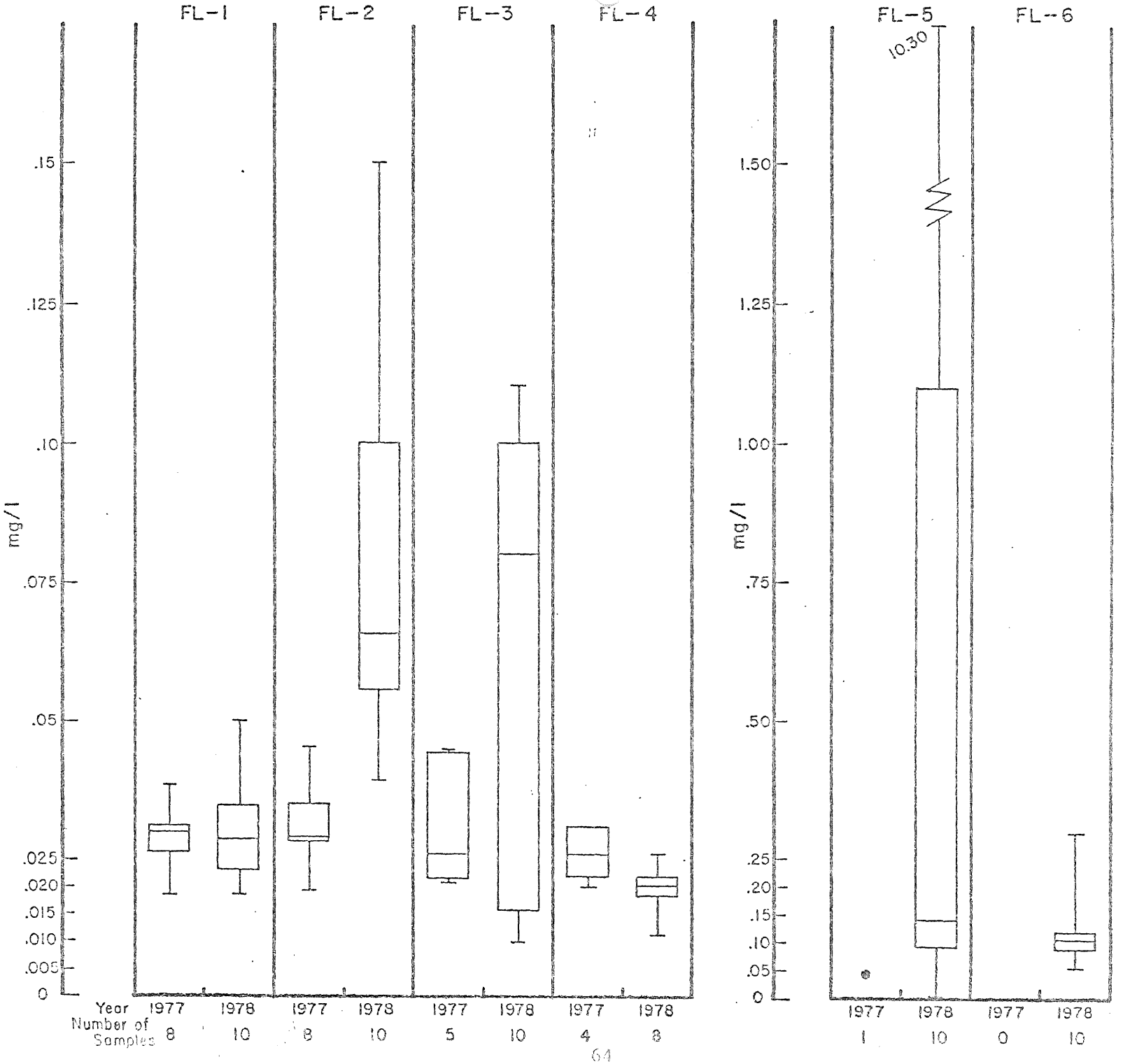


Figure 28.

BOX PLOT, [illegible]CKLE

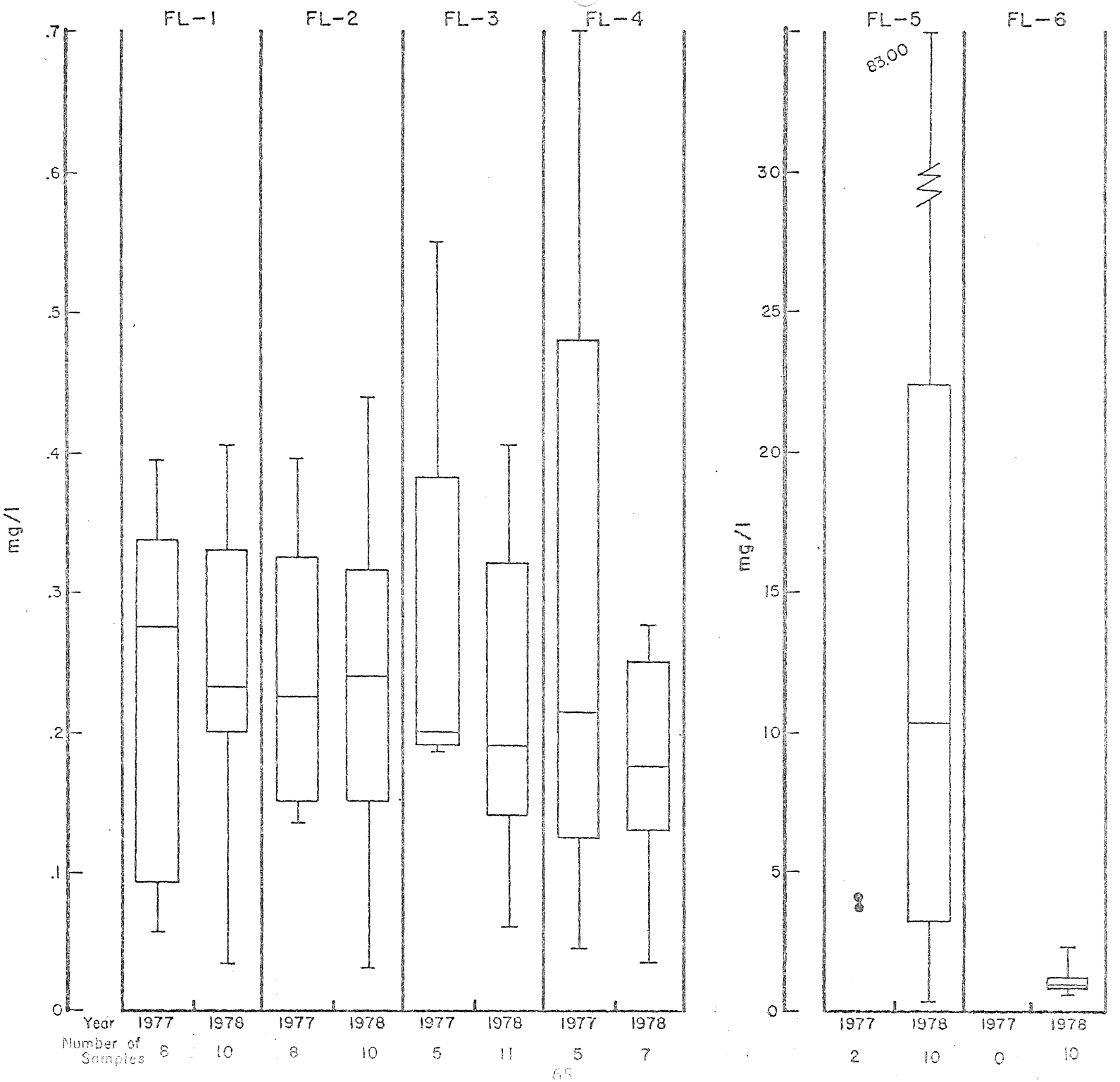


Figure 29

BOX PLOT, pH

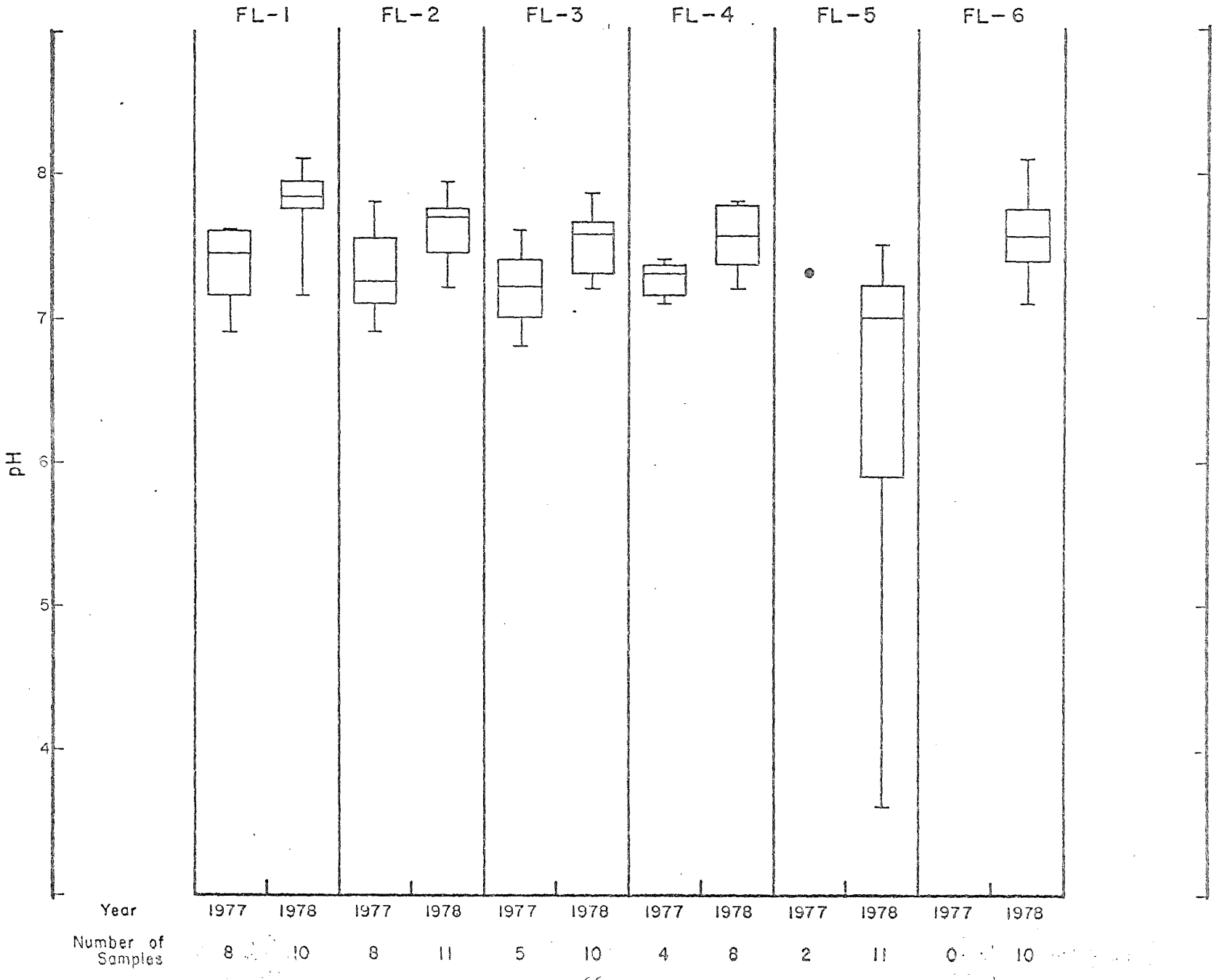
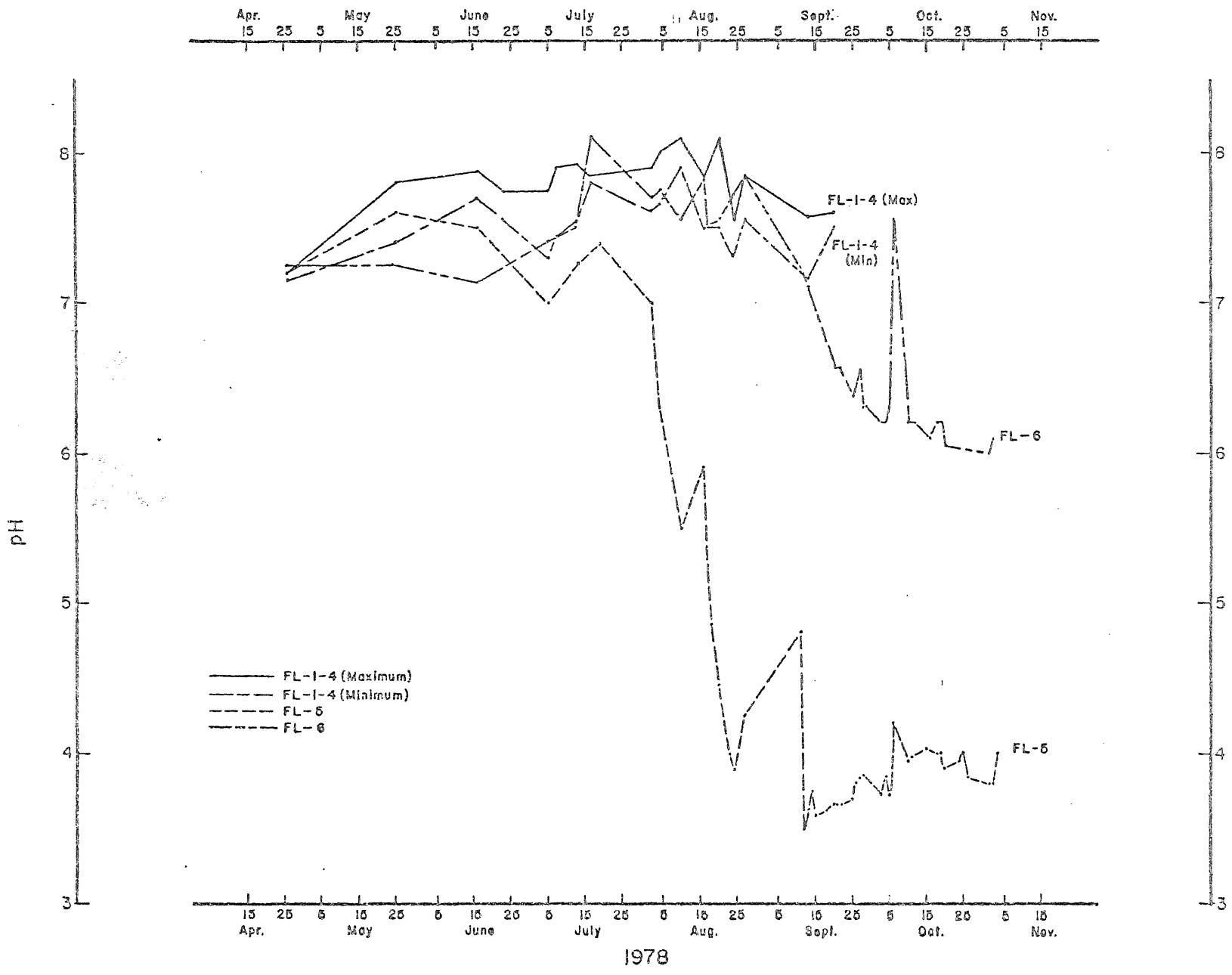


Figure 30

pH VS TIME



SUMMARY AND CONCLUSIONS

Preliminary analysis of the data indicates that for piles 1 through 4 median nickel concentrations in 1978 are comparable to those measured in 1977. There seems to be no significant differences in median nickel concentration among the first four piles.

Piles 2 and 3 exhibit increased copper concentration which appear to be the result of increased copper mobility. Organic compounds which are capable of complexing copper may have been washed out of the soil top cover, allowing more copper to remain in solution.

Piles 5 and 6 tend to behave differently than the first four. Pile 5 produced acidic leachate and the pH in pile 6 started to decrease before freeze up. When the pH declined, substantial amounts of metal were released in the leachate. The exact cause of the acidification is not known, but is being studied.

The overall runoff coefficients indicate that pile 2 had the least amount of runoff. Pile 2 was covered with topsoil and a good vegetative cover was established. Vegetative cover was sparse on the till covered piles and preliminary analysis suggests that the runoff from these piles was comparable to untreated piles. All conclusions are preliminary and analysis is continuing.

FUTURE ANALYSIS

An outline for future analysis is given below. Any recommendations or comments on the outline or suggestions for additional analyses would be appreciated.

I. Calculate mass release from each pile

A. flow calculations

1. flow records must be reanalyzed and daily flow estimates revised

2. flow records from periods of good record will be used to develop regression relations between the piles

   a. these relationships will be used to examine periods of partial and estimated records.

   b. readjustments in daily flows will be made

3. plot daily hydrographs

4. compute total outflow from each pile

B. assign "representative" concentrations

1. ideally, for each major event, there would be a flow weighted composite sample, then the mass released during that event would be

mass (mg) released	=	total (liters) flow	X	flow weighted concentration	(mg/l)

2. due to the start up nature of the project, a flow weighted sample does not exist for each storm

   a. concentration variation with time, flow, and rainfall will be studied

   b. "representative" concentrations will be chosen and applied to different portions of the hydrograph

II. Factors affecting mass release

A. top dressing

1. is there a difference between the covered and uncovered piles

2. compare the mass released by each pile for each event

B. time between events

1. plot mass release vs time between events

2. is there a relationship - does mass release increase as time between events increases

C. rainfall

1. plot mass release versus

a. rainfall (in)

b. rainfall intensity (in/hr)

2. is there a relationship - does mass release increase as rainfall and/or intensity increase

D. PH

1. plot mass release vs pH for piles 5 and 6

2. compare SO_4 release with metals release - is there evidence for a large increase in sulfide dissolution

E. age of piles

1. is there a relationship between mass release and the total amount of rainfall that has fallen on the pile.

III. Other considerations

A. Acid forming potential

1. delineate the possible cause for the acidic conditions in FL5

2. check literature references and devise experimental tests to try and isolate the cause of the acidity.

3. compare the behavior of 5 and 6, is there a trend.

B. Peak runoff vs base flow

1. How much of the flow is "peak runoff" and how much "base flow"

2. On a yearly basis is most of the mass transported during peak flow or during base flow

C. Compute release rates for each pile (kg /ton yr)

1. how does the release rate vary between piles

a. is there a correlation with the % S in the rock

2. how do the release rates compare to other studies conducted on gabbro material

3. use major cations to estimate rates of host rock weathering - is this consistent with what is already known

IV. Redesign sampling plan

A. re-evaluate parameter list

1. are there some parameters which are not providing useful information

B. re-evaluate - sampling plan

1. should more or less samples be collected?

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Appendix I Field Methods

Methods for sampling the field leach piles have undergone constant evaluation and change. Changes were made to produce a more representative sample. Documentation of these changes is necessary for proper interpretation of sample results.

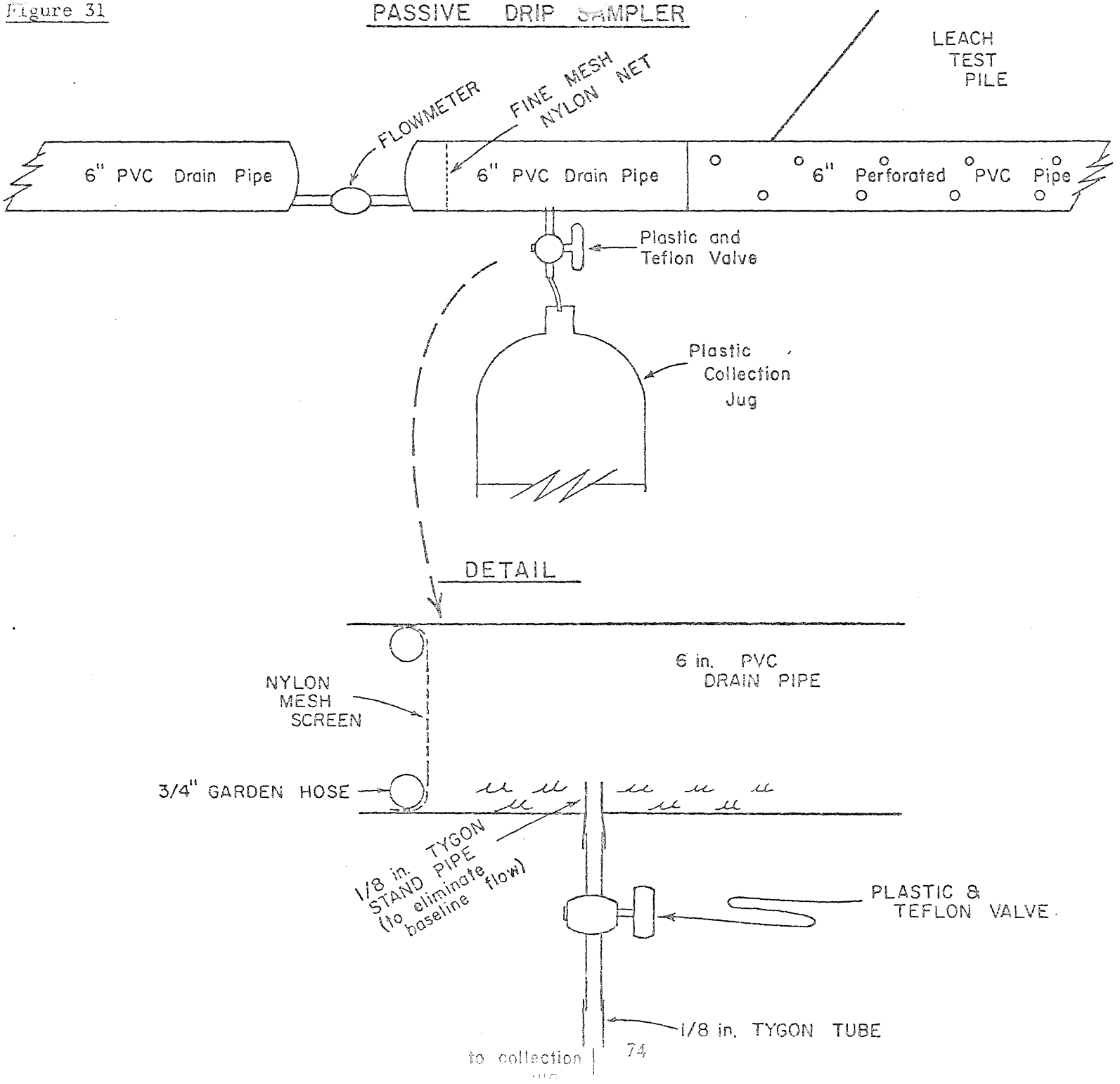
From April 26 to May 24 no volumetric discharges were obtained from individual leach piles. Instead, the discharge from the common sump (Figure 2) of the 6 leach piles was used to obtain average flow from individual piles. This method was not satisfactory because flow, and thus mass loadings, from individual piles could not be accurately calculated.

It was decided, therefore, to install low pressure, low volume flow meters in the discharge line of each leach-pile (Figure 3). The installation was completed on May 24. It soon became apparent that the flow meters were malfunctioning due to a combination of low water pressure and fine particulates. To correct this, two changes were made. A fine mesh nylon screen was installed upstream of the meter to filter particulates (Figure 31), and the position of the flow meter was lowered to give as much pressure drop as possible. These modifications were accomplished by July 14. Subsequently the meters functioned properly at high flows but continued to stop during low flows.

Release of metals is known to fluctuate with rain events, therefore, composite samples were needed to accurately estimate an average release for each rain event. To composite the flows a "passive drip sampler" (Figure 31) was developed and installed July 11. The passive drip sampler failed to operate as anticipated. Initially, the drip flow was expected to be proportional to the water level in the pipe; larger sample volumes being collected at high flow than at low flow. In practice, this did not work. The drip flow was difficult to regulate and showed very little response to the water level in the pipe. Since the drip rate did not

Figure 31

PASSIVE DRIP SAMPLER



change significantly with the flow, the composite sample tended to oversample base flow. After the filter screens (for the flow meters) were installed a standpipe was added so that only flow above base flow would be sampled. Thereafter the drip flow did not always begin with rising flow and had to be readjusted manually. Additional problems occured during backwater conditions. As the water backed up in the pipe, the drip flow continued and the sample did not represent the storm event. On several occasions, the backwater condition began during the night and by morning the five gallon collecting container had overflowed.

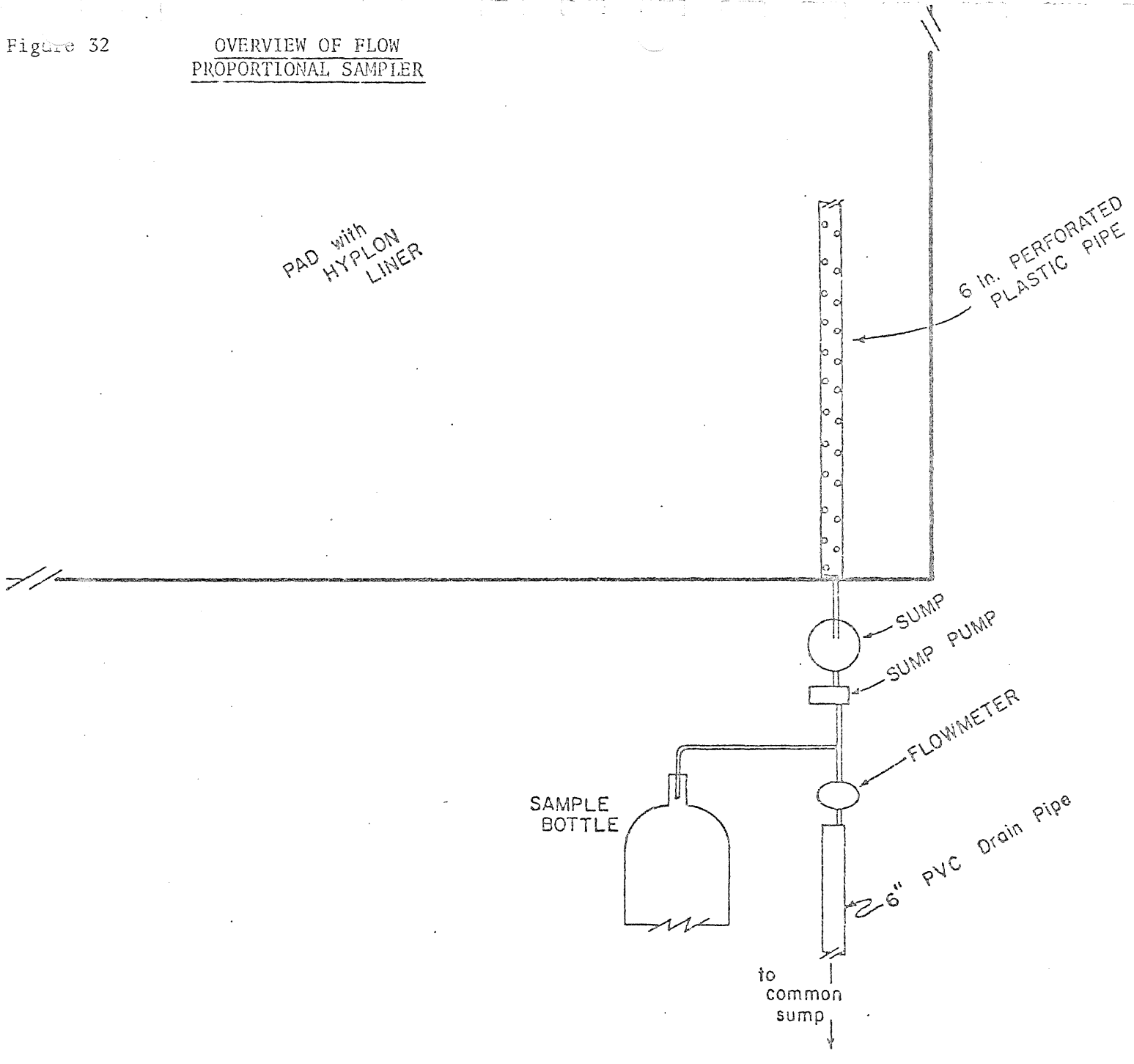
An attempt was made to solve both problems by developing a sump pump and a compositing system (Figures 3 and 32). The flow from the leach pile empties into a five gallon sump. The sump pump is controlled by a float switch in the sump. Each time the pump is activated the event is recorded on a Rustrak recorder. A flow meter installed on the pump outlet records total discharge. A small sample line has been installed in the pump discharge line. Each time the pump is activated a small aliquot is placed in the sampling container. This produces a flow weighted composite sample; more sample is collected at high flow since the pump is activated more frequently. This sampling and recording system was tested first on leach pile number 4 (installed August 11). The system operated well and it was decided to similiarly instrument the remaining piles. By November 3 instrumentation was completed on leaching piles 1,2 and 5. Instrumentation of piles 3 and 6 is nearing completion.

LABORATORIES

Analytical services for the study are provided by Serco Laboratories, Roseville, Minnesota; DNR Minerals laboratory, Hibbing, Minnesota; DNR field laboratory at Minnanax; and the Minnesota Department of Health laboratory in Minneapolis. The following is a list of the parameters analyzed by each laboratory.

Minnesota Dept. of Health	Serco Laboratories	DNR Hibbing	DNR Field Laboratory
Dissolved Organic Carbon	Chloride	Calcium	pH
	Nitrate	Magnesium	Specific Conductance
	Ammonia	Sodium	Temperature
	Total Phosphorus	Potasium	Alkalinity
	Sulfate	Copper	Dissolved Oxygen
		Nickel	
		Iron	
		Manganese	
		Cobalt	
		Zinc	

Figure 32 OVERVIEW OF FLOW PROPORTIONAL SAMPLER



Appendix II

FLOW CALCULATIONS

Flow meters were installed on each pile but did not always function satisfactorily. They were routinely read at 0800 and 1400-1600 every day during the week. The condition of the meter was noted (running, stopped, running but sticking). No readings were made on weekends and holidays.

After fine mesh screens were installed upstream of the meters, (July 14), the meters usually worked satisfactorily at high flows. Prior to July 14, the meters would sometimes jam during rain events and the flow would have to be estimated using volumetric flow measurements or through comparison with other piles. The meters rarely worked under low flow conditions. Base flows were generally computed using volumetric measurements. Normally the flow was measured between 1400-1600 and the recorded flow was assumed to be a good measure of the average daily flow rate.

Examples of the methods used to estimate daily flows are given in Table XVII. The general approach that was used was:

1 - meter readings were used whenever possible

2 - if a meter was stopped prior to a storm event, the assumption was made that it began to record when the runoff from the storm began.

3 - if the meter was recording at 1600 and still running at 0800 the next day the total flow was divided strictly by time i.e. one half of the flow was assigned to the time period 1600-2400, one half was assigned to the period 2400-0800.

4 - Weekend flows

a - if no rain occured during the weekend the flows would be interpolated from the Friday and Monday values.

b - if rain occured, meter readings would be used. Sometimes meter readings would be used to represent peak flow and base flow would be estimated from the Friday and Monday values.

5 - when there was no rain, the daily flow was computed from the volumetric measurement.

6 - in the early spring before individual meters were installed and before volumetric measurements were taken, the sump outflow was used to estimate the flow from each pile. The sump output divided by the number of piles that were flowing at that time gave the daily flow for each pile.

(A refinement could be made by weighting each pile by its collecting area. However, since the time period when these estimates were needed was relatively short and total flows were small, this refinement is probably not necessary).

EXAMPLE OF FLOW CALCULATIONS

The data for pile FL4 is shown below

(s=meter stopped, r=running; s-r, running, but sticking)

Date	Time	Meter Reading (gal)	Volumetric Flow Rate ml/min	Rainfall in Time	Daily Flow, liters
7-14	0800	6413s			
	1500	6412.9s	300	0	432
7-15	no readings			.08	432
7-16	weekend			0	281
7-17	0800	6413s		0	
	1400	6413s	90		130
7-18	0800	meter under water		1.51	
	1435	7235.2r	1020	2400-0200	3758
7-19	0800	7548s-r		.24 1400-1600	
	1455	7620.3r	2325		1274
7-20	0800	7849r		0	836

The calculations were made as follows:

Date	Rainfall	Method
7-14	0	volumetric flow used as daily average
7-15	.08	this was a small rain, the assumption was made that it was enough rain to maintain the overall flow at a level comparable to 7-14
7-16	0	since this was a weekend, no meter or volumetric measurements were made. The average of the flow on 7-15, which was assumed to be the same as the flow on 7-14, and 7-17 was used.

(cont'd)

Date	Rainfall	Method
7-17	0	volmetric flow was used as daily average
7-18	2400-0200 1.51	the assumption was that the meter began to record at 2400 the total flow from 2400-1435 was obtained from the meter the flow from 1435-2400 was obtained by time weighting the total recorded flow between 1435 and 0800, i.e. 2400-1435 = total flow from meter = 3113 l 1435 - 0800 = total flow from meter = 1184 l 1435 - 2400 = $\frac{9.5}{17.5}$ (1184) = 643 total daily flow =3113+643=3758 l
7-19	1400-1600 .24	1-the flow from 2400-0800 was obtained from the time weighted estimate as shown above 2-the flow from 0800-1500 was obtained from the meter 3-the flow from 1500-2400 was a time weighted estimate 2400-0800 $\frac{8}{17.5}$ (1184)= 541 liters 0800-1500 meter = 276 1500-2400 9/17 (867) = 457 l Total Flow=541+276+457=1274

Appendix III

WATER QUALITY DATA

All the water quality samples that were collected by the DNR in 1978 are listed in Table XVIII. On April 24, one set of samples was collected by Amax and analyzed by Serco Laboratories (Table XIX). The results of all the samples are tabulated by pile and are presented in Tables XX - XXV.

C= composite sample
G= grab sample

Table XVIII WATER QUALITY SAMPLES COLLECTED IN 1978

DATE	FLOW CONDITIONS	PILES SAMPLED 1	2	3	4	5	6
4/26	Low Flow	G	G	G	G	G	G
5/24-26	Composite over rain		C	C	C	C	C
6/6	Peak	G					
6/7	After Peak	G					
6/14	Low Flow				2G		
6/16	Low Flow	G	G	G	G	G	G
6/23	Composite over rain	3G			13G		
6/26	Tail of rain	G			G		
7/5	Tail of rain	G	G	G		G	G
7/6	Low Flow				G		
7/7	Composite over rain				24G		
7/8	Composite over rain				13G		
7/9	Tail of rain				G		
7/6-7	Peak	C					
7/7-11	Tail of rain	C					
7/12-13	Peak	C	C	C		C	C
7/13	Tail of rain				G		
7/14-19	Composite over rain	C	C	C		C	C
8/1-2	Composite over rain	C	C	C		C	C
8/2	Tail of rain					C	
8/4	Low Flow	G	G	G	G	G	G
8/10	Low Flow	G		G		G	G
8/14-16	Composite over rain	C			C		
8/16	Composite over rain	G	3G	3G	C	4G	G,C
8/17	Tail of rain					G	
8/18	Tail of rain					G	

C= composite sample
G= grab sample

DATE	FLOW CONDITIONS	PILES SAMPLED					
		1	2	3	4	5	6
8/16-17	Tail of rain				C		C
8/17-18	Composite over rain				C		
8/18-21	Low Flow				C		
8/17-23	Composite over rain	C	C	C		C	C
8/21-23	Before Peak				C		
8/23	Peak				2C	C	
8/23-25	Tail of rain	C	C	C	C	C	C
8/25-28	Composite over rain	C	C		C	C	C
9/10-13	Composite over rain		C	C	C		
9/10-11	Composite over rain				C		C
9/11	Before Peak					G	
9/11-12	Peak and after peak					C	
9/12-13	After peak	C					
9/13	Tail of rain	G	G	G	2G	G	G
10/14-17	Composite over rain	C			C		C

Table XIX
Amax Samples -- Serco

Date	4-24	4-24	4-24	4-24	4-24	4-24
Sample type	grab	grab	grab	grab	grab	grab
Time						
Site	FL-1	Fl-2	Fl-3	FL-4	FL-5	FL-6
Parameters						
Alkalinity	82	66	55	53	58	80
pH	7.8	7.8	7.6	7.5	7.7	7.7
Temperature						
Specific Conductance	967	819	642	776	1520	5150
Calcium	115	75	67	88	165	430
Magnesium	11	17	12	8.2	12	150
Sodium	110	90	58	63	162	516
Potassium	9.0	8.2	8.2	17	4.5	8.5
Copper	.025	.019	.013	.017	.039	.086
Nickel	.110	.095	.075	.090	.725	.960
Iron	.15*	7.5*	.35*	.20*	.25*	.45*
Manganese	.06	.06	.06	.09	.33	.66
Cobalt	.007	.006	.005	.007	.082	.042
Zinc	.033	.029	.017	.048	.021	.011
Sulfate	540	400	300	350	780	680
Chloride	3	3	2	5	32	569
Nitrate	2.7	4.3	3.1	1.3	28	460
Ammonia						
Total Phosphorus	0.1	.2	.1	.1	.1	0.1
Dissolved Organic Carbon	8.9	6.8	4.0	5.0	10	5.8

* Total Metal

Explanation of comment statements

1. Rain If the sample was collected during a rainfall event, the total amount of rainfall and its time of occurence is given. If the sample was collected between events, the time and amount of the preceding rainfall is given.

2. Portion of the hydrograph sampled
If the sample was a grab sample, the general position of the sample relative to the peak in stockpile runoff is given. If the sample was a composite, the portion over which the sample was collected is given.

3. Other These comments refer to field observations on pile behavior, sample appearance, etc.

Table XX WATER QUALITY DATA FL 1, 1978

FL-1 Date	4-26	6-6	6-7	6-16	6-23			6-26
Sample type	grab	grab	grab	grab	grab	grab	grab	grab
Time		2110	0900	1025	0950	1300	1525	0900
Parameters								
Alkalinity								
pH	7.15		7.60	7.84	7.49	7.82	7.85	6.85
Temperature	9.0	7.0		11.0	14.0	12.0	13.0	15.0
Specific Conductance	1120	1900	3325	2700	1775	2150	2200	2400
Calcium	112	216	253	244	189	180	181	195
Magnesium	11	19	23	22	18	17	17	20
Sodium	113	268	327	368	313	273	279	313
Potassium	13	20	23	22	23	21	20	21
Copper	.032	.030	.041	.028	.027	.024	.023	.024
Nickel	.037 [a]	.250	.272	.155 [a]	.274	.228	.250	.294
Iron	.20 [a]	.02	.03	.05	.02	.02	.03	.04
Manganese	.06	.13	.15	.14	.10	.09	.08	.09
Cobalt	.003	.009	.005	.001	.013	.012	.012	.014
Zinc	.017	.002	.030 [a]	.002	.004	.002	ND	.010
Sulfate	Not on computer				1400	1400	1000	
Chloride					6	6	3	
Nitrate								
Ammonia								
Total Phosphorus								
Dissolved Organic Carbon								
Discharge at time of Sample (ml/min)	trickle	2558	-	120	175	207	127	59
Total Volume over composite (liters	-	-	-	-	-	-	-	-

Comments

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umhos
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.

2. All metal values are filtered, unless otherwise specified.

3. a = Value appears anomalous b = reanalysis

Comments

3. Rain		.75" 1730-2400	last rain 6-6 0.75" 1730-2400	last rain 6-10 0.19" 2300-2400	——— .35" ———			last rain 6-25 .04" 1800-1830
					0745-1045 1915-2000		.29" .06"	
4. Portion of hydrograph sampled	low flow	Peak	After peak	low flow	Before peak	After peak	After peak	Tail of 0.4" rain
5. Other	Initial sample Snow Run off					Some gray matter in water		

FL-1

Date Sample type Time	7-5 Grab 0830	7-7 Comp. 1600 on 7-6 to 1630 on 7-7	7-11 Comp. 1630 on 7-7 to 1525 on 7-11	7-13 Comp. 1200 on 7-12 to 0840 on 7-13	7-18 Comp. 1530 on 7-14 to 1400 on 7-19	8-2 Comp. 8-1 1700 to 1515 on 8-2	8-4 Grab 1045	8-10 Grab 1340
Parameters								
Alkalinity								70
pH	7.75	7.90	7.70	7.92	7.80	7.90	7.80	8.10
Temperature	14.0	19				22	22	24
Specific Conductance	2100	1900	2450		1400	1100	1650	2400
Calcium	202	221	303[b]	531[a]	340	176	300	414
Magnesium	20	23	26	31	19	11	23	26
Sodium	206	222	286	395	220	127	209	317
Potassium	28	34	38	45	34	25	37	46
Copper	.018	.024	.024, .03[b]	.027	.023	.027	.034	.03
Nickel	.206	.285	.405, .37[b]	.405[a]	.255	.21	.33	.46
Iron	.01	.04	.03	.01	.01	.01	.03	.02
Manganese	.09	.09	.14	.16	.11	.09	.13	.16
Cobalt	.007	.019	.022, .03[b]	.026	.019	.010	.016	.03[b]
Zinc	ND	.005	.01[b]	.058[a]	.003	.010	.013	.025
Sulfate	900	800	1150	1500	1050	650	1200	1600
Chloride	4	4	5	4	1	1	3	4
Nitrate		5.7	5.7	6.2	3.9	3.4	5.0	-
Ammonia		.44	.64	0.55	0.5	0.7	< 0.08	< 0.13
Total Phosphorus		< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	0.1	< 0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	177 (Q at 1535)						167	70
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite a= Value appears anomalous b = reanalysis

Comments								
4. Rain	Last rain 7-1, 7-2 1.36" 1400 on 7-1 to 1100 on 7-2	1.38" 0600-1400	1.38" from 0600-1400 on 7-7; 0.15" from 1100 7-8 to 0200 7-9	0.77" from 1400-1600 and 2000-2300 on 7-12	1.51" 0001-0200	Last rain 8-1 0.95"	Last rain 8-1 0.95"	Last rain 8-1 0.95" 0.13" on 8-10 time 1100-1200
5. Portion of hydrograph	Tail of 1.36" rain	Peak	Tail	Peak	Comp. over rain 7-18	Comp. over rain 8-1	Tail of 0.95" rain	Low flow
6. Other			Comp. was 1816 mls		First of the present drip comps.			

FL-1

Date	8-16	8-16	8-24	8-25	8-28	9-13	9-13	
Sample type	Grab	Comp.	Comp.*	Comp.	Comp.	Grab	Comp.	
Time	1015	8-14 2030 to 1450 on 8-16	1025 on 8-17 to 1945 on 8-23	1945 on 8-23 to 0830 on 8-25	0830 on 8-25 to 0820 on 8-28	1020	1130 on 9-12 to 1020 on 9-13	
Parameters								
Alkalinity		42	50	47	59	41	40	
pH	8.00	7.70	8.10	7.55	7.85	7.6	7.55	
Temperature	23	16		22		14		
Specific Conductance	1400	1675	2600	2500	2700	2175	2000	
Calcium	190	149	302	300	308	254	238	
Magnesium	15	13	26	26	26	20	18	
Sodium	107	97	240	229	280	169	151	
Potassium	28	22	40	43	40	41	38	
Copper	.02	.02	.04	.04	.05	.05	.04	
Nickel	.20	.17	.36	.35	.42	.27	.28	
Iron	.02	.02	.06	.02	.04	.03	.03	
Manganese	.07	.07	.14	.14	.15	.14	.10	
Cobalt	.02	.02	.04	.03[b]	.07[b]	.02	.02	
Zinc	.01	.02	.02	.02	.02	.01	.01	
Sulfate		640	1400	1300		1100	1000	
Chloride		2	1	1		1	2	
Nitrate		2.5	5.0	4.6		-	-	
Ammonia		-	-	0.40		-	-	
Total Phosphorus		<0.1	<0.1	0.2		<0.1	<0.1	
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	3441					426		
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite a = Value appears anomalous b = reanalysis

Comments 4. Rain	1.28" 0400- 0800 .60" on 8-14 .26" on 8-15	1.28" 0400- 0800 .60" on 8-14 .26" on 8-15	Last rain 8-22,8-23 1.75" 2000 on 8-22 to 1100 on 8-23	Last rain 8-22, 8-23 1.75" 2000 on 8-22 to 1100 on 8-23	Last rain 8-27 0.20" 1200- 1400	Last rain 9-10, 11,12 2.46" total 1.00" 1200-1530 9-10 1.31" 0830-1200 9-11 .15" 0300-1000 0-12	
5. Portion of hydrograph	Peak	Comp. over rain 8-16	Comp. over rain 8-22,23	Tail of 175" rain	Comp. over rain 8-27	Tail of 2.46" rain	After Peak
6. Other							
*Mean of duplicate samples							

FL-2 Table XXI Water Quality Data, FL2, 1978

Date	4-26	5-25	6-16	7-5	7-13	7-19	8-2	8-4
Sample type	Grab	Comp.	Grab	Grab	Comp.	Comp.	Comp.	Grab
Time			1120	0330	1155 on 7-12 to 0840 on 7-13	0810 on 7-18 to 1400 on 7-19	Start of rain to 0930 on 8-2	1030
Parameters								
Alkalinity								
pH	7.20	7.70	7.72	7.5	7.92	7.80	7.75	7.75
Temperature	4.0		11.0	12			19	22
Specific Conductance	1000	1900	4200	2700		1300	2000	2500
Calcium	99	177	323	218	491	291	191	341
Magnesium	13	23	47	33	43	24	21	42
Sodium	101	241	555	313	376	189	241	410
Potassium	12	15	23	28	37	28	26	33
Copper	.038	.040	.056	.057	.049	.062	.08	.07
Nickel	.030[a]	.240	.200[a]	.316[a]	.410	.24[b]	.26	.44
Iron	.82[a]	.05	.03	.04	.03	.01	.04	.05
Manganese	.07	.16	.31	.22	.29	.16	.18	.28
Cobalt	.003	.008	.015	.021	.033	.01, .019	.01[b]	.029
Zinc	.020	.028	.075	.084	.09[b]	.033	.055	.113[a]
Sulfate				1400	800[r]	800[r]	1100[r]	1700
Chloride				34	30	41	14	19
Nitrate					23	24	19	26
Ammonia					.70	.55	.8	<.08
Total Phosphorus					<0.1	<0.1	<0.1	<0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	Trickle		192	133				100
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis
5. r = Analysis rerun, see 10-5-78 memorandum from Serco.

Comments								
4. Rain		0.29" 0630-0730	Last rain 6-10 0.10" 2300-2400	Last rain 7-1,7-2 1.36" 1400 on 7-1 to 1100 on 7-2	0.77" from 1400-1600 & 2000-2300 on 7-12	1.51" from 0001-0200 on 7-18; 0.24" on 7-19 (time unknown)	Last rain 8-1 0.95"	Last rain 8-1 0.95"
5. Portion of hydrograph		Comp. over rain 5-25	Low flow	Tail of 1.36" rain	Peak	Tail of 1.51" rain on 7-18	Comp. over rain 8-1	Tail of 0.95" rain
6. Other	Initial sample					First of the pre-set drip comps.		

FL-2

Date Sample type Time	8-16 Grab 1010	8-16 Grab 1345	8-16 Grab 1655	8-23 Comp. 1000 on 8-17 to 1940 on 8-23	8-25 Comp. 1940 on 8-23 to 0840 on 8-25	8-28 Comp. 0840 on 8-25 to 0820 on 8-28	9-13 Grab* 1045	9-13 Comp. Rain on 9-10 to 1045 on 9-13
Parameters								
Alkalinity		38	41	31	35	48	41	17
pH	7.20	7.80	7.75	7.60	7.30	7.55	7.45	7.05
Temperature	23				23		15	
Specific Conductance	975	1100	1200	1100	1600	2300	1725	950
Calcium	100	108	116	103	170	253	182	84
Magnesium	12	15	16	15	24	34	22	10
Sodium	88	95	104	86	125	196	138	74
Potassium	17	16	17	18	28	32	30	23
Copper	.14, .17[b]	.12, .14[b]	.13, .14[b]	.13, .15[b]	.11	.10	.10	.14
Nickel	.15	.18	.21	.18	.24	.38	.26	.12
Iron	.05	.05	.04	.04	.01	.01	.02	.03
Manganese	.08	.10	.10	.10	.15	.23	.16	.06
Cobalt	.01[b]	.02	.04	.02	.03	.05, .04[b]	.02	.02
Zinc	.02	.02	.04	.02	.05	.09[b]	.06	.03
Sulfate		500	500	460	680	1100	600	567
Chloride		17	17	15	25	34	18	6
Nitrate		10	11	9.4	17	24	13	-
Ammonia		-	-	-	0.64	-	<0.10	-
Total Phosphorus		0.1	0.8	< 0.1	<0.1	0.1	<0.1	< 0.1
Dissolved Organic Carbon		17						
Discharge at time of sample (ml/min)	1500	632	420				130	
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus, and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis

Comments								
4. Rain	1.28" from 0400-0800 on 8-16 also: .60" from 2030-2400 on 8-14 .26" from 0001-0100 and 1000-1130 on 8-15			.23" on 8-18 0001-0100 .60" on 8-22 2000-2400 1.15" on 8-23 0001-1100	Last rain 822 8-23 1.75" 2000 on 8-22 to 1100 on 8-23	Last rain 8-27 0.20" 1200-1400	Last rain 9-10,11, 12 2.46" total 1.00" 1200-1530 9-1 1.31" 0830-1200 9-1 .15" 0300-1000 9-1	
5. Portion of hydrograph	Peak	After Peak	After Peak	Comp. over rain 8-18, 22 8-23	Tail of 1.75" rain	Comp. over rain 8-27	Tail of 2.46" rain	Comp. over rain 9-10, 11,12
6. Other			Water was yellow colored					

*Mean of duplicate samples

FL-3

Table XXII Water Quality Data, FL3, 1978

Date Sample type Time	4-26 Grab	5-25 Comp.	6-16 Grab 1205	7-5 Grab 0845	7-13 Comp. 1210 on 7-12 to 0840 on 7-13	7-19 Comp. 1530 on 7-14 to 1400 on 7-19	8-2 Comp. 8-1 1800 to 1515 on 8-2	8-4 Grab* 1100
Parameters								
Alkalinity								
pH	7.20	7.80	7.79	7.3	7.55	7.85	7.60	7.65
Temperature	4.5		6.0	9.0			22	22
Specific Conductance	675	1200	3000	1600		1000	650	950
Calcium	62	106	214	142	352	241	96	156
Magnesium	10	24	57	34	46	28	14	24
Sodium	52	122	372[a]	102	207	-	60	88
Potassium	12	15	21	23	32	27	21	26
Copper	.023	.013	.026	.20[b]	.049	.074, .11[b]	.08	.08
Nickel	.06[b]	.042[a]	.185	.282, .36[b]	.405	.140, .33[b]	.21	.32
Iron	.70[b]	.04	.06	< .02	.03	.01	.03	.02
Manganese	.05	.09	.24	.17	.24	.15	.16	.16
Cobalt	0	.050[a]	.012	.017	.030	.020	.010	.020
Zinc	0	.020	.059	.037	.092	.055	.025	.044
Sulfate				600	1050[r]	550[r]	367[r]	500
Chloride				78	34	35	12	17
Nitrate					12	10	8.0	11
Ammonia					1.0	0.75	0.2	<0.08
Total Phosphorus					<0.1	0.1	0.2	<0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	Trickle		496[BW]	293				1000[BW]
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis
5. r = Analysis rerun, see 10-5-78 memorandum from Serco.

Comments								
4. Rain		0.29" 0630-0730	Last rain 6-10 0.10" 2300-2400	Last rain 7-1, 7-2 1.36" 1400 on 7-1 to 1100 on 7-2	0.77" from 1400-1600 & 2000-2300 on 7-12	1.51" from 0001-0200 on 7-18; 0.24" on 7-19 (time unknown)	Last rain 8-1 0.95"	Last rain 8-1 0.95"
5. Portion of hydrograph		Comp. over rain 5-25	Low flow	Tail of 1.36" rain	Peak	Tail of 1.51" rain on 7-18	Comp. over rain 8-1	Tail of 0.95" rain
6. Other	Initial sample					First of the pre-set drip comps.; comp. didn't drip over rain period. Caught tail of rain from about 0810 on 7-18 to 1400 on 7-19		

*Mean of duplicate samples

BW=backwater is present

FL-3

Date Sample type Time	8-10 grab 1340	8-16 grab 1025	8-16 grab 1400	8-16 grab 1640	8-23 Comp. 1050 on 8-17 to 1955 on 8-23	8-25 Comp. 1955 on 8-23 to 0835 on 8-25	9-13 grab 1030	9-13 Comp. Rain on 9-10 to 1020 on 9-13
Parameters								
Alkalinity	40		28	30	33	25	43	29
pH	7.90	7.60	7.65	7.60	7.50	7.30	7.3	7.30
Temperature	24	23	22	16		24	14	
Specific Conductance	1900	700	750	800	750	1150	1350	29
Calcium	348	79	89	83	75	115	147	120
Magnesium	51	13	15	16	14	23	24	19
Sodium	188	42	50	46	45	67	81	68
Potassium	49	17	20	16	17	23	28	27
Copper	.05,.07[b]	.11,.12[b]	.11,.13[b]	.10,.13[b]	.10	.08	.08	.09
Nickel	.56,.57[b]	.20,.18[b]	.24,.17[b]	.23,.22[b]	.19	.25	.30	.30
Iron	.01	.05	.08	.08	.07	.04	.04	.03
Manganese	.28	.07	.08	.09	.08	.10	.14	.10
Cobalt	.09	.02	.01	.02	< .02[b]	.01	.01	.03
Zinc	.073	.02	.02	.02	.02	.03	.04	.03
Sulfate	1200		300		300	500	750	567
Chloride	48		10		7	11	8	6
Nitrate	-		5.3		3.6	8.5	5.5	-
Ammonia	< 0.13		1.1		-	1.3	< 0.08	-
Total Phosphorus	< 0.1		0.1		0.1	< 0.1	< 0.1	< 0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	30	14,195[d]	2839	2103			451	
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite a = Value appears anomalous b = reanalysis

[d] Backwater suspected

FL-3 Date Sample type	8-10 grab	8-16 grab	8-16 grab	8-16 grab	8-23 Comp. 1050 on 8-17 to 1955 on 8-23	8-25 Comp. 1955 on 8-23 to 0835 on 8-25	9-13 grab 1030	9-13 Comp. Rain on 9-10 to 1020 on 9-13
4. Rain	last rain 8-1 0.95"	1.28" from 0400-0800 on 8-16. Also: 0.60" from 2030-2400 on 8-14 0.26" from 0001-0100 and 1000-0130 and 1000-1130			.23" on 8-18 0001- 0100 60" on 8-22 2000-2400 1.15" on 8-23 0001-1100	last rain 8-22,23 1.75" 2000 on 8-22 to 1100 on 8-23	last rain 2.46 total 1.00" 1300-1530 9-1 1.31" 0830-1200 9-1 0.15" 0300-1000 9-1	
5. Portion of hydrograph	low flow	Peak	After peak	After peak	Comp. over rain 8-18,8-22, 8-23	Tail of 1.75" rain	Tail of 2.46" rain	Comp. over rain 9-10,11,1
6. Other								

FL-4

Table XXIII Water Quality Data, FL4, 1978

Date	4-26	5-25	6-14	6-16	6-23	6-23	6-23	6-23
Sample type	Grab	Comp.	Grab*	Grab	Grab	Grab	Grab	Grab
Time			0745	0800	0950	1045	1200	1245
Parameters								
Alkalinity								
pH	7.20	7.40	7.60	7.70	7.49	7.35	7.2	7.59
Temperature	4.0		9.0	11	14.0	13.0	13.0	14.0
Specific Conductance	1100	1750	2500	2600	1300	1775	2575	2800
Calcium	121	204	228	247	134	151	207	223
Magnesium	11	20	29	30	17	19	28	31
Sodium	84	165	262	277	154	192	275	308
Potassium	27	35	37	41	27	30	39	43
Copper	.018	.012[a]	.024	.026	.022	.023	.026	.028
Nickel	.035[a]	.210	.220	.175	.172	.216	.314	.336
Iron	.42 [a]	.02	.04	.04	.01	.06	.05	.05
Manganese	.10	.19	.21	.26	.16	.17	.22	.23
Cobalt	.004	.012	.016	.015	.009	.015	.025	.028
Zinc	.048	.112	.128	.120	.017	.042	.124	.174
Sulfate					600	900	1400	1400
Chloride					17	20	27	30
Nitrate								
Ammonia								
Total Phosphorus								
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	Trickle		125	94	90	423	475	400
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = ^{o}C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous

Comments								
4. Rain		0.29" 0630-0730	Last rain 6-10 0.10" 2300-2400	Last rain 6-10 0.10" 2300-2400	Total rainfall on 6-23 = 0.35" .29" from 0745 - 1045 .06" from 1915 - 2000			
5. Portion of hydrograph	Low flow	Comp. over rain 5-25	Low flow sample just prior to rain.	Low flow				
6. Other	Initial sample				Some cloudy grey when filtered on surface of unit.		Water grey in color prior to filtering	Water not as turbid as previous sample.

*Mean of duplicate samples.

type	6-23 Grab 1420	6-23 Grab* 1540	6-23 Grab 1700	6-23 Grab 1800	6-23 Grab 1900		6-23 Grab* 2100	
eters								
alinity								
	7.45	7.45	7.82	7.55	7.8		7.75	
perature	14.0	14		7.0				
ific uctance	2650	2650	2650	2500	2625		2500	
lcium	225	223	215	228	220		216	
esium	31	30	29	32	30		30	
dium	300	296	291	306	293		292	
ssium	43	42	41	43	43		42	
pper	.026	.024	.020	.015	.020		.020	
kel	.336	.325	.314	.360	.336		.320	
n	.03	.02	.02	.03	.01		< .02	
nganese	.23	.22	.20	.24	.23		.22	
	.026	.026	.022	.026	.023		.022	
inc	.156	.152	.170	.173	.160		.154	
fate	1200	1200	1200		1300			
nloride	28	21	31		30			
trate								
monia								
otal osphorus								
Dissolved rganic rbon								
Discharge at ime of sample l/min)	280	175						
Total Volume over omposite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus, and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite

Comments							
4. Rain			Total rainfall on 0.29" from 0745 - 0.06" from 1915 -		6-23 = 0 1045 2000	.35"	
5. Portion of hydrograph							
6. Other	Clear water		Isco #2 pH taken 6-26 1350; filtered 6-26; added to Isco #3 for SO_4 Cl	Isco #3	Isco #4 pH taken 6-26 1340; filtered 6-26; added to Isco #5 for SO_4 and Cl		Isco #6 pH taken 6-26 1555; filtered at 1445 on 6-26; kept separate for SO_4 Cl

* Mean of duplicate samples.

FL-4

Date	6-23	6-26	7-6	7-7	7-7	7-7	7-7	7-7
Sample type	Grab	Grab	Grab	Grab	Grab	Grab	Grab	Grab
Time	2300	0920	0830	0400	0500	0600	0700	0800
Parameters								
Alkalinity								
pH	7.7	7.90	7.58	7.65	7.60	7.45	7.2	7.3
Temperature		17	17					
Specific Conductance	2400	2150	1825	2050	2050	2050	1200	1400
Calcium	213	186	212	234	242[b]	234	132	166
Magnesium	29	24	25	25	24[b]	27	15	17
Sodium	278	260	190,173[b]	201	220[b]	239	137	156
Potassium	41	41	52	45	43[b]	42[b]	20[b]	29[b]
Copper	.013	.021	.019	.017	.02	.02	.01	.02
Nickel	.36[b]	.260	.275	.290	.32	.31	.17	.18
Iron	.01[b]	0[a]	.03	.03	.01	.02	.04	.01
Manganese	.22	.18	.18	.20	.19	.19	.12	.13
Cobalt	.025	.019	.019	.022	.06	.06	.02	.04
Zinc	.147	.084	.073	.132	.13	.12	.06	.07
Sulfate		900	850[r]	1000[r]				550
Chloride		28	11	17				9
Nitrate								
Ammonia								
Total Phosphorus			0.1	< 0.1				< 0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)		5	88					
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis
5. r = Analysis rerun, see 10-5-78 memorandum from Serco.

Comments								
4. Rain		Last rain 6-25 .04" 1800 -1830	Last rain 7-1,7-2 1.36" 1400 on 7-1 to 1100 on 7-2	Total rainfall on 7-7 = 1.38" from 0600 - 1400				
5. Portion of hydrograph		Tail of .04" rain	Low flow after 1" rain					
6. Other	Isco #8 pH taken 6-26 1540; filtered 6-26 added to Inco #7 for SO_4 and Cl			Isco filtered 1450	Isco filtered 1455	Isco combined for SO_4 #Cl	Isco	Isco filtered 1520 combined with 0900 for general parameters

FL-4

Date Sample type Time	7-7 Grab 0830	7-7 Grab 0900	7-7 Grab 1000	7-7 Grab 1100	7-7 Grab* 1200	7-7 Grab 1300	7-7 Grab 1400	7-7 Grab 1500
Parameters								
Alkalinity								
pH	7.20	7.15	7.40	7.40	7.40	7.40	7.15	
Temperature					11	11	11	
Specific Conductance	1000	1450	1550	1275	1125	1050	950	
Calcium	176	172	179	145	126	115[b]	104	
Magnesium	18	18	19	15	13	10[b]	10	
Sodium	161	157	164	122	110	88[b]	80	
Potassium	1363[a]	29[b]	33[b]	268[a]	607[a]	22[b]	20[b]	
Copper	.02	.02	.02	.02	.02	.01	.01	
Nickel	.25	.21	.24	.20	.14	.14	.10	
Iron	.03	.07	.05	.01	.02	.02	.02	
Manganese	.12	.13	.12	.10	.08	.08	.07	
Cobalt	.02	.02	< .01	.02	.02	.01	< .02	
Zinc	.07	.07	.08	.05	.04	.04	.04	
Sulfate	600	550	600[r]	450[r]	450	400	250[r]	500
Chloride	9	9	11	8	6	6	5	5
Nitrate	2.0				1.8		1.4	
Ammonia	.76				1.1		.96	
Total Phosphorus	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	13,360		11,238	8,051	5,002	4,732	6,489	5,344
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = ^{o}C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis
5. r = Analysis rerun, see 10-5-78 memorandum from Serco.

Comments							
4. Rain				Total rainfall on 7-7 = 1.38" from 0600 - 1400			
5. Portion of hydrograph							
6. Other		Isco filtered 1530 combined with 0800 for G.P.	Isco				
*Mean of duplicate	samples.						

FL-4

Date Sample type Time	7-7 Grab 1600	7-7 Grab 1700						
Parameters								
Alkalinity								
pH	7.65	7.35						
Temperature	11	11						
Specific Conductance	950	1100						
Calcium	144	119						
Magnesium	11	11						
Sodium	84	89						
Potassium	22[b]	24[b]						
Copper	.01	.01						
Nickel	.11	.12						
Iron	.01	.03						
Manganese	.08	.09						
Cobalt	< .01	< .01						
Zinc	.04	.05						
Sulfate	430[r]	450[r]						
Chloride	5	5						
Nitrate								
Ammonia								
Total Phosphorus	< 0.1	0.1						
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	3586	2074						
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = ^{o}C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphoru and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis
5. r = Analysis rerun, see 10-5-78 memorandum from Serco.

Comments							
4. Rain	Total rainfall on 7-7 = 1.38" from 0600 - 1400						
5. Portion of hydrograph							
6. Other							

FL-4

Date	7-7	7-7	7-7	7-7	7-7		7-7	7-7
Sample type	Grab	Grab*	Grab	Grab	Grab		Grab	Grab
Time	1800	1900	2000	2100	2200		2300	2400
Parameters								
Alkalinity								
pH	7.65	7.50	7.50	7.45	7.40		7.55	7.65
Temperature	11	11	11	11	11			
Specific Conductance	1125	1150	1175	1200	1200		1175	1200
Calcium	129	128	134	136	139		132	189
Magnesium	12	12	14	14	13		13	15
Sodium	97	92	110	111	99		96	114
Potassium	35	26[b]	38	37	28[b]		32	40
Copper	.012	.015	.012	.014	.01		.01	.009
Nickel	.170	.135	.170	.150	.14		.14	.160
Iron	.05	.02	.02	.02	.02		.02	.02
Manganese	.10	.10	.10	.10	.10		.09	.11
Cobalt	.011	.01	.011	.011	< .01[b]		.01	.033
Zinc	.049	.05	.047	.056	.04		.05	.050
Sulfate		480		500			540	
Chloride		5		5			5	
Nitrate		1.5		1.5				
Ammonia		.44		.40				
Total Phosphorus		< 0.1		< 0.1			< 0.1	
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	2469	2055	1747	1822	1431			
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis

Comments								
4. Rain		Total rainfall on 7-7 = 1.38" from 0600 - 1400						
5. Portion of hydrograph								
6. Other					Filtered immedi-ately	Isco #1 filtered 7-10 1330	Isco #2 filtered 7-10 1330	Isco #3 filtered 7-10 1330

*Mean of duplicate samples.

...e ...ple type ...	7-8 Grab 0100	7-8 Grab 0200	7-8 Grab 0300	7-8 Grab 0400	7-8 Grab 0500	7-8 Grab 0600	7-8 Grab 0700	7-8 Grab 0800
...rameters								
...lkalinity								
...	7.65	7.60	7.72	6.65	7.60	7.50	7.50	7.60
...emperature								
...pecific ...nductance	1225	1250	1275	1350	1325	1325	1350	1400
Calcium	194	212	210	224	233	245	159[a]	168,175[b]
...agnesium	15	16	15	12	16	16	16	17
Sodium	111	121	118	116	121	124	113	119
...otassium	39	43	40	40	41	42	34	38
...opper	.014	.012	.007	.014	.012	.009	.02	.02
Nickel	.160	.160	.170	.225	.215	.205	.18	.18
...ron	.02	.04	.07	.04	.03	.01	.02	.04
Manganese	.12	.12	.12	.12	.13	.14	.11	.12
...alt	.011	.015	.013	.015	.015	.015	.02	.04[b]
Zinc	.049	.049	.055	.053	.061	.056	.05	.06
...ulfate							650[r]	
...hloride							6	
...itrate								
...mmonia								
Total Phosphorus							< 0.1	
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)								
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus, and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous
5. r = Analysis rerun, see 10-5-78 memorandum from Ser...
   b = reanalysis

Comments								
4. Rain	1.38"	from 0600 7-8 to 0200	- 1400 on on 7-9	7-7, 0.15" from 1100 on				
5. Portion of hydrograph								
6. Other	Isco #4 Filtered 7-10 1330	Isco #5 Filtered 7-10 1330	Isco #6 Filtered 7-10 1330	Isco #7 Filtered 7-10 1530	Isco #8 Filtered 7-10 1530	Isco #9 Filtered 7-10 1530 Lack of H_2O	Isco #10 Filtered 7-10 1530	Isco #11 Filtered 7-10 1530

FL-4

Site Sample type Time	7-8 Grab 0900	7-8 Grab 1600	7-8 Grab 2000	7-8 Grab 2300	7-9 Grab 0100	7-13 Grab 1130	8-4 Grab 1010	8-15 Comp. 8-14 to 0930 on 8-15
Parameters								
Alkalinity								39
pH	7.3	7.48	7.55	7.40	7.55	7.65	8.00	7.80
Temperature							22	21
Specific Conductance	1425	1600	1650	1700	1550		1100	1275
Calcium	303	167,177[b]	302	212,222[b]	300	273	191	201
Magnesium	20	16	21	21	20	18	13	19
Sodium	153	121	155	152	149	153	115	158
Potassium	49	37	48	42	46	44	43	39
Copper	.012	.01	.012	.02	.009	.012	.020	.02
Nickel	.255	.21[a]	.255	.26	.240	-	.20[a]	.24
Iron	.01	.01	.02	.02	.02	.02	.02	.02
Manganese	.16	.12,.16[b]	.18	.16	.16	.14	.14	.15
Cobalt	.019	.03[b]	.019	.03	.017	.015	.010	.01[b]
Zinc	.064	.06	.064	.07	.005[a]	.004[a]	.052	.07
Sulfate		950	950[r]			850	660	800
Chloride		8	7			7	3	6
Nitrate						2.2	1.2	1.2
Ammonia						0.90	< 0.08	1.2
Total Phosphorus		< 0.1	< 0.1			< 0.1	< 0.1	< 0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)							95	
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous
5. r = Analysis rerun, see 10-5-78 memorandum from Serco. b = reanalysis

Comments								
4. Rain	1.38" from 0600 - from 1100 on 7-8		1400 on 7-7, 0.15" to 0200 on 7-9			0.77" from 1400-1600 and 2000-2300 on 7-12	Last rain 8-1 0.95"	.60" from 2030-2400 on 8-14; .26" from 0001-0100 1000-1130 on 8-15
5. Portion of hydrograph						Tail of 0.77"	Tail of 0.95"	Comp. over rain 8-14,15
6. Other	Isco #12 Filtered 7-10 1530	Isco #19 Filtered 7-11 1520	Isco #23 Filtered 7-11 1520	Isco #26 Filtered 7-11 1515	Isco #28 Filtered 7-11 1515			

*Mean of duplicate samples.

FL-4

Date Sample type Time	8-16 Comp. Start of rain to 1455 on 8-16	8-17 Comp. 1455 on 8-16 to 0800 on 8-17	8-18 Comp. 0800 on 8-17 to 1500 on 8-18	8-21 Comp. 1500 on 8-18 to 0830 on 8-21	8-23 Comp. 0830 on 8-21 to 0800 on 8-23	8-23 Comp. 0800 on 8-23 to 1440 on 8-23	8-23 Comp. 1445 on 8-23 to 1925 on 8-23	8-25 Comp. 1925 on 8-23 to 0835 on 8-25
Parameters								
Alkalinity	28	29	36		29	30	31	33
pH	7.50	7.29	7.60	7.75	7.65	7.05	7.70	7.55
Temperature						23		23
Specific Conductance	825	1175	1850	2050	1400	1000	1250	1750
Calcium	118	355	219	271	155	111	147	216
Magnesium	11	41	21	27	16	11	18	21
Sodium	60	158	110	142	102	67	85	115
Potassium	26	12	42	50	31	30	36	46
Copper	.02	.03	.02	.08	.02	.02	.02	.02
Nickel	.13	.20	.24	.32	.18	.11	.14	.24
Iron	.03	.04	.02	.46[a]	.01	.01	.04	.01
Manganese	.09	.12	.16	.20	.11	.08	.10	.14
Cobalt	.02	.02	.04	.04[b]	.03	.02	.02	.03
Zinc	.03	.05	.07	.10[b]	.05	.03	.03	.06
Sulfate	100 ?	1000	800		700	500	610	800
Chloride	1	13	3		3	1	3	2
Nitrate	1.2	2.2	1.4		1.5	1.3	1.4	1.4
Ammonia	-	-	-		0.96	1.2	-	1.0
Total Phosphorus	<0.1	<0.1	<0.1		<0.1	<0.1	<0.1	<0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)								
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis

Comments								
4. Rain	1.28" 0400-0800 .60" on 8-14 .26" on 8-15	Last rain 8-16 1.28" 0400-0800	0.25" 0001-0100	Last rain 8-18 0.25" 0001-0100	.60" from 2000-2400 on 8-22 1.15" from 0001 - 1100 on 8-23			Last rain 1.75" 2000 on 8-22 to 1100 on 8-23
5. Portion of hydrograph	Comp. over rain 8-16	Tail of 1.28" rain	Tail of 1.28" rain; Comp. over rain 8-18	Low flow	Before peak	Peak	After peak	Tail of 1.75" rain
6. Other								

FL-4

Date Sample type Time	8-28 Comp.* 0855 on 8-25 to 0815 on 8-28	9-11 Comp. 9-10 to 0835 on 9-11	9-13 grab 1100	9-13 grab 1100	9-13 Comp. 9-10 rain to 1100 on 9-13			
Parameters								
Alkalinity	41.8	35	34	34				
pH	7.55	7.30	7.40	7.40	6.9			
Temperature			12	12				
Specific Conductance	2350	2300	1875	1875	1200			
Calcium	280	256	226	226	123			
Magnesium	29	26	21	21	11			
Sodium	187	219	120	119	83			
Potassium	60	59	51	52	30			
Copper	.03	.03	.03	.03	.01			
Nickel	.34	.34	.25	.25	.13			
Iron	.02	.06	.08	.03	.06			
Manganese	.19	.20	.16	.16	.09			
Cobalt	.04	.05	.05	.04	.01			
Zinc	.10	.08	.06	.07	.03			
Sulfate	1250	1200		1070	633			
Chloride	2	10		2	3			
Nitrate	2.4	-		1.4	-			
Ammonia	-	-		<0.08	-			
Total Phosphorus	<0.1	<0.1		<0.1	<0.1			
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)								
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = ^{o}C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite

Date Sample type Time	8-28 Comp.* 0835 on 8-25 to 0815 on 8-28	9-11 Comp. 9-10 to 0835 on 9-11	9-13 grab 1100	9-13 grab 1100	9-13 Comp. 9-10 rain to 1100 on 9-13			
4. Rain	Last rain 8-27 0.20" 1200-1400	~1.00" from 1200 -1530 on 9-10	Last rain 12 2.46" ~1.00" 9-10 ~1.31" 9-11 ~.15" 9-12	9-10,11, total 1200-1530 0830-1200 0500-1000				
5. Portion of Hydrograph	Comp. over rain 8-27	Comp. over rain 9-10	Tail of 2.46" rain	Comp. over rain 9-10,11, 12				
6. Other			Total Metals	Took 150 mls of each of 3 comp. and added together 9-10 to 9-11 0835 9-11 0835 to 9-11 1145 9-11 1145 to 9-13 1100		filtered metals		

* Mean of duplicate samples

FL-5

Table XXIV Water Quality Data, FL5, 1973

Date Sample type Time	4-26 grab	5-25 Comp.	6-16 grab 1115	7-5 grab 0800	7-13 Comp. 1150 on 7-12 to 0830 on 7-13	7-19 Comp. 0900 on 7-18 to 1400 on 7-19	8-2 Comp. 8-1 1700 to 0900 on 8-2	8-2 Comp. 0900 on 8-2 to 1450 on 8-2
Parameters								
Alkalinity								
pH	7.20	7.6	7.5	7.05	7.22	7.40	6.95	7.05
Temperature	6.0		13	13			20	22
Specific Conductance	1750	2500	3300	2950		2050	1850	1600
Calcium	190	279	366	325	357[b]	565,596[b]	264	288
Magnesium	11	21	26	29	27	29	19	21
Sodium	153	273	336	231	248	245	139	150
Potassium	6	10	15	13	14	16	14	12
Copper	.020	.021	.044	.14[b]	.090	.141	.25	.28
Nickel	.445	.780	.835	5.100	6.7[b]	21.5	9.22[b]	11.34[b]
Iron	.17[a]	.02	.03	.02	.04	.02	.04	.05
Manganese	.27	.89	1.40	1.29	1.67	2.40	1.94	2.22
Cobalt	.045	.136	.160	.195	.81	1.9,2.42	1.22,1.07[b]	1.39,1.28[b]
Zinc		.096	.174	.257	.230	.396	.579	.420
Sulfate				1800	1300	1500	700	850
Chloride				78	50	59	25	25
Nitrate					39	54	43	90
Ammonia					13	12	4.6	6.8
Total Phosphorus					< 0.1	< 0.1	.4	< 0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	trickle		192	370				
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite a = Value appears anomalous b = reanalysis

HL-3

Date Sample type Time	4-26 grab	5-25 Comp.	6-16 grab 1115	7-5 grab 0800	7-13 Comp. 1150 on 7-12 to 0830 on 7-13	7-19 Comp. 0900 on 7-18 to 1400 on 7-19	8-2 Comp. 8-1 1700 to 0900 on 8-2	8-2 Comp. 0900 on 8-2 to 1450 on 8-2
4. Rain		0.29"	Last rain 6-10 0.10" 2300-2400	Last rain 7-1,7-2 1.36" 1400 on 7-1 to 1100 on 7-2	0.77" from 1400-1600 and 2000-2300 on 7-12	1.51" from 0001-0200 on 7-18; 0.24" on 7-19 1400	Last rain 8-1 0.95"	Last rain 8-1 0.95"
5. Portion of hydrograph		Comp. over rain 5-25	Low flow	Tail of 1.36" rain	Peak	Tail of 1.51" rain on 7-18; First of the present drip compositors	Comp. over rain 8-1	Tail of 0.95" rain
6. Other								

FL-5

Date Sample type Time	8-4 grab 1000	8-10 grab* 1320	8-16 grab* 0945	8-16 grab 1330	8-16 grab 1700	8-17 grab 0815	8-18 grab 1300	8-23 Comp. 1100 on 8-17 to 0910 on 8-23
Parameters								
Alkalinity		7		6	10	5	4	
pH	6.30	5.50	6.10	5.80	5.80	5.25	4.85	4.45
Temperature	22	22	15	22	16		16	
Specific Conductance	2350	4000	1150	1750	1575	2100	3100	1750
Calcium	483	636	202	264	270	400[b]	350	200
Magnesium	32	47	20	27	31	40[b]	44[b]	23
Sodium	238	338	97	111	106	178[b]	354	112
Potassium	17	20	9	11	9	33	8	9
Copper	.7,.13[b]	1.28	.58[a]	1.30	1.59	2.53	2.98	1.10
Nickel	3.23[b]	3.31[b]	15.26[b]	24.07[b]	28.03[b]	40.38[b]	42.21	15.98
Iron	.05	.12	3.38	9.73	10.78	11.54	5.28	.26[b]
Manganese	3.32	4.29	2.18	3.15	3.85	5.14	5.87	2.58
Cobalt	1.77,1.94	1.96,2.43[b]	2.63	4.05	4.69	6.73	7.22	2.86
Zinc	.774	1.18,1.25[b]	.42	.66	.79	1.24	1.45	.59
Sulfate	1400	2000		900	800	800	800	1400
Chloride	44	52		19	20	27	54	17
Nitrate	83	-		38	44	33	67	33
Ammonia	7.5	8.5		4.0	-	-	-	4.0
Total Phosphorus	< 0.1	< 0.1		< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	333	147	11,356[x]	3785	2704	1714[x]	498	
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite

* mean of duplicate samples x Backwater suspected

a = Value appears anomalous b = reanalysis

FL-5

Date Sample type Time	8-4 grab	8-10 grab*	8-16 grab*	8-16 grab	8-16 grab	8-17 grab	8-18 grab	8-23 Comp. 1100 on 8- to 0910 on 8-
4. Rain	Last rain 8-1 0.95"	Last rain 8-1 0.95" 0.13" on 8-10 1100-1200	1.28" from 0400-0800 on 8-16 also: 0.60" from 2030-2400 on 8-14 0.26" from 0001-0100 and 1000-0100 and 1000-1130			Last rain 8-16 1.28" 0400-0800	0.23" 0001-0100	.23" on 8- 0001-0100 .60" on 8- 2000-2400 1.15" on 8- 0001-1100
5. Portion of hydrograph	Tail of 0.95" rain	Low flow	Peak	After peak	After peak	Tail of 1.28" rain	Tail of 0.23" rain	Before peak
6. Other								

FL-5

Date Sample type Time	8-23 Comp. 0910 on 8-23 to 1930 on 8-23	8-25 Comp. 1930 on 8-23 to 0835 on 8-25	8-28 Comp. 0835 on 8-25 to 0815 on 8-28	9-11 grab 1145	9-12 Comp. 1200 on 9-11 to 1455 on 9-12		9-15 grab 1100	
Parameters								
Alkalinity								
pH	4.00	3.90	4.25	4.80	3.50		3.62	
Temperature		21		17			13	
Specific Conductance	1800	2800	3250	1475	2650		3250	
Calcium	207	344	395	140	260		312	
Magnesium	26	42	50	20	50		65	
Sodium	102	177	220	79	133		183	
Potassium	11	15	18	9	17		21	
Copper	1.80	3.15	3.95	1.81	7.28		10.30	
Nickel	24.5	39.40	48.00	18.00	67.00		83.00	
Iron	7.75	2.78	2.38	.75	24.00[b]		25.00	
Manganese	3.40	5.40	6.47	2.90	10.00		15.00	
Cobalt	3.94	6.36	7.76	1.51	4.88		6.17	
Zinc	.66	1.18	1.49	.56	1.75		2.46	
Sulfate	700	1200	1500	700	1550		1900	
Chloride	16	22	30	8	9		15	
Nitrate	34	54	64	-	-		46	
Ammonia	-	8.3	-	-	-		0.1	
Total Phosphorus	< 0.1	< 0.1	< 0.1	0.2	< 0.1			
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)				8801[c]			721	
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis
   c Average value computed from flow meter readings over time interval 0945-1345

FL-5

Date Sample type Time	8-23 Comp. 0910 on 8-23 to 1930 on 3-23	8-23 Comp. 1830 on 8-23 to 0835 on 8-25	8-28 Comp. 0835 on 8-25 to 0815 on 8-28	9-11 grab 1145	9-12 Comp. 1200 on 9-11 to 1455 on 9-12	9-13 grab 1100	9-13 grab 1100
4. Rain	.23" on 8-18 0001-0100 .60" on 8-22 2000-2400 1.15" on 8-23 0001-1100	Last rain 8-22,23 1.75" 2000 on 8-22 to 1100 on 8-23	Last rain 8-27 0.20" 1200-1400	1.31" 0830-1200 Also: 1" from 1200-1530 9-10	Last rain 2.46" total 1.00" 1200-1530 1.31" 0830-1200 0.15" 0300-1000	9-10,11,12	9-10 9-11 9-12
5. Portion of hydrograph	Peak and after peak	Tail of 1.75" rain	Comp. over rain 8.27	Before Peak	Peak and after peak	Tail of 2.46" rain	Tail of 2.46" rain
6. Other						Total metals	Filtered metals split with Amax

FL-6 Table XXV Water Quality Data, FL6, 1978

Date Sample type Time	4-26 grab	5-25 Comp.	6-16 grab 1120	7-5 grab 0820	7-13 Comp. 1145 on 7-12 to 0850 on 7-13	7-19 Comp. 1530 on 7-14 to 0900 on 7-19	8-2 Comp. Start of rain to 1455 on 8-2	8-4 grab 0930
Parameters								
Alkalinity								
pH	7.25	7.25	7.12	7.4	7.49	8.1	7.70	7.75
Temperature	4.0		15.0	12		18	25	22
Specific Conductance	>5000		8860	4610		5700	2600	4150
Calcium	478	840	1028	455[b]	794	540	258	451
Magnesium	139	152	206	141[b]	124	93	61	120
Sodium	535	862	1106	491[b]	685	508	260	
Potassium	12	18	22	17[b]	16	12	9	12
Copper	.088	.121	.258	.060	.118	.092	.06	.11,.13[b]
Nickel	.540	.740	.665	.65,.82[b]	1.03[b]	1.0,.72[b]	.63	.93
Iron	.28	.12	.12	.02	.08	.05	.05	.04
Manganese	.60	1.14	1.64	.70	.95	.75	.62	1.04
Cobalt	.030	.062	.059	.051	.067	.052	.035	.17,.11[b]
Zinc	.022	.003	.033[a]	.006	.014	.006	.020	.030
Sulfate				2000	1450	1300	950	1700
Chloride				420	607	312	180	320
Nitrate					400	160	48	210
Ammonia					100	100	70	100
Total Phosphorus					<0.1	1.9	0.1	0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	Trickle		135	223				287
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = °C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite
4. a = Value appears anomalous b = reanalysis

FL-6

Date Sample type Time	4-26 grab	5-25 Comp.	6-16 grab 1120	7-5 grab 0800	7-13 Comp. 1145 on 7-12 to 0850 on 7-13	7-19 Comp. 1550 on 7-14 to 0900 on 7-19	8-2 Comp. Start of rain to 1435 on 8-2	8-4 grab 0930
4. Rain		0.29"	Last rain 6-10 0.10" 2300-2400	Last rain 7-1,7-2 1.36" 1400 on 7-1 to 1100 on 7-2	0.77 from 1400-1600 and 2000-2300 on 7-12	1.51" from 0001-0200 on 7-18; 0.24" on 7-19 (time unknown)	Last rain 8-1 0.95" Comp. over rain 8-1	Last rain 8-1 0.95" Tail of 0.95" rai
5. Portion of Hydrograph		Comp. over rain 5-25	Low flow	Tail of 1.36" rain	Peak	Before peak and peak		
6. Other								

H- 6

Date Sample type Time	8-10 Grab 1315	8-16 Grab 1000	8-16 Comp. Start of rain to 1440 on 8-16	8-17 Comp. 1440 on 8-16 to 1405 on 8-17	8-23 Comp. 1405 on 8-17 to 1935 on 8-23	8-25 Comp. 1935 on 8-23 to 0840 on 8-25	8-28 Grab 0840 on 8-25 to 0815 on 8-28	9-13 Grab 1600
Parameters								
Alkalinity	47		40	46	30	40	57	25
pH	7.55	7.85	7.71	7.52	7.55	7.40	7.85	7.1
Temperature	24	22				23		13
Specific Conductance	6840	2200	2250	4600	4320	4750	5290	3400
Calcium	748	231	269	396	368	373	481	367
Magnesium	178	70	78	46	112	119	135	125
Sodium	700	236	278	200	396	432	681	395
Potassium	22	6	6	14	12	12	15	14
Copper	.17	.09	.10	.13	.13	.15	.18	.28
Nickel	1.43	.87	1.00	1.48	1.54	1.70	2.15	2.25
Iron	.15	.03	.08	.05	.05	.07	.07	.06
Manganese	1.68	.75	.89	1.38	1.40	1.53	1.82	2.12
Cobalt	.28	.14	.18	.27	.24	.27	.39	.21
Zinc	.050	.03	.03	.05	.05	.05	.07	.12
Sulfate	2400		1000	1700	1600	1800	2150	2050
Chloride	600		150	190	280	220	390	180
Nitrate	-		140	120	170	140	230	-
Ammonia	160		-	-	-	98	-	-
Total Phosphorus	< 0.1		< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	0.1
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)	77	28390						667 (Q at 1135)
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = ^{o}C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite

Comments								
4. Rain	Last rain 8-1 0.95" 0.13" on 8-10 1100-1200	1.28" 0400-0800 .60" on 8-14 .26" on 8-15	1.28" 0400-0800 .60" on 8-14 .26" on 8-15	Last rain 8-16 1.28" 0400-0800	Last rain 8-22, 8-23 1.75" 2000 on 8-22 to 1100 on 8-23	Last rain 8-22, 8-23 1.75" 2000 on 8-22 to 1100 on 8-23	Last rain 8-27 0.20" 1200-1400	Last rain 9-10,11,12 2.46" total ~1.00" 1200-1530 9-10 ~1.31" 0830-1200 9-11 ~15" 0300-1000 9-12
5. Portion of hydrograph	Low flow	Peak	Comp. over rain 8-16	Tail of 1.28" rain	Comp. over rain 8-22, 8-23	Tail of 1.75" rain	Comp. over rain 8-27	Tail of 2.46" rain
6. Other								

FL- 6

Date Sample type Time	9-13 Comp. 9-10 rain to 1135 on 9-13							
Parameters								
Alkalinity	26							
pH	7.21							
Temperature								
Specific Conductance								
Calcium	408							
Magnesium	115							
Sodium	498							
Potassium	14							
Copper	.29							
Nickel	2.42							
Iron	.04							
Manganese	2.16							
Cobalt	.23							
Zinc	.12							
Sulfate	2100							
Chloride	360							
Nitrate	-							
Ammonia	-							
Total Phosphorus	0.1							
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)								
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = ^{o}C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite

Comments

4. Rain	Last rain 9-10,11, 12 2.46' total ~1.00" 1200-1530 9-10 ~1.31" 0830-1200 9-11 ~.15" 0300-1000 9-12						
5. Portion of hydrograph	Comp. over rain 9-10,11, 9-12						
6. Other							

Appendix IV

COMPUTATION OF THE PERCENT OF METALS THAT ARE RETAINED IN THE TEST PILES:

Assumptions: SO_4 is completely mobile and it is a measure of the total amount of sulfide dissolution that has occured. The goal is to compute the total sulfide dissolution rate and based on the percent of each metal sulfide in the pile, calculate the amount of each metal sulfide that has reacted. The inherent assumption is that all the sulfides are oxidized at the same rate.

The 1977 data from FL1 are used. Chemical data from Amax, as well as detailed chemical information from the regional copper nickel study are available (Tables I and III).

Filtered data was used for the metals, so that the influence of particulates could be eliminated. Since the piles are small, have fine material in them, and the samples are collected directly from the pile, fine particles may be in suspension. These particles could tend to bias the results to unusually high values.

It is assumed that all iron in solution comes from the sulfide minerals (Iron exists in the silicate minerals as well).

The calculations are shown in Table XXVI. The first column is the composition of the rock as given in Table III. The second column (Mole % of s) gives the distribution of each metal sulfide, i.e. 6.08 percent of the total sulfides contain nickel. Values in Column III were calculated by assuming all the sulfides are present as simple sulfides, i.e. NiS, CuS etc. and the amount of dissolution that would occur to yield a SO_4 concentration of 1000 mg/l was calculated.

Table XXVI Calculation of Metals Retained in Test Pile

Parameter	% determined from sample Ax 9001	Mole % of S	Concentration at base 1000 mg/l SO_4	At Base of 610 mg/l	Observed Median Concentration mg/l
$S-SO_4$	.655	100	1000	610	610
Ni	.073	6.08	37.2	22.7	.288
Cu	.306	23.6	156	95.2	.03
Zn	.016	1.20	8.18	5.0	.013
Co	.009	.750	4.60	2.8	.006
Fe(s)*	.697	61.3	356	217	<.05

* This is the iron that is in the sulfide minerals only, it does not include the iron that is in the silicates.

Rainfall Data

Table XXVII

April, 1978

PRECIPITATION AT MINNAMAX

Date	Net ppt (Inches)	Time of Rain	Remarks
1	.00		
2	.02		
3	.20		
4	.00		
5	.00		
6	.47		
7	.00		
8	.00		
9	.31		
10	.00		
11	.15		
12	.00		
13	.10		
14	.00		
15	.00		
16	.00		
17	.00		
18	.00		
19	.00		
20	.00		
21	.00		
22	.00		
23	.00		
24	.03		
25	.00		
26	.00		
27	.00		
28	.00		
29	.00		
30	.00		

From 4/1 - 5/2, rain was recorded from A.M. to A.M. with a standard rain gauge; rain during weekends would be added to the Monday rain figures.

From 5/2 on, all times midnight to midnight.

Table XXVIII May, 1978

PRECIPITATION AT MINNAMAX

Date	Net ppt (Inches)	Time of Rain	Remarks
1	.00		
2	.00		
3	.00		
4	.00		
5	.00		
6	.00		
7	.32	0300-0815	
8	.00		
9	.00		
10	.00		
11	.00		
12	.00		
13	.00		
14	.00		
15	.00		
16	.00		
17	.00		
18	.00		
19	.21	1830-2115	
20	.00		
21	.00		
22	.00		
23	.00		
24	.29	0630-0730	
25	.00		
26	.25	0500-0600	Quantity is an estimate.
27	.10	0800-0900	Quantity is an estimate.
28	1.92	0700-0830=1.86;1300-1400=.06	0700-0830 is an estimate.
29	.08	1215-1530	
30	.20	1830-2400	
31			

From 4/1 - 5/2, rain was recorded from A.M. to A.M. with a standard rain gauge; rain during weekends would be added to the Monday rain figures.

From 5/2 on, all times midnight to midnight.

Table XXIX

June, 1978

PRECIPITATION AT MINNAMAX

Date	Net ppt (Inches)	Time of Rain	Remarks
1	.48	2400-0300=.30; 1815-1845=.18	
2	.02		Not traceable on the recorder
3	.00		
4	.00		
5	.00		
6	.75	1730-2400	
7	.21	0001-0900	
8	.00		
9	Trace		
10	.10	2300-2400	
11	.00		
12	.00		
13	.00		
14	.00		
15	.00		
16	.00		
17	.00		
18	.00		
19	.00		
20	.00		
21	.00		
22	.00		
23	.35	0745-1045=.29;1915-2000=.06	
24	.00		
25	.06	1800-1830	
26	.11	1730-2030	
27	.00		
28	.13	1630-1900	
29	.00		
30	.00		

From 5/2 on, all times midnight to midnight.

Table XXX

July, 1978

PRECIPITATION AT MINNAMAX

Date	Net ppt (Inches)	Time of Rain	Remarks
1	.12	1400-2400	
2	1.24	0001-1100	
3	.00		
4	.00		
5	.00		
6	.00		
7	1.38	0600-1100=1.18"; 1200-1400=.20"	
8	.09	2300-2400 heavy	
9	.06	2400-0200 light	
10	.00		
11	.00		
12	.77	1400-1600, 2000-2300	
13	.00		
14	.00		
15	.00		
16	.20		
17	.00		
18	1.51	midnight-0200, heavy	
19	.24		
20	.00		
21	.01		
22	.29	1500-1600	
23	.00		
24	.26	1300-1600	
25	.00		
26	.10	2100-2200	
27	.00		
28	.11	2000-2100	
29	.00		
30	.00		
31	.00		

From 5/2 on, all times midnight to midnight.

Table XXXI

August, 1978

PRECIPITATION AT MINNAMAX

Date	Net ppt (Inches)	Time of Rain	Remarks
1	.95		
2	.00		
3	.00		
4	.00		
5	.00		
6	.00		
7	.00		
8	.00		
9	.00		
10	.13	1100-1200	
11	.00		
12	.13	0600-0630	
13	.00		
14	.60	2030-2400	
15	.26	2400-0100, 1000-1130	
16	1.28	0400-0800	
17	.00		
18	.23	2400-0100	
19	.00		
20	.00		
21	.00		
22	.60	2000-2400	
23	1.15	0001-1100	
24	.00		
25	.00		
26	.00		
27	.20	1200-1400	
28	.00		
29	.00		
30	.00		
31	.00		

From 5/2 on, all times midnight to midnight.

Table XXXII September, 1978

PRECIPITATION AT MINNAMAX

Date	Net ppt (Inches)	Time of Rain	Remarks
1	.00		
2	.00		
3	.00		
4	.00		
5	.00		
6	.00		
7	.00		
8	.00		
9	.00		
10	1.00	1200-1530	quantity is an estimate
11	1.31	0830-1200	quantity is an estimate
12	.15	0300-1000	quantity is an estimate; Exact total 9/10-9/12 = 2.46"
13	.00		
14	.53	0001-0600	
15	.03	0810-1200	
16	.00		
17	.00		
18	.09		
19	.06	1700-1800	
20	.00		
21	.00		
22	.00		
23	.00		
24	.00		
25	.00		
26	.03	1500-1700	
27	.00		
28	.08	1900-2000	
29	.00		
30	.00		

From 5/2 on, all times midnight to midnight.

Table XXXIII

October, 1978

PRECIPITATION AT MINNAMAX

Date	Net ppt (Inches)	Time of Rain	Remarks
1	.05"	2300-2400	
2	.17"	0001-0100=.02"; 0200-0400=.15"	
3	.00		
4	.00		
5	.32	0400-1700, drizzle	
6	.00		
7	.00		
8	.00		
9	.00		
10	.00		
11	.04	1200-1300	
12	.00		
13	.00		
14	.00		
15	.40	0800-1100	
16	.00		
17	.00		
18	.00		
19	.00		
20	.00		
21	.00		
22	.00		
23	.00		
24	.00		
25	.00		
26	.02	0800-1200	
27	.00		
28	.05	0600-1030	quantity is an estimate
29	.00		
30	.00		
31	.00		

From 5/2 on, all times midnight to midnight.

Table XXXIV November, 1978

PRECIPITATION AT MINNAMAX

Date	Net ppt (Inches)	Time of Rain	Remarks
1	.00		
2	.00		
3	.00		
4	.00		
5	.00		
6	.00		
7	.00		
8	.00		
9	.00		
10	.15	0630-1800	Snow; quantity is an estimate
11	.00		
12	.00		

From 5/2 on, all times midnight to midnight.

Appendix VI Samples Used In Box Plot

Date	1	2	3	4	5	6
4-26	grab	grab	grab	grab	grab	grab
6-16	grab at 1025	grab at 1120	grab at 1205	grab at 0800	grab at 1115	grab at 1120
7-5	grab at 0830	grab at 0830	grab at 0845	grab from 7-6	grab at 0800	grab at 0800
7-13	composite from 1200 on 7-12 to 0840 on 7-13	composite 1155 on 7-12 to 0840 on 7-13	composite from 1210 on 7-12 to 0840 on 7-13	grab at 1130 on 7-13	composite 1150 on 7-12 to 0830 on 7-13	composite from 1145 on 7-12 to 0830 on 7-13
7-18	composite from 1530 on 7-14 to 1400 on 7-19	composite 0810 on 7-18 to 1400 on 7-19	composite 1530 on 7-14 to 1400 on 7-19	no sample	composite 0900 on 7-18 to 1400 on 7-19	composite 1530 on 7-14 to 0900 on 7-19
8-2	composite 1700 on 8-1 to 1515 on 8-2	composite 1415 on 7-24 to 0930 on 8-2	composite 1800 on 8-1 to 1515 on 8-2	no sample	composite mean of two samples 1-1700 on 8-1 to 0900 on 8-2 2-0900 on 8-2 to 1450 on 8-2	composite
8-4	grab at 1045	grab at 1030	grab at 1100	grab at 1010	grab at 1000	grab at 0930
8-16	grab at 1015	grab at 1010	grab at 1025	composite	mean of 3 grabs, at 0945, 1330 1700	grab at 1000
8-23	composite 1025 on 8-17 to 1945 on 8-23	composite 1000 on 8-17 to 1940 on 8-23	composite 1050 on 8-17 to 1955 on 8-23	combined composite 1-0830, 8-21 to 0800, 8-23 2-0800, 8-23 to 1440, 8-23	composite 1100 8-17 to 0910, 8-23	composite 1405, 8-17 to 1935, 8-23
9-13	grab at 1020	grab at 1045	grab at 1030	grab at 1100	grab at 1100	grab at 1135

Appendix VII

QUALITY ASSURANCE

Duplicate, Blank Samples, Intralaboratory Splits

Six percent of all the water quality samples were masked duplicates. The data for the duplicate samples are given in Table XXXV. The results are summarized in Table XXXVI.

Distilled deionized water blanks were also analyzed as part of the quality assurance program. Table XXXVII summarizes the results.

Partial results for the intralaboratory study are given in Table XXXVIII. Except for zinc, the agreement is satisfactory.

HYPALON TESTS

The contamination potential of the Hypalon liner was evaluated in two tests. Runoff samples were collected from the tailings plot area prior to the deposition of tailings. In the second test a piece of Hypalon was soaked in distilled deionized water for 4 weeks. Some low level zinc contamination was observed (Table XXXIX).

PUMP TESTS

Distilled deionized water was passed through the compositing pumps.

The results shown in Table XL indicate some low level zinc contamination.

Duplicate Samples

Table XXXV

Date	7-7	7-7	7-8	7-8	8-28	8-28
Sample type	grab	grab	grab	grab	Composite	Composite
Sample No.	73	74	93	94	181	184
Time	1900	1900	2300	2300	0835 on 8-25	to 0815 on 8-28
	FL-4	FL-4	FL-4	FL-4	FL-4	FL-4
Parameters						
Alkalinity					41.8	41.8
pH	7.50	7.50	7.40	7.40	7.55	7.55
Temperature	11	11				
Specific Conductance	1150	1150	1700	1700	2350	2350
Calcium	129	127	211	212	287	274
Magnesium	12	12	21	21	30	28
Sodium	93	90	152	153	180	194
Potassium	348	111	43	42	57	64
Copper	.02	.01	.02	.02	.03	.03
Nickel	.15	.12	.25	.26	.33	.34
Iron	.02	.01	.03	.01	.01	.03
Manganese	.09	.10	.16	.16	.19	.19
Cobalt	.02	0	.02	.04	.03	.06
Zinc	.05	.05	.07	.07	.09	.10

Duplicate Samples

Date	8-16	8-16			6-14	6-14
Sample type	grab	grab			grab	grab
Sample number	135	138			14	15
Time	0945	0945			0745	0745
Site	FL-5	FL-5			FL-4	FL-4
Parameters						
Alkalinity						
pH	6.10	6.10			7.60	7.60
Temperature	15	15			9.0	9.0
Specific Conductance	1150	1150			2500	2500
Calcium	203	201			222	234
Magnesium	21	20			28	29
Sodium	98	96			256	267
Potassium	9	9			36	38
Copper	.59	.57			.025	.023
Nickel	15.26	15.25			.220	.219
Iron	3.44	3.31			.03	.05
Manganese	2.17	2.18			.21	.21
Cobalt	2.61	2.65			.016	.016
Zinc	.42	.42			.130	.125

Date	6-23	6-23	6-23	6-23	7-7	7-7
Sample type	grab	grab	grab	grab	grab	grab
Sample No.	31	32	41	42	56	57
Time	1540	1540	2100	2100	1200	1200
Site	FL-4	FL-4	FL-4	FL-4	FL-4	FL-4
Parameters						
Alkalinity						
pH	7.45	7.45	7.75	7.75	7.40	7.40
Temperature	14	14			11	11
Specific Conductance	2650	2650	2500	2500	1125	1125
Calcium	232	214	215	217	126	125
Magnesium	32	28	30	30	13	13
Sodium	310	283	292	291	110	111
Potassium	44	40	41	44	576	638
Copper	0?	.024	.020	.019	.02	.01
Nickel	.336	.314	.314	.325	.14	.14
Iron	.01	.02	.01	0	.04	.01
Manganese	.23	.21	.22	.22	.08	.08
Cobalt	.027	.025	.022	.021	.02	.03
Zinc	.169	.134	.153	.154	.04	.04

FL-

Duplicate Samples

Date	8-24	8-24	9-13	9-13	8-4	8-4	8-10	8-10
Sample type	Comp.	Comp.	Grab	Grab	Grab	Grab	Grab	Grab
Sample #	166	168	191	193	121	122	128	130
Time	1025 on 8-17 to 1945 on 8-23		1500	1500	1100	1100	1320	1320
Site	FL-1	FL-1	FL-2	FL-2	FL-3	FL-3	FL-5	FL-5
Parameters								
Alkalinity	50	50	41	41			7	7
pH	8.10	8.10	7.45	7.45	7.65	7.65	5.50	5.50
Temperature			15	15	22	22		22
Specific Conductance	2600	2600	1725	1725	950	950	4000	4000
Calcium	305	300	181	182	156	157	634	638
Magnesium	26	25	21	22	23	25	43	51
Sodium	243	238	139	137	88	88	336	339
Potassium	41	39	30	29	26	26	20	21
Copper	.04	.04	.10	.11	.08	.08	1.28	1.27
Nickel	.38	.35	.27	.25	.33	.32	3.31	3.31
Iron	.05	.06	.03	.02	.01	.04	.15	.08
Manganese	.14	.14	.16	.16	.16	.16	4.32	4.26
Cobalt	.05	.02	.03	.02	.023	.017	1.96	1.96
Zinc	.02	.01	.06	.06	.044	.044	1184	1180
Sulfate								
Chloride								
Nitrate								
Ammonia								
Total Phosphorus								
Dissolved Organic Carbon								
Discharge at time of sample (ml/min)								
Total Volume over composite (liters)								

1. All parameters in mg/l except as follows.
   a. Temperature = ^{o}C
   b. Specific Conductance = umho
   c. Nitrates and ammonia are mg/l as nitrogen, total phosphorus is mg/l as phosphorus and dissolved organic carbon is mg/l as carbon.
2. All metal values are filtered, unless otherwise specified.
3. Comp. = Composite

Table XXXVI QUALITY ASSURANCE SUMMARY

$$\% \text{ error} = \frac{X_1 - X_2}{\frac{(X_1 + X_2)}{2}} \times 100\%$$

Parameter	Number of Samples	Median	Maximum	Minimum
Ca	12	.96	8.1	0
Mg	12	4.4	13.3	0
Na	12	2.1	9.8	0
K	12	5.1	19.2	0
Cu	12	2.9	66.6	0
Ni	12	3.05	22	0
Zn	12	0.17	66.6	0
Co	12	3.0	200	0
Fe	12	66.3	200	0
Mn	12	0	10.5	0

Table XXXVII

BLANKS

Parameter	High mg/l	Low mg/l	Median	Mean	σ
Co	.02	0	0	.001	.005
Cu	1.01	0	0	.067	.252
Ni	0	0	0	0	0
Zn	0	0	0	0	0
Ca	31	0	0	2.07	7.73
Fe	.01	0	0	.001	.002
K	3	0	0	.200	.56
Mg	10	0	0	.667	2.49
Mn	0	0	0	0	0
Na	34	0	0	2.2	8.50
			Blanks without #172		
Co	.02	0	0	.0014	.004
Cu	0	0	0	0	0
Ni	0	0	0	0	0
Zn	0	0	0	0	0
Ca	.06	0	0	.0043	.015
Fe	0	0	0	0	0
K	0	0	0	0	0
Mg	.01	0	0	.0007	.001
Mn	0	0	0	0	0
Na	0	0	0	0	0

15 Samples

Table XXXVIII INTRALABORATORY QUALITY ASSURANCE

Date		Co	Cu	Ni	Zn	Ca	Fe	K	Mg	Mn	Na
		ug/l				Mg/l					
	Eml										
9/11	DNR	20	10	550	910	91	.03	5	72	.04	44
	Eml										
	Serco	1	8.0	450	23	115	.05	4.1	81	.03	44
	Eml										
	Erie										
% Diff. Sero											
	DNR	+100	20	18	+100	26	40	18	11	25	0
	FL5										
9/13	DNR	6170	10300	83000	2460	312	23	21	65	13	183
	FL5										
	Serco	7500	9500	78000	2480	380	20	20	76	12	200
% Diff.		18	7.7	6.0	.81	18	13	4.8	14	7.7	8.5
	FL4										
7/13	Serco	12	15	150	64					.12	
	FL4										
	DNR	15	12		4					.14	
% Diff.		20	20		100+					14	

Table XXXIX

HYPALON CONTAMINATION TESTS

Parameters	Rainwater Runoff From Hypalon-covered Pad		1.4 cm by 1.7 cm Hypalon Section immersed in water for 28 days
Calcium	< .02	< .02	.70
Magnesium	< .01	< .01	.71
Sodium	< .01	< .01	.94
Potassium	< .01	< .01	.07
Copper	< .01	< .01	< .01
Nickel	< .02	< .02	< .02
Iron	< .02	< .02	< .02
Manganese	< .02	< .02	< .02
Cobalt	< .01	< .01	< .01
Zinc	< .01	< .01	.02

Table XL

PUMP CONTAMINATION TEST

September 29, 1978

Parameters	FL1	FL2	FL3	FL5	FL6	Blank*
Calcium	< .02	< .02	< .02	< .02	< .02	< .02
Magnesium	< .01	< .01	< .01	< .01	< .01	< .01
Sodium	< .01	< .01	< .01	< .01	< .01	< .01
Potassium	< .01	< .01	< .01	< .01	< .01	.01
Copper	.02	.02	< .01	< .01	< .01	< .01
Nickel	< .02	< .02	< .02	< .02	< .02	< .01
Iron	< .02	< .02	< .02	.10	< .02	< .02
Manganese	< .02	< .02	< .02	< .02	< .02	< .02
Cobalt	< .010	< .010	< .01	< .01	< .01	< .01
Zinc	.01	.02	.02	.02	.02	.01

*Distilled, deionized water

All values in mg/l

Appendix VIII Timeline

1978 SAMPLING SEASON - GENERAL EVENTS

APRIL		
	18	All leach piles unfrozen
MAY		
	11	FL-2, 3, and 5 covered with soil
JUNE		
	22	Fertilized soil-covered leach piles
	23	Intensive leachate sampling of FL-4, .66 cm (.26") rain
	27	Seeded soil-covered leach piles
	28	Mulch applied to soil-covered leach piles
JULY		
	7	Intensive leachate sampling of FL-4, 3.88 cm (1.53") rain
	11	Compositing Drip Samplers installed
AUGUST		
	11	Pressurized Flow Meter/Recorder/Compositor (Flow Proportional Sampler) Installed at FL-4
SEPTEMBER		
OCTOBER		
NOVEMBER		
	3	Flow proportional sampler installed at FL-1,2,3, and 7
	10	Freeze-up
DECEMBER		

1977 - 1978

DETAILED EVENTS

4/20/77	FL-1, 2, 3, and 4 Constructed.
9/10/77	FL-5 Constructed.
9/30/77	FL-6 Constructed
4/18/78	All piles unfrozen.
4/26/78	First Sampling, 1978
5/02/78	Recording rain gauge started.
5/11/78	FL-3, 5, and 2 covered with soil.
5/24/78	Flow meters attached to pipelines.
5/25/78	Hand-made flow weighted composite FL-4.
6/13/78	Lowered FL-1 discharge pipe and adjusted flow meters to have more hydraulic head.
6/22/78	Fertilized soil-covered piles with 16,000 g/pile.
6/23/78	Intensive sampling FL-4 .89 cm (.35 inches) rain, also hydrograph obtained from FL-1.
6/27/78	Soil-covered piles seeded.
6/28/78	Soil-covered piles mulched.
7/07/78	Intensive sampling FL-4 3.88 cm (1.53 inches) rain, also hydrographs from FL-1, 2, 3, 4, 5, and 6.
7/11/78	Compositing drip samplers installed.
7/14/78	Filter screens placed above all flow meters.
8/11/78	Pressurized flow meter and flow Proportional Sampler Samplers) attached to FL-4; also Rustrack recorder attached
8/18/78	Pipe between Fl-3 and FL-1 lowered.
9/21/78	First hard frosts.
11/03/78	Flow Proportional Sampler installed at FL-1,2,5, and 7
11/10/78	First snow.
11/14/78	Heat tapes put on drains of piles.

Appendix IX

Reanalysis of Anomalous Values

Certain values in the original water quality data appeared anomalous. The anomalous values were reanalyzed, if a sufficient amount of the sample remained. The results of the reanalysis (performed in December) are given in Table XLI.

FL 1

Table XiI Reanalysis

Sample #	Date		Para-meter	original value mg/l		rerun value mg/l		
95	7-11		Ca	419		303		
			Cu	.024		.03		
			Ni	.405		.37		
			Co	.022		.03		
			Zn	.074		.01		
129	8-10		Co	.06		.03		
147	8-16		Co	.06		.02		
166	8-24		Co	.04		.04		
168	8-24		Co	.04		.04		
173	8-25		Co	.05		.03		
180	8-28		Co	.06		.07		
FL 2								
101	7-13		Zn	.008		.09		
108	7-19		Ni	20.5		.24		
			Co	.019		.01		
112	8-2		Co	40		.01		
			Zn	.055		.05		
137	8-16		Cu	.14		.17		
			Co	0		.01		
143	8-16		Cu	.12		.14		
150	8-16		Cu	.13		.14		
169	8-24		Cu	.13		.15		
183	8-20		Co	.05		.04		
			Zn	.09		.09		

FL 3

	Date		Parameter	original value mg/l	rerun mg/l			
3	4-26		Ni	0	.06			
			Fe	.7	.7			
48	7-5		Cu	1.0	.20			
			Ni	.282	.36			
			Fe	0	<.02			
110	7-19		Cu	.074	.11			
			Ni	.140	.33			
131	8-10		Cu	.05	.07			
			Ni	.56	.57			
139	8-16		Cu	.11	.12			
			Ni	.20	.18			
142	8-16		Cu	.11	.13			
			Ni	.24	.17			
149	8-16		Cu	.10	.13			
			Ni	.23	.22			
167	8-24		Co	0	<.02			
FL 4								
44	6-23		Ni	.116	.36			
			Fe	0	.01			
52	7-6		Na	190	173			
60	7-7		Ca	118	242			
			Mg	12	24			
			Na	103	220			
			K	225	43			

FL 4 (cont.)

	Date		Parameter	original value mg/l	rerun mg/l			
61	7-7		K	135	42			
62	7-7		K	1393	20			
63	7-7		K	348	29			
64	7-7		K	1363	no sample	left		
65	7-7		K	356	29			
66	7-7		K	278	33			
1100	7-7		K	268	no sample			
58	7-7		K	295	22			
			Na	243	88			
			Mg	27	10			
			Ca	233	115			
68	7-7		K	148	20			
70	7-7		K	95	22			
71	7-7		K	248	24			
73	7-7		K	230	29			
74	7-7				24			
77	7-7		K	289	28			
			Co	0	2.01			
90	7-8		Ca	168	175			
			Mg	17	16			
			Co	0	.04			
91	7-8		Ca	167	177			
			Mn	.12	.16			
			Co	0	.03			
93	7-8		Ca	212	222			

FL 4 (cont.)

sample #	Date		Para-meter	Original	rerun			
133	8-15		Co	.03	.01			
156	8-21		Co	.06	.04			
			Zn	1.0	.10			
			Fe	.46	missed			
FL 5								
49	7-5		Cu	.90	.14			
103	7-13		Ca	533	357			
			Ni	12.50	6.70			
			Co	1.1	.81			
128	7-19		Ca	565	596			
			Co	1.9	2.42			
			Zn	.396	1.24			
			Ni	22.5	missed			
113	8-2		Ni	2.88	9.22			
			Co	1.22	1.07			
114	8-2		Ni	3.02	11.34			
			Co	1.39	1.28			
124	8-4		Cu	.7	.13			
			Ni	3.23	3.23			
			Co	1.77	1.94			
128	8-10		Ni	3.31	3.31			
130	8-10		Co	1.96	2.43			
			Zn	1.18	1.25			
135	8-16		Ni	15.26	15.26			

FL 5 (cont)

sample #	Date		Parameter	original	rerun			
	8-16		Ni	24.07	24.07			
148	8-16		Ni	28.03	28.03			
151	8-17		Ca	165	400			
			Mg	16	40			
			Na	82	178			
			Ni	40.38	40.38			
154	8-18		Mg	115	44			
161	8-23		Fe	.26	.26			
188	9-12		Fe	24.0	24.0			
FL 6								
47	7-5		Ca	309	435			
			Mg	124	140			
			Na	476	491			
			K	10	17			
			Ni	.65	.82			
104	7-13		Ni	.400	1.08			
106	7-19		Ni	1.0	.72			
125	8-4		Cu	.11	.13			
			Co	.17	.11			

Attachment 4

Eger P, Lapakko K (1980) Stockpile Leaching and Chemical Transport at the Erie Mining Company; Dunka Site: A Data Summary for 1976-1979. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN. 117 p.

523-0759741-15

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Stockpile Leaching and Chemical Transport at the Erie Mining Company Dunka Site: A Data Summary for 1976- 1979

Minnesota Department of Natural Resources

Division of Minerals

1980

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Stockpile Leaching and Chemical Transport at the Erie Mining Company Dunka Site: A Data Summary for 1976 - 1979

Paul Eger
Kim Lapakko

MINNESOTA DEPARTMENT OF NATURAL RESOURCES

Division of Minerals

1980

FOREWORD

The purpose of this report is to summarize and present all the data that has been collected from 1976 through 1979. There is essentially no discussion of the data in this report, since detailed discussion of much of the data has already been presented elsewhere or will be presented in reports in preparation. (1-23).

ACKNOWLEDGEMENTS

We would like to thank Erie Mining Company for their cooperation in this study. Their willingness to allow us access to their property, their assistance in the field and their analyses of our many "special" samples have added greatly to the data in this report. Special thanks go to Larry Peterson for his constant complaining, good insights, and help in the field (at least on sunny days).

The data from 1976 - 1977 were collected as part of the Minnesota's Environmental Quality Board's Regional Copper-Nickel Study. Data since 1977 have been collected under a joint DNR-Erie study. Bruce Johnson and Ann Weir have been responsible for our data collection efforts and they have done an outstanding job. All our student workers and laborers have also provided much appreciated assistance. Special thanks also goes to the U. S. Geological Survey,Grand Rapids, for their continued assistance with our many field and data problems.

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1. INTRODUCTION

Water quality and flow have been monitored at the Erie Mining Company Dunka Site from 1976 to present as a joint study involving Erie Mining Company, the Regional Copper-Nickel Study (1976-1977), and the Minnesota Department of Natural Resources (DNR). The objective of this report is to present a brief summary of the data collected from 1976 through 1979. Data continues to be collected under a joint program with Erie and the DNR. The report focuses on the release of trace metals from mining stockpiles and their subsequent transport through Unnamed Creek.

Additional information on this study, and related information regarding regional background data, gabbro leaching, chemical transport, stockpile hydrology, and mitigation is availabe in other references (1-23).

2. SITE DESCRIPTION

The Dunka Mine is an open pit taconite operation. The pit is approximately 4,000 m long, 400 m wide and 110 m deep. The contact between the Duluth gabbro complex and the iron formation occurs in the mine and to remove the underlying taconite, Erie has stockpiled $\sim 20 \times 10^6$ tons of gabbro material along the eastern boundary of its pit (Figure 1). Waste rock stockplies contain less than 0.2% Cu and lean ore stockpiles greater than 0.2% Cu. The stockpiles contain from 4-10 million tons of gabbro and cover from 0.11 to 0.62 km^2 (Table 1). The entire watershed area is 1.14 km^2.

Mine dewatering and runoff from the stockpiles flow into Unnamed Creek and subsequently into Bob Bay on Birch Lake. The mine dewatering discharges are denoted as the 011 and 012 dis-

Figure 1

DUNKA PIT STUDY

STOCKPILE AND SAMPLING SITE LOCATIONS

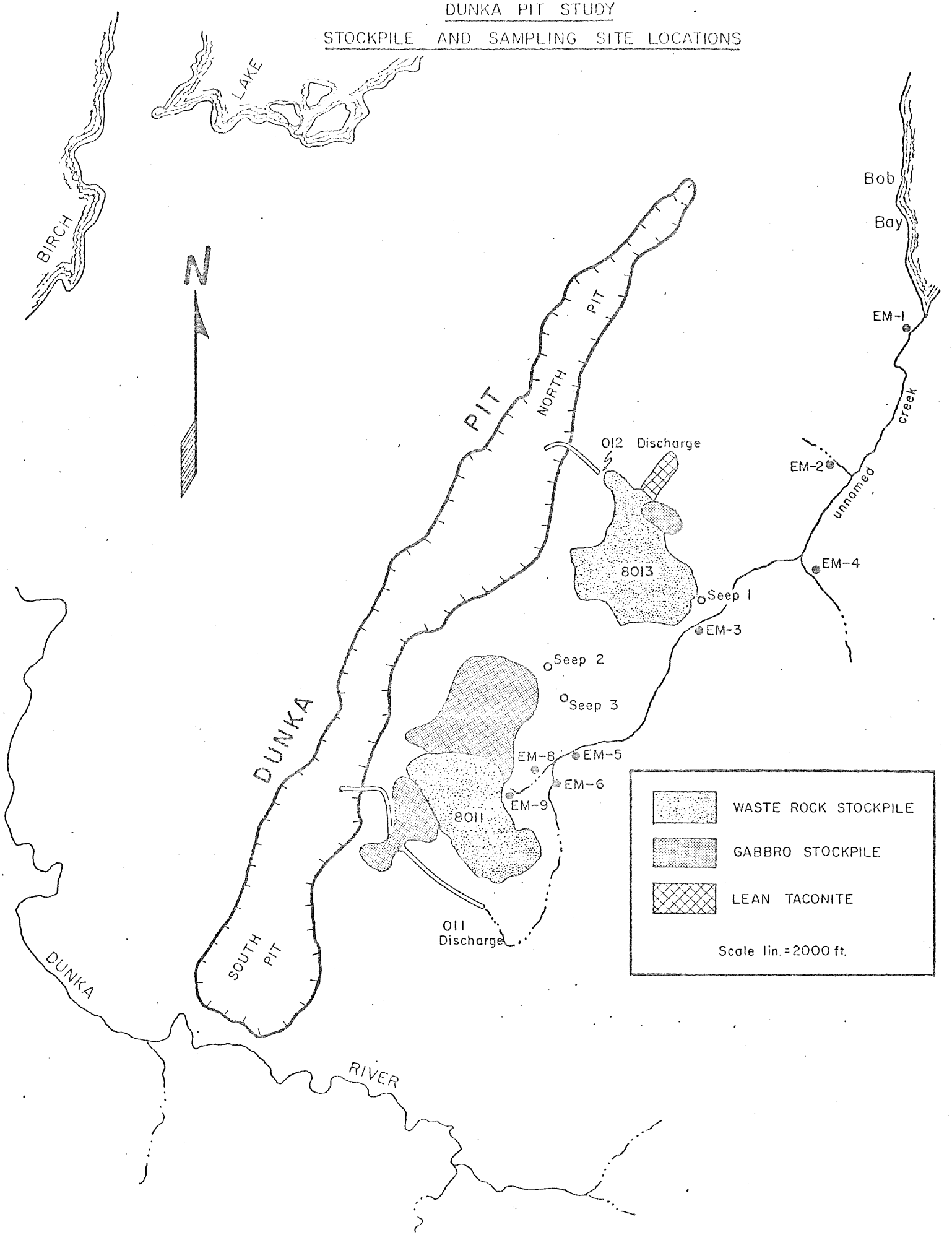


TABLE 1

Stockpile Mass and Composition, Dunka Pit *

Seep	Material	Date Started	Tons Gabbro	Area (km^2)	Composition Cu	Composition Ni
EM-8	Waste Rock	12/1965	9,976,107	0.62	0.057	0.014
Seep 1	Waste Rock	4/1967	5,578,465	0.11	0.043	0.014
Seep 3	Lean Ore	6/1967	4,190,806	0.11	0.29	0.08

* As of January 1977

Charges; waste rock seepages are denoted Em-8 and Seep 1, and lean ore seepage, Seep 3 (Figure 1). The lean ore seepage flows through a white cedar bog prior to its discharge into Unnamed Creek (Figure 1). Two stations, EM-9 and Em-8, were originally monitored, with EM-8 slightly downstream from the point of seepage discharge, Em-9. It was found that the difference in water quality and flow between those two stations was negligible, therefore, the Em-9 station was deleted. Six stream stations (Em-1 - Em-6) were established with numbering increasing with distance upstream.

Eight stations (3-1 to 3-8) were established across the Seep 3 bog where shallow water wells and piezometers were installed. (Figure 2). The shallow water wells consisted of a perforated PVC pipe (D=10.2 cm, 1=46 cm) sealed at the top with duct tape to prevent surface water leakage. The piezometers (1.5 m depth) were constructed of PVC, with a flange and a bentonite seal to prevent leakage. Peat samples were collected throughout the bog and organic bank and clastic sediment samples were collected in Unnamed Creek.

3. METHODS

Mine dewatering (011, 012) was monitored monthly by Erie Mining Company and analyzed for Cu, Ni, SO_4, and pH. Other parameters, such as oil and suspended solids, were also measured but are not discussed in this report.

The three seeps (Em-8, Seep 1, Seep 3) and stream stations were monitored on a biweekly basis. During 1976 and 1977 six stream stations were monitored. In subsequent years, stations Em-2 through Em-6 were deleted as more emphasis was placed on chemical output from the watershed rather than transport within it. Water quality samples were analyzed for trace metals (Cu

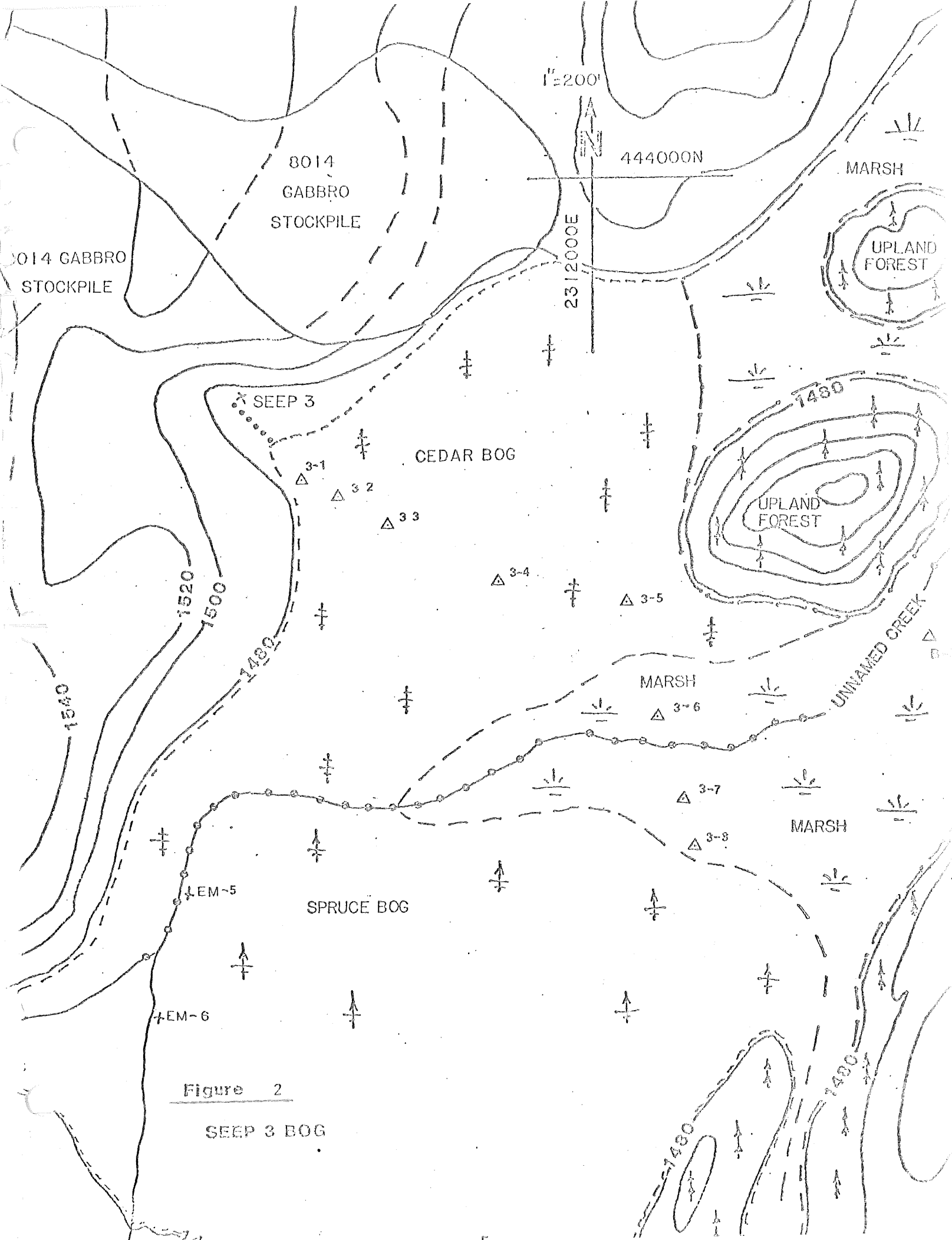


Figure 2

SEEP 3 BOG

Ni, Co, Zn, Fe, Mn), major cations (Ca, Mg, and in 1978, Na), Al (in 1979), SO_4, Cl, pH, alkalinity, dissolved organic carbon (DOC), dissolved inorganic carbon (DIC), specific conductance, and temperature. Initially both total and filtered metals were measured, but filtered metals analyses were deleted as there was no significant difference between total and filtered measurements. Nitrogen and phosphorous were analyzed in selected samples and stage readings were taken to quantify flow. More time intensive studies were conducted and are described in another report (14).

Continuous flow measurements were made during the open water (unfrozen) period at Em 8, Em 1 (1976 - present); Em 3(1976-1977); Em 5 (1977) and Seep 3 (1979 - present). Periodic flow measurements were made at Seep 1 (1976 - present); Seep 3 (1976 - 1979); Em 2 and Em 4 (1976 - 1977), and Em 5 (1976).

The bog was studied from July 1976 to August 1977 (12, 13). Water levels at the surface and in wells and piezometers were recorded, and drawdown tests were conducted at the eight piezometers. Trace metals and specific conductance were analyzed at the surface and in the wells and piezometers. At each site composite peat samples were collected from the top 20 cm and analyzed for trace metals by EDTA and acid extraction. At four stations (3-1, 3-2, 3-5, 3-8) and two control sites, trace metal concentrations were determined at approximately 25 cm depth intervals.

4. RESULTS

Several references related to this report are available to augment the comparatively brief summary which follows. These references discuss laboratory leaching studies (3, 4, 5, 6, 7), field leaching and transport studies (1, 7-15,19-23), stockpile

hydrology (16-18), stockpile reclamation (19-21), regional data (1), and transport through the Seep 3 bog (12, 13) and Unnamed Creek (14). Reports are presently being prepared on transport in Bob Bay (15), sediment data from Birch Lake and other regional lakes (22), and trace metal concentrations in aquatic vegetation (23).

4.1 STOCKPILE RUNOFF

4.1.1 Discharge

The stockpile seepages typically flowed from April to November. The median flows based on periodic measurements, at Em 8, Seep 1, and Seep 3 were 5, 1.2 and 2 l/sec, respectively (Figure 4). Additional information on stockpile hydrology is available in other reports (16-18)

4.1.2 pH

Over the first four years of the study the median pH at Em 8 (7.16) and Seep 3 (7.10) was comparable, but was significally lower at Seep 1 (6.70). The pH at the seeps varied seasonally but was gnerally within about 0.3 units of the yearly median (Figure 5). There was a tendency for pH to decrease over the course of the study (Figure 6).

4.1.3 Copper, Nickel, Cobalt, and Zinc

Median trace metal (Cu, Ni, Co, Zn) concentrations observed in the Erie stockpile leachate ranged from 10 to 10,000 times natural background levels of streams in the area (Table 2). Metal concentrations varied with stockpile composition, the chemistry of the individual metal, and time, but were generally independent of flow. Trace metal concentrations were higher in seepage from lean ore piles than waste rock leachate (Figure 7, Table 2). An exception was zinc, which had higher concentrations at Seep 1 than Seep 3. Cobalt concentrations at Seep 3 were also

Figure 3 - Interpretation of Box Plot Figures

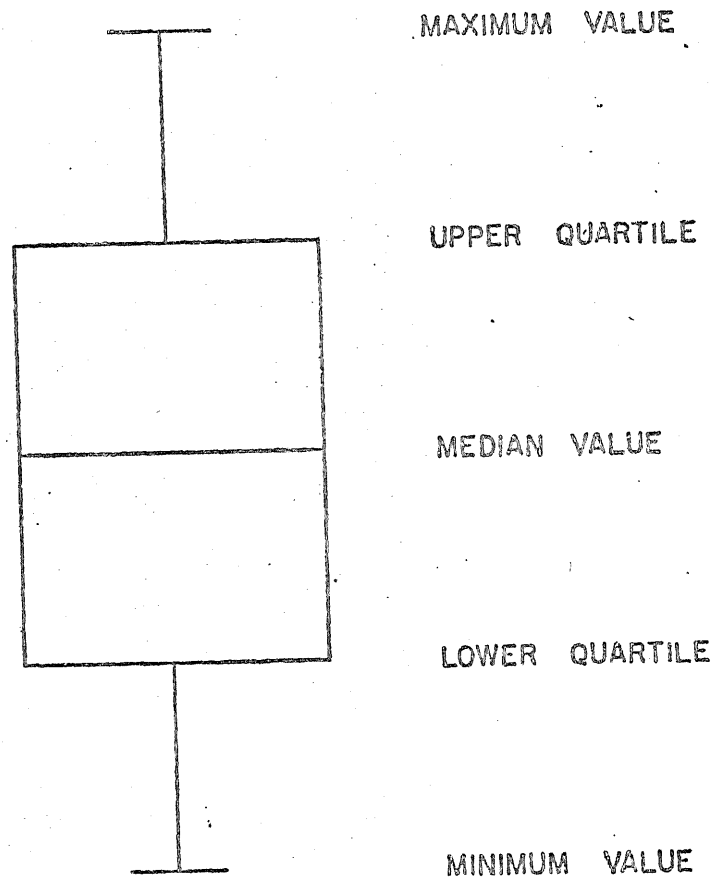
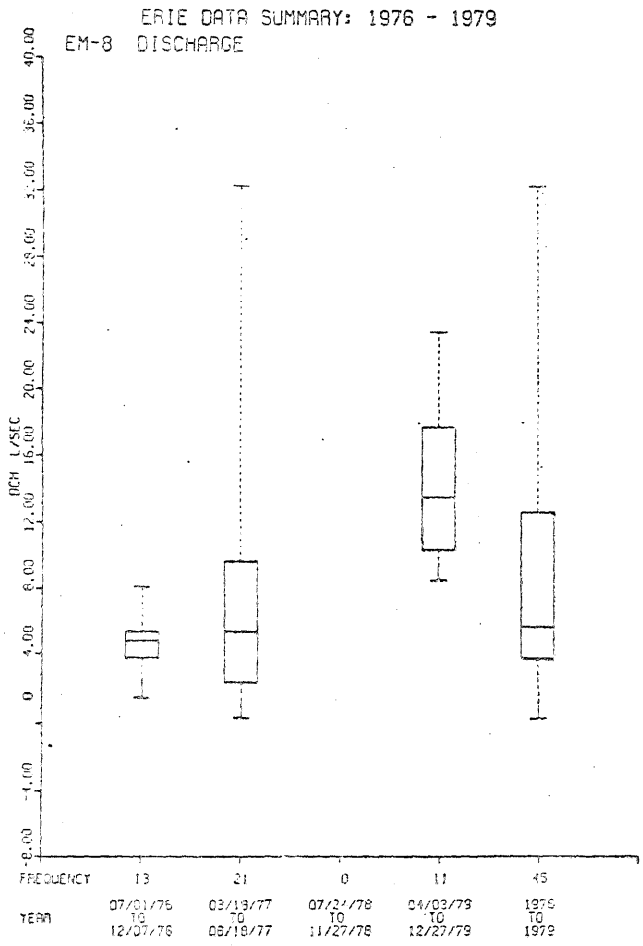
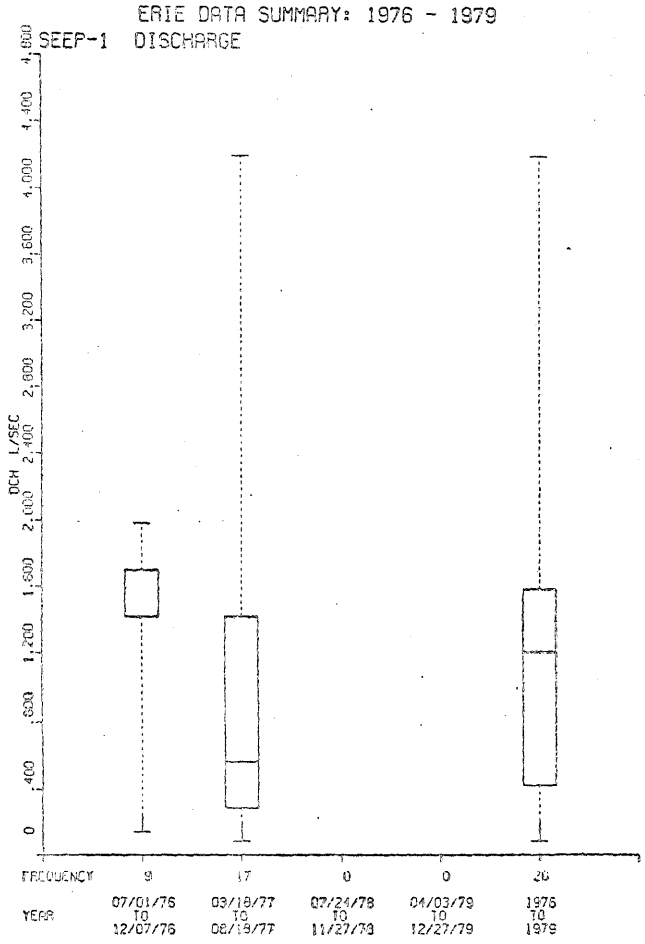
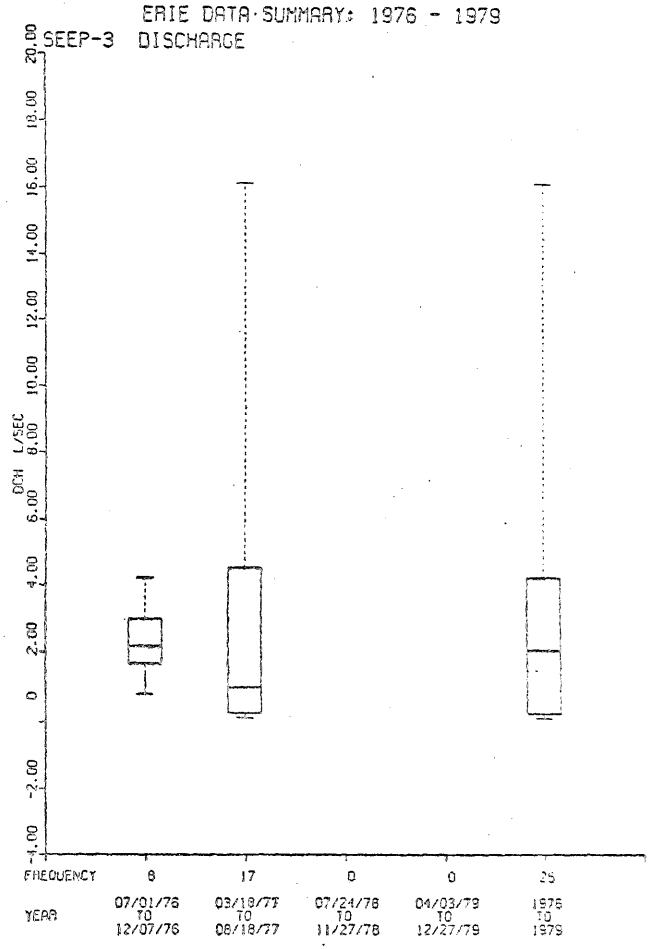


FIGURE 4







Plots are based on periodic measurements
Continuous flow records exist for Em-8 (1976-present) and Seep 3 (1979-present)

FIGURE 5

SEEP 1.-TOTAL COPPER, TOTAL NICKEL, TOTAL IRON, pH

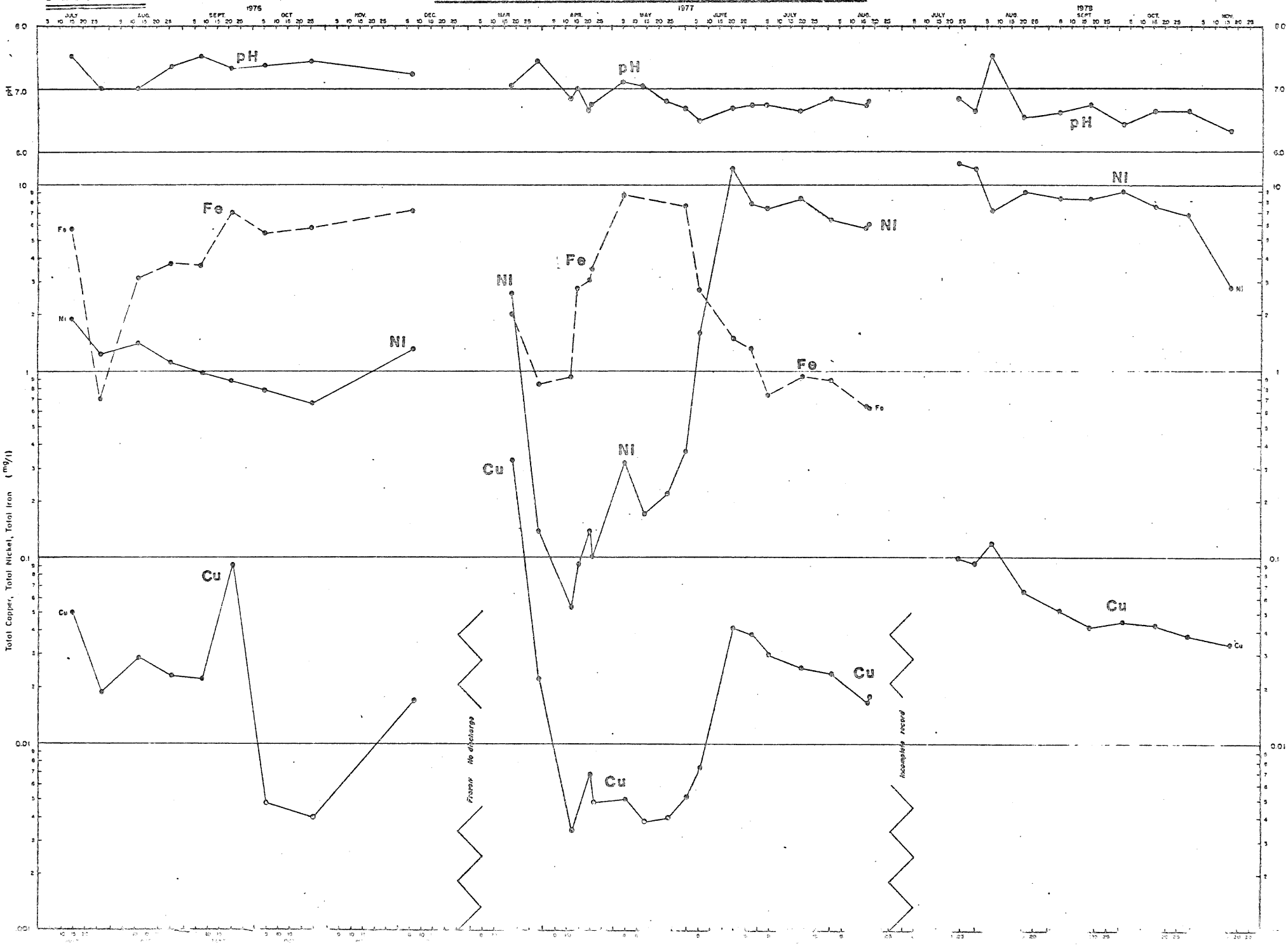
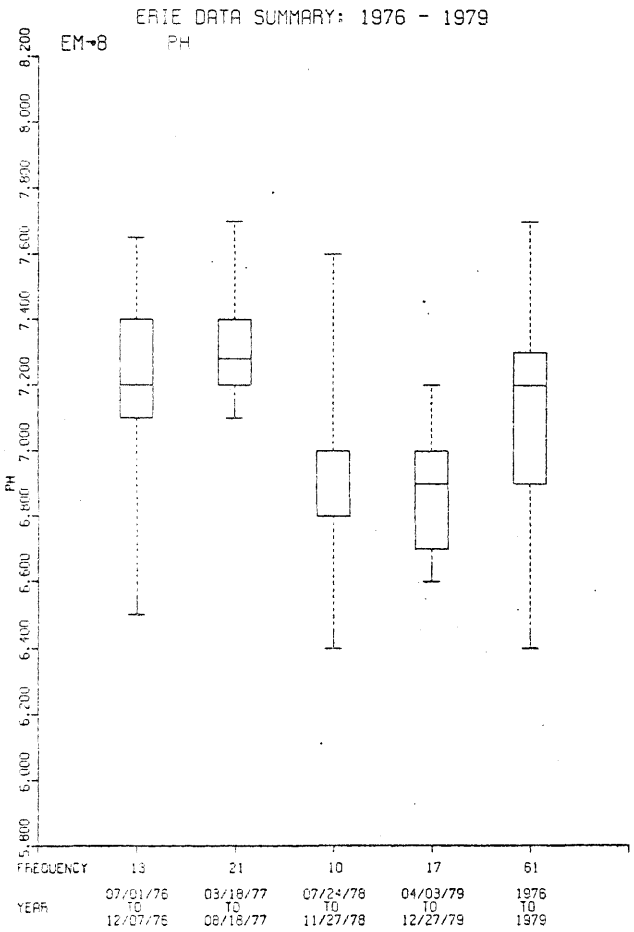
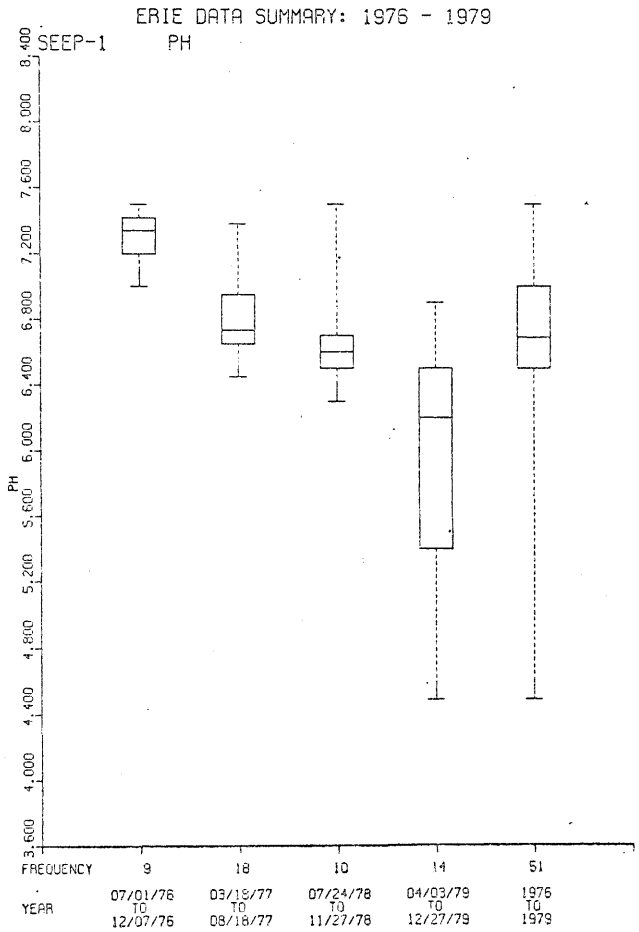


FIGURE 6


ERIE DATA SUMMARY: 1976 - 1979
EM-8
PH
PH
8.200
8.000
7.800
7.600
7.400
7.200
7.000
6.800
6.600
6.400
6.200
6.000
5.800
FREQUENCY 13 21 10 17 61
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1
PH
PH
8.400
8.000
7.600
7.200
6.800
6.400
6.000
5.600
5.200
4.800
4.400
4.000
3.600
FREQUENCY 9 18 10 14 51
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

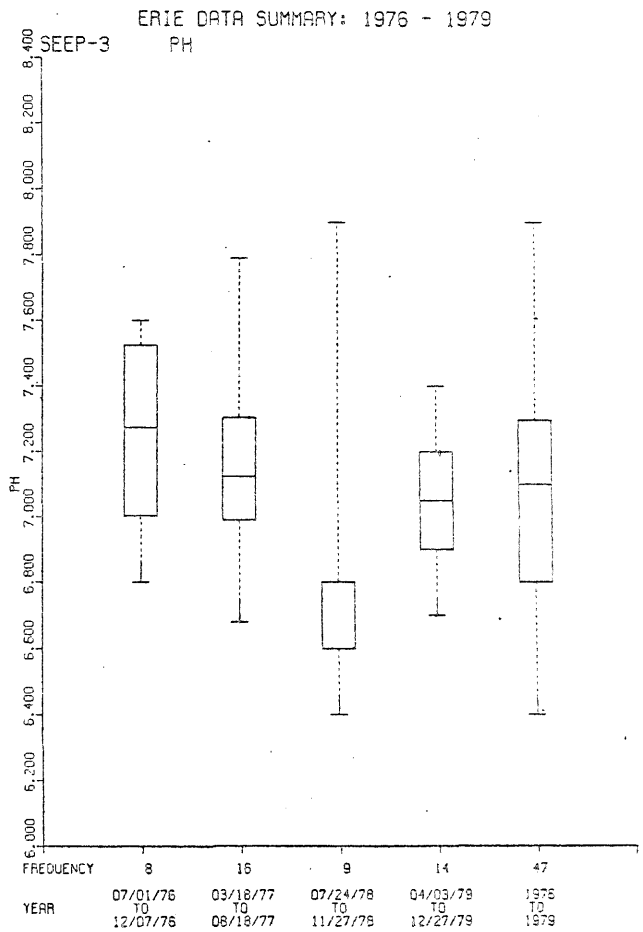

ERIE DATA SUMMARY: 1976 - 1979
SEEP-3
PH
PH
8.400
8.200
8.000
7.800
7.600
7.400
7.200
7.000
6.800
6.600
6.400
6.200
6.000
FREQUENCY 8 16 9 14 47
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

Figure 7

(Cu), (Ni) IN ERIE STOCKPILE SEEPAGE : 1976-1979

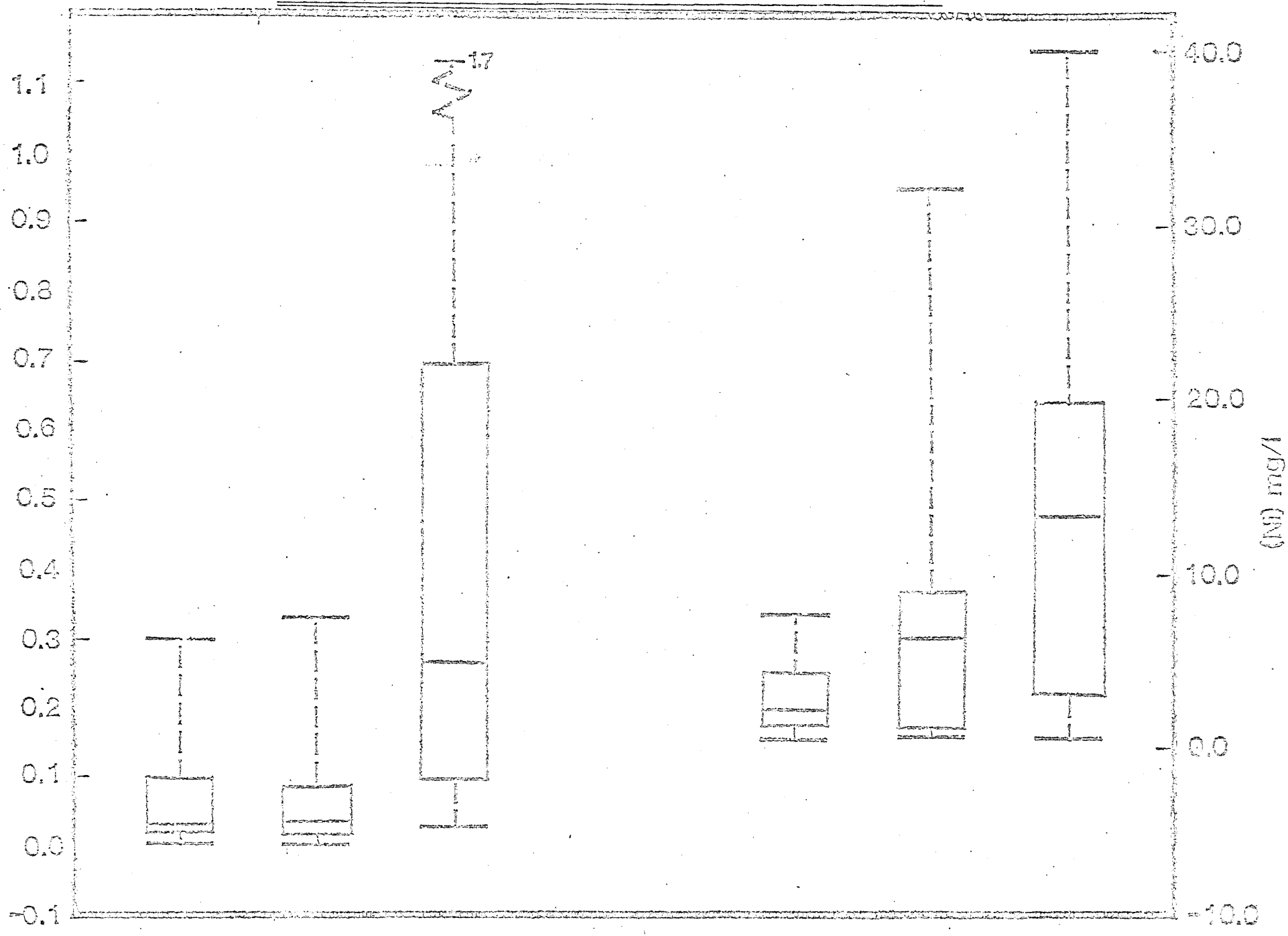


TABLE 2

COMPARISON OF SEEP WATER QUALITY
WITH NATURAL BACKGROUND:

MEDIAN VALUES[1] 1976-1979

Parameter	Waste Rock EM 8	Waste Rock Seep 1	Lean Ore Seep 3	Natural[2]
pH	7.16 (59)[3]	6.70 (51)	7.10 (48)	6.9
Sulfate	1200. (50)	3400. (42)	1364. (42)	6.6
Copper[4]	0.030 (67)	0.036 (48)	0.270 (50)	0.0013
Nickel	2.14 (62)	6.13 (50)	13.3 (51)	0.001
Cobalt	0.195 (14)	0.64 (19)	0.790 (19)	0.004
Zinc	0.10 (33)	1.45 (35)	0.18 (37)	0.002

[1]Concentrations in mg/l

[2]Regional Copper-Nickel Study

[3]Numbers in parentheses indicate number of samples

[4]Metal concentrations as total metal

significantly higher at Seep 1 than in the waste rock leachate at Em 8.

The seasonal variation of seep water quality was greatest from March to the beginning of June, prior to and during thaw. (Figures 5, 8, 9, in particular 1977 data). After this period concentrations of Cu, Ni, Co and Zn were less subject to wide fluctuation. The variation over the course of the study was not consistent. At Em 8 and Seep 1, trace metal concentrations increased, but at Seep 3 concentrations decreased (Figures 10-12).

4.1.4 Sulfate, calcium, and magnesium

The median sulfate concentration at Seep 1 was higher than those at Em 8 and Seep 3 (3400 vs 1200 and 1360 mg/1). Sulfate concentrations at Em 8 increased steadily during the 1976 season, varied widely in 1977, and were relatively stable in 1978 (Figure 13)). Seep 1 concentrations varied widely in 1977 but were relatively stable in 1976 and 1978 (Figure 14). Sulfate concentrations at Seep 3 varied widely during all three years. (Figure 15). Despite the wide seasonal fluctuations median sulfate concentrations at each of the three seeps was relatively constant over the course of the study (Figures 16-18).

From 1976 - 1978 *, median calcium concentrations were highest at Seep 3, but magnesium concentrations were highest at Seep 1 (Table 3). Calcium concentrations exceeded magnesium concentrations at Em-8 and Seep 3, but the opposite was true at

*Calcium and magnesium date reported for 1979 was anomolous due to a faulty analytical technique. (Appendix)

Figure

EM-8 TOTAL COPPER, TOTAL NICKEL, TOTAL IRON, pH

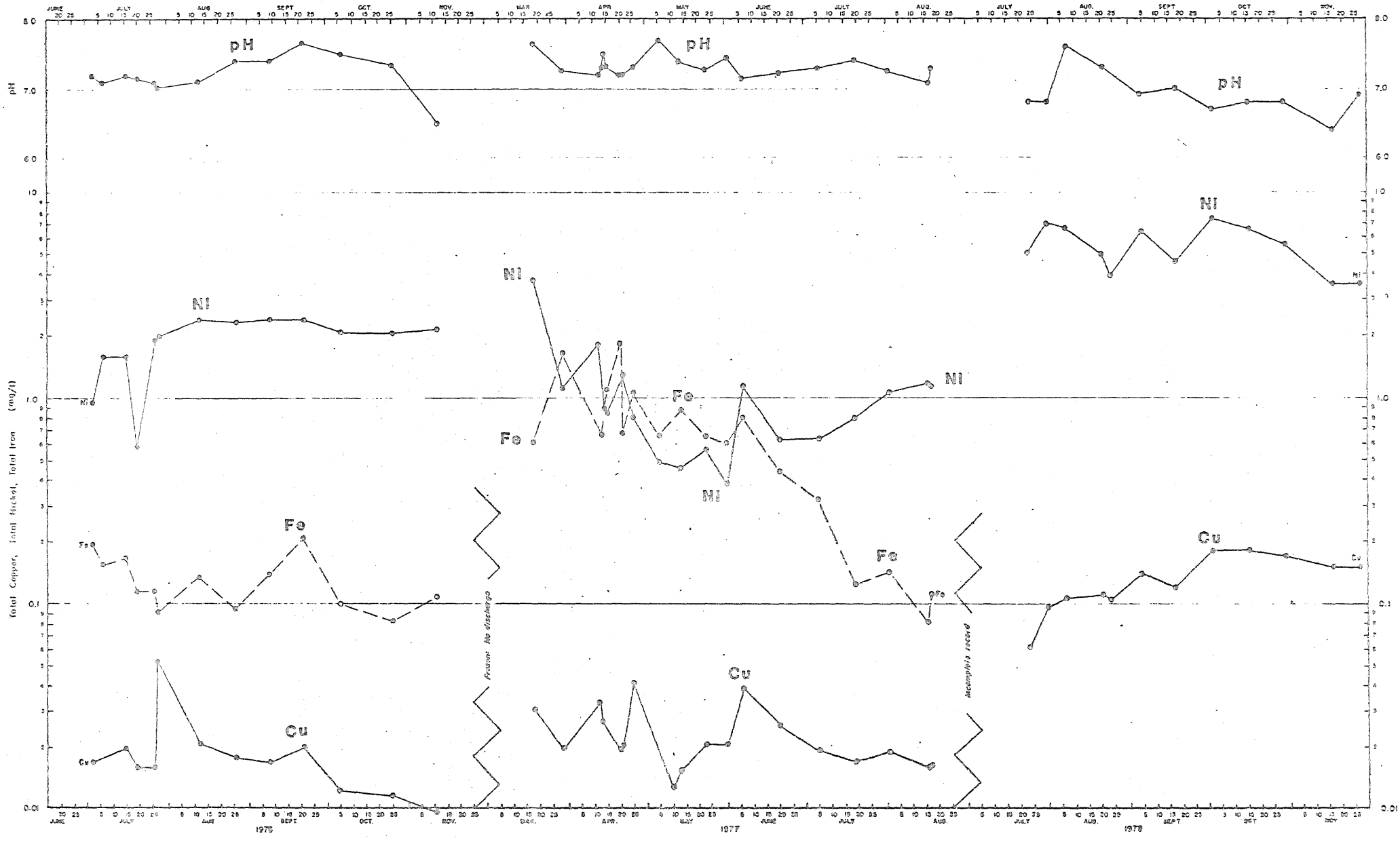


Figure 9

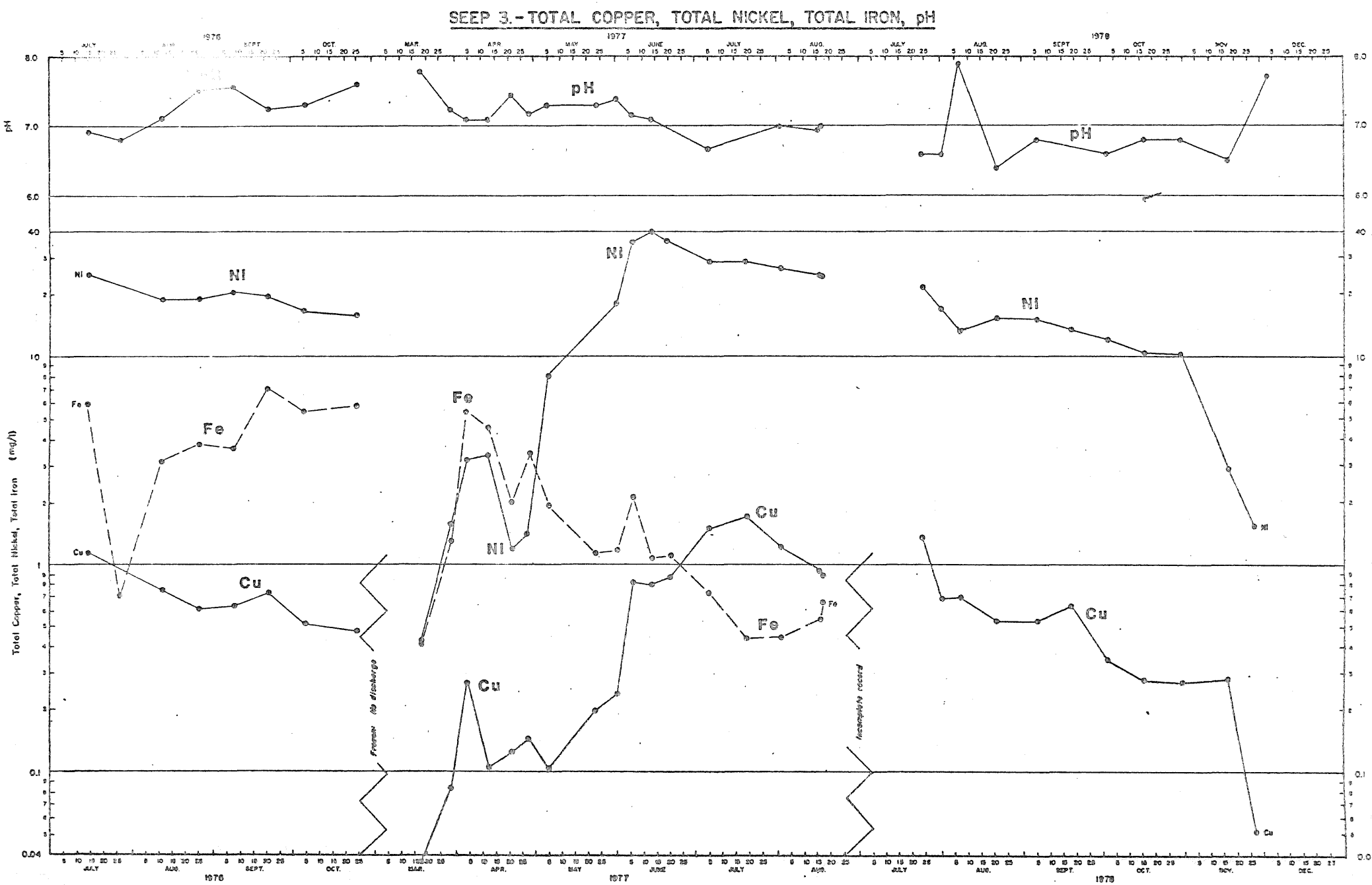
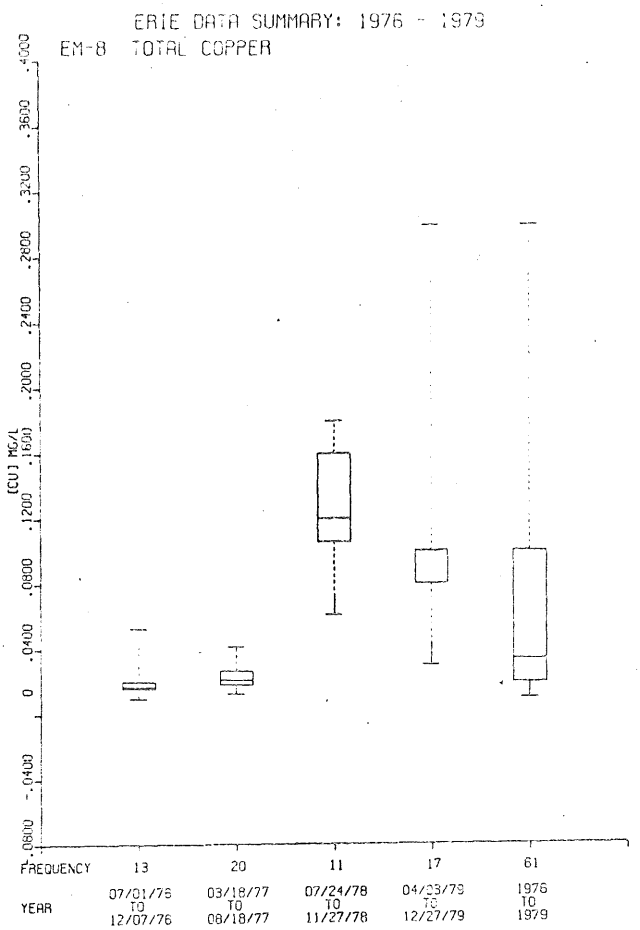
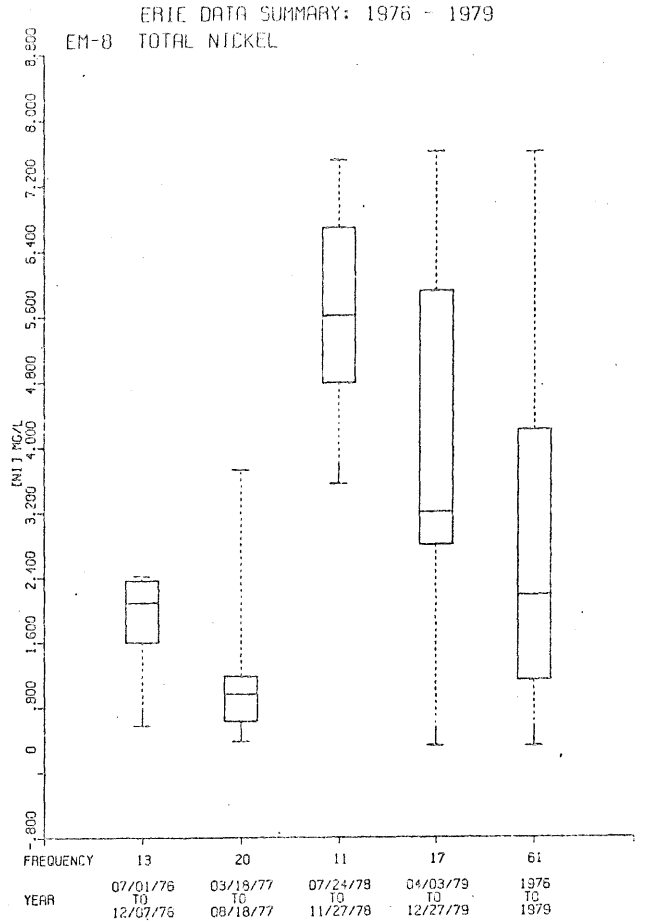
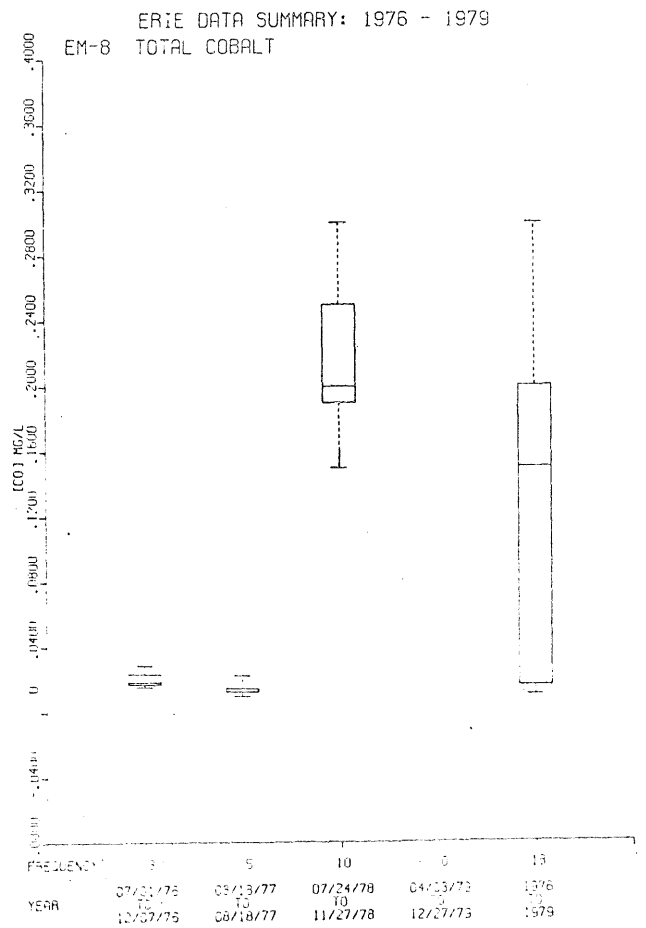

SEEP 3.-TOTAL COPPER, TOTAL NICKEL, TOTAL IRON, pH
1976
1977
1978
pH
Ni
Fe
Cu
Total Copper, Total Nickel, Total Iron (mg/l)
Frozen No discharge
Inadequately record

FIGURE 10


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL COPPER
[CU] MG/L
.4000
.3600
.3200
.2800
.2400
.2000
.1600
.1200
.0800
.0400
0
-.0400
-.0800
FREQUENCY 13 20 11 17 61
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL NICKEL
[NI] MG/L
8.800
8.000
7.200
6.400
5.600
4.800
4.000
3.200
2.400
1.600
.800
0
-.800
FREQUENCY 13 20 11 17 61
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL COBALT
[CO] MG/L
.4000
.3600
.3200
.2800
.2400
.2000
.1600
.1200
.0800
.0400
0
-.0400
FREQUENCY 5 10 13
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

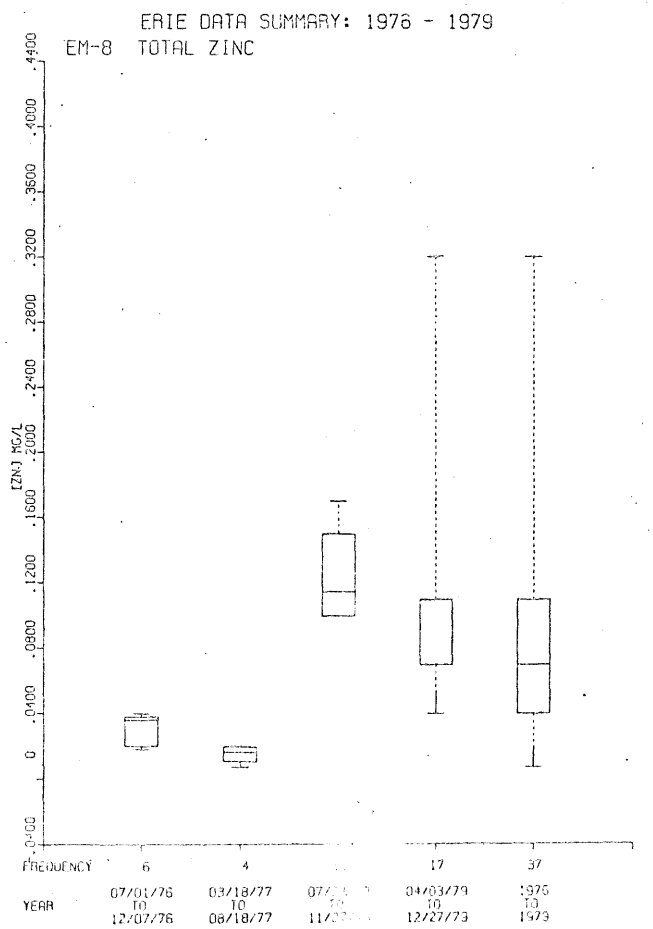
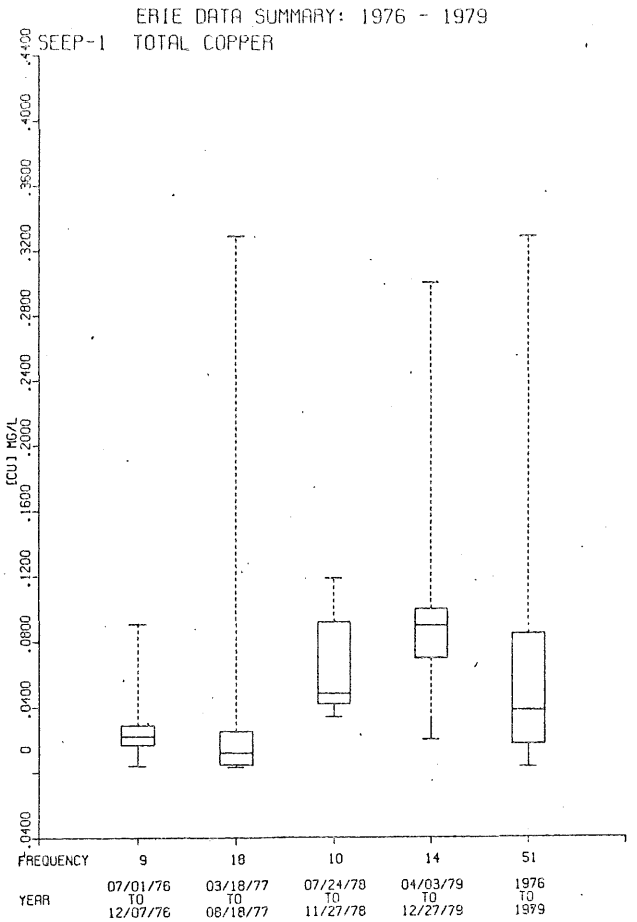
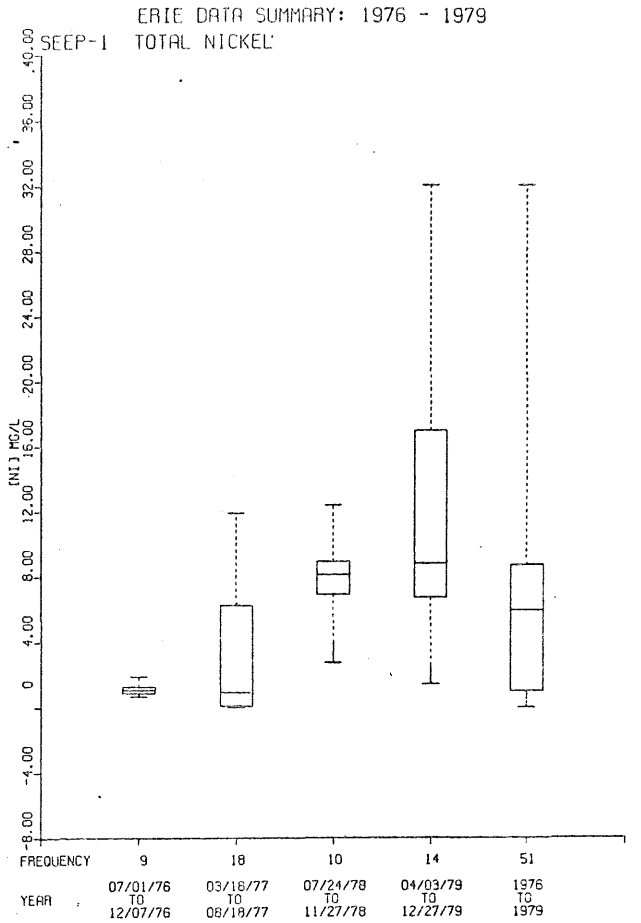
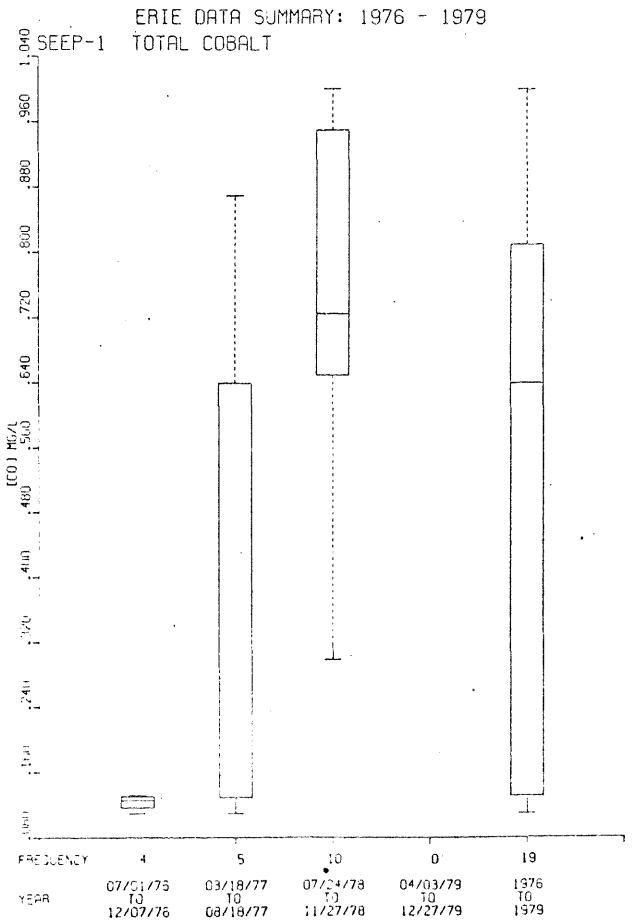

ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL ZINC
[ZN] MG/L
.4400
.4000
.3600
.3200
.2800
.2400
.2000
.1600
.1200
.0800
.0400
0
-.0400
FREQUENCY 6 4 17 37
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 04/03/79 TO 12/27/79 | 1976 TO 1979

FIGURE 11


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL COPPER
[CU] MG/L
.4400
.4000
.3600
.3200
.2800
.2400
.2000
.1600
.1200
.0800
.0400
0
-.0400
FREQUENCY 9 18 10 14 51
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL NICKEL
[NI] MG/L
40.00
36.00
32.00
28.00
24.00
20.00
16.00
12.00
8.00
4.00
0
-4.00
-8.00
FREQUENCY 9 18 10 14 51
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL COBALT
[CO] MG/L
1.040
.960
.880
.800
.720
.640
.560
.480
FREQUENCY 4 5 10 0 19
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

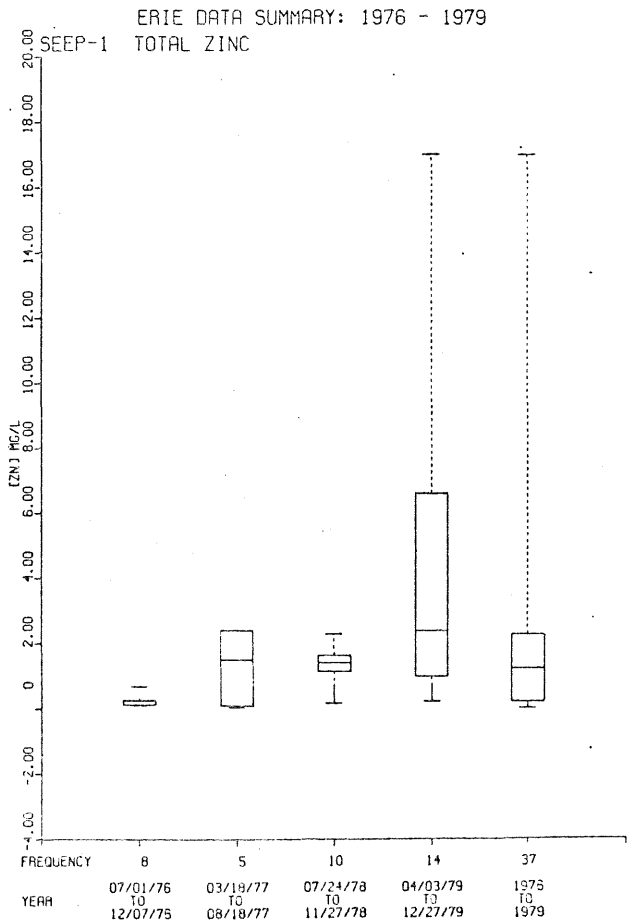
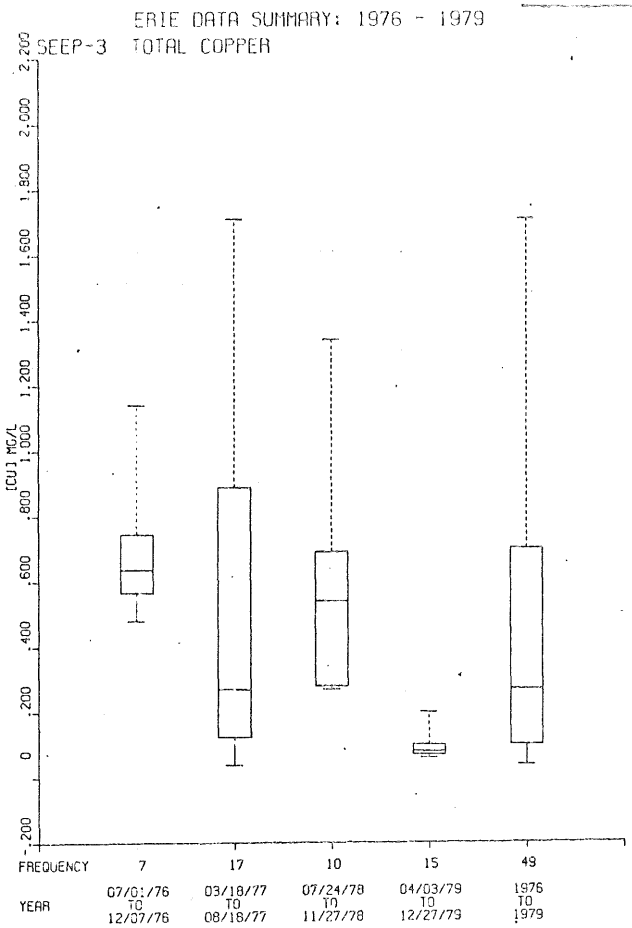
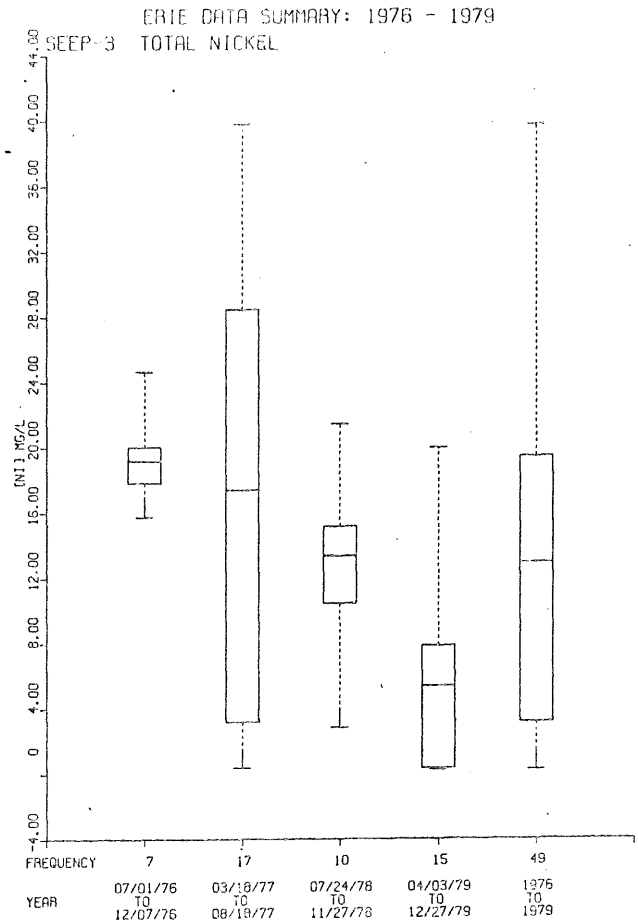
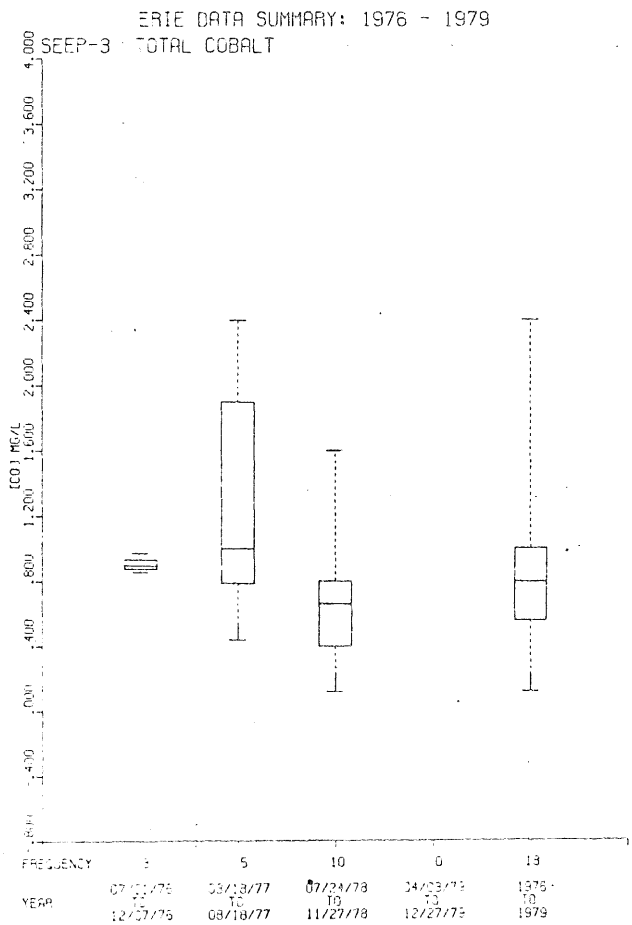
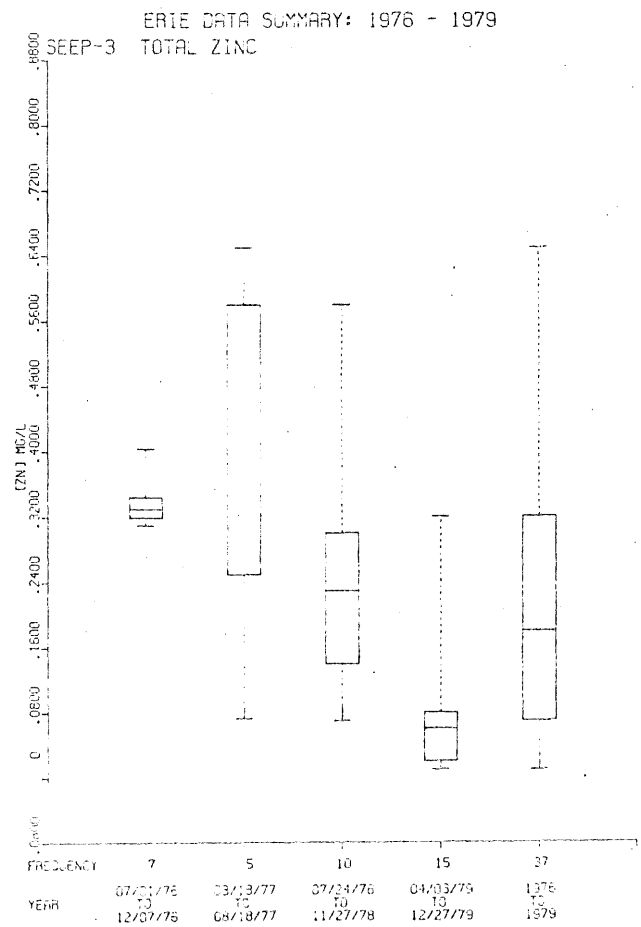

ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL ZINC
[ZN] MG/L
20.00
18.00
16.00
14.00
12.00
10.00
8.00
6.00
4.00
2.00
0
-2.00
-4.00
FREQUENCY 8 5 10 14 37
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

FIGURE 12


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL COPPER
[CU] MG/L
2.200
2.000
1.800
1.600
1.400
1.200
1.000
.800
.600
.400
.200
0
-.200
FREQUENCY 7 17 10 15 49
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL NICKEL
[NI] MG/L
44.00
40.00
36.00
32.00
28.00
24.00
20.00
16.00
12.00
8.00
4.00
0
-4.00
FREQUENCY 7 17 10 15 49
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL COBALT
[CO] MG/L
4.000
3.600
3.200
2.800
2.400
2.000
1.600
1.200
.800
.400
0
-.400
FREQUENCY 3 5 10 0 13
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL ZINC
[ZN] MG/L
.8800
.8000
.7200
.6400
.5600
.4800
.4000
.3200
.2400
.1600
.0800
0
-.0800
FREQUENCY 7 5 10 15 37
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

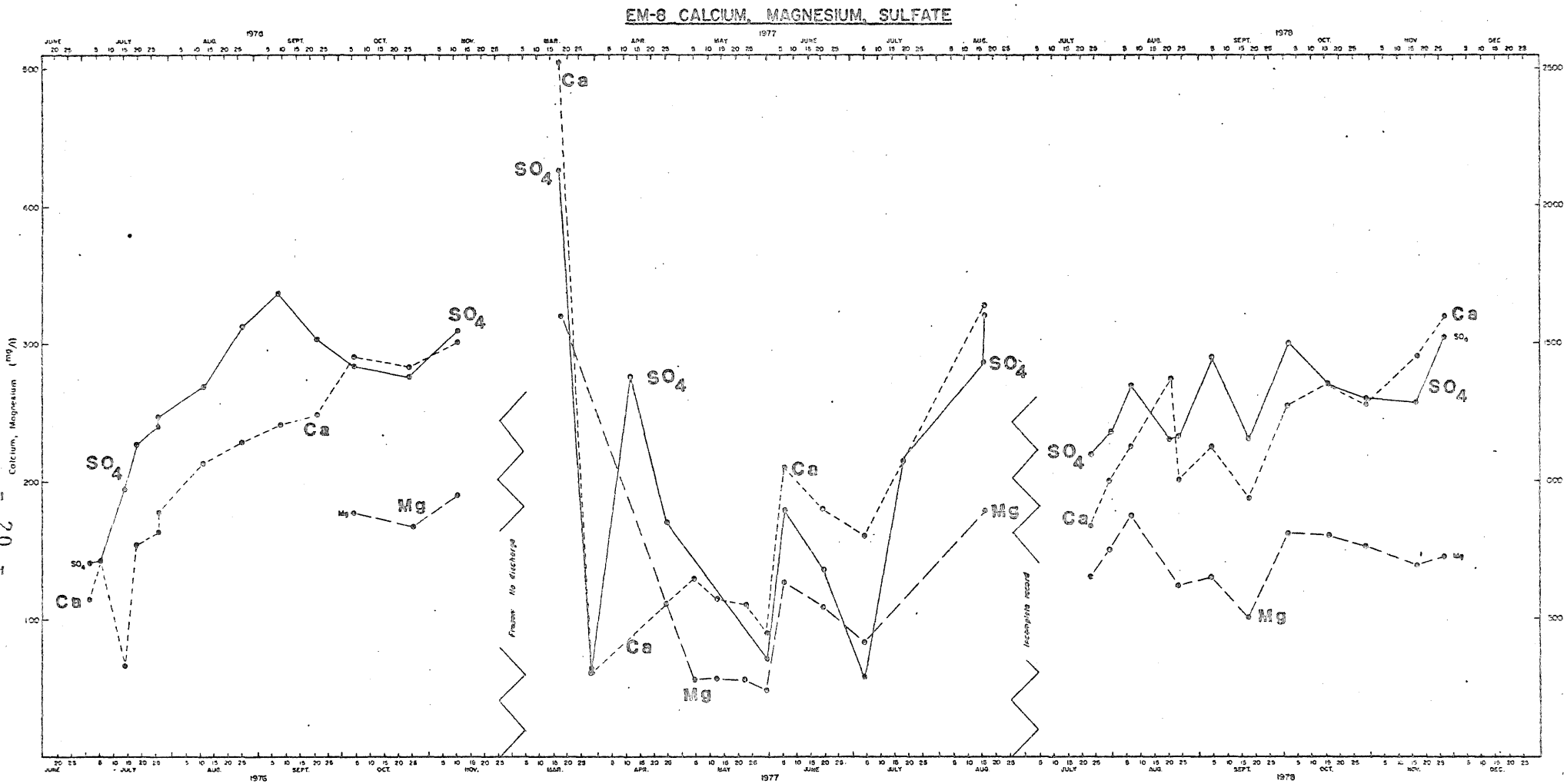

EM-8 CALCIUM, MAGNESIUM, SULFATE
Calcium, Magnesium (mg/l)
Sulfate (mg/l)
1976
1977
1978
Ca
SO4
Mg
Frozen: No discharge
Frozen: No discharge

FIGURE 13

SEEP I. – CALCIUM, MAGNESIUM, SULFATE

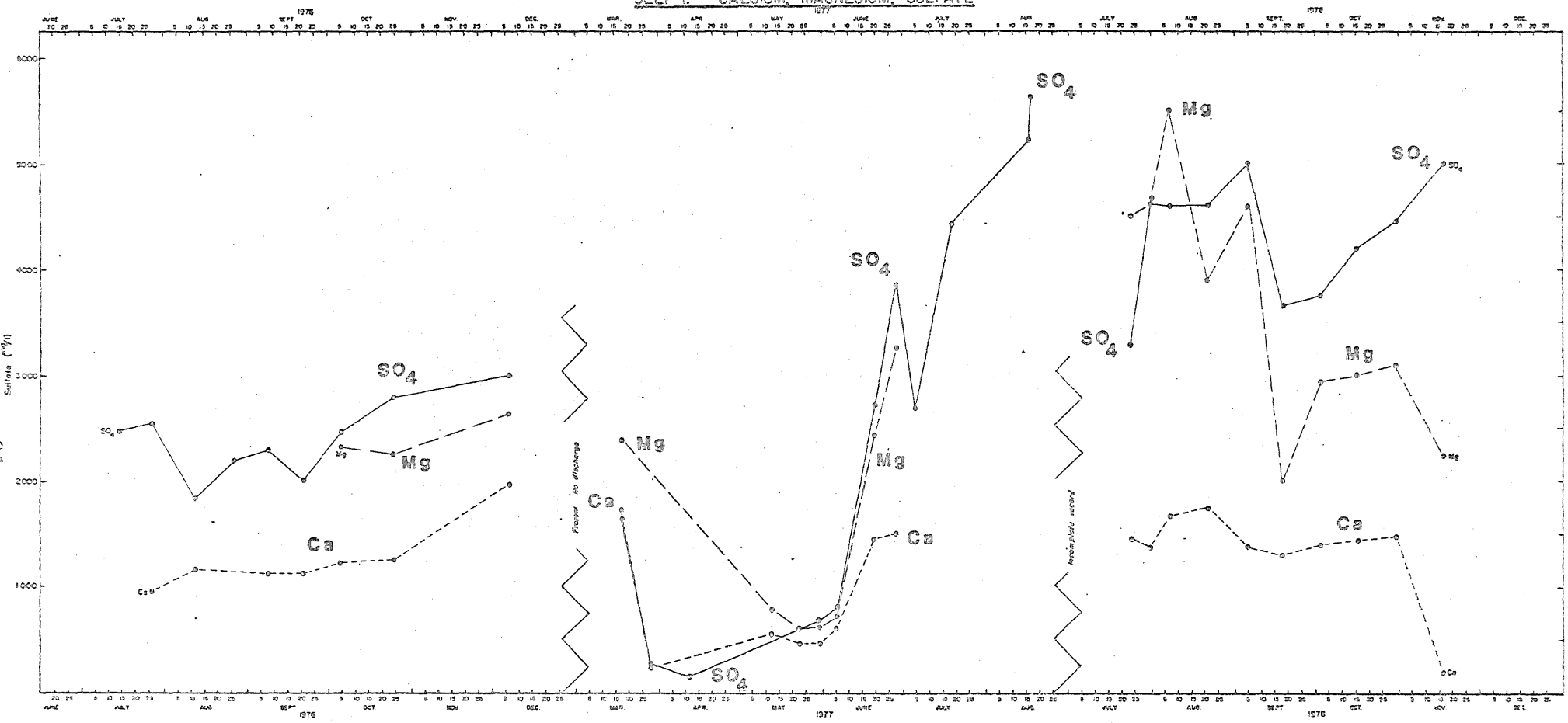


FIGURE 14

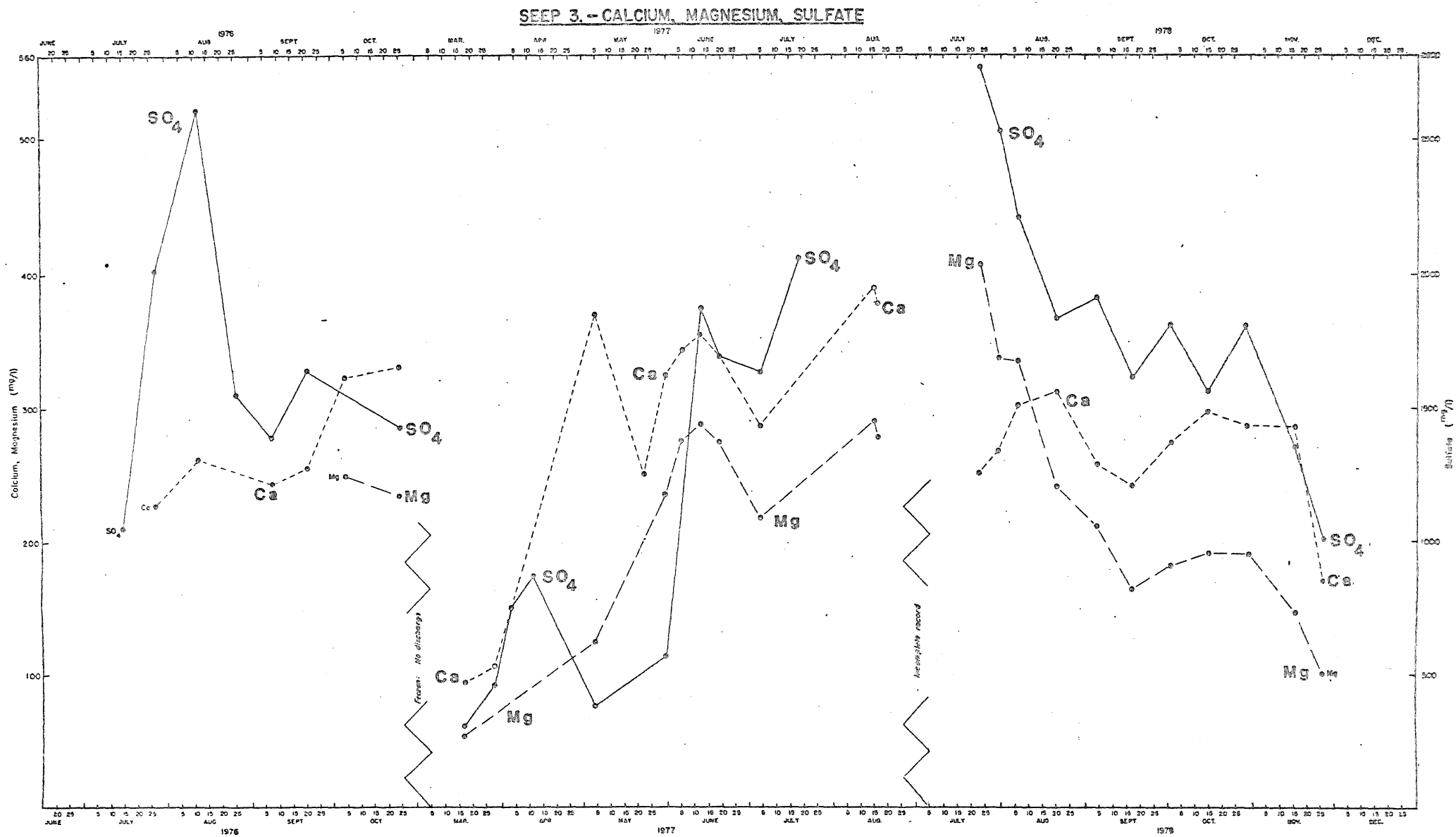
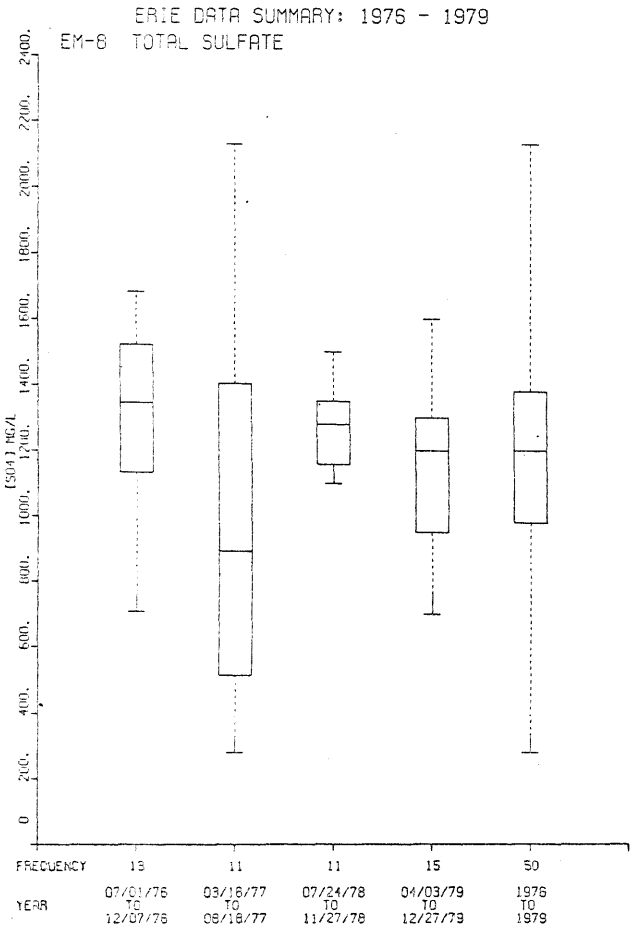
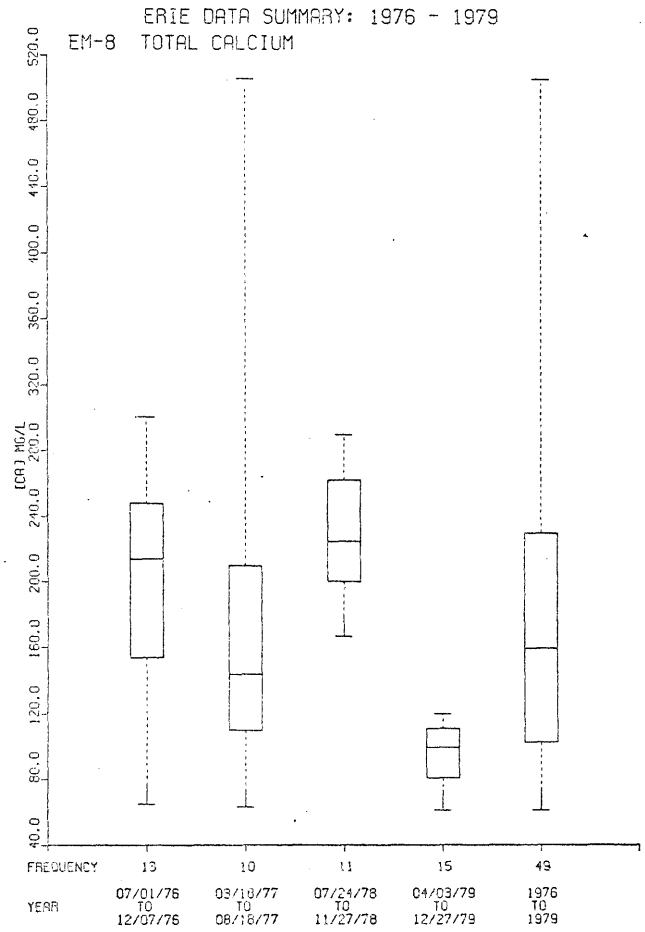

SEEP 3. - CALCIUM, MAGNESIUM, SULFATE
Calcium, Magnesium (mg/l)
Sulfate (mg/l)
1976
1977
1978
JUNE
JULY
AUG
SEPT
OCT.
MAR.
APR
MAY
JUNE
JULY
AUG.
NOV.
DEC.
SO4
Ca
Mg
From: No discharge
Incomplete record

FIGURE 15

FIGURE 16


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL SULFATE
[SO4] MG/L
0
200.
400.
600.
800.
1000.
1200.
1400.
1600.
1800.
2000.
2200.
2400.
FREQUENCY 13 11 11 15 50
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL CALCIUM
[CA] MG/L
40.0
80.0
120.0
160.0
200.0
240.0
280.0
320.0
360.0
400.0
440.0
480.0
520.0
FREQUENCY 13 10 11 15 49
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

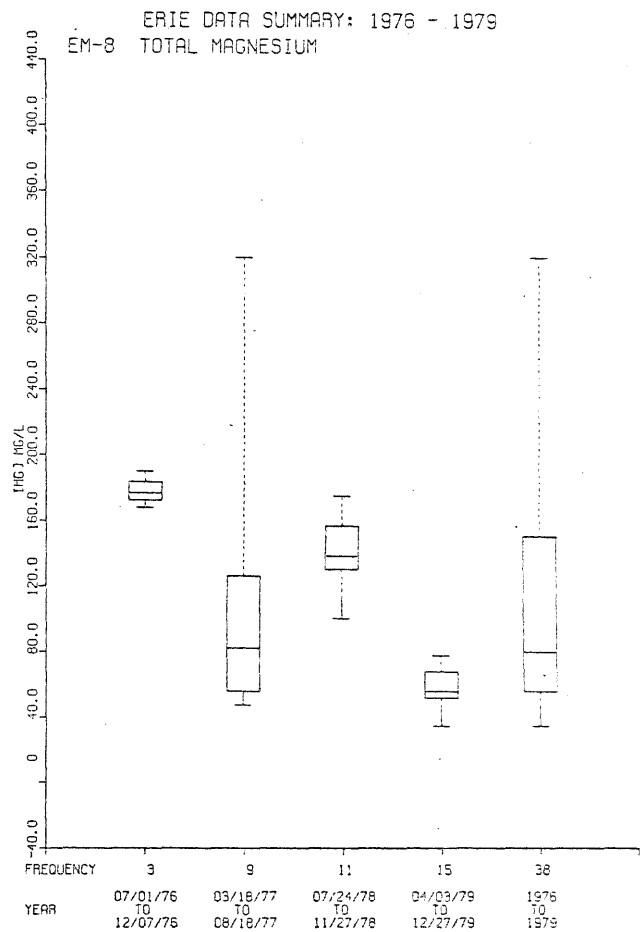
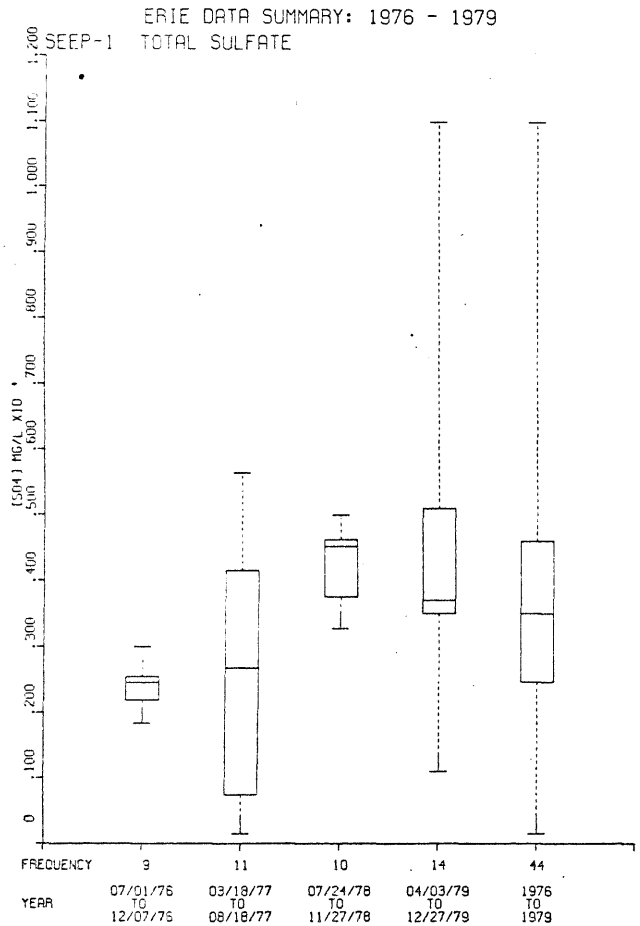
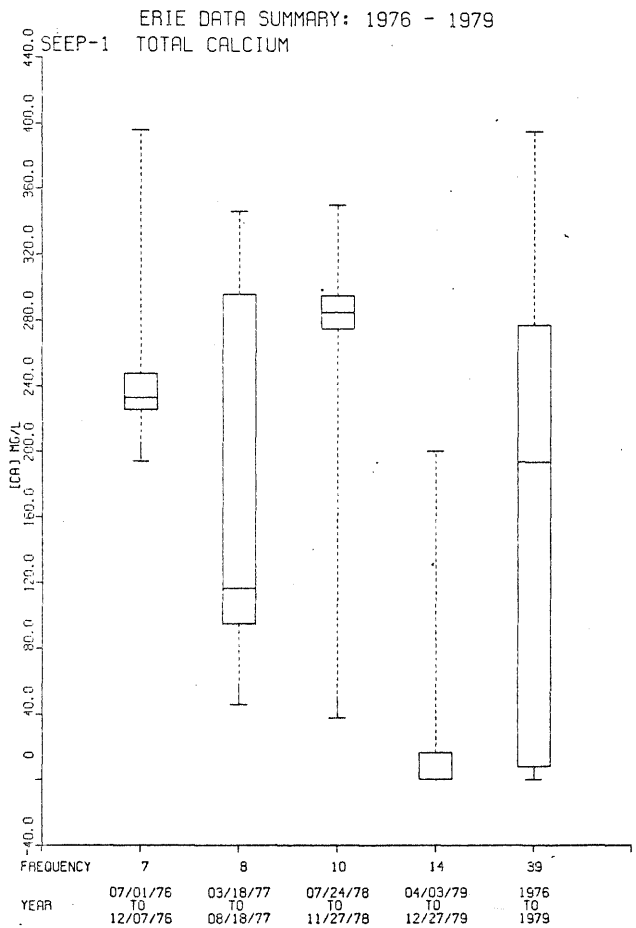

ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL MAGNESIUM
[MG] MG/L
-40.0
0
40.0
80.0
120.0
160.0
200.0
240.0
280.0
320.0
360.0
400.0
440.0
FREQUENCY 3 9 11 15 38
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

FIGURE 17


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL SULFATE
[SO4] MG/L X10
0
100
200
300
400
500
600
700
800
900
1,000
1,100
1,200
FREQUENCY 9 11 10 14 44
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL CALCIUM
[CA] MG/L
-40.0
0
40.0
80.0
120.0
160.0
200.0
240.0
280.0
320.0
360.0
400.0
440.0
FREQUENCY 7 8 10 14 39
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

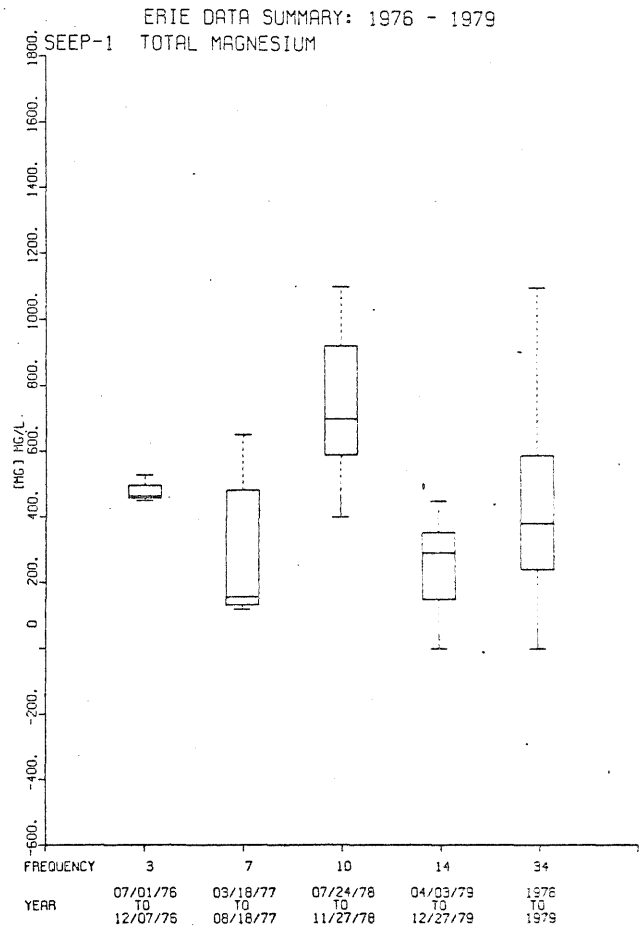
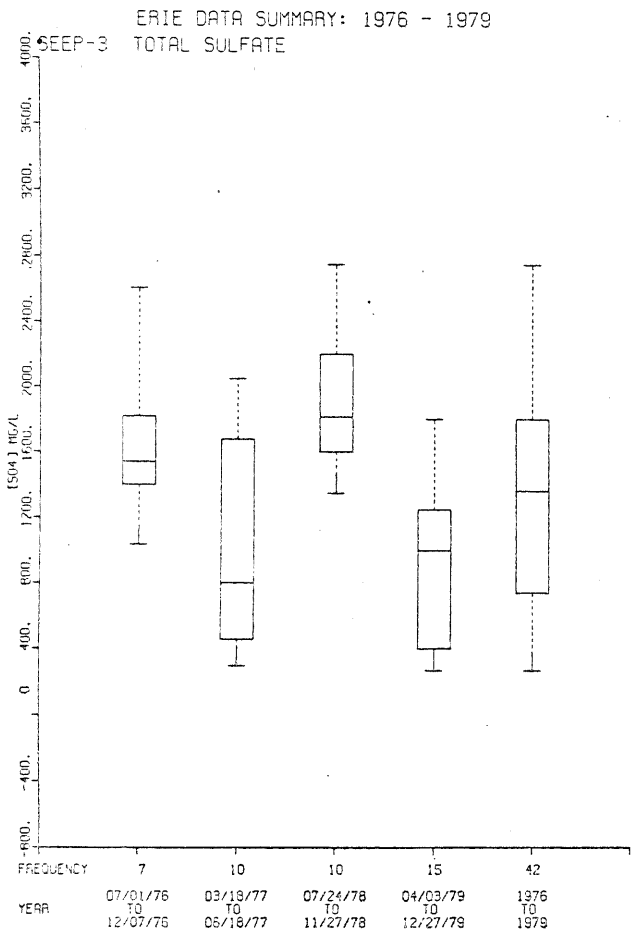
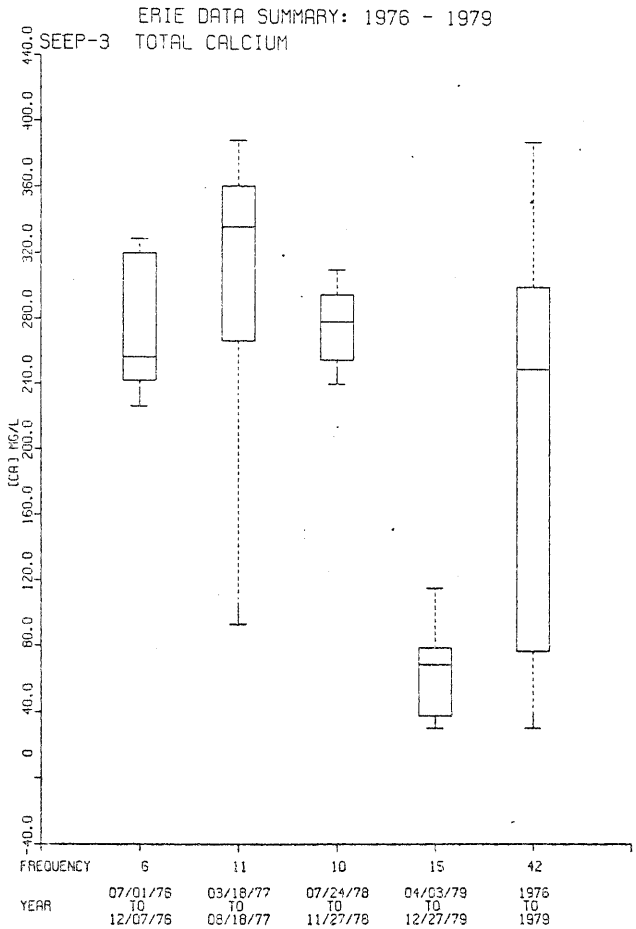

ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL MAGNESIUM
[MG] MG/L
-600.
-400.
-200.
0
200.
400.
600.
800.
1000.
1200.
1400.
1600.
1800.
FREQUENCY 3 7 10 14 34
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

FIGURE 18


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL SULFATE
[SO4] MG/L
4000.
3600.
3200.
2800.
2400.
2000.
1600.
1200.
800.
400.
0
-400.
-800.
FREQUENCY 7 10 10 15 42
YEAR 07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL CALCIUM
[CA] MG/L
440.0
400.0
360.0
320.0
280.0
240.0
200.0
160.0
120.0
80.0
40.0
0
-40.0
FREQUENCY 6 11 10 15 42
YEAR 07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

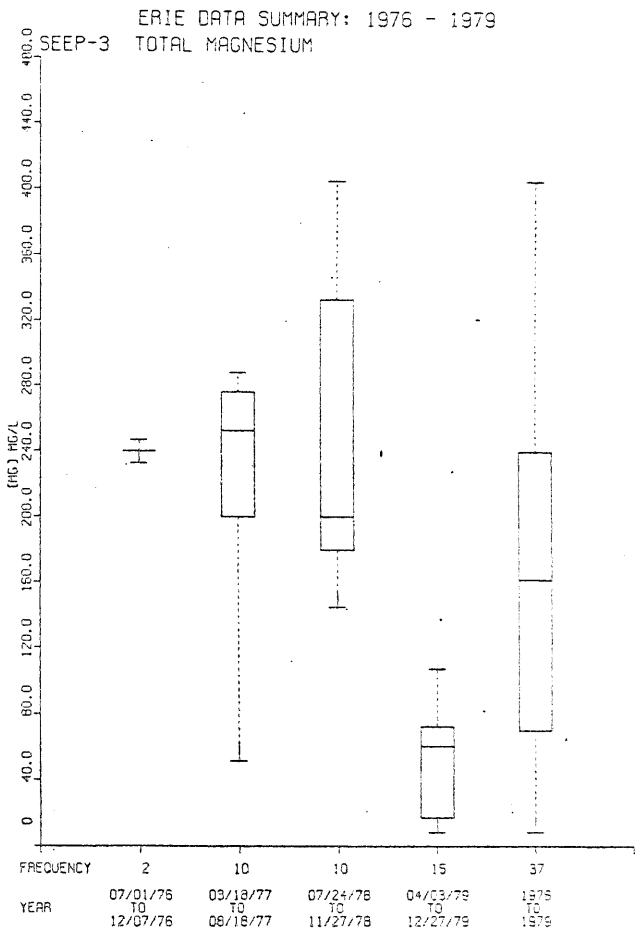

ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL MAGNESIUM
[MG] MG/L
480.0
440.0
400.0
360.0
320.0
280.0
240.0
200.0
160.0
120.0
80.0
40.0
0
FREQUENCY 2 10 10 15 37
YEAR 07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

TABLE 3

Median calcium and magnesium concentrations in stockpile runoff, 1976-1978

	Em-8	Seep 1	Seep 3
Ca	160	195	250
Mg	80	385	160

Seep 1. Despite these variations concentrations of both calcium and magnesium tended to parallel those of sulfate (Figures 13-16). This is because calcium and magnesium are released to solution by reaction with hydrogen ions, which are generated by iron sulfide oxidation.

4.1.5 Iron, manganese, and dissolved organic carbon

Median iron concentrations ranged from about 0.3 mg/l at Em 8 to 2.4 mg/l at Seep 1 (Figures 19-21). Median manganese concentrations ranged from 1.6 mg/l at Em 8 to 8.0 mg/l at Seep 1. The depressed pH at Seep 1 most likely was a factor contributing to the elevated iron and manganese concentrations. Dissolved organic carbon (DOC) concentrations were also higher at Seep 1 (approximately 25 mg/l) than at Em 8 or Seep 3 (approximately 16 mg/l). An increase in the concentration of complexing organics, would increase the trace metal concentrations.

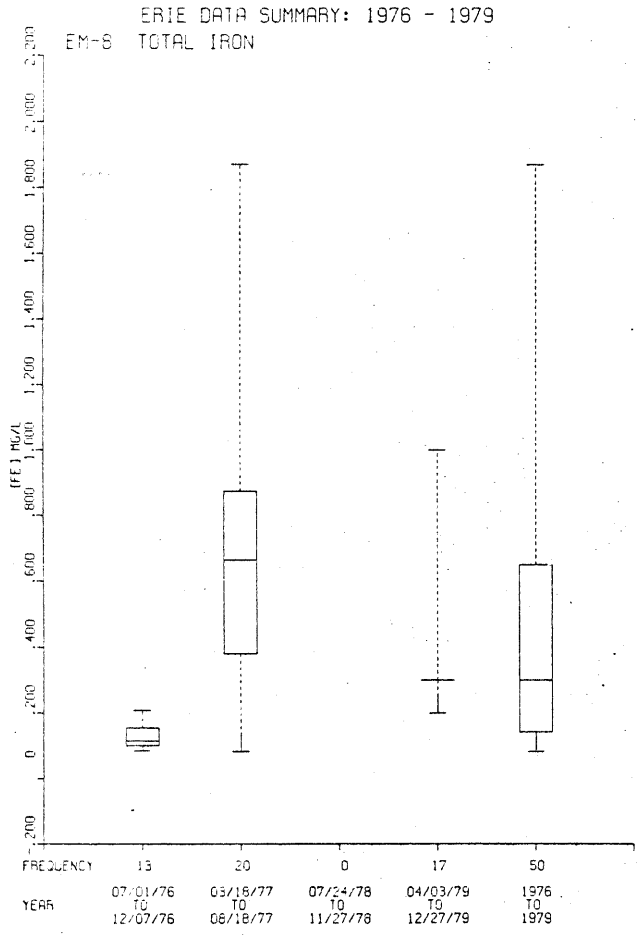
4.1.6 Specific conductance, TAlk T, Cl, P, NO_2-NO_3

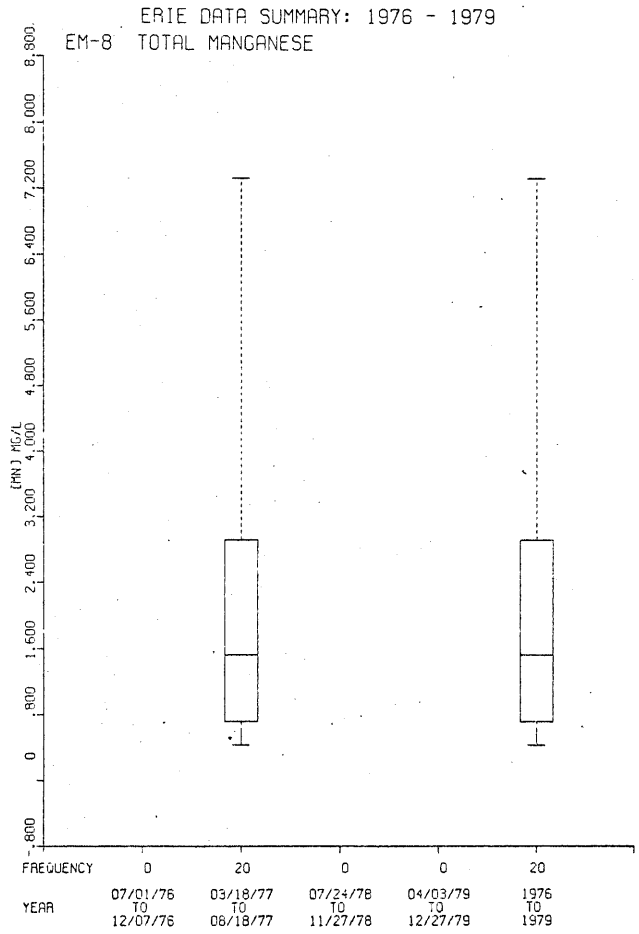
Specific conductance is an indicator of the total ionic content of the water. In the case of stockpile runoff, it reflects the concentrations of sulfate, calcium, and magnesium, since these are the major components of the runoff. Consequently, the specific conductance is highest at Seep 1 where sulfate, calcium and magnesium concentrations are highest (Figures 22-24, 16-18, Table 3).

Alkalinity was relatively constant, approximately 130 mg/l, at Em 8 (Figure 26). The alkalinity at Seep 1 decreased over the four year period, as did the pH at the site (Figures 27, 6). At Seep 3 the alkalinity was relatively constant at about 100 mg/l.

Temperatures at the seeps were typically less than 5°C throughout the summer months (Figures 23-25). The temperatures

FIGURE 19


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL IRON
(FE) MG/L
-.200
0
.200
.400
.600
.800
1.000
1.200
1.400
1.600
1.800
2.000
2.200
FREQUENCY 13 20 0 17 50
YEAR 07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL MANGANESE
(MN) MG/L
-.800
0
.800
1.600
2.400
3.200
4.000
4.800
5.600
6.400
7.200
8.000
8.800
FREQUENCY 0 20 0 0 20
YEAR 07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

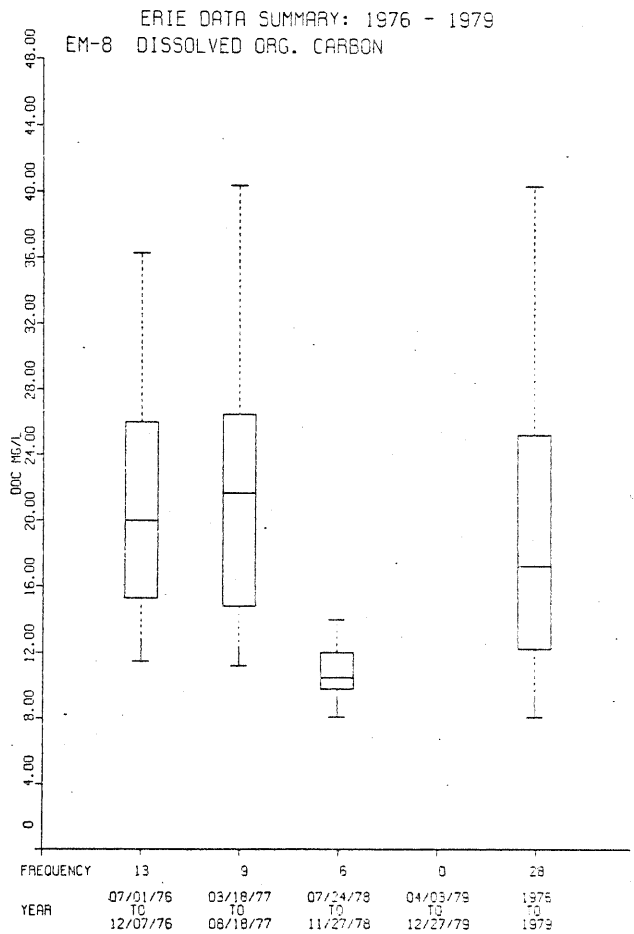
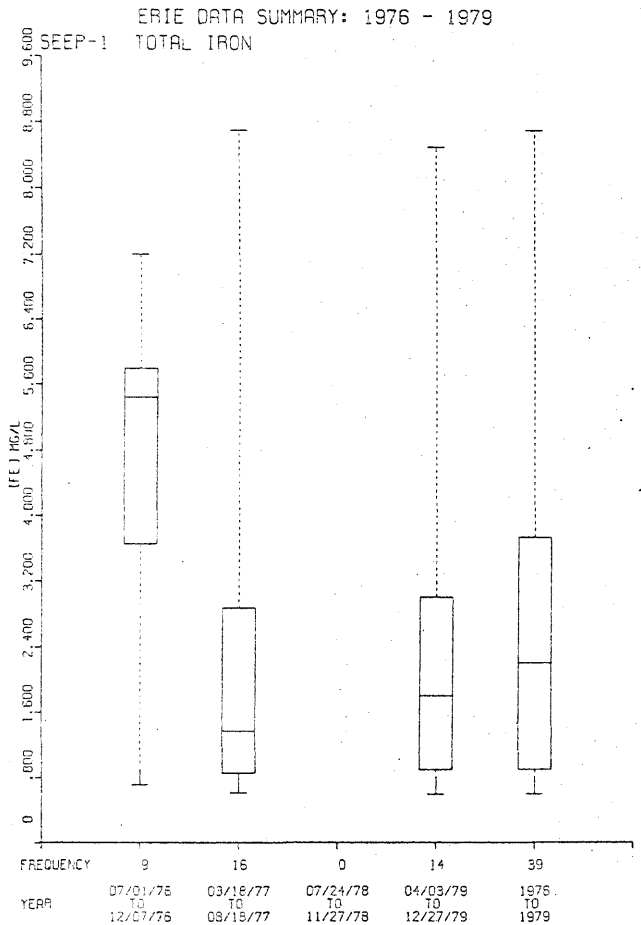
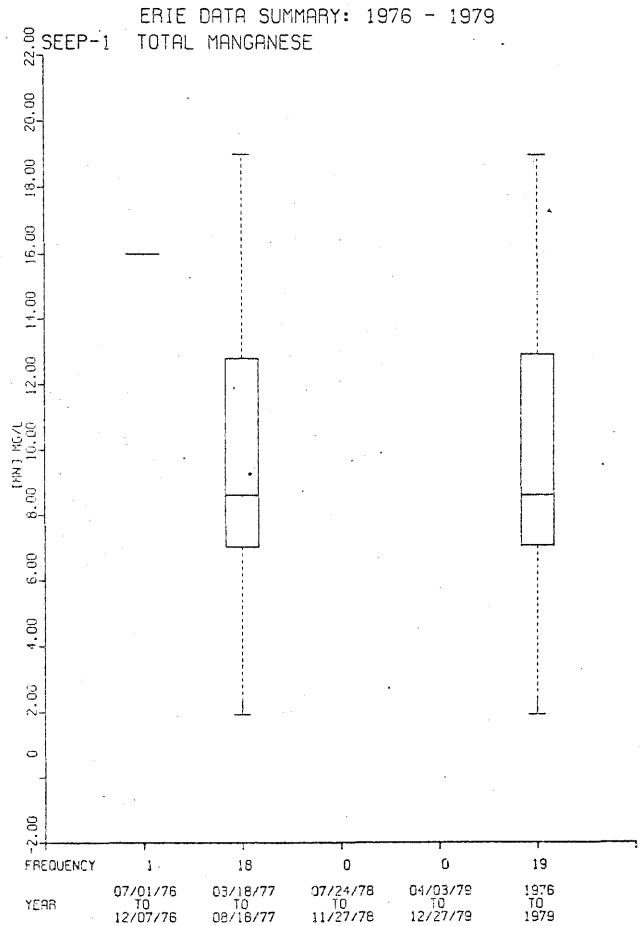

ERIE DATA SUMMARY: 1976 - 1979
EM-8 DISSOLVED ORG. CARBON
DOC MG/L
0
4.00
8.00
12.00
16.00
20.00
24.00
28.00
32.00
36.00
40.00
44.00
48.00
FREQUENCY 13 9 6 0 28
YEAR 07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

FIGURE 20


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL IRON
[FE] MG/L
0
.800
1.600
2.400
3.200
4.000
4.800
5.600
6.400
7.200
8.000
8.800
9.600
FREQUENCY 9 16 0 14 39
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL MANGANESE
[MN] MG/L
-2.00
0
2.00
4.00
6.00
8.00
10.00
12.00
14.00
16.00
18.00
20.00
22.00
FREQUENCY 1 18 0 0 19
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

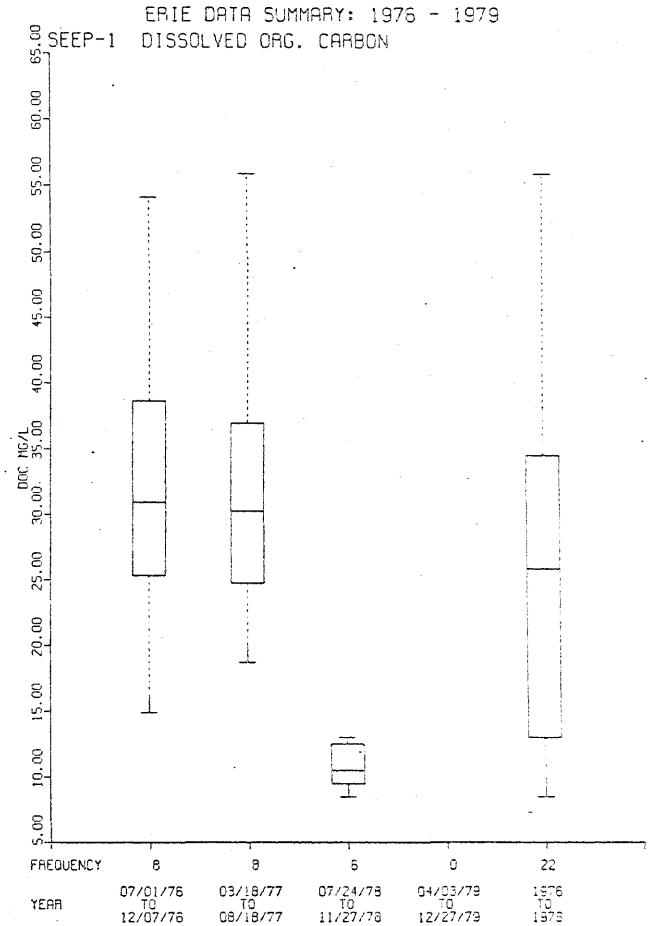
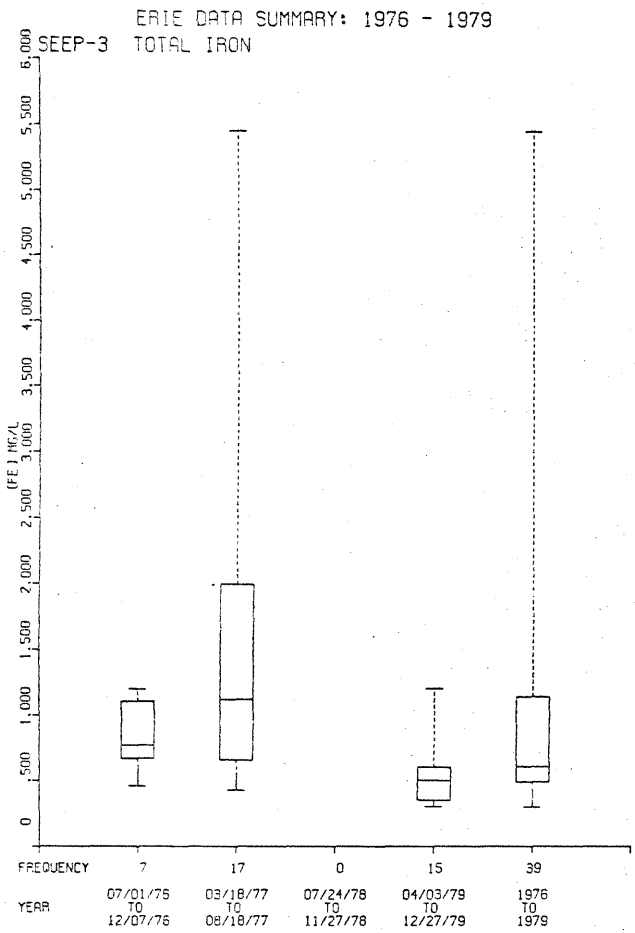
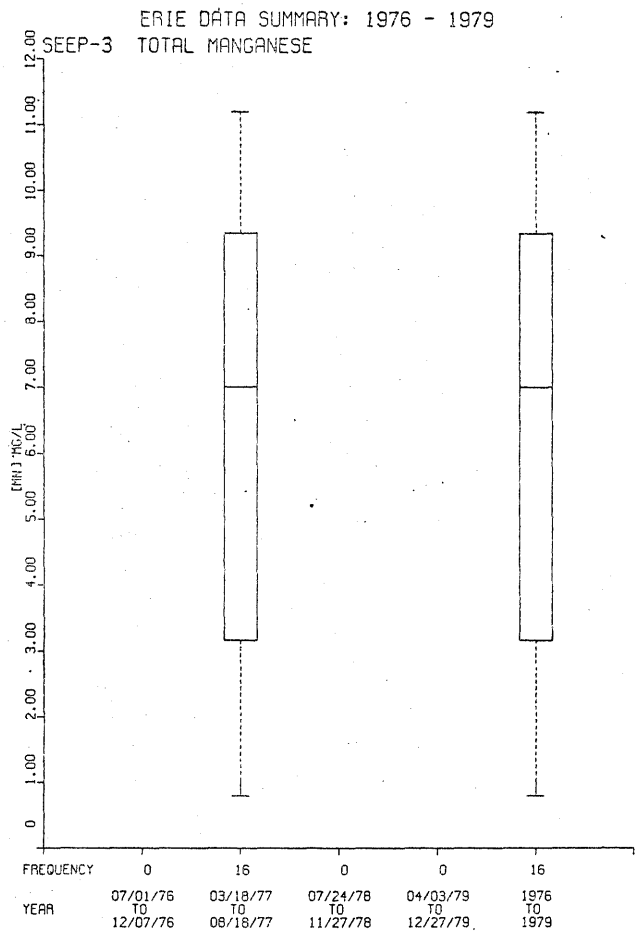

ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 DISSOLVED ORG. CARBON
DOC MG/L
5.00
10.00
15.00
20.00
25.00
30.00
35.00
40.00
45.00
50.00
55.00
60.00
65.00
FREQUENCY 8 8 6 0 22
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

FIGURE 21


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL IRON
(FE) MG/L
0
.500
1.000
1.500
2.000
2.500
3.000
3.500
4.000
4.500
5.000
5.500
6.000
FREQUENCY 7 17 0 15 39
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL MANGANESE
(MN) MG/L
0
1.00
2.00
3.00
4.00
5.00
6.00
7.00
8.00
9.00
10.00
11.00
12.00
FREQUENCY 0 16 0 0 16
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

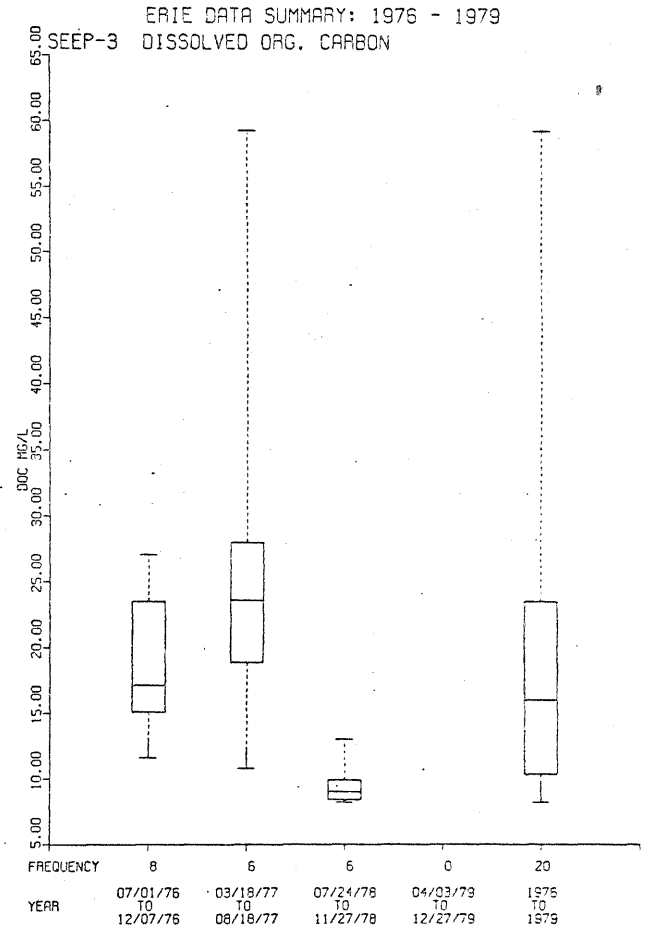
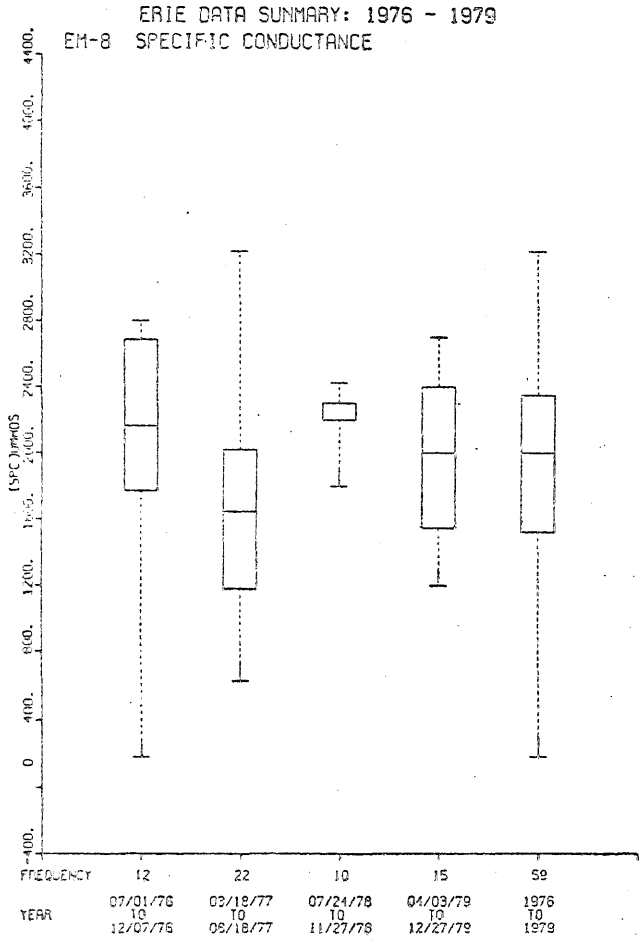
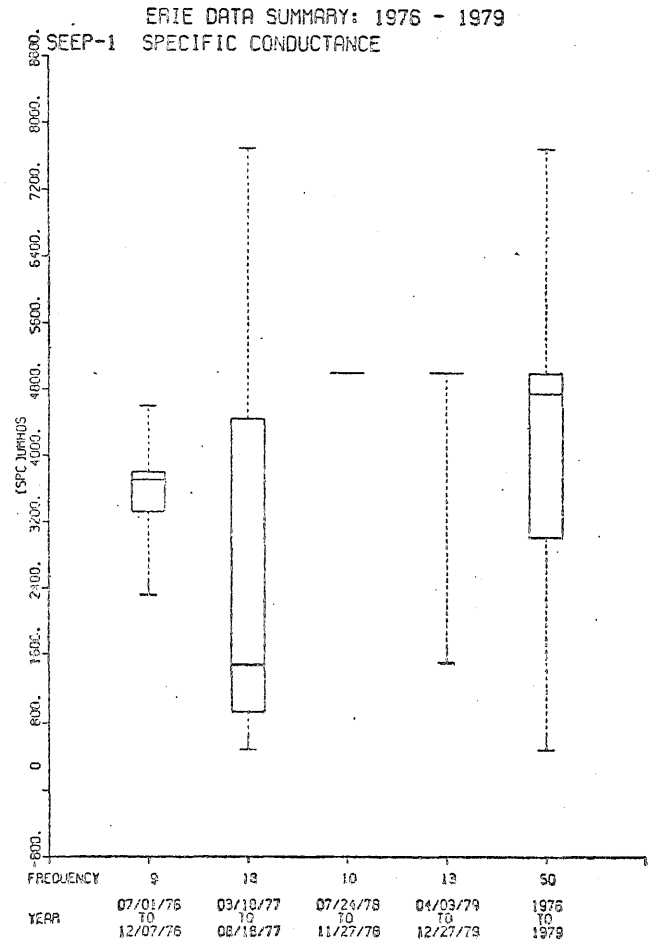
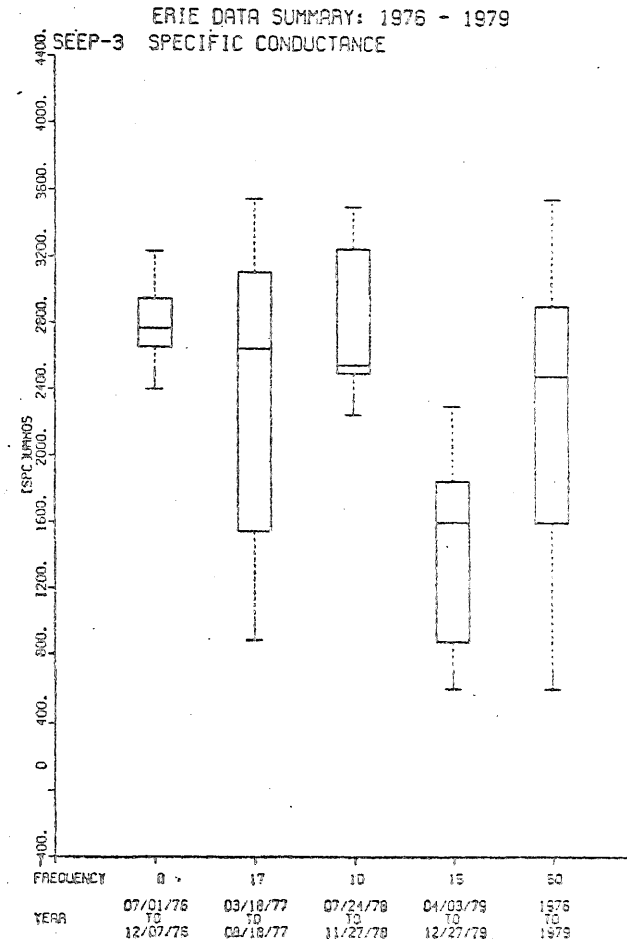

ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 DISSOLVED ORG. CARBON
DOC MG/L
5.00
10.00
15.00
20.00
25.00
30.00
35.00
40.00
45.00
50.00
55.00
60.00
65.00
FREQUENCY 8 6 6 0 20
YEAR
07/01/76 TO 12/07/76
03/18/77 TO 08/18/77
07/24/78 TO 11/27/78
04/03/79 TO 12/27/79
1976 TO 1979

FIGURE 22







Seep 1, 1978-1979, specific conductance in many of the samples was greater than 5000 which exceeded the range of the meter.

FIGURE 23

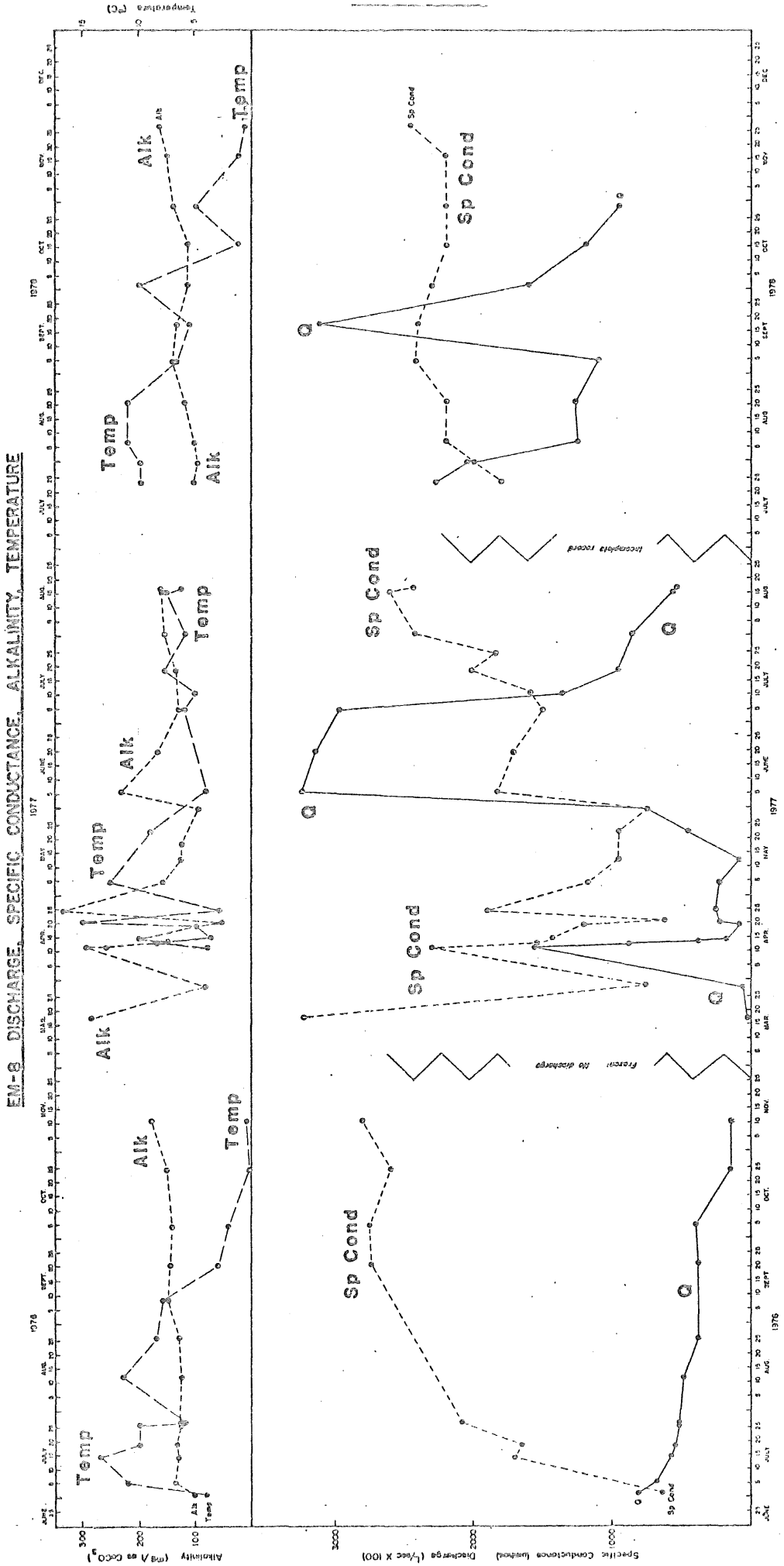

EM-8 DISCHARGE, SPECIFIC CONDUCTANCE, ALKALINITY, TEMPERATURE
Temperature (°C)
Alkalinity (mg/l as $CaCO_3$)
Specific Conductance (µmhos) Discharge (l/sec x 100)
Temp
Alk
Sp Cond
Q
Incomplete record
Frozen, No discharge
1976
1977
1978

FIGURE 24

SEEP I. – DISCHARGE, SPECIFIC CONDUCTANCE, ALKALINITY, TEMPERATURE

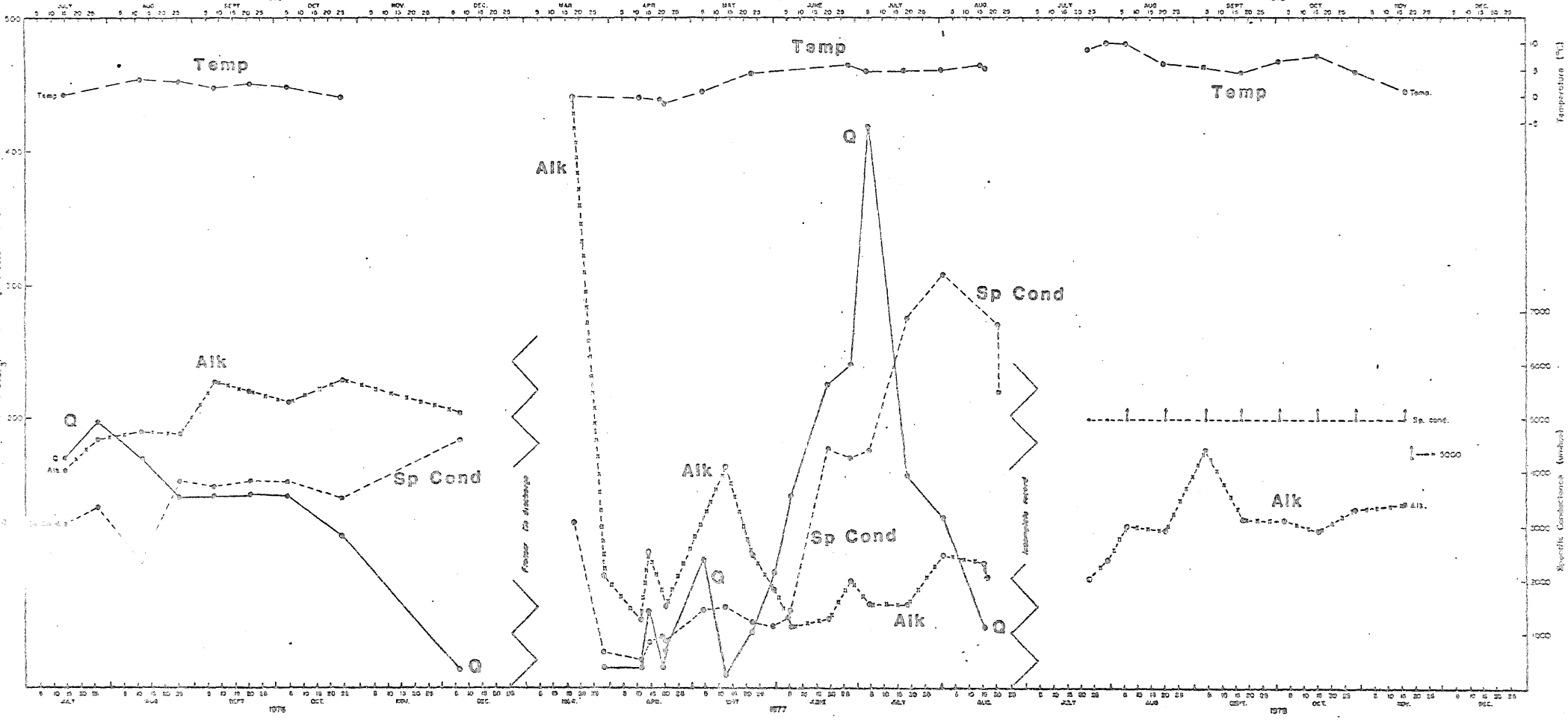


FIGURE 25

SEEP 3.-DISCHARGE, SPECIFIC CONDUCTANCE, ALKALINITY, TEMPERATURE

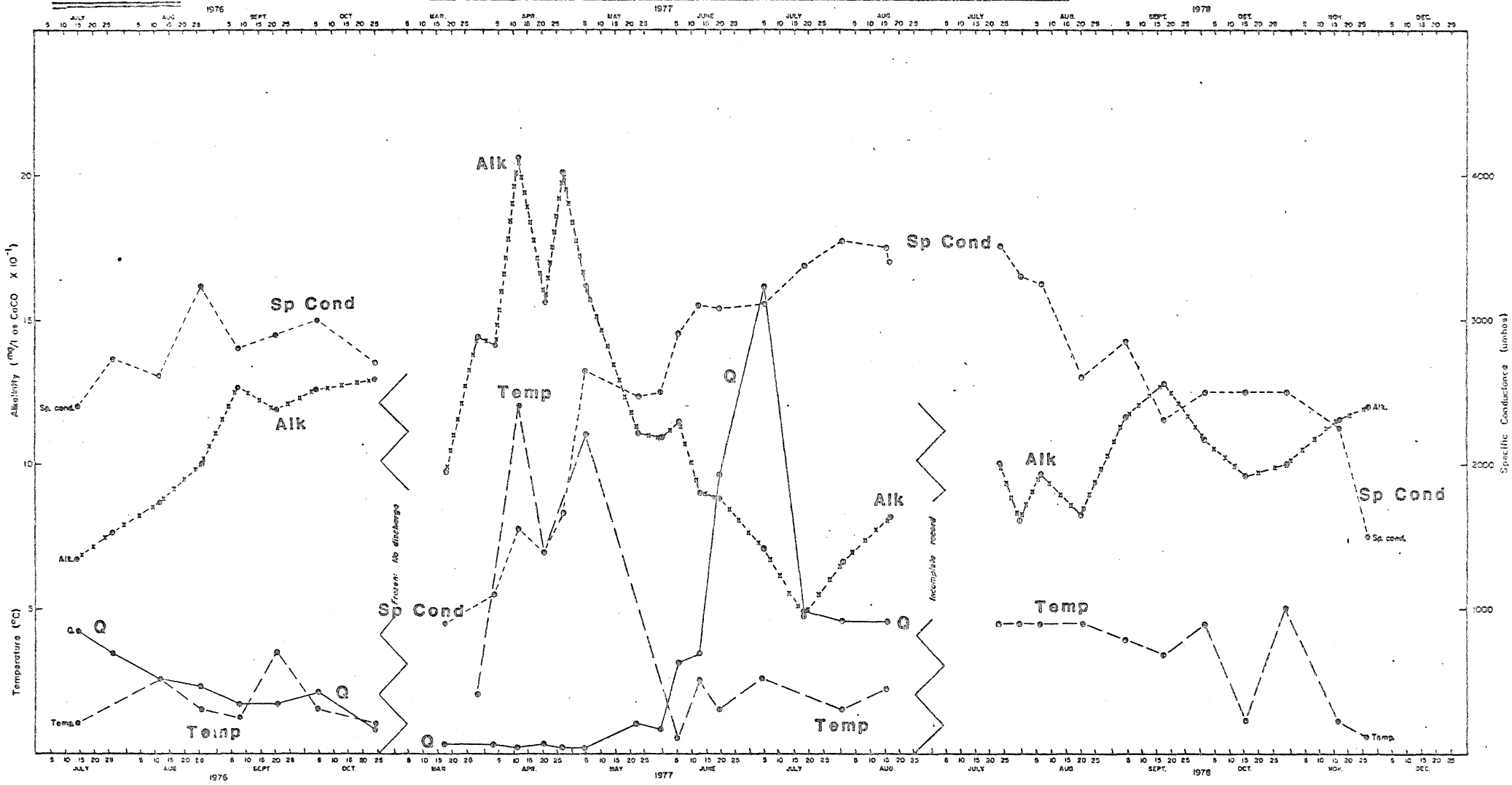
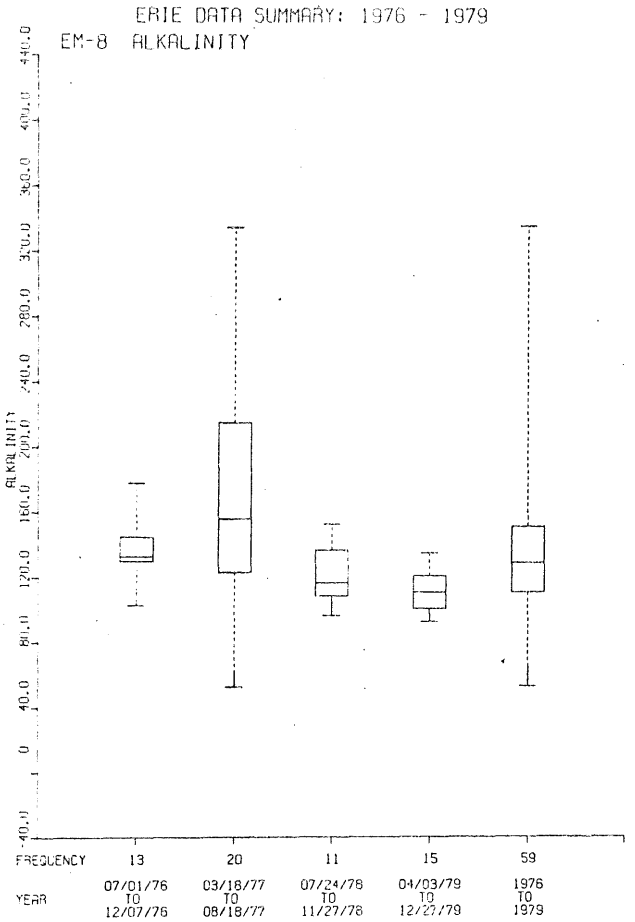
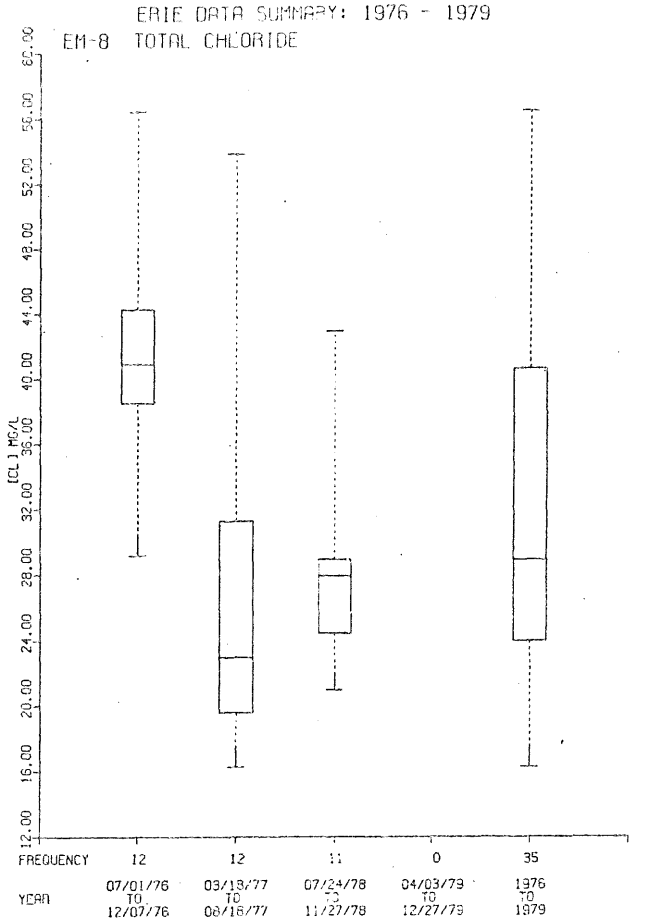
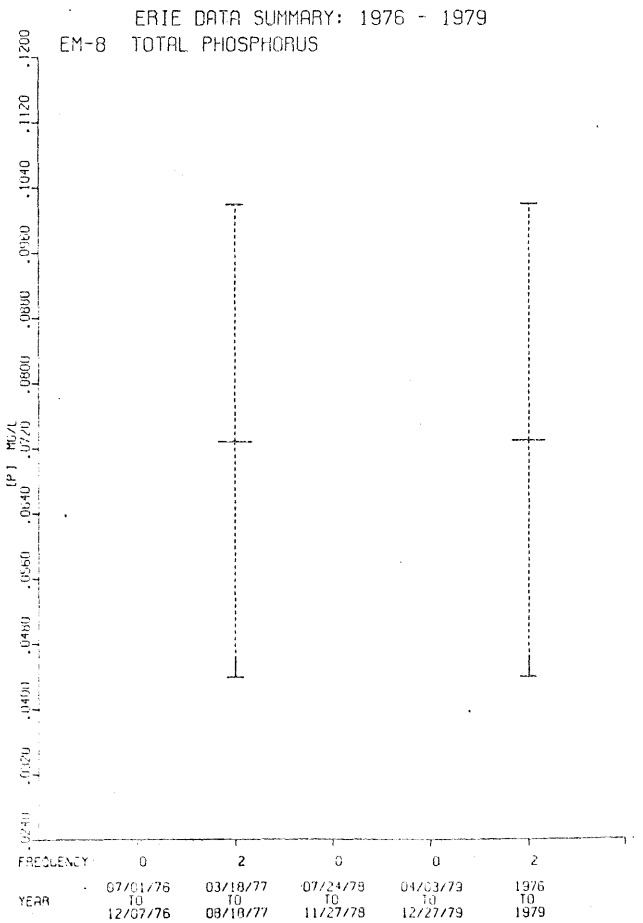


FIGURE 26


ERIE DATA SUMMARY: 1976 - 1979
EM-8 ALKALINITY
ALKALINITY
FREQUENCY 13 20 11 15 59
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL CHLORIDE
[CL] MG/L
FREQUENCY 12 12 11 0 35
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-8 TOTAL PHOSPHORUS
[P] MG/L
FREQUENCY 0 2 0 0 2
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

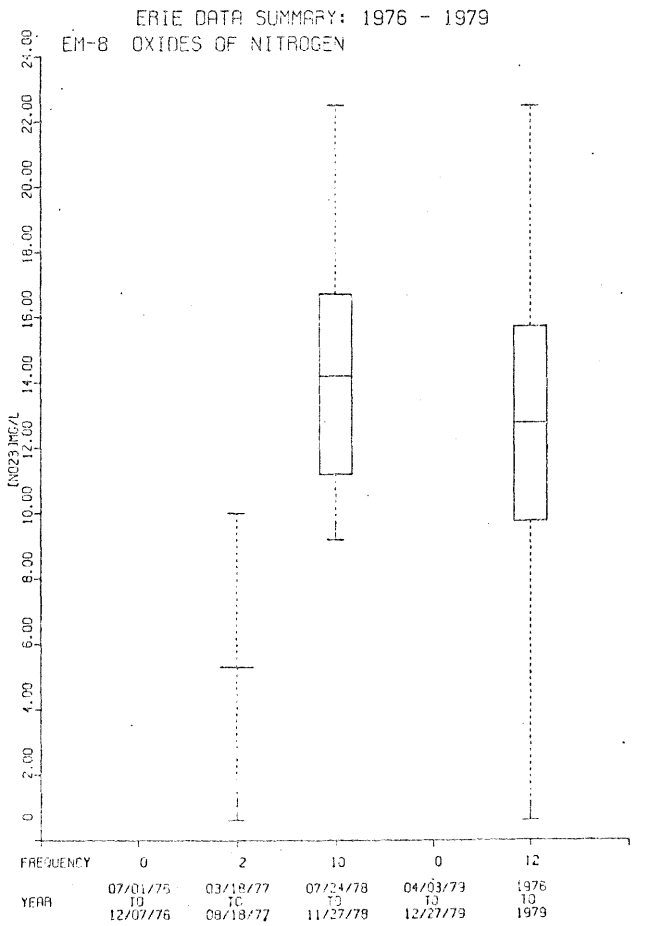
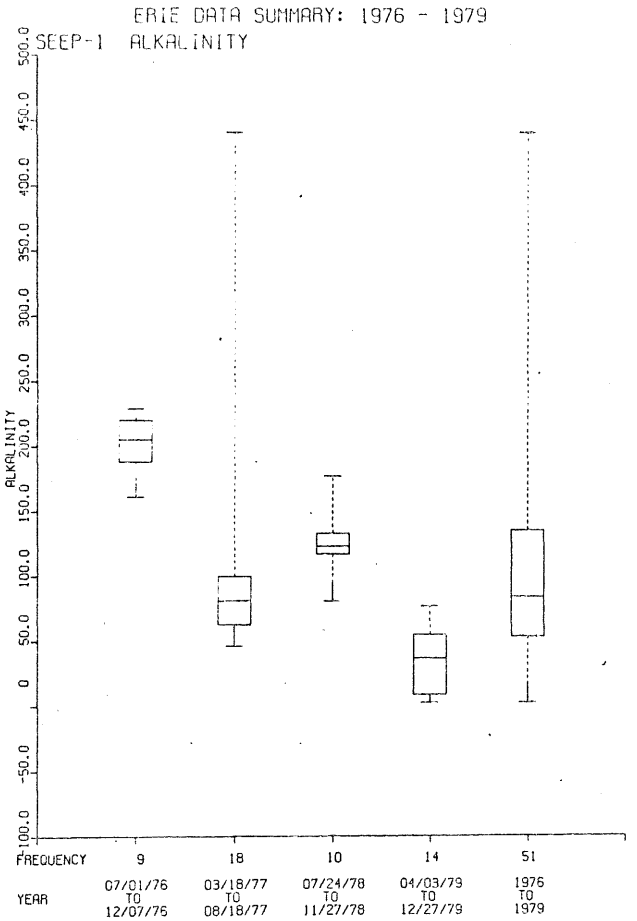
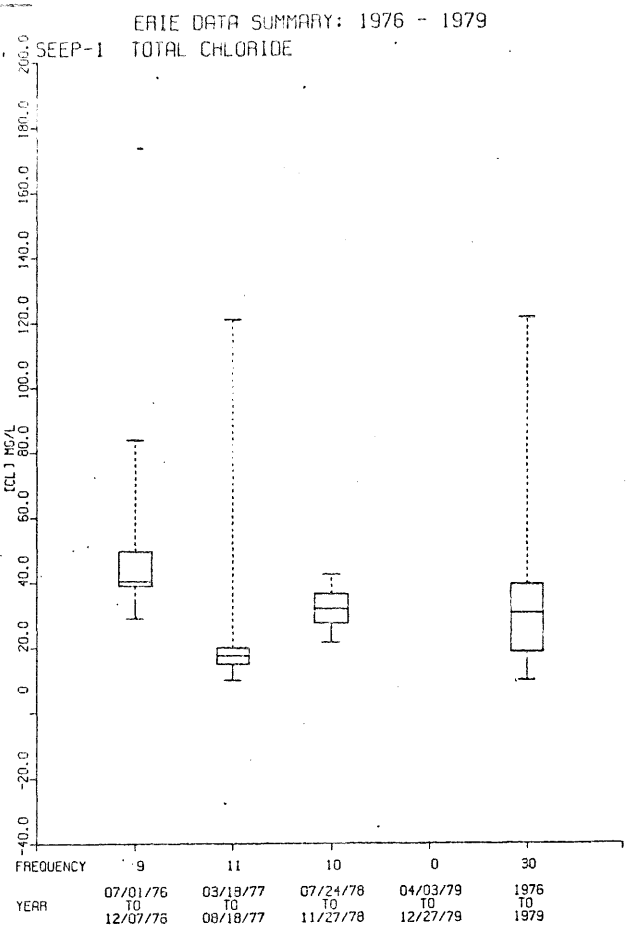
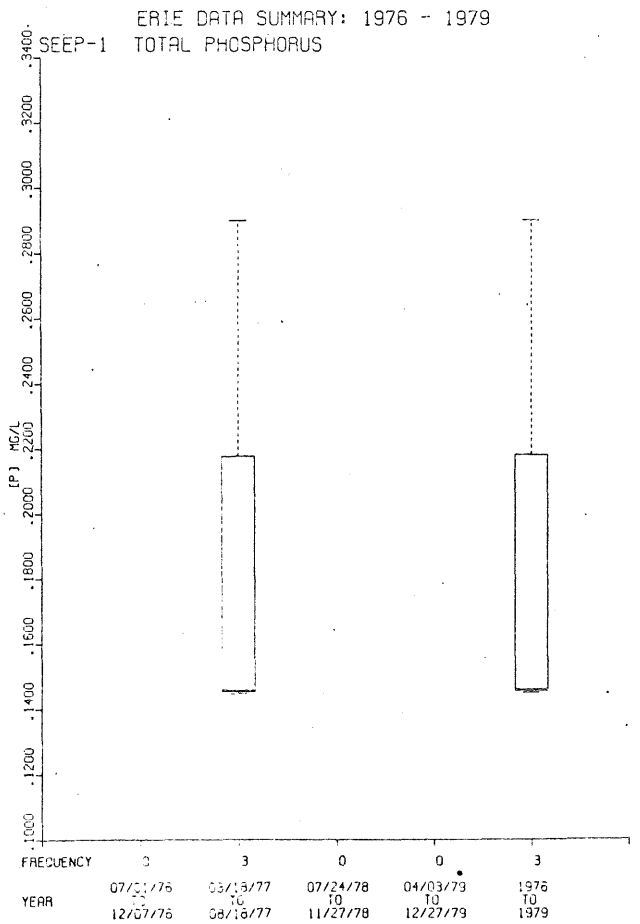
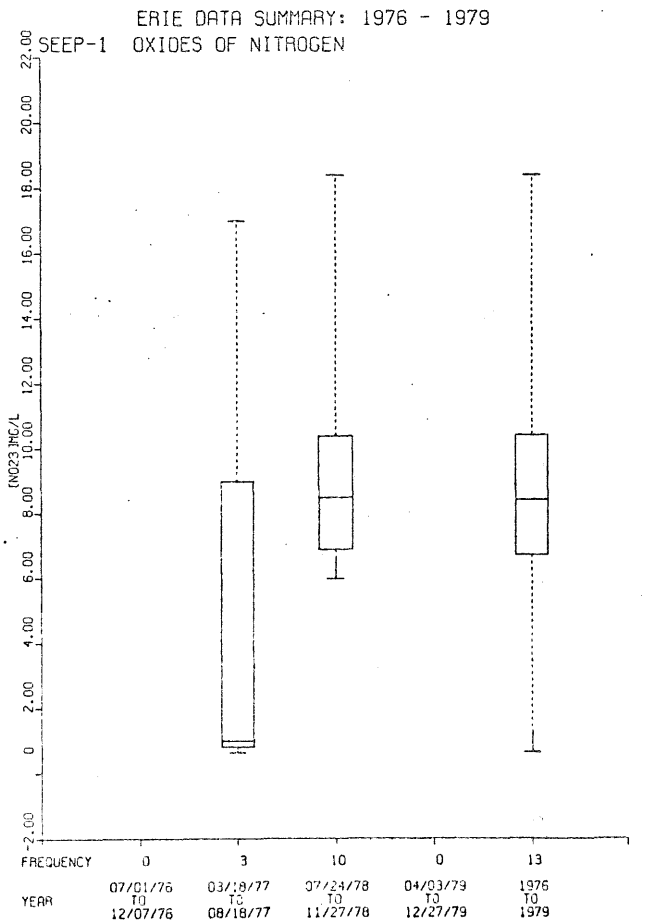

ERIE DATA SUMMARY: 1976 - 1979
EM-8 OXIDES OF NITROGEN
[NO23] MG/L
FREQUENCY 0 2 10 0 12
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

FIGURE 27


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 ALKALINITY
ALKALINITY
500.0
450.0
400.0
350.0
300.0
250.0
200.0
150.0
100.0
50.0
0
-50.0
-100.0
FREQUENCY 9 18 10 14 51
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL CHLORIDE
[CL] MG/L
200.0
180.0
160.0
140.0
120.0
100.0
80.0
60.0
40.0
20.0
0
-20.0
-40.0
FREQUENCY 9 11 10 0 30
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 TOTAL PHOSPHORUS
[P] MG/L
.3400
.3200
.3000
.2800
.2600
.2400
.2200
.2000
.1800
.1600
.1400
.1200
.1000
FREQUENCY 0 3 0 0 3
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-1 OXIDES OF NITROGEN
[NO23] MG/L
22.00
20.00
18.00
16.00
14.00
12.00
10.00
8.00
6.00
4.00
2.00
0
-2.00
FREQUENCY 0 3 10 0 13
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

at Em 8 were somewhat warmer since this station was located about 100 meters downstream of the point of discharge from the stockpile. The discharge from the stockpiles was colder than ambient temperatures because the stockpiled rock acted as an insulator.

The median chloride concentration for the study period was about 30 mg/l at each of the stockpiles. Chloride concentrations decreased over the study period, a trend most apparent at Seep 3 (Figure 28). Data on phosphorus were too limited to warrant discussion. Observed concentrations ranged from 0.04 - 7 mg/l.

Median NO_2-NO_3 concentrations at the stockpiles ranged from 6 to 13 mg/l. (Figures 26-28). Residual nitrate from blasting materials may have contributed to the total NO_2-NO_3 release.

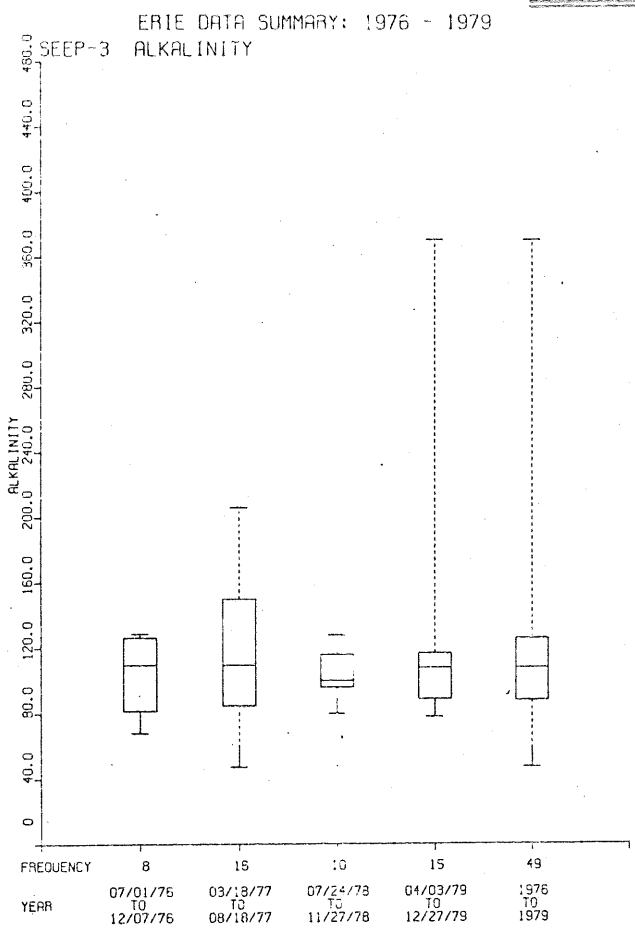
4.2 STREAM STATIONS

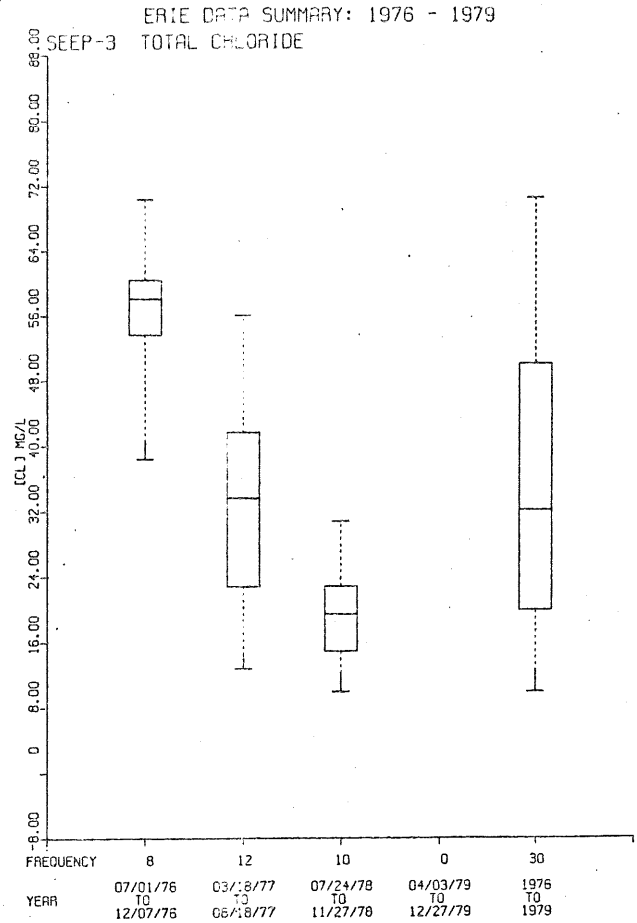
The stream stations were monitored in 1976 and 1977 except for Em 1 which has been monitored up to the present. The stations included two flows from mine dewatering (Em 6, Em 2), one source of natural runoff (Em 4), and three stations along the main stream branch (Em 5, Em 3, Em 1, Figure 29). A summary of the observed median concentrations and flows for 1976-1977 is presented in Table 4. The range of concentrations and flows, and mass fluxes at the various stations is presented in Figure 30.

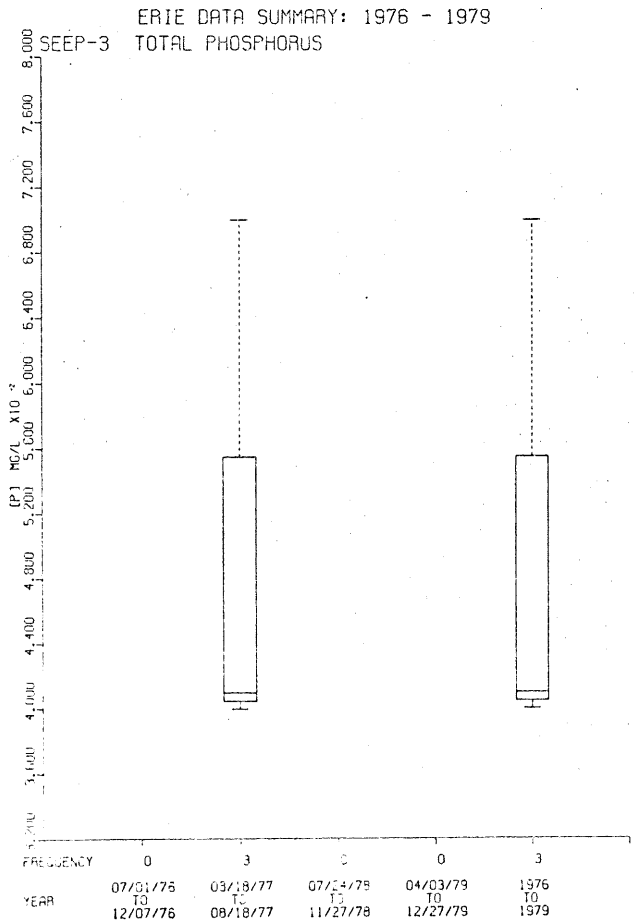
4.2.1 Mine dewatering

Mine dewatering discharges, 011 and 012, flow through stations Em 6 and Em 2, respectively, and provide flow input an order of magnitude higher than the input from stockpile runoff (Figure 30). The median pH values at both Em 6 and Em 2 was approximately 7.6, and median sulfate concentrations were 110 and 340 mg/l, respectively (Figures 31, 32). Copper and nickel

FIGURE 28


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 ALKALINITY
ALKALINITY
0
40.0
80.0
120.0
160.0
200.0
240.0
280.0
320.0
360.0
400.0
440.0
480.0
FREQUENCY 8 16 10 15 49
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL CHLORIDE
[CL] MG/L
-8.00
0
8.00
16.00
24.00
32.00
40.00
48.00
56.00
64.00
72.00
80.00
88.00
FREQUENCY 8 12 10 0 30
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 TOTAL PHOSPHORUS
[P] MG/L X10 -2
4.000
4.400
4.800
5.200
5.600
6.000
6.400
6.800
7.200
7.600
8.000
FREQUENCY 0 3 0 0 3
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

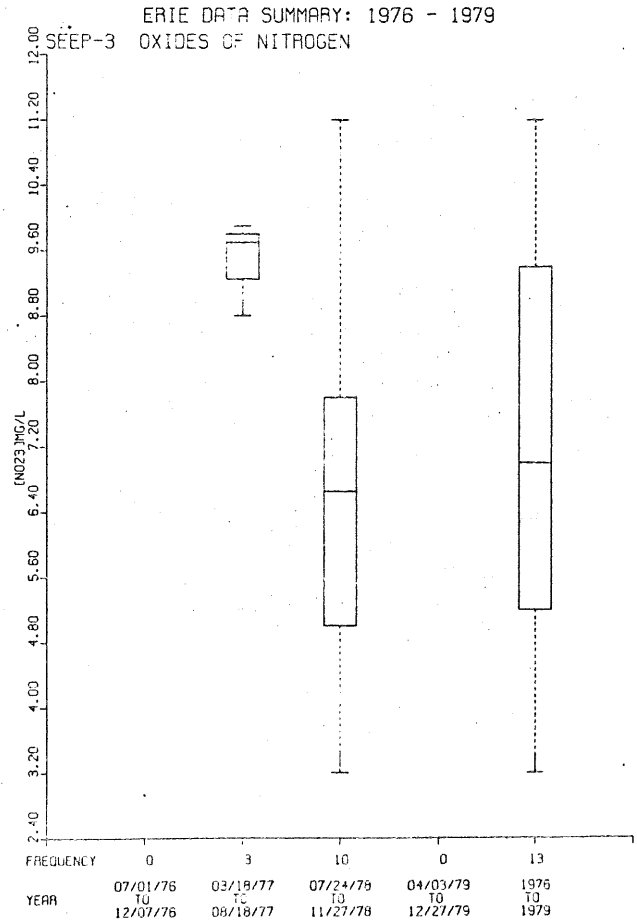

ERIE DATA SUMMARY: 1976 - 1979
SEEP-3 OXIDES OF NITROGEN
[NO23] MG/L
2.40
3.20
4.00
4.80
5.60
6.40
7.20
8.00
8.80
9.60
10.40
11.20
12.00
FREQUENCY 0 3 10 0 13
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

FIGURE 29 - Stream sampling sites.

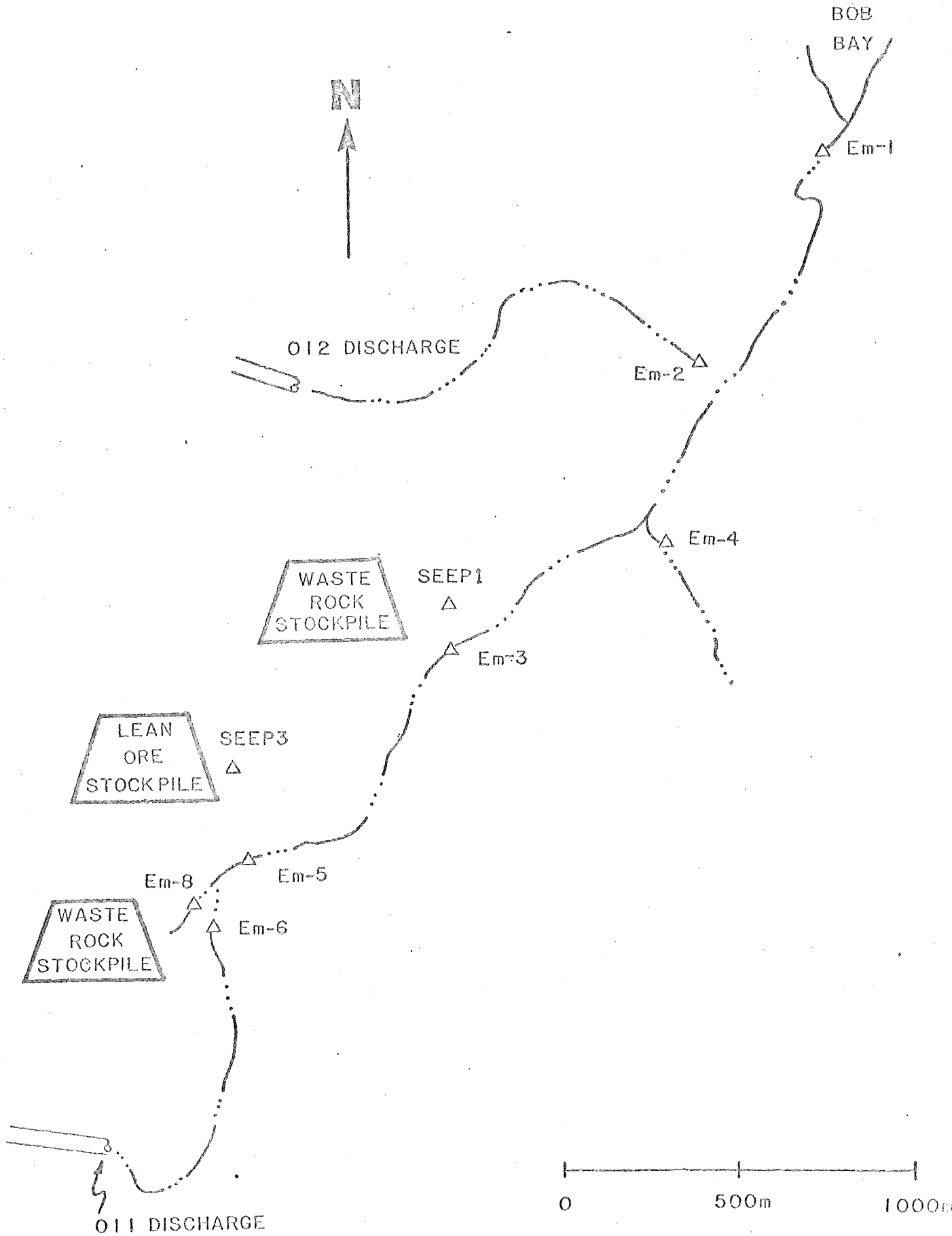
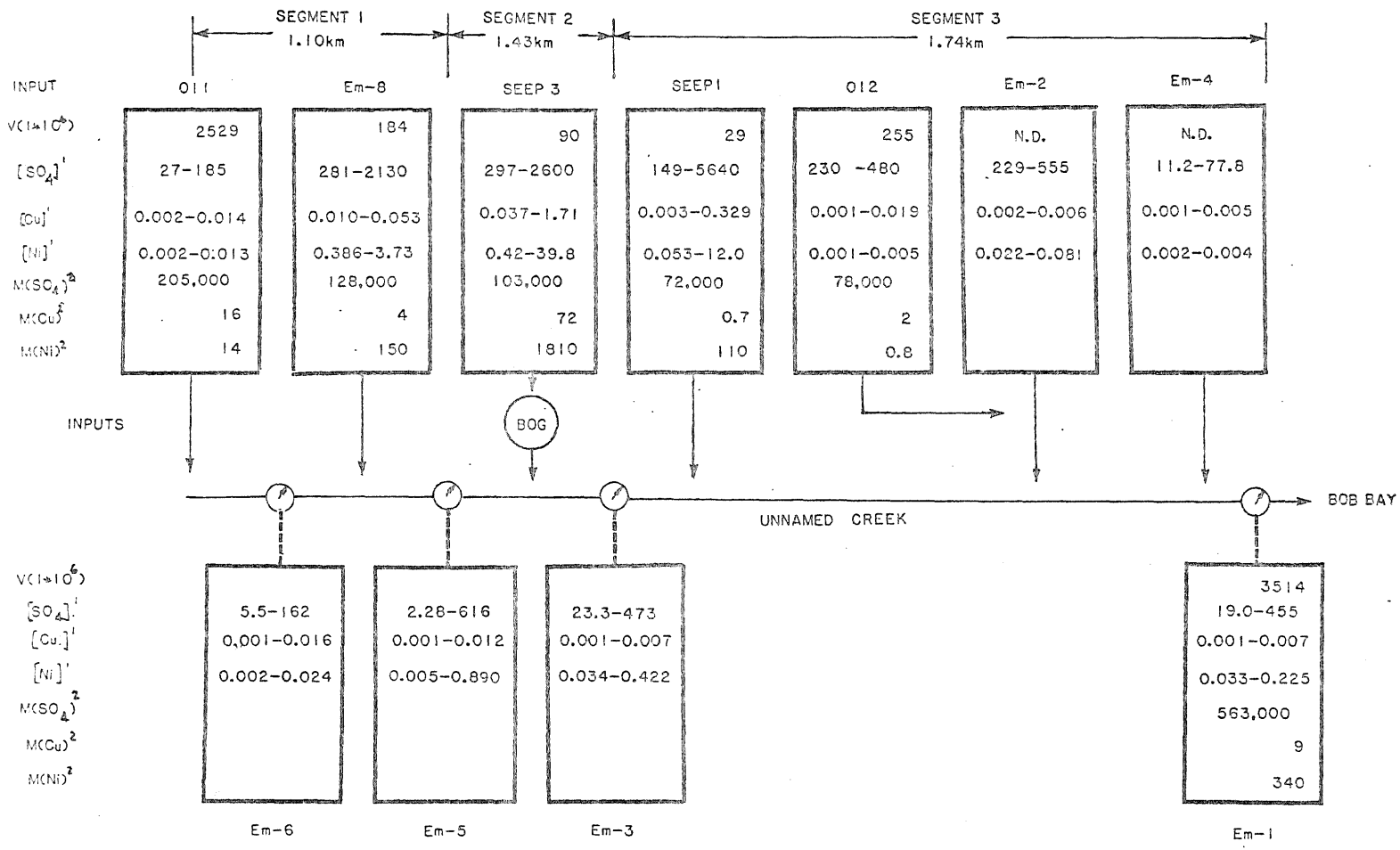


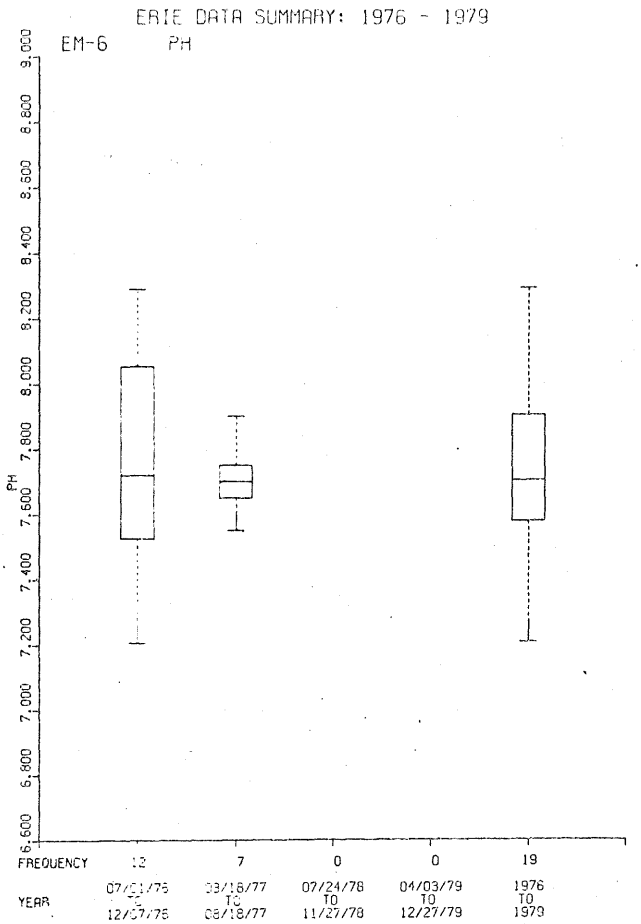
Figure 30. Concentrations and mass flux for Unnamed Creek inputs and stream stations (not to scale)

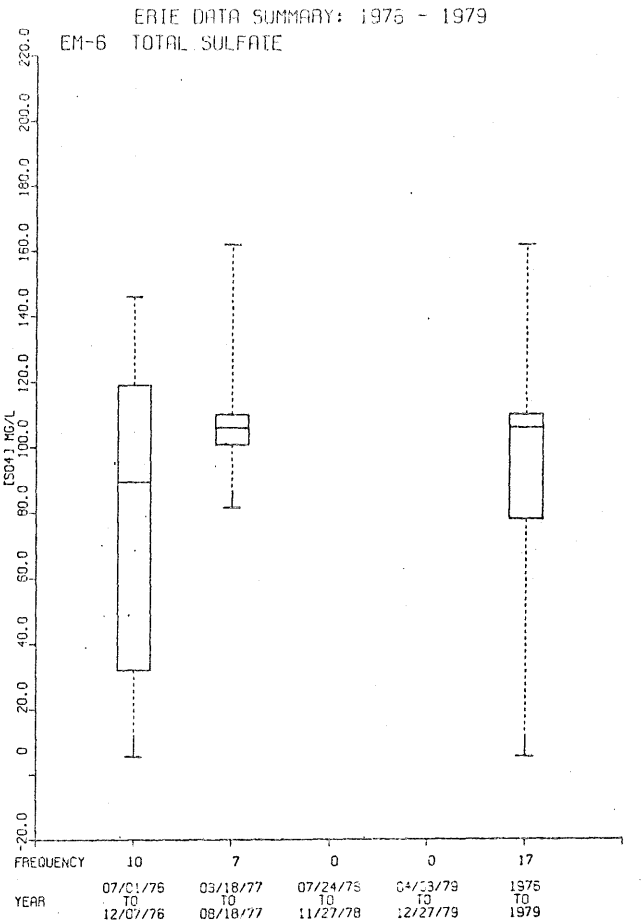


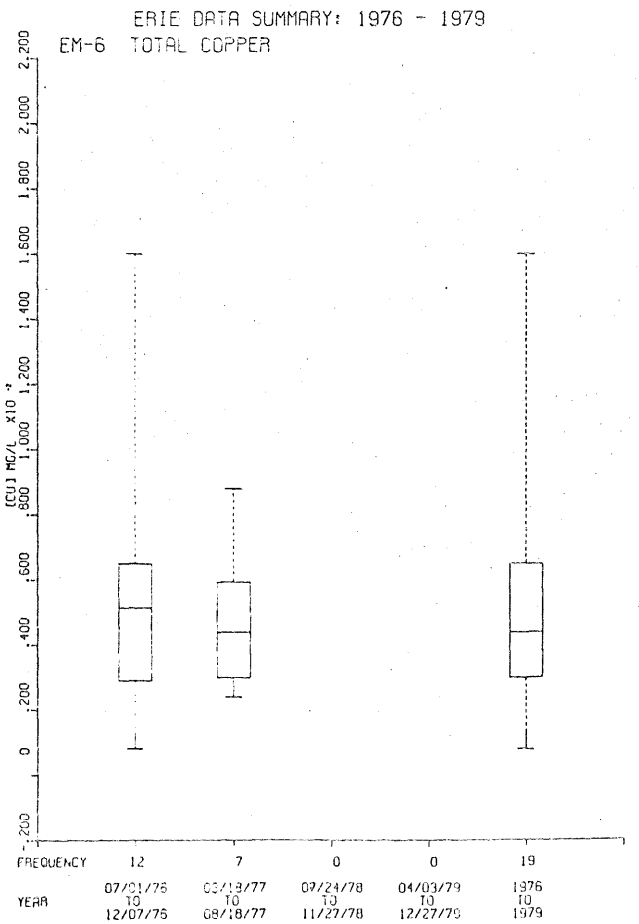
1. mg/l, range of concentrations, total metals

2. mass flux(kg) from 7/76-8/77

FIGURE 31


ERIE DATA SUMMARY: 1976 - 1979
EM-6 PH
PH
FREQUENCY 12 7 0 0 19
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-6 TOTAL SULFATE
[SO4] MG/L
FREQUENCY 10 7 0 0 17
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-6 TOTAL COPPER
[CU] MG/L X10 -3
FREQUENCY 12 7 0 0 19
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979

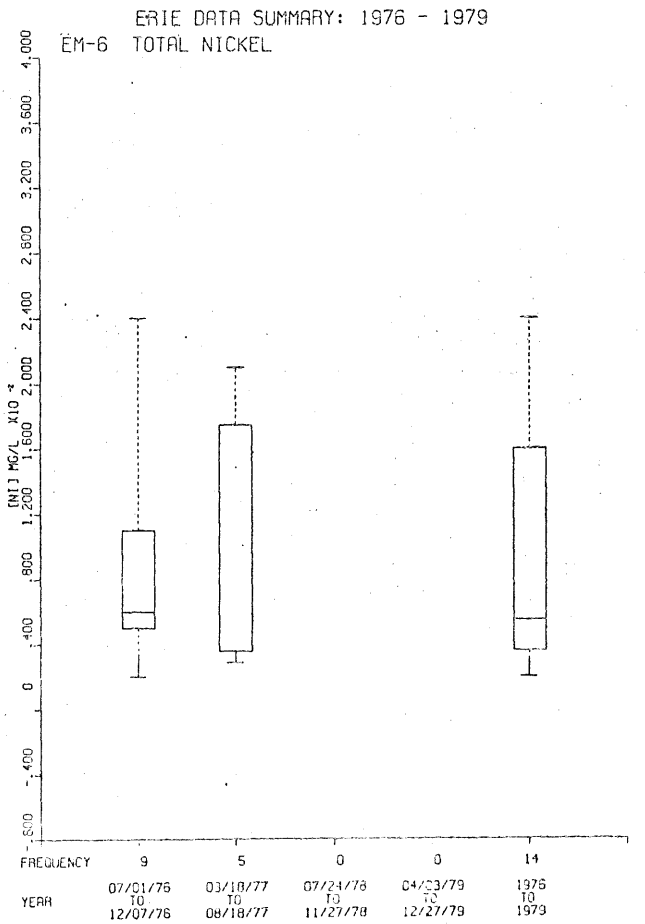
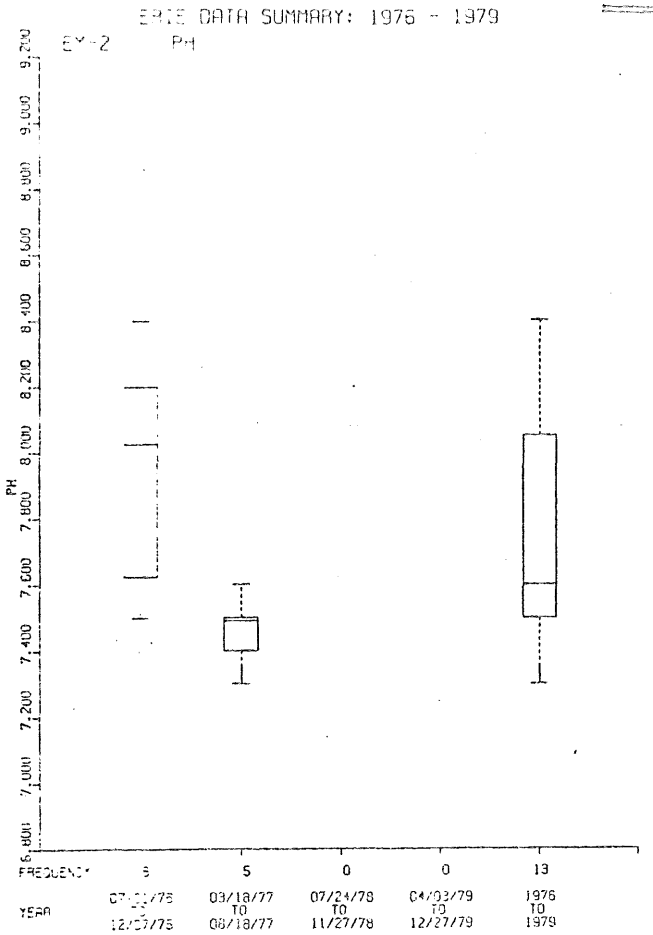
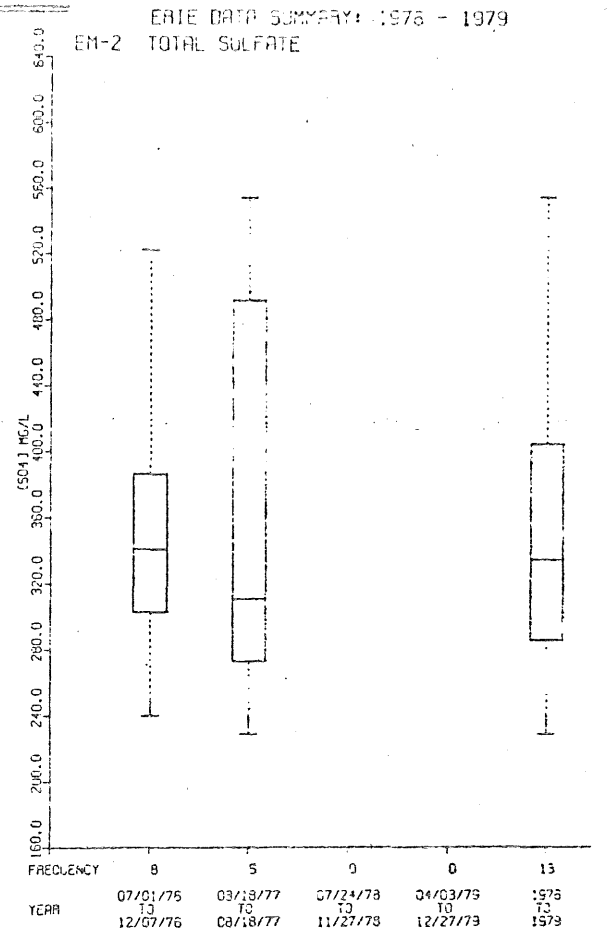
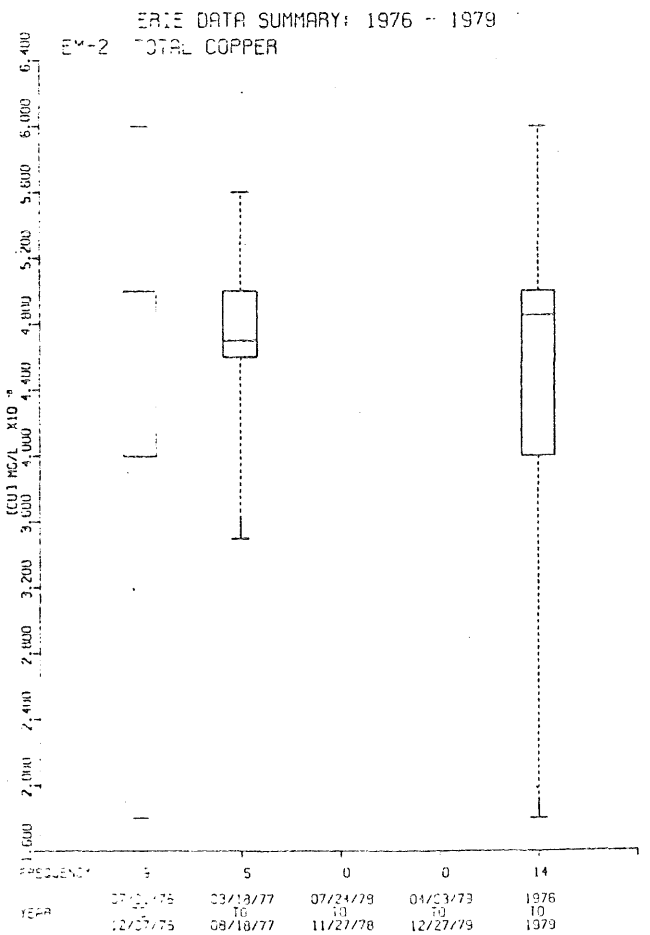
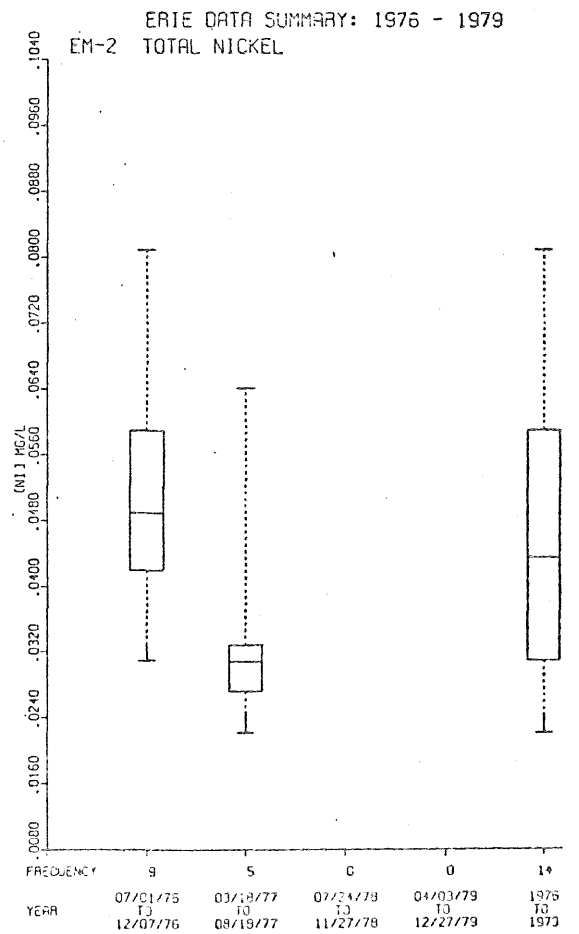

ERIE DATA SUMMARY: 1976 - 1979
EM-6 TOTAL NICKEL
[NI] MG/L X10 -3
FREQUENCY 9 5 0 0 14
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979

FIGURE 32


ERIE DATA SUMMARY: 1976 - 1979
EM-2 PH
PH
FREQUENCY 8 5 0 0 13
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-2 TOTAL SULFATE
(SO4) MG/L
FREQUENCY 8 5 0 0 13
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-2 TOTAL COPPER
(CU) MG/L X10 -3
FREQUENCY 8 5 0 0 14
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-2 TOTAL NICKEL
(NI) MG/L
FREQUENCY 9 5 0 0 14
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979

concentrations were quite low in comparison with the stockpile runoff. Median copper concentrations at both sites were 4-5 µg/l, but nickel concentrations were significantly higher at Em 2 (40µg/l) than at Em 6 (5µg/l, Figures 31, 32).

4.2.2. Natural runoff

Station Em 4 was intended to represent a natural runoff input. This objective was not attained because backwater conditions occurred and affected the water quality at the site. Few data were collected at this site, the maximum number of data points being 8.

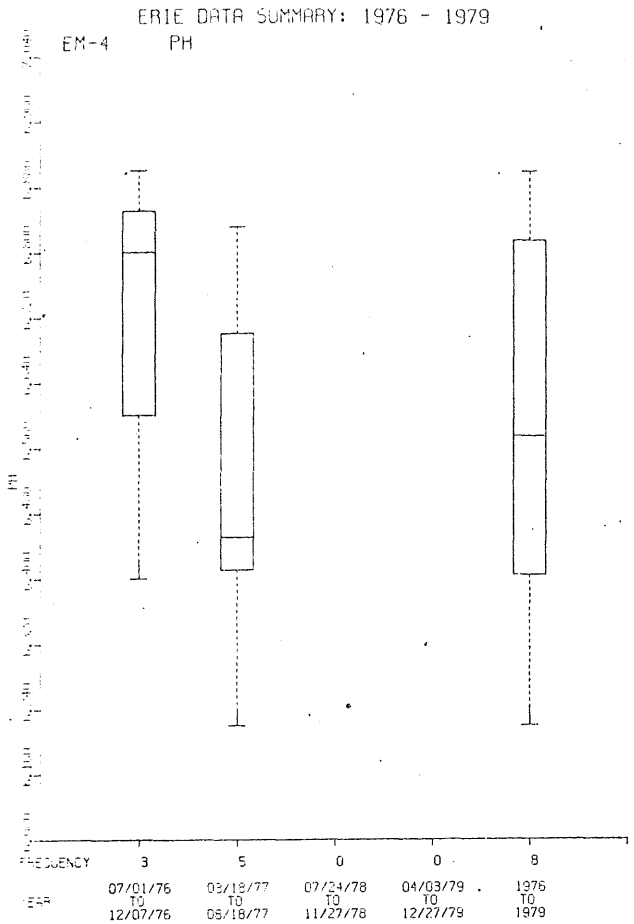
The pH at Em 4 was lower than in the stockpile runoff and mine dewatering. The median pH from 1976 to 1977 was about 6.6 (Figure 33). The three sulfate samples analyzed were all above background levels (Figure 33, Table 2). Median copper and nickel concentrations were 2 and 3 µg/l, respectively.

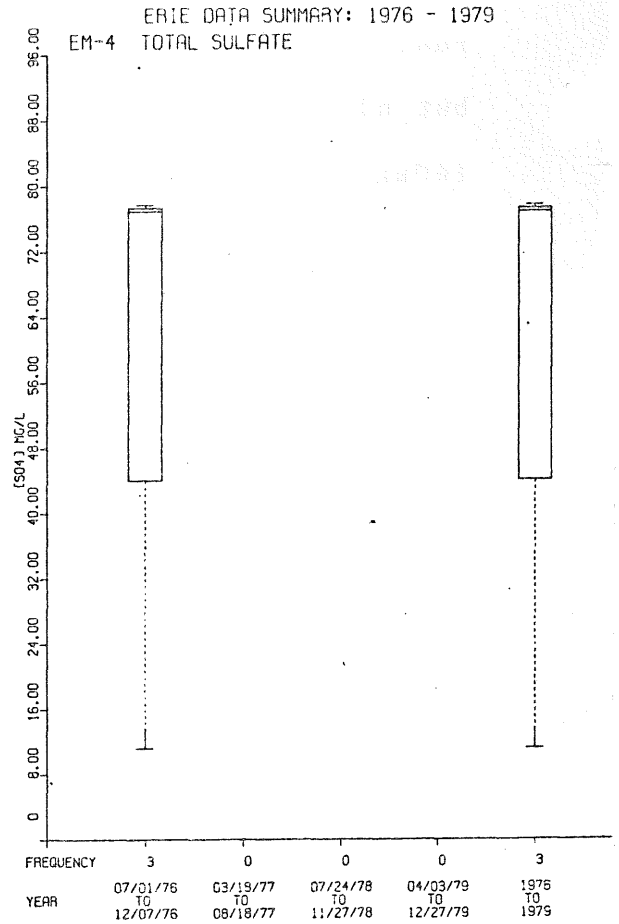
4.2.3 Stations along the main stream branch

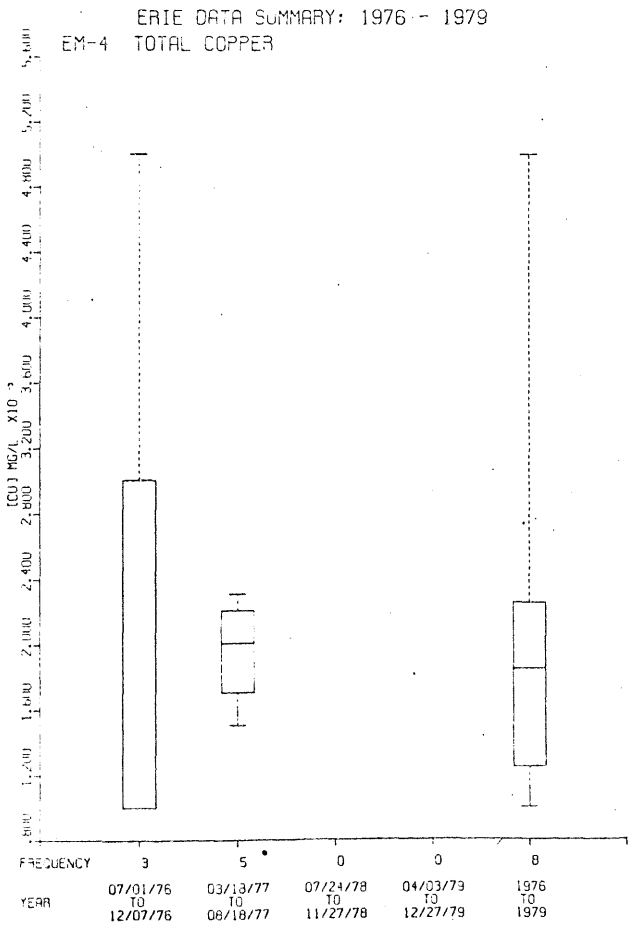
Three stations were located along the main branch of Unnamed Creek. Em 5 was located at the confluence of waste rock stockpile runoff (Em 8) and a mine dewatering discharge (Em 6, Figure 29). Em 3 was located about 1.4 km downstream from Em 5 and received flow from Seep 3 (lean ore stockpile runoff) in addition to the flow from Em 5 and natural runoff. Em 1 was located at the mouth of Unnamed Creek, about 1.7 km downstream from Em 3. In addition to the flow from Em 3, Em 1 also received waste rock stockpile runoff from Seep 1, natural runoff from Em 4 and other parts of the watershed, and mine dewatering from Em 2.

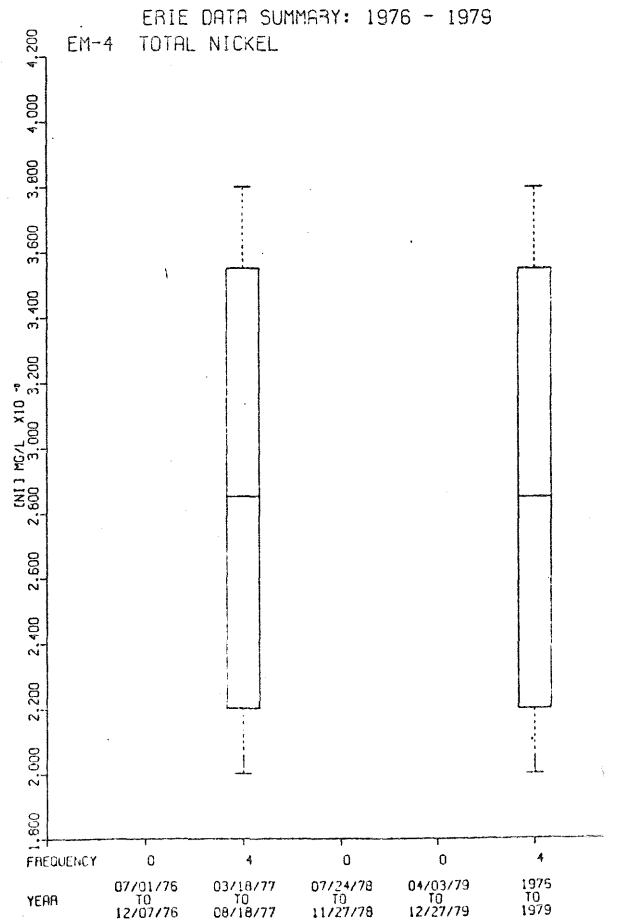
Going downstream from Em 5 to Em 3 and Em 1 the median flow

FIGURE 33


ERIE DATA SUMMARY: 1976 - 1979
EM-4 PH
PH
FREQUENCY 3 5 0 0 8
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-4 TOTAL SULFATE
(SO4) MG/L
0 8.00 16.00 24.00 32.00 40.00 48.00 56.00 64.00 72.00 80.00 88.00 96.00
FREQUENCY 3 0 0 0 3
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-4 TOTAL COPPER
(CU) MG/L X10 -3
FREQUENCY 3 5 0 0 8
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-4 TOTAL NICKEL
(NI) MG/L X10 -3
1.800 2.000 2.200 2.400 2.600 2.800 3.000 3.200 3.400 3.600 3.800 4.000 4.200
FREQUENCY 0 4 0 0 4
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

rate increased from 30 to 90 to 110 l/s (Table 4). The pH at all three sites was about the same, with median pH values ranging from 7.5 to 7.65 (Figures 34, 35, 37). Median sulfate concentrations at the three sites were 122, 155, and 160 mg/l. Mass balance calculations indicated that virtually 100% of the sulfate input was transported to the mouth of the creek (14). Copper concentrations were low at all three sites (Figures 34, 35, 37). Median concentrations at Em 5, Em 3, and Em 1 were 5, 3, and 2 μg/l. Median nickel concentrations were 51, 110, and 87 μg/l. Mass transport calculations indicated that nickel transport through the system was more efficient than copper transport (14). The majority of input metal removal occurred as flow from Seep 3 passed through a bog, and was the result of trace metal sequestration by peat (12-14).

4.2.4. Transport from the creek: Data from Em 1

As was previously mentioned, the median flow at Em 1 was 110 l/s during 1976 and 1977. This flow had an elevated ionic content as is indicated by the specific conductance (Figure 36). The high specific conductance is the result of high concentrations of sulfate, calcium, and magnesium (Figure 37). Concentrations of copper (3 μg/l and nickel (160 μg/l) were elevated above background levels (Figure 38, Table 2). Cobalt and zinc concentrations were typically in the range of 1-2.5 μg/l. Median iron and manganese concentrations were 0.2 and 0.05 mg/l, and the median DOC concentration was 12 mg/l (Figure 39). The alkalinity at Em 1 was about 100 mg/l as $CaCO_3$, and typical chloride concentrations ranged from 20-30 mg/l. The three phosphorus analyses ranged from 0.02 to 0.12 mg/l and the NO_2-NO_3 values from 6-14 mg/l (Figure 40).

Table 4 Median Values At Erie Sites, 1976-1977[1]

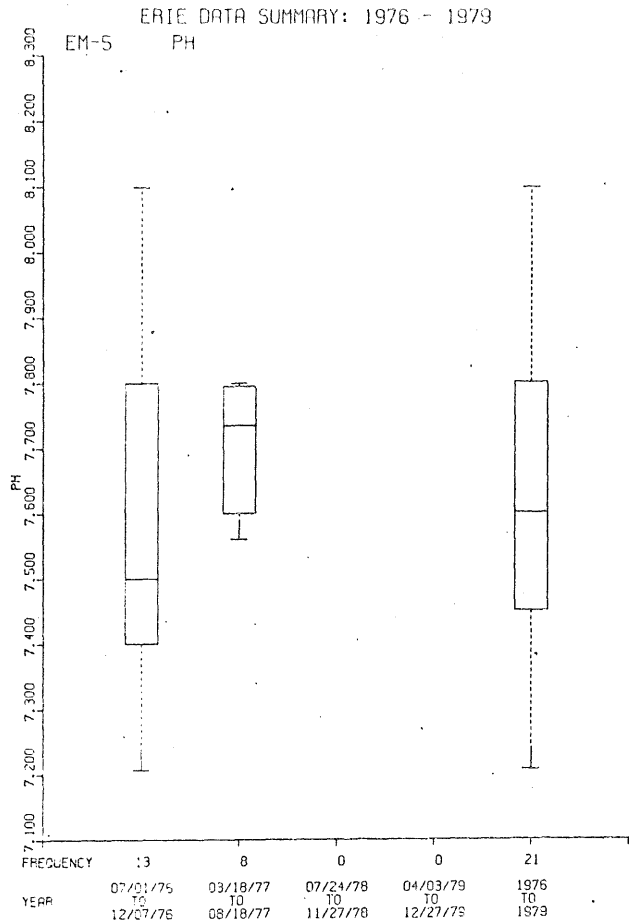
Site	Q	N	SO_4	N	Cu	N	Ni	N	Co	N	Zn	N	pH	N	DOC	N	DIC	N	TALK	N
011[2]			72	14	0.004	14	0.005	14												
EM-6	26.6[3]	19	106	17	0.004	19	0.005	14	0.002	1	0.003	1	7.70	19	9.4	14	17.9	13	94	18
EM-8	4.82	34	1140	24	0.019	33	1.18	33	0.015	8	0.020	10	7.26	34	20.1	22	24.7	22	142	33
EM-5	31.4	21	122	19	0.005	21	0.051	21	0.005	5	0.009	3	7.63	21	12.2	12	20.1	11	101	21
Seep 3	2.10	25	1420	17	0.617	24	19.0	24	0.910	8	0.331	12	7.16	24	19.0	14	28.2	14	109	24
EM-3	93.4	26	155	25	0.003	28	0.110	27	0.002	3	0.009	2	7.65	27	10.5	17	19.8	16	96	27
Seep 1	1.15	26	2470	20	0.018	27	1.10	27	0.130	9	0.249	13	6.95	27	28.0	16	25.2	15	98.8	27
012[2]			250	7	<0.003	7	0.002	7												
EM-4	14.2	8	77.0	3	0.002	8	0.003	4	-	0	-	0	6.46	8	20.3	7	4.3	7	14.1	8
EM-2	2.62	14	335	13	0.005	14	0.042	14	0.002	1	-	0	7.60	13	11.0	10	16.6	10	96.6	14
EM-1	110	37	159	35	0.002	37	0.087	35	0.002	5	0.010	2	7.50	38	12.7	19	18.5	19	88.0	36

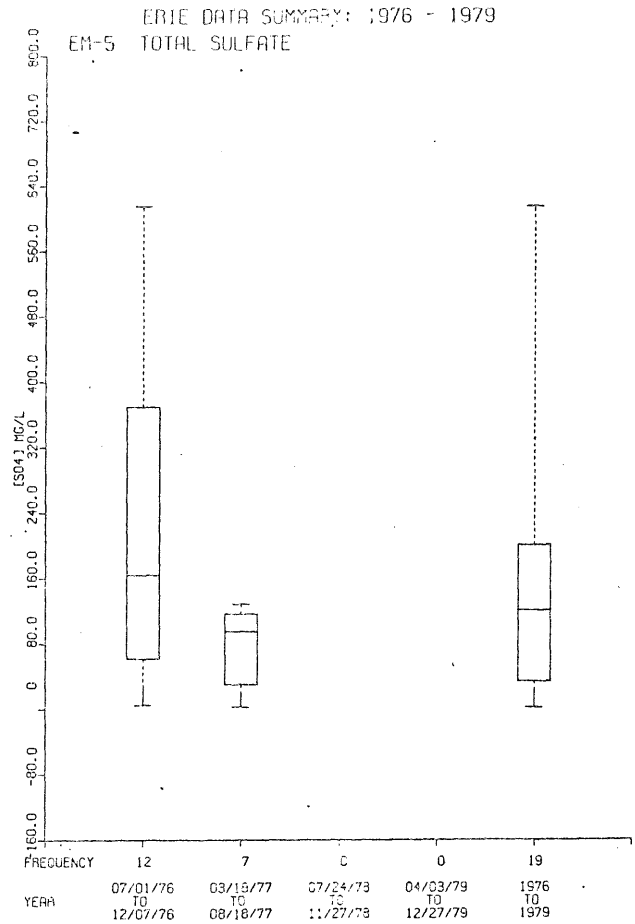
[1] Q in liters/sec, concentrations in mg/l, DOC and DIC in mg/l as C, TALK in mg/ℓ as Ca CO_3

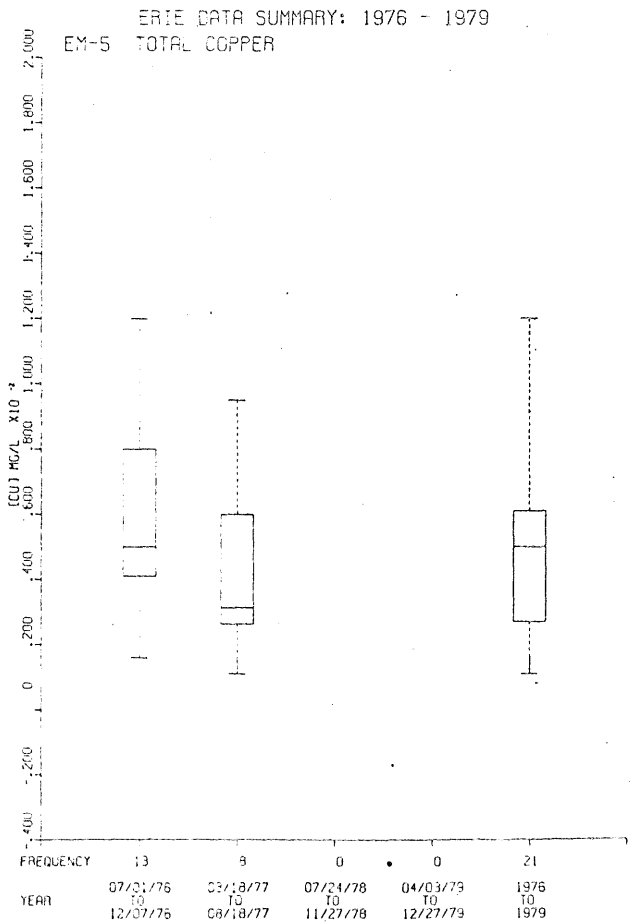
[2] Data from Erie Mining Company

[3] Calculated from data at EM-8 and EM-5

FIGURE 34


ERIE DATA SUMMARY: 1976 - 1979
EM-5 PH
PH
7.100 7.200 7.300 7.400 7.500 7.600 7.700 7.800 7.900 8.000 8.100 8.200 8.300
FREQUENCY 13 8 0 0 21
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-5 TOTAL SULFATE
[SO4] MG/L
-80.0 0 80.0 160.0 240.0 320.0 400.0 480.0 560.0 640.0 720.0 800.0
FREQUENCY 12 7 0 0 19
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-5 TOTAL COPPER
[CU] MG/L X10 -2
-.400 -.200 0 .200 .400 .600 .800 1.000 1.200 1.400 1.600 1.800 2.000
FREQUENCY 13 8 0 0 21
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

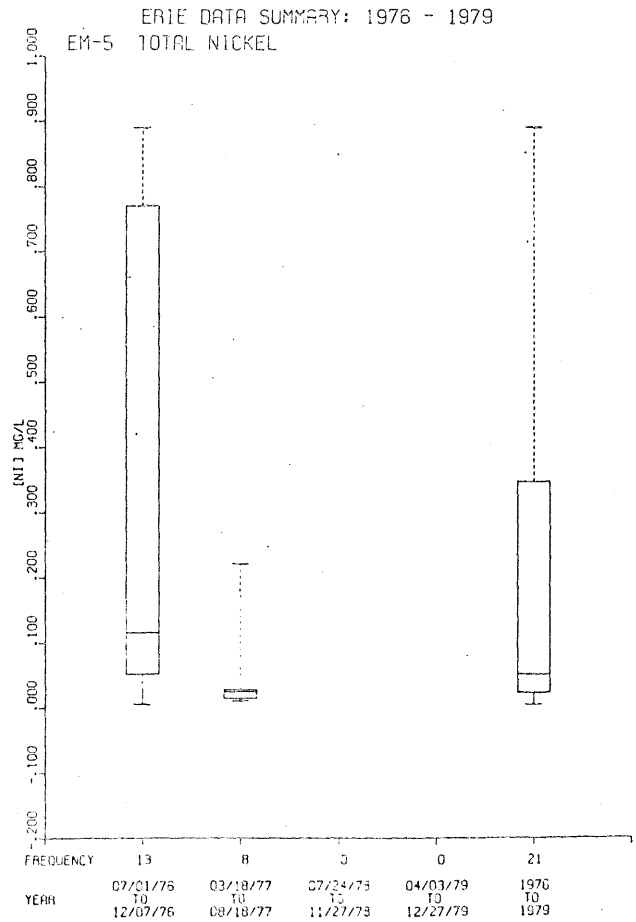
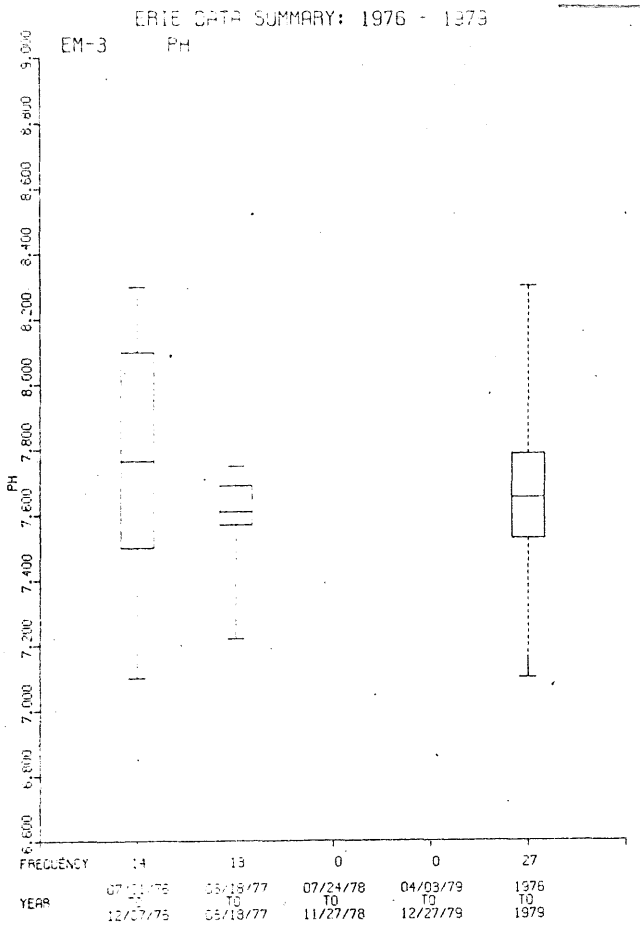
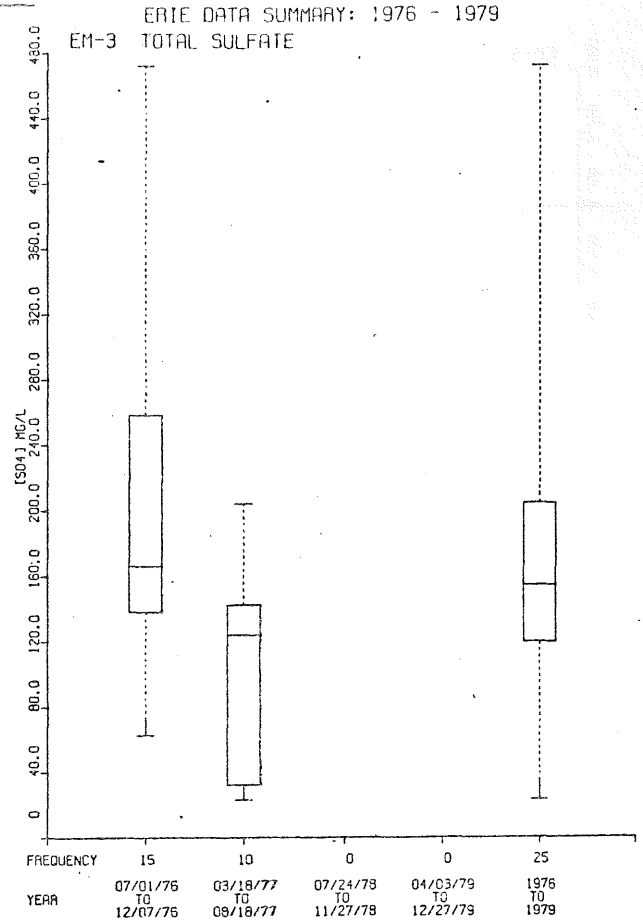
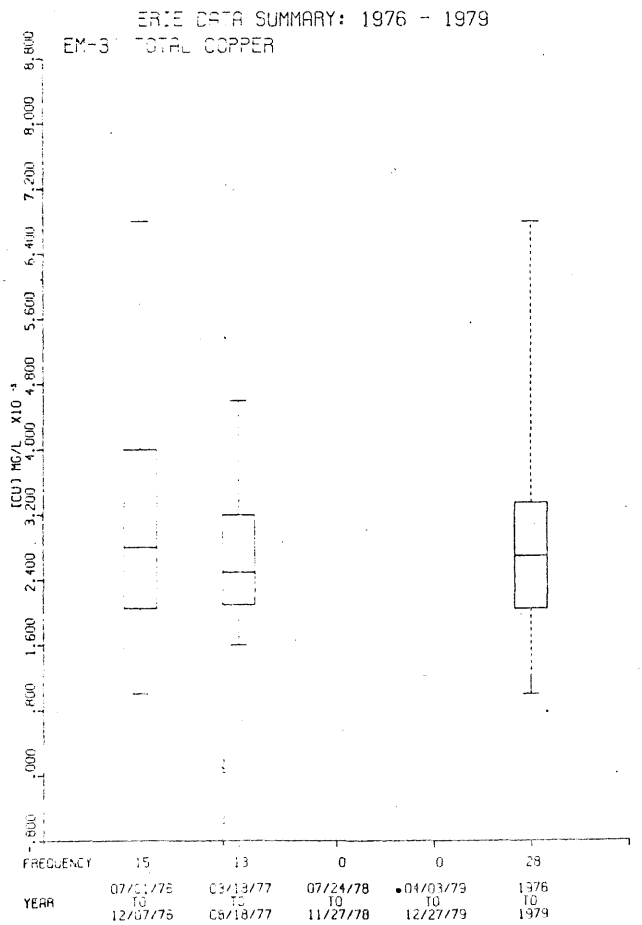

ERIE DATA SUMMARY: 1976 - 1979
EM-5 TOTAL NICKEL
[NI] MG/L
-.200 -.100 .000 .100 .200 .300 .400 .500 .600 .700 .800 .900 1.000
FREQUENCY 13 8 0 0 21
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

FIGURE 35


ERIE DATA SUMMARY: 1976 - 1979
EM-3 PH
PH
6.600 6.800 7.000 7.200 7.400 7.600 7.800 8.000 8.200 8.400 8.600 8.800 9.000
FREQUENCY 14 13 0 0 27
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-3 TOTAL SULFATE
[SO4] MG/L
0 40.0 80.0 120.0 160.0 200.0 240.0 280.0 320.0 360.0 400.0 440.0 480.0
FREQUENCY 15 10 0 0 25
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-3 TOTAL COPPER
[CU] MG/L X10 -1
-.800 .000 .800 1.600 2.400 3.200 4.000 4.800 5.600 6.400 7.200 8.000 8.800
FREQUENCY 15 13 0 0 28
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979

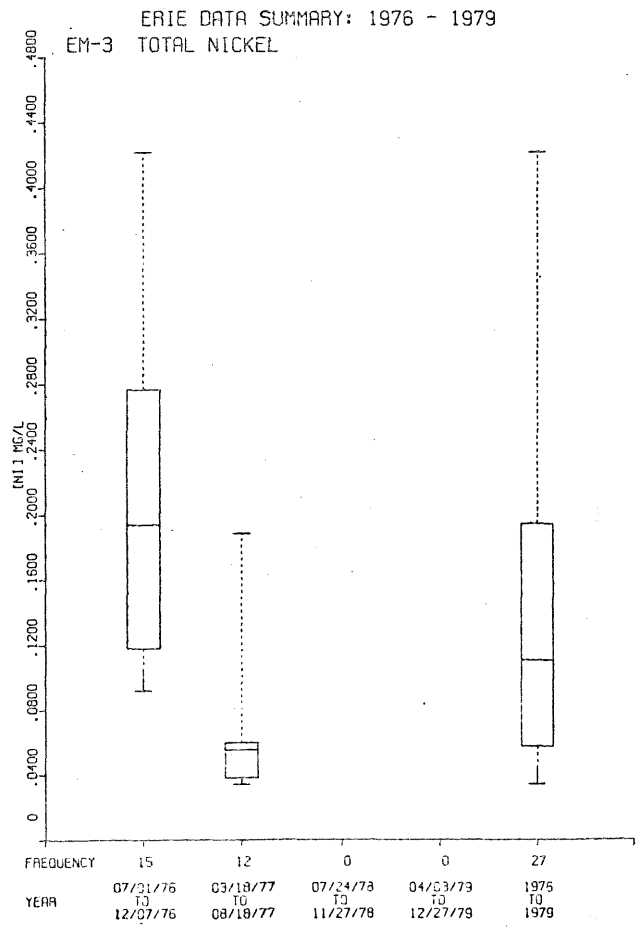
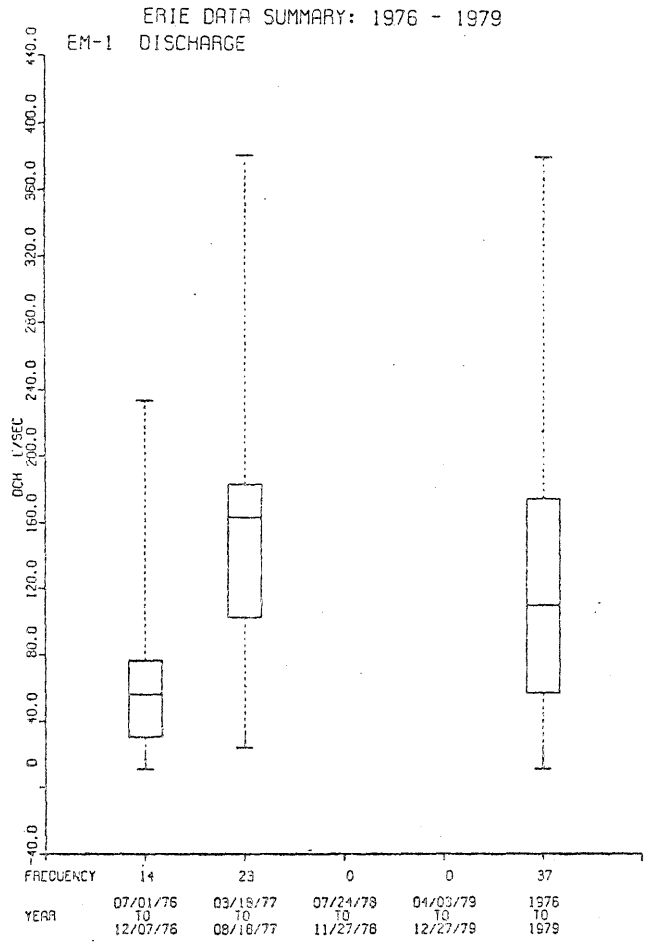
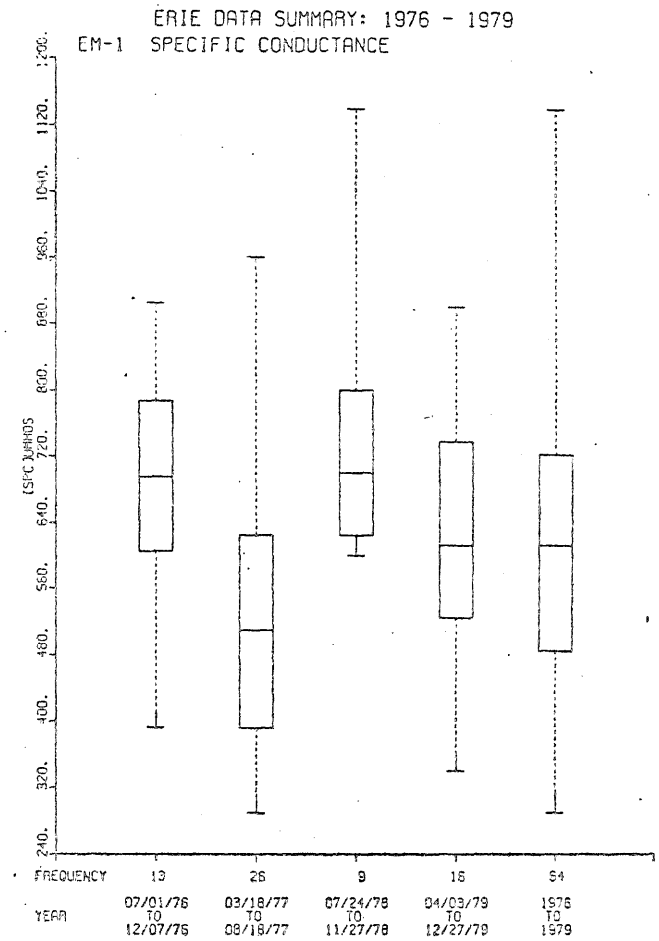

ERIE DATA SUMMARY: 1976 - 1979
EM-3 TOTAL NICKEL
[NI] MG/L
0 .0400 .0800 .1200 .1600 .2000 .2400 .2800 .3200 .3600 .4000 .4400 .4800
FREQUENCY 15 12 0 0 27
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979

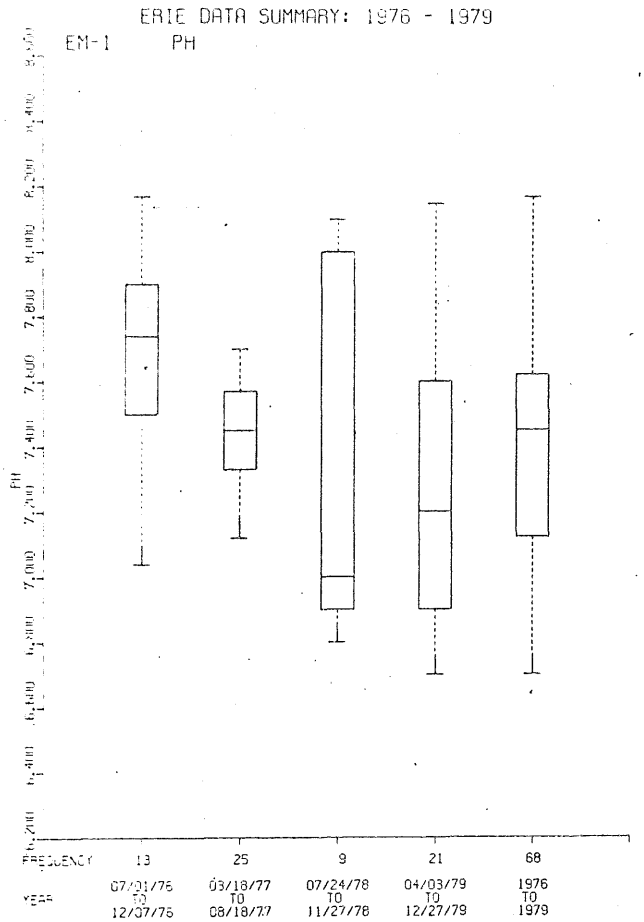
FIGURE 36

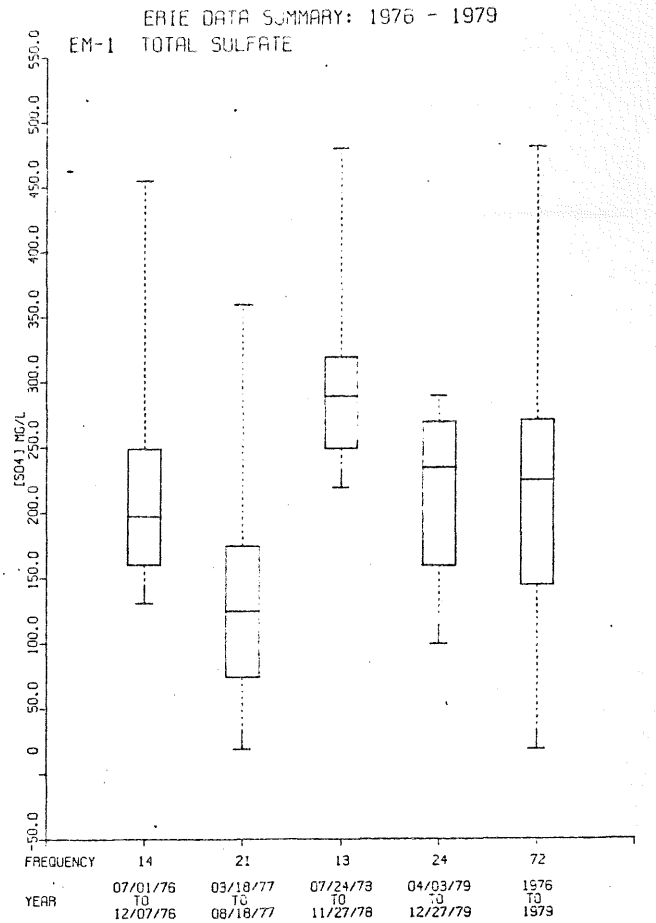


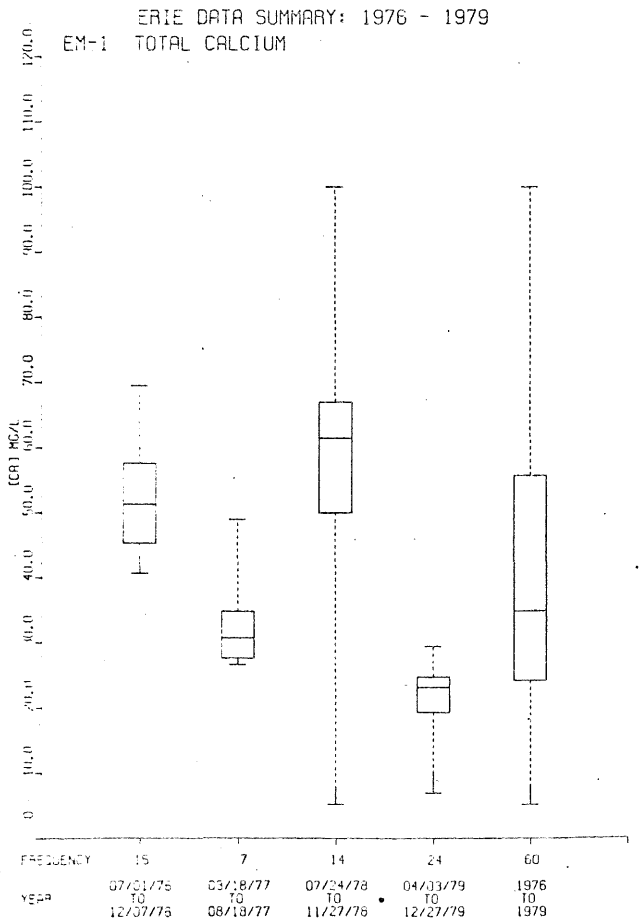


Based on periodic measurements

FIGURE 37


ERIE DATA SUMMARY: 1976 - 1979
EM-1 PH
PH
FREQUENCY 13 25 9 21 68
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL SULFATE
[SO4] MG/L
-50.0 0 50.0 100.0 150.0 200.0 250.0 300.0 350.0 400.0 450.0 500.0 550.0
FREQUENCY 14 21 13 24 72
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL CALCIUM
[CA] MG/L
0 10.0 20.0 30.0 40.0 50.0 60.0 70.0 80.0 90.0 100.0 110.0 120.0
FREQUENCY 15 7 14 24 60
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

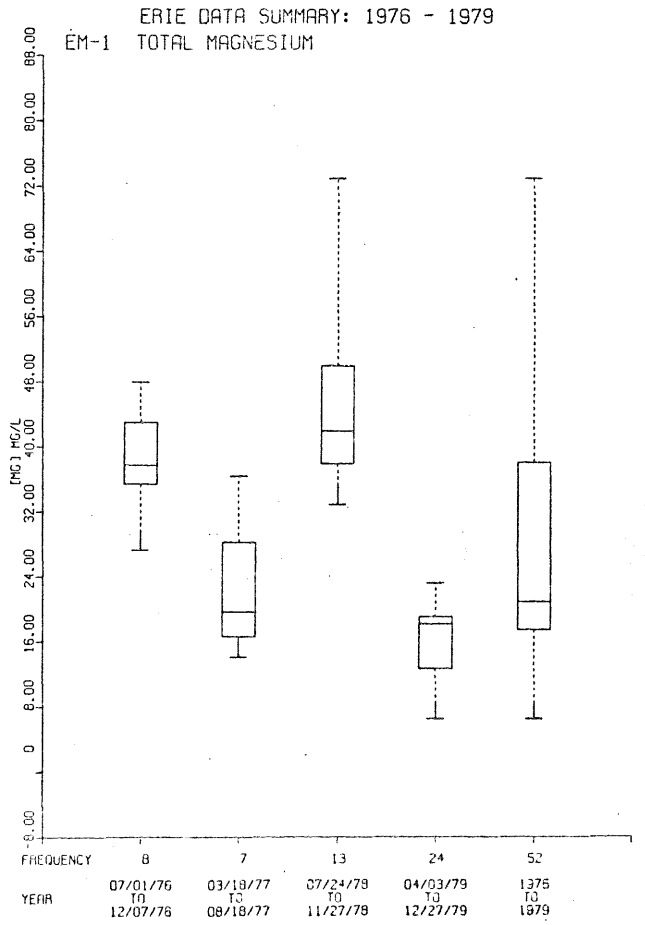
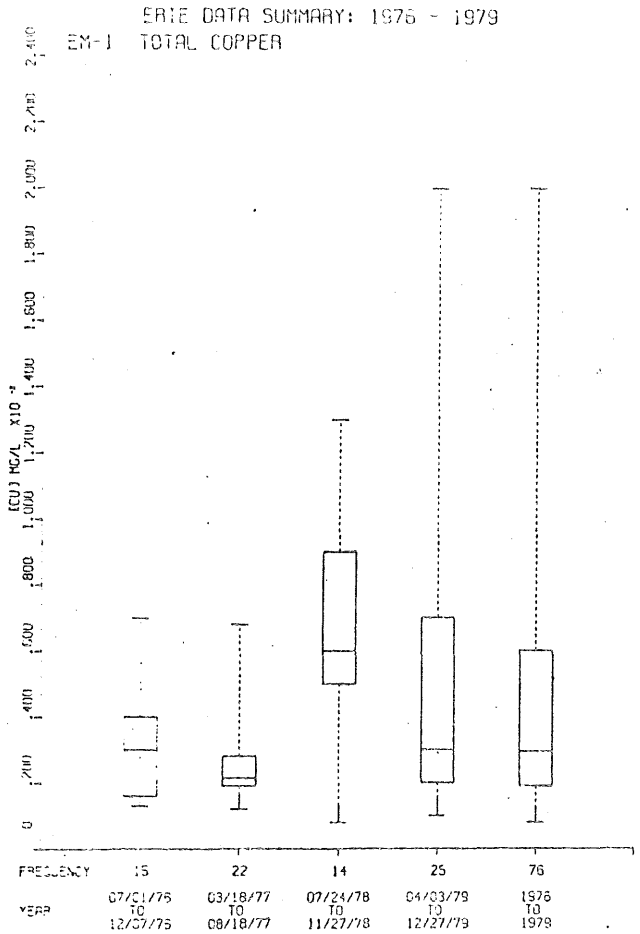
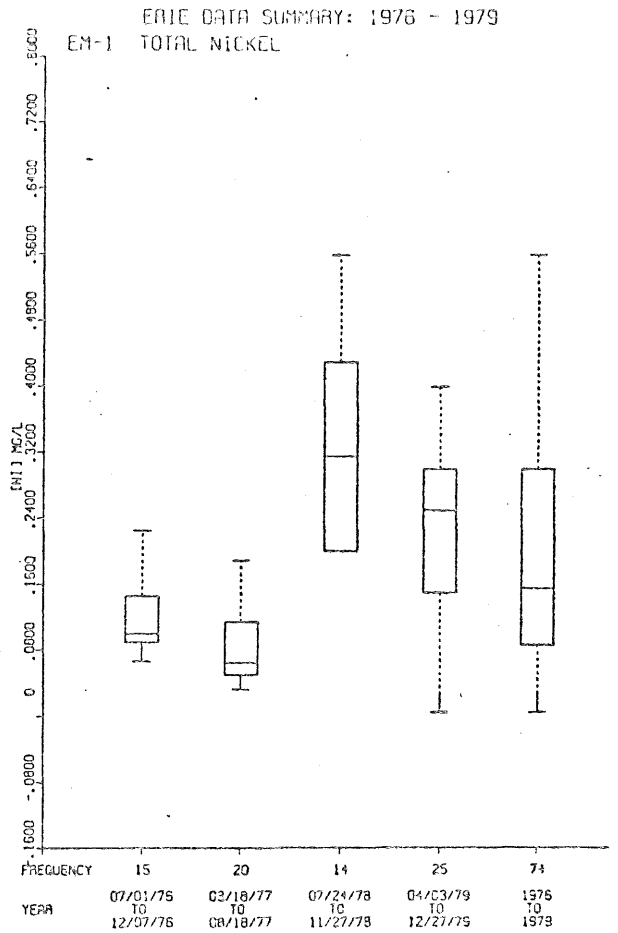
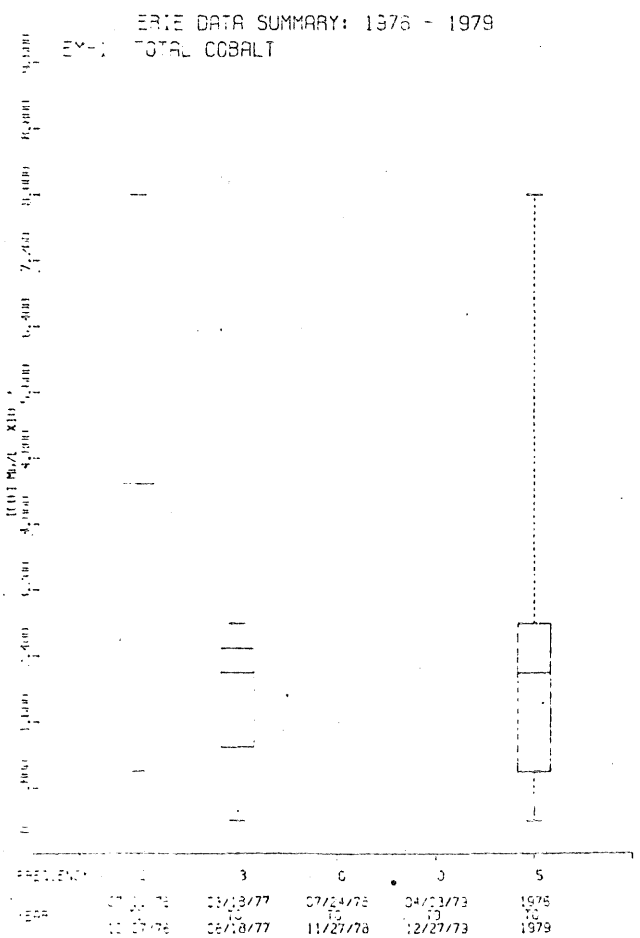

ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL MAGNESIUM
[MG] MG/L
-8.00 0 8.00 16.00 24.00 32.00 40.00 48.00 56.00 64.00 72.00 80.00 88.00
FREQUENCY 8 7 13 24 52
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

FIGURE 38


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL COPPER
[CU] MG/L X10 -2
FREQUENCY 15 22 14 25 76
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL NICKEL
[NI] MG/L
FREQUENCY 15 20 14 25 74
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL COBALT
3 0 0 5
03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979

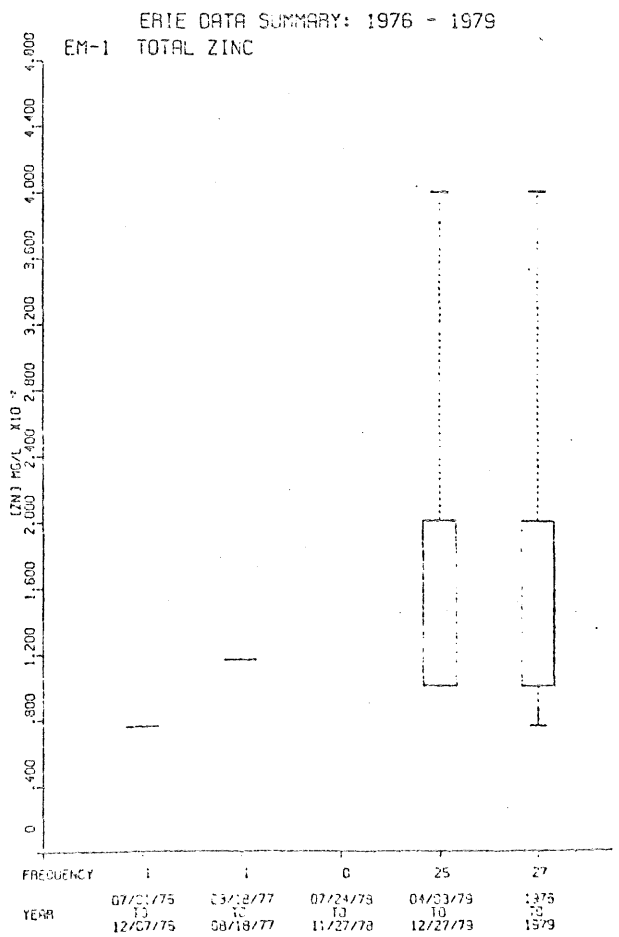
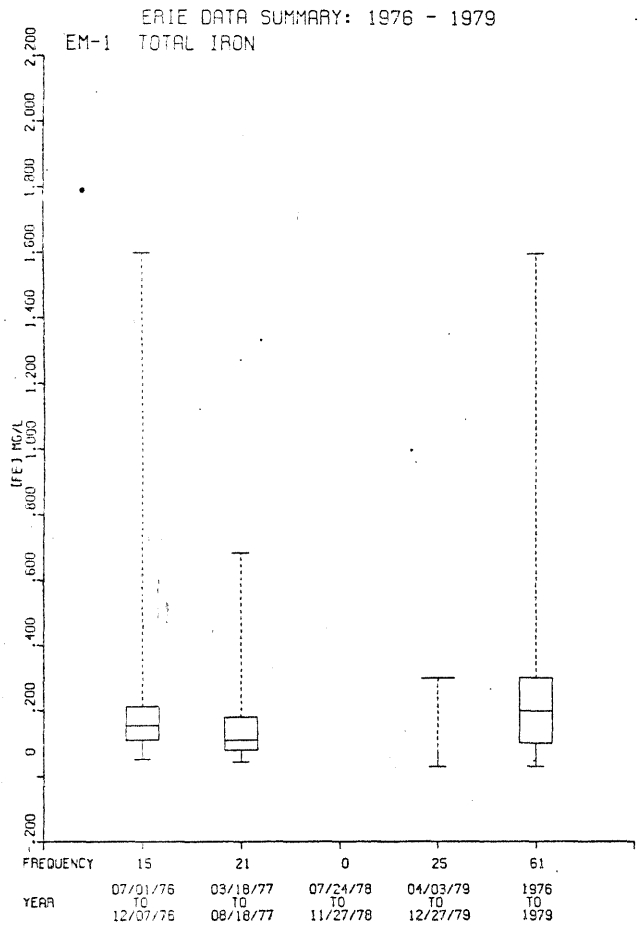
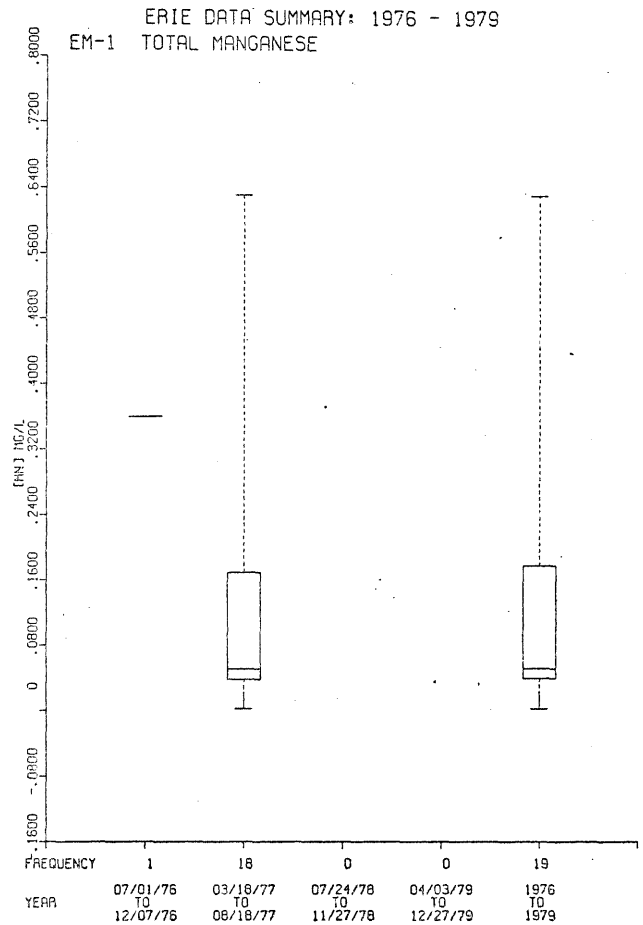

ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL ZINC
[ZN] MG/L X10 -2
FREQUENCY 1 1 0 25 27
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979

FIGURE 39


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL IRON
[FE] MG/L
2.200
2.000
1.800
1.600
1.400
1.200
1.000
.800
.600
.400
.200
0
-.200
FREQUENCY 15 21 0 25 61
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL MANGANESE
[MN] MG/L
.8000
.7200
.6400
.5600
.4800
.4000
.3200
.2400
.1600
.0800
0
-.0800
-.1600
FREQUENCY 1 18 0 0 19
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

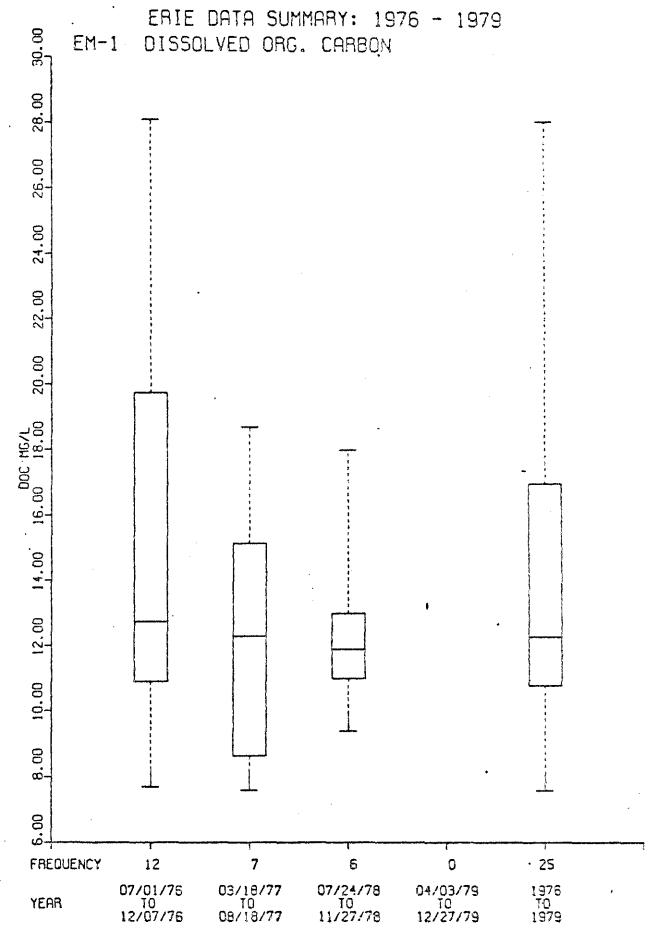
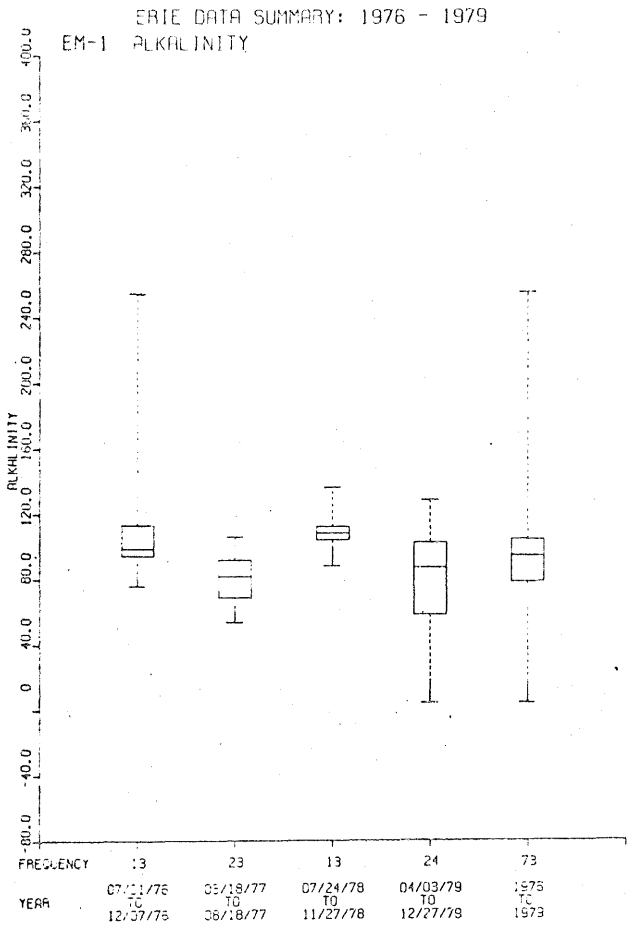
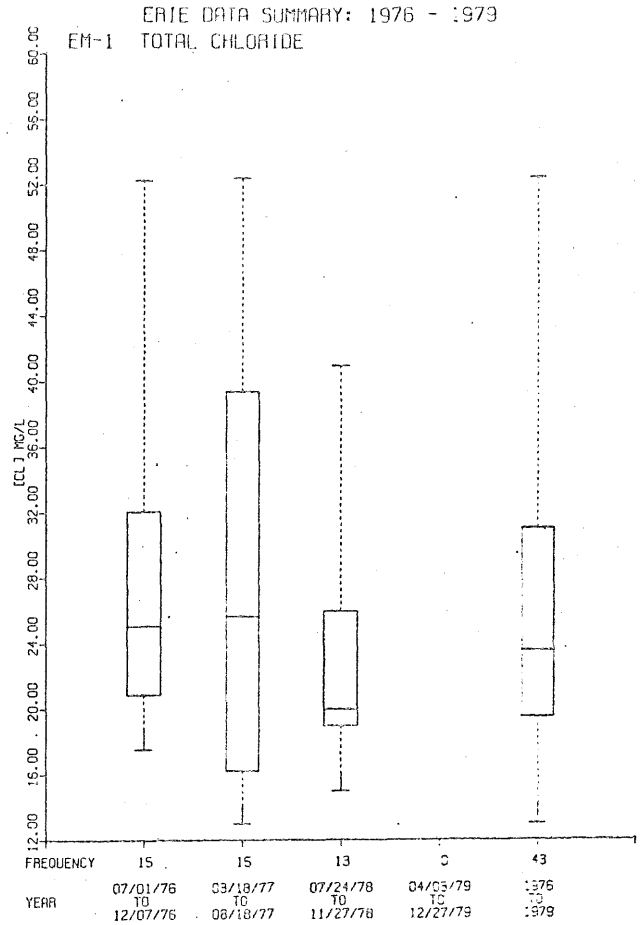
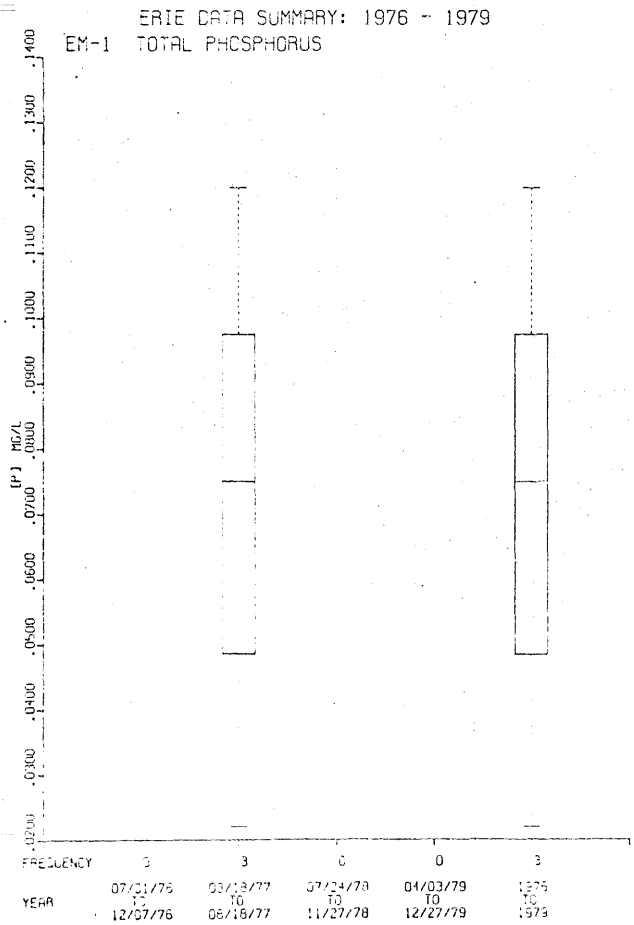
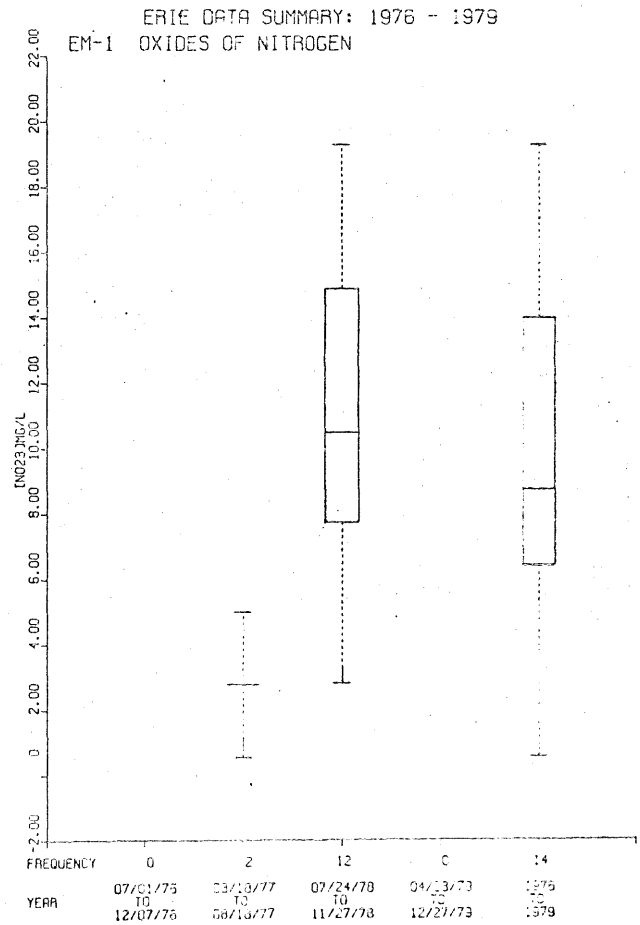

ERIE DATA SUMMARY: 1976 - 1979
EM-1 DISSOLVED ORG. CARBON
DOC MG/L
30.00
28.00
26.00
24.00
22.00
20.00
18.00
16.00
14.00
12.00
10.00
8.00
6.00
FREQUENCY 12 7 6 0 25
YEAR 07/01/76 TO 12/07/76 | 03/18/77 TO 08/18/77 | 07/24/78 TO 11/27/78 | 04/03/79 TO 12/27/79 | 1976 TO 1979

FIGURE 40


ERIE DATA SUMMARY: 1976 - 1979
EM-1 ALKALINITY
ALKALINITY
FREQUENCY 13 23 13 24 73
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL CHLORIDE
[CL] MG/L
FREQUENCY 15 15 13 0 43
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-1 TOTAL PHOSPHORUS
[P] MG/L
FREQUENCY 0 3 0 0 3
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979


ERIE DATA SUMMARY: 1976 - 1979
EM-1 OXIDES OF NITROGEN
[NO23] MG/L
FREQUENCY 0 2 12 0 14
YEAR 07/01/76 TO 12/07/76 03/18/77 TO 08/18/77 07/24/78 TO 11/27/78 04/03/79 TO 12/27/79 1976 TO 1979

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APPENDIX I

RAW DATA

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Table A1.1 Em-8 raw data, 1976-1979

SITE= EM8
YEAR= 1976

DATE	07/02	07/06	07/15	07/19	07/26	07/27	08/12
PH	7.20	7.10	7.20	7.15	7.09	7.01	7.10
ALK	103.	136.	128.	133.	130.	133.	124.
SP COND*	629.		1695.	1846.	1980.	2079.	2253.
T CENT	4.0	11.0	13.5	10.0	10.0	6.0	11.5
SO4	708.	714.	980.	1134.	1203.	1237.	1346.
CU T	.017	.018	.020	.016	.016	.053	.021
NI T	.963	1.580	1.600	.580	1.890	1.970	2.400
CO T							
ZN T					.020	.018	.035
FE T	.19	.16	.17	.11	.11	.09	.13
CU F	.016	.015	.018	.015	.015	.018	.020
NI F	.950	.498	1.560	.570	1.940	2.100	2.400
FE F	.10	.01	.06	.09	.07	.04	.04
CA	94.	143.	65.	154.	163.	178.	214.
MG							
DOC	14.5	24.5	26.0	26.3	19.0	20.0	21.4
DIC	32.9	21.5	15.7	19.0	24.5	26.5	17.4
CL	34.1	41.2	39.1	40.7	43.9	45.3	56.5
STAGE			1.39			1.37	1.37
DCH**	8.10	6.65	5.69	5.38	5.10	5.10	4.81

*=UMHOS/CM **=L/SEC

CONTINUED

Table A1.1 (Continued)

SITE= EM8
YEAR= 1976

DATE	08/26	09/08	09/21	10/05	10/25	11/11
PH	7.40	7.40	7.65	7.50	7.35	6.50
ALK	130.	150.	145.	142.	151.	178.
SP COND*	186.	2632.	2736.	2750.	2600.	2800.
T CENT	8.5	8.0	3.0	2.1	.1	.3
SO4	1564.	1685.	1523.	1420.	1380.	1550.
CU T	.018	.017	.020	.012	.012	.010
NI T	2.360	2.420	2.420	2.110	2.090	2.180
CO T			.029	.018	.016	
ZN T	.038	.037	.040			
FE T	.10	.14	.21	.10	.08	.11
CU F	.016	.016	.016	.011		
NI F	2.400	2.400	2.560	2.210		
FE F	.06	.07	.13	.04		
CA	230.	241.	248.	291.	284.	301.
MG				177.	168.	190.
DOC	26.5	36.3	11.5	14.1	15.3	15.9
DIC	19.5	10.3	37.5	33.4	37.9	37.5
CL	29.2		38.0	40.7	42.7	44.7
STAGE	1.33	1.34	1.35	1.34	1.21	1.20
DCH**	3.77	3.77	3.77	3.96	1.42	1.42

*=UMHOS/CM **=L/SEC

Table A1.1 (Continued)

SITE= EM8
YEAR= 1977

DATE	03/18	03/29	04/12	04/13	04/14	04/15	04/20	04/21	04/25	05/05	05/13
PH	7.64	7.25	7.20	7.30	7.50	7.29	7.20	7.20	7.30	7.70	7.40
ALK	286.	83.	296.	257.	150.	197.	99.	53.	335.	157.	123.
SP COND*	3220.	750.	2300.	1950.	1550.	1420.	1210.	625.	1900.	1180.	950.
T CENT			4.0		8.5	3.5	5.0	15.0	3.0	12.8	
SO4	2130.	329.	1380.						846.		
CU T	.031	.020	.034	.027	.026	.024	.019	.020	.041	.013	.015
NI T	3.730	1.120	1.830	1.290	.870	.840	1.180	1.310	.820	.490	.457
CO T										.013	
ZN T										.007	
FE T	.62	1.64	.71	.66	.87	1.09	1.87	.68	1.10	.67	.88
MN T	7.33	4.66	4.01	2.99	2.46	2.30	2.26	.98	3.35	.60	.57
CU F	.021	.012				.021		.019	.027	.009	
NI F	3.510	1.170				.820		1.330	.880	.390	
FE F	.15	1.04				.76		.55	.38	.04	
MN F	7.16	4.56				2.01		.94	3.26		
CA	506.	63.								128.	114.
MG	320.									55.	56.
DOC			11.2		12.5		18.6	22.7	21.7	14.8	40.4
DIC			54.0		41.7		20.8	13.9	94.3	33.8	27.7
CL	53.9	16.3	33.9	30.3			24.1	18.0	28.8	21.2	21.9
NO2-NO3	10.00									.62	
TOTAL P	.10									.04	
STAGE	1.09	1.10	1.62	1.48	1.32	1.22	1.14	1.25	1.26	1.24	1.14
DCH**	.28	.57	16.71	8.78	3.68	1.70	.85	2.27	2.55	2.27	.85

.*=UMHOS/CM **=L/SEC

CONTINUED

Table A1.1 (Continued)

SITE= EM8
YEAR= 1977

DATE	05/23	05/31	06/06	06/20	07/05	07/11	07/19	07/25	08/01	08/16	08/17
PH	7.27	7.45	7.16	7.22	7.28		7.41	7.20	7.25	7.10	7.30
ALK	123.	95.	233.	167.	129.		136.		154.	162.	161.
SP COND*	950.	750.	1820.	1710.	1500.	1590.	2020.	1836.	2425.	2600.	2433.
T CENT	9.0		4.0		6.1	5.0	7.7		6.0	7.7	6.3
SO4		353.	892.	674.	281.		1070.			1428.	1605.
CU T	.021	.021	.039	.026	.019		.017		.019	.016	.016
NI T	.566	.386	1.140	.630	.641		.807		1.070	1.190	1.150
CO T		.013		.023	.015						.010
ZN T		.014		.020							.019
FE T	.65	.60	.81	.44	.32		.13		.14	.08	.10
MN T	.66	.43	2.85	2.05	1.01		.68		.93	.76	.85
CU F	.017		.022								
NI F	.528		1.240								
FE F	.35		.24								
MN F	.69		2.77								
CA	110.	90.	210.	179.	160.						328.
MG	56.	47.	126.	107.	82.						178.
DOC		31.6	26.5								
DIC		7.1	22.7								
CL	21.9	17.4	32.4								
NO2-NO3											
TOTAL P											
STAGE	1.34	1.44	1.83	1.82	1.79	1.54	1.46		1.59	1.51	1.50
DCH**	4.53	7.65	32.28	31.43	29.73	13.59	9.63		8.49	5.66	5.38

*=UMHOS/CM **=L/SEC

Table A1.1 (Continued)

SITE= EM8
YEAR= 1978
CONC= MG/L

DATE	07/24	07/31	08/07	08/21	08/24	09/05	09/18
PH	6.80	6.80	7.60	7.30		6.90	7.00
ALK	104.	96.	102.	116.	122.	142.	132.
SP COND*	1800.	2000.	2200.	2200.		2425.	2400.
T CENT	10.0	10.0	11.1	11.1		6.7	5.6
SO4	1100.	1175.	1350.	1150.	1166.	1450.	1150.
CU T	.061	.096	.106	.110	.105	.140	.120
NI T	5.100	7.100	6.700	5.000	3.940	6.500	4.570
CD T	.200	.300	.150	.190		.280	.220
ZN T	.100	.160	.100	.100		.170	.120
CA	167.	200.	225.	275.	201.	225.	187.
MG	130.	150.	175.	130.	123.	130.	100.
NA	63.	63.	93.	86.		122.	72.
DOC				12.0		10.0	9.8
CL	29.0	24.0	29.0	28.0	43.0	29.0	21.0
NO2-NO3	9.60	11.20	14.50	9.20		22.50	11.70
DCH**	18.12	45.02	15.86		44.74	15.86	

*=UMHOS/CM **=L/SEC

CONTINUED

Table A1.1: (Cont.)

SITE= EM8
YEAR= 1978
CONC= MG/L

DATE	10/02	10/16	10/30	11/17	11/27
PH	6.70	6.80	6.80	6.40	6.90
ALK	112.	112.	140.	152.	164.
SP COND*	2300.	2200.	2200.	2200.	2450.
T CENT	10.0	1.1	5.0	1.1	.6
SO4	1500.	1350.	1300.	1280.	1520.
CU T	.180	.180	.170	.150	.150
NI T	7.500	6.650	5.600	3.550	3.550
CO T	.250	.200	.200	.150	.120
ZN T	.150	.130	.110	.100	.100
CA	255.	270.	255.	290.	320.
MG	163.	160.	153.	138.	145.
NA	88.	91.	87.	108.	118.
DOC	8.1	11.0	14.0		
CL	25.0	24.0	27.0	28.0	26.0
NO2-NO3	16.70	16.70	13.90	14.80	22.30
DCH**	15.86	11.89	13.59	5.95	

*=UMHOS/CM **=L/SEC

Table A1.1 (Cont.)

SITE= EM8
YEAR= 1979

DATE	04/03	04/16	05/09	05/16	05/24	06/06	06/22	07/05	07/27	08/09
PH	6.70	6.80	6.70	6.70	6.70	7.00	6.70	6.60	6.90	6.90
ALK			94.	100.	110.	120.	120.	110.	116.	126.
SP COND*			1200.	1200.	1350.	1650.	1450.	1700.	2000.	2100.
T CENT			2.0	7.0		6.0	7.0	9.0	11.0	9.0
SO4			800.	700.	800.	900.	1000.	1100.	1300.	1300.
CU T	.030	.300	.070	.070	.050	.100	.100	.100	.100	.100
NI T	1.200	.330	3.200	2.800	1.900	2.800	3.000	3.000	3.000	5.900
ZN T	.040	.320	.060	.060	.050	.070	.070	.080	.070	.120
FE T	1.00	.30	.50	.50	.30	.30	.30	.30	.30	.30
CA			112.	61.	70.	82.	70.	80.	99.	105.
MG			35.	35.	38.	50.	55.	55.	71.	66.
AL T	.10	.40	.20	.20	.08	.08	.08	.20	.08	.08
DCH**			13.51	15.83	19.65	15.35	23.47	11.07		8.52

*=UMHOS/CM **=L/SEC

CONTINUED

Table A1.1 (Cont.)

SITE= EM8
YEAR= 1979

DATE	08/28	09/13	09/27	10/10	10/19	10/29	11/08	11/29	12/04	12/26
PH	6.90	7.20	7.20	7.20	6.90	7.00	7.20	7.10	7.00	7.20
ALK	134.	126.	110.	94.	92.	100.	102.	104.	112.	124.
SP COND*	2300.	2000.	2700.	2500.	2300.	2650.	2500.	2500.	2500.	2500.
T CENT	3.0	4.0	9.0	8.0	5.0	4.0	1.0	1.0	1.0	1.0
SO4	1200.	1100.	1200.	1400.	1300.	1600.	1200.	1500.	1500.	530.
CU T	.100	.100	.100	.100	.080	.100	.100	.110	.100	.040
NI T	5.300	4.200	6.000	7.500	6.800	7.600	4.000	6.000	6.600	3.000
ZN T	.100	.070	.100	.110	.110	.120	.070	.110	.130	.070
FE T	.30	.30	.30	.20	.30	.30	.20	.20	.30	.50
CA	103.	92.	111.	117.	114.	120.	97.	108.	530.	200.
MG	74.	53.	60.	69.	59.	77.	56.	76.	190.	80.
AL T	.08	.08	.08	.08	.20	.08	.08	.08	.08	.08
DCH**	9.03	22.11	12.66				9.68		6.31	

*=UMHOS/CM **=L/SEC

Table A1.2 Seep 1 raw data, 1976-1979

SITE= SP1
YEAR= 1976

DATE	07/15	07/27	08/12	08/26	09/08	09/21	10/05	10/25	12/07
PH	7.50	7.00	7.02	7.34	7.50	7.30	7.35	7.42	7.20
ALK	161.	184.	190.	188.	227.	220.	213.	229.	205.
SP COND*	3027.	3328.	2324.	3818.	3709.	3809.	3800.	3490.	4600.
T CENT	.5		3.5	3.0	2.0	2.4	1.8	.1	
SO4	2487.	2555.	1839.	2190.	2305.	2004.	2460.	2800.	3000.
CU T	.050	.019	.029	.023	.022	.091	.005	.004	.017
NI T	1.920	1.240	1.400	1.100	.980	.884	.790	.670	1.300
CO T						.132	.129	.123	.110
ZN T		.258	.250	.250	.245	.125	.100	.100	.690
FE T	5.79	.71	3.15	3.80	3.65	7.07	5.44	5.78	7.20
MN T									16.00
CU F					.019	.009	.002		
NI F					.900	.845	.710		
FE F					1.07	5.70	2.84		
CA		194.	233.		225.	226.	246.	249.	396.
MG							464.	451.	528.
DOC	34.5	28.0	33.9	42.8	54.1	23.5	14.9	27.2	
DIC	20.7	35.5	27.6	25.2	23.4	56.0	51.3	61.2	
CL	48.6	49.8	84.3	29.3	40.7	36.3	39.3	39.8	50.0
STAGE			3.21	3.21	3.15	3.13	3.17	3.15	
DCH**	1.70	1.98	1.70	1.42	1.42	1.42	1.42	1.13	.14

*=UMHOS/CM **=L/SEC

Table A1.2 (Cont.)

SITE= SP1
YEAR= 1977

DATE	03/18	03/29	04/12	04/15	04/20	04/21	05/05	05/13	05/23
PH	7.02	7.38	6.79	6.95	6.60	6.71	7.07	7.00	6.75
ALK	441.	83.	50.	100.	72.	61.	124.	164.	98.
SP COND*	3100.	660.	500.	850.	940.	850.	1450.	1500.	1210.
T CENT	0		-.2		-.5	-1.0	1.0		4.5
SO4	1638.	265.	149.						
CU T	.329	.022	.003	.004	.007	.005	.005	.004	.004
NI T	2.580	.135	.053	.088	.136	.100	.320	.168	.213
CO T							.130		
ZN T							.037		
FE T	1.98	.84	.92	2.73	3.00	3.45	8.70		
MN T	7.05	1.94	2.31	4.31	7.28	6.38	19.00	11.65	8.05
CU F	.181	.007		.002		.005	.002		
NI F	2.320	.150		.082		.128	.290		
FE F	.44	.40		2.19		2.95	6.84		
MN F	7.11	1.64		3.81		6.33			
CA	346.	46.						111.	95.
MG	477.							156.	119.
DOC			24.3	26.5	55.9	37.1	36.8		25.2
DIC			13.0	28.2	8.8	20.5	28.6		21.2
CL	121.4	18.0	10.2	16.0	19.0	15.3	23.4	21.5	17.8
NO2-NO3	17.00	1.00					.65		
TOTAL P	.29	.15					.14		
STAGE						2.92		3.22	
DCH**		.14	.14	.57	.14	.28	.96	.08	.42

*=UMHOS/CM **=L/SEC

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Table A1.2 (Cont.)

SITE= SP1
YEAR= 1977

DATE	05/31	06/06	06/20	06/28	07/05	07/19	08/01	08/16	08/17
PH	6.65	6.45	6.64	6.69	6.69	6.60	6.79	6.70	6.75
ALK	74.	46.	52.	79.	62.	62.	99.	92.	82.
SP COND*	1150.	1420.	4450.	4253.	4410.	6900.	7700.	6800.	5508.
T CENT				6.0	5.0	5.0	5.0	6.1	5.5
SO4	677.	801.	2723.	3857.	2677.	4439.		5210.	5636.
CU T	.005	.007	.041	.038	.029	.025	.023	.016	.018
NI T	.370	1.570	12.000	7.800	7.370	8.190	6.340	5.770	5.910
CO T	.110		.870			.640			.640
ZN T	.110		1.500			2.400			2.400
FE T	7.54	2.66	1.45	1.28	.72	.91	.87	.63	.62
MN T	8.60	7.14	13.90	11.10	8.65	13.10	12.10	12.80	13.60
CU F	.004	.007							
NI F	.358	1.610							
FE F	4.83	1.48							
MN F	8.55	7.11							
CA	94.	122.	290.	301.					
MG	122.	142.	486.	652.					
DOC	18.7		34.0						
DIC	17.2								
CL	13.8	15.1							
NO2-NO3									
TOTAL P									
STAGE	3.27		3.17	3.12	3.10	3.18	3.18	3.22	3.15
DCH**	.85	1.42	2.27	2.41	4.19	1.59	1.27	.45	.45

*=UMHOS/CM **=L/SEC

Table A1.2 (Cont.)

SITE= SP1
YEAR= 1978
CONC= MG/L

DATE	07/24	07/31	08/07	08/21	09/05	09/18	10/02	10/16	10/30	11/17
PH	6.80	6.60	7.50	6.50	6.60	6.70	6.40	6.60	6.60	6.30
ALK	80.	96.	120.	116.	176.	124.	124.	116.	132.	136.
SP COND*	5000.	5000.	5000.	5000.	5000.	5000.	5000.	5000.	5000.	5000.
T CENT	8.9	10.0	10.0	6.1	5.6	4.4	6.7	7.8	4.4	1.1
SO4	3275.	4625.	4600.	4600.	5000.	3650.	3750.	4200.	4450.	5000.
CU T	.098	.092	.119	.065	.052	.042	.045	.043	.038	.034
NI T	12.500	11.800	7.000	8.900	8.200	8.200	9.000	7.500	6.800	2.800
CD T	1.000	1.000	.750	.870	.950	.650	.700	.700	.600	.300
ZN T	2.300	2.200	1.400	1.450	1.650	1.570	1.260	1.170	1.020	.180
CA	290.	275.	305.	350.	275.	260.	280.	290.	295.	38.
MG	900.	920.	1100.	780.	920.	400.	590.	600.	620.	450.
NA	135.	115.	150.	140.	152.	95.	103.	97.	105.	145.
DOC				10.0	12.5	8.5	9.5	11.0	13.0	
CL	32.0	30.0	33.0	38.0	43.0	22.0	26.0	28.0	37.0	35.0
NO2-NO3	12.00	18.40	6.70	10.40	8.00	6.00	6.90	10.00	8.40	8.60

*=UMHOS/CM

Table A1.2 (Cont.)

SITE= SP1
YEAR= 1979

DATE	05/09	05/24	06/06	06/22	07/05	07/27	08/09	08/28
PH	6.10	6.40	4.49	4.50	5.40	5.30	6.00	6.30
ALK	28.	4.	2.	2.	18.	8.	14.	50.
SP COND*	1500.		5000.	5000.	5000.	5000.	5000.	5000.
T CENT	0		4.0	5.0	7.0	5.0	4.0	6.0
SO4	1100.	4700.	11000.	3800.	5400.	5100.	6500.	3900.
CU T	.020	.100	.300	.200	.200	.100	.100	.070
NI T	1.500	21.000	32.000	17.000	6.800	5.200	21.000	13.000
ZN T	.250	6.900	17.000	6.600	3.900	3.000	4.100	15.000
FE T	.80	8.50	3.00	2.20	2.10	4.60	3.60	.60
CA	123.	201.	0.	0.	17.	0.	0.	0.
MG	20.	0.	243.	43.	262.	151.	450.	364.
AL T	.30	18.00	54.00	14.00	10.00	5.50	10.00	.70

*=UMHOS/CM

CONTINUED

Table A1.2 (Cont.)

SITE= SP1
YEAR= 1979

DATE	09/13	09/27	10/10	10/19	10/29	11/08	11/29
PH	6.50	6.70	6.90	6.60	6.50	6.00	6.80
ALK	44.	70.	76.	54.	60.	50.	64.
SP COND*	4400.	4900.	5000.	5000.	5000.	5000.	5000.
T CENT	4.0	4.0	6.0	3.0	3.0	2.0	0
SO4	3500.	3600.	3500.	3200.	3300.	3500.	3600.
CU T	.080	.090	.090	.070	.080	.060	.060
NI T	10.000	9.000	8.800	8.700	7.500	6.000	5.900
ZN T	1.800	1.500	1.000	1.100	.920	1.000	1.000
FE T	.80	1.50	.90	.90	1.30	2.80	.70
CA	0.	0.	0.	0.	0.	141.	140.
MG	323.	355.	360.	321.	331.	262.	262.
AL T	.50	.40	.20	.50	.70	.80	.30

*=UMHOS/CM

Table A1.3 Seep 2 raw data

SITE= SP2
YEAR= 1976

DATE	07/15	07/27
PH	6.99	
ALK	69.	
SP COND*	383.	
T CENT	16.0	
SO4	2661.	
CU T	.020	1.900
NI T	.104	20.000
CO T		
ZN T		.383
FE T	.67	.26
CA		226.
DOC	36.5	
DIC	3.6	
CL	16.0	

*=UMHOS/CM

SITE= SP2
YEAR= 1977

DATE	03/18
PH	7.00
ALK	29.
SP COND*	620.
NO2-NO3	3.20
TOTAL P	.07

*=UMHOS/CM

Table A1.4 Seep 3 raw data, 1976-1979

SITE= SP3
YEAR= 1976

DATE	07/15	07/27	08/12	08/26	09/08	09/21	10/05	10/25
PH	6.91	6.80	7.10	7.50	7.55	7.25	7.30	7.60
ALK	68.	77.	87.	100.	127.	119.	126.	129.
SP COND*	2399.	2732.	2614.	3234.	2807.	2899.	3000.	2700.
T CENT	1.0		2.5	1.5	1.2	3.5	1.5	1.0
SO4	1036.	2001.	2603.	1539.	1378.	1628.		1420.
CU T	1.140		.753	.617	.637	.736	.516	.481
NI T	24.700		19.000	19.200	20.600	19.500	16.710	15.770
CO T						.857	.972	.896
ZN T		.403	.350	.330	.338	.318	.320	.310
FE T	.46		.61	.73	1.07	1.20	.77	1.14
CU F		1.880	.624	.588		.504	.345	
NI F		20.000	19.000	19.300		20.000	15.420	
FE F		.24	.14	.57		.35	.12	
CA		226.	260.		242.	253.	320.	329.
MG							247.	233.
DOC	15.2	17.5	23.2	23.8	27.1	15.0	11.6	16.8
DIC	9.0	10.0	27.0	13.7	12.1	31.0	28.7	32.6
CL	62.0	58.9	70.4	38.5	57.5	50.1	57.5	58.9
STAGE	4.53			4.50	4.47		4.48	4.44
DCH**	4.25	3.40	2.55	2.27	1.70	1.70	2.10	.82

*=UMHOS/CM **=L/SEC

Table A1.4 (Cont.)

SITE= SP3
YEAR= 1977

DATE	03/18	03/29	04/04	04/12	04/21	04/27	05/05	05/23	05/31
PH	7.79	7.25	7.10	7.10	7.45	7.18	7.32	7.29	7.38
ALK	97.	144.	141.	206.	156.	201.	162.	111.	109.
SP COND*	890.	1000.	1100.	1550.	1375.	1650.	2650.	2470.	2500.
T CENT		2.0		12.0	7.0		11.0		
SO4	297.	456.	746.	857.			372.		564.
CU T	.037	.084	.270	.105	.125	.146	.104	.198	.237
NI T	.420	1.570	3.200	3.380	1.190	1.390	8.000	13.800	17.450
CO T							.440		.790
ZN T							.073		.250
FE T	.43	1.29	5.45	4.56	1.99	3.43	1.91	1.12	1.15
MN T	.80	1.98	2.74	3.60	1.77		4.51	4.60	5.62
CU F	.025	.081	.052		.080	.101	.068	.069	.178
NI F	.380	1.340	.930		1.180	1.150	10.000	13.800	18.200
FE F	.04	.92	.22		.39	.17	.55		.11
MN F	.44	1.54	.84		1.47			4.50	5.78
CA	93.	105.					368.	249.	323.
MG	52.						123.	200.	234.
DOC				26.3	28.0	59.3	20.9	10.8	
DIC				13.4	40.9	53.6	36.4	29.5	
CL	12.8	19.6	22.3	24.7	23.4	33.9	44.7	38.9	33.5
NO2-NO3	9.90	8.80					9.70		
TOTAL P	.07	.04					.04		
STAGE	4.30	4.32	4.31	4.22	4.14	4.07	4.02	4.82	4.81
DCH**	.28	.28	.28	.14	.28	.14	.14	1.02	.79

*=UMHOS/CM **=L/SEC

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Table A1.4 (Cont.)

SITE= SP3
YEAR= 1977

DATE	06/06	06/13	06/20	07/05	07/19	08/01	08/16	08/17
PH	7.15	7.10	6.99	6.68		6.98	6.95	6.99
ALK	114.	90.	88.	70.	47.	66.		81.
SP COND*	2910.	3100.	3080.	3110.	3370.	3550.	3500.	3400.
T CENT	.5	2.5	1.5	2.6		1.5	2.2	
SO4		1864.	1677.	1622.	2048.			
CU T	.826	.803	.854	1.480	1.710	1.220	.952	.889
NI T	35.900	39.800	36.200	28.900	28.500	26.700	24.600	24.400
CO T			1.900				2.400	1.000
ZN T			.650				.580	.580
FE T	2.10	1.08	1.10	.74	.45	.48	.55	.66
MN T	11.10	11.20	10.40	9.07	8.41	8.99	9.35	9.36
CU F	.612	.722	.685					
NI F	33.500	38.400	35.000					
FE F	.22	.23	.17					
MN F	9.82	10.90	10.10					
CA	342.	353.	336.	284.			388.	376.
MG	274.	286.	271.	215.			288.	276.
DOC	18.9							
DIC	28.1							
CL	56.2	46.0		39.0				
NO2-NO3								
TOTAL P								
STAGE		5.01	5.16	5.02		5.03	5.02	5.02
DCH**	3.11	3.40	9.63	16.14	4.81	4.53	4.53	4.53

*=UMHOS/CM **=L/SEC

Table A1.4 (Cont.)

SITE= SP3
YEAR= 1978
CONC= MG/L

DATE	07/24	07/31	08/07	08/21	09/05	09/18	10/02	10/16	10/30	11/17	11/27
PH	6.60	6.60	7.90	6.40	6.30		6.60	6.80	6.80	6.50	7.70
ALK	100.	80.	96.	82.	116.	128.	108.	96.	100.	116.	120.
SP COND*	3500.	3300.	3250.	2600.	2850.	2300.	2500.	2500.	2500.	2250.	1500.
T CONT	4.4	4.4	4.4	4.4	3.9	3.3	4.4	1.1	5.0	1.1	.6
SO4	2750.	2525.	2200.	1825.	1900.	1600.	1800.	1550.	1800.	1350.	1000.
CU T	1.340	.690	.700	.540	.540	.640	.350	.280	.270	.280	.053
NI T	21.500	17.000	13.300	15.200	15.200	13.500	12.200	10.500	10.500	2.900	1.550
CO T	1.600	1.000	.800	.790	.720	.555	.600	.400	.400	.120	.170
ZN T	.580	.350	.300	.230	.230	.240	.160	.140	.130	.070	.040
CA	250.	265.	300.	310.	255.	240.	272.	275.	285.	285.	170.
MG	405.	335.	333.	240.	210.	162.	180.	190.	190.	145.	100.
NA	115.	107.	130.	100.	88.	66.	76.	75.	71.	60.	43.
DOC				8.2	8.5	9.5	8.4	9.9	13.0		
CL	31.0	20.0	23.0	17.0	29.0	10.0	14.0	15.0	19.0	21.0	21.0
NO2-NO3	9.40	11.20	7.80	6.40	5.20	3.20	5.00	6.90	5.00	7.00	7.00

*=UMHOS/CM

Table A1.4 (Cont.)

SITE= SP3
YEAR= 1979

DATE	05/09	05/16	05/24	06/06	06/22	07/05	07/27	08/09
PH	6.90	7.00		6.91	6.80	7.30	6.70	7.20
ALK	108.	116.	126.	370.	128.	118.	90.	88.
SP COND*	1700.	700.	650.	750.	600.	1000.	1600.	1500.
T CENT	4.0	4.0		9.0	4.0	8.0	10.0	12.0
SO4	300.	400.	400.	370.	270.	490.	800.	1300.
CU T	.100	.070	.060	.060	.060	.080	.070	.080
NI T	.460	.290	.270	.330	.270	.400	1.400	6.700
ZN T	.020	.020	.020	.010	.010	.030	.030	.080
FE T	.40	.40	.60	.30	.30	.50	.30	.50
CA	40.	36.	34.	30.	36.	43.	69.	71.
MG	18.	18.	18.	36.	17.	29.	8.	74.
AL T	.10	.10	.10	.08	.08	.08	.08	.08

*=UMHOS/CM

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Table A1.4 (Cont.)

SITE= SP3
YEAR= 1979

DATE	08/28	09/13	09/27	10/10	10/19	10/29	11/08	11/29
PH	7.30	7.00	6.80	7.20	7.10	7.20	7.40	7.30
ALK	84.	94.	78.	84.	104.	116.	112.	152.
SP COND*	2100.	1900.	2300.	2300.	1800.	1600.	1500.	1600.
T CENT	12.0	5.0	9.0	8.0	7.0	7.0	4.0	2.0
SO4	1200.	1000.	1800.	1500.	1000.	1300.	1100.	900.
CU T	.070	.200	.200	.110	.080	.100	.080	.070
NI T	7.800	10.000	20.000	13.000	8.000	6.900	5.400	1.800
ZN T	.080	.180	.320	.130	.080	.070	.060	.030
FE T	.30	.60	.60	.50	.60	1.20	.60	.80
CA	72.	65.	116.	93.	77.	87.	81.	58.
MG	81.	71.	107.	91.	61.	71.	70.	48.
AL T	.08	.08	.08	.08	.20	.08	.08	.08

*=UMHOS/CM

Table A1.5 - Em-1 raw data, 1976-1979

SITE= EM1
YEAR= 1976

DATE	07/01	07/15	07/27	08/12	08/26	09/08	09/21	10/05	10/25
PH	7.50	7.50	7.59	7.55	7.95	7.90	8.00	7.90	8.17
ALK	76.	92.	113.	97.	105.	117.	109.	98.	95.
SP COND*	393.	905.	787.		690.	818.	606.	675.	695.
T CENT	18.0	20.0		17.0	19.1	12.2	8.0	7.3	1.0
SO4	249.	323.	455.	180.	243.	242.	277.		215.
CU T	.004	.006	.007	.004	.003	.004	.002	.002	.001
NI T	.106	.161	.171	.087	.105	.130	.100	.082	.099
CO T							.001		
ZN T									
FE T	.26	.16	.25	.17	.17	.16	.15	.14	.13
MN T									
CU F	.004	.005	.006	.003	.003	.004	.001	.005	
NI F	.108	.158	.170	.082	.095	.105	.110	.069	
FE F	.24	.12	.11	.13		.08	.10	.14	
CA	41.	48.	70.	41.	42.	51.	46.	58.	61.
MG								37.	40.
DOC	12.7	17.0	18.5	27.4	21.0	28.1	10.0	7.7	10.8
DIC	19.2	13.4	18.0	13.7	14.2	9.4	27.0	21.7	23.6
CL	31.9	52.3	43.2	32.3	20.9	29.5	27.5	22.4	20.9
STAGE		6.29	6.02	7.67	6.81	6.75	6.23	6.79	5.98
DCH**		49.27	31.71	233.61	79.00	68.24	30.58	76.45	24.92

*=UMHOS/CM **=L/SEC

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Table A1.5 (Cont.)

SITE= EM1
YEAR= 1976

DATE	10/28	10/29	11/10	11/11	12/07	12/10
PH	7.74	7.85			7.18	7.04
ALK	99.	93.			255.	255.
SP COND*	600.	482.		720.	700.	850.
T CENT	.8	.7		.0		
SO4	161.	131.	176.	174.	140.	159.
CU T	.002	.001	.002	.001	.005	.004
NI T	.092	.067	.098	.078	.220	.225
CO T					.008	
ZN T					.008	
FE T	.07	.05	.09	.07	1.60	.88
MN T					.36	
CU F						
NI F	.076					
FE F	.04					
CA	52.	45.	51.	54.	58.	62.
MG	35.	27.	38.	36.	47.	48.
DOC	12.8	11.1		11.0		
DIC	22.4	21.2		24.1		
CL	19.0	17.5	23.4	45.7	20.2	25.1
STAGE	6.79	7.72	6.64	6.92	5.60	5.98
DCH**	61.52	144.98	50.69	69.37	11.33	11.33

*=UMHOS/CM **=L/SEC

Table A1.5 (Cont.)

SITE= EM1
YEAR= 1977

DATE	01/31	02/22	02/23	02/24	03/15	03/18	03/29
PH	7.30	7.33	7.29	7.50	7.42		7.13
ALK	95.	99.	96.	95.	79.		
SP COND*	389.	369.	290.	315.	314.	530.	580.
T CENT					.5	1.5	
SO4	53.	19.	25.	23.	25.		
CU T	.007	.003	.002	.002			
NI T	.056	.056	.081	.054			
CO T	.000	.002	.003				
ZN T	.012						
FE T	.14	.38	.68	.33			
MN T	.05	.36	.63	.19			
CU F	.003			.002	.003		
NI F	.060			.033	.048		
FE F				.11	.10		
MN F				.01	.17		
CA	27.	28.	31.	27.			
MG	20.	14.	18.	15.			
DOC							
DIC							
CL	13.0	14.8	15.6	13.5	16.9		
NO2-NO3	5.00						.55
TOTAL P	.08						.12
STAGE	7.62		7.79	7.79		7.10	
DCH**			82.12	50.12		94.58	

*=UMHOS/CM **=L/SEC

CONTINUED

Table A1.5 (Cont.)

SITE= EM1
YEAR= 1977

DATE	04/08	04/11	04/19	04/22	04/25	04/27	04/29
PH	7.57	7.48	7.45	7.12	7.41	7.47	7.30
ALK	86.	88.	68.	54.	70.	65.	68.
SP COND*	480.	380.	510.	416.	475.	520.	625.
T CENT	3.0		4.0	4.0	6.0		
SO4	89.	75.	114.		150.	85.	145.
CU T	.001	.002	.001	.003	.002	.002	.002
NI T	.033	.043	.047	.067	.059		
CO T							
ZN T							
FE T	.11	.18	.08	.24	.13	.11	
MN T	.11	.21	.00	.17	.10		
CU F				.004			.002
NI F				.065			.042
FE F				.14			.09
MN F				.15			.04
CA							
MG							
DOC		13.9	18.7	16.4			7.9
DIC		8.8	9.7	13.3			18.5
CL	30.3	20.4	35.2	23.6		43.7	52.5
NO2-NO3							
TOTAL P							
STAGE	6.20	7.63	7.60	7.48	7.72	7.21	7.16
DCH**	24.07	184.90	178.39	163.38	222.85	129.69	126.57

*=UMHOS/CM **=L/SEC

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Table A1.5 (Cont.)

SITE= EM1
YEAR= 1977

DATE	05/02	05/05	05/13	05/31	06/01	06/02	06/13
PH	7.62	7.61	7.45	7.41			7.40
ALK	86.	82.	85.	81.			71.
SP COND*	580.	510.	462.	490.	392.		672.
T CENT			6.0	14.1	12.4		15.0
SO4	137.	125.	114.	145.			175.
CU T	.002	.002	.003	.003			.005
NI T	.038	.038	.063	.072			.087
FE T	.06	.09	.10	.13			.07
MN T	.05	.01		.04			.03
CU F		.002		.003		.004	
NI F		.031		.068		.086	
FE F		.04		.07		.09	
MN F		.18	.04	.04		.04	
CA			32.	38.			49.
MG			26.	31.			36.
DOC	7.6	9.4		12.3			
DIC	18.3	18.8		19.4			
CL	52.5	43.7	26.9	25.7			
TOTAL P		.02					
STAGE	7.01	7.48	6.70	7.18	8.32	8.20	7.36
DCH**	109.87	203.59	90.04	144.98	376.32	360.75	174.99

*=UMHOS/CM **=L/SEC

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Table A1.5 (Cont.)

SITE= EM1
YEAR= 1977

DATE	06/20	07/05	07/19	07/25	08/10	08/16	08/17
PH	7.70	7.22	7.56	7.55	7.62	7.62	7.65
ALK	54.	56.	79.	88.	106.	98.	
SP COND*	620.	640.	950.	960.		807.	725.
T CENT				18.0	15.0	15.5	14.5
SO4		255.	360.		245.	303.	208.
CU T	.003	.004	.002		.002	.002	.002
NI T	.103	.163	.189		.166	.169	.126
FE T	.10	.20	.09		.07	.08	.04
MN T		.04	.04		.04	.04	.14
CU F							
NI F							
FE F							
MN F	.01		.04				
CA							
MG							
DOC							
DIC							
CL							
TOTAL P							
STAGE		8.33	7.34	7.30	6.24	6.88	7.10
DCH**		381.13	171.31	180.94	57.20	127.99	167.35

*=UMHOS/CM **=L/SEC

Table A1.6 (Cont.)

SITE= EM2
YEAR= 1977

DATE	04/12	05/05	05/13	08/16	08/17
PH	7.40	7.30	7.49	7.60	7.50
ALK	96.	70.	81.	79.	102.
SP COND*	800.	900.	1000.	1450.	1374.
T CENT	4.5	10.8			13.3
SO4	229.	273.	311.	492.	555.
CU T	.004	.005	.006	.005	.005
NI T	.027	.022	.064	.033	.031
FE T	.05	.09	.08	.02	.05
MN T	.39		.01	.00	.02
CU F		.004			
NI F		.018			
FE F		.07			
CA			85.		
MG			53.		
DOC		10.9			
DIC		16.3			
CL	58.6	43.7	50.1		
STAGE	6.00	5.95	5.85	5.94	5.84
DCH**	9.34	6.09	1.70	5.52	1.42

*=UMHOS/CM **=L/SEC

SITE= EM3
YEAR= 1976

Table A1.7 Em-3 raw data, 1976-1977

DATE	07/01	07/15	07/27	08/12	08/26	09/08	09/21	10/05	10/25
PH	7.50	7.50	7.70	7.71	7.82	8.30	8.10	8.00	8.30
ALK	84.	93.	97.	88.	104.	102.	106.	96.	90.
SP COND*	410.	910.	847.	499.	725.	383.	617.	850.	800.
T CENT	16.5	16.0		19.5	17.5	16.0	9.8	6.0	.1
SO4	331.	473.	317.	122.	215.	157.	63.	293.	225.
CU T	.005	.006	.006	.003	.003	.003	.002	.001	.002
NI T	.194	.360	.422	.099	.194	.110	.127	.310	.320
CO T							.002		
ZN T									
FE T	.12	.12	.27	.08	.10	.07	.10		.08
MN T									
CU F	.004	.007		.003	.003	.005	.003	.003	
NI F	.081	.352		.095		.098	.154	.242	
FE F	.08	.06		.03	.08	.04	.03	.16	
CA	47.	65.	55.	35.	46.	31.	39.		84.
MG									49.
DOC	10.5	20.3	16.5	17.4	21.0	24.6	7.5	7.3	10.5
DIC	21.1	11.0	15.5	13.0	14.4	9.1	18.0	22.3	21.0
CL		56.9	35.1	33.9	22.5	25.7	24.0	21.8	20.0
STAGE				7.85	6.88		7.78	6.88	6.50
DCH**		33.70	46.72	178.96	38.23	73.62	126.57	27.75	9.34

*=UMHOS/CM **=L/SEC

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Table A1.7 (Cont.)

SITE= EM3
YEAR= 1976

DATE	10/26	10/27	11/10	11/11	12/07	12/10
PH	8.13	7.87		7.10	7.51	7.45
ALK	98.	97.		112.	183.	144.
SP COND*	647.	493.	550.	620.	600.	600.
T CENT	.5		.1	.1		
SO4	166.	120.	173.	155.	160.	104.
CU T	.003	.002	.002	.002	.007	.003
NI T	.244	.103	.130	.092	.220	.126
CO T					.005	
ZN T					.015	
FE T	.17	.09	.09	.16	.80	.05
MN T					.23	
CU F						
NI F						
FE F						
CA	65.	43.	52.	50.	55.	44.
MG	39.	28.	33.	30.	39.	30.
DOC	9.8	11.9		12.0	11.0	
DIC	21.9	22.2		23.6		
CL	19.0	18.1	29.5		16.5	25.2
STAGE	7.26	8.68	7.36	8.21		
DCH**	46.06	134.03	52.38	177.54	5.66	5.66

*=UMHOS/CM **=L/SEC

Table A1.7 (Cont.)

SITE= EM3
YEAR= 1977

DATE	02/22	02/23	02/24	03/14	03/18	04/22	04/25
PH	7.57	7.61	7.72	7.45	7.54	7.22	7.63
ALK	102.	99.	96.	52.	98.	74.	81.
SP COND*	345.	345.	325.	235.	500.	452.	475.
T CENT	1.0	1.0	1.1	1.0	1.5		
SO4	37.	33.	23.	30.		134.	
CU T	.002	.003	.004	.004	.002	.005	.003
NI T	.060	.038	.038	.034	.035	.055	.059
CO T							
ZN T							
FE T	.07	.25	.27	.26	.20	.40	.25
MN T	.05	.07	.19	.07	.11	.25	.20
CU F			.002	.003	.002		.003
NI F			.011	.041	.032		.054
FE F			.12	.10	.07		.03
MN F			.01	.07	.09		.25
CA	32.	30.	30.	23.	44.		
MG	18.	17.	16.	12.	26.		
DOC							7.4
DIC							19.7
CL	19.7	19.2	18.0	12.4		34.8	39.0
NO2-NO3					10.00		
TOTAL P							
STAGE	6.38	7.49	7.58	7.40	7.32		7.64
DCH**	6.23	98.54	111.85	93.16	83.82		135.63

*=UMHOS/CM **=L/SEC

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Table A1.7 (Cont.)

SITE= EM3
YEAR= 1977

DATE	04/29	05/02	05/05	05/13	08/16	08/17
PH	7.60	7.69	7.58	7.69	7.75	7.65
ALK	79.	96.	87.	94.	100.	86.
SP COND*	650.	530.	455.	380.	650.	596.
T CENT						12.7
SO4		122.	127.	143.	205.	173.
CU T	.002	.002	.003	.002	.003	.002
NI T		.044	.056	.059	.188	.174
CO T						.001
ZN T						.003
FE T	.18	.14	.27	.23	.10	.07
MN T		.13	.13	.08	.03	.01
CU F	.002		.002			
NI F	.040		.043			
FE F	.05		.10			
MN F	.13		.10			
CA				27.		
MG				22.		
DOC	7.9	6.8	8.2			
DIC	20.1	19.9	19.4			
CL	66.1	46.8	37.2	26.0		
NO2-NO3			7.20			
TOTAL P			.03			
STAGE	7.56	7.61	7.76	7.35	7.69	7.76
DCH**	123.46	131.10	155.45	94.58	129.69	140.16

*=UMHOS/CM **=L/SEC

SITE= EM4
YEAR= 1976

Table A1.8 EM-4 raw data, 1976-1977

DATE	07/01	07/15	07/27
PH	6.90	6.40	6.80
ALK	27.	32.	36.
SP COND*	56.	85.	83.
T CENT	16.0	17.0	
SO4	11.	77.	78.
CU T	.005	.001	.001
FE T	1.12	1.26	2.36
CU F	.003	.001	.001
FE F	.75	1.13	1.99
CA	3.	4.	5.
DOC	27.7	33.1	30.0
DIC	6.5	4.3	4.5
CL	1.3	3.5	1.4
STAGE		4.62	
DCH**	18.41	10.19	7.36

*=UMHOS/CM **=L/SEC

Table A1.8 (Cont.)

SITE= EM4
YEAR= 1977

DATE	04/12	04/15	04/27	05/06	05/13
PH	6.70	6.41	6.45	6.83	6.22
ALK	17.	14.	7.	7.	11.
SP COND*	85.	110.	95.	122.	170.
T CENT	4.0	2.0		2.0	
CU T	.002	.002	.002	.002	.002
NI T	.002	.002		.003	.004
FE T	.29	.16	.19	.19	.24
MN T	.20	.02		.03	.06
CU F		.003	.003	.002	
NI F			.003		
FE F		.17	.13	.14	
MN F		.02	.03		
CA					6.
MG					6.
DOC	19.3	20.3	18.1	17.9	
DIC	3.1	4.2	2.4	5.4	
CL	4.3	9.7	3.7	3.3	
NO2-NO3				.66	
TOTAL P				.03	
STAGE	5.40	5.12	5.09	5.33	5.00
DCH**	.14	28.32	14.16	28.32	14.16

*=UMHOS/CM **=L/SEC

Table A1.9 EM-4 raw data, 1976-1977

SITE= EM5
YEAR= 1976

DATE	07/01	07/15	07/27	08/12	08/26	09/08	09/21
PH	7.35	7.41	7.50	7.89	7.80	8.10	7.55
ALK	103.	101.	97.	92.	108.	105.	107.
SP COND*	496.	1023.	492.	396.	1280.	529.	756.
T CENT	14.0	14.0	16.0	17.0	14.1	15.2	8.0
SO4	339.	401.	132.	86.		38.	236.
CU T	.011	.012	.010	.006	.008	.006	.005
NI T	.425	.789	.036	.053	.791	.051	.346
CO T							.005
ZN T			.010				
FE T	.19	.08	.20	.29	.07	.16	.10
CU F	.010	.010	.005	.004	.008	.004	.005
NI F	.420	.663	.023	.046	.666	.049	.356
FE F	.10	.03	.09	.03	.02	.03	.03
CA	59.	78.	10.	30.	92.	28.	60.
MG							
DOC	9.7	18.3	15.5		17.3	24.1	8.5
DIC	25.8	14.4	15.5		14.4	8.2	25.0
CL	38.2	34.1	49.9	32.3	20.6	24.0	20.9
STAGE				6.92	5.80	6.84	
DCH**	19.25	14.16	191.70	224.83	11.04	200.76	29.45

*=UMHOS/CM **=L/SEC

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Table A1.9 (Cont.)

SITE= EM5
YEAR= 1976

DATE	10/05	10/25	11/08	11/11	12/07	12/10
PH	7.70	7.85	7.40	7.21	7.45	7.40
ALK	113.	106.	99.	124.	108.	92.
SP COND*	1300.	1130.	525.	675.	240.	195.
T CENT	3.1	.1	.6	1.2		
SO4	616.	422.	167.	161.	17.	5.
CU T	.005	.004	.002	.003	.004	.002
NI T	.890	.770	.115	.054	.011	.005
CO T	.008	.007			.001	
ZN T					.009	
FE T	.06	.07	.06	.24	.12	.18
CU F	.005					
NI F	.722					
FE F						
CA	125.	114.	52.	56.	18.	19.
MG	77.	64.	28.	33.	19.	13.
DOC	7.3	12.1	9.1	15.6	5.2	
DIC	26.7	24.4	20.1	18.1		
CL	22.9	18.6	10.2	40.7	1.5	2.6
STAGE	5.70	5.66	5.62	6.30		
DCH**	6.23	3.96	1.70	87.21	1.70	1.70

*=UMHOS/CM **=L/SEC

Table A1.9 (Cont.)

SITE= EM5
YEAR= 1977

DATE	02/22	02/23	02/24	05/05	05/13	06/28	08/16	08/17
PH	7.56	7.75	7.79	7.72	7.80	7.60	7.60	7.80
ALK	93.	104.	99.	98.	103.	89.	95.	96.
SP COND*	245.	380.	345.	450.	400.	905.	480.	461.
T CENT	7.5	1.5	2.0	9.0		11.0	12.0	12.2
SO4	2.	31.	30.	111.	96.		122.	129.
CU T	.001	.003	.003	.006	.006	.010	.003	.003
NI T	.024	.015	.013	.010	.023	.220	.029	.025
CO T								.001
ZN T								.001
FE T	.08	.28	.29	.20	.35	.31	.16	.16
MN T				.06	.05	.38	.03	.04
CU F				.002				
NI F				.010				
FE F				.05				
MN F				.04				
CA	23.	32.	32.		31.	87.		
MG	12.	18.	18.		23.	50.		
DOC				13.9				
DIC				20.8				
CL	8.3	18.0	18.2	22.5	26.9			
NO2-NO3				7.90				
TOTAL P				.03				
STAGE	5.50	6.42	6.21	6.44	6.42	5.91	6.69	6.70
DCH**	3.26	46.44	55.78	111.28	106.75	31.43	187.17	191.13

*=UMHOS/CM **=L/SEC

Table A1.10 EM-6 raw data, 1976-1977

SITE= EM6
YEAR= 1976

DATE	07/01	07/15	07/27	08/12	08/26	09/08	09/21
PH	7.50	7.50	7.55	7.81	8.01	8.20	8.10
ALK	87.	86.	95.	92.	88.	104.	99.
SP COND*	276.	521.	500.	347.	329.	367.	346.
T CENT	12.0	15.0	17.0	18.0	17.0	15.5	8.8
SO4	146.	119.	123.	74.	78.	32.	
CU T	.006	.006	.016	.009	.007	.006	.003
NI T		.024	.016	.009	.006	.011	.004
CO T							
ZN T							
FE T	.08	.03	.59	.29	.12	.21	.14
MN T							
CU F	.006	.006		.006	.005	.006	.001
NI F		.021		.016	.008	.011	.003
FE F	.07	.03		.03	.06	.03	.03
CA	25.	31.	11.	25.	22.	25.	23.
MG							
DOC	7.0	14.9	17.0	15.8	16.7	24.4	8.0
DIC	21.9	12.0	15.5	12.9	12.4	7.2	23.5
CL	43.9	32.0	49.9	31.4	15.5	23.4	17.4
DCH**	27.18	9.06	172.44	201.04	5.10	174.99	19.25

*=UMHOS/CM **=L/SEC

CONTINUED

Table A1.10 (Cont.)

SITE= EM6
YEAR= 1976

DATE	10/05	10/25	11/08	11/11	12/07
PH	7.90	8.29	7.60	7.21	7.63
ALK	92.	80.	91.	123.	
SP COND*	350.	218.	243.	600.	330.
T CENT	4.8	.1	.9	1.1	
SO4	25.	6.	101.	107.	
CU T	.002	.001	.003	.004	.004
NI T			.005	.005	.002
CO T					.002
ZN T					.003
FE T	.04	.10	.16	.19	.13
MN T					.02
CU F	.002				
NI F					
FE F	.05				
CA	28.	24.	23.	49.	
MG	15.	13.	15.	28.	
DOC	5.5	9.4	8.2	12.6	9.3
DIC	21.2	18.2	17.9	9.2	
CL	11.2	5.9	7.2	41.7	
DCH**	2.27	2.55	.28	85.80	1.42

*=UMHOS/CM **=L/SEC

Table A1.10 (Cont.)

SITE= EM6
YEAR= 1977

DATE	04/04	04/12	05/05	05/13	06/28	08/16	08/17
PH	7.70	7.79	7.71	7.70	7.55	7.60	7.90
ALK	98.	115.	94.	102.	74.	97.	94.
SP COND*	700.	430.	450.	400.	550.	395.	389.
T CENT	3.1	7.0	10.2		15.0	13.3	12.2
SO4	106.	82.	110.	96.	162.	106.	110.
CU T	.003	.009	.008	.004	.004	.003	.002
NI T	.004	.004	.003	.017	.021		
FE T	.21	.45	.38	.27	.17	.16	.15
MN T	.10	.08	.04	.02	.03	.03	.01
CU F	.003		.002				
NI F	.003		.002				
FE F	.04		.04				
MN F	.09		.02				
CA				29.	43.		
MG				23.	29.		
DOC		6.9	7.0				
DIC		22.3	20.8				
CL	63.7	29.0	39.8	26.9			
NO2-NO3			8.70				
TOTAL P			.05				
STAGE							
DCH**	90.04	59.46	109.02	105.90	18.97	177.54	185.75

*=UMHOS/CM **=L/SEC

Table A1.11 Em-9 raw data, 1976-1977 SITE= EM9 YEAR= 1976

DATE	07/06	07/15	07/19	07/26	07/27	08/12	08/26	09/08	09/21	10/18
PH	6.95	6.89	7.10	6.83	7.00	7.00	7.32	7.40	7.35	7.50
ALK	138.	132.	138.	131.	130.	133.	136.	154.	152.	157.
SP COND*		1718.	1930.	1980.	2048.	2305.	2900.	2655.	3018.	2450.
T CENT	1.0	1.0	.5	.5	.5	.5	.9	1.0	1.8	1.1
SO4	796.	973.	1144.	1119.	1095.	1381.	1619.	1529.	1386.	1500.
CU T	.020	.023	.022	.022	.022	.026	.018	.020	.019	
NI T	1.270	1.610	1.670	2.060	1.890	2.330	2.360	2.360	2.520	
CO T										
ZN T						.029	.035	.035	.043	
FE T	.30	.17	.13	.25	.11	.15	.13	.14	.21	
CU F	.017	.019	.020	.020	.021	.023	.016	.018	.018	.035
NI F	.467	1.630	1.600	1.970	1.890	2.360	2.320	2.360	2.490	2.040
FE F	.01	.06	.07	.07	.09	.06	.06	.08	.11	.46
MN F										1.30
CA	152.	62.	163.	168.	24.	228.	241.	248.	244.	284.
MG										158.
DOC	13.2	25.4	26.2	20.0	17.0	28.2	24.9	32.6	12.0	11.0
DIC	34.9	19.5	19.8	28.5	28.0	13.4	19.8	17.3	39.5	
CL	42.0	40.1	42.0	41.4	42.0	55.6	28.8		38.0	31.0
NO2-NO3										12.00

*=UMHOS/CM

Table A1.11 (Cont.)

SITE= EM9
YEAR= 1977

DATE	03/18
PH	7.39
ALK	306.
SP COND*	3490.
SO4	2317.
NI T	3.950
MN T	6.10
NI F	3.860
FE F	.11
MN F	6.15
CA	578.
MG	365.
NO2-NO3	12.00
TOTAL P	.11

*=UMHOS/CM

APPENDIX II

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Figure A2.1

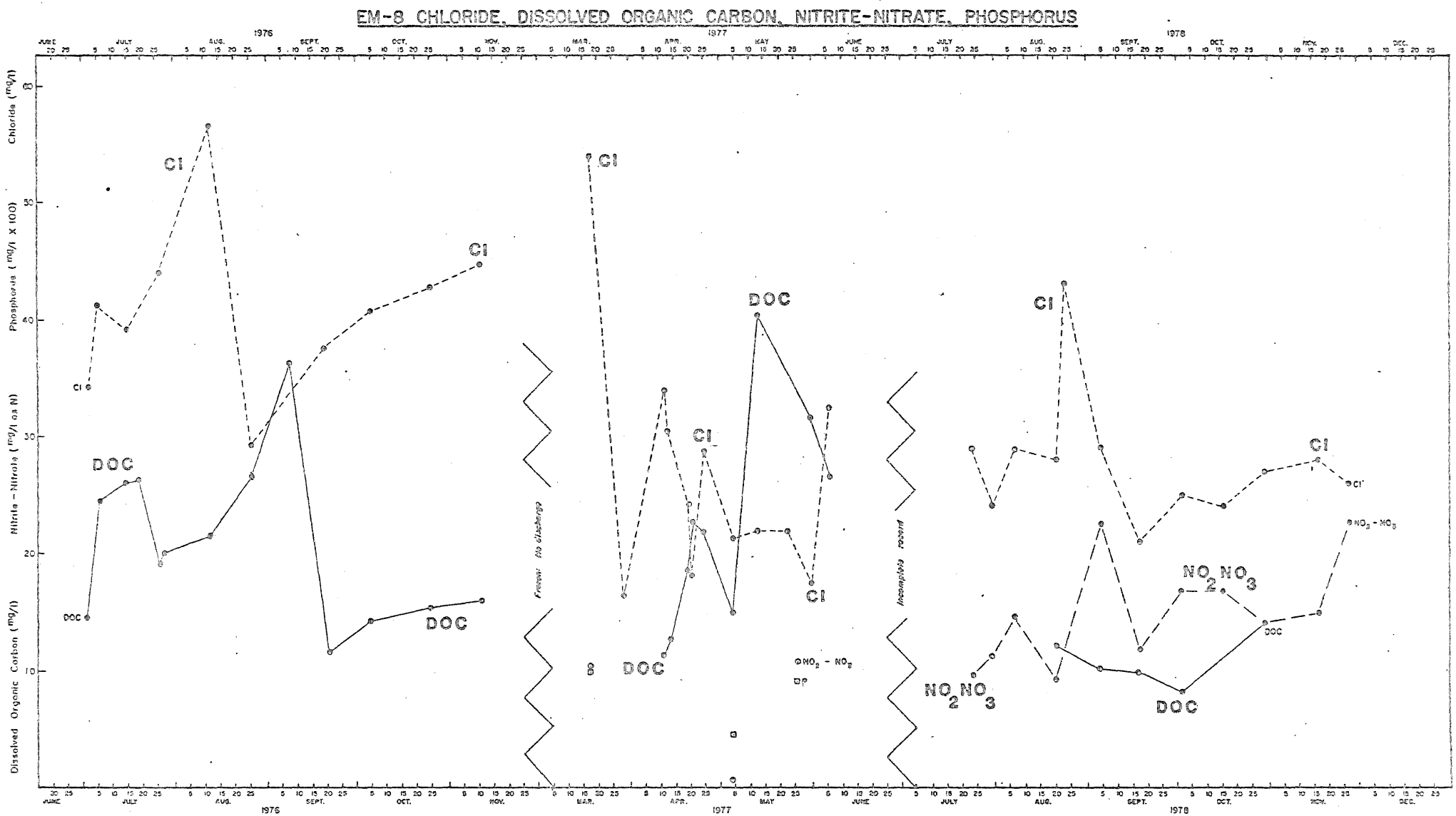

EM-8 CHLORIDE, DISSOLVED ORGANIC CARBON, NITRITE-NITRATE, PHOSPHORUS
Chloride (mg/l)
Phosphorus (mg/l X 100)
Nitrite - Nitrate (mg/l as N)
Dissolved Organic Carbon (mg/l)
1976
1977
1978
JUNE
JULY
AUG.
SEPT.
OCT.
NOV.
MAR.
APR.
MAY
DEC.
CI
DOC
NO2 NO3
Freeze: no discharge
Incomplete record

Figure A2.2

SEEP 3.- CHLORIDE, DISSOLVED ORGANIC CARBON, NITRITE - NITRATE, PHOSPHORUS

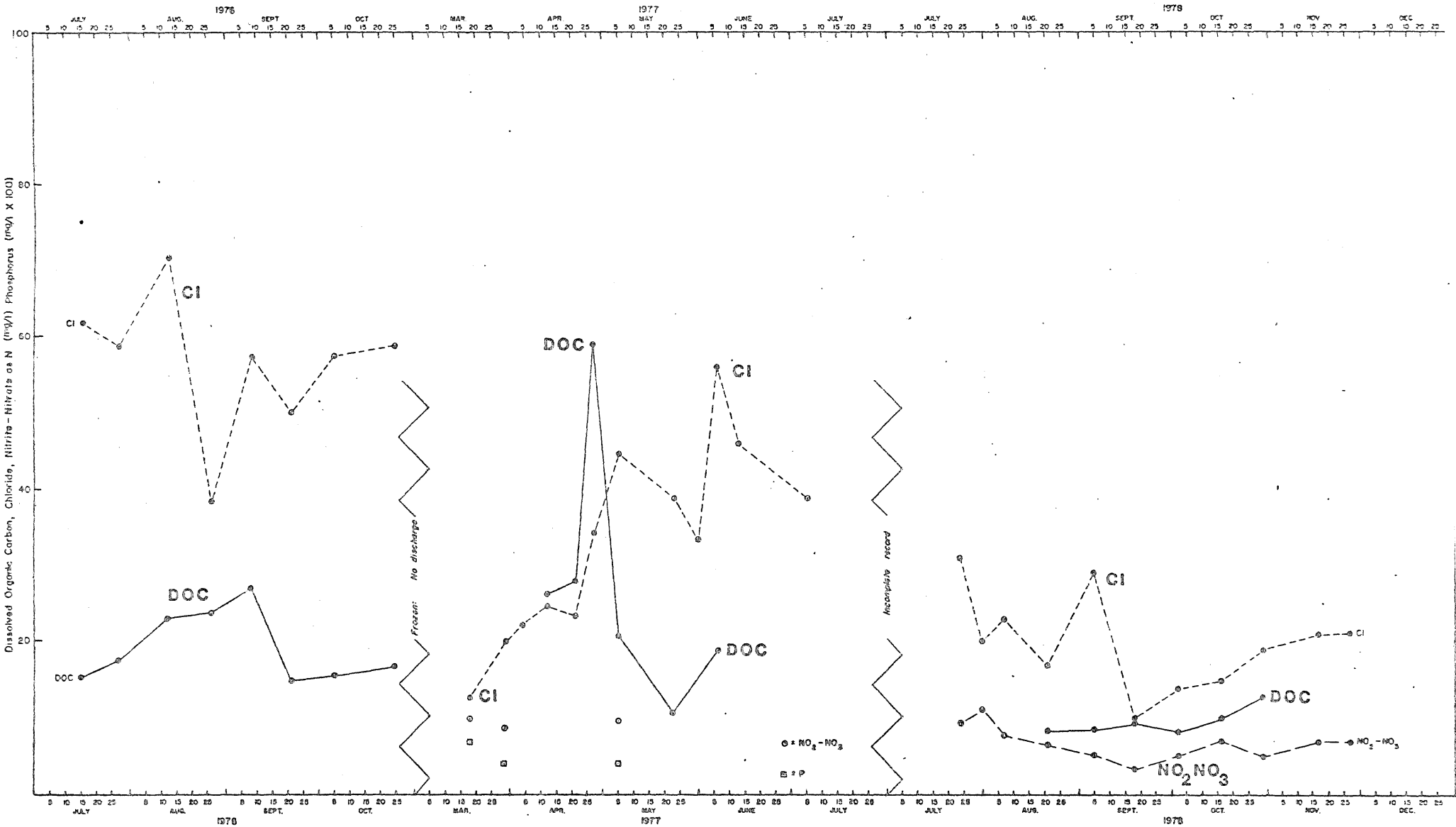


Figure A2.3

EM-1 CHLORIDE, DISSOLVED ORGANIC CARBON, NITRITE-NITRATE, PHOSPHORUS

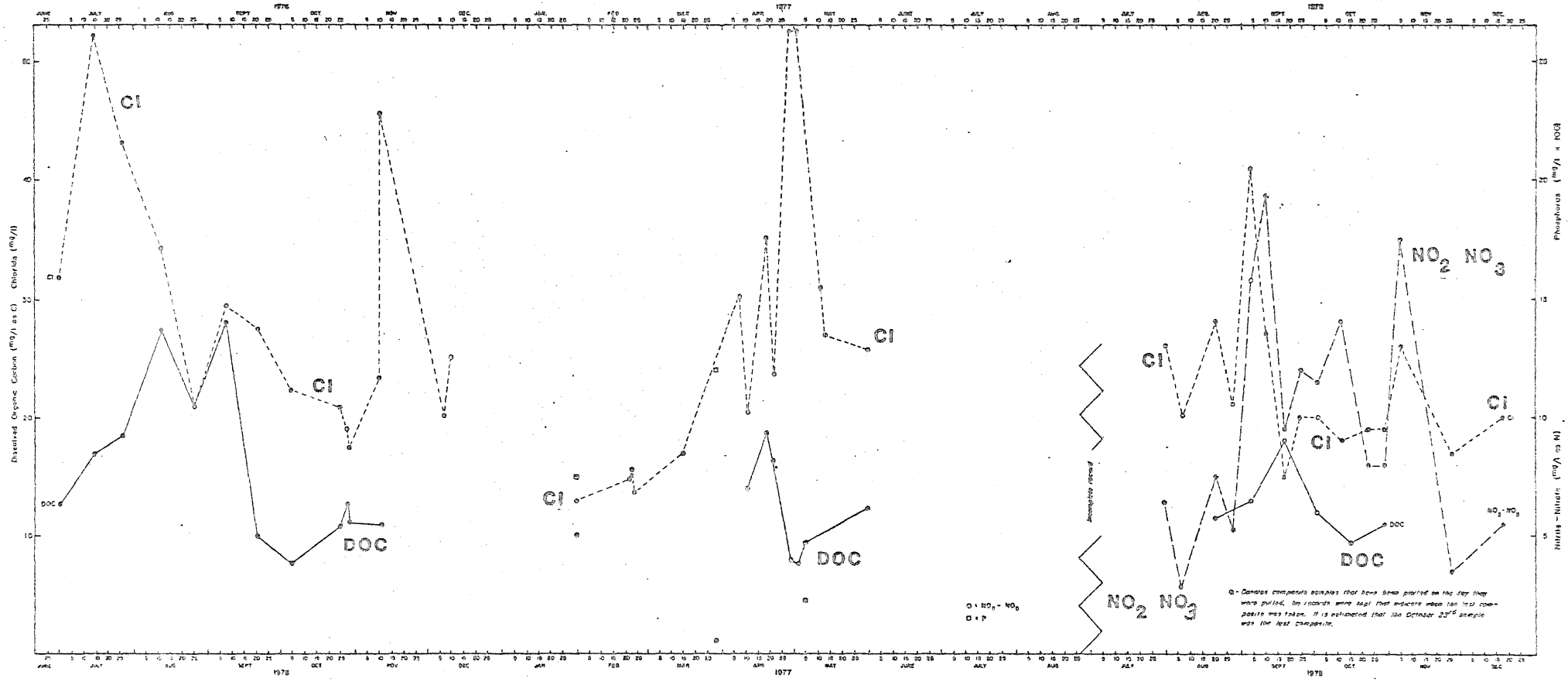


Figure A2.4

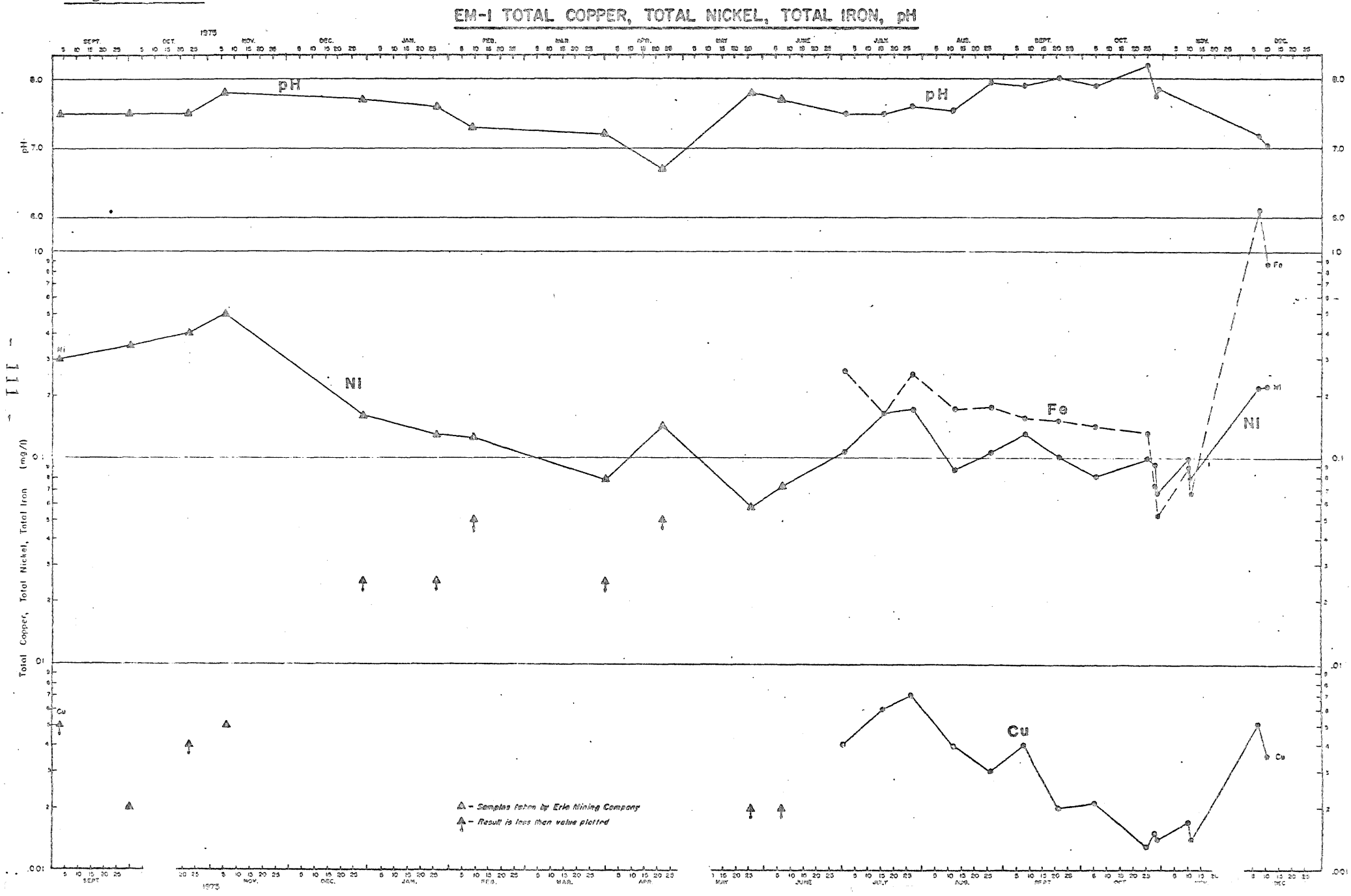

EM-1 TOTAL COPPER, TOTAL NICKEL, TOTAL IRON, pH
1975
SEPT.
OCT.
NOV.
DEC.
JAN.
FEB.
MAR.
APR.
MAY
JUNE
JULY
AUG.
SEPT.
OCT.
NOV.
DEC.
pH
8.0
7.0
6.0
Ni
Fe
Cu
10
1.0
0.1
.01
.001
Total Copper, Total Nickel, Total Iron (mg/l)
Δ - Samples taken by Erie Mining Company
Δ - Result is less than value plotted

Figure A2.5

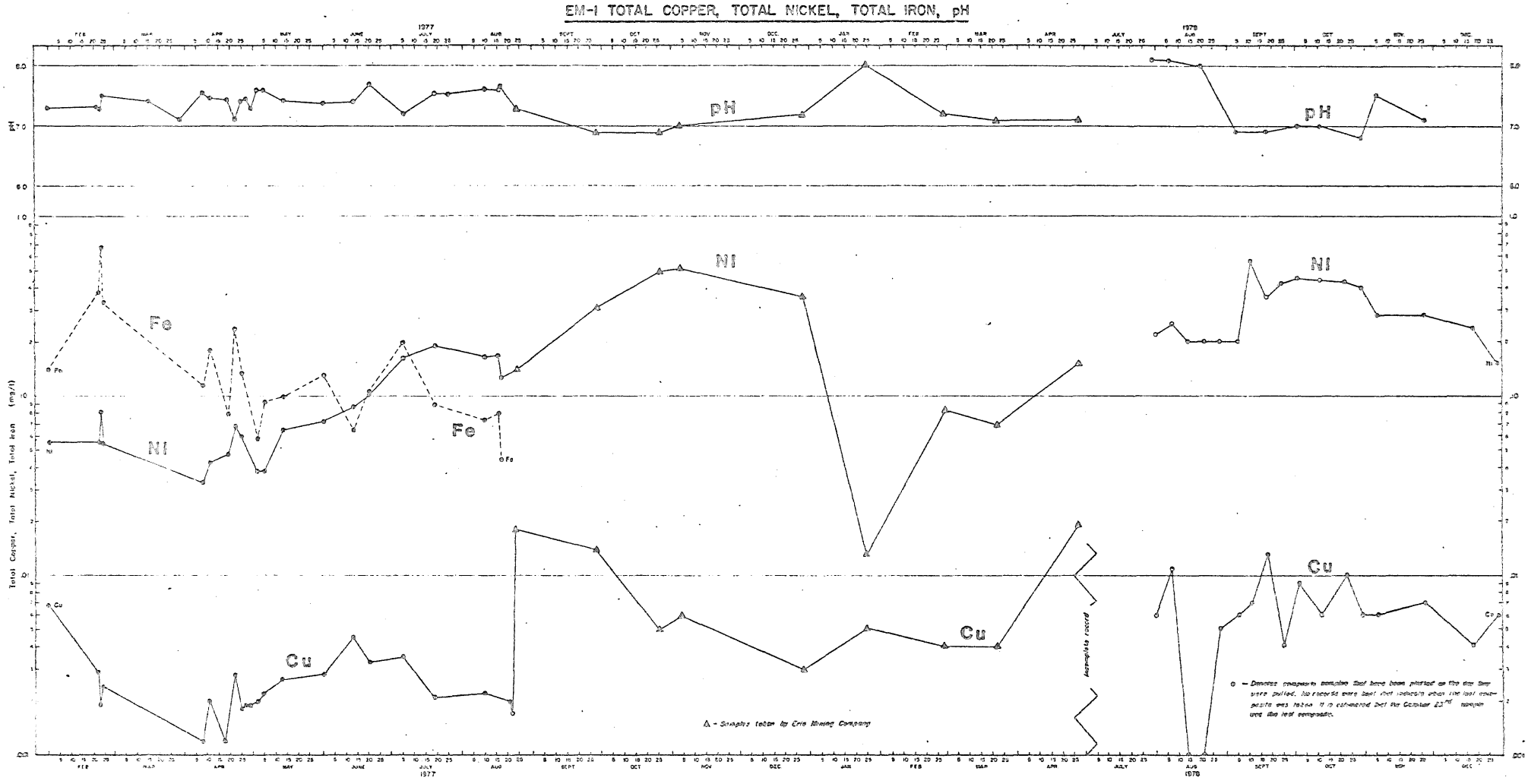

EM-1 TOTAL COPPER, TOTAL NICKEL, TOTAL IRON, pH
1977
1978
pH
Fe
Ni
Cu
Total Copper, Total Nickel, Total Iron (mg/l)
Δ - Samples taken by Erie Mining Company
hemoglobin record

Figure A2.6

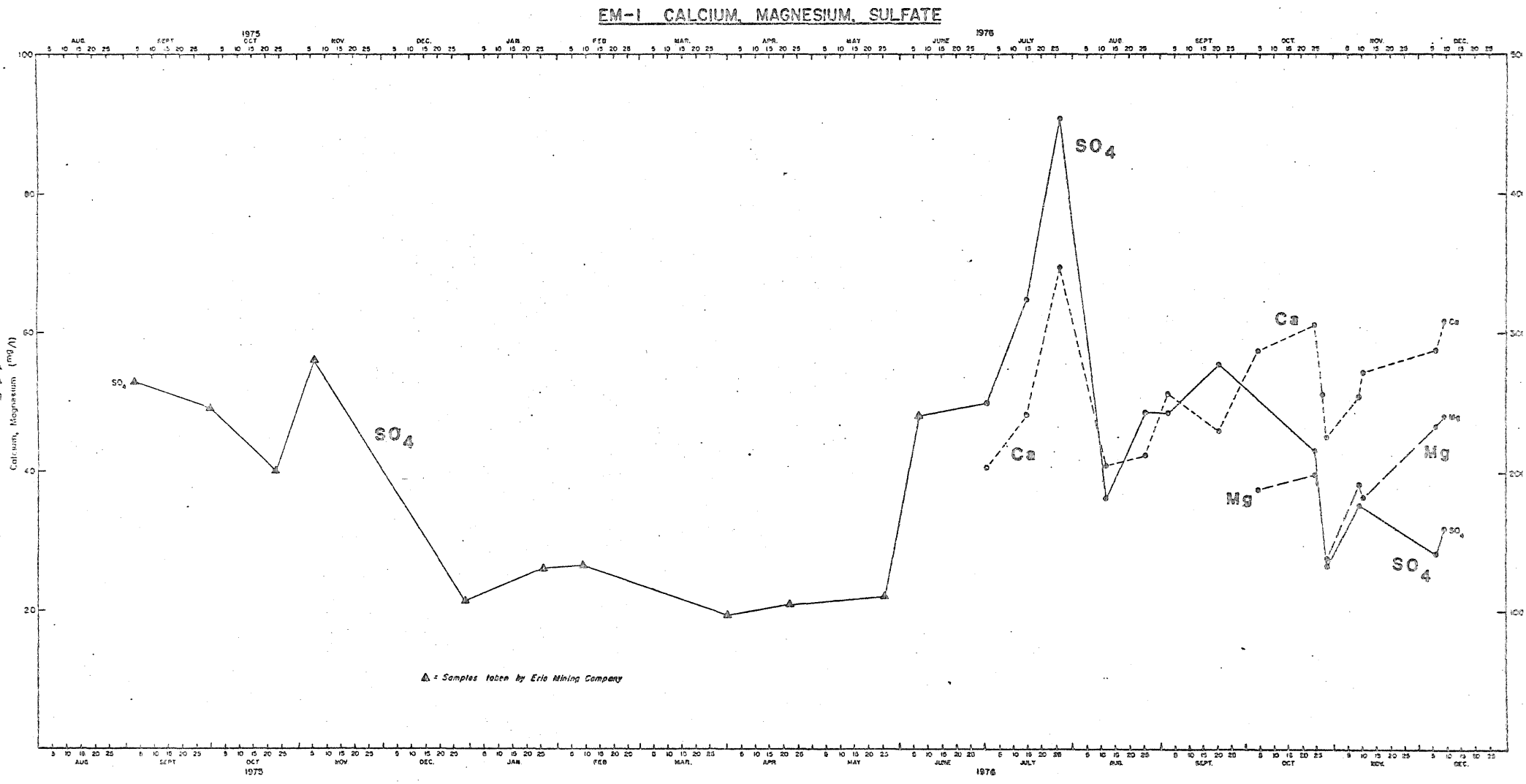

EM-I CALCIUM, MAGNESIUM, SULFATE
1975
1976
Calcium, Magnesium (mg/l)
SO4
Ca
Mg
▲ = Samples taken by Erie Mining Company

Figure A2.7

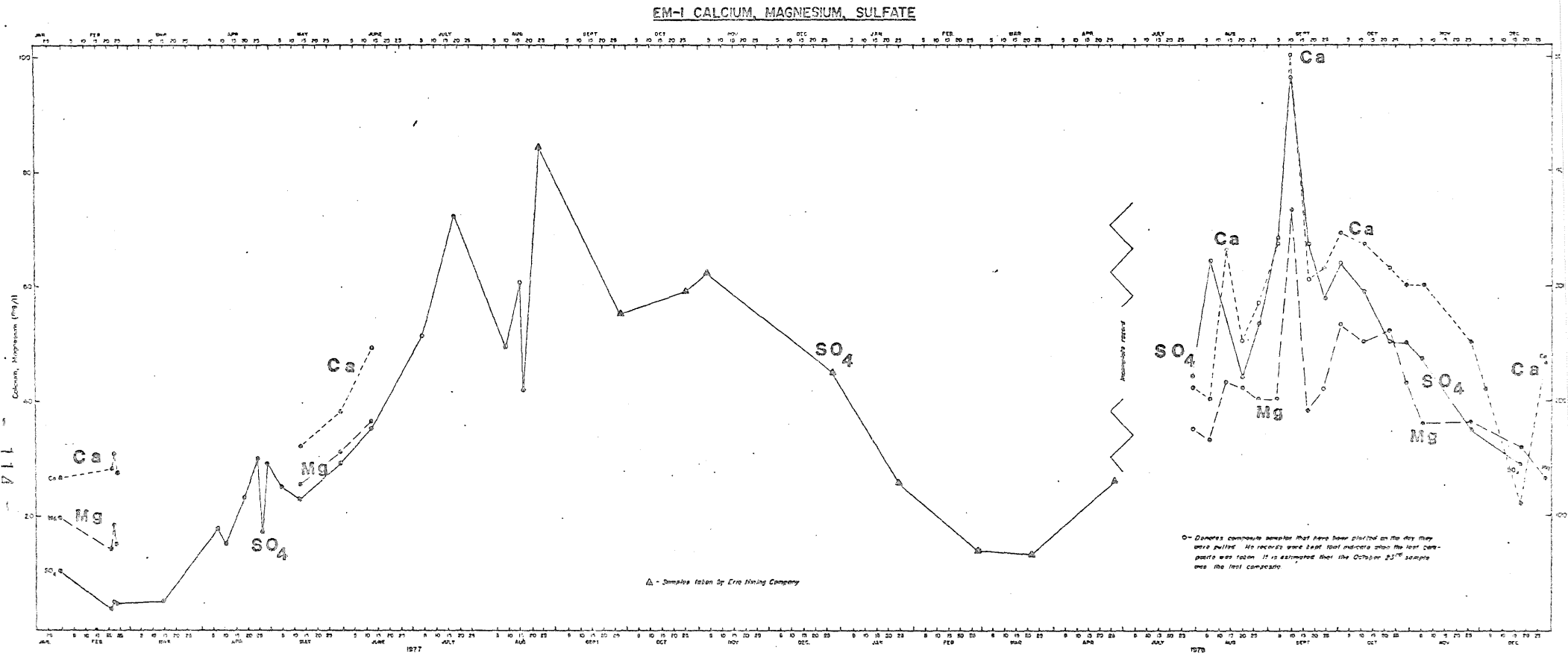
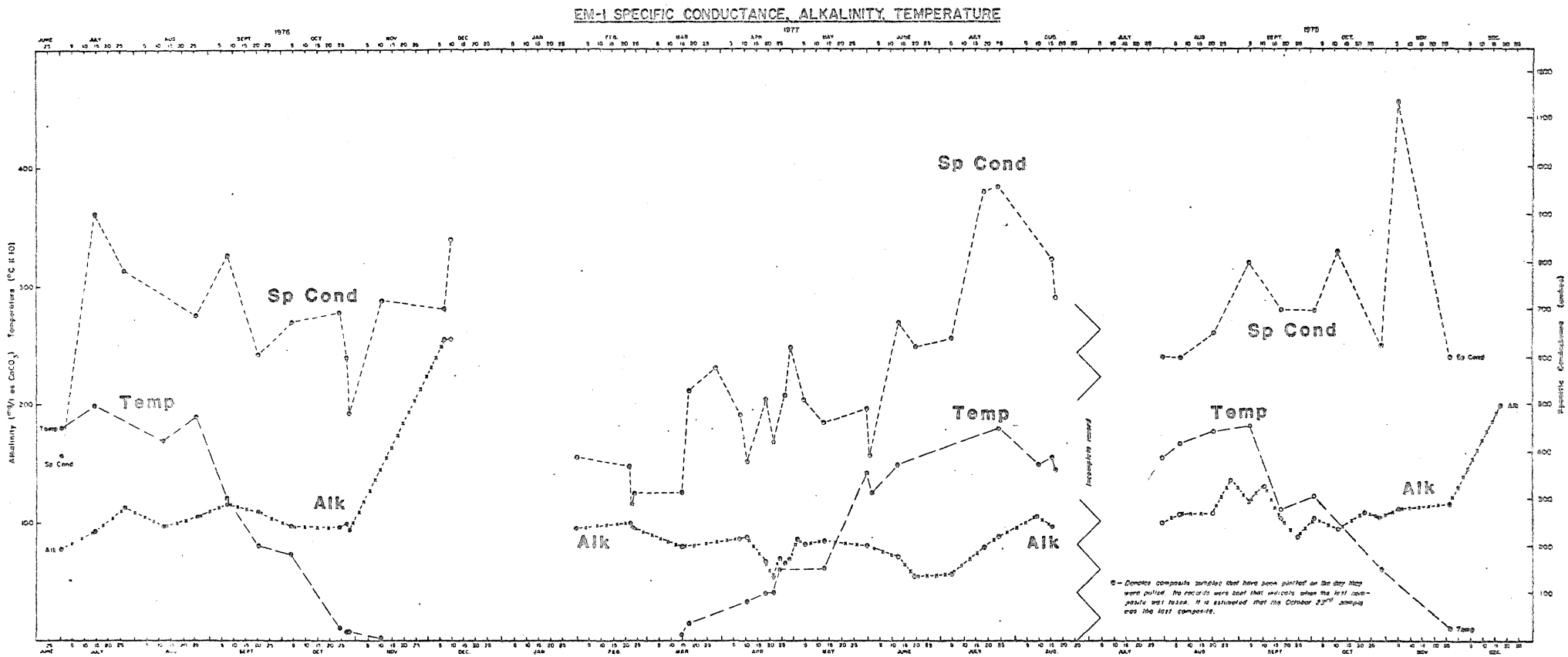

EM-I CALCIUM, MAGNESIUM, SULFATE
Calcium, Magnesium (mg/l)
Ca
Mg
SO4
Ca
Mg
SO4
SO4
Ca
Mg
SO4
Ca
Ca
Ca
SO4
Mg
SO4
Mg
Ca
Δ - Samples taken by Ero Mining Company
O- Denotes composite samples that have been plotted on the day they were pulled. No records were kept that indicate when the last composite was taken. It is estimated that the October 83rd sample was the last composite.
1977
1978

Figure A2.8


EM-1 SPECIFIC CONDUCTANCE, ALKALINITY, TEMPERATURE
1976
1977
1978
Sp Cond
Temp
Alk
Incomplete record
Alkalinity (mg/l as CaCO3) Temperature (°C x 10)
Specific Conductance (umhos)
JUNE
JULY
AUG
SEPT
OCT
NOV
DEC
JAN
FEB
MAR
APR
MAY
JUNE
JULY
AUG

APPENDIX III

ANALYTICAL PROBLEMS

Erie Mining Company
A LIMITED PARTNERSHIP
Pickands Mather & Co., Managing Agent
P.O. Box 847
Hoyt Lakes, Minnesota 55750

Phone 218-225-2171
Telex 29-4456

December 21, 1979

Mr. Kim Lapakko
Division of Minerals
Department of Natural Resources
Centennial Building
St. Paul, Minnesota 55155

Re: Erie Mining Company
NPDES Permit MN0042579

Dear Mr. Lapakko:

This letter is a confirmation of your phone conversation today with J. J. Garmaker and W. H. Boutwell of our staff, and is in reference to your letter dated December 17, 1979. We will analyze the November Seep 1 samples for sodium and include the results in the December report. For November, the calcium and magnesium hardness results in parts per million $CaCO_3$ were calculated from the atomic absorption analysis for calcium and magnesium. All future calcium and magnesium determinations will be done by the atomic absorption spectrophotometer and reported in ppm Ca and ppm Mg. Our laboratory staff discontinued the titration method for total hardness, calcium hardness, and the derived magnesium hardness after discovering an interference with the calcium hardness during analyses of the November samples. If you have questions regarding this matter, please let me know.

Very truly yours,

ERIE MINING COMPANY

J. B. Bowen

J. B. Bowen
Ass't. Manager-Staff Services

JBB:cl-RC

CC: Director, Minnesota Pollution Control Agency.
P. D. Brick

Any liability or obligation of Erie Mining Company, a limited partnership, shall be limited solely to its assets and no personal liability whatever shall attach to, or be incurred by, any partner, as such, by reason of the obligations or agreements of the limited partnership

Attachment 5

MN DNR (1996) Dunka Data Summary : 1976-1993 DRAFT. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN.

Dunka Data Summary: 1976-1993

DRAFT

March 15, 1996

Minnesota Department of Natural Resources
Division of Minerals
St. Paul, Minnesota 55155-4045

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1. Introduction.

This report summarizes the drainage quality and chemical mass release from stockpiles at the Dunka site over the period from 1976 through 1993. The data presented are those currently on computer files at the Minnesota Department of Natural Resources, Division of Minerals (MNDNR). Other data relevant to closure at the site may be available and can be added to report revisions. Data from the site were previously compiled in 1981 (Eger et al. 1981) and 1987 (MNDNR 1987). A literature review, site description, and methodology are presented in the 1981 report. A summary of MNDNR sampling in the summer of 1993 has also been compiled.

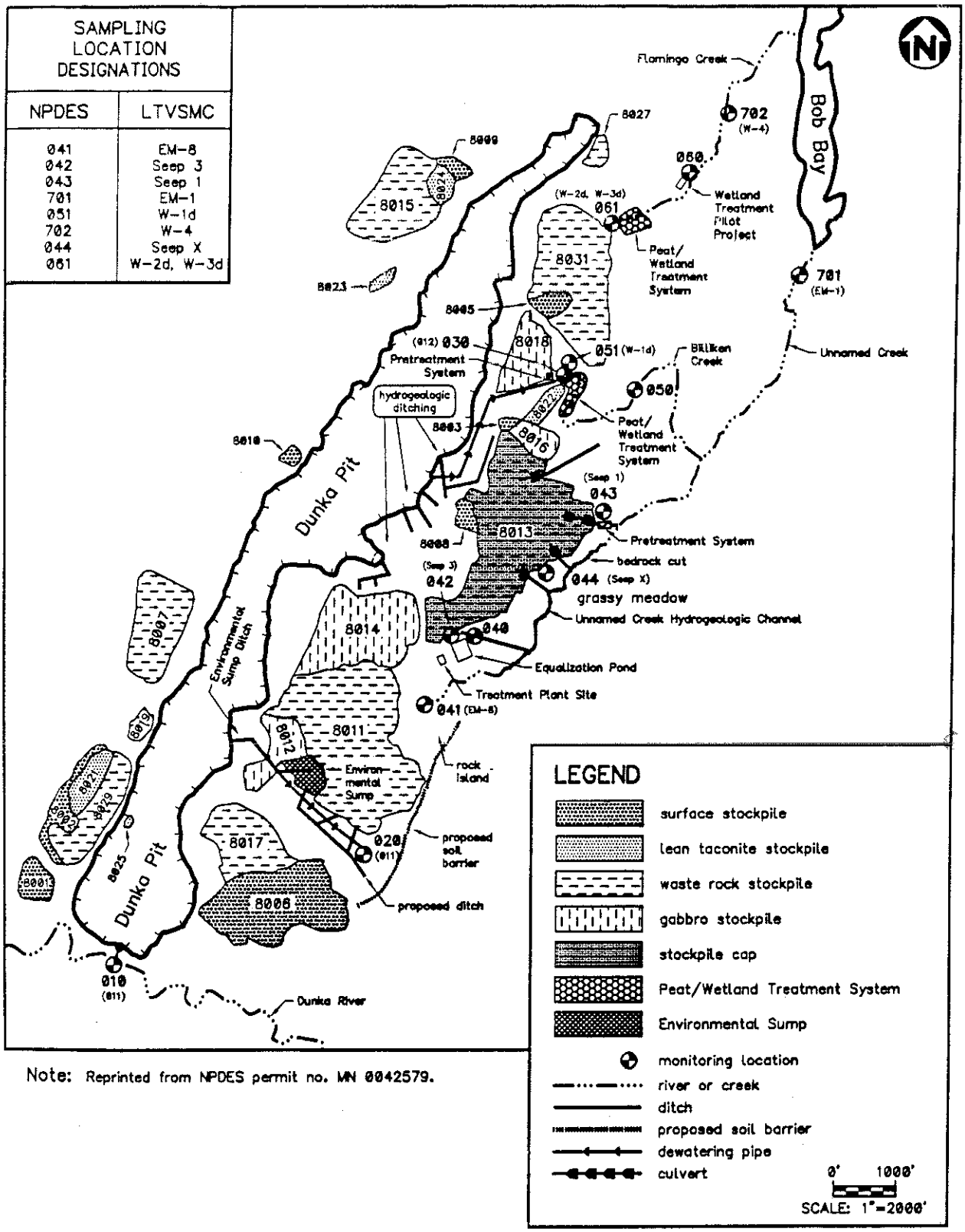
2. Site description.

Beginning in the 1960's, Duluth Complex rock was excavated and stockpiled at the Dunka site in order to access the underlying iron formation. This stripping and stockpiling was and is common practice for iron mining operations. In the mid-1970's it was discovered that trace metal concentrations were elevated in Unnamed Creek, which drains the mining watershed. The metal sulfide minerals present in the stockpiled Duluth Complex rock were determined to be the source of the metals in the creek. The mass, basal area, and composition of the piles are summarized in table 1. Additional data on the stockpiles are presented in appendix 1.

The metals are transported from the stockpiles in several well-defined seeps, as well as in diffuse flows. The seeps (and the associated stockpiles) discussed in this report are designated EM8 (stockpiles 8011, 8014), Seep 3 (8013, 8014), Seep X (8013), Seep 1 (8013), W1D (8018, 8031), W2D (8031), and W3D (8031). The drainages and stockpiles are identified in figure 1. The areas, masses, and compositions of the stockpiles in the seep watersheds are presented in table 2.

The first five of the aforementioned drainages are in the watershed of Unnamed Creek, which flows into Bob Bay on Birch Lake (figure 1). From mid-April through the end of October, EM8 (since 10/15/91), Seep 3 (5/11/90), Seep X (4/1/92), and Seep 1 (9/13/90) are pumped into the collection basin near Seep 3 (STS 1993, p. 99-100). Seep 1 flow first passes through a limestone/peat pretreatment system. Collection basin water is then treated at the lime treatment plant. W1D flows through a limestone/peat pretreatment system, then through a peat treatment system.

The W2D/W3D drainages converge a short distance from the 8031 stockpile and are treated in a limestone/peat passive treatment system. Flow from these two sites passes through station W4 and subsequently flows into Bob Bay (figure 1).



Note: Reprinted from NPDES permit no. MN 0042579.

Figure 1. Dunka Mine area map.

Table 1. Dunka rock stockpile summary. Data source (except where noted): 1993 STS Dunka Closure Report.

Stockpile No.	Classification Based on Content	Location Versus Side of Mine Pit	Size		Chemical Composition			Deposition History	
			Area[1] $ft^2 \times 10^6$	Mass LT $\times 10^6$	% NiO	% CuO	% S	Start	End
8011	Waste Rock	East	3.73	20.50	0.03	0.06[2]	0.23	1965	1985
8013	Waste Rock	East	3.58	17.75	0.03	0.09	0.24	1967	1991
8015	Waste Rock[3]	West	NA	4.80	0.02[2]	0.11[2]	1.00[2]	1968	1985
8031	Waste Rock	East	3.17	9.50	0.04	0.10	0.35	1979	1985
8027	Waste Rock	East	NA	0.14	<0.01[2]	0.01[2]	0.12[2]	1979	1979
8012 N/S	Gabbro	East	0.35	1.20	0.07[5]	0.27[4]	NA[6]	1967	1991
8014	Gabbro	East	2.40	7.00	0.09	0.28	0.73	1967	1993
8018	Gabbro	East	0.76	2.20	0.09	0.28	0.98	1979	1985

[1] Pile footprint areas determined by Steve Dewar of MNDNR.
[2] Chemical analysis based on geologic survey.
[3] Data is for the north side of the pile.
[4] 0.19% CuO based on geologic survey.
[5] 0.07% NiO based on geologic survey.
[6] 1.08% S based on geologic survey.

Table 2. Watershed characteristics summary.

Site	Watershed Area, $ft^2 \times 10^6$	Pile	Area of Pile in Watershed, $ft^2 \times 10^6$	Mass in Watershed, LT x 10^6	Composition			Deposition History	
					%NiO	%CuO	%S	Start	End
EM8	7.7[1]	8011	3.6	12.92	0.03	0.06	0.23	1965	1985
		8014	1.9	2.42	0.09	0.28	0.73	1967	1993
Seep 3	1.9	8013	0.35	0.89	0.03	0.09	0.24	1967	1991
		8014	0.40	3.46	0.09	0.28	0.73	1967	1993
Seep X	1.7	8013	1.3	8.44	0.03	0.09	0.24	1967	1991
Seep 1	0.65	8013	0.50	1.86	0.03	0.09	0.24	1967	1991
W1D	1.9	8018	0.75	2.18	0.09	0.28	0.98	1979	1985
		8031	0.35	0.04	0.04	0.10	0.35	1979	1985
W4	7.5	8027	0.12	0.14	<0.01	0.01	0.12	1979	1979
		8031	2.0	6.55	0.04	0.10	0.35	1979	1985

[1] Ditching caused the watershed area to drop from 17 x 10^6 ft^2 to 9.2 x 10^6 ft^2 in August 1991, then to 7.7 x 10^6 ft^2 in spring 1993.

3. Flow.

The accuracy of the flow data presented is limited by methods of measurement, frequency of measurement, and the period over which measurements were made. First, the methods of flow measurement were of variable accuracy. Some values were determined by continuous flow recorders while others were estimated. Second, the period of flow record varies among sites. For example, data for the EM8 drainage have been collected since 1976. In contrast, regular flow measurement at W1D began in 1986 and Seep X was not discovered until 1990.

Third, the period over which annual flow was measured was of variable length. The annual flow record at EM1 was typically from the end of April through the middle of November (table 3), and at times may have been limited by the ability to measure flow in the winter months due to ice conditions. The annual flow records at Seep 1 and Seep 3 were similar to those at EM1, while those at W1D (1986-1993) extended to the beginning of December.

The annual flow records at EM8 were similar to those at EM1 from 1979 to 1986. Subsequently, the EM8 record increased by about 3.5 months, extending from mid-February to the fourth week in December. The annual flow record at W4 also increased in 1987, although to a lesser extent (table 3). These factors, as well as watershed modifications (table 4), must be considered when interpreting flow data and applying these data to other calculations.

The annual flow volume at EM1 averaged close to 4000 million liters, typically ranging from 2000 to 6600 million liters (figure 2, table 5). The five seeps in the watershed contributed about 300 million liters per year, or six percent of the EM1 flow. EM8 contributed the most seepage flow, typically 100 to 300 million liters/yr (figure 3). The next highest contributions were from Seep X and W1D, with typical contributions of 50 to 70 and 20 to 50 million liters per year. Seep 1 and Seep 3 each typically ranged from 5 to 10 million liters per year.

The volume of stockpile drainage is largely a function of the amount of precipitation which falls onto the stockpile. To normalize for both watershed area and precipitation, annual yield coefficients were calculated as follows:

$$y = V/(PA \times 1000), \text{ where}$$

y = annual yield coefficient (dimensionless),
V = annual volume of stockpile drainage (liters),
P = annual precipitation (m), and
A = drainage area (m^2).

Table 3. *** Note JW 3/21/96: This needs to be updated to include 1993 data.

Site	Drainage Quality			Flow		
	Years	From	To	Years	From	To
EM8	75-86	03/27[2]	11/21[3]	79-86	04/18	11/10
	87-92	01/26	12/25	87-91	02/15	12/23
EM8E	88	04/12	07/11	88	NA	NA
Seep 3	75-86	04/13[2]	11/25	77-86	05/02	11/21[4]
	87-92	04/22[5]	11/22[5]	87-90	04/24	11/22
Seep X	90-92	02/17	12/23	90-92	01/08[6]	12/28[5]
Seep 1	76-86	04/18[7]	11/15[8]	86	04/18	11/20
	87-92	04/21	11/03	87-92	04/22[5]	10/28[5]
W1D	80-86	04/16[9]	11/03[10]	86	04/18	11/20
	87-92	03/30	12/12	87-92	03/29[5]	12/01[5]
050	92	04/09	11/24	NA	NA	NA
EM1	75-86	02/14	12/17[11]	77-86	04/22	11/10[11]
	87-92	01/29	12/19	87-91	04/23	11/17
W2D	79-86	04/20[12]	10/29[13]	NA	NA	NA
	88	05/17	10/19	NA	NA	NA
W3D	79-82	04/17[14]	10/29[15]	NA	NA	NA
W4	80-86	04/11	11/24	80-86	04/18[16]	11/16
	87-92	02/24	12/22	87-91	03/03	12/23

[1] Filtered metal concentrations were substituted where total concentrations were missing, then periods were determined.
[2] Excludes 1975 date.
[3] Excludes 1975 and 1977 dates.
[4] Excludes 1977 and 1985 dates.
[5] Excludes 1992 date.
[6] Excludes 1990 and 1992 dates.
[7] Excludes 1976 and 1978 dates.
[8] Excludes 1977 date.
[9] Excludes 1983 date.
[10] Excludes 1980 and 1982 dates.
[11] Excludes 1978 date.
[12] Excludes 1979, 1980, and 1983 dates.
[13] Excludes 1980, 1982, 1983, 1984, and 1986 dates.
[14] Excludes 1979 date.
[15] Excludes 1982 date.
[16] Excludes 1980 date.

Table 4. Modifications in seepage watersheds.

Environmental sump:

1980: 77 million liters pumped
1981: 106 million liters pumped
1984: 15 million liters pumped
1985: 176 million liters pumped
1986: 183 million liters pumped
1991: Ditch from environmental sump to pit completed (July 31)
1993: Ditch blasted from SE end of 8011 to environmental sump

EM8: Continuous flow (60° v-notch weir, stilling well, Stevens A-35 recorder) installed 1976.

1988: Mud wave at stockpile toe blasted (June).
New weir installed beginning in June and completed July 11, 1988.
1991: Ditch blasted from EM8-E to weir (August). ???Is this year correct???
1992: Seep pumped to basin for subsequent treatment (April 16)

Seep 3: Continuous flow equipment (60° v-notch weir, stilling well, Stevens type F recorder) installed 1977. (The 1981 report (p. 13) indicates that flow was estimated in 1978.)
1990: Pumped to basin (May 11)

Seep X: Continuous flow recording equipment installed 11 September 1991.
1992: Pumped to basin (April 27)

Seep 1: Weir installed in 1977 and recorder in 1980 (?)
1989: Erosional channel, perhaps due to beaver dam failure, reportedly lowers water table (STS 1993 Final Closure Report, p. 63)
1990: Pumped to basin (13 September).
1990-91: Ditching past grassy meadow and "bedrock cut area" or "bedrock dam" (winter).
1992: 8013 stockpile capped
1992: Limestone pretreatment system constructed (August).

W1D: Weir and recorder installed November 1991 and June 1993, respectively.
1992: Flow directed to wetland treatment system (March)
1992: Limestone-peat pretreatment system constructed (July)

EM1: Culvert and continuous recorder installed 28 July 1988.
??Was there continuous flow measurement prior to this time?
1982: EM8-4 mine dewatering switched from Unnamed Creek to Dunka River (23 June).??
1990-91: Ditching of Unnamed Creek (winter)

W2D-W3D: Continuous flow monitoring at W4 and near treatment cells??
1992: Limestone-peat treatment system installed (March)

ADDITIONAL?? Other ditching: to pit, to intercept flow. Mine dewatering? OTHER?

Table 5. Annual flow volumes at the Dunka site, 1976-1993.

Year	Precip. cm	Annual flow volume, L x 10^8								
		EM8	EM8-9	Seep 3	Seep X	Seep 1	W1D	EM1	EM1-Pumping	W4
76-77[1]	97	2.14	NP	0.90	ND	0.29	NC	35.1	7.2	NA
78[1]	70	2.96	NP	0.08	ND	0.09	NC	61.3	16.6	NA
79[1]	63	2.88	NP	0.07	ND	0.09	NC	60.6	19.3	NA
80[1]	68	2.50	0.77	0.07	ND	0.13	0.47	49.8	11.7	NA
81[1]	70	1.21	1.06	0.12	ND	0.10	0.42	57.8	21.5	2.37
82[1]	87	1.97	NP	0.09	ND	0.09	0.35	38.3	23.5	4.01
83[1]	78	1.46	NP	0.10	ND	0.07	0.26	21.6	18.1	2.79
84[1]	56	0.86	0.15	0.05	ND	0.06	0.27	18.7	14.8	1.96
85[1]	84	0.85	1.76	0.09	ND	0.12	0.47	27.1	20.6	3.01
86[1]	68	0.81	1.83	0.09	ND	0.09	0.53	22.4	19.4	2.33
87	56	1.04	NA	0.07	ND	0.06	0.43	31.5	27.6	1.08
88	79	3.96	NA	0.12	ND	0.10	0.69	65.9	61.0	1.92
89	66	4.01	NA	0.08	ND	0.21	0.42	44.4	40.1	1.53
90	71	2.35	NA	0.05	0.57[2]	0.04	0.30	40.9	39.5	1.41
91	74	2.54	NA	0.01	0.72	0.11	0.31	27.6	27.6	1.50
92	65	1.50	NA	0.01	0.57	0.07	0.38	25.4	25.4	1.15[3]
93	77	1.99	NA	0.01	0.41	0.07	0.40	15.3	15.3[4]	1.09
Average	71	2.06	1.36[5]	0.07[6]	0.57[7]	0.08[8]	0.44[9]	37.9	24.1	2.01

NP: No or sporadic pumping. ND: Not discovered until 1990
NC: Stockpile 8018, which feeds W1D, wasn't constructed until 1979.
[1] Data for 1976 to 1986 are from the 1987 Dunka Data Summary.
[2] Flow measurements at Seep X began in July 1990.
[3] Value based entirely on estimated flows.
[4] Pumping records are unavailable, but no pumping is believed to have occurred.
[5] Average excludes 1984 value.
[6] Average excludes 1976-1977 value.
[7] Average excludes 1990 value.
[8] Average of 1986 to 1993 values, excluding 1989.
[9] Average of 1985 to 1993 values.

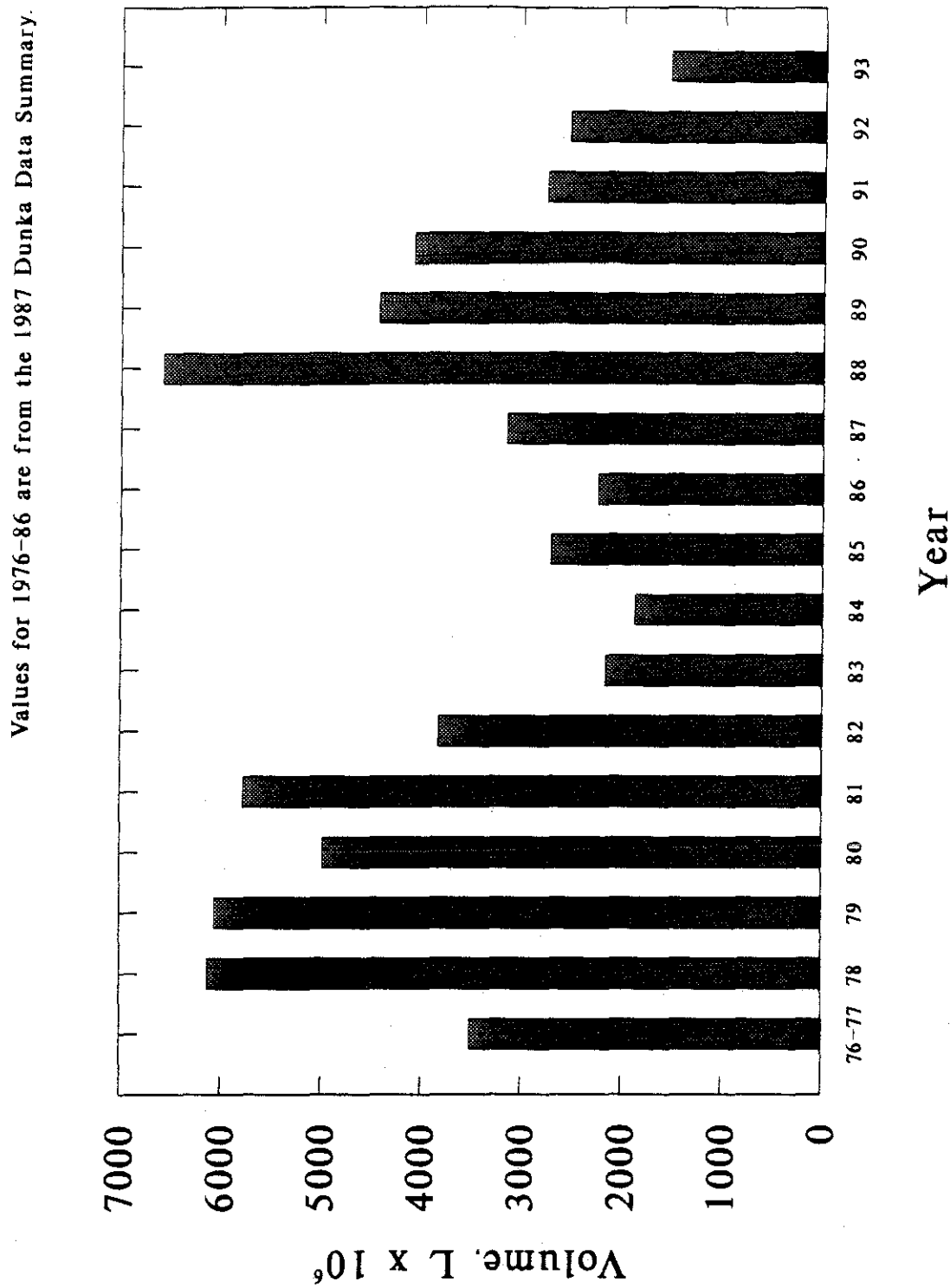


Figure 2. Annual volumes at EM1, 1976-1993.

Volume values for 1976 through 1986 are from the 1987 Dunka Data Summary. Seep X 1990 represents July – December only.

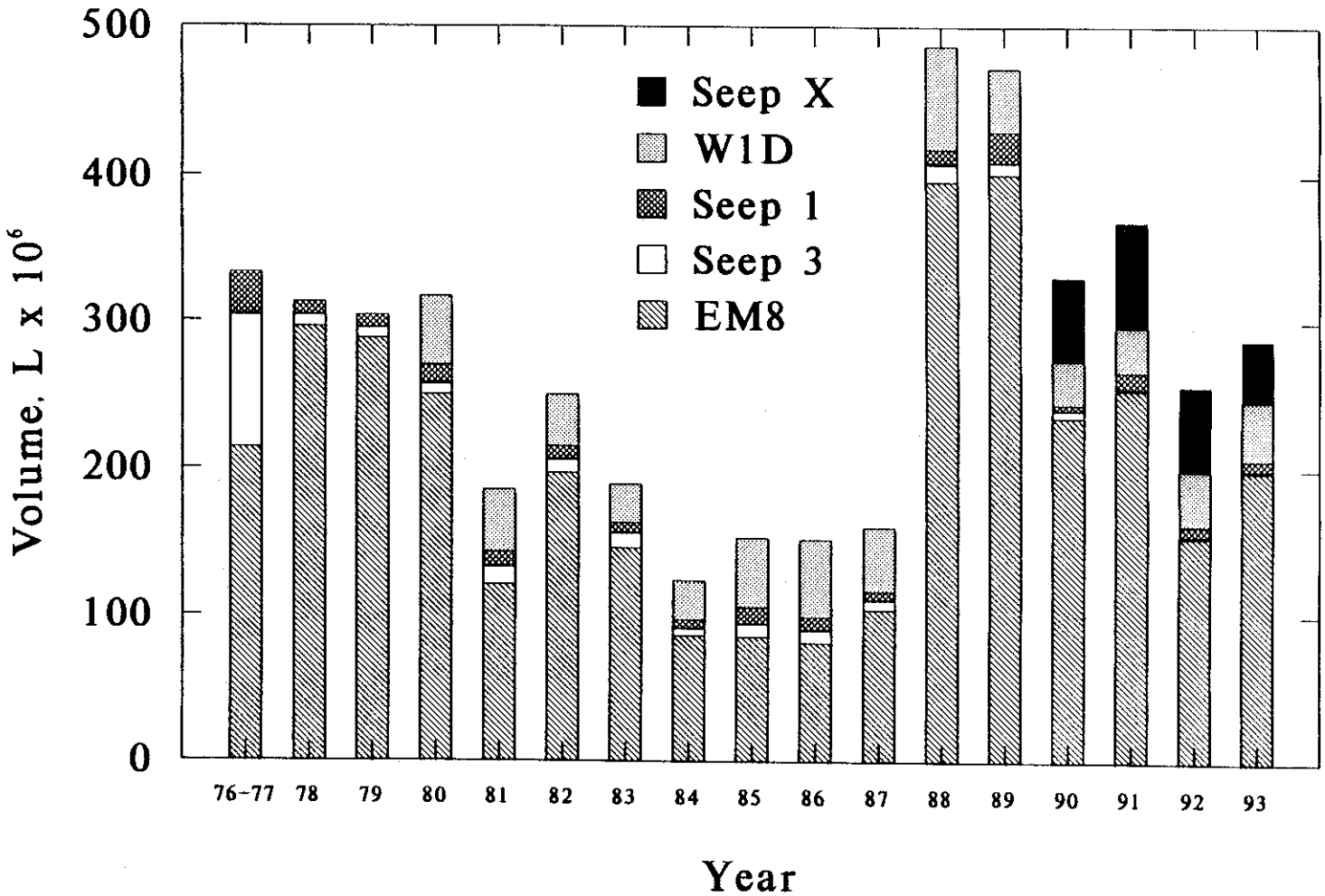


Figure 3. Annual volumes in the EM1 watershed, 1976-1993.

The annual yield coefficients represent the ratio of water output to precipitation input for the watersheds. The average annual yield coefficients at EM8 (1976-1993), Seep 1 (1986-1993), and W1D (1986-1993) were all around 0.3, and the value for Seep X (1990-1993) was 0.64 (table 6). These values are fairly consistent with the range of 0.29 to 0.52 reported for test piles at the MNDNR Babbitt research site (1978-1989 data). The average annual yield coefficients for Seep 3 (1978-1986) ranged from 0.02 to 0.05, well below the commonly observed range. Yields for the next four years were considerably higher, and the overall average yield coefficient was 0.20. The anomalously low values may have been due to errors in estimating the watershed area, unmeasured seepage or subsurface flow from the watershed, or water storage followed by release over the period from 1987 through 1990.

The yield coefficients for EM1 ranged from 0.17 to 0.84, with an average of 0.44. The flow at EM1 was affected by pumping and ditching activities in the watershed. Prior to June 23, 1982 mine dewatering was pumped into Unnamed Creek, contributing roughly 4000 million liters per year. Subsequently, the dewatering flow was pumped to the Dunka River (table 7). The area contributing flow to the environmental sump was effectively removed from the watershed in 1991 when the ditch from the sump to the pit was completed. Short-term impacts may have resulted from other activities, particularly blasting the mud wave at the toe of the 8011 stockpile and ditching to drain water away from stockpiles. (See STS 1993 for details.)

4. Water quality.

This report focuses on the water quality parameters of pH and nickel. Information on copper, cobalt, and zinc is also being compiled. EM8, W1D, and Seep X produced typical drainage pH values in the range of 6.5 to 7.5, with Seep X on the lower end of this range (figure 4). Both Seep 1 and Seep 3 produced some acidic pH values. At Seep 1 pH decreased from 1976 to 1979, when a minimum value of 4.5 was reported. The pH rebounded to the neutral range in 1980, then decreased steadily over time; the median pH in 1991 was 4.9. In 1992 sample collection was moved from the seep itself to the outflow from the limestone pretreatment bed. This modification resulted in an apparent rise in pH. At Seep 3 the pH hovered around 7.0 from 1976 through 1988. In 1989 it dropped to a minimum of 4.8, then returned to the neutral range in 1990.

Nickel concentrations at EM8 and Seep X were generally below 5 mg/L, while those at W1D typically ranged from 4 to 8 mg/L. Nickel concentrations at Seep 3 were in the range of 15 to 25 mg/L from 1975 to 1978, then decreased and remained below 10 mg/L through 1986. Concentrations subsequently increased, reaching a median value of 47 mg/L in 1988. Several values near 80 mg/L were observed from 1988 to 1990. Subsequently nickel concentrations decreased, concurrent with an increase in pH and a decrease in flow (figures 4, 3, respectively). Seep 1 nickel concentrations typically ranged from 10 to 20 mg/L, although annual maxima around 30 mg/L were common (figure 5).

The temporal variations of copper, cobalt, zinc, sulfate, calcium, and magnesium are presented in figures 6 through 11.

Table 6. Annual and average runoff coefficients at the Dunka site, 1976-1993.

Year	Precipitation cm	Site						
		EM8	Seep 3	Seep X	Seep 1	W1D	EM1[1]	W4
1976-77[2]	97	0.24	0.44	ND	0.24	NC	NA	NA
1978[2]	70	0.39	0.03	ND	0.39	NC	0.24	NA
1979[2]	63	0.42	0.02	ND	0.42	NC	0.31	NA
1980[2]	68	0.34	0.04	ND	0.34	0.34	0.17	NA
1981[2]	70	0.31	0.05	ND	0.31	0.31	0.30	0.39
1982[2]	87	0.21	0.03	ND	0.21	0.21	0.51	0.52
1983[2]	78	0.17	0.04	ND	0.17	0.17	0.48	0.40
1984[2]	56	0.25	0.03	ND	0.25	0.25	0.33	0.42
1985[2]	84	0.29	0.03	ND	0.29	0.29	0.43	0.40
1986[2]	68	0.36	0.03	ND	0.29	0.40	0.51	0.43
1987	56	0.12	0.66	ND	0.25	0.44	0.54	0.28
1988	79	0.32	0.82	ND	0.29	0.49	0.84	0.35
1989	66	0.38	0.64	ND	0.73	0.36	0.66	0.33
1990	71	0.21	0.34	1.03[3]	0.14	0.24	0.60	0.29
1991	74	0.27	0.05	0.62	0.35	0.24	0.42	0.29
1992	65	0.27	0.04	0.56	0.26	0.33	0.46	0.26
1993	77	0.34	0.08	0.34	0.20	0.30	0.24	0.20
Average	71	0.29	0.20 (0.04 excluding '76-'77 and '87-'90 values)	0.64 (0.51 excluding '90 value)	0.30 (0.25 excluding '76-'85 and '89 values)	0.31 (0.35 excluding '76-'85 values)	0.44	0.35

ND: not discovered until 1990. NA: not applicable.
NC: Stockpile 8018, which feeds W1D, wasn't discovered until 1979.
[1] EM1 runoff coefficient calculations excluded volume from mine dewatering pumping.
[2] Data for 1976 to 1986 are from the 1987 Dunka Data Summary.
[3] Calculated over the period of flow measurement, July - December, 1990.

Table 7. Pumping (in 10^6 L) to Unnamed Creek, 1977 - 1992. Includes pumping from EM8-4 and EM8-6 until 6/23/82, after which pumping from EM8-4 went to the Dunka River. Pumping data for 1977 to 1985 were taken from Erie Site Flow Estimations (ESFE). Pumping data for 1986 to 1992 were taken from DNR Division of Waters annual water use reports (permit no. 735186). Data for 1993 were unavailable, but no pumping is believed to have occured.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
1977	153	131	176	226	223	344	363	249	411	70	66	39	2451
1978	257	204	240	324	21	204	838	840	432	406	366	340	4472
1979	169	262	482	358	412	447	316	287	419	347	338	414	4251
1980	260	214	302	311	149	333	356	351	479	558	279	253	3845
1981	217	222	327	300	262	370	401	334	270	404	284	240	3631
1982	194	185	284	362	429	284	73	28	71	108	263	16	2297
1983	19	0	0	66	58	0	53	39	69	45	8	0	357
1984	0	0	0	142	108	11	47	29	14	2	7	27	387
1985	0	0	4	109	95	114	83	95	76	66	7	0	649
1986	0	0	0	95	69	20	46	37	62	51	11	5	395
1987	2	6	18	12	54	78	62	70	40	16	15	12	386
1988	8	2	24	69	37	6	9	32	106	62	87	47	490
1989	32	27	34	94	93	60	37	5	22	11	8	3	427
1990	2	2	36	11	86	6	0	0	0	0	0	0	142
1991	0	0	0	0	0	0	0	0	0	0	0	0	0
1992	0	0	0	0	0	0	0	0	0	0	0	0	0
1993	0	0	0	0	0	0	0	0	0	0	0	0	0

Note: Pumping volumes for 1986 were reported in ESFE as follows:

Jan:	0	Apr:	11	Jul:	48	Oct:	47		
Feb:	0	May:	69	Aug:	38	Nov:	18		
Mar:	0	Jun:	19	Sep:	57	Dec:	6	TOTAL:	313

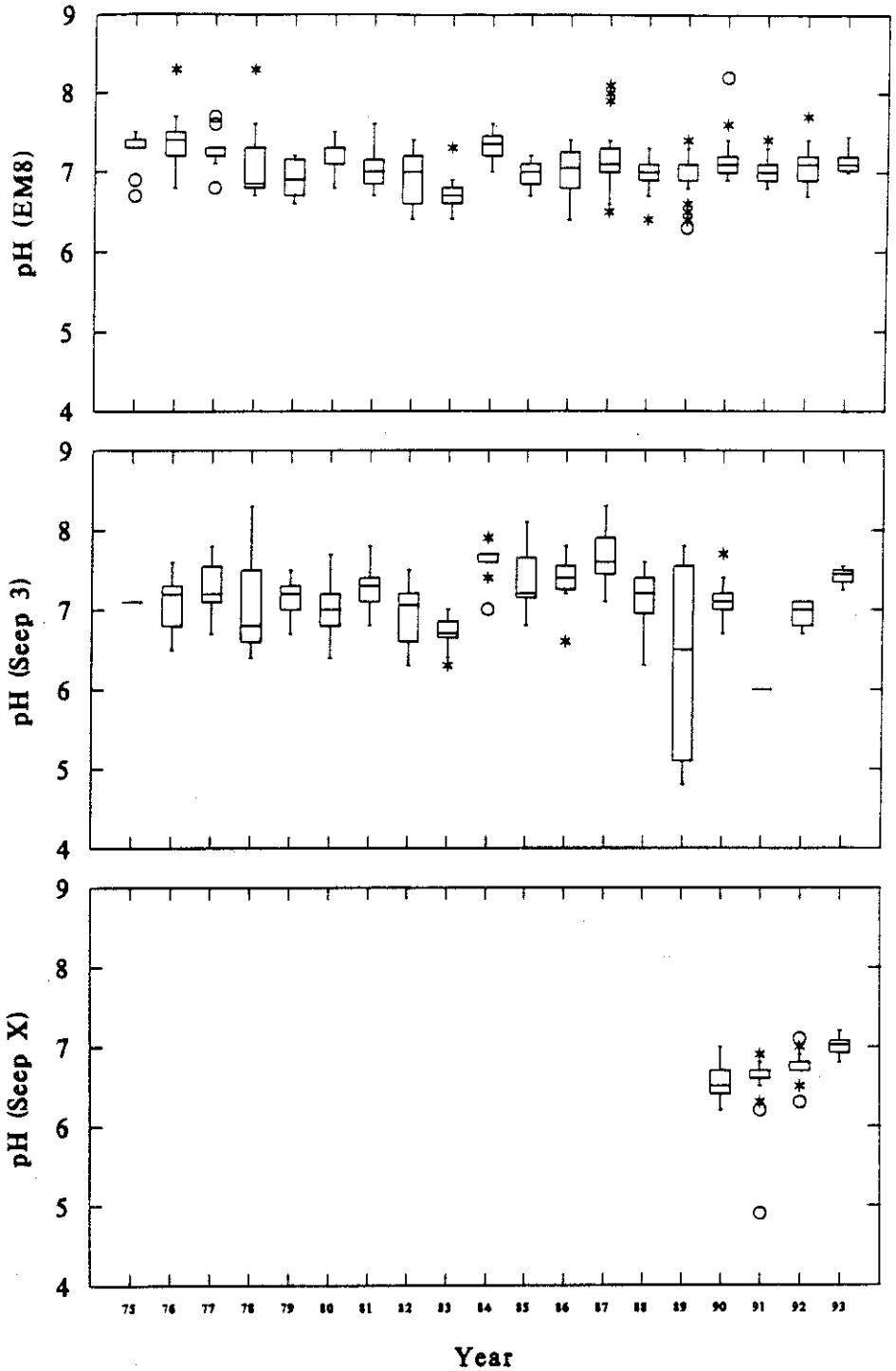


Figure 4a. pH box plots for EM8, Seep 3, and Seep X.

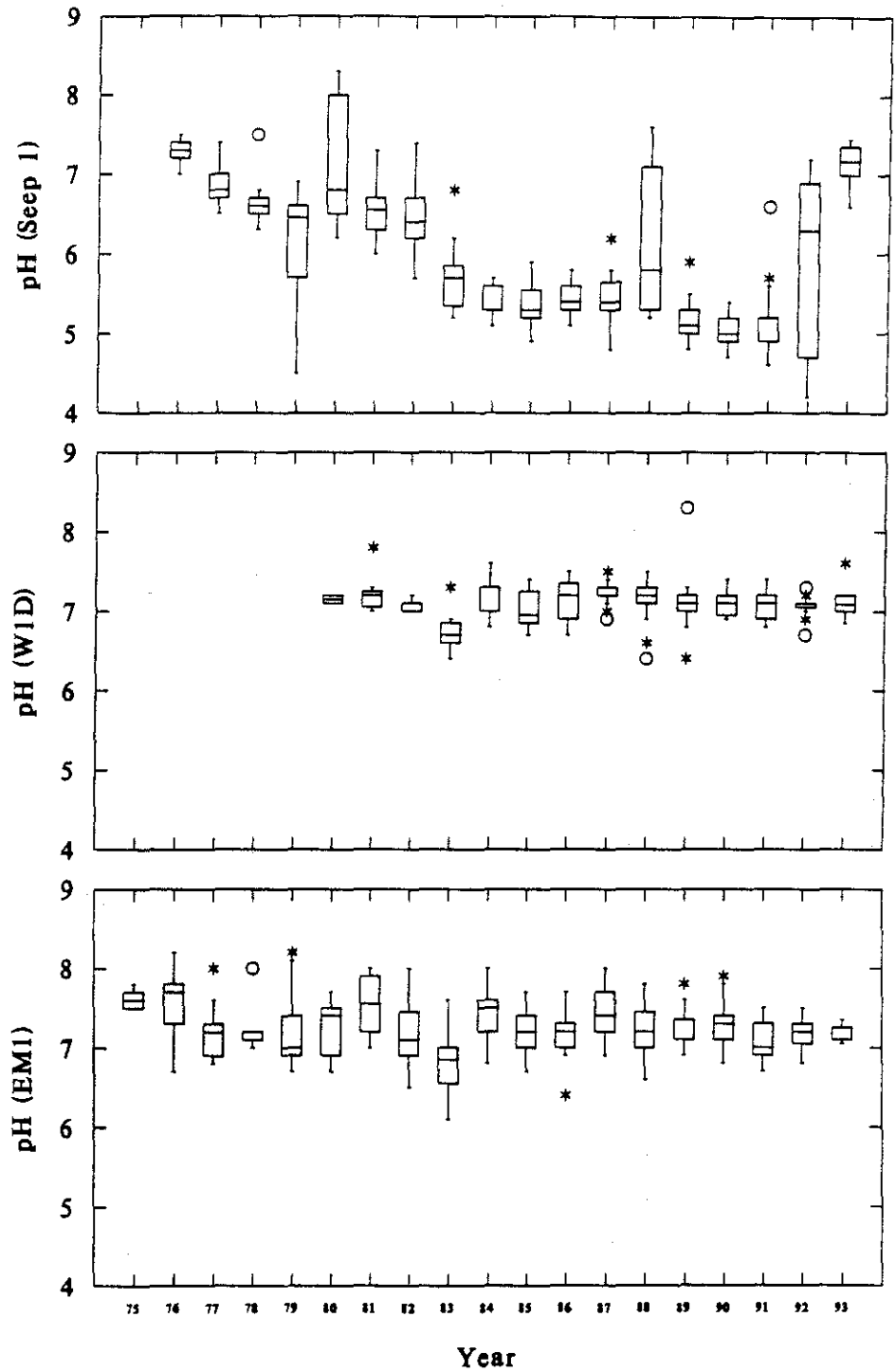


Figure 4b. pH box plots for Seep 1, W1D and EM1.

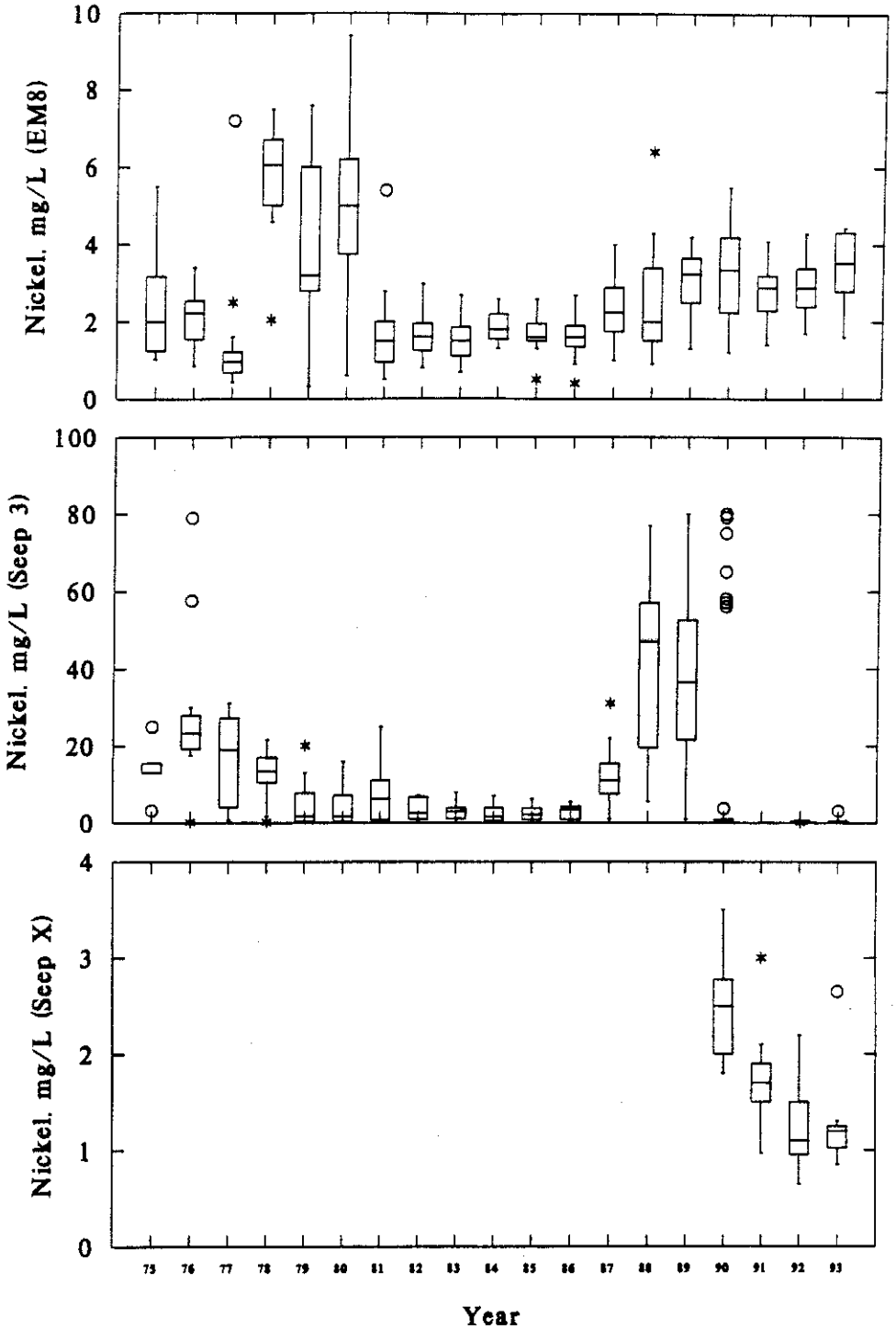


Figure 5a. Nickel box plots for EM8, Seep 3, and Seep X.

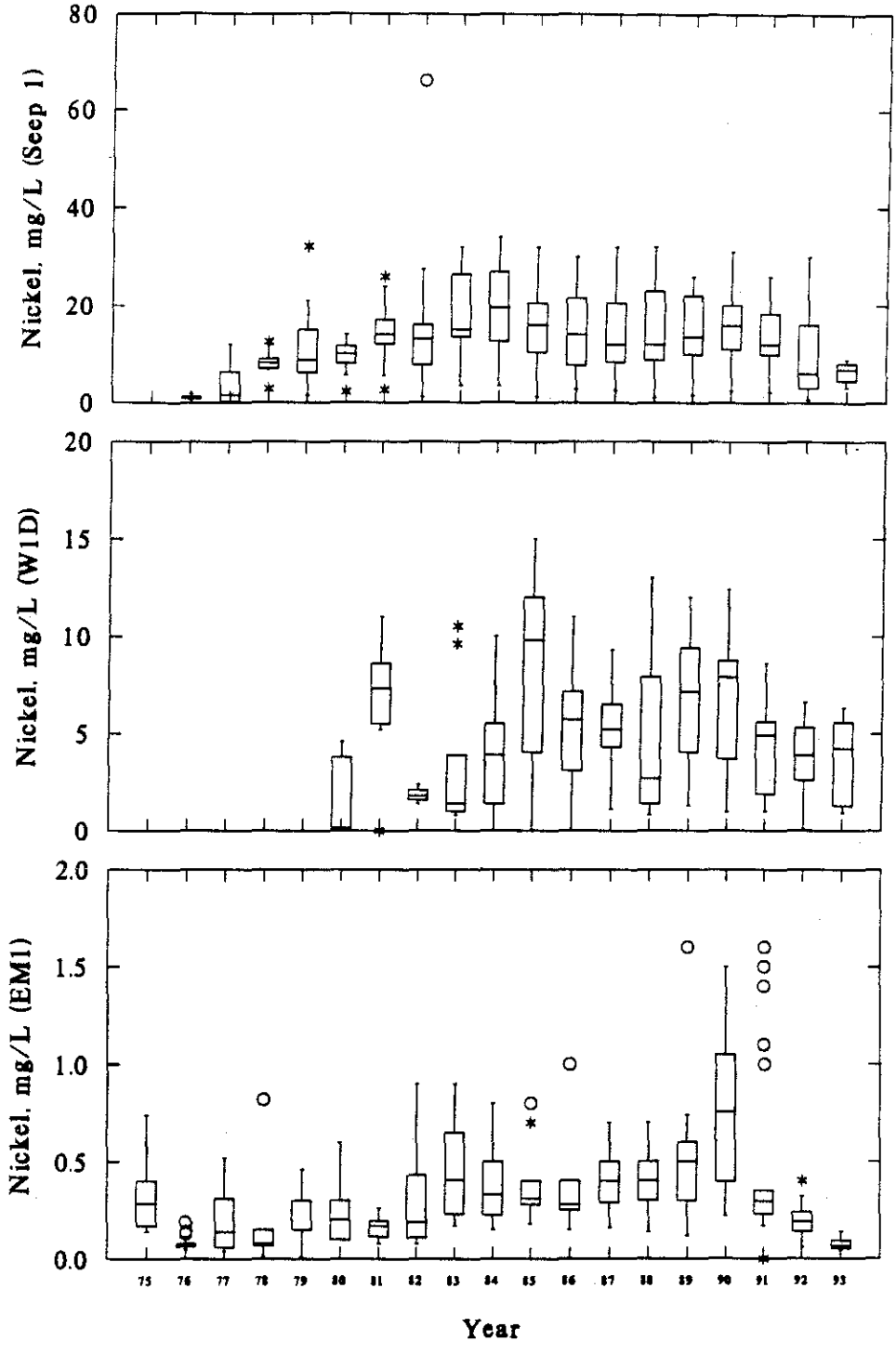


Figure 5b. Nickel box plots for Seep 1, W1D and EM1.

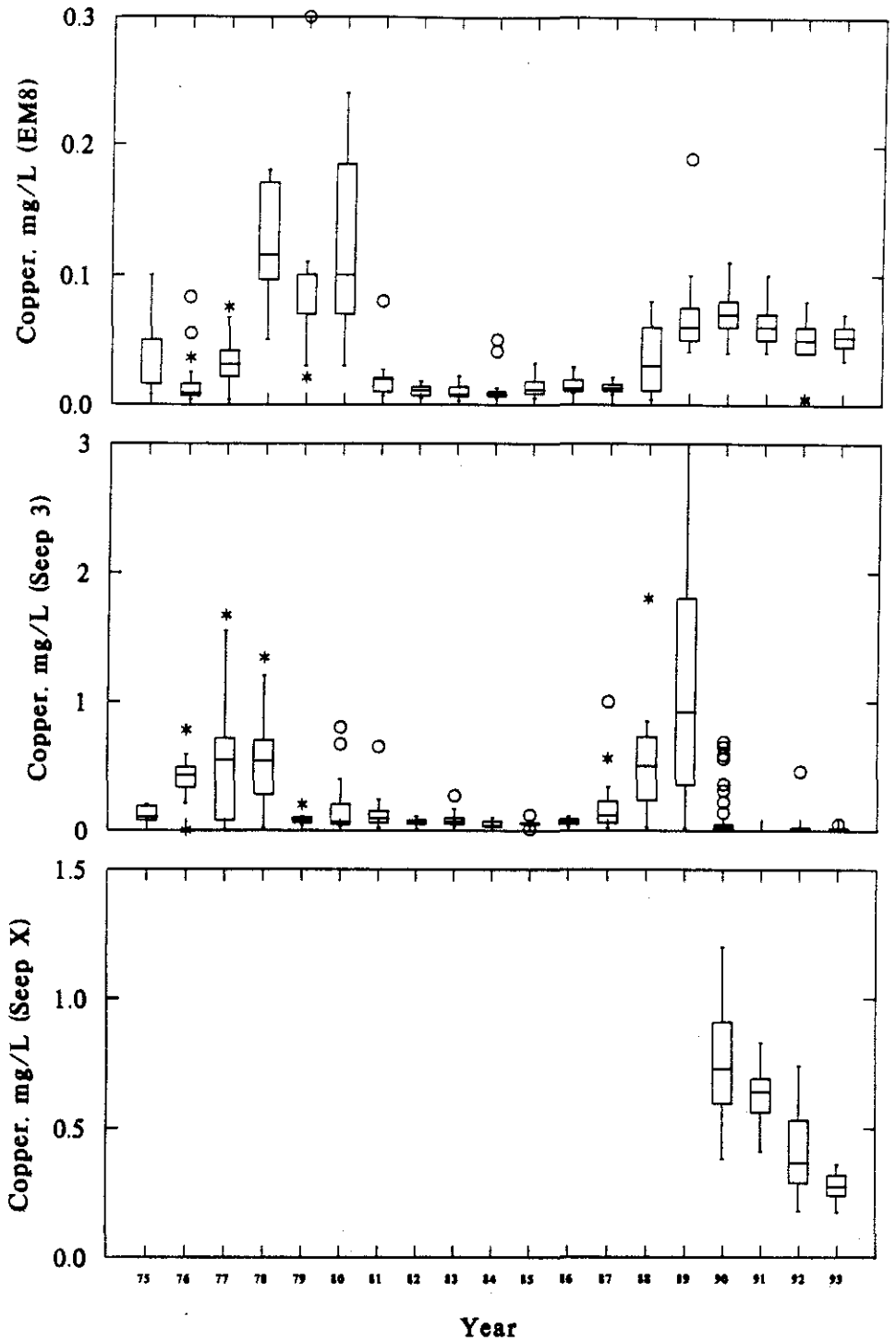


Figure 6a. Copper box plots for EM8, Seep 3, and Seep X.

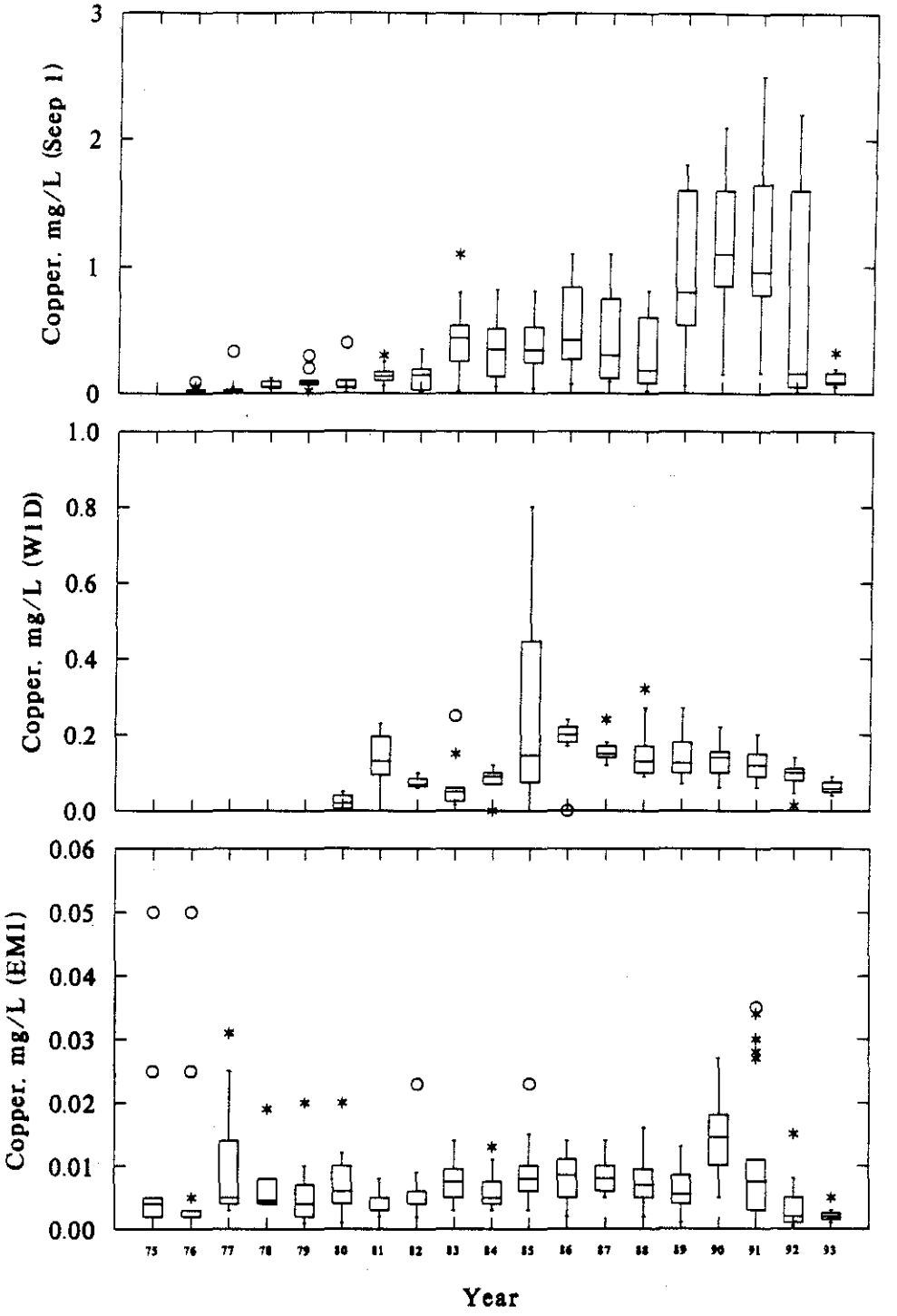


Figure 6b. Copper box plots for Seep 1, W1D and EM1.

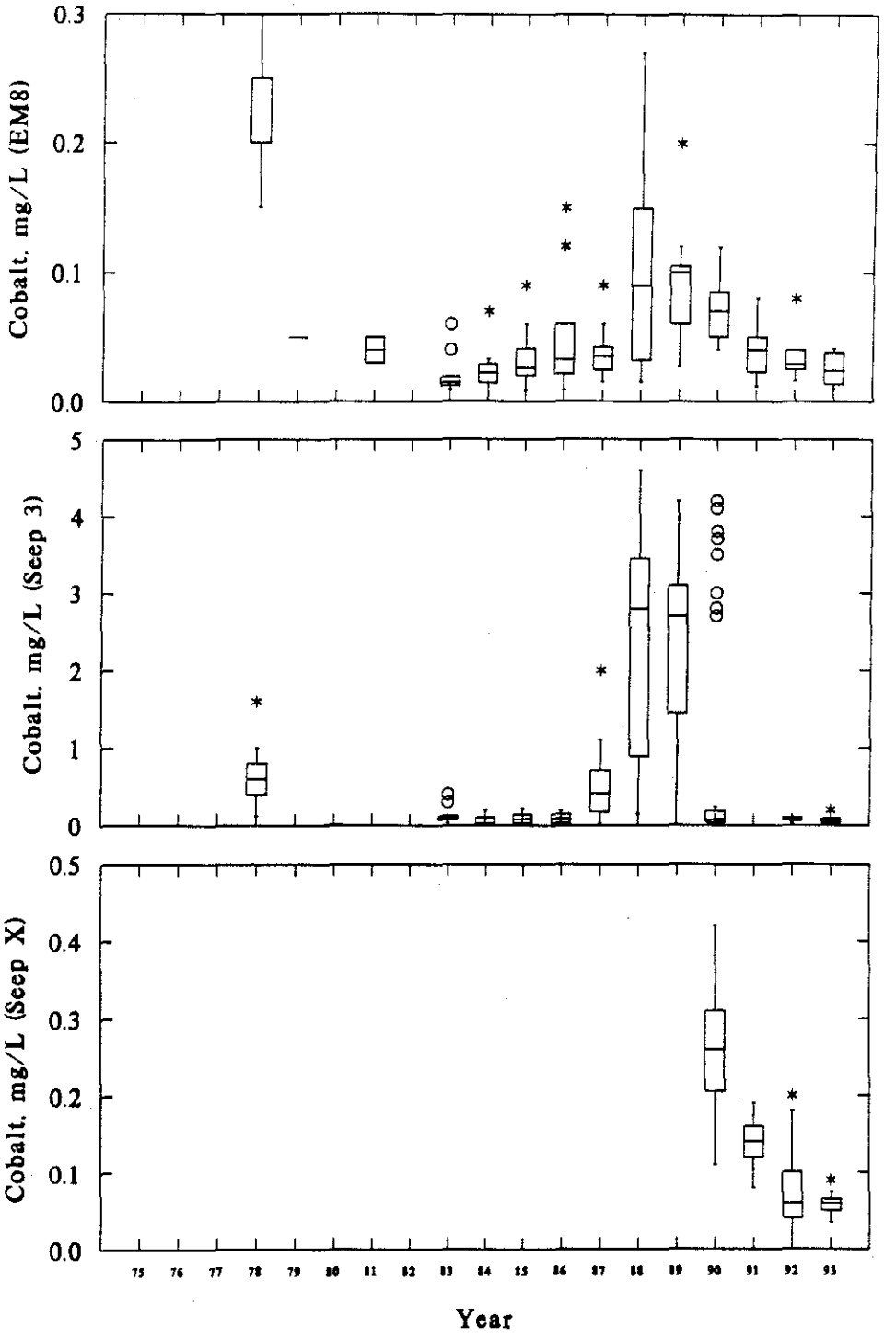


Figure 7a. Cobalt box plots for EM8, Seep 3, and Seep X.

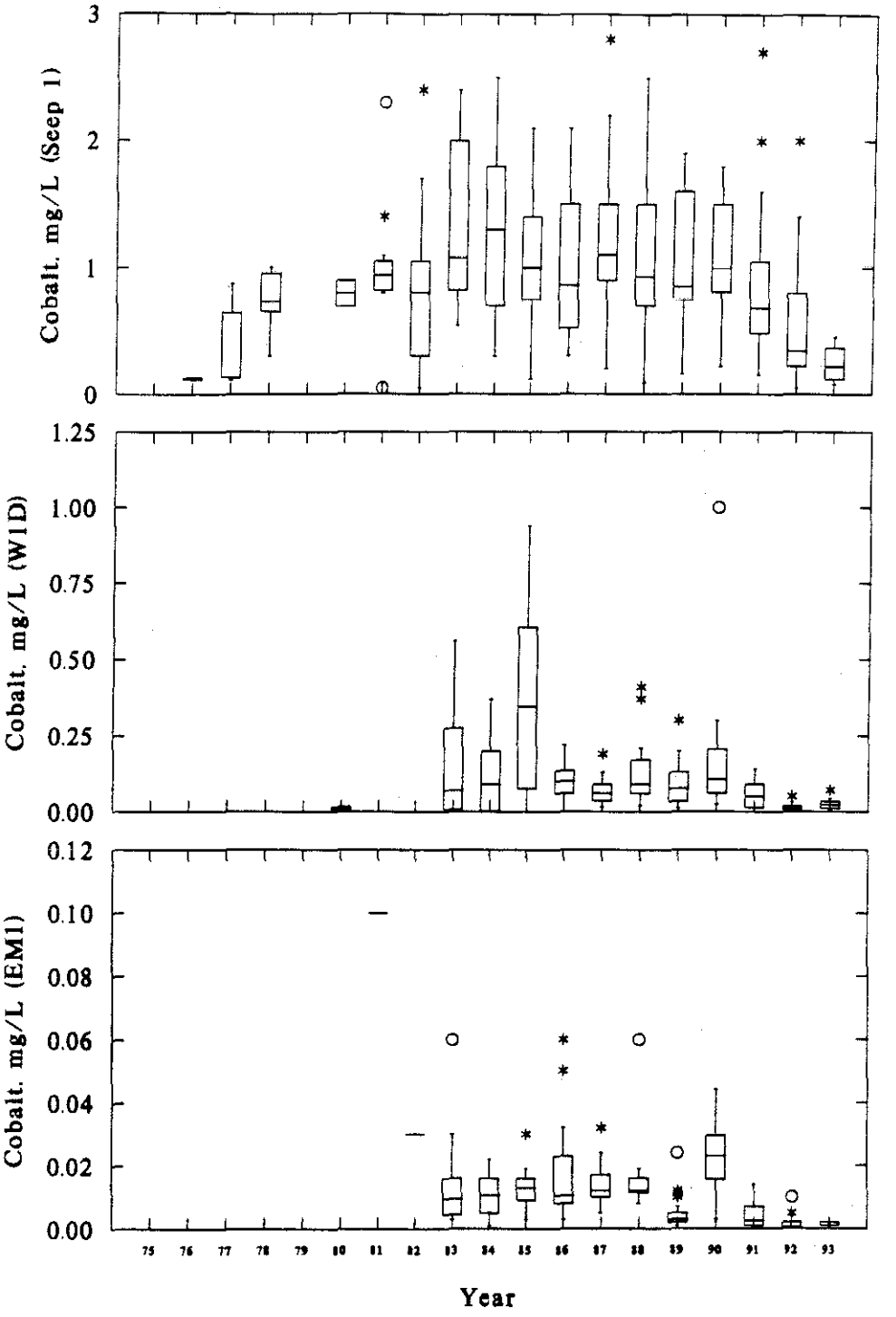


Figure 7b. Cobalt box plots for Seep 1, W1D and EM1.

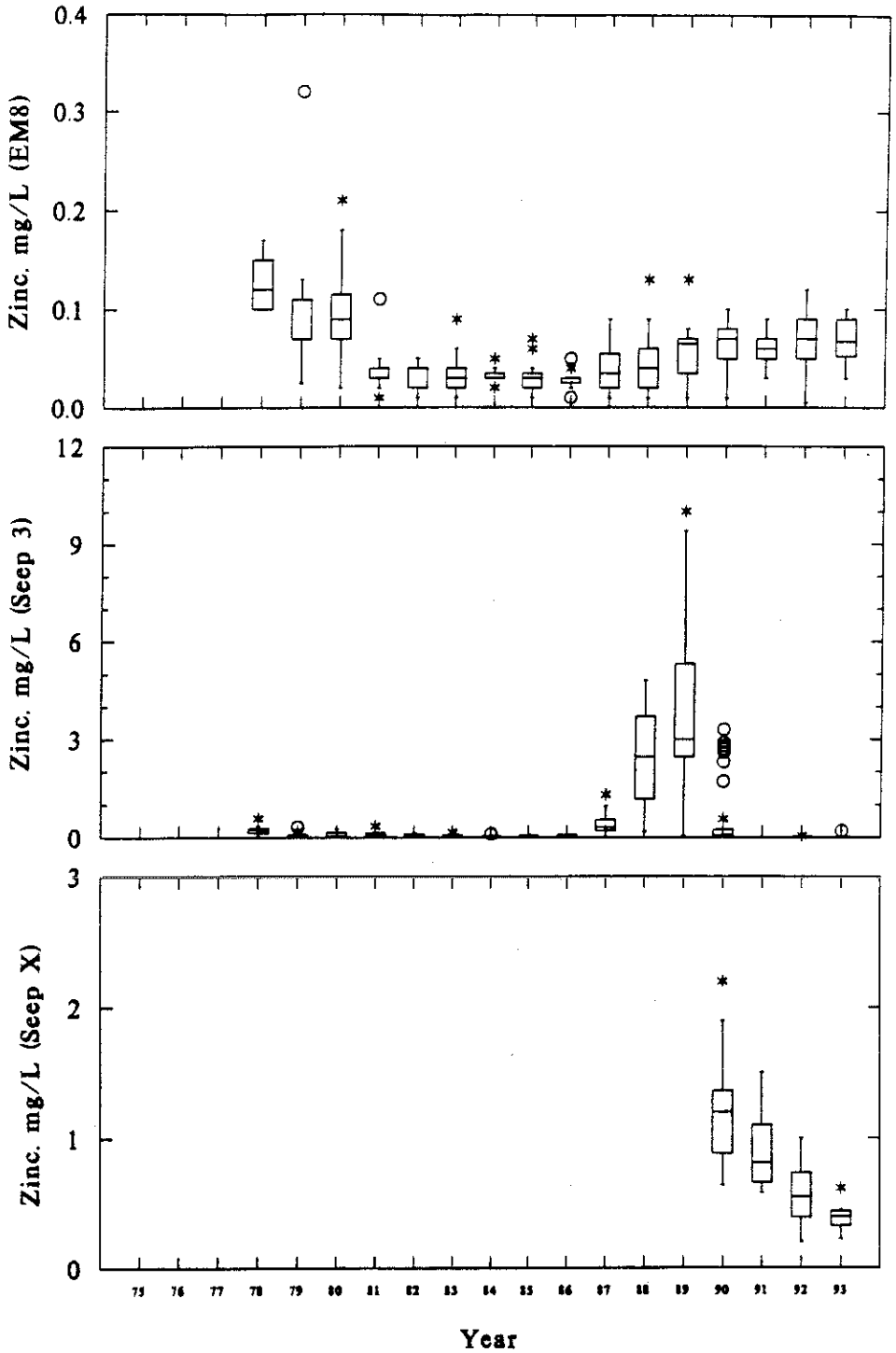


Figure 8a. Zinc box plots for EM8, Seep 3, and Seep X.

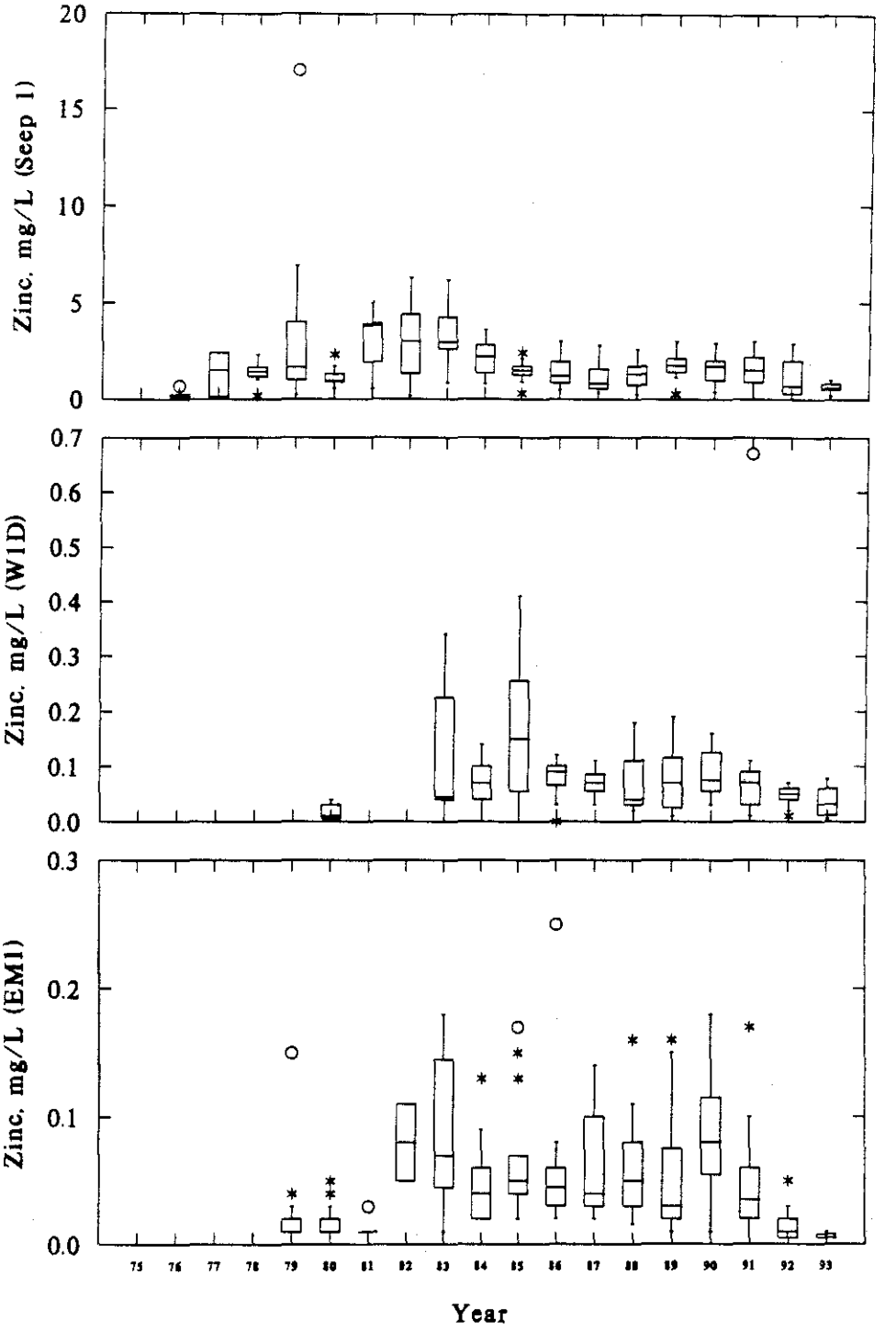


Figure 8b. Zinc box plots for Seep 1, W1D and EM1.

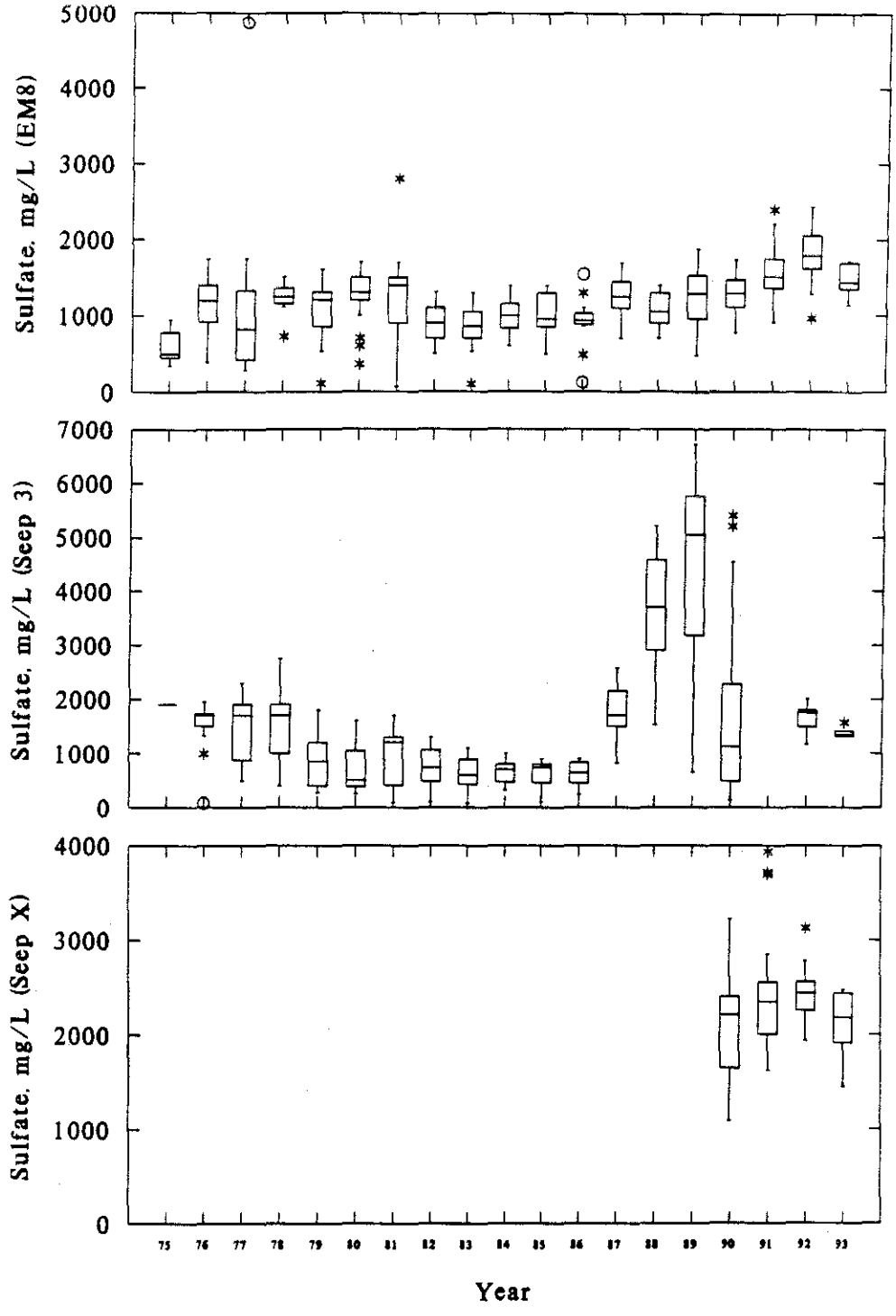


Figure 9a. Sulfate box plots for EM8, Seep 3, and Seep X.

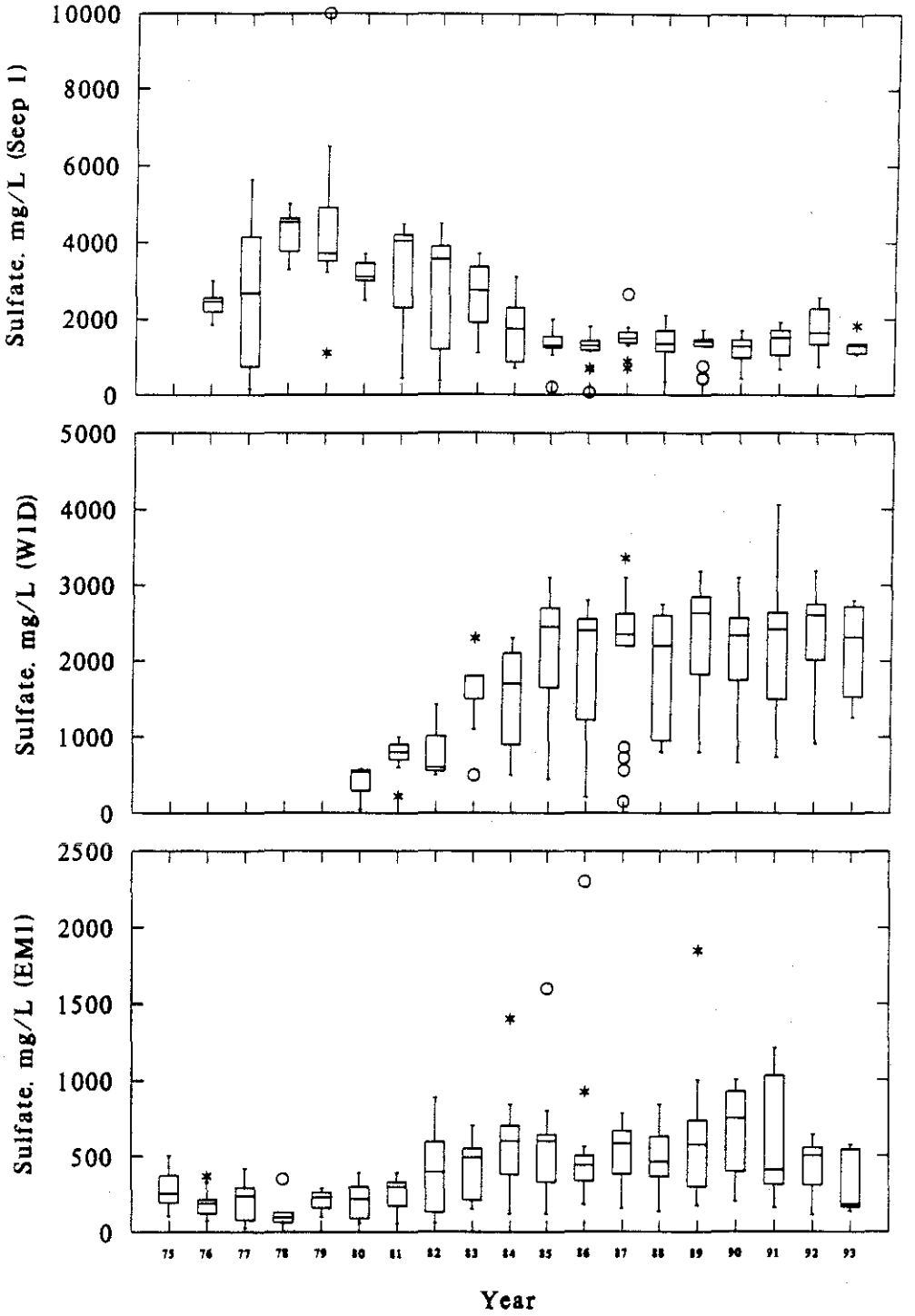


Figure 9b. Sulfate box plots for Seep 1, W1D and EM1.

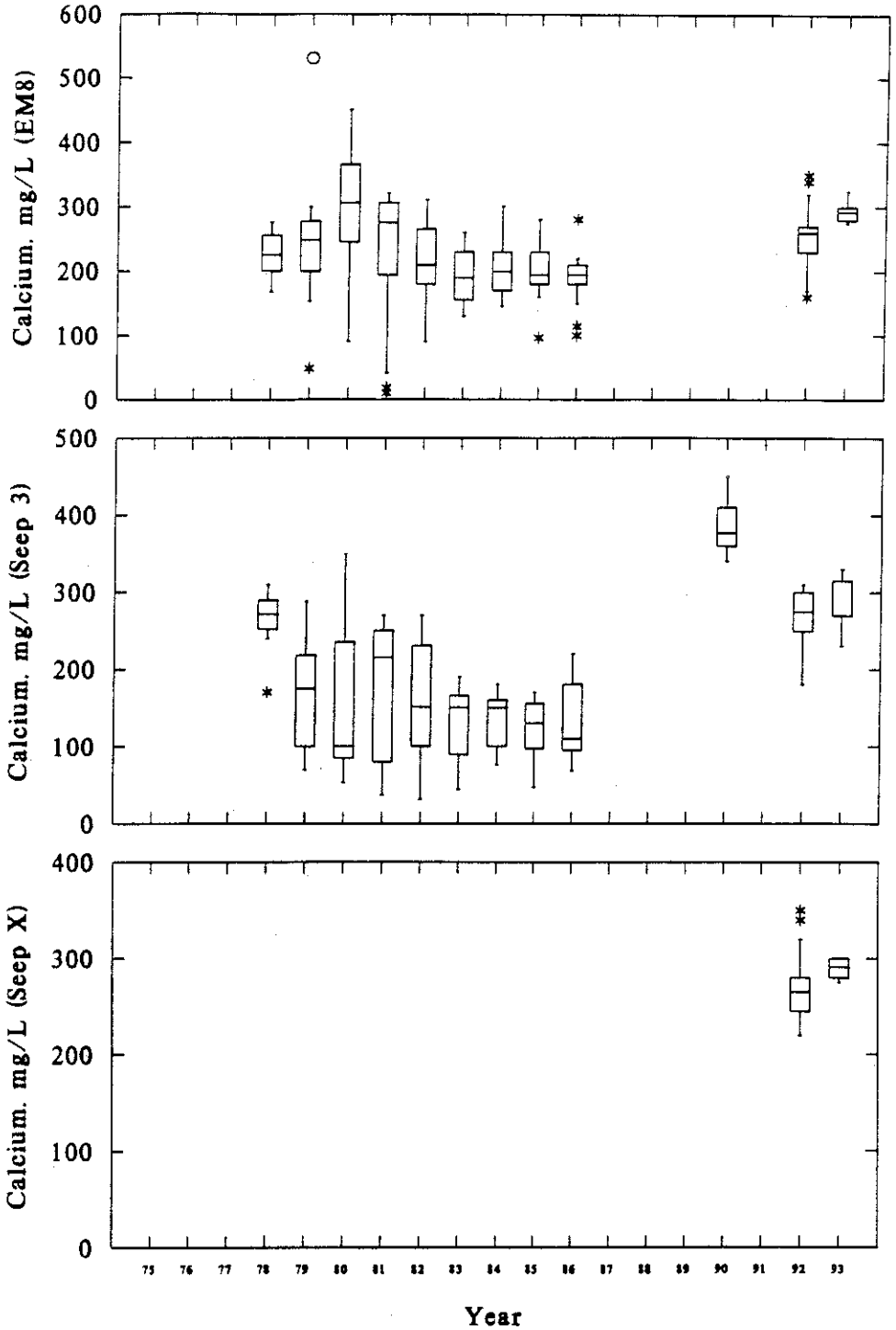


Figure 10a. Calcium box plots for EM8, Seep 3, and Seep X.

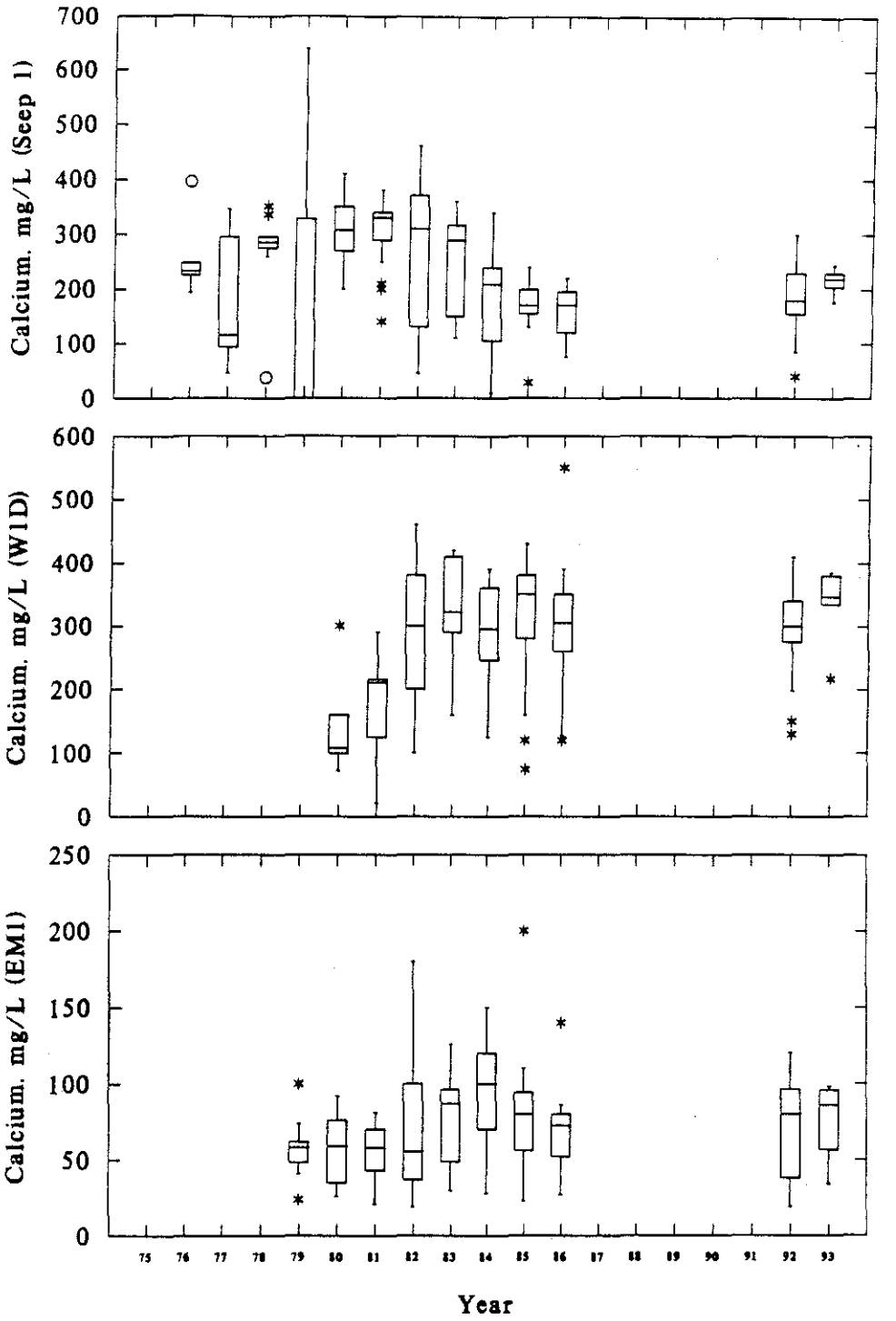


Figure 10b. Calcium box plots for Seep 1, W1D and EM1.

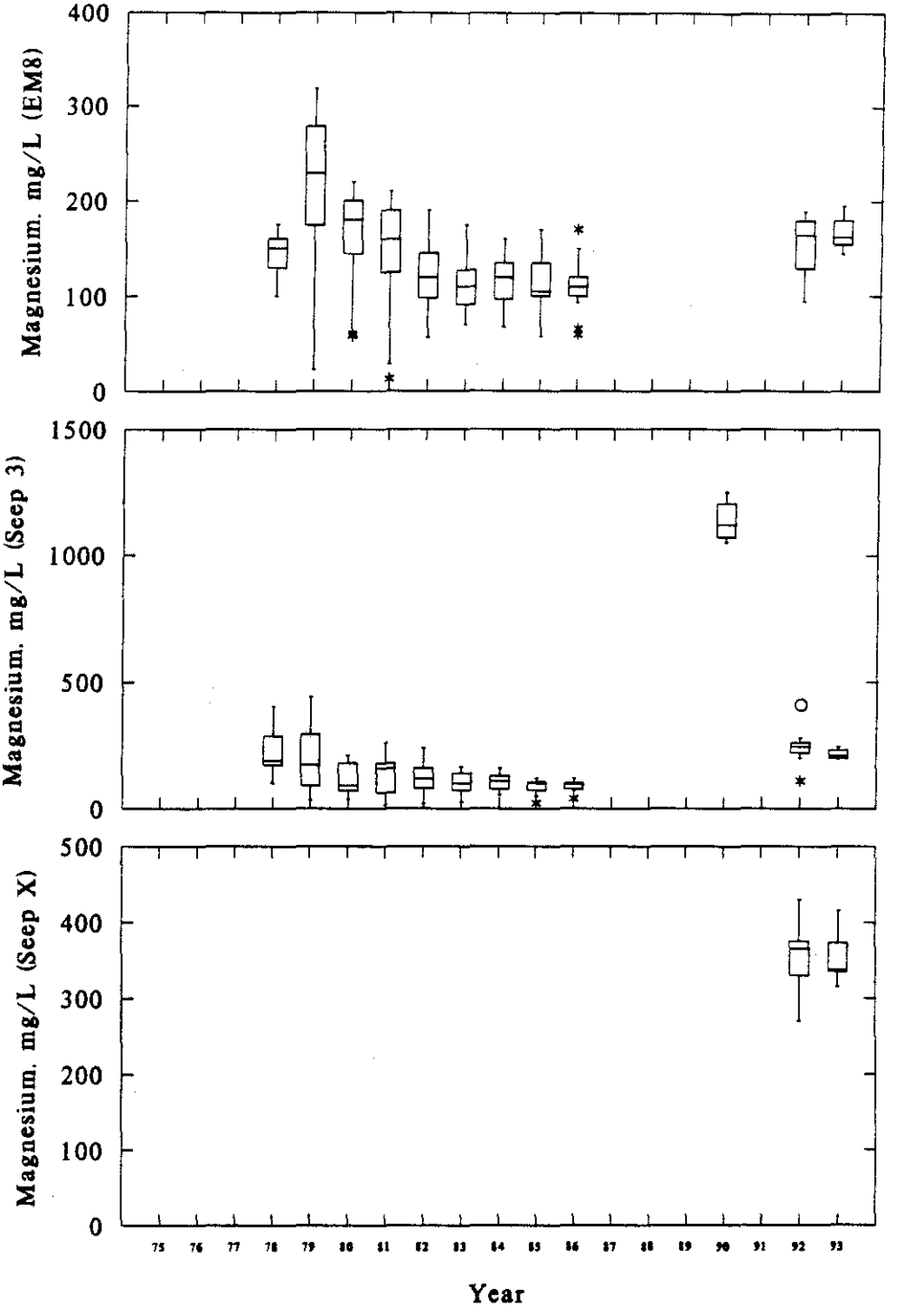


Figure 11a. Magnesium box plots for EM8, Seep 3, and Seep X.

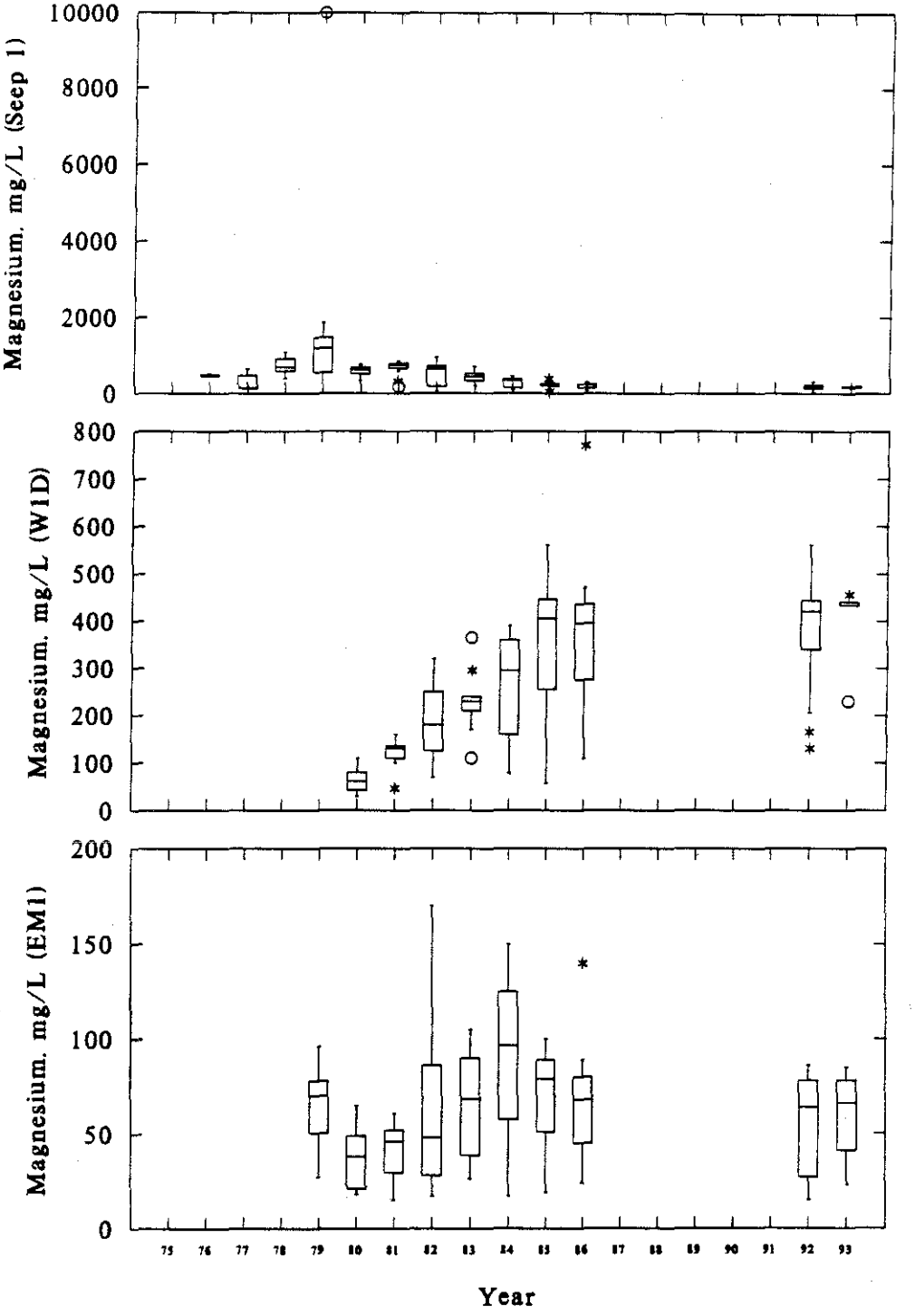


Figure 11b. Magnesium box plots for Seep 1, W1D and EM1.

5. Chemical mass release.

From 1978 to 1980 and from 1988 through 1991, EM8 was the major source of nickel in the Unnamed Creek watershed (figure 12). From 1988 through 1991 the nickel contributions from Seep 3, Seep X, Seep 1, and W1D were each about 20 to 30 percent of that from EM8 (figure 6, table 7). From 1978 to 1987 the annual nickel release at EM1 was in the approximate range of 1000 to 1500 kg. The release was considerably higher from 1988 to 1991, with a maximum release of 6000 kg in 1990 (figure 6). From 1987 through 1991 the release at EM1 was roughly two to four times that from the five monitored seeps in the watershed (figure 12, table 8). This suggests a substantial error in mass release calculations or an unmonitored source of nickel in the Unnamed Creek watershed.

Annual mass release data for copper, cobalt, zinc, sulfate, calcium, and magnesium are presented in tables 9 through 14. Annual mass release rates are presented in tables 15 through 21.

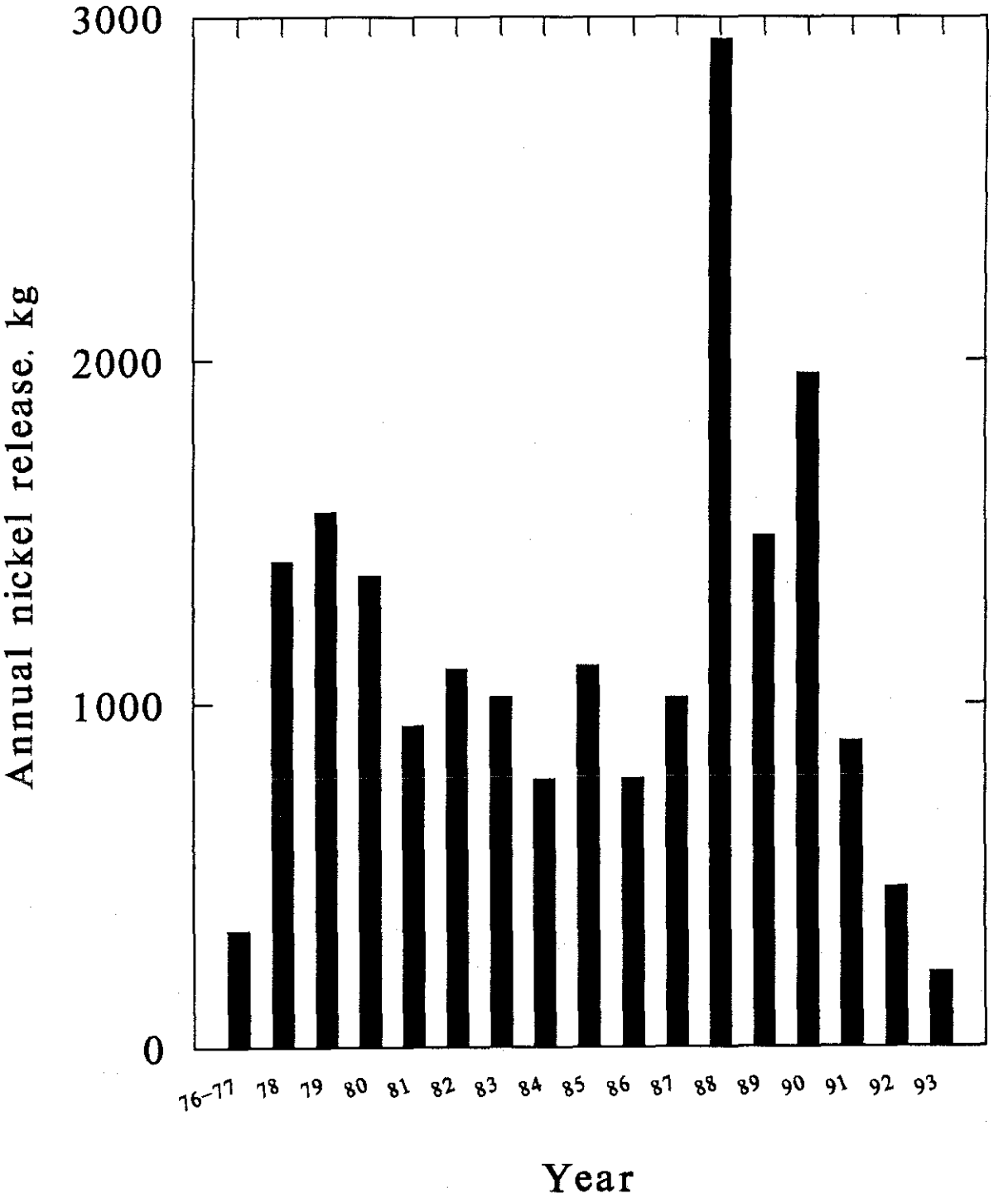


Figure 12. Annual nickel release in the EM1 watershed, 1976-1993.

Site	Annual Nickel Release, kg																
	76/77[1]	78[1]	79[1]	80[1]	81[1]	82[1]	83[1]	84[1]	85[1]	86[1]	87	88	89	90	91	92	93
EM8	150	1561	1149	1716	182	323	227	141	139	125	232	1229	1226	739	582	451	713
Seep 3	1810	95	28	66	80	42	34	15	33	26	95	559	320	166	NA	0.18	3.3
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	149	126	70	51
Seep 1	110	78	106	125	146	134	127	114	183	140	112	178	345	53	150	79	47
W1D	NA	NA	NA	NA	340	66	103	90	296	268	268	589	311	205	135	159	181
EM1	340	1417	1561	1375	936	1103	1021	781	1112	784	1020	2936	1492	1963	892	466	218
W4	NA	NA	NA	NA	3.6[2]	8.4[2]	3.6[2]	7.3[2]	11[2]	13[2]	12	45	26	19	36	6.9	2.9

[1] Values are from the 1987 Dunka Data Summary unless otherwise noted.
[2] Value is the product of the total annual volume and the median annual concentration reported in the 1987 Dunka Data Summary.

Table 8. Annual nickel release in the EM1 watershed, 1976-1993.

Site	Annual Copper Release, kg																
	76/77[1]	78[1]	79[1]	80[1]	81[2]	82[2]	83[2]	84[2]	85[2]	86[2]	87	88	89	90	91	92	93
EM8	4	30	36	32	2.4	2.2	1.2	0.69	0.97	1.1	1.4	21	25	16	16	7.5	11
Seep 3	72	4.1	0.6	1.6	1.0	0.63	0.70	0.20	0.54	0.63	1.8	12	9.6	1.1	NA	0.014	0.055
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	36	45	22	11
Seep 1	0.7	0.6	1.0	0.7	1.4	1.3	3.0	2.1	4.1	3.8	3.5	5.0	22	4.3	12	6.0	1.1
W1D	NA	NA	NA	NA	NA	NA	1.3	2.4	6.8	11	6.8	13	5.3	4.0	3.1	3.2	2.3
EM1	9	40	37	38	29	15	17	9.4	22	20	25	54	29	61	23	9.2	5.9
W4	NA	NA	NA	NA	1.9	3.2	2.2	1.4	2.7	2.6	1.5	2.4	3.1	3.6	2.1	0.89	1.0

[1] Values are from the 1981 Environmental Leaching report.
[2] Each value is the product of the total annual volume and the median annual concentration reported in the 1987 Dunka Data Summary.

Table 9. Annual copper release in the EM1 watershed, 1976-1993.

Site	Annual Cobalt Release, kg																
	76/77[1]	78[1]	79[1]	80[1]	81[1]	82[1]	83[1]	84[1]	85[1]	86[1]	87	88	89	90	91	92	93
EM8	NA	59	14	NA	7.3	6.9	6.6	2.0	2.2	2.7	3.6	51	39	18	8.8	5.0	5.8
Seep 3	NA	4.8	NA	0.035	2.2	0.86	0.85	0.12	0.72	0.77	4.3	33	20	8.5	NA	0.027	0.21
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	13	9.8	4.3	2.3
Seep 1	NA	6.5	NA	10	9.4	7.2	7.5	7.8	12	7.7	8.9	13	24	4.1	9.4	4.4	2.2
W1D	NA	NA	NA	NA	NA	NA	1.8	2.4	16	5.3	3.9	16	4.6	4.0	1.6	0.79	1.2
EM1	NA	153	NA	NA	202	115	22	19	35	22	41	97	14	94	8.5	5.2	3.1
W4	NA	NA	NA	NA	NA	6.0	2.5	2.5	3.0	3.7	2.1	5.6	0.77	3.9	0.45	0.19	0.15

[1] Each value is the product of the total annual volume and the median annual concentration reported in the 1987 Dunka Data Summary.

Table 10. Annual cobalt release in the EM1 watershed, 1976-1993.

Site	Annual Zinc Release, kg																
	76/77[1]	78[1]	79[1]	80[1]	81[1]	82[1]	83[1]	84[1]	85[1]	86[1]	87	88	89	90	91	92	93
EM8	NA	36	20	23	3.6	3.9	4.4	2.6	2.6	2.4	3.7	24	26	15	14	10	14
Seep 3	NA	1.8	0.21	0.21	0.84	0.59	0.55	0.10	0.45	0.50	3.4	41	32	6.5	NA	0.015	0.21
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	69	66	31	16
Seep 1	NA	13	15	12	38	27	21	13	18	11	8.7	19	37	5.7	14	9.0	4.6
W1D	NA	NA	NA	NA	NA	NA	1.3	1.9	7.1	4.8	3.3	7.5	3.7	3.0	2.7	1.9	1.5
EM1	NA	153	121	50	58	306	151	75	136	101	144	381	162	249	212	53	19
W4	NA	NA	NA	NA	1.2	8.0	2.8	2.0	1.5	2.3	1.2	2.3	2.0	1.5	2.0	0.84	0.59

[1] Each value is the product of the total annual volume and the median annual concentration reported in the 1987 Dunka Data Summary.

Table 11. Annual zinc release in the EM1 watershed, 1976-1993.

Site	Annual Sulfate Release, 10^3 kg																
	76/77[1]	78[1]	79[1]	80[1]	81[2]	82[2]	83[2]	84[2]	85[2]	86[2]	87	88	89	90	91	92	93
EM8	128	348	299	322	169	177	126	86	81	75	126	439	471	284	346	271	283
Seep 3	103	14	4.3	7.2	14	6.6	6.0	3.5	6.7	5.8	13	50	37	13	NA	0.72	1.2
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	107	172	143	89
Seep 1	72	41	34	41	41	32	19	10	16	12	9.7	15	27	4.5	14	12	7.9
W1D	NA	NA	NA	25[2]	34	21	39	46	115	104	106	163	101	63	67	98	76
EM1	563	1348	1001	905	645	931	887	888	1236	854	1503	2776	1744	2331	1158	1100	422
W4	NA	NA	NA	NA	78	196	140	176	250	163	92	180	151	117	142	99	105

[1] Values are from the 1981 Environmental Leaching report unless otherwise noted.
[2] Each value is the product of the total annual volume and the median annual concentration reported in the 1987 Dunka Data Summary.

Table 12. Annual sulfate release in the EM1 watershed, 1976-1993.

Site	Annual Calcium Release, 10^3 kg																
	76/77[1]	78[1]	79[1]	80[1]	81[2]	82[2]	83[2]	84[2]	85[2]	86[2]	87	88	89	90	91	92	93
EM8	NA	58	NA	86	33	41	28	17	17	16	NA	NA	NA	NA	NA	39	58
Seep 3	NA	2.1	NA	1.6	2.5	1.4	1.5	0.75	1.2	1.0	NA	NA	NA	1.5	NA	0.11	0.21
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	15	12
Seep 1	NA	2.5	NA	4.0	3.3	2.8	2.0	1.3	2.0	1.5	NA	NA	NA	NA	NA	1.3	1.3
W1D	NA	NA	NA	5.1[2]	8.8	11	8.4	8.0	16	16	NA	NA	NA	NA	NA	11	13
EM1	NA	100[2]	112[2]	69[2]	125	130	157	148	165	141	NA	NA	NA	NA	NA	176	97
W4	NA	NA	NA	NA	16	34	29	36	41	30	NA	NA	NA	NA	NA	16	18

[1] Data are from the 1981 Environmental Leaching report unless otherwise noted.
[2] Each value is the product of the total annual volume and the median annual concentration reported in the 1987 Dunka Data Summary.

Table 13. Annual calcium release in the EM1 watershed, 1976-1993.

Site	Annual Magnesium Release, 10^3 kg																
	76/77[1]	78[1]	79[1]	80[1]	81[2]	82[2]	83[2]	84[2]	85[2]	86[2]	87	88	89	90	91	92	93
EM8	NA	38	NA	44	19	24	16	10	8.9	8.9	NA	NA	NA	NA	NA	24	33
Seep 3	NA	1.7	NA	1.2	1.7	1.1	1.0	0.55	0.90	0.87	NA	NA	NA	4.7	NA	0.085	0.12
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	20	6.9
Seep 1	NA	6.8	NA	7.7	7.6	5.9	3.1	2.1	2.8	2.2	NA	NA	NA	NA	NA	1.3	1.1
W1D	NA	NA	NA	2.9[2]	5.5	6.3	6.0	8.0	19	19	NA	NA	NA	NA	NA	15	15
EM1	NA	70[2]	135[2]	44[2]	99	113	124	143	163	132	NA	NA	NA	NA	NA	147	77
W4	NA	NA	NA	NA	11	25	22	29	35	28	NA	NA	NA	NA	NA	15	17

[1] Data are from the 1981 Environmental Leaching report unless otherwise noted.
[2] Each value is the product of the total annual volume and the median annual concentration reported in the 1987 Dunka Data Summary.

Table 14. Annual magnesium release in the EM1 watershed, 1976-1993.

Site	Annual Nickel Release Rate, mg/LT																
	76/77	78	79	80	81	82	83	84	85	86	87	88	89	90	91	92	93
EM8	12	109	79	111	12	20	14	8.6	8.2	7.3	14	71	71	42	32	25	39
Seep 3	646	33	8.8	18	22	12	9.4	4.2	8.7	6.5	24	140	76	40	NA	0.041	0.75
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	25	21	11	7.9
Seep 1	73	52	71	83	97	89	85	76	122	93	75	119	203	31	88	42	25
W1D	NA	NA	NA	NA	213	33	52	45	135	122	122	268	141	93	61	72	82
EM1	17	63	65	57	36	37	34	25	35	24	32	91	45	50	22	9.3	4.3
W4	NA	NA	NA	NA	0.97	1.6	0.61	1.2	1.8	2.1	1.9	7.1	4.1	3.0	5.7	1.1	0.44

Table 15. Annual nickel release rates, 1976-1993.

Site	Annual Copper Release Rate, mg/LT																
	76/77	78	79	80	81	82	83	84	85	86	87	88	89	90	91	92	93
EM8	0.32	2.1	2.5	2.1	0.16	0.14	0.075	0.042	0.057	0.064	0.082	1.2	1.5	0.92	0.89	0.41	0.60
Seep 3	26	1.4	0.19	0.44	0.28	0.18	0.19	0.056	0.14	0.16	0.45	3.0	2.3	0.26	NA	0.0032	0.013
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	6.0	7.4	3.4	1.7
Seep 1	0.47	0.40	0.67	0.47	0.93	0.87	2.0	1.4	2.7	2.5	2.3	3.3	13	2.5	7.1	3.2	0.58
W1D	NA	NA	NA	NA	NA	NA	0.65	1.2	3.1	5.0	3.1	5.9	2.4	1.8	1.4	1.5	1.1
EM1	0.45	1.8	1.6	1.6	1.1	0.50	0.56	0.30	0.69	0.62	0.78	1.7	0.88	1.6	0.58	0.18	0.12
W4	NA	NA	NA	NA	0.51	0.62	0.37	0.24	0.43	0.41	0.24	0.38	0.49	0.57	0.33	0.13	0.15

Table 16. Annual copper release rates, 1976-1993.

Site	Annual Cobalt Release Rate, mg/LT																
	76/77	78	79	80	81	82	83	84	85	86	87	88	89	90	91	92	93
EM8	NA	4.1	0.97	NA	0.47	0.44	0.41	0.12	0.13	0.16	0.21	3.0	2.3	1.0	0.49	0.28	0.32
Seep 3	NA	1.7	NA	0.01	0.61	0.24	0.24	0.033	0.19	0.19	1.1	8.3	4.8	2.0	NA	0.0061	0.048
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	2.2	1.6	0.66	0.35
Seep 1	NA	4.3	NA	6.7	6.3	4.8	5.0	5.2	8.0	5.1	5.9	8.7	14.1	2.4	5.5	2.3	1.2
W1D	NA	NA	NA	NA	NA	NA	0.90	1.2	7.3	2.4	1.8	7.3	2.1	1.8	0.73	0.36	0.55
EM1	NA	6.8	NA	NA	7.9	3.8	0.72	0.61	1.1	0.68	1.3	3.0	0.43	2.4	0.21	0.10	0.061
W4	NA	NA	NA	NA	NA	1.2	0.42	0.42	0.48	0.59	0.33	0.89	0.12	0.62	0.071	0.029	0.023

Table 17. Annual cobalt release rates, 1976-1993.

Site	Annual Zinc Release Rate, mg/LT																
	76/77	78	79	80	81	82	83	84	85	86	87	88	89	90	91	92	93
EM8	NA	2.5	1.4	1.5	0.23	0.25	0.27	0.16	0.15	0.14	0.22	1.4	1.5	0.86	0.78	0.55	0.77
Seep 3	NA	0.62	0.066	0.058	0.23	0.16	0.15	0.028	0.12	0.13	0.85	10.3	7.6	1.6	NA	0.0034	0.048
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	12	11	4.8	2.5
Seep 1	NA	8.7	10.0	8.0	25.3	18.0	14.0	8.7	12.0	7.3	5.8	12.7	21.8	3.4	8.2	4.7	2.4
W1D	NA	NA	NA	NA	NA	NA	0.65	0.95	3.2	2.2	1.5	3.4	1.7	1.4	1.2	0.86	0.68
EM1	NA	6.8	5.1	2.1	2.3	10.2	5.0	2.4	4.3	3.1	4.5	11.8	4.9	6.4	5.3	1.1	0.38
W4	NA	NA	NA	NA	0.32	1.5	0.47	0.34	0.24	0.37	0.19	0.37	0.32	0.24	0.32	0.13	0.089

Table 18. Annual zinc release rates, 1976-1993.

Site	Annual Sulfate Release Rate, g/LT																
	76/77	78	79	80	81	82	83	84	85	86	87	88	89	90	91	92	93
EM8	10	24	21	21	11	11	7.8	5.2	4.8	4.4	7.4	26	27	16	19	15	16
Seep 3	37	4.8	1.3	2.0	3.9	1.8	1.7	1.0	1.8	1.5	3.3	13	8.8	3.1	NA	0.16	0.27
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	18	28	22	14
Seep 1	48	27	23	27	27	21	13	6.7	11	8.0	6.5	10	16	2.6	8.2	6.3	4.2
W1D	NA	NA	NA	19	21	11	20	23	52	47	48	74	46	29	31	45	35
EM1	28	60	42	38	25	31	29	29	39	27	47	86	53	60	29	22	8.4
W4	NA	NA	NA	NA	21	38	24	30	40	26	15	29	24	19	23	15	16

Table 19. Annual sulfate release rates, 1976-1993.

Site	Annual Calcium Release Rate, g/LT																
	76/77	78	79	80	81	82	83	84	85	86	87	88	89	90	91	92	93
EM8	NA	4.1	NA	5.6	2.1	2.6	1.7	1.0	1.0	0.94	NA	NA	NA	NA	NA	2.2	3.2
Seep 3	NA	0.72	NA	0.44	0.69	0.39	0.42	0.21	0.32	0.25	NA	NA	NA	0.36	NA	0.025	0.048
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	2.3	1.9
Seep 1	NA	1.7	NA	2.7	2.2	1.9	1.3	0.87	1.3	1.0	NA	NA	NA	NA	NA	0.68	0.68
W1D	NA	NA	NA	3.9	5.5	5.5	4.2	4.0	7.3	7.3	NA	NA	NA	NA	NA	5.0	5.9
EM1	NA	4.5	4.7	2.9	4.9	4.3	5.2	4.7	5.2	4.4	NA	NA	NA	NA	NA	3.5	1.9
W4	NA	NA	NA	NA	4.3	6.5	4.9	6.1	6.5	4.8	NA	NA	NA	NA	NA	2.4	2.7

Table 20. Annual calcium release rates, 1976-1993.

Site	Annual Magnesium Release Rate, g/LT																
	76/77	78	79	80	81	82	83	84	85	86	87	88	89	90	91	92	93
EM8	NA	2.7	NA	2.9	1.2	1.5	0.99	0.61	0.52	0.52	NA	NA	NA	NA	NA	1.3	1.8
Seep 3	NA	0.59	NA	0.33	0.47	0.31	0.28	0.15	0.24	0.22	NA	NA	NA	NA	NA	0.019	0.027
Seep X	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	3.1	1.1
Seep 1	NA	4.5	NA	5.1	5.1	3.9	2.1	1.4	1.9	1.5	NA	NA	NA	NA	NA	0.68	0.58
W1D	NA	NA	NA	2.2	3.4	3.2	3.0	4.0	8.6	8.6	NA	NA	NA	NA	NA	6.8	6.8
EM1	NA	3.1	5.7	1.8	3.9	3.8	4.1	4.6	5.1	4.1	NA	NA	NA	NA	NA	2.9	1.5
W4	NA	NA	NA	NA	3.0	4.8	3.7	4.9	5.6	4.4	NA	NA	NA	NA	NA	2.3	2.6

Table 21. Annual magnesium release rates, 1976-1993.

References

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STS Consultants, Ltd. 1993. LTV Steel Mining Company final closure plan. 218 p. plus appendices.

Attachment 6

Lapakko K, Olson M (2015) Scaling Laboratory Sulfate Release Rates to Operational Waste Rock Piles. In: 10th International Conference on Acid Rock Drainage & IMWA Annual Conference.

Scaling Laboratory Sulfate Release Rates to Operational Waste Rock Piles

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ABSTRACT

There is little agreement on scaling factors to be used for extrapolating laboratory dissolution test results for predicting solute release rates from proposed waste rock piles in the field. The scaling factor for a given solute is the ratio of its release rate in the field to that observed in the laboratory, and its magnitude is dependent on both the solute and site-specific variables. Scaling factors for sulfate release rates from Duluth Complex rock of moderate sulfur content were determined empirically by comparison of laboratory rates to field rates. The laboratory rates were calculated based on sulfate release during weeks 6 to 71 observed for 17 blast hole samples with sulfur contents of 0.18 to 1.64 percent. The samples were collected from the mine site from which comparative field data were generated. Annual field rates were determined over a period of 3 to 13 years for five waste rock piles, ranging in mass from 2,000,000 to 15,000,000 tons, with estimated sulfur contents of 0.24 to 0.97 percent. Laboratory and field rates were expressed per unit mass sulfur. Comparison of 17 laboratory rates and 42 annual field rates yielded in 714 distinct calculated scaling factors. These values were fit to a beta distribution for which the mean and standard deviation were 0.127 and 0.083, respectively.

Keywords: mine waste rock, mine drainage, environmental review, mine waste drainage quality prediction, laboratory scaling

INTRODUCTION

Environmental review for proposed operations requires predicting solute release from mine waste storage facilities. Such predictions can be informed by operational phase data from mine wastes of similar composition, disposed in similarly constructed facilities, and weathering under similar climatic conditions. Such information from "analog" sites is often not available. Consequently predictions are based on the scaling of laboratory data to field conditions. The magnitude of such scaling will depend on the solute being scaled, drainage pH, rock type, rock composition, method of laboratory rate generation (e.g. humidity cells vs. column tests), method of field rate generation (e.g. test plots vs. full scale storage facilities), and reaction environment variables in both the laboratory and field (e.g. temperature, leachate to rock ratio, oxygen availability). Consequently, it cannot be assumed that scaling factors quantified will be universal.

Previous Studies

Two approaches have been used for scaling laboratory rates to the field, the first of which applies scaling factors for individual mechanistic factors to account for differences such as rock composition, reaction environment, and solute transport. Considerable data and analysis for this approach was generated by studies focused on waste rock piles containing two varieties of both gneiss and schist in northern Sweden (Strömberg & Banwart, 1994; Strömberg et al., 1994; Strömberg & Banwart 1999a; Strömberg & Banwart 1999b). Apparent rates of pyrite, chalcopyrite, biotite, and plagioclase weathering in the field were found to be roughly 0.01 to 0.2 times those derived from laboratory batch tests on six different particle size fractions (Malmström et al., 2000). A "scale-dependence model" accounted for the differences between field and laboratory rates using quantitative contributions of particle size, temperature, pH, and hydrologic variables affecting solute transport.

Kempton (2012) included these factors as well as moisture content and pore gas oxygen concentration in a literature review of scaling factors for solute release from waste rock. This is less detailed consideration than that of scaling mineral dissolution rates but is adequate for application to predicting solute release from waste rock. He concluded that prediction of solute release rates based on reported ranges of scaling factors would have a high degree of uncertainty. For example, he stated that waste rock particle size distribution and its relationship to hydrologic factors affecting solute transport produced a scaling range covering a factor of 7.5.

A second approach is that of empirically determining scaling factor based on comparison of laboratory and field rates for the rock in question. Wagner et al. (2006) compared sulfate release rates from mafic and feldspathic gneiss in a northern Saskatchewan test pile (8 x 8 x 5 m high) to those derived from laboratory dissolution tests on five different size fractions of the rock (Hollins et al., 2001). Sulfate release rates (normalized for surface area) in the field were 0.3 times those observed in the laboratory, suggesting that factors such as temperature and solute transport limited sulfate release in the field. There were no humidity cell data for the typical -0.25-inch fraction used in humidity cell tests to calculate a scaling factor directly from this test to the test pile.

Shaw & Samuels (2012) conducted paired humidity cell and 30-gallon barrel tests on three porphyritic intrusive rocks and one metasedimentary rock. The particle size was similar in both tests and the average temperature and precipitation in the field were approximately 16 °C and 100 mm, respectively. Drainage pH from two of the field tests were 3.5 and 1 unit lower than the corresponding laboratory test, and values from the remaining two pairs were in fairly close agreement. Direct extrapolation of laboratory data yielded predicted sulfate concentrations that

were roughly 0.07 to 0.3 those observed in the field tests. The comparatively low laboratory rates were attributed to higher pH conditions resulting from the higher flushing rate in the laboratory. It was speculated that the higher flushing enhanced dissolution of neutralizing minerals and prevented development of acidic micro-environments in the laboratory tests.

Robertson, Barazzuol & Day (2012) compared sulfate release rates from 13 humidity cell tests on fresh drill core to field rates observed for 7.6 million tonnes of rock that had been weathering for 27 years in the field, roughly 200 km northeast of Timmons, Ontario. The rock was predominantly massive and pillowed basaltic rocks, some hydrothermally altered, and felsic and mafic intrusives. Comparison of the sulfate release rate for the median sulfur content rock (0.5 percent S) in the laboratory with that observed in the field yielded a scaling factor of 0.1. The corresponding factor assuming a 95 percentile sulfur content (1.46 percent) to be controlling release in the field yielded a scaling factor of 0.05.

Hanna & Lapakko (2012) determined scaling factors for sulfate release using data generated by 26-week humidity cell tests on seven drill core samples of finely laminated, fine-grained rock, referred to by miners locally as the lower slaty member of the Biwabik Iron Formation (Severson et al., 2010). Sulfate release rates, normalized for pyrite mass, calculated for roughly 50 million tonnes of this rock in northeastern Minnesota were generally equal to or greater than the corresponding rates in the laboratory.

Lapakko (1994) compared results from laboratory dissolution tests on fine noritic and gabbroic Duluth Complex rock (0.053 < diameter ≤ 0.149 mm) from the South Kawishiwi Intrusion to those generated by roughly 1000-tonne test piles of similar Duluth Complex rock from the Partridge River Intrusion in northeastern Minnesota. Rates in the field were compared to laboratory rates for rock of similar sulfur content and yielded scaling factors of roughly 0.1 to 0.3.

OBJECTIVE AND APPROACH

The objective of this report is to present scaling factors for sulfate release from operation scale stockpiles of Duluth Complex rock in the field relative to those that would be generated in humidity cell tests. The approach employed data that was generated in studies previously conducted by the Minnesota Department of Natural Resources (MN DNR) (Kellogg et al., 2014, MN DNR, 1996). Specifically, field data are from monitoring at the Dunka Mine and laboratory data are from dissolution tests on rock from that site. Those tests were conducted on rock with diameters in the range of 0.053 – 0.149 mm. Corresponding humidity cell rates were projected using data comparing results from the two methods.

METHODS

Field

Beginning in the 1960's, Duluth Complex rock was excavated at the Dunka taconite mine (Figure 1) and stockpiled at the site in order to access the underlying iron formation. In the mid-1970's it was discovered that sulfate and trace metal concentrations were elevated in Unnamed Creek, which drains the mining watershed, and the Duluth Complex rock was determined to be the source of these solutes. Due partly to concerns regarding water quality impacts from potential copper-nickel mining in the area, a program was

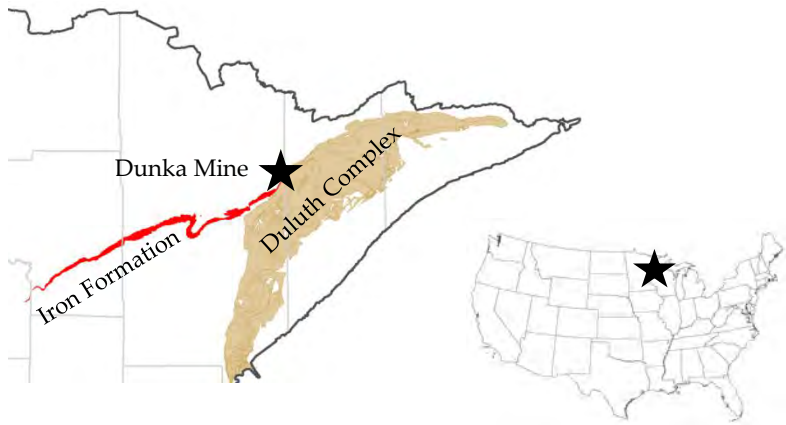


Figure 1 Location of Dunka Mine, Iron Formation and Duluth Complex in northeastern Minnesota

initiated to monitor the Duluth Complex rock seepage at the Dunka Mine. The following information was taken from a summary of waste rock composition, flow, and drainage quality data from the site (MN DNR, 1996).

The Duluth Complex rock mass within the individual seepage watersheds ranged from roughly 2 to 15 million tons, with average sulfur contents of approximately 0.2 to 1 percent (Table 1). Sulfur contents were determined based on analyses conducted on blast hole samples from 1980 to 1988. These analyses accounted for most of the rock in stockpiles 8018 and 8031, which drain to sites W1D and W4, respectively. Therefore, there is a relatively high degree of confidence in the sulfur contents and calculations based on these contents. In contrast, sulfur contents of stockpiles 8011, 8013, and 8014 were estimated based on analyses of roughly 19, 35, and 44 percent of the rock present. No sulfur analyses were submitted by the company for rock stockpiled after 1988.

Sulfate and metals are transported from the stockpiles in several well-defined seeps, as well as in diffuse flows. The seeps (and the associated stockpiles) discussed in the present report are designated EM8 (stockpiles 8011, 8014), Seep X (8013), Seep I (8013), W4 (8027, 8031), and WID (8018, 8031). Duluth Complex drainage quantity and quality data presented in this report were collected for periods of 3 to 13 years during the time span from 1979 to 1992. Since 1976, data have been collected on the drainage quality, quantity, and chemical mass release associated with stockpile drainages at the site. The Regional Copper-Nickel Study began the data collection and after 1980 monitoring was conducted by Erie Mining Company and LTV Steel Mining Company. The seasonal flow period typically extended from the middle of April to the end of November. Weirs and flow recording equipment were installed at all sites and flow was recorded automatically from roughly mid-May to mid-November. Beyond this time period flow measurements were conducted manually. Grab samples were typically collected 15 to 20 times per year for determination of pH, sulfate concentrations, and concentrations of other solutes that are not discussed in this paper.

Table 1 Duluth Complex stockpile and seep watershed characteristics

Seep	Stockpile	Deposition (start-end)	Mass in watershed,	%S	Area of pile in watershed, $ft^2 \times 10^6$	Watershed area,	Monitoring (start-end)
Seep X[1]	8013	1967-1991	8.44	0.24	1.3	1.7	1990-1992
Seep 1[2]	8013	1967-1991	1.86	0.24	0.50	0.65	1986-1992
EM8[3]	8011	1965-1986	12.92	0.23	3.6	7.7	1979-1991
W4[4]	8027[6]	1979-1979	0.14	0.12	0.12	7.5	1980-1991
W1D[5]	8018	1979-1985	2.18	0.98	0.75	1.9	1986-1992

[1] Weir and continuous flow recording equipment, installed in 1991
[2] Weir installed in 1977, recorder installed in 1980
[3] Weir and recorder installed in 1976
[4] Weir and recorder installed 1981 and 1983, respectively
[5] Weir and recorder installed 1981 and 1986, respectively
[6] Mass and composition reported by MN DNR, 1996

Laboratory

Seventeen samples were collected from blast holes at the Dunka mine (Figure 2a). These samples were characterized (chemistry, mineral content, mineral chemistry) and subjected to dissolution testing (Kellogg et al., 2014). For 15 of the 17 samples, particle size was reduced using a bucking maul, or mechanically, with a pulverizer. These samples were subjected to dissolution tests in duplicate. The remaining two samples (0.67, 0.82 %S) were stage crushed with a jaw crusher and were not replicated in dissolution testing. Particles with diameters from 0.053 to 0.149 mm (-100/+270 mesh) were retained for experimental use. For the crushed samples, sulfur content was determined by LECO furnace, metal concentrations were determined using ICP-AES, and mineral content and mineral chemistry were determined by microprobe analysis of 100 to 125 mineral grains for each sample.

Seventy-five grams of crushed rock were placed into the upper segment of a two-stage filter unit (Figure 2b). The solids were placed on a glass fiber filter that rested on a perforated plastic plate near the bottom of the reactor. At the inception of the experiment, each reactor was rinsed three times to remove any oxidation products that accumulated between sample crushing and onset of the experiment. Each week thereafter, two hundred milliliters of distilled-deionized water was added, and remained in contact with the solids for four to seven minutes, and then vacuum-pumped from the upper stage through a 0.45- micron filter on top of the lower stage of the filter unit. This procedure changed slightly beginning in July 2002. During weeks where no sample was collected, the reactors were filled with water as before, but were then gravity drained. The procedure remained the same as before for weeks during which samples were collected. Sample pH was determined weekly using either a Radiometer 29 or an Orion SA720 meter. Sulfate concentrations were determined biweekly (R1-R20) or monthly (R29-R43) at the MN DNR lab in Hibbing, MN using an HF Scientific DRT-100 nephelometer for the barium sulfate turbidimetric method (APHA et al., 1992). The sample volume was determined by weighing the sample collection flask containing the sample and subtracting the flask mass.

(a) (b) (c)

Figures 2a,b,c The first photo (a) depicts the weir at a site used for monitoring flow and water quality at the Dunka taconite mine in northeastern Minnesota, the second photo (b) depicts an MN reactor apparatus, and the third photo (c) depicts a typical humidity cell apparatus.

Between rinses, the solids remained within the reactors, allowing evaporation of retained rinse water and continuous oxidation. The reactors were stored in individual cubicles that formed a rectangular matrix within a topless housing with a perforated base; the housing was stored in a temperature and humidity controlled room (8.5 x 10.5 x 9.5 ft). Reactor experiments were initiated at three different times. For weeks 6-71 average temperatures for the three sets of samples ranged from 25.2-26.3 °C with standard deviations of 0.5-1.3 °C. Corresponding values for average relative humidity were 55.3-59.7% and 3.4-6.8%.

Calculations

For waste rock piles in the field, sulfate mass release was calculated for each day sulfate concentrations were measured. Flow data from the previous and subsequent measurements were used to determine the flow volume associated with the sulfate concentration (equation 1).

$$SO_4 \text{ mass release}, M(SO_4) = [SO_4]_i \left(\frac{Q_i+Q_{i-1}}{2}\right)\left(\frac{t_i+t_{i-1}}{2}\right) + [SO_4]_i \left(\frac{Q_i+Q_{i+1}}{2}\right)\left(\frac{t_i+t_{i+1}}{2}\right), \text{ where} \quad (1)$$

$M(SO_4)$ = mass of sulfate associated with sample i, mg.

$[SO_4]$ = sulfate concentration (mg/L),

Q = flow (liters/sec), and

t = time (seconds).

Cumulative sulfate mass release for the year was then divided by the product of the molecular weight of sulfate, annual mass of sulfur in the watershed and the time over which release occurred to determine annual average sulfate release rates (mol SO_4 (g S)$^{-1}$ s^{-1}) for each seep (Appendix 1, Table A1.1).

Annual rate of sulfate release was calculated as follows:

$(dSO4/dt)_A = \Sigma MSO4/(96.1*M(S)_{ws} * t)$, where (2)

$(dSO4/dt)_A$ = Average annual rate of sulfate release (mg/(gS*s)),

ΣMSO_4 = sum of mass sulfate release for individual sampling periods (mg), and

$M(S)_{ws}$ = mass of sulfur in the watershed (g).

The period over which flow and sulfate concentrations were determined typically extended from mid-April to the end of November. The calculation assumed that the sulfur content in each stockpile did not change over time (the final weighted average sulfur content was used), where in reality the sulfur content added to each pile varied each year.

For the 17 laboratory tests, the average sulfate release rate and associated standard deviation were determined for weeks 6 through 71 (Appendix 1, Table A1.2). The initial five weeks were omitted to eliminate contributions of sulfate generated during sample storage. Drainage pH values for most samples were circumneutral during this period, although minimum values as low as about 4.5 were observed for some of the higher sulfur sample. The mass sulfate release for each point at which sulfate concentration was determined was calculated as the product of that concentration and sample volume. For sulfate concentrations below detection (2 mg/L) a value of 1.0 mg/L was assumed. A total of 887 sulfate measurements were made for the 17 reactors (including the replicates), out of those 93 were below the detection limit. These typically occurred in the samples with S ≤ 0.41%. To determine the rate of release, normalized for sulfur content, the mass release was divided by the product of 604,800 seconds/week and the mass of sulfur present in the sample. Normalizing relative to the sulfur content allowed for direct comparison among laboratory and field rates.

Scaling factors were calculated to express the ratio of field rates to a laboratory rate from a humidity cell test (Figure 2c). This required scaling rates from the MN DNR reactors to humidity cells. This was done using data generated by subjecting Duluth Complex rock samples to testing in both reactors and humidity cells (seven and ten samples, respectively) (Lapakko, Olson, & Antonson, 2013). During weeks 6 through 71 (and throughout the first three years of testing) sulfate release rates as a function of sulfur content from the humidity cells were approximately one-third those from the reactors. Thus, the observed sulfate release rates for the reactors were divided by three to represent humidity cell rates.

A set of scaling factors was then calculated using the ratio of each of 42 annual field rates to each of the adjusted 17 laboratory rates. This generated a set of 714 distinct scaling factors (42 x 17 = 714) (Figure 3a). This data set was fit to normal, beta, and log normal distributions, and the associated mean and standard deviations were determined.

RESULTS AND DISCUSSION

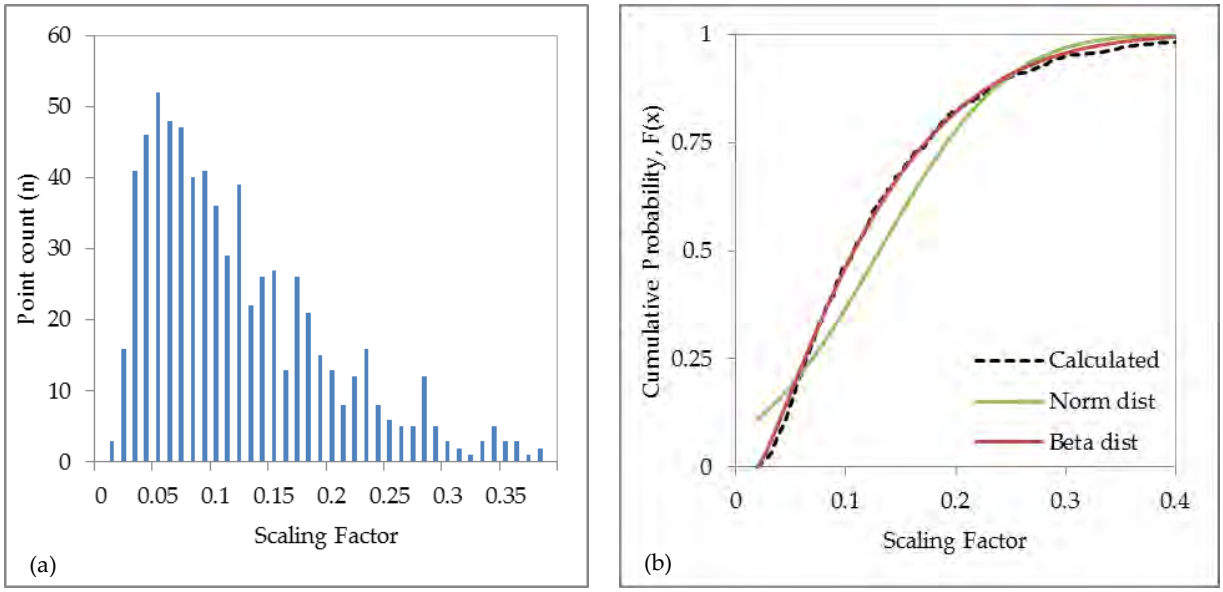
The Duluth Complex rock mass within the individual seepage watersheds ranged from roughly 2 to 15 million tons, with average sulfur contents of approximately 0.2 to 1 percent (Table 1). Because lab samples were collected from blast holes at the site, it is assumed that the mineralogy is similar to those in the lab (see below). Sulfate concentrations in the field generally ranged from 600 to 1600 mg/L and geochem equilibrium modeling of 269 seepage samples indicated that the drainages were undersaturated with respect to gypsum in all cases except one.

Sulfate release rates in the field typically ranged from 0.8 x 10^{-12} to 4 x 10^{-12} mol (g S)$^{-1}$ s^{-1}, with a median value of 1.7 x 10^{-12} mol (g S)$^{-1}$ s^{-1} (Appendix 1, Table A1.1). These values were in good agreement with calculations by a slightly different method (MN DNR, 1996). Rates calculated for Seep 1 tended to be on the lower of the range and those from W4 on the higher end. Their respective mean rates were 1.2 x 10^{-12} and 3.3 x 10^{-12} mol (g S)$^{-1}$ s^{-1}. Rates for individual seeps showed no consistent temporal trend and values for each seep fell within a fairly consistent range over the period of record. The percent standard deviation of the mean value ranged from about 35 to 55 percent (Appendix 1, Table A1.1). Rates for Seep X, Seep 1 and EM8 were in reasonable agreement with those for W4 and W1D (Appendix 1, Table A1.1). Sulfur content of the rock in the watersheds of the last two sites was determined throughout the period of stockpiling (see Methods). The agreement among sulfate release rates suggests that estimations of sulfur content made for rock in the first three watersheds were not unreasonable. Minimum annual pH values for four of the five seeps reflected circumneutral pH conditions, typically ranging from 6.2 to 6.6. Values for Seep 1 were markedly lower, generally in the range of 4.6 to 4.8 (Appendix 1, Table A1.1).

Sulfur content of samples subjected to dissolution testing ranged from 0.18 to 1.64%. Pyrrhotite was the dominate sulfide mineral, followed by chalcopyrite, cubanite, and pentlandite. Cu and Zn sulfides were also observed in trace amounts. Plagioclase (36-65%), augite (3-17%), hypersthene (5-28%), and olivine (3-21%) contributed 65-97% of mineral content in Duluth Complex samples. Eleven samples contained the plagioclase species labradorite, while six samples contain the more sodium-rich species andesine (0.18 to 1.64%S). Unleached Duluth Complex silicate minerals had the following range of stoichiometric coefficients: plagioclase $(Ca_{0.40-0.63}Na_{0.32-0.55})Al_{1.44-1.66}Si_{2.35-2.57}O_8$, augite $((Ca_{0.76-0.95}Na_{0.01-0.03})(Mg_{0.36-0.79}Fe_{0.27-0.63}Ti_{0.01-0.02})(Si_{1.95-2.00}Al_{0.03-0.1})O_6)$, hypersthene $((Mg_{0.91-1.16}Fe_{0.78-1.66})2Si_{1.97-2.00}O_6)$, and olivine $((Mg_{0.72-1.00}Fe_{0.96-1.25})Si_{0.99-1.01}O_4)$ (Kellogg et al., 2014).

Laboratory rates typically ranged from 1.1 x 10^{-11} to 2.3 x 10^{-11} mol (g S)$^{-1}$ s^{-1} with three values in the range of 2.5 x 10^{-11} to 3.9 x 10^{-11} mol (g S)$^{-1}$ s^{-1} (Appendix 1, Table A1.2). Rates for six of the samples included estimations of sulfate concentrations for values reported as less than detection (Appendix 1, Table A1.2). For three samples with seven or fewer below detection values this introduced a maximum error not exceeding two percent. The maximum error increased to eight percent for 15 below detection values and 19 percent for 35 sulfate concentrations reported below detection. Median pH values for the period of record ranged from 4.3 to 6.6, and minimum pH values were generally 0.6 to 1.3 units lower. There was little dependence of these rates on pH, as indicated by a correlation coefficient of 0.062 for the regression of sulfate release rate against the median pH values.

The overall range of scaling factors was 0.019 to 0.687 with about two thirds of the values falling between 0.053 and 0.21 (Figure 3a). Fitting the set of scaling factors to a normal distribution yielded respective values for the mean, standard deviation, and skewness of 0.13, 0.09, and 1.69, respectively. Given the asymmetry of the data relative to a normal distribution (Figure 3a), a beta distribution was applied for dynamic systems modeling for scaling at a proposed mining operation in Minnesota. Scaling factors for probabilities of 5, 25, 50, 75, and 95 percent were 0.034, 0.064, 0.11, 0.17, and 0.30, respectively (Figure 3b). The respective mean and standard deviation for this distribution were 0.127 and 0.083; minimum, maximum, alpha, and beta values were 0.019, 0.687, 1.25, and 6.50, respectively. A log-normal distribution also fit the data well with mean and standard deviation for ln(x) of -2.25 and 0.675, respectively.



Figures 3a,b a) Calculated scaling factor histogram showing point counts and b) cumulative probability of composite scaling factors compared to normal and beta distribution fits. Omitted scaling factors > 0.40 to improve resolution.

The modeling used the distribution of scaling factors to generate a corresponding distribution of sulfate release rates for the field. Such distributions are recommended by the National Academy of Sciences (National Research Council, 2007) as opposed to a single value. They are deemed more appropriate for assessment of risk by regulatory agencies, an assessment that is essential to environmental review of proposed mining operations.

It should be noted that the scaling factors presented were calculated based on field rates determined during periods of open flow (roughly April through November) and ignored winter periods when ice covered seepage areas (roughly December through March). There were few data available to compare sulfate release rates during the winter to those during the remainder of the year. The best data available were from the outflow of the watershed, site EM1, roughly 1 to 3 km downstream of the seeps. From 1975 to 1998 average sulfate concentrations during the winter were 0.71 times the average during the summer (99 and 389 measurements, respectively). From 2000 to 2014, National Pollution Discharge Elimination System (NPDES) data indicated the monthly average flow during the winter was 0.46 times that during the open flow season (52 and 116 measurements, respectively). Using these ratios allowed estimation of yearly field rate (open flow and ice cover) as 84 percent of that during open flow alone. This would in turn yield scaling factors that were 84 percent of those calculated. For example, the mean value for the beta distribution would be reduced from 0.13 to 0.11. The lower scaling factor does not, however, imply a lesser potential for environmental impact. The lower field sulfate release rates estimated for the winter months actually yielded higher sulfate concentrations in the receiving stream due to lower input flows of unimpacted water.

Beyond the statistical description of uncertainty, specific variables in the calculation were considered. First, field rates were calculated based on sulfur contents of rock within seep watersheds. As discussed above, sulfate release rates for watersheds in which sulfur content was determined throughout stockpiling were in reasonable agreement with those for watersheds with less frequent rock analyses. Thus, it appears that such estimation did not introduce substantial error.

Second, the potential role of pH was investigated. As discussed above, the laboratory rates exhibited little dependence on pH for the period of record considered. Although most of the field seeps exhibited circumneutral pH, it is possible that acidic "hot spots" with elevated sulfate release rates could develop in isolated areas of piles. However, there were no indications suggesting such areas existed. That is, there were no sudden drops in pH and release rates of sulfate remained with constant ranges (Appendix 1, Table A1.1).

Indicators of acidification were observed at Seep 3 at the Dunka mine, a seep not included in the present calculation of scaling factors. From 1975 through1988 drainage pH at the seep was near 7.0, but declined to a minimum of 4.8 in 1989 (MN DNR, 1996). Despite pH values in the typical circumneutral range, the annual release rates for nickel, copper, cobalt, and zinc increased by factors of roughly 20 to 80 from 1986 to 1988, and the sulfate release rate increased by about a factor of nine during this period. Such increases are often associated with isolated acidic zones in waste rock piles generating circumneutral pH. In contrast, sulfate release rates from each of the seeps used for scaling calculations fell within a fairly constant range (Appendix 1, Table A1.1), as discussed above. Similarly, heavy metal concentrations from these seeps did not exhibit the radical changes observed at Seep 3. Thus, it seems unlikely that acidic zones within these piles influenced rates of solute release.

Acidic drainage pH was observed at Seep 1, where minimum values were between 4.6 and 5.2 (Appendix 1, Table A1.1), but the sulfate release rate for this site was the lowest of the five sites examined. This, in conjunction with the laboratory data, suggests there is not a strong dependence of sulfate release on pH in the range of roughly 4.5 to 7.

Third, the reliability of flow measurements was considered. Flow measurements were automated from roughly the middle of May to the middle of November and measurements were made manually beyond this period. To assess the reasonability of the annual flow volumes, these values were divided by total precipitation onto the seep watershed to quantify the annual fractional yields. The average annual yield coefficients for the periods of record for EM8, Seep 1, W1D and W4 were 0.28, 0.33, 0.36, and 0.37, respectively. These values do not suggest any substantial error in flow measurement.

The average annual yield coefficient for Seep X was notably higher at 0.74. At this site the yield coefficient decreased steadily from 1.03 to 0.56 over the three-year period of record and declined to 0.34 the following year. Visual observation of flow at the site was consistent with the high values reported. It is believed that release of stored water was responsible for the initial elevated flows, although the precise mechanism is unknown. The ultimate value of 0.34 is consistent with the annual averages from the other sites and is believed to reflect "typical flow" from the pile.

Finally, the scaling factors determined are in general agreement with values previously determined empirical scaling factors for Duluth Complex and Biwabik Iron Formation rock in northeastern Minnesota. Lapakko (1994) compared sulfate release rates from South Kawishiwi Intrusion rock tested in MN DNR reactors to rates observed for 1000 tonne test piles of Partridge River Intrusion rock. The approximate range of scaling factors reported was 0.1 to 0.3. The rates for the South

Kawishiwi rock in MN DNR reactors reported in the study can be converted to Partridge River rock in humidity cells. Subsequent research indicated that transforming these rates for influences of dissolution test type (MN DNR reactor rate/humidity cell rate ~ 3) and specific intrusion (South Kawishiwi rate/Partridge River rate ~ 0.5) (Lapakko, Olson, & Antonson, 2013). The resultant range of 0.15 to 0.45 represents that for scaling sulfate release rates from humidity cells to test piles, which is higher than the 0.127 mean for scaling humidity cell rates to operational scale piles. This seems reasonable considering, for example, that reaction product transport would likely be higher in the smaller test piles.

Hanna & Lapakko (2012) determined empirical scaling factors for waste rock piles from the lower slaty unit of the Biwabik Iron Formation waste rock piles in northeastern Minnesota. The more reliable of two methods reported yielded a range of 1.0 to 1.3, that is, field rates of sulfate release per gram pyrite were higher than those observed for drill core in the laboratory. This was largely attributed to laboratory rates determined for weeks 15 to 26 of humidity cell testing. Additional testing of weathered rock yielded rates that were roughly an order of magnitude higher than those for drill core. Using these rates for determination of scaling factors would have produced values near 0.1, which is in general agreement with the values determined in the present work. The values determined in the present work generally agree with those presented for operational scale piles by Robertson, Barazzuol, & Day (2012) and not extremely deviant from other literature values presented. However, as discussed previously, comparisons of scaling factors among different sites must be done with caution due to differences in site specific environmental conditions, rock characteristics, and test methods.

SUMMARY AND CONCLUSIONS

Sulfate release rates from Duluth Complex rock were determined for 17 laboratory tests and five operational waste rock seepages in northeastern Minnesota. The 714 scaling factors, representing the ratio of field rates to those in the laboratory, were fit to a beta distribution that yielded a mean of 0.127 and a standard deviation of 0.083. This empirically-based distribution forms a foundation on which laboratory rates can be scaled to develop a probability distribution of predicted operational rates for the purpose of environmental review, consistent with recommendations by the National Academy of Sciences (National Research Council, 2007). It should be noted that the results were generated based on dissolution of a specific rock type in the laboratory and under specific conditions of climate and waste rock stockpile design in the field. Consequently, care must be taken when applying these results to other conditions. It does provide an example for development of empirically based scaling factors. Further application of this approach to other rock types and field conditions will reduce uncertainty in the presently tenuous extrapolation of laboratory test results to the field conditions for proposed mining operations. Consequently, it is recommended that operations and regulatory agencies generate, compile and analyze data to develop additional distributions for empirical scaling factors.

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APPENDIX 1. Field and laboratory release rates

Table A1.1 Annual precipitation, sulfate release rates (mol SO_4 (g S)$^{-1}$ s^{-1}), and minimum annual pH for the 5 seeps from 1979-1992

Year	Annual precip (cm)	Seep X[1]		Seep 1[2]		EM8[3]		W4[4]		W1D[5]	
		Rate	Min pH	Rate	Min pH	Rate	Min pH	Rate	Min pH	Rate	Min pH
1979	63	NA	NA	NA	NA	1.6E-12	6.6	NA	NA	NA	NA
1980	68	NA	NA	NA	NA	2.7E-12	6.8	8.4E-13	7.2	NA	NA
1981	70	NA	NA	NA	NA	1.5E-12	6.7	2.4E-12	6.6	NA	NA
1982	87	NA	NA	NA	NA	1.8E-12	6.4	4.1E-12	6.4	NA	NA
1983	78	NA	NA	NA	NA	1.2E-12	6.4	3.3E-12	6.2	NA	NA
1984	56	NA	NA	NA	NA	1.1E-12	7.0	4.0E-12	6.8	NA	NA
1985	84	NA	NA	NA	NA	8.6E-13	6.7	6.5E-12	6.5	NA	NA
1986	68	NA	NA	9.1E-13	5.1	7.8E-13	6.4	4.3E-12	6.4	1.5E-12	6.7
1987	56	NA	NA	7.5E-13	4.8	7.5E-13	6.5	2.8E-12	6.6	3.2E-12	6.9
1988	79	NA	NA	9.1E-13	5.2	2.0E-12	6.4	2.1E-12	6.7	1.7E-12	6.4
1989	66	NA	NA	2.5E-12	4.8	3.1E-12	6.3	3.3E-12	6.7	2.1E-12	6.8
1990	71	2.8E-12	6.2	8.5E-13	4.7	2.2E-12	6.9	2.9E-12	6.6	1.3E-12	6.9
1991	74	4.8E-12	6.2	1.6E-12	4.6	1.7E-12	6.8	3.1E-12	6.6	2.1E-12	6.8
1992	65	1.9E-12	6.5	8.2E-13	4.8	NA	NA	NA	NA	1.4E-12	6.7
Avg	70.4	3.2E-12	6.3	1.2E-12	4.9	1.6E-12	6.6	3.3E-12	6.6	1.9E-12	6.7
S.D.	9.3	1.5E-12	0.2	6.4E-13	0.2	7.3E-13	0.2	1.4E-12	0.2	6.6E-13	0.2

[1] Number of annual samples collected for analysis from 1990 to 1992 was 10, 23, and 10, respectively.
[2] Number of annual samples collected for analysis from 1986 to 1992 was 15, 15, 16, 14, 12, 14, and 7, respectively.
[3] Number of annual samples collected for analysis from 1979 to 1991 was 15, 13, 11, 15, 14, 14, 13, 15, 22, 20, 23, 18, and 24, respectively.
[4] Number of annual samples collected for analysis from 1980 to 1991 was 5, 15, 21, 15, 16, 14, 16, 15, 19, 23, 19, and 18, respectively.
[5] Number of annual samples collected for analysis from 1986 to 1992 was 16, 19, 15, 16, 16, 18, and 7, respectively.

Table A1.2 Summary of Dunka blast hole laboratory data (weeks 6-71). Average sulfate release rates (mol SO_4 (g S)$^{-1}$ s^{-1}) were determined for duplicated samples.

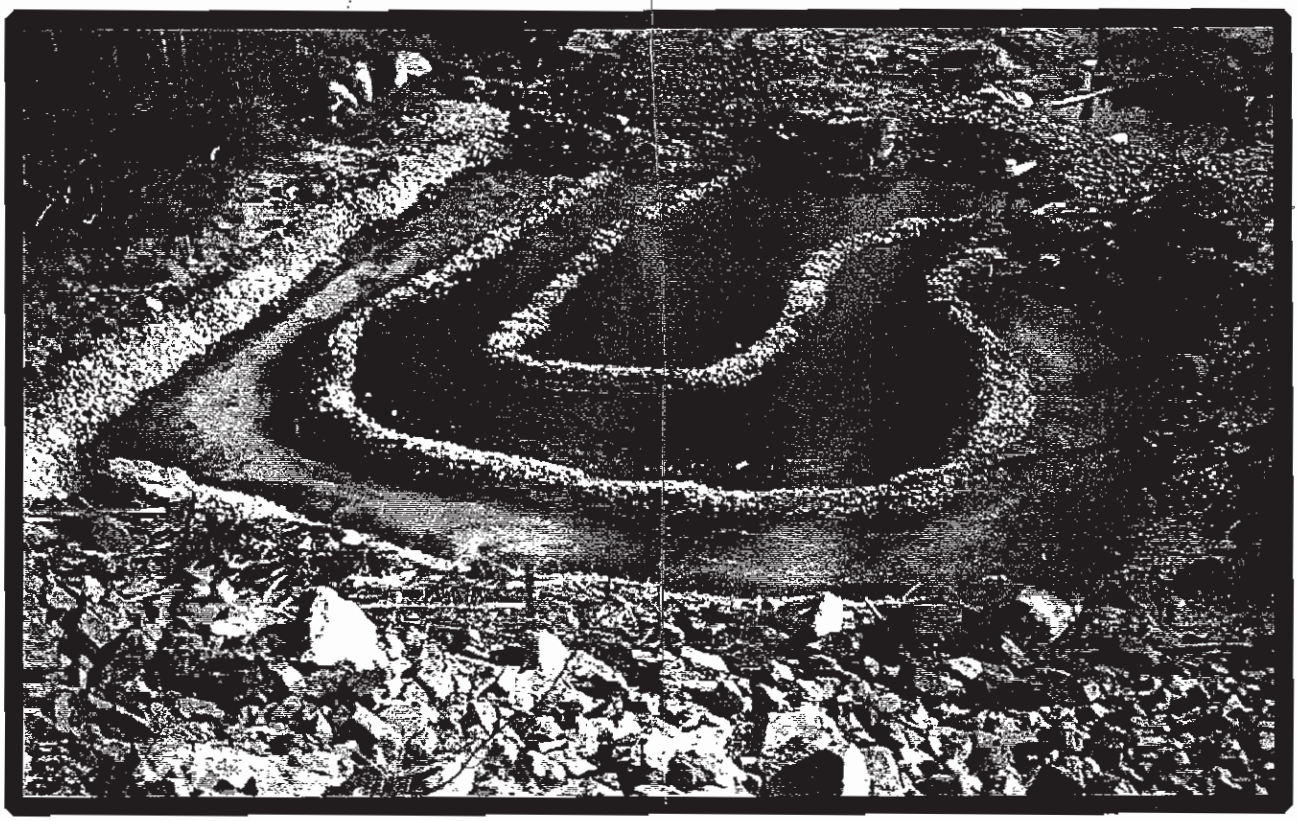
Reactor(s)	%S	Median pH	Minimum pH	Sulfate count	Avg dSO_4/dt
1,2[1]	0.18	6.55	5.97	65 (29<DL)	2.29E-11
3,4[1]	0.22	6.63	6.07	61 (15<DL)	1.79E-11
5,6[1]	0.40	6.30	4.86	64	3.89E-11
7,8[1]	0.41	5.32	4.48	64 (35<DL)	9.48E-12
9,10[1]	0.51	5.01	4.00	64 (7<DL)	1.70E-11
11,12[1]	0.54	5.17	4.10	64	2.32E-11
13,14[1]	0.57	5.34	4.09	64	2.28E-11
15,16[1]	0.58	4.69	4.04	64 (3<DL)	1.77E-11
40[3]	0.67	5.68	4.04	17	1.70E-11
17,18[1]	0.71	5.03	3.80	62 (4<DL)	1.35E-11
43[3]	0.82	5.42	4.49	17	1.19E-11
35,36[2]	1.12	4.30	3.51	43	3.01E-11
29,30[2]	1.16	4.61	3.63	44	1.36E-11
37,38[2]	1.40	4.84	3.55	44	1.42E-11
33,34[2]	1.44	4.43	3.42	42	1.15E-11
19,20[1]	1.63	4.38	3.61	64	1.60E-11
31,32[2]	1.64	4.40	3.64	44	2.54E-11
				Average	1.90E-11
				St Dev	7.52E-12

[1] Reactor initiated 2/14/89, average T = 26.25 ± 1.19, average R.H. = 57.77 ± 6.77.
[2] Reactor initiated 9/4/90, average T= 25.31 ± 1.32, average R.H. = 55.27 ± 5.79.
[3] Reactor initiated 8/12/97, average T= 25.23 ± 0.51, average R.H. = 59.65 ± 3.36.

Attachment 7

Eger P, Wagner J, Melchert G, Antonson D, Johnson A (2000) Long Term Wetland Treatment of Mine Drainage at LTV Steel Mining Company's Dunka Mine. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN.

Long Term Wetland Treatment of Mine Drainage at LTV Steel Mining Company's Dunka Mine



December 2000

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Executive Summary

A study was conducted at LTV's Dunka Mine in northeastern Minnesota to examine the long term metal removal in two wetland treatment systems that had operated for seven years. One system was an overland flow wetland that had been constructed to treat neutral drainage with an average nickel concentration of around 5 mg/L. The other was a pretreatment system which was installed to treat a drainage with an average pH of 5.4, and which contained 14.7 mg/L of nickel and 0.69 mg/L of copper.

W1D overland flow wetland

The wetland system was constructed in 1992, and the stockpile that contributed the major load to the wetland was capped with a linear low density polyethylene liner in 1995. Over 90% of the metal input to the wetland occurred before the stockpile was capped. After capping, both flow and nickel concentrations decreased. Flow decreased by 55%, while average nickel concentrations decreased by 82%, from 3.98 mg/L to 0.74 mg/L.

The overall nickel removal in the system prior to capping the stockpile averaged 89%. After capping, the input load to the wetland decreased by over 90% and the average percent removal decreased to 61%. Despite lower input concentrations after capping, nickel removal continued throughout the entire study period.

In order to examine treatment lifetime, a portion of the wetland (W1D study cell), which had accounted for 26% of the total nickel removal, was selected for detailed study. Mass balance calculations conducted for the study cell indicated that essentially all the nickel removed from the water could be accounted for by the estimated nickel mass within the substrate. There was no evidence of nickel being removed from the wetland.

Assuming that the post capping load to the wetland remains unchanged, and that new metal removal sites are formed from decaying vegetation in the wetland, there appears to be a balance between the input metal load and the yearly generation of removal sites. If this situation continues, the wetland may be self-sustaining, and treatment may continue indefinitely.

Seep 1 pretreatment system

The pretreatment system is comprised of a pool with two limestone berms, a peat-mixture substrate, and a vertical down-flow section through which the water flows prior to discharge. The system was constructed in 1992, and has generally been successful in increasing pH from an average of 5.40 to 6.95. Copper concentrations have decreased by about 70% and nickel concentrations have decreased by 55%.

The pH increase and much of the copper removal are related to dissolution of the limestone within the pretreatment system. Although it was not possible to calculate a limestone dissolution rate and an expected lifetime, there is no data to suggest that the rate of dissolution has decreased. Nickel removal within the pretreatment system averaged only 15-20%, and occurred primarily in the vertical down-flow section of the system. The major reduction in nickel load appears to be related to capping of the stockpile, and not to removal within the pretreatment system. In 1999, removal in the vertical down-flow section was only 8%, and additional data should be collected to determine if the system has reached saturation.

Introduction

Wetlands have been used to treat a variety of water quality problems, including agricultural, municipal and industrial discharges (Hammer, 1989, Moshiri, 1993). Wetlands have also been successful in treating coal and metal mine drainage and can be an attractive alternative to more conventional treatment methods (Hedin et al., 1994, Eger et al., 1996, Sobolewski, 1997). Wetlands can be less costly to build, use processes which naturally occur to remove metals from the water (e.g. adsorption, filtration), and offer a system that ideally should operate with little to no maintenance for extended periods of time. Since mine drainage problems can persist for hundreds of years, the longevity of any system is a critical issue. The lifetime of a wetland treatment system is a function of the type of removal processes occurring in the wetland. Figure 1 depicts the types of removal processes (and their typical locations) that can occur within a wetland treatment system.

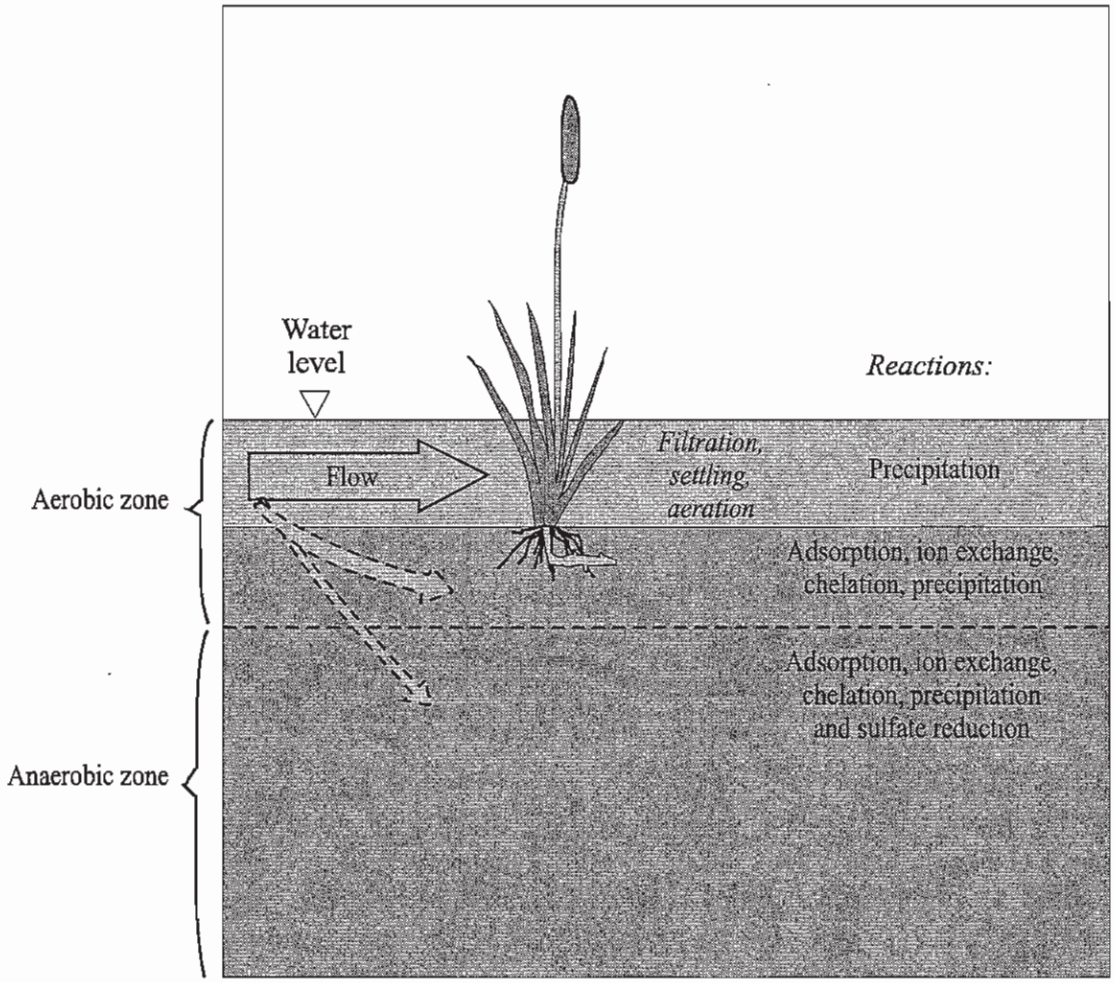


Figure 1. Metal removal processes that occur in overland flow wetlands (schematic).

Physical removal

Some of the contaminants that enter wetlands are associated with solid particles. These can be removed by sedimentation as water spreads out over the wetland and the velocity of the water decreases. As a result, particles begin to settle out and accumulate in the wetland. In addition to physical removal by settling, dense vegetation acts as a filter to remove additional material. Particulate removal can continue for an extended period of time, or until the wetland fills with sediment.

Chemical removal

Chemical processes occur in both the upper, oxygenated , aerobic zone of the wetland and in the deeper, oxygen-poor, anaerobic zone. Some removal reactions including adsorption, ion exchange, and chelation can occur in both zones. Certain processes such as the oxidation of ferrous iron to ferric iron can only occur in the aerobic zone, whereas reactions like sulfate reduction require an anaerobic condition.

Aerobic processes

When there is water on the surface of the wetland, aerobic conditions exist throughout the water column, but conditions quickly become anaerobic below the substrate surface. If the water level is below the substrate surface, the area of the substrate above the water level is usually aerobic. In addition, when vegetation is present, an aerobic zone exists around the roots of each plant as the plant transports oxygen to its roots (Grosse, 1989, Michaud and Richardson, 1989). This occurs even when the roots of the plant are totally submerged.

Most of the wetlands constructed to treat mine drainage have been used in the coal mining industry. These wetlands have typically been constructed so that flow occurs across the surface of an organic substrate which were planted with cattails, and treatment occurs in the aerobic zone (Wieder, 1989, Hedin, 1989). Iron removal primarily occurrs through the oxidation of ferrous iron to ferric iron, and the subsequent removal of iron as a ferric oxyhydroxide precipitate. In addition to iron removal, pH is reduced. This reaction will continue until the wetland fills with iron precipitate. A wetland in Pennsylvania not only operated successfully for over eight years, but treatment efficiency increased over time (Stark et al., 1994). Recently several researchers have been investigating the potential to recover the iron precipitate from wetlands for use as a coloring pigment (Hedin, 1998).

For neutral metal mine drainage, wetland systems similar to those used for coal drainage can effectively remove low levels of metals (Eger and Melchert, 1992, Eger et al., 1996, Hambley, 1996). In these systems, removal reactions primarily occur within the substrate and occur in both the aerobic and anaerobic zones. Most removal occurs in the upper 20 cm of the substrate, since the transport of metals to the deeper layers usually is limited by the rate of diffusion (Eger and Lapakko, 1988, Eger et al., 1994). Ion exchange reactions often involve the exchange of a hydrogen ion for the metal ion, therefore causing pH to decrease. As pH decreases, removal efficiency also decreases. In laboratory studies, metal removal decreased by over 50% as pH decreased from 7.4 to 4.0 (Lapakko and Eger, 1988).

The lifetime of adsorption, chelation, ion exchange reactions is a function of the amount of removal sites present in the substrate. As a result, each substrate has a finite ability to remove metals through these mechanisms. New sites can be generated in a wetland as vegetation decays, but this is a slow process; peat in northern wetlands accumulates at a rate of about 1mm/year (Craft and Richardson, 1993). Although it is theoretically possible to design a system where the input of new sites would equal the input of metals, the ratio of wetland area to flow must be about ten times larger than the ratio to provide short term treatment (Eger et al.,1994).

Anaerobic processes

In most wetlands anaerobic conditions develop in the saturated zone, below the soil/water interface. Although exchange reactions can still occur, the primary reaction which can provide long-term metal removal is sulfate reduction. Sulfate reduction reactions cannot only remove metals but can also increase pH, and therefore can treat acid mine drainage effectively. The reactions involved can be represented as follows:

$$SO_4 + 2 \text{ lactate} \rightarrow 2 \text{ acetate} + H_2S + 2HCO_3$$

$$H_2S + M^{+2} \rightarrow MS + 2H^+$$

Bacteria break down complex organic materials into small chain organic compounds (e.g. lactate). These compounds, along with sulfate, can then be used as an energy source by the sulfate reducing bacteria. Sulfate reducing bacteria are ubiquitous and tolerate a wide range of environmental conditions. Their optimal pH range has been reported to be from 5 to 9 (Postgate, 1984), but they can control their microenvironment even when the bulk solution pH is below 5. Successful sulfate reduction has been reported for a drainage with a pH as low as 2.6 (Bolis et al.,1991; Gusek, 1998).

Although anaerobic zones exist in natural wetlands, the peat material itself breaks down very slowly, so the rate of sulfate reduction is low. By constructing a wetland with a more reactive organic substrate, short-term reaction rates can be increased significantly. Spent mushroom compost has been commonly used in constructed wetlands in the eastern United State (Wieder, 1989). Experiments in Colorado and Arkansas have used other local waste products such as steer manure, rice hulls, and chicken litter as the substrate (Howard et al., 1989, Gross et al., 1991).

Sulfate reduction reactions can be modeled as a series of chemical reactions where the rate of reaction is related to the concentration of sulfate and the concentration of small chain organics. Since sulfate is generally present in high concentrations in most mine drainage, the limiting reactant is the amount of small chain organics. Lifetime estimates based on the total amount of organic material in constructed wetlands have been at least 20 to 30 years (Wildeman et al., 1993). Although these wetlands can continue to treat for extended periods of time, treatment efficiency will decrease as the more easily degraded organic material is consumed (Eger and Wagner, 1995, 2001). In several studies efficiency has been improved when a supplemental organic carbon source was added to the system (Stark et al., 1995).

Objectives

The objectives of this study were to:

- Examine metal removal in existing wetland systems at LTV's Dunka Mine in northern Minnesota
- Determine the potential of these systems to provide long-term treatment

Site description

The Dunka Mine was a large open pit taconite operation which operated from 1962 to 1995 (Figure 2). At this location the Duluth Complex, a metalliferous gabbroic intrusion, overlaid the taconite ore and was removed and stockpiled along the east side of the open pit. The Duluth Complex material contains copper, nickel, and iron sulfides, and the stockpiles contain over 50 million tons of waste rock and cover about 300 acres (120 hectares). Seeps appear at the base of the stockpiles, and generally flow continuously from early April to late November. Average flows from the various seeps have ranged from 8 to 220 gal/min (0.5 L/sec to 14 L/sec), but flows exceeding1600 gal/min (100 L/sec) have been observed after periods of heavy precipitation.

Nickel is the major trace metal in the drainage, and annual median concentrations prior to closure were on the order of 3-30 mg/L. Copper, cobalt, and zinc are also present but are generally less than 5% of the nickel values. Median pH ranges from 5.0 to 7.5, but most of the stockpile drainage has pH greater than 6.5.

Wetlands were located near every stockpile and appeared to offer potential treatment areas for each seep (Eger and Lapakko, 1989). These wetlands are typical of the many small lowland areas in northern Minnesota, and would generally be associated with any mining operation in the area.

In the mid-1980s, LTV began an extensive program to evaluate various options for mitigating the problems at this mine. The company's preferred option was a combination of passive alternatives which would reduce flow emanating from their stockpiles, and which would use wetland treatment to remove metals from the resulting drainage. In 1988, four overland flow test cells were built to investigate methods to optimize metal removal and to provide design data for the ultimate implementation of wetland treatment at this facility (Eger and Lapakko, 1989; Eger et al.,1991, 1993, 1994). Based on the results of this study, two full-scale wetland treatment systems were built in March of 1992. In addition, two pretreatment systems were installed in the summer of 1992.

By the time the Dunka mine was closed in 1995, the company had already begun to implement a closure plan. The amount of water flow through the stockpiles was reduced by routing surface and groundwater away from the piles. Infiltration into the waste rock was reduced by covering the top portion of the stockpile. Any residual drainage was to be treated in constructed wetlands (Eger et al., 2000). This study focuses on two sites constructed in 1992, where large amounts of metals had been removed; the W1D wetland treatment system, and the Seep 1 pretreatment system (Figure 2).

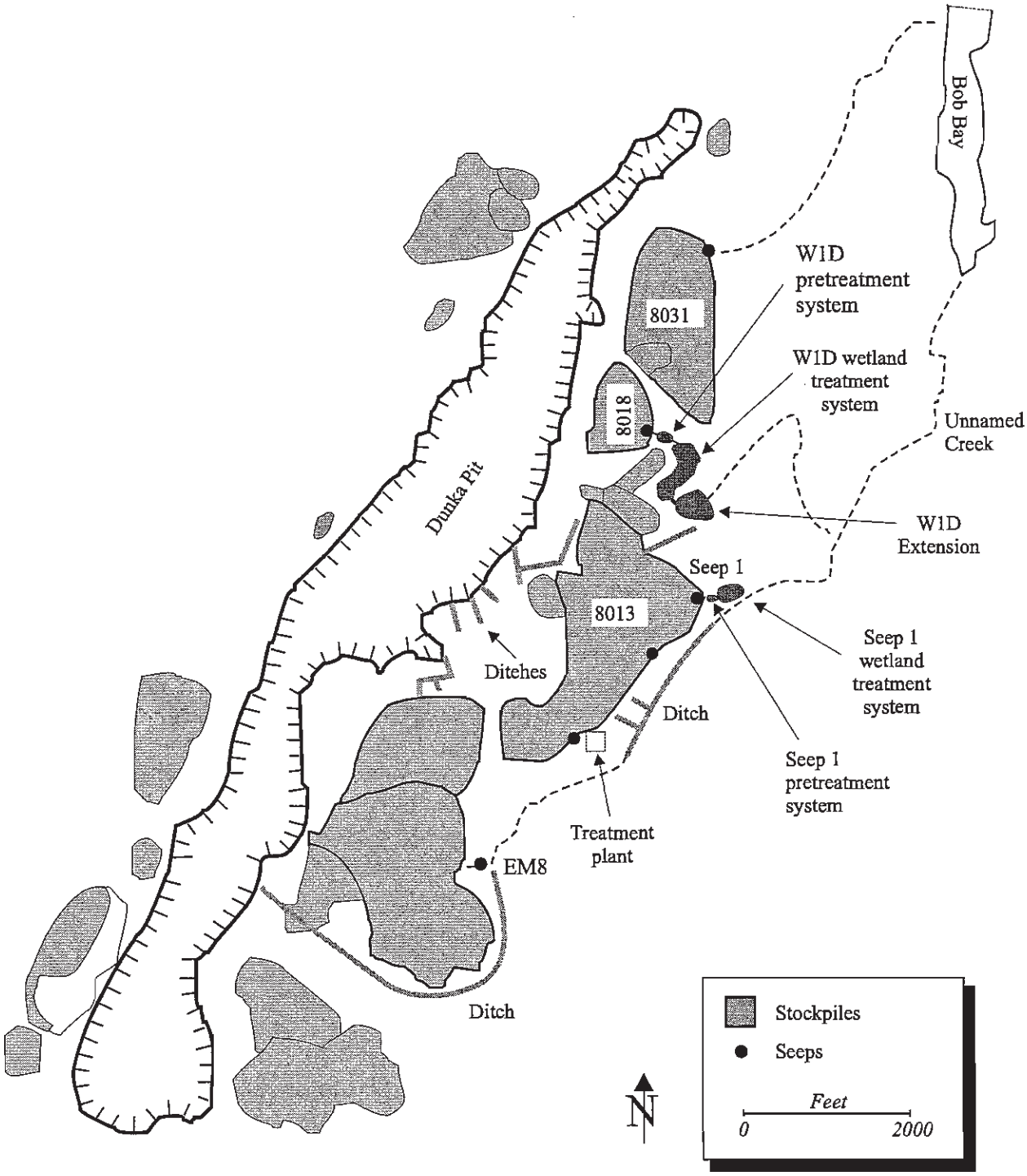


Figure 2. Location of the Seep 1 and W1D wetland treatment systems at LTV Steel Mining Co.'s Dunka Mine, Babbitt, MN.

Overland flow wetland, W1D

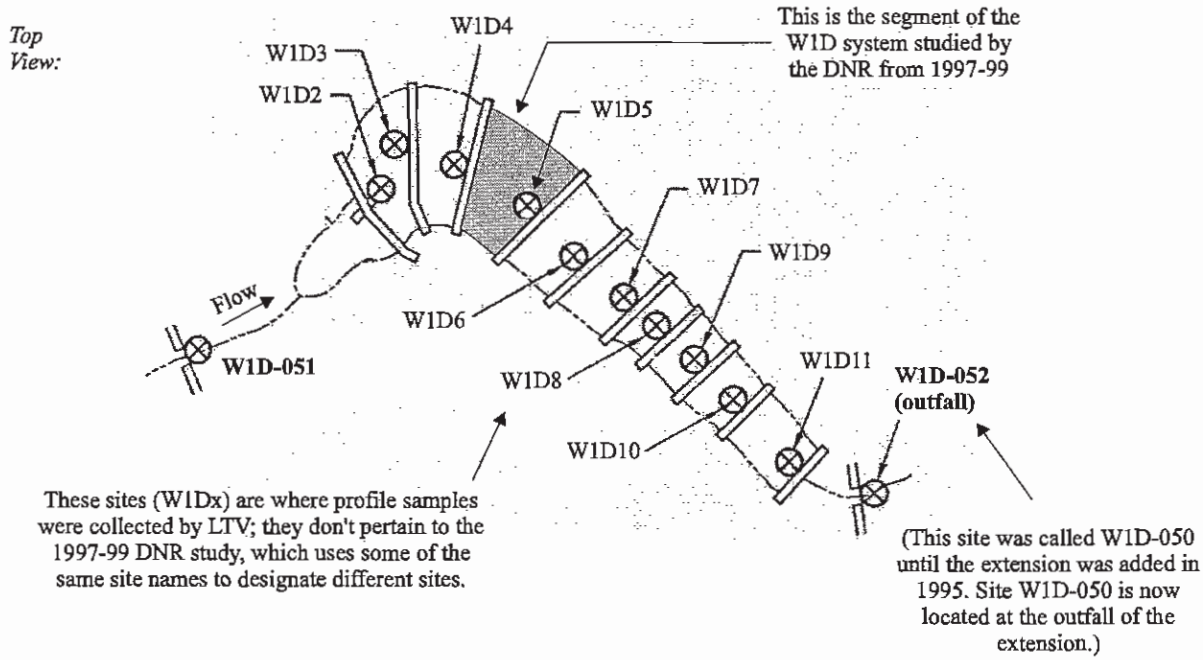
This system was built in an existing wetland. Construction began in March of 1992. The wetland systems were designed by STS Consultants, Ltd., and built by LTV Steel Mining Company (Frostman, 1992). In the summer of 1992 a pretreatment system was constructed at the base of the 8018 stockpile (Figure 2, Appendix 14).

The area was originally a combination of emergent (wet meadow) and scrub-shrub type wetlands, and the majority of the woody vegetation, which consisted primarily of alder (*alnus sp.*), was removed from the site. The basic design for the system included the construction of a series of soil berms, which were built to control water levels and to maximize contact between the drainage and the substrate (Figure 3). Soil berms were built with glacial till (sandy silt) available from a surface overburden stockpile on the property. After the berms were constructed, a one-foot (30 cm) layer of a mixture of local peat and peat screenings was applied to the entire area except the top of the berm. The screenings were a waste material generated during the processing of horticultural peat and consisted mostly of wood fragments and long peat fibers (Appendix 4). This material was selected to increase the permeability of the peat to at least 10^{-3} cm/sec and to provide available organic carbon. In the spring of 1992, the berms were hand-seeded with Japanese Millet, while the open water areas were seeded with cattails. To obtain the cattail seeds, cattail heads were placed in a container of water with a small amount of liquid soap and several large metal bolts. The mixture was agitated until the heads broke and the seeds were dispersed. The slurry was then broadcast by hand over the wetland.

The majority of the flow to this system originates from the base of the 8018 stockpile, although additional seepage from the 8031 stockpile also drains to this area (Figure 4). V-notch weirs were installed to provide continuous measurement of the input and output flows. From 1986 to 1994, May to October flows ranged from 20-40 gal/min (75-150 L/min), with peak flows exceeding 200 gal/min (750 L/min). Flows decreased after the 8018 stockpile was capped with a 30 mil LLDPE (linear low density polyethylene) membrane in 1995. May to October flows from 1996-1999 have ranged from 8-22 gal/min (30-83 L/min). Water quality samples of the inflow and outflow were collected twice per month during the period of flow (generally late March - December). From 1992 through 1994, samples were also collected from within the system about once per month (Figure 3). From 1992-94, the input to the wetland had an average pH of 7.07 and contained 3.98 mg/L nickel, 0.068 mg/L copper, 0.052 mg/L zinc and 0.036 mg/L cobalt (Table 1). Nickel concentrations decreased by over 50% in 1995 to 1.78 mg/L, and the average input concentration for 1996 to 1999 was 0.74 mg/L (Table 1).

The original W1D treatment system, constructed in the spring of 1992, covered 1.7 acres (7000 m^2) and contained a series of 9 berms. From 1992 through 1994, changes were made to the system to disperse flow, minimize channeling and improve contact between the drainage and the substrate (Appendix 3). In 1995, the system was expanded by 2.5 acres (10,000 m^2). The extension included an alternating series of overflow and underflow berms (Eger et al.,1996). Prior to construction of these berms, the original organic soils were removed; the berms were then built on the mineral soil base and compacted to minimize any future settling. This report focues on the original W1D system, and excludes the W1D extension.

Figure 3. Top view of the original W1D wetland treatment system, and cross-section (as designed and as built) of typical berms in this system.



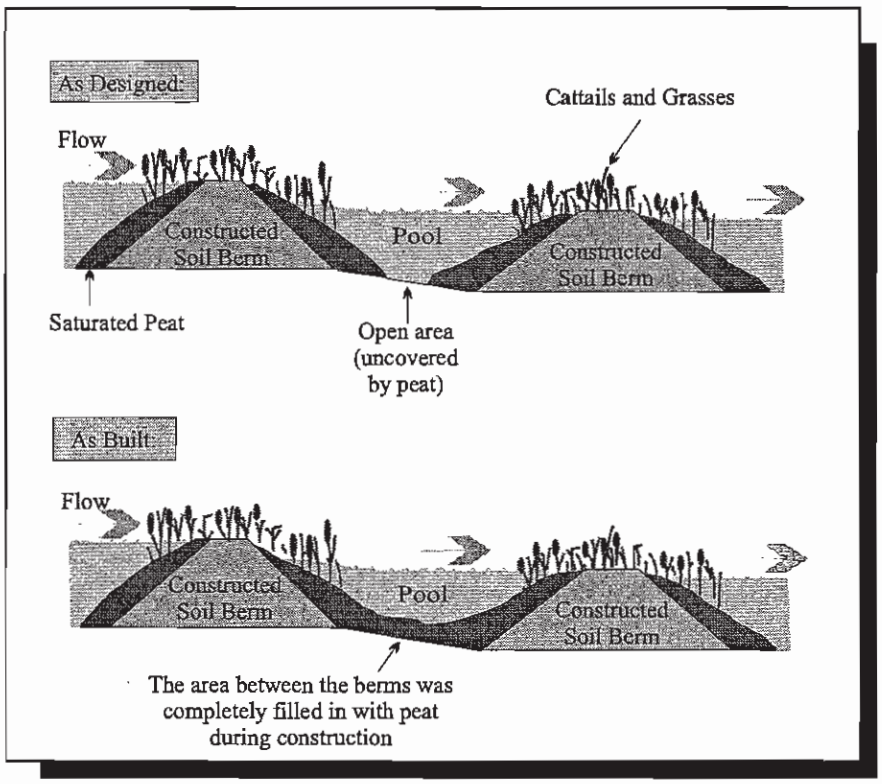


Table 1. Mean water quality data, W1D treatment system, 1992-1999.

Year	pH		SO_4		Cu		Ni		Co		Zn		Ca		Mg	
	In	Out	In	Out	In	Out	In	Out	In	Out	In	Out	In	Out	In	Out
1992	7.08	7.03	2480	1840	0.093	0.012	4.28	0.315	0.016	0.016	0.054	0.013	300	190	380	230
1993	7.11	7.14	1660	1290	0.051	0.009	3.14	0.354	0.043	0.004	0.041	0.009	220	190	220	220
1994	7.02	7.36	2120	1510	0.059	0.007	4.53	0.426	0.049	0.004	0.061	0.008	310	210	310	260
Average 1992-94	7.07	7.18	2087	1547	0.068	0.009	3.98	0.365	0.036	0.008	0.052	0.013	277	197	303	237
1995	7.05	7.30	1400	1100	0.105	0.003	1.78	0.392	0.049	0.005	0.022	0.007	240	180	240	200
1996	7.18	7.33	1600	1320	0.043	0.009	1.17	0.337	0.009	0.002	0.030	0.003	260	200	260	200
1997	7.17	7.32	1000	790	0.030	0.001	0.69	0.178	0.012	0.001	0.017	0.007	200	150	200	140
1998	7.74	7.29	970	820	0.031	0.002	0.49	0.121	0.010	0.001	0.023	0.006	200	150	200	150
1999	7.11	7.29	1140	770	0.015	0.001	0.61	0.140	0.006	0.001	0.015	0.007	210	140	210	140
Average 1996-99	7.30	7.48	1180	925	0.030	0.003	0.74	0.194	0.009	0.001	0.021	0.006	220	160	220	170

Notes: pH values are in standard units, all other concentration values are mg/L, and all parameters are total (i.e. unfiltered) values. The site called "in" is W1D-051, the historical sampling point for this drainage, and the site called "out" is the outflow from the original W1D system, which was originally called W1D-050, but which was renamed W1D-052 in 1995 after the W1D extension was put on-line. (Site W1D-050 is now at the outfall of the extension area.) Stockpile 8018 was capped in 1995, therefore 1995 isn't included in the 'post-closure' period (1996-1999).

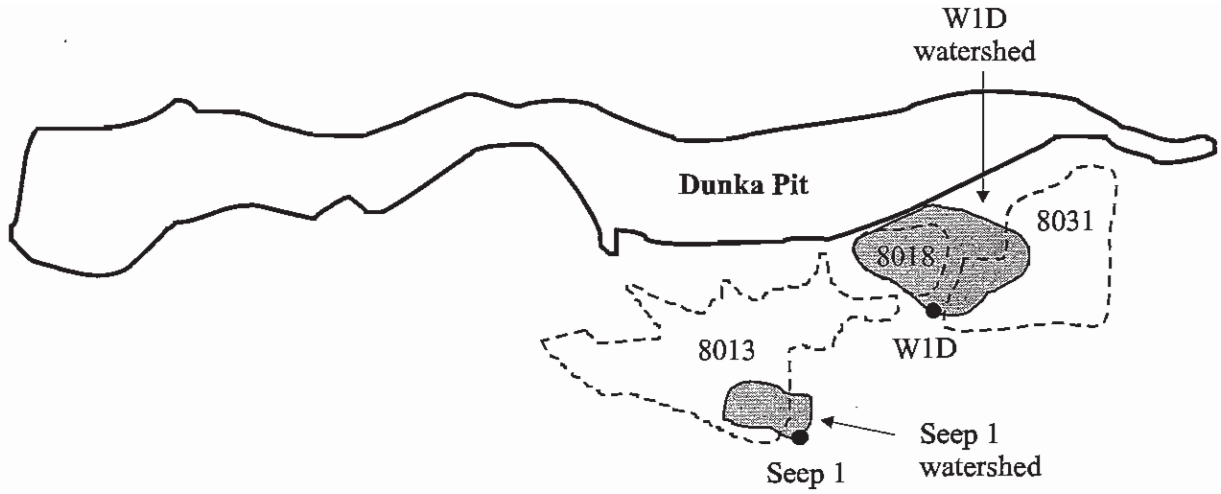


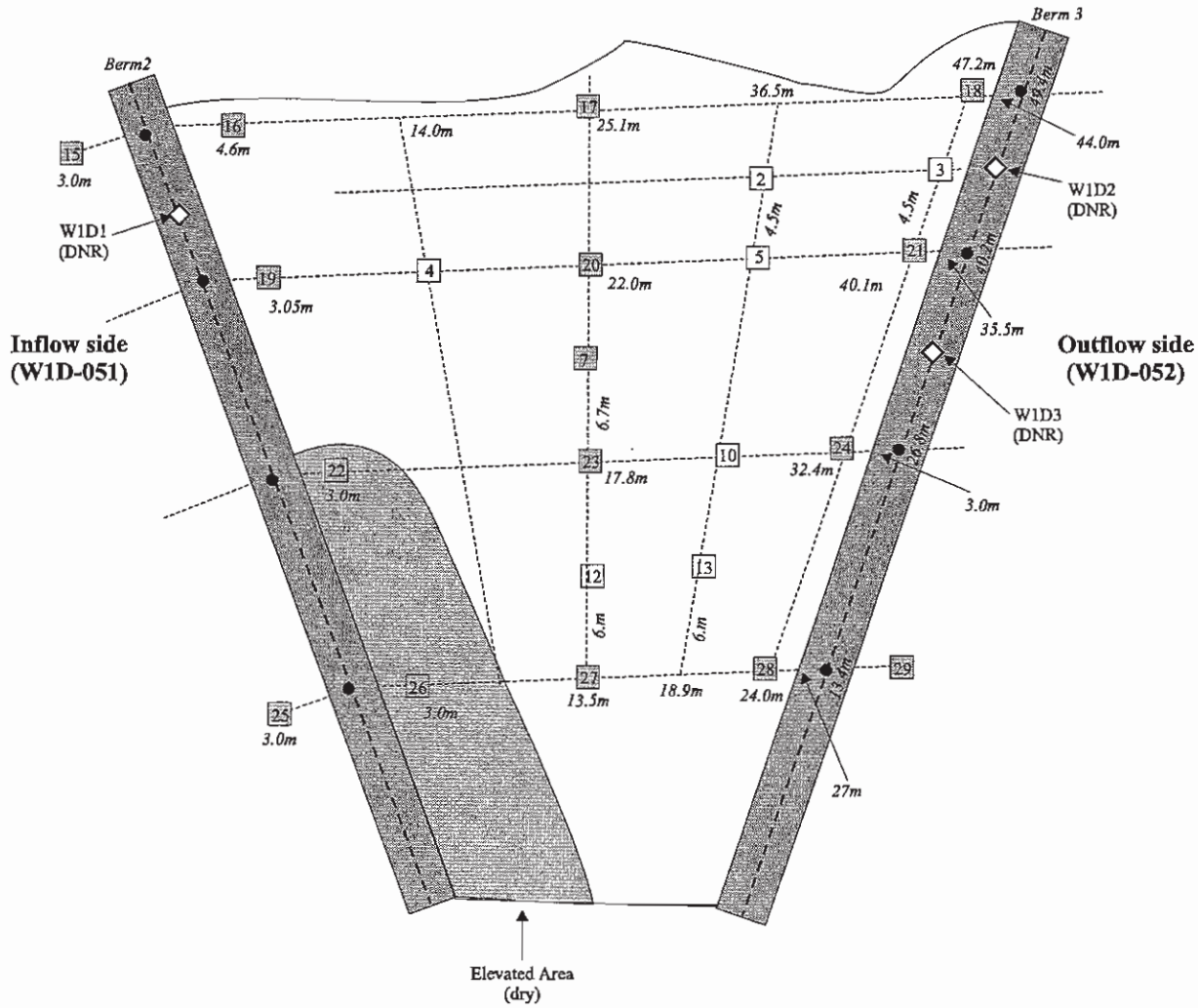
Figure 4. Watersheds and stockpile contributions to the W1D and Seep 1 seeps.

Study cell, W1D

Based on the 1992 to 1994 data collected from within the wetland, an individual cell within the original W1D system was selected for additional study (Figures 3, 5). This cell (the study cell) was chosen because it had the largest drop in nickel concentration. Based on the 1992 to1994 profile data, nickel concentrations decreased by an average of 26% as water moved through the cell (Figure 6). Since the watershed contribution to the cell was small, and since there was no evidence to support a substantial input of groundwater to the cell, there should not be a significant difference between input and output flow. As a result, mass removal was directly related to the change in concentration.

Since over 99% of the metal removal in these types of systems occurs in the substrate (Eger et al.,1994), the goal in this type of constructed wetland is to disperse flow uniformly across the wetland. Although this was the goal, it was difficult to achieve. Essentially all the flow enters the study cell at one point and leaves at two sites (Figure 5). Water depth ranged from near 0 at the berm to about 8 inches (20 cm) in the center of the cell.

The total cell area is about 16,000 ft^2 (1500 m^2), but about 2000 ft^2 (185 m^2) in the northeast corner is higher than the rest of the cell and is dry for most of the year. Vegetation in the treatment system consists primarily of cattails (*Typha sp.*), which occupy most of the study cell except for an area of open water in the center.



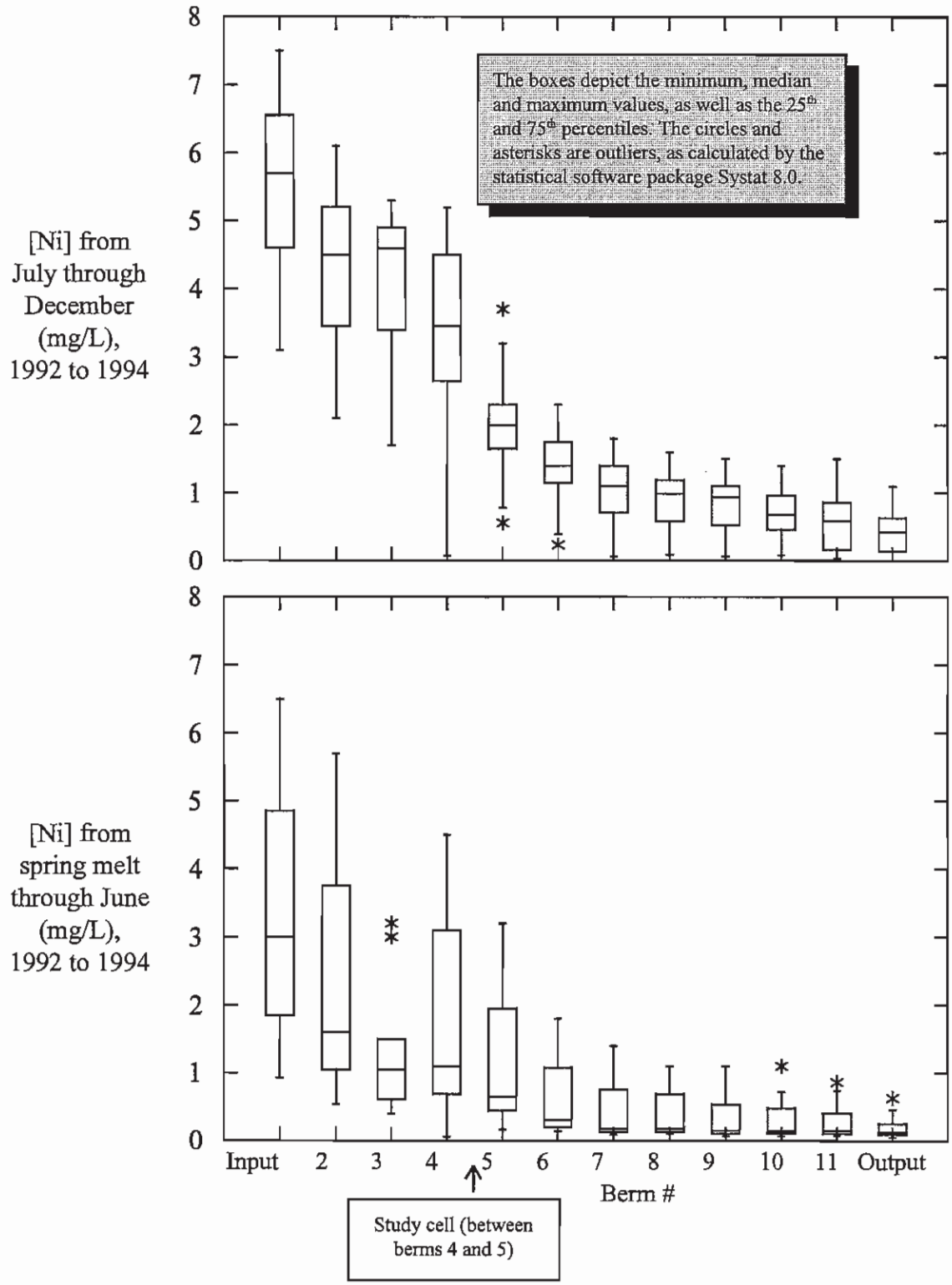
Notes:

1) The sites called W1D1, W1D2 and W1D3 are DNR site names related to the input and output of the study cell. They should not be confused with the LTV profile sites of the same name (see Figure 3 for the LTV profile site locations).

2) The grey squares indicate sites where peat cores were colleeted in 1996, and the white squares indicate 1997 peat sites. In both cases, the number inside the box indicates the site location number.

3) The distance values depicted on the horizontal dotted lines are the distances from the upstream berm, as measured along the particular transect. The values on the vertical dotted lines are the distance from the site to the appropriate horizontal transect.

Figure 5. Locations of monitoring sites (peat, water quality) in the W1D study cell.



Two plots are depicted, because input nickel concentrations typically increased in July. The bottom plot includes all data in the spring through June, while the top plot includes all data from July through December.

Figure 6. Nickel concentrations within the W1D wetland treatment system, 1992-1994.

Seep 1 pretreatment system

In 1992, LTV built a passive treatment system at Seep 1 which combined peat and limestone in an attempt to increase pH and remove metals. The Seep 1 monitoring site originally collected diffuse seepage that originated at the toe of waste rock stockpile 8013. This stockpile was constructed from 1967-1991 and contains material with an estimated composition of 0.06% copper (as CuO), 0.02 % nickel (as NiO) and 0.24% sulfur (Appendix 3). The Seep 1 watershed was estimated from the original contours in the area to be 10.6 acres (4.3 hectares), with 76% of the wateshed covered by stockpile (Figure 4, Appendix 3). Bedrock in the area is within 6.6 feet (2 meters) of the surface and, as a result, groundwater in the area ranges from 0 to about 3.3 feet (one meter) below surface. Average annual flow is on the order of 5-10 gal/min (19-38 L/min), with maximum daily peak flows on the order of 50 gal/min (200 L/min). Flow typically begins in April and continues until November, when the seep freezes. Peak flows are typically observed after heavy rainfall events that occurr during the summer; instantaneous peak flows of around 100 gal/min (400 L/min) have been recorded.

Monitoring of the drainage began in 1976. The pH was neutral (7.0-7.5), nickel concentrations were 1-2 mg/L and copper was less than 0.1 mg/L. The pH began to decline slowly, and by 1984 had stabilized in the range of 5.0-5.5. Concurrent with the pH decrease was an increase in metal concentrations; nickel concentrations increased by about an order of magnitude to 10-20 mg/L, while copper increased to 0.5-1.5 mg/L (Figure 7.) Water quality did not change substantially from 1985-1991.

Laboratory and small-scale field studies had shown that limestone was successful in increasing the pH of the drainage and reducing copper concentrations by 50-80% (Lapakko and Antonson, 1990a, 1990b). Removal of other trace metals was much lower, ranging from 10% for nickel to around 25% for cobalt. In other experiments peat had removed over 80% of both copper and nickel from drainage at this mine (Eger and Lapakko, 1988). LTV combined the limestone and the peat in a treatment system in an attempt to increase pH and remove metals. The system was built between the toe of the stockpile and the weir monitoring station. The original design was developed by STS Consultants, and modified based on suggestions made by LTV and Minnesota regulatory agencies.

Unfortunately, the size of the treatment system was constrained by the topography of the site. As a result, the overall dimensions were determined by available space, rather than a specific design size. The area was cleared of vegetation and a 10,000 ft^2 (930 m^2) pond, roughly 3 feet (0.9 m) deep, was excavated. Limestone, 3½ inch to 4 inch (7.5 to 10 cm) in diameter was placed around the perimeter of the pond, and two limestone berms were built within the pond. Each berm was about 3 feet (0.9 m) high and the berms ranged in length from about 130 to 200 feet (40 to 60 meters). Once the berms were in place, a 1-2 foot (0.3-0.6 meter) layer of a 1:2 mixture of peat and peat screenings (a waste product from the production of horticultural peat) was placed between the berms. At the outlet end of the pond a limestone under-drain was constructed and covered with a one foot (0.3 m) layer of the peat and screenings mixture. An overflow pipe was installed to handle excess flow. The design of the system is shown schematically in Figure 8.

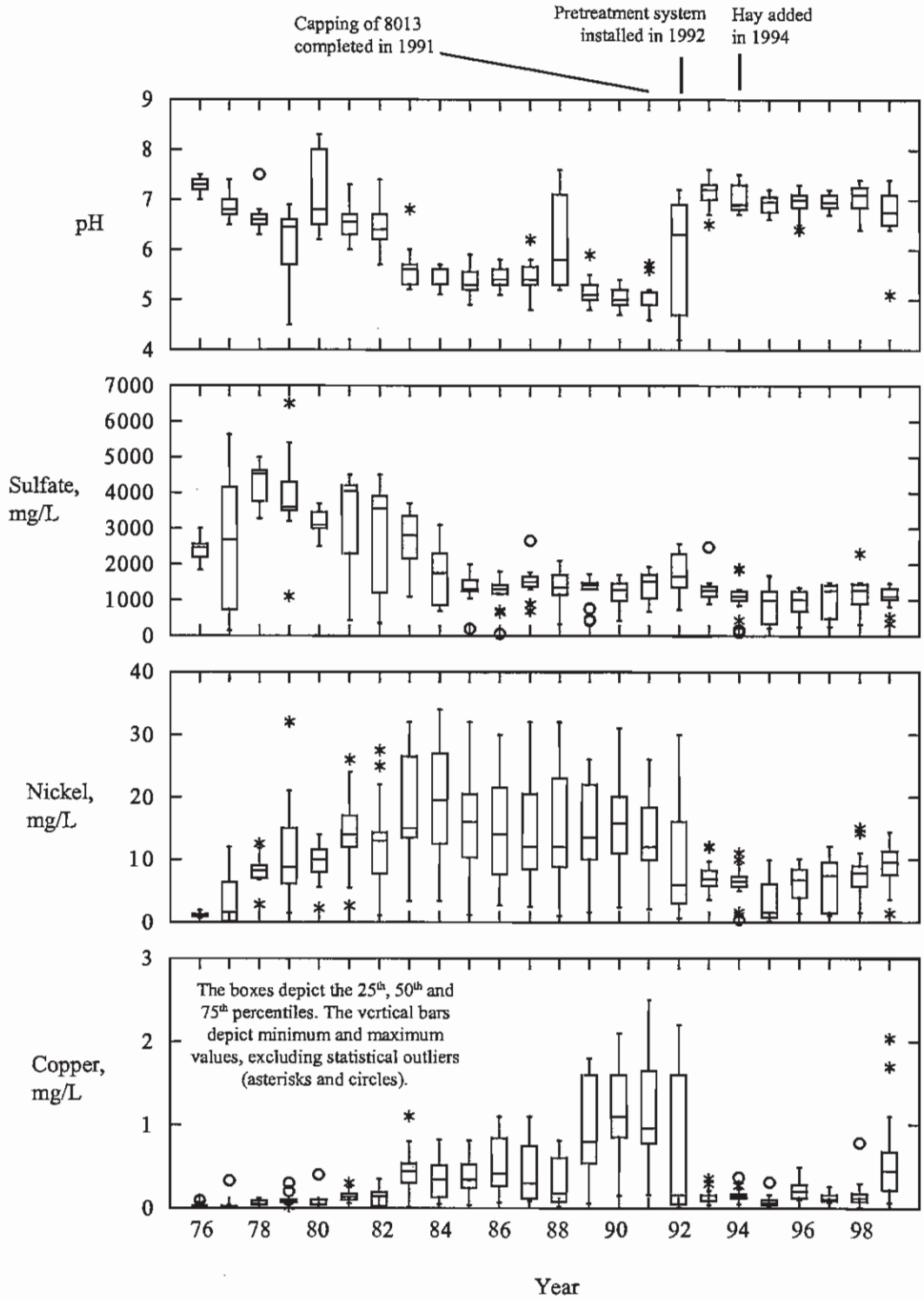


Figure 7. Box plots of pH, sulfate, nickel and copper concentrations vs. time at Seep 1 (site 043), LTV data, 1976-1999.

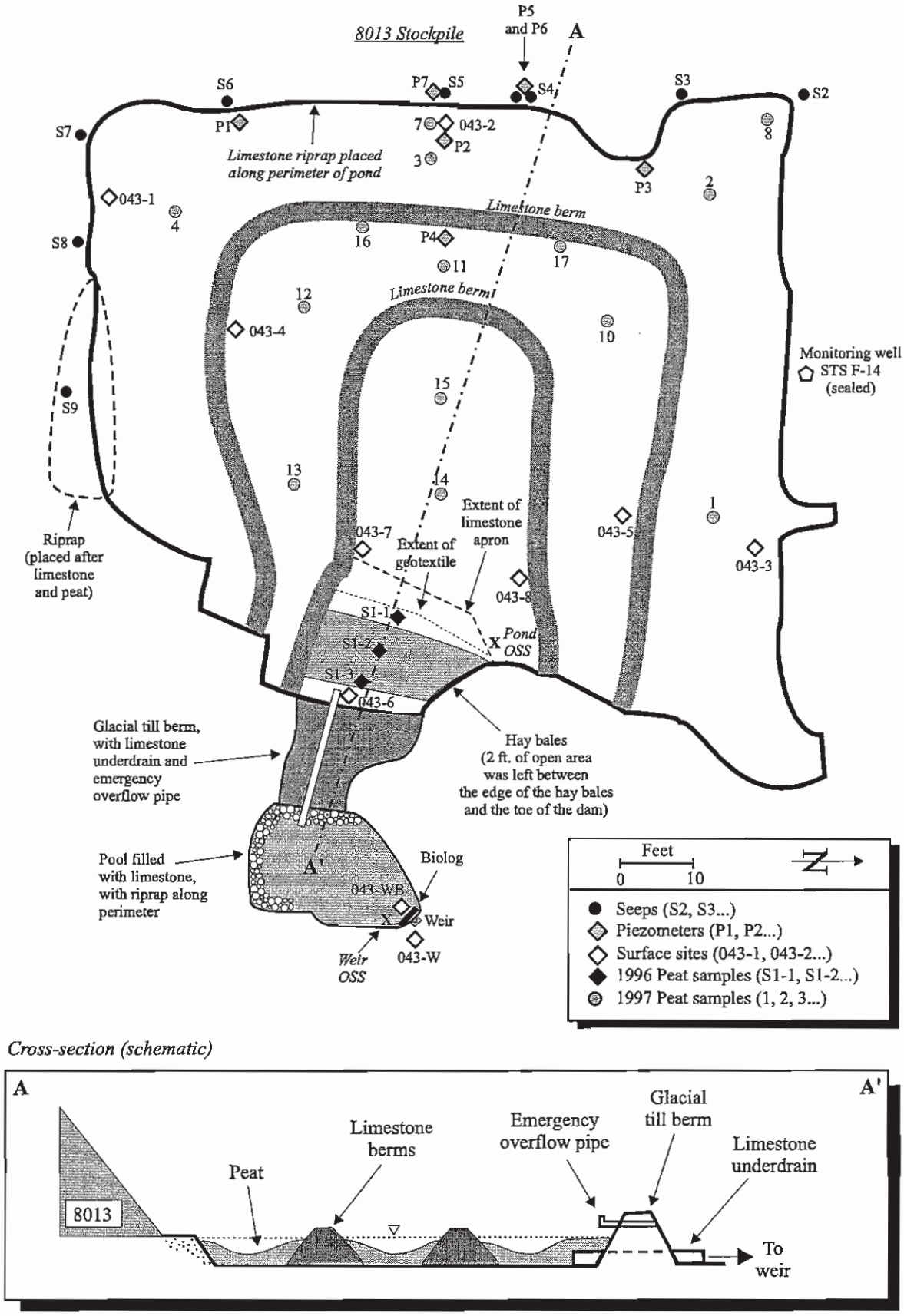


Figure 8. Seep 1 pretreatment system, with locations of monitoring sites.

In 1994, hay was added to the system in an attempt to stimulate sulfate reduction by providing a readily available food source for the sulfate reducing bacteria. In June, three bales weighing about 130 lbs (60 kg) were broken and dispersed in the vertical down-flow section of the system. In August, about 40 intact bales were placed side by side on top of the down-flow section.

Methods

W1D

Water samples of the inflow and outflow of the wetland treatment system are collected twice a month by LTV personnel. From 1992 to 1994, LTV also collected samples at each berm in the wetland (Figure 3). In 1997, input and output samples to the study cell were collected twice per month. In 1998, the plan was to monitor the study cell when nickel concentrations were elevated. Nickel concentrations at site W1D were estimated with a field colorimetric kit. If the nickel concentration exceeded 1 mg/L, then samples would be collected in the study cell. Nickel concentrations stayed low throughout the spring and early summer of 1997 (Appendix 1), so no samples were collected from the study cell. In order to provide some field data in 1998, sample collection in the study cell resumed in August. Monthly samples were collected in 1999. Samples were analyzed for pH, specific conductance, copper, nickel, calcium, magnesium, and sulfate. Water samples were brought back to the DNR laboratory, refrigerated, and analyzed the following day; pH and specific conductance were analyzed directly in the sample bottle. A portion of the remaining sample was poured directly into a 60 mL polyethylene bottle for total metals and the remainder was filtered through a 0.45 micron filter for sulfate. Metal samples were acidified with 0.2 mL of Baker Instra-Analyzed nitric acid per 50 mL of sample.

An Orion SA 720 pH meter equipped with a Ross combination pH electrode (model 8165) was used for pH analysis, and a Myron L model (model EP) conductivity meter was used to determine specific conductance. Sulfate was analyzed at the Minnesota Department of Agriculture (MDA) laboratory. Prior to October 11, 1998, sulfate was analyzed with the Ion Chromatographic Method (Wastewater Method 4500-SO_4B) with a Dionex DX300 IC. Subsequently, these parameters were measured with a Lachat QuickChem 8000 using the same methods. Metals samples were analyzed at MDA using a Varian 400 SPECTRAA atomic absorption spectrophotometer in the flame mode or a Zeeman GFAA graphite furnace. This was replaced by ICP/MS (Hewlett Packard HP4500 Series, model #G1820A) on August 22, 1999.

Continuous measurements of inflow and outflow water levels were made with a Steven's Model F recorder, and the flow was calculated from the standard equation for a 60 degree V-notch weir. Due to potential problems with the recording equipment under freezing conditions, continuous flow estimates were generally only available from May through October (Appendix 5).

Peat samples were collected in April of 1996 and 1997 (Figure 5). If the surface of the peat was frozen, samples were collected with a specially designed core tube sampler (Appendix 4). A

cylindrical cutting head (3 3/4 inches ID) was fabricated and mounted on the shaft of a power soil auger. With this sampler, samples could be collected while the substrate was frozen, which made identifying and separating layers much easier than with a standard coring device. Deeper samples were collected with a standard Macauley peat sampler (Appendix 4).

Samples were generally divided into 10 cm sections: 0-10, 10-20, and 20-30 cm. Samples were dried at 105^0C for 24 hours. The samples were then processed in a blender, sieved to -80 mesh, and the -80 mesh material was totally digested. A mixture of 5 ml of water, 10 ml of concentrated nitric acid and 2 ml of concentrated hydrochloric acid was added to each sample. The samples were sequentially microwaved for 10 minutes each at 40, 80, 120 and 160 psi. The digested samples were analyzed for trace metals by the MDA laboratory. Additional information on sample preparation and anlysis is presented in Appendix 4.

Seep 1

In general, water quality samples of the outflow were collected twice per month by LTV and analyzed for pH, specific conductance copper, nickel and sulfate. During the summer of 1993 and 1994, a series of water quality profile samples were collected by LTV from within the pretreatment pond. In 1993, DNR collected samples of the discrete input seeps at the toe of the stockpile (Figure 8).

In 1997, when the intensive study of the pretreatment system began, the sampling points within the pond and at the small input seeps were re-established (Figure 8). Additional samples were collected from within the pond to examine flow paths (Appendix 1). In 1998 and 1999, the sampling program focused on the performance of the vertical down-flow section of the bed (sites 043-6 and 043-WB; Figure 8). Site 043-WB was located about 10 feet (3 meters) upstream of the weir and immediately upstream of a "biolog" LTV had installed in 1996. The biolog contained an organic substrate designed to remove metals from solution. According to LTV, the log plugged and did not remove significant quantities of metals, but became too heavy to remove (Aagnes, personal communication, 2000).

In 1998, nine piezometers were installed to examine vertical groundwater gradients and to collect data on groundwater quality. The piezometers were constructed of ½ inch (1.27 cm) schedule 80 PVC pipe. A metal bolt was inserted in the bottom of the pipe to prevent soil from plugging the pipe while the piezometers were installed. When the piezometers had been hand driven to the desired length, a metal rod was inserted into the pipe, the metal bolt was pushed out and the pipe was opened (Appendix 8). Limited water level and water quality data were obtained in the fall of 1998 (Appendix 1).

Substrate samples from the vertical down-flow section were collected in April 1996 using either the core tube sampler or the Macauley sampler. A Macauley sampler was used to collect samples from the bottom of the pond in the pretreatment system in April 1997. Samples taken with the Macauley sampler were collected within an 8 inch (0.2 meter) circle and combined to form a composite of two

or three cores. The core tube samples were analyzed individually. The samples were dried, crushed with a mortar and pestle, sieved to -80 mesh, digested and sent to MDA for analyses.

Results, W1D overland flow wetland

Flow

Input flow was measured at the v-notch weir located about 200 feet (60 meters) from the toe of the 8018 stockpile and about 200 feet upstream of the beginning of the wetland (site W1D-051; Figure 3). Additional watershed area contributed flow to the wetland treatment system, but the majority of the nickel load originated at site W1D (Figure 4). Since continuous flow measurements were only available for May through October, the average daily flow calculated over this period has been used to compare the change in flow over time.

Average input flows ranged from 29-36 gal/min (110-136 L/min) for 1992 to 1994. Flows decreased in 1995 and remained low after the top of the 8018 stockpile was capped. Average flows for 1996 to 1999 ranged from 8 to 22 gal/min (30-83 L/min). Flow in 1999 was the highest of the post-closure flows. Over the July 4th weekend about 7 inches (17.5 cm) of rain fell at the site. A peak flow on the order of 900 gal/min (3400 L/min) was recorded, and the estimated daily flow was 360 gal/min (1360 L/min). Annual precipitation for 1999 was 35.17 in. (89.3 cm), substantially above the long-term average precipitation of 28.49 inches (72.4 cm; Table 2).

Output flows were generally greater than input flow except during hot dry periods when evapotranspiration losses were large. During the summer of 1998 the output flow was 5 to 6 gal/min (19-23 liter/min) less than the input. In July when the input flow decreased to 4-5 gal/min there was no flow at the outlet (Appendix 5). Flow into and out of the study cell was not measured directly. Since there was little contributing watershed between the beginning and end of the wetland, output flow from the entire system (site W1D-052; Figure 3) was used to represent flow through the cell.

Water quality

There was little variation in pH in the wetland (Figure 9). Both input and output pH ranged from 6.8 to 7.6. There was little difference in pH between the input and output of the study cell; the average value for both sites was around 7.2 (Table 3).

From 1992 to 1994, the input nickel concentration to the wetland treatment system typically was on the order of 1 mg/L in the spring, then increased to approximately 6 mg/L in early summer. Concentrations then remained relatively constant until the seep froze in late fall (Figure 9). The average input nickel concentration to the study cell during this period was 2.9 mg/L. As water flowed through the cell, nickel concentrations decreased by about 26%, to about 1.8 mg/L.

Table 2. Flow and precipitation data W1D treatment system, 1992-1999.

Year	Inflow - Average Daily Flow (gal/min)		Outflow - Average Daily Flow (gal/min)		Precipitation (in)	
	May-Oct	Annual Estimate	May-Oct	Annual Estimate	May-Oct	Annual
1992	33	27	44	35	17.3	26.9
1993	29	24	39	31	17.4	29.7
1994	36	31	46	38	19.2	28.7
Average 1992-1994	33	27	43	35	18.0	28.4
1995[1]	27	24	51	43	18.5	25.6
1996	22	28	23	29	18.2	34.2
1997	8	9	8	13	15.5	22.9
1998	8	8	5	4	19.2	31.1
1999	22	21	30	24	28.8	35.2
Avcrage 1996-1999	15	16	16	18	20.4	30.9

[1] The stockpile was capped in 1995.

$$\text{Avcrage Daily Flow (gal/min)} = \frac{\text{total volume (L)}}{\text{\# days x 1440 min/day x 3.785 L/gal}}$$

Notes: The number of days used for the annual estimate was 245: April 1 through November 30. Annual average precipitation for Babbitt, from 1961-1990, was 28.49 inches. Annual May-October precipitation for Babbitt (1961-1990) was 21.16 inches (data from Minnesota Climatology Working Group; www.climate.umn.edu).

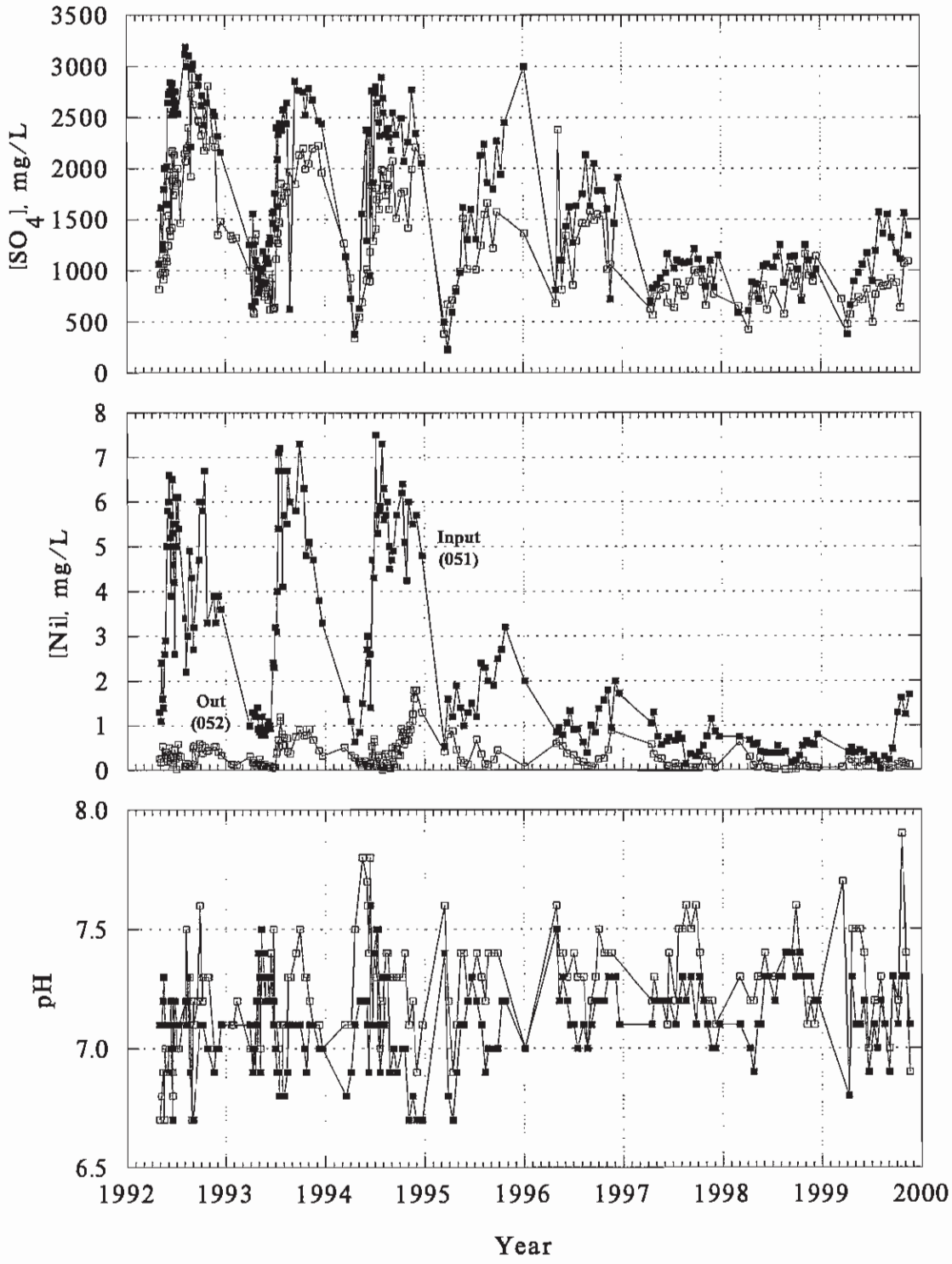


Figure 9. pH, nickel and sulfate concentrations (input and output) vs. time for the original W1D wetland treatment system.

Table 3. Mean water quality data, W1D study cell, 1997-99.

Year	n	pH		Ni		Cu		Co		Zn		Ca		Mg		SO_4	
		In	Out	In	Out	In	Out	In	Out	In	Out	In	Out	In	Out	In	Out
1997	13	7.02	7.19	0.346	0.240	0.013	0.011	0.003	0.001	0.033	0.030	180	170	160	160	860	810
1998	3	7.47	7.49	0.216	0.277	0.021	0.017	0.042	0.036	0.028	0.035	220	220	230	220	1040	910
1999	7	7.25	7.28	0.374	0.297	0.013	0.010	0.008	0.006	0.054	0.031	210	200	200	200	640	650
Ave. 1997-99	---	7.16	7.24	0.324	0.259	0.015	0.012	0.017	0.010	0.035	0.032	200	180	190	180	880	810

Notes: Nickel concentration in the W1D study cell decreased by 20–30%, a value consistent with the value measured in 1992-1994. The 1998 data is anomalous, and is likely due to the limited number of samples collected and the very low flow conditions that existed in 1998.

In 1995, input nickel concentrations decreased substantially. Input concentrations were low in the spring but only increased to 2-3 mg/L in the summer. Maximum concentrations gradually decreased to less than 1 mg/L by 1999. As a result of the decrease in nickel concentration, the nickel concentration entering the study cell also decreased. Input concentrations to the cell dropped to 0.32 mg/L in 1997 to 1999. Concentrations in the outflow were lower than the inflow and averaged 0.26 mg/L for 1997 to 1999 (Table 3, Figure 10).

Substrate samples

The initial trace metal content of the mixture of peat and peat screenings that was added to the wetland was very low; all concentrations were less than 10 mg/kg (Appendix 4). By 1996, the peat substrate had accumulated measurable amounts of copper, nickel, cobalt and zinc. Nickel concentrations were the highest and varied with depth and location within the cell. The maximum metal concentrations were generally in the top 10 cm of the core, and nickel concentrations varied from 68 mg/kg to 14,600 mg/kg (Figures 11, 12, 13). The average nickel concentration for the 0-10 cm segment in the cell was 4959 mg/kg, and decreased to 2110 mg/kg for the 20-30 cm segment. Copper, cobalt, and zinc generally followed the same pattern as nickel but concentrations were about an order of magnitude lower. The maximum copper, cobalt, and zinc concentrations were 350 to 420 mg/kg (Appendix 4).

Mass removal

Overall mass into and out of the wetland was calculated by multiplying the average concentration for the month by the average daily flow for that month (Appendix 7). Daily flow data were generally available from May through October, but for April, November and December, there were only a few individual flow readings. Flow data was collected by LTV as part of their NPDES permit (Appendix 5). Since both flow and precipitation in November and December tended to be low, the average of the limited individual measurements was assumed to be a reasonable estimate of flow. An average value may not provide a reaonable estimate of spring melt flow, since the volume and timing of flow depends on the amount of moisture in the snow pack, temperature and rainfall. However, metal concentrations during April were about one-half the summer values, so the total mass input during April even with higher flows would tend to be lower than summer months (Appendix 7). From 1992 through 1995, the May to October input mass accounted for 86% of the annual load. Although this percentage increased from 1996 to 1999, the total input load decreased dramatically during this time (Table A7.3).

The total mass removed by the wetland was the difference between the input and the output masses. Overall mass removal in the wetland ranged from 171 kg in 1994 to 3 kg in 1997, and corresponded to a percent removal that ranged from 38 to 91 percent (Table 4). Lower mass removal occurred in 1995 to 1999 due to the much lower nickel input to the wetland.

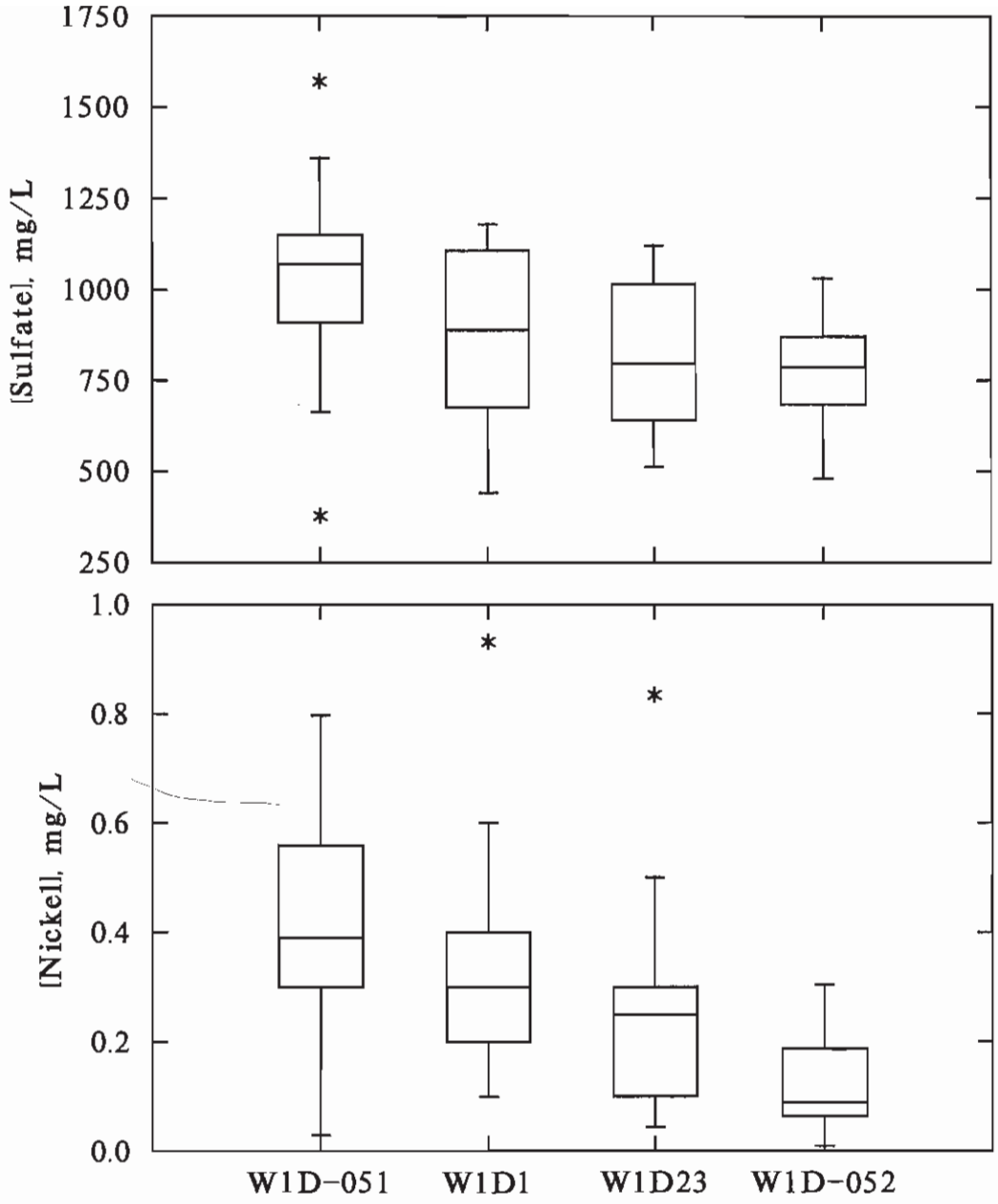
Table 4. Nickel mass removal, W1D wetland and study cell, 1992-1999.

	Entire wetland					Study cell		
	Mass into wetland (kg)	Mass out of wetland (kg)	Annual mass removal (kg)	Mass removal (%)	Cum. mass removal (kg)	Annual mass removal (kg)	Cum. mass removal (kg)	Calculation method
1992	158	16	142	90	142	37	37	1
1993	162	21	141	87	283	37	74	1
1994	190	19	171	91	454	44	118	1
1995	76	26	50	66	504	13	131	1
1996	36	15	21	58	525	5	136	1
1997	8	5	3	38	528	1	137	2
1998	5	1	4	80	532	1	138	2
1999	16	5	11	69	543	3	141	2

Calculation methods

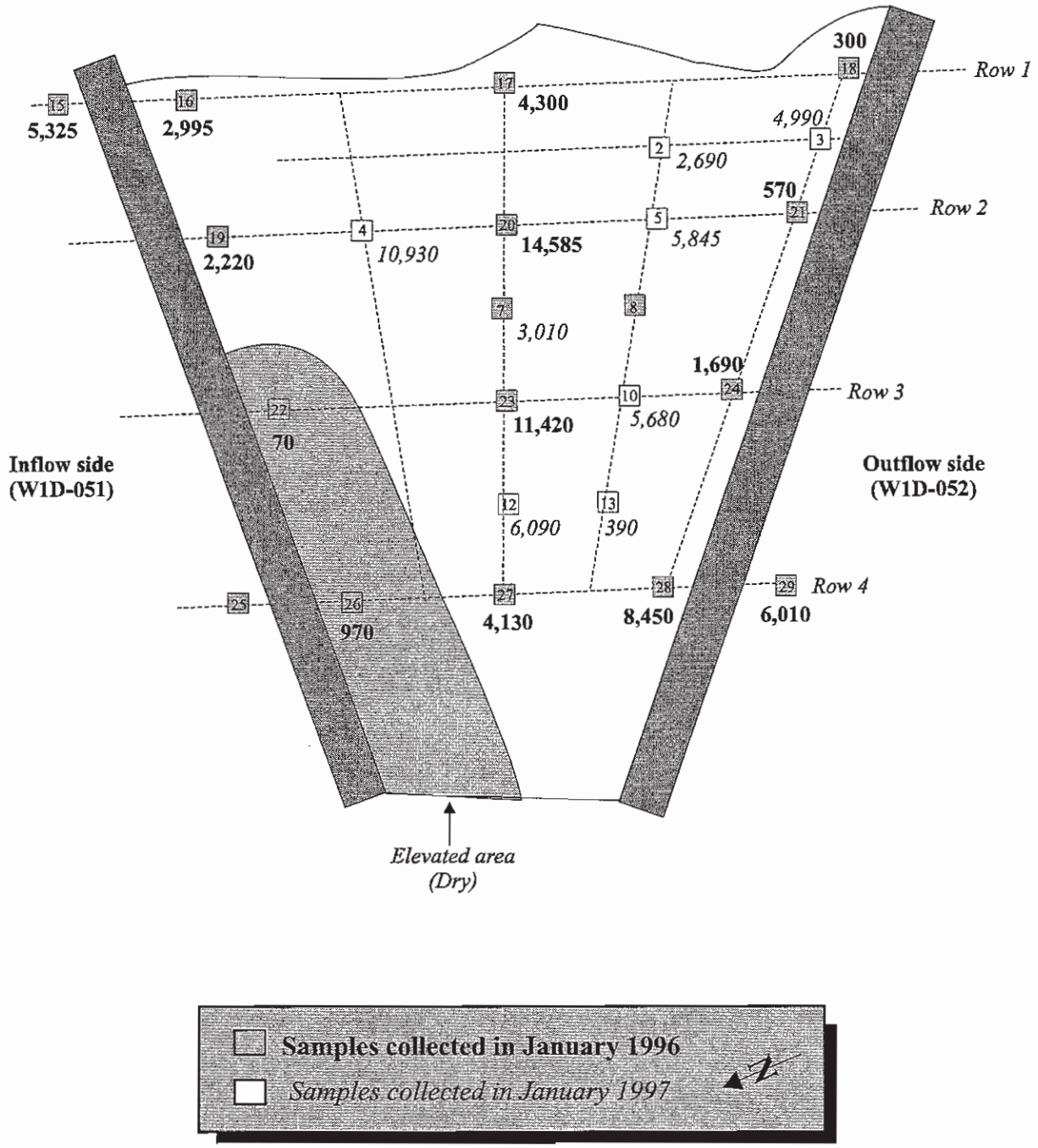
1) From 1992-94, "profile" samples were collected from surface water sites within the W1D wetland. The overall concentration decrease (i.e. from the input to the output) was determined for each day of these studies, and then the concentration decrease observed in the study cell was compared to the total value. It was determined that 26% of the overall removal occurred in this one cell, and the annual removal in the study cell (from 1992-96) was assumed to be 26% of the overall mass removal for each of those five years.

2) From 1997-99, DNR collected water quality samples at the input and output of the study cell. Flow was not measured since the input and output channels (sites W1D1, W1D2 and W1D3) were too shallow to use standard measuring equipment. In addition, diffuse flow probably occurred at other sites along the input and output berms. The water quality data could be combined with flow data from the outfall of the wetland (W1D-052) to arrive at mass estimates for the cell. (The outflow is considered to be a more accurate indicator of flow in the system than is the input, since significant water enters the system downstream of the input weir.) However, since the average decrease in nickel concentration in 1997 and 1999 (26%) was the same as in 1992-1994, calculation method 1 was used for all years.



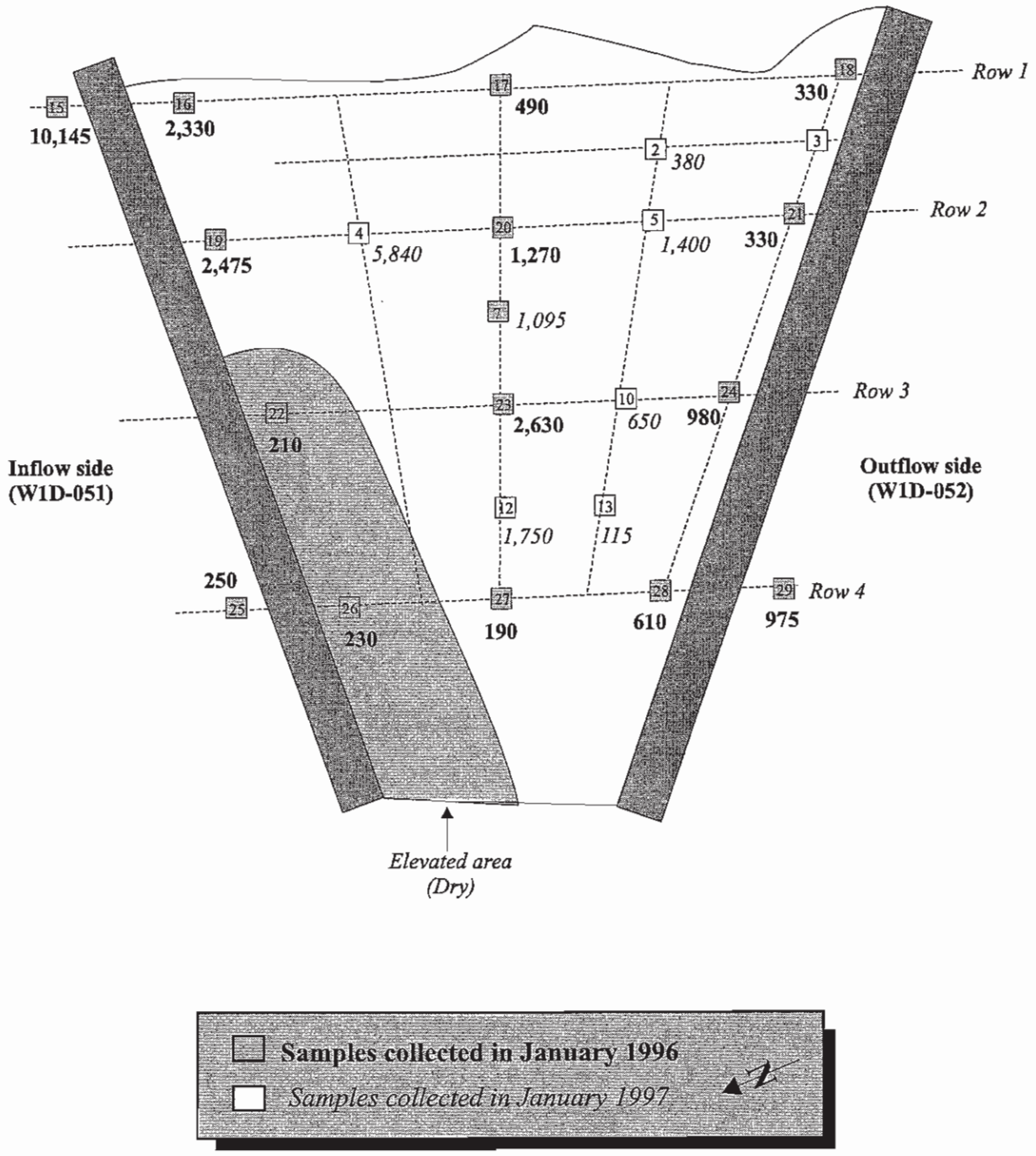
The site called W1D-051 is the input to the entire wetland, W1D1 is the input to the study cell that DNR studied from 1997-99, W1D23 is the average of two outflows from the study cell, and W1D-052 is the outfall from the original system (the extension added in 1995 is not included). The data sets for 051 and 052 have been truncated to correspond to the same time period used for W1D1 and W1D23. The boxes depict the minimum, median and maximum values, as well as the 25th and 75th percentiles; circles and asterisks indicate statistical outliers as calculated by the statistical software package Systat 8.0.

Figure 10. Box plots of nickel and sulfate concentrations in the W1D study cell, 1997-1999.



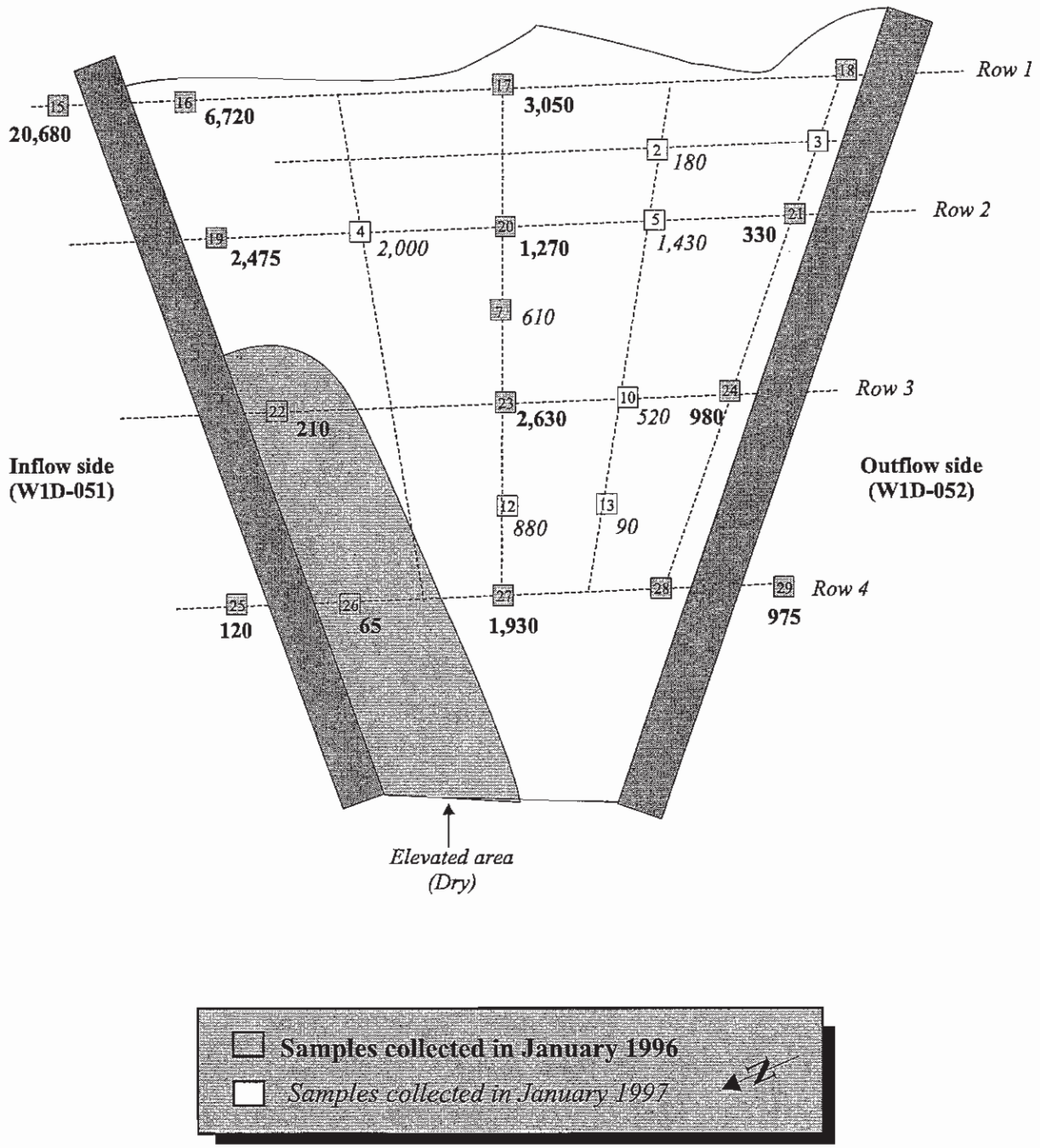
Notes: The number inside each square is the site location ID number.
Concentrations are mg/kg of dry peat.

Figure 11. Nickel concentrations in the 0-10 cm layer of peat in the W1D study cell of the W1D wetland treatment system.



Note: The number inside each square is the site location ID number.
Concentrations are mg/kg of dry peat.

Figure 12. Nickel concentrations in the 10-20 cm layer of peat in the W1D study cell of the W1D wetland treatment system.



Note: The number inside each square is the site location ID number.
Concentrations are mg/kg of dry peat.

Figure 13. Nickel concentrations in the 20-30 cm layer of peat in the W1D study cell of the W1D wetland treatment system.

W1D study cell

Mass removal in the study cell was calculated using both water quality and peat data. The mass removal in the study cell for 1992 to 1994 was estimated by using the change in concentration over the cell. Since the average change in nickel concentrations over the cell was 26%, it was assumed that 26% of the total annual mass removal occurred within the cell. Percent removal over the study cell in 1997 and 1999 also averaged 26% (Table 3). As a result, a constant percent removal of 26% was assumed for all years (Table 4). The overall nickel removal in the cell from 1992-1999 was 141 kg, with 84% of the removal occurring between 1992 and 1994. Between 1992 and 1996, when the peat samples were collected, 136 kg was removed.

Two methods were used to calculate the mass removal in the study cell from the metal concentrations in the peat. The first method used a computer model which contoured the nickel values in the peat and assigned concentrations to specific areas within the cell The second method made an overall estimate by multiplying the average nickel concentration by the mass of peat. Since the peat samples were collected in April of 1996 and 1997, the value calculated from the peat would represent the total mass removed through 1996. The total nickel mass in the substrate ranged from 112 kg for the computer model to 126 kg for the average calculation method (Table 5). Details of each calculation are provided in Appendix 7.

Results, Seep 1 pretreatment system

Flow

The entire top portion of the 8013 stockpile was capped in 1991 with a composite soil covering, which consisted of 12 inches (30 cm) of minus 3 inch (17.5 cm) buffer layer, 18 inches (45 cm) of minus ½ inch silty soil which was compacted to produce a laboratory permeability of at least 1×10^{-6} cm/sec, and 12 inches (30 cm) of soil cover (Gale, 1992). The area was fertilized, seeded and mulched in 1992, and within three years percent cover exceeded 90%.

Average May to October flow at the weir ranged from 3.5 to 10.6 gal/min (9.1 to 40.1 L/min) prior to stockpile capping and pretreatment system construction and from 2.4 to 7.9 gal/min (9-30 L/min) for the post-closure period (1992 to 1999; Table 6) Although small surface seeps were observed at the toe of the stockpile, the majority of the input enters as groundwater.

Data collected from the piezometers installed in the pretreatment system showed that water generally moves upward into the outer pool of the pretreatment and then laterally through the system (Appendix 8). At the end of the system, water flows vertically down through the substrate and limestone drain. The construction of the pretreatment system did not appear to affect the amount of groundwater that reports to the Seep 1 weir (Appendix 8).

Table 5. Mass removal estimates for the W1D study cell.

Calculation method	Peat layer	Nickel mass (kg)
(A) Combining input/output flow and nickel concentration data.	---	135.7
(B) Peat mass x mean nickel concentration of each layer	0 - 10 cm	65.0
	10 - 20 cm	33.7
	20 - 30 cm	27.7
	Total:	126.4
(C) Peat mass x concentration, as determined by computer model (Tech Base)	0 - 10 cm	71.5
	10 - 20 cm	27.1
	20 - 30 cm	13.9
	Total:	112.5
(D) Average (This includes methods B and C for the individual layers, and methods A, B and C for the total values)	0 - 10 cm	68.2
	10 - 20 cm	30.4
	20 - 30 cm	20.8
	Total:	124.9

Calculation methods

(A) Based on the results of profiles studies carried out within the W1D wetland from 1992 through 1994, it was determined that 25.8% of the overall removal within the system occurred in this single cell. The overall removal value for 1992-1996 (526 kg) was multiplied by 0.258 to yield 135.7 kg of nickel removed in the study segment.

(B) The average nickel concentration of each peat layer (0-10, 10-20 and 20-30 cm) was multiplied by the mass of dry peat assumed to be present in each 10-cm layer (13100 kg). This was based on an area of 1310 m^2 (14,098 ft^2), which was the value used in the Tech Base model (method 3), and on the assumption that the wet peat was 90% water. All available 1996 and 1997 peat data was used to calculate the means (4959 mg/kg in 0-10 cm layer, 2571 mg/kg in 10-20 cm layer, and 2112 mg/klg in 20-30 cm layer). These estimates exclude the dry peninsula area (i.e. removal is assumed to be zero in the peninsula). A few miscellaneous peat samples had unusual peat lengths (i.e. 0 - 8 cm instead of the usual 0-10 cm length), and these were grouped together with the standard core segments prior to calculating the means for each layer.

(C) A software program named Tech Base was used to calculate concentration contours within the W1D study segment, and then to calculate nickel mass in each layers based on these contours. These estimates also exclude the peninsula area (Appendix 7).

(D) Overall results from all three methods were used to calculate the average value of 124.9.

Table 6. Flow and precipitation, Seep 1, 1986-1999.

Year	May - October	
	Precipitation (in.)	Average Daily Flow (gal/min)
1986	20	6.6
1987	21.3	5.9
1988	24.5	9.9
1989	22.1	20.7
1990	16.6	3.5
1991	22.8	10.6
Average 1986-1991	21.2	7.3[a]
1992[b]	17.3	3.1
1993	17.4	6.3
1994	19.2	3.6
1995	18.5	5.8
1996	18.2	5.8
1997	15.5	3.6
1998	19.2	2.4
1999	28.8	7.9
Average 1993-1999	19.5	5.1

[a] 1989 flow data was omitted from the average because the reported volume of water at the weir exceeded the total volume of water that fell on the watershed.

[b] Excluded due to construction of treatment system.

Water quality

Prior to installation of the pretreatment system, the average pH at the weir (Seep 1 site) was 5.35, copper was 0.74 mg/L and nickel was 14.8 mg/L. In September 1992, immediately after the system was constructed, pH increased to 7.2, copper decreased to 0.02 mg/L and nickel decreased to 0.64 mg/L (Appendix 1). Until 1999 the pH of the outflow had remained above 6.5, and copper concentrations had averaged 0.1 mg/L, which corresponded to a removal efficiency of 85%. Nickel concentrations increased from the minimum value observed immediately after construction, but still averaged about 40% lower than pre-1992 levels. After the pretreatment system was constructed, sulfate concentrations averaged about 20% lower, from 1320 to 1060 mg/L (Table 7).

In 1999, about seven inches of rain fell on the July 4th weekend. Output flow was recorded as 265 gal/min (1000 L/min) and the pretreatment system overflowed. Rainfall was above average for most of the summer (Appendix 11) and water levels in the pretreatment pond were high (Appendix 8). The pH at the weir decreased substantially in 1999, and reached a minimum of 4.76 in July. The pH did not increase to above 6 until August (Figure 14). Metal concentrations also increased; nickel reached a maximum of around 15 mg/L, and copper was 1.3 mg/L. Nickel values had reached this level in both 1997 and 1998, but copper was about three times greater than the 1997-1998 values. The median nickel and copper concentrations in 1999 were the highest values observed since the pretreatment system was constructed in 1992 (Figure 7).

Before the pretreatment system was constructed, it was not possible to collect water samples from the base of the stockpile. The flow was diffuse and no channelization was observed until the water collected at the Seep 1 monitoring location. After the system was installed, small seeps at the base of the pile could be seen and sampled (Figure 8). Water quality and flows were sampled for these sites during the summer of 1993, once in 1996, and then periodically from 1997-99 (Appendix 1). In general, pH was lower and metal levels were higher than had been observed historically at the Seep 1 monitoring point. The pH ranged from 4.2 to 5.1, while copper ranged from about 1-6 mg/L and nickel from 10-55 mg/L. Flows were generally estimated qualitatively but some flow measurements were made with a portable flume device. In 1993, flows measured during late summer ranged from 0.002 to 0.02 L/sec, and accounted for only about 10-20% of the flow measured at the weir. Flows had been higher during the early summer but no quantitative measurements were made.

In 1993-1994, the pH in the outer pool samples was substantially higher than the summer average for the Seep 1 monitoring location prior to construction of the pretreatment system. The pH continued to increase as water moved across the pond (Figure 15). Copper concentrations generally decreased throughout the pond, with a 40% reduction as the water moved through the vertical down-flow section (Figure 16). Nickel and sulfate concentrations were also lower than the summer averages prior to construction, but remained relatively constant throughout the pond (Figures 17, 18). A 10% reduction in nickel concentrations was observed as water passed through the vertical down-flow section.

Table 7. Seep 1 water quality summary, 1992-99.

Year	pH	SO4	Cu	Ni	Co	Zn	Ca	Mg
Ave. 1986-1991	5.40	1320	0.69	14.7	1.08	1.41	163	215
1992	5.93	1770	0.686	9.77	0.54	1.10	190	180
1993	7.15	1300	0.135	7.05	0.26	0.69	210	170
1994	7.02	1060	0.153	6.23	0.27	0.62	190	160
1995	6.92	870	0.085	3.48	0.10	0.43	170	150
1996	6.94	930	0.237	6.32	0.182	0.773	190	130
1997	6.96	1010	0.130	6.36	0.123	0.764	190	140
1998	7.03	1180	0.164	7.46	0.136	0.891	230	160
1999	6.64	1100	0.603	9.00	0.319	1.362	210	140
Ave. 1993-1999	6.95	1060	0.215	6.56	0.198	0.790	200	150

Notes: The values showns are means of all available data for each year. pH values are standard units, all other values are mg/L. The metals values shown are totals. These data were collected by LTV and are for site 043, which is the historical measurement point for this seep, and which is now the effluent from the pretreatment pond. The pretreatment system was constructed during 1992, so the data from that year are not included in the overall average.

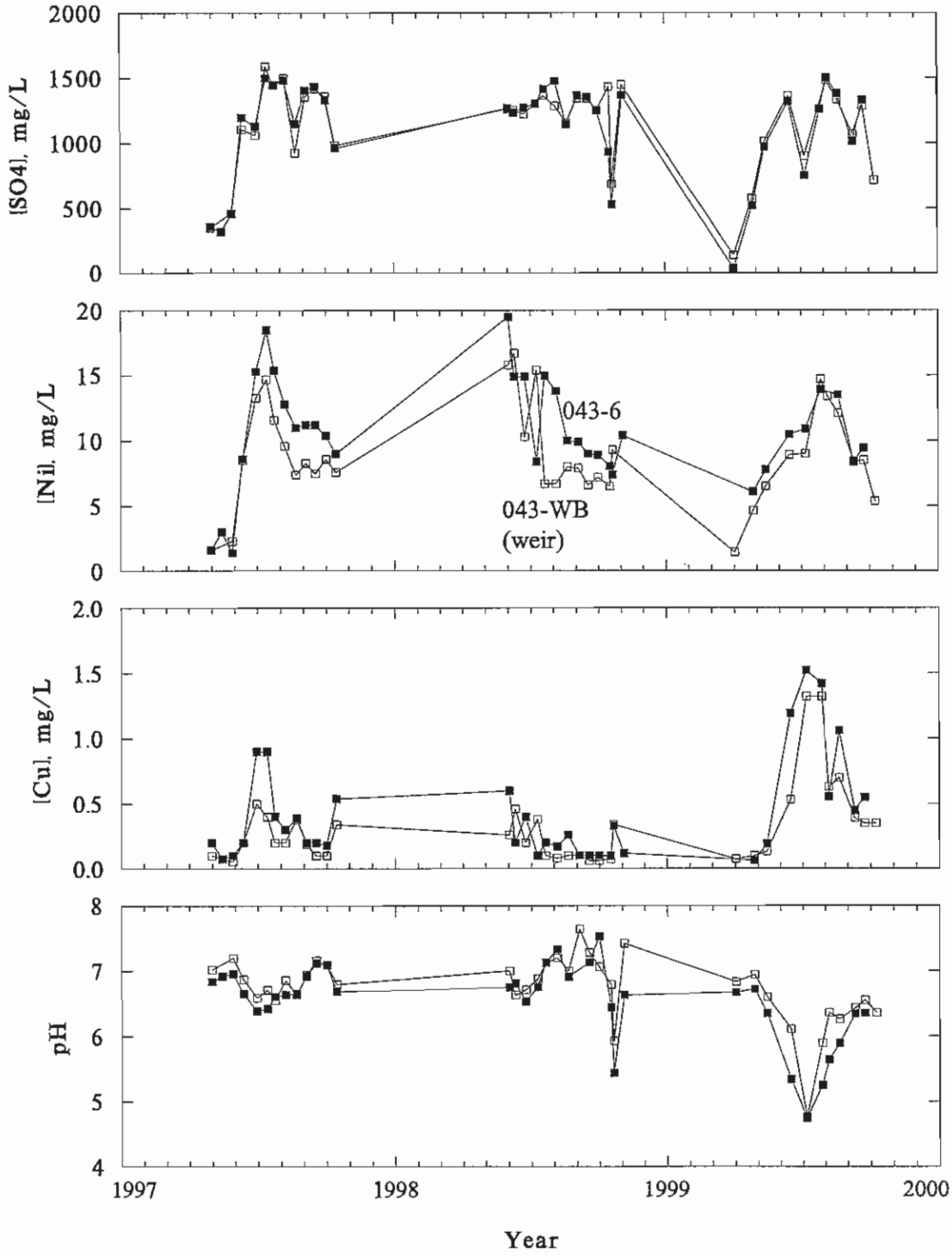


Figure 14. pH, copper, nickel and sulfate concentrations before and after the vertical down-flow section at the Seep 1 pretreatment system.

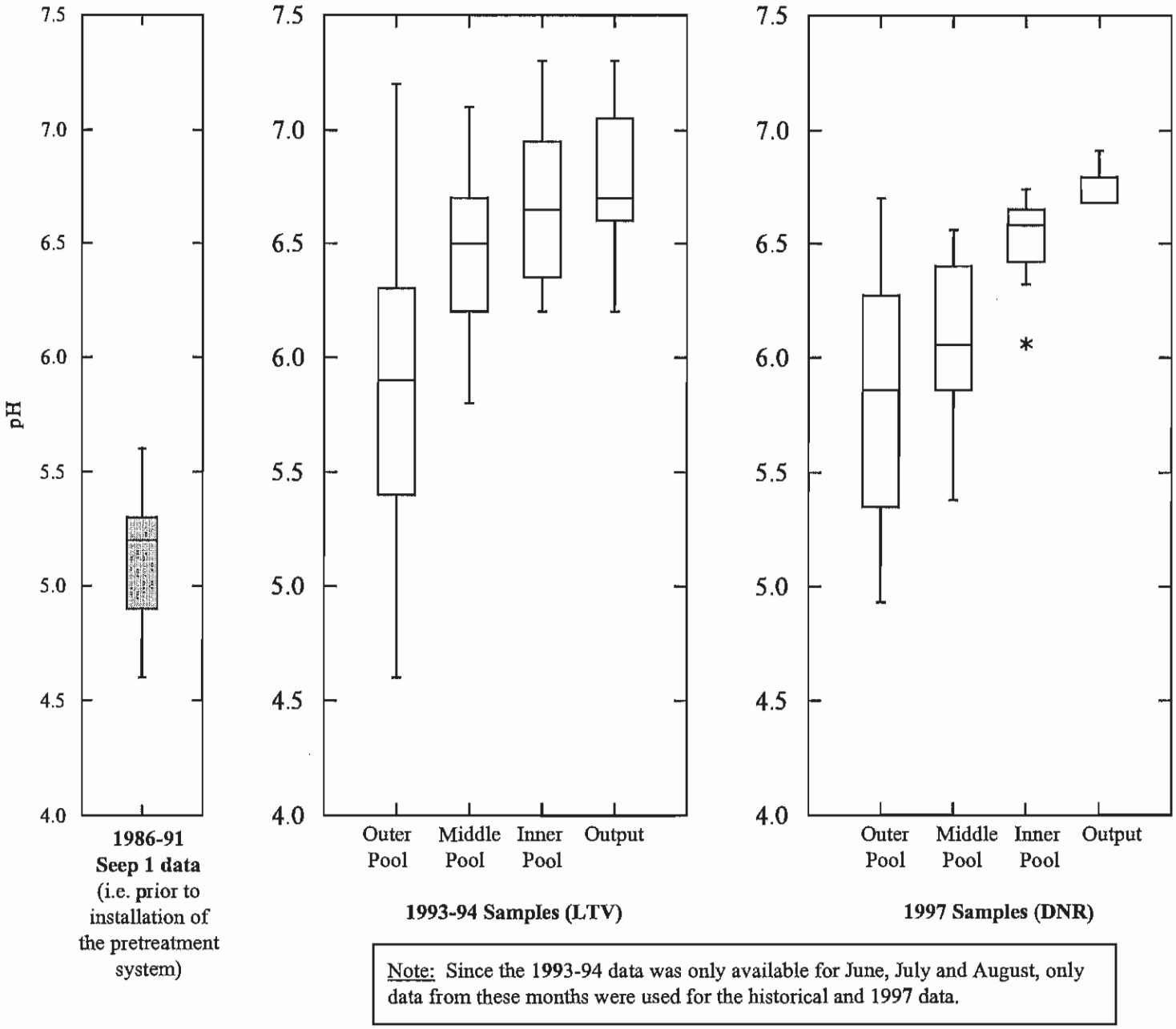


Figure 15. pH in the surface water and output of the Seep 1 pretreatment system; historical data (1986-1991), and data after construction of the pretreatment system (1993-1994 and 1997 data).

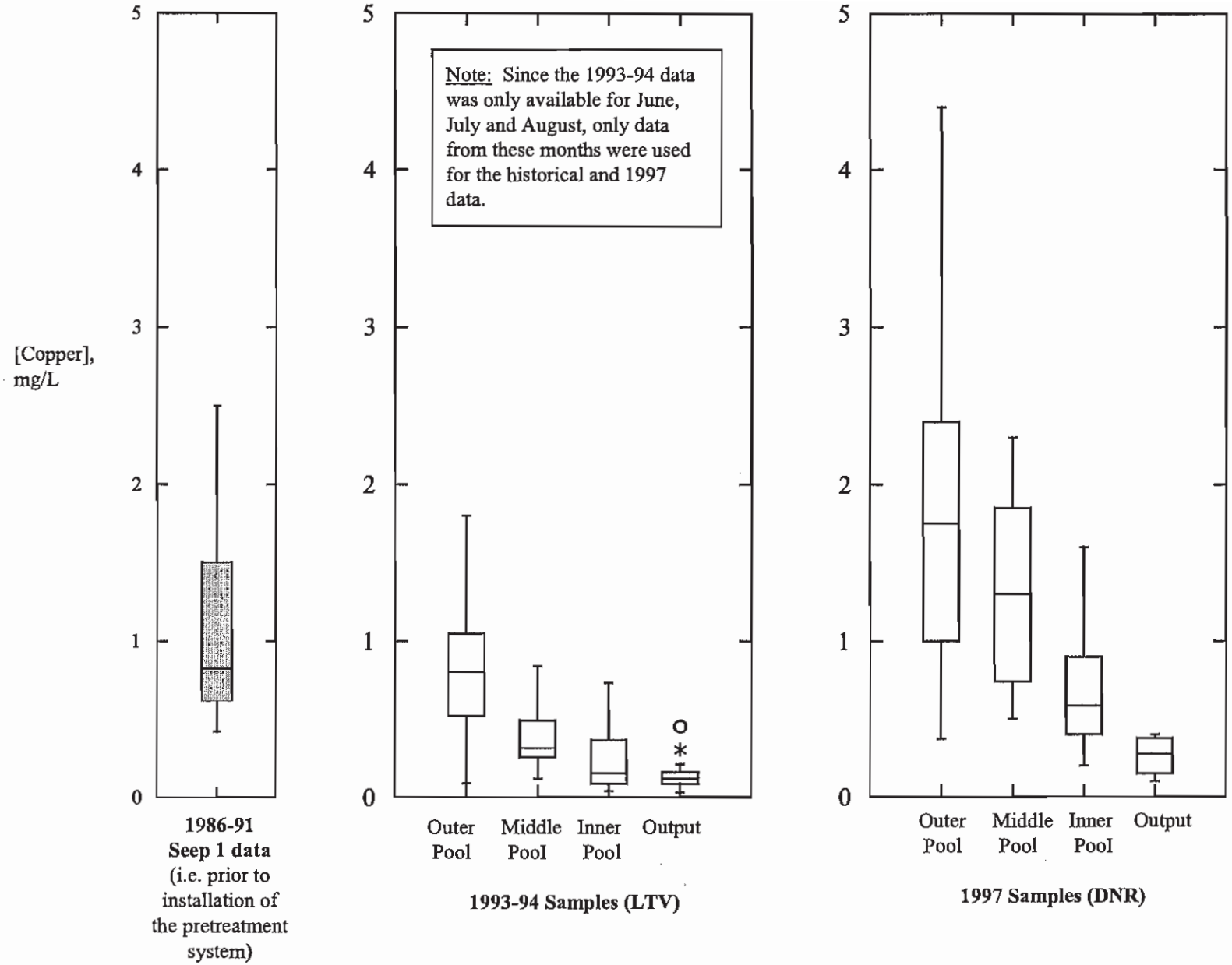


Figure 16. Copper concentrations in the surface water and output of the Seep 1 pretreatment system; historical data (1986-1991), and data after construction of the pretreatment system (1993-1994 and 1997 data).

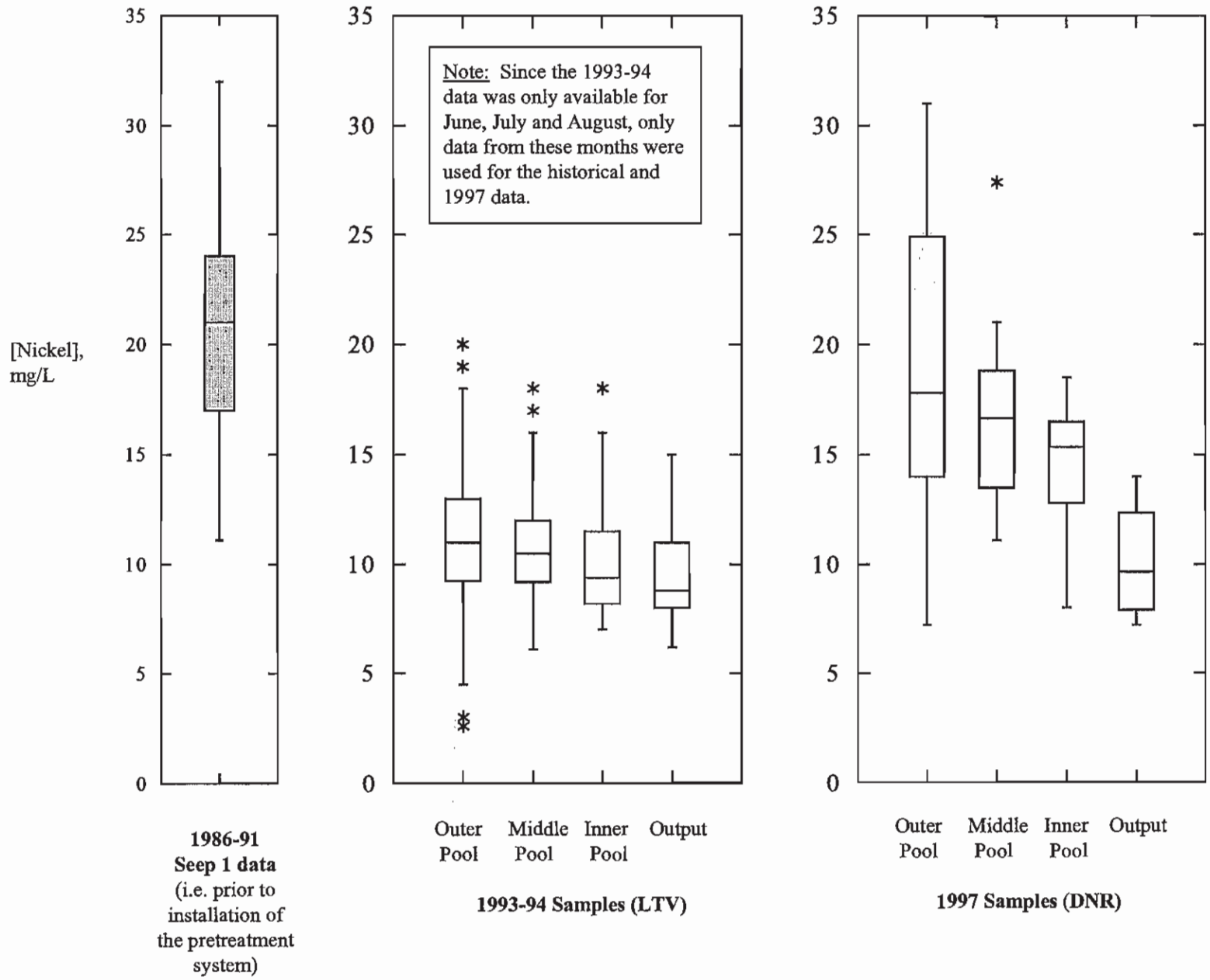


Figure 17. Nickel concentrations in the surface water and output of the Seep 1 pretreatment system; historical data (1986-1991), and data after construction of the pretreatment system (1993-1994 and 1997 data).

In 1997, pH followed the same general trend as in 1993-94, and increased as water flowed across the pond (Figure 15). While copper concentrations followed the same general trend, concentrations were not only higher than in 1993-94, but the outer pool concentrations were substantially greater than the historical data (Figure 16). Nickel concentrations were also greater than those measured in 1993-94, but were less than the historical values at the site prior to pretreatment system construction (Figure 17). For both copper and nickel, the change in concentration in the vertical down-flow section was greater than in 1993-94. Nickel decreased by 21% (from 13.6 mg/L to 10.8 mg/L) and copper decreased by 42% (from 0.52 mg/L to 0.3 mg/L). Sulfate concentrations did not change substantially within the system (Figure 18).

Samples collected from the outer pool in 1997 revealed that the water was not well mixed. Cold water from the small seeps entered the outer pool and tended to flow across the bottom of the pool to the first berm. Water would move over or through the first berm and would generally be mixed in the middle pool (Appendix 9). As a result of this stratification and the relatively constant nickel concentration in the system, sampling in 1998 and 1999 focused on the long-term removal capacity of the vertical down-flow section of the system.

In 1998, nickel concentrations decreased by about 17% in the vertical down-flow section, and a paired t-test confirmed that there was a statistically significant decrease in concentration over this section in 1997 and 1998 (Appendix 1). In 1999, the average change in nickel concentration decreased to only 8% in this section (Figure 14, Table 8).

In 1999, copper concentrations were the highest since the pretreatment system was built, but still generally decreased in the vertical down-flow section, although there was only about a 10% difference between the concentrations immediately following the July storm.

Substrate samples

Three samples were collected from the vertical down-flow section of the treatment system and 15 samples were collected and analyzed throughout the pond area to examine the metal concentrations in the substrate (Figure 8). Samples of both the peat mixture and the hay were collected. Metal concentrations in all substrate samples were two to three orders of magnitude higher than the original peat mixture and several of the samples contained more than 1% nickel (Figures 19, 20, 21). Concentrations were high throughout the substrate, but the highest concentrations were observed in the vertical down-flow section. The maximum concentrations were 14,180 mg/kg nickel, 8392 mg/kg copper, 1347 mg/kg cobalt and 3299 mg/kg zinc (Figures 19, 20, 21 and Appendix 4).

In the samples from the pond area, metal concentrations were about an order of magnitude higher in the top 10 cm of the cores than in the 20-30 cm layer.

A sample of a whitish precipitate was collected from the surface of the substrate and analyzed by Midland Research Center, Nashwauk, MN. A total digestion was performed and x-ray diffraction was used in an attempt to identify any crystalline solids. The precipitate occurs throughout the

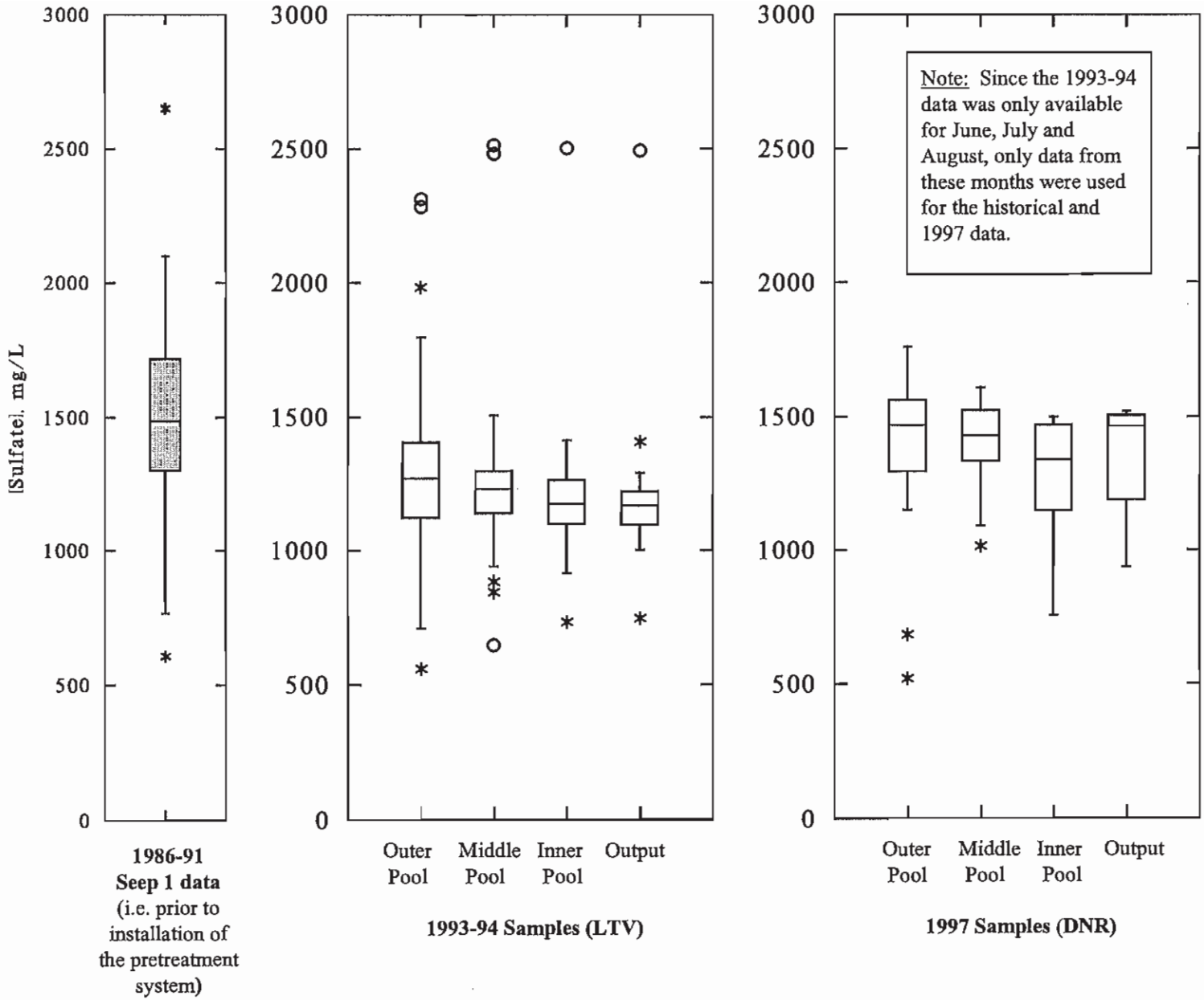


Figure 18. Sulfate concentrations in the surface water and output of the Seep 1 pretreatment system; historical data (1986-1991), and data after construction of the pretreatment system (1993-1994 and 1997 data).

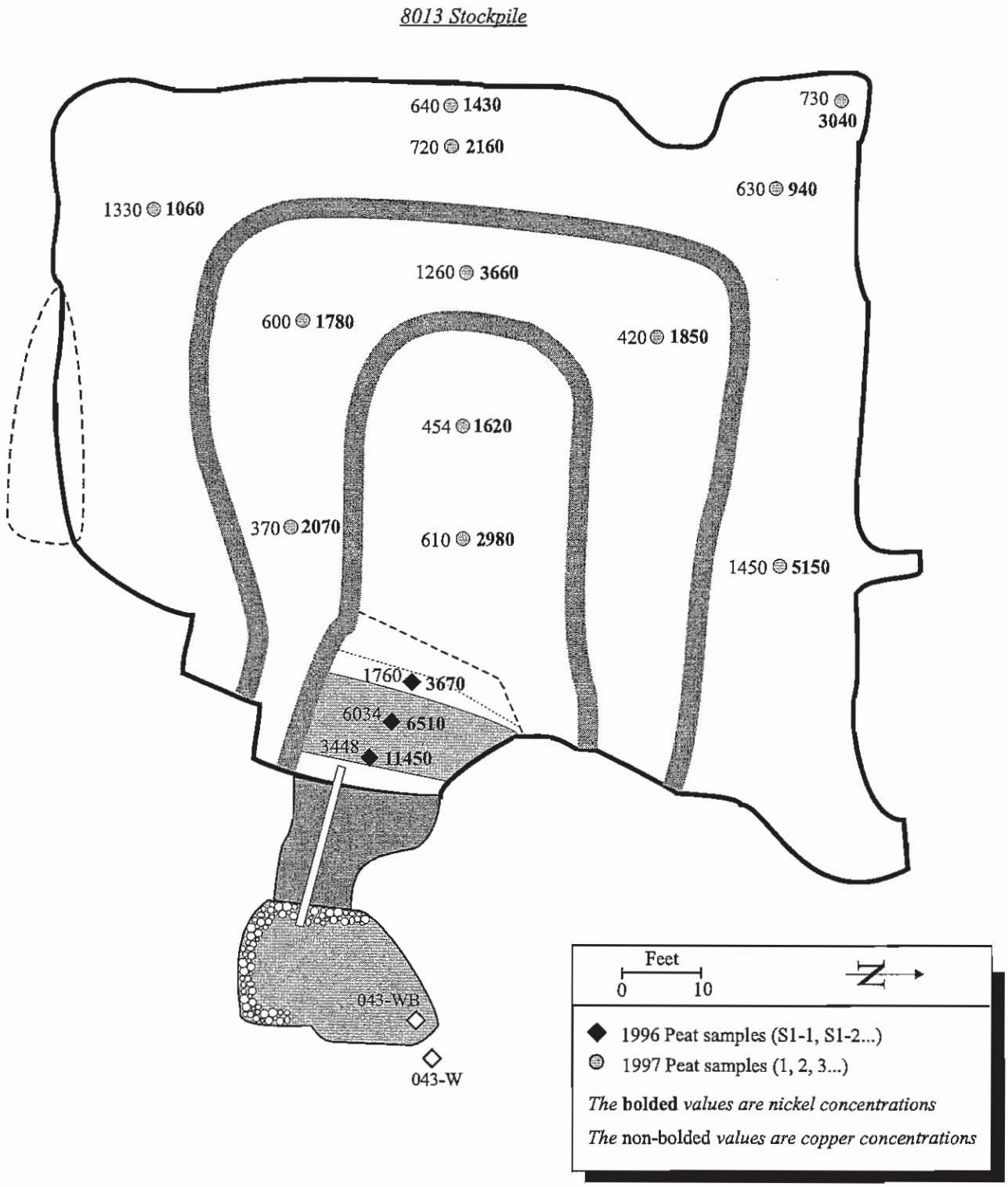


Figure 19. Seep 1 pretreatment system; nickel and copper concentrations (mg/kg) in the 0-10 cm substrate layer.

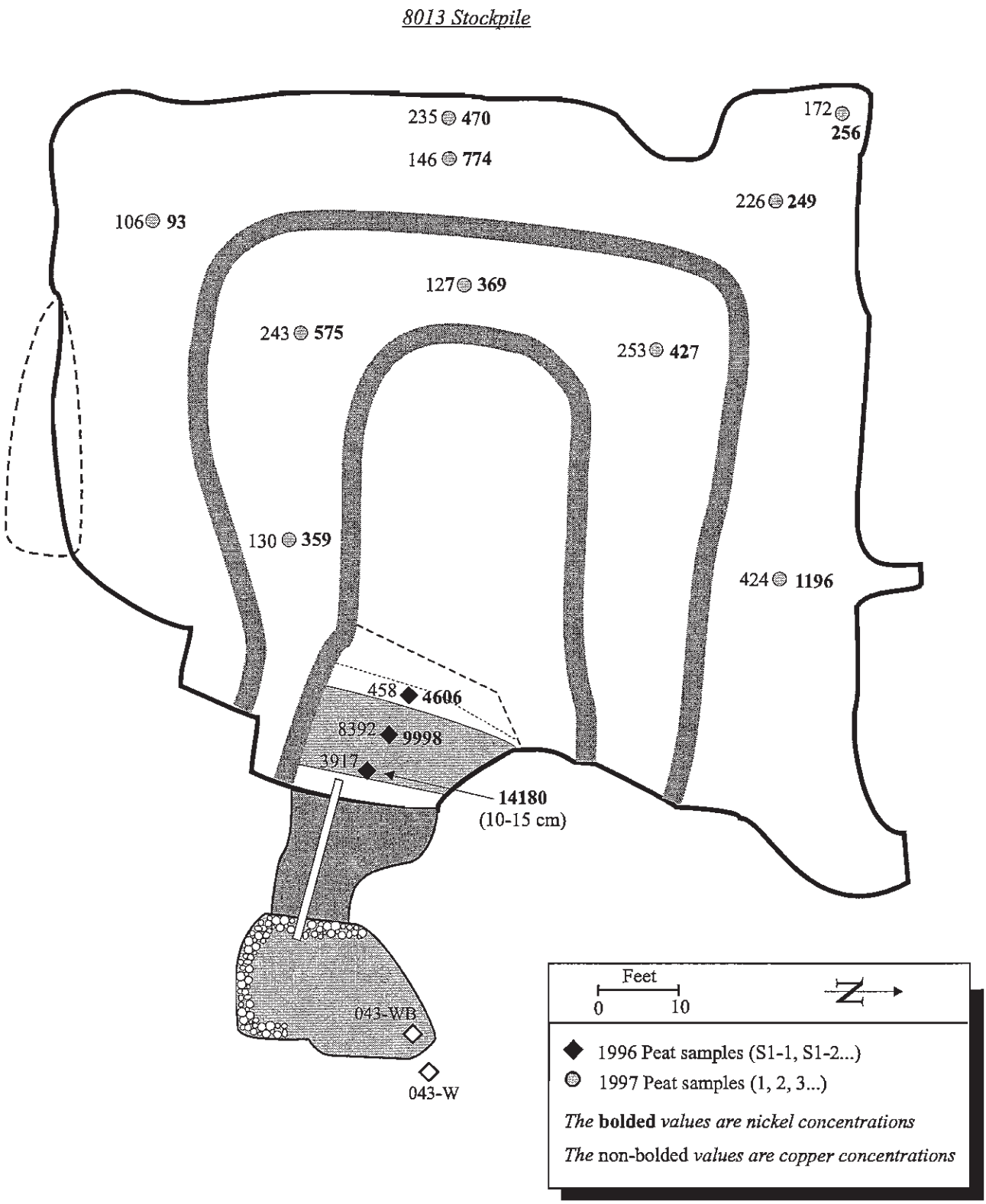


Figure 20. Seep 1 pretreatment system; nickel and copper concentrations (mg/kg) in the 10-20 cm substrate layer.

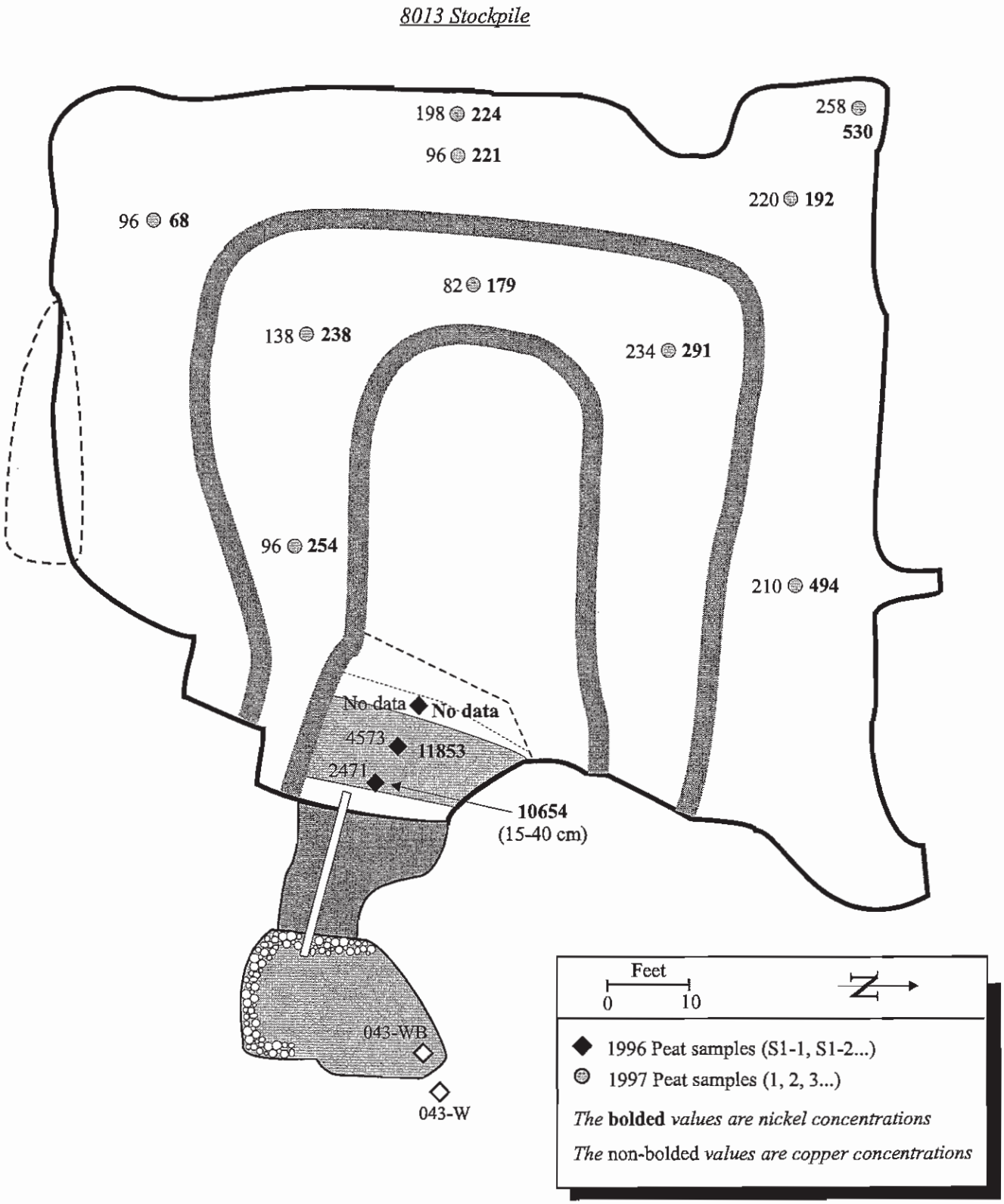


Figure 21. Seep 1 pretreatment system; nickel and copper concentrations (mg/kg) in the 20-30 cm substrate layer.

Table 8. Change in concentration over the Seep 1 vertical down-flow section.

Year	pH		Cu		Ni		SO_4		Ca		Mg	
	In	Out	In	Out	In	Out	In	Out	In	Out	In	Out
1993	6.7	6.8	0.20	0.12	10.5	9.6	1265	1250	no data	no data	no data	no data
1994	6.6	6.7	0.48	0.27	8.3	7.4	990	1010	no data	no data	no data	no data
1997	6.6	6.7	0.52	0.30	13.6	10.8	1320	1270	243	226	173	161
1998	6.9	6.9	0.28	0.23	13.8	11.4	1300	1260	272	274	202	201
1999	5.4	5.9	1.06	0.80	12.2	11.2	1240	1270	253	263	202	203

Notes: pH values are in standard units and all other values are mg/L. The values shown are based on June-August data only.

In: Site 043-6 (water above the vertical section)
Out: 1993-94: Seep 1 weir
1997-99: Site 043-WB (This site was after the vertical down-flow section but upstream of the weir and a biolog, which contained organic materials. This product was installed by LTV in 1996 and tested for its ability to remove metals from the discharge (see Figure 2).

pretreatment pond but the heaviest accumulation appears to be in the outer pool. The sample was amorphous and was dominated by aluminosilicates and clays, and contained some iron sulfate hydrates (Appendix 10). The major trace metal in the precipitate was copper (5400 mg/kg). Nickel content was 1280 mg/kg.

Mass removal

Annual mass release from the site has decreased substantially since the system was installed. Copper mass decreased from 5.4 kg to 1.9 kg, a 65% reduction, while nickel release decreased from about 124 to 45 kg, a 67% reduction (Table 9). (Copper mass release was unusually high in 1999 (Appendix 7). If 1999 data is excluded, the average release was 0.9 kg, an 85% reduction in copper load.) Since there is no single input into the system and most of the water enters as groundwater into the pond area, it is not possible to conduct a standard mass balance (input-output) for this system (Appendix 8). Since metal concentrations in the substrate decreased with depth, there was no evidence for metal removal from upwelling groundwater as it entered the pretreatment area. The majority of the metal removal appeared to occur within the pretreatment system. As a result, mass removal within the pretreatment system was calculated by comparing the estimated concentration in the outer pool with the concentration measured at the weir (Appendix 7). Using these

Table 9. Seep 1 mass release, 1986-1999.

	Before the pre-treatment system was put on line		After the pre-treatment system was put on line	
	Average conc. (mg/L)	Mass release (kg)	Average conc. (mg/L)	Mass release (kg)
pH	5.4	NAp	6.95	NAp
Cu	0.69	5.4	0.22	1.9
Ni	14.7	124	6.6	45
SO4	1,320	10,800	1,060	6,600
Mean annual flow, May-Oct ($L \times 10^6$)	7.3		5.1	

Nap = not applicable

Note: a) The "before" time period includes all water quality data from 1986-1991, and all flow data from 1986-1991 with the exception of data from 1989. The 1989 flow data was excluded because the volume of water measured at the weir exceeded the volume of water that fell on the watershed.

b) The "after" time period includes all data from 1993-1999. Copper release was anomalously high (7.9 kg) due to the July 4th rainstorm - if 1999 is excluded, the average "after" mass release is 0.9 kg instead of 1.9 kg.

assumptions, 9.4 kg of copper and 77.2 kg of nickel were removed from 1993 through the end of 1996 (Table 10). The total mass contained within the peat substrate was calculated by:

$$\text{mass of metal (kg)} = \text{area of pool } (m^2) \times \text{depth of sample (cm)} \times \text{density } (gm/cm^3) \times \text{average metal concentration mg/kg} \times 10^4\ cm^2/m^2 \times 10^{-6}\ kg/mg \times 10^{-3}\ kg/gm$$

Using this method, 11.1 kg of copper and 30.2 kg of nickel were calculated for the top 30 cm of the peat (Table 10). The highest metal concentrations, and about 36% of the total mass, were in the vertical down-flow section.

Table 10. Copper and nickel mass in substrate of the Seep 1 pretreatment system, 1992-1996.

Portion of system	Area (m^2)	Volume in each 10 cm layer of peat (m^3)	Mass of dry [A] peat in each 10 cm layer (kg)	Mean [B] [Ni], mg/kg	Niekel mass (kg)	Mean [Cu], mg/kg	Copper mass (kg)
Outer ring	337.6	33.76	3376				
0-10 cm				2,297	7.75	917	3.10
10-20 cm				506	1.71	185	0.62
20-30 cm				288	0.97	180	0.61
Total				---	10.43	---	4.33
Middle ring	195.6	19.56	1956				
0-10 cm				2,340	4.58	662	1.29
10-20 em				432	0.85	188	0.37
20-30 cm				240	0.47	138	0.27
Total				---	5.90	---	1.93
Inside ring	99.6	9.96	996				
0-10 cm				2,300	2.29	532	0.53
10-20 cm				432	0.43	188	0.19
20-30 em				240	0.24	138	0.14
Total				---	2.96	--	0.86
Vertical flow	38.3	3.83	383				
0-10 cm				7,210	2.76	3,747	1.44
10-20 cm				9,594	3.67	4,255	1.63
20-30 cm				11,254	4.31	2,348	0.90
Total				---	10.74	---	3.97
Berms	103.7	---	---	---	---	---	---
Total in substrate	774.5	67.08	6708	---	30.03	---	11.09
Mass removal calculated from water quality	---	---	---	---	77.20	---	9.40

A: Assumes 0.1 g of dry peat per cc of wet peat
B: No copper or niekel data was available for the 10-20 and 20-30 cm layers for the inner ring, so the eorresponding values from the middle ring were used for thesc calculations.
Note: The peat samples used to calculate the mean nickel values were collected 4/14/97 - 4/24/97, except for the vertical flow section, which were collected 4/4/96.

Discussion, W1D

The overall objective of the W1D study was to determine the lifetime of the wetland treatment system. LTV designed the original W1D system based on average input values for 1990-1991. Average daily flow was 20 gal/min and the average nickel concentration was 5.4 mg/L (Eger et al., 1996). Based on a wetland area of 7000 m^2, an effective removal depth of 20 cm, a peat bulk density of 0.1 gm/cm^3, a maximum removal capacity of 10,000 mg nickel/kg dry peat, and flow from April through November (245 days), the design lifetime was:

$$\text{Lifetime (years)} = \frac{\text{total removal capacity of the wetland (kg nickel)}}{\text{annual load (kg nickel/year)}}$$

$$= \frac{\text{volume of reactive peat x bulk density x removal capacity}}{\text{average daily flow x average nickel concentration x number of days of flow}}$$

$$= \frac{7000\ m^2 \times (100\ cm/m)^2 \times 20\ cm \times 0.1\ gm/cm^3 \times 10{,}000\ mg\ nickel \times 10^{-3}\ kg/mg \times 10^{-6}\ kg/mg}{20\ gal/min \times 3.785\ L/gal \times 1440\ min/day \times 5.4\ mg\ nickel/L \times 1/10^6 mg/kg \times 245\ days}$$

$$= \frac{1400\ kg}{144\ Kg} \approx 10\ \text{years}$$

In 1995 the entire top of the stockpile was covered with a 30 mil linear low density polyethylene liner (LLDPE). While this cover prevented water from contacting most of the reactive material in the stockpile, mineralized rock was still exposed on the uncovered side slopes. Flow at the W1D weir dropped 55%, from an average May to October flow of 33 gal/min (125 L/min) during 1992-1994, to 15 gal/min (57 L/min) for the post-closure period (1996-1999).

By preventing precipitation from infiltrating the stockpile and contacting the reactive material, the transport of reactive products was significantly reduced. Nickel concentrations decreased from an average of 3.98 mg/L in 1992-1994 to 0.74 mg/L for 1996-1999. Since both flow and nickel concentrations decreased, the overall load to the wetland decreased by about 90% (Table 5). By reducing the load, the estimated lifetime was increased substantially, from the initial design lifetime of 10 years to an average of 150 years in 1996-1999 (Table 11).

The ultimate goal of a passive treatment system is to provide permanent treatment. The major mode of metal removal in this system is assumed to be the same as observed in the initial test cell study (Eger et al., 1994). Over 90% of the nickel removal in the test cells occurred within the substrate, through a series of reactions (adsorption, ion exchange, chelation) associated with the organic fraction of the peat. With this type of removal mechanism, the wetland will have a finite life unless new removal sites can be generated at a rate greater than or equal to the incoming metal load (Eger et al., 1994).

New sites are generated as vegetation dies and new organic substrate accumulates. The average rate of peat accumulation in northern wetlands is about 1 mm/year (Craft and Richardson, 1993). If the

Table 11. Effect of stockpile capping on load to the W1D wetland and wetland lifetime.

Year	Annual nickel load (kg)	Percent of pre-capping load	Lifetime of wetland [1] (years)
Initial design lifetime	144	---	10
1992-1994	170	---	8
1995	76	45	18
1996	36	21	39
1997	8	5	175
1998	5	3	280 (self sustaining [2])
1999	16	9	88

[1] Lifetime is calculated by dividing the initial removal capacity of the wetland (1400 kg) by the annual load.

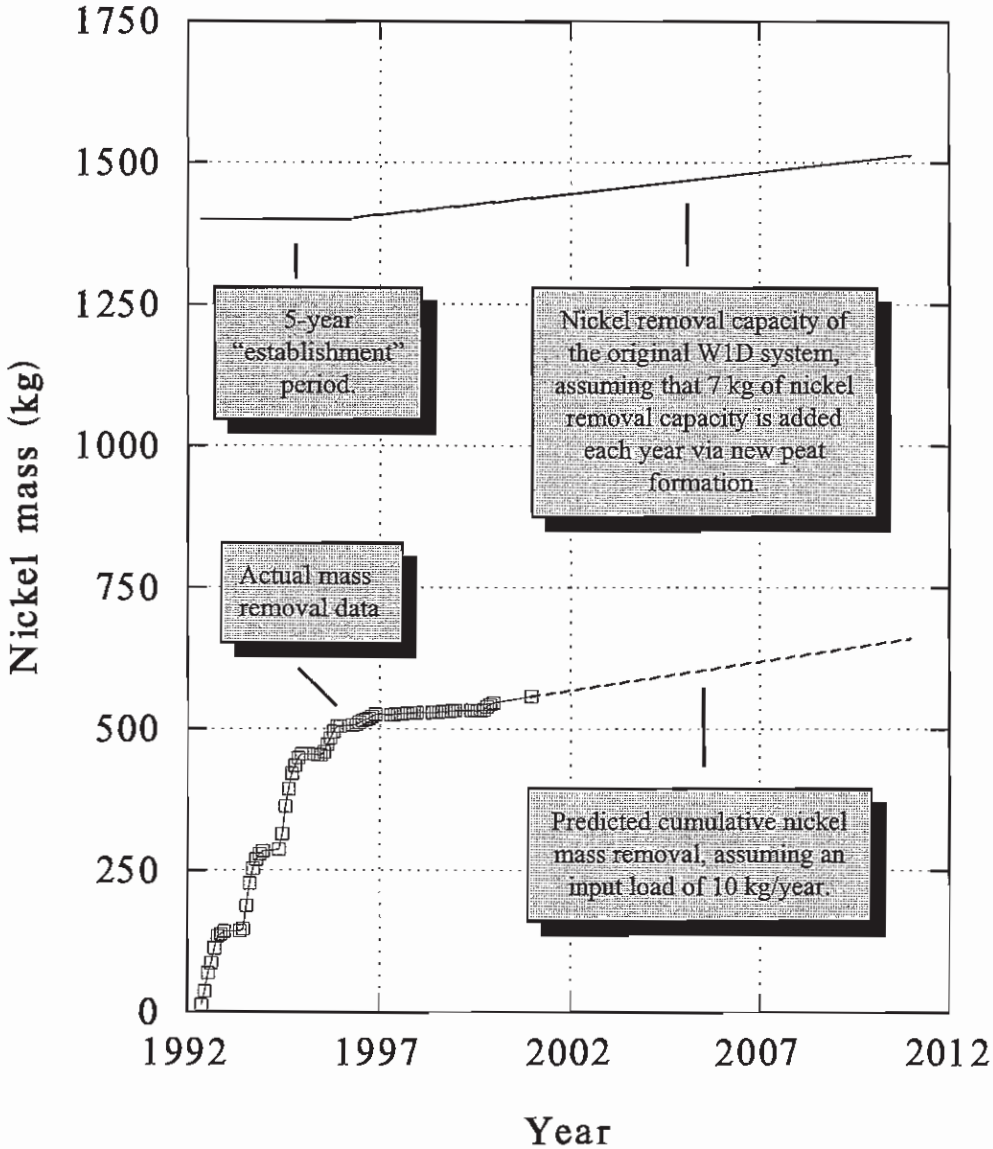
[2] Annual nickel load to the wetland is less than the estimated annual gain in nickel removal capacity (7 kg); the wetland is self-sustaining.

removal capacity of the newly accumulated material is assumed to be 10,000 mg nickel/kg, the wetland would add 7 kg of nickel removal capacity each year:

Annual gain in nickel removal capacity = rate of peat accumulation x nickel removal capacity x wetland area

$$= \frac{1\ \text{mm/year} \times 0.1\ \text{cm/mm} \times 10{,}000\ \text{mg Ni/kg} \times 7000\ \text{m}^2 \times (100\ \text{cm/m})^2 \times 0.1\ \text{gm/cm}^3 \times 10^{-3}\ \text{kg/gm}}{10^6\ \text{mg/kg}}$$

In 1998 the annual nickel input load was less than the gain in nickel removal capacity. If the annual input load is less than or equal to the annual increase in removal capacity, the wetland should be self-sustaining. In 1999 the input load was greater than the sustainable load, but the increased load was the result of above normal precipitation, particularly in July when 9.49 inches (24.1 cm) of rain fell. Using the average of the 1997-1999 input load (10 kg) as representative of the post-closure period, and assuming an annual increase of 7 kg nickel removal capacity, the projected lifetime for the wetland is about 290 years (Figure 22, Appendix 12).



This model assumes that:

1. The original W1D wetland system had a nickel removal capacity of 1400 kg.
2. Organic matter accumulates at an annual rate of 1 mm/year.
3. Newly formed organic matter has the same removal capacity as the original substrate and can remove 7 kg nickel per year.
4. It took 5 years for the vegetation to become fully established and to start contributing to substrate formation.
5. The input nickel load will stay at 10 kg/year, which was the average input from 1997-1999.

At these metal loading and organic matter deposition rates, the model predicts that the peat would become saturated in about 290 years. This simple model does not account for uneven flow distribution, which would decrease the amount of peat contact and therefore reduce system lifetime. (Additional details on the assumptions used in these calculations are provided in Appendix 12.)

Figure 22. Nickel mass removal in the W1D system, and projected nickel removal capacity.Fora

wetland to be sustainable, not only must there be new metal removal capacity generated, but the metal must be retained within the wetland. Mass balances calculated on wetland test cells demonstrated that over 99% of the removed metals were associated with the substrate and less than 1% of the total removal occurring in the vegetation (Eger et al.,1994). These results were consistent with earlier studies on metal removal in a white cedar wetland (Eger and Lapakko, 1988) and with data reported by others (Skousen et al., 1992, Wildeman et al.,1993). Nickel contained within the substrate of the study cell accounted for essentially all of the total nickel removal that was calculated from the change in water quality data. Sequential extraction tests, conducted on a series of substrate samples collected from test cells constructed at the Dunka Mine, demonstrated that only 1-2% of the nickel was water soluble and could be readily removed from the substrate (Eger et al., 1994).

Additional evidence for the permanent nature of the removal is that nickel removal in the wetland has continued despite a decrease in input concentration of almost an order of magnitude. If the nickel was weakly bound to the substrate, nickel would be released from the substrate as nickel concentrations in the water decreased, and no removal would occur. Although continuous flow data is only collected from May through October, water quality samples are collected whenever there is water flowing into or out of the wetland. Over the seven years of operation, output concentrations have rarely exceeded input values, and there has always been a net nickel removal in the wetland (Figure 9).

Discussion, Seep 1

Until 1999, the pretreatment system routinely increased pH and reduced copper and nickel concentrations. The pH at the outflow of the pretreatment system generally met discharge limits. Copper concentrations were reduced by about 70%, but still routinely exceeded the NPDES discharge limit of 0.022 mg/L. Average nickel concentrations were about 55% below historic levels, but they were also substantially greater than the discharge limit of 0.484 mg/L (Table 8). Based on the topography of the site, the construction of the pond did not alter the overall amount or direction of groundwater flow, so any reduction in concentrations at the weir monitoring site must be due to reactions that occur in the treatment system, or a reduction in the total load from the stockpile.

Typically, to analyze the performance of a treatment system, the amount of mass removal is calculated by subtracting the output mass from the input mass. Since this system does not contain a single input point and the majority of the water enters as groundwater, a standard analysis (input mass minus output mass) cannot be done. Therefore, changes in flow and concentration must be analyzed separately.

Average May to October flow, after the stockpile had been capped and the pretreatment system built, was about 30% less than the historic (1986-1991) values. Some of this reduction in flow was probably due to lower precipitation. When flow was corrected for the lower precipitation, the overall reduction in flow due to capping was estimated to be 25% (Appendix 8). The projected flow reduction, based on the total watershed area at Seep 1, the capped stockpile area, and a 40% reduction in flow for the covered portion of the stockpile, was 24% (Appendix 8). Assuming a 25% reduction in flow after capping, the annual copper load would be 4 kg and the annual nickel load would be 93 kg.

Based on these input loads and the measured output loads at the Seep 1 site after the pretreatment system was constructed, annual average removal would be about 3 kg for copper and about 50 kg for nickel. For 1993 through 1996, the total metal removal would be 12 kg of copper and 200 kg of nickel. This amount of copper was within 10% of the mass calculated to be contained within the substrate, but the nickel mass was almost seven times higher than the substrate value (Appendix 7). When the outer pool concentrations were used to represent the input, the copper removal was 9.4 kg, while the nickel mass removal decreased to 77.2 kg, a value much closer to the mass contained within the substrate (30.0 kg).

Since the nickel concentrations in the outer pool were significantly less than the historical concentrations, and provided a better correlation with the mass contained in the substrate, it appears that capping of the stockpile has not only reduced flow but may have also reduced nickel concentrations in the input seepage. Limited measurements of the contribution of the individual surface seeps to the overall load indicated that the majority of the nickel input was via groundwater, while copper enters primarily as surface flow (Table 12; Appendix 7). Any change in the input nickel concentration could not be quantified due to the lack of both wells and historical data.

Table 12. Comparison of the load from surface seeps, with the load at Seep 1 weir, August, September, 1993.

Date	(Sum of seeps / Seep 1) x 100%				Seep 1 Flow	
	Flow	Cu	Ni	Mg	L/sec	gal/min
8/17/93	23	242	53	32	0.30	4.8
8/30/93	17	206	35	21	0.37 (e)	5.8
9/16/93	13	127	25	16	0.27	4.3
Average	18	192	38	23	0.31	4.9

Notes: The instantaneous load from each surface seep was calculated by multiplying the flow (measured with a small portable flume) by the concentration. The load from all seeps with measurable flows were added to estimate the total load (Appendix 7). Samples were collected at some seeps but there was not sufficient flow to measure. The contributing seeps included S-2, S-3, S-4, S-8 and S-9 (Figure 8).

The sum of the load from the seeps was divided by the instantaneous load measured at Seep 1. When the ratio exceeds 100%, more mass entered the system through the seeps than left the system at the weir.

(e) = estimated flow, see Table A7.6

Copper entered the pretreatment system primarily through the small acidic seeps. Copper concentrations decreased as water moved through the pool, contacted the limestone berms, and increased in pH. This removal was consistent with that observed in both laboratory and field experiments conducted with water from the site. A field pilot system, using 6-mm limestone chips, was constructed at this site in 1989 (Lapakko and Antonson, 1990a). With an average residence time of about 1 hour, the average pH increased from around 5.0 to 6.8. Even with residence times as short as 15 minutes, pH always increased to 6.2 or higher. Copper removal averaged about 50%, while nickel was reduced by 10%. In laboratory limestone columns, with residence times on the order of 5 hours, pH increased to 7.5, and 80% of the copper and 10% of the nickel were removed (Lapakko and Antonson, 1990b).

The limestone used in the existing treatment system ranged in size from 7½-10 cm, and the reactive surface area was an order of magnitude less per unit mass than the limestone used in the pilot and laboratory experiments. (The larger limestone was used to avoid plugging problems that had been observed in the pilot study.) Therefore, the residence time in the treatment system would have to be increased by at least a factor of 10 to achieve results similar to the pilot study. Based on the size of the treatment system (930 m^2) and an assumed average water depth of 0.3 m, the overall residence time was calculated to have ranged from around 120 hours for the average flow to 12 hours for the typical daily maximum flow, generally meeting the criteria for an increased residence time. With the exception of 1999, the pH of the outflow was similar to that observed in the pilot system and tended to decrease as flow increased (Appendix 1).

During storms over the July 4th weekend in 1999, daily flow increased to about 265 gal/min (1000 L/min), with a peak flow of around 790 gal/min (2990 L/min). This flow rate exceeded the capacity of the vertical down-flow section. As a result, the water level rose about 1.3 feet (0.4 m) and flowed through the emergency overflow. On July 7th, the pH at the site above the vertical down-flow section (site 043-6) was only 4.74, and there was essentially no change as the water moved through this section. Flow at the weir dropped to about 35 gal/min (139 L/min) within two days of the rain, but the pH did not increase until August. The estimated residence time ranged from five hours during peak flow to 35 hours two days later. Even though flow had decreased within two days, water levels in the pond remained high. The water level was only 0.3 feet (9 cm) below the overlow level, and was about one foot (30 cm) above the limestone berms (Appendix 8). Despite increased residence time, the high water level prevented contact between the drainage and the limestone berms.

As water moved through the pretreatment system and contacted the substrate, removal reactions similar to those at the W1D wetland would be expected to occur. As the water moves through the vertical down-flow section, other reactions such as sulfate reduction may become important. The addition of the hay to the system in 1994 appears to have provided a short-term stimulus to the sulfate reduction process. Hay has been used in other studies to provide a readily available source of food for sulfate reducing bacteria (Wildeman et al., 1993). After the hay was added, sulfate and nickel concentrations decreased, but returned to the original values in less than a year.

In earlier laboratory experiments, peat columns were used to treat drainage from another stockpile at the Dunka Mine (site EM-8, Figure 2). Nickel concentrations in the outflow of the column followed a classical adsorption type breakthrough curve, except that the final concentration was about 10-20% lower than the input, and remained at that level for the duration of the experiment (Lapakko et al., 1986). This change in nickel concentration was similar to that observed in the vertical down-flow section. It is possible that sulfate reduction reactions occurring within the substrate account for this sustained removal. Nickel concentrations calculated for the peat in the columns were on the order of 20,000 mg/kg, similar to the values measured within the vertical down-flow section (Lapakko et al., 1986).

Although summer nickel removal in the vertical section averaged 18% for 1997 and 1998, removal decreased to 8% in 1999. The decrease in nickel removal could be due to slower reaction rates, a loss in treatment capacity or the result of higher than average flows. Additional data should be collected to determine if the system has reached saturation.

Conclusions

W1D

Since 1995, when the mine was closed and the stockpiles capped, nickel loads into the W1D wetland have dropped by almost an order of magnitude. Nickel has been removed every year and there has been no evidence of nickel release from the wetland. The nickel load into the wetland is now about the same as the estimated annual production of new removal sites. If conditions remain unchanged, treatment could continue indefinitely.

Seep 1

Since 1992, the pretreatment system has been effective in increasing pH and removing copper. The system appeared to provide treatment for a range of flows, but could not handle the flows following the seven inches of rain over the July 4th weekend in 1999. After this rain event, the pH of the outflow decreased and metal concentrations increased. Since much of the copper removal is related to the increase in pH, the system should continue to remove copper until the limestone dissolution rate can no longer neutralize the pH of the input drainage. Since a detailed mass balance could not be conducted on the pretreatment system, the rate of limestone dissolution and an expected lifetime could not be calculated. However, there is no data to suggest that the system's ability to neutralize the drainage has decreased.

Most of the nickel removal occurred in the vertical down-flow section, but concentrations decreased by only 10-20%. Nickel concentrations in the substrate of the vertical down-flow section were on the order of 10,000 mg/kg, a value that has been used to estimate total removal capacity. In 1999, nickel removal in the vertical down-flow section decreased to only 8%. The lower rate of removal could be due to the higher flow in 1999 or a decreased treatment efficiency. Additional data should be collected to determine if the system has reached saturation.

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Appendix 1

Water quality data

Tables

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 . . . . . A1.1

Table A1.2. Water quality data (1997-99) from the DNR study cell of the W1D treatment system A1.22

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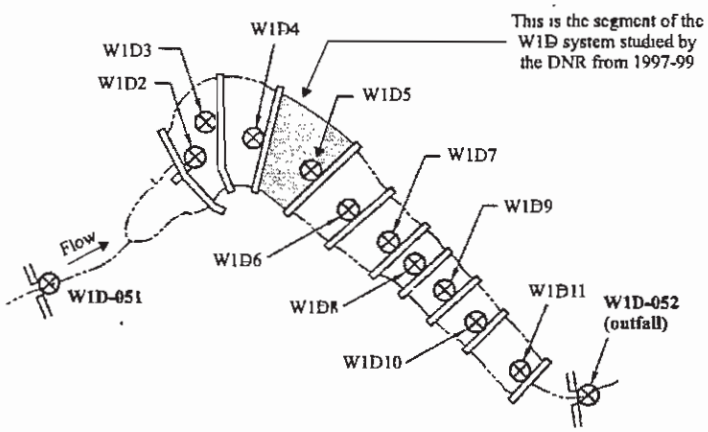
Figure A1.2. Comparison of LTV's and DNR's data (pH, Cu, Ni, SO_4) for 04 (the weir prior to the wetland treatment system), 1997-99 . . . . . A1.54

Table A1.1. W1D water quality data, collected by LTV from 1992-1999.

This file includes all W1D data (orginal system) collected by LTV. DNR also conducted a separate study of a particular segment of the treatment system; the data from that study is presented in Table A1.2.



Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
Input (W1D-051)											
W1D-051	4 29 92	7.1	0.080	1.300	0.004	0.010	0.100	.	1064.0	130.0	130.0
W1D-051	5 4 92	.	0.120	1.100	0.024	0.050	.	.	.	235.0	280.0
W1D-051	5 6 92	7.1	0.130	2.400	0.008	0.030	0.200	3.2	1613.0	270.0	310.0
W1D-051	5 11 92	7.2	0.130	1.600	0.012	0.030	.	.	1257.0	200.0	205.0
W1D-051	5 13 92	7.3	0.090	1.400	0.008	0.020	.	.	1209.0	150.0	165.0
W1D-051	5 18 92	7.1	0.120	2.600	0.008	0.030	.	.	1794.0	198.0	253.0
W1D-051	5 21 92	7.1	0.140	2.900	0.001	0.040	0.050	2.8	2002.0	240.0	270.0
W1D-051	5 26 92	7.1	0.110	5.000	0.032	0.060	.	.	1648.0	320.0	410.0
W1D-051	5 28 92	7.1	0.110	5.800	0.017	0.040	.	.	2019.0	300.0	445.0
W1D-051	6 1 92	7.1	0.100	6.000	0.019	0.040	.	.	2644.0	280.0	450.0
W1D-051	6 3 92	7.1	0.110	6.600	0.018	0.050	.	.	2728.0	370.0	490.0
W1D-051	6 8 92	7.2	0.110	5.200	0.008	0.050	.	.	2753.0	345.0	450.0
W1D-051	6 10 92	7.0	0.140	3.900	0.013	0.050	.	.	2844.0	.	.
W1D-051	6 10 92	7.1	0.140	5.700	0.014	0.050	0.200	2.8	2728.0	400.0	435.0
W1D-051	6 15 92	6.7	0.100	5.300	0.019	0.060	.	.	2820.0	300.0	560.0
W1D-051	6 15 92	7.2	0.100	6.500	0.015	0.040	.	.	2728.0	.	.
W1D-051	6 17 92	.	0.110	5.500	0.006	0.040	.	.	2532.0	.	.
W1D-051	6 17 92	7.2	0.110	4.600	0.022	0.060	0.050	5.6	2836.0	300.0	400.0
W1D-051	6 22 92	7.1	0.080	4.200	0.011	0.050	.	.	2523.0	300.0	420.0
W1D-051	6 22 92	.	0.090	5.000	0.012	0.080	.	.	2567.0	.	.
W1D-051	6 24 92	7.2	0.070	2.600	0.007	0.040	.	.	2669.0	290.0	370.0
W1D-051	6 29 92	.	0.110	5.500	0.011	0.050	.	.	2610.0	310.0	370.0
W1D-051	6 29 92	.	0.110	6.100	0.011	0.100	.	.	2644.0	.	.
W1D-051	6 30 92	7.1	0.110	5.000	0.008	0.060	.	.	2753.0	300.0	450.0
W1D-051	7 6 92	.	0.080	6.100	0.015	0.110	.	.	2541.0	.	.
W1D-051	7 9 92	7.1	0.100	5.400	0.016	0.050	0.050	0.8	2532.0	310.0	410.0
W1D-051	8 3 92	7.2	0.100	3.400	0.012	0.040	.	.	3118.0	.	.
W1D-051	8 6 92	7.1	0.090	2.200	0.011	0.030	0.050	2.0	3188.0	410.0	460.0
W1D-051	8 12 92	7.3	0.090	3.000	0.016	0.040	0.100	.	2999.0	340.0	430.0
W1D-051	8 19 92	6.9	0.070	4.900	0.011	0.040	0.050	3.2	3102.0	340.0	420.0
W1D-051	8 26 92	.	0.070	4.300	0.012	0.090	.	.	2212.0	.	.
W1D-051	9 2 92	6.7	0.045	2.700	0.019	0.060	0.100	0.8	2974.0	220.0	260.0

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-051	9 4 92	.	0.060	3.200	0.008	0.060	.	.	3023.0	.	.
W1D-051	9 23 92	.	0.060	4.700	0.060	0.060	.	.	2819.0	.	.
W1D-051	9 25 92	7.1	0.060	6.000	0.050	0.060	0.100	1.2	2895.0	310.0	470.0
W1D-051	10 5 92	.	0.060	6.000	0.034	0.100	.	.	2614.0	.	.
W1D-051	10 7 92	7.1	0.070	5.800	0.028	0.060	0.050	2.8	2715.0	350.0	430.0
W1D-051	10 15 92	.	0.050	6.700	0.011	0.100	.	.	2426.0	.	.
W1D-051	10 23 92	7.0	0.070	3.300	0.026	0.050	0.050	3.6	2648.0	310.0	440.0
W1D-051	11 17 92	6.9	0.070	3.900	0.027	0.060	0.050	.	2555.0	360.0	430.0
W1D-051	11 24 92	7.0	0.080	3.300	0.019	0.070	0.050	.	2517.0	310.0	390.0
W1D-051	12 4 92	7.0	0.080	3.900	0.003	0.060	0.050	.	2316.0	390.0	430.0
W1D-051	12 14 92	7.1	0.080	3.600	0.003	0.060	0.050	.	2157.0	300.0	380.0
W1D-051	3 29 93	7.1	0.060	1.000	0.070	0.005	0.100	4.4	1252.0	70.0	90.0
W1D-051	4 8 93	6.9	0.060	1.300	0.070	0.020	0.100	.	654.0	110.0	110.0
W1D-051	4 12 93	7.0	0.070	1.300	0.070	0.020	0.050	.	1555.0	130.0	120.0
W1D-051	4 15 93	7.0	0.057	1.270	0.052	0.020	0.500	.	1253.0	140.0	130.0
W1D-051	4 19 93	7.1	0.060	1.200	0.014	0.010	0.050	.	697.0	140.0	140.0
W1D-051	4 22 93	8.3	0.060	1.200	0.002	0.010	0.050	.	1102.0	150.0	160.0
W1D-051	4 26 93	7.2	0.040	1.400	0.007	0.005	0.050	.	1030.0	160.0	160.0
W1D-051	4 29 93	7.4	0.040	0.920	0.006	0.005	0.050	.	919.0	120.0	130.0
W1D-051	5 3 93	7.3	0.040	0.860	0.004	0.050	0.050	.	779.0	90.0	90.0
W1D-051	5 6 93	6.9	0.050	0.960	0.003	0.050	0.050	.	821.0	110.0	120.0
W1D-051	5 10 93	7.5	0.060	0.800	0.003	0.050	0.050	.	892.0	130.0	130.0
W1D-051	5 13 93	7.4	0.060	1.200	0.002	0.050	0.050	.	992.0	130.0	130.0
W1D-051	5 17 93	7.4	0.040	0.860	0.001	0.010	0.050	.	1020.0	.	.
W1D-051	5 19 93	7.3	0.050	0.840	0.004	0.050	0.050	.	1188.0	180.0	170.0
W1D-051	5 24 93	7.2	0.060	0.790	0.004	0.050	0.100	.	872.0	170.0	140.0
W1D-051	5 27 93	7.2	0.040	0.900	.	0.050	0.050	.	883.0	240.0	150.0
W1D-051	6 2 93	7.2	0.040	0.940	0.001	0.050	0.050	.	1122.0	170.0	170.0
W1D-051	6 3 93	7.3	0.040	1.000	0.001	0.010	0.050	.	1068.0	180.0	170.0
W1D-051	6 7 93	.	0.050	1.100	0.001	0.060	.	.	1073.0	190.0	180.0
W1D-051	6 10 93	.	0.050	1.000	.	.	.	.	1261.0	170.0	170.0
W1D-051	6 14 93	7.3	0.050	0.930	0.005	0.060	.	.	1319.0	.	.
W1D-051	6 23 93	7.1	0.070	2.400	0.013	0.020	0.400	.	1564.0	300.0	350.0
W1D-051	6 24 93	7.2	0.036	2.300	.	.	.	.	1461.0	.	.
W1D-051	6 28 93	7.1	0.040	2.300	.	.	.	.	1573.0	.	.
W1D-051	7 1 93	7.0	0.050	3.200	.	.	.	.	1754.0	.	.
W1D-051	7 6 93	6.9	0.040	3.100	.	.	.	.	1408.0	.	.
W1D-051	7 8 93	.	0.007	4.000	.	.	.	.	2398.0	.	.
W1D-051	7 12 93	.	0.030	5.400	.	.	.	.	2087.0	.	.
W1D-051	7 14 93	6.9	0.050	6.700	0.450	0.030	0.100	.	1626.0	410.0	450.0
W1D-051	7 15 93	6.8	0.040	7.100	.	.	.	.	2330.0	.	.
W1D-051	7 19 93	6.8	0.030	7.200	.	.	.	.	2439.0	.	.
W1D-051	7 26 93	6.8	0.030	6.700	.	.	.	.	2439.0	.	.
W1D-051	7 28 93	6.8	0.030	4.100	0.410	0.030	0.100	.	2370.0	360.0	430.0
W1D-051	8 2 93	6.8	0.026	5.700	.	.	.	.	2575.0	.	.
W1D-051	8 13 93	6.9	0.050	5.500	0.024	0.050	0.100	.	2438.0	370.0	450.0
W1D-051	8 16 93	7.1	0.030	6.700	.	.	.	.	2642.0	.	.
W1D-051	8 25 93	7.1	0.050	6.000	0.010	0.060	0.100	.	622.0	390.0	460.0
W1D-051	9 15 93	7.1	0.070	5.800	0.029	0.070	0.100	.	2852.0	360.0	430.0
W1D-051	9 29 93	7.1	0.080	7.300	0.029	0.090	0.100	.	2766.0	340.0	430.0
W1D-051	10 15 93	7.0	0.080	6.300	0.014	0.070	0.100	.	2749.0	370.0	460.0
W1D-051	10 22 93	6.9	0.070	4.800	0.009	0.060	0.100	.	2525.0	310.0	420.0
W1D-051	11 3 93	7.1	0.070	5.100	0.009	0.060	0.005	.	2783.0	330.0	410.0
W1D-051	11 17 93	7.1	0.080	4.700	0.011	0.060	0.100	.	2671.0	340.0	470.0
W1D-051	12 8 93	7.0	0.090	3.800	0.019	0.060	0.100	.	2468.0	.	.
W1D-051	12 20 93	7.0	0.090	3.300	0.035	0.060	0.100	.	2435.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-051	3 17 94	6.8	0.100	1.600	0.160	0.050	0.810	.	1136.0	.	.
W1D-051	4 4 94	6.9	0.050	1.100	0.070	0.030	0.600	.	726.0	.	.
W1D-051	4 18 94	7.1	0.050	0.640	0.050	0.010	0.100	.	379.0	.	.
W1D-051	5 6 94	7.2	0.050	0.850	0.009	0.010	0.050	.	631.0	130.0	120.0
W1D-051	5 16 94	7.2	0.060	1.500	0.008	0.010	0.100	.	1555.0	220.0	260.0
W1D-051	6 2 94	7.2	0.050	2.700	0.007	0.020	0.050	.	2376.0	290.0	350.0
W1D-051	6 3 94	7.1	0.050	3.000	.	.	.	.	1292.0	.	.
W1D-051	6 8 94	6.9	0.050	2.400	.	.	.	.	2381.0	.	.
W1D-051	6 13 94	7.1	0.050	2.600	0.004	0.020	0.050	.	2341.0	310.0	380.0
W1D-051	6 15 94	7.6	0.050	1.400	.	.	.	.	950.0	.	.
W1D-051	6 22 94	7.1	0.030	4.700	.	.	.	.	2761.0	.	.
W1D-051	6 27 94	7.4	0.040	4.300	.	.	.	.	1861.0	.	.
W1D-051	7 6 94	.	0.050	7.500	.	.	.	.	2737.0	.	.
W1D-051	7 7 94	7.5	0.060	5.700	0.080	0.060	0.100	.	2803.0	360.0	440.0
W1D-051	7 11 94	7.1	0.040	5.300	.	.	.	.	2643.0	.	.
W1D-051	7 20 94	6.9	0.050	5.800	0.050	0.060	0.100	.	2450.0	300.0	410.0
W1D-051	7 21 94	.	0.050	5.900	.	.	.	.	2319.0	.	.
W1D-051	7 29 94	.	0.070	7.300	.	.	.	.	2895.0	.	.
W1D-051	8 3 94	7.1	0.070	6.300	0.030	0.070	0.100	.	2689.0	350.0	440.0
W1D-051	8 4 94	7.3	0.040	5.600	.	.	.	.	2545.0	.	.
W1D-051	8 11 94	6.1	0.040	5.700	.	.	.	.	2331.0	.	.
W1D-051	8 18 94	7.3	0.040	6.000	.	.	.	.	2379.0	.	.
W1D-051	8 24 94	6.9	0.050	5.000	0.001	0.060	0.100	.	2390.0	340.0	430.0
W1D-051	8 25 94	6.9	0.060	4.500	.	.	.	.	2308.0	.	.
W1D-051	9 2 94	.	0.050	4.700	.	.	.	.	2178.0	.	.
W1D-051	9 8 94	7.0	0.070	4.900	0.021	0.070	0.050	.	2544.0	340.0	420.0
W1D-051	9 20 94	6.9	0.050	5.700	0.080	0.070	0.050	.	2331.0	330.0	410.0
W1D-051	10 10 94	7.0	0.090	6.200	0.040	0.100	0.050	.	2493.0	345.0	430.0
W1D-051	10 12 94	.	.	6.400	.	.	.	.	.	.	.
W1D-051	10 12 94	.	.	6.400	.	.	.	.	.	.	.
W1D-051	10 19 94	7.0	0.080	5.100	0.090	0.070	0.050	.	2072.0	320.0	380.0
W1D-051	10 27 94	.	.	4.240	.	.	.	.	.	.	.
W1D-051	11 3 94	6.7	0.080	6.000	0.090	0.090	0.050	.	2257.0	340.0	410.0
W1D-051	11 17 94	6.8	0.090	5.500	0.060	0.080	0.050	.	2773.0	330.0	420.0
W1D-051	12 1 94	6.7	0.090	5.700	0.060	0.200	0.100	.	2344.0	350.0	410.0
W1D-051	12 22 94	6.7	0.090	4.800	0.021	0.070	0.100	.	2048.0	350.0	410.0
W1D-051	3 14 95	7.4	0.030	0.530	0.038	0.020	0.100	.	499.0	90.0	80.0
W1D-051	3 27 95	6.8	0.100	1.600	0.080	0.040	0.500	.	234.0	190.0	220.0
W1D-051	3 27 95	6.8	0.100	1.600	0.080	0.040	0.400	.	230.0	200.0	220.0
W1D-051	4 13 95	6.7	0.600	1.200	0.070	0.020	0.500	.	596.0	140.0	150.0
W1D-051	4 27 95	6.9	0.080	1.900	0.210	0.020	0.100	.	798.0	150.0	160.0
W1D-051	5 12 95	7.1	0.080	1.400	0.050	0.010	0.050	.	978.0	170.0	180.0
W1D-051	5 23 95	7.1	0.050	1.000	0.050	0.010	0.100	.	1617.0	180.0	200.0
W1D-051	6 7 95	7.2	0.150	1.300	0.019	0.010	0.050	.	1303.0	310.0	230.0
W1D-051	6 21 95	7.3	0.110	1.500	0.035	0.010	0.100	.	1594.0	250.0	300.0
W1D-051	7 10 95	7.2	0.060	1.200	0.017	0.010	0.050	.	1306.0	210.0	230.0
W1D-051	7 27 95	7.1	0.040	2.400	0.040	0.020	0.100	.	2126.0	300.0	380.0
W1D-051	8 10 95	6.9	0.040	2.300	0.024	0.030	0.100	.	2236.0	356.0	409.0
W1D-051	8 21 95	7.0	0.070	2.000	0.017	0.010	0.100	.	1857.0	270.0	317.0
W1D-051	9 11 95	7.0	0.070	1.900	0.026	0.020	0.100	.	1796.0	284.0	313.0
W1D-051	9 25 95	7.0	0.040	2.500	0.037	0.030	0.100	.	2266.0	321.0	350.0
W1D-051	10 10 95	7.2	0.050	2.700	0.023	0.030	0.050	.	1937.0	290.0	310.0
W1D-051	10 23 95	7.2	0.110	3.200	0.020	0.040	0.100	.	2449.0	310.0	170.0
W1D-051	1 4 96	7.0	0.080	2.000	0.040	0.030	0.100	.	3001.0	350.0	320.0
W1D-051	4 29 96	7.5	0.046	0.853	0.019	0.029	0.060	.	809.0	.	.
W1D-051	5 9 96	7.2	0.031	0.955	0.007	0.005	0.070	.	1100.0	135.0	258.0

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-051	5 22 96	7.3	0.034	0.790	0.007	0.023	0.089	.	1100.0	165.0	161.0
W1D-051	6 7 96	7.2	0.053	1.060	0.005	0.031	0.060	.	1430.0	243.0	246.0
W1D-051	6 19 96	7.1	0.053	1.330	0.005	0.031	0.070	.	1620.0	292.0	277.0
W1D-051	7 2 96	7.1	0.033	0.908	0.009	0.021	0.100	.	1270.0	216.0	222.0
W1D-051	7 15 96	7.0	0.038	0.927	0.005	0.027	0.070	.	1630.0	222.0	231.0
W1D-051	8 6 96	7.1	0.013	0.624	0.002	0.018	0.290	.	1750.0	248.0	269.0
W1D-051	8 20 96	7.0	0.010	0.385	0.003	0.011	0.040	.	2130.0	326.0	308.0
W1D-051	9 4 96	7.1	0.037	0.999	0.001	0.027	0.080	.	1630.0	308.0	314.0
W1D-051	9 19 96	7.2	0.025	0.844	0.006	0.025	0.110	.	2045.0	322.0	301.0
W1D-051	10 2 96	7.2	0.026	1.380	0.009	0.035	0.070	.	1780.0	308.0	285.0
W1D-051	10 21 96	7.2	0.066	1.575	0.004	0.048	0.100	.	1780.0	291.0	278.0
W1D-051	11 6 96	7.3	0.054	1.795	0.012	0.049	0.080	.	1600.0	278.0	265.0
W1D-051	11 18 96	7.3	0.043	0.958	0.010	0.022	0.040	.	722.0	115.0	110.0
W1D-051	12 2 96	7.3	0.059	2.000	0.010	0.055	0.040	.	1460.0	277.0	255.0
W1D-051	12 16 96	7.1	0.069	1.720	0.008	0.056	0.040	.	1910.0	293.0	276.0
W1D-051	4 14 97	7.1	0.043	1.060	0.045	0.028	0.040	.	700.0	113.0	105.0
W1D-051	4 22 97	7.2	0.015	1.300	0.019	0.028	0.080	.	828.0	109.0	104.0
W1D-051	5 7 97	7.2	0.033	0.754	0.005	0.022	0.070	.	860.0	154.0	141.0
W1D-051	5 21 97	7.2	0.031	0.571	0.001	0.012	0.040	.	924.0	158.0	147.0
W1D-051	6 9 97	7.2	0.043	0.643	0.005	0.015	0.060	.	975.0	191.0	174.0
W1D-051	6 16 97	7.2	0.033	0.726	0.004	0.005	0.050	.	1160.0	206.0	183.0
W1D-051	7 10 97	7.1	0.025	0.658	0.002	0.022	0.070	.	1020.0	210.0	179.0
W1D-051	7 21 97	7.2	0.048	0.797	0.013	0.020	0.070	.	1100.0	232.0	189.0
W1D-051	8 4 97	7.3	0.041	0.700	0.009	0.021	0.030	.	1080.0	223.0	191.0
W1D-051	8 18 97	7.2	0.007	0.144	0.013	0.005	0.070	.	1070.0	224.0	202.0
W1D-051	9 4 97	7.3	0.016	0.354	0.012	0.005	0.110	.	1080.0	237.0	215.0
W1D-051	9 23 97	7.1	0.017	0.305	0.009	0.012	0.080	.	1210.0	246.0	218.0
W1D-051	10 6 97	7.3	0.027	0.401	0.005	0.020	0.050	.	1110.0	244.0	223.0
W1D-051	10 21 97	7.2	0.018	0.550	0.016	0.013	0.015	.	1010.0	202.0	194.0
W1D-051	11 3 97	7.1	0.023	0.755	0.017	0.014	0.070	.	846.0	194.0	165.0
W1D-051	11 20 97	7.0	0.047	1.150	0.020	0.023	0.050	.	1100.0	226.0	201.0
W1D-051	12 3 97	7.0	0.041	0.865	0.015	0.024	0.050	.	837.0	209.0	186.0
W1D-051	12 18 97	7.1	0.037	0.750	0.007	0.019	0.050	.	1150.0	221.0	194.0
W1D-051	3 3 98	7.1	0.018	0.748	0.011	0.014	-0.030	.	586.0	105.0	105.0
W1D-051	4 9 98	7.0	0.017	0.675	0.005	0.017	-0.030	.	605.0	113.0	99.8
W1D-051	4 22 98	6.9	0.037	0.566	0.002	-0.010	0.040	.	887.0	140.0	129.0
W1D-051	5 7 98	7.1	0.053	0.589	0.002	0.014	0.030	.	869.0	172.0	164.0
W1D-051	5 18 98	7.1	0.035	0.417	0.003	-0.010	.	.	719.0	146.0	129.0
W1D-051	6 3 98	7.3	0.026	0.393	0.002	0.016	0.090	.	1040.0	203.0	183.0
W1D-051	6 16 98	7.3	0.024	0.378	0.006	0.022	-0.030	.	1060.0	202.0	182.0
W1D-051	7 9 98	7.2	0.025	0.373	0.008	0.015	0.060	.	1030.0	230.0	206.0
W1D-051	7 21 98	7.3	0.049	0.551	0.007	0.018	0.060	.	1135.0	238.0	213.0
W1D-051	8 4 98	7.3	0.047	0.381	0.024	0.015	0.030	.	1250.0	263.0	237.0
W1D-051	8 18 98	7.4	0.031	0.414	-0.001	0.020	-0.030	.	881.0	200.0	176.0
W1D-051	9 10 98	7.4	0.014	0.170	0.013	0.012	0.070	.	1130.0	243.0	222.0
W1D-051	9 25 98	7.3	0.015	0.209	0.009	0.014	0.040	.	1140.0	247.0	225.0
W1D-051	10 8 98	7.4	0.014	0.318	0.015	0.020	0.140	.	1010.0	232.0	208.0
W1D-051	10 21 98	7.3	0.024	0.546	0.013	0.167	0.040	.	708.0	151.0	125.0
W1D-051	11 4 98	7.3	0.026	0.653	0.015	0.018	-0.030	.	1250.0	209.0	194.0
W1D-051	11 17 98	7.3	0.031	0.590	0.011	0.017	0.050	.	1100.0	228.0	206.0
W1D-051	12 2 98	7.2	0.035	0.565	0.004	0.027	-0.030	.	948.0	207.0	184.0
W1D-051	12 14 98	7.2	0.069	0.799	0.032	0.031	0.050	.	1010.0	201.0	186.0
W1D-051	4 8 99	6.8	0.034	0.400	0.010	0.020	0.040	.	378.0	74.4	61.7
W1D-051	4 19 99	7.3	0.023	0.505	0.006	0.031	0.060	.	663.0	124.0	114.0
W1D-051	5 3 99	7.1	0.032	0.388	0.002	0.015	0.040	.	895.0	177.0	161.0
W1D-051	5 19 99	7.1	0.024	0.450	0.005	0.022	0.050	.	977.0	196.0	174.0

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-051	6 3 99	7.2	0.022	0.390	0.007	0.016	0.090	.	1060.0	206.0	194.0
W1D-051	6 19 99	6.9	0.009	0.215	0.012	0.014	-0.030	.	1170.0	222.0	212.0
W1D-051	7 8 99	7.1	0.008	0.322	0.004	0.017	0.410	.	897.0	152.0	138.0
W1D-051	7 20 99	7.0	0.004	0.187	0.002	0.015	0.100	.	1190.0	210.0	197.0
W1D-051	8 17 99	7.1	0.008	0.295	0.002	-0.010	0.050	.	1360.0	262.0	242.0
W1D-051	8 23 99	7.2	0.001	0.030	0.002	-0.010	0.090	.	1570.0	256.0	244.0
W1D-051	9 3 99	6.9	0.003	0.220	0.001	0.010	.	.	1550.0	284.0	264.0
W1D-051	9 16 99	7.3	0.005	0.473	0.002	-0.010	.	.	1320.0	239.0	224.0
W1D-051	10 5 99	7.1	0.011	1.290	0.013	0.030	.	.	1170.0	237.0	211.0
W1D-051	10 19 99	7.3	0.017	1.620	0.012	0.026	.	.	1110.0	243.0	230.0
W1D-051	11 2 99	7.3	0.023	1.250	0.009	0.026	.	.	1560.0	259.0	259.0
W1D-051	11 17 99	7.1	0.015	1.690	0.011	0.030	.	.	1340.0	282.0	258.0

Outfall (W1D-052: outfall from original treatment system)

W1D-052	4 29 92	6.7	0.011	0.250	0.017	0.005	.	.	817.0	95.0	90.0
W1D-052	5 4 92	.	0.010	0.170	0.016	0.010	.	.	.	105.0	110.0
W1D-052	5 6 92	6.8	0.022	0.260	0.018	0.010	1.600	20.4	961.0	.	.
W1D-052	5 11 92	6.9	0.032	0.530	0.040	0.020	.	.	1095.0	170.0	160.0
W1D-052	5 13 92	6.9	0.018	0.190	0.025	0.005	.	.	985.0	120.0	106.0
W1D-052	5 15 92	6.7	0.017	0.250	0.040	0.005	.	.	911.0	71.0	90.0
W1D-052	5 18 92	7.0	0.019	0.420	0.040	0.020	.	.	1067.0	113.0	111.0
W1D-052	5 21 92	6.9	0.020	0.330	0.008	0.005	4.200	19.6	981.0	120.0	95.0
W1D-052	5 26 92	6.9	0.016	0.400	0.050	0.005	.	.	1123.0	156.0	160.0
W1D-052	5 28 92	6.9	0.012	0.200	0.060	0.005	.	.	1091.0	130.0	145.0
W1D-052	6 1 92	7.0	0.011	0.360	0.030	0.005	.	.	1244.0	150.0	210.0
W1D-052	6 3 92	7.0	0.020	0.500	0.040	0.005	.	.	1539.0	220.0	260.0
W1D-052	6 8 92	7.0	0.012	0.270	0.030	0.005	.	.	1389.0	225.0	245.0
W1D-052	6 10 92	7.2	0.015	0.180	0.040	0.010	.	.	1932.0	.	.
W1D-052	6 10 92	6.9	0.003	0.290	0.003	0.005	5.100	16.4	1339.0	225.0	230.0
W1D-052	6 15 92	6.8	0.019	0.130	0.027	0.005	.	.	1882.0	.	.
W1D-052	6 15 92	6.9	0.009	0.160	0.026	0.005	.	.	1418.0	210.0	310.0
W1D-052	6 17 92	6.4	0.000	0.220	0.005	0.010	.	.	2172.0	.	.
W1D-052	6 17 92	7.1	0.019	0.320	0.015	0.005	0.900	30.8	2153.0	240.0	310.0
W1D-052	6 22 92	7.0	0.014	0.450	0.017	0.005	.	.	1739.0	210.0	280.0
W1D-052	6 22 92	7.0	0.006	0.460	0.012	0.060	.	.	1961.0	.	.
W1D-052	6 24 92	7.1	0.017	0.450	0.013	0.005	.	.	2134.0	220.0	280.0
W1D-052	6 29 92	7.1	0.002	0.280	0.013	0.050	.	.	1872.0	.	.
W1D-052	6 29 92	7.1	0.005	0.250	0.011	0.005	.	.	1831.0	210.0	240.0
W1D-052	6 30 92	7.0	0.004	0.020	0.014	0.005	.	.	1941.0	200.0	270.0
W1D-052	7 6 92	7.0	0.005	0.580	0.008	0.060	.	.	1852.0	.	.
W1D-052	7 9 92	7.0	0.010	0.330	0.009	0.005	0.700	10.4	2000.0	200.0	260.0
W1D-052	7 17 92	.	0.011	0.300	0.014	0.005	2.200	.	1464.0	200.0	250.0
W1D-052	7 31 92	.	0.014	0.080	0.010	0.005	0.200	.	.	190.0	230.0
W1D-052	8 3 92	7.2	0.013	0.090	0.010	0.005	.	.	2144.0	.	.
W1D-052	8 6 92	7.5	0.019	0.080	0.009	0.005	0.100	9.6	2068.0	200.0	280.0
W1D-052	8 11 92	.	0.026	0.150	0.009	0.005	0.100	.	2181.0	230.0	290.0
W1D-052	8 12 92	7.2	0.011	0.140	0.013	0.005	0.100	.	2200.0	220.0	290.0
W1D-052	8 17 92	7.1	0.008	0.100	0.009	0.005	0.300	.	2400.0	200.0	260.0
W1D-052	8 19 92	7.3	0.006	0.100	0.009	0.005	0.200	.	2228.0	220.0	270.0
W1D-052	8 26 92	6.7	0.017	0.140	0.009	0.060	.	.	1922.0	.	.
W1D-052	8 28 92	6.4	0.008	0.100	0.001	0.010	.	.	2733.0	.	.
W1D-052	9 1 92	7.2	0.006	0.500	0.004	0.005	.	.	2811.0	.	.
W1D-052	9 2 92	7.2	0.004	0.530	0.006	0.040	1.000	.	.	170.0	210.0

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-052	9 4 92	7.1	0.008	0.550	0.005	0.040	.	.	2627.0	.	.
W1D-052	9 17 92	7.2	0.014	0.062	0.003	0.005	.	.	.	.	.
W1D-052	9 23 92	.	0.003	0.490	0.003	0.020	.	.	2474.0	.	.
W1D-052	9 25 92	7.6	0.010	0.590	0.004	0.005	0.900	10.4	2460.0	230.0	320.0
W1D-052	10 5 92	7.2	0.004	0.560	0.003	0.040	.	.	2323.0	.	.
W1D-052	10 7 92	7.3	0.007	0.380	0.004	0.005	0.200	5.2	2382.0	210.0	270.0
W1D-052	10 15 92	.	0.002	0.510	0.011	0.040	.	.	2175.0	.	.
W1D-052	10 23 92	7.3	0.013	0.430	0.011	0.005	0.050	6.4	2211.0	240.0	350.0
W1D-052	10 28 92	7.3	0.002	0.440	0.007	0.005	.	.	2809.0	.	.
W1D-052	11 17 92	7.0	0.022	0.520	0.016	0.005	0.200	2.0	2280.0	280.0	320.0
W1D-052	11 24 92	7.0	0.022	0.530	0.019	0.010	0.200	2.0	2212.0	250.0	300.0
W1D-052	12 4 92	7.0	0.010	0.410	0.007	0.010	0.050	0.8	1348.0	300.0	320.0
W1D-052	12 14 92	7.1	0.005	0.330	0.010	0.005	0.200	4.4	1481.0	240.0	300.0
W1D-052	1 21 93	7.1	0.009	0.140	0.002	0.020	0.300	3.6	1341.0	240.0	260.0
W1D-052	1 27 93	7.1	0.008	0.120	0.003	0.005	0.300	2.4	1307.0	210.0	260.0
W1D-052	2 9 93	7.2	0.004	0.120	0.005	0.005	0.200	6.4	1320.0	240.0	270.0
W1D-052	3 29 93	7.0	0.015	0.310	0.010	0.005	0.400	6.8	1001.0	80.0	110.0
W1D-052	4 8 93	7.0	0.024	0.170	0.007	0.005	0.200	21.6	1294.0	150.0	170.0
W1D-052	4 12 93	6.9	0.018	0.220	0.008	0.005	0.200	3.6	601.0	130.0	140.0
W1D-052	4 15 93	7.0	0.017	0.160	0.005	0.005	0.300	7.6	583.0	150.0	150.0
W1D-052	4 19 93	6.9	0.012	0.100	0.004	0.005	0.200	10.0	1287.0	160.0	170.0
W1D-052	4 22 93	7.2	0.009	0.110	0.008	0.005	0.300	0.4	1358.0	170.0	180.0
W1D-052	4 26 93	7.0	0.015	0.140	0.003	0.005	0.500	8.8	1047.0	80.0	70.0
W1D-052	4 29 93	7.2	0.012	0.170	0.002	0.005	0.050	4.0	988.0	120.0	140.0
W1D-052	5 3 93	7.4	0.015	0.240	0.002	0.005	0.500	4.8	817.0	80.0	91.0
W1D-052	5 6 93	7.0	0.012	0.120	0.003	0.005	0.400	1.6	888.0	100.0	110.0
W1D-052	5 10 93	7.3	0.015	0.120	0.004	0.005	0.400	7.2	922.0	130.0	140.0
W1D-052	5 13 93	7.3	0.011	0.090	0.006	0.005	0.600	7.6	977.0	120.0	120.0
W1D-052	5 17 93	7.3	0.016	0.080	0.003	0.005	0.200	6.4	995.0	.	.
W1D-052	5 19 93	7.3	0.017	0.100	0.001	0.005	0.200	10.0	1174.0	180.0	180.0
W1D-052	5 24 93	7.3	0.018	0.130	0.001	0.005	0.300	14.8	746.0	150.0	140.0
W1D-052	5 27 93	7.2	0.006	0.090	0.002	0.005	0.200	5.6	774.0	190.0	130.0
W1D-052	6 2 93	7.2	0.018	0.050	0.003	0.005	0.200	8.4	855.0	140.0	140.0
W1D-052	6 3 93	7.3	0.007	0.050	0.002	0.005	0.100	6.8	839.0	140.0	140.0
W1D-052	6 7 93	.	0.007	0.100	0.002	0.050	.	.	714.0	120.0	130.0
W1D-052	6 10 93	7.2	0.007	0.050	0.001	0.040	.	.	962.0	120.0	130.0
W1D-052	6 14 93	7.4	0.004	0.090	0.002	0.040	.	.	621.0	.	.
W1D-052	6 23 93	7.2	0.005	0.044	0.003	0.005	0.100	9.6	925.0	150.0	150.0
W1D-052	6 24 93	7.5	0.005	0.070	.	.	.	.	628.0	.	.
W1D-052	6 28 93	7.1	0.001	0.050	.	.	.	.	628.0	.	.
W1D-052	7 1 93	7.1	0.006	0.050	.	.	.	.	640.0	.	.
W1D-052	7 6 93	7.1	0.005	0.370	.	.	.	.	1113.0	.	.
W1D-052	7 8 93	.	0.005	0.410	.	.	.	.	1294.0	.	.
W1D-052	7 12 93	7.0	0.003	0.680	.	.	.	.	1368.0	.	.
W1D-052	7 14 93	7.0	0.012	0.520	0.002	0.005	0.300	4.4	1269.0	270.0	290.0
W1D-052	7 15 93	6.8	0.005	0.560	.	.	.	.	1479.0	.	.
W1D-052	7 19 93	6.8	0.008	1.100	.	.	.	.	1464.0	.	.
W1D-052	7 19 93	6.8	0.013	1.200	0.005	0.010	0.200	.	1464.0	230.0	290.0
W1D-052	7 26 93	6.8	0.004	0.810	.	.	.	.	1848.0	.	.
W1D-052	7 28 93	7.1	0.008	0.560	0.002	0.005	0.200	7.6	1734.0	270.0	320.0
W1D-052	8 2 93	7.1	0.004	0.740	.	.	.	.	1665.0	.	.
W1D-052	8 13 93	7.1	0.005	0.720	0.003	0.005	0.200	3.6	1818.0	290.0	350.0
W1D-052	8 16 93	7.3	0.005	0.420	.	.	.	.	1766.0	.	.
W1D-052	8 25 93	7.3	-0.001	0.370	0.003	0.005	0.200	10.8	1964.0	320.0	370.0
W1D-052	9 15 93	7.4	0.001	0.750	0.003	0.005	0.200	5.2	1846.0	260.0	320.0
W1D-052	9 29 93	7.5	0.005	0.910	0.001	0.005	0.100	4.4	2128.0	280.0	360.0

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-052	10 15 93	7.3	0.003	0.770	0.002	0.005	0.100	5.2	2194.0	310.0	390.0
W1D-052	10 22 93	7.3	0.009	0.790	0.009	0.005	0.100	5.6	1992.0	270.0	370.0
W1D-052	11 3 93	7.2	0.004	0.920	0.003	0.005	0.300	10.2	2043.0	290.0	370.0
W1D-052	11 17 93	7.1	0.004	0.680	0.003	0.005	0.500	13.2	2194.0	300.0	420.0
W1D-052	12 8 93	7.1	0.004	0.440	0.007	0.005	1.000	10.8	2221.0	270.0	350.0
W1D-052	12 20 93	7.0	0.004	0.320	0.008	0.005	1.400	15.6	1956.0	230.0	300.0
W1D-052	3 11 94	7.1	0.006	0.510	0.021	0.005	0.780	8.0	1265.0	190.0	220.0
W1D-052	4 4 94	7.1	0.001	0.330	0.006	0.005	0.900	7.2	926.0	150.0	170.0
W1D-052	4 18 94	7.5	0.011	0.280	0.006	0.005	0.200	2.4	338.0	70.0	70.0
W1D-052	5 2 94	.	.	0.209	.	.	.	.	.	.	.
W1D-052	5 5 94	.	.	0.168	.	.	.	.	.	.	.
W1D-052	5 6 94	8.2	0.007	0.190	0.001	0.010	0.100	2.0	548.0	100.0	110.0
W1D-052	5 9 94	.	.	0.119	.	.	.	.	.	.	.
W1D-052	5 12 94	.	.	0.129	.	.	.	.	.	.	.
W1D-052	5 16 94	7.8	0.005	0.168	0.001	0.005	0.100	1.0	693.0	110.0	130.0
W1D-052	5 19 94	.	.	0.128	.	.	.	.	.	.	.
W1D-052	5 23 94	.	.	0.203	.	.	.	.	.	.	.
W1D-052	5 27 94	.	.	0.197	.	.	.	.	.	.	.
W1D-052	5 31 94	.	.	0.139	.	.	.	.	.	.	.
W1D-052	6 2 94	7.7	0.003	0.100	0.002	0.005	0.100	8.4	906.0	150.0	180.0
W1D-052	6 3 94	7.6	0.002	0.100	.	.	.	.	1015.0	.	.
W1D-052	6 6 94	.	.	0.093	.	.	.	.	.	.	.
W1D-052	6 8 94	7.4	0.004	0.100	.	.	.	.	974.0	.	.
W1D-052	6 9 94	.	.	0.064	.	.	.	.	.	.	.
W1D-052	6 13 94	7.8	0.003	0.070	0.001	0.005	0.100	19.6	892.0	160.0	190.0
W1D-052	6 15 94	.	0.005	0.156	.	.	.	.	1182.0	.	.
W1D-052	6 16 94	.	.	0.137	.	.	.	.	.	.	.
W1D-052	6 20 94	.	.	0.534	.	.	.	.	.	.	.
W1D-052	6 22 94	7.1	0.006	0.280	.	.	.	.	1832.0	.	.
W1D-052	6 23 94	.	.	0.317	.	.	.	.	.	.	.
W1D-052	6 27 94	7.4	0.005	0.630	.	.	.	.	1286.0	.	.
W1D-052	6 27 94	.	.	0.697	.	.	.	.	.	.	.
W1D-052	6 30 94	.	.	0.646	.	.	.	.	.	.	.
W1D-052	7 5 94	.	.	0.201	.	.	.	.	.	.	.
W1D-052	7 6 94	7.3	0.016	0.140	.	.	.	.	1404.0	.	.
W1D-052	7 7 94	7.5	0.005	0.107	0.002	0.005	0.200	7.6	1807.0	270.0	320.0
W1D-052	7 11 94	.	.	0.293	.	.	.	.	.	.	.
W1D-052	7 11 94	7.5	0.005	0.280	.	.	.	.	1693.0	.	.
W1D-052	7 14 94	.	.	0.325	.	.	.	.	.	.	.
W1D-052	7 18 94	.	.	0.180	.	.	.	.	.	.	.
W1D-052	7 20 94	7.0	0.004	0.122	0.002	0.005	0.200	6.8	1598.0	230.0	310.0
W1D-052	7 21 94	.	.	0.103	.	.	.	.	.	.	.
W1D-052	7 21 94	7.2	0.006	0.090	.	.	.	.	1725.0	.	.
W1D-052	7 25 94	.	.	0.067	.	.	.	.	.	.	.
W1D-052	7 28 94	.	.	0.087	.	.	.	.	.	.	.
W1D-052	7 29 94	7.2	0.003	0.007	.	.	.	.	1987.0	.	.
W1D-052	8 1 94	.	.	0.050	.	.	.	.	.	.	.
W1D-052	8 3 94	7.3	0.009	0.050	0.002	0.005	0.100	8.0	1906.0	270.0	330.0
W1D-052	8 4 94	.	.	0.066	.	.	.	.	.	.	.
W1D-052	8 4 94	.	0.003	0.050	.	.	.	.	1973.0	.	.
W1D-052	8 8 94	.	.	0.365	.	.	.	.	.	.	.
W1D-052	8 11 94	.	.	0.296	.	.	.	.	.	.	.
W1D-052	8 11 94	7.4	0.003	0.310	.	.	.	.	1737.0	.	.
W1D-052	8 15 94	.	.	0.102	.	.	.	.	.	.	.
W1D-052	8 18 94	7.4	0.002	0.060	.	.	.	.	1840.0	.	.
W1D-052	8 18 94	.	.	0.056	.	.	.	.	.	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-052	8 22 94	.	.	0.052	.	.	.	.	.	.	.
W1D-052	8 24 94	7.3	0.002	.	0.002	0.005	0.100	8.8	1832.0	260.0	340.0
W1D-052	8 25 94	.	.	0.050	.	.	.	.	.	.	.
W1D-052	8 25 94	7.3	0.004	0.040	.	.	.	.	1599.0	.	.
W1D-052	8 29 94	.	.	0.041	.	.	.	.	.	.	.
W1D-052	9 2 94	.	.	0.501	.	.	.	.	.	.	.
W1D-052	9 2 94	7.3	0.002	0.380	.	.	.	.	2003.0	.	.
W1D-052	9 6 94	.	.	0.196	.	.	.	.	.	.	.
W1D-052	9 8 94	7.3	0.002	0.127	0.003	0.010	0.000	1.6	2071.0	280.0	350.0
W1D-052	9 12 94	.	.	0.068	.	.	.	.	.	.	.
W1D-052	9 15 94	.	.	0.535	.	.	.	.	.	.	.
W1D-052	9 20 94	7.3	0.006	0.506	0.002	0.005	0.100	4.4	1508.0	240.0	290.0
W1D-052	9 22 94	.	.	0.478	.	.	.	.	.	.	.
W1D-052	9 26 94	.	.	0.351	.	.	.	.	.	.	.
W1D-052	9 29 94	.	.	0.605	.	.	.	.	.	.	.
W1D-052	10 3 94	.	.	0.329	.	.	.	.	.	.	.
W1D-052	10 6 94	.	.	0.931	.	.	.	.	.	.	.
W1D-052	10 10 94	7.3	0.006	0.770	0.001	0.010	0.100	2.8	1754.0	250.0	310.0
W1D-052	10 12 94	.	.	0.854	.	.	.	.	.	.	.
W1D-052	10 12 94	.	.	0.854	.	.	.	.	.	.	.
W1D-052	10 13 94	.	.	0.858	.	.	.	.	.	.	.
W1D-052	10 17 94	.	.	0.580	.	.	.	.	.	.	.
W1D-052	10 19 94	7.4	0.005	0.660	0.001	0.010	0.100	3.2	1774.0	260.0	330.0
W1D-052	10 20 94	.	.	0.661	.	.	.	.	.	.	.
W1D-052	10 24 94	.	.	0.696	.	.	.	.	.	.	.
W1D-052	10 27 94	.	.	0.897	.	.	.	.	.	.	.
W1D-052	10 27 94	.	.	0.897	.	.	.	.	.	.	.
W1D-052	10 31 94	.	.	0.671	.	.	.	.	.	.	.
W1D-052	11 3 94	7.1	0.023	0.840	0.002	0.005	0.000	4.0	1417.0	230.0	290.0
W1D-052	11 7 94	.	.	1.009	.	.	.	.	.	.	.
W1D-052	11 10 94	.	.	0.917	.	.	.	.	.	.	.
W1D-052	11 17 94	7.2	0.060	1.100	0.001	0.010	0.000	1.6	1992.0	260.0	340.0
W1D-052	11 18 94	.	.	1.260	.	.	.	.	.	.	.
W1D-052	11 21 94	.	.	1.790	.	.	.	.	.	.	.
W1D-052	11 23 94	.	.	1.610	.	.	.	.	.	.	.
W1D-052	11 28 94	.	.	1.780	.	.	.	.	.	.	.
W1D-052	12 1 94	6.9	0.011	1.800	0.004	0.200	0.200	6.0	2208.0	300.0	370.0
W1D-052	12 22 94	7.1	0.008	1.300	0.007	0.020	0.600	9.6	2100.0	310.0	380.0
W1D-052	3 14 95	7.6	0.001	0.420	0.007	0.010	0.100	.	382.0	60.0	60.0
W1D-052	3 27 95	7.2	0.007	0.810	0.001	0.005	0.050	.	671.0	120.0	140.0
W1D-052	4 13 95	6.9	0.002	0.880	0.001	0.010	0.050	.	713.0	120.0	160.0
W1D-052	4 27 95	7.1	0.002	0.460	0.002	0.005	0.100	.	819.0	140.0	180.0
W1D-052	5 12 95	7.4	0.005	0.220	0.002	0.005	0.050	.	989.0	170.0	200.0
W1D-052	5 23 95	7.4	0.003	0.160	0.001	0.100	0.100	4.4	1507.0	140.0	170.0
W1D-052	6 7 95	7.2	0.003	0.130	0.002	0.005	0.050	8.8	1017.0	170.0	200.0
W1D-052	7 10 95	7.4	0.005	0.690	0.006	0.100	0.100	-0.4	1008.0	170.0	180.0
W1D-052	7 27 95	7.3	0.003	0.360	0.004	0.100	0.100	2.8	1243.0	190.0	220.0
W1D-052	8 10 95	7.2	0.003	0.150	0.007	0.100	0.100	5.6	1545.0	285.0	319.0
W1D-052	8 21 95	7.4	0.003	0.120	0.020	0.100	0.100	5.2	1658.0	274.0	319.0
W1D-052	9 11 95	7.4	0.002	0.240	0.002	0.005	0.050	0.4	1218.0	195.0	218.0
W1D-052	9 25 95	7.4	0.002	0.450	0.006	0.005	0.050	0.4	1573.0	252.0	283.0
W1D-052	1 4 96	7.0	-0.001	0.080	0.007	-0.010	1.800	13.6	1361.0	250.0	260.0
W1D-052	4 29 96	7.6	0.010	0.599	-0.001	0.019	0.060	-1.0	679.0	.	.
W1D-052	5 9 96	7.4	0.004	0.644	-0.001	-0.010	0.100	1.6	2380.0	106.0	139.0
W1D-052	5 22 96	7.4	0.003	0.543	0.002	0.015	0.140	1.2	814.0	115.0	116.0
W1D-052	6 7 96	7.3	0.015	0.372	-0.001	0.011	0.060	2.8	1340.0	176.0	161.0

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-052	7 2 96	7.4	0.008	0.336	-0.001	-0.010	0.080	3.2	857.0	140.0	123.0
W1D-052	7 15 96	7.3	0.011	0.207	0.002	-0.010	0.060	1.2	1290.0	193.0	176.0
W1D-052	8 6 96	7.3	-0.001	0.181	-0.001	-0.010	0.290	3.2	1460.0	247.0	234.0
W1D-052	8 20 96	7.0	0.008	0.096	0.007	-0.010	0.050	17.2	1460.0	234.0	235.0
W1D-052	9 4 96	7.2	0.002	0.089	0.002	-0.010	0.100	8.8	1580.0	241.0	229.0
W1D-052	9 19 96	7.3	0.005	0.056	-0.001	-0.010	0.090	6.4	1490.0	272.0	258.0
W1D-052	10 2 96	7.5	0.033	0.245	0.003	0.018	0.090	4.0	1550.0	217.0	251.0
W1D-052	10 21 96	7.4	0.026	0.261	0.002	0.022	0.080	4.4	1530.0	248.0	245.0
W1D-052	11 6 96	7.4	0.009	0.449	-0.001	0.022	0.080	2.0	1010.0	147.0	141.0
W1D-052	11 18 96	7.4	0.009	0.891	0.005	0.025	0.120	2.0	1060.0	170.0	169.0
W1D-052	4 14 97	7.2	0.004	0.580	0.004	0.017	0.090	1.2	625.0	88.0	86.5
W1D-052	4 22 97	7.3	-0.001	0.355	0.001	0.005	0.060	2.0	566.0	66.7	58.0
W1D-052	5 7 97	7.2	0.003	0.266	0.001	0.011	0.070	1.2	750.0	119.0	115.0
W1D-052	5 21 97	7.2	-0.001	0.253	0.001	0.005	0.090	1.2	816.0	112.0	116.0
W1D-052	6 9 97	7.1	0.001	0.102	0.001	0.005	0.080	4.4	832.0	142.0	143.0
W1D-052	6 16 97	7.4	0.002	0.077	0.001	0.016	0.050	4.8	684.0	160.0	152.0
W1D-052	7 10 97	7.2	0.002	0.143	0.001	0.012	0.070	4.8	638.0	145.0	135.0
W1D-052	7 21 97	7.5	-0.001	0.076	0.001	0.005	0.050	2.0	882.0	149.0	148.0
W1D-052	8 4 97	7.5	0.002	0.082	0.001	0.011	0.050	6.4	808.0	143.0	135.0
W1D-052	8 18 97	7.6	0.001	0.051	0.001	0.005	0.015	2.8	752.0	162.0	156.0
W1D-052	9 4 97	7.5	-0.001	0.063	0.001	0.005	0.040	2.8	892.0	175.0	174.0
W1D-052	9 23 97	7.6	-0.001	0.053	0.001	0.005	0.060	2.4	1000.0	170.0	169.0
W1D-052	10 6 97	7.4	0.001	0.067	0.001	0.005	0.060	2.4	978.0	188.0	184.0
W1D-052	10 21 97	7.2	0.001	0.313	0.001	0.005	0.050	4.0	888.0	180.0	174.0
W1D-052	11 3 97	7.2	0.002	0.299	0.001	0.005	0.050	1.2	659.0	139.0	124.0
W1D-052	11 20 97	7.2	0.003	0.196	0.001	0.005	0.015	4.4	904.0	175.0	164.0
W1D-052	12 3 97	7.1	-0.001	0.050	0.001	0.005	0.040	5.6	769.0	181.0	181.0
W1D-052	3 3 98	7.3	0.012	0.640	0.006	0.030	0.110	2.5	654.0	104.0	106.0
W1D-052	4 9 98	7.2	0.003	0.299	-0.001	0.013	-0.030	1.6	421.0	90.1	84.7
W1D-052	4 22 98	7.2	0.002	0.104	-0.001	-0.010	-0.030	6.0	748.0	116.0	115.0
W1D-052	5 7 98	7.3	-0.001	0.048	-0.001	-0.010	-0.030	10.0	804.0	144.0	150.0
W1D-052	5 18 98	7.3	0.005	0.267	-0.001	-0.010	0.040	3.0	699.0	119.0	117.0
W1D-052	6 3 98	7.4	-0.001	0.069	-0.001	0.013	0.060	4.0	856.0	149.0	142.0
W1D-052	6 16 98	7.3	0.002	0.046	-0.001	0.012	-0.030	4.0	618.0	109.0	108.0
W1D-052	7 9 98	7.3	-0.001	0.024	-0.001	-0.010	0.040	6.0	808.0	151.0	149.0
W1D-052	7 21 98	.	.	.	.	.	.	.	.	.	.
W1D-052	8 18 98	7.4	0.005	0.009	-0.001	0.026	-0.030	5.0	574.0	111.0	107.0
W1D-052	9 10 98	7.4	-0.001	0.017	-0.001	-0.010	0.030	2.4	1030.0	192.0	195.0
W1D-052	9 25 98	7.6	-0.001	0.017	-0.001	0.011	-0.030	1.2	845.0	182.0	188.0
W1D-052	10 8 98	7.4	0.002	0.079	0.002	0.015	0.080	4.5	952.0	191.0	198.0
W1D-052	10 21 98	7.3	0.003	0.206	0.001	0.015	0.130	1.6	730.0	131.0	126.0
W1D-052	11 4 98	7.1	-0.001	0.085	0.002	-0.010	0.050	4.0	1160.0	175.0	179.0
W1D-052	11 17 98	7.2	-0.001	0.047	0.001	-0.010	0.060	4.0	951.0	179.0	182.0
W1D-052	12 2 98	7.1	0.002	0.057	0.002	0.019	-0.030	5.2	893.0	165.0	167.0
W1D-052	12 14 98	7.2	-0.001	0.043	0.002	0.013	0.050	9.0	1140.0	199.0	209.0
W1D-052	3 17 99	7.7	0.002	0.063	0.002	0.021	0.330	4.0	721.0	97.8	99.2
W1D-052	4 8 99	6.8	0.015	0.304	0.003	0.023	0.090	2.5	479.0	65.3	57.0
W1D-052	4 19 99	7.5	0.003	0.229	-0.001	0.024	0.080	-1.0	573.0	76.9	70.4
W1D-052	5 3 99	7.5	-0.001	0.166	-0.001	-0.010	0.090	1.0	684.0	124.0	116.0
W1D-052	5 19 99	7.5	0.001	0.064	-0.001	0.020	0.040	3.0	737.0	130.0	131.0
W1D-052	6 3 99	7.4	-0.001	0.184	0.002	0.015	0.070	1.2	714.0	132.0	134.0
W1D-052	6 19 99	7.0	-0.001	0.095	-0.001	0.011	0.080	4.0	817.0	142.0	144.0
W1D-052	7 9 99	7.2	0.004	0.229	-0.001	0.019	0.020	.	494.0	103.0	86.8
W1D-052	7 20 99	7.2	0.001	0.187	-0.001	0.016	0.020	4.0	765.0	140.0	133.0
W1D-052	8 2 99	7.3	-0.001	0.075	0.001	-0.010	0.040	4.0	870.0	146.0	144.0
W1D-052	8 17 99	7.1	-0.001	0.110	-0.001	-0.010	0.040	3.5	845.0	187.0	178.0

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-052	9 3 99	7.0	-0.001	0.037	-0.001	0.011	.	.	854.0	170.0	162.0
W1D-052	9 16 99	7.3	-0.001	0.085	-0.001	-0.010	.	.	921.0	157.0	151.0
W1D-052	10 5 99	7.2	-0.001	0.117	-0.001	0.014	.	.	879.0	160.0	148.0
W1D-052	10 19 99	7.9	-0.001	0.172	-0.001	-0.010	.	.	640.0	173.0	164.0
W1D-052	11 2 99	7.4	-0.001	0.140	-0.001	-0.010	.	.	1070.0	187.0	188.0
W1D-052	11 17 99	6.9	-0.001	0.115	0.001	0.013	.	.	1090.0	206.0	201.0

Profile sites (sites within the system; see figure at top for site ID's)

W1D-1	6 10 92	7.5	0.025	0.170	0.000	0.005	.	.	1306.0	.	.
W1D-1	6 15 92	7.4	0.007	0.120	0.000	0.005	.	.	1377.0	.	.
W1D-1	6 17 92	.	0.001	0.160	0.000	0.005	.	.	1633.0	.	.
W1D-1	6 22 92	.	0.006	0.210	0.000	0.040	.	.	1411.0	.	.
W1D-1	6 29 92	.	0.004	0.320	0.000	0.050	.	.	1342.0	.	.
W1D-1	7 6 92	.	0.012	0.420	0.000	0.040	.	.	1171.0	.	.
W1D-1	8 26 92	.	0.005	0.380	0.006	0.060	.	.	1422.0	.	.
W1D-1	9 4 92	.	0.004	0.330	0.000	0.050	.	.	1365.0	.	.
W1D-1	9 23 92	.	0.020	4.000	0.029	0.060	.	.	2800.0	.	.
W1D-1	10 5 92	.	0.019	5.000	0.016	0.080	.	.	2610.0	.	.
W1D-1	10 15 92	.	0.005	0.470	0.000	0.040	.	.	1638.0	.	.
W1D-1	6 7 93	.	0.005	0.140	0.001	0.040	.	.	412.0	70.0	64.0
W1D-1	6 10 93	.	0.006	0.110	0.001	0.040	.	.	125.0	73.0	60.0
W1D-1	6 14 93	.	0.004	0.140	.	.	.	.	529.0	.	.
W1D-1	6 24 93	.	0.005	0.190	.	.	.	.	616.0	.	.
W1D-1	6 28 93	.	0.006	1.300	.	.	.	.	496.0	.	.
W1D-1	7 1 93	.	0.007	0.140	.	.	.	.	894.0	.	.
W1D-1	7 6 93	.	0.004	0.250	.	.	.	.	772.0	.	.
W1D-1	7 8 93	.	0.006	0.770	.	.	.	.	875.0	.	.
W1D-1	7 12 93	6.8	0.006	0.300	.	.	.	.	340.0	.	.
W1D-1	7 15 93	.	0.010	1.800	.	.	.	.	760.0	.	.
W1D-1	7 19 93	.	0.008	0.570	.	.	.	.	468.0	.	.
W1D-1	7 26 93	.	0.006	0.300	.	.	.	.	626.0	.	.
W1D-1	8 2 93	.	0.005	0.360	.	.	.	.	558.0	.	.
W1D-1	8 16 93	.	0.005	0.260	.	.	.	.	1178.0	.	.
W1D-1	6 3 94	.	0.005	0.200	.	.	.	.	451.0	.	.
W1D-1	6 8 94	.	0.030	0.400	.	.	.	.	763.0	.	.
W1D-1	6 15 94	.	0.007	0.200	.	.	.	.	398.0	.	.
W1D-1	6 22 94	.	0.008	0.470	.	.	.	.	505.0	.	.
W1D-1	6 27 94	.	0.008	0.580	.	.	.	.	451.0	.	.
W1D-1	7 6 94	.	0.013	0.670	.	.	.	.	523.0	.	.
W1D-1	7 11 94	.	0.008	0.680	.	.	.	.	548.0	.	.
W1D-1	7 21 94	.	0.014	0.490	.	.	.	.	514.0	.	.
W1D-1	7 29 94	.	0.018	4.200	.	.	.	.	2182.0	.	.
W1D-1	8 4 94	.	0.008	0.470	.	.	.	.	3136.0	.	.
W1D-1	8 11 94	.	0.008	0.370	.	.	.	.	407.0	.	.
W1D-1	8 18 94	.	0.012	0.320	.	.	.	.	1645.0	.	.
W1D-1	8 25 94	.	0.009	1.800	.	.	.	.	1206.0	.	.
W1D-1	9 2 94	.	0.009	0.300	.	.	.	.	571.0	.	.
W1D-1	10 12 94	.	.	0.270	.	.	.	.	.	.	.
W1D-1	10 12 94	.	.	0.270	.	.	.	.	.	.	.
W1D-1	10 27 94	.	.	0.256	.	.	.	.	.	.	.
W1D-10	6 10 92	7.3	0.009	0.270	0.014	0.005	.	.	2096.0	.	.
W1D-10	6 15 92	6.8	0.015	0.140	0.010	0.005	.	.	2125.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-10	6 17 92	.	0.001	0.500	0.002	0.010	.	.	2228.0	.	.
W1D-10	6 22 92	.	0.003	0.720	0.008	0.060	.	.	1981.0	.	.
W1D-10	6 29 92	.	0.002	0.480	0.005	0.050	.	.	2058.0	.	.
W1D-10	7 6 92	.	0.005	0.880	0.005	0.060	.	.	2039.0	.	.
W1D-10	8 26 92	.	0.019	0.330	0.002	0.070	.	.	2012.0	.	.
W1D-10	9 4 92	.	0.036	0.790	0.002	0.050	.	.	2737.0	.	.
W1D-10	9 23 92	.	0.010	0.690	0.000	0.020	.	.	2550.0	.	.
W1D-10	10 5 92	.	0.003	0.940	0.000	0.040	.	.	2400.0	.	.
W1D-10	10 15 92	.	0.000	1.000	0.000	0.040	.	.	2255.0	.	.
W1D-10	6 7 93	.	0.005	0.110	0.001	0.050	.	.	714.0	120.0	140.0
W1D-10	6 10 93	.	0.007	0.120	0.001	0.030	.	.	947.0	130.0	140.0
W1D-10	6 14 93	.	0.006	0.080	.	.	.	.	990.0	.	.
W1D-10	6 24 93	.	0.006	0.070	.	.	.	.	1054.0	.	.
W1D-10	6 28 93	.	0.003	0.070	.	.	.	.	1191.0	.	.
W1D-10	7 1 93	.	0.004	0.080	.	.	.	.	1153.0	.	.
W1D-10	7 6 93	.	0.006	0.490	.	.	.	.	1181.0	.	.
W1D-10	7 8 93	.	0.012	0.600	.	.	.	.	1346.0	.	.
W1D-10	7 12 93	.	0.003	0.660	.	.	.	.	1379.0	.	.
W1D-10	7 15 93	.	0.004	0.630	.	.	.	.	1474.0	.	.
W1D-10	7 19 93	.	0.006	1.400	.	.	.	.	1536.0	.	.
W1D-10	7 26 93	.	0.006	1.100	.	.	.	.	1848.0	.	.
W1D-10	8 2 93	.	0.002	0.880	.	.	.	.	1708.0	.	.
W1D-10	8 16 93	.	0.005	0.640	.	.	.	.	1880.0	.	.
W1D-10	6 3 94	.	0.003	0.140	.	.	.	.	1089.0	.	.
W1D-10	6 8 94	.	0.009	.	.	.	.	.	1110.0	.	.
W1D-10	6 15 94	.	0.005	0.150	.	.	.	.	1284.0	.	.
W1D-10	6 22 94	.	0.007	0.460	.	.	.	.	1869.0	.	.
W1D-10	6 27 94	.	0.010	1.100	.	.	.	.	1310.0	.	.
W1D-10	7 6 94	.	0.005	0.920	.	.	.	.	1647.0	.	.
W1D-10	7 11 94	.	0.008	0.990	.	.	.	.	1712.0	.	.
W1D-10	7 21 94	.	0.005	0.430	.	.	.	.	1840.0	.	.
W1D-10	7 29 94	.	0.003	0.270	.	.	.	.	2128.0	.	.
W1D-10	8 4 94	.	0.004	0.210	.	.	.	.	1991.0	.	.
W1D-10	8 11 94	.	0.005	0.680	.	.	.	.	1748.0	.	.
W1D-10	8 18 94	.	0.003	0.290	.	.	.	.	1991.0	.	.
W1D-10	8 25 94	.	0.002	0.130	.	.	.	.	1863.0	.	.
W1D-10	9 2 94	.	0.005	0.530	.	.	.	.	1979.0	.	.
W1D-10	10 12 94	.	.	1.200	.	.	.	.	.	.	.
W1D-10	10 12 94	.	.	1.200	.	.	.	.	.	.	.
W1D-10	10 27 94	.	.	1.160	.	.	.	.	.	.	.
W1D-11	6 10 92	7.3	0.010	0.210	0.016	0.005	.	.	2049.0	.	.
W1D-11	6 15 92	6.8	0.009	0.160	0.012	0.005	.	.	2068.0	.	.
W1D-11	6 17 92	.	0.000	0.430	0.002	0.010	.	.	2172.0	.	.
W1D-11	6 22 92	.	0.003	0.740	0.008	0.060	.	.	2000.0	.	.
W1D-11	6 29 92	.	0.002	0.380	0.008	0.050	.	.	1990.0	.	.
W1D-11	7 6 92	.	0.010	0.870	0.006	0.060	.	.	1990.0	.	.
W1D-11	8 26 92	.	0.003	0.220	0.003	0.060	.	.	1892.0	.	.
W1D-11	9 4 92	.	0.008	0.790	0.000	0.050	.	.	3015.0	.	.
W1D-11	9 23 92	.	0.004	0.590	0.000	0.020	.	.	2563.0	.	.
W1D-11	10 5 92	.	0.003	0.800	0.001	0.030	.	.	2378.0	.	.
W1D-11	10 15 92	.	0.000	0.830	0.002	0.040	.	.	2255.0	.	.
W1D-11	6 7 93	.	0.005	0.110	0.001	0.050	.	.	708.0	120.0	130.0
W1D-11	6 10 93	.	0.006	0.120	0.001	0.040	.	.	1037.0	130.0	140.0
W1D-11	6 14 93	.	0.004	0.080	.	.	.	.	1041.0	.	.
W1D-11	6 24 93	.	0.003	0.090	.	.	.	.	1054.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-11	6 28 93	.	0.003	0.080	.	.	.	.	1220.0	.	.
W1D-11	7 1 93	.	0.012	0.070	.	.	.	.	1172.0	.	.
W1D-11	7 6 93	.	0.008	0.470	.	.	.	.	1193.0	.	.
W1D-11	7 8 93	.	0.009	0.590	.	.	.	.	1336.0	.	.
W1D-11	7 12 93	.	0.003	0.780	.	.	.	.	1368.0	.	.
W1D-11	7 15 93	.	0.004	0.850	.	.	.	.	1575.0	.	.
W1D-11	7 19 93	.	0.006	1.500	.	.	.	.	1536.0	.	.
W1D-11	7 26 93	.	0.004	1.100	.	.	.	.	1848.0	.	.
W1D-11	8 2 93	.	0.004	0.890	.	.	.	.	1729.0	.	.
W1D-11	8 16 93	.	0.005	0.870	.	.	.	.	1922.0	.	.
W1D-11	6 3 94	.	0.004	0.130	.	.	.	.	1077.0	.	.
W1D-11	6 8 94	.	0.005	0.150	.	.	.	.	1117.0	.	.
W1D-11	6 14 94	.	0.004	0.100	.	.	.	.	1277.0	.	.
W1D-11	6 22 94	.	0.018	0.690	.	.	.	.	1980.0	.	.
W1D-11	6 27 94	.	0.008	0.860	.	.	.	.	1286.0	.	.
W1D-11	7 6 94	.	0.006	0.590	.	.	.	.	1556.0	.	.
W1D-11	7 11 94	.	0.007	0.680	.	.	.	.	1681.0	.	.
W1D-11	7 21 94	.	0.010	0.480	.	.	.	.	1806.0	.	.
W1D-11	7 29 94	.	0.002	0.110	.	.	.	.	2045.0	.	.
W1D-11	8 4 94	.	0.003	0.110	.	.	.	.	2096.0	.	.
W1D-11	8 4 94	.	0.003	0.050	.	.	.	.	2049.0	.	.
W1D-11	8 11 94	.	0.004	0.440	.	.	.	.	851.0	.	.
W1D-11	8 11 94	.	0.005	0.320	.	.	.	.	1656.0	.	.
W1D-11	8 18 94	.	0.002	0.080	.	.	.	.	2038.0	.	.
W1D-11	8 18 94	.	0.002	0.040	.	.	.	.	1898.0	.	.
W1D-11	8 25 94	.	0.001	0.040	.	.	.	.	1829.0	.	.
W1D-11	8 25 94	.	0.001	0.030	.	.	.	.	1875.0	.	.
W1D-11	9 2 94	.	0.003	0.530	.	.	.	.	2003.0	.	.
W1D-11	9 2 94	.	0.002	0.360	.	.	.	.	1956.0	.	.
W1D-11	10 12 94	.	.	0.930	.	.	.	.	.	.	.
W1D-11	10 12 94	.	.	0.930	.	.	.	.	.	.	.
W1D-11	10 27 94	.	.	1.040	.	.	.	.	.	.	.
W1D-12	10 15 92	.	0.003	0.370	0.011	0.040	.	.	2077.0	.	.
W1D-12	6 7 93	.	0.007	0.080	0.007	0.050	.	.	763.0	120.0	130.0
W1D-12	6 10 93	.	0.006	0.090	0.001	0.030	.	.	1018.0	130.0	140.0
W1D-12	6 14 93	.	0.004	0.090	.	.	.	.	995.0	.	.
W1D-12	6 24 93	.	0.005	0.050	.	.	.	.	978.0	.	.
W1D-12	6 28 93	.	0.080	0.047	.	.	.	.	1104.0	.	.
W1D-12	7 1 93	.	0.004	0.050	.	.	.	.	1071.0	.	.
W1D-12	7 6 93	.	0.004	0.240	.	.	.	.	1084.0	.	.
W1D-12	7 8 93	.	0.002	0.210	.	.	.	.	1172.0	.	.
W1D-12	7 12 93	.	0.003	0.530	.	.	.	.	1236.0	.	.
W1D-12	7 15 93	.	0.005	0.220	.	.	.	.	1225.0	.	.
W1D-12	7 19 93	.	0.005	0.690	.	.	.	.	1291.0	.	.
W1D-12	7 26 93	.	0.005	0.430	.	.	.	.	1556.0	.	.
W1D-12	8 2 93	.	0.002	0.380	.	.	.	.	1393.0	.	.
W1D-12	8 16 93	.	0.003	0.290	.	.	.	.	1429.0	.	.
W1D-12	6 3 94	.	0.003	0.120	.	.	.	.	959.0	.	.
W1D-12	6 8 94	.	0.004	0.080	.	.	.	.	948.0	.	.
W1D-12	6 15 94	.	0.005	0.090	.	.	.	.	1028.0	.	.
W1D-12	6 22 94	.	0.014	0.230	.	.	.	.	1521.0	.	.
W1D-12	6 27 94	.	0.007	0.600	.	.	.	.	1209.0	.	.
W1D-12	7 6 94	.	0.003	0.130	.	.	.	.	1332.0	.	.
W1D-12	7 11 94	.	0.004	0.030	.	.	.	.	363.0	.	.
W1D-12	7 21 94	.	0.011	0.080	.	.	.	.	1398.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-12	7 29 94	.	0.001	0.050	.	.	.	.	1357.0	.	.
W1D-12	8 4 94	.	0.006	0.080	.	.	.	.	1807.0	.	.
W1D-12	8 11 94	.	0.006	0.170	.	.	.	.	1622.0	.	.
W1D-12	8 18 94	.	0.006	0.040	.	.	.	.	1267.0	.	.
W1D-12	8 25 94	.	0.002	0.040	.	.	.	.	1362.0	.	.
W1D-12	9 2 94	.	0.002	0.150	.	.	.	.	1702.0	.	.
W1D-12	10 12 94	.	.	0.810	.	.	.	.	.	.	.
W1D-12	10 12 94	.	.	0.110	.	.	.	.	.	.	.
W1D-12	10 13 94	.	.	0.850	.	.	.	.	.	.	.
W1D-12	10 13 94	.	.	0.120	.	.	.	.	.	.	.
W1D-12	10 27 94	.	.	0.297	.	.	.	.	.	.	.
W1D-2	6 10 92	6.5	0.060	3.300	0.004	0.040	.	.	2355.0	.	.
W1D-2	6 15 92	7.1	0.050	5.700	0.006	0.040	.	.	2669.0	.	.
W1D-2	6 17 92	.	0.050	4.000	0.001	0.030	.	.	2237.0	.	.
W1D-2	6 22 92	.	0.037	4.000	0.005	0.070	.	.	2409.0	.	.
W1D-2	6 29 92	.	0.037	4.800	0.004	0.080	.	.	2346.0	.	.
W1D-2	7 6 92	.	0.026	4.700	0.005	0.090	.	.	2283.0	.	.
W1D-2	8 26 92	.	0.037	2.900	0.005	0.080	.	.	2049.0	.	.
W1D-2	9 4 92	.	0.032	2.500	0.006	0.060	.	.	2828.0	.	.
W1D-2	9 23 92	.	0.018	3.400	0.023	0.060	.	.	2766.0	.	.
W1D-2	10 5 92	.	0.011	4.100	0.020	0.080	.	.	2567.0	.	.
W1D-2	10 15 92	.	0.030	5.700	0.017	0.090	.	.	2461.0	.	.
W1D-2	6 7 93	.	0.020	0.670	0.002	0.050	.	.	796.0	120.0	130.0
W1D-2	6 10 93	.	0.030	0.540	0.001	0.040	.	.	1037.0	130.0	130.0
W1D-2	6 14 93	.	0.019	0.560	.	.	.	.	947.0	.	.
W1D-2	6 24 93	.	0.025	1.400	.	.	.	.	1282.0	.	.
W1D-2	6 28 93	.	0.017	1.300	.	.	.	.	1405.0	.	.
W1D-2	7 1 93	.	0.018	2.100	.	.	.	.	1617.0	.	.
W1D-2	7 6 93	.	0.012	2.500	.	.	.	.	1316.0	.	.
W1D-2	7 8 93	.	0.010	3.000	.	.	.	.	1654.0	.	.
W1D-2	7 12 93	.	0.019	3.600	.	.	.	.	1688.0	.	.
W1D-2	7 15 93	.	0.016	5.100	.	.	.	.	2269.0	.	.
W1D-2	7 19 93	.	0.025	5.300	.	.	.	.	2061.0	.	.
W1D-2	7 26 93	.	0.026	5.400	.	.	.	.	2104.0	.	.
W1D-2	8 2 93	.	0.018	4.500	.	.	.	.	2267.0	.	.
W1D-2	8 16 93	.	0.021	5.200	.	.	.	.	1398.0	.	.
W1D-2	6 3 94	.	0.027	1.600	.	.	.	.	1525.0	.	.
W1D-2	6 8 94	.	0.030	1.200	.	.	.	.	1781.0	.	.
W1D-2	6 15 94	.	0.024	0.900	.	.	.	.	1022.0	.	.
W1D-2	6 22 94	.	0.016	3.500	.	.	.	.	2382.0	.	.
W1D-2	6 27 94	.	0.026	3.500	.	.	.	.	1669.0	.	.
W1D-2	7 6 94	.	0.025	6.100	.	.	.	.	2084.0	.	.
W1D-2	7 11 94	.	0.030	4.400	.	.	.	.	2075.0	.	.
W1D-2	7 21 94	.	0.028	5.400	.	.	.	.	2061.0	.	.
W1D-2	7 29 94	.	0.028	6.100	.	.	.	.	2870.0	.	.
W1D-2	8 4 94	.	0.030	5.200	.	.	.	.	2402.0	.	.
W1D-2	8 11 94	.	0.030	4.600	.	.	.	.	2026.0	.	.
W1D-2	8 18 94	.	0.029	3.700	.	.	.	.	2155.0	.	.
W1D-2	8 25 94	.	0.030	4.000	.	.	.	.	2284.0	.	.
W1D-2	9 2 94	.	0.050	3.500	.	.	.	.	1794.0	.	.
W1D-2	10 12 94	.	.	5.200	.	.	.	.	.	.	.
W1D-2	10 12 94	.	.	5.200	.	.	.	.	.	.	.
W1D-2	10 27 94	.	.	2.740	.	.	.	.	.	.	.
W1D-3	6 7 93	.	0.018	0.610	0.003	0.060	.	.	801.0	120.0	130.0

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-3	6 10 93	.	0.030	0.510	.	0.040	.	.	1009.0	130.0	130.0
W1D-3	6 14 93	.	0.014	0.660	.	.	.	.	1064.0	.	.
W1D-3	6 24 93	.	0.033	1.500	.	.	.	.	1161.0	.	.
W1D-3	6 28 93	.	0.018	1.200	.	.	.	.	1372.0	.	.
W1D-3	7 1 93	.	0.022	1.700	.	.	.	.	1467.0	.	.
W1D-3	7 6 93	.	0.016	2.500	.	.	.	.	1450.0	.	.
W1D-3	7 8 93	.	0.012	3.000	.	.	.	.	1603.0	.	.
W1D-3	7 12 93	.	0.015	3.400	.	.	.	.	1774.0	.	.
W1D-3	7 15 93	.	0.014	4.600	.	.	.	.	2116.0	.	.
W1D-3	7 19 93	.	0.021	4.900	.	.	.	.	1997.0	.	.
W1D-3	7 26 93	.	0.025	5.300	.	.	.	.	2168.0	.	.
W1D-3	8 2 93	.	0.016	4.300	.	.	.	.	2224.0	.	.
W1D-3	8 16 93	.	0.024	4.900	.	.	.	.	2364.0	.	.
W1D-3	6 3 94	.	0.026	1.300	.	.	.	.	1415.0	.	.
W1D-3	6 8 94	.	0.017	0.400	.	.	.	.	1628.0	.	.
W1D-3	6 15 94	.	0.022	0.890	.	.	.	.	1304.0	.	.
W1D-3	6 22 94	.	0.017	3.000	.	.	.	.	2221.0	.	.
W1D-3	6 27 94	.	0.021	3.200	.	.	.	.	1694.0	.	.
W1D-3	7 6 94	.	0.017	5.000	.	.	.	.	1933.0	.	.
W1D-3	7 11 94	.	0.025	4.600	.	.	.	.	2315.0	.	.
W1D-3	7 21 94	.	0.023	4.900	.	.	.	.	2003.0	.	.
W1D-3	7 29 94	.	0.027	5.300	.	.	.	.	2557.0	.	.
W1D-3	8 4 94	.	0.027	4.600	.	.	.	.	2249.0	.	.
W1D-3	8 11 94	.	0.026	4.000	.	.	.	.	1921.0	.	.
W1D-3	8 18 94	.	0.030	3.600	.	.	.	.	2308.0	.	.
W1D-3	8 25 94	.	0.023	3.600	.	.	.	.	2096.0	.	.
W1D-3	9 2 94	.	0.040	3.200	.	.	.	.	1829.0	.	.
W1D-3	10 12 94	.	.	4.600	.	.	.	.	.	.	.
W1D-3	10 12 94	.	.	4.600	.	.	.	.	.	.	.
W1D-3	10 27 94	.	.	2.620	.	.	.	.	.	.	.
W1D-4	6 10 92	7.3	0.040	2.500	0.004	0.030	.	.	2264.0	.	.
W1D-4	6 15 92	6.8	0.050	4.500	0.034	0.050	.	.	2686.0	.	.
W1D-4	6 17 92	.	0.040	3.800	0.002	0.040	.	.	2444.0	.	.
W1D-4	6 22 92	.	0.018	3.200	0.007	0.050	.	.	2246.0	.	.
W1D-4	6 29 92	.	0.014	3.800	0.011	0.050	.	.	2162.0	.	.
W1D-4	7 6 92	.	0.016	3.900	0.010	0.090	.	.	2319.0	.	.
W1D-4	8 26 92	.	0.035	2.900	0.001	0.080	.	.	2078.0	.	.
W1D-4	9 4 92	.	0.024	1.900	0.004	0.050	.	.	2627.0	.	.
W1D-4	9 23 92	.	0.007	2.300	0.014	0.040	.	.	2672.0	.	.
W1D-4	10 5 92	.	0.008	3.600	0.014	0.050	.	.	2555.0	.	.
W1D-4	10 15 92	.	0.012	4.400	0.012	0.070	.	.	2395.0	.	.
W1D-4	6 7 93	.	0.014	0.580	0.002	0.050	.	.	741.0	120.0	120.0
W1D-4	6 10 93	.	-0.001	0.390	0.001	0.040	.	.	1050.0	130.0	130.0
W1D-4	6 14 93	.	0.017	0.510	.	.	.	.	976.0	.	.
W1D-4	6 24 93	.	0.017	1.100	.	.	.	.	1294.0	.	.
W1D-4	6 28 93	.	0.014	1.000	.	.	.	.	1333.0	.	.
W1D-4	7 1 93	.	0.030	1.700	.	.	.	.	1462.0	.	.
W1D-4	7 6 93	.	0.015	2.900	.	.	.	.	1223.0	.	.
W1D-4	7 8 93	.	0.017	2.900	.	.	.	.	1555.0	.	.
W1D-4	7 12 93	.	0.014	3.300	.	.	.	.	1808.0	.	.
W1D-4	7 15 93	.	0.013	4.500	.	.	.	.	2057.0	.	.
W1D-4	7 19 93	.	0.016	4.500	.	.	.	.	1912.0	.	.
W1D-4	7 26 93	.	0.027	5.200	.	.	.	.	2136.0	.	.
W1D-4	8 2 93	.	0.016	4.300	.	.	.	.	2007.0	.	.
W1D-4	8 16 93	.	0.022	4.800	.	.	.	.	2289.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-4	6 3 94	.	0.015	1.000	.	.	.	.	1408.0	.	.
W1D-4	6 8 94	.	0.002	0.060	.	.	.	.	1222.0	.	.
W1D-4	6 15 94	.	0.016	0.800	.	.	.	.	1008.0	.	.
W1D-4	6 22 94	.	0.015	3.000	.	.	.	.	2375.0	.	.
W1D-4	6 27 94	.	0.018	2.900	.	.	.	.	1531.0	.	.
W1D-4	7 6 94	.	0.016	4.700	.	.	.	.	1944.0	.	.
W1D-4	7 11 94	.	0.020	4.200	.	.	.	.	2087.0	.	.
W1D-4	7 21 94	.	0.020	4.500	.	.	.	.	2061.0	.	.
W1D-4	7 29 94	.	0.013	4.500	.	.	.	.	2569.0	.	.
W1D-4	8 4 94	.	0.013	3.300	.	.	.	.	2225.0	.	.
W1D-4	8 11 94	.	0.015	2.900	.	.	.	.	1577.0	.	.
W1D-4	8 18 94	.	0.023	2.600	.	.	.	.	2237.0	.	.
W1D-4	8 25 94	.	0.011	2.100	.	.	.	.	2143.0	.	.
W1D-4	9 2 94	.	0.022	2.700	.	.	.	.	1852.0	.	.
W1D-4	10 12 94	.	.	0.090	.	.	.	.	.	.	.
W1D-4	10 12 94	.	.	.	.	.	.	.	.	.	.
W1D-4	10 13 94	.	.	4.600	.	.	.	.	.	.	.
W1D-4	10 13 94	.	.	4.300	.	.	.	.	.	.	.
W1D-4	10 27 94	.	.	0.072	.	.	.	.	.	.	.
W1D-5	6 10 92	7.3	0.029	1.800	0.009	0.020	.	.	2592.0	.	.
W1D-5	6 15 92	6.5	0.025	2.000	0.015	0.030	.	.	2770.0	.	.
W1D-5	6 17 92	.	0.003	3.200	0.003	0.030	.	.	2471.0	.	.
W1D-5	6 22 92	.	0.009	2.100	0.011	0.070	.	.	2153.0	.	.
W1D-5	6 29 92	.	0.006	2.500	0.013	0.070	.	.	2153.0	.	.
W1D-5	7 6 92	.	0.008	2.400	0.013	0.070	.	.	2218.0	.	.
W1D-5	8 26 92	.	0.016	2.300	0.010	0.090	.	.	2128.0	.	.
W1D-5	9 4 92	.	0.051	1.600	0.007	0.060	.	.	2635.0	.	.
W1D-5	9 23 92	.	0.006	1.700	0.009	0.040	.	.	2672.0	.	.
W1D-5	10 5 92	.	0.004	2.000	0.004	0.050	.	.	2461.0	.	.
W1D-5	10 15 92	.	0.003	3.200	0.006	0.050	.	.	2231.0	.	.
W1D-5	6 7 93	.	0.001	0.480	0.002	0.060	.	.	752.0	120.0	130.0
W1D-5	6 10 93	.	-0.001	0.230	0.001	0.060	.	.	1064.0	130.0	140.0
W1D-5	6 14 93	.	0.003	0.290	.	.	.	.	1078.0	.	.
W1D-5	6 24 93	.	0.011	0.620	.	.	.	.	1243.0	.	.
W1D-5	6 28 93	.	0.006	0.410	.	.	.	.	1284.0	.	.
W1D-5	7 1 93	.	0.007	0.560	.	.	.	.	1376.0	.	.
W1D-5	7 6 93	.	0.008	1.800	.	.	.	.	1341.0	.	.
W1D-5	7 8 93	.	0.009	1.800	.	.	.	.	1505.0	.	.
W1D-5	7 12 93	.	0.006	2.200	.	.	.	.	1357.0	.	.
W1D-5	7 15 93	.	0.009	2.600	.	.	.	.	1883.0	.	.
W1D-5	7 19 93	.	0.007	2.700	.	.	.	.	1775.0	.	.
W1D-5	7 26 93	.	0.011	3.700	.	.	.	.	1650.0	.	.
W1D-5	8 2 93	.	0.008	2.600	.	.	.	.	1985.0	.	.
W1D-5	8 16 93	.	0.007	2.300	.	.	.	.	2237.0	.	.
W1D-5	6 3 94	.	0.007	0.650	.	.	.	.	1292.0	.	.
W1D-5	6 8 94	.	0.003	0.170	.	.	.	.	1403.0	.	.
W1D-5	6 15 94	.	0.010	0.500	.	.	.	.	1121.0	.	.
W1D-5	6 22 94	.	0.069	1.900	.	.	.	.	2290.0	.	.
W1D-5	6 27 94	.	0.010	1.800	.	.	.	.	1556.0	.	.
W1D-5	7 6 94	.	0.008	1.800	.	.	.	.	1760.0	.	.
W1D-5	7 11 94	.	0.011	2.300	.	.	.	.	1885.0	.	.
W1D-5	7 21 94	.	0.007	1.600	.	.	.	.	1875.0	.	.
W1D-5	7 29 94	.	0.007	2.300	.	.	.	.	2967.0	.	.
W1D-5	8 4 94	.	0.008	0.780	.	.	.	.	2073.0	.	.
W1D-5	8 11 94	.	0.012	2.000	.	.	.	.	1565.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-5	8 18 94	.	0.011	2.000	.	.	.	.	1968.0	.	.
W1D-5	8 25 94	.	0.010	1.600	.	.	.	.	2084.0	.	.
W1D-5	9 2 94	.	0.011	1.500	.	.	.	.	1968.0	.	.
W1D-5	10 12 94	.	.	1.800	.	.	.	.	.	.	.
W1D-5	10 12 94	.	.	1.800	.	.	.	.	.	.	.
W1D-5	10 27 94	.	.	1.540	.	.	.	.	.	.	.
W1D-6	6 10 92	6.3	0.006	0.280	0.007	0.005	.	.	2200.0	.	.
W1D-6	6 15 92	6.4	0.005	0.140	0.007	0.005	.	.	2601.0	.	.
W1D-6	6 17 92	.	0.001	1.700	0.002	0.020	.	.	2283.0	.	.
W1D-6	6 22 92	.	0.005	1.400	0.008	0.060	.	.	2000.0	.	.
W1D-6	6 29 92	.	0.002	0.960	0.007	0.060	.	.	2144.0	.	.
W1D-6	7 6 92	.	0.005	1.600	0.009	0.060	.	.	2190.0	.	.
W1D-6	8 26 92	.	0.035	1.300	0.005	0.060	.	.	2142.0	.	.
W1D-6	9 4 92	.	0.011	1.300	0.004	0.050	.	.	2669.0	.	.
W1D-6	9 23 92	.	0.009	1.300	0.005	0.040	.	.	2655.0	.	.
W1D-6	10 5 92	.	0.004	1.600	0.004	0.040	.	.	2473.0	.	.
W1D-6	10 15 92	.	0.002	7.300	0.003	0.050	.	.	2350.0	.	.
W1D-6	6 7 93	.	0.007	0.360	0.002	0.060	.	.	779.0	120.0	140.0
W1D-6	6 10 93	.	0.008	0.210	0.001	0.040	.	.	1041.0	130.0	140.0
W1D-6	6 14 93	.	0.005	0.170	.	.	.	.	1082.0	.	.
W1D-6	6 24 93	.	0.007	0.260	.	.	.	.	1148.0	.	.
W1D-6	6 28 93	.	0.004	0.190	.	.	.	.	1210.0	.	.
W1D-6	7 1 93	.	0.008	0.230	.	.	.	.	1327.0	.	.
W1D-6	7 6 93	.	0.004	1.100	.	.	.	.	1248.0	.	.
W1D-6	7 8 93	.	0.006	1.100	.	.	.	.	1424.0	.	.
W1D-6	7 12 93	.	0.004	1.700	.	.	.	.	1363.0	.	.
W1D-6	7 15 93	.	0.007	1.800	.	.	.	.	1683.0	.	.
W1D-6	7 19 93	.	0.004	1.800	.	.	.	.	1691.0	.	.
W1D-6	7 26 93	.	0.007	2.100	.	.	.	.	1764.0	.	.
W1D-6	8 2 93	.	0.004	1.400	.	.	.	.	1910.0	.	.
W1D-6	8 16 93	.	0.007	1.300	.	.	.	.	2058.0	.	.
W1D-6	6 3 94	.	0.004	0.310	.	.	.	.	1184.0	.	.
W1D-6	6 8 94	.	0.003	0.140	.	.	.	.	1309.0	.	.
W1D-6	6 15 94	.	0.006	0.360	.	.	.	.	1155.0	.	.
W1D-6	6 22 94	.	0.025	1.200	.	.	.	.	2092.0	.	.
W1D-6	6 27 94	.	0.010	1.800	.	.	.	.	1346.0	.	.
W1D-6	7 6 94	.	0.011	1.500	.	.	.	.	1783.0	.	.
W1D-6	7 11 94	.	0.008	2.000	.	.	.	.	1902.0	.	.
W1D-6	7 21 94	.	0.007	1.600	.	.	.	.	1794.0	.	.
W1D-6	7 29 94	.	0.004	1.000	.	.	.	.	2318.0	.	.
W1D-6	8 4 94	.	0.005	0.900	.	.	.	.	1968.0	.	.
W1D-6	8 11 94	.	0.007	1.200	.	.	.	.	1760.0	.	.
W1D-6	8 18 94	.	0.005	0.520	.	.	.	.	2131.0	.	.
W1D-6	8 25 94	.	0.004	0.390	.	.	.	.	2084.0	.	.
W1D-6	9 2 94	.	0.009	1.400	.	.	.	.	1991.0	.	.
W1D-6	10 12 94	.	.	2.000	.	.	.	.	.	.	.
W1D-6	10 12 94	.	.	2.000	.	.	.	.	.	.	.
W1D-6	10 27 94	.	.	1.570	.	.	.	.	.	.	.
W1D-7	6 10 92	6.6	0.009	0.260	0.011	0.005	.	.	2355.0	.	.
W1D-7	6 15 92	6.6	0.007	0.140	0.008	0.005	.	.	2328.0	.	.
W1D-7	6 17 92	.	0.002	0.720	0.003	0.020	.	.	2255.0	.	.
W1D-7	6 22 92	.	0.005	1.200	0.007	0.060	.	.	2209.0	.	.
W1D-7	6 29 92	.	0.003	0.790	0.006	0.070	.	.	2144.0	.	.
W1D-7	7 6 92	.	0.006	1.300	0.007	0.060	.	.	2172.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-7	8 26 92	.	0.005	0.800	0.003	0.060	.	.	1960.0	.	.
W1D-7	9 4 92	.	0.003	0.650	0.002	0.050	.	.	2669.0	.	.
W1D-7	9 23 92	.	0.003	0.850	0.000	0.030	.	.	2572.0	.	.
W1D-7	10 5 92	.	0.004	1.100	0.001	0.030	.	.	2431.0	.	.
W1D-7	10 15 92	.	0.000	1.800	0.002	0.040	.	.	2299.0	.	.
W1D-7	6 7 93	7.2	0.002	0.180	0.002	0.070	.	.	790.0	130.0	140.0
W1D-7	6 10 93	.	0.004	0.110	0.001	0.040	.	.	1037.0	130.0	140.0
W1D-7	6 14 93	7.2	0.003	0.090	.	.	.	.	1004.0	.	.
W1D-7	6 24 93	7.2	0.005	0.180	.	.	.	.	1148.0	.	.
W1D-7	6 28 93	6.9	0.001	0.100	.	.	.	.	1176.0	.	.
W1D-7	7 1 93	6.8	-0.001	0.060	.	.	.	.	1172.0	.	.
W1D-7	7 6 93	7.0	0.006	0.510	.	.	.	.	1228.0	.	.
W1D-7	7 8 93	.	0.005	0.770	.	.	.	.	1389.0	.	.
W1D-7	7 12 93	6.9	0.004	1.600	.	.	.	.	1329.0	.	.
W1D-7	7 15 93	6.8	0.006	1.200	.	.	.	.	1626.0	.	.
W1D-7	7 19 93	6.8	0.004	1.400	.	.	.	.	1691.0	.	.
W1D-7	7 26 93	6.8	0.008	1.600	.	.	.	.	1629.0	.	.
W1D-7	8 2 93	7.1	0.006	1.100	.	.	.	.	1771.0	.	.
W1D-7	8 16 93	7.0	0.003	0.830	.	.	.	.	1880.0	.	.
W1D-7	6 3 94	7.4	0.003	0.170	.	.	.	.	1171.0	.	.
W1D-7	6 8 94	7.5	0.007	0.120	.	.	.	.	1229.0	.	.
W1D-7	6 15 94	7.4	0.008	0.290	.	.	.	.	1121.0	.	.
W1D-7	6 22 94	7.2	0.007	0.930	.	.	.	.	2115.0	.	.
W1D-7	6 27 94	7.4	0.007	1.400	.	.	.	.	1383.0	.	.
W1D-7	7 6 94	7.3	0.006	1.300	.	.	.	.	1789.0	.	.
W1D-7	7 11 94	7.5	0.007	1.400	.	.	.	.	1902.0	.	.
W1D-7	7 21 94	7.1	0.006	1.100	.	.	.	.	1806.0	.	.
W1D-7	7 29 94	7.3	0.006	0.500	.	.	.	.	2272.0	.	.
W1D-7	8 4 94	7.3	0.006	0.470	.	.	.	.	2061.0	.	.
W1D-7	8 11 94	7.2	0.006	1.100	.	.	.	.	1737.0	.	.
W1D-7	8 18 94	.	0.009	0.330	.	.	.	.	2003.0	.	.
W1D-7	8 25 94	7.2	0.002	0.230	.	.	.	.	1910.0	.	.
W1D-7	9 2 94	.	0.007	1.400	.	.	.	.	1921.0	.	.
W1D-7	10 12 94	.	.	1.600	.	.	.	.	.	.	.
W1D-7	10 12 94	.	.	1.600	.	.	.	.	.	.	.
W1D-7	10 27 94	.	.	1.180	.	.	.	.	.	.	.
W1D-8	6 10 92	6.8	0.014	0.290	0.012	0.005	.	.	2283.0	.	.
W1D-8	6 15 92	6.7	0.012	0.140	0.009	0.005	.	.	2382.0	.	.
W1D-8	6 17 92	.	0.001	0.710	0.003	0.010	.	.	2246.0	.	.
W1D-8	6 22 92	.	0.006	0.770	0.008	0.070	.	.	2077.0	.	.
W1D-8	6 29 92	.	0.003	0.670	0.006	0.050	.	.	2200.0	.	.
W1D-8	7 6 92	.	0.006	1.100	0.006	0.070	.	.	1971.0	.	.
W1D-8	8 26 92	.	0.007	0.470	0.002	0.090	.	.	1953.0	.	.
W1D-8	9 4 92	.	0.008	0.840	0.002	0.040	.	.	2661.0	.	.
W1D-8	9 23 92	.	0.003	0.760	0.001	0.030	.	.	2528.0	.	.
W1D-8	10 5 92	.	0.003	1.000	0.001	0.040	.	.	2426.0	.	.
W1D-8	10 15 92	.	0.001	1.400	0.001	0.050	.	.	2231.0	.	.
W1D-8	6 7 93	.	0.004	0.140	0.001	0.090	.	.	730.0	120.0	140.0
W1D-8	6 10 93	.	0.005	0.130	0.001	0.040	.	.	976.0	130.0	140.0
W1D-8	6 14 93	.	0.002	0.100	.	.	.	.	1046.0	.	.
W1D-8	6 24 93	.	0.002	0.110	.	.	.	.	1072.0	.	.
W1D-8	6 28 93	.	0.002	0.100	.	.	.	.	1204.0	.	.
W1D-8	7 1 93	.	0.003	0.090	.	.	.	.	1187.0	.	.
W1D-8	7 6 93	.	0.006	0.510	.	.	.	.	1202.0	.	.
W1D-8	7 8 93	.	0.004	0.650	.	.	.	.	1363.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-8	7 12 93	.	0.004	1.400	.	.	.	.	1285.0	.	.
W1D-8	7 15 93	.	0.005	1.100	.	.	.	.	1581.0	.	.
W1D-8	7 19 93	.	0.005	1.400	.	.	.	.	1608.0	.	.
W1D-8	7 26 93	.	0.004	1.200	.	.	.	.	1743.0	.	.
W1D-8	8 2 93	.	0.002	1.200	.	.	.	.	1238.0	.	.
W1D-8	8 16 93	.	0.004	0.790	.	.	.	.	1922.0	.	.
W1D-8	6 3 94	.	0.002	0.180	.	.	.	.	1114.0	.	.
W1D-8	6 8 94	.	0.002	0.180	.	.	.	.	1249.0	.	.
W1D-8	6 15 94	.	0.006	0.210	.	.	.	.	1277.0	.	.
W1D-8	6 22 94	.	0.007	0.870	.	.	.	.	2123.0	.	.
W1D-8	6 27 94	.	0.006	1.100	.	.	.	.	1359.0	.	.
W1D-8	7 6 94	.	0.006	0.990	.	.	.	.	1672.0	.	.
W1D-8	7 11 94	.	0.007	1.100	.	.	.	.	1794.0	.	.
W1D-8	7 21 94	.	0.005	0.930	.	.	.	.	1821.0	.	.
W1D-8	7 29 94	.	0.003	0.480	.	.	.	.	2093.0	.	.
W1D-8	8 4 94	.	0.024	0.320	.	.	.	.	2038.0	.	.
W1D-8	8 11 94	.	0.009	0.930	.	.	.	.	1725.0	.	.
W1D-8	8 18 94	.	0.003	0.330	.	.	.	.	2084.0	.	.
W1D-8	8 25 94	.	0.002	0.230	.	.	.	.	1794.0	.	.
W1D-8	9 2 94	.	0.006	1.100	.	.	.	.	1863.0	.	.
W1D-8	10 12 94	.	.	1.600	.	.	.	.	.	.	.
W1D-8	10 12 94	.	.	1.600	.	.	.	.	.	.	.
W1D-8	10 27 94	.	.	1.180	.	.	.	.	.	.	.
W1D-9	6 10 92	7.1	0.020	0.260	0.015	0.010	.	.	2274.0	.	.
W1D-9	6 15 92	6.7	0.005	0.130	0.009	0.005	.	.	2237.0	.	.
W1D-9	6 17 92	.	0.001	0.560	0.003	0.010	.	.	2427.0	.	.
W1D-9	6 22 92	.	0.004	0.800	0.008	0.060	.	.	2134.0	.	.
W1D-9	6 29 92	.	0.003	0.510	0.005	0.060	.	.	1990.0	.	.
W1D-9	7 6 92	.	0.010	1.100	0.005	0.060	.	.	2020.0	.	.
W1D-9	8 26 92	.	0.020	0.360	0.001	0.060	.	.	1983.0	.	.
W1D-9	9 4 92	.	0.018	0.530	0.008	0.060	.	.	2575.0	.	.
W1D-9	9 23 92	.	0.006	0.750	0.001	0.020	.	.	2563.0	.	.
W1D-9	10 5 92	.	0.004	1.000	0.001	0.040	.	.	2413.0	.	.
W1D-9	10 15 92	.	0.001	1.100	0.000	0.040	.	.	2255.0	.	.
W1D-9	6 7 93	.	0.005	0.120	0.001	0.050	.	.	736.0	120.0	130.0
W1D-9	6 10 93	.	0.006	0.110	0.001	0.040	.	.	1032.0	130.0	140.0
W1D-9	6 14 93	.	0.003	0.090	.	.	.	.	1037.0	.	.
W1D-9	6 24 93	.	0.008	0.070	.	.	.	.	1029.0	.	.
W1D-9	6 28 93	.	0.003	0.100	.	.	.	.	1199.0	.	.
W1D-9	7 1 93	.	0.005	0.060	.	.	.	.	1156.0	.	.
W1D-9	7 6 93	.	0.008	0.520	.	.	.	.	1181.0	.	.
W1D-9	7 8 93	.	0.006	0.630	.	.	.	.	1327.0	.	.
W1D-9	7 12 93	.	0.004	1.400	.	.	.	.	1368.0	.	.
W1D-9	7 15 93	.	0.004	1.000	.	.	.	.	1541.0	.	.
W1D-9	7 19 93	.	0.006	1.500	.	.	.	.	1567.0	.	.
W1D-9	7 26 93	.	0.006	1.100	.	.	.	.	1733.0	.	.
W1D-9	8 2 93	.	0.007	0.940	.	.	.	.	1718.0	.	.
W1D-9	8 16 93	.	0.004	0.750	.	.	.	.	2005.0	.	.
W1D-9	6 3 94	.	0.002	0.150	.	.	.	.	1177.0	.	.
W1D-9	6 8 94	.	0.005	0.100	.	.	.	.	1262.0	.	.
W1D-9	6 15 94	.	0.007	0.180	.	.	.	.	1250.0	.	.
W1D-9	6 22 94	.	0.008	0.690	.	.	.	.	2025.0	.	.
W1D-9	6 27 94	.	0.006	1.100	.	.	.	.	1286.0	.	.
W1D-9	7 6 94	.	0.007	1.100	.	.	.	.	1709.0	.	.
W1D-9	7 11 94	.	0.015	1.100	.	.	.	.	1756.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
W1D-9	7 19 94	.	0.008	0.370	.	.	.	.	2140.0	.	.
W1D-9	7 21 94	.	0.009	0.980	.	.	.	.	1840.0	.	.
W1D-9	8 4 94	.	0.004	0.370	.	.	.	.	2143.0	.	.
W1D-9	8 11 94	.	0.006	0.900	.	.	.	.	1691.0	.	.
W1D-9	8 18 94	.	0.003	0.350	.	.	.	.	2003.0	.	.
W1D-9	8 25 94	.	0.002	0.200	.	.	.	.	1852.0	.	.
W1D-9	9 2 94	.	0.005	0.910	.	.	.	.	1898.0	.	.
W1D-9	10 12 94	.	.	1.400	.	.	.	.	.	.	.
W1D-9	10 12 94	.	.	1.400	.	.	.	.	.	.	.
W1D-9	10 27 94	.	.	1.080	.	.	.	.	.	.	.

The following sites are all upstream of the input (W1D-051)

Upstream1	6 7 93	7.1	0.280	0.620	0.008	0.060	.	.	1695.0	240.0	160.0
Upstream1	6 10 93	.	0.450	0.550	.	0.070	.	.	1096.0	270.0	290.0
Upstream1	6 14 93	6.6	0.280	0.620	0.007	0.070	.	.	1545.0	.	.
Upstream1	6 24 93	.	0.260	0.600	.	.	.	.	1484.0	.	.
Upstream1	6 28 93	6.7	0.250	0.410	.	.	.	.	1305.0	.	.
Upstream1	7 1 93	.	0.250	0.590	.	.	.	.	1413.0	.	.
Upstream1	7 6 93	.	0.030	0.400	.	.	.	.	1207.0	.	.
Upstream1	7 8 93	.	0.090	0.450	.	.	.	.	1327.0	.	.
Upstream1	7 12 93	6.5	0.060	0.360	.	.	.	.	1247.0	.	.
Upstream1	7 15 93	6.6	0.070	0.350	.	.	.	.	1351.0	.	.
Upstream1	7 19 93	6.7	0.110	0.430	.	.	.	.	2374.0	.	.
Upstream1	7 26 93	6.6	0.100	0.350	.	.	.	.	1095.0	.	.
Upstream1	8 2 93	6.7	0.050	0.320	.	.	.	.	827.0	.	.
Upstream1	8 16 93	8.0	0.040	0.100	.	.	.	.	599.0	.	.
Upstream1	6 3 94	6.7	0.190	0.460	.	.	.	.	1945.0	.	.
Upstream1	6 8 94	6.8	0.160	0.400	.	.	.	.	2265.0	.	.
Upstream1	6 15 94	6.7	0.080	0.270	.	.	.	.	2168.0	.	.
Upstream1	6 22 94	6.5	0.040	0.240	.	.	.	.	2010.0	.	.
Upstream1	6 27 94	7.1	0.030	0.300	.	.	.	.	1900.0	.	.
Upstream1	7 6 94	7.4	0.070	0.390	.	.	.	.	2049.0	.	.
Upstream1	7 11 94	7.0	0.050	0.310	.	.	.	.	1968.0	.	.
Upstream1	7 21 94	7.0	0.060	0.290	.	.	.	.	2003.0	.	.
Upstream1	7 29 94	7.2	0.050	0.290	.	.	.	.	2509.0	.	.
Upstream1	8 4 94	7.4	0.060	0.430	.	.	.	.	2319.0	.	.
Upstream1	8 11 94	6.7	0.070	0.390	.	.	.	.	1910.0	.	.
Upstream1	8 18 94	7.3	0.130	0.300	.	.	.	.	1956.0	.	.
Upstream1	8 25 94	6.6	0.140	0.310	.	.	.	.	1452.0	.	.
Upstream1	9 2 94	6.7	0.120	0.310	.	.	.	.	1542.0	.	.
Upstream2	8 26 92	.	0.080	3.900	0.140	0.110	.	.	1486.0	.	.
Upstream2	9 4 92	7.3	0.023	0.970	0.024	0.040	.	.	1388.0	.	.
Upstream2	9 23 92	.	0.310	3.900	0.150	0.130	.	.	2554.0	.	.
Upstream2	10 5 92	.	0.460	5.800	0.220	0.170	.	.	2332.0	.	.
Upstream2	10 15 92	.	0.540	9.100	0.390	0.250	.	.	2138.0	.	.
Upstream2	6 7 93	.	0.070	0.790	0.004	0.050	.	.	752.0	130.0	280.0
Upstream2	6 10 93	.	0.160	0.920	.	.	.	.	1223.0	160.0	140.0
Upstream2	6 14 93	.	0.060	0.920	0.004	0.050	.	.	1078.0	.	.
Upstream2	6 24 93	.	0.040	1.600	.	.	.	.	1256.0	.	.
Upstream2	6 28 93	.	0.120	2.000	.	.	.	.	1445.0	.	.
Upstream2	7 1 93	.	0.110	3.600	.	.	.	.	1588.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
Upstream2	7 6 93	.	0.040	3.300	.	.	.	.	1355.0	.	.
Upstream2	7 8 93	.	0.030	3.400	.	.	.	.	1615.0	.	.
Upstream2	7 12 93	.	0.070	5.100	.	.	.	.	2028.0	.	.
Upstream2	7 15 93	.	0.070	3.300	.	.	.	.	2133.0	.	.
Upstream2	7 19 93	.	0.050	2.600	.	.	.	.	1484.0	.	.
Upstream2	7 26 93	.	0.120	5.400	.	.	.	.	2395.0	.	.
Upstream2	8 2 93	.	0.130	4.300	.	.	.	.	2278.0	.	.
Upstream2	8 16 93	.	0.210	4.800	.	.	.	.	2385.0	.	.
Upstream2	6 3 94	.	0.140	3.100	.	.	.	.	1924.0	.	.
Upstream2	6 8 94	.	0.110	2.200	.	.	.	.	2035.0	.	.
Upstream2	6 15 94	.	0.030	1.200	.	.	.	.	1465.0	.	.
Upstream2	6 22 94	.	0.050	5.800	.	.	.	.	2350.0	.	.
Upstream2	6 27 94	.	0.040	4.700	.	.	.	.	1913.0	.	.
Upstream2	7 6 94	.	0.090	6.600	.	.	.	.	2545.0	.	.
Upstream2	7 11 94	.	0.040	3.600	.	.	.	.	2195.0	.	.
Upstream2	7 21 94	.	0.050	3.600	.	.	.	.	2119.0	.	.
Upstream2	7 29 94	.	0.050	3.500	.	.	.	.	2401.0	.	.
Upstream2	8 4 94	.	0.190	4.100	.	.	.	.	2073.0	.	.
Upstream2	8 4 94	.	0.190	4.000	.	.	.	.	2225.0	.	.
Upstream2	8 11 94	.	0.030	2.300	.	.	.	.	1497.0	.	.
Upstream2	8 18 94	.	0.050	1.500	.	.	.	.	1829.0	.	.
Upstream2	8 25 94	.	0.030	1.700	.	.	.	.	1852.0	.	.
Upstream2	9 2 94	.	0.013	3.100	.	.	.	.	1599.0	.	.
Upstream3	6 10 92	6.4	0.310	4.300	0.070	0.060	.	.	2678.0	.	.
Upstream3	6 15 92	7.2	0.240	6.700	0.050	0.040	.	.	2770.0	.	.
Upstream3	6 17 92	6.6	0.260	5.900	0.070	0.050	.	.	2541.0	.	.
Upstream3	6 22 92	7.1	0.210	5.400	0.050	0.090	.	.	2541.0	.	.
Upstream3	6 29 92	7.0	0.110	6.400	0.050	0.110	.	.	2274.0	.	.
Upstream3	7 6 92	6.9	0.160	6.900	0.070	0.120	.	.	2480.0	.	.
Upstream3	8 26 92	.	0.032	4.400	0.050	0.090	.	.	2233.0	.	.
Upstream3	9 4 92	6.7	0.029	2.700	0.032	0.060	.	.	2983.0	.	.
Upstream3	9 23 92	.	0.033	3.200	0.100	0.070	.	.	2676.0	.	.
Upstream3	10 5 92	7.0	0.032	4.600	0.100	0.080	.	.	2603.0	.	.
Upstream3	10 15 92	.	0.040	4.400	0.060	0.080	.	.	2323.0	.	.
Upstream3	6 7 93	.	0.040	1.200	0.005	0.060	.	.	1133.0	200.0	190.0
Upstream3	6 10 93	.	0.050	0.850	0.002	0.060	.	.	1968.0	180.0	180.0
Upstream3	6 14 93	.	0.040	1.100	0.006	0.070	.	.	1400.0	.	.
Upstream3	6 24 93	.	0.031	1.500	.	.	.	.	1297.0	.	.
Upstream3	6 28 93	.	0.030	1.600	.	.	.	.	1511.0	.	.
Upstream3	7 1 93	.	0.030	4.700	.	.	.	.	1743.0	.	.
Upstream3	7 6 93	.	0.030	4.500	.	.	.	.	1540.0	.	.
Upstream3	7 8 93	.	0.019	4.100	.	.	.	.	2456.0	.	.
Upstream3	7 12 93	.	0.028	5.500	.	.	.	.	2169.0	.	.
Upstream3	7 15 93	.	0.030	6.800	.	.	.	.	2447.0	.	.
Upstream3	7 19 93	.	0.040	8.300	.	.	.	.	2615.0	.	.
Upstream3	7 26 93	.	0.023	4.900	.	.	.	.	2298.0	.	.
Upstream3	8 2 93	.	0.030	5.900	.	.	.	.	2708.0	.	.
Upstream3	8 16 93	.	0.060	7.100	.	.	.	.	2524.0	.	.
Upstream3	6 3 94	.	0.090	3.800	.	.	.	.	2169.0	.	.
Upstream3	6 8 94	.	0.090	3.100	.	.	.	.	2359.0	.	.
Upstream3	6 15 94	.	0.030	0.880	.	.	.	.	1416.0	.	.
Upstream3	6 22 94	.	0.040	6.000	.	.	.	.	2595.0	.	.
Upstream3	6 27 94	.	0.040	4.000	.	.	.	.	2162.0	.	.
Upstream3	7 6 94	.	0.090	7.400	.	.	.	.	2629.0	.	.
Upstream3	7 11 94	.	0.040	4.600	.	.	.	.	2303.0	.	.

Table A1.1. W1D water quality data, collected by LTV from 1992-1999 (continued).

Site	Date	pH	Cu	Ni	Co	Zn	Fe	TSS	SO_4	Ca	Mg
Upstream3	7 21 94	.	0.015	2.500	.	.	.	.	2003.0	.	.
Upstream3	7 29 94	.	0.026	3.700	.	.	.	.	2354.0	.	.
Upstream3	8 4 94	.	0.080	5.500	.	.	.	.	2390.0	.	.
Upstream3	8 11 94	.	0.030	4.700	.	.	.	.	2284.0	.	.
Upstream3	8 18 94	.	0.030	4.700	.	.	.	.	2450.0	.	.
Upstream3	8 25 94	.	0.140	6.100	.	.	.	.	2308.0	.	.
Upstream3	9 2 94	.	0.100	5.400	.	.	.	.	2237.0	.	.

Note: a minus sign preceding a value indicates the value is less than detection limit. pH values are in standard units, and all other parameters are mg/L.

Table A1.2. Water quality data (1997-99) from the DNR study cell of the W1D treatment system.

Site	Year	#	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
W1D1 (the site on the influent side of the DNR study cell)											
W1D1	5 1 97	18013	1000	7.030	0.008	0.500	0.002	0.040	441.1	.	.
W1D1	5 15 97	18026	1150	7.090	0.007	0.500	0.001	0.030	550.0	.	.
W1D1	5 29 97	18041	1250	6.950	0.008	0.600	0.003	0.040	658.0	.	.
W1D1	6 12 97	18056	1600	6.890	0.003	0.300	0.008	0.020	840.0	.	.
W1D1	6 30 97	18070	1400	7.040	0.010	0.500	.	.	693.0	169.0	121.0
W1D1	7 14 97	18084	1500	7.100	0.002	0.300	.	.	903.0	168.0	151.0
W1D1	7 24 97	18100	1790	6.670	0.004	0.100	0.002	.	958.0	.	.
W1D1	8 7 97	18131	1800	7.040	0.008	0.300	0.001	.	1062.0	.	.
W1D1	8 22 97	18151	1575	6.750	<0.050	0.400	.	.	874.0	161.0	150.0
W1D1	9 4 97	18167	2000	6.860	<0.050	0.100	.	.	1105.0	206.0	187.0
W1D1	9 17 97	18182	2000	7.230	<0.050	0.300	.	.	1172.0	212.0	193.0
W1D1	10 1 97	18196	1900	7.350	<0.050	0.200	.	.	1075.0	204.0	186.0
W1D1	10 14 97	18210	1450	7.250	<0.050	0.400	.	.	768.0	140.0	126.0
W1D1	8 5 98	18228	2100	7.160	<0.050	0.100	<0.100	0.030	1108.0	230.0	237.0
W1D1	9 4 98	18234	1800	7.760	<0.050	0.100	<0.100	.	1146.0	220.0	228.0
W1D1	9 17 98	18238	2200	7.110	<0.050	0.200	<0.100	0.030	1179.0	223.0	232.0
W1D1	9 30 98	18243	2250	7.840	<0.050	0.300	<0.100	0.020	1124.0	219.0	215.0
W1D1	10 20 98	18254		.	0.004	0.380	0.011	0.033	645.0	.	.
W1D1	4 2 99	18269	900	7.410	0.047	0.933	0.036	0.081	470.0	.	.
W1D1	5 14 99	18279	1400	7.360	<0.002	0.288	<0.002	0.028	816.0	.	.
W1D1	6 15 99	18288	1600	7.220	.	.	.	.	.	.	.
W1D1	7 7 99	18294	1200	7.250	.	.	.	.	.	.	.
W1D1	8 6 99	18307	2275	.	0.007	0.258	0.002	.	.	247.0	249.0
W1D1	9 10 99	18316	1950	7.160	0.006	0.150	<0.002	.	.	214.0	200.0
W1D1	10 6 99	18325	1500	7.080	0.004	0.239	0.002	.	.	162.0	164.0

Table A1.2. Water quality data (1997-99) from the DNR study cell of the W1D treatment system (continued).

Site	Year	#	S.C.	pH	Cu	Ni	Co	Zn	SO4	Ca	Mg
W1D2 (one of the two sample sites on the effluent side of the DNR study cell)											
W1D2	5 1 97	18014	1150	7.140	0.004	0.400	<0.001	0.040	506.1	.	.
W1D2	5 15 97	18027	1150	7.130	0.004	0.500	<0.001	0.040	546.0	.	.
W1D2	5 29 97	18042	1300	7.000	0.003	0.400	<0.001	0.030	655.0	.	.
W1D2	6 12 97	18057	1500	7.220	0.002	0.200	0.004	0.020	770.0	.	.
W1D2	6 30 97	18071	1300	7.080	0.001	0.300	.	.	638.0	134.0	115.0
W1D2	7 14 97	18085	1400	7.390	0.002	0.200	.	.	824.0	152.0	145.0
W1D2	7 24 97	18101	1825	7.380	0.002	<0.100	0.001	.	911.0	.	.
W1D2	8 7 97	18132	1700	7.250	0.003	0.100	<0.001	.	960.0	.	.
W1D2	8 22 97	18152	1500	6.860	<0.050	0.300	.	.	850.0	161.0	144.0
W1D2	9 4 97	18168	1900	7.250	<0.050	0.100	.	.	1036.0	193.0	187.0
W1D2	9 17 97	18183	1950	7.290	<0.050	0.100	.	.	1090.0	197.0	192.0
W1D2	10 1 97	18197	1925	7.200	<0.050	0.100	.	.	1085.0	199.0	186.0
W1D2	10 14 97	18211	1375	7.260	<0.050	0.300	.	.	769.0	139.0	127.0
W1D2	9 4 98	18235	1900	7.770	<0.050	0.300	<0.100	.	1172.0	228.0	231.0
W1D2	9 17 98	18239	2000	7.440	<0.050	0.200	<0.100	0.020	1078.0	212.0	229.0
W1D2	10 20 98	18255		.	<0.001	0.274	0.008	0.023	599.0	.	.
W1D2	4 2 99	18268	1000	7.440	0.035	0.875	0.028	0.064	545.0	.	.
W1D2	5 14 99	18278	1375	7.440	<0.002	0.210	<0.002	0.022	775.0	.	.
W1D2	6 15 99	18287	1500	7.050	.	.	.	.	.	.	.
W1D2	7 7 99	18296	1125	7.250	.	.	.	.	.	.	.
W1D2	8 6 99	18308	2100	7.160	0.004	0.086	<0.002	.	.	244.0	249.0
W1D2	9 10 99	18317	1750	7.230	0.004	0.131	<0.002	.	.	208.0	203.0
W1D2	10 6 99	18326	1550	7.280	0.007	0.276	0.002	0.010	.	161.0	164.0

Table A1.2. Water quality data (1997-99) from the DNR study cell of the W1D treatment system (continued).

Site	Year	#	S.C.	pH	Cu	Ni	Co	Zn	SO4	Ca	Mg
W1D3 (one of the two sample sites on the effluent side of the DNR study cell)											
W1D3	5 1 97	18015	1200	7.070	0.002	0.300	<0.001	0.030	517.5	.	.
W1D3	5 15 97	18028	1150	7.120	0.004	0.500	<0.001	0.030	542.0	.	.
W1D3	5 29 97	18043	1300	7.060	0.003	0.400	<0.001	0.030	650.0	.	.
W1D3	6 12 97	18058	1300	7.130	0.002	0.200	0.002	.	796.0	.	.
W1D3	6 30 97	18072	1300	7.180	<0.001	0.300	.	.	643.0	130.0	116.0
W1D3	7 14 97	18086	1400	7.320	0.001	0.300	.	.	793.0	156.0	148.0
W1D3	7 24 97	18102	1750	7.290	0.002	0.100	0.002	.	890.0	.	.
W1D3	8 7 97	18133	1650	7.310	0.012	0.100	<0.001	.	949.0	.	.
W1D3	8 22 97	18153	1525	6.860	<0.050	0.300	.	.	807.0	156.0	140.0
W1D3	9 4 97	18169	1850	7.200	<0.050	0.100	.	.	994.0	190.0	182.0
W1D3	9 17 97	18184	1925	7.300	<0.050	0.100	.	.	1026.0	191.0	189.0
W1D3	10 1 97	18198	1875	7.260	.	.	.	.	.	.	.
W1D3	10 14 97	18212	1400	7.270	<0.050	0.400	.	.	752.0	138.0	126.0
W1D3	8 5 98	18229	2100	7.130	<0.050	0.300	<0.100	0.020	1068.0	225.0	229.0
W1D3	9 17 98	18240	2000	7.450	<0.050	0.300	<0.100	0.100	967.0	202.0	213.0
W1D3	10 20 98	18256		.	0.001	0.289	0.009	0.024	592.0	.	.
W1D3	4 2 99	18267	900	7.500	0.029	0.793	0.024	0.064	499.0	.	.
W1D3	5 14 99	18277	1450	7.500	<0.002	0.205	<0.002	0.019	781.0	.	.
W1D3	6 15 99	18286	1500	7.120	.	.	.	.	.	.	.
W1D3	7 7 99	18292	1100	7.180	.	.	.	.	.	.	.
W1D3	8 6 99	18309	2200	7.140	0.004	<0.002	<0.002	.	.	238.0	246.0
W1D3	9 10 99	18318	1750	7.250	0.004	0.154	<0.002	.	.	213.0	202.0
W1D3	10 6 99	18327	1500	7.340	0.007	0.240	<0.002	.	.	156.0	158.0

Table A1.2. Water quality data (1997-99) from the DNR study cell of the W1D treatment system (continued).

Site	Year	#	S.C.	pH	Cu	Ni	Co	Zn	SO4	Ca	Mg
W1DH (equivalent to W1D-051; the input to the entire treatment system)											
W1DH	10 20 98	18262		.	.	.	.	.	646.0	.	.
W1DH	4 2 99	18270	950	7.000	0.055	0.771	0.032	0.028	443.0	.	.
W1DH	4 27 99	18273	1500	7.160	0.036	0.547	0.004	0.029	851.0	.	.
W1DH	5 14 99	18276	1750	7.200	.	.	.	.	998.0	.	.
W1DH	6 15 99	18285	1950	6.910	.	.	.	.	.	.	.
W1DH	7 7 99	18291	1175	6.820	.	.	.	.	.	.	.
W1DH	7 28 99	18303	2100	7.080	.	<0.100	<0.100	1.980	1230.0	236.0	238.0
W1DH	8 6 99	18306	2400	7.070	0.003	0.020	<0.002	.	.	264.0	266.0
W1DH	8 20 99	18312	2200	7.080	0.010	0.252	0.004	.	.	265.0	274.0
W1DH	9 10 99	18315	1900	7.050	0.003	0.014	<0.002	.	.	215.0	198.0
W1DH	9 23 99	18321	1750	7.090	0.004	0.235	0.002	0.010	.	262.0	256.0
W1DH	10 6 99	18324	1650	7.000	.	.	.	.	.	.	.

Note: Specific conductance (S.C.) values are in microsiemens, pH values are in standard units, and all other parameters are in mg/L.

Table A1.3. Water quality data from the input (051) and output (052) of the W1D wetland treatment system, obtained during the 1993 study of the Dunka Mine conducted by the DNR.

Site	Date	S.C.	pH	SO4	Cu	Ni	Co	Zn	Ca	Mg	O.S.S.	Flow rate (gpm) Bucket Gage	Flow rate (gpm) Stream Gage
W1D-051	7 16 93	.	.	.	.	.	.	.	.	.	2.00	95.0	.
W1D-051	7 20 93	3850.0	7.20	2440.0	0.060	6.300	0.060	0.040	295.0	410.0	1.90	55.7	.
W1D-051	7 26 93	3400.0	6.69	2640.0	0.080	6.800	0.090	0.050	315.0	425.0	1.90	66.3	.
W1D-051	7 28 93	.	.	.	.	.	.	.	.	.	1.92	74.2	.
W1D-051	7 29 93	.	.	.	.	.	.	.	.	.	1.94	79.4	.
W1D-051	8 2 93	3700.0	6.82	.	0.070	7.180	0.070	0.080	325.0	399.0	1.88	56.6	.
W1D-051	8 3 93	.	.	.	.	.	.	.	.	.	1.90	67.3	.
W1D-051	8 4 93	3400.0	6.93	.	0.070	6.800	0.080	0.080	326.0	407.0	1.87	53.5	.
W1D-051	8 6 93	.	.	.	.	.	.	.	.	.	1.86	53.9	.
W1D-051	8 9 93	3150.0	7.06	.	.	.	.	.	.	.	1.98	96.3	.
W1D-051	8 11 93	.	.	.	0.020	0.930	0.030	0.020	233.0	279.0	1.86	50.4	.
W1D-051	8 13 93	.	.	.	.	.	.	.	.	.	1.85	44.4	.
W1D-051	8 17 93	3900.0	7.06	.	.	.	.	.	.	.	1.84	41.0	.
W1D-051	8 20 93	.	.	.	.	.	.	.	.	.	1.82	36.4	37.00
W1D-051	8 25 93	4200.0	6.90	.	.	.	.	.	.	.	1.83	33.8	28.00
W1D-051	8 27 93	.	.	.	.	.	.	.	.	.	1.81	34.9	.
W1D-051	8 30 93	3890.0	6.95	.	.	.	.	.	.	.	1.88	52.8	45.70
W1D-051	9 16 93	3650.0	6.94	.	.	.	.	.	.	.	1.80	30.1	27.10
W1D-052	7 16 93	.	.	.	.	.	.	.	.	.	4.66	222.0	.
W1D-052	7 20 93	2480.0	7.61	1250.0	0.020	0.930	0.020	0.010	205.0	255.0	4.34	78.4	.
W1D-052	7 26 93	2650.0	7.13	1870.0	0.030	0.800	0.040	0.010	250.0	320.0	4.38	96.4	.
W1D-052	7 28 93	.	.	.	.	.	.	.	.	.	4.46	130.0	.
W1D-052	7 29 93	.	.	.	.	.	.	.	.	.	4.48	152.0	.
W1D-052	8 2 93	2600.0	7.05	.	0.030	1.010	0.020	0.020	229.0	267.0	4.30	55.5	.
W1D-052	8 3 93	.	.	.	.	.	.	.	.	.	4.40	52.3	.
W1D-052	8 4 93	2600.0	7.30	.	0.020	0.980	0.040	0.020	235.0	279.0	4.30	63.1	.
W1D-052	8 6 93	.	.	.	.	.	.	.	.	.	4.30	64.6	.
W1D-052	8 9 93	2450.0	7.48	.	0.070	6.200	0.090	0.070	294.0	359.0	4.48	144.0	.

Table A1.3. Water quality data from the input (051) and output (052) of the W1D wetland treatment system, obtained during the 1993 study of the Dunka Mine conducted by the DNR (continued).

Site	Date	S.C.	pH	SO4	Cu	Ni	Co	Zn	Ca	Mg	O.S.S.	Flow rate (gpm) Bucket Gage	Flow rate (gpm) Stream Gage
W1D-052	8 11 93	.	.	.	.	.	.	.	.	.	4.25	35.2	.
W1D-052	8 13 93	.	.	.	.	.	.	.	.	.	4.24	34.9	.
W1D-052	8 16 93	2900.0	7.49	.	.	.	.	.	.	.	4.24	33.2	.
W1D-052	8 20 93	.	.	.	.	.	.	.	.	.	4.20	27.4	54.00
W1D-052	8 25 93	3450.0	7.32	.	.	.	.	.	.	.	4.18	19.8	18.80
W1D-052	8 27 93	.	.	.	.	.	.	.	.	.	4.20	27.7	.
W1D-052	8 30 93	2910.0	7.46	.	.	.	.	.	.	.	4.38	90.4	110.60
W1D-052	9 16 93	3000.0	7.44	.	.	.	.	.	.	.	4.20	24.5	49.60

Notes: The bucket gage values are the average value (gpm) of four buckets. S.C. values are microsiemens, metals and sulfate are mg/L, and O.S.S. values are in feet.

These samples were collected in 1993 by an automatic water sampler:

Site	Sample	Date	Time (hr)	pH	S.C.
W1D-051	1	9 2 93	1430	7.80	2410
W1D-051	2	9 2 93	1600	7.85	2400
W1D-052	1	9 2 93	1500	8.11	2800
W1D-052	2	9 2 93	1630	8.22	2780
W1D-052	3	9 2 93	1800	8.19	2800
W1D-052	4	9 2 93	1930	8.21	2800
W1D-052	5	9 2 93	2100	8.17	2810
W1D-052	6	9 2 93	2230	8.19	2880
W1D-052	7	9 2 93	2400	8.17	2850
W1D-052	8	9 3 93	130	8.17	2820
W1D-052	9	9 3 93	300	8.15	2820
W1D-052	10	9 3 93	430	8.15	2880
W1D-052	11	9 3 93	600	8.10	2880
W1D-052	12	9 3 93	730	8.11	2870
W1D-052	13	9 3 93	900	8.11	2880
W1D-052	14	9 3 93	1030	8.11	2850
W1D-052	15	9 3 93	1200	8.12	2880
W1D-052	16	9 3 93	1330	8.14	2860
W1D-052	17	9 3 93	1500	8.14	2840
W1D-052	18	9 3 93	1630	8.16	2850
W1D-052	19	9 3 93	1800	8.13	2880
W1D-052	20	9 3 93	1930	8.22	2840

Notes: Staff gage for 051-#1 was 2.01, 4.66 for 052-#1.

Table A1.4. Water quality data for individual seeps at Seep 1; from samples collected by the DNR in 1993.

Seep	Date	pH	S.C.	Sulfate	Cu	Ni	Co	Zn	Ca	Mg	OSS	Flow (L/sec)
S1	7 15 93	6.93	1550.0	.	0.28	10.60	0.61	1.02	.	.	1.28	0.485
S1	7 16 93	.	.	.	.	.	.	.	.	.	1.40	1.218
S1	7 19 93	.	.	.	.	.	.	.	.	.	1.28	0.662
S1	7 20 93	6.80	1710.0	1090.0	0.35	9.67	0.50	1.00	190.0	151.0	.	.
S1	7 26 93	.	.	.	.	.	.	.	.	.	1.28	0.719
S1	7 27 93	6.80	1900.0	1330.0	0.28	10.70	0.61	1.06	200.0	180.0	.	.
S1	7 28 93	.	.	.	.	.	.	.	.	.	1.36	0.776
S1	8 2 93	.	.	.	.	.	.	.	.	.	1.28	0.479
S1	8 3 93	.	.	.	.	.	.	.	.	.	1.28	0.562
S1	8 4 93	7.09	1700.0	955.0	0.31	10.00	0.56	1.21	188.0	156.0	1.28	0.498
S1	8 5 93	.	.	.	.	.	.	.	.	.	1.26	0.744
S1	8 9 93	7.15	1720.0	1125.0	0.25	9.83	0.53	0.98	183.0	164.0	1.34	Med
S1	8 16 93	7.39	1920.0	1150.0	0.22	10.00	0.57	1.12	228.0	182.0	1.24	0.404
S1	8 17 93	7.25	2100.0	1150.0	0.22	10.00	0.54	1.10	234.0	190.0	1.24	0.300
S1	8 30 93	7.19	2340.0	1430.0	0.13	8.97	0.46	0.78	242.0	204.0	1.25	.
S1	9 16 93	7.23	1800.0	1015.0	0.12	6.43	0.32	0.74	206.0	160.0	1.23	0.267
S1	9 28 93	7.17	1950.0	1095.0	0.09	6.27	0.26	0.58	218.0	168.0	1.23	0.240
S2	7 15 93	4.55	2100.0	.	1.42	15.80	0.97	2.74	.	.	.	Hi
S2	7 20 93	4.68	1900.0	1370.0	1.26	13.00	0.69	2.25	175.0	170.0	.	Hi
S2	7 27 93	4.46	2000.0	1340.0	1.38	15.00	0.90	3.49	219.0	209.0	.	.
S2	8 4 93	4.58	1700.0	1070.0	1.15	11.00	0.71	2.18	181.0	163.0	.	Low
S2	8 10 93	4.67	1800.0	1090.0	1.08	13.00	0.75	3.15	186.0	160.0	.	Low
S2	8 17 93	4.38	2550.0	1660.0	1.46	17.00	1.07	3.80	252.0	252.0	.	0.023
S2	8 30 93	4.64	2680.0	1710.0	1.30	17.00	1.04	3.80	254.0	248.0	.	0.023
S2	9 16 93	4.77	2450.0	1600.0	1.04	16.00	0.84	3.19	264.0	238.0	.	0.009

Table A1.4. Water quality data for individual seeps at Seep 1; from samples collected by the DNR in 1993, continued.

Seep	Date	pH	S.C.	Sulfate	Cu	Ni	Co	Zn	Ca	Mg	OSS	Flow (L/sec)
S3	7 15 93	4.41	2700.0	.	2.36	28.00	1.84	4.89	.	.	.	Med
S3	7 20 93	4.52	2650.0	1840.0	1.96	24.00	1.40	4.13	240.0	255.0	.	Med
S3	7 27 93	4.29	2800.0	2320.0	2.37	29.00	1.89	6.65	265.0	313.0	.	.
S3	8 4 93	4.38	2760.0	1880.0	2.23	24.00	1.75	6.52	268.0	301.0	.	Low
S3	8 10 93	4.34	1820.0	2030.0	2.05	25.00	1.69	6.84	280.0	276.0	.	Low
S3	8 17 93	4.20	3600.0	2420.0	2.51	30.00	2.14	7.94	336.0	370.0	.	0.006
S3	8 30 93	4.38	3460.0	2280.0	1.94	28.00	2.07	7.42	302.0	338.0	.	0.004
S4	7 15 93	4.44	3410.0	.	6.33	55.00	3.74	4.13	.	.	.	Hi
S4	7 20 93	4.55	3550.0	2640.0	5.78	48.00	3.14	3.54	325.0	325.0	.	Hi
S4	7 27 93	4.26	3000.0	2580.0	5.74	45.00	3.23	3.94	344.0	316.0	.	.
S4	8 4 93	4.50	3000.0	1880.0	5.09	40.00	3.08	4.38	332.0	313.0	.	Low
S4	8 10 93	4.46	2920.0	1980.0	4.78	43.00	2.99	4.30	312.0	298.0	.	Low
S4	8 17 93	4.43	3360.0	1960.0	4.68	42.00	3.21	4.61	338.0	338.0	.	0.017
S4	8 30 93	4.57	3320.0	2150.0	2.97	36.00	2.74	4.58	306.0	318.0	.	0.006
S4	9 16 93	4.79	3100.0	2180.0	2.57	30.00	2.03	3.16	352.0	316.0	.	0.002
S5	7 15 93	4.47	1600.0	.	4.43	27.00	1.87	1.59	.	.	.	Low
S5	7 20 93	4.57	1620.0	1035.0	4.02	29.00	1.81	1.49	160.0	115.0	.	Low
S5	7 27 93	4.23	1450.0	1005.0	3.50	24.00	1.65	1.59	148.0	112.0	.	.
S5	8 4 93	4.51	1150.0	600.0	3.31	20.00	1.45	1.33	123.0	84.0	.	0.000
S5	8 10 93	4.51	1210.0	635.0	2.97	22.00	1.99	1.30	114.0	78.0	.	0.000
S5	8 17 93	4.49	1320.0	690.0	2.70	19.00	1.49	1.29	130.0	84.0	.	0.000
S5	8 30 93	4.56	1510.0	880.0	2.46	19.00	1.48	1.36	146.0	104.0	.	0.000
S5	9 16 93	4.69	2500.0	1640.0	3.23	27.00	1.98	2.09	290.0	214.0	.	0.000

Table A1.4. Water quality data for individual seeps at Seep 1; from samples collected by the DNR in 1993, continued.

Seep	Date	pH	S.C.	Sulfate	Cu	Ni	Co	Zn	Ca	Mg	OSS	Flow (L/sec)
S6	7 15 93	4.51	1600.0	.	3.20	19.70	1.49	1.60	.	.	.	Low
S6	7 20 93	4.64	1410.0	890.0	2.97	20.00	1.36	1.33	160.0	90.0	.	Low
S6	7 27 93	4.32	1300.0	850.0	2.54	16.00	1.22	1.45	151.0	88.0	.	.
S6	8 4 93	4.57	1300.0	730.0	2.64	16.00	1.30	1.42	146.0	88.0	.	0.000
S6	8 10 93	4.53	1400.0	735.0	2.49	17.00	1.32	1.44	154.0	86.0	.	0.000
S6	8 17 93	4.49	1490.0	900.0	2.53	17.00	1.40	1.54	164.0	94.0	.	0.000
S6	8 30 93	4.58	1390.0	700.0	2.08	14.00	1.16	1.27	142.0	84.0	.	0.000
S6	9 16 93	4.79	2100.0	1080.0	2.43	17.00	1.30	1.46	194.0	116.0	.	0.000
S7	7 15 93	4.57	2450.0	.	5.21	25.00	2.12	2.61	.	.	.	Med
S7	7 20 93	4.78	2200.0	1360.0	3.94	23.00	1.71	2.01	285.0	165.0	.	Med
S7	7 27 93	4.38	2100.0	1380.0	3.52	19.00	1.62	2.17	282.0	165.0	.	.
S7	8 4 93	4.62	2000.0	1420.0	3.74	20.00	1.72	2.06	279.0	155.0	.	Low
S7	8 10 93	4.84	1880.0	1115.0	1.59	19.00	1.53	2.12	242.0	126.0	.	0.000
S7	8 17 93	4.67	2000.0	1430.0	1.63	19.00	1.62	2.15	238.0	132.0	.	0.000
S7	8 30 93	4.81	2020.0	1130.0	1.29	17.00	1.48	1.95	218.0	132.0	.	Low
S8	7 15 93	4.61	1950.0	.	1.45	11.90	0.70	2.50	.	.	.	Low
S8	7 20 93	4.72	1780.0	1140.0	1.20	9.51	0.55	1.95	175.0	155.0	.	Low
S8	7 27 93	4.38	2150.0	1300.0	1.61	13.00	0.80	2.84	227.0	205.0	.	.
S8	8 4 93	4.63	1900.0	1315.0	1.42	10.00	0.69	2.26	196.0	181.0	.	Med
S8	8 10 93	4.61	1900.0	1180.0	1.30	11.00	0.73	2.23	202.0	162.0	.	Low
S8	8 17 93	4.58	2250.0	1469.0	1.44	12.00	0.84	2.40	234.0	210.0	.	0.014
S8	8 30 93	4.58	2620.0	1550.0	1.50	14.00	0.95	2.70	260.0	230.0	.	0.023
S8	9 16 93	4.74	2000.0	1365.0	1.14	9.90	0.68	2.04	216.0	190.0	.	0.015

Table A1.4. Water quality data for individual seeps at Seep 1; from samples collected by the DNR in 1993, continued.

Seep	Date	pH	S.C.	Sulfate	Cu	Ni	Co	Zn	Ca	Mg	OSS	Flow (L/sec)
S9	7 15 93	4.70	1800.0	.	1.07	9.90	0.58	2.28	.	.	.	Med
S9	7 20 93	5.08	1600.0	1030.0	0.68	6.73	0.38	1.63	145.0	140.0	.	Med
S9	7 27 93	4.62	2000.0	1280.0	1.07	10.00	0.64	2.59	209.0	198.0	.	.
S9	8 4 93	4.76	1750.0	1085.0	0.91	9.14	0.54	2.06	176.0	169.0	.	Low
S9	8 10 93	4.82	1800.0	1040.0	1.78	8.90	0.54	2.00	174.0	150.0	.	0.000
S9	8 17 93	4.65	2150.0	1430.0	1.16	11.00	0.78	2.36	222.0	204.0	.	0.000
S9	8 30 93	4.66	2530.0	1590.0	1.18	12.00	0.84	2.57	252.0	228.0	.	0.008
S9	9 16 93	4.88	2000.0	1300.0	0.88	8.73	0.61	1.87	202.0	186.0	.	0.008
S10	7 15 93	4.57	2350.0	.	2.60	23.00	1.44	3.97	.	.	.	0.000
S10	7 20 93	4.70	2240.0	1440.0	2.14	20.00	1.14	3.00	235.0	195.0	.	0.000
S10	7 27 93	4.40	2210.0	1320.0	2.24	20.00	1.20	3.78	254.0	221.0	.	.
S10	8 4 93	4.66	1900.0	1335.0	1.98	19.00	1.26	2.90	221.0	185.0	.	0.000
S10	8 10 93	4.70	1790.0	1065.0	1.60	14.00	1.16	2.43	198.0	136.0	.	0.000
S10	8 17 93	4.69	2250.0	735.0	1.41	22.00	1.50	2.59	252.0	194.0	.	0.000
S10	8 30 93	4.68	2400.0	1400.0	1.77	20.00	1.40	2.61	256.0	184.0	.	0.000

Notes: All flow measurements were taken under low flow conditions, with the exception of S1, which was bucket gaged. S10 had no visual flow, so water quality samples were taken from standing water. Qualitative flow descriptors may be estimated as follows:

Low = <0.063 L/sec (<1 gpm)
Med = 0.063 to 0.505 L/sec (1 to 8 gpm)
Hi = >0.505 L/sec (>8 gpm)

Specific conductance values are in microsiemens, pH values are in standard units, and all other water quality parameters are in mg/L.

Table A1.5. Water quality data for Seep 1; samples collected by the DNR from 1997-1999.

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO4	Ca	Mg
Pool Samples											
043-1	5 1 97	18000	800.0	6.64	0.100	1.800	0.100	0.280	284.4	.	.
043-1	5 15 97	18016	800.0	6.69	0.200	2.200	0.065	0.540	456.0	.	.
043-1	5 29 97	18029	1000.0	6.62	0.200	4.600	0.200	0.650	601.0	.	.
043-1	6 12 97	18044	2500.0	5.81	2.000	26.400	1.400	2.640	1594.0	.	.
043-1	6 30 97	18059	2100.0	4.93	4.400	26.700	.	.	1598.0	276.0	217.0
043-1	7 14 97	18073	3000.0	5.44	2.400	19.700	.	.	1499.0	287.0	199.0
043-1	7 24 97	18087	2200.0	6.27	1.000	14.900	0.700	1.600	1302.0	.	.
043-1	8 7 97	18116	1800.0	6.30	0.900	14.000	0.600	1.370	1440.0	.	.
043-1	8 22 97	18134	1790.0	5.84	1.600	11.700	.	.	1150.0	230.0	161.0
043-1	9 4 97	18154	1950.0	6.39	0.890	14.300	.	.	1393.0	267.0	181.0
043-1	9 17 97	18170	1700.0	6.37	1.610	13.200	.	.	1116.0	241.0	125.0
043-1	10 1 97	18185	2200.0	5.92	1.100	13.000	.	.	1555.0	274.0	214.0
043-1	10 14 97	18199	1200.0	6.50	0.720	9.400	.	.	795.0	178.0	80.9
043-2	5 1 97	18001	600.0	6.82	0.100	1.800	0.100	0.300	272.9	.	.
043-2	5 15 97	18017	750.0	6.85	0.100	2.100	0.050	0.390	407.0	.	.
043-2	5 29 97	18030	750.0	6.76	0.200	4.100	0.100	0.440	424.0	.	.
043-2	6 12 97	18045	2500.0	5.88	2.100	25.400	1.300	2.680	1524.0	.	.
043-2	6 30 97	18060	2000.0	5.39	1.900	18.200	.	.	1230.0	218.0	170.0
043-2	7 14 97	18074	2700.0	6.14	1.300	17.400	.	.	1295.0	246.0	178.0
043-2	7 24 97	18088	2075.0	6.19	1.300	15.400	0.700	1.900	1384.0	.	.
043-2	8 7 97	18117	1850.0	6.44	0.900	14.000	0.600	1.690	1324.0	.	.
043-2	8 22 97	18135	975.0	6.37	0.900	7.500	.	.	519.0	116.0	65.7
043-2	9 4 97	18155	2100.0	6.54	0.860	14.600	.	.	1392.0	255.0	191.0
043-2	9 17 97	18171	2125.0	6.54	1.200	15.600	.	.	1402.0	277.0	177.0
043-2	10 1 97	18186	2100.0	6.23	1.100	16.200	.	.	1422.0	274.0	189.0
043-2	10 14 97	18200	1350.0	6.73	0.340	6.800	.	.	816.0	159.0	102.0

Table A1.5. Water quality data for the Seep 1 samples collected by DNR from 1997-1999 (continued).

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
043-3	5 1 97	18002	500.0	7.04	0.063	0.500	0.017	0.100	143.1	.	.
043-3	5 15 97	18018	500.0	6.98	0.100	1.500	0.056	0.260	207.0	.	.
043-3	5 29 97	18031	400.0	7.17	0.039	0.300	0.007	0.060	133.0	.	.
043-3	6 12 97	18046	2600.0	5.04	3.200	31.000	1.900	3.170	1760.0	.	.
043-3	6 30 97	18061	2200.0	5.35	2.700	24.900	.	.	1531.0	260.0	215.0
043-3	7 14 97	18075	2500.0	5.09	2.900	24.300	.	.	1627.0	300.0	228.0
043-3	7 24 97	18089	2425.0	5.19	2.300	19.900	1.100	2.400	1563.0	.	.
043-3	8 7 97	18118	2050.0	6.05	1.400	16.000	0.800	2.460	1538.0	.	.
043-3	8 22 97	18136	1325.0	6.70	0.370	7.200	.	.	681.0	236.0	94.6
043-3	9 4 97	18156	2350.0	6.30	1.180	17.000	.	.	1604.0	275.0	222.0
043-3	9 17 97	18172	2275.0	6.70	0.500	14.600	.	.	1518.0	283.0	212.0
043-3	10 1 97	18187	2200.0	6.37	0.800	16.000	.	.	1572.0	292.0	213.0
043-3	10 14 97	18201	1500.0	6.62	0.360	7.900	.	.	917.0	178.0	118.0
043-4	5 1 97	18003	.	.	0.100	1.700	0.065	0.270	.	.	.
043-4	5 15 97	18019	750.0	6.94	0.100	2.100	0.042	0.380	370.0	.	.
043-4	5 29 97	18032	1050.0	6.91	0.074	3.200	0.063	0.500	616.0	.	.
043-4	6 12 97	18047	2400.0	5.91	1.500	21.000	1.000	2.580	1531.0	.	.
043-4	6 30 97	18062	2000.0	5.38	2.200	18.600	.	.	1395.0	247.0	204.0
043-4	7 14 97	18076	2600.0	5.86	1.600	17.000	.	.	1609.0	285.0	221.0
043-4	7 24 97	18090	2390.0	6.28	0.700	15.100	0.600	1.700	1390.0	.	.
043-4	8 7 97	18119	2050.0	6.56	0.500	13.300	0.500	1.540	1518.0	.	.
043-4	8 22 97	18137	1775.0	5.98	1.400	11.100	.	.	1092.0	224.0	150.0
043-4	9 4 97	18157	2250.0	6.68	0.500	12.600	.	.	1451.0	270.0	196.0
043-4	9 17 97	18173	2225.0	6.75	0.480	13.100	.	.	1472.0	281.0	197.0
043-4	10 1 97	18188	2000.0	6.60	0.600	12.400	.	.	1367.0	264.0	178.0
043-4	10 14 97	18202	1475.0	5.96	1.130	9.500	.	.	888.0	162.0	110.0
043-5	5 1 97	18004	700.0	6.94	0.100	1.300	0.057	0.260	294.4	.	.
043-5	5 15 97	18020	650.0	6.97	0.077	1.600	0.039	0.270	.	.	.
043-5	5 29 97	18033	950.0	6.80	0.500	5.800	0.200	0.580	512.0	.	.

Table A1.5. Water quality data for the Seep 1 samples collected by DNR from 1997-1999 (continued).

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
043-5	6 12 97	18048	2500.0	5.86	2.300	27.400	1.400	2.820	1601.0	.	.
043-5	6 30 97	18063	1900.0	5.56	2.100	19.000	.	.	1279.0	226.0	180.0
043-5	7 14 97	18077	2400.0	6.13	1.200	18.400	.	.	1487.0	275.0	200.0
043-5	7 24 97	18091	2310.0	6.37	1.000	16.300	0.600	1.800	1418.0	.	.
043-5	8 7 97	18120	2100.0	6.51	0.600	13.700	0.500	1.570	1445.0	.	.
043-5	8 22 97	18138	1600.0	6.43	0.780	11.100	.	.	1015.0	209.0	144.0
043-5	9 4 97	18158	2100.0	6.73	0.560	13.500	.	.	1450.0	271.0	203.0
043-5	9 17 97	18174	2200.0	6.83	0.400	13.300	.	.	1463.0	274.0	202.0
043-5	10 1 97	18189	2100.0	6.79	0.400	.	.	.	1432.0	273.0	187.0
043-5	10 14 97	18203	1400.0	6.71	0.400	8.100	.	.	879.0	166.0	111.0
043-6	5 1 97	18005	800.0	6.84	0.200	1.600	0.031	0.200	360.8	.	.
043-6	5 15 97	18021	700.0	6.92	0.076	3.000	0.043	0.290	318.0	.	.
043-6	5 29 97	18034	850.0	6.96	0.100	1.400	0.008	0.120	459.0	.	.
043-6	6 12 97	18049	1900.0	6.65	0.200	8.600	0.052	0.270	1197.0	.	.
043-6	6 30 97	18064	1700.0	6.39	0.900	15.300	.	.	1128.0	210.0	158.0
043-6	7 14 97	18078	2400.0	6.42	0.900	18.500	.	.	1500.0	281.0	205.0
043-6	7 24 97	18092	2375.0	6.61	0.400	15.400	0.400	1.500	1447.0	.	.
043-6	8 7 97	18121	2000.0	6.64	0.300	12.800	0.300	1.460	1482.0	.	.
043-6	8 22 97	18139	1750.0	6.65	0.390	11.000	.	.	1149.0	239.0	157.0
043-6	9 4 97	18159	2100.0	6.92	0.200	11.200	.	.	1405.0	263.0	190.0
043-6	9 17 97	18175	2175.0	7.12	0.200	11.200	.	.	1432.0	273.0	198.0
043-6	10 1 97	18190	2050.0	7.09	0.180	10.400	.	.	1333.0	258.0	178.0
043-6	10 14 97	18204	1500.0	6.69	0.540	9.000	.	.	964.0	177.0	124.0
043-6	6 2 98	18215	2200.0	6.75	0.600	19.500	0.600	1.960	1268.0	255.0	223.0
043-6	6 10 98	18218	2100.0	6.81	0.200	14.900	0.200	1.400	1235.0	258.0	206.0
043-6	6 24 98	18220	1850.0	6.53	0.400	14.900	0.300	1.630	1272.0	231.0	183.0
043-6	7 10 98	18222	2100.0	6.75	0.100	8.400	0.100	0.920	1302.0	279.0	196.0
043-6	7 21 98	18224	2100.0	7.13	0.200	15.000	0.200	1.360	1413.0	292.0	216.0
043-6	8 5 98	18226	2300.0	7.33	0.170	13.800	0.100	1.100	1475.0	326.0	218.0
043-6	8 20 98	18230	1900.0	6.91	0.260	10.000	0.200	1.420	1143.0	269.0	173.0

Table A1.5. Water quality data for the Seep 1 samples collected by DNR from 1997-1999 (continued).

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
043-6	9 4 98	18232	2000.0	.	0.100	9.900	.	1.220	1361.0	298.0	194.0
043-6	9 17 98	18236	2100.0	7.13	0.100	9.000	0.100	1.030	1353.0	294.0	193.0
043-6	9 30 98	18241	2000.0	7.53	0.100	8.900	0.100	1.080	1250.0	280.0	180.0
043-6	10 16 98	18244	1600.0	6.44	0.098	8.060	0.116	0.932	932.0	.	.
043-6	10 20 98	18252	1300.0	5.44	0.330	7.400	0.629	.	529.0	.	.
043-6	11 3 98	18263	2150.0	6.63	0.118	10.400	0.285	1.950	1366.0	.	.
043-6	4 2 99	18265	110.0	6.67	.	.	.	.	36.9	.	.
043-6	4 27 99	18271	900.0	6.72	0.062	6.080	0.142	0.609	519.0	.	.
043-6	5 14 99	18274	1400.0	6.35	0.193	7.771	0.160	0.991	971.0	.	.
043-6	6 15 99	18283	2000.0	5.34	1.191	10.470	0.533	1.513	1318.0	.	.
043-6	7 7 99	18289	1275.0	4.74	1.520	10.900	0.647	1.670	752.0	.	.
043-6	7 28 99	18301	1700.0	5.25	1.420	13.900	0.890	1.990	1261.0	231.0	183.0
043-6	8 6 99	18304	2150.0	5.64	0.555	.	.	.	1500.0	276.0	220.0
043-6	8 20 99	18310	1900.0	5.89	1.060	13.500	0.790	2.130	1378.0	253.0	203.0
043-6	9 10 99	18313	1450.0	6.34	0.450	8.400	0.410	1.410	1011.0	199.0	139.0
043-6	9 23 99	18319	2150.0	6.35	0.550	9.420	0.480	1.630	1326.0	278.0	178.0
043-6	10 8 99	18322	.	.	.	.	.	.	.	.	.
043-7	5 1 97	18006	800.0	7.00	0.100	1.500	0.063	0.260	354.9	.	.
043-7	5 15 97	18022	750.0	7.03	0.066	1.700	0.032	0.270	352.0	.	.
043-7	5 29 97	18035	900.0	7.09	0.100	3.400	0.071	0.360	483.0	.	.
043-7	6 12 97	18050	2000.0	6.72	0.400	15.900	0.500	1.600	1228.0	.	.
043-7	6 30 97	18065	1800.0	6.32	1.100	15.800	.	.	1148.0	231.0	166.0
043-7	7 14 97	18079	2400.0	6.53	0.800	17.800	.	.	1469.0	278.0	206.0
043-7	7 24 97	18093	2325.0	6.54	0.500	15.000	0.500	1.600	1388.0	.	.
043-7	8 7 97	18122	2000.0	6.74	0.300	12.900	0.400	1.260	1486.0	.	.
043-7	8 22 97	18140	1690.0	6.61	0.570	11.100	.	.	1136.0	233.0	152.0
043-7	9 4 97	18160	2050.0	6.90	0.300	11.600	.	.	1385.0	270.0	188.0
043-7	9 17 97	18176	2200.0	7.05	0.200	11.500	.	.	1446.0	269.0	193.0
043-7	10 1 97	18191	2000.0	7.01	0.210	10.600	.	.	1327.0	260.0	178.0
043-7	10 14 97	18205	1525.0	6.72	0.550	8.900	.	.	920.0	175.0	119.0

Table A1.5. Water quality data for the Seep 1 samples collected by DNR from 1997-1999 (continued).

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
043-8	5 1 97	18007	780.0	7.03	0.100	1.400	0.058	0.240	336.0	.	.
043-8	5 15 97	18023	750.0	7.01	0.066	1.800	0.036	0.280	350.0	.	.
043-8	5 29 97	18036	900.0	6.98	0.300	4.600	0.200	0.450	485.0	.	.
043-8	6 12 97	18051	2100.0	6.69	0.600	16.500	0.500	1.580	1290.0	.	.
043-8	6 30 97	18066	1900.0	6.06	1.600	16.600	.	.	1185.0	209.0	167.0
043-8	7 14 97	18080	2800.0	6.41	1.000	18.400	.	.	1494.0	278.0	201.0
043-8	7 24 97	18094	2225.0	6.55	0.600	15.400	0.500	1.700	1442.0	.	.
043-8	8 7 97	18123	2000.0	6.69	0.400	13.200	0.400	1.330	1470.0	.	.
043-8	8 22 97	18141	1225.0	6.47	0.600	8.000	.	.	754.0	147.0	99.8
043-8	9 4 97	18161	2100.0	6.90	0.300	11.900	.	.	1362.0	265.0	191.0
043-8	9 17 97	18177	2175.0	7.03	0.290	11.900	.	.	1459.0	273.0	197.0
043-8	10 1 97	18192	2100.0	7.01	0.220	10.800	.	.	1345.0	262.0	180.0
043-8	10 14 97	18206	1500.0	6.70	0.450	8.400	.	.	919.0	172.0	119.0
043-W	5 1 97	18008	800.0	7.00	0.100	1.500	0.050	0.200	358.5	.	.
043-W	7 14 97	18081	2700.0	6.68	0.400	14.000	.	.	1442.0	275.0	197.0
043-W	7 24 97	18095	.	.	0.200	10.700	0.300	1.000	1521.0	.	.
043-W	8 7 97	18124	2050.0	6.91	0.100	8.600	0.200	0.680	1490.0	.	.
043-W	8 22 97	18142	1475.0	6.68	0.350	7.200	.	.	936.0	191.0	129.0
043-W	9 4 97	18162	2100.0	6.97	0.160	7.800	.	.	1331.0	266.0	181.0
043-W	9 17 97	18178	2080.0	7.13	0.100	8.200	.	.	1395.0	266.0	186.0
043-W	10 1 97	18193	2100.0	7.10	0.100	8.400	.	.	1357.0	269.0	180.0
043-W	10 14 97	18207	1600.0	6.83	0.330	7.700	.	.	963.0	182.0	127.0
043-W	4 2 99	18266	300.0	6.83	.	.	.	.	138.0	.	.
043-W	4 27 99	18272	.	.	.	.	.	.	576.0	.	.
043-W	5 14 99	18275	.	.	.	.	.	.	1012.0	.	.
043-WB	5 1 97	18009	800.0	7.02	0.100	1.600	0.053	0.230	350.4	.	.
043-WB	5 29 97	18037	850.0	7.20	0.055	2.300	0.027	0.240	462.0	.	.
043-WB	6 12 97	18052	1800.0	6.87	0.200	8.500	0.094	0.570	1106.0	.	.
043-WB	6 30 97	18067	1800.0	6.59	0.500	13.300	.	.	1062.0	213.0	160.0

Table A1.5. Water quality data for the Seep 1 samples collected by DNR from 1997-1999 (continued).

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
043-WB	7 14 97	18082	1400.0	6.71	0.400	14.700	.	.	1589.0	277.0	199.0
043-WB	7 24 97	18096	2350.0	6.55	0.200	11.600	0.300	1.100	1449.0	.	.
043-WB	8 7 97	18125	2050.0	6.86	0.200	9.600	0.200	0.860	1497.0	.	.
043-WB	8 22 97	18143	1500.0	6.64	0.380	7.400	.	.	927.0	187.0	125.0
043-WB	9 4 97	18163	2150.0	6.94	0.190	8.300	.	.	1354.0	269.0	184.0
043-WB	9 17 97	18179	2100.0	7.16	0.100	7.500	.	.	1418.0	266.0	187.0
043-WB	10 1 97	18194	2100.0	7.10	0.100	8.600	.	.	1360.0	269.0	180.0
043-WB	10 14 97	18208	1650.0	6.80	0.340	7.600	.	.	985.0	186.0	128.0
043-WB	6 2 98	18216	2250.0	7.00	0.260	15.800	0.300	1.480	1262.0	252.0	212.0
043-WB	6 10 98	18219	2150.0	6.63	0.460	16.700	0.300	1.730	1253.0	255.0	209.0
043-WB	6 24 98	18221	1850.0	6.71	0.200	10.300	0.200	1.240	1224.0	238.0	187.0
043-WB	7 10 98	18223	2050.0	6.88	0.380	15.400	0.300	1.600	1305.0	277.0	198.0
043-WB	7 21 98	18225	2100.0	7.12	0.100	6.700	-0.100	0.640	1364.0	302.0	207.0
043-WB	8 5 98	18227	2300.0	7.20	0.080	6.700	-0.100	0.590	1284.0	321.0	213.0
043-WB	8 20 98	18231	1800.0	7.00	0.100	8.000	0.100	1.000	1157.0	275.0	181.0
043-WB	9 4 98	18233	2000.0	7.64	0.100	7.900	0.100	0.880	1341.0	298.0	194.0
043-WB	9 17 98	18237	2100.0	7.28	0.060	6.600	-0.100	0.660	1342.0	294.0	198.0
043-WB	9 30 98	18242	2000.0	7.06	0.060	7.200	0.100	0.790	1251.0	283.0	181.0
043-WB	10 16 98	18245	1650.0	6.79	0.073	6.510	0.054	0.687	1431.0	.	.
043-WB	10 20 98	18253	1500.0	5.93	0.340	9.300	0.570	0.001	687.0	.	.
043-WB	11 3 98	18264	2150.0	7.42	.	.	.	.	1446.0	.	.
043-WB	4 2 99	18266	300.0	6.83	0.074	1.434	0.097	0.237	138.0	.	.
043-WB	4 27 99	18272	1000.0	6.94	0.100	4.631	0.134	0.455	576.0	.	.
043-WB	5 14 99	18275	1550.0	6.60	0.131	6.509	0.127	0.918	1012.0	.	.
043-WB	6 15 99	18284	2025.0	6.11	0.533	8.912	0.340	1.168	1360.0	.	.
043-WB	7 7 99	18290	1525.0	4.76	1.320	9.000	0.536	1.280	901.0	.	.
043-WB	7 28 99	18302	2000.0	5.89	1.320	14.700	0.880	1.970	.	250.0	199.0
043-WB	8 6 99	18305	2100.0	6.36	0.630	13.400	0.620	1.570	1480.0	282.0	208.0
043-WB	8 20 99	18311	2000.0	6.26	0.700	12.100	0.600	1.720	1333.0	258.0	202.0
043-WB	9 10 99	18314	1600.0	6.43	0.390	8.360	0.380	1.250	1070.0	231.0	153.0
043-WB	9 23 99	18320	1800.0	6.55	0.350	8.510	0.360	1.380	1284.0	284.0	183.0

Table A1.5. Water quality data for the Seep 1 samples collected by DNR from 1997-1999 (continued).

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
043-WB	10 8 99	18323	1200.0	6.36	0.350	5.360	0.300	0.970	717.0	151.0	99.3
Piezometer Samples											
P1	10 16 98	18246	.	.	0.003	0.062	0.021	0.044	.	.	.
P2	10 16 98	18247	.	.	0.009	0.160	0.033	0.108	.	.	.
P3	10 16 98	18248	.	.	0.050	9.810	0.099	0.138	.	.	.
P4	10 16 98	18249	.	.	0.005	0.155	0.027	0.031	.	.	.
P5	10 16 98	18250	.	.	0.003	33.300	1.810	0.043	.	.	.
P6	10 16 98	18251	.	.	0.660	31.000	1.610	1.910	.	.	.
Seeps											
S2	5 1 97	18010	700.0	5.43	0.200	3.400	0.200	0.450	315.5	.	.
S2	5 15 97	18024	750.0	5.25	0.300	3.700	0.200	0.550	396.0	.	.
S2	5 29 97	18038	700.0	5.31	0.300	3.000	0.100	0.490	347.0	.	.
S2	6 12 97	18053	3300.0	4.46	3.700	38.400	2.400	4.700	2335.0	.	.
S2	6 30 97	18068	2700.0	4.47	2.300	23.400	.	.	1935.0	294.0	264.0
S2	7 14 97	18083	2000.0	4.43	2.000	20.200	.	.	1969.0	287.0	240.0
S2	7 24 97	18097	2900.0	4.44	2.000	19.900	1.200	4.200	2069.0	.	.
S2	8 7 97	18126	2700.0	4.47	2.100	20.100	1.200	4.940	2215.0	.	.
S2	8 22 97	18144	1950.0	4.70	1.400	13.600	.	.	1411.0	207.0	192.0
S2	9 4 97	18164	2900.0	4.55	2.060	19.400	.	.	2203.0	308.0	288.0
S2	9 17 97	18180	3025.0	4.62	2.160	21.800	.	.	2450.0	334.0	321.0
S2	10 1 97	18195	1875.0	4.55	.	.	.	.	1050.0	202.0	184.0
S2	10 14 97	18209	2100.0	4.68	1.360	12.900	.	.	1434.0	224.0	179.0
S2	6 2 98	18217	2100.0	4.61	1.750	18.300	1.100	2.470	1200.0	219.0	194.0
S2	10 20 98	18257	.	.	0.536	10.000	0.909	2.770	850.0	.	.
S2	5 14 99	18280	3075.0	.	1.818	17.430	1.322	2.819	2337.0	.	.
S2	7 7 99	18295	1900.0	4.58	1.600	.	.	.	.	.	.
S2	7 7 99	18296	2150.0	4.47	1.710	14.100	1.080	2.540	1385.0	276.0	196.0

Table A1.5. Water quality data for the Seep 1 samples collected by DNR from 1997-1999 (continued).

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
S3	5 29 97	18039	1000.0	4.76	0.600	5.400	0.400	1.510	583.0	.	.
S3	6 12 97	18054	4000.0	4.38	4.300	54.300	3.700	5.990	2959.0	.	.
S3	6 30 97	18069	3200.0	4.42	2.400	28.700	.	.	2411.0	363.0	330.0
S3	7 24 97	18098	2750.0	4.39	2.200	24.200	1.600	4.100	1965.0	.	.
S3	8 7 97	18127	2700.0	4.42	2.400	24.600	1.700	4.760	2182.0	.	.
S3	8 22 97	18145	2200.0	4.51	1.950	19.500	.	.	1678.0	217.0	229.0
S3	9 5 97	18165	3500.0	4.44	2.670	31.000	.	.	2807.0	334.0	357.0
S3	10 20 98	18258	1750.0	4.61	.	11.600	0.586	2.590	855.0	.	.
S3	7 7 99	18297	2750.0	4.35	2.400	21.000	1.640	3.910	1847.0	311.0	253.0
S4	8 7 97	18128	1850.0	4.52	3.000	21.100	1.500	2.040	1430.0	.	.
S4	8 22 97	18146	1725.0	4.63	2.900	23.900	.	.	1196.0	195.0	150.0
S4	10 20 98	18259	2150.0	4.68	1.520	21.900	.	2.030	1116.0	.	.
S5	8 22 97	18147	2125.0	4.56	5.700	27.000	.	.	1572.0	287.0	164.0
S5	10 20 98	18260	2450.0	4.67	2.880	35.400	1.880	2.460	1214.0	.	.
S5	5 14 99	18282	4500.0	.	9.816	52.340	4.473	3.369	.	.	.
S5	7 7 99	18298	2225.0	4.58	4.320	23.500	2.020	2.140	1426.0	219.0	200.0
S6	8 7 97	18129	1490.0	4.52	3.900	14.600	1.200	1.180	1063.0	.	.
S6	8 22 97	18148	1325.0	4.62	3.480	14.700	.	.	985.0	183.0	76.5
S6	7 7 99	18299	1825.0	4.51	4.590	16.100	1.520	1.660	1135.0	274.0	112.0
S7	7 24 97	18099	2600.0	4.35	7.300	22.700	2.000	2.400	1751.0	.	.
S7	8 7 97	18130	2250.0	4.44	6.300	19.800	1.700	2.220	1889.0	.	.
S7	8 22 97	18149	1650.0	4.57	4.100	15.000	.	.	1201.0	240.0	100.0
S7	9 17 97	18181	2125.0	4.61	4.660	20.000	.	.	1620.0	327.0	132.0
S7	10 20 98	18261	2050.0	6.55	6.000	.	.	.	32.6	.	.
S7	5 12 99	18281	4200.0	.	8.622	34.610	3.228	2.674	3416.0	.	.
S7	7 7 99	18300	2675.0	4.37	6.680	28.400	2.510	2.480	1777.0	380.0	206.0

Table A1.5. Water quality data for the Seep 1 samples collected by DNR from 1997-1999 (continued).

Site	Date	Sample	S.C.	pH	Cu	Ni	Co	Zn	SO_4	Ca	Mg
S8	5 1 97	18011	1200.0	5.02	0.300	2.200	0.100	0.940	634.0	.	.
S8	8 22 97	18150	1900.0	4.70	1.400	10.900	.	.	1496.0	230.0	196.0
S9	5 1 97	18012	1100.0	4.98	0.200	2.200	0.100	0.870	594.8	.	.
S9	5 15 97	18025	1150.0	4.90	0.300	2.400	0.100	0.960	670.0	.	.
S9	5 29 97	18040	1300.0	4.93	0.300	3.300	0.200	1.250	790.0	.	.
S9	6 12 97	18055	2300.0	4.46	1.800	18.100	1.200	3.500	1577.0	.	.
S9	9 4 97	18166	2500.0	4.62	1.390	10.600	.	.	1791.0	264.0	227.0

Note: pH values are in standard units, specific conductance values are in microsiemens, and all other values are mg/L. Site locations are depicted in Figure 8.

Table A1.6. Historical Seep 1 data (1985-1991), and 1993-94 data from the surface sites within the Seep 1 pretreatment system.

Note: The "historic" values (i.e. before construction of the pretreatment system) presented here are from site 043 before construction of the pretreatment system. pH values are in standard units, all other values are mg/L.

Site	Date	Cu	Ni	Co	Zn	Ca	SO4	pH
Historic	6 14 85	0.81	32.0	1.4	1.6	200.0	1850	5.3
Historic	6 24 85	0.57	21.0	2.1	2.4	150.0	1300	5.3
Historic	7 12 85	0.81	27.0	1.4	1.8	200.0	1700	4.9
Historic	7 25 85	0.60	21.0	1.9	2.1	180.0	1600	5.1
Historic	8 16 85	0.48	20.0	1.3	1.6	240.0	1400	5.2
Historic	8 28 85	0.42	19.0	1.4	1.5	230.0	1300	5.2
Historic	6 13 86	1.10	25.0	1.8	2.2	130.0	1400	5.2
Historic	6 24 86	1.00	24.0	1.6	2.1	170.0	1400	5.1
Historic	7 9 86	0.84	21.0	1.5	1.9	180.0	1200	5.3
Historic	7 28 86	0.84	22.0	1.5	2.0	200.0	1600	5.3
Historic	8 11 86	0.62	12.0	1.4	1.3	200.0	1300	5.5
Historic	8 21 86	0.56	18.0	1.4	1.3	210.0	1450	5.6
Historic	6 11 87	0.90	22.0	1.6	1.8	.	1700	5.4
Historic	6 23 87	1.10	31.0	2.2	2.3	.	2650	5.3
Historic	7 15 87	0.72	17.0	1.1	1.3	.	1400	5.3
Historic	7 28 87	0.77	26.0	1.6	1.7	.	1780	5.4
Historic	8 13 87	0.55	19.0	1.4	1.4	.	1750	5.4
Historic	8 24 87	0.44	14.0	1.0	1.1	.	1600	5.5
Historic	6 1 88	0.74	32.0	2.5	2.0	.	2100	5.3
Historic	6 29 88	0.62	23.0	1.5	1.6	.	1700	5.2
Historic	7 13 88	0.81	30.0	2.4	2.1	.	1925	5.3
Historic	7 28 88	0.49	26.0	2.1	1.5	.	1950	5.5
Historic	8 11 88	0.60	25.0	1.9	2.6	.	1800	5.2
Historic	8 25 88	0.60	15.0	1.0	1.9	.	1275	5.3
Historic	6 8 89	1.80	24.0	1.9	2.1	.	1450	5.0
Historic	6 22 89	0.80	12.0	0.7	1.1	.	765	5.0
Historic	7 6 89	1.60	20.0	1.4	2.5	.	1400	5.0
Historic	7 26 89	1.60	24.0	1.6	3.0	.	1675	4.8
Historic	8 8 89	1.20	22.0	1.7	2.1	.	1725	4.9
Historic	8 28 89	1.30	20.0	1.4	1.9	.	1450	5.1
Historic	6 6 90	0.76	11.1	0.8	1.5	.	850	5.0
Historic	6 21 90	1.10	11.3	0.8	1.6	.	1300	4.8
Historic	7 3 90	1.65	17.0	1.5	2.0	.	605	4.9
Historic	7 18 90	1.88	20.0	1.7	2.6	.	1285	4.8
Historic	8 3 90	2.10	31.0	1.8	2.9	.	1700	4.7
Historic	8 31 90	1.40	21.0	1.3	2.0	.	1310	4.9
Historic	6 5 91	2.50	23.0	2.7	3.0	.	1730	4.6
Historic	6 19 91	1.80	21.0	1.6	2.6	.	1206	4.9

Table A1.6. Historical Seep 1 data (1985-1991), and 1993-94 data from the surface sites within the Seep 1 pretreatment system (continued).

Site	Date	Cu	Ni	Co	Zn	Ca	SO4	pH
Historic	7 3 91	0.78	12.0	0.5	0.8	.	902	5.2
Historic	7 3 91	0.78	12.0	0.5	0.8	.	890	5.2
Historic	7 18 91	1.70	19.6	1.1	2.6	.	1711	4.9
Historic	8 8 91	1.60	17.0	0.9	2.1	.	1710	4.9
Historic	8 8 91	1.60	13.0	0.9	2.1	.	1732	4.9
Historic	8 29 91	1.40	12.0	0.7	1.9	.	1522	4.6
Inf-1	6 7 93	0.54	7.8	.	.	.	1111	6.8
Inf-1	6 11 93	1.00	11.0	.	.	.	1655	.
Inf-1	6 15 93	0.94	12.0	.	.	.	1420	.
Inf-1	6 25 93	0.82	11.0	.	.	.	1161	5.0
Inf-1	6 29 93	1.30	11.0	.	.	.	1358	5.0
Inf-1	7 2 93	1.50	13.0	.	.	.	1421	.
Inf-1	7 7 93	1.10	11.0	.	.	.	1248	.
Inf-1	7 9 93	0.82	7.9	.	.	.	1095	4.6
Inf-1	7 13 93	0.09	2.6	.	.	.	1220	.
Inf-1	7 16 93	0.86	10.0	.	.	.	1329	.
Inf-1	7 20 93	0.50	10.0	.	.	.	1143	5.9
Inf-1	7 29 93	0.39	5.8	.	.	.	971	6.3
Inf-1	8 3 93	0.54	7.6	.	.	.	1337	5.7
Inf-1	8 6 93	0.09	4.5	.	.	.	1429	6.0
Inf-1	8 10 93	0.69	8.6	.	.	.	1230	4.8
Inf-1	8 17 93	0.67	10.0	.	.	.	1530	5.7
Inf-1	8 20 93	0.88	9.9	.	.	.	1391	5.3
Inf-1	6 30 94	0.97	6.1	.	.	.	1031	5.4
Inf-1	7 15 94	1.40	9.4	.	.	.	1392	5.3
Inf-1	7 28 94	0.73	.	.	.	.	1496	5.8
Out	6 7 93	0.03	6.2	.	.	.	1002	7.2
Out	6 11 93	0.05	6.6	.	.	.	1290	7.2
Out	6 15 93	0.06	11.0	.	.	.	1191	7.0
Out	6 25 93	0.11	11.0	.	.	.	1131	6.5
Out	6 29 93	0.09	8.8	.	.	.	1191	6.9
Out	7 2 93	0.09	11.0	.	.	.	1243	.
Out	7 7 93	0.13	10.0	.	.	.	1100	.
Out	7 9 93	0.14	8.9	.	.	.	1052	.
Out	7 13 93	0.20	14.0	.	.	.	1127	.
Out	7 16 93	0.16	11.0	.	.	.	1203	.
Out	7 20 93	0.16	15.0	.	.	.	1165	6.5
Out	7 29 93	0.21	9.0	.	.	.	1095	6.2
Out	8 3 93	0.13	8.0	.	.	.	1172	6.7
Out	8 6 93	0.13	7.6	.	.	.	1139	6.7
Out	8 10 93	0.10	8.3	.	.	.	2492	6.7

Table A1.6. Historical Seep 1 data (1985-1991), and 1993-94 data from the surface sites within the Seep 1 pretreatment system (continued).

Site	Date	Cu	Ni	Co	Zn	Ca	SO4	pH
Out	8 17 93	0.09	8.4	.	.	.	1203	7.3
Out	8 20 93	0.08	8.1	.	.	.	1408	7.1
Out	6 30 94	0.45	6.8	.	.	.	745	6.7
Out	7 15 94	0.30	8.0	.	.	.	1060	6.4
Out	7 28 94	0.05	.	.	.	.	1237	6.9
SP-1	6 7 93	0.55	9.1	.	.	.	1247	.
SP-1	6 11 93	0.77	9.9	.	.	.	1568	5.5
SP-1	6 15 93	0.32	10.0	.	.	.	1058	6.2
SP-1	6 25 93	0.78	11.0	.	.	.	1169	5.5
SP-1	6 29 93	0.82	11.0	.	.	.	1376	6.5
SP-1	7 2 93	0.45	9.5	.	.	.	1288	.
SP-1	7 7 93	0.98	11.0	.	.	.	1251	.
SP-1	7 9 93	1.40	9.8	.	.	.	1247	4.6
SP-1	7 13 93	1.10	11.0	.	.	.	1324	.
SP-1	7 16 93	1.60	12.0	.	.	.	1390	.
SP-1	7 20 93	1.20	18.0	.	.	.	1374	5.5
SP-1	7 29 93	1.30	12.0	.	.	.	1135	4.8
SP-1	8 3 93	0.50	8.6	.	.	.	1085	6.6
SP-1	8 6 93	1.00	11.0	.	.	.	1796	6.1
SP-1	8 10 93	1.20	15.0	.	.	.	1342	4.8
SP-1	8 17 93	0.72	9.6	.	.	.	1483	6.1
SP-1	8 20 93	0.25	8.5	.	.	.	1316	6.6
SP-1	6 30 94	1.00	6.2	.	.	.	939	5.3
SP-1	7 15 94	1.40	9.9	.	.	.	1225	5.3
SP-1	7 28 94	0.66	.	.	.	.	1435	6.2
SP-2	6 7 93	0.14	6.0	.	.	.	790	7.2
SP-2	6 11 93	0.45	9.5	.	.	.	1265	6.6
SP-2	6 15 93	0.41	11.0	.	.	.	1154	6.4
SP-2	6 25 93	0.41	11.0	.	.	.	999	5.9
SP-2	6 29 93	0.29	7.3	.	.	.	969	6.8
SP-2	7 2 93	0.39	10.0	.	.	.	1005	.
SP-2	7 7 93	0.96	10.0	.	.	.	1225	.
SP-2	7 9 93	1.50	16.0	.	.	.	1280	4.8
SP-2	7 13 93	0.65	14.0	.	.	.	998	.
SP-2	7 16 93	0.71	13.0	.	.	.	1390	.
SP-2	7 20 93	0.36	11.0	.	.	.	717	6.2
SP-2	7 29 93	1.40	15.0	.	.	.	1691	5.0
SP-2	8 3 93	0.14	3.0	.	.	.	558	6.8
SP-2	8 6 93	0.45	10.0	.	.	.	1310	6.2
SP-2	8 10 93	0.57	13.0	.	.	.	2311	5.9
SP-2	8 17 93	0.18	7.5	.	.	.	897	6.9
SP-2	8 20 93	0.16	8.3	.	.	.	2282	6.8

Table A1.6. Historical Seep 1 data (1985-1991), and 1993-94 data from the surface sites within the Seep 1 pretreatment system (continued).

Site	Date	Cu	Ni	Co	Zn	Ca	SO4	pH
SP-2	6 30 94	0.70	7.4	.	.	.	708	6.7
SP-2	7 15 94	0.82	9.1	.	.	.	995	5.6
SP-2	7 28 94	0.64	.	.	.	.	1153	6.3
SP-3	6 7 93	0.54	13.0	.	.	.	1333	7.1
SP-3	6 11 93	0.91	13.0	.	.	.	1579	6.5
SP-3	6 15 93	0.43	11.0	.	.	.	1227	6.1
SP-3	6 25 93	0.84	15.0	.	.	.	1183	5.6
SP-3	6 29 93	0.59	11.0	.	.	.	1260	6.6
SP-3	7 2 93	0.67	13.0	.	.	.	1184	.
SP-3	7 7 93	0.83	13.0	.	.	.	1274	.
SP-3	7 9 93	1.30	20.0	.	.	.	1592	5.3
SP-3	7 13 93	1.00	18.0	.	.	.	1307	.
SP-3	7 16 93	1.40	19.0	.	.	.	1734	.
SP-3	7 20 93	1.00	19.0	.	.	.	1280	5.9
SP-3	7 29 93	1.50	17.0	.	.	.	1090	5.0
SP-3	8 3 93	1.10	15.0	.	.	.	1443	6.2
SP-3	8 6 93	0.82	14.0	.	.	.	1658	5.9
SP-3	8 10 93	1.10	17.0	.	.	.	1388	5.5
SP-3	8 17 93	1.20	12.0	.	.	.	1982	6.1
SP-3	8 20 93	0.79	16.0	.	.	.	1742	6.2
SP-3	6 30 94	1.80	15.0	.	.	.	970	6.2
SP-3	7 15 94	0.87	10.0	.	.	.	1066	5.9
SP-3	7 28 94	0.73	.	.	.	.	1254	6.2
SP-4	6 7 93	0.13	9.3	.	.	.	1122	.
SP-4	6 11 93	0.12	9.2	.	.	.	1448	.
SP-4	6 15 93	0.14	11.0	.	.	.	1272	.
SP-4	6 25 93	0.33	13.0	.	.	.	1177	6.1
SP-4	6 29 93	0.23	11.0	.	.	.	1279	6.7
SP-4	7 2 93	0.32	12.0	.	.	.	1294	.
SP-4	7 7 93	0.46	12.0	.	.	.	1187	.
SP-4	7 9 93	0.54	15.0	.	.	.	1230	6.0
SP-4	7 13 93	0.57	16.0	.	.	.	1274	.
SP-4	7 16 93	0.42	13.0	.	.	.	1396	.
SP-4	7 20 93	0.49	17.0	.	.	.	1291	6.2
SP-4	7 29 93	0.52	10.0	.	.	.	1041	5.8
SP-4	8 3 93	0.35	9.8	.	.	.	1199	7.0
SP-4	8 6 93	0.28	10.0	.	.	.	1294	6.4
SP-4	8 10 93	0.30	9.7	.	.	.	2481	6.3
SP-4	8 17 93	0.19	9.0	.	.	.	1379	7.0
SP-4	8 20 93	0.17	7.7	.	.	.	1333	6.8
SP-4	6 30 94	0.49	6.1	.	.	.	644	6.7
SP-4	7 15 94	0.66	8.1	.	.	.	882	6.1
SP-4	7 28 94	0.25	.	.	.	.	1177	6.7

Table A1.6. Historical Seep 1 data (1985-1991), and 1993-94 data from the surface sites within the Seep 1 pretreatment system (continued).

Site	Date	Cu	Ni	Co	Zn	Ca	SO4	pH
SP-5	6 7 93	0.13	8.7	.	.	.	1122	.
SP-5	6 11 93	0.16	9.4	.	.	.	1298	.
SP-5	6 15 93	0.27	12.0	.	.	.	1304	.
SP-5	6 25 93	0.26	12.0	.	.	.	1119	6.3
SP-5	6 29 93	0.26	11.0	.	.	.	1270	6.7
SP-5	7 2 93	0.22	11.0	.	.	.	1193	.
SP-5	7 7 93	0.41	11.0	.	.	.	1159	.
SP-5	7 9 93	0.31	9.2	.	.	.	940	.
SP-5	7 13 93	0.69	16.0	.	.	.	1209	.
SP-5	7 16 93	0.51	12.0	.	.	.	1291	.
SP-5	7 20 93	0.69	18.0	.	.	.	1230	6.2
SP-5	7 29 93	0.46	11.0	.	.	.	1020	6.0
SP-5	8 3 93	0.39	9.3	.	.	.	1199	7.1
SP-5	8 6 93	0.27	8.8	.	.	.	1150	6.5
SP-5	8 10 93	0.30	10.0	.	.	.	2513	6.3
SP-5	8 17 93	0.46	11.0	.	.	.	1507	6.6
SP-5	8 20 93	0.26	8.4	.	.	.	1431	6.8
SP-5	6 30 94	0.82	7.9	.	.	.	843	6.5
SP-5	7 15 94	0.84	9.9	.	.	.	1130	6.1
SP-5	7 28 94	0.31	.	.	.	.	1296	6.9
SP-6	6 7 93	0.07	7.0	.	.	.	1019	.
SP-6	6 11 93	0.05	8.3	.	.	.	1323	.
SP-6	6 15 93	0.17	12.0	.	.	.	1275	.
SP-6	6 25 93	0.12	11.0	.	.	.	1164	6.4
SP-6	6 29 93	0.08	9.0	.	.	.	1225	6.8
SP-6	7 2 93	0.09	12.0	.	.	.	1231	.
SP-6	7 7 93	0.33	11.0	.	.	.	1128	.
SP-6	7 9 93	0.08	7.4	.	.	.	914	.
SP-6	7 13 93	0.60	16.0	.	.	.	1230	.
SP-6	7 16 93	0.40	12.0	.	.	.	1252	.
SP-6	7 20 93	0.47	18.0	.	.	.	1161	6.3
SP-6	7 29 93	0.21	10.0	.	.	.	1095	6.2
SP-6	8 3 93	0.13	8.1	.	.	.	1106	7.3
SP-6	8 6 93	0.14	8.0	.	.	.	1150	6.6
SP-6	8 10 93	0.14	9.6	.	.	.	2502	6.5
SP-6	8 17 93	0.22	9.3	.	.	.	1322	7.0
SP-6	8 20 93	0.18	9.4	.	.	.	1414	7.0
SP-6	6 30 94	0.73	7.4	.	.	.	731	6.7
SP-6	7 15 94	0.68	9.2	.	.	.	1060	6.2
SP-6	7 28 94	0.04	.	.	.	.	1186	6.9

Note: pH values are standard units, other parameters are mg/L.

Table A1.7. T-tests of water quality data from the input and output of the study cell in the W1D treatment system.

The data used for these statistics were from samples collected by DNR in 1997-1999, and the data were matched up by date prior to running the t-tests. The site called W1D1 is the input site to the study cell, and W1D23 is the average of the two output sites from the cell (which are called W1D2 and W1D3). The data sets for the input and output have been altered so that they represent the same time periods each year.

```
PAIRED SAMPLES T-TEST ON      NI1      VS      NI23     WITH     21 CASES
MEAN DIFFERENCE =        0.076
SD DIFFERENCE =        0.103
T =       3.415 DF =   20 PROB =       0.003

PAIRED SAMPLES T-TEST ON      CU1      VS      CU23     WITH     14 CASES
MEAN DIFFERENCE =        0.002
SD DIFFERENCE =        0.003
T =       1.989 DF =   13 PROB =       0.068

PAIRED SAMPLES T-TEST ON      CO1      VS      CO23     WITH     21 CASES
MEAN DIFFERENCE =        0.002
SD DIFFERENCE =        0.004
T =       2.516 DF =   20 PROB =       0.021

PAIRED SAMPLES T-TEST ON      ZN1      VS      ZN23     WITH      8 CASES
MEAN DIFFERENCE =        0.002
SD DIFFERENCE =        0.014
T =       0.343 DF =    7 PROB =       0.742

PAIRED SAMPLES T-TEST ON     SO41      VS     SO423     WITH     18 CASES
MEAN DIFFERENCE =       43.044
SD DIFFERENCE =       57.702
T =       3.165 DF =   17 PROB =       0.006

PAIRED SAMPLES T-TEST ON      CA1      VS      CA23     WITH     12 CASES
MEAN DIFFERENCE =        9.583
SD DIFFERENCE =       11.239
T =       2.954 DF =   11 PROB =       0.013

PAIRED SAMPLES T-TEST ON      MG1      VS      MG23     WITH     12 CASES
MEAN DIFFERENCE =        2.792
SD DIFFERENCE =        4.003
T =       2.416 DF =   11 PROB =       0.034
```

Table A1.8. T-tests of 1997-99 water quality data from the vertical downflow section of the Seep 1 pretreatment system.

The data used for these statistics were from samples collected by DNR in 1997-1999, and the data were matched up by date prior to running the t-tests.

```
THE FOLLOWING RESULTS ARE FOR:
             YEAR      =       97.000


PAIRED SAMPLES T-TEST ON      SC6     VS     SCWB    WITH    12 CASES

MEAN DIFFERENCE =        87.500
SD DIFFERENCE =       305.536
T =        0.992 DF =   11 PROB =      0.342

THE FOLLOWING RESULTS ARE FOR:
             YEAR      =       98.000


PAIRED SAMPLES T-TEST ON      SC6     VS     SCWB    WITH    13 CASES

MEAN DIFFERENCE =       -15.385
SD DIFFERENCE =        68.874
T =       -0.805 DF =   12 PROB =       0.436

THE FOLLOWING RESULTS ARE FOR:
             YEAR      =       99.000


PAIRED SAMPLES T-TEST ON      SC6     VS     SCWB    WITH    10 CASES

MEAN DIFFERENCE =       -86.500
SD DIFFERENCE =       183.909
T =       -1.487 DF =    9 PROB =       0.171

THE FOLLOWING RESULTS ARE FOR:
             YEAR      =       97.000


PAIRED SAMPLES T-TEST ON      PH6     VS     PHWB    WITH    12 CASES

MEAN DIFFERENCE =        -0.122
SD DIFFERENCE =         0.117
T =       -3.595 DF =   11 PROB =       0.004

THE FOLLOWING RESULTS ARE FOR:
             YEAR      =       98.000


PAIRED SAMPLES T-TEST ON      PH6     VS     PHWB    WITH    12 CASES
```

Table A1.8. T-tests of 1997-99 water quality data from the vertical downflow section of the Seep 1 pretreatment system (continued).

```
MEAN DIFFERENCE =        -0.137
SD DIFFERENCE =         0.327
T =      -1.448 DF =   11 PROB =       0.175

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       99.000


PAIRED SAMPLES T-TEST ON      PH6     VS     PHWB    WITH    10 CASES

MEAN DIFFERENCE =        -0.344
SD DIFFERENCE =         0.271
T =      -4.019 DF =    9 PROB =       0.003

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       97.000


PAIRED SAMPLES T-TEST ON     SO46     VS    SO4WB    WITH    12 CASES

MEAN DIFFERENCE =        24.783
SD DIFFERENCE =        77.919
T =       1.102 DF =   11 PROB =       0.294

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       98.000


PAIRED SAMPLES T-TEST ON     SO46     VS    SO4WB    WITH    13 CASES

MEAN DIFFERENCE =       -34.462
SD DIFFERENCE =       160.000
T =      -0.777 DF =   12 PROB =       0.452

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       99.000


PAIRED SAMPLES T-TEST ON     SO46     VS    SO4WB    WITH     9 CASES

MEAN DIFFERENCE =       -38.011
SD DIFFERENCE =        64.983
T =      -1.755 DF =    8 PROB =       0.117

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       97.000
```

Table A1.8. T-tests of 1997-99 water quality data from the vertical downflow section of the Seep 1 pretreatment system (continued).

```
PAIRED SAMPLES T-TEST ON      CU6     VS     CUWB    WITH    12 CASES

MEAN DIFFERENCE =        0.145
SD DIFFERENCE =        0.158
T =       3.187 DF =   11 PROB =       0.009

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       98.000


PAIRED SAMPLES T-TEST ON      CU6     VS     CUWB    WITH    12 CASES

MEAN DIFFERENCE =        0.037
SD DIFFERENCE =        0.174
T =       0.739 DF =   11 PROB =       0.476

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       99.000


PAIRED SAMPLES T-TEST ON      CU6     VS     CUWB    WITH     9 CASES

MEAN DIFFERENCE =        0.170
SD DIFFERENCE =        0.226
T =       2.256 DF =    8 PROB =       0.054

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       97.000


PAIRED SAMPLES T-TEST ON      NI6     VS     NIWB    WITH    12 CASES

MEAN DIFFERENCE =        2.117
SD DIFFERENCE =        1.663
T =       4.408 DF =   11 PROB =       0.001

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       98.000


PAIRED SAMPLES T-TEST ON      NI6     VS     NIWB    WITH    11 CASES

MEAN DIFFERENCE =        2.695
SD DIFFERENCE =        3.164
T =       2.825 DF =   10 PROB =       0.018

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       99.000
```

Table A1.8. T-tests of 1997-99 water quality data from the vertical downflow section of the Seep 1 pretreatment system (continued).

```
PAIRED SAMPLES T-TEST ON      NI6     VS     NIWB    WITH     8 CASES

MEAN DIFFERENCE =        0.965
SD DIFFERENCE =        0.903
T =       3.021 DF =    7 PROB =       0.019


THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       97.000


PAIRED SAMPLES T-TEST ON      ZN6     VS     ZNWB    WITH     5 CASES

MEAN DIFFERENCE =        0.110
SD DIFFERENCE =        0.376
T =       0.655 DF =    4 PROB =       0.548

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       98.000


PAIRED SAMPLES T-TEST ON      ZN6     VS     ZNWB    WITH    11 CASES

MEAN DIFFERENCE =        0.250
SD DIFFERENCE =        0.402
T =       2.067 DF =   10 PROB =       0.066

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       99.000


PAIRED SAMPLES T-TEST ON      ZN6     VS     ZNWB    WITH     8 CASES

MEAN DIFFERENCE =        0.225
SD DIFFERENCE =        0.147
T =       4.340 DF =    7 PROB =       0.003

THE FOLLOWING RESULTS ARE FOR:
             YEAR     =       97.000


PAIRED SAMPLES T-TEST ON      CA6     VS     CAWB    WITH     7 CASES

MEAN DIFFERENCE =        4.857
SD DIFFERENCE =       21.798
T =       0.590 DF =    6 PROB =       0.577
```

Table A1.8. T-tests of 1997-99 water quality data from the vertical downflow section of the Seep 1 pretreatment system (continued).

```
THE FOLLOWING RESULTS ARE FOR:

              YEAR      =       98.000

 PAIRED SAMPLES T-TEST ON      CA6     VS     CAWB    WITH    10 CASES

 MEAN DIFFERENCE =        -1.300
 SD DIFFERENCE =         4.990
 T =       -0.824 DF =    9 PROB =       0.431

 THE FOLLOWING RESULTS ARE FOR:
              YEAR      =       99.000

 PAIRED SAMPLES T-TEST ON      CA6     VS     CAWB    WITH     5 CASES

 MEAN DIFFERENCE =       -13.600
 SD DIFFERENCE =        11.803
 T =       -2.577 DF =    4 PROB =       0.062

 THE FOLLOWING RESULTS ARE FOR:
              YEAR      =       97.000

 PAIRED SAMPLES T-TEST ON      MG6     VS     MGWB    WITH     7 CASES

 MEAN DIFFERENCE =         6.714
 SD DIFFERENCE =        12.419
 T =        1.430 DF =    6 PROB =       0.203

 THE FOLLOWING RESULTS ARE FOR:
              YEAR      =       98.000

 PAIRED SAMPLES T-TEST ON      MG6     VS     MGWB    WITH    10 CASES

 MEAN DIFFERENCE =         0.200
 SD DIFFERENCE =         6.197
 T =        0.102 DF =    9 PROB =       0.921

 THE FOLLOWING RESULTS ARE FOR:
              YEAR      =       99.000

 PAIRED SAMPLES T-TEST ON      MG6     VS     MGWB    WITH     5 CASES

 MEAN DIFFERENCE =        -4.400
 SD DIFFERENCE =        11.459
 T =       -0.859 DF =    4 PROB =       0.439
```

Table A1.9. Anomalous data.

Site	Sample #	Date	Parameter	Reported value	Expected value
Seep 1					
043-1	18134	8/22/97	Mg	1.6 [A]	160
043-6	18244	10/16/98	Ni	0	9.0
043-6	18252	10/20/98	NI	0	9.0
043-6	18304	8/6/99	Ni	0.085	13.0
043-6	18304	8/6/99	Co	0.008	0.8
043-5	18020	5/15/97	SO4	3.5	300
043-5	18189	10/1/97	Ni	4.00	10.0
043-6	18232	9/4/98	Co	2.40	0.2
043-6	18252	10/20/98	Zn	0.001	1.0
043-6	18265	4/2/99	4" of water observed above ice; anomalously low values (SC=110, pH=6.67, Cu=0.06, Ni=0.30, Co=0.018, Zn=0.080, SO4=36.9) are probably due to dilution.		
043-6	18304	8/6/99	Ni	0.08	13.0
043-6	18304	8/6/99	Co	0.008	0.8
043-6	18322	10/8/99	SC, pH, Cu, Ni, Co, Zn, SO4, Ca and Mg were all anomalous (390, 6.77, 0.09, 1.31, 0.10, 0.21, 162, 44.2 and 24.4, respectively)		

[A] This value was changed to 160 in the file called DNRSEEP1.sys

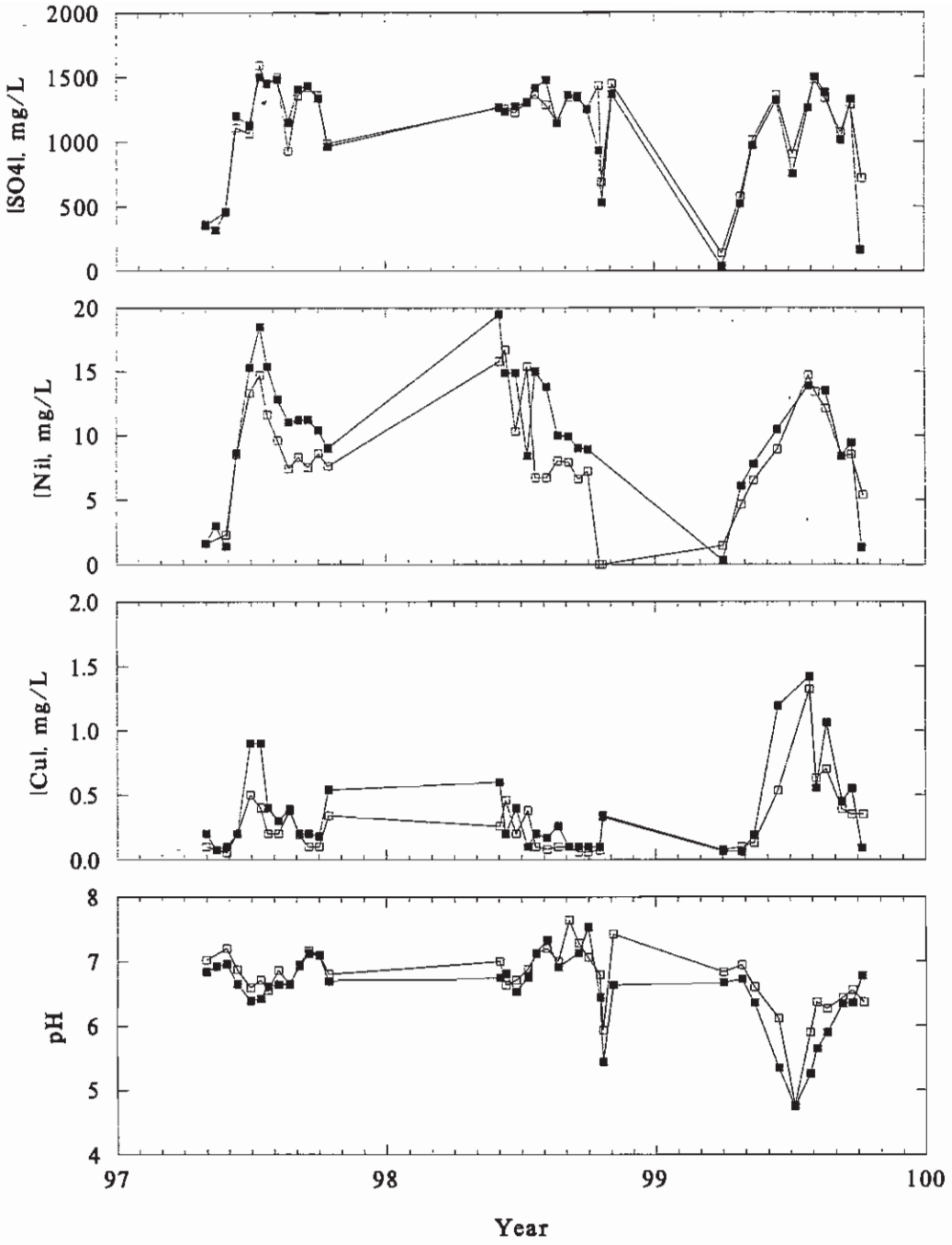


Figure A1.1. Comparison of pH, copper, nickel and sulfate concentrations at **043-6** (the site prior to the vertical flow section; dark squares) and **043-WB** (the weir; open squares) at Seep 1.

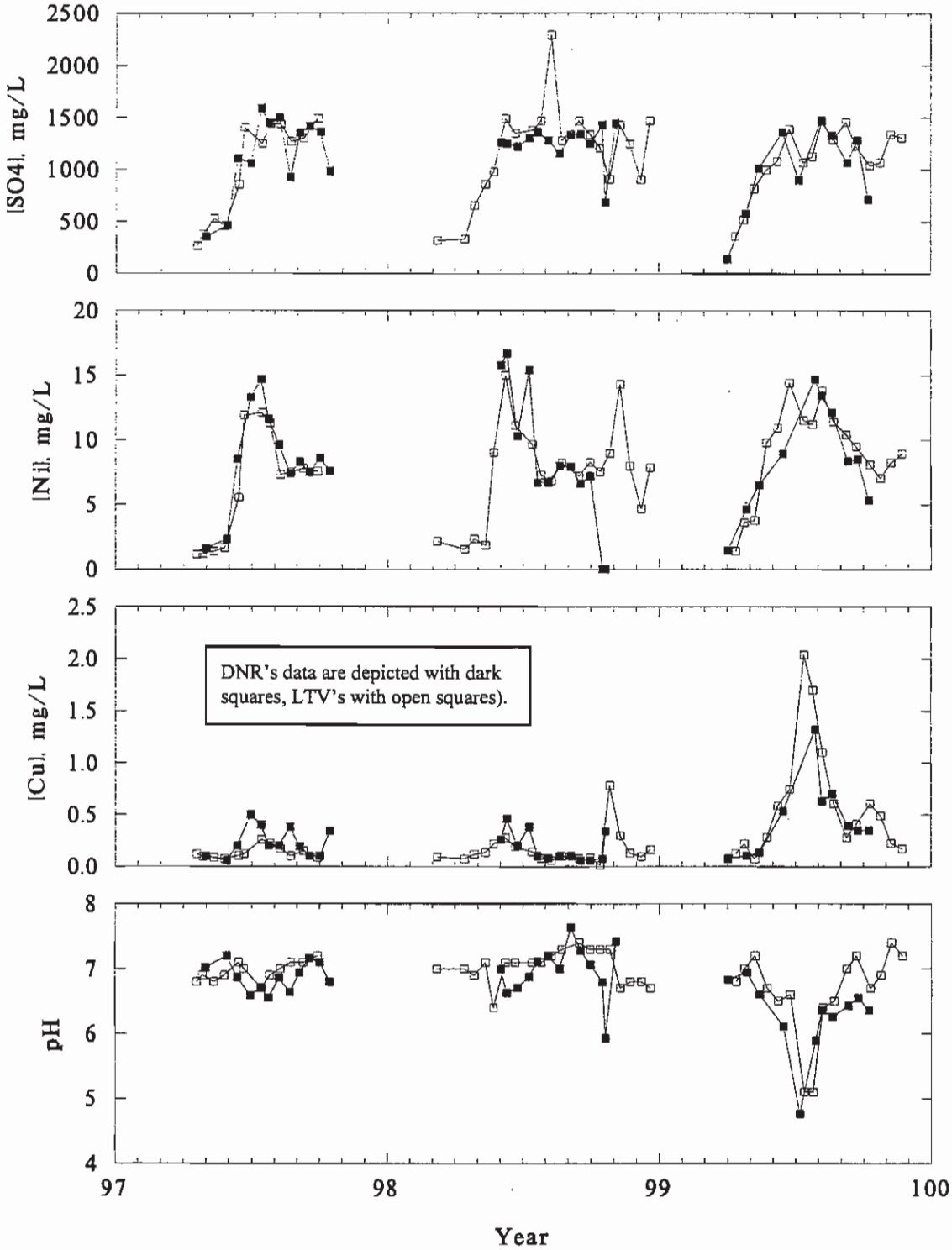


Figure A1.2. Comparison of LTV's and DNR's data (pH, Cu, Ni, SO_4) for 043, 1997-99.

Appendix 2

Summary statistics of W1D and Seep 1 water quality data

Page

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data A2.1

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data A2.10

Table A2.3. Summary statistics of water quality data from W1D sites other than W1D-051 and W1D-052 . . . . . A2.18

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Table A2.5. Summary statistics of Seep 1 (site 043) water quality data A2.61

Table A2.6. Summary statistics of surface water quality data from sites within the Seep 1 pretreatment system A2.78

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Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August A2.92

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data.

Site W1D-051

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-051
YEAR = 92.000

TOTAL OBSERVATIONS: 43

	PH	SC	CUT	CUF	NIT
N OF CASES	32	0	43	0	43
MINIMUM	6.700	.	0.045	.	1.100
MAXIMUM	7.300	.	0.140	.	6.700
RANGE	0.600	.	0.095	.	5.600
MEAN	7.081	.	0.093	.	4.284
STANDARD DEV	0.135	.	0.025	.	1.584
MEDIAN	7.100	.	0.090	.	4.600

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	43	0	43	0
MINIMUM	.	0.001	.	0.010	.
MAXIMUM	.	0.060	.	0.110	.
RANGE	.	0.060	.	0.100	.
MEAN	.	0.016	.	0.054	.
STANDARD DEV	.	0.012	.	0.022	.
MEDIAN	.	0.012	.	0.050	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	17	11	42	0	31
MINIMUM	0.050	0.800	1064.000	.	130.000
MAXIMUM	0.200	5.600	3188.000	.	410.000
RANGE	0.150	4.800	2124.000	.	280.000
MEAN	0.079	2.618	2484.190	.	296.387
STANDARD DEV	0.050	1.401	518.189	.	66.920
MEDIAN	0.050	2.800	2629.000	.	300.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	31	43	43	0
MINIMUM	130.000	165.000	194.000	.
MAXIMUM	560.000	165.000	194.000	.
RANGE	430.000	0.000	0.000	.
MEAN	381.065	165.000	194.000	.
STANDARD DEV	99.514	0.000	0.000	.
MEDIAN	420.000	165.000	194.000	.

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-051
YEAR = 93.000

TOTAL OBSERVATIONS: 45

	PH	SC	CUT	CUF	NIT
N OF CASES	41	0	45	0	45
MINIMUM	6.800	.	0.007	.	0.790
MAXIMUM	8.300	.	0.090	.	7.300
RANGE	1.500	.	0.083	.	6.510
MEAN	7.112	.	0.051	.	3.139
STANDARD DEV	0.266	.	0.018	.	2.319
MEDIAN	7.100	.	0.050	.	2.300

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	32	0	33	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.450	.	0.090	.
RANGE	.	0.450	.	0.085	.
MEAN	.	0.043	.	0.041	.
STANDARD DEV	.	0.104	.	0.023	.
MEDIAN	.	0.010	.	0.050	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	31	1	45	0	30
MINIMUM	0.005	4.400	622.000	.	70.000
MAXIMUM	0.500	4.400	2852.000	.	410.000
RANGE	0.495	0.000	2230.000	.	340.000
MEAN	0.097	4.400	1660.600	.	222.000
STANDARD DEV	0.099	.	735.401	.	107.588
MEDIAN	0.100	4.400	1461.000	.	175.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	30	45	45	0
MINIMUM	90.000	165.000	194.000	.
MAXIMUM	470.000	165.000	194.000	.
RANGE	380.000	0.000	0.000	.
MEAN	247.333	165.000	194.000	.
STANDARD DEV	146.357	0.000	0.000	.
MEDIAN	170.000	165.000	194.000	.

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-051
YEAR = 94.000

TOTAL OBSERVATIONS: 36

	PH	SC	CUT	CUF	NIT
N OF CASES	29	0	33	2	36
MINIMUM	6.100	.	0.030	0.027	0.640
MAXIMUM	7.600	.	0.100	0.030	7.500
RANGE	1.500	.	0.070	0.003	6.860
MEAN	7.017	.	0.059	0.029	4.529
STANDARD DEV	0.289	.	0.018	0.002	1.914
MEDIAN	7.000	.	0.050	0.029	5.050

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	19	0	19	0
MINIMUM	1.300	0.001	.	0.010	.
MAXIMUM	3.100	0.160	.	0.200	.
RANGE	1.800	0.160	.	0.190	.
MEAN	2.200	0.049	.	0.061	.
STANDARD DEV	1.273	0.041	.	0.044	.
MEDIAN	2.200	0.050	.	0.060	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	19	0	33	2	16
MINIMUM	0.050	.	379.000	1292.000	130.000
MAXIMUM	0.810	.	2895.000	2381.000	360.000
RANGE	0.760	.	2516.000	1089.000	230.000
MEAN	0.140	.	2118.727	1836.500	312.813
STANDARD DEV	0.204	.	676.792	770.039	59.441
MEDIAN	0.100	.	2341.000	1836.500	335.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	16	36	36	0
MINIMUM	120.000	165.000	194.000	.
MAXIMUM	440.000	165.000	194.000	.
RANGE	320.000	0.000	0.000	.
MEAN	382.500	165.000	194.000	.
STANDARD DEV	82.745	0.000	0.000	.
MEDIAN	410.000	165.000	194.000	.

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-051
YEAR = 95.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	17	0	17	0	17
MINIMUM	6.700	.	0.030	.	0.530
MAXIMUM	7.400	.	0.600	.	3.200
RANGE	0.700	.	0.570	.	2.670
MEAN	7.053	.	0.105	.	1.778
STANDARD DEV	0.191	.	0.132	.	0.684
MEDIAN	7.100	.	0.070	.	1.600

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	17	0	17	0
MINIMUM	.	0.017	.	0.010	.
MAXIMUM	.	0.210	.	0.040	.
RANGE	.	0.193	.	0.030	.
MEAN	.	0.049	.	0.022	.
STANDARD DEV	.	0.046	.	0.011	.
MEDIAN	.	0.037	.	0.020	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	17	0	17	0	17
MINIMUM	0.050	.	230.000	.	90.000
MAXIMUM	0.500	.	2449.000	.	356.000
RANGE	0.450	.	2219.000	.	266.000
MEAN	0.153	.	1401.294	.	236.529
STANDARD DEV	0.153	.	731.457	.	76.136
MEDIAN	0.100	.	1594.000	.	250.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	17	17	17	0
MINIMUM	80.000	165.000	194.000	.
MAXIMUM	409.000	165.000	194.000	.
RANGE	329.000	0.000	0.000	.
MEAN	248.176	165.000	194.000	.
STANDARD DEV	90.167	0.000	0.000	.

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data (continued).

MEDIAN	230.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-051
YEAR = 96.000

TOTAL OBSERVATIONS: 18

	PH	SC	CUT	CUF	NIT
N OF CASES	18	0	18	0	18
MINIMUM	7.000	.	0.010	.	0.385
MAXIMUM	7.500	.	0.080	.	2.000
RANGE	0.500	.	0.070	.	1.615
MEAN	7.178	.	0.043	.	1.172
STANDARD DEV	0.131	.	0.019	.	0.475
MEDIAN	7.200	.	0.041	.	0.979

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	18	0	18	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.040	.	0.056	.
RANGE	.	0.040	.	0.051	.
MEAN	.	0.009	.	0.030	.
STANDARD DEV	.	0.009	.	0.014	.
MEDIAN	.	0.007	.	0.028	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	18	0	18	0	17
MINIMUM	0.040	.	722.000	.	115.000
MAXIMUM	0.290	.	3001.000	.	350.000
RANGE	0.250	.	2279.000	.	235.000
MEAN	0.084	.	1598.167	.	258.176
STANDARD DEV	0.056	.	524.754	.	68.020
MEDIAN	0.070	.	1625.000	.	278.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	17	18	18	0
MINIMUM	110.000	165.000	194.000	.
MAXIMUM	320.000	165.000	194.000	.
RANGE	210.000	0.000	0.000	.

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data (continued).

MEAN	257.412	165.000	194.000	.
STANDARD DEV	54.082	0.000	0.000	.
MEDIAN	269.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-051
YEAR = 97.000

TOTAL OBSERVATIONS: 18

	PH	SC	CUT	CUF	NIT
N OF CASES	18	0	18	0	18
MINIMUM	7.000	.	0.007	.	0.144
MAXIMUM	7.300	.	0.048	.	1.300
RANGE	0.300	.	0.041	.	1.156
MEAN	7.167	.	0.030	.	0.694
STANDARD DEV	0.091	.	0.012	.	0.293
MEDIAN	7.200	.	0.032	.	0.713

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	18	0	18	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.045	.	0.028	.
RANGE	.	0.045	.	0.023	.
MEAN	.	0.012	.	0.017	.
STANDARD DEV	.	0.010	.	0.007	.
MEDIAN	.	0.010	.	0.020	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	18	0	18	0	18
MINIMUM	0.015	.	700.000	.	109.000
MAXIMUM	0.110	.	1210.000	.	246.000
RANGE	0.095	.	510.000	.	137.000
MEAN	0.059	.	1003.333	.	199.944
STANDARD DEV	0.022	.	141.040	.	41.113
MEDIAN	0.055	.	1045.000	.	209.500

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	18	18	18	0
MINIMUM	104.000	165.000	194.000	.
MAXIMUM	223.000	165.000	194.000	.

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data (continued).

RANGE	119.000	0.000	0.000	.
MEAN	178.389	165.000	194.000	.
STANDARD DEV	34.563	0.000	0.000	.
MEDIAN	187.500	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-051
YEAR = 98.000

TOTAL OBSERVATIONS: 19

	PH	SC	CUT	CUF	NIT
N OF CASES	19	0	19	0	19
MINIMUM	6.900	.	0.014	.	0.170
MAXIMUM	7.400	.	0.069	.	0.799
RANGE	0.500	.	0.055	.	0.629
MEAN	7.232	.	0.031	.	0.491
STANDARD DEV	0.138	.	0.015	.	0.172
MEDIAN	7.300	.	0.026	.	0.546

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	19	0	19	0
MINIMUM	.	-0.001	.	-0.010	.
MAXIMUM	.	0.032	.	0.167	.
RANGE	.	0.033	.	0.177	.
MEAN	.	0.010	.	0.023	.
STANDARD DEV	.	0.008	.	0.036	.
MEDIAN	.	0.008	.	0.017	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	18	0	19	0	19
MINIMUM	-0.030	.	586.000	.	105.000
MAXIMUM	0.140	.	1250.000	.	263.000
RANGE	0.170	.	664.000	.	158.000
MEAN	0.029	.	966.211	.	196.316
STANDARD DEV	0.050	.	198.994	.	46.164
MEDIAN	0.040	.	1010.000	.	203.000

	MGT	TOTHARD	ALK	FLOWLM

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data (continued).

N OF CASES	19	19	19	0
MINIMUM	99.800	165.000	194.000	.
MAXIMUM	237.000	165.000	194.000	.
RANGE	137.200	0.000	0.000	.
MEAN	177.568	165.000	194.000	.
STANDARD DEV	41.478	0.000	0.000	.
MEDIAN	184.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-051
YEAR = 99.000

TOTAL OBSERVATIONS: 16

	PH	SC	CUT	CUF	NIT
N OF CASES	16	0	16	0	16
MINIMUM	6.800	.	0.001	.	0.030
MAXIMUM	7.300	.	0.034	.	1.690
RANGE	0.500	.	0.033	.	1.661
MEAN	7.113	.	0.015	.	0.608
STANDARD DEV	0.154	.	0.010	.	0.533
MEDIAN	7.100	.	0.013	.	0.395

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	16	0	16	0
MINIMUM	.	0.001	.	-0.010	.
MAXIMUM	.	0.013	.	0.031	.
RANGE	.	0.012	.	0.041	.
MEAN	.	0.006	.	0.015	.
STANDARD DEV	.	0.004	.	0.014	.
MEDIAN	.	0.005	.	0.017	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	16	0	16	0	16
MINIMUM	-0.030	.	378.000	.	74.400
MAXIMUM	0.410	.	1570.000	.	284.000
RANGE	0.440	.	1192.000	.	209.600
MEAN	0.085	.	1138.125	.	213.963
STANDARD DEV	0.092	.	328.183	.	58.242
MEDIAN	0.070	.	1170.000	.	229.500

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	16	16	10	16
MINIMUM	61.700	165.000	194.000	10.514
MAXIMUM	264.000	1800.000	194.000	141.938

Table A2.1. Summary statistics of water quality data from the input (W1D-051) of the W1D wetland treatment system, 1992-1999 data (continued).

RANGE	202.300	1635.000	0.000	131.424
MEAN	198.981	718.125	194.000	59.305
STANDARD DEV	57.140	742.338	0.000	41.449
MEDIAN	211.500	165.000	194.000	56.512

Note: The parameters with a T suffix are total values, parameters with an F suffix are filtered values. TOTHARD is total hardness, FLOWLM is liters/min, and values preceded with a hyphen are less than the detection limit.

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data.

Site W1D-052

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-052
YEAR = 92.000

TOTAL OBSERVATIONS: 52

	PH	SC	CUT	CUF	NIT
N OF CASES	46	2	52	0	52
MINIMUM	6.400	2480.000	0.000	.	0.020
MAXIMUM	7.600	2590.000	0.032	.	0.590
RANGE	1.200	110.000	0.032	.	0.570
MEAN	7.026	2535.000	0.012	.	0.315
STANDARD DEV	0.232	77.782	0.007	.	0.165
MEDIAN	7.000	2535.000	0.011	.	0.310

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	52	0	52	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.060	.	0.060	.
RANGE	.	0.060	.	0.055	.
MEAN	.	0.016	.	0.013	.
STANDARD DEV	.	0.013	.	0.016	.
MEDIAN	.	0.011	.	0.005	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	20	13	48	0	35
MINIMUM	0.050	0.800	817.000	.	71.000
MAXIMUM	5.100	30.800	2811.000	.	300.000
RANGE	5.050	30.000	1994.000	.	229.000
MEAN	0.925	10.646	1840.875	.	193.429
STANDARD DEV	1.403	8.893	553.348	.	53.008
MEDIAN	0.200	9.600	1936.500	.	210.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	35	52	52	0
MINIMUM	90.000	165.000	194.000	.
MAXIMUM	350.000	165.000	194.000	.
RANGE	260.000	0.000	0.000	.
MEAN	234.914	165.000	194.000	.
STANDARD DEV	77.187	0.000	0.000	.

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data (continued).

MEDIAN	260.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-052
YEAR = 93.000

TOTAL OBSERVATIONS: 49

	PH	SC	CUT	CUF	NIT
N OF CASES	47	0	49	0	49
MINIMUM	6.800	.	-0.001	.	0.044
MAXIMUM	7.500	.	0.024	.	1.200
RANGE	0.700	.	0.025	.	1.156
MEAN	7.145	.	0.009	.	0.354
STANDARD DEV	0.180	.	0.006	.	0.320
MEDIAN	7.100	.	0.007	.	0.170

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	38	0	38	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.010	.	0.050	.
RANGE	.	0.010	.	0.045	.
MEAN	.	0.004	.	0.009	.
STANDARD DEV	.	0.002	.	0.011	.
MEDIAN	.	0.003	.	0.005	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	35	34	49	0	36
MINIMUM	0.050	0.400	583.000	.	80.000
MAXIMUM	1.400	21.600	2221.000	.	320.000
RANGE	1.350	21.200	1638.000	.	240.000
MEAN	0.313	7.488	1286.776	.	192.778
STANDARD DEV	0.262	4.319	499.339	.	74.512
MEDIAN	0.200	6.800	1287.000	.	175.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	36	49	49	0
MINIMUM	70.000	165.000	194.000	.
MAXIMUM	420.000	165.000	194.000	.
RANGE	350.000	0.000	0.000	.
MEAN	222.806	165.000	194.000	.
STANDARD DEV	104.943	0.000	0.000	.
MEDIAN	175.000	165.000	194.000	.

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-052
YEAR = 94.000

TOTAL OBSERVATIONS: 88

	PH	SC	CUT	CUF	NIT
N OF CASES	31	0	33	2	87
MINIMUM	6.900	.	0.001	0.003	0.007
MAXIMUM	8.200	.	0.060	0.004	1.800
RANGE	1.300	.	0.060	0.001	1.793
MEAN	7.355	.	0.007	0.004	0.426
STANDARD DEV	0.266	.	0.010	0.001	0.432
MEDIAN	7.300	.	0.005	0.004	0.280

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	19	0	19	0
MINIMUM	0.110	0.001	.	0.005	.
MAXIMUM	0.110	0.021	.	0.200	.
RANGE	0.000	0.020	.	0.195	.
MEAN	0.110	0.004	.	0.017	.
STANDARD DEV	0.000	0.005	.	0.044	.
MEDIAN	0.110	0.002	.	0.005	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	19	19	33	2	19
MINIMUM	0.000	1.000	338.000	974.000	70.000
MAXIMUM	0.900	19.600	2208.000	1015.000	310.000
RANGE	0.900	18.600	1870.000	41.000	240.000
MEAN	0.209	5.947	1508.636	994.500	215.263
STANDARD DEV	0.258	4.355	491.458	28.991	71.598
MEDIAN	0.100	6.000	1693.000	994.500	240.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	19	88	88	0
MINIMUM	70.000	165.000	194.000	.
MAXIMUM	380.000	165.000	194.000	.
RANGE	310.000	0.000	0.000	.
MEAN	264.737	165.000	194.000	.
STANDARD DEV	95.357	0.000	0.000	.
MEDIAN	310.000	165.000	194.000	.

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-052
YEAR = 95.000

TOTAL OBSERVATIONS: 13

	PH	SC	CUT	CUF	NIT
N OF CASES	13	0	13	0	13
MINIMUM	6.900	.	0.001	.	0.120
MAXIMUM	7.600	.	0.007	.	0.880
RANGE	0.700	.	0.006	.	0.760
MEAN	7.300	.	0.003	.	0.392
STANDARD DEV	0.178	.	0.002	.	0.261
MEDIAN	7.400	.	0.003	.	0.360

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	13	0	13	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.020	.	0.100	.
RANGE	.	0.020	.	0.095	.
MEAN	.	0.005	.	0.042	.
STANDARD DEV	.	0.005	.	0.048	.
MEDIAN	.	0.002	.	0.010	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	13	8	13	0	13
MINIMUM	0.050	-0.400	382.000	.	60.000
MAXIMUM	0.100	8.800	1658.000	.	285.000
RANGE	0.050	9.200	1276.000	.	225.000
MEAN	0.077	3.400	1103.308	.	175.846
STANDARD DEV	0.026	3.186	396.754	.	64.708
MEDIAN	0.100	3.600	1017.000	.	170.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	13	13	13	0
MINIMUM	60.000	165.000	194.000	.
MAXIMUM	319.000	165.000	194.000	.
RANGE	259.000	0.000	0.000	.
MEAN	203.769	165.000	194.000	.
STANDARD DEV	72.024	0.000	0.000	.
MEDIAN	200.000	165.000	194.000	.

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-052
YEAR = 96.000

TOTAL OBSERVATIONS: 15

	PH	SC	CUT	CUF	NIT
N OF CASES	15	0	15	0	15
MINIMUM	7.000	.	-0.001	.	0.056
MAXIMUM	7.600	.	0.033	.	0.891
RANGE	0.600	.	0.034	.	0.835
MEAN	7.327	.	0.009	.	0.337
STANDARD DEV	0.162	.	0.009	.	0.246
MEDIAN	7.400	.	0.008	.	0.261

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	15	0	15	0
MINIMUM	.	-0.001	.	-0.010	.
MAXIMUM	.	0.007	.	0.025	.
RANGE	.	0.008	.	0.035	.
MEAN	.	0.002	.	0.003	.
STANDARD DEV	.	0.003	.	0.015	.
MEDIAN	.	0.002	.	-0.010	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	15	15	15	0	14
MINIMUM	0.050	-1.000	679.000	.	106.000
MAXIMUM	1.800	17.200	2380.000	.	272.000
RANGE	1.750	18.200	1701.000	.	166.000
MEAN	0.213	4.707	1324.067	.	196.857
STANDARD DEV	0.443	4.964	414.386	.	54.898
MEDIAN	0.090	3.200	1361.000	.	205.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	14	15	15	0
MINIMUM	116.000	165.000	194.000	.
MAXIMUM	260.000	165.000	194.000	.
RANGE	144.000	0.000	0.000	.
MEAN	195.500	165.000	194.000	.
STANDARD DEV	53.904	0.000	0.000	.
MEDIAN	202.500	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-052

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data (continued).

YEAR = 97.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	17	0	17	0	17
MINIMUM	7.100	.	-0.001	.	0.050
MAXIMUM	7.600	.	0.004	.	0.580
RANGE	0.500	.	0.005	.	0.530
MEAN	7.318	.	0.001	.	0.178
STANDARD DEV	0.170	.	0.002	.	0.148
MEDIAN	7.200	.	0.001	.	0.102

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	17	0	17	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.004	.	0.017	.
RANGE	.	0.003	.	0.012	.
MEAN	.	0.001	.	0.007	.
STANDARD DEV	.	0.001	.	0.004	.
MEDIAN	.	0.001	.	0.005	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	17	17	17	0	17
MINIMUM	0.015	1.200	566.000	.	66.700
MAXIMUM	0.090	6.400	1000.000	.	188.000
RANGE	0.075	5.200	434.000	.	121.300
MEAN	0.055	3.153	790.765	.	146.747
STANDARD DEV	0.021	1.679	126.504	.	33.973
MEDIAN	0.050	2.800	808.000	.	149.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	17	17	17	0
MINIMUM	58.000	165.000	194.000	.
MAXIMUM	184.000	165.000	194.000	.
RANGE	126.000	0.000	0.000	.
MEAN	142.029	165.000	194.000	.
STANDARD DEV	34.317	0.000	0.000	.
MEDIAN	148.000	165.000	194.000	.

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-052
YEAR = 98.000

TOTAL OBSERVATIONS: 18

	PH	SC	CUT	CUF	NIT
N OF CASES	17	0	17	0	17
MINIMUM	7.100	.	-0.001	.	0.009
MAXIMUM	7.600	.	0.012	.	0.640
RANGE	0.500	.	0.013	.	0.631
MEAN	7.294	.	0.002	.	0.121
STANDARD DEV	0.125	.	0.003	.	0.159
MEDIAN	7.300	.	0.002	.	0.057

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	17	0	17	0
MINIMUM	.	-0.001	.	-0.010	.
MAXIMUM	.	0.006	.	0.030	.
RANGE	.	0.007	.	0.040	.
MEAN	.	0.000	.	0.006	.
STANDARD DEV	.	0.002	.	0.014	.
MEDIAN	.	-0.001	.	0.012	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	17	17	17	0	17
MINIMUM	-0.030	1.200	421.000	.	90.100
MAXIMUM	0.130	10.000	1160.000	.	199.000
RANGE	0.160	8.800	739.000	.	108.900
MEAN	0.026	4.353	816.647	.	147.476
STANDARD DEV	0.054	2.433	196.873	.	35.244
MEDIAN	0.040	4.000	808.000	.	149.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	17	18	18	0
MINIMUM	84.700	165.000	194.000	.
MAXIMUM	209.000	165.000	194.000	.
RANGE	124.300	0.000	0.000	.
MEAN	148.394	165.000	194.000	.
STANDARD DEV	38.836	0.000	0.000	.
MEDIAN	149.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-052

Table A2.2. Summary statistics of water quality data from the output (W1D-052) of the W1D wetland treatment system, 1992-1999 data (continued).

YEAR = 99.000
TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	17	0	17	0	17
MINIMUM	6.800	.	-0.001	.	0.037
MAXIMUM	7.900	.	0.015	.	0.304
RANGE	1.100	.	0.016	.	0.267
MEAN	7.288	.	0.001	.	0.140
STANDARD DEV	0.287	.	0.004	.	0.072
MEDIAN	7.300	.	-0.001	.	0.117

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	17	0	17	0
MINIMUM	.	-0.001	.	-0.010	.
MAXIMUM	.	0.003	.	0.024	.
RANGE	.	0.004	.	0.034	.
MEAN	.	-0.000	.	0.007	.
STANDARD DEV	.	0.001	.	0.014	.
MEDIAN	.	-0.001	.	0.013	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	17	10	17	0	17
MINIMUM	0.020	-1.000	479.000	.	65.300
MAXIMUM	0.330	4.000	1090.000	.	206.000
RANGE	0.310	5.000	611.000	.	140.700
MEAN	0.074	2.620	773.706	.	141.000
STANDARD DEV	0.070	1.709	173.629	.	39.216
MEDIAN	0.060	3.250	765.000	.	142.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	17	17	11	6
MINIMUM	57.000	165.000	194.000	13.142
MAXIMUM	201.000	1340.000	194.000	91.997
RANGE	144.000	1175.000	0.000	78.854
MEAN	135.729	506.765	194.000	40.741
STANDARD DEV	39.848	482.655	0.000	28.782
MEDIAN	144.000	165.000	194.000	36.799

Note: The parameters with a T suffix are total values, parameters with an F suffix are filtered values. TOTHARD is total hardness, FLOWLM is liters/min, and values preceded with a hyphen are less than the detection limit.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052.

- The sites called Upstream1, etc. are from sites upstream of site W1D-051 (the input to the wetland system; see figure 4).
- The sites called W1D-1, W1D-2, etc. are profile sites within the original system, and
- The site called W1D-050 is the outfall from the enlarged W1D system.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Upstream1
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	9	0	14	0	14
MINIMUM	6.500	.	0.030	.	0.100
MAXIMUM	8.000	.	0.450	.	0.620
RANGE	1.500	.	0.420	.	0.520
MEAN	6.833	.	0.166	.	0.439
STANDARD DEV	0.469	.	0.127	.	0.147
MEDIAN	6.700	.	0.105	.	0.420

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	3	0
MINIMUM	.	0.007	.	0.060	.
MAXIMUM	.	0.008	.	0.070	.
RANGE	.	0.001	.	0.010	.
MEAN	.	0.008	.	0.067	.
STANDARD DEV	.	0.001	.	0.006	.
MEDIAN	.	0.008	.	0.070	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	599.000	.	240.000
MAXIMUM	.	.	2374.000	.	270.000
RANGE	.	.	1775.000	.	30.000
MEAN	.	.	1326.071	.	255.000
STANDARD DEV	.	.	413.263	.	21.213
MEDIAN	.	.	1316.000	.	255.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	160.000	165.000	194.000	.
MAXIMUM	290.000	165.000	194.000	.
RANGE	130.000	0.000	0.000	.
MEAN	225.000	165.000	194.000	.
STANDARD DEV	91.924	0.000	0.000	.
MEDIAN	225.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Upstream1
YEAR = 94.000

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	14	0	14	2	14
MINIMUM	6.500	.	0.030	0.070	0.240
MAXIMUM	7.400	.	0.190	0.180	0.460
RANGE	0.900	.	0.160	0.110	0.220
MEAN	6.936	.	0.089	0.125	0.335
STANDARD DEV	0.305	.	0.050	0.078	0.066
MEDIAN	6.900	.	0.070	0.125	0.310

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.420	.	.	.	.
MAXIMUM	0.450	.	.	.	.
RANGE	0.030	.	.	.	.
MEAN	0.435	.	.	.	.
STANDARD DEV	0.021	.	.	.	.
MEDIAN	0.435	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1452.000	1945.000	.
MAXIMUM	.	.	2509.000	2265.000	.
RANGE	.	.	1057.000	320.000	.
MEAN	.	.	1999.714	2105.000	.
STANDARD DEV	.	.	276.788	226.274	.
MEDIAN	.	.	1985.500	2105.000	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	14	14	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Upstream2
YEAR = 92.000

TOTAL OBSERVATIONS: 5

	PH	SC	CUT	CUF	NIT
N OF CASES	1	0	5	0	5
MINIMUM	7.300	.	0.023	.	0.970
MAXIMUM	7.300	.	0.540	.	9.100
RANGE	0.000	.	0.517	.	8.130

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MEAN	7.300	.	0.283	.	4.734
STANDARD DEV	.	.	0.227	.	2.990
MEDIAN	7.300	.	0.310	.	3.900

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	5	0	5	0
MINIMUM	.	0.024	.	0.040	.
MAXIMUM	.	0.390	.	0.250	.
RANGE	.	0.366	.	0.210	.
MEAN	.	0.185	.	0.140	.
STANDARD DEV	.	0.135	.	0.077	.
MEDIAN	.	0.150	.	0.130	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	5	0	0
MINIMUM	.	.	1388.000	.	.
MAXIMUM	.	.	2554.000	.	.
RANGE	.	.	1166.000	.	.
MEAN	.	.	1979.600	.	.
STANDARD DEV	.	.	517.891	.	.
MEDIAN	.	.	2138.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	5	5	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Upstream2
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.030	.	0.790
MAXIMUM	.	.	0.210	.	5.400
RANGE	.	.	0.180	.	4.610
MEAN	.	.	0.091	.	3.002
STANDARD DEV	.	.	0.052	.	1.580
MEDIAN	.	.	0.070	.	3.300

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.004	.	0.050	.
MAXIMUM	.	0.004	.	0.050	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

RANGE	.	0.000	.	0.000	.
MEAN	.	0.004	.	0.050	.
STANDARD DEV	.	0.000	.	0.000	.
MEDIAN	.	0.004	.	0.050	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	752.000	.	130.000
MAXIMUM	.	.	2395.000	.	160.000
RANGE	.	.	1643.000	.	30.000
MEAN	.	.	1643.929	.	145.000
STANDARD DEV	.	.	518.735	.	21.213
MEDIAN	.	.	1536.000	.	145.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	140.000	165.000	194.000	.
MAXIMUM	280.000	165.000	194.000	.
RANGE	140.000	0.000	0.000	.
MEAN	210.000	165.000	194.000	.
STANDARD DEV	98.995	0.000	0.000	.
MEDIAN	210.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Upstream2
YEAR = 94.000

TOTAL OBSERVATIONS: 15

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	15	2	15
MINIMUM	.	.	0.013	0.060	1.200
MAXIMUM	.	.	0.190	0.120	6.600
RANGE	.	.	0.177	0.060	5.400
MEAN	.	.	0.074	0.090	3.400
STANDARD DEV	.	.	0.058	0.042	1.531
MEDIAN	.	.	0.050	0.090	3.500

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	2.300	.	.	.	.
MAXIMUM	3.200	.	.	.	.
RANGE	0.900	.	.	.	.
MEAN	2.750	.	.	.	.
STANDARD DEV	0.636	.	.	.	.
MEDIAN	2.750	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	15	2	0
MINIMUM	.	.	1465.000	1924.000	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MAXIMUM	.	.	2545.000	2035.000	.
RANGE	.	.	1080.000	111.000	.
MEAN	.	.	2001.467	1979.500	.
STANDARD DEV	.	.	322.289	78.489	.
MEDIAN	.	.	2035.000	1979.500	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	15	15	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Upstream3
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT
N OF CASES	8	0	11	0	11
MINIMUM	6.400	.	0.029	.	2.700
MAXIMUM	7.200	.	0.310	.	6.900
RANGE	0.800	.	0.281	.	4.200
MEAN	6.863	.	0.132	.	4.991
STANDARD DEV	0.272	.	0.108	.	1.390
MEDIAN	6.950	.	0.110	.	4.600

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.032	.	0.040	.
MAXIMUM	.	0.100	.	0.120	.
RANGE	.	0.068	.	0.080	.
MEAN	.	0.064	.	0.077	.
STANDARD DEV	.	0.021	.	0.025	.
MEDIAN	.	0.060	.	0.080	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	2233.000	.	.
MAXIMUM	.	.	2983.000	.	.
RANGE	.	.	750.000	.	.
MEAN	.	.	2554.727	.	.
STANDARD DEV	.	.	224.492	.	.
MEDIAN	.	.	2541.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Upstream3
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.019	.	0.850
MAXIMUM	.	.	0.060	.	8.300
RANGE	.	.	0.041	.	7.450
MEAN	.	.	0.034	.	4.146
STANDARD DEV	.	.	0.011	.	2.500
MEDIAN	.	.	0.030	.	4.600

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	3	0	3	0
MINIMUM	.	0.002	.	0.060	.
MAXIMUM	.	0.006	.	0.070	.
RANGE	.	0.004	.	0.010	.
MEAN	.	0.004	.	0.063	.
STANDARD DEV	.	0.002	.	0.006	.
MEDIAN	.	0.005	.	0.060	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	1133.000	.	180.000
MAXIMUM	.	.	2708.000	.	200.000
RANGE	.	.	1575.000	.	20.000
MEAN	.	.	1986.357	.	190.000
STANDARD DEV	.	.	540.673	.	14.142
MEDIAN	.	.	2068.500	.	190.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	180.000	165.000	194.000	.
MAXIMUM	190.000	165.000	194.000	.
RANGE	10.000	0.000	0.000	.
MEAN	185.000	165.000	194.000	.
STANDARD DEV	7.071	0.000	0.000	.
MEDIAN	185.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

SITE$ = Upstream3
YEAR = 94.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	14
MINIMUM	.	.	0.015	0.060	0.880
MAXIMUM	.	.	0.140	0.070	7.400
RANGE	.	.	0.125	0.010	6.520
MEAN	.	.	0.060	0.065	4.456
STANDARD DEV	.	.	0.037	0.007	1.653
MEDIAN	.	.	0.040	0.065	4.650

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	2.800	.	.	.	.
MAXIMUM	3.800	.	.	.	.
RANGE	1.000	.	.	.	.
MEAN	3.300	.	.	.	.
STANDARD DEV	0.707	.	.	.	.
MEDIAN	3.300	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1416.000	2169.000	.
MAXIMUM	.	.	2629.000	2359.000	.
RANGE	.	.	1213.000	190.000	.
MEAN	.	.	2261.357	2264.000	.
STANDARD DEV	.	.	293.720	134.350	.
MEDIAN	.	.	2305.500	2264.000	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	14	14	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-050
YEAR = 95.000

TOTAL OBSERVATIONS: 8

	PH	SC	CUT	CUF	NIT
N OF CASES	8	0	8	0	8
MINIMUM	7.000	.	0.002	.	0.039

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MAXIMUM	7.500	.	0.012	.	0.540
RANGE	0.500	.	0.010	.	0.501
MEAN	7.313	.	0.007	.	0.199
STANDARD DEV	0.181	.	0.004	.	0.208
MEDIAN	7.350	.	0.007	.	0.100

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	8	0	8	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.002	.	0.010	.
RANGE	.	0.002	.	0.005	.
MEAN	.	0.001	.	0.006	.
STANDARD DEV	.	0.001	.	0.002	.
MEDIAN	.	0.001	.	0.005	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	8	8	8	0	8
MINIMUM	0.050	0.800	865.000	.	150.000
MAXIMUM	0.300	4.800	1187.000	.	197.000
RANGE	0.250	4.000	322.000	.	47.000
MEAN	0.131	2.650	974.250	.	174.250
STANDARD DEV	0.080	1.417	128.858	.	18.783
MEDIAN	0.100	3.200	898.000	.	178.500

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	8	8	8	0
MINIMUM	140.000	165.000	194.000	.
MAXIMUM	220.000	165.000	194.000	.
RANGE	80.000	0.000	0.000	.
MEAN	181.375	165.000	194.000	.
STANDARD DEV	27.375	0.000	0.000	.
MEDIAN	183.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-050
YEAR = 96.000

TOTAL OBSERVATIONS: 18

	PH	SC	CUT	CUF	NIT
N OF CASES	18	0	18	0	18
MINIMUM	7.000	.	-0.001	.	0.050
MAXIMUM	7.700	.	0.017	.	0.533
RANGE	0.700	.	0.018	.	0.483
MEAN	7.439	.	0.007	.	0.193
STANDARD DEV	0.185	.	0.005	.	0.135
MEDIAN	7.450	.	0.006	.	0.126

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	18	0	18	0

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.005	.	0.028	.
RANGE	.	0.005	.	0.023	.
MEAN	.	0.001	.	0.012	.
STANDARD DEV	.	0.001	.	0.008	.
MEDIAN	.	0.001	.	0.010	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	18	18	18	0	18
MINIMUM	0.040	-1.000	114.000	.	62.400
MAXIMUM	0.300	5.600	1580.000	.	240.000
RANGE	0.260	6.600	1466.000	.	177.600
MEAN	0.106	2.839	982.500	.	175.133
STANDARD DEV	0.058	1.665	353.075	.	48.035
MEDIAN	0.100	3.000	1042.500	.	179.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	18	18	18	0
MINIMUM	78.000	165.000	194.000	.
MAXIMUM	240.000	165.000	194.000	.
RANGE	162.000	0.000	0.000	.
MEAN	167.833	165.000	194.000	.
STANDARD DEV	40.083	0.000	0.000	.
MEDIAN	176.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-050
YEAR = 97.000

TOTAL OBSERVATIONS: 24

	PH	SC	CUT	CUF	NIT
N OF CASES	24	0	24	0	24
MINIMUM	7.000	.	-0.002	.	0.049
MAXIMUM	7.700	.	0.005	.	0.363
RANGE	0.700	.	0.007	.	0.314
MEAN	7.404	.	0.001	.	0.143
STANDARD DEV	0.220	.	0.002	.	0.077
MEDIAN	7.500	.	0.001	.	0.125

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	24	0	24	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.004	.	0.038	.
RANGE	.	0.004	.	0.033	.
MEAN	.	0.001	.	0.009	.
STANDARD DEV	.	0.001	.	0.007	.
MEDIAN	.	0.001	.	0.005	.

	FEF	TSS	SO4T	SO4F	CAT

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

N OF CASES	24	24	24	0	24
MINIMUM	0.030	-1.000	568.000	.	82.700
MAXIMUM	0.770	15.600	1300.000	.	205.000
RANGE	0.740	16.600	732.000	.	122.300
MEAN	0.134	3.667	896.958	.	154.971
STANDARD DEV	0.145	3.484	210.232	.	33.005
MEDIAN	0.100	2.400	832.000	.	156.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	24	24	24	0
MINIMUM	71.500	165.000	194.000	.
MAXIMUM	212.000	165.000	194.000	.
RANGE	140.500	0.000	0.000	.
MEAN	153.104	165.000	194.000	.
STANDARD DEV	37.511	0.000	0.000	.
MEDIAN	150.500	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-050
YEAR = 98.000

TOTAL OBSERVATIONS: 23

	PH	SC	CUT	CUF	NIT
N OF CASES	23	0	23	0	23
MINIMUM	6.900	.	-0.001	.	0.019
MAXIMUM	7.700	.	0.009	.	0.288
RANGE	0.800	.	0.010	.	0.270
MEAN	7.383	.	0.001	.	0.073
STANDARD DEV	0.229	.	0.003	.	0.061
MEDIAN	7.400	.	-0.001	.	0.054

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	23	0	23	0
MINIMUM	.	-0.001	.	-0.010	.
MAXIMUM	.	1.900	.	0.021	.
RANGE	.	1.901	.	0.031	.
MEAN	.	0.082	.	0.004	.
STANDARD DEV	.	0.396	.	0.013	.
MEDIAN	.	-0.001	.	0.011	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	23	23	23	0	23
MINIMUM	-0.030	-1.000	299.000	.	73.600
MAXIMUM	0.280	8.400	1060.000	.	201.000
RANGE	0.310	9.400	761.000	.	127.400
MEAN	0.070	1.722	779.913	.	147.722
STANDARD DEV	0.091	3.249	179.481	.	28.924
MEDIAN	0.050	1.000	776.000	.	143.000

	MGT	TOTHARD	ALK	FLOWLM

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

N OF CASES	23	23	23	0
MINIMUM	70.400	165.000	194.000	.
MAXIMUM	194.000	165.000	194.000	.
RANGE	123.600	0.000	0.000	.
MEAN	147.974	165.000	194.000	.
STANDARD DEV	29.113	0.000	0.000	.
MEDIAN	145.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-050
YEAR = 99.000

TOTAL OBSERVATIONS: 22

	PH	SC	CUT	CUF	NIT
N OF CASES	22	0	22	0	22
MINIMUM	6.800	.	-0.001	.	0.038
MAXIMUM	7.800	.	0.008	.	0.209
RANGE	1.000	.	0.009	.	0.171
MEAN	7.382	.	0.001	.	0.098
STANDARD DEV	0.254	.	0.002	.	0.049
MEDIAN	7.400	.	0.001	.	0.093

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	22	0	22	0
MINIMUM	.	-0.001	.	-0.010	.
MAXIMUM	.	0.002	.	0.021	.
RANGE	.	0.003	.	0.031	.
MEAN	.	-0.000	.	0.005	.
STANDARD DEV	.	0.001	.	0.013	.
MEDIAN	.	-0.001	.	0.011	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	22	6	22	0	22
MINIMUM	-0.030	2.000	299.000	.	57.800
MAXIMUM	0.460	9.000	1120.000	.	200.000
RANGE	0.490	7.000	821.000	.	142.200
MEAN	0.082	4.133	789.273	.	144.745
STANDARD DEV	0.112	2.515	220.909	.	37.184
MEDIAN	0.060	3.650	814.000	.	145.500

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	22	22	16	22
MINIMUM	52.300	165.000	194.000	5.257
MAXIMUM	206.000	1100.000	194.000	935.739
RANGE	153.700	935.000	0.000	930.482
MEAN	143.255	400.045	194.000	155.678
STANDARD DEV	39.190	394.888	0.000	228.654
MEDIAN	144.500	165.000	194.000	88.054

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-1
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	7.400	.	0.001	.	0.120
MAXIMUM	7.500	.	0.025	.	5.000
RANGE	0.100	.	0.024	.	4.880
MEAN	7.450	.	0.010	.	1.053
STANDARD DEV	0.071	.	0.008	.	1.723
MEDIAN	7.450	.	0.006	.	0.330

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.000	.	0.005	.
MAXIMUM	.	0.029	.	0.080	.
RANGE	.	0.029	.	0.075	.
MEAN	.	0.005	.	0.040	.
STANDARD DEV	.	0.009	.	0.025	.
MEDIAN	.	0.000	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	1171.000	.	.
MAXIMUM	.	.	2800.000	.	.
RANGE	.	.	1629.000	.	.
MEAN	.	.	1643.182	.	.
STANDARD DEV	.	.	543.279	.	.
MEDIAN	.	.	1411.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-1
YEAR = 93.000

TOTAL OBSERVATIONS: 14

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

	PH	SC	CUT	CUF	NIT
N OF CASES	1	0	14	0	14
MINIMUM	6.800	.	0.004	.	0.110
MAXIMUM	6.800	.	0.010	.	1.800
RANGE	0.000	.	0.006	.	1.690
MEAN	6.800	.	0.006	.	0.474
STANDARD DEV	.	.	0.002	.	0.500
MEDIAN	6.800	.	0.006	.	0.280

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.040	.
MAXIMUM	.	0.001	.	0.040	.
RANGE	.	0.000	.	0.000	.
MEAN	.	0.001	.	0.040	.
STANDARD DEV	.	0.000	.	0.000	.
MEDIAN	.	0.001	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	125.000	.	70.000
MAXIMUM	.	.	1178.000	.	73.000
RANGE	.	.	1053.000	.	3.000
MEAN	.	.	617.786	.	71.500
STANDARD DEV	.	.	264.785	.	2.121
MEDIAN	.	.	587.000	.	71.500

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	60.000	165.000	194.000	.
MAXIMUM	64.000	165.000	194.000	.
RANGE	4.000	0.000	0.000	.
MEAN	62.000	165.000	194.000	.
STANDARD DEV	2.828	0.000	0.000	.
MEDIAN	62.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-1
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	17
MINIMUM	.	.	0.005	0.006	0.200
MAXIMUM	.	.	0.030	0.006	4.200
RANGE	.	.	0.025	0.000	4.000
MEAN	.	.	0.011	0.006	0.703
STANDARD DEV	.	.	0.006	0.000	0.975

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MEDIAN	.	.	0.009	0.006	0.400

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.220	.	.	.	.
MAXIMUM	0.420	.	.	.	.
RANGE	0.200	.	.	.	.
MEAN	0.320	.	.	.	.
STANDARD DEV	0.141	.	.	.	.
MEDIAN	0.320	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	398.000	451.000	.
MAXIMUM	.	.	3136.000	763.000	.
RANGE	.	.	2738.000	312.000	.
MEAN	.	.	950.000	607.000	.
STANDARD DEV	.	.	825.237	220.617	.
MEDIAN	.	.	535.500	607.000	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-10
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	6.800	.	0.000	.	0.140
MAXIMUM	7.300	.	0.036	.	1.000
RANGE	0.500	.	0.036	.	0.860
MEAN	7.050	.	0.009	.	0.613
STANDARD DEV	0.354	.	0.011	.	0.288
MEDIAN	7.050	.	0.005	.	0.690

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.000	.	0.005	.
MAXIMUM	.	0.014	.	0.070	.
RANGE	.	0.014	.	0.065	.
MEAN	.	0.004	.	0.037	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

STANDARD DEV	.	0.005	.	0.024	.
MEDIAN	.	0.002	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	1981.000	.	.
MAXIMUM	.	.	2737.000	.	.
RANGE	.	.	756.000	.	.
MEAN	.	.	2225.545	.	.
STANDARD DEV	.	.	243.584	.	.
MEDIAN	.	.	2125.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-10
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.002	.	0.070
MAXIMUM	.	.	0.012	.	1.400
RANGE	.	.	0.010	.	1.330
MEAN	.	.	0.005	.	0.495
STANDARD DEV	.	.	0.002	.	0.430
MEDIAN	.	.	0.006	.	0.545

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.030	.
MAXIMUM	.	0.001	.	0.050	.
RANGE	.	0.000	.	0.020	.
MEAN	.	0.001	.	0.040	.
STANDARD DEV	.	0.000	.	0.014	.
MEDIAN	.	0.001	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	714.000	.	120.000
MAXIMUM	.	.	1880.000	.	130.000
RANGE	.	.	1166.000	.	10.000

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MEAN	.	.	1314.357	.	125.000
STANDARD DEV	.	.	347.174	.	7.071
MEDIAN	.	.	1268.500	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	140.000	165.000	194.000	.
MAXIMUM	140.000	165.000	194.000	.
RANGE	0.000	0.000	0.000	.
MEAN	140.000	165.000	194.000	.
STANDARD DEV	0.000	0.000	0.000	.
MEDIAN	140.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-10
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	16
MINIMUM	.	.	0.002	0.004	0.130
MAXIMUM	.	.	0.010	0.005	1.200
RANGE	.	.	0.008	0.001	1.070
MEAN	.	.	0.005	0.005	0.616
STANDARD DEV	.	.	0.002	0.001	0.415
MEDIAN	.	.	0.005	0.005	0.495

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.130	.	.	.	.
MAXIMUM	0.130	.	.	.	.
RANGE	0.000	.	.	.	.
MEAN	0.130	.	.	.	.
STANDARD DEV	0.000	.	.	.	.
MEDIAN	0.130	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1089.000	1089.000	.
MAXIMUM	.	.	2128.000	1110.000	.
RANGE	.	.	1039.000	21.000	.
MEAN	.	.	1682.929	1099.500	.
STANDARD DEV	.	.	345.618	14.849	.
MEDIAN	.	.	1794.000	1099.500	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-11
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	6.800	.	0.000	.	0.160
MAXIMUM	7.300	.	0.010	.	0.870
RANGE	0.500	.	0.010	.	0.710
MEAN	7.050	.	0.005	.	0.547
STANDARD DEV	0.354	.	0.004	.	0.275
MEDIAN	7.050	.	0.003	.	0.590

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.000	.	0.005	.
MAXIMUM	.	0.016	.	0.060	.
RANGE	.	0.016	.	0.055	.
MEAN	.	0.005	.	0.035	.
STANDARD DEV	.	0.005	.	0.022	.
MEDIAN	.	0.003	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	1892.000	.	.
MAXIMUM	.	.	3015.000	.	.
RANGE	.	.	1123.000	.	.
MEAN	.	.	2215.636	.	.
STANDARD DEV	.	.	330.467	.	.
MEDIAN	.	.	2068.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-11
YEAR = 93.000

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.003	.	0.070
MAXIMUM	.	.	0.012	.	1.500
RANGE	.	.	0.009	.	1.430
MEAN	.	.	0.005	.	0.543
STANDARD DEV	.	.	0.003	.	0.467
MEDIAN	.	.	0.005	.	0.530

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.040	.
MAXIMUM	.	0.001	.	0.050	.
RANGE	.	0.000	.	0.010	.
MEAN	.	0.001	.	0.045	.
STANDARD DEV	.	0.000	.	0.007	.
MEDIAN	.	0.001	.	0.045	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	708.000	.	120.000
MAXIMUM	.	.	1922.000	.	130.000
RANGE	.	.	1214.000	.	10.000
MEAN	.	.	1338.500	.	125.000
STANDARD DEV	.	.	347.428	.	7.071
MEDIAN	.	.	1278.000	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	130.000	165.000	194.000	.
MAXIMUM	140.000	165.000	194.000	.
RANGE	10.000	0.000	0.000	.
MEAN	135.000	165.000	194.000	.
STANDARD DEV	7.071	0.000	0.000	.
MEDIAN	135.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-11
YEAR = 94.000

TOTAL OBSERVATIONS: 22

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	19	2	22
MINIMUM	.	.	0.001	0.004	0.030
MAXIMUM	.	.	0.018	0.007	1.040
RANGE	.	.	0.017	0.003	1.010

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MEAN	.	.	0.005	0.006	0.395
STANDARD DEV	.	.	0.004	0.002	0.340
MEDIAN	.	.	0.004	0.006	0.340

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.160	.	.	.	.
MAXIMUM	0.160	.	.	.	.
RANGE	0.000	.	.	.	.
MEAN	0.160	.	.	.	.
STANDARD DEV	0.000	.	.	.	.
MEDIAN	0.160	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	19	2	0
MINIMUM	.	.	851.000	1077.000	.
MAXIMUM	.	.	2096.000	1117.000	.
RANGE	.	.	1245.000	40.000	.
MEAN	.	.	1688.211	1097.000	.
STANDARD DEV	.	.	384.771	28.284	.
MEDIAN	.	.	1829.000	1097.000	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	22	22	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-12
YEAR = 92.000

TOTAL OBSERVATIONS: 1

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	1	0	1
MINIMUM	.	.	0.003	.	0.370
MAXIMUM	.	.	0.003	.	0.370
RANGE	.	.	0.000	.	0.000
MEAN	.	.	0.003	.	0.370
STANDARD DEV	.	.	.	.	.
MEDIAN	.	.	0.003	.	0.370

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	1	0	1	0
MINIMUM	.	0.011	.	0.040	.
MAXIMUM	.	0.011	.	0.040	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

RANGE	.	0.000	.	0.000	.
MEAN	.	0.011	.	0.040	.
STANDARD DEV	.	.	.	.	.
MEDIAN	.	0.011	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	1	0	0
MINIMUM	.	.	2077.000	.	.
MAXIMUM	.	.	2077.000	.	.
RANGE	.	.	0.000	.	.
MEAN	.	.	2077.000	.	.
STANDARD DEV	.	.	.	.	.
MEDIAN	.	.	2077.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	1	1	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	.	.	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-12
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.002	.	0.047
MAXIMUM	.	.	0.080	.	0.690
RANGE	.	.	0.078	.	0.643
MEAN	.	.	0.010	.	0.243
STANDARD DEV	.	.	0.020	.	0.201
MEDIAN	.	.	0.005	.	0.215

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.030	.
MAXIMUM	.	0.007	.	0.050	.
RANGE	.	0.006	.	0.020	.
MEAN	.	0.004	.	0.040	.
STANDARD DEV	.	0.004	.	0.014	.
MEDIAN	.	0.004	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	763.000	.	120.000

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MAXIMUM	.	.	1556.000	.	130.000
RANGE	.	.	793.000	.	10.000
MEAN	.	.	1165.357	.	125.000
STANDARD DEV	.	.	208.534	.	7.071
MEDIAN	.	.	1138.000	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	130.000	165.000	194.000	.
MAXIMUM	140.000	165.000	194.000	.
RANGE	10.000	0.000	0.000	.
MEAN	135.000	165.000	194.000	.
STANDARD DEV	7.071	0.000	0.000	.
MEDIAN	135.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-12
YEAR = 94.000

TOTAL OBSERVATIONS: 19

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	19
MINIMUM	.	.	0.001	0.004	0.030
MAXIMUM	.	.	0.014	0.005	0.850
RANGE	.	.	0.013	0.001	0.820
MEAN	.	.	0.005	0.005	0.215
STANDARD DEV	.	.	0.004	0.001	0.253
MEDIAN	.	.	0.005	0.005	0.120

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.070	.	.	.	.
MAXIMUM	0.100	.	.	.	.
RANGE	0.030	.	.	.	.
MEAN	0.085	.	.	.	.
STANDARD DEV	0.021	.	.	.	.
MEDIAN	0.085	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	363.000	948.000	.
MAXIMUM	.	.	1807.000	959.000	.
RANGE	.	.	1444.000	11.000	.
MEAN	.	.	1276.786	953.500	.
STANDARD DEV	.	.	370.329	7.778	.
MEDIAN	.	.	1344.500	953.500	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	19	19	0

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-2
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	6.500	.	0.011	.	2.500
MAXIMUM	7.100	.	0.060	.	5.700
RANGE	0.600	.	0.049	.	3.200
MEAN	6.800	.	0.035	.	4.100
STANDARD DEV	0.424	.	0.014	.	1.055
MEDIAN	6.800	.	0.037	.	4.000

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.001	.	0.030	.
MAXIMUM	.	0.023	.	0.090	.
RANGE	.	0.022	.	0.060	.
MEAN	.	0.009	.	0.065	.
STANDARD DEV	.	0.007	.	0.021	.
MEDIAN	.	0.005	.	0.070	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	2049.000	.	.
MAXIMUM	.	.	2828.000	.	.
RANGE	.	.	779.000	.	.
MEAN	.	.	2451.818	.	.
STANDARD DEV	.	.	236.655	.	.
MEDIAN	.	.	2409.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

SITE$ = W1D-2
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.010	.	0.540
MAXIMUM	.	.	0.030	.	5.400
RANGE	.	.	0.020	.	4.860
MEAN	.	.	0.020	.	2.941
STANDARD DEV	.	.	0.005	.	1.897
MEDIAN	.	.	0.019	.	2.750

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.040	.
MAXIMUM	.	0.002	.	0.050	.
RANGE	.	0.001	.	0.010	.
MEAN	.	0.002	.	0.045	.
STANDARD DEV	.	0.001	.	0.007	.
MEDIAN	.	0.002	.	0.045	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	796.000	.	120.000
MAXIMUM	.	.	2269.000	.	130.000
RANGE	.	.	1473.000	.	10.000
MEAN	.	.	1560.071	.	125.000
STANDARD DEV	.	.	480.226	.	7.071
MEDIAN	.	.	1511.000	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	130.000	165.000	194.000	.
MAXIMUM	130.000	165.000	194.000	.
RANGE	0.000	0.000	0.000	.
MEAN	130.000	165.000	194.000	.
STANDARD DEV	0.000	0.000	0.000	.
MEDIAN	130.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-2
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	17
MINIMUM	.	.	0.016	0.014	0.900

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MAXIMUM	.	.	0.050	0.016	6.100
RANGE	.	.	0.034	0.002	5.200
MEAN	.	.	0.029	0.015	3.932
STANDARD DEV	.	.	0.007	0.001	1.604
MEDIAN	.	.	0.029	0.015	4.000

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	1.300	.	.	.	.
MAXIMUM	1.600	.	.	.	.
RANGE	0.300	.	.	.	.
MEAN	1.450	.	.	.	.
STANDARD DEV	0.212	.	.	.	.
MEDIAN	1.450	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1022.000	1525.000	.
MAXIMUM	.	.	2870.000	1781.000	.
RANGE	.	.	1848.000	256.000	.
MEAN	.	.	2009.286	1653.000	.
STANDARD DEV	.	.	444.371	181.019	.
MEDIAN	.	.	2068.000	1653.000	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-3
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.012	.	0.510
MAXIMUM	.	.	0.033	.	5.300
RANGE	.	.	0.021	.	4.790
MEAN	.	.	0.020	.	2.791
STANDARD DEV	.	.	0.006	.	1.779
MEDIAN	.	.	0.018	.	2.750

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	1	0	2	0

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MINIMUM	.	0.003	.	0.040	.
MAXIMUM	.	0.003	.	0.060	.
RANGE	.	0.000	.	0.020	.
MEAN	.	0.003	.	0.050	.
STANDARD DEV	.	.	.	0.014	.
MEDIAN	.	0.003	.	0.050	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	801.000	.	120.000
MAXIMUM	.	.	2364.000	.	130.000
RANGE	.	.	1563.000	.	10.000
MEAN	.	.	1612.143	.	125.000
STANDARD DEV	.	.	504.415	.	7.071
MEDIAN	.	.	1535.000	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	130.000	165.000	194.000	.
MAXIMUM	130.000	165.000	194.000	.
RANGE	0.000	0.000	0.000	.
MEAN	130.000	165.000	194.000	.
STANDARD DEV	0.000	0.000	0.000	.
MEDIAN	130.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-3
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	17
MINIMUM	.	.	0.017	0.007	0.400
MAXIMUM	.	.	0.040	0.016	5.300
RANGE	.	.	0.023	0.009	4.900
MEAN	.	.	0.024	0.012	3.495
STANDARD DEV	.	.	0.006	0.006	1.480
MEDIAN	.	.	0.024	0.012	3.600

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.360	.	.	.	.
MAXIMUM	1.300	.	.	.	.
RANGE	0.940	.	.	.	.
MEAN	0.830	.	.	.	.
STANDARD DEV	0.665	.	.	.	.
MEDIAN	0.830	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

N OF CASES	0	0	14	2	0
MINIMUM	.	.	1304.000	1415.000	.
MAXIMUM	.	.	2557.000	1628.000	.
RANGE	.	.	1253.000	213.000	.
MEAN	.	.	1962.357	1521.500	.
STANDARD DEV	.	.	361.960	150.614	.
MEDIAN	.	.	1968.000	1521.500	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-4
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	6.800	.	0.007	.	1.900
MAXIMUM	7.300	.	0.050	.	4.500
RANGE	0.500	.	0.043	.	2.600
MEAN	7.050	.	0.024	.	3.345
STANDARD DEV	0.354	.	0.015	.	0.857
MEDIAN	7.050	.	0.018	.	3.600

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.001	.	0.030	.
MAXIMUM	.	0.034	.	0.090	.
RANGE	.	0.033	.	0.060	.
MEAN	.	0.010	.	0.055	.
STANDARD DEV	.	0.009	.	0.018	.
MEDIAN	.	0.010	.	0.050	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	2078.000	.	.
MAXIMUM	.	.	2686.000	.	.
RANGE	.	.	608.000	.	.
MEAN	.	.	2404.364	.	.
STANDARD DEV	.	.	210.282	.	.
MEDIAN	.	.	2395.000	.	.

	MGT	TOTHARD	ALK	FLOWLM

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-4
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	-0.001	.	0.390
MAXIMUM	.	.	0.030	.	5.200
RANGE	.	.	0.031	.	4.810
MEAN	.	.	0.017	.	2.691
STANDARD DEV	.	.	0.007	.	1.781
MEDIAN	.	.	0.016	.	2.900

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.040	.
MAXIMUM	.	0.002	.	0.050	.
RANGE	.	0.002	.	0.010	.
MEAN	.	0.001	.	0.045	.
STANDARD DEV	.	0.001	.	0.007	.
MEDIAN	.	0.001	.	0.045	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	741.000	.	120.000
MAXIMUM	.	.	2289.000	.	130.000
RANGE	.	.	1548.000	.	10.000
MEAN	.	.	1560.214	.	125.000
STANDARD DEV	.	.	481.412	.	7.071
MEDIAN	.	.	1508.500	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	120.000	165.000	194.000	.
MAXIMUM	130.000	165.000	194.000	.
RANGE	10.000	0.000	0.000	.
MEAN	125.000	165.000	194.000	.
STANDARD DEV	7.071	0.000	0.000	.
MEDIAN	125.000	165.000	194.000	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-4
YEAR = 94.000

TOTAL OBSERVATIONS: 19

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	18
MINIMUM	.	.	0.002	0.003	0.060
MAXIMUM	.	.	0.023	0.009	4.700
RANGE	.	.	0.021	0.006	4.640
MEAN	.	.	0.016	0.006	2.685
STANDARD DEV	.	.	0.005	0.004	1.662
MEDIAN	.	.	0.016	0.006	2.900

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.080	.	.	.	.
MAXIMUM	1.100	.	.	.	.
RANGE	1.020	.	.	.	.
MEAN	0.590	.	.	.	.
STANDARD DEV	0.721	.	.	.	.
MEDIAN	0.590	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1008.000	1222.000	.
MAXIMUM	.	.	2569.000	1408.000	.
RANGE	.	.	1561.000	186.000	.
MEAN	.	.	1874.214	1315.000	.
STANDARD DEV	.	.	459.528	131.522	.
MEDIAN	.	.	2002.500	1315.000	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	19	19	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-5
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

N OF CASES	2	0	11	0	11
MINIMUM	6.500	.	0.003	.	1.600
MAXIMUM	7.300	.	0.051	.	3.200
RANGE	0.800	.	0.048	.	1.600
MEAN	6.900	.	0.015	.	2.255
STANDARD DEV	0.566	.	0.015	.	0.545
MEDIAN	6.900	.	0.008	.	2.100

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.003	.	0.020	.
MAXIMUM	.	0.015	.	0.090	.
RANGE	.	0.012	.	0.070	.
MEAN	.	0.009	.	0.053	.
STANDARD DEV	.	0.004	.	0.021	.
MEDIAN	.	0.009	.	0.050	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	2128.000	.	.
MAXIMUM	.	.	2770.000	.	.
RANGE	.	.	642.000	.	.
MEAN	.	.	2407.636	.	.
STANDARD DEV	.	.	238.491	.	.
MEDIAN	.	.	2461.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-5
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	-0.001	.	0.230
MAXIMUM	.	.	0.011	.	3.700
RANGE	.	.	0.012	.	3.470
MEAN	.	.	0.007	.	1.592
STANDARD DEV	.	.	0.003	.	1.139
MEDIAN	.	.	0.007	.	1.800

	NIF	COT	COF	ZNT	ZNF

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.060	.
MAXIMUM	.	0.002	.	0.060	.
RANGE	.	0.002	.	0.000	.
MEAN	.	0.001	.	0.060	.
STANDARD DEV	.	0.001	.	0.000	.
MEDIAN	.	0.001	.	0.060	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	752.000	.	120.000
MAXIMUM	.	.	2237.000	.	130.000
RANGE	.	.	1485.000	.	10.000
MEAN	.	.	1466.429	.	125.000
STANDARD DEV	.	.	402.774	.	7.071
MEDIAN	.	.	1366.500	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	130.000	165.000	194.000	.
MAXIMUM	140.000	165.000	194.000	.
RANGE	10.000	0.000	0.000	.
MEAN	135.000	165.000	194.000	.
STANDARD DEV	7.071	0.000	0.000	.
MEDIAN	135.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-5
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	17
MINIMUM	.	.	0.003	0.003	0.170
MAXIMUM	.	.	0.069	0.009	2.300
RANGE	.	.	0.066	0.006	2.130
MEAN	.	.	0.013	0.006	1.532
STANDARD DEV	.	.	0.016	0.004	0.628
MEDIAN	.	.	0.010	0.006	1.800

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.150	.	.	.	.
MAXIMUM	0.660	.	.	.	.
RANGE	0.510	.	.	.	.
MEAN	0.405	.	.	.	.
STANDARD DEV	0.361	.	.	.	.
MEDIAN	0.405	.	.	.	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1121.000	1292.000	.
MAXIMUM	.	.	2967.000	1403.000	.
RANGE	.	.	1846.000	111.000	.
MEAN	.	.	1843.357	1347.500	.
STANDARD DEV	.	.	463.534	78.489	.
MEDIAN	.	.	1880.000	1347.500	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-6
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	6.300	.	0.001	.	0.140
MAXIMUM	6.400	.	0.035	.	7.300
RANGE	0.100	.	0.034	.	7.160
MEAN	6.350	.	0.008	.	1.716
STANDARD DEV	0.071	.	0.010	.	1.922
MEDIAN	6.350	.	0.005	.	1.300

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.002	.	0.005	.
MAXIMUM	.	0.009	.	0.060	.
RANGE	.	0.007	.	0.055	.
MEAN	.	0.006	.	0.041	.
STANDARD DEV	.	0.002	.	0.022	.
MEDIAN	.	0.005	.	0.050	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	2000.000	.	.
MAXIMUM	.	.	2669.000	.	.
RANGE	.	.	669.000	.	.
MEAN	.	.	2337.000	.	.
STANDARD DEV	.	.	230.618	.	.
MEDIAN	.	.	2283.000	.	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-6
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.004	.	0.170
MAXIMUM	.	.	0.008	.	2.100
RANGE	.	.	0.004	.	1.930
MEAN	.	.	0.006	.	0.980
STANDARD DEV	.	.	0.002	.	0.721
MEDIAN	.	.	0.007	.	1.100

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.040	.
MAXIMUM	.	0.002	.	0.060	.
RANGE	.	0.002	.	0.020	.
MEAN	.	0.001	.	0.050	.
STANDARD DEV	.	0.001	.	0.014	.
MEDIAN	.	0.001	.	0.050	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	779.000	.	120.000
MAXIMUM	.	.	2058.000	.	130.000
RANGE	.	.	1279.000	.	10.000
MEAN	.	.	1409.143	.	125.000
STANDARD DEV	.	.	365.302	.	7.071
MEDIAN	.	.	1345.000	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	140.000	165.000	194.000	.
MAXIMUM	140.000	165.000	194.000	.
RANGE	0.000	0.000	0.000	.
MEAN	140.000	165.000	194.000	.
STANDARD DEV	0.000	0.000	0.000	.
MEDIAN	140.000	165.000	194.000	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-6
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	17
MINIMUM	.	.	0.003	0.001	0.140
MAXIMUM	.	.	0.025	0.005	2.000
RANGE	.	.	0.022	0.004	1.860
MEAN	.	.	0.008	0.003	1.170
STANDARD DEV	.	.	0.006	0.003	0.640
MEDIAN	.	.	0.007	0.003	1.200

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.140	.	.	.	.
MAXIMUM	0.310	.	.	.	.
RANGE	0.170	.	.	.	.
MEAN	0.225	.	.	.	.
STANDARD DEV	0.120	.	.	.	.
MEDIAN	0.225	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1155.000	1184.000	.
MAXIMUM	.	.	2318.000	1309.000	.
RANGE	.	.	1163.000	125.000	.
MEAN	.	.	1772.643	1246.500	.
STANDARD DEV	.	.	377.547	88.388	.
MEDIAN	.	.	1848.000	1246.500	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-7
YEAR = 92.000

TOTAL OBSERVATIONS: 11

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	6.600	.	0.000	.	0.140
MAXIMUM	6.600	.	0.009	.	1.800
RANGE	0.000	.	0.009	.	1.660
MEAN	6.600	.	0.004	.	0.874
STANDARD DEV	0.000	.	0.002	.	0.469
MEDIAN	6.600	.	0.004	.	0.800

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.000	.	0.005	.
MAXIMUM	.	0.011	.	0.070	.
RANGE	.	0.011	.	0.065	.
MEAN	.	0.005	.	0.039	.
STANDARD DEV	.	0.003	.	0.023	.
MEDIAN	.	0.003	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	1960.000	.	.
MAXIMUM	.	.	2669.000	.	.
RANGE	.	.	709.000	.	.
MEAN	.	.	2308.545	.	.
STANDARD DEV	.	.	199.233	.	.
MEDIAN	.	.	2299.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-7
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	12	0	14	0	14
MINIMUM	6.800	.	-0.001	.	0.060
MAXIMUM	7.200	.	0.008	.	1.600
RANGE	0.400	.	0.009	.	1.540
MEAN	6.975	.	0.004	.	0.695
STANDARD DEV	0.166	.	0.002	.	0.595
MEDIAN	6.950	.	0.004	.	0.640

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.040	.
MAXIMUM	.	0.002	.	0.070	.
RANGE	.	0.002	.	0.030	.
MEAN	.	0.001	.	0.055	.
STANDARD DEV	.	0.001	.	0.021	.
MEDIAN	.	0.001	.	0.055	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	790.000	.	130.000
MAXIMUM	.	.	1880.000	.	130.000
RANGE	.	.	1090.000	.	0.000
MEAN	.	.	1347.857	.	130.000
STANDARD DEV	.	.	325.726	.	0.000
MEDIAN	.	.	1278.500	.	130.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	140.000	165.000	194.000	.
MAXIMUM	140.000	165.000	194.000	.
RANGE	0.000	0.000	0.000	.
MEAN	140.000	165.000	194.000	.
STANDARD DEV	0.000	0.000	0.000	.
MEDIAN	140.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-7
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	12	0	14	2	17
MINIMUM	7.100	.	0.002	0.003	0.120
MAXIMUM	7.500	.	0.009	0.003	1.600
RANGE	0.400	.	0.007	0.000	1.480
MEAN	7.317	.	0.006	0.003	0.889
STANDARD DEV	0.127	.	0.002	0.000	0.541
MEDIAN	7.300	.	0.007	0.003	1.100

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.140	.	.	.	.
MAXIMUM	0.200	.	.	.	.
RANGE	0.060	.	.	.	.
MEAN	0.170	.	.	.	.
STANDARD DEV	0.042	.	.	.	.
MEDIAN	0.170	.	.	.	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1121.000	1171.000	.
MAXIMUM	.	.	2272.000	1229.000	.
RANGE	.	.	1151.000	58.000	.
MEAN	.	.	1744.286	1200.000	.
STANDARD DEV	.	.	370.630	41.012	.
MEDIAN	.	.	1854.000	1200.000	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-8
YEAR = 92.000

TOTAL OBSERVATIONS: 11

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	6.700	.	0.001	.	0.140
MAXIMUM	6.800	.	0.014	.	1.400
RANGE	0.100	.	0.013	.	1.260
MEAN	6.750	.	0.006	.	0.741
STANDARD DEV	0.071	.	0.004	.	0.359
MEDIAN	6.750	.	0.006	.	0.760

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.001	.	0.005	.
MAXIMUM	.	0.012	.	0.090	.
RANGE	.	0.011	.	0.085	.
MEAN	.	0.005	.	0.042	.
STANDARD DEV	.	0.004	.	0.028	.
MEDIAN	.	0.003	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	1953.000	.	.
MAXIMUM	.	.	2661.000	.	.
RANGE	.	.	708.000	.	.
MEAN	.	.	2268.909	.	.
STANDARD DEV	.	.	221.499	.	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MEDIAN	.	.	2246.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITES = W1D-8
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.002	.	0.090
MAXIMUM	.	.	0.006	.	1.400
RANGE	.	.	0.004	.	1.310
MEAN	.	.	0.004	.	0.637
STANDARD DEV	.	.	0.001	.	0.534
MEDIAN	.	.	0.004	.	0.580

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.040	.
MAXIMUM	.	0.001	.	0.090	.
RANGE	.	0.000	.	0.050	.
MEAN	.	0.001	.	0.065	.
STANDARD DEV	.	0.000	.	0.035	.
MEDIAN	.	0.001	.	0.065	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	730.000	.	120.000
MAXIMUM	.	.	1922.000	.	130.000
RANGE	.	.	1192.000	.	10.000
MEAN	.	.	1296.929	.	125.000
STANDARD DEV	.	.	321.395	.	7.071
MEDIAN	.	.	1221.000	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	140.000	165.000	194.000	.
MAXIMUM	140.000	165.000	194.000	.
RANGE	0.000	0.000	0.000	.
MEAN	140.000	165.000	194.000	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

STANDARD DEV	0.000	0.000	0.000	.
MEDIAN	140.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-8
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	17
MINIMUM	.	.	0.002	0.003	0.180
MAXIMUM	.	.	0.024	0.004	1.600
RANGE	.	.	0.022	0.001	1.420
MEAN	.	.	0.006	0.004	0.784
STANDARD DEV	.	.	0.006	0.001	0.485
MEDIAN	.	.	0.006	0.004	0.930

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.160	.	.	.	.
MAXIMUM	0.200	.	.	.	.
RANGE	0.040	.	.	.	.
MEAN	0.180	.	.	.	.
STANDARD DEV	0.028	.	.	.	.
MEDIAN	0.180	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1114.000	1114.000	.
MAXIMUM	.	.	2123.000	1249.000	.
RANGE	.	.	1009.000	135.000	.
MEAN	.	.	1714.714	1181.500	.
STANDARD DEV	.	.	339.042	95.459	.
MEDIAN	.	.	1794.000	1181.500	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-9
YEAR = 92.000

TOTAL OBSERVATIONS: 11

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

	PH	SC	CUT	CUF	NIT
N OF CASES	2	0	11	0	11
MINIMUM	6.700	.	0.001	.	0.130
MAXIMUM	7.100	.	0.020	.	1.100
RANGE	0.400	.	0.019	.	0.970
MEAN	6.900	.	0.008	.	0.645
STANDARD DEV	0.283	.	0.007	.	0.333
MEDIAN	6.900	.	0.005	.	0.560

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	11	0	11	0
MINIMUM	.	0.000	.	0.005	.
MAXIMUM	.	0.015	.	0.060	.
RANGE	.	0.015	.	0.055	.
MEAN	.	0.005	.	0.039	.
STANDARD DEV	.	0.005	.	0.023	.
MEDIAN	.	0.005	.	0.040	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	11	0	0
MINIMUM	.	.	1983.000	.	.
MAXIMUM	.	.	2575.000	.	.
RANGE	.	.	592.000	.	.
MEAN	.	.	2261.000	.	.
STANDARD DEV	.	.	215.911	.	.
MEDIAN	.	.	2255.000	.	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	11	11	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-9
YEAR = 93.000

TOTAL OBSERVATIONS: 14

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	0	14
MINIMUM	.	.	0.003	.	0.060
MAXIMUM	.	.	0.008	.	1.500
RANGE	.	.	0.005	.	1.440
MEAN	.	.	0.005	.	0.599
STANDARD DEV	.	.	0.002	.	0.523

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

MEDIAN	.	.	0.006	.	0.575

	NIF	COT	COF	ZNT	ZNF
N OF CASES	0	2	0	2	0
MINIMUM	.	0.001	.	0.040	.
MAXIMUM	.	0.001	.	0.050	.
RANGE	.	0.000	.	0.010	.
MEAN	.	0.001	.	0.045	.
STANDARD DEV	.	0.000	.	0.007	.
MEDIAN	.	0.001	.	0.045	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	0	2
MINIMUM	.	.	736.000	.	120.000
MAXIMUM	.	.	2005.000	.	130.000
RANGE	.	.	1269.000	.	10.000
MEAN	.	.	1330.643	.	125.000
STANDARD DEV	.	.	346.237	.	7.071
MEDIAN	.	.	1263.000	.	125.000

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	2	14	14	0
MINIMUM	130.000	165.000	194.000	.
MAXIMUM	140.000	165.000	194.000	.
RANGE	10.000	0.000	0.000	.
MEAN	135.000	165.000	194.000	.
STANDARD DEV	7.071	0.000	0.000	.
MEDIAN	135.000	165.000	194.000	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D-9
YEAR = 94.000

TOTAL OBSERVATIONS: 17

	PH	SC	CUT	CUF	NIT
N OF CASES	0	0	14	2	17
MINIMUM	.	.	0.002	0.004	0.100
MAXIMUM	.	.	0.015	0.004	1.400
RANGE	.	.	0.013	0.000	1.300
MEAN	.	.	0.006	0.004	0.728
STANDARD DEV	.	.	0.003	0.000	0.453
MEDIAN	.	.	0.006	0.004	0.900

	NIF	COT	COF	ZNT	ZNF
N OF CASES	2	0	0	0	0
MINIMUM	0.110	.	.	.	.
MAXIMUM	0.160	.	.	.	.
RANGE	0.050	.	.	.	.
MEAN	0.135	.	.	.	.

Table A2.3. Summary statistics of water quality data from W1D sites other than 051 and 052 (continued).

STANDARD DEV	0.035	.	.	.	.
MEDIAN	0.135	.	.	.	.

	FEF	TSS	SO4T	SO4F	CAT
N OF CASES	0	0	14	2	0
MINIMUM	.	.	1177.000	1177.000	.
MAXIMUM	.	.	2143.000	1262.000	.
RANGE	.	.	966.000	85.000	.
MEAN	.	.	1716.571	1219.500	.
STANDARD DEV	.	.	340.578	60.104	.
MEDIAN	.	.	1798.000	1219.500	.

	MGT	TOTHARD	ALK	FLOWLM
N OF CASES	0	17	17	0
MINIMUM	.	165.000	194.000	.
MAXIMUM	.	165.000	194.000	.
RANGE	.	0.000	0.000	.
MEAN	.	165.000	194.000	.
STANDARD DEV	.	0.000	0.000	.
MEDIAN	.	165.000	194.000	.

Note: The parameters with a T suffix are total values, parameters with an F suffix are filtered values. TOTHARD is total hardness, FLOWLM is liters/min, and values preceded with a hyphen are less than the detection limit. pH values are in standard units, specific conductance (SC) values are in microsiemens, flow (FLOWLM) values are liters/min, and all other parameters are mg/L.

Table A2.4 Summary statistics of water quality data from the DNR study cell of the W1D treatment system.

Input (DNR Site# W1D-1)

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D1

TOTAL OBSERVATIONS: 25

	SC	PH	CU	NI	CO
N OF CASES	24	23	23	23	16
MINIMUM	900.000	6.670	0.001	0.100	0.001
MAXIMUM	2275.000	7.840	0.047	0.933	0.050
RANGE	1375.000	1.170	0.046	0.833	0.049
MEAN	1649.583	7.157	0.015	0.324	0.017
STANDARD DEV	392.087	0.275	0.012	0.194	0.022
MEDIAN	1600.000	7.110	0.008	0.300	0.003

	ZN	SO4	CA	MG
N OF CASES	10	20	14	14
MINIMUM	0.020	441.100	140.000	121.000
MAXIMUM	0.081	1179.000	247.000	249.000
RANGE	0.061	737.900	107.000	128.000
MEAN	0.035	879.355	198.214	188.500
STANDARD DEV	0.017	241.999	31.981	41.398
MEDIAN	0.030	888.500	209.000	190.000

Output (DNR Site# W1D-2)

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D2

TOTAL OBSERVATIONS: 23

	SC	PH	CU	NI	CO
N OF CASES	22	22	21	21	14
MINIMUM	1000.000	6.860	0.001	0.050	0.001
MAXIMUM	2100.000	7.770	0.035	0.875	0.050
RANGE	1100.000	0.910	0.035	0.825	0.050
MEAN	1557.955	7.250	0.012	0.257	0.011
STANDARD DEV	324.164	0.188	0.012	0.186	0.018
MEDIAN	1500.000	7.250	0.004	0.210	0.001

Table A2.4 Summary statistics of water quality data from the DNR study cell of the W1D treatment system.

	ZN	SO4	CA	MG
N OF CASES	9	18	12	12
MINIMUM	0.010	506.100	134.000	115.000
MAXIMUM	0.064	1172.000	244.000	249.000
RANGE	0.054	665.900	110.000	134.000
MEAN	0.030	822.728	185.667	181.000
STANDARD DEV	0.016	213.440	35.605	42.980
MEDIAN	0.023	799.500	195.000	186.500

Output (DNR Site# W1D-3)

THE FOLLOWING RESULTS ARE FOR:
SITE$ = W1D3

TOTAL OBSERVATIONS: 23

	SC	PH	CU	NI	CO
N OF CASES	22	22	20	20	14
MINIMUM	900.000	6.860	0.001	0.001	0.001
MAXIMUM	2200.000	7.500	0.029	0.793	0.050
RANGE	1300.000	0.640	0.029	0.792	0.050
MEAN	1551.136	7.226	0.011	0.269	0.010
STANDARD DEV	345.677	0.151	0.011	0.174	0.018
MEDIAN	1500.000	7.225	0.004	0.295	0.001

	ZN	SO4	CA	MG
N OF CASES	8	17	11	11
MINIMUM	0.019	499.000	130.000	116.000
MAXIMUM	0.100	1068.000	238.000	246.000
RANGE	0.081	569.000	108.000	130.000
MEAN	0.040	780.382	181.364	177.182
STANDARD DEV	0.028	184.110	36.269	42.951
MEDIAN	0.030	793.000	190.000	182.000

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

Note: The site called Seep 1 (043) was originally the actual Seep 1 outfall, but a pretreatment system was started in 1992, and starting in August 1992 the data shown for 043 are actually for the effluent from the limestone pretreatment system.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 76.000

TOTAL OBSERVATIONS: 9

	pH	ALK	SC	SO4	CUT
N OF CASES	9	9	9	9	9
MINIMUM	7.000	161.000	2324.000	1839.000	0.004
MAXIMUM	7.500	229.000	4600.000	3000.000	0.091
MEAN	7.289	201.889	3545.000	2404.444	0.029
STANDARD DEV	0.190	22.784	628.336	367.369	0.027

	NIT	COT	ZNT	FEF	CA
N OF CASES	9	4	8	9	7
MINIMUM	0.670	0.110	0.100	0.700	194.000
MAXIMUM	1.920	0.132	0.690	7.200	396.000
MEAN	1.143	0.124	0.254	4.744	252.714
STANDARD DEV	0.379	0.010	0.190	2.091	65.711

	MG	FLOWMGD
N OF CASES	3	0
MINIMUM	451.000	.
MAXIMUM	528.000	.
MEAN	481.000	.
STANDARD DEV	41.219	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 77.000

TOTAL OBSERVATIONS: 17

	pH	ALK	SC	SO4	CUT

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

N OF CASES	17	17	17	11	17
MINIMUM	6.500	46.000	500.000	149.000	0.003
MAXIMUM	7.400	441.000	7700.000	5636.000	0.329
MEAN	6.812	104.706	3105.941	2552.000	0.034
STANDARD DEV	0.226	91.436	2473.321	2007.647	0.077

	NIT	COT	ZNT	FEF	CA
N OF CASES	17	5	5	15	8
MINIMUM	0.053	0.110	0.040	0.600	46.000
MAXIMUM	12.000	0.870	2.400	8.700	346.000
MEAN	3.471	0.478	1.290	2.320	175.625
STANDARD DEV	3.867	0.340	1.169	2.493	116.402

	MG	FLOWMGD
N OF CASES	7	0
MINIMUM	119.000	.
MAXIMUM	652.000	.
MEAN	307.714	.
STANDARD DEV	223.442	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 78.000

TOTAL OBSERVATIONS: 10

	pH	ALK	SC	SO4	CUT
N OF CASES	10	0	10	10	10
MINIMUM	6.300	.	5000.000	3275.000	0.034
MAXIMUM	7.500	.	5000.000	5000.000	0.119
MEAN	6.660	.	5000.000	4315.000	0.063
STANDARD DEV	0.327	.	0.000	584.190	0.030

	NIT	COT	ZNT	FEF	CA
N OF CASES	10	10	10	0	10
MINIMUM	2.800	0.300	0.180	.	38.000
MAXIMUM	12.500	1.000	2.300	.	350.000
MEAN	8.270	0.752	1.420	.	268.800
STANDARD DEV	2.699	0.216	0.600	.	85.706

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	MG	FLOWMGD
N OF CASES	10	0
MINIMUM	400.000	.
MAXIMUM	1100.000	.
MEAN	728.000	.
STANDARD DEV	230.063	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 79.000

TOTAL OBSERVATIONS: 16

	pH	ALK	SC	SO4	CUT
N OF CASES	16	16	15	16	16
MINIMUM	4.500	-2.000	1500.000	1100.000	0.020
MAXIMUM	6.900	80.000	5000.000	10000.000	0.300
MEAN	6.106	38.500	4720.000	4281.250	0.106
STANDARD DEV	0.772	28.636	904.118	1920.840	0.070

	NIT	COT	ZNT	FEF	CA
N OF CASES	16	0	16	16	16
MINIMUM	1.500	.	0.250	0.400	-1.000
MAXIMUM	32.000	.	17.000	8.500	640.000
MEAN	11.231	.	3.348	2.219	136.313
STANDARD DEV	7.841	.	4.153	2.075	216.538

	MG	FLOWMGD
N OF CASES	16	0
MINIMUM	1.000	.
MAXIMUM	9999.000	.
MEAN	1555.438	.
STANDARD DEV	2320.214	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 80.000

TOTAL OBSERVATIONS: 18

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	pH	ALK	SC	SO4	CUT
N OF CASES	14	15	15	15	17
MINIMUM	6.200	28.000	2050.000	2500.000	-0.010
MAXIMUM	8.300	68.000	4300.000	3700.000	0.400
MEAN	7.107	49.733	3866.667	3166.667	0.075
STANDARD DEV	0.763	11.997	594.218	359.894	0.089

	NIT	COT	ZNT	FEF	CA
N OF CASES	17	2	17	16	17
MINIMUM	2.200	0.690	0.580	0.200	200.000
MAXIMUM	14.000	0.900	2.300	3.800	410.000
MEAN	9.669	0.795	1.118	0.731	308.706
STANDARD DEV	3.011	0.148	0.426	0.870	57.250

	MG	FLOWMGD
N OF CASES	17	0
MINIMUM	340.000	.
MAXIMUM	763.000	.
MEAN	599.706	.
STANDARD DEV	117.811	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 81.000

TOTAL OBSERVATIONS: 18

	pH	ALK	SC	SO4	CUT
N OF CASES	16	18	17	18	18
MINIMUM	6.000	14.000	1650.000	440.000	0.060
MAXIMUM	7.300	96.000	5000.000	4500.000	0.300
MEAN	6.550	32.333	4406.471	3263.333	0.144
STANDARD DEV	0.327	18.458	995.314	1378.733	0.058

	NIT	COT	ZNT	FEF	CA
N OF CASES	18	15	17	2	18
MINIMUM	2.600	0.050	0.540	0.300	140.000
MAXIMUM	26.000	2.300	5.000	0.400	380.000
MEAN	14.506	0.995	3.205	0.350	302.222
STANDARD DEV	5.491	0.458	1.521	0.071	63.483

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	MG	FLOWMGD
N OF CASES	18	0
MINIMUM	180.000	.
MAXIMUM	860.000	.
MEAN	695.556	.
STANDARD DEV	177.175	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 82.000

TOTAL OBSERVATIONS: 18

	pH	ALK	SC	SO4	CUT
N OF CASES	18	0	16	18	18
MINIMUM	5.700	.	750.000	360.000	0.014
MAXIMUM	7.400	.	5000.000	4504.000	0.350
MEAN	6.494	.	3421.875	2790.222	0.142
STANDARD DEV	0.505	.	1727.519	1547.159	0.106

	NIT	COT	ZNT	FEF	CA
N OF CASES	18	18	18	0	18
MINIMUM	1.100	-0.100	0.160	.	45.000
MAXIMUM	66.000	2.400	6.300	.	460.000
MEAN	15.306	0.831	2.949	.	267.222
STANDARD DEV	14.659	0.617	1.824	.	138.810

	MG	FLOWMGD
N OF CASES	18	0
MINIMUM	62.000	.
MAXIMUM	960.000	.
MEAN	524.556	.
STANDARD DEV	298.170	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 83.000

TOTAL OBSERVATIONS: 11

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	pH	ALK	SC	SO4	CUT
N OF CASES	10	0	11	11	11
MINIMUM	5.200	.	1550.000	1100.000	0.016
MAXIMUM	6.800	.	4600.000	3700.000	1.100
MEAN	5.660	.	3468.182	2686.364	0.469
STANDARD DEV	0.470	.	981.586	831.893	0.289

	NIT	COT	ZNT	FEF	CA
N OF CASES	11	11	11	8	11
MINIMUM	3.400	0.540	0.840	0.200	110.000
MAXIMUM	32.000	2.400	6.200	1.600	360.000
MEAN	18.127	1.325	3.485	0.675	260.273
STANDARD DEV	8.573	0.684	1.492	0.585	86.433

	MG	FLOWMGD
N OF CASES	11	0
MINIMUM	190.000	.
MAXIMUM	710.000	.
MEAN	459.545	.
STANDARD DEV	146.739	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 84.000

TOTAL OBSERVATIONS: 14

	pH	ALK	SC	SO4	CUT
N OF CASES	14	0	14	14	14
MINIMUM	5.100	.	775.000	700.000	0.050
MAXIMUM	5.700	.	3100.000	3100.000	0.820
MEAN	5.471	.	2187.500	1597.143	0.387
STANDARD DEV	0.220	.	840.830	784.341	0.263

	NIT	COT	ZNT	FEF	CA
N OF CASES	14	14	14	13	14
MINIMUM	3.400	0.300	0.780	-0.100	9.000
MAXIMUM	34.000	2.500	3.600	0.600	340.000
MEAN	18.514	1.330	2.141	0.223	184.643
STANDARD DEV	10.469	0.775	0.917	0.183	95.477

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	MG	FLOWMGD
N OF CASES	14	0
MINIMUM	84.000	.
MAXIMUM	450.000	.
MEAN	282.857	.
STANDARD DEV	128.604	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 85.000

TOTAL OBSERVATIONS: 15

	pH	ALK	SC	SO4	CUT
N OF CASES	15	0	15	15	15
MINIMUM	4.900	.	385.000	200.000	0.038
MAXIMUM	5.900	.	2700.000	2000.000	0.810
MEAN	5.360	.	1999.000	1356.667	0.393
STANDARD DEV	0.253	.	620.381	413.118	0.226

	NIT	COT	ZNT	FEF	CA
N OF CASES	15	15	15	15	15
MINIMUM	1.200	0.120	0.300	0.100	28.000
MAXIMUM	32.000	2.100	2.400	1.100	240.000
MEAN	15.747	1.101	1.451	0.440	173.200
STANDARD DEV	7.877	0.510	0.505	0.329	52.311

	MG	FLOWMGD
N OF CASES	15	0
MINIMUM	30.000	.
MAXIMUM	380.000	.
MEAN	226.667	.
STANDARD DEV	88.694	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 86.000

TOTAL OBSERVATIONS: 15

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	pH	ALK	SC	SO4	CUT
N OF CASES	15	0	15	15	15
MINIMUM	5.100	.	960.000	60.000	0.070
MAXIMUM	5.800	.	2550.000	1800.000	1.100
MEAN	5.440	.	1930.667	1198.000	0.529
STANDARD DEV	0.213	.	491.771	432.488	0.350

	NIT	COT	ZNT	FEF	CA
N OF CASES	15	15	15	15	15
MINIMUM	2.700	0.300	0.500	0.200	75.000
MAXIMUM	30.000	2.100	3.000	1.100	220.000
MEAN	14.493	1.045	1.361	0.400	152.267
STANDARD DEV	8.493	0.598	0.723	0.245	50.286

	MG	FLOWMGD
N OF CASES	15	0
MINIMUM	80.000	.
MAXIMUM	300.000	.
MEAN	203.867	.
STANDARD DEV	69.395	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 87.000

TOTAL OBSERVATIONS: 15

	pH	ALK	SC	SO4	CUT
N OF CASES	15	0	15	15	15
MINIMUM	4.800	.	1225.000	700.000	0.090
MAXIMUM	6.200	.	2900.000	2650.000	1.100
MEAN	5.473	.	2263.333	1502.333	0.449
STANDARD DEV	0.339	.	402.987	433.803	0.379

	NIT	COT	ZNT	FEF	CA
N OF CASES	15	15	15	15	0
MINIMUM	2.500	0.200	0.330	0.200	.
MAXIMUM	32.000	2.800	2.800	2.900	.
MEAN	14.827	1.216	1.133	0.640	.
STANDARD DEV	9.387	0.657	0.741	0.729	.

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	MG	FLOWMGD
N OF CASES	0	0
MINIMUM	.	.
MAXIMUM	.	.
MEAN	.	.
STANDARD DEV	.	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 88.000

TOTAL OBSERVATIONS: 17

	pH	ALK	SC	SO4	CUT
N OF CASES	17	1	17	17	17
MINIMUM	5.200	3.000	625.000	340.000	0.017
MAXIMUM	7.600	3.000	2925.000	2100.000	0.810
MEAN	6.088	3.000	2050.882	1330.882	0.332
STANDARD DEV	0.882	.	642.770	492.079	0.271

	NIT	COT	ZNT	FEF	CA
N OF CASES	17	17	17	17	0
MINIMUM	1.000	0.090	0.250	-0.100	.
MAXIMUM	32.000	2.500	2.600	1.000	.
MEAN	14.324	1.114	1.268	0.211	.
STANDARD DEV	9.548	0.712	0.686	0.258	.

	MG	FLOWMGD
N OF CASES	0	0
MINIMUM	.	.
MAXIMUM	.	.
MEAN	.	.
STANDARD DEV	.	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 89.000

TOTAL OBSERVATIONS: 14

	pH	ALK	SC	SO4	CUT

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

N OF CASES	14	0	14	14	14
MINIMUM	4.800	.	640.000	430.000	0.060
MAXIMUM	5.900	.	2525.000	1725.000	1.800
MEAN	5.193	.	1894.286	1271.071	0.922
STANDARD DEV	0.354	.	576.264	417.079	0.593

	NIT	COT	ZNT	FEF	CA
N OF CASES	14	14	14	14	0
MINIMUM	1.600	0.160	0.250	0.100	.
MAXIMUM	26.000	1.900	3.000	0.500	.
MEAN	14.971	1.057	1.673	0.144	.
STANDARD DEV	7.923	0.574	0.773	0.115	.

	MG	FLOWMGD
N OF CASES	0	0
MINIMUM	.	.
MAXIMUM	.	.
MEAN	.	.
STANDARD DEV	.	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 90.000

TOTAL OBSERVATIONS: 13

	pH	ALK	SC	SO4	CUT
N OF CASES	13	0	13	13	13
MINIMUM	4.700	.	720.000	425.000	0.150
MAXIMUM	5.400	.	2750.000	1700.000	2.100
MEAN	5.031	.	2053.462	1182.308	1.177
STANDARD DEV	0.210	.	621.079	391.384	0.550

	NIT	COT	ZNT	FEF	CA
N OF CASES	13	13	13	13	0
MINIMUM	2.400	0.220	0.390	0.100	.
MAXIMUM	31.000	1.800	2.900	0.300	.
MEAN	15.389	1.112	1.590	0.169	.
STANDARD DEV	7.161	0.458	0.754	0.063	.

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	MG	FLOWMGD
N OF CASES	0	0
MINIMUM	.	.
MAXIMUM	.	.
MEAN	.	.
STANDARD DEV	.	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 91.000

TOTAL OBSERVATIONS: 15

	pH	ALK	SC	SO4	CUT
N OF CASES	15	0	10	15	15
MINIMUM	4.600	.	840.000	674.000	0.160
MAXIMUM	5.700	.	3260.000	1927.000	2.500
MEAN	5.013	.	2220.000	1376.200	1.167
STANDARD DEV	0.318	.	976.058	406.959	0.652

	NIT	COT	ZNT	FEF	CA
N OF CASES	15	15	15	15	0
MINIMUM	2.100	0.150	0.030	0.100	.
MAXIMUM	26.000	2.700	3.000	0.400	.
MEAN	13.240	0.911	1.469	0.133	.
STANDARD DEV	7.054	0.708	0.917	0.082	.

	MG	FLOWMGD
N OF CASES	0	0
MINIMUM	.	.
MAXIMUM	.	.
MEAN	.	.
STANDARD DEV	.	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 92.000

TOTAL OBSERVATIONS: 18

	pH	ALK	SC	SO4	CUT

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

N OF CASES	18	13	0	15	18
MINIMUM	4.200	7.000	.	741.000	0.008
MAXIMUM	7.200	180.000	.	2575.000	2.200
MEAN	5.928	47.077	.	1770.400	0.686
STANDARD DEV	1.123	54.680	.	533.258	0.804

	NIT	COT	ZNT	FEF	CA
N OF CASES	18	18	18	15	16
MINIMUM	0.640	0.050	0.050	-0.100	40.000
MAXIMUM	30.000	2.000	2.900	2.300	300.000
MEAN	9.766	0.544	1.095	0.300	187.188
STANDARD DEV	9.169	0.527	0.954	0.621	67.181

	MG	FLOWMGD
N OF CASES	16	0
MINIMUM	40.000	.
MAXIMUM	310.000	.
MEAN	178.313	.
STANDARD DEV	69.572	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 93.000

TOTAL OBSERVATIONS: 15

	pH	ALK	SC	SO4	CUT
N OF CASES	15	0	0	15	15
MINIMUM	6.500	.	.	894.000	0.040
MAXIMUM	7.600	.	.	2483.000	0.350
MEAN	7.147	.	.	1302.133	0.135
STANDARD DEV	0.288	.	.	374.947	0.088

	NIT	COT	ZNT	FEF	CA
N OF CASES	15	15	15	15	15
MINIMUM	3.600	0.080	0.350	-0.100	140.000
MAXIMUM	12.000	0.680	1.100	0.300	260.000
MEAN	7.047	0.260	0.695	0.013	213.333
STANDARD DEV	2.157	0.172	0.224	0.136	33.736

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	MG	FLOWMGD
N OF CASES	15	0
MINIMUM	120.000	.
MAXIMUM	210.000	.
MEAN	173.333	.
STANDARD DEV	28.702	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 94.000

TOTAL OBSERVATIONS: 21

	pH	ALK	SC	SO4	CUT
N OF CASES	21	0	0	21	21
MINIMUM	6.700	.	.	102.000	0.050
MAXIMUM	7.500	.	.	1869.000	0.370
MEAN	7.019	.	.	1063.571	0.153
STANDARD DEV	0.256	.	.	429.591	0.070

	NIT	COT	ZNT	FEF	CA
N OF CASES	21	21	21	21	21
MINIMUM	0.400	0.030	0.080	-0.100	26.000
MAXIMUM	11.000	0.570	1.100	0.900	270.000
MEAN	6.233	0.226	0.621	0.048	191.762
STANDARD DEV	2.663	0.138	0.333	0.223	63.219

	MG	FLOWMGD
N OF CASES	21	0
MINIMUM	10.000	.
MAXIMUM	270.000	.
MEAN	156.571	.
STANDARD DEV	57.783	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 95.000

TOTAL OBSERVATIONS: 16

	pH	ALK	SC	SO4	CUT

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

N OF CASES	16	0	0	16	16
MINIMUM	6.600	.	.	215.000	0.026
MAXIMUM	7.200	.	.	1679.000	0.310
MEAN	6.919	.	.	872.188	0.085
STANDARD DEV	0.187	.	.	488.060	0.071

	NIT	COT	ZNT	FEF	CA
N OF CASES	16	16	16	16	16
MINIMUM	0.150	0.011	-0.010	-0.100	60.000
MAXIMUM	9.900	0.290	1.700	1.300	309.000
MEAN	3.483	0.100	0.426	0.150	169.938
STANDARD DEV	3.427	0.086	0.540	0.337	77.698

	MG	FLOWMGD
N OF CASES	16	0
MINIMUM	30.000	.
MAXIMUM	280.000	.
MEAN	145.750	.
STANDARD DEV	68.945	.

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 96.000

TOTAL OBSERVATIONS: 15

	pH	ALK	SC	SO4	CUT
N OF CASES	15	0	0	15	15
MINIMUM	6.400	.	.	248.000	0.103
MAXIMUM	7.300	.	.	1350.000	0.489
MEAN	6.940	.	.	929.267	0.237
STANDARD DEV	0.247	.	.	360.121	0.137

	NIT	COT	ZNT	FEF	CA
N OF CASES	15	15	15	15	15
MINIMUM	1.440	-0.001	0.137	-0.030	85.600
MAXIMUM	10.100	0.429	1.370	2.550	289.000
MEAN	6.325	0.182	0.773	0.205	190.907
STANDARD DEV	2.858	0.109	0.383	0.650	66.091

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	MG	FLOWMGD
N OF CASES	15	14
MINIMUM	31.200	0.008
MAXIMUM	195.000	0.032
MEAN	127.500	0.014
STANDARD DEV	54.551	0.007

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 97.000

TOTAL OBSERVATIONS: 12

	pH	ALK	SC	SO4	CUT
N OF CASES	12	0	0	12	12
MINIMUM	6.700	.	.	263.000	0.071
MAXIMUM	7.200	.	.	1490.000	0.258
MEAN	6.958	.	.	1006.083	0.130
STANDARD DEV	0.151	.	.	475.037	0.058

	NIT	COT	ZNT	FEF	CA
N OF CASES	12	12	12	12	12
MINIMUM	1.120	0.020	0.102	-0.030	46.800
MAXIMUM	12.100	0.331	1.777	0.060	290.000
MEAN	6.358	0.123	0.764	0.038	193.717
STANDARD DEV	4.220	0.095	0.522	0.023	94.795

	MG	FLOWMGD
N OF CASES	12	12
MINIMUM	28.100	0.000
MAXIMUM	210.000	0.021
MEAN	136.142	0.005
STANDARD DEV	68.971	0.006

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 98.000

TOTAL OBSERVATIONS: 19

	pH	ALK	SC	SO4	CUT

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

N OF CASES	19	0	0	19	19
MINIMUM	6.400	.	.	321.000	0.011
MAXIMUM	7.400	.	.	2290.000	0.783
MEAN	7.032	.	.	1179.316	0.164
STANDARD DEV	0.260	.	.	457.224	0.167

	NIT	COT	ZNT	FEF	CA
N OF CASES	19	19	19	0	19
MINIMUM	1.550	0.014	0.183	.	64.300
MAXIMUM	15.000	0.403	1.830	.	325.000
MEAN	7.459	0.136	0.891	.	229.363
STANDARD DEV	3.776	0.103	0.486	.	77.252

	MG	FLOWMGD
N OF CASES	19	19
MINIMUM	41.300	0.000
MAXIMUM	208.000	0.026
MEAN	155.032	0.005
STANDARD DEV	50.782	0.006

THE FOLLOWING RESULTS ARE FOR:
SITE$ = Seep1
YEAR = 99.000

TOTAL OBSERVATIONS: 16

	pH	ALK	SC	SO4	CUT
N OF CASES	16	0	0	16	16
MINIMUM	5.100	.	.	358.000	0.070
MAXIMUM	7.400	.	.	1470.000	2.040
MEAN	6.644	.	.	1098.000	0.603
STANDARD DEV	0.669	.	.	315.622	0.565

	NIT	COT	ZNT	FEF	CA
N OF CASES	16	16	16	16	16
MINIMUM	1.380	0.033	0.319	-0.030	43.500
MAXIMUM	14.400	0.666	1.990	0.400	295.000
MEAN	8.998	0.319	1.362	0.085	210.531
STANDARD DEV	3.608	0.182	0.524	0.111	66.516

Table A2.5. Summary statistics of Seep 1 (site 043) water quality statistics, 1992-1999.

	MG	FLOWMGD
N OF CASES	16	16
MINIMUM	30.100	0.000
MAXIMUM	199.000	0.042
MEAN	142.388	0.010
STANDARD DEV	46.111	0.009

Note: The parameters with a T suffix are total values, parameters with an F suffix are filtered values, FLOWMGD is flow in mgd, and values preceded with a hyphen are less than the detection limit.

Table A2.6. Summary statistics of water quality data from surface water sites within the Seep 1 pretreatment system.

(This table includes the 1993-94 data collected by LTV, as well as the 1997-99 data collected by DNR.)

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-1

TOTAL OBSERVATIONS: 33

	SC	PH	Cu	Ni	CO
N OF CASES	13	28	33	32	6
MINIMUM	800.000	4.600	0.100	1.800	0.065
MAXIMUM	3000.000	6.690	4.400	26.700	1.400
RANGE	2200.000	2.090	4.300	24.900	1.335
MEAN	1772.308	5.904	1.064	11.719	0.511
STANDARD DEV	668.495	0.635	0.796	5.411	0.510
MEDIAN	1800.000	6.100	0.980	11.000	0.400

	Zn	SO4	CA	MG
N OF CASES	6	33	7	7
MINIMUM	0.280	284.400	178.000	80.900
MAXIMUM	2.640	1796.000	287.000	217.000
RANGE	2.360	1511.600	109.000	136.100
MEAN	1.180	1237.315	250.429	168.271
STANDARD DEV	0.877	326.295	37.801	50.218
MEDIAN	1.010	1302.000	267.000	181.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-2

TOTAL OBSERVATIONS: 33

	SC	PH	Cu	Ni	Co
N OF CASES	13	29	33	32	6
MINIMUM	600.000	4.800	0.100	1.800	0.050
MAXIMUM	2700.000	7.200	2.100	25.400	1.300
RANGE	2100.000	2.400	2.000	23.600	1.250
MEAN	1682.692	6.310	0.716	10.975	0.475

Table A2.6. Summary statistics of water quality data from surface water sites within the Seep 1 pretreatment system (continued).

STANDARD DEV	710.222	0.570	0.525	5.184	0.492
MEDIAN	2000.000	6.400	0.640	10.500	0.350

	Zn	SO4	CA	MG
N OF CASES	6	33	7	7
MINIMUM	0.300	272.900	116.000	65.700
MAXIMUM	2.680	2311.000	277.000	191.000
RANGE	2.380	2038.100	161.000	125.300
MEAN	1.233	1124.512	220.714	153.243
STANDARD DEV	0.996	463.571	61.378	49.078
MEDIAN	1.065	1154.000	246.000	177.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-3

TOTAL OBSERVATIONS: 33

	SC	PH	Cu	Ni	Co
N OF CASES	13	29	33	32	6
MINIMUM	400.000	5.000	0.039	0.300	0.007
MAXIMUM	2600.000	7.170	3.200	31.000	1.900
RANGE	2200.000	2.170	3.161	30.700	1.893
MEAN	1755.769	6.100	1.071	14.441	0.647
STANDARD DEV	819.626	0.640	0.771	6.571	0.769
MEDIAN	2200.000	6.200	0.870	15.000	0.428

	Zn	SO4	CA	MG
N OF CASES	6	33	7	7
MINIMUM	0.060	133.000	178.000	94.600
MAXIMUM	3.170	1982.000	300.000	228.000
RANGE	3.110	1849.000	122.000	133.400
MEAN	1.408	1283.033	260.571	186.086
STANDARD DEV	1.417	452.127	42.182	55.203
MEDIAN	1.330	1333.000	275.000	213.000

Table A2.6. Summary statistics of water quality data from surface water sites within the Seep 1 pretreatment system (continued).

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-4

TOTAL OBSERVATIONS: 33

	SC	PH	Cu	Ni	Co
N OF CASES	12	25	33	32	6
MINIMUM	750.000	5.380	0.074	1.700	0.042
MAXIMUM	2600.000	7.000	2.200	21.000	1.000
RANGE	1850.000	1.620	2.126	19.300	0.958
MEAN	1913.750	6.384	0.541	11.238	0.378
STANDARD DEV	563.246	0.441	0.493	4.390	0.390
MEDIAN	2025.000	6.400	0.460	11.050	0.283

	Zn	SO4	CA	MG
N OF CASES	6	32	7	7
MINIMUM	0.270	370.000	162.000	110.000
MAXIMUM	2.580	2481.000	285.000	221.000
RANGE	2.310	2111.000	123.000	111.000
MEAN	1.162	1253.094	247.571	179.429
STANDARD DEV	0.926	356.639	43.146	37.868
MEDIAN	1.020	1285.000	264.000	196.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-5

TOTAL OBSERVATIONS: 33

	SC	PH	Cu	Ni	Co
N OF CASES	13	25	33	31	6
MINIMUM	650.000	5.560	0.077	1.300	0.039
MAXIMUM	2500.000	7.100	2.300	27.400	1.400
RANGE	1850.000	1.540	2.223	26.100	1.361
MEAN	1762.308	6.505	0.559	11.487	0.466
STANDARD DEV	644.634	0.381	0.498	5.032	0.512
MEDIAN	2100.000	6.510	0.400	11.000	0.350

Table A2.6. Summary statistics of water quality data from surface water sites within the Seep 1 pretreatment system (continued).

	Zn	SO4	CA	MG
N OF CASES	6	32	7	7
MINIMUM	0.260	294.400	166.000	111.000
MAXIMUM	2.820	2513.000	275.000	203.000
RANGE	2.560	2218.600	109.000	92.000
MEAN	1.217	1234.356	242.000	175.286
STANDARD DEV	1.025	363.673	42.887	35.032
MEDIAN	1.075	1250.000	271.000	187.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-6

TOTAL OBSERVATIONS: 57

	SC	PH	Cu	Ni	Co
N OF CASES	36	47	55	53	27
MINIMUM	110.000	4.740	0.040	1.400	0.008
MAXIMUM	2400.000	7.530	1.420	19.500	2.400
RANGE	2290.000	2.790	1.380	18.100	2.392
MEAN	1750.972	6.563	0.324	10.466	0.348
STANDARD DEV	533.625	0.559	0.308	3.723	0.477
MEDIAN	1950.000	6.650	0.200	10.000	0.200

	Zn	SO4	CA	MG
N OF CASES	27	55	22	22
MINIMUM	0.001	318.000	177.000	124.000
MAXIMUM	2.130	2502.000	326.000	223.000
RANGE	2.129	2184.000	149.000	99.000
MEAN	1.054	1174.505	260.000	187.045
STANDARD DEV	0.633	339.513	34.734	25.706
MEDIAN	1.100	1231.000	266.000	191.500

Table A2.6. Summary statistics of water quality data from surface water sites within the Seep 1 pretreatment system (continued).

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-7

TOTAL OBSERVATIONS: 13

	SC	PH	Cu	Ni	Co
N OF CASES	13	13	13	13	6
MINIMUM	750.000	6.320	0.066	1.500	0.032
MAXIMUM	2400.000	7.090	1.100	17.800	0.500
RANGE	1650.000	0.770	1.034	16.300	0.468
MEAN	1726.154	6.789	0.400	10.592	0.261
STANDARD DEV	570.633	0.244	0.303	5.404	0.229
MEDIAN	2000.000	6.740	0.300	11.500	0.236

	Zn	SO4	CA	MG
N OF CASES	6	13	7	7
MINIMUM	0.260	352.000	175.000	119.000
MAXIMUM	1.600	1486.000	278.000	206.000
RANGE	1.340	1134.000	103.000	87.000
MEAN	0.892	1086.377	245.143	171.714
STANDARD DEV	0.664	424.844	35.951	29.250
MEDIAN	0.810	1228.000	260.000	178.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-8

TOTAL OBSERVATIONS: 13

	SC	PH	Cu	Ni	Co
N OF CASES	13	13	13	13	6
MINIMUM	750.000	6.060	0.066	1.400	0.036
MAXIMUM	2800.000	7.030	1.600	18.400	0.500
RANGE	2050.000	0.970	1.534	17.000	0.464
MEAN	1735.000	6.733	0.502	10.685	0.282
STANDARD DEV	642.900	0.300	0.414	5.586	0.213
MEDIAN	2000.000	6.700	0.400	11.900	0.300

Table A2.6. Summary statistics of water quality data from surface water sites within the Seep 1 pretreatment system (continuied).

	Zn	SO4	CA	MG
N OF CASES	6	13	7	7
MINIMUM	0.240	336.000	147.000	99.800
MAXIMUM	1.700	1494.000	278.000	201.000
RANGE	1.460	1158.000	131.000	101.200
MEAN	0.930	1068.538	229.429	164.971
STANDARD DEV	0.679	444.785	53.376	39.983
MEDIAN	0.890	1290.000	262.000	180.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-W

TOTAL OBSERVATIONS: 32

	SC	PH	Cu	Ni	Co
N OF CASES	9	24	29	28	3
MINIMUM	300.000	6.200	0.030	1.500	0.050
MAXIMUM	2700.000	7.300	0.450	15.000	0.300
RANGE	2400.000	1.100	0.420	13.500	0.250
MEAN	1689.444	6.839	0.159	8.993	0.183
STANDARD DEV	743.011	0.275	0.108	2.679	0.126
MEDIAN	2050.000	6.865	0.130	8.400	0.200

	Zn	SO4	CA	MG
N OF CASES	3	32	6	6
MINIMUM	0.200	138.000	182.000	127.000
MAXIMUM	1.000	2492.000	275.000	197.000
RANGE	0.800	2354.000	93.000	70.000
MEAN	0.627	1148.922	241.500	166.667
STANDARD DEV	0.403	388.430	42.824	30.559
MEDIAN	0.680	1168.500	266.000	180.500

Table A2.6. Summary statistics of water quality data from surface water sites within the Seep 1 pretreatment system (continued).

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-WB

TOTAL OBSERVATIONS: 36

	SC	PH	Cu	Ni	Co
N OF CASES	36	36	34	35	27
MINIMUM	300.000	4.760	0.055	1.434	0.027
MAXIMUM	2350.000	7.640	1.320	16.700	0.880
RANGE	2050.000	2.880	1.265	15.266	0.853
MEAN	1765.278	6.728	0.281	8.886	0.248
STANDARD DEV	465.460	0.517	0.255	3.832	0.215
MEDIAN	1925.000	6.815	0.200	8.360	0.200

	Zn	SO4	CA	MG
N OF CASES	27	35	23	23
MINIMUM	0.001	138.000	151.000	99.300
MAXIMUM	1.970	1589.000	321.000	213.000
RANGE	1.969	1451.000	170.000	113.700
MEAN	0.961	1136.497	257.304	182.057
STANDARD DEV	0.519	350.083	40.966	30.087
MEDIAN	0.918	1262.000	269.000	187.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Historic

TOTAL OBSERVATIONS: 44

	SC	PH	Cu	Ni	Co
N OF CASES	0	44	44	44	44
MINIMUM	.	4.600	0.420	11.100	0.520
MAXIMUM	.	5.600	2.500	32.000	2.700
RANGE	.	1.000	2.080	20.900	2.180
MEAN	.	5.118	1.053	20.568	1.476
STANDARD DEV	.	0.251	0.513	5.882	0.495
MEDIAN	.	5.200	0.825	21.000	1.450

Table A2.6. Summary statistics of water quality data from surface water sites within the Seep 1 pretreatment system (continued).

	Zn	SO4	CA	MG
N OF CASES	44	44	12	0
MINIMUM	0.880	605.000	130.000	.
MAXIMUM	3.000	2650.000	240.000	.
RANGE	2.120	2045.000	110.000	.
MEAN	1.923	1501.091	190.833	.
STANDARD DEV	0.532	370.618	31.176	.
MEDIAN	1.950	1486.000	200.000	.

Table A2.7. Summary statistics of all 1993 surface water quality data for the outer pool of the Seep 1 pretreatment system.

Note: All 1993-94 data from sites 043-1, 04302 and 043-3 were included in these statistics.

THE FOLLOWING RESULTS ARE FOR:
MONTH = June

TOTAL OBSERVATIONS: 18

	CU	NI	CO	ZN	CA
N OF CASES	18	18	0	0	0
MINIMUM	0.140	6.000	.	.	.
MAXIMUM	1.800	15.000	.	.	.
RANGE	1.660	9.000	.	.	.
MEAN	0.653	10.411	.	.	.
STANDARD DEV	0.371	2.618	.	.	.
MEDIAN	0.570	11.000	.	.	.

	MG	SO4
N OF CASES	0	18
MINIMUM	.	708.000
MAXIMUM	.	1579.000
RANGE	.	871.000
MEAN	.	1155.222
STANDARD DEV	.	236.504
MEDIAN	.	1176.000

THE FOLLOWING RESULTS ARE FOR:
MONTH = July

TOTAL OBSERVATIONS: 27

	CU	NI	CO	ZN	CA
N OF CASES	27	24	0	0	0
MINIMUM	0.360	9.100	.	.	.
MAXIMUM	1.600	20.000	.	.	.
RANGE	1.240	10.900	.	.	.
MEAN	0.993	13.346	.	.	.
STANDARD DEV	0.368	3.529	.	.	.
MEDIAN	0.980	12.500	.	.	.

Table A2.7. Summary statistics of all 1993 surface water quality data for the outer pool of the Seep 1 pretreatment system (continued).

	MG	SO4
N OF CASES	0	27
MINIMUM	.	717.000
MAXIMUM	.	1734.000
RANGE	.	1017.000
MEAN	.	1255.704
STANDARD DEV	.	215.419
MEDIAN	.	1254.000

THE FOLLOWING RESULTS ARE FOR:
MONTH = August

TOTAL OBSERVATIONS: 15

	CU	NI	CO	ZN	CA
N OF CASES	15	15	0	0	0
MINIMUM	0.140	3.000	.	.	.
MAXIMUM	1.200	17.000	.	.	.
RANGE	1.060	14.000	.	.	.
MEAN	0.679	11.233	.	.	.
STANDARD DEV	0.389	3.824	.	.	.
MEDIAN	0.720	11.000	.	.	.

	MG	SO4
N OF CASES	0	15
MINIMUM	.	558.000
MAXIMUM	.	2311.000
RANGE	.	1753.000
MEAN	.	1506.200
STANDARD DEV	.	478.565
MEDIAN	.	1443.000

Note: values are mg/L

Table A2.8. Summary statistics of all 1997 surface water quality data for the outer pool of the Seep 1 pretreatment system.

Note: All 1997 data from sites 043-1, 04302 and 043-3 were included in these statistics.

THE FOLLOWING RESULTS ARE FOR:
MONTH = May

TOTAL OBSERVATIONS: 9

	Cu	Ni	Co	Zn	CA
N OF CASES	9	9	9	9	0
MINIMUM	0.039	0.300	0.007	0.060	.
MAXIMUM	0.200	4.600	0.200	0.650	.
RANGE	0.161	4.300	0.193	0.590	.
MEAN	0.122	2.100	0.077	0.336	.
STANDARD DEV	0.062	1.440	0.058	0.192	.
MEDIAN	0.100	1.800	0.065	0.300	.

	MG	SO4
N OF CASES	0	9
MINIMUM	.	133.000
MAXIMUM	.	601.000
RANGE	.	468.000
MEAN	.	325.378
STANDARD DEV	.	157.361
MEDIAN	.	284.400

THE FOLLOWING RESULTS ARE FOR:
MONTH = June

TOTAL OBSERVATIONS: 6

	Cu	Ni	Co	Zn	CA
N OF CASES	6	6	3	3	3
MINIMUM	1.900	18.200	1.300	2.640	218.000
MAXIMUM	4.400	31.000	1.900	3.170	276.000
RANGE	2.500	12.800	0.600	0.530	58.000
MEAN	2.717	25.433	1.533	2.830	251.333
STANDARD DEV	0.962	4.151	0.321	0.295	29.956
MEDIAN	2.400	25.900	1.400	2.680	260.000

Table A2.8. Summary statistics of all 1997 surface water quality data for the outer pool of the Seep 1 pretreatment system (continued).

	MG	SO4
N OF CASES	3	6
MINIMUM	170.000	1230.000
MAXIMUM	217.000	1760.000
RANGE	47.000	530.000
MEAN	200.667	1539.500
STANDARD DEV	26.577	173.848
MEDIAN	215.000	1562.500

THE FOLLOWING RESULTS ARE FOR:
MONTH = July

TOTAL OBSERVATIONS: 6

	Cu	Ni	Co	Zn	CA
N OF CASES	6	6	3	3	3
MINIMUM	1.000	14.900	0.700	1.600	246.000
MAXIMUM	2.900	24.300	1.100	2.400	300.000
RANGE	1.900	9.400	0.400	0.800	54.000
MEAN	1.867	18.600	0.833	1.967	277.667
STANDARD DEV	0.766	3.486	0.231	0.404	28.184
MEDIAN	1.800	18.550	0.700	1.900	287.000

	MG	SO4
N OF CASES	3	6
MINIMUM	178.000	1295.000
MAXIMUM	228.000	1627.000
RANGE	50.000	332.000
MEAN	201.667	1445.000
STANDARD DEV	25.106	139.021
MEDIAN	199.000	1441.500

Table A2.8. Summary statistics of all 1997 surface water quality data for the outer pool of the Seep 1 pretreatment system (continued).

THE FOLLOWING RESULTS ARE FOR:
MONTH = August

TOTAL OBSERVATIONS: 6

	Cu	Ni	Co	Zn	CA
N OF CASES	6	6	3	3	3
MINIMUM	0.370	7.200	0.600	1.370	116.000
MAXIMUM	1.600	16.000	0.800	2.460	236.000
RANGE	1.230	8.800	0.200	1.090	120.000
MEAN	1.012	11.733	0.667	1.840	194.000
STANDARD DEV	0.435	3.659	0.115	0.560	67.617
MEDIAN	0.900	12.850	0.600	1.690	230.000

	MG	SO4
N OF CASES	3	6
MINIMUM	65.700	519.000
MAXIMUM	161.000	1538.000
RANGE	95.300	1019.000
MEAN	107.100	1108.667
STANDARD DEV	48.864	417.802
MEDIAN	94.600	1237.000

THE FOLLOWING RESULTS ARE FOR:
MONTH = September

TOTAL OBSERVATIONS: 6

	Cu	Ni	Co	Zn	CA
N OF CASES	6	6	0	0	6
MINIMUM	0.500	13.200	.	.	241.000
MAXIMUM	1.610	17.000	.	.	283.000
RANGE	1.110	3.800	.	.	42.000
MEAN	1.040	14.883	.	.	266.333
STANDARD DEV	0.379	1.291	.	.	15.731
MEDIAN	1.035	14.600	.	.	271.000

Table A2.8. Summary statistics of all 1997 surface water quality data for the outer pool of the Seep 1 pretreatment system (continued).

	MG	SO4
N OF CASES	6	6
MINIMUM	125.000	1116.000
MAXIMUM	222.000	1604.000
RANGE	97.000	488.000
MEAN	184.667	1404.167
STANDARD DEV	34.098	165.051
MEDIAN	186.000	1397.500

THE FOLLOWING RESULTS ARE FOR:
MONTH = October

TOTAL OBSERVATIONS: 6

	Cu	Ni	Co	Zn	CA
N OF CASES	6	6	0	0	6
MINIMUM	0.340	6.800	.	.	159.000
MAXIMUM	1.100	16.200	.	.	292.000
RANGE	0.760	9.400	.	.	133.000
MEAN	0.737	11.550	.	.	225.833
STANDARD DEV	0.337	4.100	.	.	60.101
MEDIAN	0.760	11.200	.	.	226.000

	MG	SO4
N OF CASES	6	6
MINIMUM	80.900	795.000
MAXIMUM	214.000	1572.000
RANGE	133.100	777.000
MEAN	152.817	1179.500
STANDARD DEV	59.399	374.898
MEDIAN	153.500	1169.500

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-1
YEAR = 93.000

TOTAL OBSERVATIONS: 17

	SC	PH	Cu	Ni	Co
N OF CASES	0	12	17	17	0
MINIMUM	.	4.600	0.250	8.500	.
MAXIMUM	.	6.600	1.600	18.000	.
RANGE	.	2.000	1.350	9.500	.
MEAN	.	5.733	0.879	11.000	.
STANDARD DEV	.	0.724	0.391	2.378	.
MEDIAN	.	5.800	0.820	11.000	.

	Zn	SO4	CA	MG
N OF CASES	0	17	0	0
MINIMUM	.	1058.000	.	.
MAXIMUM	.	1796.000	.	.
RANGE	.	738.000	.	.
MEAN	.	1320.529	.	.
STANDARD DEV	.	180.875	.	.
MEDIAN	.	1316.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-1
YEAR = 94.000

TOTAL OBSERVATIONS: 3

	SC	PH	Cu	Ni	Co
N OF CASES	0	3	3	2	0
MINIMUM	.	5.300	0.660	6.200	.
MAXIMUM	.	6.200	1.400	9.900	.
RANGE	.	0.900	0.740	3.700	.
MEAN	.	5.600	1.020	8.050	.
STANDARD DEV	.	0.520	0.370	2.616	.
MEDIAN	.	5.300	1.000	8.050	.

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

	Zn	SO4	CA	MG
N OF CASES	0	3	0	0
MINIMUM	.	939.000	.	.
MAXIMUM	.	1435.000	.	.
RANGE	.	496.000	.	.
MEAN	.	1199.667	.	.
STANDARD DEV	.	248.969	.	.
MEDIAN	.	1225.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-1
YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	1790.000	4.930	0.900	11.700	0.600
MAXIMUM	3000.000	6.300	4.400	26.700	1.400
RANGE	1210.000	1.370	3.500	15.000	0.800
MEAN	2231.667	5.765	2.050	18.900	0.900
STANDARD DEV	460.887	0.520	1.286	6.474	0.436
MEDIAN	2150.000	5.825	1.800	17.300	0.700

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	1.370	1150.000	230.000	161.000
MAXIMUM	2.640	1598.000	287.000	217.000
RANGE	1.270	448.000	57.000	56.000
MEAN	1.870	1430.500	264.333	192.333
STANDARD DEV	0.677	175.934	30.238	28.589
MEDIAN	1.600	1469.500	276.000	199.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-2
YEAR = 93.000

TOTAL OBSERVATIONS: 17

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

	SC	PH	Cu	Ni	Co
N OF CASES	0	13	17	17	0
MINIMUM	.	4.800	0.140	3.000	.
MAXIMUM	.	7.200	1.500	16.000	.
RANGE	.	2.400	1.360	13.000	.
MEAN	.	6.269	0.539	10.329	.
STANDARD DEV	.	0.724	0.406	3.334	.
MEDIAN	.	6.400	0.410	10.000	.

	Zn	SO4	CA	MG
N OF CASES	0	17	0	0
MINIMUM	.	558.000	.	.
MAXIMUM	.	2311.000	.	.
RANGE	.	1753.000	.	.
MEAN	.	1225.941	.	.
STANDARD DEV	.	485.683	.	.
MEDIAN	.	1154.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-2
YEAR = 94.000

TOTAL OBSERVATIONS: 3

	SC	PH	Cu	Ni	Co
N OF CASES	0	3	3	2	0
MINIMUM	.	5.600	0.640	7.400	.
MAXIMUM	.	6.700	0.820	9.100	.
RANGE	.	1.100	0.180	1.700	.
MEAN	.	6.200	0.720	8.250	.
STANDARD DEV	.	0.557	0.092	1.202	.
MEDIAN	.	6.300	0.700	8.250	.

	Zn	SO4	CA	MG
N OF CASES	0	3	0	0
MINIMUM	.	708.000	.	.
MAXIMUM	.	1153.000	.	.
RANGE	.	445.000	.	.

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

MEAN	.	952.000	.	.
STANDARD DEV	.	225.595	.	.
MEDIAN	.	995.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-2
YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	975.000	5.390	0.900	7.500	0.600
MAXIMUM	2700.000	6.440	2.100	25.400	1.300
RANGE	1725.000	1.050	1.200	17.900	0.700
MEAN	2016.667	6.068	1.400	16.317	0.867
STANDARD DEV	602.840	0.386	0.502	5.851	0.379
MEDIAN	2037.500	6.165	1.300	16.400	0.700

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	1.690	519.000	116.000	65.700
MAXIMUM	2.680	1524.000	246.000	178.000
RANGE	0.990	1005.000	130.000	112.300
MEAN	2.090	1212.667	193.333	137.900
STANDARD DEV	0.522	354.099	68.420	62.655
MEDIAN	1.900	1309.500	218.000	170.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-3
YEAR = 93.000

TOTAL OBSERVATIONS: 17

	SC	PH	Cu	Ni	Co
N OF CASES	0	13	17	17	0
MINIMUM	.	5.000	0.430	11.000	.
MAXIMUM	.	7.100	1.500	20.000	.
RANGE	.	2.100	1.070	9.000	.

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

MEAN	.	6.000	0.942	15.059	.
STANDARD DEV	.	0.566	0.302	2.883	.
MEDIAN	.	6.100	0.910	15.000	.

	Zn	SO4	CA	MG
N OF CASES	0	17	0	0
MINIMUM	.	1090.000	.	.
MAXIMUM	.	1982.000	.	.
RANGE	.	892.000	.	.
MEAN	.	1426.824	.	.
STANDARD DEV	.	246.888	.	.
MEDIAN	.	1333.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-3
YEAR = 94.000

TOTAL OBSERVATIONS: 3

	SC	PH	Cu	Ni	Co
N OF CASES	0	3	3	2	0
MINIMUM	.	5.900	0.730	10.000	.
MAXIMUM	.	6.200	1.800	15.000	.
RANGE	.	0.300	1.070	5.000	.
MEAN	.	6.100	1.133	12.500	.
STANDARD DEV	.	0.173	0.582	3.536	.
MEDIAN	.	6.200	0.870	12.500	.

	Zn	SO4	CA	MG
N OF CASES	0	3	0	0
MINIMUM	.	970.000	.	.
MAXIMUM	.	1254.000	.	.
RANGE	.	284.000	.	.
MEAN	.	1096.667	.	.
STANDARD DEV	.	144.462	.	.
MEDIAN	.	1066.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-3

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	1325.000	5.040	0.370	7.200	0.800
MAXIMUM	2600.000	6.700	3.200	31.000	1.900
RANGE	1275.000	1.660	2.830	23.800	1.100
MEAN	2183.333	5.570	2.145	20.550	1.267
STANDARD DEV	466.280	0.665	1.070	8.265	0.569
MEDIAN	2312.500	5.270	2.500	22.100	1.100

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	2.400	681.000	236.000	94.600
MAXIMUM	3.170	1760.000	300.000	228.000
RANGE	0.770	1079.000	64.000	133.400
MEAN	2.677	1450.000	265.333	179.200
STANDARD DEV	0.428	386.229	32.332	73.554
MEDIAN	2.460	1550.500	260.000	215.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-4
YEAR = 93.000

TOTAL OBSERVATIONS: 17

	SC	PH	Cu	Ni	Co
N OF CASES	0	10	17	17	0
MINIMUM	.	5.800	0.120	7.700	.
MAXIMUM	.	7.000	0.570	17.000	.
RANGE	.	1.200	0.450	9.300	.
MEAN	.	6.430	0.327	11.453	.
STANDARD DEV	.	0.424	0.152	2.618	.
MEDIAN	.	6.350	0.320	11.000	.

	Zn	SO4	CA	MG
N OF CASES	0	17	0	0

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

MINIMUM	.	1041.000	.	.
MAXIMUM	.	2481.000	.	.
RANGE	.	1440.000	.	.
MEAN	.	1335.118	.	.
STANDARD DEV	.	311.856	.	.
MEDIAN	.	1279.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-4
YEAR = 94.000

TOTAL OBSERVATIONS: 3

	SC	PH	Cu	Ni	Co
N OF CASES	0	3	3	2	0
MINIMUM	.	6.100	0.250	6.100	.
MAXIMUM	.	6.700	0.660	8.100	.
RANGE	.	0.600	0.410	2.000	.
MEAN	.	6.500	0.467	7.100	.
STANDARD DEV	.	0.346	0.206	1.414	.
MEDIAN	.	6.700	0.490	7.100	.

	Zn	SO4	CA	MG
N OF CASES	0	3	0	0
MINIMUM	.	644.000	.	.
MAXIMUM	.	1177.000	.	.
RANGE	.	533.000	.	.
MEAN	.	901.000	.	.
STANDARD DEV	.	267.007	.	.
MEDIAN	.	882.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-4
YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	5	6	6	3

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

MINIMUM	1775.000	5.380	0.500	11.100	0.500
MAXIMUM	2600.000	6.560	2.200	21.000	1.000
RANGE	825.000	1.180	1.700	9.900	0.500
MEAN	2202.500	5.995	1.317	16.017	0.700
STANDARD DEV	309.576	0.401	0.624	3.602	0.265
MEDIAN	2220.000	5.945	1.450	16.050	0.600

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	1.540	1092.000	224.000	150.000
MAXIMUM	2.580	1609.000	285.000	221.000
RANGE	1.040	517.000	61.000	71.000
MEAN	1.940	1422.500	252.000	191.667
STANDARD DEV	0.560	182.602	30.806	37.072
MEDIAN	1.700	1456.500	247.000	204.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-5
YEAR = 93.000

TOTAL OBSERVATIONS: 17

	SC	PH	Cu	Ni	Co
N OF CASES	0	9	17	17	0
MINIMUM	.	6.000	0.130	8.400	.
MAXIMUM	.	7.100	0.690	18.000	.
RANGE	.	1.100	0.560	9.600	.
MEAN	.	6.500	0.356	11.106	.
STANDARD DEV	.	0.339	0.164	2.542	.
MEDIAN	.	6.500	0.300	11.000	.

	Zn	SO4	CA	MG
N OF CASES	0	17	0	0
MINIMUM	.	940.000	.	.
MAXIMUM	.	2513.000	.	.
RANGE	.	1573.000	.	.
MEAN	.	1291.471	.	.
STANDARD DEV	.	342.856	.	.
MEDIAN	.	1209.000	.	.

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-5
YEAR = 94.000

TOTAL OBSERVATIONS: 3

	SC	PH	Cu	Ni	Co
N OF CASES	0	3	3	2	0
MINIMUM	.	6.100	0.310	7.900	.
MAXIMUM	.	6.900	0.840	9.900	.
RANGE	.	0.800	0.530	2.000	.
MEAN	.	6.500	0.657	8.900	.
STANDARD DEV	.	0.400	0.300	1.414	.
MEDIAN	.	6.500	0.820	8.900	.

	Zn	SO4	CA	MG
N OF CASES	0	3	0	0
MINIMUM	.	843.000	.	.
MAXIMUM	.	1296.000	.	.
RANGE	.	453.000	.	.
MEAN	.	1089.667	.	.
STANDARD DEV	.	229.178	.	.
MEDIAN	.	1130.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-5
YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	1600.000	5.560	0.600	11.100	0.500
MAXIMUM	2500.000	6.510	2.300	27.400	1.400
RANGE	900.000	0.950	1.700	16.300	0.900
MEAN	2135.000	6.143	1.330	17.650	0.833
STANDARD DEV	339.632	0.371	0.706	5.617	0.493
MEDIAN	2205.000	6.250	1.100	17.350	0.600

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	1.570	1015.000	209.000	144.000
MAXIMUM	2.820	1601.000	275.000	200.000
RANGE	1.250	586.000	66.000	56.000
MEAN	2.063	1374.167	236.667	174.667
STANDARD DEV	0.665	204.539	34.269	28.378
MEDIAN	1.800	1431.500	226.000	180.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-6
YEAR = 93.000

TOTAL OBSERVATIONS: 17

	SC	PH	Cu	Ni	Co
N OF CASES	0	9	17	17	0
MINIMUM	.	6.200	0.050	7.000	.
MAXIMUM	.	7.300	0.600	18.000	.
RANGE	.	1.100	0.550	11.000	.
MEAN	.	6.678	0.205	10.476	.
STANDARD DEV	.	0.370	0.156	2.941	.
MEDIAN	.	6.600	0.140	9.600	.

	Zn	SO4	CA	MG
N OF CASES	0	17	0	0
MINIMUM	.	914.000	.	.
MAXIMUM	.	2502.000	.	.
RANGE	.	1588.000	.	.
MEAN	.	1265.353	.	.
STANDARD DEV	.	340.417	.	.
MEDIAN	.	1225.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-6
YEAR = 94.000

TOTAL OBSERVATIONS: 3

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

	SC	PH	Cu	Ni	Co
N OF CASES	0	3	3	2	0
MINIMUM	.	6.200	0.040	7.400	.
MAXIMUM	.	6.900	0.730	9.200	.
RANGE	.	0.700	0.690	1.800	.
MEAN	.	6.600	0.483	8.300	.
STANDARD DEV	.	0.361	0.385	1.273	.
MEDIAN	.	6.700	0.680	8.300	.

	Zn	SO4	CA	MG
N OF CASES	0	3	0	0
MINIMUM	.	731.000	.	.
MAXIMUM	.	1186.000	.	.
RANGE	.	455.000	.	.
MEAN	.	992.333	.	.
STANDARD DEV	.	234.926	.	.
MEDIAN	.	1060.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-6
YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	1700.000	6.390	0.200	8.600	0.052
MAXIMUM	2400.000	6.650	0.900	18.500	0.400
RANGE	700.000	0.260	0.700	9.900	0.348
MEAN	2020.833	6.560	0.515	13.600	0.251
STANDARD DEV	303.487	0.121	0.307	3.537	0.179
MEDIAN	1950.000	6.625	0.395	14.050	0.300

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	0.270	1128.000	210.000	157.000
MAXIMUM	1.500	1500.000	281.000	205.000
RANGE	1.230	372.000	71.000	48.000
MEAN	1.077	1317.167	243.333	173.333
STANDARD DEV	0.699	176.612	35.698	27.429

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

MEDIAN	1.460	1322.000	239.000	158.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-6
YEAR = 98.000

TOTAL OBSERVATIONS: 7

	SC	PH	Cu	Ni	Co
N OF CASES	7	7	7	7	7
MINIMUM	1850.000	6.530	0.100	8.400	0.100
MAXIMUM	2300.000	7.330	0.600	19.500	0.600
RANGE	450.000	0.800	0.500	11.100	0.500
MEAN	2078.571	6.887	0.276	13.786	0.243
STANDARD DEV	157.737	0.267	0.170	3.650	0.172
MEDIAN	2100.000	6.810	0.200	14.900	0.200

	Zn	SO4	CA	MG
N OF CASES	7	7	7	7
MINIMUM	0.920	1143.000	231.000	173.000
MAXIMUM	1.960	1475.000	326.000	223.000
RANGE	1.040	332.000	95.000	50.000
MEAN	1.399	1301.143	272.857	202.143
STANDARD DEV	0.339	111.109	30.372	18.916
MEDIAN	1.400	1272.000	269.000	206.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-6
YEAR = 99.000

TOTAL OBSERVATIONS: 5

	SC	PH	Cu	Ni	Co
N OF CASES	5	5	4	4	3
MINIMUM	1275.000	4.740	0.555	10.470	0.533
MAXIMUM	2150.000	5.890	1.420	13.900	0.890
RANGE	875.000	1.150	0.865	3.430	0.357
MEAN	1805.000	5.372	1.057	12.193	0.738
STANDARD DEV	338.378	0.435	0.366	1.757	0.184

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

MEDIAN	1900.000	5.340	1.126	12.200	0.790

	Zn	SO4	CA	MG
N OF CASES	3	5	3	3
MINIMUM	1.513	752.000	231.000	183.000
MAXIMUM	2.130	1500.000	276.000	220.000
RANGE	0.617	748.000	45.000	37.000
MEAN	1.878	1241.800	253.333	202.000
STANDARD DEV	0.323	287.792	22.502	18.520
MEDIAN	1.990	1318.000	253.000	203.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-7
YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	1690.000	6.320	0.300	11.100	0.400
MAXIMUM	2400.000	6.740	1.100	17.800	0.500
RANGE	710.000	0.420	0.800	6.700	0.100
MEAN	2035.833	6.577	0.612	14.750	0.467
STANDARD DEV	280.721	0.153	0.293	2.389	0.058
MEDIAN	2000.000	6.575	0.535	15.400	0.500

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	1.260	1136.000	231.000	152.000
MAXIMUM	1.600	1486.000	278.000	206.000
RANGE	0.340	350.000	47.000	54.000
MEAN	1.487	1309.167	247.333	174.667
STANDARD DEV	0.196	158.481	26.577	28.024
MEDIAN	1.600	1308.000	233.000	166.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-8
YEAR = 97.000

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	1225.000	6.060	0.400	8.000	0.400
MAXIMUM	2800.000	6.690	1.600	18.400	0.500
RANGE	1575.000	0.630	1.200	10.400	0.100
MEAN	2041.667	6.478	0.800	14.683	0.467
STANDARD DEV	510.065	0.234	0.438	3.692	0.058
MEDIAN	2050.000	6.510	0.600	15.950	0.500

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	1.330	754.000	147.000	99.800
MAXIMUM	1.700	1494.000	278.000	201.000
RANGE	0.370	740.000	131.000	101.200
MEAN	1.537	1272.500	211.333	155.933
STANDARD DEV	0.189	280.572	65.531	51.500
MEDIAN	1.580	1366.000	209.000	167.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-W
YEAR = 93.000

TOTAL OBSERVATIONS: 17

	SC	PH	Cu	Ni	Co
N OF CASES	0	12	17	17	0
MINIMUM	.	6.200	0.030	6.200	.
MAXIMUM	.	7.300	0.210	15.000	.
RANGE	.	1.100	0.180	8.800	.
MEAN	.	6.833	0.115	9.582	.
STANDARD DEV	.	0.339	0.050	2.378	.
MEDIAN	.	6.800	0.110	8.900	.

	Zn	SO4	CA	MG
N OF CASES	0	17	0	0
MINIMUM	.	1002.000	.	.
MAXIMUM	.	2492.000	.	.

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

RANGE	.	1490.000	.	.
MEAN	.	1247.294	.	.
STANDARD DEV	.	333.784	.	.
MEDIAN	.	1172.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-W
YEAR = 94.000

TOTAL OBSERVATIONS: 3

	SC	PH	Cu	Ni	Co
N OF CASES	0	3	3	2	0
MINIMUM	.	6.400	0.050	6.800	.
MAXIMUM	.	6.900	0.450	8.000	.
RANGE	.	0.500	0.400	1.200	.
MEAN	.	6.667	0.267	7.400	.
STANDARD DEV	.	0.252	0.202	0.849	.
MEDIAN	.	6.700	0.300	7.400	.

	Zn	SO4	CA	MG
N OF CASES	0	3	0	0
MINIMUM	.	745.000	.	.
MAXIMUM	.	1237.000	.	.
RANGE	.	492.000	.	.
MEAN	.	1014.000	.	.
STANDARD DEV	.	249.205	.	.
MEDIAN	.	1060.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-W
YEAR = 97.000

TOTAL OBSERVATIONS: 4

	SC	PH	Cu	Ni	Co
N OF CASES	3	3	4	4	2
MINIMUM	1475.000	6.680	0.100	7.200	0.200
MAXIMUM	2700.000	6.910	0.400	14.000	0.300

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

RANGE	1225.000	0.230	0.300	6.800	0.100
MEAN	2075.000	6.757	0.263	10.125	0.250
STANDARD DEV	612.883	0.133	0.138	2.957	0.071
MEDIAN	2050.000	6.680	0.275	9.650	0.250

	Zn	SO4	CA	MG
N OF CASES	2	4	2	2
MINIMUM	0.680	936.000	191.000	129.000
MAXIMUM	1.000	1521.000	275.000	197.000
RANGE	0.320	585.000	84.000	68.000
MEAN	0.840	1347.250	233.000	163.000
STANDARD DEV	0.226	276.086	59.397	48.083
MEDIAN	0.840	1466.000	233.000	163.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-WB
YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	1400.000	6.550	0.200	7.400	0.094
MAXIMUM	2350.000	6.870	0.500	14.700	0.300
RANGE	950.000	0.320	0.300	7.300	0.206
MEAN	1816.667	6.703	0.313	10.850	0.198
STANDARD DEV	350.238	0.136	0.131	2.842	0.103
MEDIAN	1800.000	6.675	0.290	10.600	0.200

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	0.570	927.000	187.000	125.000
MAXIMUM	1.100	1589.000	277.000	199.000
RANGE	0.530	662.000	90.000	74.000
MEAN	0.843	1271.667	225.667	161.333
STANDARD DEV	0.265	273.175	46.318	37.018
MEDIAN	0.860	1277.500	213.000	160.000

THE FOLLOWING RESULTS ARE FOR:

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

DNRSITE$ = 043-WB
YEAR = 98.000

TOTAL OBSERVATIONS: 7

	SC	PH	Cu	Ni	Co
N OF CASES	7	7	7	7	7
MINIMUM	1800.000	6.630	0.080	6.700	0.050
MAXIMUM	2300.000	7.200	0.460	16.700	0.300
RANGE	500.000	0.570	0.380	10.000	0.250
MEAN	2071.429	6.934	0.226	11.371	0.186
STANDARD DEV	188.982	0.208	0.149	4.479	0.118
MEDIAN	2100.000	7.000	0.200	10.300	0.200

	Zn	SO4	CA	MG
N OF CASES	7	7	7	7
MINIMUM	0.590	1157.000	238.000	181.000
MAXIMUM	1.730	1364.000	321.000	213.000
RANGE	1.140	207.000	83.000	32.000
MEAN	1.183	1264.143	274.286	201.000
STANDARD DEV	0.455	64.878	29.279	12.715
MEDIAN	1.240	1262.000	275.000	207.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = 043-WB
YEAR = 99.000

TOTAL OBSERVATIONS: 5

	SC	PH	Cu	Ni	Co
N OF CASES	5	5	4	5	4
MINIMUM	1525.000	4.760	0.533	8.912	0.340
MAXIMUM	2100.000	6.360	1.320	14.700	0.880
RANGE	575.000	1.600	0.787	5.788	0.540
MEAN	1930.000	5.876	0.796	11.622	0.610
STANDARD DEV	230.082	0.648	0.356	2.602	0.221
MEDIAN	2000.000	6.110	0.665	12.100	0.610

	Zn	SO4	CA	MG

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

N OF CASES	4	4	3	3
MINIMUM	1.168	901.000	250.000	199.000
MAXIMUM	1.970	1480.000	282.000	208.000
RANGE	0.802	579.000	32.000	9.000
MEAN	1.607	1268.500	263.333	203.000
STANDARD DEV	0.336	253.194	16.653	4.583
MEDIAN	1.645	1346.500	258.000	202.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Historic
YEAR = 85.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	0	6	6	6	6
MINIMUM	.	4.900	0.420	19.000	1.300
MAXIMUM	.	5.300	0.810	32.000	2.100
RANGE	.	0.400	0.390	13.000	0.800
MEAN	.	5.167	0.615	23.333	1.583
STANDARD DEV	.	0.151	0.164	5.086	0.331
MEDIAN	.	5.200	0.585	21.000	1.400

	Zn	SO4	CA	MG
N OF CASES	6	6	6	0
MINIMUM	1.500	1300.000	150.000	.
MAXIMUM	2.400	1850.000	240.000	.
RANGE	0.900	550.000	90.000	.
MEAN	1.833	1525.000	200.000	.
STANDARD DEV	0.350	227.486	32.863	.
MEDIAN	1.700	1500.000	200.000	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Historic
YEAR = 86.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

N OF CASES	0	6	6	6	6
MINIMUM	.	5.100	0.560	12.000	1.400
MAXIMUM	.	5.600	1.100	25.000	1.800
RANGE	.	0.500	0.540	13.000	0.400
MEAN	.	5.333	0.827	20.333	1.533
STANDARD DEV	.	0.186	0.209	4.761	0.151
MEDIAN	.	5.300	0.840	21.500	1.500

	Zn	SO4	CA	MG
N OF CASES	6	6	6	0
MINIMUM	1.300	1200.000	130.000	.
MAXIMUM	2.200	1600.000	210.000	.
RANGE	0.900	400.000	80.000	.
MEAN	1.800	1391.667	181.667	.
STANDARD DEV	0.400	135.708	29.269	.
MEDIAN	1.950	1400.000	190.000	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Historic
YEAR = 87.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	0	6	6	6	6
MINIMUM	.	5.300	0.440	14.000	1.000
MAXIMUM	.	5.500	1.100	31.000	2.200
RANGE	.	0.200	0.660	17.000	1.200
MEAN	.	5.383	0.747	21.500	1.483
STANDARD DEV	.	0.075	0.238	6.221	0.431
MEDIAN	.	5.400	0.745	20.500	1.500

	Zn	SO4	CA	MG
N OF CASES	6	6	0	0
MINIMUM	1.100	1400.000	.	.
MAXIMUM	2.300	2650.000	.	.
RANGE	1.200	1250.000	.	.
MEAN	1.600	1813.333	.	.
STANDARD DEV	0.429	432.281	.	.
MEDIAN	1.550	1725.000	.	.

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Historic
YEAR = 88.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	0	6	6	6	6
MINIMUM	.	5.200	0.490	15.000	1.000
MAXIMUM	.	5.500	0.810	32.000	2.500
RANGE	.	0.300	0.320	17.000	1.500
MEAN	.	5.300	0.643	25.167	1.900
STANDARD DEV	.	0.110	0.114	5.981	0.569
MEDIAN	.	5.300	0.610	25.500	2.000

	Zn	SO4	CA	MG
N OF CASES	6	6	0	0
MINIMUM	1.500	1275.000	.	.
MAXIMUM	2.600	2100.000	.	.
RANGE	1.100	825.000	.	.
MEAN	1.950	1791.667	.	.
STANDARD DEV	0.394	287.518	.	.
MEDIAN	1.950	1862.500	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Historic
YEAR = 89.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	0	6	6	6	6
MINIMUM	.	4.800	0.800	12.000	0.750
MAXIMUM	.	5.100	1.800	24.000	1.900
RANGE	.	0.300	1.000	12.000	1.150
MEAN	.	4.967	1.383	20.333	1.458
STANDARD DEV	.	0.103	0.360	4.457	0.395
MEDIAN	.	5.000	1.450	21.000	1.500

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

	Zn	SO4	CA	MG
N OF CASES	6	6	0	0
MINIMUM	1.100	765.000	.	.
MAXIMUM	3.000	1725.000	.	.
RANGE	1.900	960.000	.	.
MEAN	2.117	1410.833	.	.
STANDARD DEV	0.634	343.153	.	.
MEDIAN	2.100	1450.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Historic
YEAR = 90.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	0	6	6	6	6
MINIMUM	.	4.700	0.760	11.100	0.840
MAXIMUM	.	5.000	2.100	31.000	1.800
RANGE	.	0.300	1.340	19.900	0.960
MEAN	.	4.850	1.482	18.567	1.332
STANDARD DEV	.	0.105	0.498	7.397	0.414
MEDIAN	.	4.850	1.525	18.500	1.400

	Zn	SO4	CA	MG
N OF CASES	6	6	0	0
MINIMUM	1.500	605.000	.	.
MAXIMUM	2.900	1700.000	.	.
RANGE	1.400	1095.000	.	.
MEAN	2.100	1175.000	.	.
STANDARD DEV	0.551	387.840	.	.
MEDIAN	2.000	1292.500	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Historic
YEAR = 91.000

TOTAL OBSERVATIONS: 8

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

	SC	PH	Cu	Ni	Co
N OF CASES	0	8	8	8	8
MINIMUM	.	4.600	0.780	12.000	0.520
MAXIMUM	.	5.200	2.500	23.000	2.700
RANGE	.	0.600	1.720	11.000	2.180
MEAN	.	4.900	1.520	16.200	1.153
STANDARD DEV	.	0.227	0.560	4.546	0.714
MEDIAN	.	4.900	1.600	15.000	0.990

	Zn	SO4	CA	MG
N OF CASES	8	8	0	0
MINIMUM	0.880	890.000	.	.
MAXIMUM	3.000	1732.000	.	.
RANGE	2.120	842.000	.	.
MEAN	2.026	1425.375	.	.
STANDARD DEV	0.781	371.756	.	.
MEDIAN	2.170	1616.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Inf-1
YEAR = 93.000

TOTAL OBSERVATIONS: 17

	SC	PH	Cu	Ni	Co
N OF CASES	0	11	17	17	0
MINIMUM	.	4.600	0.090	2.600	.
MAXIMUM	.	6.800	1.500	13.000	.
RANGE	.	2.200	1.410	10.400	.
MEAN	.	5.555	0.749	9.041	.
STANDARD DEV	.	0.680	0.378	2.750	.
MEDIAN	.	5.700	0.820	10.000	.

	Zn	SO4	CA	MG
N OF CASES	0	17	0	0
MINIMUM	.	971.000	.	.
MAXIMUM	.	1655.000	.	.
RANGE	.	684.000	.	.
MEAN	.	1297.000	.	.

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

STANDARD DEV	.	173.614	.	.
MEDIAN	.	1329.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = Inf-1
YEAR = 94.000

TOTAL OBSERVATIONS: 3

	SC	PH	Cu	Ni	Co
N OF CASES	0	3	3	2	0
MINIMUM	.	5.300	0.730	6.100	.
MAXIMUM	.	5.800	1.400	9.400	.
RANGE	.	0.500	0.670	3.300	.
MEAN	.	5.500	1.033	7.750	.
STANDARD DEV	.	0.265	0.339	2.333	.
MEDIAN	.	5.400	0.970	7.750	.

	Zn	SO4	CA	MG
N OF CASES	0	3	0	0
MINIMUM	.	1031.000	.	.
MAXIMUM	.	1496.000	.	.
RANGE	.	465.000	.	.
MEAN	.	1306.333	.	.
STANDARD DEV	.	244.050	.	.
MEDIAN	.	1392.000	.	.

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S2
YEAR = 97.000

TOTAL OBSERVATIONS: 6

	SC	PH	Cu	Ni	Co
N OF CASES	6	6	6	6	3
MINIMUM	1950.000	4.430	1.400	13.600	1.200
MAXIMUM	3300.000	4.700	3.700	38.400	2.400
RANGE	1350.000	0.270	2.300	24.800	1.200
MEAN	2591.667	4.495	2.250	22.600	1.600

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

STANDARD DEV	525.753	0.102	0.771	8.374	0.693
MEDIAN	2700.000	4.465	2.050	20.150	1.200

	Zn	SO4	CA	MG
N OF CASES	3	6	3	3
MINIMUM	4.200	1411.000	207.000	192.000
MAXIMUM	4.940	2335.000	294.000	264.000
RANGE	0.740	924.000	87.000	72.000
MEAN	4.613	1989.000	262.667	232.000
STANDARD DEV	0.378	320.809	48.336	36.661
MEDIAN	4.700	2019.000	287.000	240.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S2
YEAR = 98.000

TOTAL OBSERVATIONS: 1

	SC	PH	Cu	Ni	Co
N OF CASES	1	1	1	1	1
MINIMUM	2100.000	4.610	1.750	18.300	1.100
MAXIMUM	2100.000	4.610	1.750	18.300	1.100
RANGE	0.000	0.000	0.000	0.000	0.000
MEAN	2100.000	4.610	1.750	18.300	1.100
STANDARD DEV	.	.	.	.	.
MEDIAN	2100.000	4.610	1.750	18.300	1.100

	Zn	SO4	CA	MG
N OF CASES	1	1	1	1
MINIMUM	2.470	1200.000	219.000	194.000
MAXIMUM	2.470	1200.000	219.000	194.000
RANGE	0.000	0.000	0.000	0.000
MEAN	2.470	1200.000	219.000	194.000
STANDARD DEV	.	.	.	.
MEDIAN	2.470	1200.000	219.000	194.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S2
YEAR = 99.000

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

TOTAL OBSERVATIONS: 2

	SC	PH	Cu	Ni	Co
N OF CASES	2	2	1	2	1
MINIMUM	1900.000	4.470	1.710	4.500	1.080
MAXIMUM	2150.000	4.580	1.710	14.100	1.080
RANGE	250.000	0.110	0.000	9.600	0.000
MEAN	2025.000	4.525	1.710	9.300	1.080
STANDARD DEV	176.777	0.078	.	6.788	.
MEDIAN	2025.000	4.525	1.710	9.300	1.080

	Zn	SO4	CA	MG
N OF CASES	1	2	1	1
MINIMUM	2.540	1141.000	276.000	196.000
MAXIMUM	2.540	1385.000	276.000	196.000
RANGE	0.000	244.000	0.000	0.000
MEAN	2.540	1263.000	276.000	196.000
STANDARD DEV	.	172.534	.	.
MEDIAN	2.540	1263.000	276.000	196.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S3
YEAR = 97.000

TOTAL OBSERVATIONS: 5

	SC	PH	Cu	Ni	Co
N OF CASES	5	5	5	5	3
MINIMUM	2200.000	4.380	1.950	19.500	1.600
MAXIMUM	4000.000	4.510	4.300	54.300	3.700
RANGE	1800.000	0.130	2.350	34.800	2.100
MEAN	2970.000	4.424	2.650	30.260	2.333
STANDARD DEV	676.018	0.051	0.941	13.828	1.185
MEDIAN	2750.000	4.420	2.400	24.600	1.700

	Zn	SO4	CA	MG
N OF CASES	3	5	2	2
MINIMUM	4.100	1678.000	217.000	229.000

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

MAXIMUM	5.990	2959.000	363.000	330.000
RANGE	1.890	1281.000	146.000	101.000
MEAN	4.950	2239.000	290.000	279.500
STANDARD DEV	0.959	485.034	103.238	71.418
MEDIAN	4.760	2182.000	290.000	279.500

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S3
YEAR = 99.000

TOTAL OBSERVATIONS: 1

	SC	PH	Cu	Ni	Co
N OF CASES	1	1	1	1	1
MINIMUM	2750.000	4.350	2.400	21.000	1.640
MAXIMUM	2750.000	4.350	2.400	21.000	1.640
RANGE	0.000	0.000	0.000	0.000	0.000
MEAN	2750.000	4.350	2.400	21.000	1.640
STANDARD DEV	.	.	.	.	.
MEDIAN	2750.000	4.350	2.400	21.000	1.640

	Zn	SO4	CA	MG
N OF CASES	1	1	1	1
MINIMUM	3.910	1847.000	311.000	253.000
MAXIMUM	3.910	1847.000	311.000	253.000
RANGE	0.000	0.000	0.000	0.000
MEAN	3.910	1847.000	311.000	253.000
STANDARD DEV	.	.	.	.
MEDIAN	3.910	1847.000	311.000	253.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S4
YEAR = 97.000

TOTAL OBSERVATIONS: 2

	SC	PH	Cu	Ni	Co
N OF CASES	2	2	2	2	1
MINIMUM	1725.000	4.520	2.900	21.100	1.500

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

MAXIMUM	1850.000	4.630	3.000	23.900	1.500
RANGE	125.000	0.110	0.100	2.800	0.000
MEAN	1787.500	4.575	2.950	22.500	1.500
STANDARD DEV	88.388	0.078	0.071	1.980	.
MEDIAN	1787.500	4.575	2.950	22.500	1.500

	Zn	SO4	CA	MG
N OF CASES	1	2	1	1
MINIMUM	2.040	1196.000	195.000	150.000
MAXIMUM	2.040	1430.000	195.000	150.000
RANGE	0.000	234.000	0.000	0.000
MEAN	2.040	1313.000	195.000	150.000
STANDARD DEV	.	165.463	.	.
MEDIAN	2.040	1313.000	195.000	150.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S5
YEAR = 97.000

TOTAL OBSERVATIONS: 1

	SC	PH	Cu	Ni	Co
N OF CASES	1	1	1	1	0
MINIMUM	2125.000	4.560	5.700	27.000	.
MAXIMUM	2125.000	4.560	5.700	27.000	.
RANGE	0.000	0.000	0.000	0.000	.
MEAN	2125.000	4.560	5.700	27.000	.
STANDARD DEV	.	.	.	.	.
MEDIAN	2125.000	4.560	5.700	27.000	.

	Zn	SO4	CA	MG
N OF CASES	0	1	1	1
MINIMUM	.	1572.000	287.000	164.000
MAXIMUM	.	1572.000	287.000	164.000
RANGE	.	0.000	0.000	0.000
MEAN	.	1572.000	287.000	164.000
STANDARD DEV	.	.	.	.
MEDIAN	.	1572.000	287.000	164.000

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S5
YEAR = 99.000

TOTAL OBSERVATIONS: 1

	SC	PH	Cu	Ni	Co
N OF CASES	1	1	1	1	1
MINIMUM	2225.000	4.580	4.320	23.500	2.020
MAXIMUM	2225.000	4.580	4.320	23.500	2.020
RANGE	0.000	0.000	0.000	0.000	0.000
MEAN	2225.000	4.580	4.320	23.500	2.020
STANDARD DEV	.	.	.	.	.
MEDIAN	2225.000	4.580	4.320	23.500	2.020

	Zn	SO4	CA	MG
N OF CASES	1	1	1	1
MINIMUM	2.140	1426.000	219.000	200.000
MAXIMUM	2.140	1426.000	219.000	200.000
RANGE	0.000	0.000	0.000	0.000
MEAN	2.140	1426.000	219.000	200.000
STANDARD DEV	.	.	.	.
MEDIAN	2.140	1426.000	219.000	200.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S6
YEAR = 97.000

TOTAL OBSERVATIONS: 2

	SC	PH	Cu	Ni	Co
N OF CASES	2	2	2	2	1
MINIMUM	1325.000	4.520	3.480	14.600	1.200
MAXIMUM	1490.000	4.620	3.900	14.700	1.200
RANGE	165.000	0.100	0.420	0.100	0.000
MEAN	1407.500	4.570	3.690	14.650	1.200
STANDARD DEV	116.673	0.071	0.297	0.071	.
MEDIAN	1407.500	4.570	3.690	14.650	1.200

	Zn	SO4	CA	MG

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

N OF CASES	1	2	1	1
MINIMUM	1.180	985.000	183.000	76.500
MAXIMUM	1.180	1063.000	183.000	76.500
RANGE	0.000	78.000	0.000	0.000
MEAN	1.180	1024.000	183.000	76.500
STANDARD DEV	.	55.154	.	.
MEDIAN	1.180	1024.000	183.000	76.500

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S6
YEAR = 99.000

TOTAL OBSERVATIONS: 1

	SC	PH	Cu	Ni	Co
N OF CASES	1	1	1	1	1
MINIMUM	1825.000	4.510	4.590	16.100	1.520
MAXIMUM	1825.000	4.510	4.590	16.100	1.520
RANGE	0.000	0.000	0.000	0.000	0.000
MEAN	1825.000	4.510	4.590	16.100	1.520
STANDARD DEV	.	.	.	.	.
MEDIAN	1825.000	4.510	4.590	16.100	1.520

	Zn	SO4	CA	MG
N OF CASES	1	1	1	1
MINIMUM	1.660	1135.000	274.000	112.000
MAXIMUM	1.660	1135.000	274.000	112.000
RANGE	0.000	0.000	0.000	0.000
MEAN	1.660	1135.000	274.000	112.000
STANDARD DEV	.	.	.	.
MEDIAN	1.660	1135.000	274.000	112.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S7
YEAR = 97.000

TOTAL OBSERVATIONS: 3

	SC	PH	Cu	Ni	Co

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

N OF CASES	3	3	3	3	2
MINIMUM	1650.000	4.350	4.100	15.000	1.700
MAXIMUM	2600.000	4.570	7.300	22.700	2.000
RANGE	950.000	0.220	3.200	7.700	0.300
MEAN	2166.667	4.453	5.900	19.167	1.850
STANDARD DEV	480.451	0.111	1.637	3.889	0.212
MEDIAN	2250.000	4.440	6.300	19.800	1.850

	Zn	SO4	CA	MG
N OF CASES	2	3	1	1
MINIMUM	2.220	1201.000	240.000	100.000
MAXIMUM	2.400	1889.000	240.000	100.000
RANGE	0.180	688.000	0.000	0.000
MEAN	2.310	1613.667	240.000	100.000
STANDARD DEV	0.127	363.980	.	.
MEDIAN	2.310	1751.000	240.000	100.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S7
YEAR = 99.000

TOTAL OBSERVATIONS: 1

	SC	PH	Cu	Ni	Co
N OF CASES	1	1	1	1	1
MINIMUM	2675.000	4.370	6.680	28.400	2.510
MAXIMUM	2675.000	4.370	6.680	28.400	2.510
RANGE	0.000	0.000	0.000	0.000	0.000
MEAN	2675.000	4.370	6.680	28.400	2.510
STANDARD DEV	.	.	.	.	.
MEDIAN	2675.000	4.370	6.680	28.400	2.510

	Zn	SO4	CA	MG
N OF CASES	1	1	1	1
MINIMUM	2.480	1777.000	380.000	206.000
MAXIMUM	2.480	1777.000	380.000	206.000
RANGE	0.000	0.000	0.000	0.000
MEAN	2.480	1777.000	380.000	206.000
STANDARD DEV	.	.	.	.
MEDIAN	2.480	1777.000	380.000	206.000

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S8
YEAR = 97.000

TOTAL OBSERVATIONS: 1

	SC	PH	Cu	Ni	Co
N OF CASES	1	1	1	1	0
MINIMUM	1900.000	4.700	1.400	10.900	.
MAXIMUM	1900.000	4.700	1.400	10.900	.
RANGE	0.000	0.000	0.000	0.000	.
MEAN	1900.000	4.700	1.400	10.900	.
STANDARD DEV	.	.	.	.	.
MEDIAN	1900.000	4.700	1.400	10.900	.

	Zn	SO4	CA	MG
N OF CASES	0	1	1	1
MINIMUM	.	1496.000	230.000	196.000
MAXIMUM	.	1496.000	230.000	196.000
RANGE	.	0.000	0.000	0.000
MEAN	.	1496.000	230.000	196.000
STANDARD DEV	.	.	.	.
MEDIAN	.	1496.000	230.000	196.000

THE FOLLOWING RESULTS ARE FOR:
DNRSITE$ = S9
YEAR = 97.000

TOTAL OBSERVATIONS: 1

	SC	PH	Cu	Ni	Co
N OF CASES	1	1	1	1	1
MINIMUM	2300.000	4.460	1.800	18.100	1.200
MAXIMUM	2300.000	4.460	1.800	18.100	1.200
RANGE	0.000	0.000	0.000	0.000	0.000
MEAN	2300.000	4.460	1.800	18.100	1.200
STANDARD DEV	.	.	.	.	.
MEDIAN	2300.000	4.460	1.800	18.100	1.200

Table A2.9. Summary statistics of Seep 1 data broken down by year, and using only data from June through August (continued).

	Zn	SO4	CA	MG
N OF CASES	1	1	0	0
MINIMUM	3.500	1577.000	.	.
MAXIMUM	3.500	1577.000	.	.
RANGE	0.000	0.000	.	.
MEAN	3.500	1577.000	.	.
STANDARD DEV	.	.	.	.
MEDIAN	3.500	1577.000	.	.

Note: pH values are in standard units, specific conductance values are in microsiemens, and all other values are mg/L.

Appendix 3

Stockpile Information

Page

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Section A3.2. Seep 1 wetland treatment system and the 8013 stockpile A3.4

Table A3.1. Stockpile characteristics A3.6

Table A3.2. Watershed estimates A3.7

Section A3.1.

W1D Wetland Treatment System and the 8018 Stockpile

General History of the 8018 Stockpile

The 8018 was started in 1979. The stockpile would receive Duluth Complex material that contained more than about 0.20% CuO or 0.08% NiO. Most of the stockpile would be placed on upland soils, and any wetland areas would be covered with waste rock that did not contain copper or nickel prior to stockpiling mineralized Duluth Complex.

Both surface and groundwater samples were collected prior to stockpile construction, but only surface water continues to be sampled at the W1D weir site.

Chemical composition of the 8018 was determined by analysis of drill hole samples. Prior to blasting, a series of holes were drilled. The cutting from each of these holes was collected and analyzed, and provides an estimate of the composition of each blast.

The average original composition of the stockpile was 0.28% CuO, 0.09 NiO, and 0.98% S. Based on previous laboratory and field studies with Duluth Complex material, it was believed that this stockpile would eventually produce acid drainage. If acid conditions developed, the overland wetland treatment system would not effectively treat the seepage.

As part of the closure plan, LTV selected stockpile caps based on the potential for long-term water quality problems. Since this pile could potentially produce acid, a 30 mil synthetic membrane LLDPE was selected as the cap. The stockpile was not designed to be capped at closure, and was built with angles of repose side slopes (slope ~ 1:1). These slopes could not be covered, so LTV initiated a program to cover and extend these slopes with inert material. Reject material (cobbles and larger) from the screening of surface material were used to extend the stockpile to eliminate the bottom lift. A silty material was then added as the final cover. The goal was to extend the side slopes far enough so that the top cover would extend over all the mineralized material. This was completed in 1992. Based on a visual inspection, about 50 percent of the side slopes were covered with the silty material.

Water that infiltrates the side slopes will not come in contact with any mineralized material and by covering the slopes with soil, the entry of oxygen into the pile can be restricted. In large rock stockpile, oxygen can enter at the toe through the coarse rubble zone and move upward through the pile like a chimney. The fine soil layer can reduce this effect and slow the oxygen supply to the pile. Capping of the top of the stockpile with low density polyethylene was completed September 1, 1995.

History of 8018 Stockpile

May 1979: Construction of stockpile begins

November 1979: Initial water and peat samples collected (for background see Appendix **xx**)

1992: Reject cobbles from screening operation at 8006 stockpile placed on second lift of stockpile to create single lift – eastern side slope eliminated grade southern half of lift 2 to life 1 cover eastern side slope with silty soil, based on visual estimates about 50% of the slope was covered

1995: Install flexible membrane cover (LLDPE, linear low density polyethylene, 30 mil)

History of W1D Wetland Treatment System

1992: System was constructed in March – went on-line spring melt.
Modifications to system (during flow season):
- rip raps placed on first berm
- installed 4x40 treated lumber across berms, 2, 3, 4
- installed pretreatment system

1992: A pretreatment system was installed upstream of 051 (input to the wetland system). An attempt was made to reduce channelization at each of the berms by placing wood beams to across each dike. The peat in the system was also regraded to correct channelization.

1993: Replaced peat in last 4 cells. (press to melt) Added drop pipe. Raised berm 5 (larger glacial till berm). Raised berm to decrease head in upper section. Several berms at the downstream end of the system utilized a different peat mixture (Michigan/Peatrex peat), LTV felt that this mixture did not provide good metal removal. These peats were replaced with a peat mixture consisting of one-part processing screenings (sphagnum) and one-part Mining Area 2WX peat (reed sedge).

1994: Hay bale added to two berms near outflow of system in June (see field notes from Dunka visit, Appendix 9).

More detailed information on the initial construction and monitoring can be found in the DNR report "The Environmental Leaching of Stockpiles Containing Copper-Nickel Sulfide Minerals" (DNR 1981).

Section A3.2.

Seep 1 Wetland Treatment System and the 8013 Stockpile

Seep 1 is located at the NE corner of the top of stockpile 8013. 8013 is a waste rock stockpile (0.09% CuO, 0.03% NiO, 0.24% S) that was started in 1967 and completed in 1991. At the time of the completion it contained approximately 17,750,000 long tons of material, and covered 2,300,000 ft^2 (52.8 acres; 21.37 ha).

The seep exits the toe of the 8013 over a wide area in the form of many small seepages, which originally formed a large wet area adjacent to 8013. In 1992, a peat/limestone pretreatment system was constructed to capture this diffuse seepage and treat it, and in 1995 a wetland treatment system was constructed to receive the effluent from the pretreatment system and provide further treatment.

October 1974:	8013 very small, growing from west, not in Seep 1 watershed yet.
Prior to April 1976:	Entire first lift of 8013 complete.
1977-1980:	Filing first lift on south end of stockpile (out of watershed).
1981:	Started second lift from south (small unit in watershed).
1982-1985:	90% of second life completed.
1986-1988:	No activity.
1989:	First ditching activity intended to influence flow rate at Seep 1 (Closure Report, Vol. 1, p. 63).
1990-1991:	The ditching of Unnamed Creek past the grassy meadow and through the bedrock outcrop occurred, and cut-off ditches were constructed on both sides of the grassy meadow.
May 1991:	Remainder of second lift (NW area) had been added by this time. This remainder was probably added late 1990 or early 1991.
Fall 1991:	Capping completed.
Spring 1992:	Stockpile hydroseed was fertilized. Barrier layer compacted -1/2 inch screened till from 8006 stockpile; cover -3 inches glacial till.

August 11, 1992:	Peat/limestone pretreatment system was started. This is located between 8013 and the weir. The site historically called 043 (i.e., the raw seepage) is now located at the effluent of the pretreatment system, and can, therefore, not be considered representative of raw, untreated Seep 1 seepage. 8013 was capped in the fall of 1991 by local contractors with compacted fill with a runoff drainage conveyance system with discharge culverts to collect and convey precipitation off the stockpile. Two basic materials were used: 3 inch compacted till, 1/2 inch till (see page 4 of "review of 8013"). 8013 was then planted with oats and a standard mineland reclamation mix of grasses and clover to increase evaportranspiration.
June 1994:	Three hay bales were scattered in the pretreatment system to encourage sulfate reduction.
August 1994:	Hay bales were placed above the vertical down-flow section.
1996:	Peat samples collected from pretreatment system
1997:	DNR started monitoring program of pretreatment system. Peat samples collected from pretreatment system (see detailed timeline, Appendix 6).

Table A3.1. Stockpile characteristics.

Stockpile	Type	Size - Footprint (acres)	Mass (long tons)	Chemical Composition			Deposition History	
				% CuO	% NiO	% S	Start	End
8013	Waste rock	78	17,750,000	0.09	0.03	0.24	1967	1991
8018	Gabbro	18	2,200,000	0.28	0.09	0.98	1979	1985

All data except footprint is from the Dunka Closure Plan submitted 2/11/93.

The footprint of each stockpile was calculated by Steve Dewar from a May 1991 air photo using a dot grid.

Chemical data for the 8018 stockpile was based on drill hole cutting chemical analysis for almost the entire stockpile and should be quite accurate.

Chemical data for the 8013 stockpile was based on drill hole cutting chemical analysis. No drill cutting analysis for sulfur was available prior to 1980 so the sulfur composition of the 8013 is based only on data after 1980. Prior to 19809 some samples were analyzed to determine if the material was "waste rock" < 0.20% CuO or "gabbro" 0.20% CuO. Hornfels, which are higher in sulfur S > 1%, but low in copper and nickel and are acid generating, have been observed on the slope of the 8013 stockpile and likely contribute to the low pH of the small surface seeps.

Table A3.2. Watershed estimates.

Seep #	Pre-Ditching	Post-Ditching	% Reduction	Stockpile	Pre-Ditching	Post-Ditching
	Watershed size (10^6 ft^2)	Watershed size (10^6 ft^2)		Watershed Under Stockpile	Watershed Under Stockpile	Watershed Under Stockpile
EM8	16.6	10/7.7[a]	40/54[a]	8011	3.6	3.6
				8014	1.9	1.9
				8012	0.5	0
Seep 3	1.9	1.9	0	8013	0.35	0.35
				8014	0.4	0.4
Seep X	1.7	1.7	0	8013	1.7	1.7
Seep X (likely)	2.4	2.1	13	8013	2.1	2.1
Seep 1 (low)	.47	.47	0	8013	.35	.35
Seep 1 (high)	.82	.82	0	8013	.65	.65
W1D (historieal) weir (input to wetland)	1.9	1.9	0	8018	.75	.75
				8031	.35	.35
W1D, Original Outflow[b] of first wetland	1.4	1.4	0	8031	.10	.10
				8018	.10	.10
				8022	.4	.4
EM1	99	89.5	10			

[a]First value is the result of ditching the environmental sump. The second value is with the addition of the ditch along the south side of 8011.
[b]This is the additional watershed area that contribute to outflow of the wetland. (This does not include the 1995 extension.) To get the total watershed area add the W1D historical value.

Appendix 4

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Section A4.1. Collection of frozen and unfrozen peat cores/samples at W1D, Dunka A4.2

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Section A4.1.
Collection of frozen and unfrozen peat cores/samples at W1D, Dunka

DEPARTMENT: Natural Resources
Division of Minerals

STATE OF MINNESOTA

Office Memorandum

DATE: March 26, 1996

TO: Paul Eger, Principal Engineer

FROM: Glenn Melchert, Hydrologist

PHONE: (218) 262-7343 FAX: (218) 262-7328

SUBJECT: Collection of frozen and unfrozen peat cores/samples at W1D, Dunka.

Core R1-4 was collected on 3/13 and the rest of rows 1 and 4 were collected on 3/18/96. About 3' of snow covered the area. Mike Ellett provided assistance.

The numbering convention was four rows (lines) from berm 2 to berm 3 numbered in order with row 1 being the farthest across the wetland and row 4 being closest to the parking/road area. There were five sample sites along each row, labeled so that the most upstream site was 1 and the farthest downstream site was 5 (see attached figure).

A 300' tape was stretched between the rebar stakes at either end of the berms. Then the locations of each row were staked. The spacing was 13.1 m, 26.15 m, 39.2 m and 48.2 m along berm 2 and 13.4 m, 26.8 m, 40.2 m and 49,4 m along berm 3. A rod was used to assure the stakes were directly over the wooden plank running the length of each berm. These stakes were used to align the sample locations along each row. Samples 1 and 2 were collected 10 ft. upstream and 10 ft. downstream of berm 2. Samples 4 and 5 were located similarly on either side of berm 3. Sample 3 was the midpoint between the berms.

H_2S odors were frequently observed. The odors were the strongest in areas where there was ice or water above the peat. This was common at sites 3, 4 and 1. The odor usually was not detected until after augering through the ice.

Sample R1-2 was sampled 15 ft. downstream of berm 2 instead of 10 ft. Later, the stake marking this site was found. The stake was located 21' downstream and no sample was taken at that site.

Sample R1-3 and R4-3 were sampled 25.0 m and 13.5 m, respectively, downstream from berm 2.

Sample #	Ice/H_2O Above Peat (cm)	Sample Collected[1] (cm)	Sampling Device	Comments
R1-1	40 cm	0-10 10-20 20-30	M.C. M.C. M.C.	no roots no roots no roots
R1-2	4 cm	0-18 20-30	C.B. M.C.	roots
R1-3	15-20	0-10 10-20 20-30	M.C. M.C. M.C.	no roots no roots no roots
R1-4	11 cm	0-18	C.B.	roots throughout/gravel at 18 cm.
R1-5	0	0-10	C.B.	roots in upper half/tree roots stopped auger at 10 cm.
R4-1		0-13 13-20 20-30	C.B. M.C. M.C.	roots throughout
R4-2	0	0-14 14-20 20-30	C.B. M.C. M.C.	roots throughout silt at 40 cm.
R4-3	4-13	0-13* 13-23 23-33 33-43	C.B. M.C. M.C. M.C.	clear ice and duckweed mats few to no roots/13 cm. is top of peat
R4-4	15	0-10 10-20	M.C. M.C.	no roots no roots/silt at 22 cm.
R4-5	4	0-23* 20-30	C.B. M.C.	cattail and some roots/top 4 cm. is clear ice silt at 30 cm.

[1]Measured from top of peat.
*Includes ice at top of core. Must be removed.
M.C. = Macaulay sampler. Sample typically is composite of 2 cores.
C.B. = Power auger core barrel. 4" diam.

cc: Jon Wagner

Section A4.2. Peat Sampling - W1D and Seep 1 Pretreatment

DEPARTMENT: Natural Resources
Division of Minerals

STATE OF MINNESOTA

Office Memorandum

DATE: April 22, 1996

TO: Paul Eger, Principal Engineer

FROM: Glenn Melchert, Hydrologist

PHONE: (218) 262-7343 FAX: (218) 262-7328

SUBJECT: Collection of additional peat cores/samples on April 4, 1996, at W1D and Seep 1, Dunka

There was about two feet of snow on the wetland. Most of the snow was firm enough to walk on. Although the snow pack has decreased by about 1/3, no significant runoff has occurred yet. Mike Ellett assisted with sampling.

The sample numbering convention is the same as indicated in my March 26, 1996, memo.

Samples R2-3 and R3-3 were sampled 22.0 m and 17.8 m (mid-points), respectively, downstream from berm 2.

Three samples were collected at the Seep 1 pre-treatment under drain bed. Sites were spaced in a row 6 feet apart with site 3 being about 6 feet from the toe of the dam (see attached figure).

Flow was not detected at either wetland.

GM:djm

cc: J. Wagner

Sample #	Ice/H_2O Above Peat (cm)	Sample Collected[1] (cm)	Sampling Device	Comments
R2-2	0	0-20 20-30	C.B. M.C.	Unfrozen, grassy area, water at surface of peat. Roots in top 12 cm.
R2-3	15	0-10 10-20 20-30	M.C. M.C. M.C.	Strong H_2S after breaking through ice.
R2-4	21	0-10 10-20 20-30	M.C. M.C. M.C.	Occasional white hair-like roots from 0-30 cm.
R3-2	0	0-20 20-25	C.B. Grab	Site has cattails, grass and dogwood. Grass and cattail roots in sample. Some roots. Gravel at 25 cm.
R3-3	14	0-10 10-20 20-30	M.C. M.C. M.C.	Strong H_2S after breaking through ice. No roots. No roots. No roots.
R3-4	10	0-10 10-20 20-28	M.C. M.C. M.C.	Sparse roots 0-28 cm. Gravel at 28 cm.
S1-1 (Seep 1)	40	0-10 10-20	M.C. M.C.	Peat. Has yellow-green blobs at surface. Has some quartz and feldspar grains. Black peat, rock at 22 cm.
S1-2	10	0-10 10-20 20-30 30-40	M.C. M.C. M.C. M.C.	Hay. Hay. Hay/peat mix. Peat. Rock at 60 cm.
S1-3	24	0-25* 15-40	C.B. M.C.	Top 10 cm. is ice. Bottom 15 cm. is hay. Mostly hay. Rock at 42 cm.

[1] Measured from top of peat except where *.
* Includes ice at top of core which must be removed.
M.C. Macaulay sampler. Sample typically is composite of 2-3 cores.
C.B. Power auger core barrel. 4" ID.

DEPARTMENT: Natural Resources
Division of Minerals

STATE OF MINNESOTA

Office Memorandum

DATE: May 15, 1997
March 13, 1998

TO: Paul Eger, Principal Engineer

FROM: Glenn Melchert, Hydrologist

PHONE: (218)262-7343 FAX: (218)262-7328

SUBJECT: Peat sampling: W1D and seep 1 pretreatment. April 11, 14, 17, 24 and May 1, 1997

I was assisted by Julie McCormick on April 24 and May 1.

A major runoff event from snow melt occurred April 4-6. This was followed by a cold snap on April 7 & 8 where lows were near 0°F. By April 17, most of the ice was gone from W1d except near site 3 which still had about 6" of ice (this area receives less sun). Sites 1, 4, and 6 are about 1 meter upstream of a silt fence. Site 11 is at the edge of the upland peninsula. All samples were collected at intervals of 10 cm with a Macaulay sampler.

W1D. On April 11, ice was thickest at the southeastern portion (sites 2, 3, 5), around 12 inches or more, and was thinnest or absent near point A where the flow occurs. Most of the ponded area was covered with a medium density of cattails. Holes about 20 cm in diameter were chiseled to give access for sampling with the Macaulay sampler. The diameter of the Macaulay sample chamber was 4.0 cm in diameter and 50 cm long. The leading drive point was about 10 cm long -- the sample chamber began above this. Usually 4 samples of each interval were composited to give a wet sample weight of at least 70 grams. Only near sites 2, 3, and 5 and 6, 9, 11, did the ice extend into the upper portion of the peat. For site 9 and 11, there was a few cm of ice then frozen peat. Below the surface ice the peat was completely unfrozen. H_2S was detected at nearly every site after a hole was chopped through the ice.

The total peat thickness at the various sites indicates there was probably less than a foot of peat that existed naturally prior to construction of the system. Two feet of loose, fresh peat, more or less, was placed over the existing soil when the system was built. At some sample locations, black peat was encountered just above the mineral sediments. The black peat is interpreted to be original peat/soil. This was found at sites 2 (5 cm thick), 5 (10 cm thick), and 13 (25 cm thick).

Metals analyses of the black peat indicates substantially higher levels than the peat immediately overlying it. This phenomenon also occurred at site 7.

Site 4 appears to be an area that washed out shortly after construction and was subsequently filled with hay bales and more peat.

May 15, 1997
March 13, 1998

Seep 1. No H_2S was detected at any of the sample sites except for a minor amount at sites 4, 7, and 16. Upon turning the Macaulay, air bubbles up welled around the sampler at most of the sites. The greatest amount of bubbles were at sites 4 and 7. Most of the bubbles are believed to be H_2S, but is only detectable immediately downwind of the site due to sparse vegetation.

On April 11, water levels had dropped back to near normal. Perched ice ledges were observed in the pretreatment pool indicating approximate maximum water depth from snow melt a few days prior. Staff gauge reading was 1.44'. The perched ice mark was at 2.70', which is about 0.6' below the overflow spill notch. There was not any perched ice in the weir pool.

By April 17, the northeast pool (site 1) and along the east edge, the ice was less than an inch thick, whereas the west edge along the stockpile was over a foot thick.

On April 24, there was still thick ice on the west and southern portions of the outer pool. By May 1, all the ice had melted.

Generally, some of the white precipitate (up to about 2 cm) overlying the peat was included with the 0-10 cm sample.

GM/djm

WID Sample Sites.

Sample #	Date	Sample Device	Water Depth (cm)	Peat Thickness (cm)	Peat Intervals Sampled	Comments (Depths are from top of peat)
1					NS	
2	4/14/97	MC	17	31	0-30	Gray silt, peat has brown tint from 10-27 cm. Black peat below 27 cm.
3	4/17/97	MC	23	10	0-10	H_2S odor. Brown gravel at 10 cm. Numerous cattails. Numerous fine roots 0-10 cm.
4	4/14/97	MC	15	47±	0-40	Reed canary and cattails. This is an area of primary flow although not channelized. High H_2S. Hay? from 25-40 cm. Brown peat 18-25 cm. Sample brown peat 20-28 cm. (separate sample). Resistance at 47 cm.
5	4/11/97	MC	26	30-44	0-40	Rock at 30 cm. Few roots. Black peat with woody roots at 35-45 cm.
6	4/11/97		10±		NS	
7	4/11/97	MC	19	58	0-58	Gray brown silt at 58 cm. Few roots in upper 10 cm. Lowest sample is 40-58 cm.
8	4/11/97		20±		NS	
9	4/11/97		2±		NS	
10	4/11/97	MC	22	53	0-30	Rock at 53 cm. Few to no roots. Wood fragments 0-30 cm. Sparse cattails.
11	4/11/97		2±		NS	Edge of high ground.
12	4/11/97	MC	15	59	0-30	Rock at 59 cm.
13	4/11/97	MC	14	71-76	0-76	Brown silt at 76 cm. Numerous fibrous white roots 0-20 cm. Few roots 20-30. Black peat with plant fragments at 50-75 cm. Cattail patch.
14	4/11/97	MC	22	48	0-30	Few to no roots. Soupy. Little recovery from 0-10 cm. Rock at 48 cm. This is the same site as R2-3 sample (#20) from 1996.

MC = Macaulay sediment sampler

Seep 1 Sample Sites -- Pretreatment Bed.

Sample #	Date	Sample Device	Water Depth (cm)	Peat Thickness (cm)	Intervals Sampled	Comments (Depths are from top of peat)
1	4/17/97	MC	28	57	0-50	No ice here. Waded from north bank. 3 cm. of creamy precipitate. Sparse to medium cattails with few other plants. Sand at 57 cm. Took ppt sample.
2	4/17/97	MC	42	26	0-6 ppt[a] 6-10 peat 10-20 20-26	Upper 6 cm. is cream, orange and brown precipitate. Mostly cream colored. Brown material is fine sand comprising 25% by volume. Some roots 10-20 cm. 2 cm. cream blob at 10 cm. probably pulled down by sampler.
3	4/17/97	MC	42	48	0-48	2-3 cm. precipitate at surface. Brown sand at 48 cm. Moderate cattails. Spare roots 0-20 cm.
4	4/17/97	MC	42	39	0-39	6 cm. precipitate at surface. Upper 5 cm. is cream colored then 1 cm. orange layer adjacent to black peat/goo for 1 cm. then fibrous peat. Lots of wood at 25-40 cm. Gray sand at 39 cm.
5					NS	
6					NS	
7	5/1/97	MC	19-24	46	0-46	Site midway between seeps 5 & 6 and 70 cm. from waters edge. Gray coarse sand at 46 cm. 2 cm. of cream colored ppt with orange surface coat. Medium cattails, few roots at 10-20 cm. Numerous H2S bubbles occurred when sampler penetrated near the peat/mineral interface.
8	5/1/97	MC	23	44	0-40	Near seep 4. Area has about 5 cm. ppt like #7. Sparse to no vegetation. Gray sand at 44 cm. Gray silt 20-21 cm. discarded. Peat often washed out at 18-26 and 34-40 cm. due to sticks in upper 10 cm. Some ppt contamination in 30-40 cm.
9					NS	

Sample #	Date	Sample Device	Water Depth (cm)	Peat Thickness (cm)	Intervals Sampled	Comments (Depths are from top of peat)
10	4/14/97	MC	53	19-30	0-30	Black peat with fibrous roots decreasing to sparse roots at 30 cm. Occasional white blobs 0-20 cm (probably vertical mixing from MC). Peat at 10-30 cm. had brown tint. Cattails. Very little wood in the peat compared to W1d.
11	4/14/97	MC	32	42	0-40	Roots in top 15 cm. Silt at 42 cm.
12	4/14/97	MC	34	61	0-50	Roots 0-20 cm., sparse cattails. Lots of orange flocculent stirred up. Some had deposited on lower ice layer. Silt at 61 cm.
13	4/14/97	MC	38	41	0-40	0-10 cm is darker with roots. Generated air bubbles when turning the sampler causing suspension of white flocculent. Gray sand/silt at 41 cm.
14	4/14/97	MC	53	11	0-11	Black muck. One sample had cream colored amorphores globs in the upper 2-3 cm. In a separate sample, the globs spilled as sampler was pulled from the water. Orange-brown silt/clay at 11-13; gray silt/clay 13-16. 2 ppt samples. Brown sand at 16-20. Grades to dark gray sand/silt from 20-24 cm.
15	4/17/97	MC	32	23	0-20	Small amount of precipitate on peat. Medium stand of cattails (1-1.5' spacing). Hard gray sand/silt.
16	4/24/97	MC	8	67	0-67	Thick cattail patch. Gas bubbling when sampling, H2S odor. Heavy roots to 7 cm. Sparse roots 7-24 cm. Peat was loose and wet from 50-67 cm. Gray sand. 0-10 cm. interval is grab sample. Samples taken 1.3-1.6 m. from berm center. Exposed portion of berm was about 1 m. wide.
17	4/24/97	MC	12	78	0-78	Medium cattail patch. Sparse roots 0-20 cm. Dirty sand layer at 64-68 cm. Gray to brown sand/silt. Sampled 1.6-1.9 m. from berm center. Berm was .95 m. wide. Ppt sample.

[a]split in field

Note: There was some whitish precipitate in most of the samples. The precipitate was colloidal, so tended to stick to substrate, so it is generally included in samples.

Section A4.3.
Seep 1 - Notes for Peat Samples

Samples in Vertical Down-flow Section

The power auger was used to cut through the hay and collect the substrate samples. The hay was black below the water, but stems were still visible. On a volumetric basis, the stems made up the majority of the sample. The blackest material was probably a combination of decomposition and precipitation and comprised most of the -80 mesh materials.

The samples were dried and crushed manually with a mortar and pestle. Normally substrate samples are processed in a Wiley mill, but this would have turned all the stems into a -80 mesh powder and would tend to lower the metal concentrations. Each sample was processed by the same person and each sample was ground approximately the same (visual observation).

Samples were screened to -80 mesh, and most of the hay stayed in the +80 mesh fraction and was discarded. Several samples of the +80 mesh fraction were analyzed. Metal concentrations were lower than in the -80 mesh material. Nickel concentrations ranged from 4082 to 5235 mg/Kg in the +80 mesh, and from 6510 to 9998 mg/Kg in the -80 mesh material.

April 1996 Seep 1 peat samples

Sample	Depth	Lab #	Calculated +80 Fraction (%)
S1-1	0-10	10627	52
	10-20	10628	60
S1-2	0-10	10629	57
	10-20	30	56
	20-30	31	47
	30-40	32	47
S1-3	0-10	53	61
	10-15	54	61
	15-40	10633	74

Sample 10656 and 10657 are +80 fractions for 51-2 (0-10) and 51-2 (10-20), respectively.

The % are up to 5% higher due to probably losses during sieving that is not accounted for here because the +80 was not actually weighed, but calculated as a residual.

Table A4.1. Peat samples, comparison of metal levels in +80 mesh with -80 mesh size fraction.

Row	Sample		-80 mesh					+80 mesh					
	Lab IO	#	Depth	Ni	Cu	Co	Zn	Sample #	Depth	Ni	Cu	Co	Zn
S1	10629	2	0-10	6510	6034	995	1626	10656	0-10	4082	3838	769	1098
S1	10630	2	10-20	9998	8392	1397	2620	10657	10-20	5235	4713	1178	1619

Section A4.4.
Dunka Peat Processing

Dunka Peat Processing 1996 (W1d & Seep 1) 3/98

- Use Mettler electronic scale in lab.

Note: Samples consist of 1) core barrel samples : 4" dia core 10 cm long collected frozen -- approx. 750 g. and 2) Macauley samples: composite unfrozen cores collected "field moist" meaning free water has drained off. Approx. 40-60 g samples.

Processing Procedures:

1) Thaw enough samples for 1 or 2 days work.
2) The core barrel samples need to be quartered lengthwise using a sharp knife or the clean bandsaw. Place one quarter into a labeled baggie and refreeze the remaining core. (Original plans were to blend the wet peat in a blender, but this did not work -- probably due to insufficient water content).
3) Use entire contents of Macauley (1 quart baggie) samples.
4) Take first sample. Remove large roots, stones and wood.
5) Squeeze baggie to thoroughly mix blended peat.
6) Withdraw a sample -- about 20 grams (range 20-22 g.) -- use 40 g. for large samples -- and place in small aluminum pie tin. (Weigh pie tin prior to this (to 0.00 g.) and record tin weight on tin lip.) Precisely weigh peat and tin together and record on data sheet. Set aside to place in oven.
7) Sequential sample. Using a plastic boat, tare this to zero on electronic scale. Place exactly 10.00 g. (range 9.90-10.10) of wet peat into the plastic boat. Then pour the peat into a new quart size baggie. Seal baggie after removing all air and freeze. Pre-label baggie with 1) sample date, 2) sample site (e.g. R1-4 10-20 cm.), and 3) actual weight of peat (e.g. 10.05 g.)
8) pH sample. Leave remaining sample in original baggie, remove all air and seal. Then freeze.
9) Repeat above process for each sample (do only enough for 1 days batch).
10) Take tins with peat and place in oven (in lab). Set temp. at 105° C., turn on and let dry for 24 hours. Ask for assistance with oven. Place all of the tins from the day in at the same time.
11) After 24 hours remove tins from oven, weigh peat and tin together and record. Place dry peat in brown envelope and label with lab # only. Calculate the peat dry weight by subtracting the "tin wt." from the "peat and tin dry wt.")
12) Place dry peat in blender and mix. If sample volume is too small, blender process will not work, which was the case for spring 1996 samples. Instead, dry peat was manually ground in a large porcelain mortal and pastel. Every sample was ground to approximately the same consistency. The only problem occurred in a few samples that had 1-3 mm size sand grains, which made it difficult to crush all the material. Note which samples have sand or pebbles. Return material to the brown envelope.

Sample for Ashing

13) If dry peat is less than 2.1 g., then remove 0.500 g. of peat from the envelope and place into the white crucible. If peat weight is more than 2.1 g., then remove 1.00° g. and place into the

white crucible.

- Procedure for removing peat sample from the envelope: Hold envelope so all the peat goes to one corner. Take the stainless steel spatula and obtain a representative sample by pushing the spatula to the bottom of the envelope and withdraw the sample. This way there should be some heavy and some light peat on the spatula. Weigh the empty white ashing dish and record its weight to 3 decimals. Then tare the scale (Mettler) and gradually sprinkle the peat into the ashing dish until 0.500 or 1.000 g is added. Record actual weight. Note: Recording to 3 decimals gives ±2% error.

14) Set the ashing dishes aside in pans and cover until ready to put into the ashing oven.
15) Place dry blended or crushed peat into stainless steel 80 mesh sieve (don't use brass or bronze ones) with a stainless steel collection pan under the sieve. Sieve the peat by hand by tapping and rubbing your fingers over the peat.
16) Weigh the dry peat that has passed through the 80 mesh sieve. Use aluminum pans. The plastic boats have too much static. If the peat weighs more than 1.5 g. (although it is desirable to get at least 2.1 g.), then discard the coarser peat. Try to sieve each sample the same way. Place fine peat in envelope (reuse same one) after weighing. Record the final weight of the sieved peat placed in the envelope.
17) Clean sieve and pan with compressed air and repeat steps 13-16 for each sample.
18) The samples in the brown envelopes are now done and can be sent to Ag. labeled with lab # only. Enter lab # , date, sample #, and -80 mesh metals in red lab book.

Ashing Method

1. Weigh each porcelain dish. Keep in order.
2. Weigh out 0.50 or 1.00 g of dry, unsieved peat into porcelain dishes.
3. Peat is ignited in a muffle furnace at 550°C for 1 hour.
4. After 1 hour, turn off oven and let cool for an hour or so.
5. Weigh crucible and ash together.
6. Subtract the crucible weight from this to get the ash weight.
7. Percent ash (mineral residue) = $\frac{ash\ wt.}{sample\ wt.} \times 100$

pH Measurement

1. Place 15 g of wet peat in 50 or 100 ml beaker.
2. Add 15 ml of distilled water.
3. Stir peat and water with a glass rod and let sit.
4. Stir again at 15 min. and 30 min.
5. Let sit for 30 minutes. Total time is 60 minutes.
6. Measure pH by placing the probe about half way into the solution. Ask for assistance with the pH meter/probe.

Percent Moisture

$$\frac{wet\ weight\ (g) - dry\ weight\ (g)}{wet\ weight\ (g)} \times 100$$

Digestion Method

1. About 0.5 g dry sample.
2. Digested in microwave with 5 mL RO water, 10 mL HNO_3, 2 ml HCL at 40, 80, 120 and 160 psi sequentially for 10 minutes at each pressure.
3. Samples were filtered and brought to 50 mL volume.
4. Analyzed on AA.

Listed below are peat digestion methods used previously for analysis of total metals.

Date: **9/88**
Ref: 1991 and 1996 DNR Wetland Treatment Reports.
Preparation: 0-20 cm peat samples
20-50 cm peat samples
105°C for 24 hrs. -- oven
processed with blender to break up clumps
sieved to -80 mesh (>70% of spl was -80)

Total Digestion ref: 1991 and 1996 Wetland Treatment Reports.

0.5 g sample
1 mL HCl
2 mL HNO_3
0.5 mL HF
digested at 90°C for 2 hr.
adjusted to 50 mL vol., then AA

Date: **5/91**
Ref: 1996 DNR Wetland Treatment Report.
Preparation: 0-20 & 20-50 cm peat samples
- Same process as 9/88 except a microwave digestion was used.

- Reruns in 4/93

Total digestion (from lab sheet)

0.5 g sample
5 mL H_2O DD
10 mL HNO_3
2 mL HCL
microwave 15 min.
final volume 50 mL
analyze

Date: **3/92**
Ref: Zena's writeup
- frozen peat core, 2-4 cm segments.
sequential according to Wieder, 1991

Total Digestion

10 g wet peat -- ash for 1 hr. at 300°C and 3 hrs. at 800°C
Add 10 mL H_2O_2 and evap. at 90°C -- repeat until no more effervescence with H_2O_2 addition.

Digestion: (Weider 92)
Add 10 mL HCl to dry peat.
Place in block digester or sand bath -- raising T to 200°C and evap.
Add 25 mL DDW and filter thru Whatman 42 filter.
Bring vol. to 100 mL with DDW.
Analyze.

Plant Tissue Digestions (Typha, Lemna, Sedge, etc.)
Ref: 1991 DNR Wetland Treatment Report, Appendix 13

Sedge and grass leaves and lemna
Date: 1988-89
- oven dry 24 hrs. @ 80°C.
- dry ashed 2 hrs. at 550°C.
- add 2N HCl -- warmed, filtered and brought to vol.
AA analysis.

Typha
Date: 1987, 1989
- rinsed, sectioned into leaves, roots, rhizomes
- air dry
- oven dry 24 hrs. @ 80°C.
- ground in Wiley mill with 20 mesh screen.
- Ashed 2 hrs. At 550°C.
Digested with HNO_3
AA

Peat Processing Procedure - 2 Days (Spring 1997)

DAY ONE

1. Thaw baggies of frozen peat. Organize according to site and depth of sample.

2. Mix contents of baggie well and remove any sticks and/or rocks from sample.

3. Tare plastic weighing dish to zero using the analytical scale and measure out 10.000g (9.900-10.100) of peat. Pour this sample into a new baggie labeled with date, site, and actual weight.(ex. 5/23/97, S1-1, 10-20cm, 10.002g) Seal baggie, making sure to remove all air, and freeze. This sample will be used for sequential analysis.

4. Place 15g of peat in 50ml or 100ml beaker. Set aside for pH.

5. Label a metal tin and record weight to three decimal places. Place approximately 40g of wet peat into tin. Record actual weight of tin + wet peat. Set aside.

6. Repeat steps 1-5 for remaining samples. (You can usually complete approximately 35-45 samples a day.)

7. Place tins with wet peat in oven at 100C overnight.

8. pH: Add 15ml of distilled water to each beaker and stir with glass rod. Stir all beakers again at 15min and again at 30min. At 60min measure and record pH. To measure pH, tip beaker so that the peat solution is deep enough for the pH probe. Make sure to immerse probe completely in solution, but do not allow the probe to touch the bottom of the beaker.

DAY TWO

9. Remove tins with dry peat from oven and cool, measure and record dry weight of each tin, (tin + dry sample). Remove dry peat from tin using a spatula and place into small manilla envelope labeled with lab # only.

10. Blend peat sample in Oster blender for 15 seconds. Use brush to remove fine particles from side and lid of blender and place ground peat back into envelope. Clean blender, lid and brush with compressed air. Repeat for each sample.

11. Ashing: Place 1.00g of ground peat from envelope into preweighed/predried ashing crucible, record actual sample weight. Ash at 550C for one hour. Remove crucibles from muffle using tongs and place into dessicator, let cool one hour. Measure and record ash weight.

<u>Procedure for removing peat sample from the envelope</u>: Hold envelope so all the peat goes to one corner. Take the stainless steel spatula and obtain a representative sample by pushing the spatula to the bottom of the envelope and withdraw the sample. This way there should be some heavy and some light peat on the spatula. Weigh the empty white ashing dish and record its weight to 3 decimals.

Then tare the scale (Mettler) and gradually sprinkle the peat into the ashing dish until 0.500 or 1.000 g is added. Record actual weight. Note: Recording to 3 decimals gives ±2% error.

12. Place remaining dry blended peat from envelope into an 80 mesh SS sieve with collection pan underneath, seive with hands for approximately 15 seconds by tapping and rubbing.

13. Tare empty envelope, place -80 mesh peat from the collection pan into envelope weigh and record sample weight. This should give the -80 sample weight only.

14. Tare new envelope labeled with lab# and "+80". Place +80 mesh peat (that which did not go through mesh) into envelope, weigh and record. This should give the +80 sample weight only.

15. Clean mesh and pan with compressed air and repeat for each sample.

17. Send -80 mesh samples to Ag department for metals.

Calculations

Calculate % moisture and % ash;

$$\% \text{ moisture} = \frac{((\text{tin+wetpeat wt})-(\text{tin wt}))-((\text{tin+drypeat wt})-(\text{tin wt}))}{((\text{tin+wetpeat wt})-(\text{tin wt}))} \times 100$$

$$\% \text{ ash} = \frac{(\text{ash wt - crucible wt})}{(\text{preash wt - crucible wt})} \times 100$$

Table A4.2. Metal concentrations in peat samples collected from the DNR study cell W1D treatment system in March-April 1996.

Row	Site	Sample #	Peat depth (cm)			Metal concentration in peat (mg/kg)			
			Min	Max	Avg.	Cu	Ni	Co	Zn
1	1	10597	0	10	5	351.8	5325.9	52.6	180.8
1	1	10598	10	20	15	183.0	10145.2	105.4	206.8
1	1	10599	20	30	25	205.7	20683.2	181.1	254.9
1	2	10641	0	10	5	59.9	2995.8	10.9	127.8
1	2	10642	10	18	14	37.8	2329.6	16.9	67.7
1	2	10600	20	30	25	41.8	6720.1	28.9	130.6
1	3	10601	0	10	5	41.7	4296.5	35.7	95.3
1	3	10602	10	20	15	24.5	486.5	16.7	35.3
1	3	10603	20	30	25	68.6	3049.5	58.6	91.5
1	4	10643	0	10	5	17.8	298.8	6.7	22.6
1	4	10644	10	18	14	32.9	328.1	11.9	26.9
2	2	10649	0	10	5	107.8	2223.4	49.4	93.0
2	2	10650	10	20	15	20.9	2475.0	11.9	55.8
2	2	10610	20	30	25	31.6	597.5	16.8	46.4
2	3	10611	0	10	5	229.9	14585.8	153.3	212.0
2	3	10612	10	20	15	45.5	1272.0	18.8	50.4
2	3	10613	20	30	25	79.7	3451.9	51.8	86.7
2	4	10614	0	10	5	22.7	567.6	13.8	48.4
2	4	10615	10	20	15	24.8	331.6	12.9	32.8
2	4	10616	20	30	25	20.6	173.1	9.8	32.4

Table A4.2. Metal concentrations in peat samples collected from the DNR study cell W1D treatment system in March-April1996 (continued).

Row	Site	Sample #	Peat depth (cm)			Metal concentration in peat (mg/kg)			
			Min	Max	Avg.	Cu	Ni	Co	Zn
3	2	10651	0	10	5	18.8	68.4	7.9	23.8
3	2	10652	10	20	15	17.8	212.2	10.9	25.7
3	2	10623	20	25	22	113.5	321.6	18.9	39.8
3	3	10617	0	10	5	123.4	11422.7	80.9	167.8
3	3	10618	10	20	15	47.4	2631.0	29.6	60.3
3	3	10619	20	30	25	31.6	1723.6	21.7	42.5
3	4	10624	0	10	5	46.5	1694.0	30.7	75.3
3	4	10625	10	20	15	34.9	977.6	18.9	48.9
3	4	10626	20	30	25	55.8	530.6	15.9	43.8

Row	Site	Sample #	Peat depth (cm)			Metal concentration in peat (mg/kg)			
			Min	Max	Avg.	Cu	Ni	Co	Zn
4	1	10645	0	10	5	105.6	98.6	19.9	37.8
4	1	10639	13	20	16	29.8	254.0	10.9	27.9
4	1	10634	20	30	25	22.8	118.3	11.9	35.7
4	2	10646	0	10	5	20.9	973.3	22.9	76.7
4	2	10640	14	20	17	22.8	231.7	9.9	30.8
4	2	10635	20	30	25	27.9	65.8	12.9	32.9
4	3	10638	0	8	4	252.3	131.0	395.0	306.2
4	3	10604	0	10	5	20.8	187.4	11.9	39.6
4	3	10605	10	20	15	34.7	931.7	27.7	60.5
4	3	10606	20	30	25	18.8	62.6	7.9	38.7
4	4	10607	0	10	5	113.7	450.7	62.3	166.2
4	4	10608	10	20	15	45.5	607.9	14.8	51.4
4	5	10647	0	10	5	43.8	006.5	16.9	150.3
4	5	10648	10	23	16	73.9	975.5	17.7	49.3
4	5	10609	20	30	25	67.2	453.6	17.7	41.5

Table A4.3. Metal concentrations in peat samples collected from the DNR study cell of the W1D treatment system, April 1997.

Date	Site	Depth (cm) min	max	mean	#	pH	Cu	Ni	Co	Zn	Ash	Moisture
4 14 97	2	0	10	5	10793	6.31	68.6	2690	27.1	113.0	32.60	92.11
4 14 97	2	10	20	15	10794	6.73	28.7	382	5.0	53.0	56.78	80.97
4 14 97	2	20	30	25	10795	6.94	39.3	184	5.0	54.4	54.90	81.19
4 17 97	3	0	10	5	10792	7.03	156.6	4990	71.4	174.6	67.44	86.66
4 14 97	4	0	10	5	10796	6.53	263.8	10930	95.0	416.7	33.43	88.87
4 14 97	4	10	20	15	10797	6.47	147.3	5840	45.0	186.2	39.48	87.40
4 14 97	4	20	30	25	10798	6.66	73.7	2000	22.0	103.5	44.25	84.36
4 14 97	4	30	40	35	10799	6.82	92.6	1496	38.4	111.0	41.46	85.16
4 14 97	4	20	28	24	10800	6.61	43.8	915	8.0	67.2	54.75	80.85
4 11 97	5	0	10	5	10801	6.89	70.2	5845	46.5	139.4	48.17	80.87
4 11 97	5	10	20	15	10802	6.93	34.6	1402	23.9	57.7	66.37	79.30
4 11 97	5	20	30	25	10803	7.09	63.4	1429	44.5	68.9	45.99	84.02
4 11 97	5	30	40	35	10804	6.79	264.9	2240	185.3	156.7	54.27	81.79
4 11 97	7	0	10	5	10805	6.84	42.0	3010	17.6	86.1	39.34	87.51
4 11 97	7	10	20	15	10806	6.94	24.9	1095	5.0	59.8	57.92	82.91
4 11 97	7	20	30	25	10807	7.01	23.4	609	4.0	58.2	50.05	83.76
4 11 97	7	30	40	35	10808	7.13	22.0	488	4.0	49.2	61.58	82.45
4 11 97	7	40	58	49	10809	6.99	36.4	1187	19.1	63.7	53.98	82.72
4 11 97	10	0	10	5	10810	6.99	63.0	5680	30.4	115.6	51.72	85.18
4 11 97	10	10	20	15	10811	7.21	28.3	653	4.0	53.6	47.12	83.34
4 11 97	10	20	30	25	10812	6.95	35.0	520	5.0	60.0	60.76	82.85
4 11 97	12	0	10	5	10813	7.29	65.6	6090	30.2	113.4	62.58	77.65
4 11 97	12	10	20	15	10814	7.22	40.2	1747	8.0	68.0	50.50	81.50
4 11 97	12	20	30	25	10815	7.17	29.7	878	4.0	55.4	51.20	82.61

Table A4.3. Metal concentrations in peat samples collected from the DNR study cell of the W1D treatment system, April 1997 (continued).

Date	Site	Depth (cm) min	max	mean	#	pH	Cu	Ni	Co	Zn	Ash	Moisture
4 11 97	13	0	10	5	10816	7.00	30.6	391	4.0	48.2	47.84	82.72
4 11 97	13	10	20	15	10817	7.02	39.1	115	4.0	45.8	69.11	76.99
4 11 97	13	20	30	25	10818	6.89	46.9	92	4.0	54.8	66.76	74.84
4 11 97	13	30	40	35	10819	7.01	47.3	111	4.0	54.8	59.09	73.94
4 11 97	13	40	50	45	10820	7.10	38.7	82	4.0	51.7	63.86	73.38
4 11 97	13	50	60	55	10821	7.15	118.6	600	33.3	81.0	52.64	82.94
4 11 97	13	60	76	68	10822	7.35	154.8	508	40.1	86.4	54.14	85.37
4 11 97	14	0	10	5	10823	6.87	215.2	21010	142.4	373.6	53.42	85.06
4 11 97	14	10	20	15	10824	6.88	159.0	15230	96.4	238.7	43.56	88.21
4 11 97	14	20	30	25	10825	6.90	106.9	91	80.5	155.6	46.04	88.06

Table A4.4. 1997 peat samples collected from the Seep 1 pretreatment area.

Date	Site	Depth (cm) min	max	mean	#	pH	Cu	Ni	Co	Zn	Ash	Moisture
4 17 97	1	0	10	5	10828	6.38	1451.0	5151	327.0	643.0	55.01	82.05
4 17 97	1	10	20	15	10829	6.55	424.0	1196	73.1	192.0	58.31	79.58
4 17 97	1	20	30	25	10830	6.27	210.0	494	36.0	90.1	73.78	70.00
4 17 97	1	30	40	35	10831	6.49	219.0	248	17.0	59.0	83.80	63.91
4 17 97	1	40	50	45	10832	6.81	216.0	198	11.0	52.5	90.76	49.38
4 17 97	2	6	10	8	10833	6.82	633.0	938	72.7	178.0	77.61	72.52
4 17 97	2	10	20	15	10834	6.99	226.0	249	12.8	64.2	76.87	69.86
4 17 97	2	20	26	23	10835	7.10	220.0	192	9.5	53.9	84.62	55.75
4 17 97	3	0	10	5	10836	6.56	724.0	2159	149.0	260.0	77.79	68.80
4 17 97	3	10	20	15	10837	6.68	146.0	774	60.0	84.8	58.07	78.00
4 17 97	3	20	30	25	10838	6.70	96.0	221	23.2	54.0	69.19	67.73
4 17 97	3	30	40	35	10839	6.39	82.0	109	13.2	47.0	64.75	73.08
4 17 97	3	40	48	44	10840	6.39	305.0	256	40.2	77.5	84.29	59.04
4 14 97	4	0	10	5	10841	6.71	1334.0	1056	72.5	211.0	70.90	77.60
4 14 97	4	10	20	15	10842	6.81	106.0	93	10.5	44.0	64.59	73.86
4 14 97	4	20	30	25	10843	6.86	96.2	68	9.2	42.6	71.10	72.09
4 14 97	4	30	40	35	10844	6.68	155.0	113	29.7	46.3	80.87	67.31
5 1 97	7	0	10	5	10845	6.76	637.0	1427	97.0	180.0	74.02	74.41
5 1 97	7	10	20	15	10846	6.87	235.0	470	38.6	91.5	63.50	79.67
5 1 97	7	20	30	25	10847	6.99	198.0	224	23.0	58.6	65.49	77.27
5 1 97	7	30	40	35	10848	6.89	234.0	196	20.0	48.1	75.31	71.66
5 1 97	7	40	50	45	10849	6.67	247.0	179	23.5	85.4	78.99	66.13

Table A4.4. 1997 peat samples collected from the Seep 1 pretreatment area (continued).

Date	Site	Depth (cm) min	max	mean	#	pH	Cu	Ni	Co	Zn	Ash	Moisture
5 1 97	8	0	10	5	10850	6.91	730.0	3045	243.0	329.0	73.38	73.98
5 1 97	8	10	20	15	10851	6.97	172.0	256	28.4	60.7	66.54	75.28
5 1 97	8	20	30	25	10852	6.98	258.0	530	48.1	79.2	51.09	75.53
5 1 97	8	30	40	35	10853	7.07	105.0	682	65.2	38.9	59.68	73.81
4 14 97	10	0	10	5	10854	6.83	418.0	1850	111.0	277.0	70.61	76.31
4 14 97	10	10	20	15	10855	6.70	253.0	427	27.0	88.8	79.68	70.81
4 14 97	10	20	30	25	10856	6.79	234.0	291	21.2	64.7	79.30	62.59
4 14 97	11	0	10	5	10857	6.72	1256.0	3655	222.0	568.0	62.28	81.80
4 14 97	11	10	20	15	10858	6.78	127.0	369	22.0	63.2	62.87	75.53
4 14 97	11	20	30	25	10859	6.59	81.8	179	9.1	41.8	77.43	74.31
4 14 97	11	30	40	35	10860	6.35	97.5	135	16.7	44.1	69.69	73.94
4 14 97	12	0	10	5	10861	6.70	599.0	1775	122.0	325.0	59.26	81.69
4 14 97	12	10	20	15	10862	6.91	243.0	575	45.6	107.0	73.68	77.98
4 14 97	12	20	30	25	10863	6.88	138.0	238	18.1	55.4	63.92	75.14
4 14 97	12	30	40	35	10864	6.85	90.8	155	8.6	40.1	71.46	72.74
4 14 97	12	40	50	45	10865	6.66	124.0	145	10.1	35.8	75.93	67.27
4 14 97	13	0	10	5	10866	6.89	367.0	2067	122.0	337.0	66.87	80.56
4 14 97	13	10	20	15	10867	6.99	130.0	359	21.3	74.8	64.98	78.83
4 14 97	13	20	30	25	10868	6.91	96.1	254	14.3	62.9	60.62	75.71
4 14 97	13	30	40	35	10869	6.76	134.0	297	22.2	70.4	73.18	73.90

Table A4.4. 1997 peat samples collected from the Seep 1 pretreatment area (continued).

Date	Site	Depth (cm) min	max	mean	#	pH	Cu	Ni	Co	Zn	Ash	Moisture
4 14 97	14	0	10	5	10870	6.65	606.0	2980	155.0	444.0	80.71	72.55
4 17 97	15	0	10	5	10871	6.79	454.0	1617	102.0	294.0	78.84	69.87
4 17 97	15	10	20	15	10872	6.65	158.0	278	28.8	136.0	82.88	62.49

Note: pH values are standard units, metals are mg/kg (ppm).

Table A4.5 Total metal concentrations in substrate vertical down-flow section, April 1996.

Sample	Distance From Terminal Berm (m)	Depth of Sample (cm)	Substrate Type	Metal concentrations (μ g metal/gm dry substrate)			
				Ni	Co	Cu	Zn
Initial Material	NA_p	NA_p	Peat	5	2	5	7
S1-1	6	0-10	Peat	3667	325	1759	984
		10-20		4606	202	458	1461
S1-2	4	0-10	Hay	6510	995	6034	1626
		10-20		9998	1347	8392	2620
		20-30	Hay/Peat	11853	671	4573	2926
		30-40	Peat	8674	354	648	1727
S1-3	2	0-10	Hay	11451	1302	3448	3162
		10-15		14180	768.5	3917	3299
		15-40	Hay/Peat	10654	489	2471	1928

NA_p = Not applicable

Appendix 5

Flow, W1D

Page

Section A5.1. Description of flow measurement and estimates A5.2

Table A5.1 Annual precipitation and flow at W1D A5.4

Section A5.2. July 1999 staff gage readings and flow calculations for W1D A5.5

Section A5.3. W1D Flow corrections; September 30 and October 1, 1995; October 11-13, 1997 . . . . . A5.7

Note: This appendix contains information on how flow measurements and flow estimates were made, as well as information on OSS readings and LTV's rating equation. Average daily flow and total flow per month for both W1D and Seep 1 are included in the mass release tables in Appendix 7.

Section A5.1. Description of flow measurement and estimates.

Daily flow measurements are generally only available from May 1 to October 31. The type of water level recorders used by LTV (Stevens Model F) use a float system which does not work well under freezing conditions. In order to standardize data collection and to minimize equipment problems, LTV decided to install the recorders on or near May 1 and remove them around October 31.

Flow at the seep generally begins in late March or early April and can continue into November. Fall flow typically can be characterized by a linear decrease from the last flow measurement to zero at freeze-up.

Spring melt flows are much more difficult to estimate. Flow depends on the amount of moisture in the snow pack, temperature, and rainfall. Limited annual data existed for site EM8 (seepage from Stockpile 8011) in 1979 and 1980. Although these years may not represent "average" conditions, they provide a range of conditions. In 1979 the snow pack contained 4.39 inches of water, spring flow began around April 10, and peaked around the 20th of April. Rainfall for the rest of the year was below normal and there were no major rain events during the year. For this year, spring melt accounted for 18% of the total annual flow.

In 1980, the snow pack contained only 2.74 inches of water and the spring flow was much less than in 1979. Large rain events (daily rainfall > 2 inches) in September and October produced a large amount of runoff, and for this year spring melt accounted for only 2% of the total flow.

Calculation of annual average flow values. LTV reports monthly flow values as part of their NPDES permit. During May to October, these are based on daily measurements obtained from water level recorders. During April, November, and December, the staff gage is read when samples are collected (about twice per month) and a monthly average is calculated based on the spot readings. Total annual flow is estimated by adding all the values reported by LTV in their NPDES report. No corrections have been made for spring or fall flow.

To calculate an average annual daily flow, it is assumed that all the reported monthly volumes are reasonable estimates of the flow and that there are 244 days of flow (April 1 - November 30).

There are several problems with the method:

1. While two staff readings in November may give a reasonable estimate of average flow, it is unlikely that two readings accurately estimates spring melt.

2. Flow measurements are estimated from a standard weir equation. No actual flow measurements are taken to verify the equation. It is not clear how often the gages are surveyed to ensure that there has not been any movement of the staff gage and/or the basin. In 1993, the DNR made flow measurements at the inlet and outlet of the original W1D wetland treatment system. There was about a 15% difference between measured values and values using LTV's standard equation.

3. The number of days of flow is not constant, but varies. In some years, there has been flow out of the wetland into December.

May to October is the more accurate flow data, while total annual flows are estimates. Based on data from small uncovered stockpiles at the AMAX exploration site near Babbitt (about two miles southwest of the Dunka Mine), flow for May through October accounted for 77% of the total annual flow.

To calculate an average flow for the May through October data, the total flow for each month reported on the NPDES forms were added and divided by the total number of days:

$$\text{Average daily flow (gal/min)} = \frac{\text{total flow (gal)}}{\text{244 days (1440 min/day)}}$$

Often flow data for April was not estimated on the NPDES forms.

Table A5.1. Annual precipitation and flow at W1D.

Year	Annual Precip.	May-Oct Precip	Annual volume (m^3)		Cumulative volume (m^3)	
			Input (051)	Output (052)	Input (051)	Output (052)
1992	26.90	17.30	35,602[a]	47,039	35,602	47,039
1993	29.64	17.43	32,306	40,760	66,111	67,600
1994	28.66	19.16	40,772	50,507	106,663	136,307
1995	25.60	18.49	31,320	56,941	156,479	195,249
1996	34.15	18.23	37,157	37,726	193,636	232,978
1997	22.91	15.46	11,673	16,724	205,510	249,702
1998	31.12	19.16	10,571	6,166	216,081	255,667
1999	35.17	28.80	28,327	32,434	244,409	288,301
Avg. Precip.[b]	28.57	20.84				

Note: These values include all flow measured and reported by LTV. The precipitation data are based on a composite of various data sources; see Table A11.1 for details.

In an average year there is ~ 244 days of flow (April 1 - November 30), then to convert from m^3/year to gpm, multiply entries by:

$$\frac{x\ m^3/\text{year} \times 10^3\ L/m^3}{244\ \text{days/year} \times 3.785\ L/\text{gal} \times 1440\ \text{min/day}} = 0.00076$$

[a]No flow reported for April, average flow based on May 1 - November 15.

[b]Based on 56 years of data from Babbitt (Watson, 1978).

Section A5.2. July 1999 staff gage readings and flow calculations for W1D.

The table on this page presents staff gage readings and flow calculations for W1D-051 for July 1999 (approximately 7 inches of rain fell during the July 4th weekend.) The chart on the next page (page A5.6) is the corresponding strip chart for this time period. (These data were not changed; the information is presented to chronicle peak flow during this period of high precipitation.)

LTV Mining Company- Dunka

Wetland Flow Monitoring

Month of July, 1999

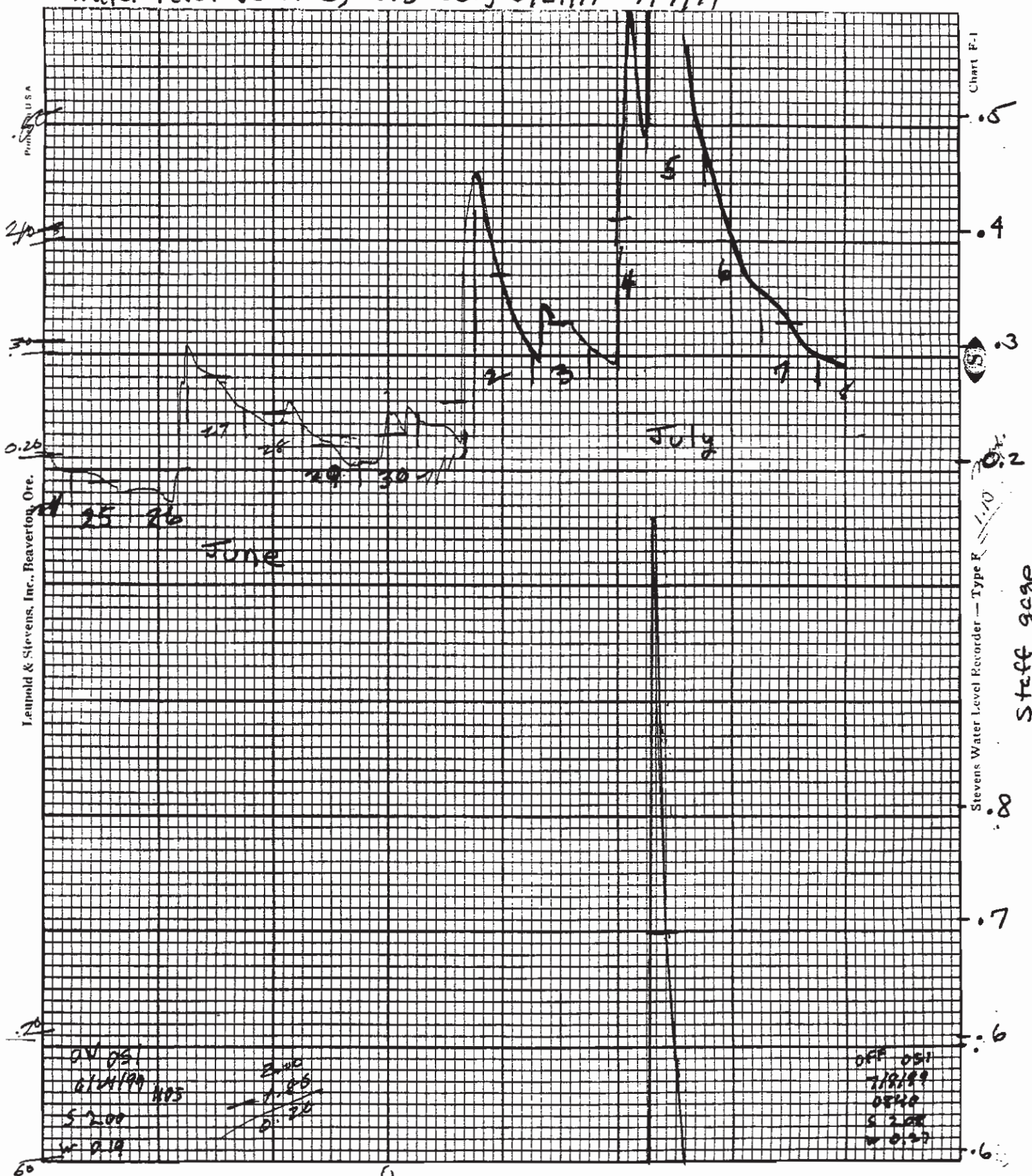
Location: 051 60 degree V-notch Weir

Date:	Head Feet	CFS	GPM	MGD
07/01/1999	0.25	0.0451	20.24	0.0291
07/02/1999	0.36	0.1122	50.36	0.0725
07/03/1999	0.33	0.0903	40.51	0.0583
07/04/1999	0.42	0.1650	74.04	0.1066
07/05/1999	0.79	0.8004	359.24	0.5173
07/06/1999	0.39	0.1371	61.51	0.0886
07/07/1999	0.32	0.0836	37.51	0.0540
07/08/1999	0.32	0.0836	37.51	0.0540
07/09/1999	0.35	0.1046	46.93	0.0676
07/10/1999	0.31	0.0772	34.65	0.0499
07/11/1999	0.27	0.0547	24.53	0.0353
07/12/1999	0.25	0.0451	20.24	0.0291
07/13/1999	0.29	0.0654	29.33	0.0422
07/14/1999	0.26	0.0497	22.32	0.0321
07/15/1999	0.26	0.0497	22.32	0.0321
07/16/1999	0.37	0.1202	53.93	0.0777
07/17/1999	0.28	0.0599	26.87	0.0387
07/18/1999	0.26	0.0497	22.32	0.0321
07/19/1999	0.25	0.0451	20.24	0.0291
07/20/1999	0.24	0.0407	18.27	0.0263
07/21/1999	0.23	0.0366	16.43	0.0237
07/22/1999	0.22	0.0328	14.70	0.0212
07/23/1999	0.26	0.0497	22.32	0.0321
07/24/1999	0.23	0.0366	16.43	0.0237
07/25/1999	0.23	0.0366	16.43	0.0237
07/26/1999	0.22	0.0328	14.70	0.0212
07/27/1999	0.22	0.0328	14.70	0.0212
07/28/1999	0.24	0.0407	18.27	0.0263
07/29/1999	0.22	0.0328	14.70	0.0212
07/30/1999	0.22	0.0328	14.70	0.0212
07/31/1999	0.21	0.0292	13.09	0.0188

6/24 7

WID-051, 6/24/99 - 7/7/99

Water level vs time; WID-051; 6/24/99 - 7/7/99



1999

Section A5.3. W1D Flow corrections; September 30 and October 1, 1995; October 11-13, 1997.

LTV misread the strip charts for W1D-051 on September 30-October 1, 1995, and from October 11-13, 1997. The tables on this page present LTV's staff gage readings and flow calculations for the 1995 data, with the corresponding strip chart on the next page (page A5.8). LTV's data for October 11-13, 1997 are presented on page A5.9, with the corresponding strip chart presented on page A5.10.

Flow Worksheet For 051

ENTER NUMBER OF DAYS IN THE MONTH 30

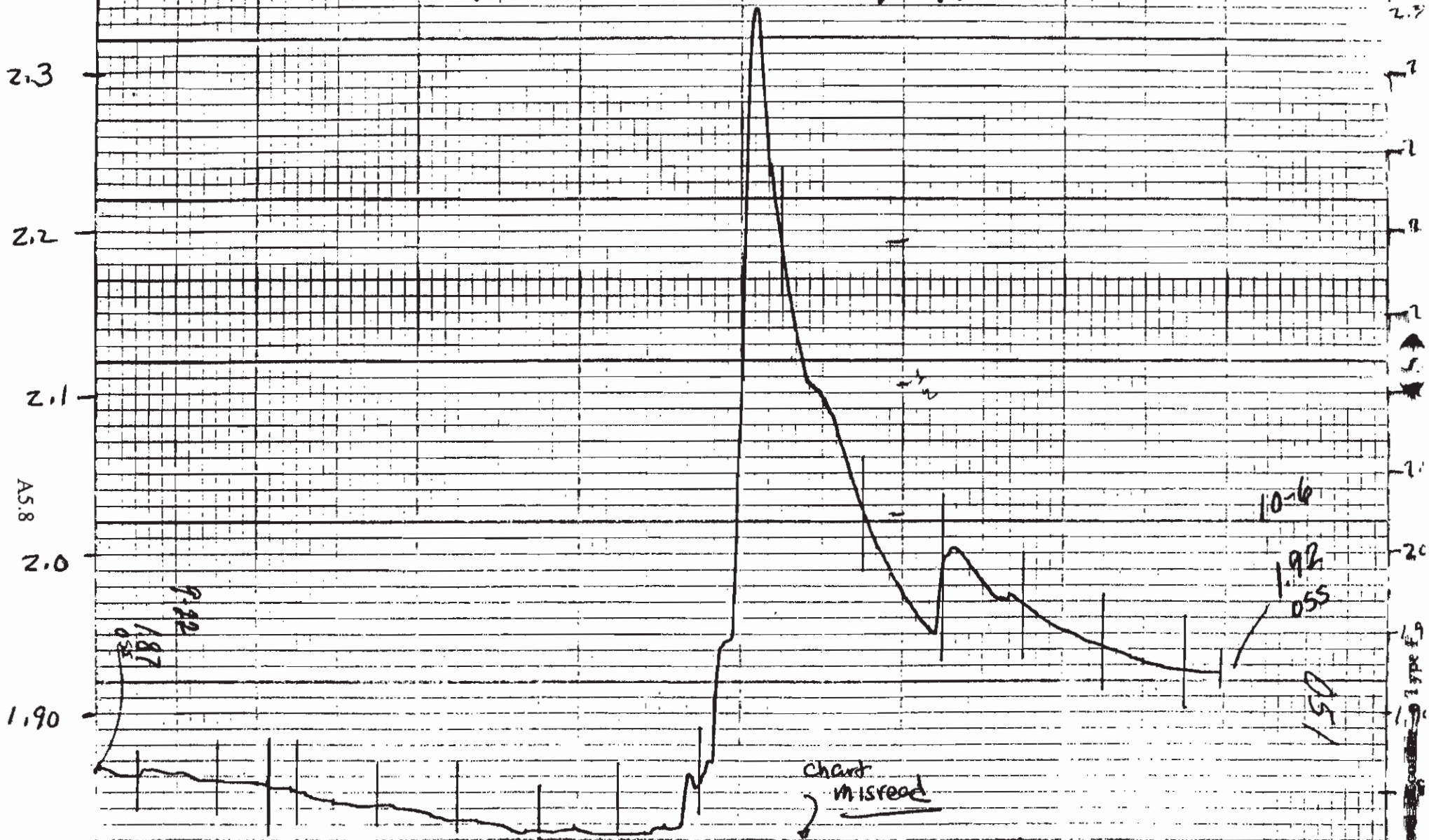
Date:		Height	GPM:	MGD:
Sept 1–5		1.89	40.43	0.058
	2	1.87	34.58	0.050
	3	1.89	40.43	0.058
	4	1.9	43.56	0.063
	5	1.87	34.58	0.050
	6	1.94	57.53	0.083
	7	1.99	78.36	0.113
	8	1.91	46.84	0.067
	9	1.89	40.43	0.058
	10	1.88	37.44	0.054
	11	1.87	34.58	0.050
	12	1.87	34.58	0.050
	13	1.87	34.58	0.050
	14	1.87	34.58	0.050
Sept 15–5		1.89	40.43	0.058
	16	1.92	50.25	0.072
	17	1.88	37.44	0.054
	18	1.87	34.58	0.050
	19	1.88	37.44	0.054
	20	1.87	34.58	0.050
	21	1.87	34.58	0.050
	22	1.86	31.86	0.046
	23	1.86	31.86	0.046
	24	1.86	31.86	0.046
	25	1.85	29.27	0.042
	26	1.84	26.81	0.039
	27	1.83	24.48	0.035
	28	1.83	24.48	0.035
	29	1.83	24.48	0.035
Sept 30–5		3	1608.13	2.316

Flow Worksheet For 051

ENTER NUMBER OF DAYS IN THE MONTH 31

Date:		Height	GPM:	MGD:
Oct 1–5		2.1 3	1608.13	2.316
	2	1.98	11+ 73.88	0.164 0.106
	3	1.98	73.88	0.106
	4	1.96	65.40	0.094
	5	1.94	57.53	0.083
	6	1.92	50.25	0.072
	7	1.92	50.25	0.072
	8	1.91	46.84	0.067
	9	1.91	46.84	0.067
	10	1.91	46.84	0.067
	11	1.9	43.56	0.063
	12	1.9	43.56	0.063
	13	1.9	43.56	0.063
	14	1.9	43.58	0.063
Oct 15–5		1.88	37.44	0.054
	16	1.88	37.44	0.054
	17	1.87	34.58	0.050
	18	1.86	31.86	0.046
	19	1.87	34.58	0.050
	20	1.87	34.58	0.050
	21	1.87	34.58	0.050
	22	1.87	34.58	0.050
	23	1.88	37.44	0.054
	24	1.91	46.84	0.067
	25	1.93	53.82	0.077
	26	1.93	53.82	0.077
	27	1.96	65.40	0.094
	28	1.95	61.39	0.088
	29	1.9	43.58	0.063
Oct 30–5		1.86	37.44	0.054
	31	1.86	31.86	0.046

Water Level vs Time; W10-051; [redacted]
September 22 - October 6, 1995

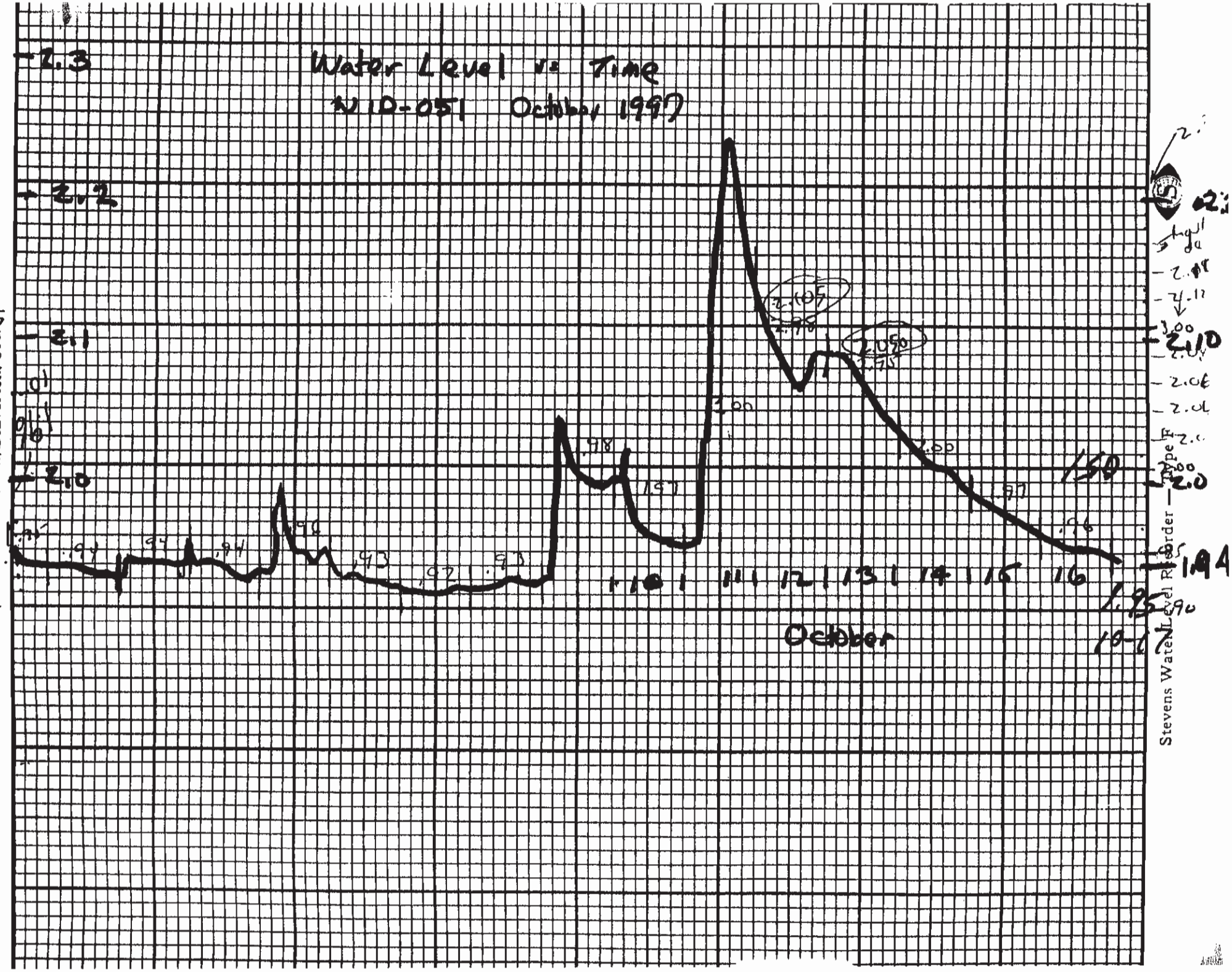


1.86	1.86	1.86	1.85	1.84	1.83	1.83	1.83	3.00	3.00	1.98	1.98	1.96	1.94	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	30	10/1	2	[illegible]	[illegible]	5	6

Flow Worksheet For 051 O=1.76

ENTER NUMBER OF DAYS IN THE MONT 31

Date:		Height	GPM:		MGD:
Oct 1, 1997		1.95	10.17		0.015
2		1.94	8.88		0.013
3		1.94	8.88		0.013
4		1.94	8.88		0.013
5		1.96	11.56		0.017
6		1.93	7.70		0.011
7		1.92	6.62		0.010
8	055	1.93	7.70		0.011
9	(Correct Values)	1.98	14.67	MGD	0.021
10		1.97	13.06		0.019
11	2.09	3	1106.55	0.041	1.593
12	2.105	2.98	1062.47	0.043	1.530
13	2.050	2.95	998.35	.032	1.438
14		2	18.24		0.026
Oct 15, 1997		1.97	13.06		0.019
16		1.96	11.56		0.017
17		1.95	10.17		0.015
18		1.95	10.17		0.015
19		1.94	8.88		0.013
20		1.93	7.70		0.011
21		1.92	6.62		0.010
22		1.92	6.62		0.010
23		1.92	6.62		0.010
24		1.91	5.63		0.008
25		1.9	4.74		0.007
26		1.9	4.74		0.007
27		1.9	4.74		0.007
28		1.9	4.74		0.007
29		1.9	4.74		0.007
Oct 30, 1997		1.9	4.74		0.007
31		1.9	4.74		0.007


Water Level vs Time
WID-051 October 1997
2.3
2.2
2.1
2.0
October
Stevens Water Level Recorder — Type F
Leupold & Stevens, Inc., Beaverton, Ore.

A510

Appendix 6

Timeline for 1997-1999 W1D and Seep 1 Monitoring Program

Page

Section A1.1 1997-99 DNR Monitoring program of the Seep 1 pretreatment and the W1D wetland treatment system A6.2

1997-99 DNR Monitoring program of the Seep 1 pretreatment and the W1D wetland treatment system

10/5/95
Survey of W1D to identify flow paths and water levels in study cell (see Appendix 9).

3/13/95 and 3/18/96
Collected peat samples in W1D study cell.

8/21/96
W1D: Walked berms 2 and 3. The flow paths appeared the same as last fall. Noted vegetation types.

Seep 1: Sampled pH, SC and total metals from seeps 2-9 except for #8.

10/29/96
W1D: Took SC and water (total nickel) samples at one location on berm 2 (42.0 m from west stake) and two locations on berm 3 (5.0 and 44.2 m from west stake).

4/11/97
W1D: Collected peat samples.

Seep 1: Peak snow melt runoff probably occurred 4/5-6. A cold snap started the evening of 4/6. Perched ice in the pretreatment system indicated water had risen to about 0.6 ft (ice on the staff gauge was at 2.70') below the overflow invert this past week. No perched ice was observed in the weir pool.

4/14/97
W1D: Collected peat samples. Downstream weir was washed out (bypassing about 30 gpm est).

Seep 1: Observed flow in the adjacent capping culvert. Collected peat samples. Observed flow in all the capping culverts (stockpile 8013) except culvert 4. The top of stockpile 8013 in the vicinity of culvert 4 has no snow. I saw some snow patches in the swales of stockpile 8014.

4/17/97
W1D: Collected a peat sample. The weir below the peat berms (middle weir) is still washed out.

Seep 1: Collected peat samples. Observed some flow in the capping culvert adjacent to seep 1. Observed 3-4 cm of precipitate in the vicinity of peat sample #1. The material was really fluffy and was layered. It was described from the surface downward as follows: brown layer (1mm), white cream color (1 cm), brown layer (1mm), white cream color (1 cm), brown layer (4mm), orange layer (2mm), brownish cream layer (5mm), orange layer (1mm), then black peat.

4/24/97
W1D: Took weir and drop pipe measurements only.

Seep 1: There still is some ice on the west and south perimeters. There was also some frost in the hay bales yet. Noted flow at seeps 2, 3, and 6-9. No water samples were collected. Collected peat samples from sites 16 and 17.

5/1/97
W1D: Routine water sampling. Site 2 (W1D-2) on berm 3 is a new site this year.

Seep 1: Seeps 2, 8, and 9 had flow (sampled). Routine water sampling. The system has no snow or ice remaining except a small amount of snow at the base of the stockpile. Collected peat samples at sites 7 and 8.

5/15/97
W1D: Routine sampling. The measured distances from the west rebar rod to sample points W1D-1, W1D-2, and W1D-3 are 41.9 m, 32.6 m, and 44.4 m, respectively.

Seep 1: Only seeps 2 and 9 had flow (sampled). Routine sampling.

5/29/97
W1D: Routine sampling.

Seep 1: Seeps 2, 3, and 9 had flow (sampled). Routine sampling. Lots of cattail growth. Noticed an orange, oil-like scum floating on the water surface in the vicinity of 043-1.

6/12/97
W1D: First stop was at the drop pipe. Both notches were obstructed with debris so we cleared it and allowed the water level to equalize. The west notch (left--facing upstream) was at .05 feet and the east notch (right) was at .07 feet. Samples were taken at W1D-1 and W1D-2; a sample was taken just upstream of the planck at W1D-3 because there was no water flowing over the planck. The flow at W1D-1 was estimated to be 5 gpm with a water depth on the planck of .05 feet. The flow at W1D-2 was minimal, and the crack between the plancks was almost clogged. The weir oss was at 4.17 feet with a water depth (weir notch) of 0.07 feet. No debris obstructed the weir, and the flow was gauged at 0.5 liters per 6 seconds.

Seep 1: Last stop of the day was at seep 1. The weir oss was at 1.44 feet with a water depth of 0.12 feet. No debris obstructed the weir, and the flow was gauged at 0.5 liters per 2 seconds. The weir oss is on the downstream side of the black bag of peat with a head drop across the bag of 0.13 feet. In the pretreatment pond, the OSS was 1.50 feet. The distance to the water from the right (east) and left notches of the drop pipe were 2.14 and 2.15 feet, respectively, and 2.92 feet to the top of the pipe. Samples were taken at 043-1 thru 043-8 and at the weir upstream of the

black bag. Flow estimates at the seeps were as follows: Seep 2 1 gpm, seep 3 0.3 gpm, seep 4 damp, seep 5 trickle, seep 6 trickle 100-500 ml/min., seep 7 slightly more than seep 6, seep 8 damp, seep 9 1 gpm. Seep 7 had rusty rocks at the edge of water. Noticed a white/blue/green ppt in the outer pool. Noticed underwater "streams" between seeps 4 and 5 with a partly eroded channel with a brown ppt on top and a white ppt underneath. Noticed evidence of other underwater flow 2 meters east of seep 4 and at the peninsula by seep 3. Each flow appears to emerge at the base (about 1 ft under water) of the rip rap supporting the outer slope. Noted aquatic insects throughout the treatment site.

6/30/97

W1D: The weather on this day was extremely humid with a temp. of 75 - 80 degrees and overcast sky. The drop pipe at W1D was blocked with debris so I cleaned it out and took samples at W1D-1, W1D-2, W1D-3 while it equalized. The flow at W1D-1 was approx. 10-15 gpm with a depth of water on the planck of 1 inch. W1D-2 had a flow of approx. 5 gpm and a depth of water on the plank of 0.02 inches. W1D-3 had a flow of less than 1 gpm and no water flowing over the planck. All samples were taken up stream of the sampling points. The cattails in the wetland were in full bloom. When I would bump against them, I noticed that they would discharge large amounts of yellow pollen into the air. The depth of water had greatly increased from 6/12/97 because of the increase in precipitation events during the two-week period. After I allowed the flow to equalize in the drop pipe, I took measurements of the depth of water flowing over the notch. The west notch (left) was at 0.12 feet and the east notch (right) was at 0.15 feet. Next I stopped at the weir between the treatment wetlands and discharge to the stream. The weir oss was at 4.28 feet, and the depth of water flowing over the weir was .19 feet. The weir was level and in good working order. There was no debris obstructing the flow over the weir, which indicates that the flow was high enough to keep it open. I gauged the flow coming out of the weir using a 15-liter bucket and a stop watch. The flow was 10.3 liters per 11.55 seconds.

Seep 1: At 1220 I arrived at seep 1 with the weather changing to light showers. The weir oss was at 1.48 feet with a depth of water flowing over the weir of 0.16 feet. I gauged the flow coming over the weir to be 500 ml/1.5 seconds. Seep 1 oss was at 1.6 feet, and it appeared to be level and in good working order. I sampled pool spls 043-1 through 043-8 and seeps 2 and 3. I also estimated the flow coming from the seeps around the treatment site as follows: Seep 2 5 - 7 gpm, seep 3 less than 1 gpm, seep 4 less than 1 gpm, seep 5 less than 1 gpm, seep 6 was wet, seep 7 less than 1 gpm, seep 8 less than 1 gpm, seep 9 was dry. The water level in the treatment cell was much higher than it was in the previous sampling event. The cattails were not as far along in blooming as they were at W1D treatment site. Only a few of the cattails were blooming and not as much pollen as at W1D. I observed an increase in aquatic insects and amphibians. I did not observe the submerged "streams" by seeps 4 and 5 as I did on 6/12/97.

7/14/97

W1D: The weather was 70 degrees with cloudy skies and the possibility of thunder storms. I sampled Wld-1, W1D-2, and W1D-3 and estimated the flow at W1D-1 to be 1-2 gpm There was no water flowing over the planck at W1D-2 and W1D-3. The drop pipe at W1D was .05 feet at

the west (left) notch and .07 at the east (right) notch. The water level at W1D was considerably lower than during the past weeks. The cattails were approximately 6 feet tall, and the reproductive portion of the cattail was close to maturity. The weir oss was at 4.18 feet with a flow gauged at 500 ml/4.84 seconds. The weir was not obstructed and had a depth of water on it of .08 feet.

Seep 1: Seep 1 was the last stop I made at Dunka. I sampled pool samples 043-1 through 043-8 plus weir before black bag. The weir oss was at 1.45 feet and had a depth of water on it of 0.12 feet. The seep 1 oss was at 1.60 feet. Seeps 2 and 7 were the only seeps that had enough flow to sample. I estimated the flow at seep 2 to be 1-2 gpm and seep 7 to be less than 1 gpm

<u>7/24/97</u>
W1D: Routine sampling. The reproductive portion of the cattails were mature with a long narrow part on top of the structure. There was a slight trickle coming over the weir.

Seep 1: The water at the weir/before the black bag smelled organic (slight hydrogen sulfide smell). After the weir, the water was building up and becoming stagnate. The flow going through the weir was stopping in the pipe, and a weak eddy was forming. The outer part of the horseshoe had an organic film on top of the water. The precipitate was lightest on the outer portion of the horseshoe with it getting increasingly darker as one progresses through to the effluent. That could indicate increased decay at end of treatment. I was able to use the flow trough to gauge accurately the flows at the seeps. I sampled seeps 2, 3, and 7. There was a stream of "dry" ppt. coming from the ground at seep 2 and between seeps 2 and 3, but smaller than the first.

<u>7/28/97</u>
Seep 1: Eger and Knuckey conduct a pH and SC survey in the pretreatment pond. Collected total metals samples at new sites.

<u>8/7/97</u>
W1D: Recorded flow measurements for the first time this summer at the upstream weir (historical LTV weir). Flow was gauged at 35 liters/minute. The water level has increased considerably since last sampling period. Vegetation has also increased. Weir—no flow going over the weir, only slightly damp. Flow appears to be bypassing the weir on the right as seepage due to the porous material used in construction. I observed an increase in frogs at the weir.

Seep 1: The water is backing up at the downstream side of the weir. It could be that the drain pipe going under the road is plugged. At 043-4 there is a trail of rust leading back to the limestone berm. Used the flow trough to gauge the flow at the seeps. Sampled seeps 2 through 7.

8/21/97

W1D: Routine sampling of W1D and Seep 1. LTV weir—flow was gauged at 104 liters/minute. W1D—water level was much higher than in past sampling periods. The water was flowing over the planck at all the sampling sites.

Seep 1: There was no head drop at the weir so the water was still backing up. No smell of hydrogen sulfide. The inner horseshoe had approximately 0.5 feet of water over the limestone berm. The outer most horseshoe had approximately 0.2 feet over the limestone berm. I observed an increase in amphibians and aquatic insects. The water level in the pretreatment area was the highest that it had been all season. I sampled seeps 2 through 8 and used the flow trough to gauge the flow of the seeps.

9/4/97

W1D: LTV weir—flow was gauged at 22 liters/minute. W1D—water has decreased substantially since last sampling period. The cattails were starting to turn light brown, and some were starting to seed (seeds were blowing off in the wind). Weir--some water is seeping through the berm at the toe on the right side facing the road. There was no channelized flow, but I estimated the flow lost to the leak to be 150 milliliters/minute.

Seep 1: The water was not backing up at the weir as in the past few months. No smell of hydrogen sulfide, possibly because of the decrease in water levels. Was not able to use flow trough to gauge flow because there was no flow at the end of the seeps. All the water was infiltrating into soil before it could reach the treatment cell, but I was able to sample seeps 2 and 3. Conducted a specific conductivity and pH survey of Seep 1 using a peristaltic pump to pull samples. I collected the sample in a lab bottle and did S.C. and pH on-site. I was not able to finish survey because of time constraints, and it was very difficult to obtain sample. It would be better if I had an extra person to help conduct survey using this method.

9/17/97

W1D: LTV weir—flow was gauged at 36 liters/minute. W1D—no change since last sampling period. Weir—still having bypassing problems at weir.

Seep 1: Flow was backing up at the weir again. I sampled seeps 2 and 7; all other seeps not able to sample because lack of water. I estimated the flow at the seeps because I forgot the flow trough although flow was very low, and I probably could not have used it. Was not able to conduct S.C. profile of Seep 1. Next sampling period I will bring a pipe to mount the S.C. probe so I don't have to wade in the water. This method proved to be a disaster because the water in the seep is about 4 feet deep.

10/1/97

W1D: LTV weir—flow was gauged at 60 liters/minute. W1D—cattails are mostly brown with slight green left. Some heads on the cattails were missing indicating seeding has begun. Water has come up since last sample period. Weir—could not get graduated cylinder all the way under the flow from the weir, so I measured only about 80% of the flow.

Seep 1: Water is over 043-8 berm. Cattails are mostly brown with some green left. PPT has not changed significantly since the start of fall. There was no head drop across the weir. Water was backing up at the weir, and the water depth in the outlet pond is the highest I have seen it this year. Treatment system across the road is also high—high oss does not mean high flow. Conducted S.C. survey at seep 1.

10/14/97

W1D: LTV weir—the flow was gauged at 102 liter/minute. W1D—there was an organic smell throughout the treatment cell. The cattails are completely dead and seeding is occurring. Weir—the flow was gauged at 240 liters/minute, the highest this year.

Seep 1: Water level is very high, but seeps are not flowing; this could be because of the high precipitation events last weekend. On the average there were 8 inches of water on the limestone berm throughout the treatment area. The cattails are dead but not as far along as W1D; they were not seeding yet. Pumped ppt into bucket for a wet analysis. Finished SC survey of points I missed last sampling period.

3/17/98

W1D: G. Melchert and M. Lubotina collected peat samples with a Macaulay sampler from the W1D vertical flow peat bed (pretreatment).

4/30/98

W1D: Measured flow of historical and middle weirs, drop pipe, and W1D(1-3).

Seep 1: Measured flow of S-1 weir, pond weir, and collected weir-B and 043-6 samples. Only Seep 2 and Seep 9 were flowing.

5/8/98

8013: installed culvert ring, tripod, and battery. Slope of culvert = 0.09'/2' (1st full segment), 0.10'/2' (2nd full segment in).

8018: installed culvert band in South culvert. Slope=0.10'/2'. Water in culvert but no flow. North culvert had flow, water coming from the soil approximately 1 Lpm.

W1D: usual flow measurements, Historical weir Ni=1.4mg/L (HACH kit). Middle weir was clogged with a clump of algae, did not wait to stabilize.

5/29/98

8013: Melchert started CR500 data logger approximately 1330. Starflow not hooked up yet.

W1D: usual flow measurements Ni for W1D-1 was 0.67mg/L (HACH).

Seep 1: usual flow measurements, samples collected 043 wier before and 043-6. Seeps 2-5, and Seep 9 were flowing.

6/10/98

Seep 1: usual flow measurements, collected samples at 043-WB and 043-6. Removed metal pole from pipe near 043-6. Seeps 2, 4, 5 & 9 were flowing.

W1D: usual flow measurements, Ni sample @ W1D-1 = 0.61 mg/l (HACH).

6/24/98

W1D: usual flow meas., Ni sample @ W1D-1 = 0.66 mg/l. Water is up but not much compared to last weeks rain. Cattails over 6' tall and have seed heads.

Seep 1: usual flow meas., 043-6 & 043- WB samples collected, strong sulphur smell. All of Seeps 2 - 9 (except S7) had some flow.

7/10/98
Seep 1: usual flow measurements, collected samples at 043-WB and 043-6. Seeps 2 - 9 (except S4 & 8) had some flow.

W1D: usual flow measurements, drop pipe clogged by cattails, water may be flowing around middle weir when flow is high.

7/21/98
W1D: usual measurements for flow.

Seep 1: usual measurements for flow. Seeps 2-9 all some flow except S 8 & S 9. Water in pond unusually clear, sulphur smell is gone.

8/4/98

W1D: usual measurements for flow (dead fish odor at drop pipe). Samples collected @ W1D-1 & 3, wetland very dry, samples collected at puddles within 1 m of sampling sites. Standing water in wetland below middle weir.

Seep 1: Unable to measure flow due to blocked culvert (11.5 cm deep), flow seems to be circular, berms that are not supposed to be covered with water are covered. 043-6 and 043-WB samples collected. Seeps 4, 5 & 6 are flowing.

8/19/98

W1D: usual measurements for flow, and water samples.

Seep 1: usual measurements for flow.

9/3/98

P. Eger, G. Melchert, and M.Crozier visited wetland cells at treatment plant. GM downloaded flow data from 8018 and 8013. Visited pre-treatment area (of W1D), measured specific conductance, Ni and water level

9/4/98

Ni sampling at W1D pre-treatment area. Ni ranged from 0 - 1.63 mg/L

9/16/98

W1D: usual measurements for flow, and water samples at W1D-1, W1D-2, and W1D-3

Seep 1: wier pipe clogged, minuscule new flow <0.3cm headdrop across weir
Piezometers installed

9/29/98

W1D: usual measurements for flow (clogged exit point middle weir, seep still moving through on right), samples at W1D-1, W1D-2, and W1D-3

Seep 1: weir backed up again
Piezometers sampled

10/15/98

W1D: usual measurements for flow, samples at W1D-1 W1D-, and W1D-3.

Seep 1: weir backed up a little, water samples, seeps dry
Piezometers sampled

10/20/98
Big rain last Friday
W1D: usual measurements for flow, samples at W1D-1, W1D-2, and W1D-3. Pre-treatment samples

Seep 1: weir is very backed up, < 1cm drop at weir, seeps flowing

11/3/98
Frost and ice covered ponds.

W1D: usual measurements for flow, samples at W1D-1, W1D-2, and W1D-3.

Seep 1: routine flow measurements water level has come down, ice thick enough to stand on Piezometers depth measured but not sampled

1999
4/2/99
G. Melchert and A. Johnson spring site visit.
8013 capping: culvert flowing, approximately 0.07 feet ± 0.02, uncovered rain gage tripped once @ 1200. Ground saturated with water, frost just starting to come out of the ground.
8018 capping: Solar panel face down on ground, culvert 0.06 feet water flowing through

Seep 1: usual measurements for flow, ice in way of normal 043-WB site so sampled just before weir. Seep 1-2 flowing about 100mls/min.

W1D: usual measurements for flow, samples at W1D-1, W1D-2, and W1D-3, middle weir some flow around right side, historical weir 2 to 3 inch hole under weir flowing around steel part of weir.

4/27/99
8018 capping: no flow in culvert, computer battery died so unable to donwload flow data, unmeasurable water in culvert.
8013 capping: rain gage empty, downloaded rain data at 1057

Seep 1: read staff gages, sampled 043-6 and 043-WB.

W1D: sampled historical weir.

5/7/99
8018 capping: unmeasurable flow in culvert, reprogrammed flow datalogger with 8018Dec8.scm program at 1234.
8013 capping: download of rain gage data at 1416, flow datalogger not responding to computer program so no information of flow data through the culvert, some water in ridges of the culvert.

5/14/99
8018 capping: little bit of water in gullies of culvert

Seep 1: read staff gages, water level lower, algae and plants growing near weir, sampled 043-6 and 043-WB, seeps in order of flow seep1-2,7,5,6,8.

W1D: historical weir had green algae all over, had to clear out prior to measuring flow depth, algae in wetland and cattail seeds made it difficult to get clean samples for W1D-1, W1D-2 and W1D-3 difficult.

6/15/99
light rain
8018 capping: downloaded flow data at 1410, little bit of water in culvert ridges with green slime.
8013: culvert dry, still unable to get flow datalogger to respond to computer, downloaded rain data at 1500.

Seep 1: routine recording of flow an collection of water samples, water over limestone berms.

W1D: routine flow data and water measurements W1D-1, W1D-2 and W1D-3

7/7/99
Heavy rainstorm on 4th of July.
8018 capping: computer not working no flow data downloaded, little bit of water in culvert ridges.
8013: culvert dry, some debree over grate so flow must have been fast and/or high, still unable to get datalogger to respond to computer, downloaded rain data at 1443.
Seep 1:routine flow measurements and water samples, pond very full, overflow pipe (pond side) partially in the water but not overflowing at this time, horsetail grown up around the boardwalk to the weir, sampled seeps
W1D: routine flow measurements, good flow over boardwalk at sampling sites in wetland.

7/13/99
thunderstorms in AM
8018 capping: download of flow data at 1438, slight flow into culvert from right side

7/28/99
rain earlier in week
8018 capping: download of flow data at 1148, slight flow into culvert from right side
8013 capping: download of rain gage data 1212

Seep 1:routine sampling of flow data and water samples, water level down in pond and precipitate and decaying plants.

W1D: routine flow measurements.

8/6/99
8018 capping: download of flow data at 1408, no flow into culvert
8013 capping: download of rain gage data 1430, removed flow meter from culvert for repairs

Seep 1:routine flow measurements and water samples.

W1D: routine flow measurements, and water samples in wetland

8/20/99
Seep 1:routine flow measurements and water samples

W1D: routine flow measurements

9/10/99
Seep 1:routine flow measurements and water samples

W1D: routine flow measurements, and water samples in wetland

9/23/99
8018 capping: download of flow data at 0850, no flow into culvert
8013 capping: download of rain gage data

Seep 1:routine flow measurements and water samples

W1D: routine flow measurements, and water samples in wetland

10/8/99
Seep 1:routine flow measurements and water samples.
W1D: routine flow measurements

10/28/99
8018 capping: download of flow data at 1024, no flow into culvert
8013 capping: download of rain gage data, emptied rain gage and covered with plastic for the winter.

Seep 1:routine flow measurements

W1D: routine flow measurements

11/16/99
8018 capping: downloaded flow data before winter

Appendix 7

Mass removal calculations

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Attachment A7.1. Methods used to estimate mass removal in the W1D wetland, using input/output flow and concentration data.

Nickel and sulfate mass removal was calculated for the W1D wetland treatment system using flow and water quality data from the historical input to the system (W1D-051) and from the outfall of the system (W1D-052).

An extension to the system was added in 1995, and the outfall from the extension area is now called W1D-050, which is the same named used for the original outfall (now called W1D-052) before the extension was added. This report focuses solely on the original portion of the system, so sites W1D-051 and W1D-052 are the relevant sites.

Flow at this site is recorded by level recorders (Stephens Type F), which record the staff gage reading at each weir, and which operate continuously once they are installed. The data from the strip chart is broken down into 24-hour increments, and then an estimate is made of the average flow for each time period. (These estimates appear to be based on visual approximations of the charts.) These daily values are then converted into flow values via a standard weir equation, and the total flow for the month is determined by totaling these daily values. The value LTV reports in their NPDES reports are these total monthly values divided by the number of days in the month.

For accurate flow measurements, elevation measurements of the staff gage and v-notch should be taken every year after ice out. If the staff gage has shifted it should be reset or corrections should be made. Flow measurements should be made and a rating curve developed which relates staff readings to flow.

LTV does not take any flow measurements, and it is unclear how often elevation measurements are made. In 1993 a comparison of actual flow measurements with LTV'standard weir equation indicated that there was about a 15% difference between measured and estimated flow.

The level recorders typically don't get installed until late April or early May, and are typically removed at the end of October, after freezing conditions resume. The flow values reported for April, November and December are generally the result of two or three visual observations made of the staff gage at the time that water quality samples are collected, and are therefore much more prone to error than data from the level recorders. However, since there is generally little flow or precipitation in November and December, an average value based on spot readings provides a reasonable estimate of average flow.

A more imporant question is how much flow goes unrecorded in the spring, prior to micrologger installation. When the snowpack in the W1D watershed melts (typically in April), there is usually a spike in flow through both monitoring sites (051 and 052). If the visual staff gage observations in the spring happen to coincide with this spike, the flow reported as the average flow for that month may be too high, since it assumes this 'spike' condition persists for two weeks (until the next observation is made). And if this flow spike occurs when no one is there to observe it, the flow

reported as average flow for that month will be too low, and could be significantly higher.

In the W1D system, however, input (i.e. W1D-051) concentrations have historically been low in the spring (approximately 1 mg/L before watershed manipulations began at W1D), and increased dramatically in June and July, to about 7-10 mg/L in July. The fact that most of the potentially unrecorded flow at this site occurs at the time of year when input nickel concentrations are low means that the unreported nickel mass is probably minor in comparison to the summer mass load.

(The possibility exists that nickel concentrations in the first melt water may be somewhat elevated, since it may rinse off some of the oxidation products that occur in the sulfide-bearing waste rock stockpiles in the W1D watershed. In general, however, nickel concentrations have been at their lowest in April, leading to the assumption that this rinse-off nickel load is probably low.)

There were also two time periods when the flow data reported for a particular month seemed too high in relation to the corresponding precipitation and to the data from the other weir:

1. September 30 and October 1, 1995. The flow values reported for W1D-051 were too high because the strip chart was misread. The staff gage values for both days was reported as 3.00, while the correct value for both days was approximately 2.1. This means that the reported value of 2.316 mgd (for both days) was actually 0.164 mgd, dropping the total flow for September from 14,647 to 6,586 m^3, and dropping the total October flow from 16427 to 8213 m^3. This in turn dropped the annual total for 1995 from 56,941 m^3 to 31,319 m^3.

   (The corresponding values for 052 were also high for these two days, but those values appear to accurately reflect flow that resulted from precipitation during that time period, and are not the result of a misread chart.)

2. October 11-13, 1997. Reported values were 2.98 and 2.95, corresponding to flow rates of 1.593, 1.530 and 1.438 mgd, respectively. It appears that the correct values are 2.09, 2.105 and 2.050, which correspond to 0.041, 0.043 and 0.034, respectively. This changed the total monthly flow from 4902 m^3 to 1756 m^3, and the annual total from 15,019 to 11,874 m^3.

Using these adjusted flow values, the total input and output mass load to the system was calculated. The average monthly flows were multiplied by the average monthly concentration, using all available concentration data. (For the spring and fall periods, when a value was reported for that month, flow was assumed to have occurred for the entire month.) While the flow data generally reflect daily values, however, the water quality data used in these mass calculations consist primarily of two grab samples collected each month, meaning that concentration increases and decreases may not have been reflected in the monthly averages. Table A7.1 and A7.2 summarize overall mass removal for the original W1D treatment system. Table A7.3 compares annual flow and mass estimates with values calculated for May to October (the period of accurate flow data).

Table A7.1. Nickel input, output and mass removal in the W1D wetland treatment system, 1992-99.

Date	Volume (m^3) In	Volume (m^3) Out	[Ni], mg/L In	[Ni], mg/L Out	Input mass (kg) Month	Input mass (kg) Annual	Input mass (kg) Cum.	Output mass (kg) Month	Output mass (kg) Annual	Output mass (kg) Cum.	Removed (kg)	Cum. Rem. (kg)
may 92	5162.7	6922.7	2.850	0.306	14.714	14.714	14.714	2.118	2.118	2.118	12.595	12.595
june 92	4996.2	6585.9	5.180	0.289	25.880	40.594	40.594	1.903	4.022	4.022	23.977	36.572
july 92	6101.4	8096.1	5.750	0.323	35.083	75.677	75.677	2.615	6.637	6.637	32.468	69.041
aug 92	5045.4	6805.4	3.560	0.113	17.962	93.639	93.639	0.769	7.406	7.406	17.193	86.233
sept 92	7494.3	9992.4	4.150	0.454	31.101	124.740	124.740	4.537	11.942	11.942	26.565	112.798
oct 92	4458.7	5984.0	5.450	0.464	24.300	149.040	149.040	2.777	14.719	14.719	21.523	134.321
nov 92	1135.5	1362.6	3.600	0.525	4.088	153.128	153.128	0.715	15.434	15.434	3.372	137.694
dec 92	1408.0	1290.6	3.750	0.370	5.280	158.408	158.408	0.478	15.912	15.912	4.803	142.496
may 93	1642.6	8448.1	0.901	0.121	1.480	1.480	159.888	1.022	1.022	16.934	0.458	142.954
jun 93	2611.6	3292.9	1.496	0.063	3.907	5.387	163.795	0.207	1.230	17.142	3.700	146.654
jul 93	9386.8	14432.2	5.278	0.626	49.544	54.931	213.339	9.035	10.264	26.176	40.509	187.163
aug 93	7157.4	5162.7	5.975	0.563	42.766	97.697	256.105	2.907	13.171	29.083	39.859	227.022
sep 93	4655.5	5109.7	6.550	0.830	30.494	128.190	286.598	4.241	17.412	33.324	26.253	253.275
oct 93	3285.3	2933.3	5.550	0.780	18.234	146.424	304.832	2.288	19.700	35.612	15.946	269.220
nov 93	2043.9	794.8	4.900	0.800	10.015	156.439	314.847	0.636	20.336	36.248	9.379	278.600
dec 93	1525.3	586.6	3.550	0.380	5.415	161.854	320.262	0.223	20.559	36.471	5.192	283.792
may 94	2933.3	4106.7	1.175	0.165	3.447	3.447	323.709	0.678	0.677	37.148	2.769	286.561
jun 94	10219.5	13626.0	3.014	0.280	30.802	34.249	354.511	3.815	4.493	40.964	26.986	313.547
jul 94	8096.1	9738.8	6.250	0.154	50.601	84.849	405.111	1.500	5.992	42.463	49.101	362.648
aug 94	5632.0	4576.0	5.517	0.113	31.072	115.922	436.184	0.517	6.509	42.980	30.555	393.203
sep 94	6131.7	9992.4	5.100	0.375	31.272	147.193	467.455	3.747	10.257	46.728	27.525	420.728
oct 94	2933.3	3989.3	5.668	0.743	16.626	163.820	484.082	2.964	13.221	49.692	13.662	434.390
nov 94	3065.8	2952.3	5.750	1.288	17.629	181.448	501.710	3.803	17.023	53.494	13.826	448.216
dec 94	1760.0	1525.3	5.250	1.550	9.240	190.688	510.950	2.364	19.388	55.859	6.876	455.092
apr 95	794.8	2157.4	1.550	0.670	1.232	1.232	512.182	1.445	1.445	57.304	-0.213	454.878
may 95	586.6	7626.7	1.200	0.190	0.704	1.936	512.886	1.449	2.894	58.753	-0.745	454.133
jun 95	113.5	6699.4	1.400	0.130	0.159	2.095	513.045	0.871	3.765	59.624	-0.712	453.421
jul 95	4928.0	6570.7	1.800	0.525	8.871	10.966	521.916	3.450	7.215	63.074	5.421	458.842
aug 95	6453.4	6218.7	2.150	0.135	13.875	24.841	535.791	0.840	8.054	63.913	13.035	471.878
sep 95	6586.0	8629.8	2.200	0.345	14.489	39.330	550.280	2.977	11.031	66.890	11.512	483.389
oct 95	8213.0	15722.8	2.950	0.800	24.228	63.558	574.508	12.578	23.610	79.469	11.650	495.039
nov 95	3292.9	2611.6	3.450	0.800	11.361	74.918	585.868	2.089	25.699	81.558	9.271	504.310
dec 95	352.0	704.0	2.350	0.800	0.827	75.746	586.696	0.563	26.262	82.121	0.264	504.574

Table A7.1. Nickel input, output and mass removal in the W1D wetland treatment system, 1992-99 (continued).

Date	Volume (m^3) In	Volume (m^3) Out	[SO_4], mg/L In	[SO_4], mg/L Out	Input mass (kg) Month	Input mass (kg) Annual	Input mass (kg) Cum.	Output mass (kg) Month	Output mass (kg) Annual	Output mass (kg) Cum.	Removed (kg)	Cum. Rem. (kg)
apr 96	12036.3	13739.5	0.853	0.599	10.267	10.267	596.963	8.230	8.230	90.351	2.037	506.611
may 96	2698.7	2698.7	0.873	0.594	2.356	12.623	599.319	1.603	9.833	91.954	0.753	507.364
jun 96	3747.1	1135.5	1.195	0.372	4.478	17.100	603.796	0.422	10.256	92.377	4.055	511.420
jul 96	5045.4	5162.7	0.918	0.272	4.632	21.732	608.428	1.404	11.660	93.781	3.227	514.647
aug 96	3637.3	3050.7	0.505	0.138	1.837	23.569	610.265	0.421	12.081	94.202	1.416	516.063
sep 96	3065.8	2838.7	0.922	0.073	2.827	26.396	613.092	0.207	12.288	94.409	2.619	518.682
oct 96	3520.0	7626.7	1.478	0.253	5.203	31.598	618.294	1.930	14.218	96.339	3.273	521.955
nov 96	3406.5	1476.1	1.377	0.670	4.691	36.289	622.985	0.989	15.207	97.328	3.702	525.657
apr 97	2611.6	6245.2	0.787	0.468	2.055	2.055	625.040	2.923	2.923	100.251	-0.867	524.790
may 97	1408.0	2112.0	0.663	0.260	0.934	2.989	625.974	0.549	3.472	100.800	0.384	525.174
jun 97	1703.2	2157.4	0.685	0.090	1.167	4.156	627.141	0.194	3.666	100.994	0.973	526.147
jul 97	821.3	352.0	0.728	0.109	0.598	4.754	627.739	0.038	3.704	101.032	0.560	526.706
aug 97	1290.6	704.0	0.422	0.067	0.545	5.298	628.283	0.047	3.751	101.079	0.498	527.204
sep 97	794.8	113.5	0.330	0.058	0.262	5.561	628.546	0.007	3.758	101.086	0.256	527.460
oct 97	1756.2	3050.7	0.476	0.190	0.836	6.397	629.382	0.580	4.338	101.666	0.256	527.716
nov 97	1135.5	1930.3	0.953	0.248	1.082	7.479	630.464	0.479	4.816	102.144	0.603	528.319
dec 97	352.0	58.6	0.808	0.050	0.284	7.763	630.748	0.003	4.819	102.147	0.281	528.601
may 98	1290.6	1290.6	0.503	0.158	0.649	0.649	631.397	0.204	0.204	102.351	0.445	529.046
jun 98	1135.5	908.4	0.386	0.057	0.438	1.088	631.836	0.052	0.256	102.403	0.387	529.433
jul 98	938.6	117.3	0.462	0.024	0.434	1.521	632.269	0.003	0.259	102.406	0.431	529.863
aug 98	1290.6	352.0	0.398	0.009	0.514	2.035	632.783	0.003	0.262	102.409	0.511	530.374
sep 98	1249.0	227.1	0.190	0.017	0.237	2.272	633.020	0.004	0.266	102.413	0.233	530.607
oct 98	2581.3	2464.0	0.432	0.143	1.115	3.387	634.135	0.352	0.618	102.765	0.763	531.370
nov 98	794.8	454.2	0.622	0.066	0.494	3.882	634.630	0.030	0.648	102.795	0.464	531.835
dec 98	1290.6	352.0	0.682	0.050	0.880	4.762	635.510	0.018	0.666	102.813	0.863	532.697
may 99	1760.0	4106.7	0.419	0.115	0.737	0.738	636.248	0.472	0.472	103.285	0.265	532.962
jun 99	1589.7	2384.5	0.303	0.140	0.482	1.219	636.729	0.334	0.806	103.619	0.148	533.110
jul 99	6570.7	12554.8	0.255	0.208	1.676	2.895	638.405	2.611	3.417	106.230	-0.936	532.174
aug 99	2346.7	1760.0	0.162	0.093	0.380	3.275	638.785	0.164	3.581	106.394	0.216	532.391
sep 99	4882.6	5791.0	0.347	0.061	1.694	4.969	640.479	0.353	3.934	106.747	1.341	533.732
oct 99	4458.7	3754.7	1.455	0.145	6.487	11.457	646.967	0.544	4.479	107.292	5.943	539.675
nov 99	2611.6	908.4	1.470	0.128	3.839	15.296	650.806	0.116	4.595	107.408	3.723	543.398
dec 99	4106.7	1173.3	0.063	.	0.259	15.555	651.065	.	.	.	.	.

Note: The original flow data for Sept and Oct 1995 (for 0510 were mis-read from the strip charts. The data in this table include the corrected data.

Table A7.2. Sulfate input, output and mass removal in the W1D wetland treatment system, 1992-99.

Date	Volume (m^3) In	Volume (m^3) Out	[SO_4], mg/L In	[SO_4], mg/L Out	Input mass (kg) Month	Input mass (kg) Annual	Input mass (kg) Cum.	Output mass (kg) Month	Output mass (kg) Annual	Output mass (kg) Cum.	Removed (kg)	Cum. Rem. (kg)
may 92	5162.7	6922.765	1649	1027	8513.3	8513.3	8513.3	7109.6	7109.6	7109.6	1403.6	1403.6
june 92	4996.2	6585.900	2691	1769	13444.7	21958.1	21958.1	11650.4	18760.1	18760.1	1794.3	3197.9
july 92	6101.4	8096.115	2536	1772	15473.2	37431.3	37431.3	14346.3	33106.4	33106.4	1126.8	4324.8
aug 92	5045.4	6805.430	2923	2234	14747.7	52179.0	52179.0	15203.3	48309.7	48309.7	0.0	4324.8
sept 92	7494.3	9992.400	2927	2593	21935.8	74114.8	74114.8	25910.2	74220.0	74220.0	0.0	4324.8
oct 92	4458.7	5984.085	2600	2380	11592.6	85707.5	85707.5	14242.1	88462.1	88462.1	0.0	4324.8
nov 92	1135.5	1362.600	2536	2246	2879.6	88587.1	88587.1	3060.4	91522.5	91522.5	0.0	4324.8
dec 92	1408.0	1290.685	2236	1414	3148.3	91735.5	91735.5	1825.0	93347.6	93347.6	1323.3	5648.1
may 93	1642.6	8448.120	931	911	1529.3	1529.3	93264.8	7696.2	7696.2	101043.8	0.0	5648.1
jun 93	2611.6	3292.950	1305	771	3408.2	4937.5	96673.0	2538.8	10235.1	103582.7	869.3	6517.5
jul 93	9386.8	14432.205	2094	1367	19655.9	24593.5	116329.0	19728.8	29963.9	123311.5	0.0	6517.5
aug 93	7157.4	5162.740	2069	1803	14808.7	39402.2	131137.7	9308.4	39272.3	132619.9	5500.3	12017.8
sep 93	4655.5	5109.750	2809	1987	13077.4	52479.6	144215.2	10153.0	49425.4	142773.0	2924.3	14942.2
oct 93	3285.3	2933.375	2637	2093	8663.5	61143.2	152878.7	6139.5	55564.9	148912.6	2523.9	17466.1
nov 93	2043.9	794.850	2727	2118	5573.7	66716.9	158452.4	1683.4	57248.4	150596.0	3890.2	21356.4
dec 93	1525.3	586.675	2451	2088	3738.6	70455.5	162191.1	1224.9	58473.4	151821.0	2513.6	23870.0
may 94	2933.3	4106.725	1093	620	3206.1	3206.1	165397.2	2546.1	2546.1	154367.2	660.0	24530.0
jun 94	10219.5	13626.000	1994	1155	20377.6	23583.8	185774.9	15738.0	18284.1	170105.2	4639.6	29169.7
jul 94	8096.1	9738.805	2641	1702	21381.8	44965.7	207156.8	16575.4	34859.6	186680.7	4806.3	33976.1
aug 94	5632.0	4576.065	2440	1814	13742.2	58707.9	220899.0	8300.9	43160.6	194981.6	5441.2	39417.4
sep 94	6131.7	9992.400	2351	1860	14415.6	73123.6	235314.7	18585.8	61746.4	213567.5	0.0	39417.4
oct 94	2933.3	3989.390	2282	1764	6693.9	79817.5	242008.6	7037.2	68783.7	220604.8	0.0	39417.4
nov 94	3065.8	2952.300	2515	1704	7710.6	87528.1	249719.2	5030.7	73814.4	225635.5	2679.8	42097.3
dec 94	1760.0	1525.355	2196	2154	3865.0	91393.1	253584.3	3285.6	77100.1	228921.1	579.4	42676.7
apr 95	794.8	2157.450	697	766	554.0	554.0	254138.3	1652.6	1652.6	230573.7	0.0	42676.7
may 95	586.6	7626.775	1297	1248	760.9	1314.9	254899.2	9518.2	11170.8	240092.0	0.0	42676.7
jun 95	113.5	6699.450	1448	1017	164.4	1479.3	255063.6	6813.3	17984.1	246905.3	0.0	42676.7
jul 95	4928.0	6570.760	1716	1125	8456.5	9935.9	263520.2	7392.1	25376.2	254297.4	1064.4	43741.1
aug 95	6453.4	6218.755	2046	1601	13203.7	23139.6	276723.9	9956.2	35332.4	264253.6	3247.4	46988.6
sep 95	6586.0	8629.800	2031	1395	13376.0	36515.6	306473.9	12038.5	47371.0	276292.2	1337.4	48326.1
oct 95	8213.0	15722.890	2193	884	18011.0	54526.6	342498.1	13899.0	61270.1	290191.2	4111.9	52438.0
nov 95	3292.9	2611.650	2193	884	7221.4	61748.0	349719.5	2308.6	63578.7	292499.9	4912.7	57350.8
dec 95	352.0	704.010	2193	884	771.9	62520.0	350491.4	622.3	64201.1	293122.3	149.6	57500.4

Table A7.2. Sulfate input, output and mass removal in the W1D wetland treatment system, 1992-99 (continued).

Date	Volume (m^3) In	Volume (m^3) Out	[SO_4], mg/L In	[SO_4], mg/L Out	Input mass (kg) Month	Input mass (kg) Annual	Input mass (kg) Cum.	Output mass (kg) Month	Output mass (kg) Annual	Output mass (kg) Cum.	Removed (kg)	Cum. Rem. (kg)
apr 96	12036.3	13739.550	809	679	9737.3	9737.3	360228.8	9329.1	9329.1	302451.4	408.2	57908.6
may 96	2698.7	2698.705	1100	1597	2968.5	12705.9	363197.4	4309.8	13638.9	306761.3	0.0	57908.6
jun 96	3747.1	1135.500	1525	1340	5714.4	18420.3	368911.8	1521.5	15160.5	308282.8	4192.8	62101.4
jul 96	5045.4	5162.740	1450	1073	7315.8	25736.1	376227.6	5539.6	20700.1	313822.4	1776.2	63877.6
aug 96	3637.3	3050.710	1940	1460	7056.5	32792.7	383284.2	4454.0	25154.2	318276.5	2602.4	66480.1
sep 96	3065.8	2838.750	1837	1535	5631.9	38424.6	388916.1	4357.4	29511.6	322634.0	1274.4	67754.6
oct 96	3520.0	7626.775	1780	1540	6265.6	44690.3	395181.8	11745.2	41256.9	334379.2	0.0	67754.6
nov 96	3406.5	1476.150	1161	1035	3954.9	48645.3	399136.8	1527.8	42784.7	335907.0	2427.1	70181.7
apr 97	2611.6	6245.250	785	595	2050.1	2050.1	401186.9	3715.9	3715.9	339622.9	0.0	70181.7
may 97	1408.0	2112.030	892	783	1255.9	3306.0	402442.9	1653.7	5369.6	341276.7	0.0	70181.7
jun 97	1703.2	2157.450	1067	758	1817.3	5123.4	404260.2	1635.3	7004.9	342912.0	182.0	70363.8
jul 97	821.3	352.005	1060	760	870.6	5994.0	405130.9	267.5	7272.5	343179.5	603.1	70966.9
aug 97	1290.6	704.010	1075	780	1387.4	7381.5	406518.3	549.1	7821.6	343728.7	838.3	71805.2
sep 97	794.8	113.550	1145	946	910.1	8291.6	407428.4	107.4	7929.0	343836.1	802.6	72607.9
oct 97	1756.2	3050.710	1060	933	1861.6	10153.2	409290.1	2846.3	10775.3	346682.4	0.0	72607.9
nov 97	1135.5	1930.350	973	781	1104.8	11258.1	410394.9	1507.6	12282.9	348190.0	0.0	72607.9
dec 97	352.0	58.668	993	769	349.5	11607.6	410744.4	45.1	12328.0	348235.1	304.4	72912.3
may 98	1290.6	1290.685	794	751	1024.8	1024.8	411769.2	969.3	969.3	349204.4	55.4	72967.8
jun 98	1135.5	908.400	1050	737	1192.2	2217.0	412961.5	669.4	1638.7	349873.9	522.7	73490.6
jul 98	938.6	117.335	1082	808	1015.6	3232.7	413977.2	94.8	1733.6	349968.7	920.8	74411.5
aug 98	1290.6	352.005	1065	574	1374.5	4607.3	415351.7	202.0	1935.6	350170.8	1172.5	75584.0
sep 98	1249.0	227.100	1135	937	1417.6	6024.9	416769.4	212.7	2148.4	350383.6	1204.8	76788.9
oct 98	2581.3	2464.035	859	841	2217.3	8242.3	418986.8	2072.2	4220.6	352455.8	145.1	76934.0
nov 98	794.8	454.200	1175	1055	933.9	9176.3	419920.8	479.1	4699.8	352935.0	454.7	77388.8
dec 98	1290.6	352.005	979	1016	1263.5	10439.9	421184.3	357.6	5057.5	353292.6	905.9	78294.7
may 99	1760.0	4106.725	936	710	1647.3	1647.3	422831.7	2915.7	2915.7	356208.4	0.0	78294.7
jun 99	1589.7	2384.550	1115	765	1772.5	3419.8	424604.2	1824.1	4739.9	358032.6	0.0	78294.7
jul 99	6570.7	12554.845	1043	629	6853.3	10273.2	431457.5	7896.9	12636.9	365929.6	0.0	78294.7
aug 99	2346.7	1760.025	1465	857	3437.9	13711.1	434895.5	1508.3	14145.2	367437.9	1929.5	80224.3
sep 99	4882.6	5791.050	1440	890	7031.0	20742.1	441926.5	5154.0	19299.3	372592.0	1876.9	82101.3
oct 99	4458.7	3754.720	1140	760	5082.9	25825.0	447009.4	2853.5	22152.9	375445.5	2229.3	84330.6
nov 99	2611.6	908.400	1450	1080	3786.8	29611.9	450796.3	981.0	23133.9	376426.6	2805.8	87136.5

Note: The original flow data for Sept and Oct 1995 (for 0510 were mis-read from the strip charts. The data in this table include the corrected data.

Table A7.3. Comparison of May-October flow and mass vs. annual estimates, W1D, 1992-1999.

Year	Precipitation (in)			Input Flow (10^3 L)			Output Flow (10^3 L)			Input Nickel Mass (kg)			Output Nickel Mass (kg)		
	May-Oct	Annual	% Diff.	May-Oct	Annual	% Diff.	May-Oct	Annual	% Diff.	May-Oct	Annual	% Diff.	May-Oct	Annual	% Diff.
1992	17.3	26.9	35.7	33,258	35,802	7.1	44,386	47,039	5.6	134.3	158.4	15.2	14.7	15.9	7.5
1993	17.4	29.7	41.4	28,739	32,308	11.0	39,379	40,760	3.4	146.4	161.9	9.6	19.7	20.6	4.4
1994	19.2	28.7	33.1	35,946	40,772	11.8	46,029	50,507	8.9	163.8	190.7	14.1	13.2	19.4	31.9
1995	18.5	25.6	27.7	26,881	31,320	17.4	51,468	56,941	9.6	63.6	75.7	16.0	22.2	26.3	15.6
1996	18.2	34.2	46.8	21,714	37,157	41.5	22,513	37,728	40.3	21.3	36.3	41.3	6.0	15.2	60.5
1997	15.5	22.9	32.3	7,775	11,873	34.5	8,490	16,724	49.2	4.3	7.8	44.9	1.4	4.8	70.8
1998	19.2	31.1	38.3	8,485	10,571	19.7	5,359	6,165	13.1	3.4	4.8	29.2	0.6	0.7	14.3
1999	28.8	35.2	18.2	21,608	28,326	23.7	30,351	32,433	6.4	11.5	15.6	26.3	4.4	4.7	6.4
Ave. 92-99	20.8	28.6	34.2	23,051	28,516	20.8	28,497	36,037	17.0	68.6	81.4	24.6	10.3	13.4	26.4

* Flow at W1D is monitored with level recorders that, due to freezing conditions, are typically installed in early May and removed by early November. The "annual" columns include any data reported for the spring (i.e. before May) and fall (i.e. after October), which are generally based on visual observations of staff gage levels made when personnel are collecting the (usually twice monthly) water quality samples, with the assumption that flow occurred for the entire month at the reported flow rate.

Attachment A7.2. Methods used to estimate mass removal in the DNR study cell of the W1D wetland.

Section A7.2.1 Using input/output flow and concentration data.

The W1D system was started in 1992. The original outfall was called W1D-050, but when the extension to the system was put on-line in July 1995, the name of the original outfall was changed from 050 to 052. The site now referred to as 050 is the outfall from the expansion area.

An estimate of nickel removal was made using average monthly flow and nickel concentrations in the input (051) and output (052). The flow values reflect the average of the daily values, while the water quality samples usually reflect two grab samples that were collected each month. Table A7.2 summarizes the nickel mass removal calculations made from the water quality/quantity data.

As depicted in Figure A7.1, 526 kg of nickel was calculated to have been removed in the entire system (not including the extension) from the start of the system in 1992 until the completion of the 1996 season. Profiles water quality samples were collected from 1992 through 1994 in the original system, when samples were collected simultaneously throughout the system. After excluding any data prior to July (since input nickel concentrations were typically low prior to July), the portion of the system focused on in this report was found to be responsible for 25.8% of the overall removal by the system.

$$526 \text{ kg} \times 0.258 = 135.7 \text{ kg nickel}$$

Section A7.2.2 Multiplying the peat mass in the cell by the mean nickel concentration in each peat layer.

The second method used to calculate nickel removal in the DNR study cell was to multiply the mass of peat present in the cell by the average nickel concentration of all samples (in each peat layer).

Area Planimeter measurements of the DNR stucy cell indicated an area of approximately 15,000 square feet. (This value is for the entire segment, and didn't exclude the dry peninsula that exists on the west side of the segment.)

The computer model (Tech Base) used to estimate the nickel mass in the peat came up with a similar number (16,000 square feet). But it appears that because the computer model didn't extrapolate past the final data points on the west side, Tech Base may have underestimated the actual area. Some of this excluded area is the dry peninsula on the north side of the section, but some should probably have been included. But for these mass calculations we assume there are 16,000 square feet.

$16000 \text{ ft}^2 = 1487 \text{ m}^2$

$1487 \text{ m}^2 \times 0.1 \text{ m}$ (in each 10 cm segment of the peat) $= 148.7 \text{ m}^3$

Figure A7.1. Nickel mass input, output and removal in the original W1D wetland treatment system.

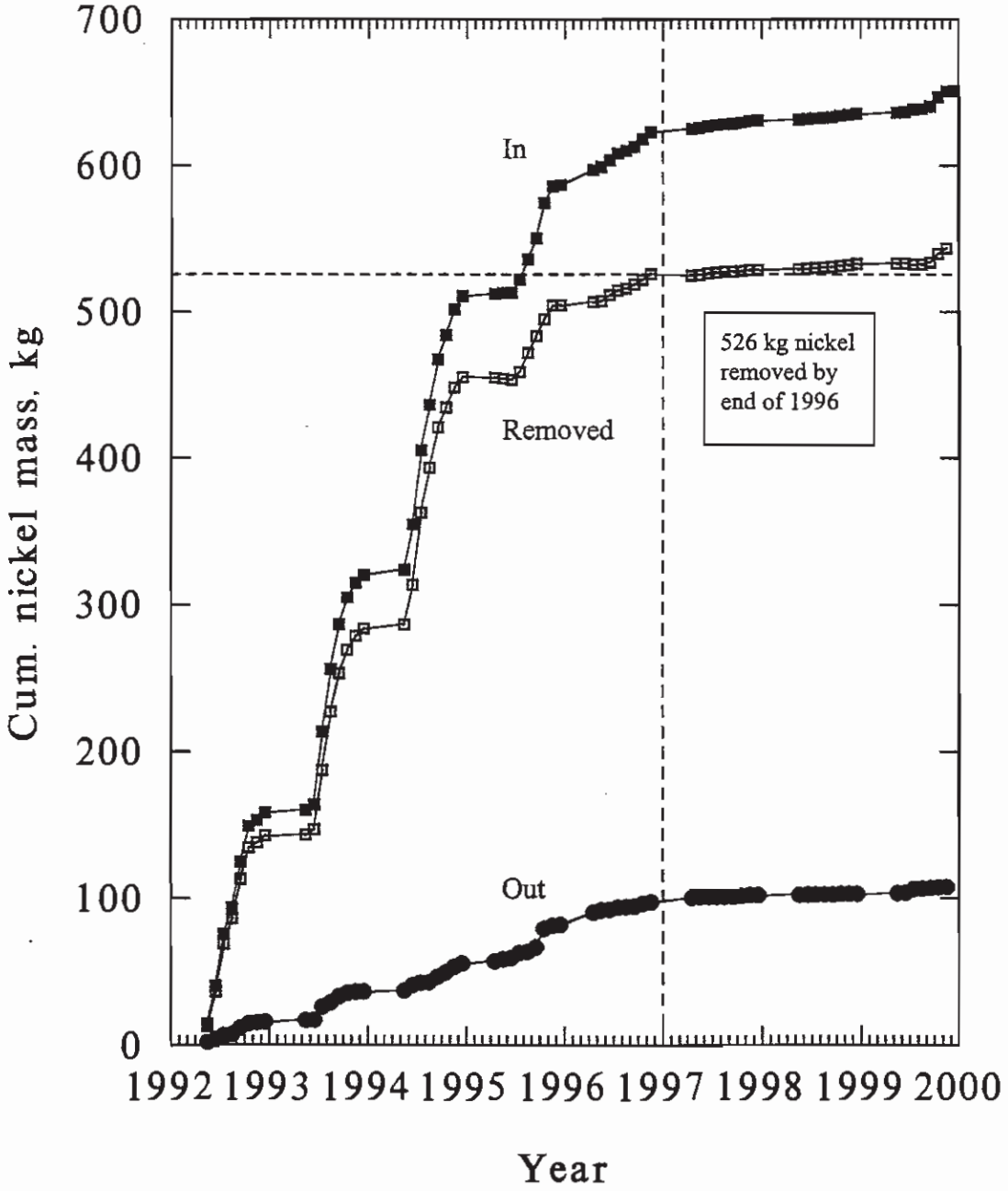


Table A7.4. Mass removal estimates for the DNR study cell of the original W1D system.

Calculation method	Peat layer	Nickel mass (kg)
(A) Combining input/output flow and nickel concentration data.	---	135.7
(B) Peat mass x mean nickel concentration of each layer	0 - 10 cm	65.0
	10 - 20 cm	33.7
	20 - 30 cm	27.7
	Total:	126.4
(C) Peat mass x concentration (as determined by Tech Base)	0 - 10 cm	71.5
	10 - 20 cm	27.1
	20 - 30 cm	13.9
	Total:	112.5
(D) Average (This includes methods A and B for the individual layers, and methods A, B and C for the total values)	0 - 10 cm	68.2
	10 - 20 cm	30.4
	20 - 30 cm	20.8
	Total:	124.9

(A) Based on the results of profiles studies carried out within the W1D wetland from 1992 through 1994, it was determined that 25.8% of the overall removal within the system occurs in this single segment. The overall removal value for 1992-1996 (526 kg) was multiplied by 0.258 to yield 135.7 kg of nickel removed in the study segment.

(B) The average nickel concentration of each peat layer (0-10, 10-20 and 20-30 cm) was multiplied by the mass of dry peat assumed to be present in each 10-cm layer (13100 kg). This was based on an area of 1310 m^2 (14,098 ft^2), which was the value used in the Tech Base model (method 3), and on the assumption that the wet peat was 90% water. All available 1996 and 1997 peat data was used to calculate the means (4959 mg/kg in 0-10 cm layer, 2571 mg/kg in 10-20 cm layer, and 2112 mg/klg in 20-30 cm layer). These estimates exclude the dry peninsula area (i.e. removal is assumed to be zero in the peninsula). A few miscellaneous peat samples had unusual peat lengths (i.e. 0 - 8 cm instead of the usual 0-10 cm length), and these were grouped together with the standard core segments prior to calculating the means for each layer.

(C) Dan Steinbrink of the MDNR used a software program used Tech Base to calculate concentration contours within the W1D study segment, and then to calculate nickel mass in each layers based on these contours. These estimates also exclude peninsula area.

(D) Overall results from all three methods were used to calculate average value of 1218.5.

Density The peat was assumed to be 90% water, so there are 100 kg of dry peat for every cubic meter of peat. 148.7 m^3 x 100 = 14870 kg of dry peat.

Average nickel concentrations The mean nickel concentration of the 0-10 cm segments of the 1997 peat samples was 6740mg/kg, or 6.74 g/kg (see appendix 4 for data about metals in peat). The nickel mass in this layer would be 100.2 kg. The mean nickel concentration of the 20-30 cm segment was 3310 mg/kg, or 49.2 kg. The mean nickel in the 0-30 layer was 910, or 13.5 kg. The total for these three segments, which the pilot study at W2d/3D had shown to be responsible for the vast majority of nickel removal via adsorption in the peat, is 162.9 kg nickel.

Section A7.2.3 Tech Base calculations, which are based on concentration contour lines.

The third method used to estimate nickel mass present in the DNR study cell was a software program called Tech Base, which calculates concentration contour lines, and then uses those contour lines to calculate the mass of nickel present in each contour interval (Appedix 4).

The wetland was divided into 20 ft by 20 ft grid and the model determined an average concentration for each cell (Ni mg/gm).

To convert into mass, a bulk density of 0.1 g dry peat/cm^3 was assumed.

The area of each cell is 20 ft x 20 ft and the sampling interval is 10 cm (.328 ft) so the volume of a cell is

$$20 \text{ ft} \times 20 \text{ ft} \times 0.328 \text{ ft} = 131.2 \text{ ft}^3$$

The mass of metal in grams in each cell is

$$\frac{2.83 \text{ kg dry peat}}{\text{ft}^3 \text{ of wetland}} \times 132.2 \text{ ft}^3 \text{ wetland} \times \frac{\text{mg metal}}{\text{kg dry peat}} \times \frac{1}{1000 \text{ mg/yr}}$$

Tech Base calculated that a total of 112.5 kg of nickel is present in the top 30 cm of the peat in the study cell (Table A7.2). This is low in comparison to the 146.5 kg estimate arrived at via the input/output data, and to the 126.4 kg value arrived at if you simply multiply the peat mass by the average metal concentrations. But the three estimates are remarkably similar when you take into account the limitations and assumptions inherent in simple models such as these, which include the relatively low number of water and peat samples, the potential for hydrological short-circuiting, and the always real possibility of laboratory/field error.

Attachment A7.3 Mass loads at Seep 1 (site 043, which is now the outfall of the pre-treatment system).

Until 1992, when the Seep 1 pretreatment system was built, the site called 043 was the actual seepage from the 8013 stockpile. But when the pretreatment system was put on line, the location of site 043 became the output of the pretreatment system (and prior to the wetland treatment system, which was started in 1995, and which is not discussed in this report). Thus all data reported for 043 from 1993 and on represents the effluent from the pretreatment system, and not the actual raw seepage.

A flow recorder was installed at the Seep 1 weir in 1986, so accurate estimations of flow are available from 1986-1991. The construction of the pretreatment system in 1992, however, makes it now impossible to collect truly representative "input" water quality and flow data. The seepage at this site occurs at a variety of small, diffuse seepages along the toe of 8013, and it is difficult to capture all of this diffuse flow. Attempts have been made to measure flow using small hand-held flumes, but only estimated data from 1993 are available. Two different methods were thus used to estimate the mass removed by the pretreatment system from the time of its construction through the 1996 field season.

1. Peat metal mass. Peat cores were collected in January 1996 from the vertical flow portion of the pretreatment system (Figure 8), and from the inner pool, middle pool and outer pool of the system in January 1997. Each peat core was divided into 10 cm segments (0-10 cm, 10-20 cm, 20-30 cm, etc.), and then those peat splits were processed and analyzed for metals content. Some individual cores had peat deeper than 30 cm, but those deeper layers were omitted from these calculations. Metal concentrations generally decreased with depth, so including deeper layers would add little to the total mass.

Mean concentrations were calculated for each peat layer in each pool, using all available data, and then mass calculations were made by multiplying the mean concentration in each segment by the dry peat weight estimated for that segment (Table 10). The total mass in the system was then estimated by adding the values from each 10 cm peat layer.

2. Using water quality data. **Output mass** was calculated for each month after the pretreatment system was started in 1992. This was done by multiplying the average concentration for that month by the total flow reported for 043 during that month in LTV's NPDES reports. These values should be quite accurate since they are based on daily flow data recorded at the weir.

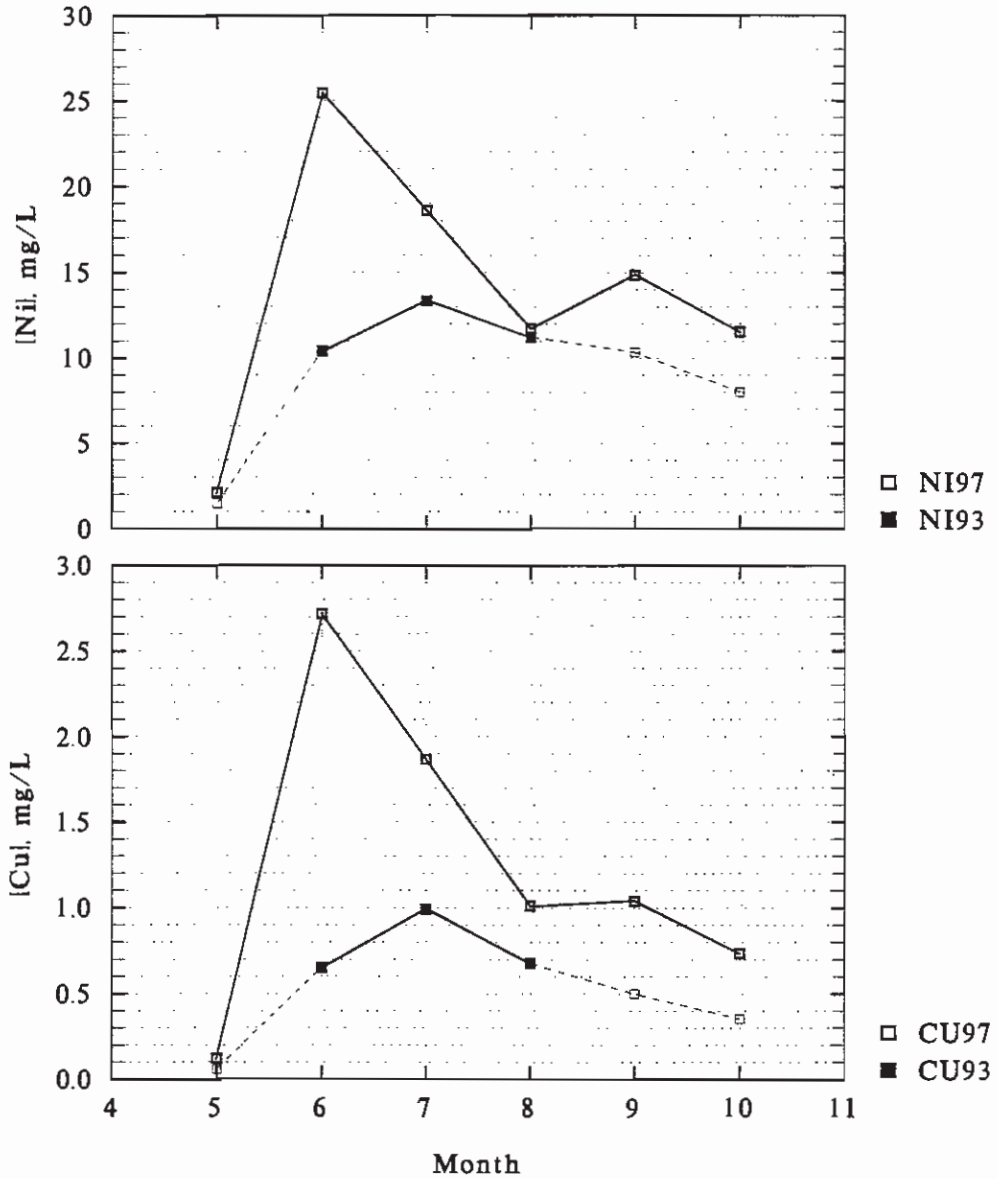
To estimate **input mass**, however, two assumptions were made. First, the average monthly input concentrations were assumed to be the same as the average monthly value observed in 1993-94 in the outer pool. This assumes that the samples around the edge of the outer pool represent the metal input to the pretreatment system. Based on water level data from the piezometers, groundwater enters the system in the outer pool. Since metal concentrations in substrate samples decreased with depth, there did not appear to be any removal until water had entered the pretreatment system.

Samples were also collected from the outer pool in 1997, but these samples were collected a year after the first of the peat samples were collected, while the 1993-94 samples were collected prior to peat collection. The 1997 values were also substantially higher than the 1993-94 values. Unfortunately, the 1993-94 samples were collected only during June, July and August. To estimate the "mean" values for May, September and October to be used in calculating the input masses, the average difference between the two sets of samples for June-August was calculated for each parameter. For copper the average 1993 value was 48.1% of the 1997 values, while the nickel value was 69.5% of the 1997 value (Figure A7.2).

To calculate the "mean" values to be used for May, September and October in the input mass calculations, the average value observed in 1997 was multiplied by these conversion factors. The values calculated for May, September and October were 1.46, 10.34 and 8.03 mg/L, respectively, while the copper values were 0.05, 0.50 and 0.35 mg/L, respectively. These monthly averages were then multiplied by the monthly flow reported for each month to arrive at a monthly input mass value, and then the monthly values were added to arrive at the annual mass removal.

An alternative estimation method would be to use the average monthly values observed before the system was constructed (i.e. using 1986-91 data), but this was rejected because nickel concentrations in the outer pool in 1993-94 were significantly less than the historical values from the seep. Since there was no evidence of removal from groundwater, it is likely that the nickel concentrations have decreased since 1991 due to the capping of 8013, which was completed in the fall of 1991. Also, while some flow occurs via discrete seeps at the toe of 8013, using these as an indicator of flow into the pretreatment system is unrealistic because a substantial portion of the flow observed at the weir results from groundwater flow.

Initially, mass removal calculations were done using the average of the 1993-94 and 1997 values. Using this approach, the total nickel removal was calculated as 130.5 kg. All mass removal estimates using the water quality data were at least twice the amound of nickel calculated to be in the substrate.



Notes: The only 1993 data available is from June through August. The averages for these months (using all available data from 043-1, 043-2 and 043-3) are depicted by the dark squares in the bottom line of each figure. To estimate 1993 concentrations for the other months, the average difference between the 1993 and 1997 data was calculated for each parameter (for June, July and August). For copper the average 1993 value was 48.1% of the 1997 values, while the nickel value was 69.5% of the 1997 value. The 1997 values were then multiplied by these conversion factors to get the estimated values for May, September and October. The exception was September, when the 1997 value was anomalously high; the value calculated for 1993 omits the anomalous value.

Figure A7.2. Mean copper and nickel concentrations in the surface water of the outer pool in 1993-94 and in 1997.

Table A7.5. Monthly mass load calculations for Seep 1, 1992-99.

	Monthly flow		Average Conc. (mg/L)			Mass (kg)			Cum. mass (kg)		
Date	MGD	Liters	Ni	Cu	SO^4	Ni	Cu	SO^4	Ni	Cu	SO^4
May 1992	0.015	1760025	14.450	1.270	1200	25.43	2.235	2112.0	25.432	2.235	2112.0
June 1992	0.009	1021950	24.000	1.800	1820	24.52	1.840	1859.9	49.959	4.075	3971.9
July 1992	0.009	1056015	13.800	1.400	1340	14.57	1.478	1415.0	64.532	5.553	5387.0
Aug 1992	0.006	704010	3.763	0.278	1540	2.64	0.196	1084.1	67.181	5.749	6471.2
Sept 1992	0.016	1816800	4.725	0.100	1980	8.58	0.182	3597.2	75.766	5.931	10068.4
Oct 1992	0.005	586675	5.233	0.071	2320	3.07	0.042	1361.0	78.836	5.972	11429.5
May 1993	0.010	1173350	7.467	0.073	1080	8.76	0.086	1267.2	8.761	0.086	1267.2
Jun 1993	0.011	1249050	7.467	0.073	1080	9.32	0.091	1348.9	18.088	0.177	2616.1
Jul 1993	0.019	2229365	8.850	0.320	1120	19.73	0.713	2496.8	37.818	0.890	5113.0
Aug 1993	0.007	821345	8.650	0.190	1350	7.10	0.156	1108.8	44.922	1.046	6221.8
Sep 1993	0.004	454200	7.200	0.130	1840	3.27	0.059	835.7	48.193	1.106	7057.6
Oct 1993	0.003	352005	6.467	0.090	1360	2.27	0.032	478.7	50.469	1.137	7536.3
May 1994	0.004	469340	1.350	0.105	300	0.63	0.049	140.8	0.634	0.050	140.8
Jun 1994	0.011	1249050	8.575	0.130	1490	10.71	0.162	1861.0	11.344	0.212	2001.8
Jul 1994	0.003	352005	8.950	0.320	1220	3.15	0.113	429.4	14.495	0.325	2431.3
Aug 1994	0.001	117335	6.750	0.115	1160	0.79	0.013	136.1	15.287	0.338	2567.4
Sep 1994	0.008	908400	7.200	0.210	1130	6.54	0.191	1026.4	21.827	0.529	3593.9
Oct 1994	0.004	469340	5.967	0.167	1120	2.80	0.078	525.6	24.628	0.607	4119.5
May 1995	0.001	117335	0.700	0.040	480	0.08	0.005	56.3	0.082	0.005	56.3
Jun 1995	0.001	113550	0.230	0.040	1240	0.02	0.005	140.8	0.108	0.009	197.1
Jul 1995	0.008	938680	4.550	0.070	1230	4.27	0.066	1154.5	4.379	0.075	1351.6
Aug 1995	0.007	821345	3.350	0.058	1390	2.75	0.048	1141.6	7.130	0.123	2493.3
Sep 1995	0.011	1249050	7.650	0.125	1120	9.55	0.156	1398.9	16.686	0.279	3892.3
Oct 1995	0.022	2581370	9.450	0.235	1020	24.39	0.607	2632.9	41.080	0.885	6525.3
May 1996	0.010	1173350	2.795	0.161	460	3.28	0.189	539.7	3.279	0.189	539.7
Jun 1996	0.010	1135500	5.920	0.193	970	6.72	0.219	1101.4	10.001	0.409	1641.1
Jul 1996	0.012	1408020	9.045	0.482	960	12.73	0.679	1351.6	22.737	1.087	2992.8
Aug 1996	0.006	704010	8.425	0.256	1240	5.93	0.180	872.9	28.668	1.267	3865.8
Sep 1996	0.004	454200	7.335	0.117	1320	3.33	0.053	599.5	32.000	1.321	4465.3
Oct 1996	0.008	938680	8.565	0.292	1200	8.04	0.274	1126.4	40.040	1.595	5591.8

Table A7.5. Monthly mass load calculations for Seep 1, 1992-99 (continued).

Date	Monthly flow		Average Conc. (mg/L)			Mass (kg)			Cum. mass (kg)		
	MGD	Liters	Ni	Cu	SO^4	Ni	Cu	SO^4	Ni	Cu	SO^4
May 1997	0.007	821345	1.500	0.078	490	1.23	0.064	402.4	1.232	0.064	402.4
Jun 1997	0.012	1362600	8.715	0.110	1130	11.87	0.150	1539.7	13.107	0.214	1942.1
Jul 1997	0.004	469340	11.700	0.239	1340	5.49	0.112	628.9	18.598	0.326	2571.1
Aug 1997	0.003	352005	7.400	0.135	1350	2.60	0.048	475.2	21.203	0.373	3046.3
Sep 1997	0.001	113550	7.680	0.116	1400	0.87	0.013	158.9	22.075	0.387	3205.2
Oct 1997	0.004	469340	7.680	0.116	1400	3.60	0.054	657.0	25.679	0.441	3862.3
May 1998	0.005	586675	5.443	0.178	920	3.19	0.104	539.7	3.194	0.104	539.7
Jun 1998	0.004	454200	13.050	0.234	1420	5.92	0.106	644.9	9.121	0.211	1184.7
Jul 1998	0.001	117335	8.465	0.109	1420	0.99	0.013	166.6	10.114	0.223	1351.3
Aug 1998	0.000	46934	7.510	0.079	1780	0.35	0.004	83.5	10.467	0.227	1434.8
Sep 1998	0.001	113550	7.725	0.080	1400	0.87	0.009	158.9	11.344	0.236	1593.8
Oct 1998	0.009	1056015	8.240	0.397	1060	8.70	0.419	1119.3	20.045	0.655	2713.2
May 1999	0.016	1877360	6.770	0.175	910	12.71	0.329	1708.3	12.710	0.329	1708.3
Jun 1999	0.007	794850	12.650	0.665	1240	10.05	0.529	985.6	22.765	0.858	2694.0
Jul 1999	0.022	2581370	11.350	1.870	1100	29.29	4.827	2839.5	52.063	5.685	5533.5
Aug 1999	0.005	586675	12.600	0.854	1380	7.39	0.501	809.6	59.456	6.186	6343.1
Sep 1999	0.009	1021950	9.940	0.344	1340	10.15	0.352	1369.4	69.614	6.537	7712.5
Oct 1999	0.009	1056015	7.580	0.550	1060	8.00	0.581	1119.3	77.618	7.118	8831.9

Note: The values given for "MGD" are the average monthly flow rates, which were calculated by totaling the flow during the month, and then dividing by the number of days in the month.

Table A7.6. Flow, nickel and sulfate mass at Seep 1; comparison between May-October data and annual estimates.

Year	Precipitation (cm)			Flow (m3)			Nickel mass (kg)			Sulfate mass (kg)		
	May-oct	Annual	% Diff.	May-oct	Annual	% Diff.	May-oct	Annual	% Diff.	May-oct	Annual	% Diff.
1986	50.8	67.8	25.0	6,628	8,104	18.2	114.7	120.7	5.0	8,899	9,824	9.4
1987	54.0	60.3	10.4	5,942	6,397	7.1	110.5	112.6	1.9	9,478	9,919	4.4
1988	62.3	81.1	23.2	9,697	10,265	5.5	177.4	179.4	1.1	14,508	14,871	2.4
1989	56.1	69.2	19.0	20,693	21,487	3.7	343.2	346.1	0.8	26,878	27,468	2.1
1990	42.2	64.5	34.5	3,478	4,955	29.8	51.2	54.7	6.4	4,185	4,812	13.0
1991	57.8	73.0	20.8	10,613	11,749	9.7	148.2	151.6	2.2	13,928	14,723	5.4
1992	43.9	61.3	28.4	3,107	3,902	20.4	14.3	15.4	7.1	6,043	6,599	8.4
1993	44.3	63.8	30.6	6,279	7,218	13.0	50.5	53.8	6.1	7,536	8,728	13.7
1994	48.7	67.6	28.0	3,565	7,661	53.5	24.6	28.8	14.6	4,120	4,965	17.0
1995	47.0	60.6	22.5	5,821	6,631	12.2	41.1	43.7	6.0	6,525	6,903	5.5
1996	46.3	77.7	40.4	5,813	8,653	32.8	40.0	47.3	15.4	5,592	6,710	16.7
1997	39.3	56.7	30.7	3,588	5,405	33.6	25.7	32.2	20.2	3,862	5,180	25.4
1998	48.7	69.0	29.4	2,375	2,719	12.7	20.0	23.2	13.8	2,713	3,152	13.9
1999	73.2	90.0	18.7	7,918	9,512	16.8	77.6	88.7	12.5	8,832	10,334	14.5
Ave. 1987-99	51.0	68.8	25.8	6,823	8,190	19.2	88.5	92.7	8.1	8,793	9,585	10.9

Note: The pretreatment system was constructed in 1992, during which time flow measurements were not made. The data shown here for 1992 includes only data from April, August, September and October.

Table A7.7. Nickel and copper mass removal in the Seep 1 pretreatment system, using 1993-94 data from the outer pool to represent input concentrations.

Date	Liters	Output [Ni], mg/L	Input [Ni], mg/L	Output [Cu], mg/L	Input [Cu], mg/L	Nickel mass (kg) In	Nickel mass (kg) Out	Nickel mass (kg) Rem.	Nickel mass (kg) Cum.rem.	Nickel mass (kg) Annual	Copper mass (kg) In	Copper mass (kg) Out	Copper mass (kg) Rem.	Copper mass (kg) Cum. rem.	Copper mass (kg) Annual
aug 1992	704010	3.763	11.5	0.278	0.850	8.096	2.649	5.447	5.447	5.447	0.598	0.196	0.403	0.403	0.403
sept 1992	1816800	4.725	12.6	0.100	0.770	22.892	8.584	14.307	19.754	19.754	1.399	0.182	1.217	1.620	1.620
oct 1992	586675	5.233	9.8	0.071	0.550	5.749	3.070	2.679	22.434	22.434	0.323	0.042	0.281	1.901	1.901
may 1993	1173350	7.467	1.8	0.073	0.090	2.112	8.761	0.000	22.434	0.000	0.106	0.086	0.020	1.921	0.020
jun 1993	1249050	7.467	17.9	0.073	1.680	22.358	9.327	13.031	35.465	13.031	2.098	0.091	2.007	3.928	2.027
jul 1993	2229365	8.850	16.0	0.320	1.430	35.670	19.730	15.940	51.405	28.971	3.188	0.713	2.475	6.403	4.502
aug 1993	821345	8.650	11.5	0.190	0.850	9.445	7.105	2.341	53.746	31.312	0.698	0.156	0.542	6.945	5.044
sep 1993	454200	7.200	12.6	0.130	0.770	5.723	3.270	2.453	56.198	33.764	0.350	0.059	0.291	7.236	5.335
oct 1993	352005	6.467	9.8	0.090	0.550	3.450	2.276	1.173	57.372	34.938	0.194	0.032	0.162	7.397	5.496
may 1994	469340	1.350	1.8	0.105	0.090	0.845	0.634	0.211	57.583	1.385	0.042	0.049	0.000	7.397	0.000
jun 1994	1249050	8.575	17.9	0.130	1.680	22.358	10.711	11.647	69.230	13.032	2.098	0.162	1.936	9.333	1.936
jul 1994	352005	8.950	16.0	0.320	1.430	5.632	3.150	2.482	71.712	15.514	0.503	0.113	0.391	9.724	2.327
aug 1994	117335	6.750	11.5	0.115	0.850	1.349	0.792	0.557	72.269	16.071	0.100	0.013	0.086	9.810	2.413
sep 1994	908400	7.200	12.6	0.210	0.770	11.446	6.540	4.905	77.175	20.977	0.699	0.191	0.509	10.319	2.922
oct 1994	469340	5.967	9.8	0.167	0.550	4.600	2.801	1.799	78.974	22.776	0.258	0.078	0.180	10.499	3.102
may 1995	117335	0.700	1.8	0.040	0.090	0.211	0.082	0.129	79.103	0.129	0.011	0.005	0.006	10.505	0.006
jun 1995	113550	0.230	17.9	0.040	1.680	2.033	0.026	2.006	81.109	2.135	0.191	0.005	0.186	10.691	0.192
jul 1995	938680	4.550	16.0	0.070	1.430	15.019	4.271	10.748	91.857	12.883	1.342	0.066	1.277	11.968	1.469
aug 1995	821345	3.350	11.5	0.058	0.850	9.445	2.752	6.694	98.551	19.577	0.698	0.048	0.651	12.618	2.119
sep 1995	1249050	7.650	12.6	0.125	0.770	15.738	9.555	6.183	104.734	25.760	0.962	0.156	0.806	13.424	2.925
oct 1995	2581370	9.450	9.8	0.235	0.550	25.297	24.394	0.903	105.637	26.663	1.420	0.607	0.813	14.237	3.738
may 1996	1173350	2.795	1.8	0.161	0.090	2.112	3.280	0.000	105.637	0.000	0.106	0.189	0.000	14.237	0.000
jun 1996	1135500	5.920	17.9	0.193	1.680	20.325	6.722	13.603	119.240	13.603	1.908	0.219	1.688	15.925	1.688
jul 1996	1408020	9.045	16.0	0.482	1.430	22.528	12.736	9.793	129.033	23.396	2.013	0.679	1.335	17.260	3.023
aug 1996	704010	8.425	11.5	0.256	0.850	8.096	5.931	2.165	131.198	25.561	0.598	0.180	0.418	17.678	3.441
sep 1996	454200	7.335	12.6	0.117	0.770	5.723	3.332	2.391	133.589	27.952	0.350	0.053	0.297	17.975	3.738
oct 1996	938680	8.565	9.8	0.292	0.550	9.199	8.040	1.159	134.749	29.112	0.516	0.274	0.242	18.217	3.980
may 1997	821345	1.500	1.8	0.078	0.090	1.478	1.232	0.246	134.995	0.246	0.074	0.064	0.010	18.227	0.010
jun 1997	1362600	8.715	17.9	0.110	1.680	24.391	11.875	12.515	147.511	12.762	2.289	0.150	2.139	20.366	2.149
jul 1997	469340	11.700	16.0	0.239	1.430	7.509	5.491	2.018	149.529	14.780	0.671	0.112	0.559	20.925	2.708
aug 1997	352005	7.400	11.5	0.135	0.850	4.048	2.605	1.443	150.972	16.223	0.299	0.048	0.252	21.177	2.960
sep 1997	113550	7.680	12.6	0.116	0.770	1.431	0.872	0.559	151.531	16.782	0.087	0.013	0.074	21.251	3.034
oct 1997	469340	7.680	9.8	0.116	0.550	4.600	3.605	0.995	152.526	17.777	0.258	0.054	0.204	21.455	3.238

Table A7.7. Nickel and copper mass removal in the Seep 1 pretreatment system, using 1993-94 data from the outer pool to represent input concentrations (continued).

Date	Liters	Output [Ni], mg/L	Input [Ni], mg/L	Output [Cu], mg/L	Input [Cu], mg/L	Nickel mass (kg) In	Nickel mass (kg) Out	Nickel mass (kg) Rem.	Nickel mass (kg) Cum.rem.	Nickel mass (kg) Annual	Copper mass (kg) In	Copper mass (kg) Out	Copper mass (kg) Rem.	Copper mass (kg) Cum. rem.	Copper mass (kg) Annual
may 1998	586675	5.443	1.8	0.178	0.090	1.056	3.193	0.000	152.526	0.000	0.053	0.104	0.000	21.455	0.000
jun 1998	454200	13.050	17.9	0.234	1.680	8.130	5.927	2.203	154.728	2.202	0.763	0.106	0.657	22.112	0.657
jul 1998	117335	8.465	16.0	0.109	1.430	1.877	0.993	0.884	155.613	3.087	0.168	0.013	0.155	22.267	0.812
aug 1998	46934	7.510	11.5	0.079	0.850	0.540	0.352	0.187	155.800	3.274	0.040	0.004	0.036	22.303	0.848
sep 1998	113550	7.725	12.6	0.080	0.770	1.431	0.877	0.554	156.353	3.827	0.087	0.009	0.078	22.381	0.926
oct 1998	1056015	8.240	9.8	0.397	0.550	10.349	8.702	1.647	158.001	5.475	0.581	0.419	0.162	22.543	1.088
may 1999	1877360	6.770	1.8	0.175	0.090	3.379	12.710	0.000	158.001	0.000	0.169	0.329	0.000	22.543	0.000
jun 1999	794850	12.650	17.9	0.665	1.680	14.228	10.055	4.173	162.174	4.173	1.335	0.529	0.807	23.350	0.807
jul 1999	2581370	11.350	16.0	1.870	1.430	41.302	29.299	12.003	174.177	16.176	3.691	4.827	0.000	23.350	0.807
aug 1999	586675	12.600	11.5	0.854	0.850	6.747	7.392	0.000	174.177	16.176	0.499	0.501	0.000	23.350	0.807
sep 1999	1021950	9.940	12.6	0.344	0.770	12.877	10.158	2.718	176.896	18.895	0.787	0.352	0.435	23.785	1.242
oct 1999	1056015	7.580	9.8	0.550	0.550	10.349	8.005	2.344	179.240	21.239	0.581	0.581	0.000	23.785	1.242

Figure A7.8. Comparison of load from surface seeps to pretreatment system with overall load at weir, August, September, 1993.

8/17/93

Site	Flow (L/sec)	Concentrations - Mg/L			Loading Rate - Mg/sec		
		Cu	Ni	Mg	Cu	Ni	Mg
S-2	.023	1.46	17	252	.034	.39	5.8
S-3	.006	2.51	30	370	.015	.18	2.2
S-4	.017	4.68	42	338	.080	.71	5.7
S-5,6,7[a]	.002e	2.3	18	103	.005	.04	.2
S-8	.014	1.44	12	210	.02	.17	2.9
S-9	.008e[b]	1.16	11	204	.01	.09	1.6
Total	.070				.16	1.58	18.4
Seep 1	.3	0.22	10	190	.066	3	57

∑Seeps; Q ~ 23% of Seep 1

[a] No flow was reported, but samples were collected, concentrations are averaged for the sites, lowest reported flow for an individual seep was 0.002 L/sec, assume flow from each seep is less than 0.002, assume total flow from all 3 seeps = 0.002, overall contribution to total load is small.

[b]No flow was reported on 8/17 - assumed flow was same as on 8/30 and 9/16.

8/30/93

Site	Flow (L/sec)	Concentrations - Mg/L			Loading Rate - Mg/sec		
		Cu	Ni	Mg	Cu	Ni	Mg
S-2	.023	1.3	17	248	.03	.39	5.7
S-3	.004	1.9	28	338	.008	.11	1.4
S-4	.006	2.97	36	318	.018	.22	1.9
S-8	.023	1.5	14	230	.034	.32	5.3
S-9	.008	1.18	12	228	.009	.10	1.8
Total	.064				.099	1.14	16.1
Seep 1	.368e[a]	.13	8.97	204	.048	3.3	75

[a]Estimated from LTV daily flow data, by multiplying LTV flow by 0.875, which was the average ratio for 8/17 and 9/16 of DNR gaged flow/LTV reported.

Table A7.8. Comparison of load from surface of seeps to pretreatment system with overall load at weir, August, September, 1993 (continued).

9/16/93

Site	Flow (L/sec)	Concentrations - Mg/L			Loading Rate - Mg/sec		
		Cu	Ni	Mg	Cu	Ni	Mg
S-2	.009	1.04	16	238	.009	.14	2.1
S-3	No data reported						
S-4	.002	2.57	30	316	.005	.06	.6
S-8	.015	1.14	9.9	190	.017	.15	2.8
S-9	.008	.088	8.7	186	.007	.07	1.5
Total	.034				.0387	.42	7
Seep 1	.27	0.12	6.4	160	.03	1.7	43

∑ Seeps Q ~ 13% Seep 1

Table A7.9. Flow estimates, Seep 1, August 1993.

Date	Flow (L/sec)		
	DNR gaged value	LTV's reported value	DNR/LTV
8/17/93	0.30	0.35	0.86
8/30/93	0.368 (est.)	0.42	$\bar{x} = 0.875$
9/16/93	0.267	0.30	0.89

Notes: To compute the contribution of the small surface seeps to the overall flow at Seep 1, Seep 1 flow had to be estimated for 8/30/93. This was done by using an average ratio between the DNR gaged value and the LTV reported value, which was obtained from a standard weir equation.

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Section A8.1

Elevation of Overflow, Seep 1 Pretreatment System

10/28/97 surveyed site

The elevation of the overflow pipe was measured in October of 1997 and 1998. The OSS staff reading at overflow averaged 3.50 ft. (3.52 ft. in 1997, 3.49 ft. in 1998).

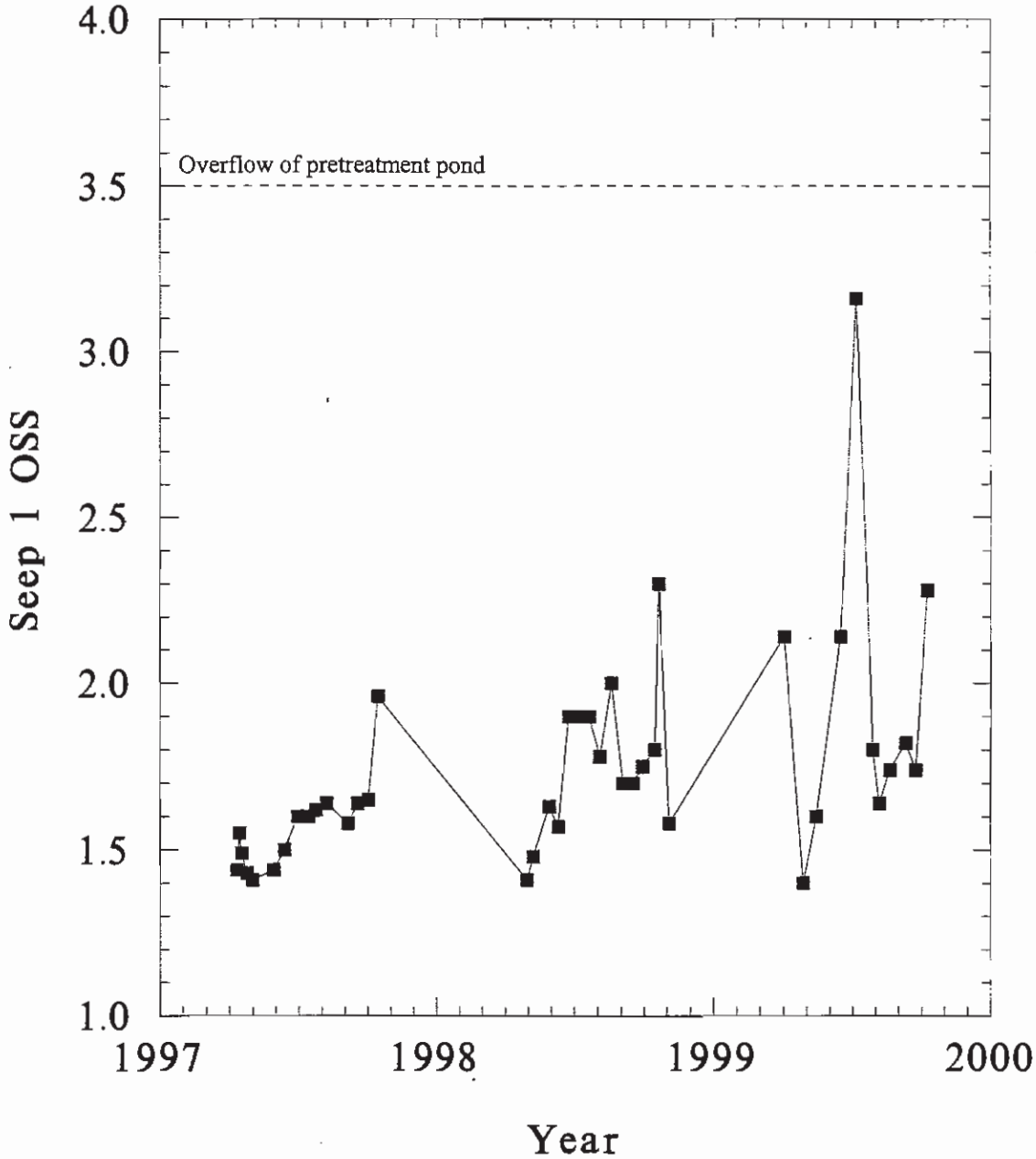


Figure A8.1. Staff gage readings (OSS) vs. time for the Seep 1 pretreatment system, 1997-99.

Section A8.2

Piezometers – Seep 1 pretreatment

Memo

Date: 9/18/00

To: Paul Eger

From: Glenn Melchert

Subject: Piezometers–Seep 1 Pre-treatment

Installation Notes

Piezometers 1 through 7 were installed on 9/16/98. Each pipe was 5 feet long and consisted of ½ inch diameter schedule 80 gray pvc. Piezometers 1- 4 were driven 1 foot into the mineral soil. Piezometers 5, 6 and 7 were driven into the soil exposed between the pretreatment pond and the waste rock stockpile. Piezometers 2d and 3d were installed on 10/26/98 and 10/23/98, respectively. Piezometer 2d was installed 1 foot deeper than #2 and about a foot away. Piezometer 3d was installed in a similar manner. Piezometers 2d and 3d were longer than 5 feet. Each piezometer was open at each end (no screen) and driven into the ground with a hammer. The drive end of each pipe was covered by a steel bolt. Once the proper depth was attained, the bolt was driven out the bottom of the piezometer with a steel rod that was placed inside the pipe and pushed or hammered down about 2 inches. See schematic and Figure 8 in the text for details.

Table 1. Construction details for each piezometer.

Piezometer	Installed	Distance from rebar in NW corner (ft)	Distance from west shore (ft)	Dip (0 degrees is horizontal)
1	9/16/98	83	6	90
2	9/16/98	48	4	90
2d	10/26/98	48	nm	90
3	9/16/98	20	4.5	90
3d	10/23/98	20	nm	90
4	9/16/98	58	20[a]	90
5	9/16/98	37	1 ft landward	8.5
6	9/16/98	36	1.5 ft landward	45
7	9/16/98	58	2.5 ft landward	90

[a] Approximately 3 feet from the limestone berm.

Piezometer data comparing water levels inside the piezometers to the pond level. Positive values for delta h means the water in the piezometer was higher than the pond level. All values are in feet.

Piezometer	9/29/98[d]		10/1/98		10/15/98		10/26/98		12/4/98[c]	
	top of pipe to pond	delta h	top of pipe to pond	delta h	top of pipe to pond	delta h	top of pipe to pond	delta h	top of pipe to pond	delta h
1	.63	+.42	.47	+.16	.44	+.15	.58	+.22	.70	+.09
2	1.08	+.29	1.08	+.07	1.05	+.10	1.16	+.13	1.26	+.05
2d							1.57	Nm	1.66	+.17
3	1.06	+.46	.95	+.12	.93	+.11	1.05	+.14	1.17	+.09
3d										
4	1.13	0	1.16	-.29	1.13	+.03	1.27	+.44[a]		
5	.63	+.63[b]	.60	+.60[b]	.58	+.58[b]	.70	+.70[b]		
6	1.48	+.40	1.45	+.24	1.43	+.19	1.55	+.40		
7	1.92	+.45	1.89	+.09	1.87	+.49	1.99	+.81		
OSS	1.75		1.78		1.80		1.68		1.54	

[a] data appears anomalous.
[b] Water always slowly dripped from this piezometer, so this value is conservative.
[c] These data were collected when the pond and the water in the piezometers was frozen.
[d] These data are suspect because they were collected using a 1/4" dowel as a dip stick inside the piezometer. The dowel likely displaced some water making it appear higher than actual. For the other dates, piezometer water level was measured with a bubble tube, so no displacement occurred.

Table Notes:

1. The distance to water (pond) for piezometers 5, 6, and 7 was measured on 10/1/98 only (these piezometers are on land). Values for other dates were calculated based on the relationship determined on 10/1/98.

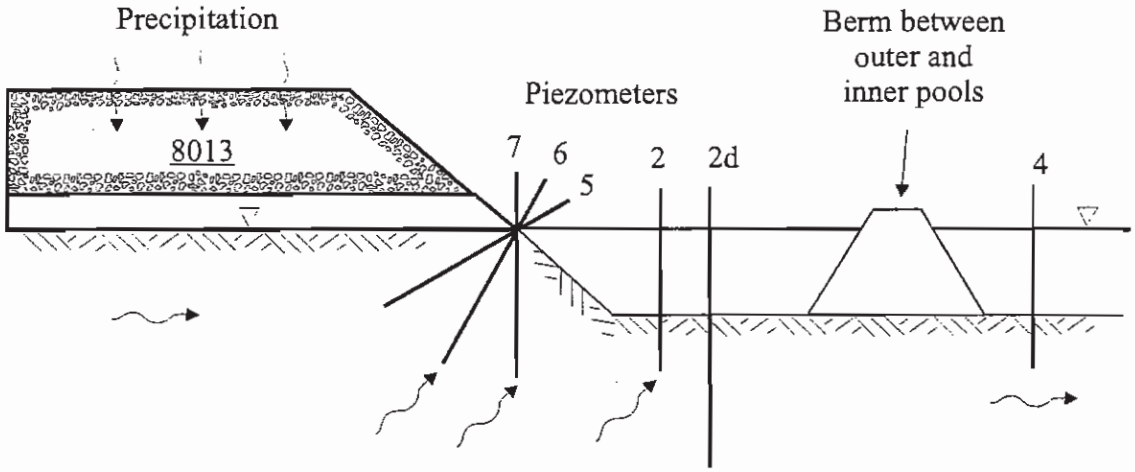
2. On 10/15/98 the piezometers were sampled after measuring water levels. Four hours later, all the piezometers except #4 and #7 had recovered to within 1.5" of original levels.

3. On 10/26/98 the outlet pipe (goes under the road) from the weir was unclogged.

Summary

Limited data from piezometer 4 suggest that most of the time, there is no vertical up welling of groundwater over most of the pond. There is a small upward gradient beneath the portion of the pond adjacent to the rock stockpile (piezometers 1, 2, and 3). The upward gradient is much larger just up slope of the pre-treatment pond (piezometers 5, 6, and 7). This upward gradient causes seeps to occur in this area. This situation also suggests that there may be substantial amounts of groundwater that enters the pre-treatment pool across the interface between the soil and the western edge of the pre-treatment pool.

Cross-section



Top View

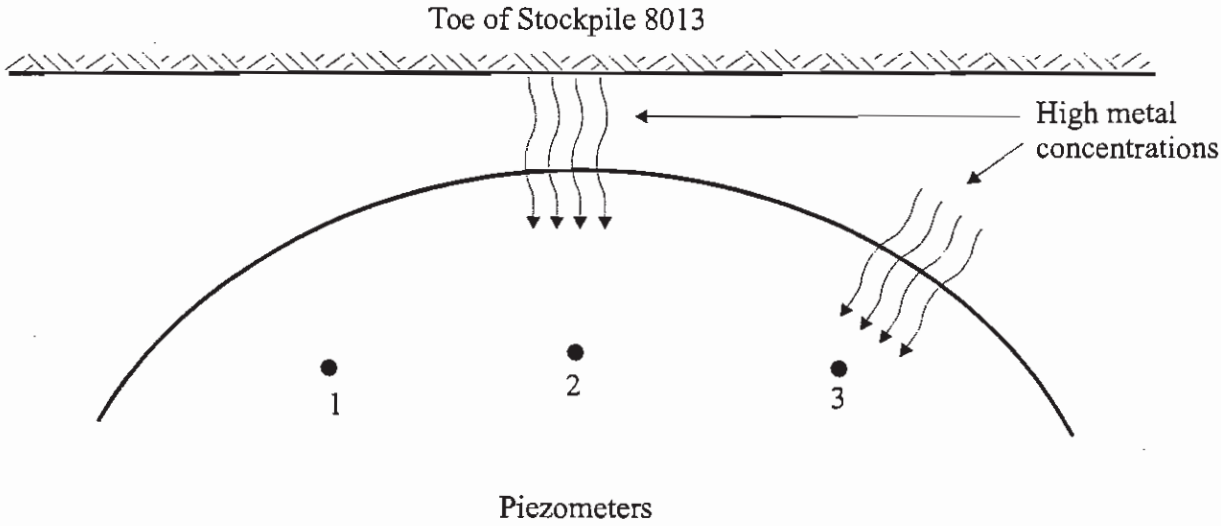


Figure A8.2. Hydrology model of stockpile 8013 and the Seep 1 pretreatment system (schematic).

Table A8.1. Summary of piezometers installation and gradients.

Piezometer	Location	Depth	Approx. Gradient Fall 1998
1	Outer pool	~ 1' below substrate, in mineral	~ 1.2-1.8" upward
2			
3			
4	Middle pool		0
5	Bank, toe of stockpile, ~ 8.5° angle into bank	mineral	~ 6-7" upward
6	Bank, toe of stockpile, ~ 45° angle into bank	mineral	~ 3" upward
7	Bank, toe of stockpile, vertical angle	mineral	~ 6" upward

Table A8.2. Piezometer water quality data, October 1998.

10/16/98

Piezometer	SC	pH	Ni	Cu	Co	Zn
1	2150	6.82	.062	.003	.021	.044
2	3350	6.96	.159	.009	.033	.108
3	1900	6.53	9.8	.06	.099	.138
4	1600	7.13	.155	.005	.027	.031
5	2200	6.56	33	.003	1.81	.043
6	2250	5.14	31	.66	1.61	1.91

Note: all trace metals in mg/L.

Section A8.3

Groundwater Flow at Seep 1, Dunka Mine

Problem: Has construction (August 1992) of the pretreatment pond upstream of the weir affected flow at the weir?

Assumptions:

1. Water table contours approximate topographic contours.
2. The permeability of bedrock, except for fracture zones, is insignificant and flow through fractures is relatively insignificant on a volumetric basis.
3. Groundwater discharge occurs in the vicinity (upstream of the historical weir) of seep 1. This is supported by observations prior to excavation of the pretreatment site that revealed there was no distinct channel between the toe of the stockpile and the weir-only pockets of water here and there. This, along with the fact that the slope in this area is around 3-4% increasing to around 5% under the stockpile toe. Water table elevations in STS well F-14 ranged 0.4 to 0.9 ft. below ground level. This well is located less than 50 ft. north of the drainage swale leading to seep 1.
4. The bedrock feature immediately upstream of the weir (at the site of the under drain) extends 100 ft. or more north and south which would effectively contain nearly all groundwater flow in this area and cause it to move through the weir site with a substantial portion being gaged.
5. The aquifer (overburden) is approximately 7 feet thick. This is average of three nearby wells (9.0, 7.5, 5.5).

Information Known:

From STS Consultants Ltd.; Hydrogeologic Study, 1988; and Supplement to Hydrogeologic Report, 1988.

Seep 1 weir	elev. notch	1451.7 (3/88)
	elev. top	1452.7
	northing	445,495
	easting	2,313,720
Well STS	F-11	2100 ft. WSW of seep 1 weir
Well STS	F-14	5 ft. N. of pretreatment pond (110-130' NW of seep 1 weir)
Well STS	F-15	1000 ft. NW of seep 1 weir

Table 1. Well information collected 11/87 through 10/88.

Well #	Ground Elevation	Average Water Level Below Ground	Well Depth (ft.)	Overburden Thickness (ft.)	Kh[1] (10^{-3}cm/s)	Kh[2] (10^{-3} cm/s)	Screened Interval
F-11	1498.3	1.6 (.4-2.8)	8.3	9	3.8	.67 (.2-1.6)	3.3-8.3
F-14	1456.8	0.6 (.4-1.0)	8.0	7.5	NM	6.1 (1.9-11)	3.0-8.0
F-15	1491.4	2.4(.7-5.5)	5.5	5.5	NM	.61 (.1-1.3)	2.5-5.5

[1] Based on slug test.
[2] Mean value based on grain size. Values in parentheses represent the range of values calculated.
NM Not measured.

Table 2. Grain size data.

Well #	Sample Interval (ft.)	d_{50} (mm)	-200 mesh
F-11	7.5-9	0.4	30%
F-14	5-7	1.0	17%
F-15	2.5-4.5	0.08	48%

Table 3. Soil descriptions.

Well #	Interval	Soil Classification
F-11	0-4.5 4.5-7.0 7-9	SM-ML SP-SM SM
F-14	0-7.5	SW-SM
F-15	0-5.5	SM-ML

Note : STS used the following for their analysis in the "Northern shallow bedrock" region which includes seep 1.

$K = 5 \times 10^{-3}$cm/s
i = .025
porosity = .30

Groundwater Gradients (based on surface topography - Dunka Pit Area, Aug. 1, 1964):

- Pretreatment pond area (in a westerly direction, prior to excavation).

$$i = \frac{1465 - 1460}{80''} = .036$$

- F-14 to weir. (Note: topo contour values ~5' greater than surveyed values -- used survey.)

$$i = \frac{1456.2 - 1451.7}{130'} = .035$$

- Pond Perimeter

$$i = \frac{1475 - 1465}{75'} = .133 \quad \text{(northerly)}$$

$$i = \frac{1470 - 1465}{100'} = .05 \quad \text{(westerly)}$$

$$i = \frac{1470 - 1465}{80'} = .065 \quad \text{(southerly)}$$

- Overall flow path (headwater to pond) $\frac{1490 - 1465}{670} = .037 = .037$

- New gradients along pond perimeter after pretreatment pond construction (subtract 4.5' from original Δh).

$$i = \frac{1475 - 1460.5}{75} = .193 \quad \text{(northerly)}$$

$$i = \frac{1470 - 1460.5}{100} = .095 \quad \text{(westerly)}$$

$$i = \frac{1470 - 1460.5}{80} = .119 \quad \text{(southerly)}$$

Groundwater flow prior to construction of pretreatment pond:

$i = .036$
$K = 6.1 \times 10^{-3}$cm/s
A = 7' thick aquifer x 200' wide valley (based on contours)
$Q = KiA$
$Q = 6.1 \times 10^{-3}$cm/s x .036 x 7' x 200' (60 s/min x .0328ft/cm x 7.481gal/ft^3) = 4.5 gpm

4.5 gpm is consistent with baseflow measured at the weir (limited data 8/93).

If groundwater flow is confined to the "notch" between bedrock knobs (site of under drain), determine gradient necessary to transmit 4.5 gpm. Area of notch = 7' deep (est.) by 10' wide.

$$i = \frac{Q}{KA}$$

$$= \frac{4.5\ gpm}{(6.1\ x\ 10^{-3})(70\ ft^2)} \div 14.723$$

$$i = 0.716$$

This kind of gradient is plausible in the area just upstream of the current weir pool.

Groundwater flow after construction of pretreatment pond:

Based on the presence of seeps and estimated flow paths, groundwater discharge probably occurs along the north, west and south sides of the pretreatment pond. The pond was excavated about 2 ft. deeper than the weir invert elevation (1451.7). The ground elevation at well F-14 is 1456.8' or about 5 feet higher than the weir. Although not surveyed, it appears this elevation carries around most of the perimeter. Since the water table in this area was about 0.5' or less below ground surface (well F-14), the differential head has increased about 4.5' as compared to pre-excavation conditions.

<u>Pretreatment Pond Dimensions</u> (paced)

North side	100'
West (stockpile) side	110'
South side	60'

Calculation of lateral seepage into pretreatment pond. $Q = K \frac{(h_1 - h_2)}{L} A$

West side $Q = 6.1 \times 10^{-3} cm/s \left(\frac{1470 - 1460.5}{100}\right)(2)(110)(14.723)$
$= 1.9$ gpm

North side $Q = 6.1 \times 10^{-3} \left(\frac{1475 - 1460.5}{75}\right)(2)(100)(14.723)$
$= 3.5$ gpm

South side $Q = 6.1 \times 10^{-3} \left(\frac{1470 - 1460.5}{80}\right)(2)(60)(14.723)$
$= 1.3$ gpm

Total = 6.7 gpm

Discussion:

This calculated flow (6.7 gpm) is 50% higher than the 4.5 gpm calculated earlier for pre-existing aquifer flow at seep 1, but given the uncertainty of some of the estimates above and that these two values are in the general range of late summer baseflow (4-5 gpm), they may be about equal. Lateral seepage into the pond may be over estimated if the gradients from the north and south directions (based on topographic contours) are not really that steep. This is likely because water table gradients typically are smaller than surface topography in areas of relatively steep surface slopes. The gradient in the westward direction is believed fairly accurate since the primary recharge area is in that direction which would tend to keep the water table near the surface. There may be errors in the cross-sectional area of the pre-existing flow calculation.

Limited data for the years 1988-1991 (STS 1988) indicate that baseflow ranges around 1-5 gpm. This fairly large fluctuation may be explained by the small recharge area and ground watershed affecting this seep. During periods of drought or low precipitation, groundwater recharge ceases after which baseflow is maintained by the slow dewatering of the aquifer.

If the estimate of lateral seepage into the pond is about equal to or even greater than baseflow, then groundwater discharge due to up swelling from the bottom of the pond is minimal or nonexistent. If hydraulic conductivity or the groundwater gradients are over estimated by as little as a factor of 2, then upwelling could be a significant contributor to flow. The presence of a vertical gradient (positive or negative) could be verified by placing nested piezometers within the pond.

The groundwater flow calculations are believed optimistic because:

1. Hydraulic conductivity was estimated based on grain size analysis (average of 3 different calculations) of well F-14. In well F-7, not listed here, the grain size K and the slug test K match well, but do not match very well in well F-11. The data from F-11 was confounded because the screened interval over which the slug test was done included a sandier zone that was not considered in the grain size analysis. Well F-14 had the highest average K of 3 nearby wells, indicating K could be lower in some areas of the seep 1 watershed.

2. Hydraulic gradients used are probably maximums.

   <u>Flow through vertical peat bed</u>

   Assume free flow through limestone.

   K of peat = 1.7×10^{-1} cm/s (STS)

   $$Q = K \Delta \frac{\Delta h}{L} A$$

- Calculate head differential to transmit 5 gpm, 20 and 65 gpm

$$\Delta h\ (5\ \text{gpm}) = \frac{QL}{KA} = \frac{5(1)}{(.17)(24x20')} \times \frac{1}{14.723} = .004\ \text{ft}$$

$\Delta h(20\text{gpm}) = .02$ ft.

$\Delta h(65\ \text{gpm}) = .05$ ft.

Section A8.4

Seep 1 Flow Summary

Seep 1, located at the extreme west side of waste rock stockpile 8013, was first detected in 19976-1977. Flow at this seep typically starts in April and ends in early November, with no flow occurring during the winter.

Data

A 1981 DNR report (Eger et al. 1981) states that periodic measurements were made at Seep 1 in 1976 using a mini current meter, but that since the water depth was often too shallow to use a meter (depth less than 1 inch), it was at those times "necessary to estimate flow using floats as an indicator of flow velocity." Staff gage readings were also taken occasionally, but were deemed unreliable because of erratic results. The values that are presented in the 1986 DNR report were thus calculated by assuming that Seep 1 runoff coefficients were the same as for EM8 and by using a watershed area of 44,390 m^2, which was determined by cut and weigh.

A V-notch weir and a Stevens Type F flow recorder were installed in 1986, so that more reliable flow estimates are available starting with 1986. To calculate flow from 1986, on LTV estimates the daily gage readings from the continuously recording charts, and then used a standard weir equation to convert these values into flow values (reported in CFS). LTV has not actually measured flow to compare with the estimates made from the standard equation. A 1993 DNR analysis of Seep 1 flow indicated that the equation used by LTV underestimates flow by approximately 10%. However, this degree of potential error is still much smaller than for the 1976-1995 data. To arrive at the average flow values that are presented in LTV's NPDES discharge monitoring reports, the daily values are summed up for each month, and then this summed monthly total is divided by the number of days in the month to arrive at an average monthly value.

The Stevens recorder is generally installed in late April or early May, and then removed in late October or early November. After the recorder is removed in the fall, occasional staff gage readings are taken manually on the days when water quality samples are collected, until the seep completely freezes up. Using these point estimates for fall flow is reasonable since flow tends to decline gradually over this period. It is less accurate in the spring when the melting snow pack in the Seep 1 watershed could conceivably pass the system unmeasured.

Ditching

In the original version of the Dunka Closure Report it says that the first ditching activity that influenced the flow rate occurred in 1989. Accurate flow in 1989 was 20 gpm, but fell to 6 gpm in 1990, a decrease that LTV says was due to this ditching activity. Additional ditching in the winter of 1990-1991 "assisted in maintaining flow at 6 gpm during 1991 and 8 gpm during 1992." LTV also states (on page 63) that Seep 1 has a base flow that results from groundwater

arising from wetland located south and upgradient of Seep 1, which migrates "through an erosional channel in the original topographic feature." It is also stated that this base flow "may influence water quality," so that, even if all flow through the 8013 was terminated, an appreciable mass load of copper and nickel (and other pollutants) would still arise from this seep.

Though the effects of the ditching on flow volumes at Seep 1 can not be discounted entirely, it appears that the high flow values observed during 1989, indicating that the 1989 flow was overestimated for some reason.

- The total annual flows calculated for three or four years prior to 1989 are very similar to the three or four years following 1989 flow was overestimated for some reason.

- Runoff coefficients calculated for each year since 1986 are consistently about 0.3 (i.e., 30% of the precipitation that falls on the Seep 1 watershed passes through the weir), yet 1989 values are abnormally high, with some of the monthly values actually being over 1.0 (i.e., there was more flow during those months than would have occurred had all of the precipitation falling on the watershed passed through the weir).

- Precipitation during 1989 was relatively normal, even a bit low, so that, unless the effective Seep 1 watershed area was somehow altered, no large surge of precipitation accounts for the abnormally high flow observed for this year.

Treatment Areas

A limestone pretreatment area was constructed during August 4-11, 1992. From this point on, the site referred to in NPDES reports as 043 refers to water samples collected after passing through the pretreatment area. Efforts to quantify flow from the seep itself (i.e., prior to entering the pretreatment area) have been unsuccessful due to the proximity of the pretreatment area to the stockpile toe, and to the relatively diffuse nature of this particular seep.

A wetland treatment area was constructed in 1995 (construction was completed in June). The outfall from this treatment system is now called 043-1.

Winter Flow

In the 1981 DNR report, it is stated that "spring runoff (of the seeps in general) was less than 20% of the total annual flow in 1979 and 1980, the only years for which complete annual records exist," with the value for 1980 being only 2 percent. It is also stated that "there was no flow during the winter."

References

Eger, P. Lapakko K., and Weir, A. 1981. The environmental leaching of stockpiles containing copper-nickel sulfide materials. Minnesota Department of Natural Resources, St. Paul, MN. 62 p.

Section A8.5

Residence Time Estimate, Seep 1 Pretreatment System

Assumptions: total area = 930 m^2
average water depth = 0.3 m

total volume = 279,000 Liters

peak daily flow on July 5, 1999 = 1000 L/min (265 gal/m)

residence ~ $\frac{\text{279,000 Liters}}{\text{1000 Liter /m}}$ ~ 5 hours

2 days later flow dropped to 132 L/min (35 gal/min)

Using the same assumptions: residence time = 35 hours

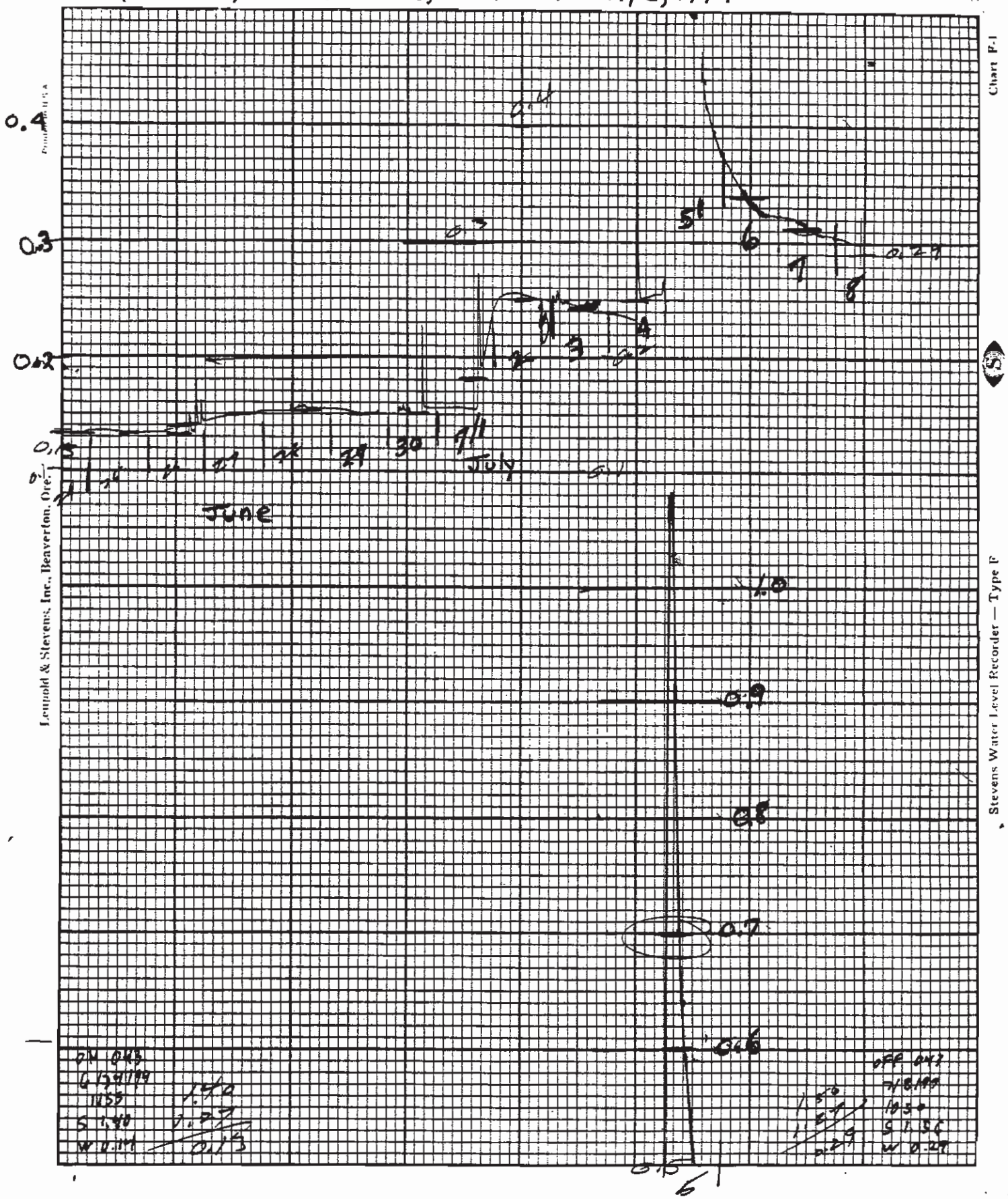
These numbers probably underestimate the total residence time during this period since the pond was flooded. The water depth was about one foot deeper, so the total volume in the pond increased and, therefore, the residence time would also increase.

The high water level prevented adequate contact between the input and the limestone.

Section A8.6
July 1999 Staff Gage Readings and Flow Calculations

In 1999, over the 4th of July rainfall, about 7 inches of rain fell. This section contains a copy of the water level vs time chart and the daily flows calculated by LTV. LTV calculates the flow from the water level with a standard weir equation.

Seep 1 Weir, OSS vs Time, June 24 - July 8, 1999



LTV Mining Company- Dunka

Wetland Flow Monitoring

Month of July, 1999

Location: 043 60 degree V-notch Weir

Date:	Head Feet	CFS	GPM	MGD
07/01/1999	0.18	0.0198	8.90	0.0128
07/02/1999	0.25	0.0451	20.24	0.0291
07/03/1999	0.24	0.0407	18.27	0.0263
07/04/1999	0.25	0.0451	20.24	0.0291
07/05/1999	0.70	0.5916	265.50	0.3823
07/06/1999	0.34	0.0973	43.65	0.0629
07/07/1999	0.31	0.0772	34.65	0.0499
07/08/1999	0.29	0.0654	29.33	0.0422
07/09/1999	0.29	0.0654	29.33	0.0422
07/10/1999	0.28	0.0599	26.87	0.0387
07/11/1999	0.26	0.0497	22.32	0.0321
07/12/1999	0.24	0.0407	18.27	0.0263
07/13/1999	0.23	0.0366	16.43	0.0237
07/14/1999	0.22	0.0328	14.70	0.0212
07/15/1999	0.21	0.0292	13.09	0.0188
07/16/1999	0.23	0.0366	16.43	0.0237
07/17/1999	0.23	0.0366	16.43	0.0237
07/18/1999	0.22	0.0328	14.70	0.0212
07/19/1999	0.20	0.0258	11.58	0.0167
07/20/1999	0.19	0.0227	10.19	0.0147
07/21/1999	0.18	0.0198	8.90	0.0128
07/22/1999	0.17	0.0172	7.72	0.0111
07/23/1999	0.17	0.0172	7.72	0.0111
07/24/1999	0.17	0.0172	7.72	0.0111
07/25/1999	0.16	0.0148	6.63	0.0095
07/26/1999	0.16	0.0148	6.63	0.0095
07/27/1999	0.15	0.0126	5.64	0.0081
07/28/1999	0.15	0.0126	5.64	0.0081
07/29/1999	0.14	0.0106	4.75	0.0068
07/30/1999	0.14	0.0106	4.75	0.0068
07/31/1999	0.13	0.0088	3.95	0.0057
Total				1.04
Average			23.26	0.0335

Appendix 9

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Section A9.1

Site Visit by Paul Eger, W1D Wetland and Seep 1 pretreatment system, August 2, 1994

W1D Wetland

I visited the W1D treatment system with Gene Halberg, from LTV, and we first went to the section of the treatment system where they have installed hay bales. They have placed hay bales on two of the berms below the large berm that was built in 1993 to reduce the gradient in the first half of the cell. Three inches of hay were placed on the center of the berm, and then the bales were placed on top of that hay. The first layer of hay was placed to avoid erosion of the material. The bales were placed about one foot apart. There are three rows of hay over one foot apart, and the spacing between each bale is about 6-8 inches. The bales were just placed; they were not staked. They are starting to collapse now, as they absorb water. 'The space between seems a little bit less as the bales start to collapse and fill in the gap. On the first berm, the berm closest to the input, there are three rows of hay bales. Now this was a berm where the water was channelized last year on the far east end, and fairly discrete channel. The hay bales have seemed to spread the water over about one-third of the berm and have dammed water up behind the hay bales. There is a whitish precipitate being produced as the water flows through the hay bales, and there is a strong odor of H_2S in this area. Gene said that after the large rain storms in June, the flow was high. The water actually came over the bales. Gene believes that the bales tended to reduce the erosion in the area, and they have not had to repair the berms this year, since the hay was put in.

In the second, or the next berm where hay bales are placed, there are five hay bales placed the same way as in the first area. They start in on the berm with the five rows. That berm was a little wider with less vegetation. They used all the hay bales. They put in about 300 hay bales in the total system, so there are two berms of hay bales. The pool below the hay bales is quite turbid. What we need to do is take some pH in the areas of precipitation and take a sample of the precipitate and analyze it to see if there are any metals associated with that, or if it is strictly an organic precipitate.

I talked to Gene about their sampling program. They are still running a profile on this site, but they have changed their sampling program of the year. In the first year of operation, which would have been in 1992, a profile sample is taken where the flow is going over the berms. In 1993 it was a mixture of sampling of a flow over the berms and also in the pools. Gene said they would have taken some samples across the width of the pool at three different locations and not found much difference. So, this year they are strictly sampling the pools of water. They are trying to alternate sides of the pool they are sampling on. So, if they sample the east side of berms 3 and 4, then they will sample the left sides of the pool between berms 4 and 5 stagger that as they go through the system to do the profile.

Gene also mentioned that they looked at total versus filtered nickel and didn't see much difference in the samples. The hay bales have been in about six weeks, and there seems to have been an improved performance of the W1D system. In the last four or five cells in 1993 the nickel concentration did not drop from the large glacial till berm to the outlet weir. It was thought that this was due to the channelization of flow that occurred in that portion of the system. The hydraulic gradient is fairly steep in this area, and as a result of high flows, channels developed. The flow went down the east side of the system in 1993 with minimal amount of contact. I suggested that this year they raise the berm prior to the weir, which would lower the hydraulic gradient and install hay bales. LTV did not raise the berm but they have installed hay bales. I have not talked to Dennis Koschak. My guess is that he may not have wanted to raise the berm because vegetation is finally getting established. The northern portion of the system has extremely dense vegetation. This is the area above, north of the large berm with the three pipes that control the water flow from that berm. The cattails are starting to develop seed heads this year, and the vegetation seems to be helping in dispersing flow. Since the hay has been put in, the concentration drops by about 50% as it moves through the hay bales. But there is a very, very large drop in the very last cell, right prior to the weir. [This could be due to the sample location (although nickel does drop substantially at weir).]

We took a look at the pool above the limestone weir, and the water is very turbid, a muddy, brown color. As we stood there watching, we saw bubbles coming out from the bottom of the pond, apparently bringing sediment up with the bubble. This last cell seems to be a little larger than some of the other cells. We do not know what its depth is and about 6-7 bales of hay was added to this last cell. Hay bales were taken to the area; about two inches was peeled off and just flung into the pond. This hay drifted around for a while but then sunk. There seems to be a lot of biological activity in the cell, and it's possible that this is accounting for the large drop in concentration. Concentrations have gone from around .5 mg/L coming in to about 0.8 mg/L as it exists the system.

Now the flow today was very low. We estimated the flow was 20 gallons/min. The water prior to reaching the outlet weir flows through a stand of vegetation. At this flow rate the vegetation seems to be very successful in taking out the turbidity, and the water coming out was quite clear.

We talked a little about the pumping that has been going on at the site. The pump that comes form the input weir, the W1D site, can only pump 28 gallons/min. The bucket engaged to find the pumping rate, and then they record the one and off time, since that pump is turned on and off manually. To find the total, we multiplied 28 times the number of minutes that the system operated.

Gene provided sine data on the most recent sampling through the wetland. There is a very large drop in the concentration in the northern part of the system in the upper parts of the wetland, the nickel concentration drops from about 4.5 to 1.6 mg/L. It is not quite clear why this occurs, although the vegetation is very dense in that area, and at these low flows, it seems like it's a very good distribution or could be a very good distribution of flow in this area.

Two photos were taken of the system. One was taken facing south showing where the hay bales were placed at the outlet weir, standing at the big glacial till berm, while one was taken facing north of the stockpile showing the dense vegetation, and the other was taken south. Additional work needs to be done to investigate the zones of nickel removal, particularly what is occurring in the last pond. It could be that sine there is organic material place din the bottom of the pond, and there is organic material and nutrients being washed out of the hay above that pond. We have created a situation where we get a lot of biological activity, and this is what is responsible for the nickel removal.

Seep 1

Gene gave me some data from the most recent profile that was done in Seep 1. The Seep 1 inflows are the same points that LTV measured last year. They have not incorporated our points into their monitoring program. The limestone in some of the berms appears to have subsided or settled. Last year the limestone was all above water. This year there are several areas where there is limestone below the water. There potentially seems to be less contact between the water and limestone, if the water is actually short circuiting coming through those subsided areas. Above 5-6 bales of hay were added in a manner similar to the last pool at W1D, so the hay was just peeled off and thrown into the last section of the seep treatment area. What it looked like was that all that hay has just floated to the front of the piled cell and have piled up along the berm. In order to get some additional sulfate reduction, we need to have the water flowing through the hay. I suggested to Gene we take the first part of the cell and stack hay bales, almost making it like a walking surface of hay bales in that first portion. That would hopefully give us more contact in the hay and give us more material, increase the reactions that are occurring in the system. The pH at the weir was 6.4, so it has dropped substantially and the nickel is about 8 or 9 mg/L. It seems like we are having less treatment efficiency than in previous years. There is a fair amount of precipitate in the Seep 1 area on the south side, kind of a milky precipitate, similar to what has been observed at that site in previously. Some of the limestone is being coated with this precipitate. In one area the limestone upstream has this whitish precipitate and then downstream has just a brown organic precipitate, possibly some kind of attached algae growing on the limestone. I picked up some rocks and it did not seem like this material was totally coating the limestone surface. So, there should be some availability of the surface.

Section A9.2
Notes From 10/5/95 Survey of the W1D Wetland Treatment System

On 10/5/95, Glenn Melchert and Jon Wagner surveyed the upper portion of the W1D system (i.e. that portion of the system between the culvert and the glacial till berm). The weather was cool and overcast, with some drizzle and air temperatures of about 45° F. The 050 staff gage read 4.40', with about 6" of flow in the weir.

The riprap berm just downstream of the culvert is designated in this report as Berm 1. The next three berms, which are all upstream of the glacial till berm, are designated as Berms 2-4, with Berm 4 being the berm just upstream of the glacial till berm.

Rebar stakes were placed at the (stockpile) end of each of the four berms, and rebar stakes were also placed at the far ends of berms 2 and 3 (we only had six rebar stakes with us). These stakes were placed to provide a reference point from which to measure across the berm. Hopefully these stakes will remain visible when snow covers the wetland, so that peat samples can be collected from appropriate spots. Additional (lathe) stakes were placed at a few sites where it seems that we may want to collect peat samples; the location of these stakes are noted in Table 1.

After placing the rebar stakes, we stretched a 300-ft tape measure across the width of the berms, and noted the total width of each of the four berms. We then noted where flow seemed to be occurring with relation to each of the four berms, and where channelization was present. There had been significant rainfall prior to our visit so that flow was quite high, and it seems likely that the flow paths that we noted are where flow occurs at low flows as well.

We also investigated the water and peat depths at various distances (usually 5, 10 and 15 feet) both upstream and downstream of each berm; the results of these surveys are presented in Table 2. In general, the peat was quite thin on top of the berms (i.e., 1 to 4 inches), with peat depth increasing to depths of 2 or 3 feet on the sides of the berms. However, due to subsidence of the berms it was not always easy to determine where the tops of the berms were located.

Because the power auger that will be used to collect the peat cores in the winter can be easily damaged if it encounters non-peat material, care should be taken to ensure that peat depth at each sampling location is known prior to sample collection, so that we avoid damaging the auger. This will obviously be most important when sampling near the riprap berm.

It should be noted that the wetland seems to extend closer to the 8031 than is shown in the map prepared by LTV; it seemed to get to within about 20 feet of the stockpile toe, with standing water present in this area of the wetland. The following is a short description of what we observed at each of the four berms:

Berm 1 (the riprap berm)

The total width of the berm was 54.0 meters. The top of the riprap was almost entirely under 3 or 4 inches of water, with the exception of a small area in the center of the berm, where there was a rise that extended out of the water. Obviously no peat samples can be collected from the top of this berm.

Upstream of the berm there was a strip of peat about 10 feet wide; any peat samples collected upstream of the berm will be from this strip (i.e. far enough upstream so that rocks won't be encountered, but not so far up that the sample will come from an open water area). See Table 2 for details of where to collect peat samples.

On the upstream side of the berm, the major flow was between 19.0 and 21.0 meters, with the 21.0 to 26.0 interval pretty dry (due to a hummock). From 26.0 to 54.0 meters, most of the upstream area was largely flooded.

On the downstream side, flow was noted between 17.0 and 18.5 meters. We put a stake 10' downstream of the edge of the riprap (at 18.5 meters); peat at this spot was 18" deep. (The riprap is about 18 ft. wide at 18.5 meters.) The downstream side was pretty much dry between 20.5 and 26.0 meters. At 33.2 meters there was a noticeable channel about 10' downstream, with a large delta behind it. There were another two comparable channels at 42.0 and 45.7 meters (with a hummock separating these two channels).

Berm 2

The total width of the berm was 52.3 meters. We stretched the tape measure across the berm along the wooden plank that now serves as a walkway, but which was originally placed to even out the flow across the berm to lessen the likelihood of washouts of the berm. Upstream of this plank (and thus the tape measure), standing water was present across most of the width of the berm. But the downstream side was quite dry from 0.0 to about 20.0 meters, so that one could walk on this peat and not get your boots wet (and if this area was dry during a period of high flow, it certainly remains dry during lower-flow conditions). The plank was acting like a dam, with little visible flow being transmitted over/through the plank except near the far side (i.e. away from the stockpile).

Near the far side there were several obvious flow channels downstream, where it appeared that most of the standing water upstream of the plank was being routed. There was disperse flow between 37.2 and 48.2 meters, with a peak at 48.2 (and about 0.5 meters wide). There was also flow between 40.0 and 41.9 meters, with a peak in the 40.0 to 41.0 interval.

We placed lathe stakes at several sites in these channels that seem good candidates for peat samples; Table 1 identifies the location of these stakes.

Berm 3

Total length of the berm was 53.6 meters. On the line between the stakes, the peat is wet almost from the stockpile edge of the wetland. Flow starts at 3.9 m and continues to 6.2 m, with most between 4.3 and 4.7 m. (This is broad overland flow over the wooden planks.) In fact, the wooden planks again seem to be largely working as planned in this berm, with water damming up on the upstream side of the plank, and with flow fairly evenly distributed across the width of the plank. (Water was observed broadly spread both upstream and downstream of the planks in this area).

The other area where flow is noticeably going over the plank are between 42.3 and 48.3 meters (flow in this area was noticeably stronger than between 4.3 - 4.7 meters). Flow is coming over the plank pretty evenly over this stretch, but downstream the flow path channelizes and meanders; at 10' downstream, the area of most flow is between 44.5 and 46.8 m.

Berm 4

The total length of the berm was 46.6 meters. There was a bit of flow noticeable at 5.5 to 8.0 meters, with most of this at 6.0 meters. There is also minor overplank flow between 12.0 and 12.8 meters, with a white precipitate present on the peat at 12.1 meters. At both of these flows, flow paths are pretty much perpendicular to the berm.

The berm is pretty much dry between 12.8 and 22.3 meters. Flow is again noticeable from 22.3 meters to the (far) edge of the wetland, with three major areas of flow at 27.6, 40.5 and 45.0 meters; the strongest flow was at 27.6 (with rapid flow and a distinct channel), with the other two flows weaker and about 30 cm wide each.

Watershed determination:

An attempt was made to determine which of the surrounding area contributed to flow in the wetland. On the far side (away from the stockpiles), the surrounding ground rises pretty steeply to a broad crest, limiting the amount of contributing watershed area. However, water falling on the side of the ridge facing the wetland obviously enters the system; my estimation is that the width of this area, from wetland edge to halfway across the crest of the ridge, is about 20 feet.

On the side of the wetland facing the stockpile, a ridge exists on the wetland-side of the road which largely prevents flow from the road (and flow coming off the stockpile) from entering most of the wetland. This ridge extends down the length of the wetland until the last cell (just upstream of the original output weir), where it ends. It appears that, in this last cell, flow from the surrounding area is able to enter the system. It was our thought that it may be possible to extend the ridge by 20 feet or so, so that the final cell is isolated from this surface flow. (It would likely require only a few dumptruck loads to accomplish this.)

As we have noticed before, outflow tends to be more than inflow during precipitation events. Partly this is due to rain falling directly on the wetland, but it is also likely due to water entering the last cell from the road/stockpile areas. By preventing this source of (uncounted) input water, and the dilution of the input water that results from it, it would be possible to get a better handle on how the wetland system is actually performing.

At the top end of the wetland, it was apparent that some of the rain that falls on the road area enters the system; Figure 1 identifies the approximate contributing watershed area.

Table 1. Location of lathe stakes that designate potential peat collection sites.

Stake #	Berm	Location along the width of the berm (meters)	Feet upstream of the berm	Feet downstream of the berm
1	2	39.8	~ 0 (in flow channel)	--
2	2	48.2	"	--
3	2	48.0	--	15
4	4	11.6	7	–

Table 2. Peat/water depths near the first four berms of the W1D treatment system.

Berm No.	Distance from the stockpile side of the wetland (meters)	Distance (ft.) upstream [1]	Distance (ft.) downstream [1]	Water depth (in.)	Peat depth (in.)	Comments
1	20.5	10 15	-- --	4 12	2 3	In major flow channel
1	34.0	10 15	-- --	5 5	17 19	In flooded area
1	44.7	-- -- --	10 15 20	1 1 NA	7 24 NA	At this location, riprap extends 7' downstream. A stump was hit 1 ft. deep at 20' downstream
2	16.0	5 -- --	-- 5 10	3 dry dry	30 18 18	Water too deep to go more than 5' upstream
2	33.0	5 -- -- -- --	-- 5 10 15 20	14 dry dry dry 0.5	8 12 15 30 18	Water too deep to go more than 5' upstream
2	39.8	5 -- --	-- 5 10	14 1 1	6 8 20	In flow channel. Water too deep to go 10' up

Berm No.	Distance from the stockpile side of the wetland (meters)	Distance (ft.) upstream [1]	Distance (ft.) downstream [1]	Water depth (in.)	Peat depth (in.)	Comments
2	48.0	5	--	12	5	
		--	5	1	8	In flow channel. Water too deep to go 10' up
		--	10	4	24	
		--	15	2	24	
3	4.5	0	0	2	4	In flow path
		8	--	NA	24	
		--	5	NA	5	
		--	10	NA	24	
3	23.3	5	--	NA	1	In flow path
		10	--	8	14	
		--	5	NA	12	
		--	10	6	24	
3	44.5	5	--	4	8	This is the major flow path of this berm. The peat at 10' up felt gravelly. At 10' down, water is too deep to sample.
		10	--	12	12	
		--	5	6	18	
4	6.0	5	--	1	26	In flow path.
		--	5	1	2	
		--	10	1	2	
		--	15	2	8	
		--	20	2	8	

Table 2. Peat/water depths near the first four berms of the W1D treatment system (continued).

Berm #	Distance from the stockpile side of the wetland (meters)	Distance (ft.) upstream	Distance (ft.) downstream	Water depth (in.)	Peat depth (in.)	Comments
4	12.0	5	--	1	48	
		10	--	dry	44	
		--	5	3	33	
4	25.0	5	--	2	42	Lots of flow seen here; meanders downstream.
		10	--	4	39	
		--	5	2	1	
		--	10	2	2	
		--	15	2	10	
		--	20	4	36	
4	34.0	5	--	4	24	This location was chosen arbitrarily. Some rocks were felt in the upstream peat.
		10	--	4	24	
		--	5	2	8	
		--	10	4	2	
		--	15	8	4	
		--	20	4	16	

Notes: Berm 1 is the riprap berm just downstream from the culvert, and Berm 4 is the Berm just upstream of the glacial till berm. All upstream/downstream distances are measured from a line drawn between the stakes that mark the ends of the berms (usually rebar). The center of the berm is estimated by the crest of the berm. For berms with 4 x 4's, the center corresponds to the location of the 4 x 4.

Table 3. Locations of wells.

Berm	Meters from stockpile side	Feet upstream	Feet downstream
1	24.5	10	--
2	14.9	2	--
2	40.7	2	--
3	23.3	2	--
3	44.5	2	--
4	35.6	--	7

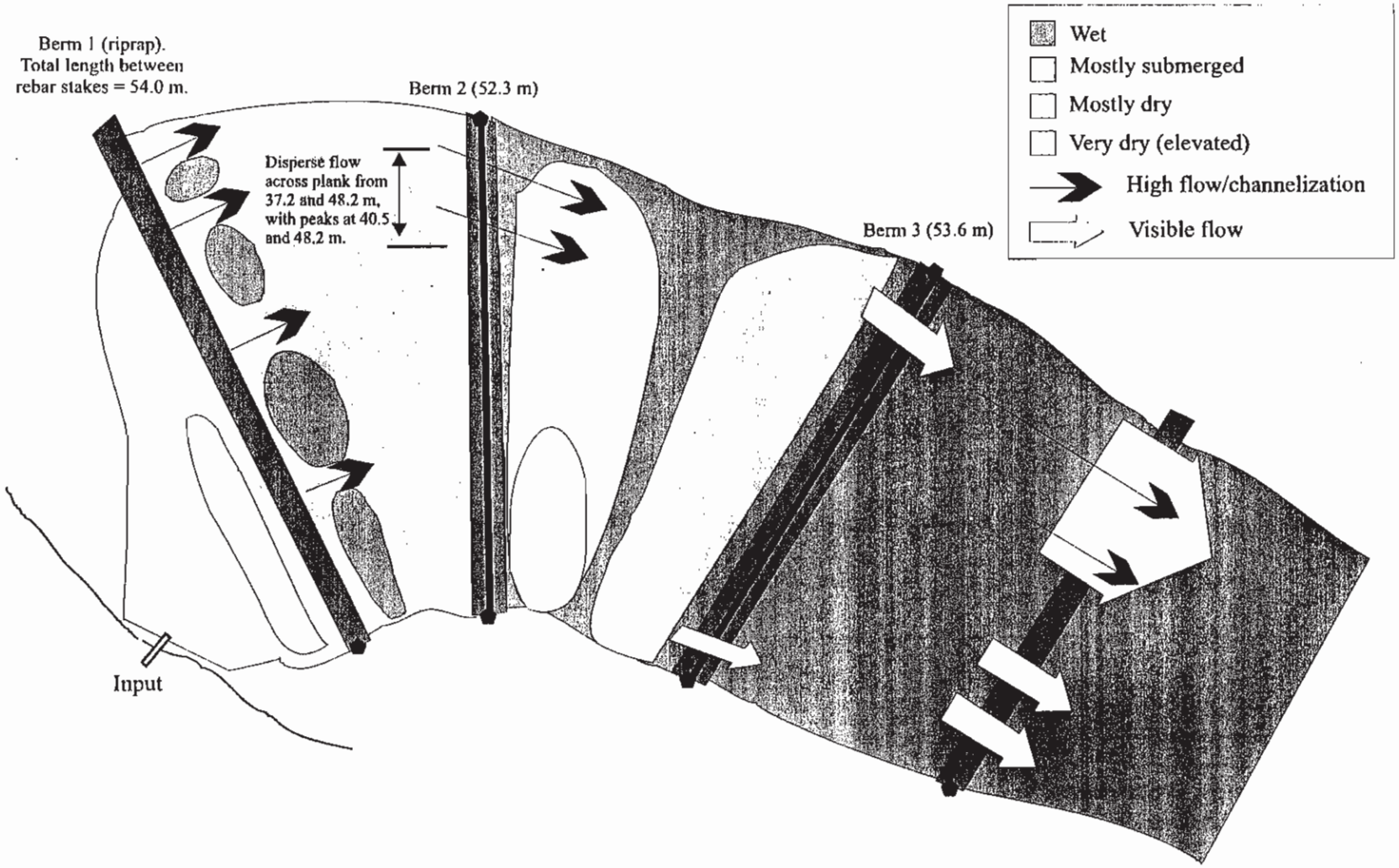


Figure A9.1. W1D wetland treatment system; flow patterns observed on 10/5/95.

Section A9.3
Comments on Seep 1 - Peat/Limestone Treatment System

1. Construction began on Tuesday, August 4, 1992. Construction was completed on Tuesday, August 11, 1992.

2. Construction details:

   a) Limestone berm crests will be one ft. above weir.
   b) Peat height will not be even with the weir (one ft. below limestone crest).
      Note: peat was actually placed at two levels. Peat near the limestone was placed approximately even with the weir, but midway between the limestone berms, the peat was 6-12 inches lower. These peat trenches were typically 3-6 ft. wide.
   c) The area was excavated to approximately 2 ft. below the weir, which amounted to approximately 3-4 ft. excavated near the stockpile.
   d) The outermost limestone "ring" functions as a riprap zone to stabilized the mineral soils. There is no ponding upgradient of this ring. This limestone riprap does not extend above original grade and much of it is about 1 ft. below grade.
      Note: probably all of the seeps here emerge from the boundary between the organic/humic surface layer (up to one ft. thick) and the underlying mineral soils.
   e) The outer ring was completed first, then the inner rings, and finally the limestone underdrain/berm. The backhoe was perched on an unexcavated peninsula near the center of the site to dig and place the outer limestone. The backhoe gradually moved back as the inner rings were excavated and completed. This was preferable over excavating the entire area first and then trying to place the peat and limestone from an operational standpoint, according to Don Markwardt.
   f) The limestone apron and drain is 1 ft. thick. The top of the apron is about 1 ft. below the weir.

3. The only bedrock encountered was the bedrock ridge just upstream of the weir.

4. There was quite a bit of silty to sandy till mixed with the limestone in many places, especially in the two outermost limestone rings. This was because LTV was low on limestone and in order to get as much limestone as possible soil was incorporated in the loads. Limestone at the weir pool and the apron and underdrain was from a new shipment and it appeared relatively free of soil.

5. The limestone was on the order of 4-6 inches in diameter and subrounded. Eighty to ninety percent of the surface of each limestone rock was coated with a limestone powder layer. The coating looked like powdered sugar and was quite thin.

6. The final berm (with the underdrain) is covered with riprap (not limestone). This riprap was placed after the limestone and peat on either side, so there is not a concern for short-circuiting here. The core of the final berm is material that was excavated from the site. It was a clay silt.

7. Peat overlying the limestone apron is 12-16 inches thick.

8. All the peat used consisted of two parts screenings to one part Area 5.

9. The invert of the bypass pipe is two feet above the peat.

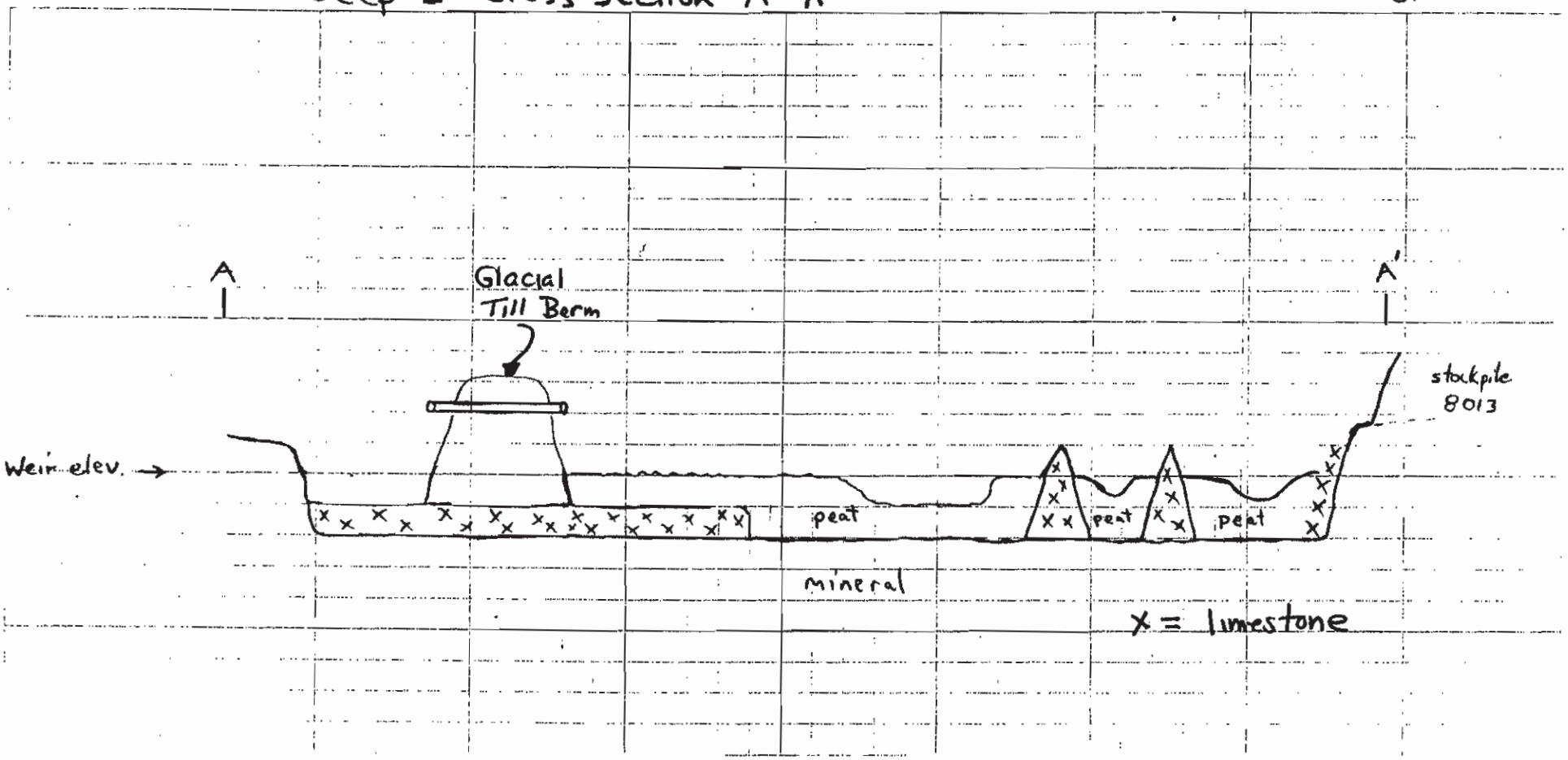
10. The working length (normal to flow direction) of the final berm underdrain is 11 ft. maximum. This was controlled by bedrock. The working width (parallel to flow direction) was 17-18 ft. The limestone apron was roughly 20 ft. x 24 ft. The size of the pool between the final berm and the weir had maximum dimensions of 18 ft. by 27 ft. The bottom of this pool was filled with limestone.

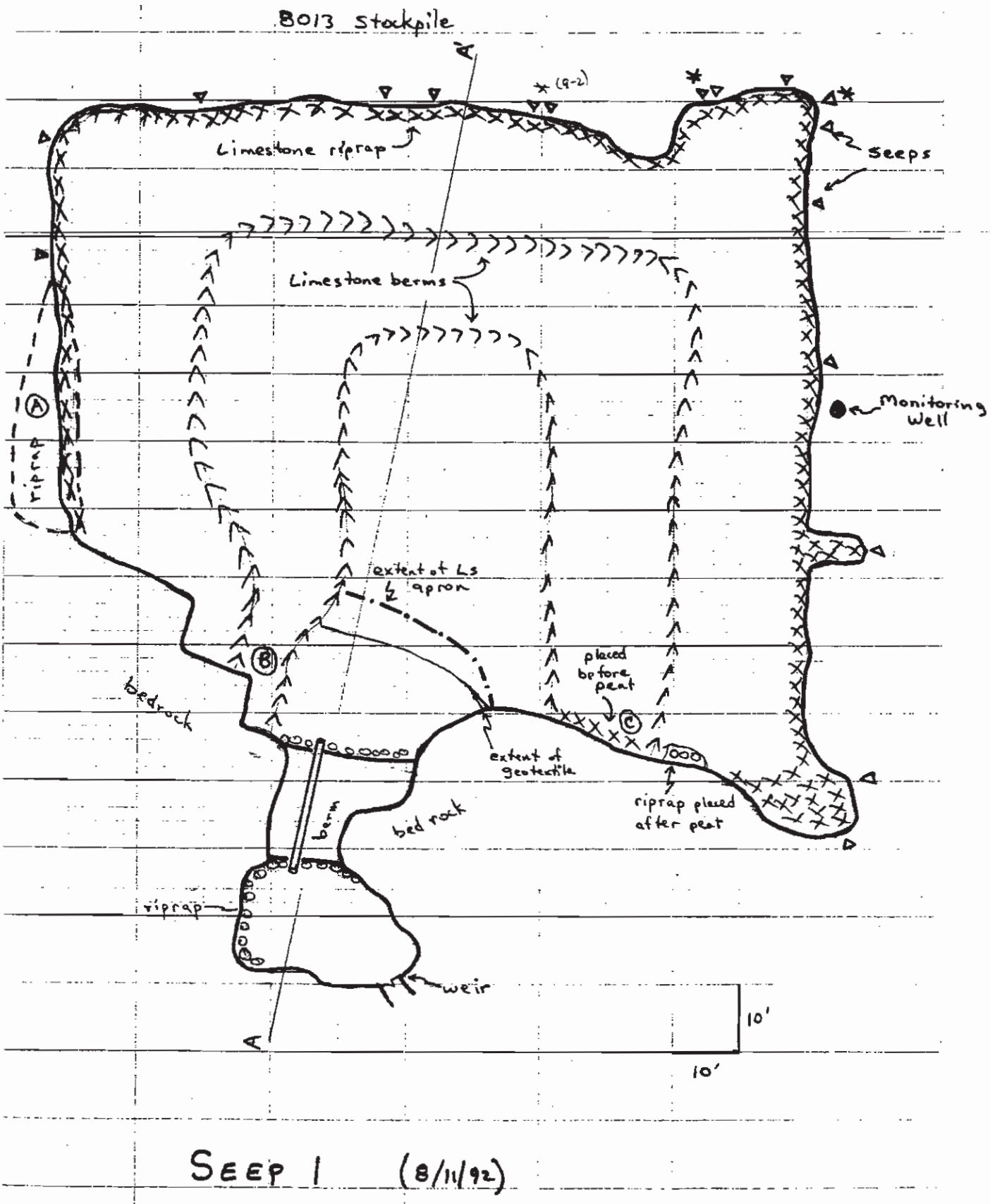
11. The limestone berms are quite small. They are teepee shaped in cross section.

12. See attached figure:

    a) The perimeter of the excavation upstream of the final berm was scaled out, as well as the steep locations. Critical dimensions of the final berm and the limestone apron were also scaled out.
    b) The riprap at the far left (A) was placed after the limestone and peat. The riprap was placed to stabilize the area which was along side the 24 inch culvert from the capping project. There probably are seeps under this riprap.
    c) Seeps observed at 1300 on August 11, 1992 are noted with triangles. Significant rainfall occurred at about 0700 on August 11. The most significant seeps (.5 to 1.0 gpm est) are designated with triangles and *. All the other seeps were primarily wet areas. Except for area A, nearly all of the seeps were found and located.
    d) Site B is an area where the two limestone berms could be interconnected. Limestone does extend between the two berms, probably because they are so close together, but I could not determine how thick the limestone connection might be. This site was covered with peat.
    e) Site C apparently was placed as riprap to stabilize the bank. This material was placed before the peat and may provide for short-circuiting.
    f) Riprap (not limestone) on and around the final berm was placed after the peat was placed.

Seep 1 Cross Section A–A′ 8/92



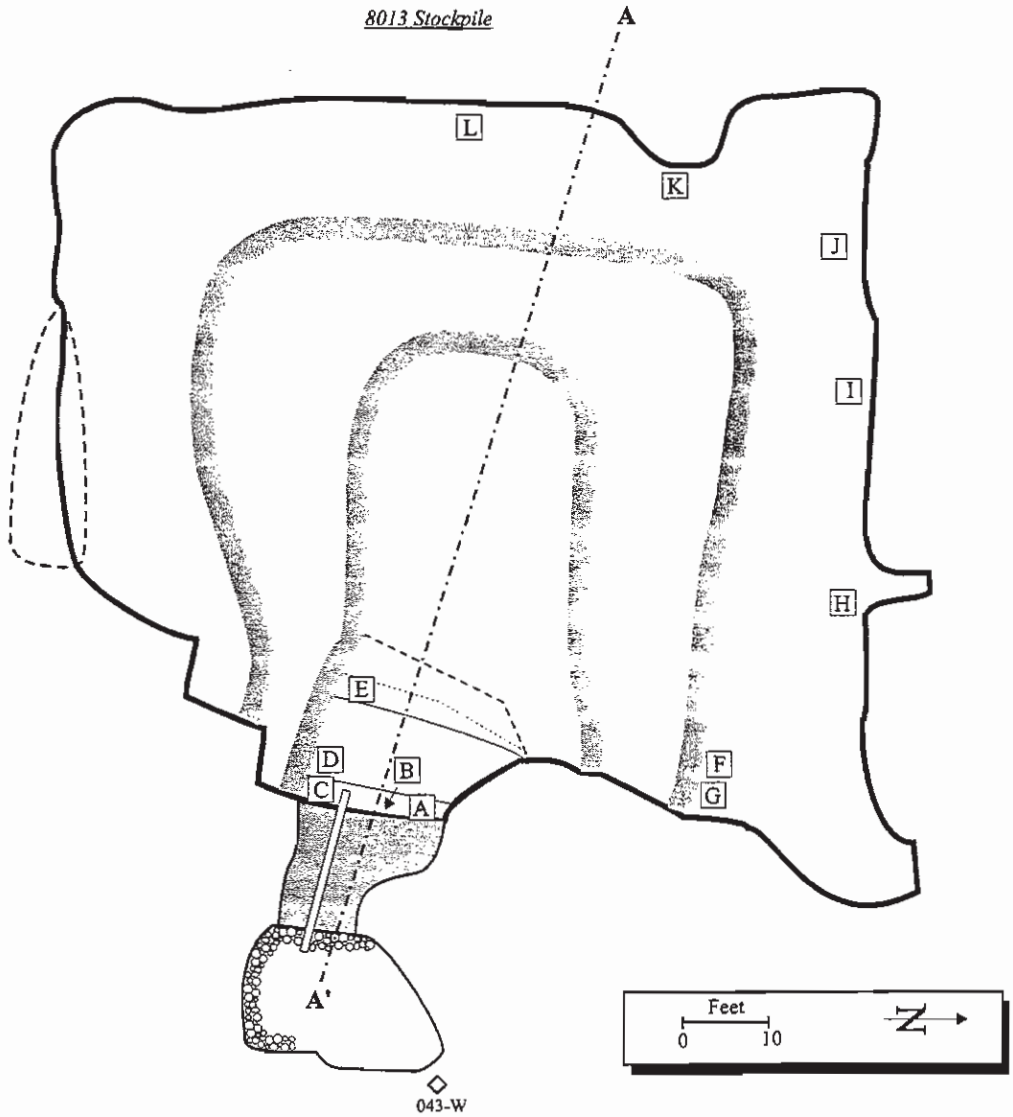

8013 Stockpile
(9-2)
Limestone riprap
seeps
Limestone berms
Monitoring Well
riprap
A
B
C
extent of Ls apron
placed before peat
bedrock
extent of geotextile
riprap placed after peat
berm
bed rock
riprap
weir
10'
10'

SEEP 1 (8/11/92)

Section A9.4

Survey of Seep 1 Pretreatment System, July, September 1997

Additional samples were collected from the Seep 1 pretreatment system to determine if concentrations were uniform with depth. Since the seeps are cold (<40° F) and specific conductance is high, it was possible that the seeps would enter the pool and flow to the bottom due to their higher density. Samples were collected at several locations throughout the system. Surface samples were collected about one inch below the surface and bottom samples were collected about one inch above the bottom. The largest difference between surface and bottom concentrations occurred in the outer pool near the toe of the stockpile (Figure A9.1). Difference in water quality decreased as water moved through the system (Tables A9.1 and A9.2).



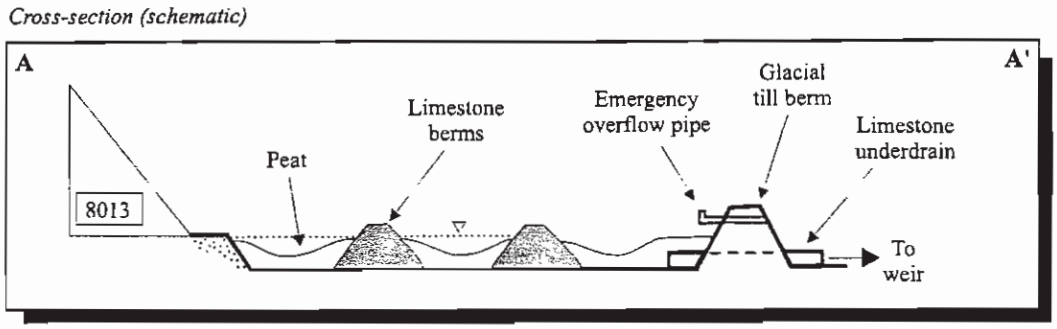
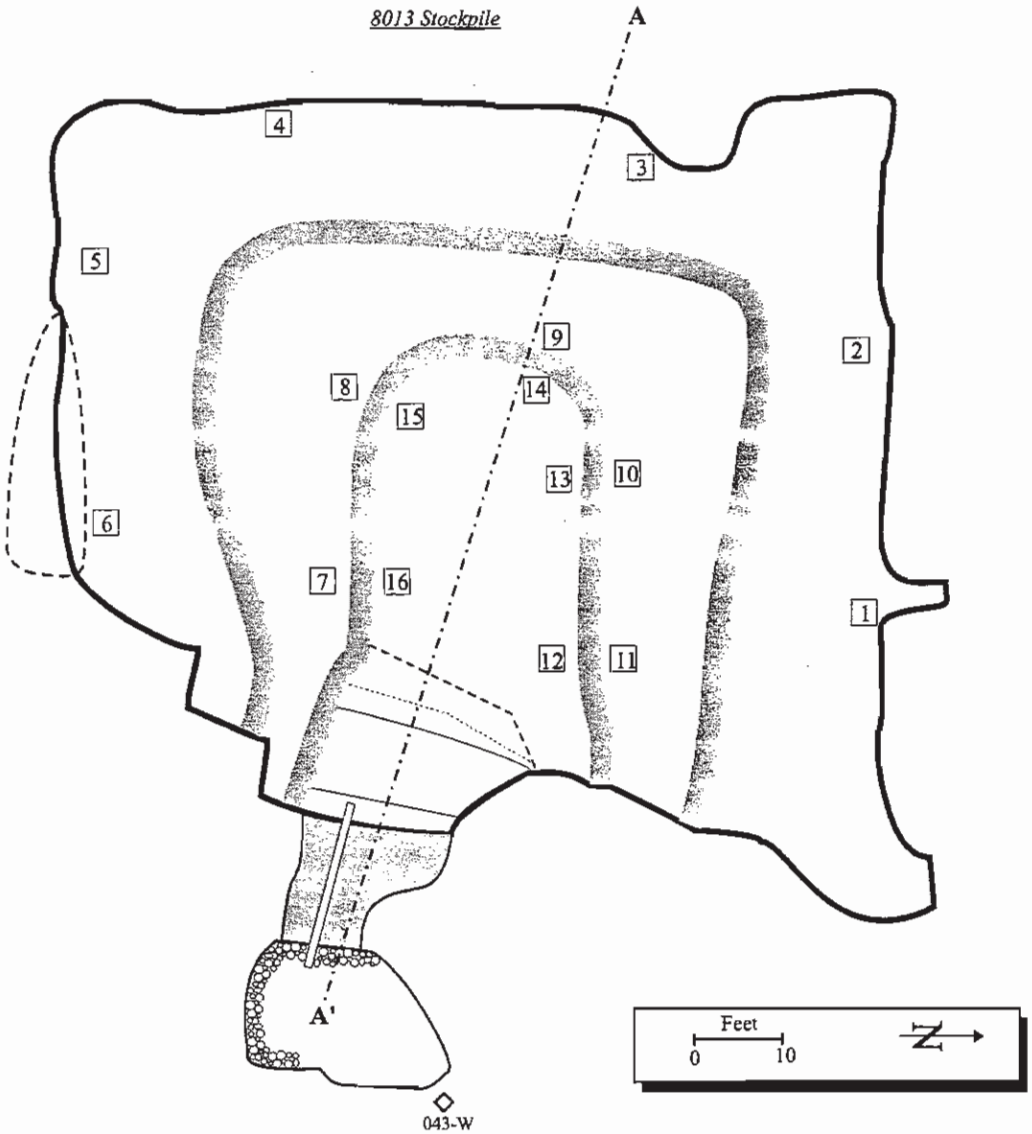


Figure A9.2. Seep 1 pretreatment system, locations of sample sites, 7-24-97.

Table A9.1. Water quality data, Seep 1 pretreatment system, 7-24-97.

Site	Location	Depth	Co (ppm)	Cu (ppm)	Ni (ppm)	Zn (ppm)
L	~ 3 feet from edge of pond	Surface	1.60	3.40	24.00	2.12
K	~ 3 feet from edge of pond	Surface	0.70	1.10	13.90	1.64
K	~ 3 feet from edge of pond	Bottom	1.70	3.60	24.50	2.56
J	~ 3 feet from edge of pond	Surface	0.40	0.70	10.90	1.27
I	~ 3 feet from edge of pond	Surface	0.60	1.10	13.60	1.63
I	~ 3 feet from edge of pond	Deep	1.20	2.40	14.90	2.53
H	~ 3 feet from edge of pond	Surface	1.10	2.10	19.10	2.39
G	At first limestone berm, accumulation of white precipitate	Bottom	1.20	2.20	19.60	2.45
F	At first limestone berm, accumulation of white precipitate	Surface	1.00	1.80	17.20	2.23
E	At upstream edge of hay bates	Surface	0.40	0.50	14.50	1.54
D	~ 2 feet toward pool from site C, visible flow	Surface	0.40	0.30	13.10	1.20
C	Upstream of outlet berm, south side	Surface	0.40	0.50	14.60	1.44
B	Upstream of outlet berm, middle	Surface				
A	Upstream of outlet berm, north side	Surface	0.40	0.40	14.50	1.40

For site locations, see Figures A9.2 and A9.3.



Cross-section (schematic)

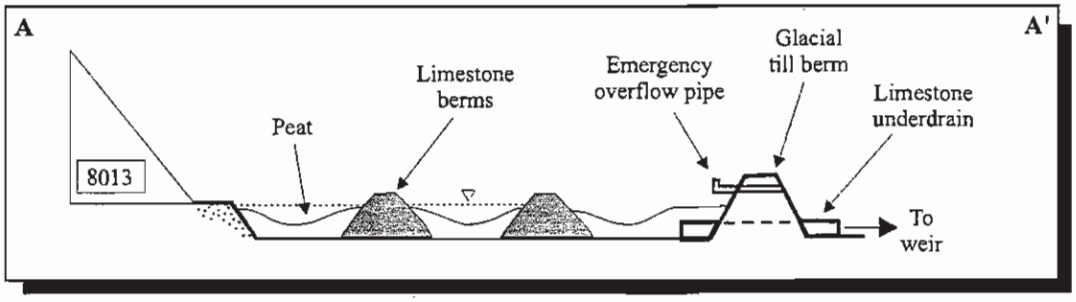


Figure A9.3. Seep 1 pretreatment system, locations of sample sites, 9-4-97.

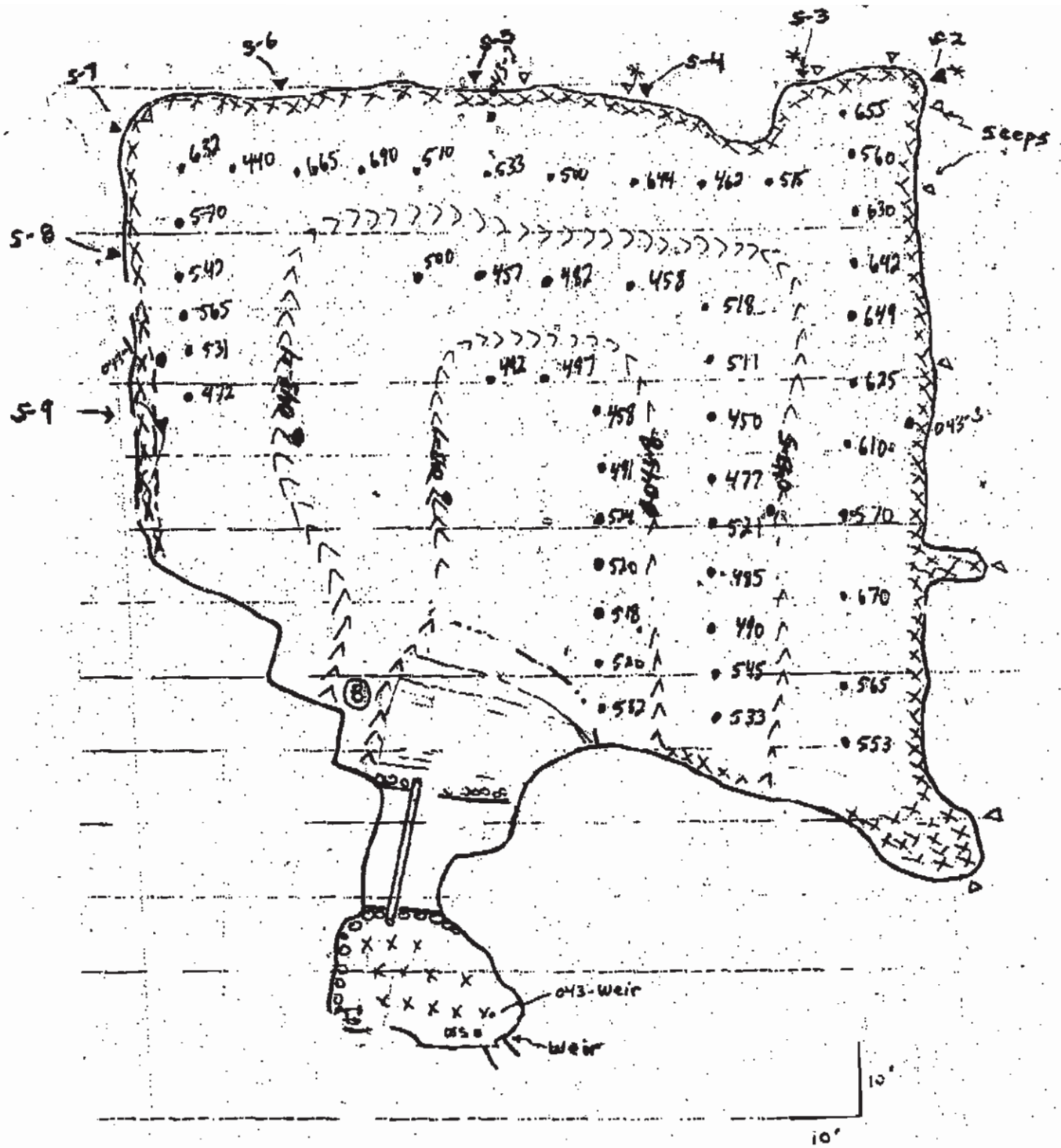
Table A9.2. Water quality data, Seep 1 pretreatment system, 9/4/97.

Site	Location	Depth	Specific Conductance (microsiemens)	pH
1	outer pool	surface	2150	6.04
1a	outer pool	bottom	2450	5.63
2	outer pool	surface	2290	6.01
2a	outer pool	bottom	2490	5.87
3	outer pool	surface	2010	6.33
3a	outer pool	bottom	2575	6.26
4	outer pool	surface	2100	6.88
4a	outer pool	bottom	2400	6.18
5	outer pool	surface	2250	6.67
5a	outer pool	bottom	2410	6.65
6	outer pool	surface	2250	6.50
6a	outer pool	bottom	2500	6.26
7	middle pool	surface	2250	6.74
7a	middle pool	bottom	2150	7.2
8	middle pool	surface	2150	7.19
8a	middle pool	bottom	2150	7.30
9	middle pool	surface	2100	7.38
9a	middle pool	bottom	2100	7.39
10	middle pool	surface	2250	7.29
10a	middle pool	bottom	2150	7.30
11	middle pool	surface	2250	7.32
11a	middle pool	bottom	2200	7.38

Section A9.5

Specific Conductance Survey, October 1997

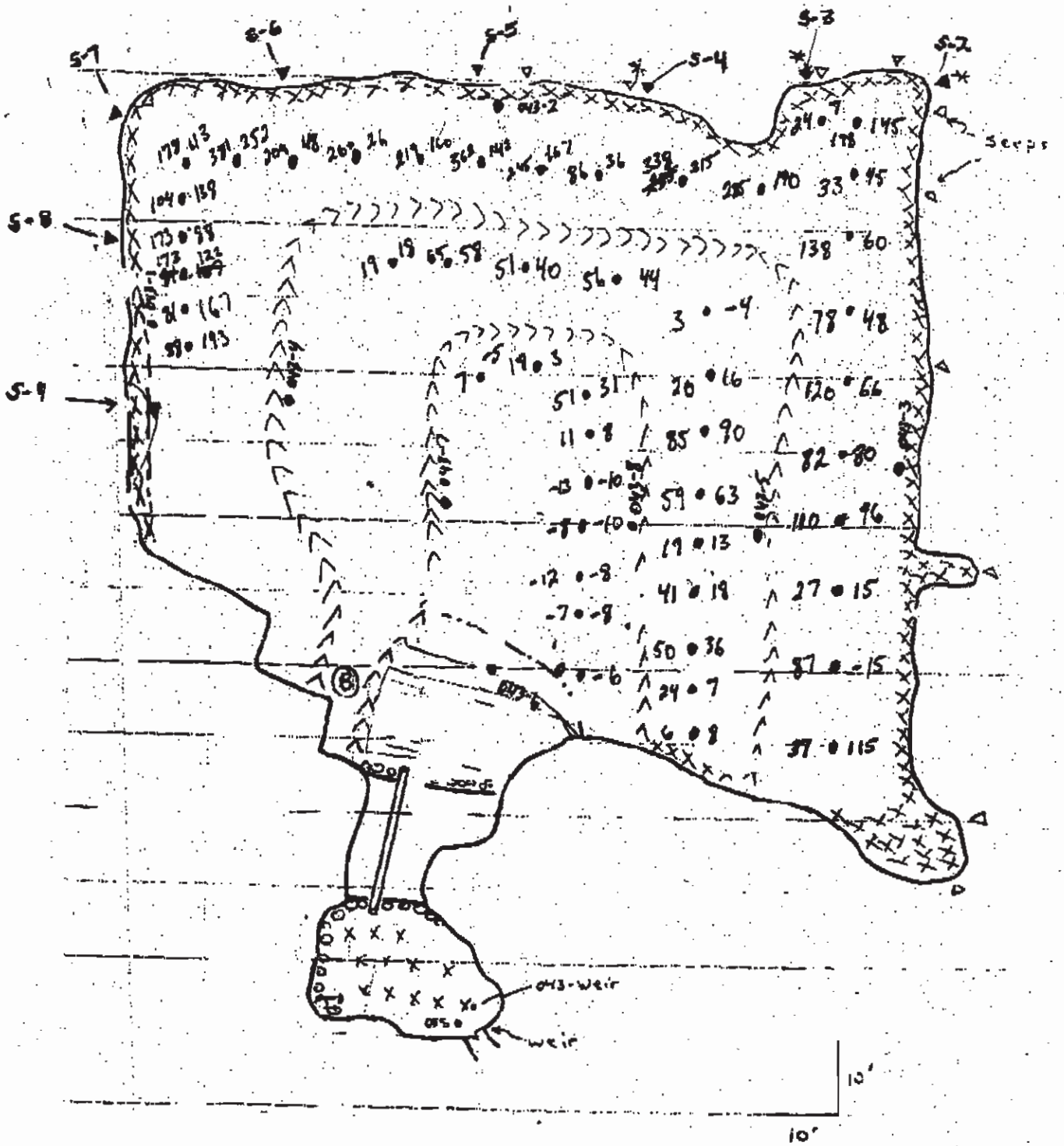
On October 1, 1997, a detailed specific conductance survey was conducted. The relative surface values are presented in Figure A9.4. To get the actual field readings add 1000 to the values shown in the figure. The relative readings for the middle and bottom are shown in Figure A9.5. To get the field reading the surface value at that location must be added to the values in the figure. These readings were most likely made with a YSI conductance meter, and have not been corrected for temperature. They provide relative comparisons, but are not true values.



Notes: (1) Add 1000 to all values; (2) Sample points are approximately 3 steps apart ~ 7 ft.; and (3) slight trickle from S-2 ~ 10 ml/min; all other seeps were dry or slightly damp.

* = substantial seeps	∇ = seep	▼ = monitored seep	x = limestone riprap	○ = local riprap

Figure A9.4. Specific conductance, surface samples, Seep 1 pretreatment system, 10/1/97



Notes: (1) Middle level is on right of sample point, bottom is on left of sample point; (2) surface value was subtracted from middle & bottom values; (3) measurement was taken 1" from bottom.

* = substantial seeps	∇ = seep	∇ = monitored seep	x = limestone riprap	○ = local riprap

Figure A9.5. Specific conductance, middle and bottom samples, 10/1/97.

Appendix 10

Analysis of Precipitate Collected From Seep 1 Pretreatment System

MIDLAND RESEARCH CENTER

A WHOLLY OWNED SUBSIDIARY OF MIDLAND STANDARD INCORPORATED
POST OFFICE BOS 67
NASHWAUK, MN 55769-0067
PHONE (218) 885-1951
FAX (218) 885-1955

April 28, 1998

Mr. Paul Eger
MnDNR-Division of Minerals
500 Lafayette Road
St. Paul, MN 55155-4045

RE: Report on DNR Sample 11066
(Seep; 1 PPT; 10-28-97)

X-ray Diffraction (XRD)

A small portion of the solids were removed from the sample container, filtered, and allowed to dry. A portion of this was placed on a glass slide and an X-ray Diffraction (XRD) pattern was run to identify mineralogy. No heating or other drying of the sample was done before the XRD pattern.

The pattern had few peaks, making identification somewhat uncertain. The pattern in general was broadly domed indicating that much of the phases present were poorly crystalline or amorphous and consequently not identifiable with XRD. A second run was done after drying at 100°C for several days. The peaks were sharpened a bit with several new peaks.

The probable phases are two (or more?) hydrated iron sulfates. These have the following formulas.

Iron sulfate hydrate (XRD 21-930) $Fe_{14}S_{18}O_{75}.nH_2O$ (predominant)

Iron sulfate hydroxide hydrate (XRD 16-935) $Fe(OH)SO_4.5H_2O$

The second phase is also known as fibroferrite.

The second XRD pattern indicates a small amount of probable 10 angstrom sheet silicates/clays. The iron sulfates above are not major components. Clays and aluminosilicates are the probable primary phases. The material has a very low density.

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ST. PAUL

Chemistry Analyses

Chemex Labs, Inc. performed whole rock and trace element analyses (see results). After a long process of filtering and drying about 3/4 of the entire sample, only seven grams of material was produced. After consultation with chemex, they believed that enough material was available to do most, if not all, requested analyses.

The sample contained 2.36% sulfur, and about .05% inorganic C (presumably as CO_2). The sample contained predominantly Al_2O_3 and H_2O. There was 20.95% SiO_2 compared with 36.68% Al_2O_3.

Of the major cations, it contained the most iron (1.85% Fe_2O_3). The sample contained 9.15% moisture (loosely held), with 19.85% water of crystallization.

With regard to other elements of interest, the sample contained anomalous Cu (5400 ppm), Ni (1280 ppm), Zn (440 ppm), U (28.5 ppm), and the rare earths.

Discussion

The chemistry and XRD agree pretty well. There is an excess of alumina. The most aluminous clays, if they contain all the silica, would still leave some alumina left, probably as hydrous aluminum oxide gels. Since the 10 angstrom clay peak is so weak, probably most everything is in the form of hydrous, poorly crystalline gels. The iron sulfates appear to be the most "crystalline" substances. Such gels are probably good candidates for adsorbing all kinds of anomalous metals and cations. Beside the base-metal amounts, I was surprised at the amount of U and rare earth elements. With regard to rock types, these elements tend to be naturally concentrated in felsic rather than mafic or ultramafic rocks.

Aluminum oxides are rather strange in that they tend to be amphoteric. In acid conditions, they behave as a base. In high PH conditions, they behave as acids (they tend to dissolve in strong acids or bases; precipitate in-between).

There probably isn't much material left for them to analyze. If you would like anything else analyzed for, let me know. I still have a quarter of the material left in the refrigerator. That could also be used. Call if you have questions.

Sincerely,

Barry Frey

Barry Frey
Technical Consultant
cc: Dave Antonson DNR-Minerals Hibbing

Chemex Labs, Inc.

Analytical Chemists * Geochemists * Registered Assayers

994 Glendale Ave., Unit 3, Sparks
Nevada, U.S.A. 89431
PHONE: 702-356-5395 FAX: 702-355-0179

To: MIDLAND RESEARCH CENTER

P.O. BOX 67
NASHWAUK, MINNESOTA
55769

A9816614

Comments: ATTN: BARRY FREY

CERTIFICATE A9816614

(AAW) - MIDLAND RESEARCH CENTER

Project: 101.177
P.O. # :

Samples submitted to our lab in Sparks, NV.
This report was printed on 21-APR-98.

SAMPLE PREPARATION

CHEMEX CODE	NUMBER SAMPLES	DESCRIPTION
217	1	Geochem ring entire sample
200	1	Whole rock fusion

ANALYTICAL PROCEDURES

CHEMEX CODE	NUMBER SAMPLES	DESCRIPTION	METHOD	DETECTION LIMIT	UPPER LIMIT
594	1	Al2O3 %: Whole rock	ICP-AES	0.01	100.00
588	1	CaO %: Whole rock	ICP-AES	0.01	100.00
590	1	Cr2O3 %: Whole Rock	ICP-AES	0.01	100.00
586	1	Fe2O3(total) %: Whole rock	ICP-AES	0.01	100.00
821	1	K2O %: Whole rock	ICP-AES	0.01	100.00
593	1	MgO %: Whole rock	ICP-AES	0.01	100.00
596	1	MnO %: Whole rock	ICP-AES	0.01	100.00
599	1	Na2O %: Whole rock	ICP-AES	0.01	100.00
597	1	P2O5 %: Whole rock	ICP-AES	0.01	100.00
592	1	SiO2 %: Whole rock	ICP-AES	0.01	100.00
595	1	TiO2 %: Whole rock	ICP-AES	0.01	100.00
475	1	L.O.I. %: @ 1000 deg.C	FURNACE	0.01	100.00
540	1	Total %	CALCULATION	0.01	105.00
2840	1	Ba ppm: ICP-MS	ICP-MS	1	10000
2841	1	Cs ppm: ICP-MS	ICP-MS	1	10000
2842	1	Hf ppm: ICP-MS	ICP-MS	1	10000
2843	1	La ppm: ICP-MS	ICP-MS	1	10000
2844	1	Nb ppm: ICP-MS	ICP-MS	1	10000
2845	1	Rb ppm: ICP-MS	ICP-MS	1	10000
2846	1	Sr ppm: ICP-MS	ICP-MS	1	10000
2847	1	Ta ppm: ICP-MS	ICP-MS	1	10000
2848	1	Y ppm: ICP-MS	ICP-MS	1	10000
2849	1	Zr ppm: ICP-MS	ICP-MS	1	10000
818	1	Crystalline water	Leco RMC100	0.01	100.0
819	1	Surface moisture	Leco RMC100	0.01	100.0
1380	1	S %: Leco furnace	LECO-IR DETECTOR	0.01	100.0
1381	1	C %: Inorganic	LECO-GASOMETRIC	0.05	100.0

The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project

Statement required by Nevada State Law NRS 519

Chemex Labs, Inc.

Analytical Chemists * Geochemists * Registered Assayers

994 Glendale Ave., Unit 3, Sparks
Nevada, U.S.A. 89431
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To: MIDLAND RESEARCH CENTER

P O. BOX 67
NASHWAUK, MINNESOTA
55769

Project : 101.177
Comments: ATTN: BARRY FREY

Page Number : 1-A
Total Pages : 1
Certificate Date: 21-APR-98
Invoice No. : I9816614
P.O. Number :
Account : AAW

* PLEASE NOTE

CERTIFICATE OF ANALYSIS A9816614

SAMPLE	PREP CODE		Al2O3 %	CaO %	Cr2O3 %	Fe2O3 %	K2O %	MgO %	MnO %	Na2O %	P2O5 %	SiO2 %	TiO2 %	LOI %	TOTAL %	Ba ppm
MRC-98-421	217	200	36.68	0.61	< 0.01	1.85	0.17	0.47	0.04	0.46	0.10	20.95	0.09	36.26	97.68	62

CERTIFICATION: Hart Bichler

* ANALYTICAL DATA MAY BE LOW DUE TO A HIGH MOISTURE CONTENT.

Chemex Labs, Inc.

Analytical Chemists * Geochemists * Registered Assayers

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To: MIDLAND RESEARCH CENTER

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55769

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Invoice No. : I9816614
P.O. Number :
Account :AAW

* PLEASE NOTE

CERTIFICATE OF ANALYSIS A9816614

SAMPLE	PREP CODE		Cs ppm	Hf ppm	La ppm	Nb ppm	Rb ppm	Sr ppm	Ta ppm	Y ppm	Zr ppm	+H2O %	-H2O %	S % Total	C % inorg
MRC-98-421	217	200	< 1	< 1	74	< 1	10	35	1	120	7	19.85	9.15	2.36	0.05

CERTIFICATION: Hart Bichler

* ANALYTICAL DATA MAY BE LOW DUE TO A HIGH MOISTURE CONTENT.

Chemex Labs, Inc.

Analytical Chemists * Geochemists * Registered Assayers

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To: MIDLAND RESEARCH CENTER

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NASHWAUK, MINNESOTA
55769

Project : 101.177
Comments: ATTN: BARRY FREY

Certificate Date: 23-APR-98
Invoice No. : 19816616
P.O. Number :
Account : AAW

CERTIFICATE OF ANALYSIS A9816616

SAMPLE	PREP CODE		Rb ppm	Sm ppm	Ag ppm	Sr ppm	Ta ppm	Tb ppm	Tl ppm	Th ppm	Tm ppm	Sn ppm	W ppm	U ppm	V ppm	Yb ppm	Y ppm	Zn ppm	Zr ppm
MRC-98-421	299	297	8.6	11.3	< 1	34.3	< 0.5	2.4	< 0.5	3	1.5	< 1	< 1	28.5	15	10.3	119.5	440	7.5

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SAMPLE	PREP CODE		Ba ppm	Ce ppm	Cs ppm	Co ppm	Cu ppm	Dy ppm	Er ppm	Eu ppm	Gd ppm	Ga ppm	Hf ppm	Ho ppm	La ppm	Pb ppm	Lu ppm	Nd ppm	Ni ppm	Nb ppm	Pr ppm
MRC-98-421	299	297	60.0	81.5	0.6	61.0	5400	15.3	9.9	3.5	14.9	4	< 1	3.3	74.0	5	1.7	60.5	1280	< 1	16.6

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(AAW) - MIDLAND RESEARCH CENTER

Project: 101.177
P.O. # :

Samples submitted to our lab in Sparks, NV.
This report was printed on 23-APR-98.

SAMPLE PREPARATION

CHEMEX CODE	NUMBER SAMPLES	DESCRIPTION
299	1	Pulp; prepped on other workorder
297	1	Meta-borate fusion charge

ANALYTICAL PROCEDURES

CHEMEX CODE	NUMBER SAMPLES	DESCRIPTION	METHOD	DETECTION LIMIT	UPPER LIMIT
2855	1	Ba ppm: ICP-MS	ICP-MS	0.5	10000
2501	1	Ce ppm: ICP-MS	ICP-MS	0.5	10000
2858	1	Cs ppm: ICP-MS	ICP-MS	0.1	10000
2859	1	Co ppm: ICP-MS	ICP-MS	0.5	10000
2860	1	Cu ppm: ICP-MS	ICP-MS	5	10000
2502	1	Dy ppm: ICP-MS	ICP-MS	0.1	1000
2503	1	Er ppm: ICP-MS	ICP-MS	0.1	1000
2504	1	Eu ppm: ICP-MS	ICP-MS	0.1	1000
2505	1	Gd ppm: ICP-MS	ICP-MS	0.1	1000
2861	1	Ga ppm: ICP-MS	ICP-MS	1	1000
2842	1	Hf ppm: ICP-MS	ICP-MS	1	10000
2506	1	Ho ppm: IPC-MS	ICP-MS	0.1	1000
2507	1	La ppm: ICP-MS	ICP-MS	0.5	10000
2862	1	Pb ppm: ICP-MS	ICP-MS	5	10000
2508	1	Lu ppm: ICP-MS	ICP-MS	0.1	1000
2509	1	Nd ppm: ICP-MS	ICP-MS	0.5	10000
2863	1	Ni ppm: ICP-MS	ICP-MS	5	10000
2844	1	Nb ppm: ICP-MS	ICP-MS	1	10000
2510	1	Pr ppm: ICP-MS	ICP-MS	0.1	1000
2864	1	Rb ppm: ICP-MS	ICP-MS	0.2	10000
2511	1	Sm ppm: ICP-MS	ICP-MS	0.1	1000
2865	1	Ag ppm: ICP-MS	ICP-MS	1	1000
2867	1	Sr ppm: ICP-MS	ICP-MS	0.1	10000
2868	1	Ta ppm: ICP-MS	ICP-MS	0.5	10000
2512	1	Tb ppm: ICP-MS	ICP-MS	0.1	1000
2869	1	Tl ppm: ICP-MS	ICP-MS	0.5	1000
2550	1	Th ppm: ICP-MS	ICP-MS	1	1000
2513	1	Tm ppm: ICP-MS	ICP-MS	0.1	1000
2870	1	Sn ppm: ICP-MS	ICP-MS	1	10000
2871	1	W ppm: ICP-MS	ICP-MS	1	10000
2549	1	U ppm: ICP-MS	ICP-MS	0.5	1000
2872	1	V ppm: ICP-MS	ICP-MS	5	10000
2514	1	Yb ppm: ICP-MS	ICP-MS	0.1	1000
2873	1	Y ppm: ICP-MS	ICP-MS	0.5	10000
2874	1	Zn ppm: ICP-MS	ICP-MS	5	10000
2875	1	Zr ppm: ICP-MS	ICP-MS	0.5	10000

The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project Statement required by Nevada State Law NRS 519

Appendix 11

Page

Table A11.1 Annual and monthly precipitation totals (inches) at the Dunka Mine, 1992-1999 . . . . . A11.1

Table A11.1. Annual and monthly precipitation totals (inches) at the Dunka Mine, 1992-99.

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	May-Oct Total *	Annual Total *
1992	---	---	---	---	---	1.97	3.94	3.50	3.46	---	---	---	17.30	24.15
	0.63	**1.35**	**0.56**	**1.75**	**3.66**	**3.71**	**3.43**	**4.06**	**4.42**	**0.77**	**1.10**	**1.46**	**20.05**	**26.90**
1993	---	---	---	---	1.94	2.49	6.71	2.60	2.76	0.93	---	---	17.43	25.12
	0.99	**0.28**	**0.39**	**2.63**	**2.42**	**2.83**	**6.92**	**4.61**	**3.65**	**1.52**	**2.08**	**1.32**	**21.95**	**29.64**
1994	---	---	---	---	1.46	8.75	2.66	1.22	---	---	---	---	19.16	26.63
	0.59	**0.27**	**0.80**	**4.28**	**3.09**	**5.83**	**4.53**	**2.67**	**2.76**	**2.31**	**1.12**	**0.41**	**21.19**	**28.66**
1995	---	---	---	---	0.58	0.55	6.45	3.70	3.94	---	---	---	18.49	23.88
	0.72	**0.74**	**0.86**	**1.41**	**1.58**	**0.81**	**6.15**	**4.96**	**3.44**	**3.27**	**0.40**	**1.26**	**20.21**	**25.60**
1996	---	---	---	---	0.58	0.72	6.29	3.76	4.94	1.94	---	---	18.23	30.59
	2.24	**1.86**	**0.36**	**2.08**	**1.51**	**6.62**	**4.08**	**2.00**	**3.68**	**3.90**	**3.39**	**2.43**	**21.79**	**34.15**
1997	---	---	---	---	---	3.76 [A]	1.21	3.23	1.59	2.53	---	---	15.46	22.34
	2.24	**0.25**	**1.51**	**0.78**	**2.54**	**4.36**	**2.26**	**2.52**	**2.35**	**2.00**	**1.68**	**0.42**	**16.03**	**22.91**
1998	---	---	---	---	---	2.26	1.48	---	---	---	---	---	19.16	27.17
	0.75	**1.19**	**1.04**	**1.20**	**3.47**	**4.18**	**3.51**	**3.88**	**3.78**	**4.29**	**2.32**	**1.51**	**23.11**	**31.12**
1999	---	---	---	---	3.29 [B]	3.66	9.49	3.07	3.19 [C]	---	---	---	28.80	35.43
	1.01	**0.68**	**1.77**	**2.41**	**3.41**	**3.30**	**8.95**	**3.71**	**6.80**	**2.37**	**0.34**	**0.42**	**28.54**	**35.17**
Average	---	---	---	---	1.59	3.10	4.78	3.12	3.75	2.30	---	---	19.38	26.91
	1.15	**0.83**	**0.91**	**2.07**	**2.71**	**3.96**	**4.98**	**3.55**	**3.86**	**2.55**	**1.55**	**1.15**	**21.61**	**29.27**

A: 6/9/97 - 6/30/97
B: 5/11/99 - 5/31/99
C: 9/1/99 - 9/23/99 (since 2.62 in. fell on 9/26/99, according to NOAA, the value of 6.80 was used when calculating the May-Oct and annual totals for the top line (Dunka data).

(Continued on next page)

* The column called total uses all Dunka data when available, and then fills in missing months (mostly during winter) to get an annual total.

Note: The **bottom line** shown for each year (the values in **bold**) are results based on climatoligical data obtained from NOAA, which is accesible at the State Climatologist's web site ((http://www.climatc.umn.edu). There are several weather stations near the Dunka Mine (including Winton, Embarrass, Ely and Tower), but data isn't always available year-round for all these stations. The web site, after being given a site location, automatically calculated which station had the most relevant data. The **top line** of data for each year is the results from actual measurements at Dunka.

Appendix 12

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Section A12.1 Background A12.2

Section A12.2 Wetland lifetime A12.4

Section A12.1 - Background

The primary removal mechanism the W1D is assumed to be the removal of metals by association with the organic fraction of the peat. Removal mechanisms are assumed to be the same as those in small wetland test cells (Eger et al., 1994). Peat contains a complex mixture of organic compounds with a series of functional groups. These groups provide a variety of sites for metals to bind. In the test cells, sulfide precipitation was much less important in the overall removal of nickel. These results were based on the changes in water quality observed in the treatment cell. The pH of the drainage decreased as it passed through the cell; the result of the exchange of metal ions in the water for hydrogen ions on the peat. In contrast, there was no statistically significant difference between the input and output sulfate concentration, which implies that the overall rate of sulfate reduction was small.

The minor amount of removal associated with sulfide precipitation is similar to that reported in studies by Wieder for peat wetlands, in which minimal accumulation of sulfur was found (Wieder and Lang 1986). Wieder suggests that although metal sulfides may form, they are later oxidized and converted to other forms. The oxidation of sulfide precipitates may not be a major concern in a constructed wetland if the water level is maintained so that the substrate remains saturated. In compost based wetlands, however, significant amounts of reduced and elemental sulfur have been measured (Hedin et al, 1989). Compost provides a readily decomposable organic substrate, and sulfate reduction rates up to 1,200 $mmol/m^3/d$ have been measured (Reynolds et al., 1991). More typical rates are on the order of several hundred millimoles per cubic meter per day (Eger 1992).

The specific mechanisms of metal removal are important because they affect the overall lifetime of the wetland. Long-term treatment of mine drainage is an important regulatory issue, and the ability of wetlands to provide continued treatment has not been demonstrated. As a result, the Office of Surface Mining has required back-up chemical treatment for wetlands built to control coal mine drainage (O.S.M. 1988). If the removal is primarily due to removal by organics, then the system lifetime is limited by the total amount of removal sites that are available in the top portion of the wetland. If the primary removal mechanism was sulfate reduction, the process would continue as long as there was an organic food source and an input of sulfate. Sulfate reduction would be more likely to offer long-term treatment. Although additional adsorption sites will be generated annually as plants die and decompose, the formation of new sites is slow. To provide a balance between the input of metals and the formation of new removal sites in the study cell, input flow and load had to be reduced by about an order of magnitude.

Another method to increase the treatment life of the wetland is to construct the system so that the surface peat can be replaced. At the W1D site the peat mixture was placed on top of an existing wetland (Frostman et al., 1993). New material can be added when the removal capacity of the mixture is exhausted.

Another factor that limits the removal in natural wetlands is the transport of metals to reaction sites. Metal concentrations generally decreased with depth at both sites. In natural wetlands, flow occurs primarily across the surface, generally within the upper 30 cm (Romanov 1968). This occurs as a result of a minimal vertical hydraulic gradient and a decrease in hydraulic conductivity with depth. However, wetlands can be constructed to encourage vertical flow and provide more contact with the substrate. This type of wetland requires additional engineering design, but the increase in treatment per unit area can be significant (Eger and Melchert 1992).

Appendix A12.2 - Wetland Lifetime

Assumptions:

- Wetland area 7000 m^2 (original W1D wetland only)
- Removal depth of 20 cm
- Maximum concentration of 10,000 mg Ni/Kg dry peat
- Bulk density 0.1 gm dry peat/cm^3.

Basis of Assumptions

- Depth of removal 20 cm for over land flow wetlands
- Metal concentrations in the substrate decrease with depth. In a study of a natural wetland (Eger and Lapakko, 1988), the highest concentrations were in the top 20 cm of the wetland. Metal measurements made every 2 cm in samples collected from wetland test cells (Eger et al, 1974) showed that concentrations generally decreased with depth.
- At the W1D wetland the highest concentrations were in the 0-10 cm layer, although even in the 20-30 cm layer concentrations were 100 to 1,000 times the initial values.

Peat Density: 0.1 gms

Bulk densities for horticultural peats ranged from 0.07 gm/cm^3 for sphagnum to 0.64 gm/cm^3 for well decomposed peat.

Bulk Density

Peat Type	lbs/cu ft	g/cc
Sphagnum	4.5 - 7.0	0.07 - 0.11
Hypnum	5.0 - 10.0	0.08 - 0.16
Reed-sedge (low lime)	10.0 - 15.0	0.16 - 0.24
Reed-sedge (high lime)	10.0 - 18.0	0.16 - 0.29
Decomposed peat	20.0 - 40.0	0.32 - 0.64

Lucas, R.E., Rieske P.E., and Farnham, R.S. Peats for soil improvement and soil mixes. Michigan State University, Cooperative Extension Services, 11 pp.

Maximum nickel concentration = 10,000 mg/Kg.

Measured concentrations in field samples exceeded this value as well as estimated concentrations in laboratory columns (Lapakko, et al, 1986). This should be a reasonable estimate.

Cattail establishment, five years.

Cattails in a wetland in Pennsylvania reached a stable cattail density after six growing seasons (Stark et al, 1994).

- Initial removed capacity of the wetland = 1400 Kg nickel
- = volume of peat x bulk density x nickel removal capacity
- = area of wetland x effective removal depth x bulk density x nickel removal capacity
- = $700\ m^2 \times (100\ cm/mm)^2 \times 20\ cm \times 0.1\ gm/cm^3$
  $\times\ 10{,}000\ mg\ Ni/Kg\ dry\ peat \times \frac{1}{10^3 gm/Kg}$
- = $1400 \times 10^6\ mg\ Ni \times \frac{1}{10^6\ mg/Kg}$
- = 1400 Kg Nickel

After five years, vegetation is well established and new sites begin to accumulate and increase the removal capacity by 7 Kg Ni/year. (This assumes that new peat accumulates at a rate of 1 mm/year, and has the same removal capacity as the existing substrate). Long term post closure input load to the wetland is assumed to be 10 Kg nickel/year, which is the average of the input for 1997 through 1999.

To solve for the lifetime of the wetland, $t\ 1400 + 7\ t = 525 + 10\ t$ where 525 Kg is the amount of initial removal capacity consumed in the first five years.

10 Kg is the average annual input load
7 Kg is the annual increase in removal capacity
so t = 292 years

Appendix 13

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Section A13.1 W1D treatment system A13.2

Figure A13.1 W1D treatment system expansion (put on line July 1995) A13.7

Section A13.2 Seep 1 pretreatment system A13.9

Section A13.1

This appendix contains miscellaneous notes and calculations primarily relating to the design, operation, and performance of the W1D wetland. It is a compilation of old files. Although this report focused on the original part of the system constructed in 1992, this appendix also contains some discussion on the extension constructed in 1995.

Observations on the W1D Treatment System

Removal increased in this section in the summer of 1994 after hay bales had been placed on several berms. The bales helped to disperse flow and H_2S was detected. Concentration decreased substantially during this time period, although removal decreased as temperatures began to decrease in the fall.

There was not enough contact time in the hay bale area itself to provide for reaction, but it is possible that organics were washed from the hay into the pool areas downstream where there was enough residence time to allow sulfate reduction to occur.

There was a notable smell of H_2S in the area after the hay had been installed and evidence of biological reactions occurring in the downstream pools.

The hay also stabilized the system with respect to high flows and helped distribute the flow.

Lower part of 1D system beyond glacial till berm is only about one-half the size of the upper system; therefore, expect a lower rate of removal.

Cause of excess channelization in lower part of system; berms are narrower so have a higher average velocity.

Design of W2D/3D system meets requirements based on test cell data. Drop across berm in 2D/3D is less than in 1D, ½ to ¾ ft drop where in W1D, the design called for a foot, in absence of rip rap will scour.

Dye Studies

1993 dye study conducted by NRRI indicated more channelization in lower half of system and less contact time

4 days prior to glacial till berm, 3 days below

NRRI conducted two dye studies at W1D in 1993. Both occurred during periods of relatively low flow.

first started on Aug 16
took about 7 days for the peak to reach outflow

average outflow for this period was 23 gpm (range 15 - 35) using this flow rate the effective volume of the system is V = 232,000 gallons

The key sample points are at the input culvert under the road, the three pipes through the large till berm, and the outflow. It is at these points that all the flow is captured.

Summer Test

Note: There are some strange things in the flow data file. This is particularly noticed in input file. Need to check ppt. Our flow measurements show a downward trend while LTV shows an upward trend. In addition, the staff readings are higher for LTV than they are for the spot readings taken when we bucket gaged. Flow estimates have been made from LTV files and bucket gaging.

Estimated input flow during 7-day period: 30-40 gpm

Estimated outflow: 20-30 gpm

Time of travel was computed by the time to peak concentration

Time from input to till berm:

minimum 93 hours
other times 118-160
about 4 days if assume most of the flow goes through the 2 end pipes. These are the ones with the time of travel of 93-118 hours

from till berm to outflow is on the order of 46-71 hours or 2 to 3 days

If we assume that the most flow goes through the pipe with the shortest travel time, then
flow from the input culvert to the till berm is about 4 days
flow from till berm to outlet weir is about 3 days
total time 7 days

Design Calculations

The final size of the wetland must consider both metal and hydraulic loading and should be the largest area calculated.

Data from the test cells indicated that the minimum residence time was two days.

Residence time (days) = Volume of water (m^3) / Flow (m^3/day)
= Area of wetland x average water depth / flow

Wetland area (m^2) = residence time (days) x flow (m^3/ day) / average water depth (m)

Assuming a water depth of 5 cm, an average flow of 20 gallons per minute and a minimum residence time of two days, the minimum area would be 47,000 ft^2. The required area also needs to be calculated based on the rate of metal removal. Removal rates are calculated on an areal basis by dividing the mass of metal removed by the area and the number of days of operation.

areal removal rate (mg/ m^2/ day) = mass removed (mg)/ wetland area (m^2) / days

Nickel rates in the test cells ranged from 4 mg/ m^2/ day for the majority of the cells to 8 mg/ m^2/ day in the cell where the peat and peat screenings mixture was added. The area of the wetland can be calculated from

area (m^2) = average input mass (mg/ day) / areal rate of removal (mg/m^2/ day)

Using an average flow of 20 gallons per minute, a nickel concentration of 5 mg/L and an areal rate of 8 mg/ m^2/ day, an area of 68,000 ft^2 would be required. Since these calculations are based on average flow conditions, the wetland may not be sufficiently sized to treat higher flows resulting during periods of high rainfall which produces substantially larger input flows.

Actual residence time calculated by NRRI using dye were higher than calculated from the simple model. The actual W1D wetland does not have a uniform water depth or a uniform distribution of flow.

Design of Original W1D Wetland Treatment System

STS consultants designed system based on the 1990-91 flow data.

Problems:

1. This is a period with reported flow lower than long term average.

2. Based on our measurements of flow in 1993, LTV flow estimates were about 15% low. We do not know if this condition existed in 1990 and 1991, but there has been concern about the accuracy of LTV flow records.

   The difference in flow between LTV and DNR was on the order of 5 -15 gpm. At low flows the % error was higher.

   example LTV 11 gpm % difference = 39
   DNR rating curve 18

LTV 65 % difference = 16
rating curve 77

3. Initially the size of the system was too small, even based on the low flow estimates. After comments were submitted, the system was resized, but still the size was based on a 20 gpm average flow.

   A 40 gpm was probably more realistic and better reflected long-term averages.
   A complete analysis of flow had never been completed.

Construction began in March of 1992

System Design

- Nine berms, each about 1 foot high at successively lower elevations, length about 700'
- First berm elevation 1485
- Berms drop by 1 foot intervals down to 1477
- Outlet weir was not installed initially
- Gradient toward end of system
  - 3' drop in 200', 1.5% drop from berm 5 to 8
- initial gradients
  - 2' in about 200' (first 3 berms) from berm 1 to 3
  - 2' in 175 from berm 3 to 5
  - from 8 to 9 1' in 100'

Peat Mixture

LTV area 5 plus Michigan peat screenings in last three cells. Other areas had area 5 peat plus screenings from Minnesota sphagnum

Vegetation

- berms hand seeded with Japanese millet
- open areas seeded with cattail seed

- took heads of cattail and mixed in solution of soap and water; add few bolts and agitated.
- bolts helped break up heads; soap helped to disperse seeds.

total area of peat is 75,000 ft sq

1992

attempted to better distribute flow by

- rip rap on first berm
- installed treated 4 x 4 treated lumber to berms 2, 3, 4

decided peat in last 4 cells was not proper pH
installed pretreatment above weir

Based on field observations, it was determined that some flow bypasses pretreatment system

prior to start of 1993 season peat in last 4 cells was replaced.

at 1481 berm, raised berm (glacial till berm) and added pipes (this is berm 5)

this reduced the gradient, took about 18" of drop out of the system at this elevation?

need elevation of drop structure pipes and large berm

inlet slots at 1481.5
raised berm to 1482

so only lowered water level 6"

<u>Historical Flow at W1D</u>

Prior to 1986, there were no flow records for the W1D site. Flow for these periods was estimated from runoff coefficient for EM-8 watershed.

This method may have been OK prior to pumping of environmental sump and development of EM 8-E.

Best to use data from 86-91 since this was measured at site

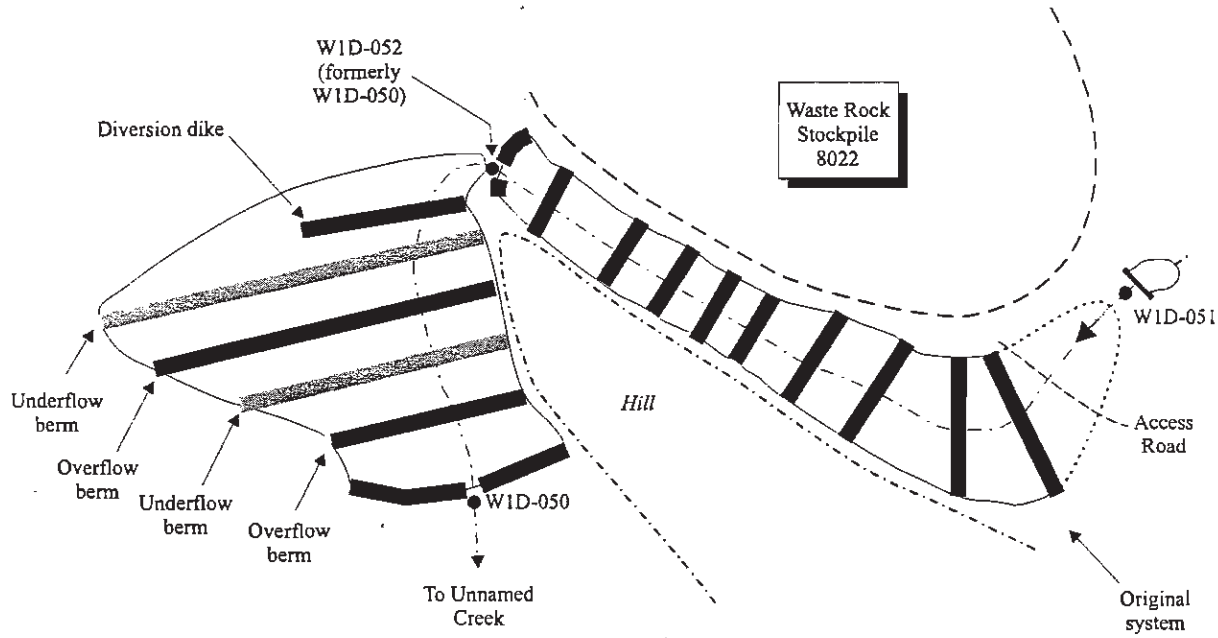

Figure A13.1. W1D treatment system expansion (put on line July 1995).

Average flow for 86-89

total 0.520 L X 10^8

assume an average of 225 days of flow get yearly average of 42 gpm

average flow for 90-91

25 gpm

average flow for the entire period 36 gpm

<u>Notes on W1D Extension - Constructed in 1995</u>

- Berms are spaced 75 ft. apart
- Alternate between under flow and overflow
- About 25 ft. of peat is placed on each side of the berm, this provides an area of around 25 ft. of open water between each set of berms

- New design, rather than have flow over the berms, standpipes will be used to get water flow through berm, plan to use rip rap on top of till berms.

- LTV removed existing peat so berms can be built on mineral soil and minimize settling. In the original system, berms were built directly on peat.

- The estimated hydraulic conductivity for the peat and screenings mixture seems very optimistic. The closure document has some lab data which yields conductivity on the order of 0.1 cm/sec.

- Even with a hydraulic conductivity of this magnitude, only a portion of the flow will flow through the underdrain. The estimates are on the order of .1-1 gpm.

<u>Calculation of flow in underflow berms</u>

It is unlikely that all the flow will be transmitted in the underdrain. Provisions need to be made for flow over the top of these berms.

- First underflow berm
  - length 400 ft., depth of limestone 1.5 ft.
  - cross sectional area: 600 sq. ft.

- Hydraulic head
  - invert elevations for pipes through overflow berms not given, it appears that inverts will be about .5 ft. below the top of the berm. This will give a head of .5 ft.

  - width of berms is about 50 ft.
  - gradient .5 / 50 = .01

- Flow rate

  - assume permeability of .001 cm/sec
  - Q = KAH/L

    $$\frac{.001 \text{ cm/sec} \times 600 \text{ sq. ft.} \times .01 \times 60 \text{ sec/min.}}{30.48 \text{ cm/ft.}}$$

    .012 cu ft/min x 7.48 gal/ cu. ft
    .09 gpm
  - if permeability was increased it could increase flow, but to get substantial flow through underdrain, i.e., 9 pgm, would need permeability of .1 cm/sec.

Section A13.2
Seep 1 Pretreatment System

Flow Reduction, Seep 1

	May - October		
	ppt (in)	Flow	Flow/ppt
Pre-capping 1986-1991	21.2	7.3[a]	.34
Post-capping 1993-1999	19.5	5.1	.26
1993-1998	18	4.6	.25
Projected[b]	21.2	5.4	

[a]1989 flow data omitted since more water was reported at the weir than fell on the watershed.
[b]The lower flow in 1993-1998 is due in part to less precipitation. If it is assumed that the ratio of flow/ppt is constant, the projected flow at 21.2 inches of ram can be estimated by:

$$\frac{\text{Flow}}{\text{ppt}} = .25 = \frac{X}{21.2}$$

$$X = 5.4$$

% reduction in flow after capping

$$\frac{7.3 - 5.4}{7.3} \times 100\% = 26\%$$

if the difference in rainfall is not included the reduction in flow is 37%.

If 1999 is included (it was excluded due to the unusual high rainfall on July, May-October rainfall was 28.8 in.

- without correction for precipitation: 31%
- with correction for precipitation: $.26 = \frac{X}{21.2}$; $X = 5.5$ % = 25

If we assume a 25% reduction in flow:

- Average mass for 1986-1991 (1989 excluded due to problems with flow data = 124)
  - Nickel = 124
  - Copper = 5.4

- Reduction due to flow (assumes no change in concentration)
  - Nickel = 93 Kg
  - Copper = 4 Kg

- Average mass at weir for 1993-1998
  - Nickel = 38
  - Copper = 0.9

- Annual estimated removal
  - Nickel 93-38 = 55
  - Copper 4-0.9 = 3.1

- Total estimated removal for 1993-1996 (corresponds to peat samples)
  - Nickel = 220 Kg
  - Copper = 12.4 Kg

Nickel is much greater than the removal calculated by using outer pool concentrations (77 Kg) and the metal values in the substrate (30 Kg).

Copper is close to both the outer pool estimates 11.1 Kg and the substrate estimate 9.4 Kg.

This discrepancy suggests that the overall load of nickel has decreased as a result of capping, but not copper. During 1993, the entire copper load could be accounted for from the small surface seepage at the top of the stockpile.

Appendix 14
W1D Pretreatment System

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Section A14.1.
W1D Pretreatment System

A pretreatment system was installed at the toe of the 8018 in 1992. The goal was to remove metals before they entered the W1D wetland treatment system. In 1993 the DNR conducted sampling in and around the system and determined that water was bypassing the pretreatment area (DNR data summary, 1993).

In March 1998, substrate samples were collected from the system to examine metal removal. The objective was to determine if the lower loads into the wetland were due to a large metal uptake in the pretreatment system.

Although nickel was elevated in the substrate, the relatively low concentrations, 1000-3000 mg/Kg, and small area did not account for the change in nickel load.

Table A14.1. Description of W1D pretreatment peat samples, March 17, 1998.

Site	Ice Thickness (cm)	Water Depth (cm)	Total Peat depth (cm)	Intervals (cm)	Description	Comments
1	13		100	0-10	Ice chips	Ice to 13 cm, peat frozen to ~ 36 cm; heavy Typha
				10-20	Ice chips	
				20-30	Few ice chips, then McCaully	
				30-60	Peat got moist ~ 50 cm, was dry prior	
				60-100		
2	22		110	0-2	Blk fibrous peat	Heavy Typha
				2-8	Gray silt - discarded	
				8+	Blk peat	
3	10		110	0-10	White fibrous roots	Thick Typha
				10-20	Some fibrous roots	
				20+	Minimal roots	
4	15	24	102	40-50	Occ. 1-2 silty, gray ½" dia. chunks throughout profile. Sand in peat	Medium Typha
5	10		100		Strong odor, hydrogen sulfide	No Typha

Each sample was split into three segments: 0-20, 20-50, 50-100, except at Site 1 where intervals analyzed were 0-10, 10-20, 20-30, 30-60, and 60-100.

Table A14.2. Chemical analyses, 1999 W1D pretreatment peat samples.

Site	Percent moisture	Percent ash	pH	Cu	Ni	Sample
1: 0-10 cm	72	43	6.21	153	1510	11533
10-20 cm	69	68	5.78	86.3	265	11534
20-30 cm	74	67	5.90	45.8	105	11535
30-60 cm	72	62	6.15	78.2	49.7	11536
60-100 cm	51	62	6.21	77.4	53.8	11537
2: 0-20 cm	72	73	6.05	221	428	11538
20-50 cm	73	68	6.23	206	146	11539
50-100 cm	73	64	6.63	95.8	96.0	11540
3: 0-20 cm	75	66	6.50	797	2730	11541
20-50 cm	77	43	6.86	118	148	11542
50-100 cm	72	67	6.92	96.9	126	11543
4: 0-20 cm	75	67	6.15	96.2	304	11544
20-50 cm	73	64	6.52	85.3	96.7	11545
50-100 cm	73	73	6.58	54.5	68.8	11546
5:. 0-20 cm	75	60	7.00	165	1720	11547
20-50 cm	76	64	7.12	266	190	11548
50-100 cm	71	66	7.23	60.0	124	11549
6: 0-20 cm	84	61	6.33	125	1030	11550
20-50 cm	76	68	6.59	51.4	95.4	11551
50-100 cm	73	65	6.92	59.2	96.5	11552

Note: copper and nickel values are mg of metal per kg of dry peat.

July 28, 1992

Mr. Dennis Koschak
LTV Steel Mining Company
P. O. Box 847
Hoyt Lakes, MN 55750

STS Project 94000-D

Re: Seep W-1d Pretreatment System at the Dunka Mine - LTV Steel Mining Company
NPDES Permit No. MN0042579

Dear Mr. Koschak:

On July 24, 1992, the Minnesota Pollution Control Agency (MPCA) approved use of a pretreatment system at Seep W-1d in the inert pad material above the existing peat/wetland treatment system. The pretreatment system will not alter the treatment system or water reporting to monitoring point 051. All water discharged through monitoring point 051 is treated through the peat/wetland treatment system at outfall 050.

Enclosed is a plan and specification which should be used to construct the pretreatment system. Originally, Type I compost was planned to be the organic stratum within the system. The Minnesota Department of Natural Resources (MDNR) has, to date, identified that the only sources for Type I compost are in St. Cloud and the Twin Cities. These materials would not be acceptable due to costs and potential alien contaminants that might be in these urban materials. As such, the plan provided to you identifies peat as the organic treatment media. The peat should have the following properties:

- Two parts Minnesota Sphagnum Inc. peat with one part Area 5 peat or one part Coarse Agri-peat from Aitken, Minnesota with one part Area 5 peat
- pH greater than 5.0
- Von Post greater than 3.0
- Organic content by dry weight greater than 85%
- Hydraulic conductivity minimum 1 meter/day

If you or the MPCA have any questions concerning the plan, please do not hesitate to contact us.

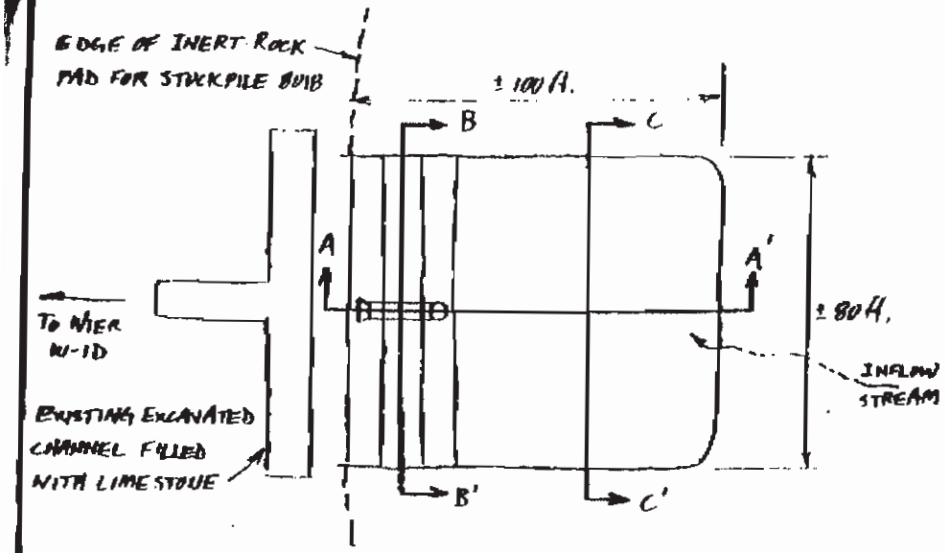
Respectfully,

STS CONSULTANTS, LTD.

Theodore M. Frostman
Principal Scientist

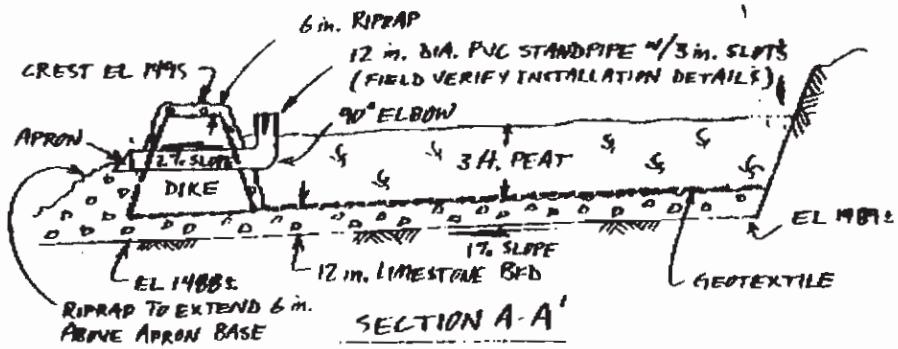
Stephan M. Gale, P.E.
Principal Engineer

SMG/dn
Enc.

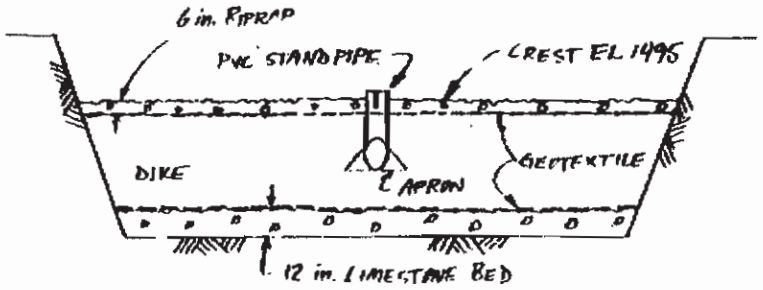


PLAN (No Scale)

SECTIONS, No Scale
Vertical Exaggeration ≈ 4 x



SECTION A-A'



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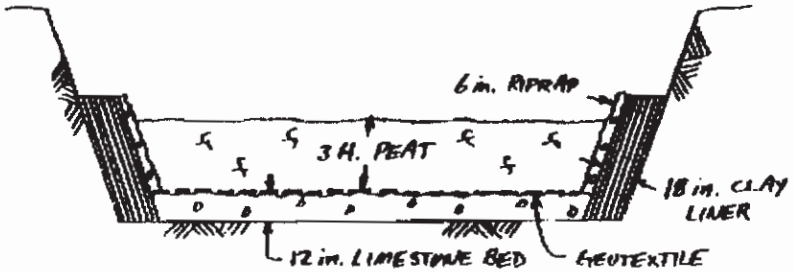


SECTION C-C'

NOTES

A. Construction Sequence

1. Excavate treatment basin to dimensions shown with 1% minimum slope @ bottom, 1.5H:1.0V sideslopes.
2. Place 18-inch compacted clay liner 7 sides. 9 inch lifts compacted to 95% ASTM D-698.
3. Place 12-inch limestone bed.
4. Place geotextile over limestone bed.
5. Construct dike in 12-inch lifts compacted to 95% ASTM D-698, 1.5H:1.0V sideslopes. Install PVC standpipe, culvert & apron. Field verify installation details.
6. Place geotextile over dike
7. Place 6-inches riprap over clay liner and dike.
8. Place 3-foot loose organic medium in treatment basin as directed by Owner.
9. Place additional limestone @ toe-of-dike to 6-inches above apron base and build into existing channel.

B. Materials

1. Limestone bed & riprap material to consist of minus 6 inch to plus 2-inch non-dolomitic crushed limestone.
2. Dike material to consist of minus 1/2-inch glacial till, screened 8986 stockpile material).
3. Organic medium to consist of Peat, Class I compost or equal as approved by Engineer.
4. Clay liner material to consist of clay borrow from Ely or Aurora site.
5. Geotextile to consist of non-woven needle-punched geotextile conforming to MnDOT 3733 Type 4 physical properties, 12-inch overlaps @ seams.
6. PVC standpipe & culvert to consist of 12-inch diameter PVC pipe with three 3-inch wide vertical slots at top-of liner

OUTFALL 050-W-1D PRE-TREATMENT
DUNKA MINE
LTV STEEL MINING COMPANY

SCALE NONE

Section A14.3.
Field Notes, 8/6/92 and 8/11/92

8/6/92

1. There is roughly an 18" drop from the top of the stockpile pad to the top of the peat.

2. Clearly, most of the Q does not surface in this system.

3. LTV, or someone, should determine whether runoff control structures are necessary just north of point A on the map. I did not check this out, but my recollection is it is likely that significant quantities of runoff and sediment could wash into the peat bed.

4. The peat bed "pond" shows evidence of being 100% saturated at some time in the past. Peat at the downstream side has definitely settled about one ft. based on the perched peat high water mark on the riprap. There is less subsidence indicated on the upstream end.

5. There appears to be a slight slope to the peat bed.

6. At the downstream end, the peat level appears to be 1-2 ft. higher than undisturbed land surface at the collection trench.

7. Seepage at point B, although significant, appears much less than at weir. Weir was bucket gaged at 26.5 gpm.

8/11/92

8. Hoover excavated a ~12 ft x 14 ft pit at point B on figure. Pit was excavated from rock pad and not the peat. At 1430 hours water in pit was turbid with no signs of flow.

9. Noted significant flowage (many gpm) at point C. This was not present on 8/6.

10. Saw a small head drop from pit to the peat. This is due to a small (3") peat berm at the edge of the pit.

11. Riser pipe not installed yet.

12. Pond was full this time. Water level now just below pipe. Saw evidence that water was 1-2" higher and spilled into the pipe. This probably occurred earlier today due to heavy rain this morning.

13. All peat was saturated. Even with water level high, there was still a noticeable gradient across the peat surface for the upstream half of the system.

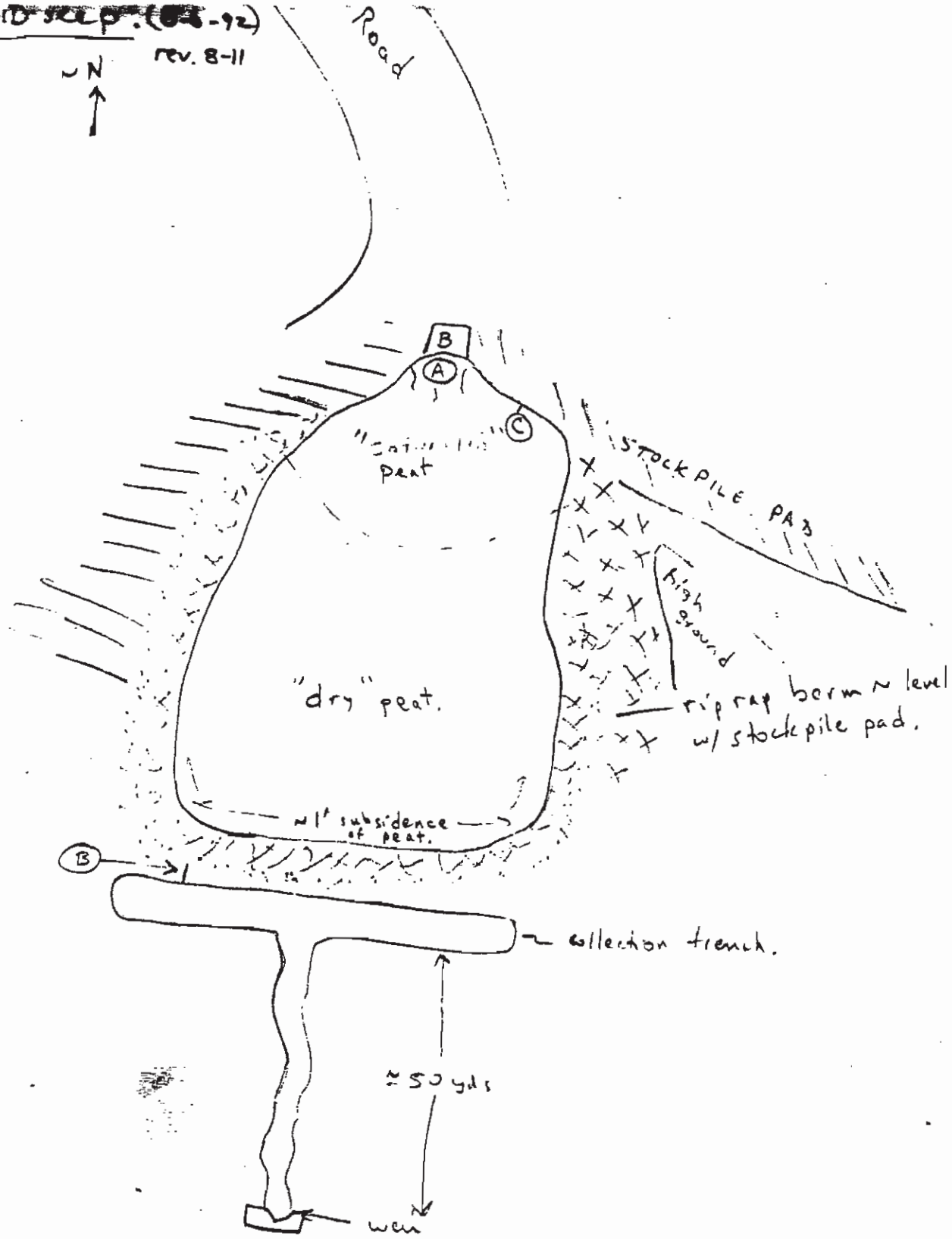
14. The surface of the downstream half of the system was flat. Near the spill pipe there was a 2-3" thick floating mat of peat. There was 9-12" depth below the mat before peat occurred.

15. Water in the collection trench was ~ 1 ft. deep. There was geotextile in the trench.

16. Minor seepage observed similar to earlier visit.

17. All exposed riprap is local in origin (i.e., not limestone).

W-1D SCLP: ([redacted]-92)
rev. 8-11



Notes

(A) = zone of very minor trickles of water. Do not see water coming from pad. Could be a zone of upwelling.

(B) = seepage – can be sampled

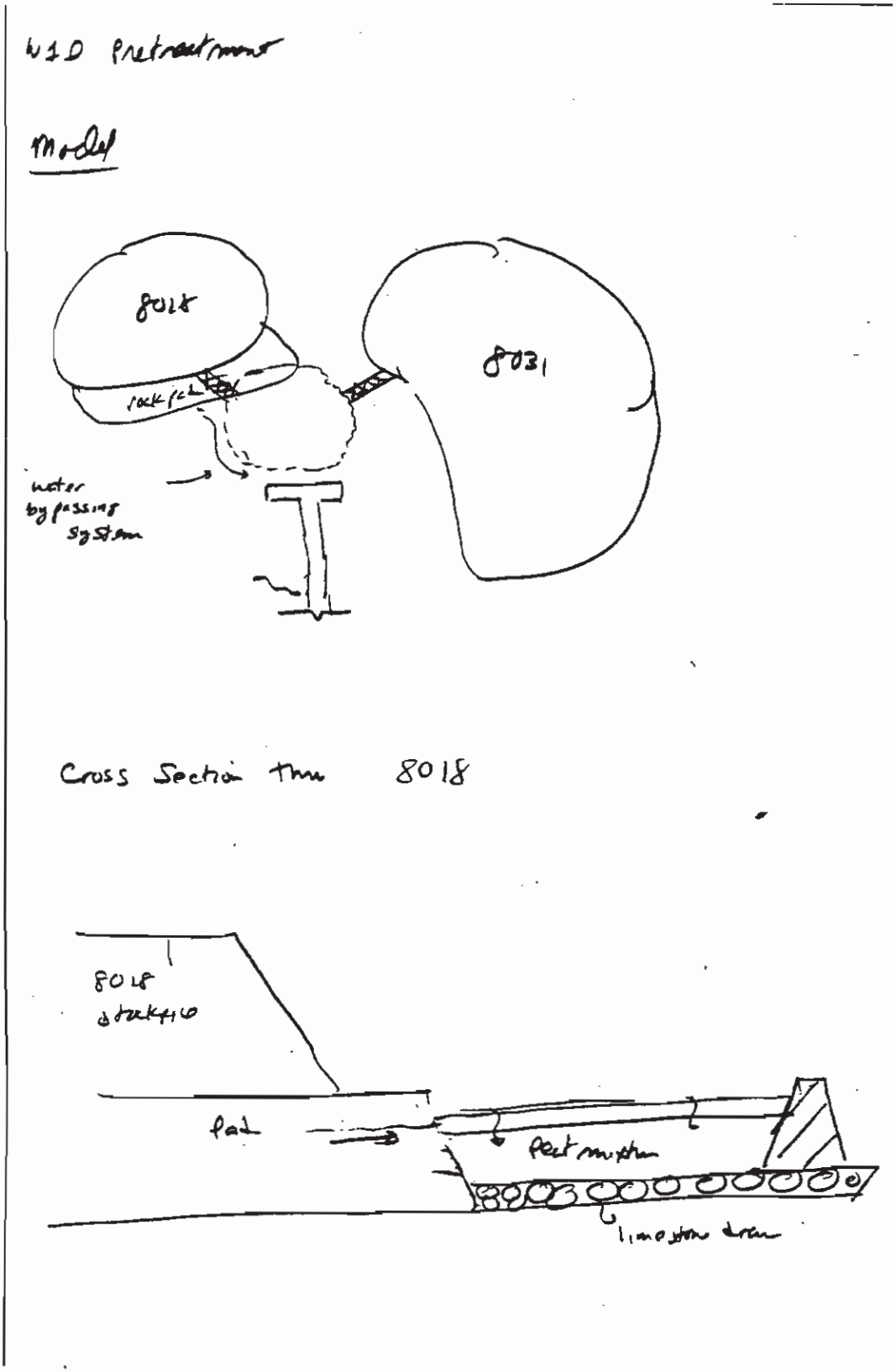
Section A14.4
Water Quality Sampling

In 1993 samples were collected in and around the W1D pretreatment bed in an attempt to determine treatment efficiency. Since water bypassed the system and most of the water enters as groundwater, it was not possible to easily determine the performance of the system.

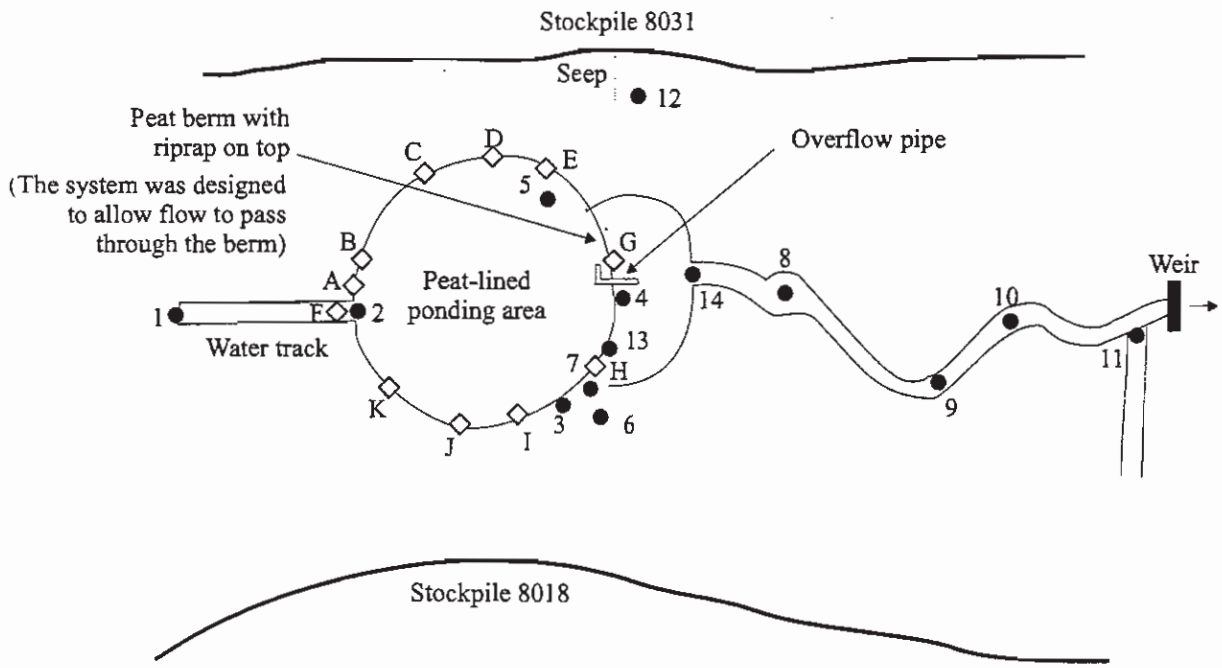
Figure A14.1 provides a schematic of the system. The sampling sites for both 1993-1994 and for a survey in 1998 are shown in Figure A14.2.

Table A14.3 provides a description of the sites sampled on 7/22/94.

Figure A14.1. W1D pretreatment system, schematic.



22-141 50 SHEETS
22-142 100 SHEETS
22-144 200 SHEETS
AMPAD



Note The numbered sites (black circles) have been sampled in the past, while the sites with letters (white diamonds) were sampled during the survey conducted at 1530 hours on 9/3/98.

Figure A14.2. Sample locations, W1D pretreatment system.

Table A14.3. Site Descriptions, 1993, 1994.

1. Standing H_2O at beginning at ditch
2. Pond side at limestone
3. Rapid seep coming through pad – 10' west of treatment berm
4. Pool area – next to outflow culvert
5. SE corner of treatment pond
6. Small seep that adjoins #3 above
7. Random site where a small seep appeared to be separated; then joining flow with #3 and #6
8. Random – larger open area
9. Random – sharp bend in stream bend
10. Random – mid-stream
11. Random – appeared as if a flow may be hidden under vegetation
12. Seep – standing H_2O in intermittent seep, next to 8018 stockpile.

Note: Downstream sites are approximately 3-10 yards apart.

Table A14.4. Drainage quality in the area of the W1D pretreatment system, DNR sampling 1993.

Site	Date	pH s.u.	S.C. µS/cm	SO4 mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Ca mg/L	Mg mg/L	Flow L/s
W1D-3	7/29/93	6.87	2900	2080	0.10	8.08	0.17	0.10	281	344	Med
	8/5/93	7.01	4020	2680	0.14	10.00	0.21	0.14	361	482	
	8/12/93	NA									0.400
	8/17/93										0.309
W1D-6	7/29/93	6.60	2400	1530	0.04	2.62	0.07	0.04	232	269	No
	8/5/93	6.23	1100	585	0.04	1.69	0.08	0.04	85	89	Med
W1D-7	7/29/93	7.16	2900	920	0.05	6.92	0.13	0.08	276	326	No
	8/5/93	7.33	3400	2060	0.05	6.03	0.12	0.07	309	366	Low
	8/12/93	NA									0.021
	8/17/93										0.010
W1D-11	7/29/93	7.13	2800	1910	0.07	5.73	0.07	0.07	278	318	Low
	8/5/93	7.05	3500	2320	0.11	7.96	0.09	0.08	330	410	Med
W1D-12	7/29/93	6.87	3000	1960	0.05	1.96	.012	0.33	304	356	No
	8/5/93	6.49	3520	2380	0.03	2.29	0.09	0.36	342	421	Low
W1D-13	7/29/93	7.18	2950	1870	0.07	6.34	0.12	0.08	285	332	Med
	8/5/93	7.49	3150	2130	0.05	3.84	0.08	0.04	301	326	
	8/12/93	NA									0.164
	8/17/93										0.076

Table A14.4. Drainage quality in the area of the W1D pretreatment system, DNR sampling 1993.

Site	Date	pH s.u.	S.C. μS/cm	SO4 mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Ca mg/L	Mg mg/L	Flow L/s
W1D-14	7/29/93	6.38	2650	1730	0.07	5.71	0.10	0.07	284	323	Low
	8/5/93	7.27	3500	2400	0.07	6.66	0.12	0.07	323	383	Med

NA - not analyzed.

Note: Metals values are total concentrations.
All flow measurements were taken under relatively low flow conditions. Qualitative flow descriptors for all sites except W1D-14 may be estimated as:

No = no flow
Low = <0.063 L/s (<1 gal/min)
Med = 0.063 to 0.505 L/s (1 to 8 gal/min)

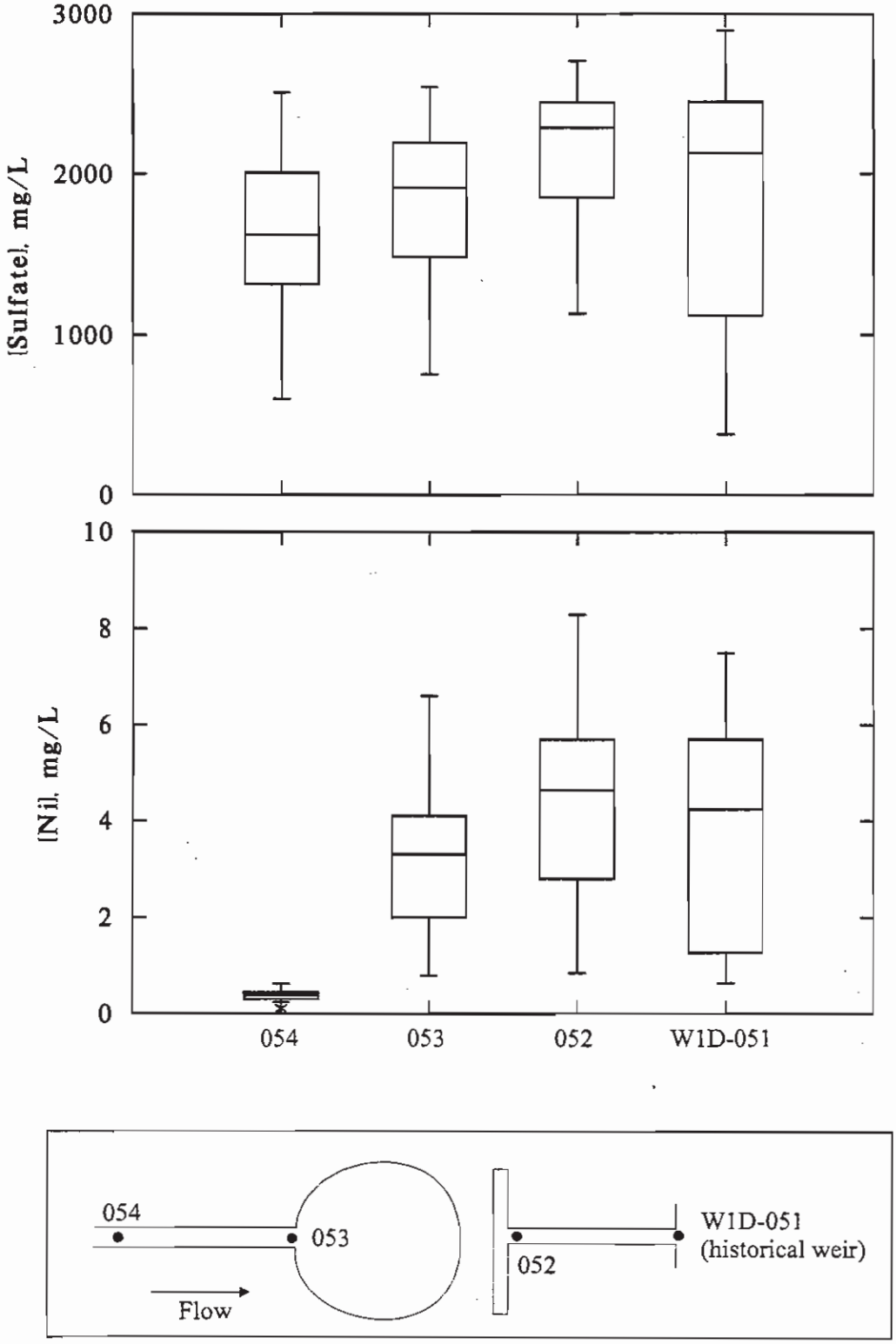


Figure A14.3. Nickel and sulfate box plot, W1D pretreatment, 1993-94.

Figure A14.4. Specific conductance survey of W1D pretreatment system, 7/22/94.

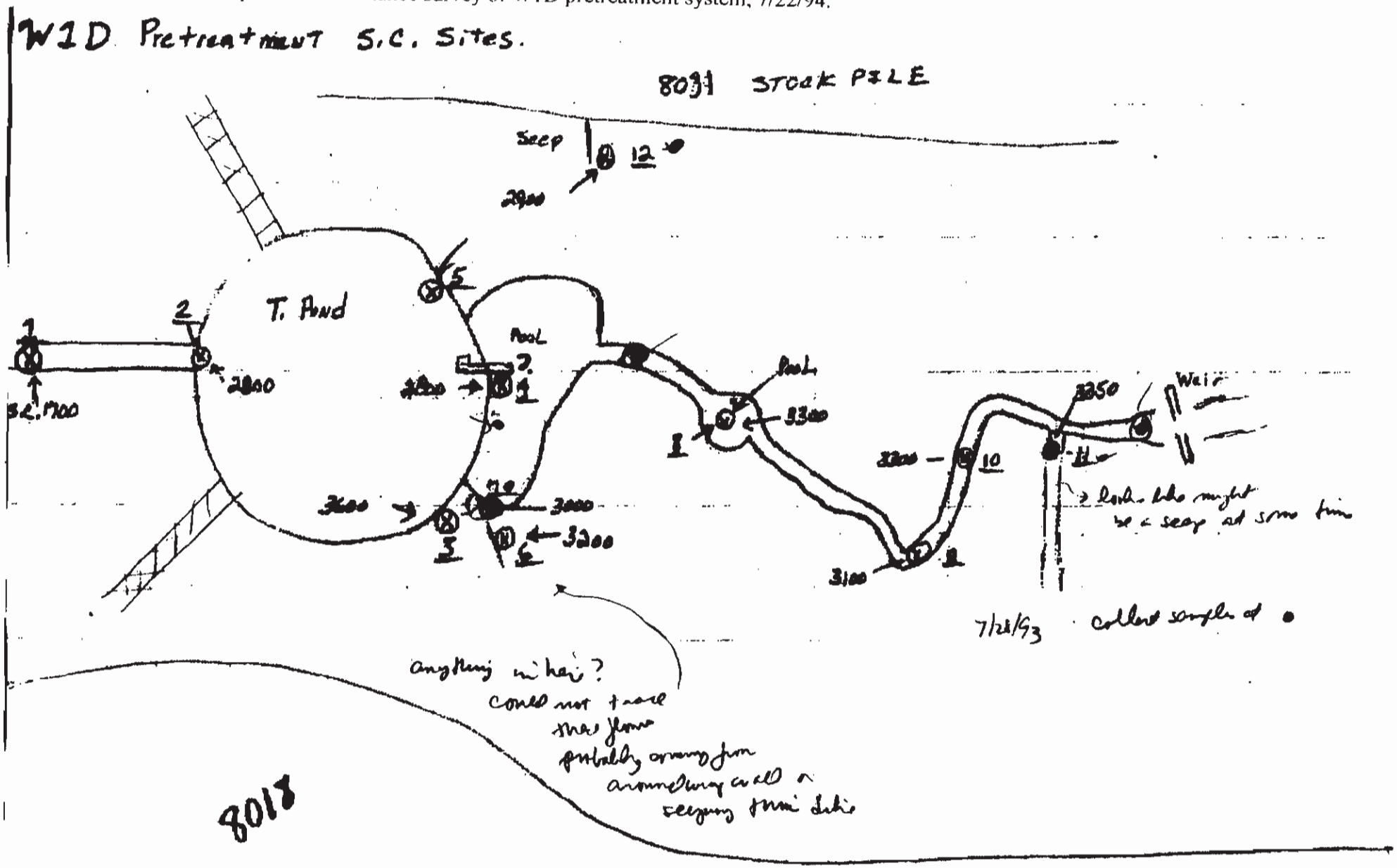


Table A14.5. Flow and water quality in the W1D pretreatment system, 9/3/98.

Site	S.C.	Q (est.)	Ni (ppm)
2	1600	Slight	1.63
A	2000	Slight	-
B	2300	Slight	-
C	1850	-	-
D	1900	-	-
E	1900	-	-
F	1000	-	-
G	1900	-	1.15
H	1900	-	-
I	2000	-	-

Site	S.C.	Q (est.)	Ni (ppm)
J	1950	-	-
K	1900	-	-
3	1800	1 L/min	2.08
6	1800	Slight	-
7	1750	Slight	-
11	1900	None	0.37
13	2000	0.3 L/min	0.12
14	1900	None	0.80
Weir	1900	13.7 L/min	0.11

Attachment 8

ITRC (Interstate Technology & Regulatory Council) (2010) Dunka Mine, Minnesota. Washington, D.C.

CASE STUDY

Dunka Mine
Minnesota

August 2010

Prepared by
The Interstate Technology & Regulatory Council
Mining Waste Team

Permission is granted to refer to or quote from this publication with the customary acknowledgment of the source. The suggested citation for this document is as follows:

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The ITRC Mining Waste Team would like to acknowledge Paul Eger from the Minnesota Department of Natural Resources for preparing this case study.

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DUNKA MINE, MINNESOTA

1. SITE INFORMATION

1.1 Contacts

Paul Eger
Minnesota Department of Natural Resources
Telephone: 651-259-5384
E-mail: a.paul.eger@state.mn.us

1.2 Name, Location, and Description

The Dunka Mine is a large, open-pit taconite mine located in northern Minnesota at the eastern end of the Biwabik Iron Formation. The mine covers approximately 160 hectares and has a depth of around 100 m. It sits along the western edge of a small watershed (920 hectares), which is drained by a small stream (Unnamed Creek, Figure 1-1). The watershed is typical for this area of Minnesota and is characterized by a series of upland ridges and low areas containing wetlands. Sulfide-containing waste material from the mine was stockpiled along the eastern edge of the mine and adjacent to these wetlands.

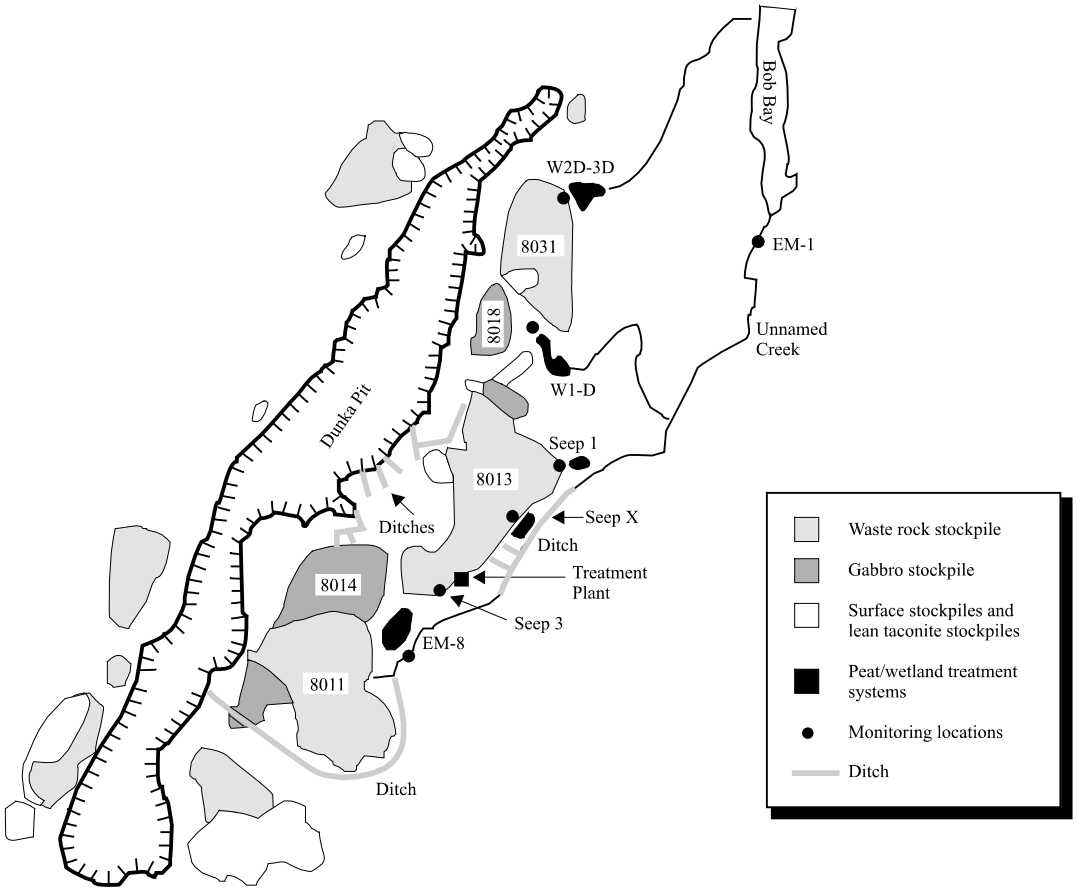

Figure 1-1. Dunka Mine waste location schematic.

The sulfide material came from an igneous intrusive rock formation known as the Duluth Complex. This formation is a gabbro which contained disseminated copper and nickel sulfides and overlies some of the iron ore at this site. During 30 years of operation, over 50 million metric tons of sulfide-containing waste was removed and stockpiled on 120 hectares of land adjacent to the mine. As water infiltrated this material, drainage with elevated copper, nickel, cobalt, and zinc was produced. Due to shallow water tables, precipitation that infiltrates the stockpiles appears as seepage in many places along the toes of the stockpiles. Although some of this seepage is diffuse, the majority of the flow occurs as discrete flow, which generally flows continuously from spring melt in April till freeze-up in late November. Average flows from the various seepages prior to closure have ranged from 3 L/min to 840 L/min, but flows exceeding 6000 L/min had been observed after periods of heavy precipitation. Annual median concentrations in the drainage from the stockpiles of Duluth Complex have ranged 1.5–50 mg/L for nickel, <0.1–1 mg/L for copper, <0.1–3 mg/L for cobalt, and <0.1–4 mg/L for zinc. Most of the drainage had pH greater than 6.5, but two sites produced acidic drainage.

2. REMEDIAL ACTION AND TECHNOLOGIES

In 1986, LTV conducted a preliminary feasibility study to determine the best method to mitigate the drainage problem at the Dunka Mine, examining both active treatment systems (lime treatment, reverse osmosis) and passive alternatives (limiting infiltration into stockpiles, wetland treatment) (Barr Engineering 1986). An active treatment plant which would treat all the stockpile drainage was projected to have a capital cost of $8.5 million and an annual operating cost of $1.2 million. The passive alternative was projected to cost $4 million to construct but only $40,000 in annual maintenance. Since mine drainage problems can persist for over 100 years, LTV decided to pursue passive alternatives.

However, acidic drainage developed in 1989 at a site (Seep 3) which had previously been neutral (Figure 1-1). As a result, the company was forced to construct a lined equalization pond and an active treatment plant to treat seeps emanating from some of the stockpiles on the eastern side of the mine. LTV's goal was still to have a completely passive system, and when the decision to close the mine was made in 1991, LTV began to develop and implement a final closure plan. This plan included reducing infiltration into the stockpile and treating all residual drainage with wetland treatment systems. Since there was a limited area available for construction and wetland systems function best under low-flow conditions, it was important that the flow rates be reduced as much as practical. This was accomplished by reducing the upstream watershed which may contribute to the drainages and by capping the stockpiles to limit infiltration of precipitation.

Watershed Reduction, Diversion

LTV conducted an extensive investigation of the surface and groundwater in the area and identified several areas where surface water was believed to flow through Duluth Complex stockpiles. Drainage ditches were constructed to route water away from the piles (Figure 1-1). Although most of these were relatively shallow ditches (~2 m), the ditch at the southern end of the mine, around the 8011 stockpile, was much larger. To isolate a relatively large upstream watershed (83 hectares), a 670 m long ditch, up to 18 m deep, was excavated through bedrock into the mine. The original plan was to drill and blast to fracture the bedrock and create a drain.

The drain did not have sufficient permeability to drain the area, and as a result, the bedrock had to be excavated. This ditch cost over $600,000 and accounted for the majority of the total ditching budget.

Based on the hydrological study, LTV also concluded that by channelizing flow in Unnamed Creek it could lower the water level in the surrounding wetlands and, as a result, reduce the amount of water in contact with the stockpile in these wetlands. The channelization project included straightening and deepening portions of the stream and required cutting through bedrock at several locations (Figure 1-1).

Limiting Infiltration into Stockpiles, Capping

Once the amount of surface and groundwater entering the piles had been minimized, the next step was to limit the volume of precipitation entering the stockpiles. Minnesota's mineland reclamation rules require that the tops and benches of rock stockpiles that do not produce water quality problems be covered with 2 ft of soil and vegetated. When water quality problems exist, additional steps to limit infiltration must be taken. Traditionally, infiltration is decreased by covering the waste with several feet of clay. For this mine, the nearest clay source is over 40 miles away, and the high transportation cost to the site made this approach too expensive. As a result, a pilot test was conducted to investigate the effectiveness of various cover systems (Eger, Antonson, and Udoh 1990). These results, in addition to computer model runs using the EPA HELP model (Schroeder et al. 1984), were used to design an approach to selecting covers for each of the stockpiles based on the chemical composition of the pile and the observed water quality in the drainage.

Stockpiles that contained Duluth Complex material with less than 0.2% CuO and less than 0.05% NiO and that produced neutral drainage were covered with 2 ft of native soil, with at least 10% silt (-200 mesh), screened to minus 15 cm. Stockpiles containing more mineralized material and/or with acid drainage were covered with a barrier layer of compacted soil or a 30-mil low-density polyethylene liner (Table 2-1). Since these stockpiles were constructed prior to the realization that a water quality problem would develop, the piles were constructed to place the maximum amount of material in the minimum area. Stockpiles were generally built in 10–15 m lifts with 45º side slopes. Only the flat top portions of the stockpiles could be economically covered.

Table 2-1. Stockpile capping summary, LTV Dunka Mine

Stockpile	Area Capped (hectares)	Stockpile Classification	Cap	Costs	
				Total	$/hectare
8011	14	Waste rock	Screened soil, no barrier	445,000	32,000
8031	22	Waste rock	Screened soil, no barrier	835,700	38,000
8013	21	Waste rock	Compacted soil barrier	1,180,000	56,000
8014	18	Gabbro	Screened soil. A flexible membrane liner was placed over a mineralized portion of the pile.	963,000	54,000
8018	3.7	Gabbro	Flexible membrane liner	455,000	123,000

Constructed Treatment Wetlands

In 1988, four pilot wetland cells were built to determine the feasibility of using wetland treatment to remove metals from stockpile drainage at the Dunka Mine. Nickel removal was on the order of 90%, and the cells were capable of producing water that met water quality standards (Eger et al. 1994, 1993). Based on these results, LTV designed and constructed two wetland systems in 1992 (Eger and Melchert 1992, Frostman 1992). These systems were 4200 m^2 (W2D/3D) and 7000 m^2 (W1D) and were designed to treat water with 1–5 mg/L nickel and flows on the order of 75 L/min.

These systems were constructed with a series of pools and soil berms to reduce the hydraulic gradient and contained a substrate which was a mixture of peat and peat screenings (a waste product from the production of horticultural peat) between the berms. In 1995, one of the original wetlands (W1D) was increased by 10,000 m^2, and two new systems (Seep 1, Seep X) were built (Eger, Wagner, and Melchert 1996). These new systems also contained a series of soil berms, but every other berm in the cell had a limestone cobble base to permit underflow through the berm.

In 1997, another wetland (EM8) was designed and constructed to maintain water level at a uniform depth of 10–15 cm, better distribute flow, and maximize treatment area. Input flow is controlled by three adjustable stainless steel weirs, and adjustable stop log style weirs were built on all of the berms in the system. A summary of the size, water quality, and cost for each of the wetlands is presented in Table 2-2.

3. PERFORMANCE

Flow Reductions Related to Ditching and Infiltration Control

Since reducing the upstream watershed and limiting infiltration into the stockpile will decrease the total flow from the stockpile, the effectiveness of these steps must be considered together. Diverting upstream flow reduced the watershed contributing to flow at Seep X and EM8 by 13% and 54%, respectively. Most of the other ditching did not affect the watersheds which contributed to the major discrete seepages but reduced the overall watershed through the stockpiles. The largest watershed affected was upstream of the 8011 stockpile, in which 83 hectares were removed from the watershed.

Stockpile capping began in 1991 when 21 hectares of stockpile 8013 was covered with a cap consisting of 27 cm of compacted soil and 60 cm of soil cover (Gale 1992). The top of the stockpile was extensively contoured with a complex of ridges and ditches and four 76 cm culverts were installed to carry water off the pile. Observations and periodic measurements indicate that a substantial quantity of water is routed off the piles, particularly during spring melt (G. Halberg, LTV Steel Mining Co., Hoyt Lakes, Minn., personal communication, 1997). Flow monitoring instrumentation was installed in 1998 so that quantitative data can be collected. Since 1989, LTV has capped about 76 hectares of stockpiles containing Duluth Complex material, with capping material ranging from uncompacted soil to 30 mil polyethylene membranes.

Table 2-2. Wetland treatment summary, LTV Dunka Mine

Site	Time	Size m^2	Flow	Design Factors - Input					Performance Factors - Output					Cost ($)
				Metal Concentrations (mg/L)					Metal Concentrations (mg/L)					
			L/min	pH	Ni	Cu	Co	Zn	pH	Ni	Cu	Co	Zn	
W2D/3D	1992-97	4200	75	7.0	1.9	0.05	0.02	0.05	7.0	.08	.004	.002	.008	75,000
W1D	1992-97	7000	150	7.07	3.98	.068	.036	.052	7.18	.70	.010	.008	.013	125,000
Expanded	1995-97	17,000	130	7.14	1.20	.059	.023	.017	7.38	.18	.005	.001	.011	244,000
Seep 1	1995-97	2500	20	6.94	5.39	.15	.13	.65	7.23	1.85	.05	.04	.29	192,000
Seep X	1995-97	10,000	100	7.03	1.50	.33	.08	.48	7.13	.61	.08	.02	.21	282,000
EM8	1998 (Jan-Sept)	16,000	310	7.20	1.6	.029	.015	.059	7.01	.902	.011	.006	.033	290,000
Water Quality Standard									6.5-8.5	.213	.023	.05	.343	
													Total Cost	$1,200,000

Note: The flow and metal concentration values presented are the average values for the periods of record.

Flow measurements for most of the seeps were made from water level measurements. All seeps were instrumented with a Stevens level recorder and a 60° v-notch weir. Continuous flow measurements are generally available from May 1 to October 31. Freezing temperatures before and after these dates often makes collection of continuous data difficult. As a result, flow during spring melt is generally not recorded. Although this flow can be substantial, 74% of the annual precipitation in northeast Minnesota occurs during the months of May through October. In the past, estimates of spring and fall flow have been made (Eger, Lapakko, and Weir 1981), but for purposes of comparing the effectiveness of the stockpile capping program, only the data representing actual flow measurements have been used.

The sites with the best long-term record and the largest changes to the watershed contributing to the drainage were EM8 and W1D. In the early 1990s the watershed that contributes to EM8 was reduced by 54% by an extensive and expensive ditching program, and in 1996 and 1997 the flat portions of the stockpile were covered with native soil and vegetated. Since 2006, average volume for May through October was around 100 million L. Prior to any activity which might have dramatically altered the flow at EM8 (1977–1979), the flow for this same period was 240 million L, more than twice the post-reclamation flow.

At W1D, around 60% of the total area of the stockpile was covered with 30 mil polyethylene. Since the stockpile contributes 40% of the total area to the monitoring weir, an overall flow reduction of around 25% was expected. Data from the first two years after capping indicate that flow from May through October decreased 36%, from an average of 36 million to 23 million L.

Reductions in infiltration for the covered portions were predicted to range from 40% for native soil to over 90% for flexible membrane liners. Since the side slopes were not covered, the total reduction in flow would be a function of the ratio of the area of the top to the total area of the stockpile.

Wetland Treatment

All the wetlands at the Dunka Mine have been effective at removing metals, but some are not always effective at achieving compliance with permit limits. Annual average metal removal has ranged around 60%–90% for nickel, 70%–90% for copper, 70%–90% for cobalt, and 50%–75% for zinc (Table 2-2). The ability to achieve compliance is largely a function of the wetland's size and the hydraulic and metal loading rate. The Dunka wetlands range in size 2500–17,000 m^2, and the average nickel loading rate ranges 15–70 $mg/m^2/day$. Pilot-scale tests indicated that adequate nickel removal could be achieved with a residence time as low as two days, and that nickel removal rates were on the order of 40–80 $mg/m^2/day$ (Eger et al. 1994). With the exception of the system constructed in the fall of 1997 (EM8), the wetland systems were constructed with a series of berms and pools that ranged in depth 0.3–0.6 m. This design increases the overall residence time in the systems but decreases the area where the drainage is in direct contact with the substrate. Flow channels have developed at the low spots in the berms, further reducing contact with the substrate. The newly constructed system at EM8 was built to control the distribution of flow and to maximize contact with the substrate. This system is expected to have much less flow channelization and provide better metal removal than the older wetlands.

Although the wetland systems are generally effective at removing metals, efficiency tends to decrease as temperatures decrease in the fall. In general, stockpile flow ceases during the winter as precipitation accumulates as snow and seep flow freezes. Most of the stockpile drainage ceases in the winter, but some winter flow has been observed at site Seep X and EM8. Limited data from the outflow of the Seep X system suggest that treatment decreases substantially during winter, with outflow concentrations approaching inlet concentrations in December (Figure 3-1).

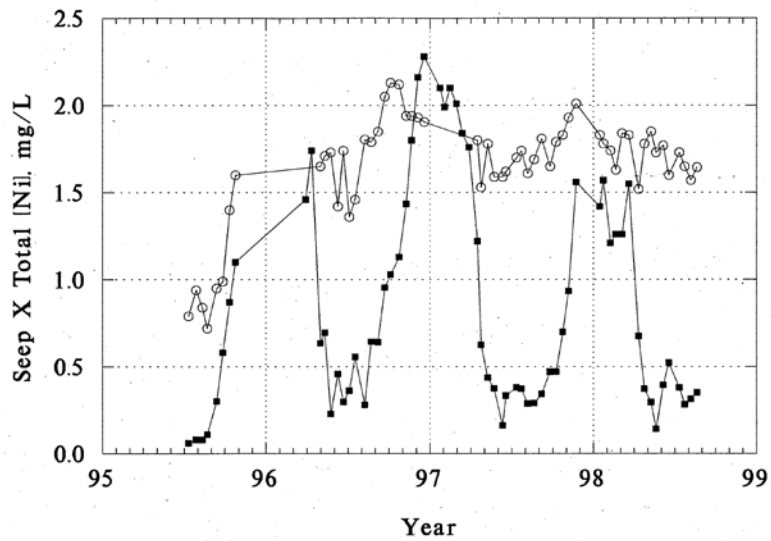

Figure 3-1. Nickel concentrations in the outflow from Seep X. Dark squares are output concentrations, and open circles are input concentrations. Concentrations increase in the fall and approach inlet concentrations in December.

Nickel removal in the test cells and the WID system also decrease in the fall (Eger et al. 1994; Eger, Wagner, and Melchert 1996). Winter operation data for other wetland systems are limited, but year-round copper removal has been reported in British Columbia (Sobolewski, Gormely, and Kistritz 1995), and metal removal was also observed throughout the winter in the Big 5 Tunnel pilot cells in Colorado (Wildeman et al. 1991). Recent work in Montana also found that metal removal occurred during winter operation (J. Pantano, Arco Mining, Anaconda, Mont., personal communication, 1997). Additional data is needed on the LTV systems to determine whether these systems can meet water quality standards throughout the year.

4. COSTS

Mine closure is an expensive undertaking. LTV has spent over $6.5 million to close the Dunka Mine. Capping the stockpiles containing Duluth Complex material was the most expensive cost at around $3.9 million. Costs ranged from $32,000/hectare for a soil cover to $123,000/hectare for flexible membrane liners. These costs generally included some reshaping of the stockpiles. Covering the stockpiles with 2 ft of native soil, as required by mineland reclamation laws, generally costs about $20,000–25,000/hectare.

A total of $1.2 million was spent to construct the wetland systems. The design of these systems has changed over time; costs have ranged from $18/m^2 for the systems built in 1992 and 1997, to $24-28/m^2 for the systems with underdrains built in 1995.

Active treatment for the southern portion of the mine (which began in 1992), cost around $1.3 million to construct and has an annual operating cost of $200,000. Maintenance costs for the passive systems generally decrease as vegetation becomes established on the stockpiles and in the wetlands, and by the fourth season, the only maintenance required was the annual inspections in the spring and fall. LTV has projected an annual maintenance cost of less than $2,000, or less than 1% of the cost of the actual plant. With an annual savings of $198,000, LTV will recover the cost of the wetland treatment systems in about six to seven years.

5. REGULATORY CHALLENGES

Although all the wetland treatment systems removed metals, removal varied substantially between systems and among metals and only two systems were able to meet the original water quality–based standard. As a result, regulatory flexibility was needed to allow passive treatment to continue.

Standards were raised to include the hardness of the discharges, and then variances were granted for two of the sites. The basic requirement was that the discharge could not be acutely toxic. In order to ensure that the variance was not causing impacts to the receiving stream, the company was required to increase its biological monitoring of the receiving stream.

Water Quality Standards

In Minnesota, all effluents must be nontoxic. Therefore at a minimum, metals concentrations must be below the Final Acute Values (FAV, Minnesota Rules, Chap. 7050). Since metal toxicity decreases as hardness increases, permissible metal concentrations generally increase with hardness up to a maximum allowable hardness of 400 mg/L as $CaCO_3$. For the metals at Dunka, the final acute values range from 126 µg/L for copper to 9136 µg/L for nickel (Table 5-1). Due to limited toxicity data on cobalt, no provision for hardness is included in the rules. LTV conducted a site-specific bioassay study, which demonstrated that cobalt toxicity decreased as hardness increased.

Originally, permit standards for the mine were based on chronic toxicity values, which were up to an order of magnitude lower than acute values (Table 5-1). When the company went bankrupt several years after the mine had closed, it sought a variance for several of the discharges. The new permit based standards on FAV include biological monitoring in the receiving stream to insure that the higher concentrations were not producing any adverse impacts on the aquatic system. Since the seeps at Dunka contain a mixture of metals, the current permit assumed that the toxicity of the effluent would be equal to the sum of the toxicities from the individual metals. Therefore, to ensure that the effluent was nontoxic, the permit requires the following:

$$\sum \frac{\text{Concentration of metal}}{\text{Final acute value}} \leq c1$$

Table 5-1. Minnesota water quality standards
Concentration in µg/L; hardness in mg/L $CaCO_3$.

Trace metal	Standard	Hardness, 50	Hardness, 100	Hardness, 200	Hardness, 400
Copper	CS	6.4	9.8	15	23
	MS	9.2	18	34	63
	FAV	18	35	68	126
Nickel	CS	88	158	283	508
	MS	789	1418	2549	4568
	FAV	1578	2836	5098	9136
Zinc	CS	59	106	191	343
	MS	65	117	211	3784
	FAV	130	234	421	7567
Cobalt*	CS	2.8	2.8	2.8	2.8
	MS	436	436	436	436
	FAV	872	872	872	872

Standards: CS = chronic standard, MS = maximum standard; FAV = final acute value.
CS is defined as "the highest water concentration of a toxicant to which organisms can be exposed indefinitely without causing chronic toxicity." This is considered the ambient in stream water quality standard, which must be met on an average basis.
MS is defined as "the highest concentration of a toxicant in water to which aquatic organisms can be exposed for a brief time with zero to slight mortality. The MS equals the FAV divided by 2." This is considered the ambient in stream concentration that cannot be exceeded on any given day.
FAV is defined as "an estimate of the concentration of a pollutant corresponding to the cumulative probability of 0.05 in the distribution of all the acute toxicity values for the genera or species from the acceptable acute toxicity tests conducted on a pollutant." By rule, any wastewater discharge must not exceed these standards at end-of-pipe at any time.
*LTV conducted site-specific testing and demonstrated that cobalt toxicity was a function of hardness. The cobalt chronic value for the Dunka Mine was increased to 50 µg/L.

Additional variances were granted for the Seep 1 and Seep X systems for spring flow and the company instituted a contingency plan to avoid water quality violations. If the concentrations in the outfall exceed the variance levels, water is collected and pumped to the beginning of the W1D system. Although this procedure has been effective, it is costly and cumbersome.

6. STAKEHOLDER CHALLENGES

None reported.

7. OTHER CHALLENGES AND LESSONS LEARNED

Part of the explanation for the difference in performance of the wetland treatment systems at the Dunka Mine appears to be the size of the treatment systems. Before the systems were built, pilot tests were conducted to determine the ability of wetlands to treat the stockpile drainage and to develop criteria to design full scale systems (Eger and Lapakko 1989; Eger et al. 1991, 1993, 1994; Eger, Wagner, and Melchert. 1996). Based on these tests, the average areal removal rate for nickel was about 40 mg nickel/m^2/day. Using this value and the estimated nickel loading to

the wetland, the area required to treat each seep was calculated. Area ranged from about 0.4 hectare (4100 m^2) for Seep 1 to about 3.2 hectares (32,200 m^2) for EM8 (Table 7-1).

Table 7-1. Wetland sizing

Seep	Input			Area (m^2)		
	Average flow (L/min)	Average concentration (mg/L)	Daily load (mg/d × 10^3)	Calculated*	As built	Expanded
W2D/3D						
• Original	75	2	216	5,000	4200[a]	
• After reclamation	45 est[b]	2	130	3,000		
W1D						
• Original	150	4	864	23,000	7000	17,000[c]
• After reclamation	40	0.8		1,200		
Seep 1	20	5.4	156	4100	2500	
Seep X	100	1.5	216	5000	10,000	
EM8[d]	400	2.1	1210	32,200	16,000	

* Area required. Based on an areal removal rate for nickel of 40 mg/m^2 day, which is average from test cells.

[a] This is the size of the constructed portion. There is about an additional 2,000 m^2 of natural wetland before the discharge monitoring point (SD-006).

[b] The 8031 was covered with 2 ft of local soil and revegetated. After successful reclamation, flow is reduced by about 40% (based on field data from test stockpiles and HELP model).

[c] Wetland was expanded in 1995.

[d] Flow data from 1997. Data 2001–2003: average flow 1140, which is higher than prereclamation flow data being checked.

For Seep 1, Seep X, the systems were built after the stockpile was capped in the fall of 1991, so there were no changes due to reclamation activity.

Note: Average flow is based on May–October records when continuous flow records are available.

In general, adequate treatment was achieved when the size of the wetland exceeded the calculated area. Originally the W1D system was not effective in meeting the original permit requirements, but performance improved after the system was expanded (Eger, Wagner, Melchert 1996). After the stockpile was reclaimed, the original part of the system met the criteria and currently meets the original water quality standard.

One of the most common reasons for the failure of wetland treatment systems is that the systems were too small (ITRC 2003). Wetlands can be effective, but in general, sufficient land must be available. Insufficient area was available at both Seep 1 and EM8, and the current systems are only about half of the calculated size.

Although the size of the Seep X wetland appears to be adequate for nickel removal, the effective treatment size is substantially smaller than the total size of the system. This system contains a series of soil berms that reduces the effective treatment area by about 20%. Field observations revealed that the flow is not equally divided between the two cells and that the northern portion of the wetland receives about three times as much water as the southern section.

Water is routed through the overflow berms by a series of pipes, which further tends to concentrate flow and lessen contact with the substrate. In addition, the limestone drains were not effective in transmitting flow through underflow berms. The hydraulic gradient in the system is only about 1%, and the permeability of the peat mixture is at best on the order of 10^{-3} cm/sec. Under these conditions, the drains could only transmit a few percent of the flow. Visual observation confirmed that a large volume of water was moving over the top of the underflow berms. Seep X also flows during most of the winter, and channelization increases dramatically as areas with little water movement freeze. Metal removal decreases too less than 10% during the winter, and the FAV of outfall is essentially the same as the input. Channelization has also been observed in the Seep 1 system, and despite the inclusion of level maintenance structures in EM8, flow has not been evenly distributed partially due to a lack of maintenance and adjustment of the level control structures.

In 2004 substantial renovation was performed on the Seep 1 and Seep X systems. The goal was to provide better flow distribution and increase the contact of the drainage with the peat substrate. Berms, which had subsided, were raised by adding limestone to the surface, and additional peat was added to some of the open water areas to increase the amount of reactive surface area. Maintenance work on the EM8 system was scheduled to occur during the summer of 2005.

In contrast, maintenance requirements for the W2D/3D and the W1D expanded system have been minimal. The systems were built with sufficient area to treat the original flow, and water quality and reclamation activities have further reduced the input load so that both systems are estimated to be have lifetimes well in excess of 100 years. Based on a model that calculates the addition of new removal sites, it appears that for the W1D system, there are enough new removal sites generated each year to remove the annual nickel load from the stockpile (Eger and Wagner 2002). As a result, the wetland can theoretically remove nickel in perpetuity.

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Attachment 9

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FIELD DISSOLUTION OF TEST PILES OF DULUTH COMPLEX ROCK

Report to the US Bureau of Mines
Salt Lake City Research Center

Minnesota Department of Natural Resources
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September 1993

FIELD DISSOLUTION OF

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INTRODUCTION

In 1974, AMAX Exploration Inc. reached an agreement with Bear Creek Mining Company and assumed their leases for properties located in T60N, R12W, about four miles southeast of Babbitt, Minnesota. The state owns the land and mineral rights on much of the property, and AMAX requested permission to construct an exploratory shaft. In 1974, very little information existed on the potential drainage problems associated with copper-nickel development in Minnesota. The state and AMAX agreed to commence field leaching tests on Duluth Complex rock.

In 1975 shaft development began, and in 1977 AMAX constructed six test stockpiles containing lean ore material. Water quality samples were collected periodically from each pile in 1977 and the data were compiled by AMAX (1978). The test pile studies were conducted from 1978 through the present by the Minnesota Department of Natural Resources (MNDNR), Division of Minerals. Kennecott Minerals Company assumed the leases in 1981 and terminated them in 1989.

This report presents data on the quantity and quality of drainage from the test piles. It is the third in a series of reports on dissolution of Duluth Complex rock. The two previous reports describe laboratory experiments on the dissolution of drill core and blast hole samples of Duluth Complex rock (Lapakko, 1993a; Lapakko, 1993b). The results from the laboratory and field studies will be used by the Salt Lake City Research Center to aid in development of a mathematical dissolution model and for comparison with laboratory data generated at the Center.

OBJECTIVES

As presented in Cooperative Agreement No. C0219003, the objectives of this report on the drainage quality from field test piles of Duluth Complex rock are to:

a) describe the variation of drainage quality with respect to time;
b) determine the chemical mass release of sulfate, calcium, and magnesium;
c) describe the variation of drainage quality and mass release with respect to flow;
d) describe the variation of drainage quality and mass release with respect to sulfur content;
e) determine the buffering capacities, or neutralization potentials, of the Duluth Complex rock samples examined; and
f) compare field results with those generated in the laboratory.

METHODS

Climate

Annual precipitation in the study area averages 72.1 cm (Hickok and Associates, 1977), and the average watershed runoff is 26.2 cm/yr (Siegel and Ericson, 1980). Temperatures are extreme, averaging -14°C in January and 19.1°C in July, with an annual mean of 3.6°C; the ground is covered with snow for an average of 140 days per year (Hickok and Associates, 1977).

Test Pile Construction

Materials

Six test stockpiles containing 820 to 1300 metric tons of low grade copper-nickel material were constructed in 1977 (figure 1). Stockpiles 1 to 4 were constructed of rock mined during the shaft sinking operation, while piles 5 and 6 contain material from underground drifts. The chemical compositional ranges for the piles are 0.30 to 0.35 percent Cu, 0.083 to 0.085 percent Ni, and 0.63 to 1.41 percent S (table 1). A grab sample was taken from test pile FL1 for more extensive chemical (table 2) and mineralogical analyses. The major sulfide minerals present in the gabbro are pyrrhotite, chalcopyrite, cubanite, and pentlandite, but over 90 percent of the rock is composed of silicate minerals. The major silicate minerals are plagioclase, pyroxenes, and olivine (table 3).

Approximately 40 percent of test pile FL4 was removed in 1982, and a representative sample, weighing over three tons, was collected for analysis of particle size distribution, specific surface area, chemistry, and mineralogy. Particle size distribution was determined by manual measurement, box screen, and a Sweco[1] separator. Boulders were measured with a tape measure and removed manually. The remaining material was raked through screens (0.9 by 1.2m) which were sloped about 20 percent, and had openings of 152, 76.2 and 25.4 mm. The material finer than 25.4 mm was then sieved with a Sweco Vibro-Energy Separator, with screen openings of 19.0, 12.7, 6.40, 2.00, and 1.41 mm. A representative sample of the less than 2.00 mm (10 mesh) material was then dry sieved on a Ro-Tap Sieve Shaker. Masses of 500 to 600 g were sieved for 20 minutes with screen openings of 0.50, 0.177, 0.149, 0.105, 0.074, and 0.053 mm. Finally, representative samples of each of the seven finest fractions were submitted to the Minnesota Department of Transportation (DOT) for a standard dry sieve/hydrometer analysis. Sieve openings of 0.84, 0.42, 0.250, 0.149, 0.105, and 0.074 mm were used. Hydrometer analysis was conducted on the less than 0.074 mm fraction to separate the less than 0.053 mm fraction. The particle size distribution data presented in table 4 are based on the unmodified coarse material separation plus the fine material dry-sieving as modified by the DOT results.

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[1] Reference to specific trade names does not imply endorsement by the MNDNR.

The sand, silt, and clay fractions were analyzed for specific surface area, chemistry, and mineralogy; the coarse fraction was excluded because its specific surface area is relatively small, and is therefore less influential in chemical reactions. The values determined represent both the Duluth Complex solids and precipitates which accumulated during leaching in the stockpile. The specific surface area of the fine fractions was roughly inversely proportional to the particle size diameter (table 5). The pH decreased and the sulfide and trace metal content increased with decreasing particle size of the gabbro material (table 5). The finer sand, silt, and clay fractions had S, Cu, and Ni contents above the 0.634 pct S, 0.348 pct Cu, and 0.083 pct Ni levels estimated for the FL4 stockpile as a whole.

X-ray diffraction (XRD) analyses were conducted on the 0.053 to 0.075 mm, the 0.075 to 0.105 mm, and the 0.105 to 0.149 mm size fractions of the rock. Approximately half of this material was plagioclase (with a Na:Ca ratio of about 2:3) and half a mixture of olivine (with a Fe:Mg ratio of 1:4 to 1:9), pyroxene, and biotite. Lesser amounts of magnetite, amphibole, chlorite, and possibly smectite and/or vermiculite were also identified. The phyllosilicate minerals (biotite, chlorite, smectite, vermiculite) were more common in the finer size fractions.

Reclamation Treatments

Three of the piles were left uncovered, as controls, and the remaining three piles were covered (May 1978) with 18 to 29 cm of soil obtained from a nearby borrow pit. Pile 2 was covered with fertile topsoil obtained from the top 30 cm of the borrow pit, while piles 3 and 5 were covered by infertile coarse sand. In 1980, an additional 30 cm of sandy till was added to pile 5.

In June of 1978, all covered piles were fertilized with 10-20-30 fertilizer (392 kg/ha), broadcast seeded, and mulched with straw (3.4 m/ha). Minnesota Highway #5, United States Forest Service wildlife mix, and native seed mixes were applied at rates of 34 and 67 kg/ha. In 1979, an additional plot containing jack pine, red pine, trembling aspen, and an upland willow was established on each of the covered piles. Additional seed was also applied to piles 3 and 5. When the sandy till was applied to pile 5 in 1980, the entire pile was reseeded with a mixture of alfalfa, U.S. Forest Service wildlife mix, and sweet clover at rates of 45, 67, and 113 kg/ha, respectively. A 13-12-12 fertilizer (730 kg/ha) was added to piles 2, 3, and 5 at this time. Additional details on vegetation, fertilization, as well as the application of lime-stabilized sludge to pile FL3 (1980, 1982) and a sodium-lauryl-based bactericide to FL5 (1982), are available in Eger and Lapakko (1985) and from the MNDNR.

In 1982 approximately 40 percent of test pile FL4 was removed for use in another study of stockpile reclamation techniques (Eger et al., 1984). Piles FL1, FL3, and FL6 were partially covered by plastic at the end of the 1989 field season, for the practical purpose of reducing the volume of drainage which the MNDNR was required to treat. The plastic covers were shredded during 1990 by exposure to the elements. A more robust Hypalon cover was placed onto FL3, thereby terminating flow from this pile at the end of the 1989 field season.

Water Sampling and Analysis

Volumes of input precipitation were measured with a standard rain gauge (Science Associates, Inc. NWS Spec. 450.230). Time and duration of rain events were obtained using a recording rain gauge (Science Associates, Inc., NWS Spec. 450.220) through 1984.

For drainage collection, each pile is underlain by an impervious Hypalon liner (30 mil) which is sloped toward a 15.2 cm perforated plastic pipe (figure 2). The drainage collected from the piles flows to a common sump. A limited number of water quality samples were collected in 1977, but no flow measurements were made. Since methods in this year deviated from those over the remainder of the study and the data were sparse, the results are not addressed in the text. During 1978, each pile was fitted with a cumulative flow meter, event recorder, and a flow-weighted-composite sampler (figure 3). For each pump discharge, a fixed volume of sample was placed into the compositing container. (The timing of flow was more precisely registered on a Rustrak event recorder (Model 292-4) from 1978 through 1984.)

The composite samples were analyzed weekly through 1990 and bi-weekly subsequently. Samples were filtered through 0.45-micron filters to remove suspended materials prior to analysis. Routine analyses included pH, alkalinity, specific conductance, sulfate, copper, nickel, cobalt, zinc, calcium, and magnesium, although calcium and magnesium were not analyzed in 1983. (Some samples were also analyzed for sodium, potassium, chloride, and dissolved organic carbon.) Specific conductance was analyzed using a Myron L conductivity meter. pH was determined using a Radiometer 29 pH meter from 1978 to 1988, and subsequently using an Orion SA 720 with a Ross combination electrode (8165). Sulfate was analyzed using either the barium sulfate turbidimetric technique (APHA et al., 1975) or occasionally, for low concentrations, a Technicon autoanalyzer. Metals were analyzed with a Perkin Elmer 603 atomic absorption spectrophotometer.

Calculations

From the weekly and bi-weekly analyses of composite samples, monthly-flow-weighted-mean concentrations were calculated. This concentration and the volume of drainage which occurred during the period were assigned to the date in the middle of the period. The mass of sulfate, calcium, and magnesium released during the period was calculated as the product of the flow-weighted-mean concentration and the drainage volume. Missing concentrations were estimated for the mass release calculation as the time-weighted mean of the previous and subsequent concentrations. Missing values occurring at the beginning (or end) of the year were estimated as the first (or last) measured concentration in that year.

Annual mass release was calculated as the sum of the mass release values for the individual periods for each year. Annual-flow-weighted-mean concentrations were calculated by dividing the annual mass release by the annual flow. The annual rates of release were calculated as the annual mass release divided by the product of the number of days of flow

and the mass of the pile (metric tons). Average rates of release for the entire period of record were calculated by dividing the total mass release by the number of days of flow.

Neutralization potentials (NP) were calculated to determine the amount of acid neutralized by the solids prior to the drainage pH decreasing and, for the duration of the period of record, remaining below pH values of 7, 6, 5, 4.5, 4, and 3.5. The acid neutralized was calculated by two methods: as the a) cumulative sulfate release (expressed as mg $CaCO_3$/g rock) and b) cumulative calcium and magnesium release (expressed as mg $CaCO_3$/g rock) prior to the drainage pH decreasing and remaining below the specified pH value. If the drainage pH from a given solid never decreased permanently below a specified value, the NP was reported as "greater than" the total sulfate release for the period of record. Additional detail on calculation of mass release and neutralization potential are presented in appendix 1.

RESULTS AND DISCUSSION

The generation of acidic drainage is the major water quality concern associated with mine wastes. Acid is produced as a result of the oxidation of iron sulfide minerals. The rate of this oxidation, and the attendant acid production, is proportional to the iron sulfide surface area available for reaction. The solids examined in this study were mined from a shaft and drifts in Duluth Complex rock. It is assumed the iron sulfide surface area is roughly proportional to the solid-phase-sulfur content. This assumption is predicated on the concept that the size distribution of rocks in the various piles is similar, and the grain size and degree of liberation of sulfide minerals does not vary greatly among the piles. The non-sulfide mineralogy of the rock was also assumed to be fairly uniform, although some exceptions most likely occurred. This suggests that the acid-neutralizing ability of the solids was somewhat uniform.

The test piles typically produced drainage between the middle of March and the middle of November, with an average flow season of 245 days. As was the case in laboratory studies, and consistent with the aforementioned assumptions, drainage pH decreased as the solid-phase sulfur content increased. The variation of drainage quality and chemical mass release with time, solid-phase sulfur content, and drainage volume is presented first. This presentation, in conjunction with the NP values determined for the test piles, provides the background necessary for comparison of the field results with those from the laboratory. The period of record for test pile 4 was only five years, as compared with 12 to 14 years for the remaining five piles. Although tabular and graphical data for this pile are presented, they are not discussed due to the short period of record.

Drainage Quality

Annual median pH and annual-flow-weighted-mean concentrations of sulfate, calcium, and magnesium are presented to describe the variation of drainage quality with respect to time,

solid-phase-sulfur content, and annual drainage volume. These values are presented in table 6. Detail on drainage quality variation over shorter periods is presented in appendix 2.

The pH of drainage from the test piles generally decreased over time, and decreased as the solid phase sulfur content increased (figure 4). At the beginning of 1978, the pH of drainage from all piles was between 7 and 8. For the three piles containing 0.63-percent sulfur, drainage pH decreased gradually from circumneutral to the range of 4.8 to 5.3. The pH of drainage from the piles containing 0.79 and 1.4 percent sulfur decreased and plateaued within a range of about 0.2 to 0.3 pH units. The pH of drainage from the 0.79-percent sulfur pile decreased until 1982 and subsequently oscillated in the typical range of 4.0 to 4.3. The pH of drainage from the 1.41-percent sulfur pile decreased to a lower level, from a median value of 5.5 in 1978 to a typical range of 3.4 to 3.6 from 1979 to 1991. The flow-weighted-median values actually increased from 1982 (3.40) to 1991 (3.60).

In general, annual-flow-weighted-mean sulfate concentrations for the individual piles increased during the first three to four years and then oscillated within a fairly constant range; increased as the solid-phase-sulfur content increased (figure 5); and were independent of flow (appendix 2). Some deviations from the general trends were observed. Most notably, sulfate concentrations in drainage from the 1.41-percent-sulfur pile increased to peak levels between 1981 and 1985, and then decreased. From 1987 to 1991, the flow-weighted-mean sulfate concentrations from this pile were comparable to, or lower than, those from the 0.79-percent-sulfur pile. Sulfate concentrations from the 0.79-percent sulfur pile increased over time (figure 5). However, these temporal variations from 1981 through 1991 were relatively small. In general the maximum annual-flow-weighted-mean sulfate concentrations were typically within a factor of two of the minimum value observed during this period.

Annual-flow-weighted-mean magnesium concentration variations over time were generally parallel to those observed for sulfate (figure 6), and were independent of flow volume (appendix 2). With respect to solid-phase-sulfur concentration, annual-flow-weighted-mean magnesium concentrations were lowest for the 0.63-percent-sulfur piles and in a similar range for the two piles of higher sulfur content. For the individual piles, annual-flow-weighted-mean magnesium concentrations tended to decrease as the annual median pH decreased.

Annual-flow-weighted-mean calcium concentrations were typically fairly constant, ranging from about 4 to 7 millimoles per liter (figure 7). As implied by the constant range, there was little dependence of annual-flow-weighted-mean calcium concentrations on time of dissolution, sulfur content, or flow. The values for the 1.41-percent-sulfur pile appeared to parallel the decreasing trend observed for sulfate concentrations from 1985 to 1991. As a result of the temporal parallelism of magnesium and sulfate concentrations, the sum of calcium and magnesium concentrations tended to parallel sulfate concentrations over time (figure 8).

Annual Rates of Release

The annual rate of sulfate release from each pile oscillated within a fairly constant range (tables 7-12). In general, annual sulfate release increased with the solid-phase-sulfur content of the pile, although the rates for the 0.79-percent-sulfur pile were comparable to or exceeded those for the 1.41-percent-sulfur pile from 1984 to 1991 (figure 9). Sulfate release also increased with annual flow (appendix 4), and temporal variations in the annual rate were largely due to variations in annual flow. A graphic example of the flow dependence is the low sulfate-release rate from the 0.79-percent-sulfur pile in 1990, during which the pile was partially covered with plastic. The drainage volume and sulfate mass release were consequently low.

Variations in magnesium release followed similar trends. Similar to the sulfate-release rates from 1984 through 1991, magnesium release from the 0.79-percent-sulfur pile was usually higher than that from the 1.41-percent-sulfur pile (figure 10). In contrast, the annual rates of calcium release typically fell into the range of one to three moles per ton per day (figure 11). They did not vary with time or solid-phase-sulfur content, but did increase linearly with annual flow (appendix 4). The average rates of sulfate, calcium, and magnesium release for the period of record are presented in table 13.

Neutralization Potential

The NP values calculated based on the sum of calcium and magnesium release ranged from 55 to 90 percent of those calculated based on sulfate release, which was deemed excessive. The combined copper and nickel sulfide content comprised 15 to 35 percent of the total sulfide mineral content in the test piles, a considerably higher fraction than for the laboratory samples. It was concluded that a significant amount of the sulfate release from the test piles was due to oxidation of trace metal sulfides (as opposed to iron sulfides). Since this oxidation does not necessarily lead to acid production, the sulfate release over-estimates acid production. Consequently, the neutralization potential data presented are based on the sum of calcium and magnesium release.

The pH of drainages from the 0.63-percent rock did not decrease permanently below pH 5.0, although the pH of drainage from pile 3 appeared to be near this point at the end of 1989. The approximate ranges of NP values determined for endpoint pH values of 7.0 (NP_{pH7}) and 6.0 (NP_{pH6})were 0.2 to 0.4 mg $CaCO_3$/g rock and 0.4 to 1.3 mg $CaCO_3$/g rock, respectively (table 14). The 1.3 mg $CaCO_3$/g rock value determined for pile 3 is assumed to be near the NP_{pH5} for this pile.

The lowest NP endpoint pH values for the piles with sulfur contents of 0.79 and 1.41 were pH 4.5 and pH 4.0, respectively. The NP_{pH7} and NP_{pH6} values for these piles were lower than those from the lower-sulfur rock (table 14). Furthermore, the NP values for the 1.41-percent rock were lower than those for the 0.79-percent rock. This trend of decreasing NP with increasing sulfur content was also observed with laboratory samples.

Comparison of Field and Laboratory Data

Certain similarities were apparent between the field data and laboratory data (table 15).

1) Drainage pH generally decreased, or decreased and plateaued, as the time of dissolution increased. A subtle variation on this trend was a possible slight increase in the pH of drainage from the 1.41-percent-sulfur pile from 1982 to 1991 (median annual pH values of 3.40 and 3.60, respectively).

2) In both laboratory and field drainage pH values decreased as solid-phase-sulfur content increased.

3) The rates of sulfate release were fairly constant over time. The ratio of maximum-sulfate-release rate to minimum-sulfate-release rate for a laboratory sample was typically less than 3:1. This ratio was observed to increase if drainage pH decreased below about 4. The corresponding ratios for the test piles ranged from 2.7:1 to 3.9:1.

4) In both laboratory and field tests the rate of sulfate release increased with solid-phase-sulfur content. Sulfate release rates from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

5) Rates of calcium release were relatively constant over time, although it must be noted that laboratory data on calcium release are limited.

6) Rates of calcium release were relatively constant with respect to sulfur content. The rates of calcium release from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

7) Rates of magnesium release increased as time increased, although it must be noted that laboratory data on magnesium release are limited.

8) Rates of magnesium release increased as solid-phase-sulfur content increased. The rates of magnesium release from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

9) Neutralization potential values were low and decreased with increasing sulfur content.

Field and laboratory data can also be quantitatively compared based on drainage pH, neutralization potentials, and rates of release. The minimum drainage pH values observed for the 0.63-percent-sulfur rock in the field ranged from 4.71 to 4.96, and were reasonably consistent with the laboratory relationship between drainage pH and solid-phase-sulfur content. However, the minimum pH of drainages from the test piles with sulfur contents of 0.79 and 1.41 percent sulfur were about one unit lower than expected based on laboratory

data (figure 12). (The minimum pH values observed were typically isolated excursions, and recurrent low pH values were typically 0.1 to 0.2 units higher.)

The discrepancy between laboratory and field drainage pH data suggests that the laboratory tests either inhibited the rate of acid production or enhanced the rate of acid neutralization relative to the field tests. Drainage pH in the field may have been lower due to a higher iron sulfide surface area than indicated by the sulfur content alone. Twelve percent of the rock in the piles was smaller than two millimeters in diameter (table 4). The specific surface areas of this material were in the range of 0.6 to 4.7 m^2/g (table 5). This was considerably higher than the 0.061 to 0.239 m^2/g and 0.020 to 0.039 m^2/g ranges estimated for the gravel and boulder/cobble fractions. The sulfur content of the smaller size fractions ranged from 0.67 to 1.94 percent (table 5), as compared to the bulk sulfur content of 0.63. Therefore, the fraction of the total available surface area contributed by the small particles is quite high relative to the larger particles. Due to their higher sulfur content, a greater fraction of the surface area is composed of iron sulfides. Thus, the fraction of the rock comprised of iron sulfide was most likely higher than indicated by the sulfur content of 0.63 percent. In contrast, the solids in the laboratory study were in the size range $0.053 <$ particle diameter ≤ 0.149 mm, and the potential of preferentially concentrating sulfides in a smaller size fraction was limited.

The acid neutralization may have been enhanced in the laboratory due to the consistently small particle size. As is the case with sulfide mineral oxidation, the dissolution of silicate minerals is a surface reaction. If the specific surface area of silicate minerals present in the laboratory samples exceeded that in the test piles, the laboratory solids would be expected to neutralize acid more efficiently.

The neutralization potentials of the test piles were lower than those for laboratory samples, which is consistent with the lower drainage pH levels observed in the field. The NP values for the 0.63-percent-sulfur rock were roughly 25 to 50 percent of those observed for laboratory samples of similar sulfur content (table 16). Although the NP_{pH6} and NP_{pH7} values from pile 3 were higher than this range, the NP_{pH5} value was 46 percent of the corresponding value for laboratory samples.

The field NP values for the 0.79-percent-sulfur rock were 13 to 54 percent of the corresponding values for laboratory samples of similar sulfur content. The ratios of field to laboratory NP were lowest for the 1.41-percent-sulfur rock, typically five percent or less than those determined in the laboratory.

The low field NP values may have been influenced by factors similar to those discussed regarding drainage pH. First, preferential flow in the field piles would reduce the amount of silicate minerals which could contribute to acid neutralization. For example, if flow through the pile contacted only half the silicate minerals, the observed neutralization potential of the rock would be half the actual value for the entire pile. Second, if the specific surface area of silicate minerals present in the laboratory samples exceeded that in

the test piles, the laboratory solids would be expected to neutralize acid more efficiently. Third, the ratio of available reactive surface area of sulfide minerals to that of silicate minerals present in the test pile rock may have been greater than that in the laboratory. This would be equivalent to increasing the sulfur content of rock in the laboratory. As was observed in the laboratory, neutralization potential tended to decrease as sulfur content increased.

For the entire period of record, average rates of sulfate release ranged from 2.0 to 10 millimoles per metric ton per day, and increased as sulfur content and drainage volume increased. The average rates of calcium release ranged from 2.1 to 2.4 millimoles per ton per day and were fairly constant with respect to sulfur content. Average rates of magnesium release ranged from 0.55 to 3.7 millimoles per ton per day and were lowest for the low-sulfur solids and highest for the solids containing 0.79-percent sulfur.

The rates of sulfate, calcium, and magnesium release observed for the test piles were compared to laboratory rates for the Duluth Complex blast hole samples of similar sulfur content. The laboratory rates were from the last 15 to 36 weeks of the period of record. The ratios of field rates to laboratory rates, or retardation factors, ranged from 0.053 to 0.46, with most values in the range of about 0.1 to 0.36 (table 17). This indicates that the laboratory rates were roughly three to ten times those in the field.

Several factors most likely contributed to the slower rates observed in the field. First, the rates of sulfide and silicate mineral dissolution (per unit mass rock) may be slower than those observed in the laboratory. As discussed previously with respect to drainage pH, the surface area of sulfide and silicate minerals per unit mass rock in the field most likely differs from that in the laboratory. This would affect the rates of sulfide oxidation and silicate mineral dissolution, since both are surface reactions. The reaction environment in the field, where temperatures are colder, may also retard reaction rates. Furthermore, potential lower levels of oxygen within the test piles may limit the rate of sulfide mineral oxidation.

Limited transport of reaction products in the field also may have reduced rates of release. Hydrologic variables which were not influential in the laboratory could have contributed to this transport limitation. The volume of rinse water per unit mass solids yielded approximately 300 L per ton rock per day of drainage in the laboratory, roughly three orders of magnitude greater than the yields of 0.27 to 0.38 L per ton rock per day in the field. The relatively low flow in the field may have limited transport. This is supported by the observation that release rates in the field increased with flow. Furthermore, the flow of the laboratory rinse water through the solids was fairly uniform, whereas preferential flow was probable in the field setting. Preferential flow would limit the transport of reaction products from some rock surfaces. This would essentially reduce the amount of rock in the test piles which was actually contributing to the observed chemical release.

The degree of transport in the field may also be limited by chemical precipitation of dissolution reaction products. In particular, the formation of gypsum might limit the rates

of sulfate and calcium release. Gypsum formation was not a problem in the laboratory since the high rinse water to solid ratio produced relatively low concentrations of sulfate and calcium in the drainages.

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The test piles were constructed by AMAX Exploration Inc. Since 1978, sampling and sampling maintenance has been the responsibility of several people: Bruce Johnson, Anne Jagunich, and, most recently, Cal Jokela and David Antonson. Paul Eger has supervised these activities. Jon Wagner provided much of the data input and Jim Porter was responsible for the most recent data management, as well as tabular and graphical output.

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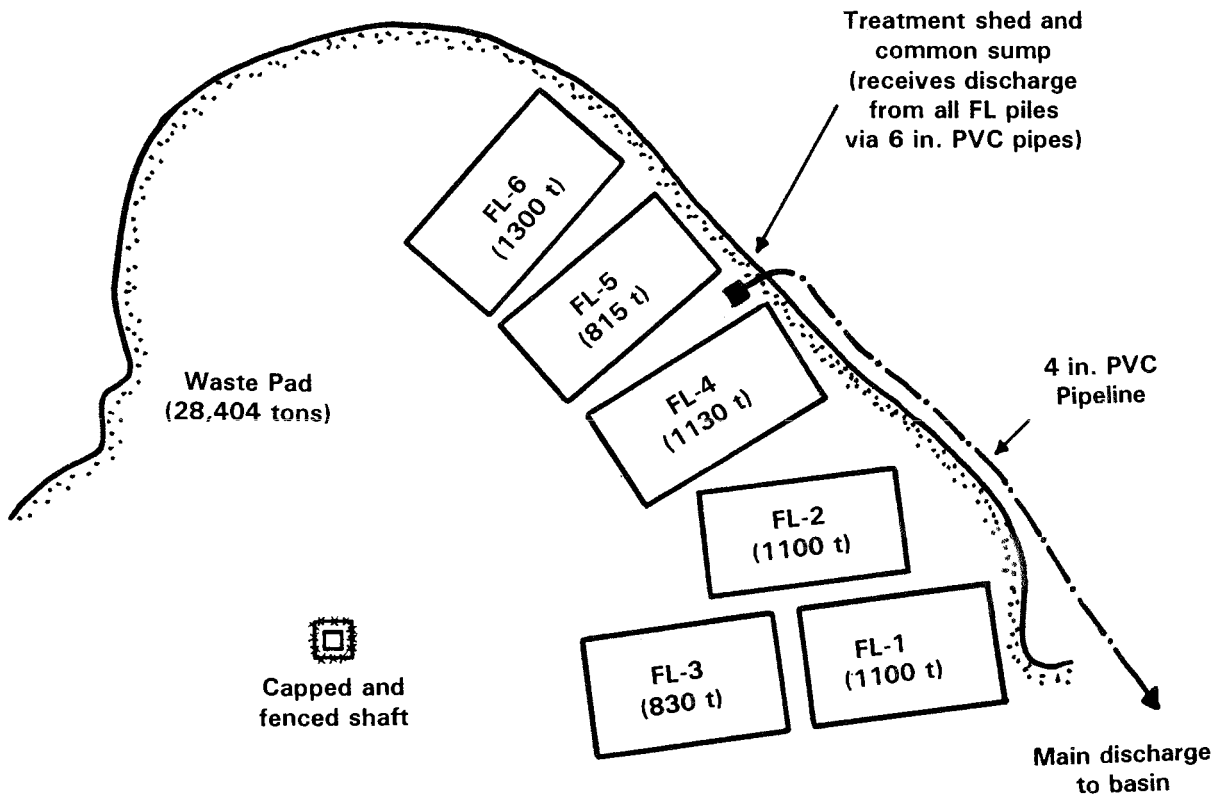

Figure 1. Test piles and drainage collection system layout.

Forty percent of FL-4 was removed in 1982 (leaving approximately 680 of the original 1130 t of material in the pile), thereby terminating the period of record. FL-3 was covered with a synthetic liner at the end of the 1989 field season, thereby terminating the periof of record. Additional information on the test piles is presented in table 1.

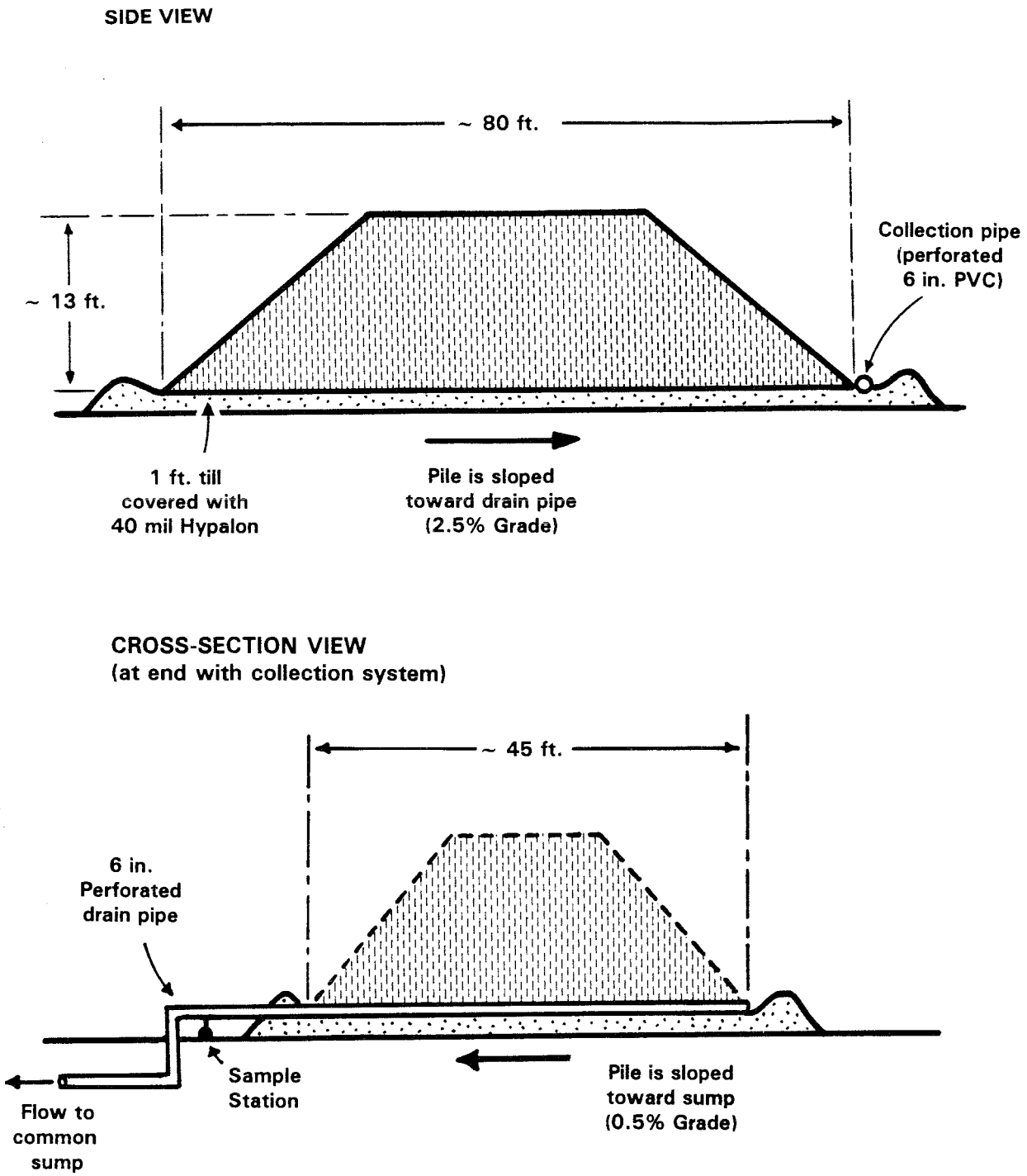

Figure 2. Leachate collection system beneath test piles (schematic; not to scale).

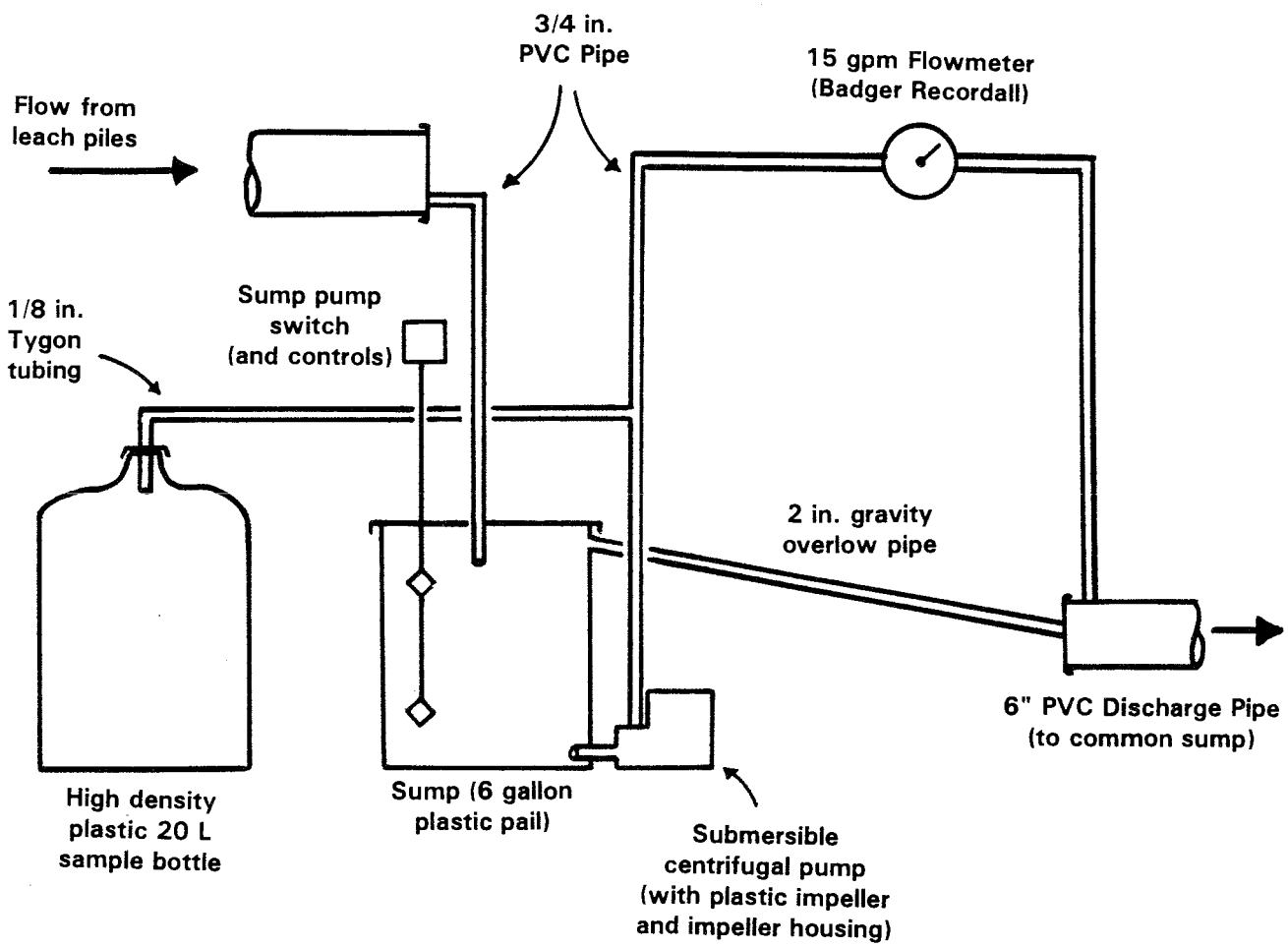

Figure 3. Flow-proportional sampler.

Figure 4. Annual median pH over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Drainage pH decreased over time and as solid-phase sulfur content increased.

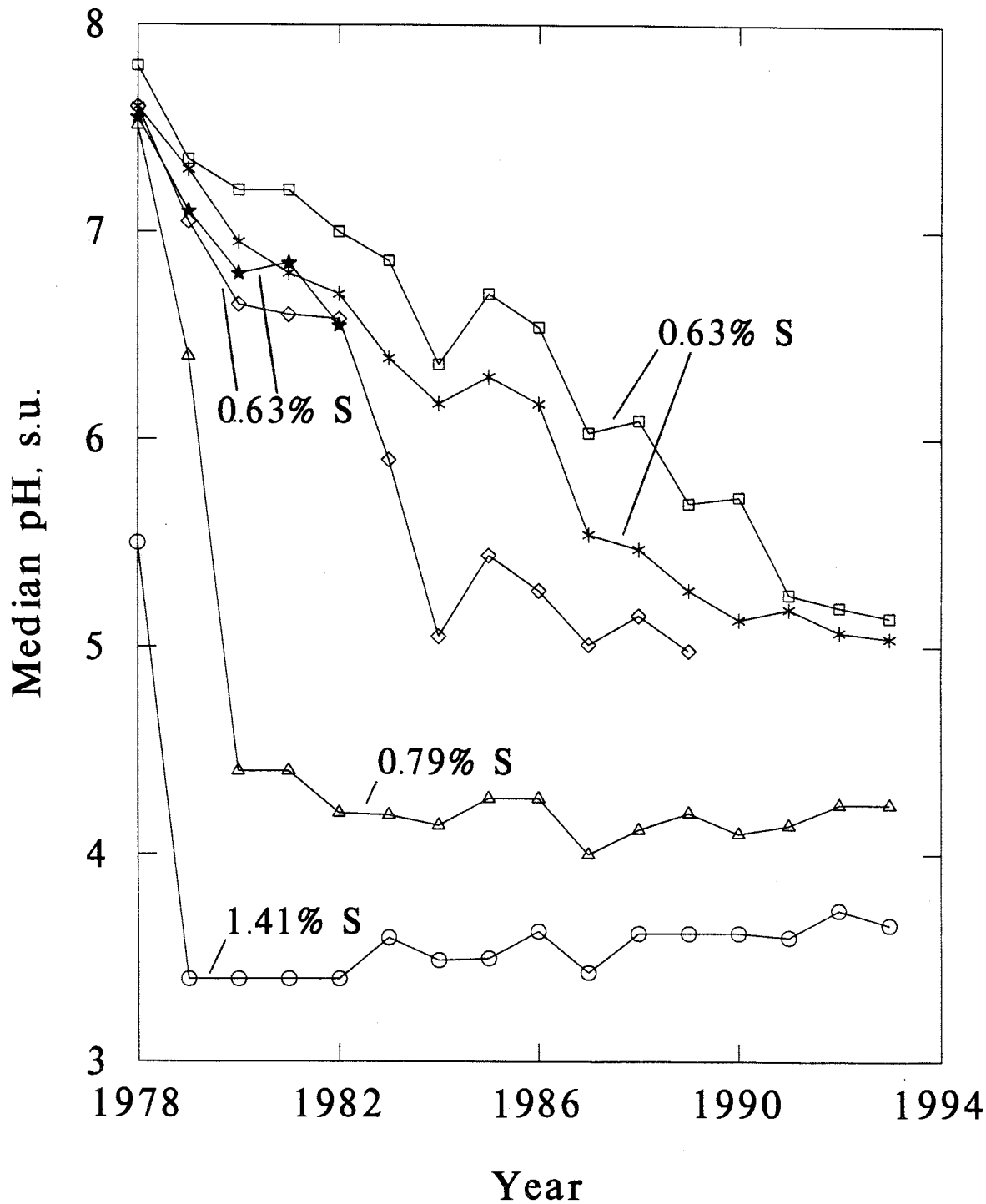

Figure 5. Annual-flow-weighted-mean sulfate concentration over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Sulfate concentrations generally increased with solid-phase sulfur content and were within a constant range over time.

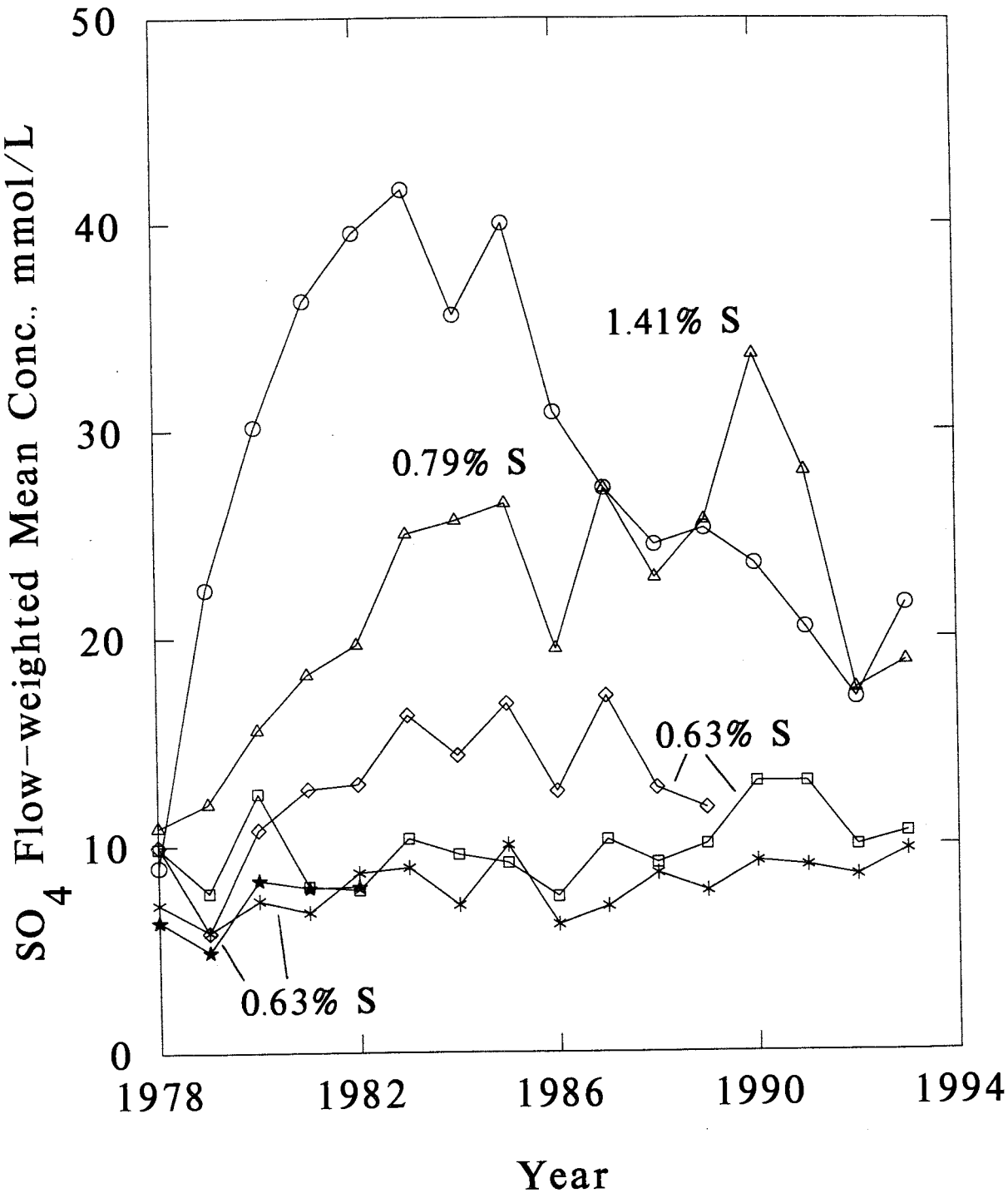

Figure 6. Annual-flow-weighted-mean magnesium concentration for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Magnesium concentrations generally increased with solid-phase sulfur content and time.

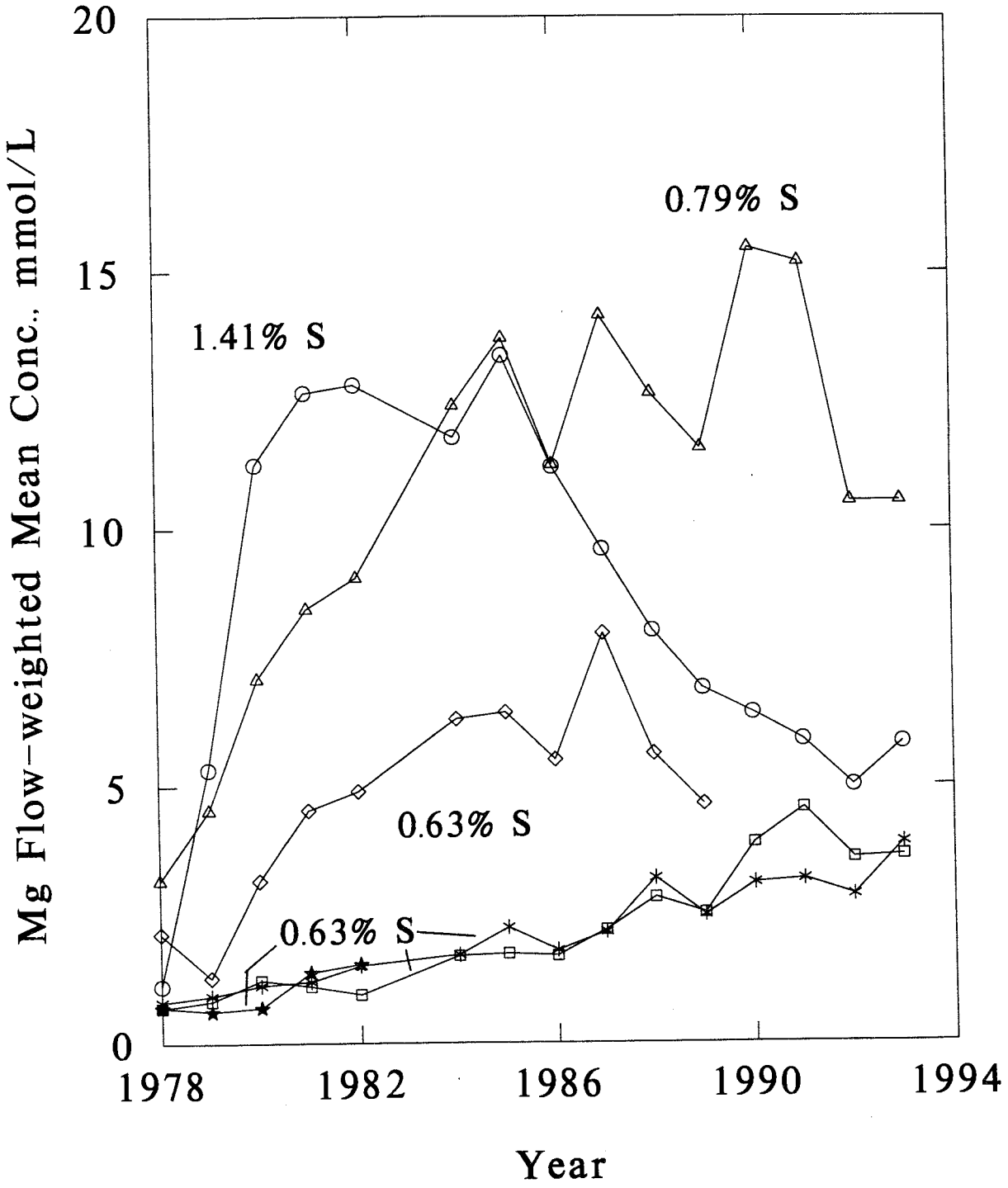

Figure 7. Annual-flow-weighted-mean calcium concentration for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (▵), and FL5 (○).

Calcium concentrations were fairly constant with respect to solid-phase sulfur content and time.

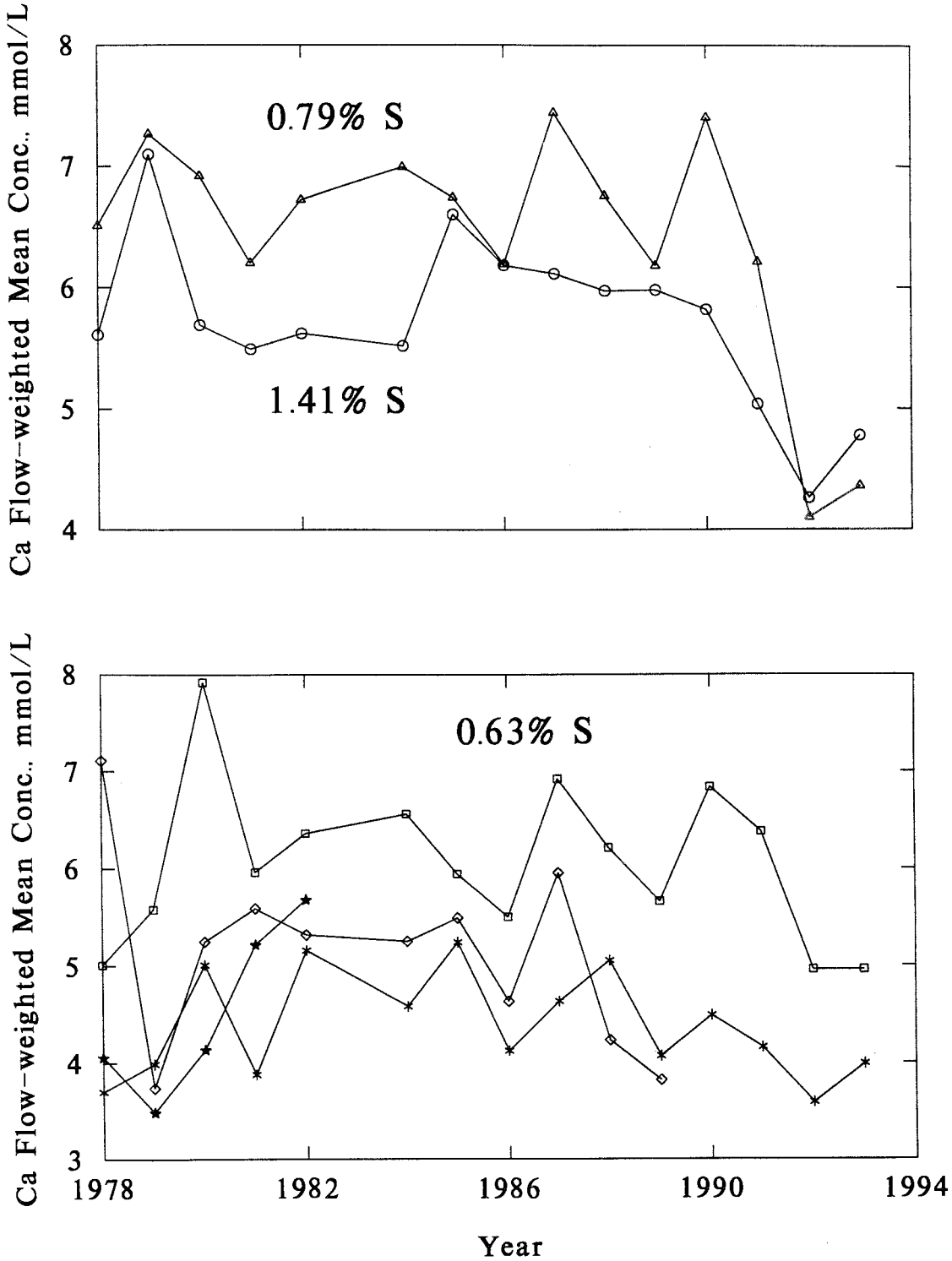

Figure 8. Annual-flow-weighted-mean calcium plus magnesium concentration for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

The sum of calcium and magnesium concentrations generally increased with solid-phase sulfur content and increased slightly over time.

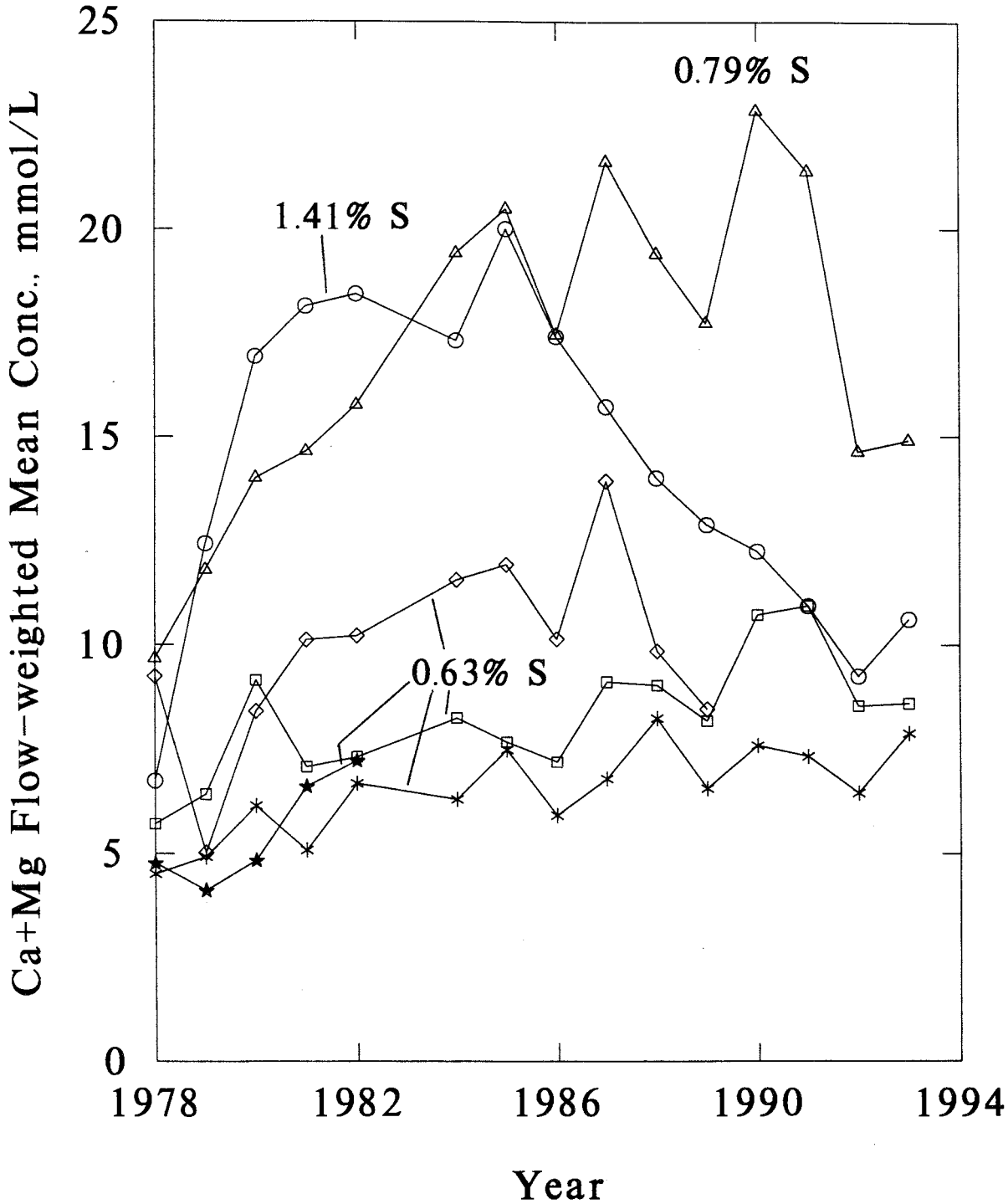

Figure 9. Annual sulfate release rate over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Sulfate release rates generally increased with solid-phase sulfur content and were within a constant range over time.

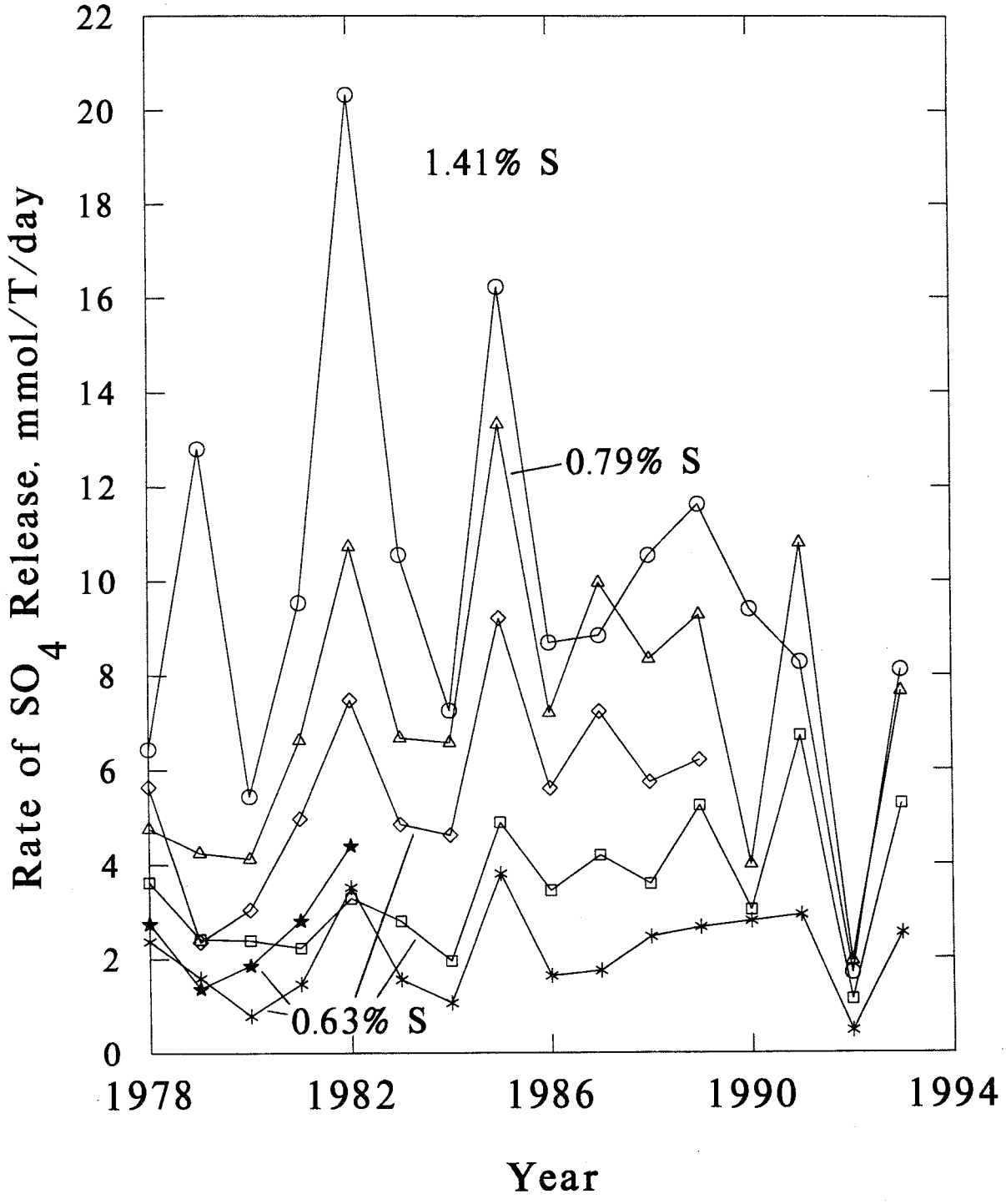

Figure 10. Annual magnesium release rate over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Magnesium release rates increase with solid-phase sulfur content and increased slightly over time.

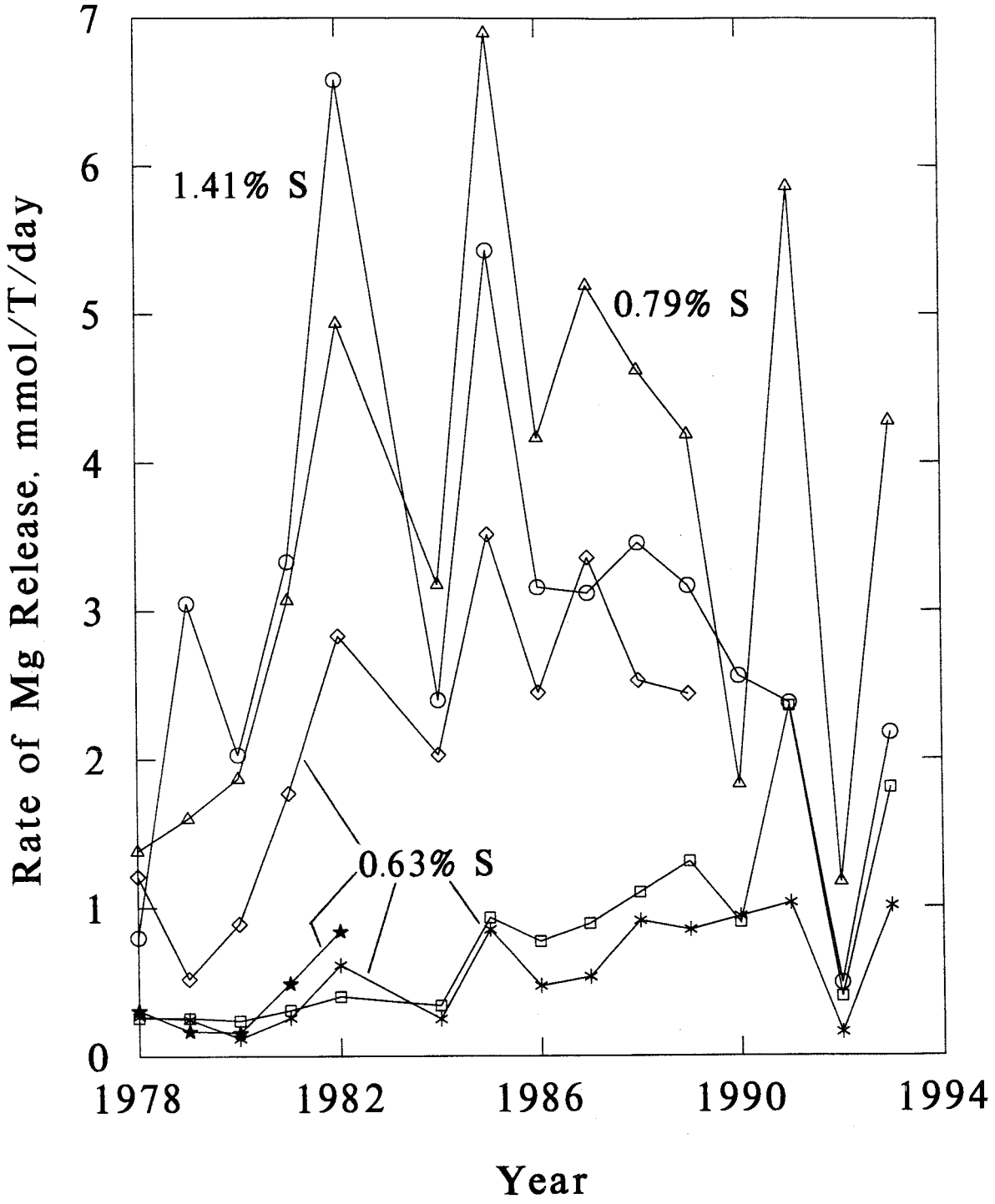

Figure 11. Annual calcium release rate over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Calcium release rates were fairly constant with respect to solid-phase sulfur content and time.

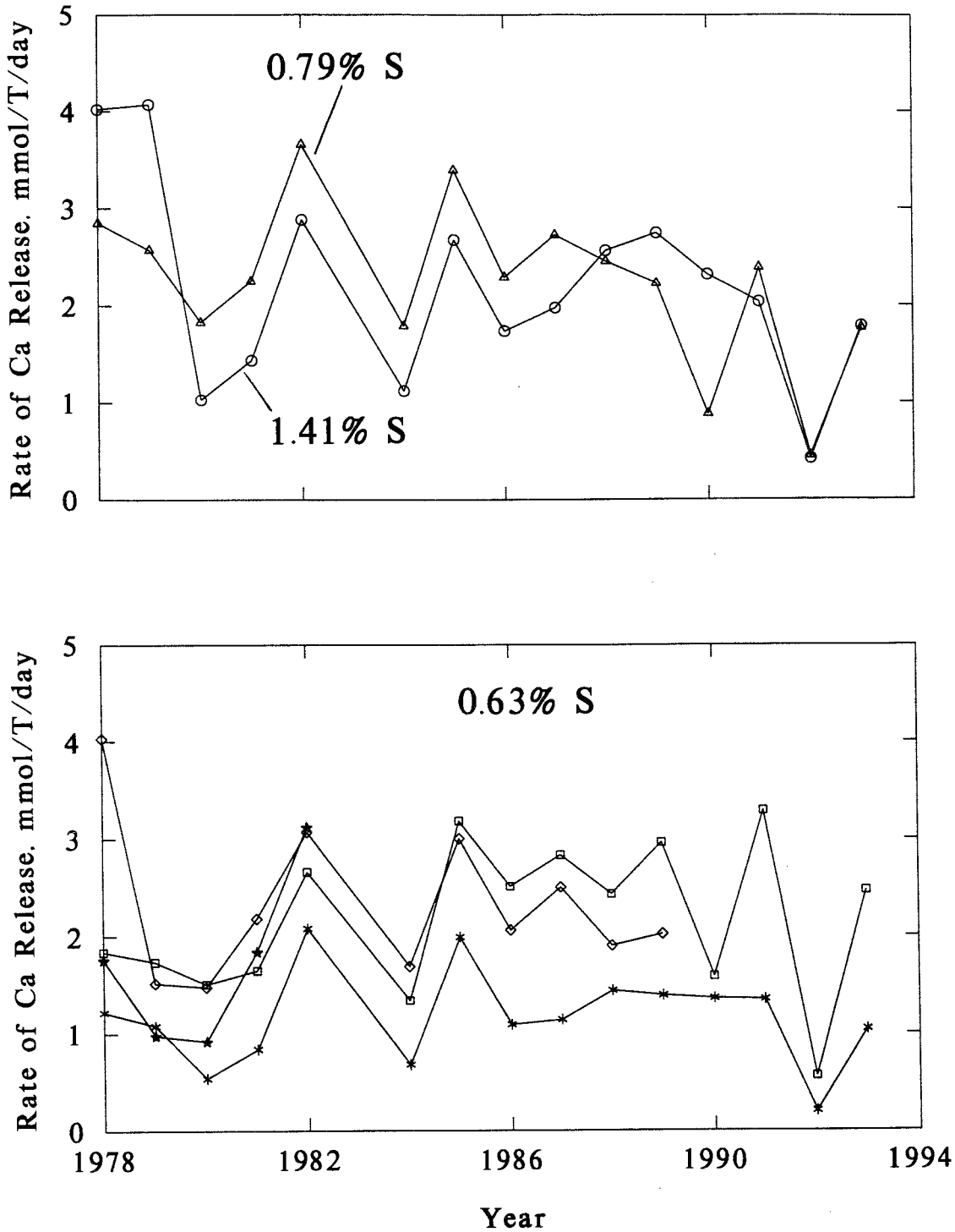

Figure 12. Comparison of minimum drainage pH in field (▲) and laboratory (○).

Field data represents overall recurrent minimum pH from the Duluth Complex test piles. Laboratory data represents drainage pH after 150 weeks of dissolution of Duluth Complex blast hole samples. The curve was generated with the log smoothing feature of the Systat statistical software package.

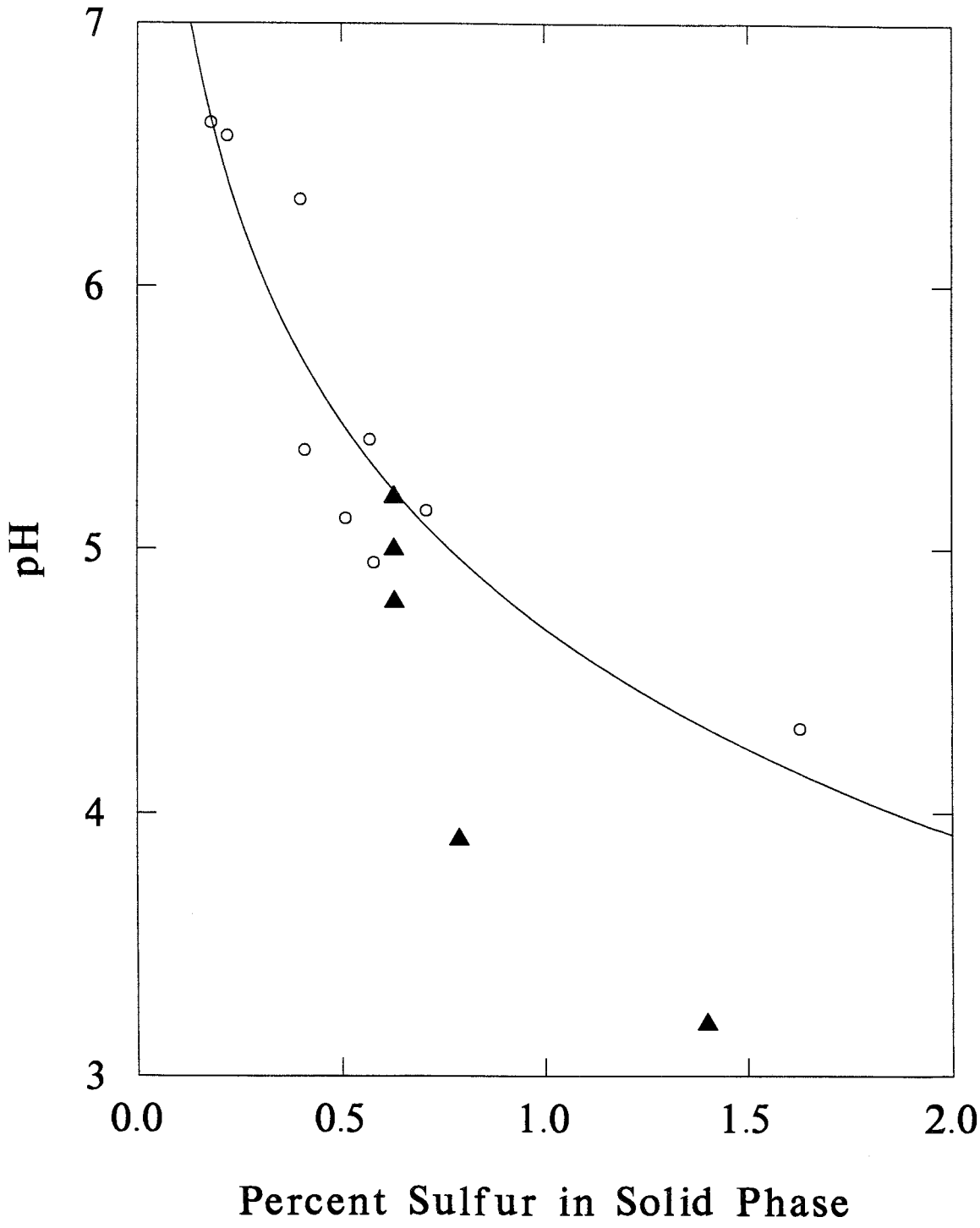

Table 1. Summary of chemical and physical characteristics of test piles.

	Pile 1	Pile 2	Pile 3	Pile 4	Pile 6	Pile 5
Date completed	4/20/77	4/20/77	4/20/77	4/20/77	9/30/77	9/10/77
% S	0.63	0.63	0.63	0.63	0.79	1.41
% Cu	0.35	0.35	0.35	0.35	0.34	0.30
% Ni	0.083	0.083	0.083	0.083	0.084	0.085
Mass, T	1100	1100	830	1130[1]	1300	815
Volume, m^3	540	530	400	830[1]	630	400
Collecting area, m^2	290	340	300	330[1]	320	340
Surface area, m^2	330	450	430	350[1]	390	370
Cover material	None	Topsoil	Glacial Till (coarse sand)	None	None	Sandy till over coarse sand
Avg. cover depth, cm	0	23	34	0	0	54
Vegetated	No	Yes	Yes	No	No	Yes

[1] Approximately 40 pct of FL4 was removed in September, 1982.

Table 2. Chemical composition of a grab sample from test pile 1.

Element	Concentration	Unit
S	0.655	Pct
Cu	0.306	Pct
Ni	0.073	Pct
Fe (S)[1]	0.696	Pct
Co	0.009	Pct
Zn	0.016	Pct
Si (SiO_2)	22.80 (48.81)	Pct
Al (Al_20_3)	8.98 (16.96)	Pct
Fe (FeO)	9.22 (11.86)	Pct
Mg (MgO)	4.31 (7.15)	Pct
Ca (CaO)	5.55 (7.76)	Pct
Na (Na_20)	2.00 (2.70)	Pct
K (K_20)	0.32 (0.39)	Pct
Ti ($Ti0_2$)	1.41 (2.35)	Pct
P (P_2O_5)	0.02 (0.05)	Pct
Mn (MnO)	0.12 (0.16)	Pct
Cr (Cr_20_3)	0.03 (0.15)	Pct
B	662.00	ppm
Ba	1173.00	ppm
Be	1.00	ppm
Sr	279.00	ppm
Pb	0	ppm
Ag	1.35	ppm
As	0	ppm
Hg	1.00	ppm
Mo	0.50	ppm
Cd	0	ppm
V	276.50	ppm
Th	6.40	ppm
Zr	80.00	ppm

[1] Iron associated with sulfur.

Table 3. Mineralogical composition of a grab sample from test pile 1 (volume percent).

Opaques[1]	5.098
Chalcopyrite-cubanite	0.769
Pentlandite	0.037
Pyrrhotite	0.844
Ilmenite-magnetite	3.447
Graphite	-
Plagioclase	59.112
Sericite	2.176
Olivine	10.510
Clinopyroxene	11.185
Orthopyroxene	3.716
Monocrystalline amphibole	3.567
Fibrous amphibole	0.288
Chlorite	1.136
Serpentine	0.257
Iddingsite	0.075
Talc	-
Biotite	1.738
Smectite	0.021
Caladonite	-
Spinel	-
Myrmekite	-
Apatite	0.085
Epidote	0.953
Allanite	-
Calcite	0.056
Quartz	-
Cordierite	0.027

[1] The value shown for opaques is the sum of the five following values.

Table 4. Particle size distribution rock from test pile 4.

	Particle diameter (mm)	Percent finer than
Boulders and cobbles	305	89.0
	152	82.2
Gravel	76.2	67.7
	25.4	38.3
	19.0	33.8
	12.7	27.1
	6.4	19.2
Sand	2.00	12.2
	1.41	10.8
	0.84	10.6
	.425	7.7
	.250	6.1
	.149	4.6
	.105	3.4
Silt and clay	.074	3.1
	.053	1.4 to 1.7[1]

Note: Boulders, cobbles, gravel, sand, and silt refer to U.S. Department of Agriculture (USDA) classification definitions.

[1] Estimated based on hydrometer analysis of the less than 0.074 mm fraction.

Table 5. Characterization of fine fractions, joint disposal bin gabbro material.

Size Fraction mm	pH	Total Sulfur %	Sulfide %	Cu %	Ni %	Co %	Zn %	Specific Surface Area m^2/g
Sand								
0.50-2.00	6.56	0.67	0.65	0.337	0.062	0.012	0.015	0.60
.177-.50	6.37	0.80	0.75	0.381	0.078	0.013	0.015	0.78
.149-.177	6.18	0.88	0.80	0.391	0.103	0.015	0.015	1.1
.105-.149	NA	1.12	1.05	0.495	0.147	0.019	0.017	1.6
.075-.105	5.95	1.37	1.30	0.585	0.221	0.022	0.019	1.7
Silt and clay								
.053-.075	5.78	1.65	1.57	0.647	0.253	0.024	0.019	2.6
<.053	5.77	1.94	1.86	0.814	0.295	0.027	0.023	4.7

NA = not available.

Table 6. Annual median pH, flow-weighted mean concentrations, and drainage volume for Duluth Complex test piles.

Pile	Sulfur %	Parameter	Flow-weighted Mean Concentration, mg/L, and Volume, 10^3 L															
			1978	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993
FL1	0.63	pH	7.80	7.35	7.20	7.20	7.00	6.86	6.36	6.70	6.54	6.03	6.09	5.69	5.72	5.25	5.19	5.14
		SO4	949	743	1202	772	755	993	920	880	723	986	879	965	1254	1253	958	1021
		Ca	201	224	318	239	255	NA	263	238	221	277	249	227	274	256	199	199
		Mg	17	20	30	27	23	NA	41	42	41	53	69	61	94	111	87	88
		Volume	83.1	86.6	53.8	83.8	118.6	82.4	66.9	133.4	119.6	104.8	106.3	122.5	60.1	121.0	27.5	125.0
FL2	0.63	pH	7.60	7.30	6.95	6.80	6.70	6.39	6.17	6.30	6.17	5.54	5.47	5.27	5.13	5.18	5.07	5.04
		SO4	689	561	707	652	834	858	685	963	593	676	829	748	884	863	818	938
		Ca	148	160	201	156	207	NA	183	210	165	186	203	163	180	167	144	160
		Mg	20	23	28	29	37	NA	41	54	43	52	78	60	75	77	69	94
		Volume	70.2	75.6	29.5	67.2	100.4	52.8	40.4	94.6	69.3	63.2	78.9	80.2	78.5	76.0	13.5	65.2
FL3	0.63	pH	7.60	7.05	6.65	6.60	6.58	5.90	5.05	5.44	5.27	5.01	5.15	4.98	NA	NA	NA	NA
		SO4	957	557	1035	1223	1244	1566	1378	1620	1210	1653	1225	1132	NA	NA	NA	NA
		Ca	285	150	211	224	213	NA	211	220	186	238	170	153	NA	NA	NA	NA
		Mg	52	31	77	110	119	NA	153	156	134	194	137	113	NA	NA	NA	NA
		Volume	90.8	87.1	58.3	92.4	125.6	68.5	78.1	103.1	89.2	81.4	94.1	94.9	NA	NA	NA	NA
FL4	0.63	pH	7.55	7.10	6.80	6.85	6.55	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
		SO4	610	470	801	765	771	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
		Ca	163	140	166	209	228	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
		Mg	17	15	17	34	37	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
		Volume	92.2	74.0	53.2	109.9	147.5	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
FL6	0.79	pH	7.51	6.40	4.40	4.40	4.20	4.19	4.14	4.27	4.27	4.00	4.12	4.20	4.10	4.14	4.24	4.24
		SO4	1046	1154	1496	1752	1891	2406	2469	2550	1873	2615	2201	2466	3234	2692	1683	1811
		Ca	261	292	277	249	270	NA	280	270	248	299	271	248	297	249	164	175
		Mg	77	110	172	205	220	NA	302	334	274	344	308	281	376	369	256	256
		Volume	117.2	118.2	82.3	130.5	117.2	96.0	99.2	148.3	114.4	108.1	117.1	102.0	36.2	106.8	31.9	120.4
FL5	1.41	pH	5.50	3.40	3.40	3.40	3.40	3.60	3.49	3.50	3.63	3.43	3.62	3.62	3.62	3.60	3.73	3.66
		SO4	862	2147	2900	3488	3805	4007	3424	3852	2973	2618	2355	2429	2264	1968	1642	2079
		Ca	225	285	228	220	225	NA	221	265	248	245	239	240	233	202	171	192
		Mg	27	129	273	308	311	NA	287	326	273	234	195	168	156	143	122	142
		Volume	120.5	117.3	35.7	58.9	105.8	57.2	49.6	83.2	54.5	60.1	70.9	82.5	76.4	70.2	17.8	69.7

Table 7. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL1 (0.63% S, 1100 T, no cover). Percent of total original sulfur released = 6.76.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	83100	206	9	821	416	58	3.62	1.84	0.26	7.80
1979	86600	252	33	670	483	71	2.42	1.74	0.26	7.35
1980	53800	256	28	673	426	66	2.39	1.51	0.23	7.20
1981	83800	275	28	673	499	93	2.22	1.65	0.31	7.20
1982	118600	258	28	932	755	113	3.28	2.66	0.40	7.00
1983	82400	277	10[2]	852	NA	NA	2.80	NA	NA	6.86
1984	66879	298	11	640	439	113	1.95	1.34	0.34	6.36
1985	133396	227	9	1221	793	231	4.89	3.18	0.93	6.70
1986	119551	238	9	900	658	202	3.44	2.51	0.77	6.54
1987	104768	233	9	1075	725	229	4.19	2.83	0.89	6.03
1988	106298	247	10	972	660	300	3.58	2.43	1.10	6.09
1989	122463	213	9	1230	693	308	5.25	2.96	1.31	5.69
1990	60101[3]	235	9	784	411	233	3.03	1.59	0.90	5.72
1991	120980	213	9	1577	772	552	6.73	3.29	2.36	5.25
1992	27465	221	16	274	136	98	1.13	0.56	0.40	5.19
1993	124968	228	17	1327	620	454	5.29	2.47	1.81	5.14

[1] Rates were calculated over the period of flow.
[2] Calcium and magnesium were not measured in 1983.
[3] FL1 was partially covered with plastic during 1990.

Table 8. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL2 (0.63% S, 1100 T, cover of topsoil). Percent of total original sulfur released = 3.94.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	70200	193	6	503	260	57	2.37	1.22	0.27	7.60
1979	75600	253	32	441	302	70	1.58	1.09	0.25	7.30
1980	29500	250	27	217	148	33	0.79	0.54	0.12	6.95
1981	67200	283	28	456	261	81	1.46	0.84	0.26	6.80
1982	100400	226	16	872	518	151	3.51	2.08	0.61	6.70
1983	52800	277	9[2]	472	NA	NA	1.55	NA	NA	6.39
1984	40431	247	10	288	185	69	1.06	0.68	0.25	6.17
1985	94649	227	9	948	496	212	3.80	1.99	0.85	6.30
1986	69334	238	9	428	286	124	1.63	1.09	0.47	6.17
1987	63221	233	9	445	293	136	1.74	1.14	0.53	5.54
1988	78948	252	10	681	399	252	2.46	1.44	0.91	5.47
1989	80188	213	9	624	326	199	2.66	1.39	0.85	5.27
1990	78458	235	9	722	352	243	2.79	1.36	0.94	5.13
1991	76029	213	9	683	316	241	2.92	1.35	1.03	5.18
1992	13481	221	16	115	48	39	0.47	0.20	0.16	5.07
1993	65210	228	17	636	260	253	2.54	1.04	1.01	5.04

[1] Rates were calculated over the period of flow.
[2] Calcium and magnesium were not measured in 1983.

Table 9. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL3 (0.63% S, 830 T, cover of glacial till (coarse sand)). Percent of total original sulfur released = 8.36.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	90800	193	5	904	646	194	5.64	4.03	1.21	7.60
1979	87100	258	34	505	326	111	2.36	1.52	0.52	7.05
1980	58300	249	30	628	306	184	3.04	1.48	0.89	6.65
1981	92400	285	31	1176	516	418	4.97	2.18	1.77	6.60
1982[2]	125600	262	21	1626	668	615	7.48	3.07	2.83	6.58
1983	68500	277	10[3]	1116	NA	NA	4.85	NA	NA	5.90
1984	78088	292	11	1120	410	493	4.62	1.69	2.03	5.05
1985	103131	227	9	1738	566	664	9.22	3.00	3.52	5.44
1986	89161	241	10	1123	413	491	5.61	2.06	2.45	5.27
1987	81412	233	9	1400	484	650	7.24	2.50	3.36	5.01
1988	94121	252	10	1200	398	529	5.74	1.90	2.53	5.15
1989[4]	94924	217	9	1118	363	440	6.21	2.02	2.44	4.98

[1] Rates were calculated over the period of flow.
[2] Fertilizer and 1500 gallons of lime-stabilized sludge were applied on May 19, 1982.
[3] Calcium and magnesium were not measured in 1983.
[4] FL3 was covered with plastic and subsequently hypalon, thereby terminating flow from the pile at the end of the 1989 field season.

Table 10. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL4 (0.63% S, 1130 T, no cover). Percent of total original sulfur released = 1.55.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	92200	188	7	585	374	65	2.75	1.76	0.31	7.55
1979	74000	233	26	362	258	46	1.37	0.98	0.17	7.10
1980	53200	211	18	444	220	37	1.86	0.92	0.16	6.80
1981	109900	276	29	875	574	152	2.81	1.84	0.49	6.85
1982	147500	238	15	1183	838	227	4.40	3.12	0.84	6.55

[1] Rates were calculated over the period of flow.

Table 11. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL6 (0.79% S, 1300 T, no cover). Percent of total original sulfur released = 11.30.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	117200	206	6	1275	764	370	4.76	2.85	1.38	7.51
1979	118200	257	36	1420	860	534	4.25	2.57	1.60	6.40
1980	82300	239	28	1281	570	582	4.12	1.83	1.87	4.40
1981	130500	276	30	2379	809	1103	6.63	2.25	3.07	4.40
1982	177200	250	19	3486	1191	1606	10.73	3.66	4.94	4.20
1983	96000	277	10[2]	2404	NA	NA	6.68	NA	NA	4.19
1984	99158	298	12	2548	693	1233	6.58	1.79	3.18	4.14
1985	148259	227	9	3934	1000	2037	13.33	3.39	6.90	4.27
1986	114434	238	9	2230	708	1291	7.21	2.29	4.17	4.27
1987	108134	227	9	2943	805	1533	9.97	2.73	5.19	4.00
1988	117127	247	10	2682	791	1483	8.35	2.46	4.62	4.12
1989	101975	217	9	2617	630	1181	9.28	2.23	4.19	4.20
1990	36196[3]	235	9	1218	268	560	3.99	0.88	1.83	4.10
1991	106787	213	9	2991	663	1622	10.80	2.39	5.86	4.14
1992	31850	221	16	558	130	335	1.94	0.45	1.17	4.24
1993	120417	228	17	2269	525	1270	7.66	1.77	4.28	4.24

[1] Rates were calculated over the period of flow.
[2] Calcium and magnesium were not measured in 1983.
[3] FL6 was partially covered with plastic during 1990.

Table 12. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL5 (1.41% S, 815 T, cover of sandy till over coarse sand). Percent of total original sulfur released = 8.58.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	120500	206	9	1081	676	135	6.44	4.03	0.80	5.50
1979	117300	251	34	2621	833	624	12.81	4.07	3.05	3.40
1980	35700	243	30	1077	203	402	5.44	1.03	2.03	3.40
1981	58900	275	29	2138	323	746	9.54	1.44	3.33	3.40
1982	105800	253	20	4189	595	1356	20.32	2.89	6.58	3.40
1983[2]	57200	277	27[3]	2385	NA	NA	10.56	NA	NA	3.60
1984	49625	299	12	1768	274	586	7.26	1.12	2.40	3.49
1985	83220	252	10	3336	549	1115	16.24	2.67	5.43	3.50
1986	54523	238	9	1687	337	613	8.70	1.74	3.16	3.63
1987	60084	227	9	1637	367	578	8.85	1.98	3.12	3.43
1988	70886	202	10	1737	423	569	10.55	2.57	3.46	3.62
1989	82485	220	9	2085	493	569	11.63	2.75	3.17	3.62
1990	76442	235	9	1801	445	491	9.40	2.32	2.56	3.62
1991	70171	213	9	1437	354	413	8.28	2.04	2.38	3.60
1992	17775	221	16	304	76	89	1.69	0.42	0.49	3.73
1993	69732	228	17	1508	333	406	8.12	1.79	2.18	3.66

[1] Rates were calculated over the period of flow.
[2] Bactericide was applied in July, 1983.
[3] Calcium and magnesium were not measured in 1983.

Table 13. Average rates of mass release and minimum drainage pH.

Percent Sulfur	Period of Record	Minimum pH[1]	Average Observed Rates of Release,[2] mmol/T•day			
			SO_4	Ca	Mg	Ca + Mg
0.63 (FL1)	1978-1993[3]	4.96	3.4	2.1	0.79	2.9
0.63 (FL2)	1978-1993	4.76	2.0	1.1	0.55	1.7
0.63 (FL3)	1978-1989	4.71	5.5	2.4	2.1	4.5
0.79 (FL6)	1978-1993[3]	3.70	7.0	2.2	3.6	5.8
1.41 (FL5)	1978-1993	3.08	9.8	2.1	3.0	5.1

[1] The minimum pH values usually occurred as isolated excursions. The recurrently observed minimum pH values were typically 0.2 to 0.3 units higher.

[2] This represents the release rate during the period of flow. 1983 was not included for the rates of Ca and Mg release, since these parameters were not analyzed in this year.

[3] Partially covered by plastic during 1990.

Table 14. Neutralization potentials as determined by drainage quality of Duluth Complex test piles.

Sulfur %	Pile	Neutralization Potential[1], mg $CaCO_3$/g rock					
		pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
0.63	FL1	0.346	0.781	>1.059	>1.059	>1.059	>1.059
0.63	FL2	0.179	0.393	>0.642	>0.642	>0.642	>0.642
0.63	FL3	0.442	1.254	>1.340	>1.340	>1.340	>1.340
0.79	FL6	0.157	0.172	0.226	1.702	>2.074	>2.074
1.41	FL5	0.044	0.044	0.096	0.130[2]	0.136[2]	>1.889

1 Neutralization potential (NP) was calculated using the cumulative calcium and magnesium release (see methods section of report).

2 Neutralization potential was calculated as of 1978, although higher pH values were observed later. Two pH values of 4.6 and 4.0 were observed in spring runoff in 1980 and 1989, respectively. Five pH values between 4.0 and 4.4 were observed in 1992.

Table 15. Qualitative comparison of test pile and laboratory data.

	Laboratory Drill Core[1]	Laboratory Blast Holes[2]	Field Test Piles
Drainage pH:			
as time increased	pH decreased	pH decreased	gradually decreased (0.63 pct S); decreased and plateaued (10.79 pct S); decreased and slightly increased (1.41 pct S)
as pct S increased	pH decreased	pH decreased	pH decreased
Sulfate Release Rate:			
as time increased max. rate: min. rate[3]	fairly constant, 1:1 to 3:1 for pH >4.5, 6:1 for some samples with pH~4	fairly constant, 1:1 to 3:1 for pH >4, 6:1 to 7:1 for pH <4	fairly constant, 2.7:1 to 3.9:1
as pct S increased	increased	increased	increased
as flow increased	no data	no data	increased
Calcium Release Rate:			
as time increased	fairly constant	limited data	fairly constant
as pct S increased	fairly constant	fairly constant	fairly constant
as flow increased	no data	no data	increased
Magnesium Release Rate:			
as time increased	increased	limited data	increased
as pct S increased	increased	increased	general increase although release from 0.79 pct S rock exceeded that from 1.41 pct S rock
as flow increased	no data	no data	increased

1 Lapakko, 1993a.
2 Lapakko, 1993b.
3 Ratio of maximum release rate to minimum release rate for a solid.

Table 16. Comparison of field and laboratory neutralization potentials.

Percent S	pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
Laboratory neutralization potential, mg $CaCO_3$/g rock[1]						
0.63	0.67	1.5	2.9	>2.9	>2.9	>2.9
0.79	0.29	0.75	1.7	>1.8	>1.8	>1.8
1.41	0.26	0.79	>1.7	>2.9	>3.1	>3.1
Ratio of field NP to laboratory NP						
0.63(FL1)	0.52	0.52	NAp	NAp	NAp	NAp
0.63(FL2)	0.27	0.26	NAp	NAp	NAp	NAp
0.63(FL3)	0.66	0.84	NAp	NAp	NAp	NAp
0.79(FL6)	0.54	0.23	0.13	NAp	NAp	NAp
1.41(FL5)	0.17	0.056	<0.056	<0.045	<0.044	NAp

NAp: not applicable

[1] See appendix 1 for sources of laboratory data.

Table 17. Comparison of field and laboratory blast hole release rates. Rates expressed in mole (g rock)$^{-1}$ s^{-1} x 10^{14}.

Percent Sulfur	Field Rate			Lab Rates			Retardation Factor			
	SO_4	Ca	Mg	SO_4	Ca	Mg	SO_4	Ca	Mg	Avg.
0.63[1] (FL1)	3.9	2.4	0.91	25[1]	6.7[1]	12[1]	0.16	0.36	0.076	0.20
0.63[1] (FL2)	2.3	1.3	0.64	25	6.7	12	0.093	0.19	0.053	0.11
0.63[1] (FL3)	6.4	2.8	2.4	25	6.7	12	0.26	0.42	0.20	0.29
0.79[2] (FL6)	8.1	2.6	4.2	26[2]	6.1[2]	12[2]	0.31	0.42	0.35	0.36
1.41[3] (FL5)	11	2.4	3.5	66[3]	19[3]	21[3]	0.17	0.13	0.17	0.16

1 The arithmetic mean of average rates from blast hole samples with sulfur contents of 0.57, 0.58, and 0.71 percent (150 week period of record) was used for comparison with 0.63-percent-sulfur rock. Minimum pH values for these samples ranged from 4.82 to 5.38 as compared to 4.7 to 5.0 in the field.

2 Rates were linearly extrapolated based on sulfur content using average rates from blast hole samples with sulfur contents of 0.71 and 1.16 percent (150- and 69-week periods of record, respectively). Minimum pH values for these samples ranged from 4.6 to 5.0, about one unit higher than the pH 3.7 minimum value for the field pile.

3 The arithmetic mean of average rates from blast hole samples with sulfur contents of 1.40 and 1.44 percent was used (69 week period of record) for comparison with the 1.41-percent-sulfur pile. Minimum drainage pH for these samples ranged from 4.4 to 4.9, about 1.5 units higher than the minimum drainage pH for the field pile and and one unit higher than its recurrent minimum pH.

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Appendix 2: Drainage Quality

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Appendix 3: Mass Release

Table A3.1. Percent of total original sulfur released.

Text A1.1. Neutralization potential calculation.

Individual drainage volume measurements were missing for given periods of record. Approximately half of the samples in the 1978 period of record for each test pile did not have an associated drainage volume value. One or two measurements were missing in 1982 and 1984 for each test pile. Calculation of the annual drainage volumes for this report omitted the missing values. When compared with the annual drainage volumes for 1978 through 1984 found in Eger and Lapakko (1985), discrepancies were revealed between the two sets of values. While most of the calculated values were generally five to twenty percent lower than the associated values in Eger and Lapakko (1985), the calculated 1978 values were several times lower.

To account for the discrepancies, the mass release values obtained using the calculated volumes were multiplied by adjusting factors. The factors were ratios of the annual volumes obtained in Eger and Lapakko (1985) over the calculated annual volumes for the respective piles. Adjusted mass release rates were calculated with the adjusted mass release values. The resulting rates compared favorably with those obtained in Eger and Lapakko (1985) (Table A1.2). The adjusted annual mass release and rates of release are used throughout this report. The annual drainage volumes for 1978 to 1984 obtained in Eger and Lapakko (1985) are used throughout the report as well.

The adjusted cumulative mass release values allowed more accurate determination of neutralization potentials (NP). These were calculated to determine the amount of acid neutralized by the solids prior to the drainage pH decreasing and, for the duration of the experiment, remaining below pH values of 7, 6, 5, 4.5, 4, and 3.5. The acid neutralized was calculated by two methods: as the a) cumulative sulfate release (expressed as mg $CaCO_3$/g rock) and b) cumulative calcium and magnesium release (expressed as mg $CaCO_3$/g rock) prior to the drainage pH decreasing and remaining below the specified pH value. If the drainage pH from a given solid never decreased permanently below a specified value, the NP was reported as "greater than" the total sulfate release for the period of record.

(a) Cumulative sulfate release to a given pH threshold:

$$NP = (\Sigma SO_4 \times 1.04) / m_p, \text{ where}$$

NP	=	neutralization potential, in mg $CaCO_3$/g rock
ΣSO_4	=	cumulative sulfate release to a given pH threshold, in milligrams
1.04	=	100 mg $CaCO_3$ / 96.1 mg SO_4
m_p	=	mass of pile, in grams

(b) Cumulative calcium plus magnesium release to a given pH threshold:

$$NP = [(\Sigma Ca \times 2.49) + (\Sigma Mg \times 4.12)] / m_p, \text{ where}$$

ΣCa	=	cumulative calcium release to a given pH threshold, in milligrams
2.49	=	100 mg $CaCO_3$ / 40.1 mg Ca
ΣMg	=	cumulative magnesium release to a given pH threshold, in milligrams
4.12	=	100 mg $CaCO_3$ / 24.3 mg Mg

Two steps were taken to increase the accuracy of the NP values by obtaining a more continuous record of cumulative release. First, mass release had to be estimated for 1977, when the piles were constructed. Mass release was estimated as the product of the mass of the pile, the number of days between the construction of the pile and freeze-up in November, 1977, and the mean rate of release from the pile for subsequent years. Rates of release from FL1 for 1978 to 1991 were used as representative rates for FL2 and FL3, because FL1 was uncovered for its entire period of record, while FL2 and FL3 were uncovered only in 1977.

The second step taken was to estimate mass release for 1983, when no calcium or magnesium concentrations were measured. Mass release for 1983 was estimated as the mean of the 1982 and 1984 flow-weighted-mean concentrations multiplied by the 1983 drainage volume. The estimated 1983 mass release was added only to those NP values corresponding to pH thresholds reached later than 1983. The pH 7 threshold for FL1 was reached in May, 1983, after one-fifth of the field season had passed. For that case, one-fifth of the estimated 1983 mass release was added to the NP_{pH7}.

Table A1.1. Monthly and annual precipitation at the Duluth Complex test piles site, 1978 - 1993.

Month	Precipitation, cm[1]															
	1978	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993
January	1.70	1.30	2.44	1.24	2.36	3.68[2]	0.43	1.14	1.80	1.17	3.07	3.23	2.36	0.00	0.00	0.00
February	0.81	4.85	1.68	3.40	1.07	1.68[2]	1.63	1.27	2.34	1.45	0.76	1.04	2.34	0.00	0.00	0.00
March	1.70	6.98	2.31	1.88	1.78	1.68[2]	0.61	1.88	1.12	0.89	6.10	3.33	5.46	3.07	0.00	0.00
April	3.25	2.50	1.90	11.2	2.41	1.40	3.00	6.81	6.76	0.56	0.23	1.30	10.4	1.83	1.55	5.49
May	9.35	9.73	1.32	2.46	11.1	4.24	3.48	10.7	4.80	14.5	7.65	8.15	2.18	8.13	9.60	6.65
June	4.90	10.1	9.09	15.5	6.65	5.13	15.2	12.1	11.0	3.12	6.58	18.3	13.3	12.1	7.04	9.98
July	19.1	7.04	6.27	4.29	16.4	8.51	4.80	8.41	10.2	16.8	6.50	2.84	6.43	13.4	8.59	19.0
August	14.1	5.89	13.4	4.39	12.5	12.4	6.43	9.73	9.12	9.45	30.5	11.2	3.58	6.25	13.5	9.78
September	8.23	5.08	10.8	5.97	9.70	8.48	4.83	16.2	13.3	8.05	6.27	10.7	7.77	13.3	10.9	6.07
October	2.59	7.59	3.66	9.55	13.3	11.0	7.16	4.75	2.39	2.06	4.83	4.93	8.99	4.67	2.39	0.64
November	2.46	1.40	2.84	1.60	5.21	8.86[2]	1.70	6.07	5.26	1.32	5.77	2.84	0.08	0.00	0.00	0.00
December	1.37	0.89	3.10	2.01	4.04	3.76[2]	4.27	1.04	0.94	0.97	2.84	1.40	1.57	0.00	0.00	0.00
TOTAL	69.6	63.3	58.8	63.4	86.5	70.9	53.5	80.1	69.0	60.3	81.1	69.3	64.5	62.8	60.0[3]	65.5[4]

[1] From Kennecott raingage unless otherwise noted.
[2] From Babbitt raingage.
[3] Includes 6.45 cm of spring runoff from snowpack.
[4] Includes 7.92 cm of spring runoff from snowpack.

Table A1.2. Comparison of adjusted rates and those obtained in the 1985 Heavy Metals Study. See Methods for more information.

Pile	Year	Adj. SO_4 Rate mmol/T/day	Adj. Ca Rate mmol/T/day	Adj. Mg Rate mmol/T/day	HMS SO_4 Rate mmol/T/day	HMS Ca Rate mmol/T/day	HMS Mg Rate mmol/T/day	Adj. SO_4/ HMS SO_4	Adj. Ca/ HMS Ca	Adj. Mg/ HMS Mg
FL1	1978	3.62	1.84	0.26	4.47	2.49	0.33	0.81	0.74	0.77
	1979	2.42	1.74	0.26	2.39	1.75	0.26	1.01	1.00	0.99
	1980	2.39	1.51	0.24	2.39	1.50	0.23	1.00	1.01	1.01
	1981	2.23	1.65	0.31	2.19	1.65	0.30	1.02	1.00	1.01
	1982	3.28	2.66	0.40	2.71	2.49	0.39	1.21	1.07	1.01
	1983	2.79	NA	NA	2.81	NA	NA	0.99	NA	NA
FL2	1978	2.37	1.22	0.27	3.23	1.87	0.37	0.73	0.65	0.71
	1979	1.59	1.08	0.25	1.46	1.10	0.26	1.09	0.99	0.98
	1980	0.79	0.54	0.12	0.80	0.55	0.12	0.98	0.98	0.99
	1981	1.46	0.84	0.26	1.46	0.85	0.26	1.00	0.99	0.99
	1982	3.51	2.08	0.61	3.02	1.95	0.58	1.16	1.07	1.06
	1983	1.55	NA	NA	1.56	NA	NA	0.99	NA	NA
FL3	1978	5.64	4.03	1.21	3.54	2.29	0.66	1.60	1.76	1.84
	1979	2.36	1.52	0.52	2.50	1.57	0.53	0.94	0.97	0.97
	1980	3.04	1.48	0.89	3.02	1.47	0.91	1.01	1.01	0.98
	1981	4.97	2.18	1.77	4.99	2.17	1.77	1.00	1.01	1.00
	1982	7.48	3.07	2.83	6.87	2.99	2.67	1.09	1.03	1.06
	1983	4.85	NA	NA	4.89	NA	NA	0.99	NA	NA
FL4	1978	2.75	1.76	0.31	3.75	2.37	0.40	0.73	0.74	0.78
	1979	1.37	0.98	0.17	1.35	1.00	0.18	1.01	0.98	0.94
	1980	1.86	0.92	0.16	1.87	0.92	0.16	0.99	1.00	1.00
	1981	2.81	1.84	0.49	2.81	1.85	0.49	1.00	0.99	1.00
	1982	4.40	3.12	0.84	3.95	2.99	0.78	1.11	1.04	1.08

Table A1.2. Comparison of adjusted rates and those obtained in the 1985 Heavy Metals Study. See Methods for more information.

Pile	Year	Adj. SO_4 Rate mmol/T/day	Adj. Ca Rate mmol/T/day	Adj. Mg Rate mmol/T/day	HMS SO_4 Rate mmol/T/day	HMS Ca Rate mmol/T/day	HMS Mg Rate mmol/T/day	Adj. SO_4/ HMS SO_4	Adj. Ca/ HMS Ca	Adj. Mg/ HMS Mg
FL6	1978	4.76	2.85	1.38	7.91	5.74	2.26	0.60	0.50	0.61
	1979	4.25	2.57	1.60	4.06	2.49	1.60	1.05	1.03	1.00
	1980	4.12	1.83	1.87	4.06	1.85	1.89	1.02	0.99	0.99
	1981	6.63	2.25	3.07	6.66	2.27	3.09	1.00	0.99	1.00
	1982	10.73	3.66	4.94	9.16	3.24	4.53	1.17	1.13	1.09
	1983	6.68	NA	NA	6.66	NA	NA	1.00	NA	NA
FL5	1978	6.44	4.02	0.80	9.47	5.99	1.11	0.68	0.67	0.72
	1979	12.81	4.07	3.05	12.49	3.99	2.96	1.03	1.02	1.03
	1980	5.44	1.03	2.03	5.41	1.02	2.02	1.01	1.00	1.01
	1981	9.54	1.44	3.33	9.57	1.45	3.33	1.00	1.00	1.00
	1982	20.32	2.88	6.58	19.77	2.74	6.58	1.03	1.05	1.00
	1983	10.56	NA	NA	10.41	NA	NA	1.02	NA	NA

Table A1.3. Laboratory neutralization potential data (for values listed as "greater than" averages represent "best guess").

Percent Sulfur	Sample Type[1]	Period of Record (weeks)	Neutralization Potential					
			pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
For comparison with 0.63 percent sulfur rock in field								
0.57	B	150	1.32	2.0	>3.1	>3.1	>3.1	>3.1
0.58	B	150	0.35	1.4	2.9	>3.0	>3.0	>3.0
0.71	B	150	0.35	1.1	>2.6	>2.6	>2.6	>2.6
Avg.			0.67	1.5	2.9	>2.9	>2.9	>2.9
For comparison with 0.79 percent sulfur rock in field								
0.71	B	150	0.35	1.1	>2.6	>2.6	>2.6	>2.6
0.80	D	17	0.32	>0.39	>0.39	>0.39	>0.39	>0.39
0.92	D	34	0.19	0.40	0.77	>0.93	>0.93	>0.93
Avg.			0.29	0.75[2]	1.7[3]	>1.8[4]	>1.8[4]	>1.8[4]
For comparison with 1.41 percent sulfur rock in field								
1.35	D	30	0	0	0	3.1	>3.7	>3.7
1.40	B	78	0.41	1.6	>3.0	>3.0	>3.0	>3.0
1.44	B	78	0.38	0.77	2.0	>2.7	>2.7	>2.7
Avg.			0.26	0.79	>1.7	>2.9	>3.1	>3.1

[1] D:drill core, B:blast hole
[2] (1.1 + 0.4)/2
[3] (2.6 + 0.77)/2
[4] (2.6 + 0.93)/2

Table A1.4. Release rates for drill core samples (Lapakko, 1993a).

Percent Sulfur	Period (weeks)	Rates, mole (g rock)$^{-1}$s^{-1}			Minimum pH
		SO_4	Ca	Mg	
0.47	2-13	13	19	3.3	6.85
0.59	7-17	22	26	6.4	7.30
0.80	6-17	36	34	7.9	6.75
0.80	9-17	35	30	9.3	6.90
0.92	23-34	82	28	28	4.60
0.92	27-34	42	17	19	4.90
1.17[1]	11-17	45	(78)[2]	12	8.30
1.17[1]	13-17	58	(83)[2]	13	8.20
1.24	2-21	49	26	14	5.70
1.24	2-21	45	25	14	5.70
1.26[3]	1-21	27	20	3.5	6.20
1.26[3]	1-21	28	19	3.7	5.95
1.35	6-25	93	18	20	4.20
1.47[4]	13-21	190	37	31	4.40
1.47[4]	14-21	310	39	38	4.25
1.74[4]	11-21	350	82	73	4.30
1.74[4]	14-21	170	59	50	4.55
1.87	17-34	150	59	59	5.05
1.87	17-34	160	53	72	4.90
2.01	17-30	380	24	15	3.70
2.17[5]	15-21	190	29	42	4.30
2.17[5]	9-21	76	19	17	4.65
2.57[5]	16-21	110	8.2	11	4.20
2.57[5]	16-21	170	14	13	4.10

1 Sample contained three percent $CaCO_3$ which is atypical of Duluth Complex rock.

2 Parentheses indicate anomalous value.

3 Sample underwent oxidation previously in a test pile.

4 Sample was amphibole-rich and thus atypical of Duluth Complex rock.

5 Contained a significant amount of quartz and was, therefore, atypical of Duluth Complex rock.

Table A1.5. Release rates for blast hole samples during period of Ca, Mg analysis (Lapakko, 1993b).

Percent Sulfur	Period, weeks		Rates, mole (g rock)$^{-1}$s^{-1}			Minimum pH
	SO_4	Ca, Mg	SO_4	Ca	Mg	
0.18	134-150	134-150	13	7.8	6.3	6.29
0.18	136-150	132-148	12	7.4	6.9	6.30
0.22	134-150	134-146	11	6.7	8.9	6.52
0.22	134-150	132-148	11	10	7.6	6.33
0.40	135-150	134-150	24	25	0.79	6.17
0.40	134-150	136-148	22	23	0.79	6.30
0.41	136-150	127-150	14	6.0	5.4	5.30
0.41	134-150	132-144	14	5.6	5.0	5.32
0.51	136-150	127-150	16	6.2	6.0	5.08
0.51	136-150	132-148	18	6.3	5.0	4.98
0.57	134-150	134-150	24	6.3	12	5.38
0.57	137-150	132-148	26	8.0	14	5.35
0.58	140-150	134-150	28	7.6	12	4.85
0.58	136-150	132-148	32	9.4	14	4.82
0.71	135-150	127-146	22	4.9	13	4.98
0.71	137-150	132-148	20	4.0	10	5.05
1.12	35-69	53-65	(100)[2]	(62)	(5.8)	4.62
1.12	37-69	51-65	(97)	(61)	(7.0)	4.75
1.16	53-69	47-69	48	11	17	4.69
1.16	50-69	51-67	51	17	16	4.60
1.40	35-57	46-65	79	29	21	4.78
1.40	33-69	51-67	75	25	24	4.92
1.44	37-69	53-65	56	9.8	19	4.59
1.44	36-69	51-65	56	11	20	4.41
1.63	117-145[1]	138-150	62	8.1	23	4.37
1.63	119-150	127-148	180	12	35	3.90
1.64	55-69	53-69	76	18	30	4.42
1.64	25-69	51-67	95	21	32	4.32

1 dSO_4/dt = 83 for wks 146-150

2 Parentheses indicate anomalous value.

Table A1.6. Comparison of field and laboratory drill core data.

Percent Sulfur	Field Rate			Min Field pH	Lab Rates			Min Lab pH	Retardation Factor				
	SO_4	Ca	Mg		SO_4	Ca	Mg		SO_4	Ca	Mg	Ca+Mg	Avg.
0.63 (FL1)[1]	3.9	2.4	0.91	4.96	25	27	6.7	6.8-7.3	0.16	0.089	0.14	0.098	0.13
0.63 (FL2)[1]	2.3	1.3	0.64	4.76	25	27	6.7	6.8-7.3	0.092	0.048	0.096	0.058	0.079
0.63 (FL3)[1]	6.4	2.8	2.4	4.71	25	27	6.7	6.8-7.3	0.26	0.10	0.36	0.15	0.24
0.79[2]	8.1	2.6	4.2	3.70	36	32	8.6	6.8-6.9	0.23	0.081	0.49	0.17	0.27
1.41[3]	11	2.4	3.5	3.08	170	28	27	4.2-4.4	0.065	0.086	0.13	0.11	0.094

1 Lab rates for 0.63-percent-sulfur determined by interpolation of rates from 0.59 and 0.80-percent-sulfur samples (17 week period of record) and assuming rates varied linearly with sulfur content.

2 Lab rates for 0.79-percent-sulfur from 0.80-percent-sulfur sample (17 week period of record).

3 Lab rates for 1.41-percent-sulfur sample determined by averaging rates for samples with sulfur contents of 1.35 (34 week period of record) and 1.47 (21 week period of record) percent.

Figure A2.1. pH over time for Duluth Complex test pile FL1 (0.63% S).

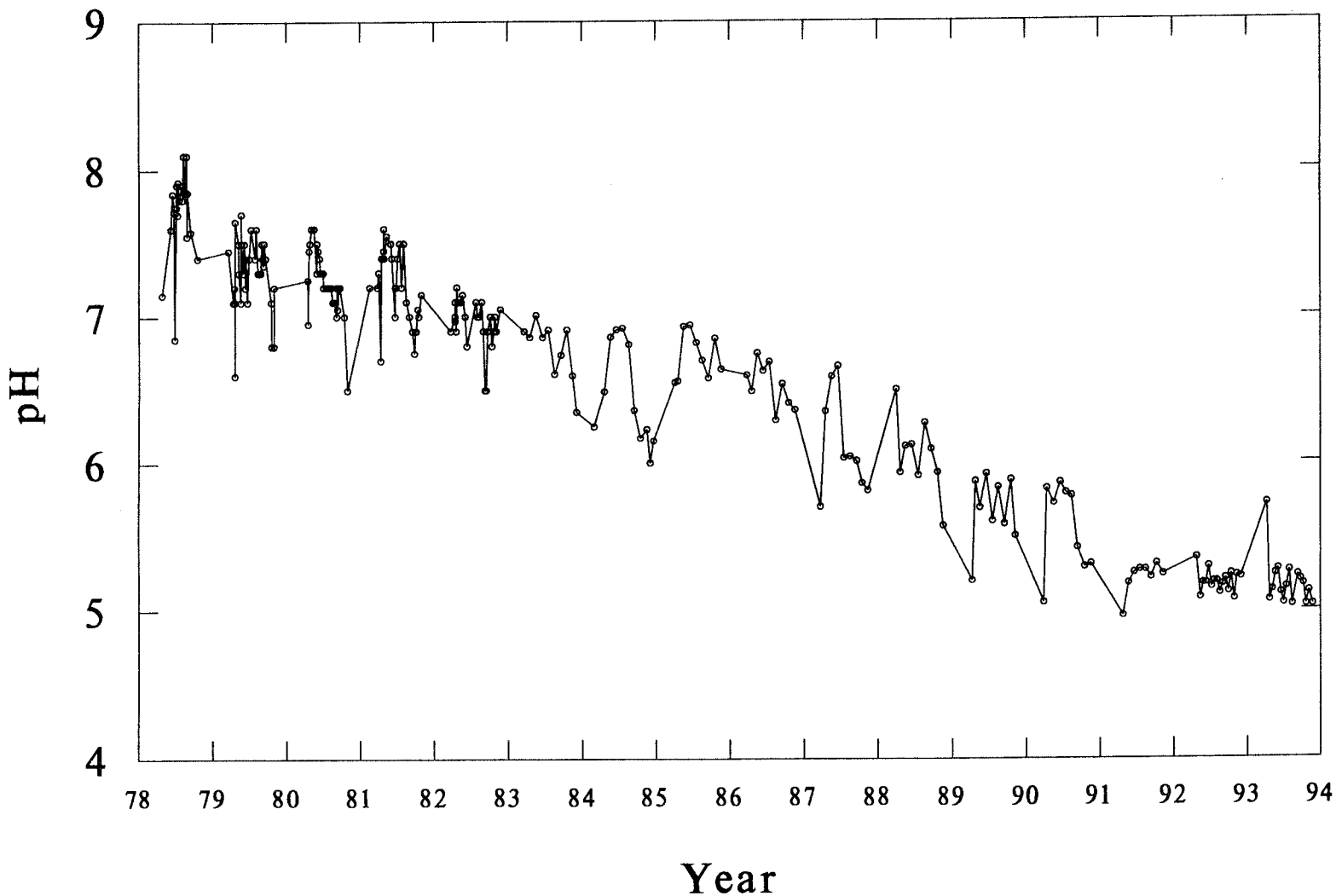

Figure A2.2. Sulfate concentrations over time for Duluth Complex test pile FL1 (0.63% S).

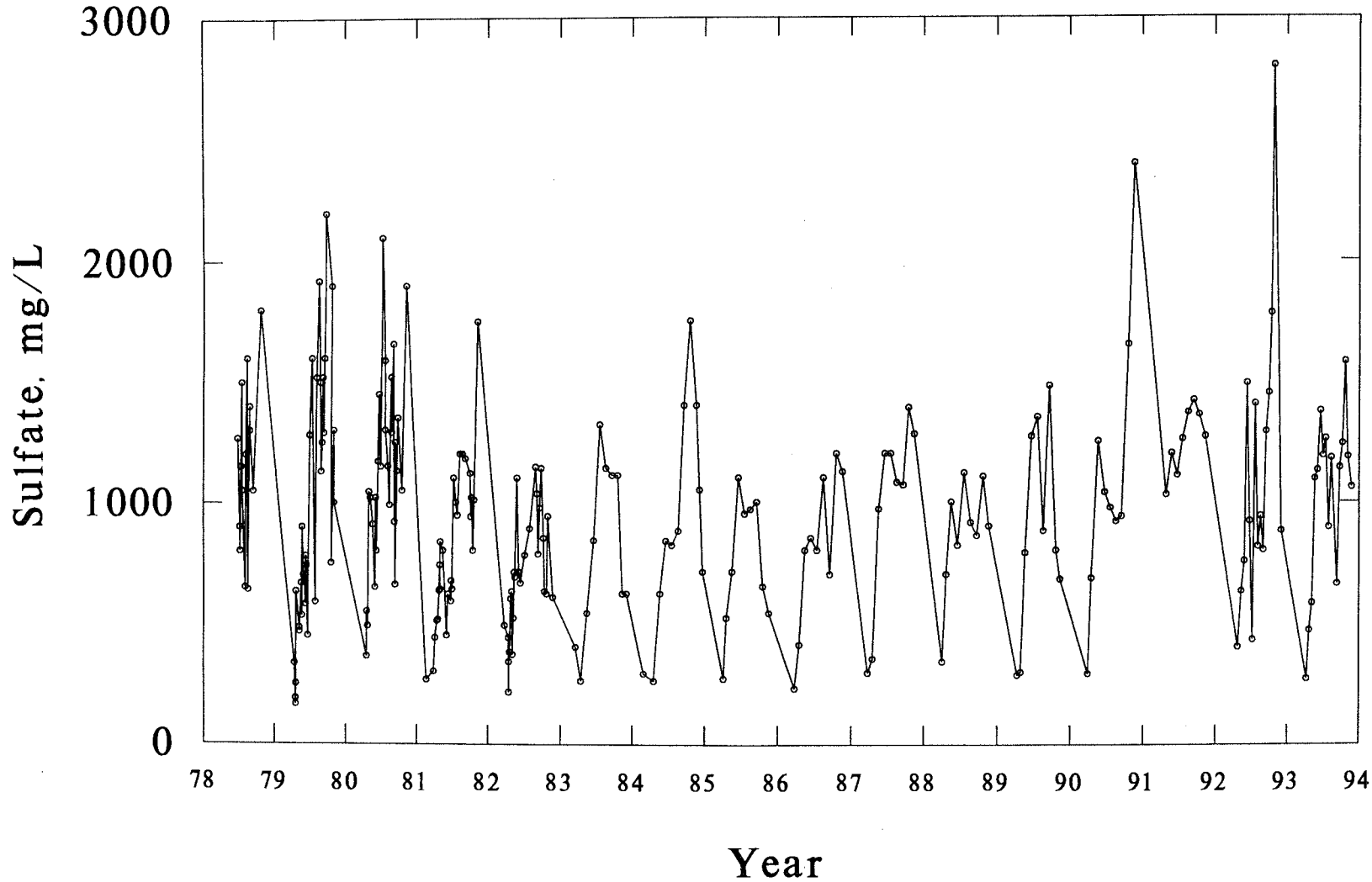

Figure A2.3. Calcium concentrations over time for Duluth Complex test pile FL1 (0.63% S).

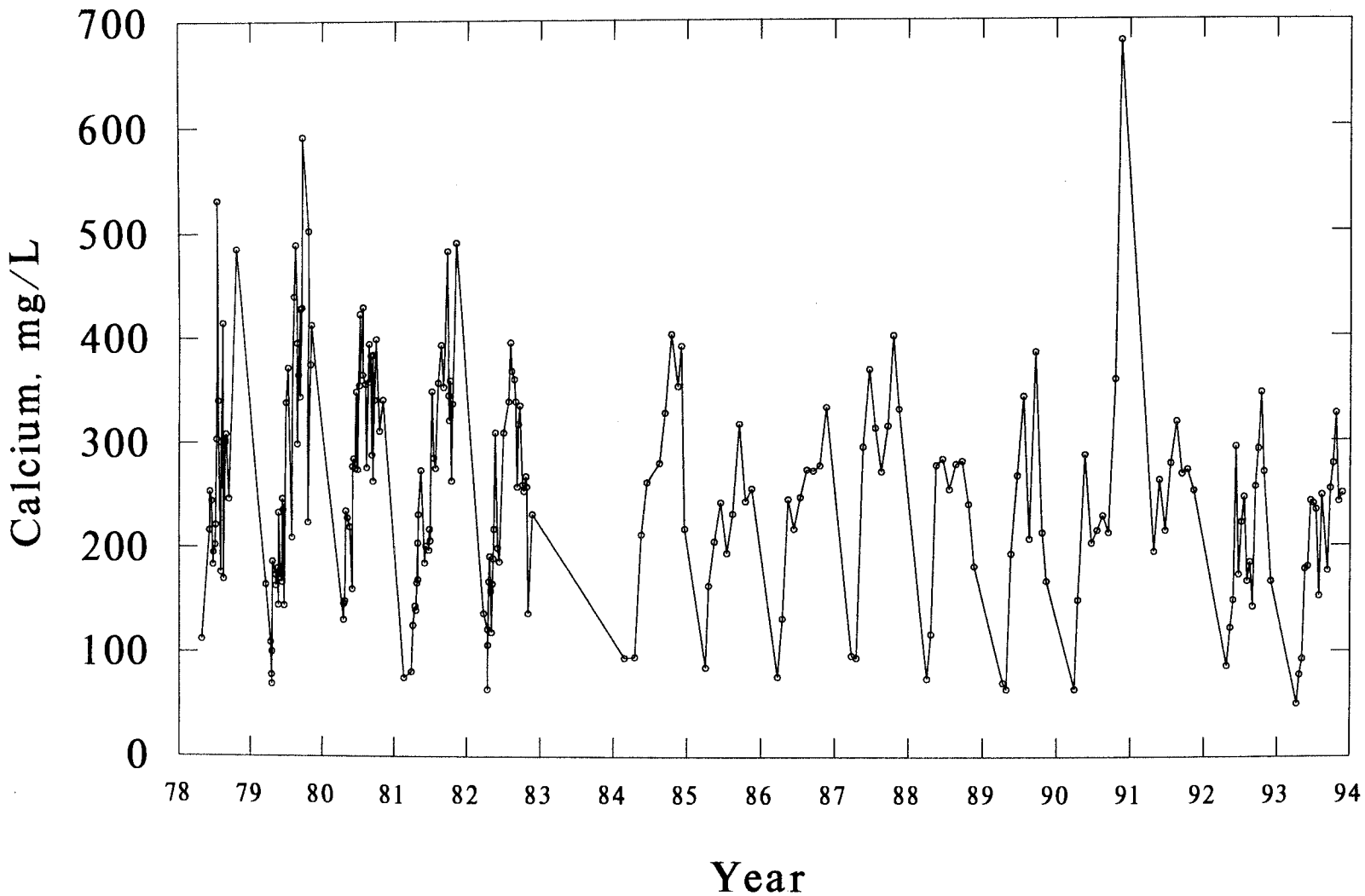

Figure A2.4. Magnesium concentrations over time for Duluth Complex test pile FL1 (0.63% S).

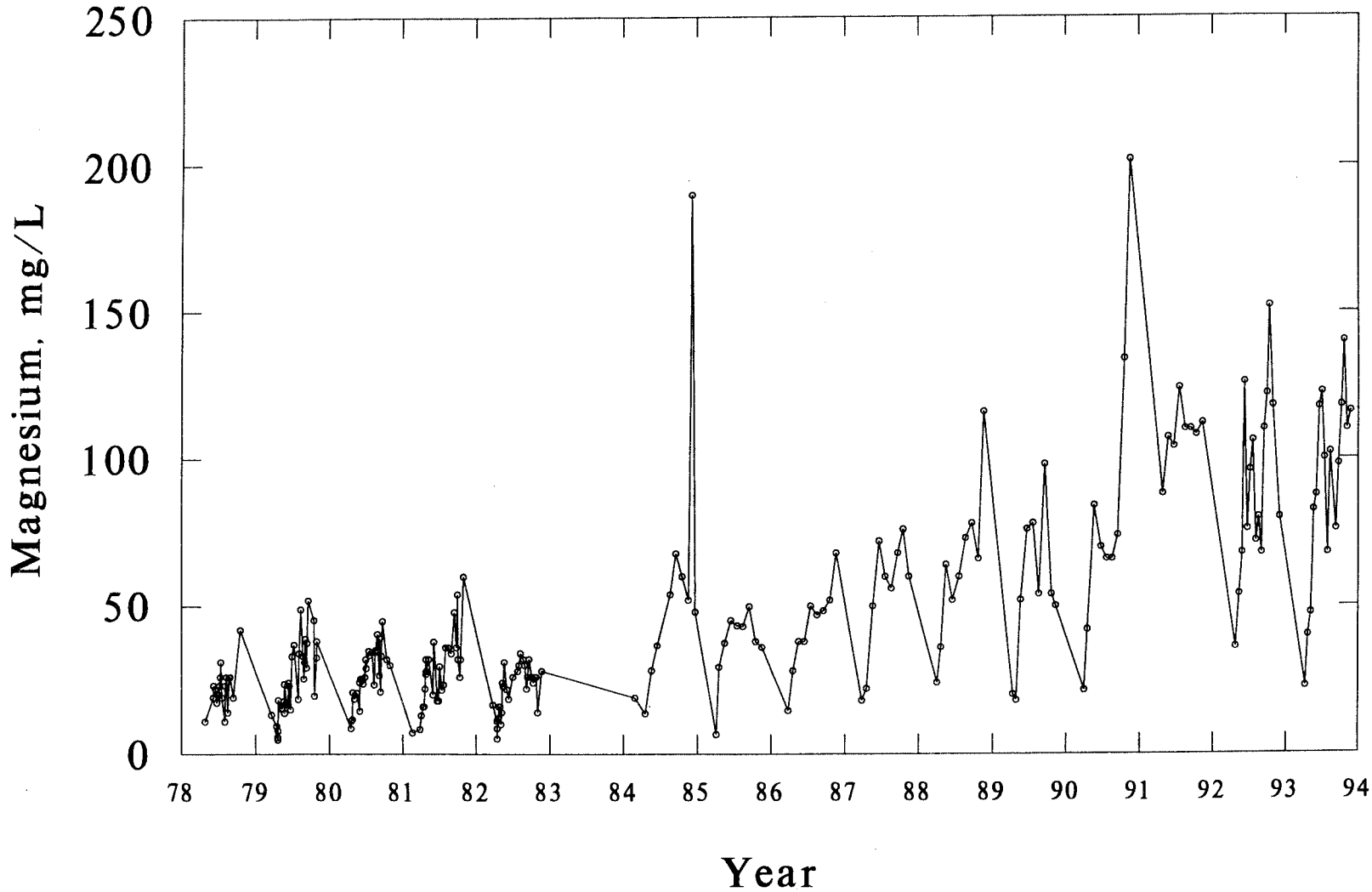

Figure A2.5. pH over time for Duluth Complex test pile FL2 (0.63% S).

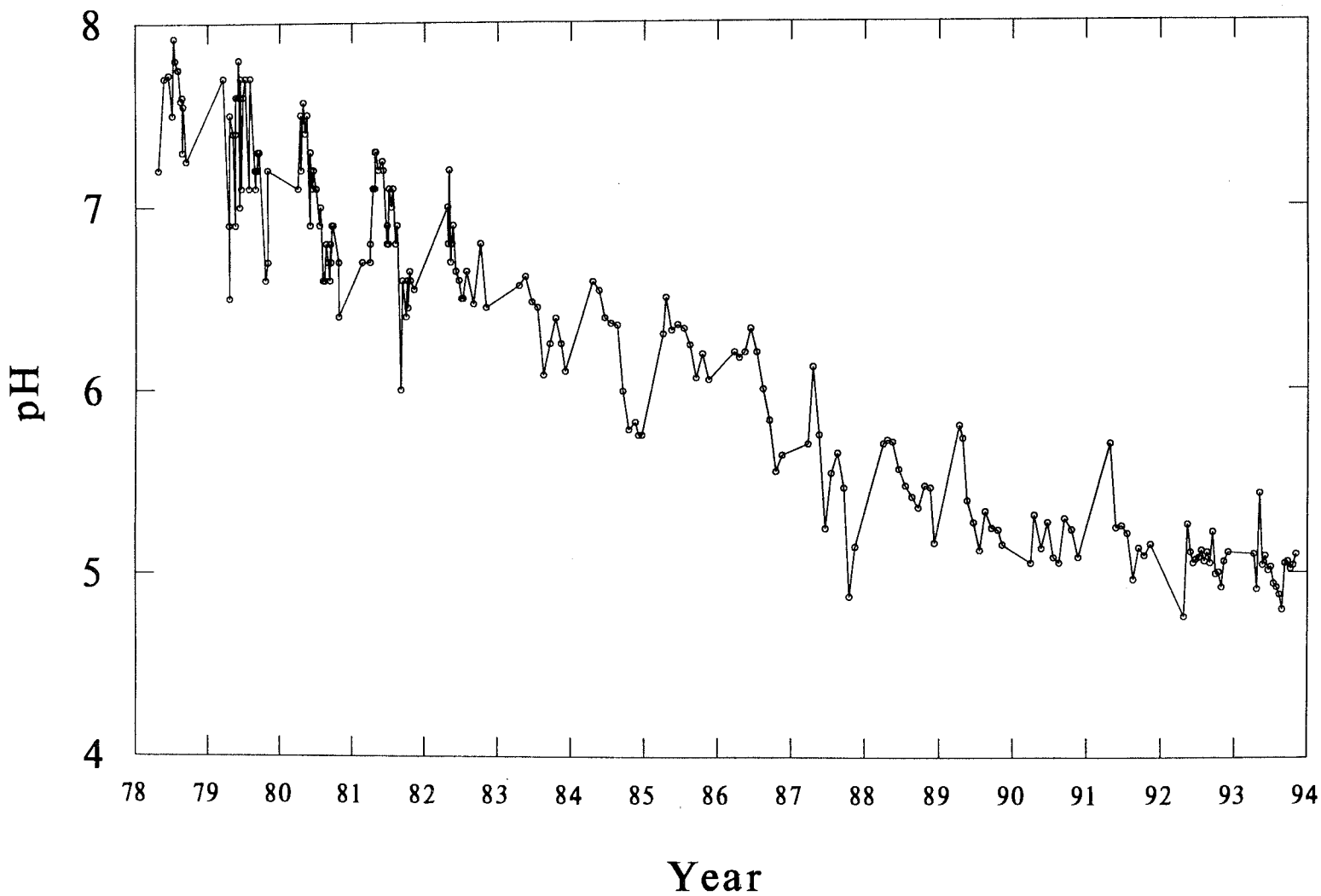

Figure A2.6. Sulfate concentrations over time for Duluth Complex test pile FL2 (0.63% S).

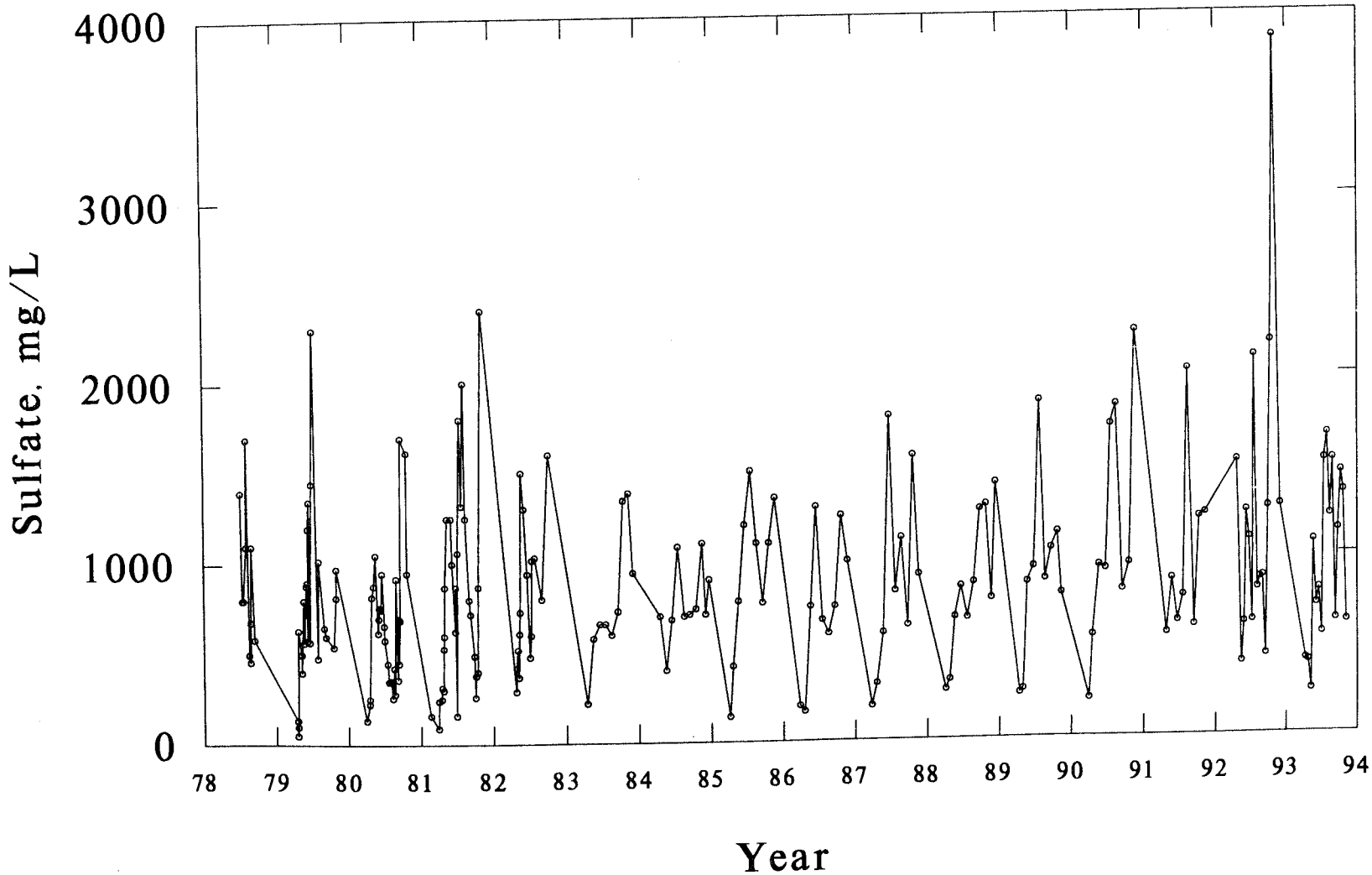

Figure A2.7. Calcium concentrations over time for Duluth Complex test pile FL2 (0.63% S).

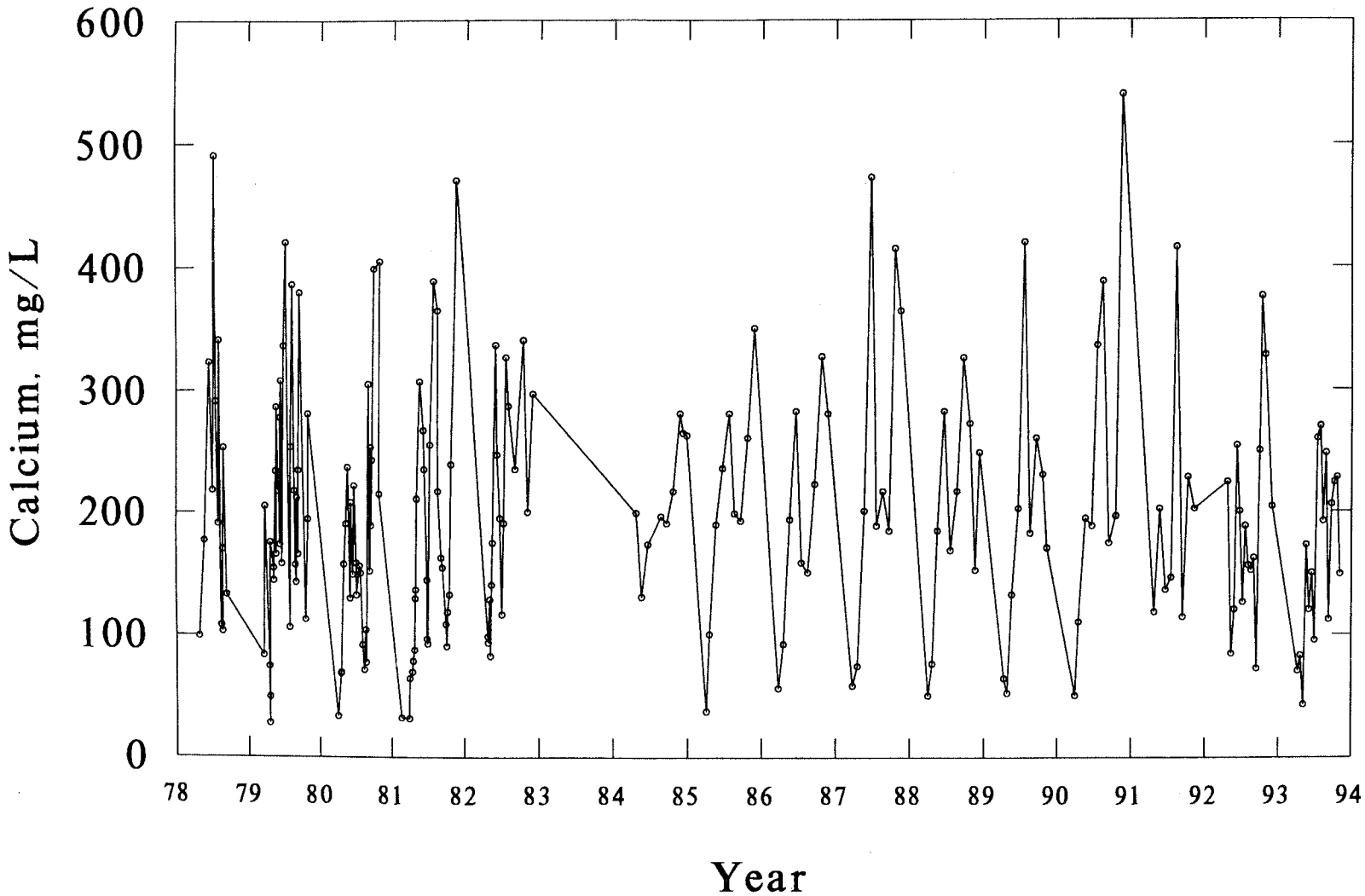

Figure A2.8. Magnesium concentrations over time for Duluth Complex test pile FL2 (0.63% S).

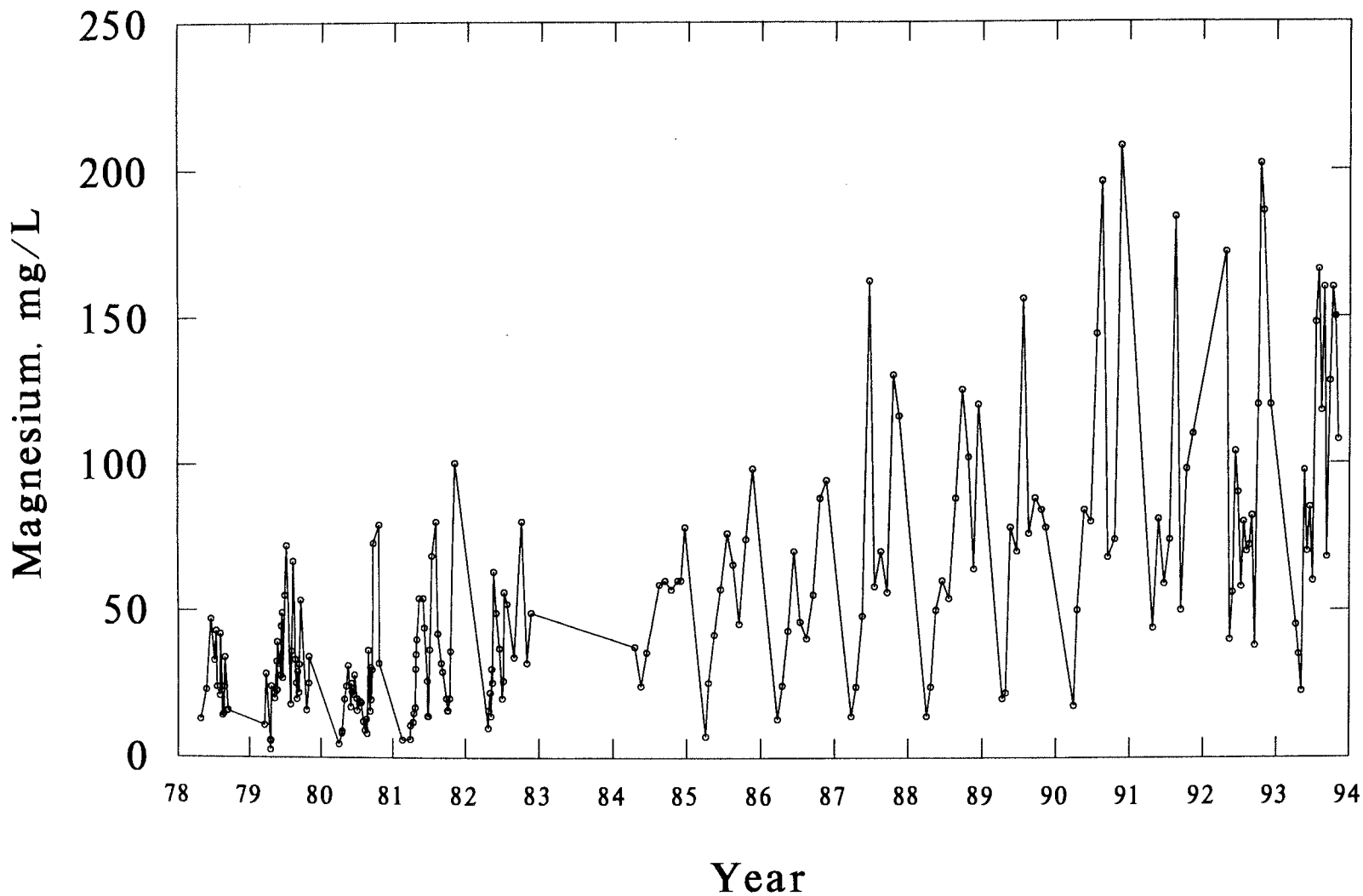

Figure A2.9. pH over time for Duluth Complex test pile FL3 (0.63% S).

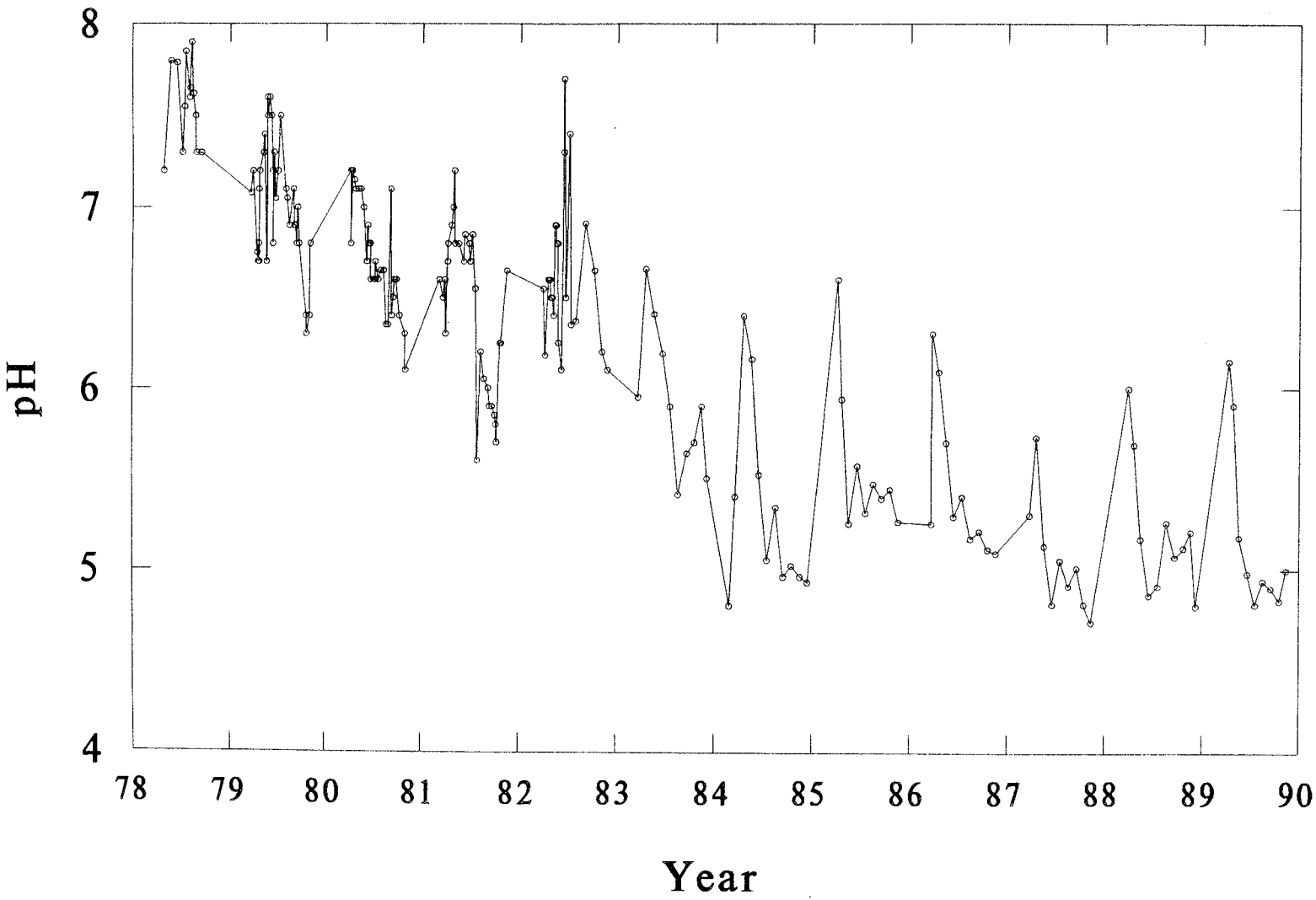

Figure A2.10. Sulfate concentrations over time for Duluth Complex test pile FL3 (0.63% S).

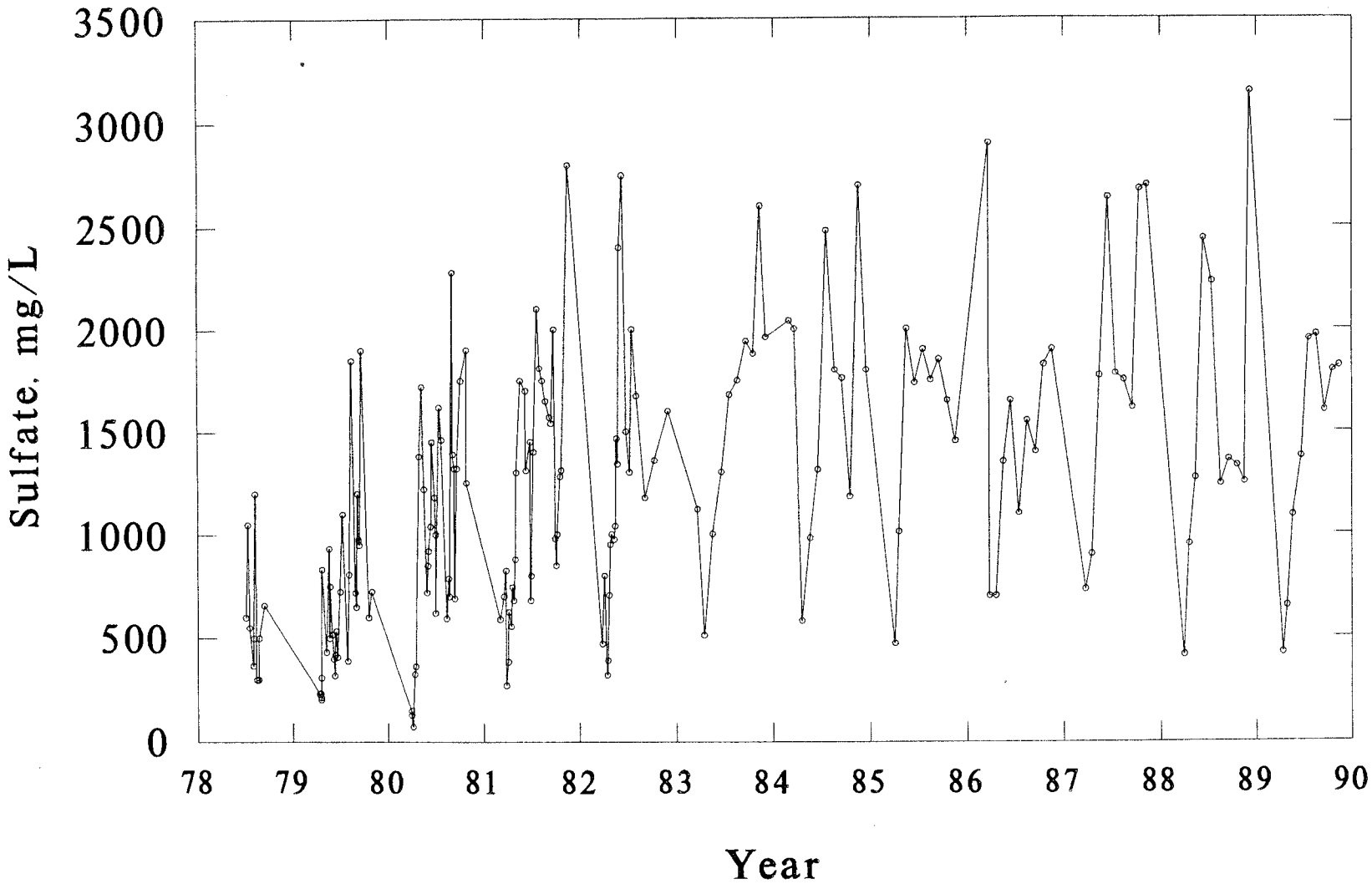

Figure A2.11. Calcium concentrations over time for Duluth Complex test pile FL3 (0.63% S).

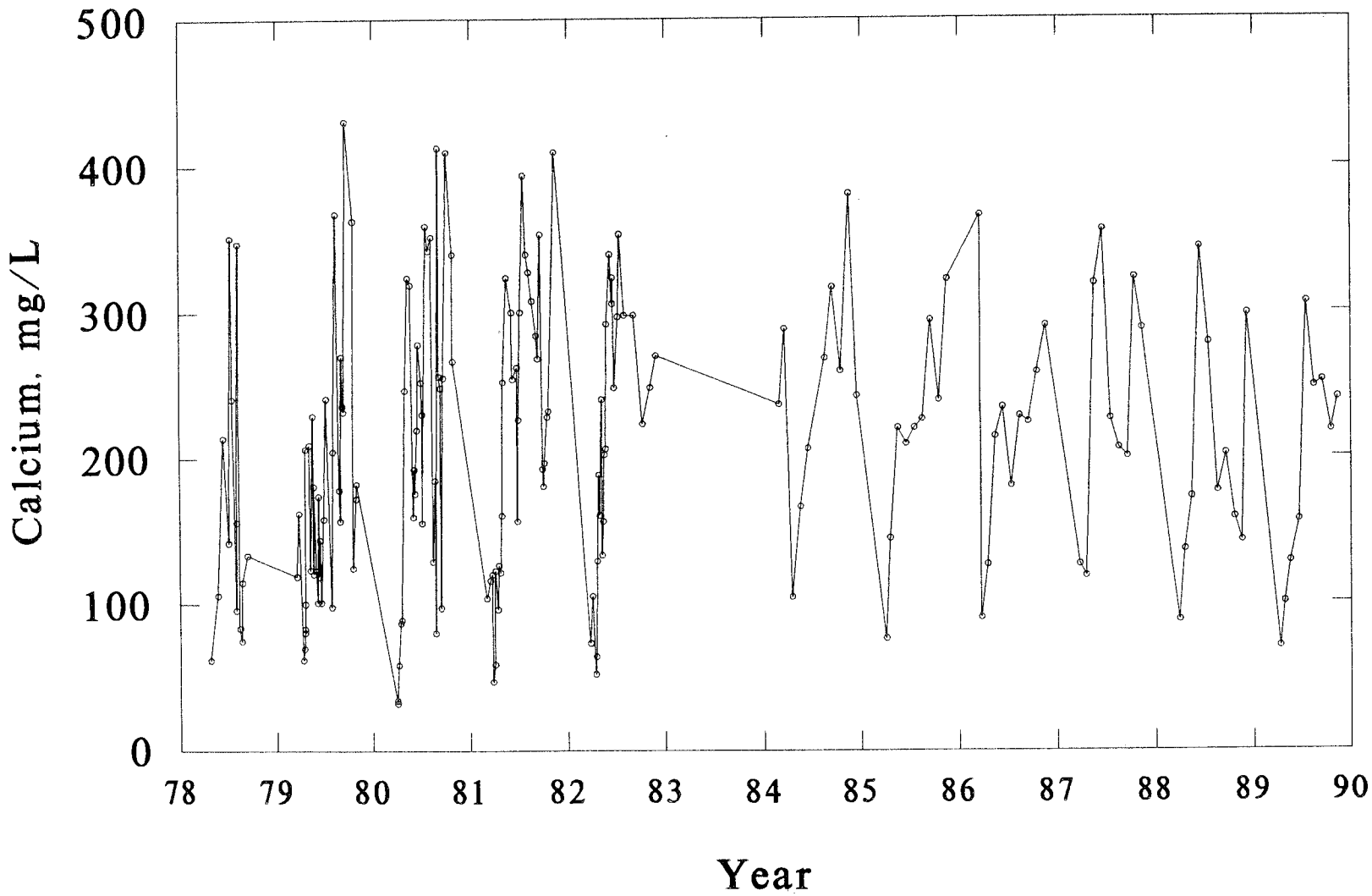

Figure A2.12. Magnesium concentrations over time for Duluth Complex test pile FL3 (0.63% S).

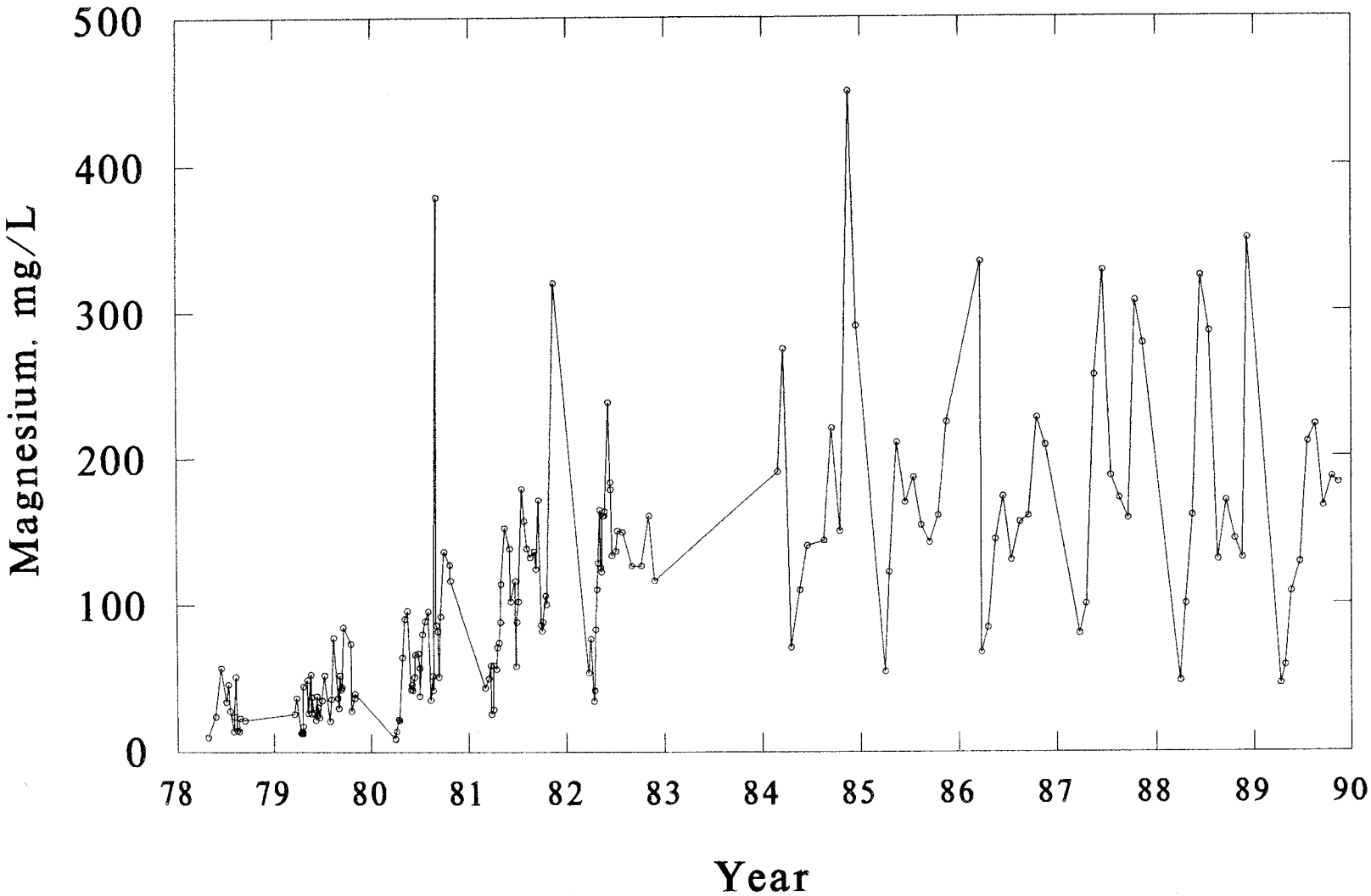

Figure A2.13. pH over time for Duluth Complex test pile FL4 (0.63% S).

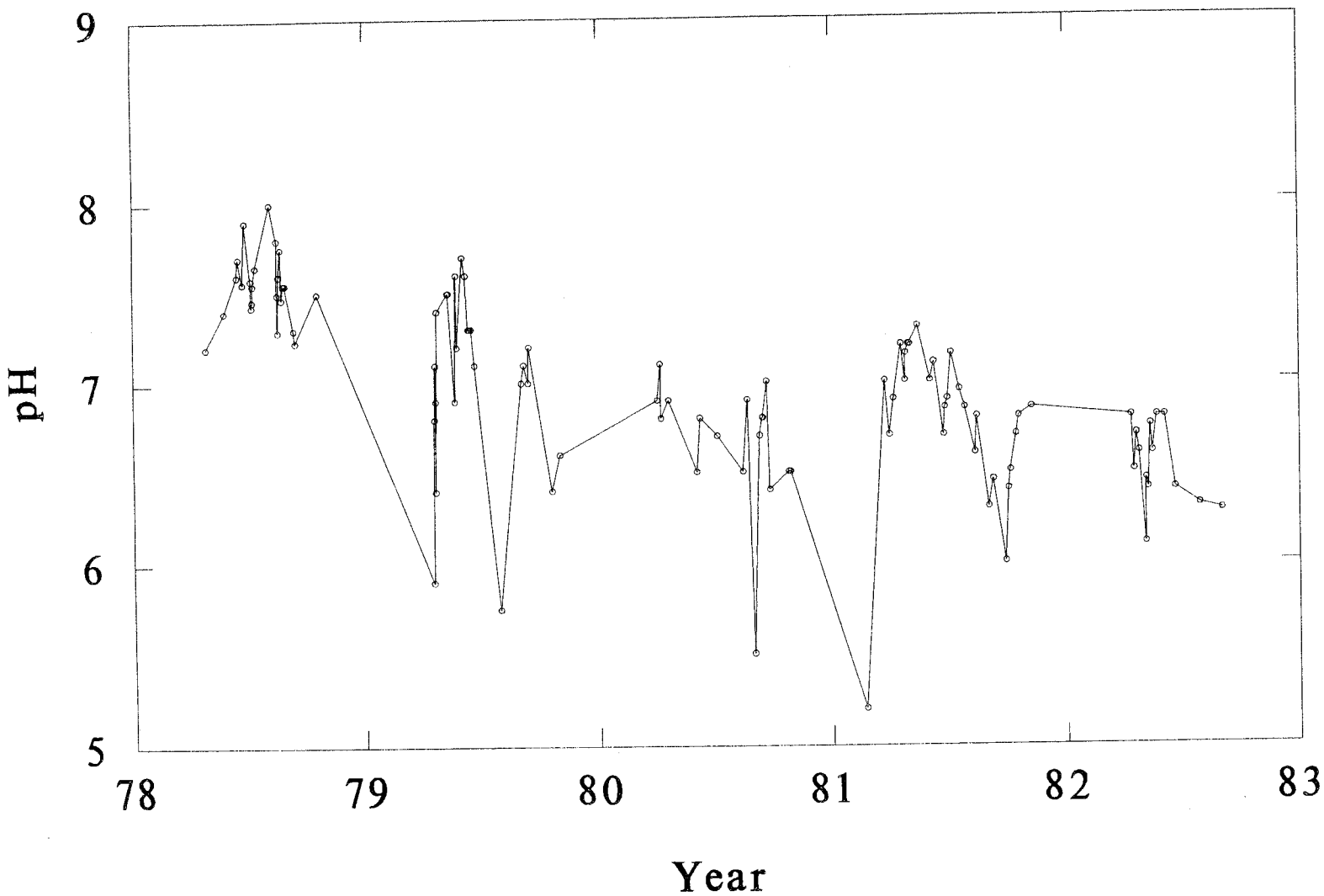

Figure A2.14. Sulfate concentrations over time for Duluth Complex test pile FL4 (0.63% S).

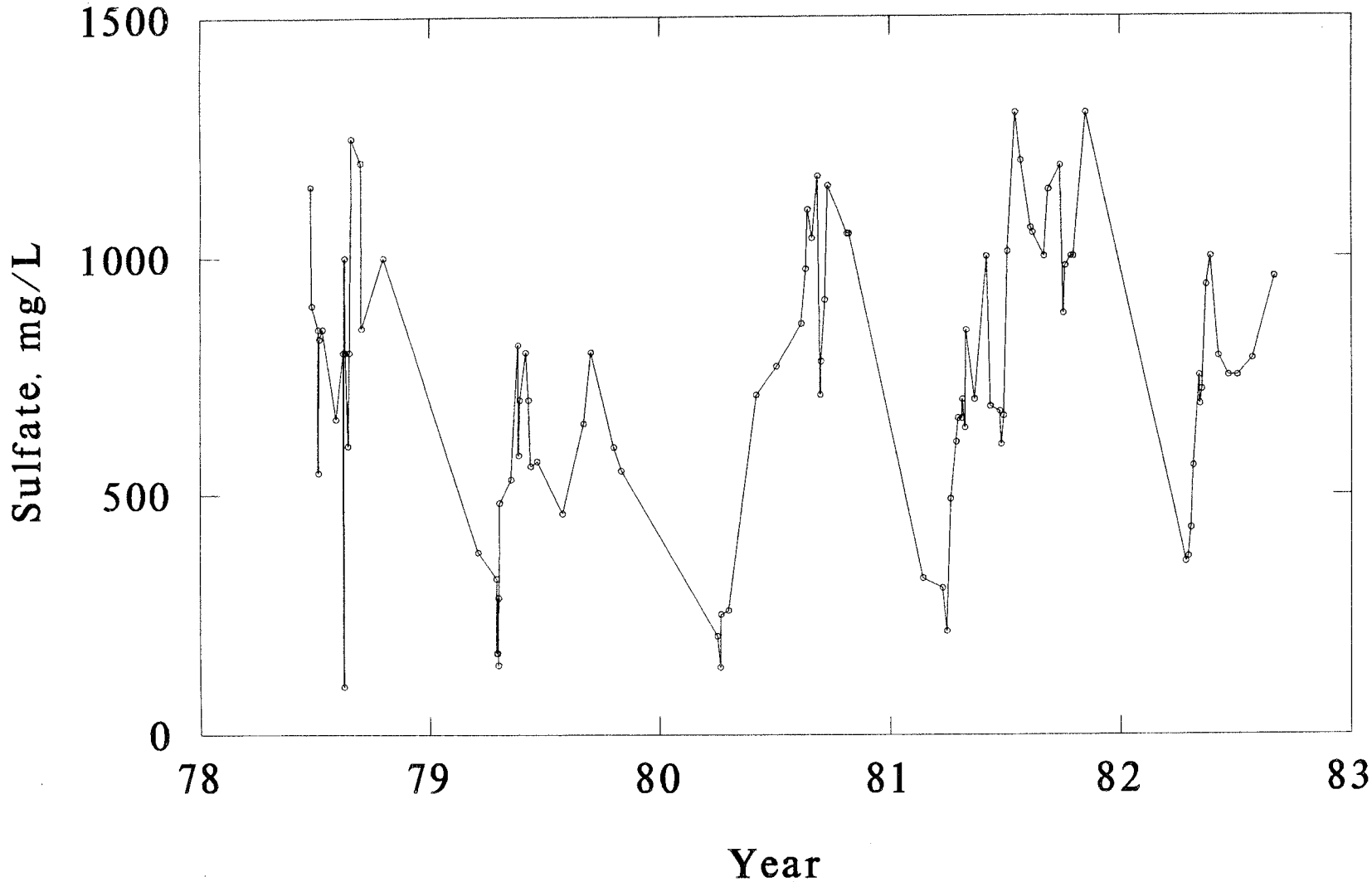

Figure A2.15. Calcium concentrations over time for Duluth Complex test pile FL4 (0.63% S).

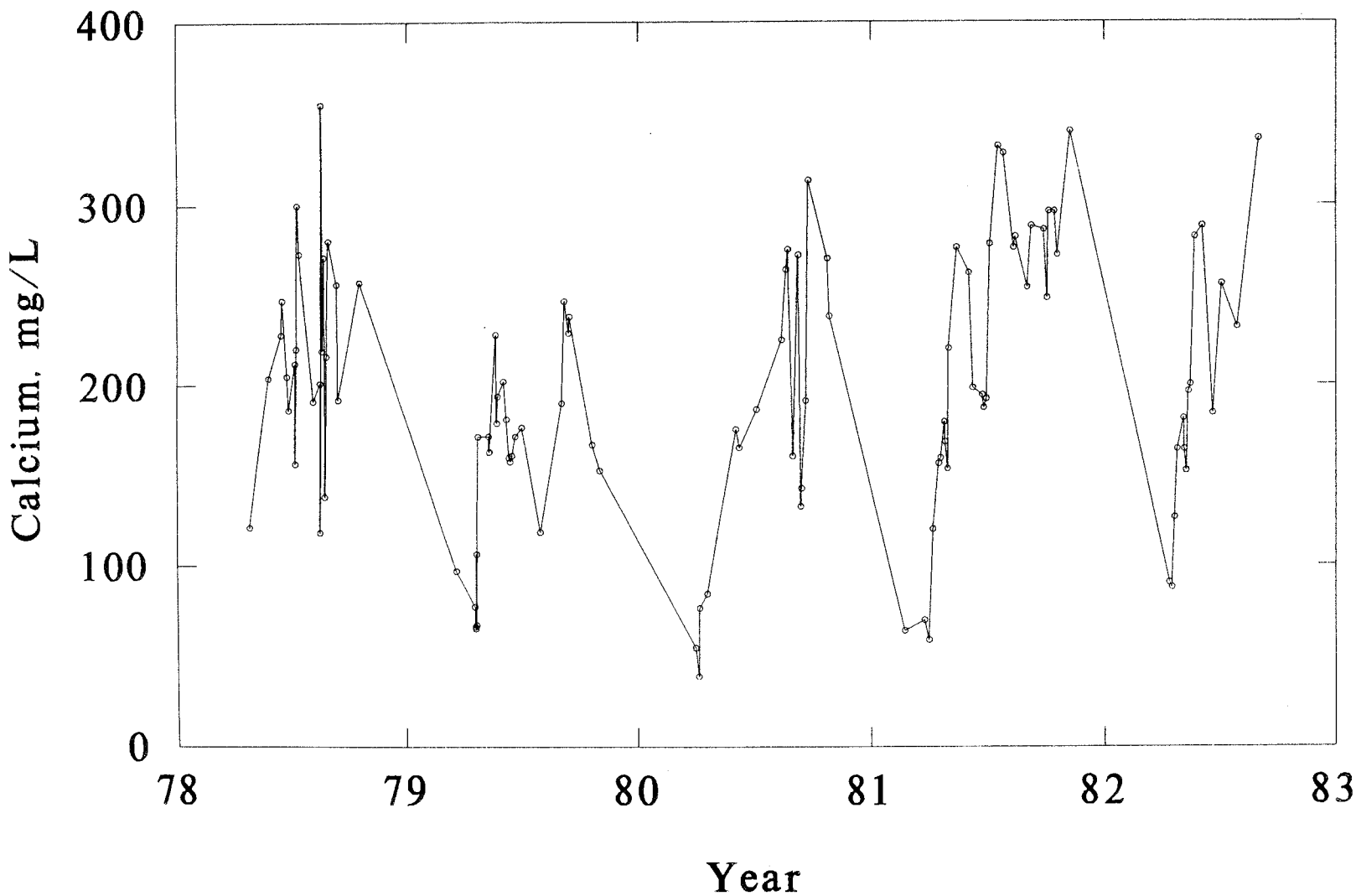

Figure A2.16. Magnesium concentrations over time for Duluth Complex test pile FL4 (0.63% S).

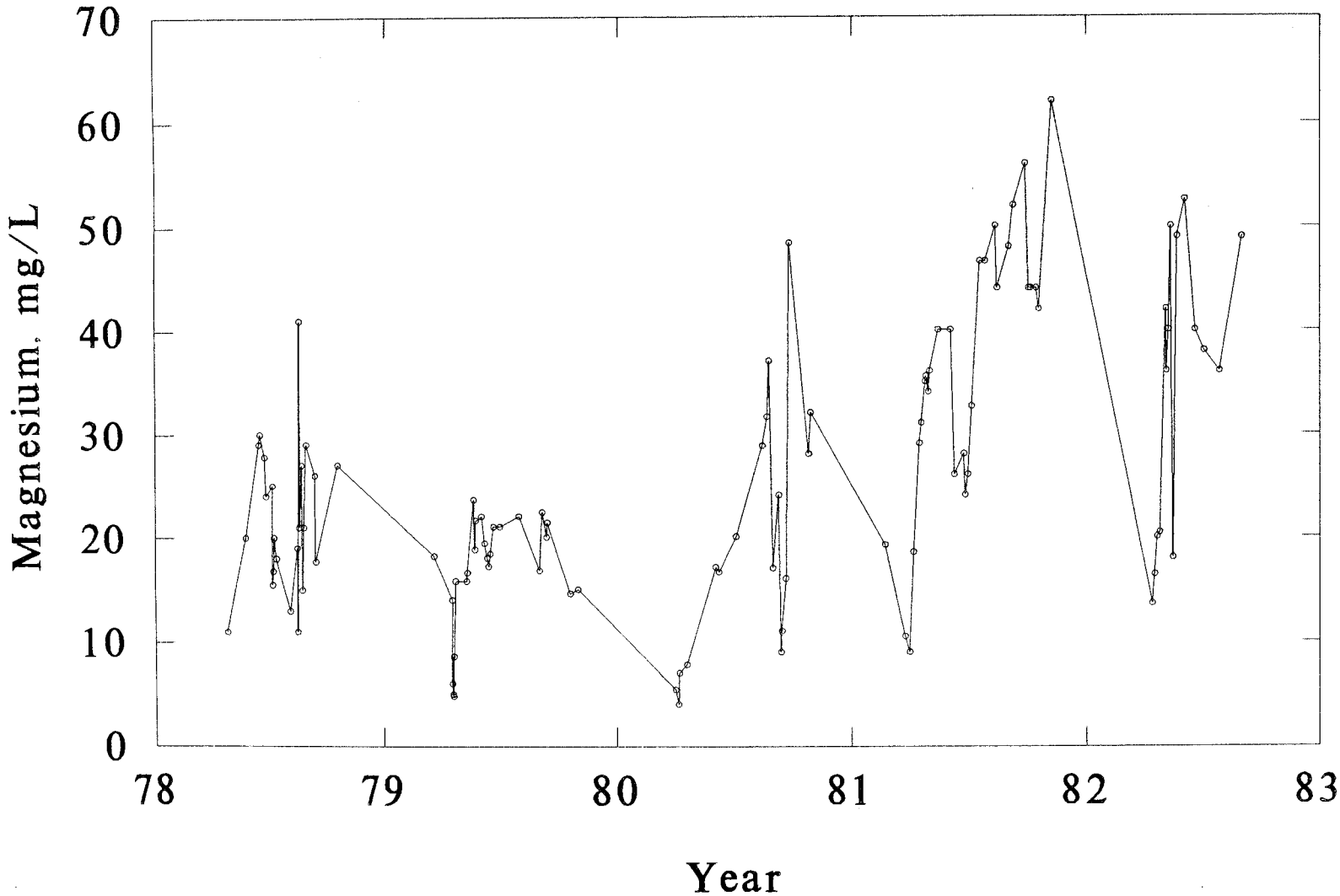

Figure A2.17. pH over time for Duluth Complex test pile FL6 (0.79% S).

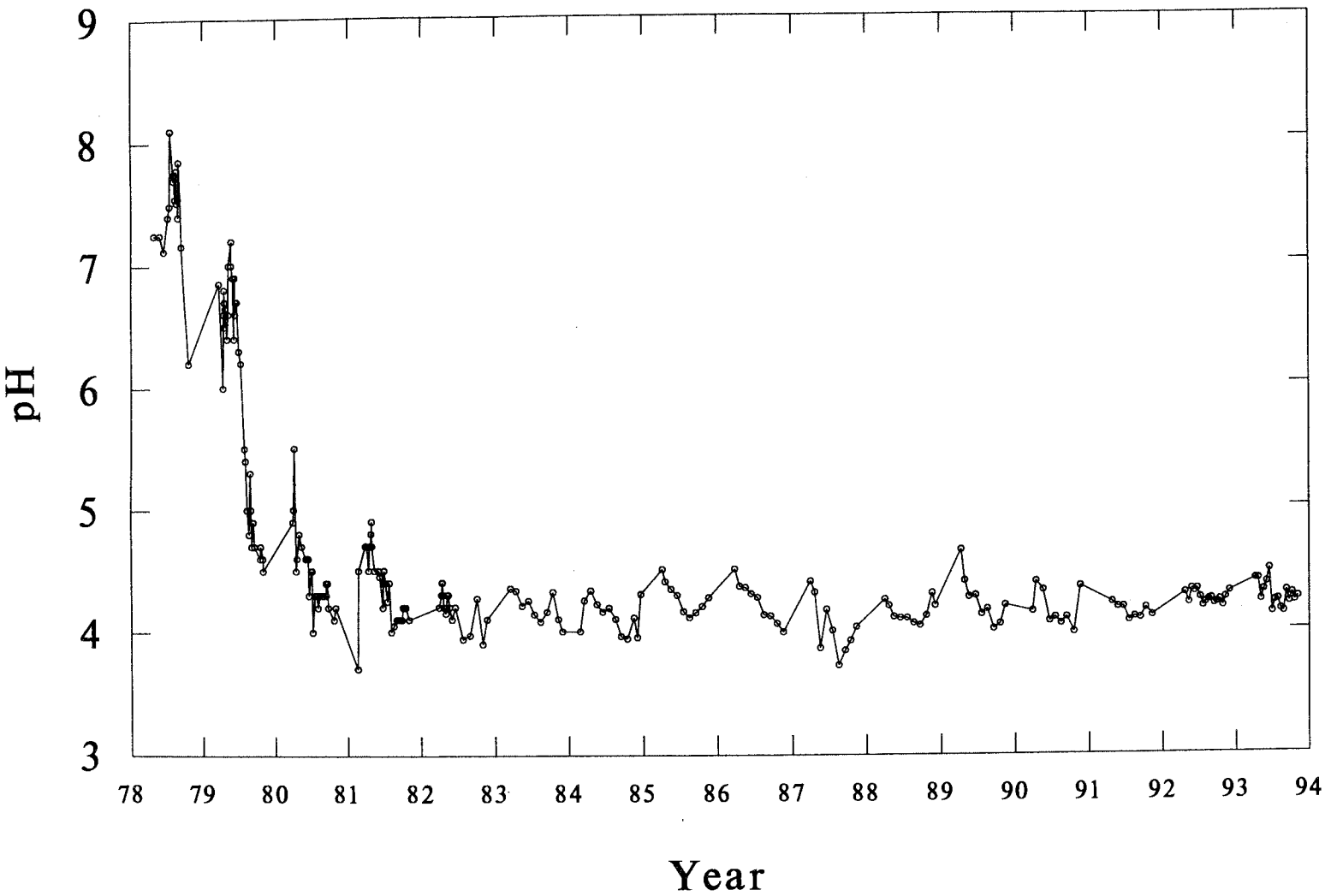

Figure A2.18. Sulfate concentrations over time for Duluth Complex test pile FL6 (0.79% S).

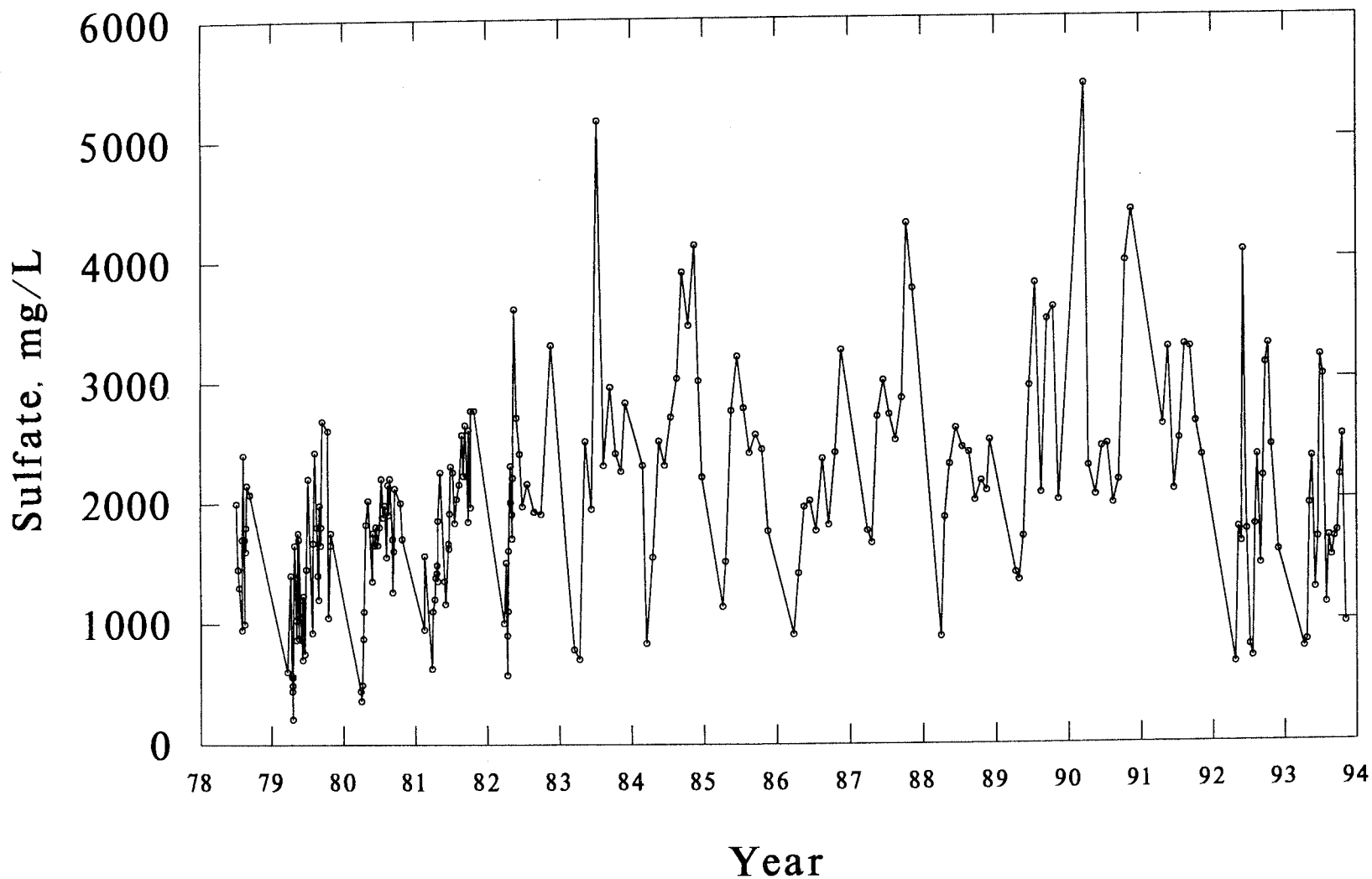

Figure A2.19. Calcium concentrations over time for Duluth Complex test pile FL6 (0.79% S).

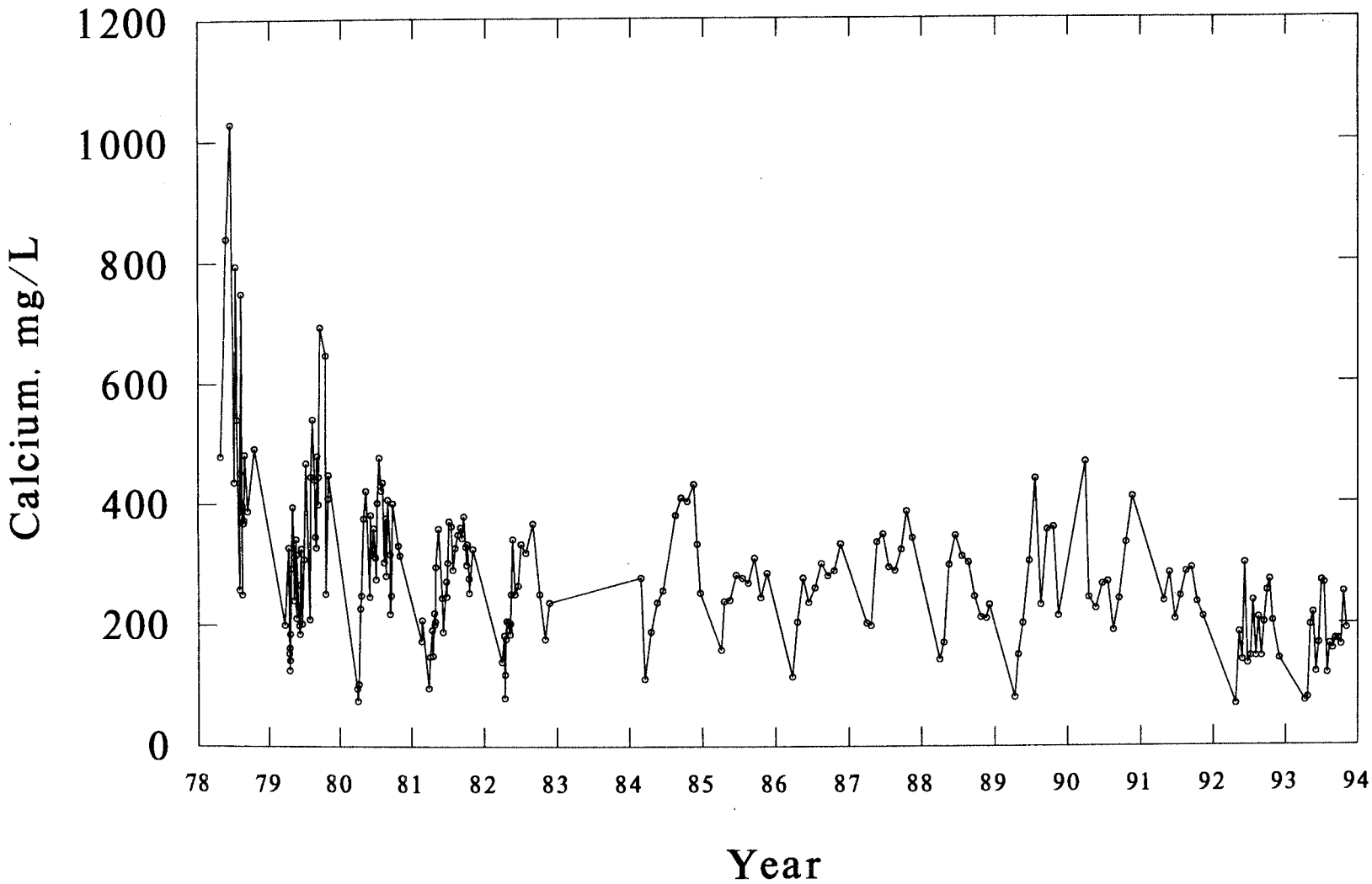

Figure A2.20. Magnesium concentrations over time for Duluth Complex test pile FL6 (0.79% S).

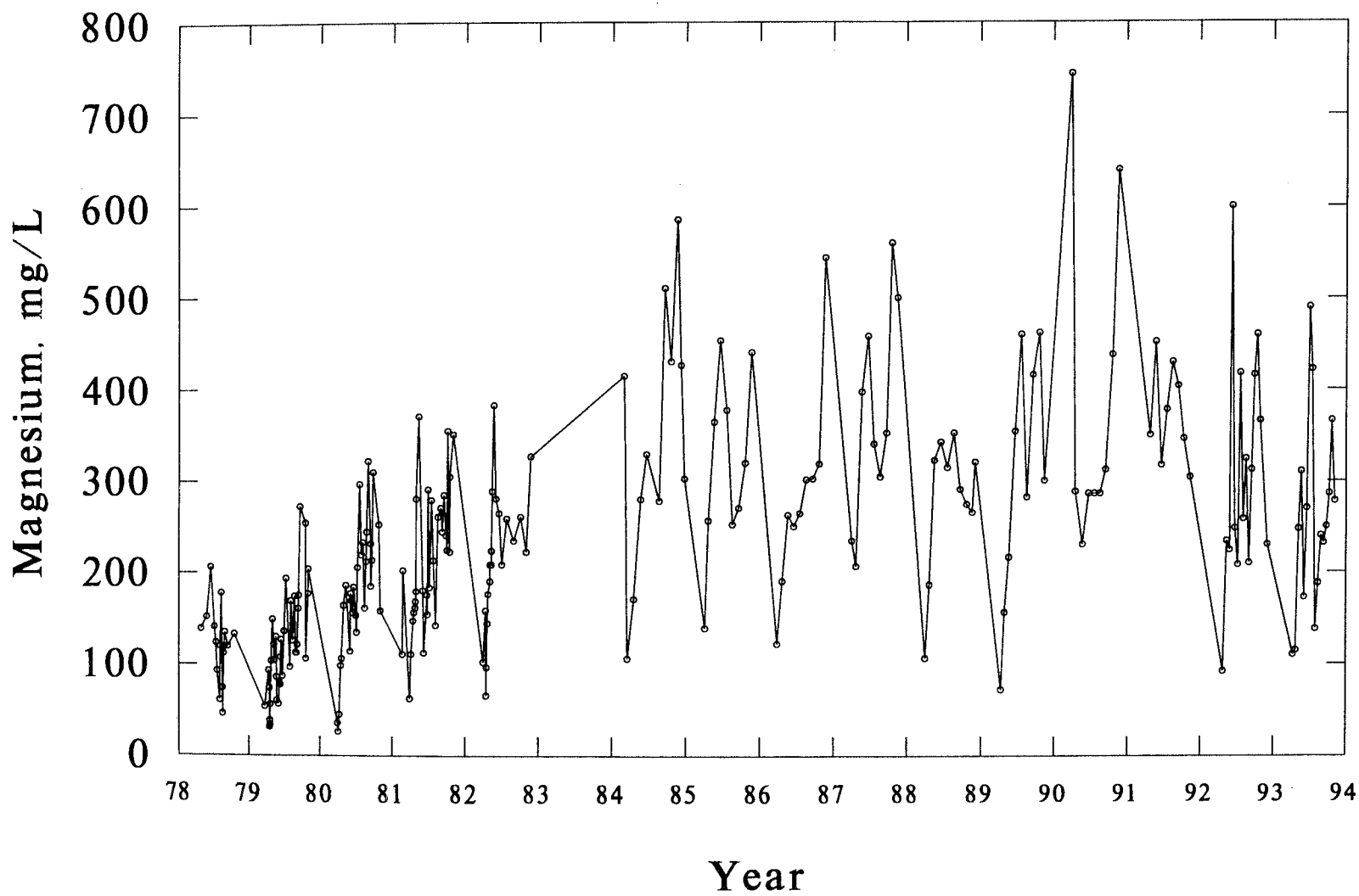

Figure A2.21. pH over time for Duluth Complex test pile FL5 (1.41% S).

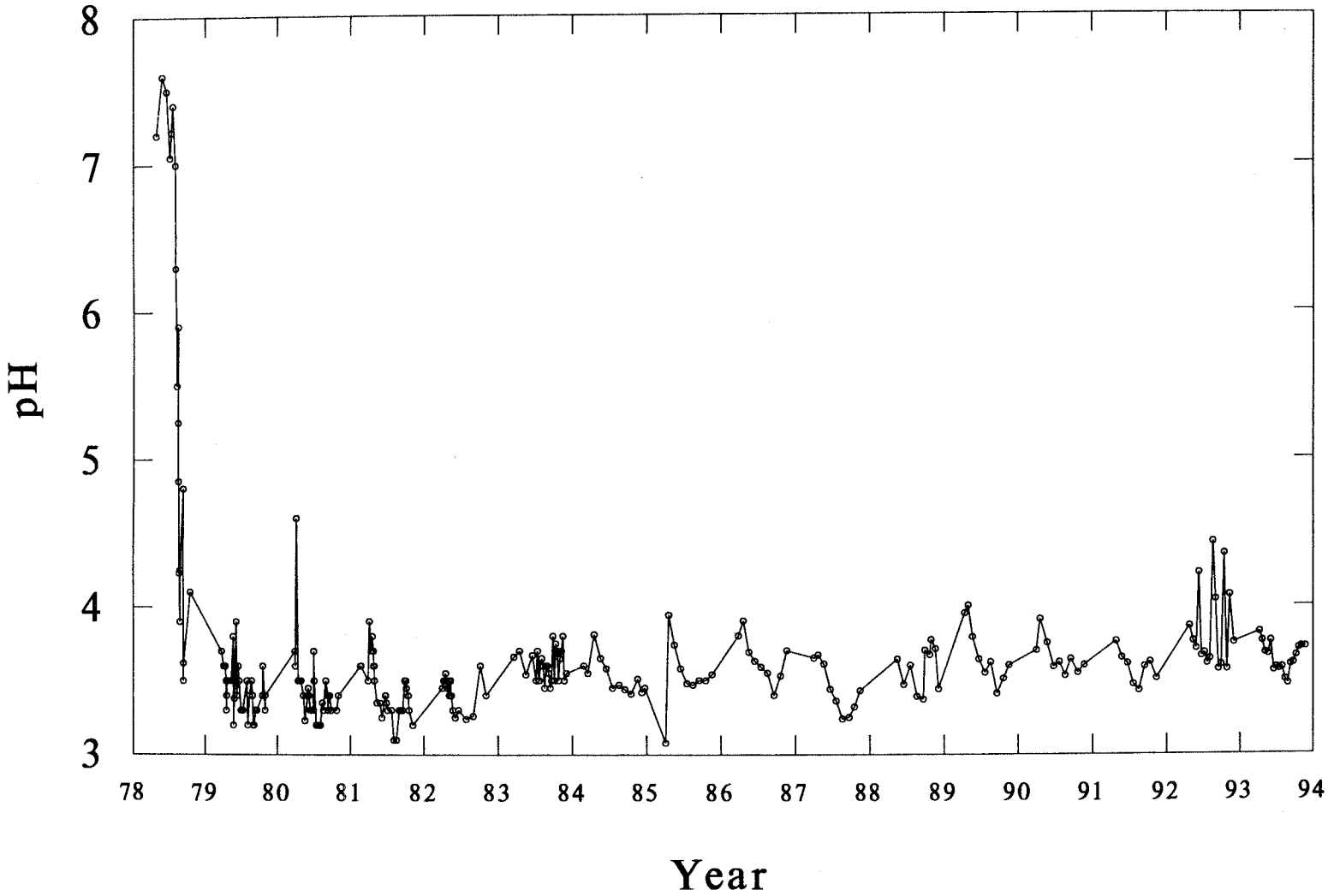

Figure A2.22. Sulfate concentrations over time for Duluth Complex test pile FL5 (1.41% S).

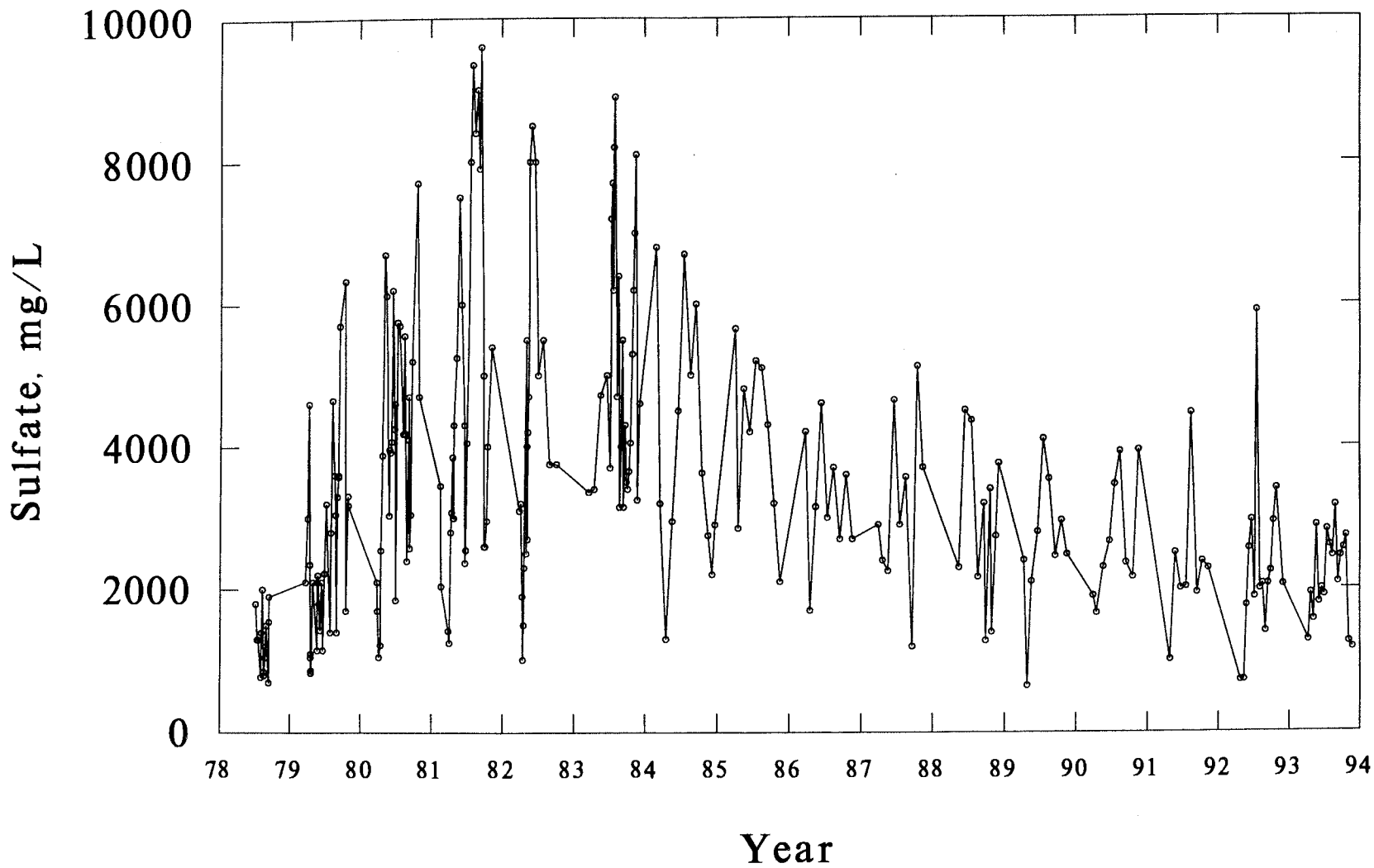

Figure A2.23. Calcium concentrations over time for Duluth Complex test pile FL5 (1.41% S).

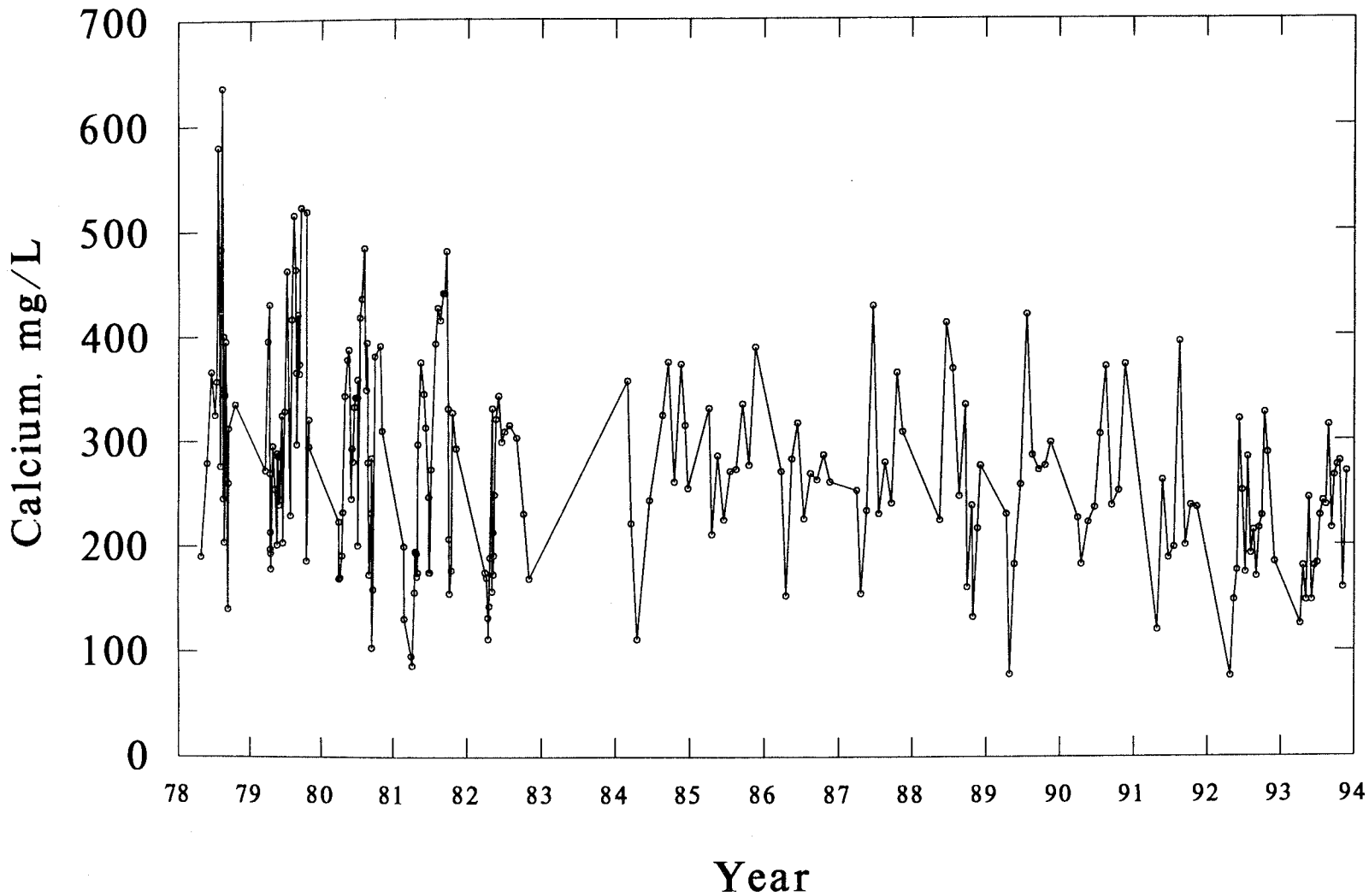

Figure A2.24. Magnesium concentrations over time for Duluth Complex test pile FL5 (1.41% S).

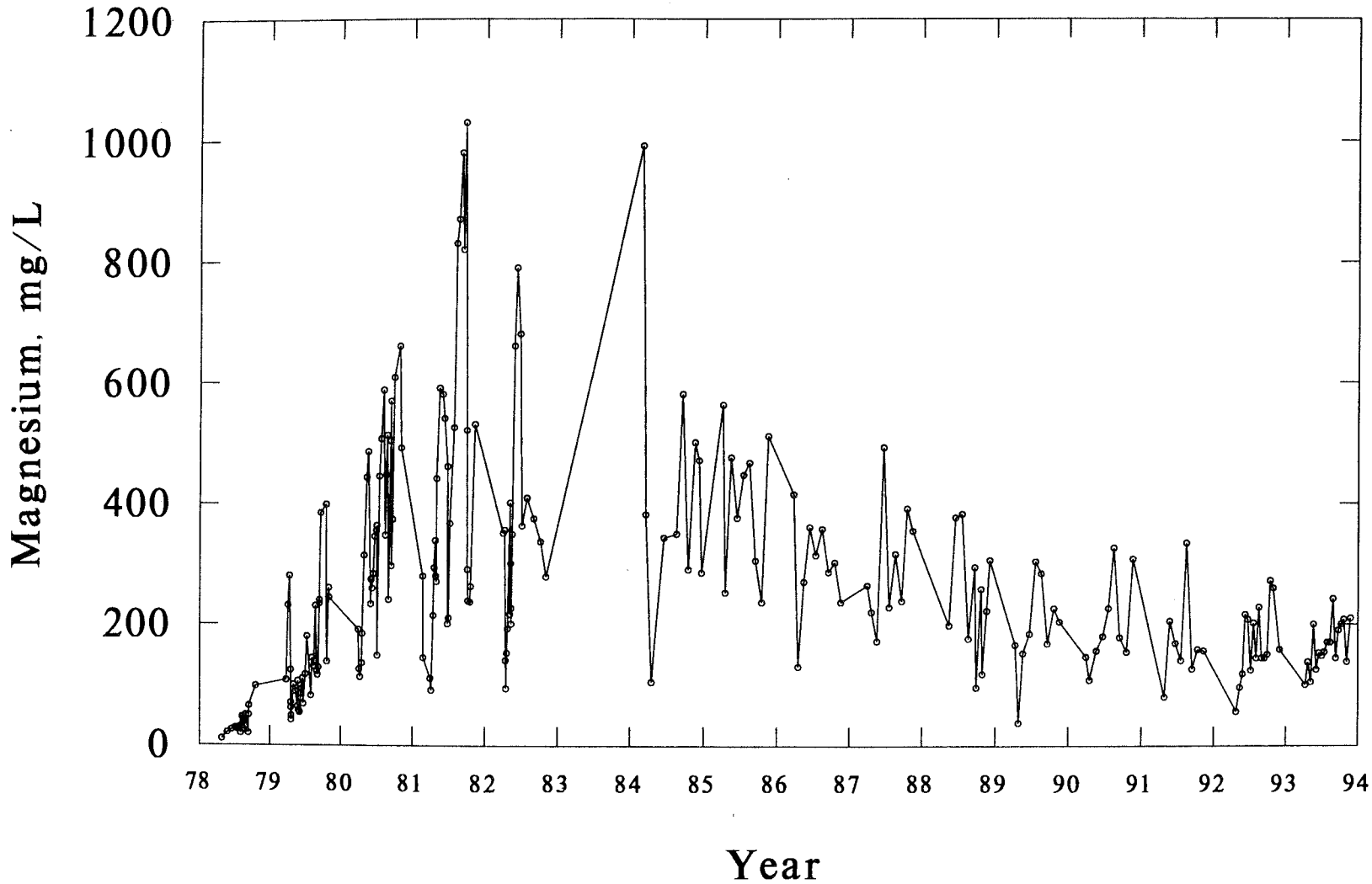

Figure A2.25. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual volume for Duluth Complex test pile FL1 (0.63% S).

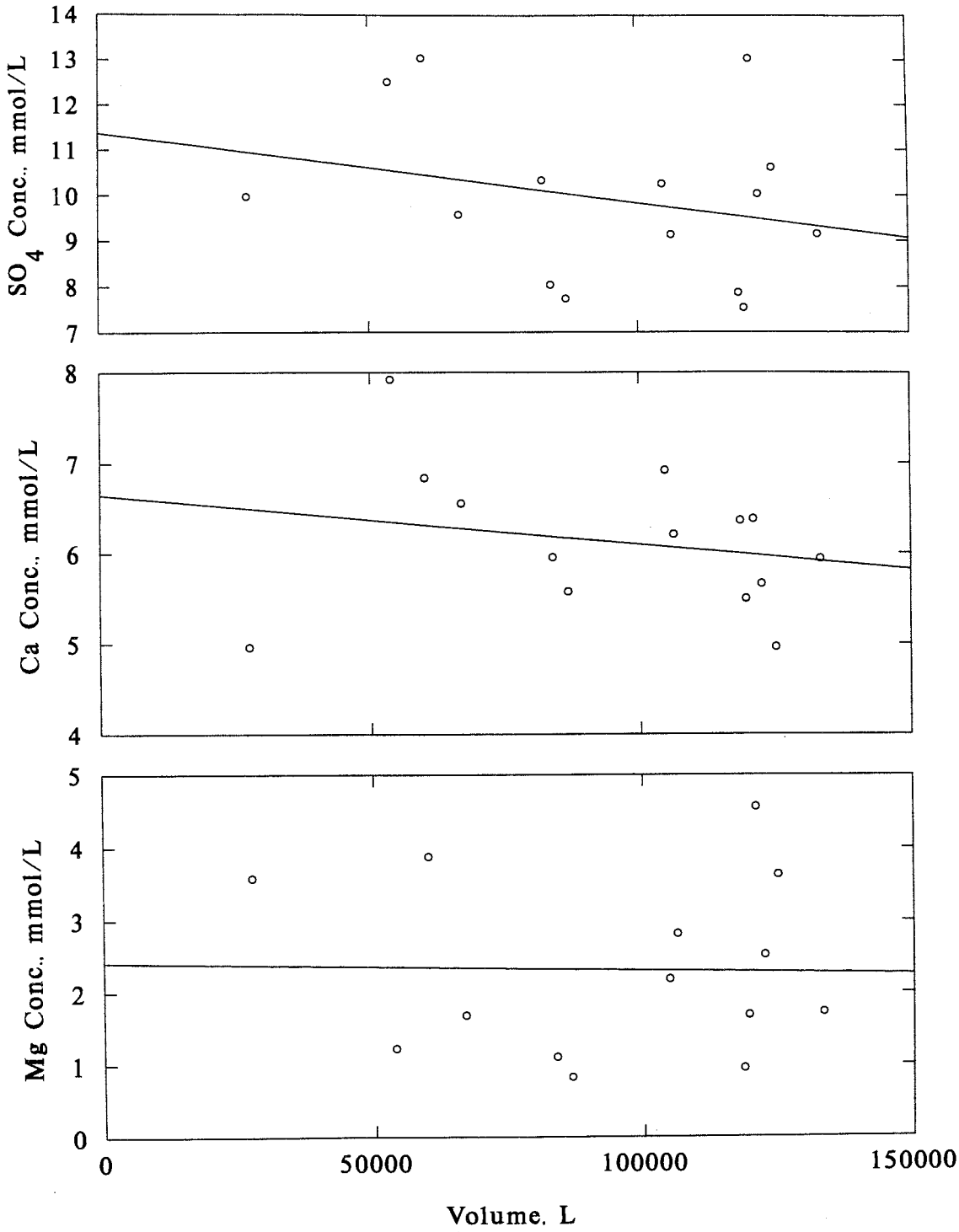

Figure A2.26. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual volume for Duluth Complex test pile FL1 (0.63% S).

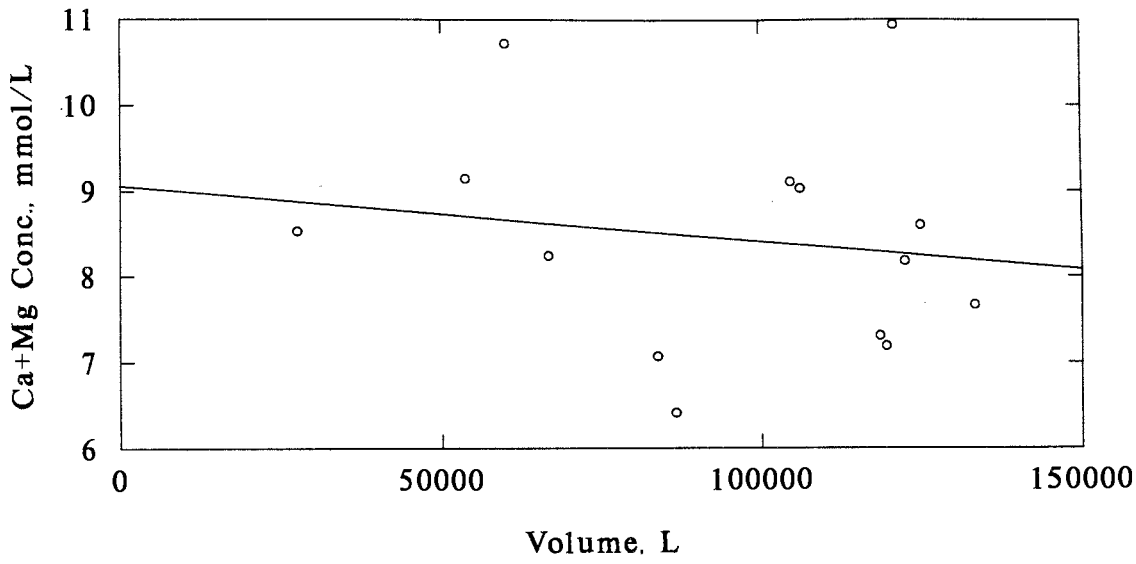

Figure A2.27. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual volume for Duluth Complex test pile FL2 (0.63% S).

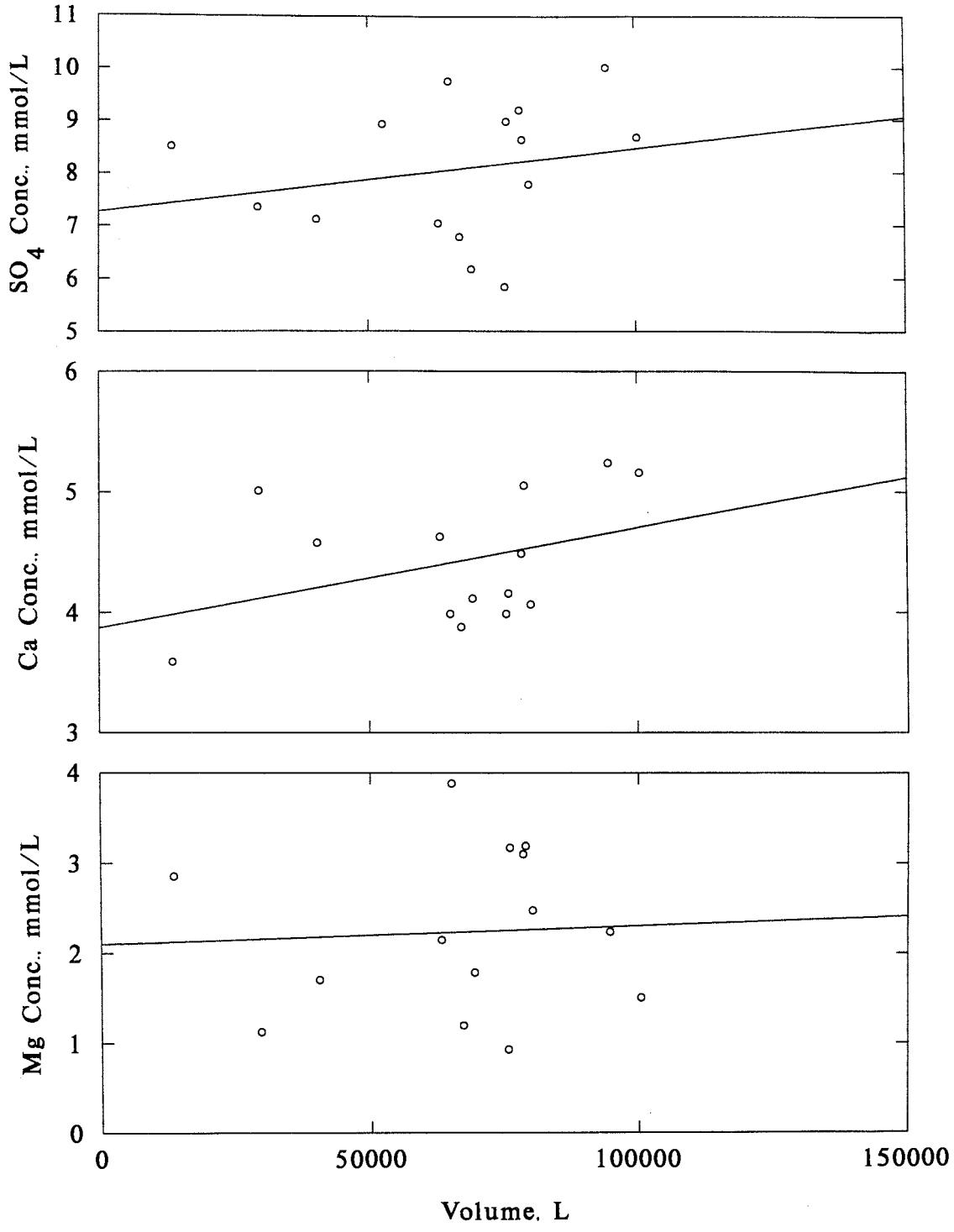

Figure A2.28. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual volume for Duluth Complex test pile FL2 (0.63% S).

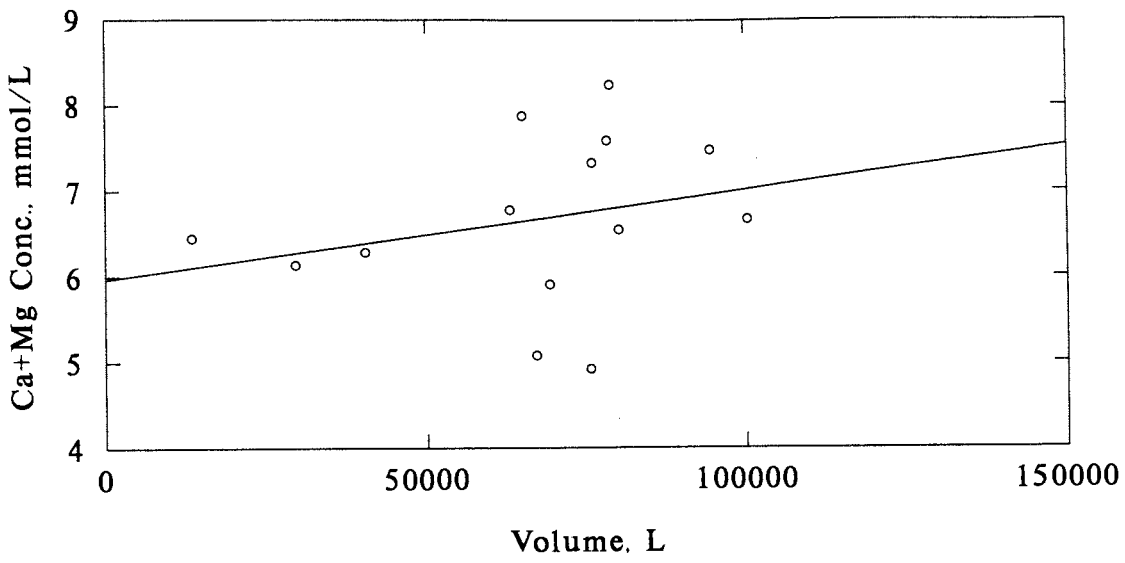

Figure A2.29. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL3 (0.63% S).

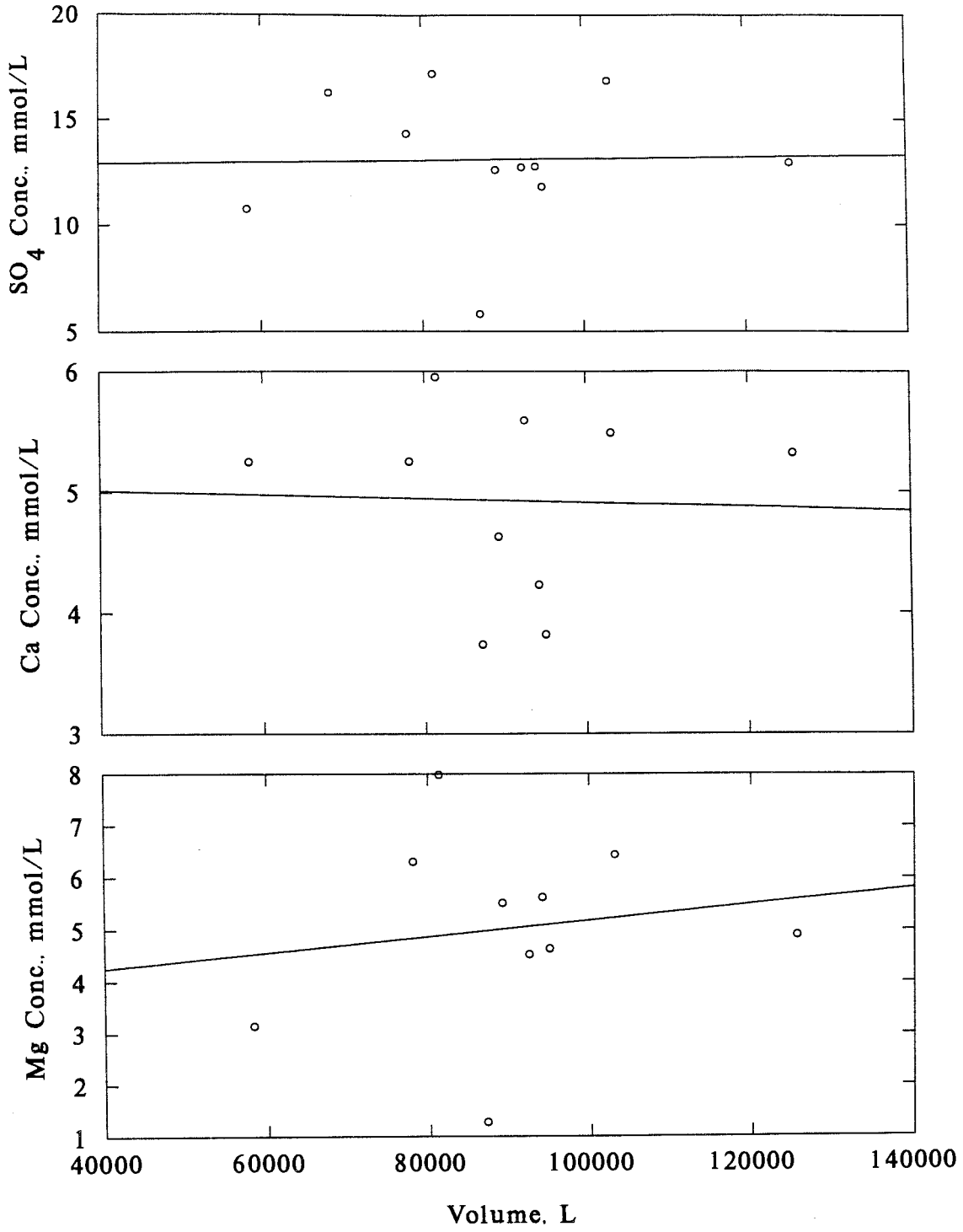

Figure A2.30. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL3 (0.63% S).

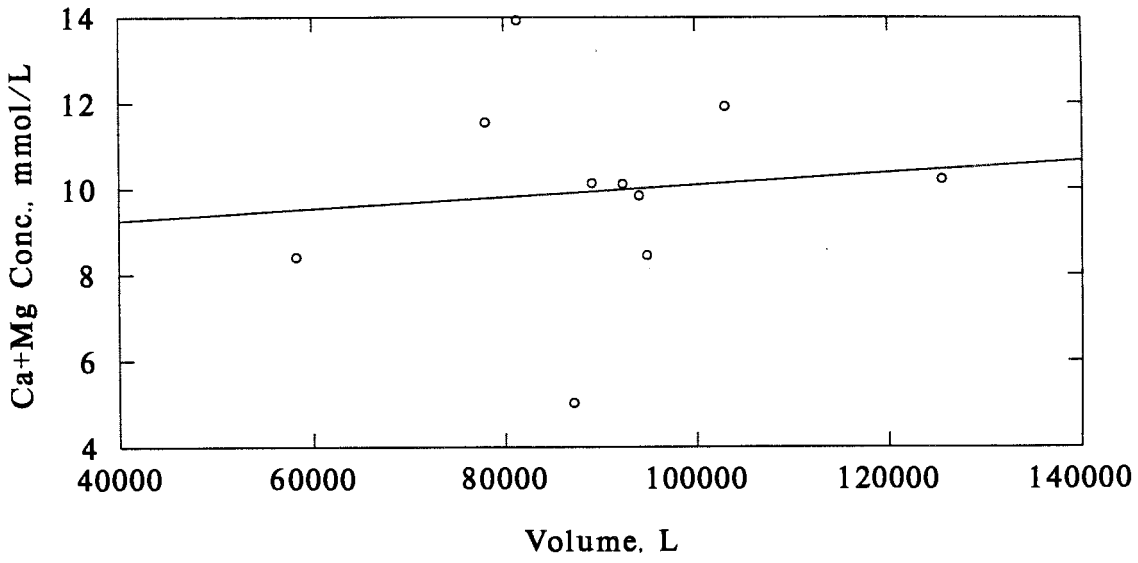

Figure A2.31. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual volume for Duluth Complex test pile FL6 (0.79% S).

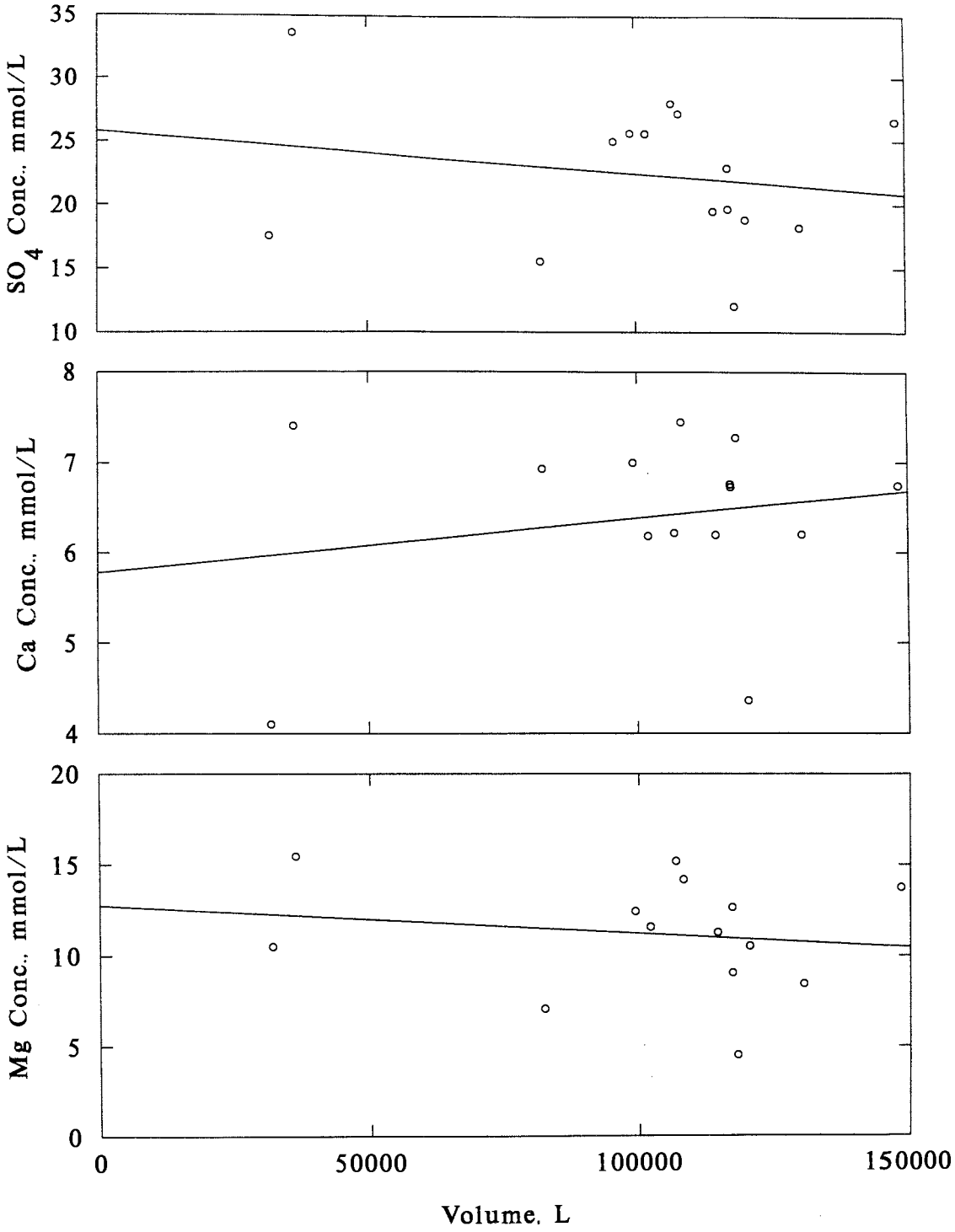

Figure A2.32. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual volume for Duluth Complex test pile FL6 (0.79% S).

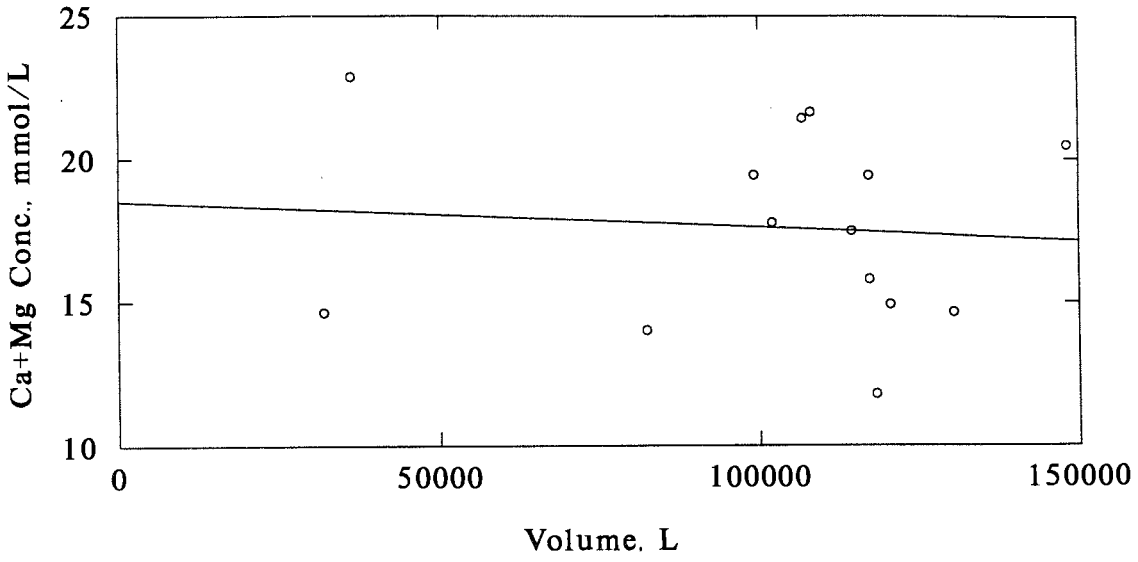

Figure A2.33. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual volume for Duluth Complex test pile FL5 (1.41% S).

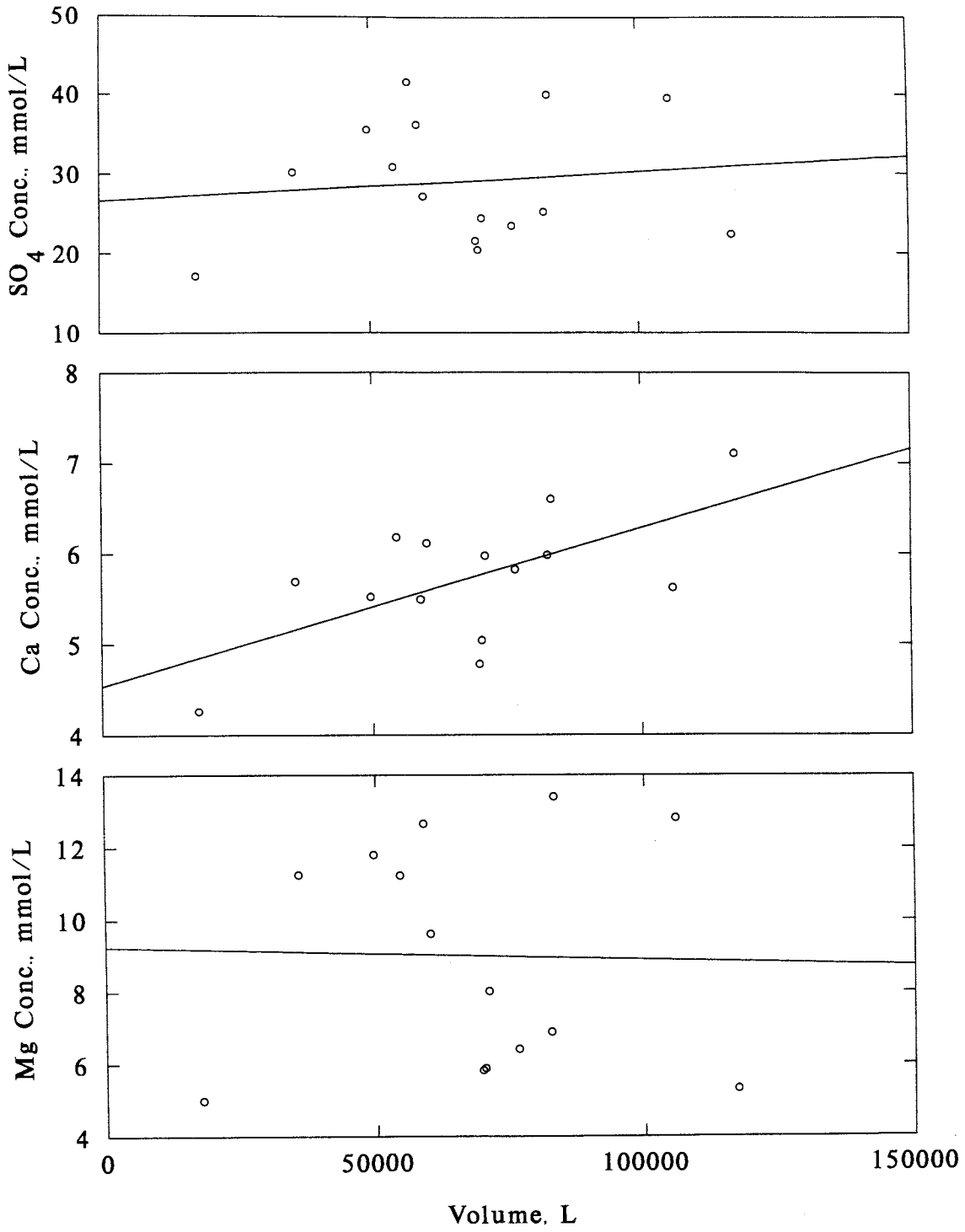

Figure A2.34. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual volume for Duluth Complex test pile FL5 (1.41% S).

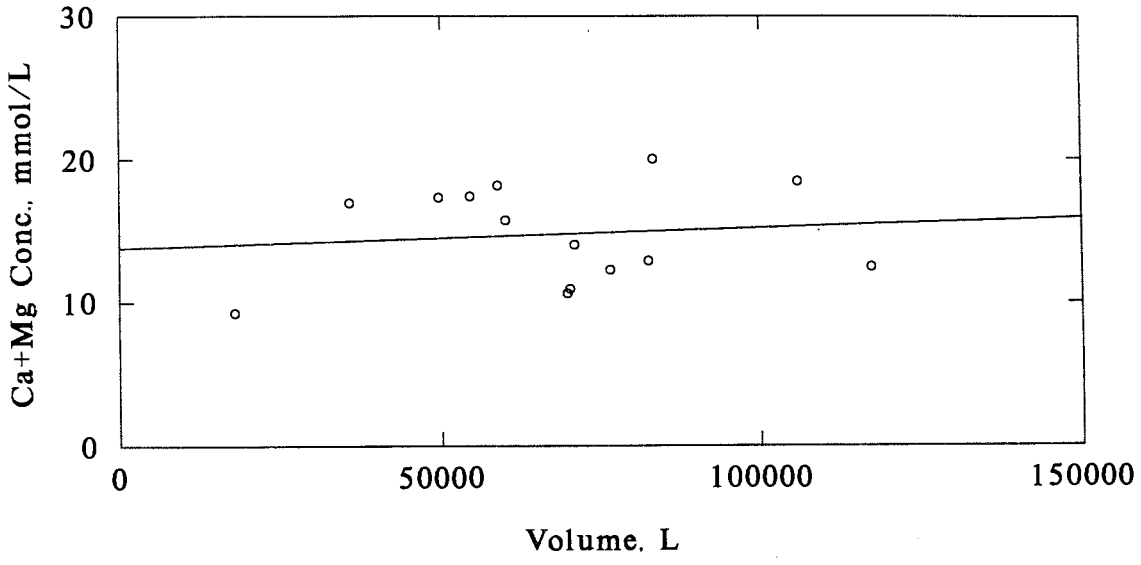

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.15	NA	1120	NA	112.0	11.0	113.0	13.0	0.032	NA	0.003	0.017	NA	0.06	NA	NA	NA	NA	NA	NA	0	9.0
06/06/78	NA	NA	1900	NA	216.0	19.0	268.0	20.0	0.030	0.250	0.009	0.002	0.02	0.13	NA	NA	NA	NA	NA	2558	0	7.0
06/07/78	7.60	NA	3325	NA	253.0	23.0	327.0	23.0	0.041	0.272	0.005	0.030	0.03	0.15	NA	NA	NA	NA	NA	NA	0	NA
06/16/78	7.84	NA	2700	NA	244.0	22.0	368.0	22.0	0.028	0.155	0.001	0.002	0.05	0.14	NA	NA	NA	NA	NA	120	0	11.0
06/23/78	7.72	NA	2042	1267	183.3	17.3	288.3	21.3	0.025	0.251	0.012	0.003	0.02	0.09	5.0	NA	NA	NA	NA	170	0	13.0
06/26/78	6.85	NA	2400	NA	195.0	20.0	313.0	21.0	0.024	0.294	0.014	0.010	0.04	0.09	NA	NA	NA	NA	NA	59	0	15.0
07/05/78	7.75	NA	2100	900	202.0	20.0	206.0	28.0	0.018	0.206	0.007	NA	0.01	0.09	4.0	NA	NA	NA	NA	177	0	14.0
07/07/78	7.90	NA	1900	800	221.0	23.0	222.0	34.0	0.024	0.285	0.019	0.005	0.04	0.09	4.0	5.7	0.44	0.08	NA	NA	0	19.0
07/11/78	7.70	NA	2450	1150	303.0	26.0	286.0	38.0	0.027	0.388	0.026	0.010	0.03	0.14	5.0	5.7	0.64	0.08	NA	NA	0	NA
07/13/78	7.92	NA	NA	1500	531.0	31.0	395.0	45.0	0.027	0.405	0.026	NA	0.01	0.16	4.0	6.2	0.55	0.08	NA	NA	0	NA
07/18/78	7.80	NA	1400	1050	340.0	19.0	220.0	34.0	0.023	0.255	0.019	0.003	0.01	0.11	1.0	3.9	0.50	0.08	NA	NA	0	NA
08/02/78	7.90	NA	1100	650	176.0	11.0	127.0	25.0	0.027	0.210	0.010	0.010	0.01	0.09	1.0	3.4	0.70	0.08	NA	NA	0	22.0
08/04/78	7.80	NA	1650	1200	300.0	23.0	209.0	37.0	0.034	0.330	0.016	0.013	0.03	0.13	3.0	5.0	0.06	0.10	NA	167	0	22.0
08/10/78	8.10	70.0	2400	1600	414.0	26.0	317.0	46.0	0.030	0.460	0.030	0.025	0.02	0.16	4.0	NA	0.10	0.08	NA	70	0	24.0
08/16/78	7.85	42.0	1538	640	169.5	14.0	102.0	25.0	0.020	0.185	0.020	0.015	0.02	0.07	2.0	2.5	NA	0.08	NA	3441	0	19.5
08/24/78	8.10	50.0	2600	1400	302.0	26.0	240.0	40.0	0.040	0.360	0.040	0.020	0.06	0.14	1.0	5.0	NA	0.08	NA	NA	0	NA
08/25/78	7.55	47.0	2500	1300	300.0	26.0	229.0	43.0	0.040	0.350	0.030	0.020	0.02	0.14	1.0	4.6	0.40	0.20	NA	NA	0	22.0
08/28/78	7.85	59.0	2700	NA	308.0	26.0	280.0	40.0	0.050	0.420	0.070	0.020	0.04	0.15	NA	NA	NA	NA	NA	NA	0	NA
09/13/78	7.58	40.5	2088	1050	246.0	19.0	160.0	39.5	0.045	0.275	0.020	0.010	0.03	0.12	1.5	NA	NA	0.08	NA	426	0	14.0
10/17/78	7.40	41.0	3207	1800	485.0	42.0	322.0	41.0	0.050	0.540	0.070	0.040	0.02	0.17	8.0	6.5	0.64	0.20	NA	NA	0	NA
03/19/79	7.45	21.0	807	NA	164.0	13.2	43.0	13.0	0.030	0.330	0.040	0.040	0.02	0.08	3.0	NA	NA	NA	2.9	2631	1986	NA
04/13/79	7.10	41.0	693	335	109.0	9.4	36.8	11.2	0.028	0.210	0.012	0.030	0.02	0.05	4.0	NA	NA	NA	3.6	2616	390	NA
04/18/79	7.20	33.0	436	190	78.0	5.6	27.0	6.0	0.023	0.106	0.008	0.020	0.02	0.04	1.0	1.3	NA	NA	NA	3665	3966	NA
04/19/79	7.10	35.0	360	165	69.0	4.8	27.0	5.0	0.025	0.102	0.008	0.010	0.02	0.03	2.0	0.8	0.12	NA	NA	4652	4029	NA
04/20/79	6.60	39.0	534	250	100.0	7.8	55.0	7.2	0.025	0.138	0.080	0.020	0.02	0.05	2.0	1.0	0.10	NA	3.1	3664	2274	NA
04/22/79	7.65	27.0	1194	633	185.9	18.1	69.8	15.5	0.028	0.315	0.020	0.040	0.02	0.09	4.0	2.2	0.23	NA	NA	4153	1200	NA
05/09/79	7.50	NA	983	483	173.0	15.2	60.6	14.8	0.025	0.250	0.015	0.020	0.02	0.08	4.0	NA	NA	NA	NA	685	815	NA
05/10/79	7.30	NA	1057	467	162.8	15.6	57.6	15.4	0.027	0.260	0.015	0.030	0.01	0.10	3.0	NA	NA	NA	NA	6572	2979	NA
05/20/79	7.10	NA	1144	667	179.6	17.8	75.6	16.2	0.031	0.320	0.020	0.030	0.03	0.10	3.0	NA	NA	NA	6.9	2991	784	NA
05/22/79	7.70	NA	897	533	144.4	13.8	60.4	14.0	0.027	0.220	0.015	0.020	0.01	0.09	1.0	NA	NA	NA	6.0	1386	848	NA
05/23/79	7.50	NA	1720	900	232.4	23.6	118.4	21.4	0.031	0.340	0.023	0.030	0.02	0.13	3.0	NA	NA	NA	5.0	2400	596	NA
06/01/79	7.30	NA	1185	700	170.2	17.2	88.8	18.4	0.033	0.290	0.020	0.020	0.03	0.12	2.0	NA	NA	NA	6.9	2468	737	NA
06/06/79	7.50	NA	1204	675	174.0	17.0	53.0	21.2	0.028	0.260	0.017	0.010	0.01	0.09	3.0	NA	NA	NA	NA	2230	941	NA
06/10/79	7.40	54.0	1168	580	166.0	16.4	49.9	20.4	0.031	0.225	0.011	0.010	0.01	0.09	3.0	NA	NA	NA	NA	2143	1098	20.0
06/12/79	7.20	45.0	1687	780	245.8	24.0	67.0	32.8	0.028	0.280	0.013	0.020	0.01	0.12	4.0	NA	NA	NA	NA	1181	416	NA
06/15/79	7.40	48.4	1488	740	234.8	23.0	68.4	28.6	0.028	0.270	0.009	0.010	0.01	0.11	5.0	NA	NA	NA	NA	545	494	20.0
06/21/79	7.10	30.0	1030	450	144.0	15.0	60.0	19.0	0.020	0.170	0.012	0.010	0.04	0.07	2.0	NA	NA	NA	NA	2343	1073	15.0
06/30/79	7.40	NA	1349	1280	338.0	33.0	160.0	32.0	0.030	0.370	0.025	0.020	0.02	0.11	4.0	NA	NA	NA	12.0	488	149	19.0
07/10/79	7.60	37.0	2120	1600	371.0	37.0	164.0	37.0	0.030	0.380	0.025	0.020	0.01	0.15	4.0	NA	NA	NA	11.0	1132	137	18.0
07/30/79	7.40	20.8	1221	589	208.5	18.5	50.0	22.0	0.020	0.210	0.014	0.010	0.01	0.07	3.0	NA	NA	NA	6.3	3449	2274	23.0
08/04/79	7.60	26.8	2094	1520	439.0	34.0	106.0	41.0	0.020	0.390	0.025	0.020	0.02	0.12	3.0	NA	NA	NA	6.1	1314	212	23.0
08/13/79	7.30	31.2	2562	1920	488.8	49.0	260.0	52.0	0.040	0.530	0.023	0.030	0.03	0.17	5.0	NA	NA	NA	11.0	345	47	13.0
08/23/79	7.30	24.8	1976	1500	394.8	33.2	174.0	38.6	0.030	0.430	0.023	0.030	0.02	0.13	5.0	NA	NA	NA	9.1	371	151	9.0
08/28/79	7.30	18.0	1642	1130	298.0	25.4	95.0	29.8	0.040	0.320	0.015	0.030	0.05	0.09	28.0	NA	NA	NA	8.3	548	458	20.0
09/01/79	7.50	23.2	2014	1250	364.0	31.0	80.2	33.0	0.040	0.450	0.020	0.030	0.03	0.10	5.0	NA	NA	NA	7.8	3710	1929	22.0
09/05/79	7.40	25.2	2145	1520	427.2	39.0	134.4	42.2	0.040	0.490	0.008	0.030	0.02	0.12	4.0	NA	NA	NA	10.0	742	550	18.0
09/10/79	7.35	24.0	1794	1290	343.4	29.2	135.0	35.9	0.035	0.415	0.019	0.030	0.02	0.10	5.0	NA	NA	NA	8.9	2608	516	17.5
09/13/79	7.50	24.0	2112	1600	428.2	37.6	165.6	41.0	0.050	0.520	0.023	0.030	0.01	0.12	5.0	NA	NA	NA	9.3	1408	500	16.0
09/19/79	7.40	25.2	3010	2200	590.8	52.0	232.0	53.2	0.050	0.660	0.060	0.040	0.03	0.12	8.0	NA	NA	NA	12.0	886	141	11.5
10/18/79	7.10	20.8	2603	1900	502.0	45.4	204.0	46.4	0.050	0.640	0.030	0.040	0.01	0.12	11.0	NA	NA	NA	11.0	352	733	NA
10/19/79	6.80	19.2	1318	750	223.0	19.6	56.0	20.8	0.040	0.460	0.021	0.030	0.01	0.05	3.0	NA	NA	NA	5.9	821	1222	NA

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
10/31/79	6.80	20.8	1959	1300	374.0	32.6	90.0	30.8	0.041	0.740	0.026	0.040	0.02	0.10	3.0	NA	NA	NA	7.2	4315	1513	NA
11/01/79	7.20	24.8	2364	1000	412.0	38.2	114.0	34.2	0.030	0.820	0.060	0.050	0.02	0.10	6.0	NA	NA	NA	8.5	2695	966	10.0
04/17/80	7.25	32.2	790	364	130.2	8.6	21.8	8.4	0.030	0.260	0.030	0.030	0.02	0.06	1.0	NA	NA	NA	NA	314	454	NA
04/18/80	6.95	38.0	NA	550	145.4	11.2	28.6	10.8	0.020	0.040	0.020	0.050	0.02	0.05	2.0	NA	NA	NA	2.4	927	907	NA
04/23/80	7.45	45.4	NA	490	148.0	11.6	22.6	11.0	0.010	0.390	0.040	0.050	0.02	0.06	2.0	NA	NA	NA	2.2	927	216	NA
04/27/80	7.50	52.7	1442	1045	233.7	20.8	58.5	17.0	0.020	0.505	0.040	0.040	0.03	0.12	3.0	NA	NA	NA	NA	2131	157	12.5
05/05/80	7.60	45.1	1330	1020	227.0	18.6	49.8	14.8	0.020	0.370	0.040	0.030	0.02	0.12	3.0	NA	NA	NA	4.5	420	201	11.0
05/18/80	7.60	46.2	1300	910	218.4	20.6	71.4	20.0	0.030	0.400	0.040	0.030	0.01	0.15	9.0	NA	NA	NA	NA	806	50	19.0
05/31/80	7.30	30.8	950	650	159.3	14.5	41.1	16.0	0.030	0.300	0.012	0.020	0.02	0.07	5.0	NA	NA	NA	6.3	1798	1190	NA
06/01/80	7.50	36.9	1575	1020	276.2	24.2	57.5	23.6	0.060	0.520	0.022	0.030	0.01	0.10	5.0	NA	NA	NA	6.2	893	441	NA
06/07/80	7.45	32.0	1475	800	283.6	25.4	67.2	25.6	0.020	0.450	0.014	0.020	0.03	0.02	5.0	NA	NA	NA	NA	799	189	NA
06/14/80	7.40	29.7	1700	1170	274.0	25.0	70.0	26.0	0.030	0.460	0.029	0.020	0.06	0.10	5.0	NA	NA	NA	NA	579	139	NA
06/18/80	7.30	31.9	2200	1450	348.0	23.7	101.0	32.8	0.040	0.530	0.012	0.030	0.03	0.12	6.0	NA	NA	NA	NA	367	46	NA
06/27/80	7.30	25.3	1700	1150	273.0	26.0	72.0	28.0	0.043	0.039	0.036	0.030	0.01	0.08	5.0	NA	NA	NA	8.2	996	245	NA
07/02/80	7.30	33.0	2300	2100	422.0	32.0	99.0	33.0	0.024	0.660	0.034	0.050	0.03	0.11	5.0	NA	NA	NA	8.8	1060	1272	NA
07/04/80	7.20	19.6	1850	NA	354.0	29.0	65.0	27.0	0.030	0.670	0.040	0.040	0.02	0.01	NA	NA	NA	NA	NA	727	1017	NA
07/17/80	7.20	28.6	2500	1590	428.4	34.8	190.3	42.4	0.040	0.630	0.040	0.040	0.03	0.12	5.5	NA	NA	NA	NA	1022	356	NA
07/20/80	7.20	30.8	2100	1300	363.8	33.4	169.0	37.8	0.040	0.550	0.040	0.040	0.02	0.10	6.0	NA	NA	NA	NA	1026	171	NA
08/03/80	7.20	24.2	2100	1150	355.2	34.4	131.6	37.4	0.060	0.540	0.026	0.060	0.04	0.13	6.0	NA	NA	NA	10.0	496	311	NA
08/12/80	7.20	9.8	1500	990	275.0	23.4	57.4	25.2	0.050	0.600	0.080	0.040	0.09	0.10	5.0	NA	NA	NA	6.1	1995	1239	NA
08/19/80	7.10	18.7	2000	1520	393.4	34.6	85.2	34.8	0.050	1.010	0.050	0.050	0.02	0.13	5.0	NA	NA	NA	NA	1643	350	NA
08/22/80	NA	NA	1850	1290	357.2	35.2	76.8	30.0	0.032	1.220	0.060	0.050	0.04	0.11	4.0	NA	NA	NA	NA	1344	657	NA
08/29/80	7.10	23.4	1850	1658	382.2	40.6	94.4	34.0	0.060	0.890	0.060	0.060	0.07	0.10	4.0	NA	NA	NA	7.5	8112	3456	NA
09/07/80	7.00	20.4	1500	920	287.0	26.6	80.6	30.0	0.035	0.660	0.040	0.050	0.04	0.08	4.0	NA	NA	NA	NA	2703	1050	NA
09/09/80	7.20	22.1	1760	1250	382.8	39.2	93.2	33.8	0.036	0.860	0.047	0.050	0.06	0.10	3.0	NA	NA	NA	NA	1147	361	14.0
09/12/80	7.05	20.0	1215	660	262.0	21.0	55.6	22.4	0.038	0.680	0.050	0.040	0.02	0.08	4.0	NA	NA	NA	4.9	3536	1890	16.0
09/18/80	7.20	21.0	1535	1130	340.0	33.2	84.0	29.2	0.046	0.740	0.033	0.050	0.06	0.08	4.0	NA	NA	NA	NA	7896	2008	12.0
09/23/80	7.20	21.3	2117	1350	398.0	45.0	122.0	10.8	0.050	0.840	0.070	0.060	0.04	0.08	4.0	NA	NA	NA	6.9	1938	588	14.0
10/15/80	7.00	20.4	1719	1050	310.0	32.0	87.8	30.0	0.050	0.700	0.040	0.060	0.03	0.07	5.0	NA	NA	NA	7.0	1741	101	8.0
10/31/80	6.50	17.0	1773	1900	340.0	30.0	65.0	29.0	0.070	1.590	0.070	0.090	0.02	0.10	4.0	NA	NA	NA	NA	1257	48	11.0
02/18/81	7.20	18.7	525	266	74.8	7.2	16.2	NA	0.020	0.630	0.030	0.040	NA	NA	1.0	NA	NA	NA	NA	832	722	8.0
03/27/81	7.20	19.2	600	300	80.6	8.3	15.8	NA	0.020	0.710	0.040	0.050	NA	NA	NA	NA	NA	NA	3.0	4742	2270	NA
04/03/81	7.30	27.2	800	440	124.8	13.0	20.8	NA	0.050	1.470	0.060	0.070	NA	NA	NA	NA	NA	NA	NA	4410	1459	NA
04/13/81	6.70	17.6	900	510	143.0	16.0	22.0	NA	0.090	1.290	0.060	0.060	NA	NA	NA	NA	NA	NA	NA	2441	1050	NA
04/17/81	7.40	30.4	1000	515	139.0	16.0	23.0	NA	0.040	1.280	0.050	0.050	NA	NA	NA	NA	NA	NA	3.6	1696	988	NA
04/23/81	7.40	28.8	1350	635	165.0	22.0	30.0	NA	0.040	1.260	0.060	0.060	NA	NA	NA	NA	NA	NA	NA	2350	2515	NA
04/27/81	7.45	29.2	1500	740	203.0	32.0	39.0	NA	0.040	1.350	0.070	0.060	NA	NA	NA	NA	NA	NA	NA	6162	2089	2.0
04/28/81	7.60	36.0	1250	640	168.0	27.0	36.0	NA	0.040	1.070	0.060	0.060	NA	NA	NA	NA	NA	NA	4.3	3785	1698	NA
04/29/81	7.40	37.0	1430	835	230.0	28.0	46.0	NA	0.040	1.050	0.080	0.070	NA	NA	NA	NA	NA	NA	NA	3323	1040	NA
05/12/81	7.55	38.4	1587	800	272.0	32.0	68.0	NA	0.030	0.830	0.040	0.070	NA	NA	NA	NA	NA	NA	6.2	3686	203	17.0
06/01/81	7.50	26.0	1064	450	184.0	20.0	46.0	NA	0.040	0.490	0.030	0.040	NA	NA	NA	NA	NA	NA	NA	2626	790	16.0
06/06/81	7.40	13.3	1202	620	200.0	38.0	56.0	22.0	0.040	0.880	0.050	0.050	NA	NA	NA	NA	NA	NA	3.6	1840	207	18.0
06/22/81	7.00	15.2	1112	590	196.0	18.0	32.0	NA	0.050	1.120	0.060	0.050	NA	NA	NA	NA	NA	NA	NA	4148	2105	NA
06/24/81	7.20	16.0	1216	675	216.0	18.0	30.0	NA	0.060	1.280	0.070	0.050	NA	NA	NA	NA	NA	NA	NA	3274	1541	16.0
06/28/81	7.20	15.2	1143	640	205.0	18.0	32.0	NA	0.040	1.040	0.050	0.040	NA	NA	NA	NA	NA	NA	3.0	4690	2922	15.0
07/04/81	7.40	21.4	1786	1100	348.0	29.6	65.6	NA	0.040	1.100	0.060	0.050	NA	NA	NA	NA	NA	NA	NA	2154	117	21.0
07/15/81	7.50	15.0	1586	1000	284.0	21.6	65.8	NA	0.060	0.670	0.050	0.040	NA	NA	NA	NA	NA	NA	NA	606	40	19.0
07/24/81	7.20	14.2	1464	945	274.0	23.4	48.2	NA	0.070	0.800	0.060	0.050	NA	NA	2.0	NA	NA	NA	3.7	1094	238	19.0
08/04/81	7.50	13.4	1798	1200	356.0	36.0	76.0	NA	0.050	0.980	0.050	0.090	NA	NA	NA	NA	NA	NA	NA	722	75	18.0
08/17/81	7.10	12.2	1936	1200	392.0	36.0	65.0	NA	0.060	1.220	0.080	0.080	NA	NA	NA	NA	NA	NA	NA	602	20	22.0
09/01/81	7.00	9.6	2275	1180	352.0	34.0	56.0	NA	0.140	1.840	0.060	0.140	NA	NA	NA	NA	NA	NA	NA	772	100	NA

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/16/81	6.90	4.8	2700	NA	482.0	48.0	75.0	NA	0.140	2.700	0.150	0.200	NA	NA	NA	NA	NA	NA	NA	148	5	NA
09/26/81	6.75	5.6	1850	1120	344.0	36.0	42.0	NA	0.270	4.300	0.190	0.230	NA	NA	NA	NA	NA	NA	6.7	1037	587	NA
10/01/81	6.90	6.4	1800	940	320.0	54.0	32.0	NA	0.390	6.500	0.290	0.300	NA	NA	NA	NA	NA	NA	NA	3043	1895	NA
10/04/81	6.90	8.8	1700	1020	358.0	32.0	40.0	30.0	0.280	5.750	0.230	0.260	NA	NA	NA	NA	NA	NA	NA	2642	1354	NA
10/13/81	7.05	10.8	1500	800	262.0	26.0	40.0	26.0	0.140	2.980	0.140	0.160	NA	NA	NA	NA	NA	NA	3.5	3622	1799	NA
10/17/81	7.00	11.2	1700	1010	336.0	32.0	50.0	32.0	0.170	3.820	0.180	0.180	NA	NA	NA	NA	NA	NA	NA	8856	1213	NA
10/30/81	7.15	6.4	2600	1750	490.0	60.0	108.0	46.0	0.120	2.040	0.090	0.110	NA	NA	NA	NA	NA	NA	6.2	3104	271	NA
03/24/82	6.90	10.8	950	490	136.0	16.6	14.0	12.0	0.390	7.700	0.300	0.250	NA	NA	NA	NA	NA	NA	NA	715	NA	NA
04/13/82	7.00	10.8	850	440	121.0	11.2	18.0	10.0	0.160	3.940	0.160	0.160	NA	NA	NA	NA	NA	NA	NA	1090	NA	NA
04/14/82	6.97	7.7	500	215	63.8	5.2	8.0	6.0	0.070	1.430	0.055	0.060	NA	NA	NA	NA	NA	NA	NA	4141	NA	NA
04/15/82	7.10	15.3	700	340	106.0	8.6	12.0	8.0	0.110	2.220	0.090	0.090	NA	NA	NA	NA	NA	NA	NA	2854	NA	NA
04/19/82	6.90	14.4	825	380	166.0	12.0	35.0	7.2	0.130	3.840	0.130	0.110	NA	NA	NA	NA	NA	NA	NA	912	NA	NA
04/23/82	7.20	18.0	1050	600	190.0	16.0	20.0	14.0	0.110	3.420	0.140	0.110	NA	NA	NA	NA	NA	NA	NA	1238	NA	NA
04/28/82	7.10	19.8	1175	630	157.0	16.0	22.0	16.0	0.080	2.840	0.110	0.080	NA	NA	NA	NA	NA	NA	NA	3274	NA	NA
05/03/82	7.10	12.6	750	370	118.0	10.0	16.0	10.0	0.100	1.610	0.090	0.070	NA	NA	NA	NA	NA	NA	NA	409	NA	NA
05/07/82	7.10	18.9	1030	520	164.0	14.0	22.0	12.0	0.080	3.050	0.140	0.090	NA	NA	NA	NA	NA	NA	NA	5167	NA	NA
05/11/82	7.10	18.0	1225	710	188.0	24.0	24.0	20.0	0.080	4.150	0.150	0.120	NA	NA	NA	NA	NA	NA	NA	3819	NA	NA
05/14/82	7.10	21.6	1200	690	216.0	22.8	26.6	20.4	0.060	3.000	0.110	0.080	0.04	0.12	NA	NA	NA	NA	NA	4038	NA	NA
05/21/82	7.15	NA	1800	1100	308.0	31.0	48.7	25.8	0.055	2.835	0.105	0.090	0.04	0.14	NA	NA	NA	NA	NA	1688	NA	NA
06/02/82	7.00	12.6	1275	710	198.0	21.8	39.0	14.0	0.050	1.660	0.070	0.070	0.04	0.11	NA	NA	NA	NA	NA	1253	NA	NA
06/11/82	6.80	NA	1175	665	185.0	18.5	23.0	18.0	0.050	3.245	0.125	0.090	0.03	0.14	NA	NA	NA	NA	NA	719	NA	NA
07/02/82	NA	NA	NA	780	308.0	26.0	36.0	26.0	0.060	3.540	0.160	0.080	0.01	0.13	NA	NA	NA	NA	NA	19809	NA	NA
07/26/82	7.10	12.6	1400	890	338.0	28.0	44.0	28.0	0.080	2.520	0.110	0.070	0.01	0.11	NA	NA	NA	NA	NA	1382	NA	NA
08/02/82	7.00	NA	1900	NA	394.0	30.0	42.0	32.0	0.090	3.400	0.150	0.070	0.02	0.13	NA	NA	NA	NA	NA	8025	NA	NA
08/09/82	7.00	17.1	2100	NA	367.0	34.0	62.0	36.0	0.100	5.170	0.220	0.090	0.02	0.13	NA	NA	NA	NA	NA	1817	NA	NA
08/23/82	7.10	8.1	1850	1145	359.0	32.0	48.0	34.0	0.130	7.020	0.310	0.120	0.02	0.17	NA	NA	NA	NA	NA	1790	NA	NA
08/30/82	6.90	9.9	1850	1035	338.0	30.0	44.0	30.0	0.120	5.940	0.270	0.120	0.30	0.17	NA	NA	NA	NA	NA	1147	NA	NA
09/08/82	6.50	10.8	1350	785	256.0	22.0	26.0	22.0	0.280	7.890	0.370	0.150	0.02	0.22	NA	NA	NA	NA	NA	4009	NA	NA
09/13/82	6.50	11.2	1300	975	316.0	26.0	32.0	28.0	0.190	7.350	0.280	0.140	0.02	0.19	NA	NA	NA	NA	NA	4190	NA	NA
09/20/82	6.90	NA	1900	1140	334.0	32.0	46.0	32.0	0.190	7.300	0.290	0.140	0.01	0.19	NA	NA	NA	NA	NA	810	NA	NA
10/04/82	7.00	16.2	1425	850	258.0	26.0	34.0	26.0	0.180	6.170	0.300	0.110	0.02	0.17	NA	NA	NA	NA	NA	15259	NA	NA
10/11/82	6.80	15.3	1400	630	252.0	24.0	38.0	26.0	0.110	4.570	0.150	0.080	0.01	0.12	NA	NA	NA	NA	NA	3782	NA	NA
10/21/82	6.90	13.0	1000	620	266.0	26.0	38.0	26.0	0.110	4.390	0.170	0.080	0.19	0.12	NA	NA	NA	NA	NA	1200	NA	NA
10/25/82	7.00	13.1	1830	940	256.0	26.0	34.0	28.0	0.150	5.640	0.240	0.100	0.03	0.14	NA	NA	NA	NA	NA	708	NA	NA
11/01/82	6.90	9.0	1000	NA	136.0	14.0	20.0	14.0	0.120	4.660	0.210	0.070	0.02	0.10	NA	NA	NA	NA	NA	1730	NA	NA
11/22/82	7.05	4.5	1400	605	230.0	28.0	32.0	NA	0.120	5.090	0.150	0.120	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
03/18/83	6.90	13.3	745	400	NA	NA	NA	NA	0.300	5.650	0.240	0.180	NA	NA	NA	NA	NA	NA	NA	2495	NA	NA
04/14/83	6.86	14.8	544	260	NA	NA	NA	NA	0.130	2.910	0.120	0.130	NA	NA	NA	NA	NA	NA	NA	4984	NA	NA
05/15/83	7.01	15.3	910	540	NA	NA	NA	NA	0.070	3.260	0.140	0.110	NA	NA	NA	NA	NA	NA	NA	6196	NA	NA
06/16/83	6.86	11.8	1348	840	NA	NA	NA	NA	0.070	4.870	0.190	0.100	NA	NA	NA	NA	NA	NA	NA	2168	NA	NA
07/14/83	6.91	12.6	1663	1320	NA	NA	NA	NA	0.260	7.470	0.280	0.230	NA	NA	NA	NA	NA	NA	NA	11710	NA	NA
08/14/83	6.61	7.5	1497	1140	NA	NA	NA	NA	0.730	9.950	0.400	0.340	NA	NA	NA	NA	NA	NA	NA	14114	NA	NA
09/15/83	6.74	7.1	1507	1110	NA	NA	NA	NA	0.780	10.220	0.460	0.350	NA	NA	NA	NA	NA	NA	NA	8997	NA	NA
10/13/83	6.91	9.8	1714	1110	NA	NA	NA	NA	0.520	8.350	0.370	0.240	NA	NA	NA	NA	NA	NA	NA	24777	NA	NA
11/09/83	6.60	5.4	900	620	NA	NA	NA	NA	0.570	6.470	0.290	0.220	NA	NA	NA	NA	NA	NA	NA	3459	NA	NA
11/30/83	6.35	5.4	900	620	NA	NA	NA	NA	0.830	13.350	0.540	0.360	NA	NA	NA	NA	NA	NA	NA	738	NA	NA
02/24/84	6.25	6.3	580	290	93.9	18.9	9.6	NA	0.730	12.000	0.500	0.300	NA	NA	NA	NA	NA	NA	NA	1222	NA	NA
03/15/84	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
04/15/84	6.49	4.9	429	260	94.7	13.6	8.2	NA	0.390	4.850	0.210	0.200	NA	NA	NA	NA	NA	NA	NA	6389	NA	NA
05/16/84	6.86	10.7	865	620	210.6	28.1	17.5	NA	0.360	9.000	0.290	0.200	NA	NA	NA	NA	NA	NA	NA	10099	NA	NA
06/13/84	6.91	7.9	1065	838	260.1	36.6	27.7	NA	0.590	12.400	0.430	0.290	NA	NA	NA	NA	NA	NA	NA	21645	NA	NA

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
07/14/84	6.92	9.0	1145	820	NA	NA	25.0	NA	0.300	9.600	0.350	0.190	NA	NA	NA	NA	NA	NA	NA	5190	NA	NA
08/15/84	6.81	6.8	1395	880	278.4	53.9	27.2	NA	0.870	24.000	0.880	0.380	NA	NA	NA	NA	NA	NA	NA	5678	NA	NA
09/12/84	6.36	3.3	1425	1400	326.6	67.8	33.5	NA	1.540	16.250	1.020	0.530	NA	NA	NA	NA	NA	NA	NA	1809	NA	NA
10/13/84	6.17	4.3	1425	1750	401.2	60.0	31.5	NA	3.300	27.900	1.790	0.790	NA	NA	NA	NA	NA	NA	NA	10556	NA	NA
11/14/84	6.23	4.9	1050	1400	352.0	52.0	32.4	NA	1.860	18.100	0.560	0.540	NA	NA	NA	NA	NA	NA	NA	2430	NA	NA
12/01/84	6.00	2.1	975	1050	390.0	190.0	25.0	NA	5.400	23.300	1.030	0.780	NA	NA	NA	NA	NA	NA	NA	409	NA	NA
12/18/84	6.15	4.2	875	710	216.0	48.0	20.0	NA	2.490	23.800	0.860	0.570	NA	NA	NA	NA	NA	NA	NA	1452	NA	NA
04/02/85	6.55	1.1	420	270	85.0	6.6	5.6	NA	0.470	3.090	0.120	0.120	NA	NA	NA	NA	NA	NA	NA	308	NA	NA
04/16/85	6.56	7.6	845	520	162.0	29.3	12.4	NA	1.770	14.500	0.550	0.440	NA	NA	NA	NA	NA	NA	NA	16654	NA	NA
05/14/85	6.93	6.9	1200	710	204.0	37.5	19.3	NA	0.670	12.300	0.450	0.270	NA	NA	NA	NA	NA	NA	NA	10821	NA	NA
06/14/85	6.94	2.6	1425	1100	241.0	45.1	25.4	NA	1.170	15.200	0.510	0.350	NA	NA	NA	NA	NA	NA	NA	33032	NA	NA
07/15/85	6.82	7.6	1350	950	193.0	43.4	28.5	NA	0.840	12.900	0.480	0.310	NA	NA	NA	NA	NA	NA	NA	12354	NA	NA
08/13/85	6.70	3.3	1350	968	230.2	43.1	25.0	NA	1.400	17.900	0.690	0.450	NA	NA	NA	NA	NA	NA	NA	12188	NA	NA
09/13/85	6.58	1.6	1475	1000	316.0	49.8	26.0	NA	1.750	16.100	0.630	0.450	NA	NA	NA	NA	NA	NA	NA	28330	NA	NA
10/15/85	6.85	5.2	1275	650	242.0	38.0	22.0	NA	1.300	10.600	0.320	0.300	NA	NA	NA	NA	NA	NA	NA	18471	NA	NA
11/15/85	6.64	1.1	1175	540	254.0	36.0	24.0	NA	0.800	7.250	0.220	0.220	NA	NA	NA	NA	NA	NA	NA	1238	NA	NA
03/23/86	6.60	1.1	570	230	76.0	14.6	5.8	NA	2.090	13.000	0.460	0.280	NA	NA	NA	NA	NA	NA	NA	18543	NA	NA
04/16/86	6.49	2.8	775	410	131.0	28.1	9.6	NA	2.400	17.800	0.580	0.380	NA	NA	NA	NA	NA	NA	NA	12687	NA	NA
05/14/86	6.75	5.2	1025	800	244.0	38.0	14.0	NA	1.540	15.900	0.600	0.360	NA	NA	NA	NA	NA	NA	NA	12884	NA	NA
06/11/86	6.63	5.6	1175	850	216.0	38.0	24.0	NA	1.270	14.100	0.520	0.330	NA	NA	NA	NA	NA	NA	NA	6703	NA	NA
07/12/86	6.69	5.9	1375	800	246.0	50.0	20.0	NA	2.740	20.400	0.800	0.520	NA	NA	NA	NA	NA	NA	NA	19489	NA	NA
08/13/86	6.29	4.2	1550	1100	272.0	47.0	26.0	NA	3.580	23.100	0.900	0.430	NA	NA	NA	NA	NA	NA	NA	11835	NA	NA
09/14/86	6.54	3.0	1500	700	271.0	48.4	22.0	NA	3.460	18.700	0.800	0.360	NA	NA	NA	NA	NA	NA	NA	23641	NA	NA
10/16/86	6.41	5.2	1925	1200	276.0	52.0	26.0	NA	2.720	19.800	0.700	0.490	NA	NA	NA	NA	NA	NA	NA	4621	NA	NA
11/16/86	6.36	3.4	1750	1125	332.0	68.0	26.0	NA	5.700	23.000	0.870	0.640	NA	NA	NA	NA	NA	NA	NA	9148	NA	NA
03/22/87	5.70	6.1	575	294	96.0	18.0	8.0	NA	3.600	19.800	0.540	0.390	NA	NA	NA	NA	NA	NA	NA	10068	NA	NA
04/15/87	6.35	4.6	725	352	94.0	22.0	8.0	NA	3.750	14.600	0.590	0.420	NA	NA	NA	NA	NA	NA	NA	3009	NA	NA
05/16/87	6.59	6.0	1500	970	294.0	50.0	20.0	NA	4.900	21.000	0.800	0.800	NA	NA	NA	NA	NA	NA	NA	32721	NA	NA
06/16/87	6.66	6.6	1900	1200	368.0	72.0	34.0	NA	1.970	16.000	0.500	0.290	NA	NA	NA	NA	NA	NA	NA	2328	NA	NA
07/15/87	6.03	7.4	1850	1200	312.0	60.0	28.0	NA	4.800	20.500	0.800	0.560	NA	NA	NA	NA	NA	NA	NA	25288	NA	NA
08/15/87	6.04	5.4	1600	1080	270.0	56.0	26.0	NA	5.400	27.000	0.900	0.680	NA	NA	NA	NA	NA	NA	NA	12176	NA	NA
09/16/87	6.01	4.3	2275	1070	314.0	68.0	26.0	NA	6.930	29.000	0.920	0.700	NA	NA	NA	NA	NA	NA	NA	16525	NA	NA
10/13/87	5.86	1.8	2150	1390	400.0	76.0	34.0	NA	4.700	25.000	0.780	0.650	NA	NA	NA	NA	NA	NA	NA	1764	NA	NA
11/10/87	5.81	2.9	1850	1280	330.0	60.0	30.0	NA	5.900	26.000	0.980	0.750	NA	NA	NA	NA	NA	NA	NA	889	NA	NA
03/28/88	6.50	5.1	675	340	74.0	24.0	6.0	NA	6.800	16.400	0.900	1.000	NA	NA	NA	NA	NA	NA	NA	7169	NA	NA
04/17/88	5.93	2.3	970	700	116.0	36.0	8.0	NA	9.500	33.000	1.400	1.500	NA	NA	NA	NA	NA	NA	NA	5008	NA	NA
05/12/88	6.11	6.7	1500	1000	276.0	64.0	18.0	NA	8.500	41.000	1.600	1.600	NA	NA	NA	NA	NA	NA	NA	11014	NA	NA
06/12/88	6.12	4.8	1320	820	282.0	52.0	18.0	NA	5.100	33.000	1.200	1.300	NA	NA	NA	NA	NA	NA	NA	3160	NA	NA
07/15/88	5.91	5.3	1450	1120	253.0	60.0	21.0	NA	9.000	49.000	1.800	1.900	NA	NA	NA	NA	NA	NA	NA	3565	NA	NA
08/16/88	6.27	7.1	1575	915	277.0	73.0	26.0	NA	5.600	38.500	1.100	0.800	NA	NA	NA	NA	NA	NA	NA	56169	NA	NA
09/16/88	6.09	3.0	1600	860	280.0	78.0	26.0	NA	5.200	42.000	1.200	0.900	NA	NA	NA	NA	NA	NA	NA	11276	NA	NA
10/18/88	5.93	3.6	1575	1105	239.0	66.0	20.0	NA	8.350	36.500	1.600	1.750	NA	NA	NA	NA	NA	NA	NA	3346	NA	NA
11/15/88	5.57	2.4	1200	900	180.0	116.0	18.0	NA	12.700	38.000	1.800	1.700	NA	NA	NA	NA	NA	NA	NA	5583	NA	NA
11/30/88	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	8	NA	NA
04/07/89	5.20	1.0	530	284	70.0	20.0	8.0	NA	5.200	20.500	0.800	0.700	NA	NA	NA	NA	NA	NA	NA	26374	NA	NA
04/24/89	5.87	3.7	575	296	64.0	18.0	6.0	NA	4.000	18.000	0.800	0.800	NA	NA	NA	NA	NA	NA	NA	7483	NA	NA
05/16/89	5.69	1.3	1325	790	192.0	52.0	14.0	NA	8.000	37.000	1.700	1.700	NA	NA	NA	NA	NA	NA	NA	13668	NA	NA
06/17/89	5.92	6.3	1725	1270	266.0	76.0	28.0	NA	8.000	46.000	1.800	1.200	NA	NA	NA	NA	NA	NA	NA	37191	NA	NA
07/17/89	5.60	4.4	1900	1350	342.0	78.0	30.0	NA	6.700	30.000	1.100	1.100	NA	NA	NA	NA	NA	NA	NA	3959	NA	NA
08/14/89	5.83	5.1	1400	880	206.0	54.0	20.0	NA	7.500	41.000	1.400	1.200	NA	NA	NA	NA	NA	NA	NA	4285	NA	NA
09/14/89	5.58	5.5	2100	1480	384.0	98.0	30.0	NA	13.100	39.000	2.300	1.600	NA	NA	NA	NA	NA	NA	NA	26192	NA	NA

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
10/16/89	5.88	2.3	1400	800	212.0	54.0	22.0	NA	5.900	25.000	1.900	1.000	NA	NA	NA	NA	NA	NA	NA	2566	NA	NA
11/06/89	5.50	2.4	1175	680	166.0	50.0	16.0	NA	5.300	25.000	2.600	1.000	NA	NA	NA	NA	NA	NA	NA	745	NA	NA
03/26/90	5.05	3.8	600	290	64.0	21.4	7.4	NA	10.200	19.000	10.200	1.010	NA	NA	NA	NA	NA	NA	NA	742	NA	NA
04/12/90	5.82	1.1	975	685	148.0	42.0	10.0	NA	9.430	32.000	1.540	1.540	NA	NA	NA	NA	NA	NA	NA	5984	NA	NA
05/17/90	5.72	2.1	1800	1250	286.0	84.0	20.0	NA	10.000	39.000	1.910	1.780	NA	NA	NA	NA	NA	NA	NA	7127	NA	NA
06/18/90	5.86	2.0	1685	1040	202.0	70.0	22.0	NA	7.780	30.000	1.270	1.140	NA	NA	NA	NA	NA	NA	NA	10465	NA	NA
07/16/90	5.79	3.2	1650	975	214.0	66.0	20.0	NA	7.820	27.000	1.250	1.190	NA	NA	NA	NA	NA	NA	NA	6972	NA	NA
08/13/90	5.77	1.6	1575	920	228.0	66.0	22.0	NA	5.210	21.000	1.010	0.930	NA	NA	NA	NA	NA	NA	NA	1544	NA	NA
09/10/90	5.42	3.2	1715	940	212.0	74.0	18.0	NA	20.000	56.000	2.820	2.270	NA	NA	NA	NA	NA	NA	NA	3978	NA	NA
10/15/90	5.29	2.7	2600	1650	358.0	134.0	30.0	NA	35.000	93.000	4.350	3.100	NA	NA	NA	NA	NA	NA	NA	22191	NA	NA
11/16/90	5.31	1.9	3875	2400	680.0	202.0	54.0	NA	29.000	88.000	4.040	3.550	NA	NA	NA	NA	NA	NA	NA	1098	NA	NA
04/08/91	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	4179	NA	NA
04/22/91	4.96	2.5	1650	1030	194.0	88.0	16.0	NA	19.900	68.000	2.600	1.800	NA	NA	NA	NA	NA	NA	NA	8005	NA	NA
05/20/91	5.18	2.5	2375	1200	262.0	107.0	20.0	NA	20.300	68.000	2.400	1.500	NA	NA	NA	NA	NA	NA	NA	11983	NA	NA
06/17/91	5.25	2.0	2050	1110	214.0	104.0	20.0	NA	27.000	71.000	3.000	1.600	NA	NA	NA	NA	NA	NA	NA	11987	NA	NA
07/15/91	5.27	3.1	2725	1260	278.0	124.0	26.0	NA	21.000	59.000	2.400	1.500	NA	NA	NA	NA	NA	NA	NA	32415	NA	NA
08/12/91	5.27	2.4	2550	1370	318.0	110.0	34.0	NA	18.000	54.000	2.300	1.300	NA	NA	NA	NA	NA	NA	NA	3463	NA	NA
09/09/91	5.22	3.6	2675	1420	268.0	110.0	28.0	NA	30.000	84.000	3.400	2.000	NA	NA	NA	NA	NA	NA	NA	19792	NA	NA
10/07/91	5.31	3.6	2575	1360	272.0	108.0	32.0	NA	21.000	67.000	2.700	1.700	NA	NA	NA	NA	NA	NA	NA	7025	NA	NA
11/07/91	5.24	3.1	2475	1270	252.0	112.0	30.0	NA	27.000	80.000	3.200	2.100	NA	NA	NA	NA	NA	NA	NA	22131	NA	NA

NA: Not analyzed.

'Dates represent intermediate dates for measurement periods.

Table A2.1.1. Drainage quality data from Duluth Complex test pile FL1, 1992-1993 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH	Alk. mg/L	Lab Acy. mg/l.	Calc. Acy. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/23/92	5.35	3.8	39	54.0	875	400	86.0	36.0	10.60	10.50	<1	8.60	24	1.00	0.80	0.03	1.19	NA	NA	NA	NA	NA	1513	NA	NA
05/11/92	5.08	3.2	60	105.1	1340	630	122.0	54.0	13.40	12.00	<1	14.90	46	1.90	1.20	0.02	2.06	NA	NA	NA	NA	NA	2724	NA	NA
05/26/92	5.18	1.9	70	99.7	1380	755	148.0	68.0	18.80	18.20	<1	16.50	42	1.60	1.00	0.03	1.88	NA	NA	NA	NA	NA	4053	NA	NA
06/08/92	5.18	3.8	100	140.3	2470	1490	294.0	126.0	31.80	31.00	<1	20.00	61	2.30	1.40	0.04	2.38	NA	NA	NA	NA	NA	798	NA	NA
06/22/92	5.29	2.6	63	101.9	1600	920	172.0	76.0	21.80	23.00	<1	16.70	43	1.60	1.10	0.03	1.87	NA	NA	NA	NA	NA	914	NA	NA
07/07/92	5.15	2.1	85	145.5	2030	430	222.0	96.0	26.60	28.00	<1	24.00	60	2.50	1.40	0.04	2.53	NA	NA	NA	NA	NA	3350	NA	NA
07/20/92	5.19	3.1	90	119.6	2250	1405	246.0	106.0	29.80	30.00	<1	19.60	50	2.10	1.20	0.04	2.02	NA	NA	NA	NA	NA	1223	NA	NA
08/03/92	5.19	2.6	50	93.5	1180	815	166.0	72.0	23.00	23.00	<1	15.90	39	1.70	1.00	0.02	1.72	NA	NA	NA	NA	NA	1118	NA	NA
08/17/92	5.11	2.1	63	124.7	1900	940	184.0	80.0	22.00	25.00	<1	21.00	49	3.20	2.60	0.10	2.25	NA	NA	NA	NA	NA	1354	NA	NA
08/31/92	5.17	5.0	65	110.7	1400	800	142.0	68.0	17.80	23.00	<1	18.50	46	1.90	1.10	0.10	2.01	NA	NA	NA	NA	NA	1875	NA	NA
09/14/92	5.21	2.1	78	136.6	1920	1290	256.0	110.0	27.80	31.00	<1	24.00	55	2.40	1.40	0.01	2.41	NA	NA	NA	NA	NA	4141	NA	NA
09/28/92	5.12	3.0	80	127.3	2300	1450	292.0	122.0	33.00	34.00	<1	20.00	54	2.20	1.20	0.02	2.09	NA	NA	NA	NA	NA	2579	NA	NA
10/12/92	5.24	3.0	81	138.2	2800	1780	346.0	152.0	40.80	37.00	<1	21.00	59	2.40	1.40	0.01	2.16	NA	NA	NA	NA	NA	539	NA	NA
10/27/92	5.07	2.6	75	125.5	2300	2800	270.0	118.0	31.40	31.00	<1	17.70	55	2.20	1.20	0.01	2.13	NA	NA	NA	NA	NA	388	NA	NA
11/09/92	5.23	3.2	NA	NA	2850	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	129	NA	NA
11/30/92	5.22	2.7	95	112.4	1500	880	166.0	80.0	19.20	22.00	<1	18.30	47	2.00	1.20	0.02	2.06	NA	NA	NA	NA	NA	767	NA	NA
04/06/93	5.72	2.0	NA	NA	500	268	50.0	22.5	6.00	6.00	<1	5.87	14	0.52	0.39	0.01	0.63	NA	NA	NA	NA	NA	3698	NA	NA
04/22/93	5.06	2.5	NA	NA	880	468	77.5	40.0	7.50	8.00	<1	10.20	30	1.32	0.73	0.03	1.25	NA	NA	NA	NA	NA	3550	NA	NA
05/06/93	5.13	2.5	NA	NA	975	580	92.5	47.5	10.25	9.00	<1	13.00	39	1.70	0.87	0.05	1.58	NA	NA	NA	NA	NA	10848	NA	NA
05/20/93	5.24	2.0	NA	NA	1500	1095	177.5	82.5	14.75	16.00	<1	16.00	57	2.20	1.05	0.04	1.97	NA	NA	NA	NA	NA	3588	NA	NA
06/03/93	5.27	3.0	NA	NA	1720	1130	180.0	87.5	17.50	16.75	<1	25.00	66	2.50	1.35	0.05	2.24	NA	NA	NA	NA	NA	6639	NA	NA
06/17/93	5.11	2.5	NA	NA	1725	1375	242.5	117.5	21.75	21.00	<1	31.00	83	3.17	1.64	0.03	2.77	NA	NA	NA	NA	NA	7366	NA	NA
07/01/93	5.04	2.0	NA	NA	2100	1190	240.0	122.5	22.50	23.50	<1	33.00	80	3.07	1.58	0.04	2.64	NA	NA	NA	NA	NA	8691	NA	NA
07/14/93	5.15	0.0	NA	NA	2000	1260	234.0	100.0	22.40	26.40	<1	18.00	53	1.93	1.05	0.02	2.07	NA	NA	NA	NA	NA	20482	NA	NA
07/29/93	5.26	3.5	NA	NA	1350	895	152.0	68.0	15.00	19.00	<1	16.00	42	1.57	0.89	0.02	1.69	NA	NA	NA	NA	NA	13028	NA	NA
08/12/93	5.03	3.5	NA	NA	2050	1180	248.0	102.0	22.20	27.60	<1	21.00	57	1.98	1.22	0.02	2.30	NA	NA	NA	NA	NA	21862	NA	NA
08/26/93	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	3777	NA	NA
09/09/93	5.23	2.5	NA	NA	1600	660	176.0	76.0	17.20	21.20	<1	18.00	44	1.69	1.03	0.01	1.95	NA	NA	NA	NA	NA	10886	NA	NA
09/23/93	5.20	4.0	NA	NA	1900	1140	254.0	98.0	22.80	27.20	<1	20.00	52	1.94	1.20	0.03	2.28	NA	NA	NA	NA	NA	4315	NA	NA
10/07/93	5.17	2.5	NA	NA	2210	1240	278.0	118.0	26.00	29.60	<1	23.00	64	2.19	1.42	0.02	2.64	NA	NA	NA	NA	NA	2608	NA	NA
10/20/93	5.03	2.0	NA	NA	2800	1580	326.0	140.0	29.40	33.20	<1	25.00	72	2.60	1.41	0.02	2.67	NA	NA	NA	NA	NA	999	NA	NA
11/04/93	5.12	2.3	NA	NA	2200	1185	242.0	110.0	22.20	24.40	<1	25.00	68	2.67	1.33	0.53	2.62	NA	NA	NA	NA	NA	2112	NA	NA
11/22/93	5.03	2.0	NA	NA	2400	1060	250.0	116.0	24.60	24.20	<1	26.00	65	2.67	1.46	0.04	2.51	NA	NA	NA	NA	NA	519	NA	NA

NA: Not analyzed.

'Dates represent intermediate dates for measurement periods.

Table A2.2. Drainage quality data from Duluth Complex test pile FL2, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.20	NA	1000	NA	99.0	13.0	101.0	12.0	0.038	NA	0.003	0.020	NA	0.07	NA	NA	NA	NA	NA	NA	NA	4.0
05/25/78	7.70	NA	1900	NA	177.0	23.0	241.0	15.0	0.040	0.240	0.008	0.028	0.05	0.16	NA	NA	NA	NA	NA	NA	NA	NA
06/16/78	7.72	NA	4200	NA	323.0	47.0	555.0	23.0	0.056	0.200	0.015	0.075	0.03	0.31	NA	NA	NA	NA	NA	496	NA	11.0
07/05/78	7.50	NA	2700	1400	218.0	33.0	313.0	28.0	0.057	0.316	0.021	0.084	0.04	0.22	34.0	NA	NA	NA	NA	293	NA	12.0
07/13/78	7.92	NA	NA	800	491.0	43.0	376.0	37.0	0.049	0.410	0.033	0.090	0.03	0.29	30.0	12.0	1.00	0.08	NA	NA	NA	NA
07/19/78	7.80	NA	1300	800	291.0	24.0	189.0	28.0	0.062	0.240	0.014	0.033	0.01	0.16	41.0	10.0	0.75	0.10	NA	NA	NA	NA
08/02/78	7.75	NA	2000	1100	191.0	21.0	241.0	26.0	0.080	0.260	0.010	0.060	0.04	0.18	14.0	8.0	0.20	0.20	NA	NA	NA	19.0
08/04/78	7.75	NA	2500	1700	341.0	42.0	410.0	33.0	0.070	0.440	0.029	0.113	0.05	0.28	19.0	11.0	0.06	0.08	NA	1000	NA	22.0
08/16/78	7.58	39.5	1092	500	108.0	14.3	95.7	16.7	0.140	0.180	0.023	0.027	0.05	0.09	17.0	NA	0.10	0.08	NA	30	NA	23.0
08/23/78	7.60	31.0	1100	460	103.0	15.0	86.0	18.0	0.140	0.180	0.020	0.020	0.04	0.10	15.0	5.3	1.10	0.10	NA	6379	NA	NA
08/25/78	7.30	35.0	1600	680	170.0	24.0	125.0	28.0	0.110	0.240	0.030	0.050	0.01	0.15	25.0	3.6	NA	0.10	NA	NA	NA	23.0
08/28/78	7.55	48.0	2300	1100	253.0	34.0	196.0	32.0	0.100	0.380	0.045	0.090	0.01	0.23	34.0	8.5	1.30	0.08	NA	NA	NA	NA
09/13/78	7.25	29.0	1338	584	133.0	16.0	106.0	26.5	0.120	0.190	0.020	0.050	0.03	0.11	12.0	5.5	0.06	0.08	NA	451	NA	15.0
03/19/79	7.70	NA	1043	NA	83.2	11.0	43.8	15.8	0.119	0.170	0.016	0.050	0.06	0.08	NA	NA	NA	NA	12.0	2612	1891	NA
03/27/79	NA	NA	NA	NA	205.0	28.4	86.0	24.6	0.180	0.460	0.060	0.060	0.05	0.18	NA	NA	NA	NA	NA	53	NA	NA
04/17/79	6.90	22.0	388	136	74.2	5.8	25.6	11.8	0.108	0.134	0.008	0.030	0.06	0.05	10.0	136.0	NA	NA	14.0	5151	3480	NA
04/18/79	6.90	16.5	177	51	27.5	2.6	18.0	6.1	0.091	0.040	0.004	0.020	0.05	0.03	5.5	1.2	NA	NA	10.5	6056	3654	NA
04/20/79	6.50	21.0	296	100	49.0	5.6	22.0	8.0	0.113	0.080	0.008	0.020	0.06	0.05	12.0	2.2	0.45	NA	11.0	3263	3830	NA
04/22/79	7.50	18.0	1163	633	175.4	24.2	82.8	18.6	0.100	0.300	0.020	0.060	0.04	0.16	30.0	11.0	0.11	NA	NA	5020	1296	NA
05/09/79	7.40	NA	1217	500	154.6	22.0	83.0	18.1	0.084	0.295	0.020	0.040	0.04	0.14	26.0	NA	NA	NA	NA	825	1083	NA
05/10/79	7.40	NA	1009	400	144.4	20.0	59.6	16.6	0.100	0.250	0.015	0.040	0.03	0.13	22.0	NA	NA	NA	NA	6810	3129	NA
05/20/79	6.90	NA	1562	800	233.4	32.6	116.8	23.2	0.100	0.330	0.023	0.050	0.03	0.18	25.0	NA	NA	NA	14.0	3611	1034	NA
05/22/79	7.40	NA	1128	567	165.8	22.8	83.6	18.4	0.180	0.280	0.020	0.040	0.04	0.14	20.0	NA	NA	NA	16.0	1639	1041	NA
05/23/79	7.60	NA	1976	NA	286.0	39.2	174.4	28.6	0.100	0.440	0.027	0.070	0.21	0.23	NA	NA	NA	NA	4.7	2286	609	NA
06/02/79	7.60	NA	1683	883	217.2	31.8	164.2	24.0	0.160	0.340	0.023	0.050	0.07	0.18	23.0	NA	NA	NA	16.0	1518	474	NA
06/06/79	7.80	NA	1749	900	220.8	33.8	85.0	29.6	0.170	0.300	0.021	0.050	0.03	0.17	23.0	NA	NA	NA	NA	1999	531	NA
06/10/79	7.60	45.0	1414	580	173.4	27.8	71.8	24.6	0.190	0.270	0.013	0.030	0.06	0.13	19.0	NA	NA	NA	NA	2021	1211	22.0
06/12/79	7.00	43.0	2317	1200	277.0	44.6	107.2	40.0	0.132	0.370	0.017	0.070	0.02	0.29	27.0	NA	NA	NA	NA	1033	361	NA
06/16/79	7.70	51.0	2365	1350	307.4	49.1	179.0	39.8	0.131	0.415	0.029	0.080	0.03	0.23	31.0	NA	NA	NA	NA	1344	368	25.0
06/20/79	7.10	36.0	1363	570	158.0	27.0	139.0	22.0	0.130	0.220	0.017	0.040	0.04	0.12	15.0	NA	NA	NA	NA	2200	1011	18.0
06/30/79	7.60	59.0	2536	1450	336.0	55.0	339.0	31.0	0.070	0.430	0.036	0.090	0.03	0.13	31.0	NA	NA	NA	16.0	1132	170	12.0
07/10/79	7.70	45.0	3294	2300	420.0	72.0	430.0	38.0	0.060	0.590	0.052	0.110	0.02	0.34	28.0	NA	NA	NA	NA	1249	142	19.0
07/30/79	7.10	17.8	980	480	106.0	18.0	88.0	17.0	0.110	0.170	0.014	0.030	0.03	0.09	16.0	NA	NA	NA	14.0	2779	2096	22.0
08/04/79	7.70	30.8	1826	1020	253.0	36.0	183.0	32.0	0.070	0.360	0.030	0.070	0.07	0.19	15.0	NA	NA	NA	9.2	443	65	23.0
08/13/79	NA	NA	NA	NA	385.6	66.6	408.0	38.6	0.050	0.650	0.035	0.080	0.04	0.30	NA	NA	NA	NA	NA	91	11	NA
08/23/79	NA	NA	NA	NA	217.2	33.2	200.0	25.8	0.100	0.470	0.027	0.070	0.04	0.21	NA	NA	NA	NA	14.0	114	70	NA
08/28/79	7.20	13.6	1338	NA	157.0	25.2	132.6	20.0	0.150	0.360	0.021	0.060	0.07	0.15	NA	NA	NA	NA	15.0	326	293	23.0
09/01/79	7.10	18.4	1087	650	142.8	19.8	84.2	19.8	0.150	0.280	0.020	0.050	0.05	0.10	9.0	NA	NA	NA	13.0	1526	905	21.0
09/04/79	7.20	20.8	1380	NA	211.4	29.0	101.4	25.8	0.120	0.370	0.020	0.070	0.02	0.16	NA	NA	NA	NA	13.0	1775	179	20.0
09/10/79	7.20	15.2	1134	600	165.6	22.0	104.0	21.2	0.150	0.340	0.020	0.050	0.03	0.14	9.0	NA	NA	NA	14.0	1268	249	17.0
09/13/79	7.30	17.6	1587	NA	233.8	31.6	156.0	26.8	0.150	0.450	0.026	0.090	0.05	0.20	NA	NA	NA	NA	14.0	367	211	18.0
09/19/79	7.30	20.8	2592	NA	379.0	53.4	238.2	37.2	0.060	0.760	0.070	0.120	0.02	0.30	NA	NA	NA	NA	13.0	129	22	18.0
10/19/79	6.60	12.8	914	542	112.5	16.1	64.0	14.9	0.195	0.365	0.020	0.040	0.06	0.10	5.0	NA	NA	NA	14.0	1726	2456	NA
10/31/79	6.70	13.6	1319	817	194.0	25.2	84.0	20.8	0.160	0.490	0.050	0.090	0.05	0.16	4.0	NA	NA	NA	13.0	2835	1118	NA
11/01/79	7.20	17.6	1659	975	280.0	34.2	98.0	25.8	0.110	0.550	0.060	0.080	0.02	0.16	10.0	NA	NA	NA	13.0	1215	637	NA
04/01/80	7.10	17.3	350	135	33.2	4.6	17.8	7.8	0.135	0.095	0.010	0.030	0.08	0.04	3.5	NA	NA	NA	10.0	1582	382	10.5
04/16/80	7.50	35.0	600	225	68.2	8.2	33.8	13.2	0.130	0.130	0.030	0.060	0.03	0.06	5.0	NA	NA	NA	NA	856	831	NA
04/17/80	7.20	33.5	NA	253	69.0	9.0	30.2	10.3	0.150	0.210	0.020	0.050	0.07	0.07	5.5	NA	NA	NA	12.0	1893	774	NA
04/29/80	7.57	41.5	1105	820	157.1	19.8	46.0	16.0	0.100	0.360	0.025	0.060	0.02	0.12	9.0	NA	NA	NA	NA	1117	163	13.0
05/08/80	7.40	34.1	1230	880	190.0	24.2	58.6	18.8	0.070	0.450	0.040	0.060	0.02	0.14	10.0	NA	NA	NA	9.5	541	51	11.0
05/18/80	7.50	30.8	1500	1050	236.0	31.2	92.0	23.0	0.050	0.600	0.050	0.100	0.03	0.18	23.0	NA	NA	NA	NA	371	26	19.0

Table A2.2. Drainage quality data from Duluth Complex test pile FL2, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
05/31/80	6.90	16.5	875	620	129.4	17.2	45.5	16.2	0.030	0.300	0.022	0.020	0.02	0.10	14.0	NA	NA	NA	15.0	1703	1084	NA
06/03/80	7.30	25.3	1400	700	207.4	25.2	68.8	21.6	0.110	0.570	0.030	0.100	0.02	0.13	15.0	NA	NA	NA	9.8	174	50	NA
06/06/80	7.20	23.1	1300	760	174.6	22.2	63.2	20.0	0.150	0.490	0.025	0.090	0.03	0.12	14.0	NA	NA	NA	NA	371	235	NA
06/13/80	7.10	20.9	1200	750	148.8	21.4	65.6	19.6	0.150	0.450	0.022	0.080	0.03	0.11	12.0	NA	NA	NA	NA	265	197	NA
06/19/80	7.20	23.1	1650	950	221.0	28.0	98.8	23.4	0.110	0.580	0.021	0.110	0.06	0.15	16.0	NA	NA	NA	NA	23	12	NA
06/30/80	7.10	17.6	1150	660	158.0	20.0	58.0	17.0	0.130	0.480	0.070	0.100	0.05	0.13	10.0	NA	NA	NA	10.0	114	38	NA
07/02/80	7.10	15.3	950	580	132.0	16.0	45.0	16.0	0.170	0.380	0.022	0.090	0.04	0.10	9.0	NA	NA	NA	NA	246	96	NA
07/17/80	6.90	17.6	1150	450	155.6	19.0	58.4	19.6	0.140	0.500	0.030	0.090	0.03	0.14	10.0	NA	NA	NA	NA	95	124	NA
07/23/80	7.00	17.6	1100	350	149.8	18.6	66.4	19.2	0.110	0.470	0.060	0.090	0.02	0.15	9.0	NA	NA	NA	NA	95	525	NA
08/03/80	6.60	12.1	725	355	91.4	12.2	35.0	14.8	0.260	0.400	0.022	0.070	0.05	0.12	9.0	NA	NA	NA	16.0	106	95	NA
08/12/80	6.60	9.8	600	258	70.9	9.3	25.7	12.8	0.260	0.340	0.015	0.050	0.10	0.11	7.0	NA	NA	NA	14.0	1026	525	NA
08/19/80	6.80	10.2	700	424	103.6	13.0	28.8	16.0	0.190	0.450	0.024	0.080	0.05	0.16	6.0	NA	NA	NA	NA	379	106	NA
08/22/80	NA	NA	600	280	77.1	8.1	24.1	11.7	0.220	0.390	0.050	0.070	0.07	0.12	6.0	NA	NA	NA	NA	428	329	NA
08/29/80	6.70	15.3	1450	920	304.0	36.4	68.2	27.0	0.170	0.830	0.080	0.160	0.06	0.22	11.0	NA	NA	NA	10.0	2271	1473	NA
09/07/80	6.60	10.2	843	360	151.4	15.8	44.0	16.4	0.190	0.610	0.050	0.110	0.04	0.16	7.0	NA	NA	NA	NA	1450	1000	NA
09/10/80	6.80	11.5	1238	700	252.2	30.6	71.0	22.8	0.180	0.860	0.060	0.150	0.04	0.23	10.0	NA	NA	NA	NA	345	88	13.2
09/12/80	6.70	12.0	969	450	188.4	19.6	47.6	17.4	0.210	0.590	0.050	0.110	0.06	0.16	9.0	NA	NA	NA	13.0	2309	1578	17.5
09/18/80	6.90	12.3	1231	690	241.8	29.8	70.6	20.2	0.240	0.760	0.060	0.130	0.08	0.16	11.0	NA	NA	NA	NA	5330	1938	13.0
09/23/80	6.90	12.8	2527	1700	398.0	72.8	204.0	40.0	0.200	1.780	0.140	0.250	0.04	0.31	25.0	NA	NA	NA	13.0	1821	612	15.0
10/22/80	6.70	13.6	2664	1620	404.0	79.0	330.0	48.4	0.240	1.910	0.120	0.310	0.04	0.35	29.0	NA	NA	NA	13.0	1628	322	10.0
10/23/80	6.40	8.5	1763	950	214.0	32.0	170.0	23.0	0.190	1.260	0.090	0.200	0.04	0.25	31.0	NA	NA	NA	NA	795	268	12.0
02/18/81	6.70	8.5	408	160	31.8	6.0	39.0	NA	0.210	0.295	0.020	0.060	NA	NA	10.0	NA	NA	NA	NA	1760	1300	6.0
03/29/81	6.70	8.8	330	90	31.2	6.2	24.7	NA	0.210	0.220	0.020	0.050	NA	NA	NA	NA	NA	NA	15.0	6719	1726	NA
03/30/81	6.80	12.8	525	240	63.7	10.9	20.8	NA	0.280	0.490	0.030	0.080	NA	NA	NA	NA	NA	NA	NA	4236	1700	NA
04/13/81	NA	NA	800	250	69.0	12.0	21.0	NA	0.290	0.770	0.060	0.080	NA	NA	NA	NA	NA	NA	NA	2142	1281	NA
04/16/81	7.10	14.0	800	315	78.0	15.0	22.0	NA	0.250	0.920	0.060	0.090	NA	NA	NA	NA	NA	NA	15.0	1427	212	NA
04/23/81	7.10	13.6	900	300	87.0	17.0	27.0	NA	0.260	0.970	0.050	0.090	NA	NA	NA	NA	NA	NA	NA	2423	2478	NA
04/25/81	7.10	19.2	1155	530	129.0	30.0	36.0	NA	0.240	1.180	0.080	0.100	NA	NA	NA	NA	NA	NA	NA	4857	1589	NA
04/27/81	7.30	24.0	1250	600	136.0	35.0	48.0	NA	0.260	1.260	0.080	0.100	NA	NA	NA	NA	NA	NA	14.0	3869	2091	NA
04/30/81	7.30	24.0	1596	870	210.0	40.0	80.0	NA	0.200	1.700	0.100	0.140	NA	NA	NA	NA	NA	NA	NA	3206	994	NA
05/12/81	7.20	23.0	2179	1250	306.0	54.0	166.0	NA	0.130	2.100	0.140	0.170	NA	NA	NA	NA	NA	NA	14.0	3539	203	17.5
06/01/81	7.25	14.0	2116	1250	266.0	54.0	200.0	NA	0.160	1.160	0.070	0.150	NA	NA	NA	NA	NA	NA	NA	2370	705	17.0
06/06/81	7.20	18.2	1959	1000	234.0	44.0	174.0	NA	0.170	0.960	0.070	0.130	NA	NA	NA	NA	NA	NA	13.0	1522	150	18.0
06/22/81	6.80	11.2	1270	625	144.0	26.0	102.0	NA	0.200	0.710	0.060	0.090	NA	NA	NA	NA	NA	NA	NA	2411	1030	16.0
06/24/81	6.90	12.0	793	870	96.0	14.0	46.0	NA	0.250	0.550	0.040	0.060	NA	NA	NA	NA	NA	NA	NA	1874	993	17.0
06/28/81	6.80	11.2	718	160	92.0	14.0	36.0	NA	0.300	0.640	0.040	0.070	NA	NA	NA	NA	NA	NA	18.0	2612	1938	15.0
07/04/81	7.10	13.5	1630	1060	254.0	36.6	98.0	NA	0.270	1.370	0.090	0.140	NA	NA	NA	NA	NA	NA	NA	1037	53	22.0
07/15/81	7.00	11.7	2769	1800	388.0	68.4	259.6	NA	0.140	2.550	0.150	0.220	NA	NA	NA	NA	NA	NA	NA	575	26	19.0
07/24/81	7.10	10.5	2142	1320	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	21.0	NA	NA	NA	9.1	492	122	20.0
08/04/81	6.80	8.4	2864	2000	364.0	80.0	154.0	NA	0.190	2.440	0.160	0.310	NA	NA	NA	NA	NA	NA	NA	246	25	18.0
08/13/81	6.90	12.2	1926	1250	216.0	42.0	184.0	28.0	0.130	1.260	0.110	0.170	NA	NA	NA	NA	NA	NA	NA	64	6	22.0
08/31/81	6.00	8.0	1700	800	162.0	32.0	140.0	NA	0.420	1.230	0.090	0.160	NA	NA	NA	NA	NA	NA	15.0	454	382	NA
09/07/81	6.60	6.8	1800	720	154.0	29.0	124.0	NA	0.280	1.190	0.080	0.160	NA	NA	NA	NA	NA	NA	NA	284	255	NA
09/26/81	6.40	4.8	1093	490	108.0	20.0	70.0	NA	0.380	1.060	0.060	0.140	NA	NA	NA	NA	NA	NA	11.0	768	530	NA
10/01/81	6.60	4.8	750	260	90.0	16.0	32.0	NA	0.350	0.880	0.070	0.120	NA	NA	NA	NA	NA	NA	NA	1968	1257	NA
10/04/81	6.45	4.4	825	380	118.0	16.0	34.0	16.0	0.310	0.960	0.080	0.160	NA	NA	NA	NA	NA	NA	NA	852	1060	NA
10/13/81	6.65	5.0	950	400	132.0	20.0	34.0	18.0	0.310	1.370	0.100	0.200	NA	NA	NA	NA	NA	NA	9.7	3937	1603	NA
10/17/81	6.60	5.6	1600	870	238.0	36.0	58.0	28.0	0.510	4.200	0.240	0.360	NA	NA	NA	NA	NA	NA	NA	3896	810	NA
11/05/81	6.55	6.4	3800	2400	470.0	100.0	304.0	56.0	1.000	5.200	0.300	0.510	NA	NA	NA	NA	NA	NA	9.6	2839	99	NA
04/24/82	7.00	14.4	620	290	98.0	10.0	32.0	12.0	0.130	0.570	0.030	0.110	NA	NA	NA	NA	NA	NA	NA	3691	NA	NA
04/26/82	6.80	11.5	870	420	93.0	16.0	29.5	14.0	0.315	1.770	0.100	0.140	NA	NA	NA	NA	NA	NA	NA	4047	NA	NA

Table A2.2. Drainage quality data from Duluth Complex test pile FL2, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
05/03/82	7.20	10.8	1000	520	128.0	22.0	56.0	16.0	0.160	1.850	0.110	0.170	NA	NA	NA	NA	NA	NA	NA	969	NA	NA
05/07/82	6.70	9.0	780	370	82.0	14.0	20.0	12.0	0.320	1.800	0.110	0.130	NA	NA	NA	NA	NA	NA	NA	4789	NA	NA
05/11/82	6.80	12.2	1150	610	140.0	30.0	36.0	16.0	0.440	3.230	0.160	0.190	NA	NA	NA	NA	NA	NA	NA	3937	NA	NA
05/14/82	6.80	13.5	1450	730	174.0	25.4	40.4	14.6	0.320	2.100	0.100	0.140	0.03	0.15	NA	NA	NA	NA	NA	2623	NA	NA
05/21/82	6.90	15.3	2500	1500	336.0	63.0	143.0	32.0	0.410	4.030	0.210	0.270	0.04	0.24	NA	NA	NA	NA	NA	2248	NA	NA
06/02/82	6.65	12.6	2100	1300	246.0	49.0	134.0	26.0	0.350	2.500	0.130	0.200	0.04	0.20	NA	NA	NA	NA	NA	1787	NA	NA
06/18/82	6.60	11.7	1775	940	194.0	37.0	124.0	25.0	0.280	1.920	0.100	0.160	0.03	0.15	NA	NA	NA	NA	NA	390	NA	NA
07/02/82	6.50	8.1	950	480	116.0	20.0	56.0	16.0	0.350	1.300	0.060	0.100	0.03	0.11	NA	NA	NA	NA	NA	1098	NA	NA
07/08/82	6.50	9.0	1100	600	190.0	26.0	46.0	18.0	0.440	2.350	0.130	0.130	0.06	0.15	NA	NA	NA	NA	NA	3373	NA	NA
07/12/82	NA	NA	NA	1015	326.0	56.0	144.0	30.0	0.460	3.030	0.200	0.180	0.01	0.23	NA	NA	NA	NA	NA	4119	NA	NA
07/26/82	6.65	9.7	2171	1030	286.0	52.0	146.0	32.0	0.320	4.110	0.250	0.180	0.05	0.26	NA	NA	NA	NA	NA	6568	NA	NA
08/30/82	6.47	6.1	1356	800	234.0	34.0	76.0	24.0	0.460	2.040	0.130	0.160	0.06	0.17	NA	NA	NA	NA	NA	5765	NA	NA
10/04/82	6.80	8.8	2888	1600	340.0	80.0	216.0	42.0	0.800	5.890	0.380	0.250	0.07	0.38	NA	NA	NA	NA	NA	4096	NA	NA
11/01/82	6.45	5.2	1662	NA	199.0	32.0	76.0	16.0	0.360	2.205	0.125	0.100	0.03	0.13	NA	NA	NA	NA	NA	1643	NA	NA
11/22/82	NA	NA	NA	NA	296.0	49.0	192.0	NA	0.545	2.715	0.155	0.160	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
04/14/83	6.57	7.0	522	220	NA	NA	NA	NA	0.450	1.820	0.100	0.130	NA	NA	NA	NA	NA	NA	NA	9591	NA	NA
05/15/83	6.62	7.3	1115	580	NA	NA	NA	NA	0.590	3.470	0.200	0.270	NA	NA	NA	NA	NA	NA	NA	3826	NA	NA
06/16/83	6.48	6.2	1171	660	NA	NA	NA	NA	0.390	2.030	0.130	0.150	NA	NA	NA	NA	NA	NA	NA	1778	NA	NA
07/14/83	6.45	5.2	1164	660	NA	NA	NA	NA	0.620	3.580	0.190	0.240	NA	NA	NA	NA	NA	NA	NA	3993	NA	NA
08/14/83	6.08	3.9	1064	600	NA	NA	NA	NA	0.830	3.920	0.190	0.280	NA	NA	NA	NA	NA	NA	NA	6223	NA	NA
09/15/83	6.25	3.7	1200	730	NA	NA	NA	NA	1.140	4.710	0.280	0.380	NA	NA	NA	NA	NA	NA	NA	5095	NA	NA
10/13/83	6.39	5.0	1739	1340	NA	NA	NA	NA	2.120	6.830	0.400	0.460	NA	NA	NA	NA	NA	NA	NA	16559	NA	NA
11/09/83	6.25	2.7	1025	1380	NA	NA	NA	NA	1.580	4.770	0.270	0.360	NA	NA	NA	NA	NA	NA	NA	5053	NA	NA
11/30/83	6.10	2.7	1325	940	NA	NA	NA	NA	1.520	5.450	0.300	0.400	NA	NA	NA	NA	NA	NA	NA	276	NA	NA
04/15/84	6.59	9.9	936	700	198.3	37.5	45.8	NA	0.410	3.210	0.170	0.400	NA	NA	NA	NA	NA	NA	NA	7737	NA	NA
05/16/84	6.54	4.9	641	400	130.1	24.2	19.9	NA	0.670	4.120	0.140	0.190	NA	NA	NA	NA	NA	NA	NA	7835	NA	NA
06/13/84	6.39	4.9	950	680	172.7	35.6	32.7	NA	1.340	5.440	0.260	0.290	NA	NA	NA	NA	NA	NA	NA	11065	NA	NA
07/14/84	6.36	6.1	1460	1080	NA	NA	71.9	NA	1.300	6.330	0.240	0.300	NA	NA	NA	NA	NA	NA	NA	2710	NA	NA
08/15/84	6.35	4.8	1260	700	195.4	58.6	53.2	NA	0.950	4.820	0.270	0.270	NA	NA	NA	NA	NA	NA	NA	2172	NA	NA
09/12/84	5.99	3.4	1150	710	190.0	59.9	64.2	NA	1.250	5.310	0.310	0.330	NA	NA	NA	NA	NA	NA	NA	1086	NA	NA
10/13/84	5.78	2.7	1075	740	215.9	56.9	29.2	NA	1.920	8.500	0.600	0.540	NA	NA	NA	NA	NA	NA	NA	5443	NA	NA
11/14/84	5.82	3.2	1140	1100	280.0	60.0	51.0	NA	2.500	8.850	0.410	0.640	NA	NA	NA	NA	NA	NA	NA	1162	NA	NA
11/30/84	5.75	2.1	1125	710	264.0	60.0	54.0	NA	2.500	11.060	0.400	0.620	NA	NA	NA	NA	NA	NA	NA	98	NA	NA
12/18/84	5.75	2.1	1175	900	262.0	78.0	46.0	NA	3.900	14.400	0.830	0.810	NA	NA	NA	NA	NA	NA	NA	1123	NA	NA
04/02/85	6.30	4.5	260	140	37.2	7.2	8.4	NA	0.410	1.260	0.030	0.230	NA	NA	NA	NA	NA	NA	NA	867	NA	NA
04/16/85	6.50	7.6	650	420	100.0	25.4	17.8	NA	1.010	5.620	0.270	0.390	NA	NA	NA	NA	NA	NA	NA	15760	NA	NA
05/14/85	6.32	5.7	1215	780	189.0	41.6	27.5	NA	2.210	10.100	0.490	0.530	NA	NA	NA	NA	NA	NA	NA	7403	NA	NA
06/14/85	6.35	3.5	1525	1200	235.0	57.0	42.8	NA	3.170	11.600	0.490	0.520	NA	NA	NA	NA	NA	NA	NA	21991	NA	NA
07/15/85	6.33	6.2	1900	1500	280.0	76.0	97.0	NA	3.180	10.600	0.490	0.540	NA	NA	NA	NA	NA	NA	NA	8334	NA	NA
08/13/85	6.24	5.0	1425	1100	198.0	65.4	64.0	NA	1.960	6.990	0.360	0.390	NA	NA	NA	NA	NA	NA	NA	4129	NA	NA
09/13/85	6.06	3.2	1275	772	192.0	45.3	35.8	NA	1.890	8.670	0.420	0.450	NA	NA	NA	NA	NA	NA	NA	16188	NA	NA
10/15/85	6.19	3.5	1675	1100	260.0	74.0	58.0	NA	3.230	9.060	0.360	0.440	NA	NA	NA	NA	NA	NA	NA	17240	NA	NA
11/15/85	6.05	2.2	1925	1350	350.0	98.0	88.0	NA	3.340	9.200	0.390	0.510	NA	NA	NA	NA	NA	NA	NA	2737	NA	NA
03/23/86	6.20	5.0	440	195	56.0	13.0	8.2	NA	0.930	4.300	0.170	0.190	NA	NA	NA	NA	NA	NA	NA	17252	NA	NA
04/16/86	5.17	4.5	675	165	92.0	24.4	12.6	NA	1.620	7.300	0.310	0.290	NA	NA	NA	NA	NA	NA	NA	9550	NA	NA
05/14/86	6.20	3.8	1000	750	193.0	43.0	19.0	NA	2.940	13.300	0.480	0.390	NA	NA	NA	NA	NA	NA	NA	9357	NA	NA
06/11/86	6.33	5.1	1750	1300	282.0	70.0	64.0	NA	304.000	11.700	0.550	0.530	NA	NA	NA	NA	NA	NA	NA	2623	NA	NA
07/12/86	6.20	4.2	1125	675	158.0	46.0	36.0	NA	2.030	6.900	0.340	0.320	NA	NA	NA	NA	NA	NA	NA	4326	NA	NA
08/13/86	6.00	3.5	1150	600	150.0	40.4	36.0	NA	2.270	7.300	0.400	0.270	NA	NA	NA	NA	NA	NA	NA	3619	NA	NA
09/14/86	5.83	3.0	1475	750	222.0	55.2	32.0	NA	4.400	14.000	0.700	0.440	NA	NA	NA	NA	NA	NA	NA	11941	NA	NA
10/16/86	5.55	2.5	2150	1250	327.0	88.0	66.0	NA	6.200	19.000	0.800	0.770	NA	NA	NA	NA	NA	NA	NA	4372	NA	NA

Table A2.2. Drainage quality data from Duluth Complex test pile FL2, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date[1]	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
11/16/86	5.64	3.1	1950	1000	280.0	94.0	62.0	NA	8.000	19.400	0.930	0.870	NA	NA	NA	NA	NA	NA	NA	6294	NA	NA
03/22/87	5.70	6.1	350	190	58.0	14.0	10.0	NA	1.100	3.900	0.150	0.170	NA	NA	NA	NA	NA	NA	NA	9599	NA	NA
04/15/87	6.12	4.3	600	316	74.0	24.0	12.0	NA	1.670	5.900	0.280	0.320	NA	NA	NA	NA	NA	NA	NA	1957	NA	NA
05/16/87	5.75	4.3	1100	600	200.0	48.0	26.0	NA	3.400	10.900	0.500	0.310	NA	NA	NA	NA	NA	NA	NA	24103	NA	NA
06/16/87	5.24	3.2	2750	1800	473.0	162.0	103.0	NA	7.300	17.900	0.800	0.690	NA	NA	NA	NA	NA	NA	NA	2165	NA	NA
07/15/87	5.54	4.1	1475	830	188.0	58.0	34.0	NA	4.900	15.900	0.800	0.660	NA	NA	NA	NA	NA	NA	NA	12335	NA	NA
08/15/87	5.65	3.5	1600	1125	216.0	70.0	46.0	NA	5.200	15.100	0.700	0.670	NA	NA	NA	NA	NA	NA	NA	4500	NA	NA
09/16/87	5.46	3.2	1650	640	184.0	56.0	32.0	NA	3.870	12.370	0.550	0.560	NA	NA	NA	NA	NA	NA	NA	6953	NA	NA
10/13/87	4.87	1.9	2850	1580	415.0	130.0	82.0	NA	9.000	31.000	1.250	1.150	NA	NA	NA	NA	NA	NA	NA	1041	NA	NA
11/10/87	5.14	2.4	2525	920	364.0	116.0	78.0	NA	8.300	27.000	1.110	1.110	NA	NA	NA	NA	NA	NA	NA	568	NA	NA
03/28/88	5.70	3.0	400	275	50.0	14.0	6.0	NA	1.500	5.200	0.300	0.500	NA	NA	NA	NA	NA	NA	NA	6737	NA	NA
04/17/88	5.72	3.3	675	330	76.0	24.0	8.0	NA	2.400	7.800	0.500	0.800	NA	NA	NA	NA	NA	NA	NA	3384	NA	NA
05/12/88	5.71	4.0	1200	680	184.0	50.0	220.0	NA	4.700	14.600	0.900	1.400	NA	NA	NA	NA	NA	NA	NA	7127	NA	NA
06/12/88	5.56	3.6	1390	850	282.0	60.0	24.0	NA	6.200	22.000	1.100	1.600	NA	NA	NA	NA	NA	NA	NA	1605	NA	NA
07/15/88	5.47	3.5	1175	675	168.0	54.0	28.0	NA	5.000	13.000	1.000	1.300	NA	NA	NA	NA	NA	NA	NA	836	NA	NA
08/16/88	5.41	2.4	1550	870	216.0	88.0	40.0	NA	6.500	37.000	1.300	1.200	NA	NA	NA	NA	NA	NA	NA	43138	NA	NA
09/16/88	5.35	3.2	2025	1275	326.0	125.0	79.0	NA	6.650	35.500	1.250	1.250	NA	NA	NA	NA	NA	NA	NA	7926	NA	NA
10/18/88	5.47	3.2	1975	1300	272.0	102.0	51.0	NA	5.900	20.500	1.100	1.400	NA	NA	NA	NA	NA	NA	NA	1893	NA	NA
11/15/88	5.46	2.9	1275	780	152.0	64.0	28.0	NA	5.700	22.000	1.200	1.400	NA	NA	NA	NA	NA	NA	NA	5182	NA	NA
12/05/88	5.16	2.2	2125	1420	248.0	120.0	42.0	NA	13.400	53.000	2.600	2.900	NA	NA	NA	NA	NA	NA	NA	1120	NA	NA
04/07/89	5.80	4.4	500	248	64.0	20.0	10.0	NA	2.700	13.200	0.700	0.700	NA	NA	NA	NA	NA	NA	NA	20677	NA	NA
04/24/89	5.73	3.6	565	270	52.0	22.0	8.0	NA	2.900	15.000	0.700	0.800	NA	NA	NA	NA	NA	NA	NA	6249	NA	NA
05/16/89	5.39	3.6	1100	865	132.0	78.0	20.0	NA	20.000	61.000	3.800	4.100	NA	NA	NA	NA	NA	NA	NA	7907	NA	NA
06/17/89	5.27	3.3	1475	950	202.0	70.0	32.0	NA	7.600	44.000	2.000	1.600	NA	NA	NA	NA	NA	NA	NA	22827	NA	NA
07/17/89	5.12	3.2	2650	1870	420.0	156.0	76.0	NA	12.100	46.000	1.900	2.600	NA	NA	NA	NA	NA	NA	NA	1900	NA	NA
08/14/89	5.33	3.3	1450	880	182.0	76.0	38.0	NA	5.500	26.000	0.900	1.000	NA	NA	NA	NA	NA	NA	NA	1874	NA	NA
09/14/89	5.24	3.6	1750	1050	260.0	88.0	32.0	NA	10.000	39.000	2.700	2.100	NA	NA	NA	NA	NA	NA	NA	14311	NA	NA
10/16/89	5.23	2.0	1500	1140	230.0	84.0	44.0	NA	8.400	29.000	2.600	1.900	NA	NA	NA	NA	NA	NA	NA	2562	NA	NA
11/06/89	5.15	2.6	1340	800	170.0	78.0	34.0	NA	6.500	24.000	2.800	1.500	NA	NA	NA	NA	NA	NA	NA	1881	NA	NA
03/26/90	5.05	2.4	500	210	50.0	17.6	6.6	NA	2.700	9.120	0.480	0.600	NA	NA	NA	NA	NA	NA	NA	6601	NA	NA
04/12/90	5.31	3.0	982	565	110.0	50.0	12.0	NA	9.260	32.000	1.840	1.850	NA	NA	NA	NA	NA	NA	NA	19731	NA	NA
05/17/90	5.13	2.7	1600	950	194.0	84.0	24.0	NA	11.000	36.000	2.120	2.170	NA	NA	NA	NA	NA	NA	NA	11597	NA	NA
06/18/90	5.27	2.0	1750	930	188.0	80.0	32.0	NA	7.210	25.000	1.220	1.420	NA	NA	NA	NA	NA	NA	NA	11752	NA	NA
07/16/90	5.08	3.9	2850	1730	336.0	144.0	50.0	NA	14.000	42.000	2.430	2.680	NA	NA	NA	NA	NA	NA	NA	6283	NA	NA
08/13/90	5.05	3.0	3110	1840	388.0	196.0	70.0	NA	14.000	40.000	2.330	2.460	NA	NA	NA	NA	NA	NA	NA	1272	NA	NA
09/10/90	5.29	2.1	1545	815	174.0	68.0	30.0	NA	6.020	20.000	1.160	1.240	NA	NA	NA	NA	NA	NA	NA	3376	NA	NA
10/15/90	5.23	2.8	1700	960	196.0	74.0	28.0	NA	11.000	36.000	2.010	1.800	NA	NA	NA	NA	NA	NA	NA	16476	NA	NA
11/16/90	5.08	2.2	3750	2250	540.0	208.0	74.0	NA	22.000	69.000	3.740	3.860	NA	NA	NA	NA	NA	NA	NA	1370	NA	NA
04/08/91	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	712	NA	NA
04/22/91	5.70	4.0	1025	570	118.0	44.0	16.0	NA	4.400	17.000	0.700	0.800	NA	NA	NA	NA	NA	NA	NA	10579	NA	NA
05/20/91	5.24	3.2	1925	870	202.0	81.0	20.0	NA	9.900	36.000	1.500	1.200	NA	NA	NA	NA	NA	NA	NA	13471	NA	NA
06/17/91	5.25	2.0	1250	635	136.0	59.0	18.0	NA	6.900	23.000	1.100	0.800	NA	NA	NA	NA	NA	NA	NA	5185	NA	NA
07/15/91	5.21	2.3	1700	775	146.0	74.0	20.0	NA	9.000	30.000	1.400	1.000	NA	NA	NA	NA	NA	NA	NA	17070	NA	NA
08/12/91	4.96	2.0	3525	2030	416.0	184.0	72.0	NA	18.000	58.000	2.700	2.100	NA	NA	NA	NA	NA	NA	NA	1639	NA	NA
09/09/91	5.13	3.0	1350	610	114.0	50.0	20.0	NA	8.000	30.000	1.400	0.900	NA	NA	NA	NA	NA	NA	NA	9693	NA	NA
10/07/91	5.09	3.0	2375	1210	228.0	98.0	38.0	NA	13.000	43.000	2.100	1.700	NA	NA	NA	NA	NA	NA	NA	3051	NA	NA
11/07/91	5.15	3.0	1875	1230	202.0	110.0	30.0	NA	22.000	69.000	3.400	2.600	NA	NA	NA	NA	NA	NA	NA	14629	NA	NA

NA: Not analyzed.

[1]Dates represent intermediate dates for measurement periods.

Table A2.2.1. Drainage quality data from Duluth Complex test pile FL2, 1992-1993 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH	Alk. mg/L	Lab Acy. mg/L	Calc. Acy. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/23/92	4.76	3.8	80	177.4	2087	1520	224.0	172.0	49.20	31.00	1	17.80	75	3.20	2.30	0.10	3.79	NA	NA	NA	NA	NA	3	NA	NA
05/11/92	5.26	3.2	30	44.3	1030	400	84.0	40.0	13.20	12.70	<1	5.60	21	1.00	0.80	0.06	1.23	NA	NA	NA	NA	NA	3229	NA	NA
05/26/92	5.11	3.2	50	71.0	1120	620	120.0	56.0	16.00	15.70	<1	10.30	31	1.40	1.10	0.14	1.77	NA	NA	NA	NA	NA	2372	NA	NA
06/08/92	5.05	3.2	70	117.2	2200	1240	254.0	104.0	30.40	27.00	<1	16.50	50	2.20	1.90	0.07	2.67	NA	NA	NA	NA	NA	433	NA	NA
06/22/92	5.07	3.1	53	89.9	1860	1090	200.0	90.0	28.40	25.00	<1	12.40	39	2.00	1.40	0.04	2.12	NA	NA	NA	NA	NA	398	NA	NA
07/07/92	5.08	2.1	55	64.9	1290	630	126.0	58.0	20.20	19.20	<1	9.00	29	1.40	1.00	0.07	1.57	NA	NA	NA	NA	NA	1270	NA	NA
07/20/92	5.12	2.1	50	86.0	1780	2100	188.0	80.0	27.40	24.00	<1	11.30	38	1.90	1.40	0.07	1.95	NA	NA	NA	NA	NA	333	NA	NA
08/03/92	5.06	2.1	33	66.3	1270	810	156.0	70.0	25.40	22.00	<1	9.50	29	1.50	1.20	0.06	1.63	NA	NA	NA	NA	NA	489	NA	NA
08/17/92	5.11	2.1	65	68.8	1600	865	152.0	72.0	26.60	23.00	<1	9.90	30	1.50	1.20	0.05	1.73	NA	NA	NA	NA	NA	402	NA	NA
08/31/92	5.05	5.0	33	70.6	1600	875	162.0	82.0	27.40	23.00	<1	9.00	32	1.50	1.20	0.04	1.67	NA	NA	NA	NA	NA	551	NA	NA
09/14/92	5.22	3.2	18	41.2	800	440	72.0	38.0	13.40	15.20	<1	6.00	19	1.00	0.70	0.09	1.10	NA	NA	NA	NA	NA	1655	NA	NA
09/28/92	4.99	2.0	73	126.7	2200	1260	250.0	120.0	37.00	32.00	<1	17.50	54	2.60	1.90	0.07	2.90	NA	NA	NA	NA	NA	1409	NA	NA
10/12/92	5.00	2.6	100	168.9	3400	2180	376.0	202.0	64.00	44.00	<1	23.00	71	3.60	2.60	0.08	3.85	NA	NA	NA	NA	NA	324	NA	NA
10/27/92	4.92	2.6	85	153.4	3120	3860	328.0	186.0	60.60	40.00	<1	19.90	66	3.20	2.10	0.04	3.54	NA	NA	NA	NA	NA	178	NA	NA
11/09/92	5.06	3.0	NA	NA	3950	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	67	NA	NA
11/30/92	5.11	3.0	93	97.7	1950	1270	204.0	120.0	39.40	28.00	<1	11.50	44	2.10	1.50	0.04	2.35	NA	NA	NA	NA	NA	368	NA	NA
04/08/93	5.10	10.0	NA	NA	780	410	70.0	45.0	11.00	12.00	<1	4.84	20	0.72	0.57	0.06	0.91	NA	NA	NA	NA	NA	6707	NA	NA
04/22/93	4.91	5.0	NA	NA	875	400	82.5	35.0	9.00	9.00	<1	4.26	18	0.83	0.55	0.09	0.81	NA	NA	NA	NA	NA	3936	NA	NA
05/06/93	5.43	3.0	NA	NA	525	240	42.5	22.5	6.50	6.50	<1	3.19	12	0.57	0.39	0.13	0.54	NA	NA	NA	NA	NA	8660	NA	NA
05/20/93	5.04	2.0	NA	NA	1600	1070	172.5	97.5	20.25	17.50	<1	13.00	56	2.60	1.42	0.05	2.57	NA	NA	NA	NA	NA	3528	NA	NA
06/03/93	5.09	3.0	NA	NA	1300	715	120.0	70.0	17.25	13.25	<1	10.00	38	1.72	1.10	0.09	1.77	NA	NA	NA	NA	NA	3524	NA	NA
06/17/93	5.01	2.5	NA	NA	1300	800	150.0	85.0	21.00	16.25	<1	12.00	43	1.90	1.26	0.07	1.97	NA	NA	NA	NA	NA	3206	NA	NA
07/01/93	5.03	1.0	NA	NA	1050	555	95.0	60.0	15.25	13.75	<1	11.00	34	1.54	1.00	0.09	1.62	NA	NA	NA	NA	NA	3304	NA	NA
07/14/93	4.94	NA	NA	NA	2480	1520	260.0	148.0	33.40	32.40	2	24.00	80	3.44	2.30	0.09	4.14	NA	NA	NA	NA	NA	13693	NA	NA
07/29/93	4.92	3.5	NA	NA	2525	1660	270.0	166.0	40.80	34.60	2	21.00	75	3.33	2.18	0.12	3.89	NA	NA	NA	NA	NA	7142	NA	NA
08/12/93	4.88	5.0	NA	NA	2000	1210	192.0	118.0	33.40	27.40	<1	13.00	51	2.02	1.55	0.08	2.80	NA	NA	NA	NA	NA	2660	NA	NA
08/26/93	4.80	2.5	NA	NA	2410	1520	248.0	160.0	45.20	33.60	<1	16.00	63	2.48	1.91	0.07	3.44	NA	NA	NA	NA	NA	1457	NA	NA
09/09/93	5.05	2.0	NA	NA	1300	630	112.0	68.0	20.40	18.80	<1	8.98	30	1.31	0.98	0.07	1.80	NA	NA	NA	NA	NA	4171	NA	NA
09/23/93	5.06	2.0	NA	NA	2000	1130	206.0	128.0	38.20	30.00	<1	14.00	53	2.21	1.69	0.07	3.02	NA	NA	NA	NA	NA	1151	NA	NA
10/07/93	5.02	2.5	NA	NA	2550	1450	224.0	160.0	44.60	33.20	<1	18.00	62	2.69	1.73	0.08	3.13	NA	NA	NA	NA	NA	685	NA	NA
10/20/93	5.04	2.0	NA	NA	2500	1340	228.0	150.0	40.00	30.20	<1	16.00	61	2.45	1.66	0.05	2.98	NA	NA	NA	NA	NA	254	NA	NA
11/04/93	5.10	2.5	NA	NA	2100	620	149.0	108.0	27.60	22.00	<1	10.00	41	1.94	1.13	0.06	2.15	NA	NA	NA	NA	NA	753	NA	NA
11/22/93	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	379	NA	NA

NA: Not analyzed.
'Dates represent intermediate dates for measurement periods.

Table A2.3. Drainage quality data from Duluth Complex test pile FL3, from 1978-1989 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.20	NA	675	NA	62.0	10.0	52.0	12.0	0.023	0.060	NA	NA	0.70	0.05	NA	NA	NA	NA	NA	NA	NA	4.5
05/25/78	7.80	NA	1200	NA	106.0	24.0	122.0	15.0	0.013	0.042	0.050	0.020	0.04	0.09	NA	NA	NA	NA	NA	NA	NA	NA
06/16/78	7.79	NA	3000	NA	214.0	57.0	372.0	21.0	0.026	0.185	0.012	0.059	0.06	0.24	NA	NA	NA	NA	NA	192	NA	6.0
07/05/78	7.30	NA	1600	600	142.0	34.0	102.0	23.0	0.200	0.321	0.017	0.037	0.02	0.17	78.0	NA	NA	NA	NA	133	NA	9.0
07/13/78	7.55	NA	NA	1050	352.0	46.0	207.0	32.0	0.049	0.405	0.030	0.092	0.03	0.24	34.0	23.0	0.70	0.08	NA	NA	NA	NA
07/19/78	7.85	NA	1000	550	241.0	28.0	NA	27.0	0.092	0.235	0.020	0.060	0.01	0.15	35.0	24.0	0.55	0.08	NA	NA	NA	NA
08/02/78	7.60	NA	650	367	96.0	14.0	60.0	21.0	0.080	0.210	0.010	0.025	0.03	0.16	12.0	19.0	0.80	0.08	NA	NA	NA	22.0
08/04/78	7.65	NA	950	500	156.0	24.0	88.0	26.0	0.080	0.320	0.020	0.044	0.02	0.16	17.0	26.0	0.06	0.08	NA	100	NA	22.0
08/10/78	7.90	40.0	1900	1200	348.0	51.0	188.0	49.0	0.060	0.565	0.090	0.073	0.01	0.28	48.0	10.5	NA	0.45	17.0	851	NA	24.0
08/16/78	7.62	29.0	750	300	83.7	14.7	46.0	17.7	0.117	0.207	0.017	0.020	0.07	0.08	10.0	9.4	NA	0.08	NA	NA	NA	20.3
08/23/78	7.50	33.0	750	300	75.0	14.0	45.0	17.0	0.100	0.190	0.016	0.020	0.07	0.08	7.0	17.0	0.64	0.08	NA	NA	NA	NA
08/25/78	7.30	25.0	1150	500	115.0	23.0	67.0	23.0	0.080	0.250	0.010	0.030	0.04	0.10	11.0	24.0	NA	0.10	NA	NA	NA	24.0
09/13/78	7.30	36.0	690	659	133.5	21.5	74.5	27.5	0.085	0.300	0.020	0.040	0.04	0.12	7.0	13.0	0.08	0.08	NA	130	NA	14.0
03/19/79	7.08	NA	794	NA	118.8	25.9	50.1	15.9	0.080	0.790	0.050	0.110	0.08	0.10	NA	NA	NA	NA	9.3	3179	1765	NA
03/27/79	7.20	40.0	1233	NA	162.0	36.8	73.8	22.2	0.060	0.780	0.070	0.110	0.02	0.12	NA	NA	NA	NA	NA	1877	442	NA
04/13/79	6.75	NA	484	233	61.9	12.8	29.6	10.1	0.106	0.296	0.024	0.040	0.13	0.06	5.0	NA	NA	NA	NA	1877	442	NA
04/17/79	6.70	28.0	587	235	69.8	13.8	32.2	10.8	0.088	0.292	0.026	0.050	0.09	0.08	8.0	NA	NA	NA	5.3	3039	3576	NA
04/19/79	6.80	23.0	503	215	80.5	13.4	41.5	9.3	0.064	0.220	0.020	0.040	0.12	0.07	5.0	1.5	0.10	NA	NA	4341	4304	NA
04/19/79	7.10	27.0	532	205	83.0	12.6	31.0	8.6	0.072	0.238	0.020	0.050	0.05	0.07	4.0	1.5	NA	NA	11.0	3312	4304	NA
04/20/79	6.70	33.0	623	310	100.0	17.6	42.0	11.0	0.069	0.278	0.032	0.050	0.06	0.09	4.0	3.4	0.10	NA	4.7	4126	2592	NA
04/22/79	7.20	19.0	1356	833	206.4	44.6	121.6	24.8	0.056	0.920	0.051	0.120	0.03	0.17	11.0	6.9	0.04	NA	NA	8607	1630	NA
05/07/79	7.30	NA	1562	NA	209.2	49.0	131.6	27.6	0.046	0.890	0.047	0.110	0.03	0.18	NA	NA	NA	NA	NA	479	308	NA
05/10/79	7.40	NA	959	433	123.2	26.4	62.4	17.0	0.068	0.520	0.031	0.070	0.03	0.11	10.0	NA	NA	NA	NA	7070	3099	NA
05/20/79	6.70	NA	1711	933	229.2	52.6	130.8	29.6	0.060	0.750	0.047	0.120	0.02	0.16	13.0	NA	NA	NA	5.9	2619	900	NA
05/22/79	7.60	NA	1537	500	120.4	26.2	60.8	16.8	0.090	0.440	0.023	0.080	0.04	0.08	6.0	NA	NA	NA	7.6	1559	784	NA
05/23/79	7.50	NA	1331	750	180.4	37.4	92.2	24.4	0.058	0.620	0.040	0.080	0.04	0.12	7.0	NA	NA	NA	4.9	2449	838	NA
06/01/79	7.60	NA	998	517	122.8	26.6	59.8	17.4	0.100	0.460	0.023	0.060	0.08	0.10	6.0	NA	NA	NA	6.7	1938	369	NA
06/06/79	7.50	NA	824	400	101.0	21.8	33.4	15.6	0.081	0.370	0.021	0.050	0.03	0.08	4.0	NA	NA	NA	NA	1768	515	NA
06/10/79	7.20	35.0	725	320	173.8	38.0	50.8	38.8	0.100	0.310	0.010	0.040	0.02	0.05	4.0	NA	NA	NA	NA	1771	938	20.0
06/12/79	6.80	33.0	912	420	118.8	25.2	36.0	22.8	0.071	0.410	0.013	0.070	0.01	0.07	5.0	NA	NA	NA	NA	1011	361	NA
06/15/79	7.30	33.0	1081	533	143.8	30.4	42.4	23.4	0.061	0.470	0.013	0.050	0.01	0.08	7.0	NA	NA	NA	NA	329	325	20.0
06/21/79	7.05	30.5	823	409	101.0	23.5	42.0	19.0	0.070	0.340	0.021	0.040	0.02	0.06	4.0	NA	NA	NA	NA	2195	1070	16.5
06/30/79	7.20	29.0	1175	725	158.0	35.0	68.0	25.0	0.050	0.490	0.032	0.070	0.02	0.24	5.0	NA	NA	NA	5.6	1389	169	12.0
07/10/79	7.50	26.0	1649	1100	241.0	52.0	105.0	37.0	0.040	1.700	0.046	0.090	0.01	0.16	12.0	NA	NA	NA	5.3	734	74	18.0
07/30/79	7.10	12.0	772	390	98.0	21.0	31.0	19.0	0.090	0.580	0.040	0.060	0.01	0.10	1.0	NA	NA	NA	7.7	2597	1907	23.0
08/04/79	7.05	13.4	1277	809	204.5	36.0	53.0	33.0	0.080	0.980	0.045	0.105	0.03	0.15	4.0	NA	NA	NA	9.6	602	138	23.0
08/13/79	6.90	12.0	2064	1850	368.0	77.6	150.8	55.2	0.100	1.460	0.090	0.180	0.04	0.23	10.0	NA	NA	NA	5.2	295	46	14.0
08/28/79	7.10	12.4	1172	720	177.8	36.8	50.0	27.2	0.170	1.190	0.080	0.140	0.01	0.20	14.0	NA	NA	NA	8.2	208	161	21.0
09/01/79	6.90	12.0	1002	650	156.6	29.8	36.6	23.2	0.150	1.280	0.062	0.110	0.01	0.13	4.0	NA	NA	NA	6.1	3895	1538	20.0
09/05/79	6.90	12.0	1575	1200	269.6	52.0	69.0	36.6	0.140	1.640	0.090	0.150	0.01	0.17	5.0	NA	NA	NA	6.2	500	292	18.0
09/10/79	6.80	9.6	1401	975	235.6	42.6	77.4	33.0	0.140	1.440	0.110	0.130	0.02	0.17	6.0	NA	NA	NA	5.7	1983	408	18.0
09/13/79	7.00	11.2	1499	950	231.8	44.0	80.6	32.8	0.150	1.630	0.090	0.130	0.03	0.18	6.0	NA	NA	NA	5.3	1332	408	17.0
09/19/79	6.80	12.0	2598	1900	430.8	84.8	144.0	56.8	0.180	2.570	0.140	0.260	0.02	0.26	12.0	NA	NA	NA	5.0	863	109	15.0
10/17/79	6.40	7.2	2192	NA	363.2	73.4	122.8	48.0	0.190	2.560	0.140	0.270	0.01	0.29	NA	NA	NA	NA	4.6	337	200	NA
10/19/79	6.30	8.0	938	600	124.4	28.0	44.0	18.8	0.190	1.630	0.070	0.140	0.03	0.13	3.0	NA	NA	NA	5.9	1177	1297	NA
10/31/79	6.40	9.6	1233	725	172.0	36.5	54.2	23.9	0.220	2.840	0.140	0.195	0.02	0.19	3.0	NA	NA	NA	4.5	3963	1450	NA
11/01/79	6.80	12.8	1258	NA	182.0	39.6	60.0	25.6	0.190	2.540	0.120	0.180	0.02	0.15	NA	NA	NA	NA	4.5	3316	944	12.0
03/31/80	7.20	19.0	355	150	34.1	9.2	7.1	5.9	0.090	0.960	0.045	0.070	0.11	0.07	2.0	NA	NA	NA	4.0	1571	496	10.7
04/01/80	6.80	12.6	335	130	32.2	8.8	10.6	6.2	0.120	0.810	0.060	0.060	0.07	0.05	2.0	NA	NA	NA	3.1	1041	775	NA
04/06/80	7.20	21.3	500	73	58.2	14.2	14.0	8.8	0.110	1.000	0.040	0.090	0.04	0.09	2.0	NA	NA	NA	NA	2271	1300	NA
04/14/80	7.15	21.0	NA	326	86.7	22.3	23.6	12.0	0.105	1.490	0.080	0.100	0.03	0.12	7.0	NA	NA	NA	NA	1590	515	NA

Table A2.3. Drainage quality data from Duluth Complex test pile FL3, from 1978-1989 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/18/80	7.10	23.0	NA	364	88.8	21.6	24.8	11.8	0.120	2.100	0.100	0.130	0.02	0.18	4.0	NA	NA	NA	2.9	3449	703	NA
04/29/80	7.10	22.0	1860	1380	246.4	64.2	87.2	30.8	0.290	4.310	0.210	0.300	0.01	0.35	10.0	NA	NA	NA	NA	2082	180	11.0
05/08/80	7.10	20.9	2352	1720	324.0	90.4	136.0	42.6	0.260	4.290	0.240	0.370	0.02	0.34	13.0	NA	NA	NA	3.5	863	130	11.5
05/18/80	7.00	15.4	2300	1220	319.2	95.8	152.6	43.0	0.230	4.620	0.260	0.430	0.03	0.38	21.0	NA	NA	NA	NA	856	57	19.0
05/31/80	6.70	12.1	1300	720	159.2	42.7	57.9	24.3	0.270	3.140	0.180	0.250	0.01	0.23	36.0	NA	NA	NA	7.1	1673	775	NA
06/03/80	6.90	13.8	1500	850	191.8	45.8	57.2	27.4	0.250	3.790	0.210	0.280	0.01	0.26	46.0	NA	NA	NA	4.9	613	191	NA
06/06/80	6.80	11.0	1400	920	175.2	42.0	55.0	25.6	0.280	3.050	0.180	0.240	0.02	0.21	39.0	NA	NA	NA	NA	598	254	NA
06/13/80	6.80	11.0	1700	1040	219.2	50.9	73.9	30.3	0.315	3.370	0.195	0.290	0.03	0.24	28.0	NA	NA	NA	NA	413	207	NA
06/16/80	6.60	9.9	2200	1450	278.0	66.0	97.6	35.4	0.320	4.520	0.260	0.390	0.02	0.30	30.0	NA	NA	NA	NA	443	82	NA
06/27/80	6.60	11.0	2000	1180	252.0	67.0	89.0	35.0	0.230	3.490	0.230	0.360	0.02	0.28	18.0	NA	NA	NA	8.1	610	187	NA
07/02/80	6.60	11.0	1700	1000	230.0	57.0	66.0	29.0	0.460	3.760	0.270	0.370	0.04	0.30	27.0	NA	NA	NA	4.9	681	722	NA
07/02/80	6.70	8.5	1200	620	155.0	38.0	38.0	20.0	0.430	3.020	0.180	0.280	0.02	0.24	35.0	NA	NA	NA	NA	803	577	NA
07/13/80	6.60	12.1	2600	1620	359.6	80.0	133.6	51.0	0.560	5.350	0.280	0.480	0.02	0.41	30.0	NA	NA	NA	NA	598	108	NA
07/23/80	6.65	12.1	2350	1460	342.6	88.8	201.6	49.9	0.505	5.195	0.300	0.480	0.02	0.45	23.0	NA	NA	NA	NA	538	45	NA
08/03/80	6.65	11.0	NA	NA	352.4	95.4	162.0	50.4	0.530	6.240	0.310	0.540	0.02	0.51	NA	NA	NA	NA	5.9	189	80	NA
08/12/80	6.35	5.1	1100	595	128.8	35.3	44.6	21.4	0.670	3.570	0.200	0.300	0.06	0.34	12.0	NA	NA	NA	8.1	837	521	NA
08/20/80	6.35	5.1	1300	788	184.2	51.6	46.2	27.8	1.170	7.630	0.400	0.560	0.01	0.65	17.0	NA	NA	NA	NA	1030	166	NA
08/22/80	NA	NA	1400	700	80.0	42.0	42.0	20.0	1.360	7.040	0.410	0.590	0.04	0.62	21.0	NA	NA	NA	NA	655	269	NA
08/29/80	7.10	28.0	4850	2280	413.0	379.0	591.0	48.0	0.750	5.340	0.310	0.360	0.06	0.43	925.0	NA	NA	NA	9.5	1514	2500	NA
09/03/80	6.40	7.6	2020	1390	256.0	86.0	123.0	40.0	1.650	12.550	0.670	0.760	0.03	0.95	50.0	NA	NA	NA	NA	2105	512	NA
09/08/80	6.50	7.7	1912	1320	248.0	82.0	116.0	39.0	2.120	13.890	0.770	0.810	0.02	1.10	44.0	NA	NA	NA	NA	1336	758	15.0
09/12/80	6.60	11.0	1448	690	97.0	51.0	80.0	25.0	1.620	9.450	0.550	0.530	0.04	0.76	52.0	NA	NA	NA	4.0	3184	1512	18.0
09/19/80	6.60	11.0	2057	1320	255.0	92.0	128.0	39.0	2.660	12.780	0.670	0.720	0.05	0.90	77.0	NA	NA	NA	NA	7809	1846	13.0
10/01/80	6.40	10.2	2984	1750	410.0	136.0	181.0	NA	2.720	17.700	0.800	1.010	0.03	1.04	77.0	NA	NA	NA	3.4	2506	244	14.0
10/22/80	6.30	13.6	2575	1900	340.0	127.0	178.0	52.0	2.330	12.300	0.650	0.900	0.02	0.89	116.0	NA	NA	NA	4.3	1408	327	10.0
10/25/80	6.10	6.8	2205	1250	266.3	116.3	125.3	43.7	3.970	19.500	0.940	1.090	0.03	1.29	56.0	NA	NA	NA	NA	2044	321	10.0
03/02/81	6.60	6.8	900	590	103.4	43.6	42.2	NA	1.100	13.000	0.620	0.550	NA	NA	15.0	NA	NA	NA	NA	284	100	NA
03/16/81	6.50	6.4	1200	700	115.7	49.9	55.9	NA	1.300	23.000	0.950	0.800	NA	NA	NA	NA	NA	NA	NA	931	1393	NA
03/23/81	6.60	7.2	1224	825	119.6	58.5	60.7	NA	1.410	21.000	0.920	0.870	NA	NA	NA	NA	NA	NA	NA	821	526	9.0
03/26/81	6.30	5.6	600	272	46.8	25.9	19.5	NA	0.910	8.200	0.390	0.380	NA	NA	NA	NA	NA	NA	7.2	1158	723	NA
04/02/81	6.70	12.0	775	385	58.5	28.9	22.1	NA	1.590	15.000	0.690	0.550	NA	NA	NA	NA	NA	NA	NA	2055	506	NA
04/03/81	6.80	12.0	1350	625	122.2	58.8	66.3	NA	5.150	33.000	1.450	1.150	NA	NA	NA	NA	NA	NA	NA	3710	1229	NA
04/13/81	NA	NA	1100	555	96.0	56.0	33.0	NA	3.630	17.000	0.800	0.660	NA	NA	NA	NA	NA	NA	NA	2839	993	NA
04/17/81	6.90	10.4	1600	745	126.0	71.0	56.0	NA	3.730	19.000	0.860	0.710	NA	NA	NA	NA	NA	NA	NA	2680	1199	NA
04/23/81	7.00	14.4	1500	680	121.0	74.0	56.0	NA	2.300	13.000	0.620	0.550	NA	NA	NA	NA	NA	NA	NA	2063	2237	NA
04/28/81	7.20	16.0	1850	880	160.0	88.0	65.0	NA	2.760	17.000	0.830	0.730	NA	NA	NA	NA	NA	NA	3.2	4467	2436	NA
04/30/81	6.80	14.8	2176	1300	252.0	114.0	92.0	NA	3.680	25.800	1.230	1.070	NA	NA	NA	NA	NA	NA	4.0	3350	1200	10.0
05/13/81	6.80	8.8	2545	1750	324.0	152.0	120.0	NA	4.010	22.000	1.440	1.310	NA	NA	NA	NA	NA	NA	3.6	4115	100	17.0
06/02/81	6.70	9.1	2406	1700	300.0	138.0	118.0	NA	3.250	21.000	1.270	1.170	NA	NA	NA	NA	NA	NA	NA	1586	473	17.0
06/06/81	6.85	9.1	2157	1310	254.0	102.0	100.0	NA	3.290	18.000	1.190	1.060	NA	NA	NA	NA	NA	NA	4.1	2555	67	18.0
06/22/81	6.80	14.4	1250	1450	262.0	116.0	128.0	NA	2.840	19.000	1.110	0.880	NA	NA	NA	NA	NA	NA	NA	3482	1273	16.0
06/26/81	6.70	12.8	1330	680	156.0	58.0	42.0	NA	4.050	19.840	0.950	0.630	NA	NA	NA	NA	NA	NA	NA	3013	958	15.0
06/28/81	6.70	14.4	1862	800	226.0	88.0	70.0	NA	4.880	20.000	1.230	0.800	NA	NA	40.0	NA	NA	NA	3.5	4130	2493	15.0
07/04/81	6.85	13.5	2098	1400	300.0	102.0	88.8	NA	4.850	25.000	1.150	0.960	NA	NA	NA	NA	NA	NA	NA	2600	385	22.0
07/15/81	6.55	6.2	3046	2100	394.0	178.6	158.6	NA	6.200	37.000	1.650	1.530	NA	NA	NA	NA	NA	NA	NA	1283	120	20.0
07/25/81	5.60	NA	2564	1810	340.0	156.6	134.6	NA	5.400	30.000	1.430	1.360	NA	NA	23.0	NA	NA	NA	3.1	961	177	21.0
08/04/81	6.20	4.4	2574	1750	328.0	138.0	118.0	NA	5.500	30.000	1.470	1.320	NA	NA	NA	NA	NA	NA	NA	863	66	19.0
08/17/81	6.05	5.4	2485	1650	308.0	132.0	108.0	52.0	5.270	25.000	1.300	1.440	NA	NA	NA	NA	NA	NA	NA	723	46	22.0
09/01/81	6.00	4.8	2575	1570	284.0	136.0	98.0	NA	5.450	26.000	1.240	1.610	NA	NA	NA	NA	NA	NA	7.4	348	100	NA
09/07/81	5.90	3.2	2800	1540	268.0	124.0	88.0	NA	6.240	25.000	1.330	1.610	NA	NA	NA	NA	NA	NA	NA	390	129	17.0
09/16/81	5.90	3.8	3300	2000	354.0	171.0	115.0	NA	10.010	41.500	2.040	2.440	NA	NA	NA	NA	NA	NA	NA	265	22	NA

Table A2.3. Drainage quality data from Duluth Complex test pile FL3, from 1978-1989 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/27/81	5.85	2.4	1775	980	192.0	86.0	50.0	NA	6.300	22.000	1.140	1.330	NA	NA	NA	NA	NA	NA	4.7	598	251	NA
10/01/81	5.80	2.0	1700	850	180.0	82.0	32.0	NA	10.670	34.000	1.710	1.620	NA	NA	NA	NA	NA	NA	NA	2279	1459	NA
10/04/81	5.70	2.4	1700	1000	196.0	88.0	34.0	28.0	12.000	36.000	1.880	1.730	NA	NA	NA	NA	NA	NA	NA	1287	1013	NA
10/14/81	6.25	4.8	2150	1280	228.0	106.0	64.0	40.0	9.000	39.000	1.940	1.550	NA	NA	NA	NA	NA	NA	3.4	3165	1597	NA
10/18/81	6.25	4.0	2000	1310	232.0	100.0	56.0	38.0	9.000	36.000	1.880	1.430	NA	NA	NA	NA	NA	NA	NA	10569	1210	NA
11/09/81	6.65	11.2	4500	2800	410.0	320.0	240.0	84.0	10.980	62.000	2.480	2.250	NA	NA	NA	NA	NA	NA	3.5	4997	108	NA
03/25/82	6.55	7.2	900	470	73.0	53.4	20.0	14.0	4.140	19.000	0.940	0.690	NA	NA	NA	NA	NA	NA	NA	2839	NA	NA
04/01/82	6.18	5.2	1325	800	105.0	76.4	28.0	18.0	6.275	31.000	1.470	1.490	NA	NA	NA	NA	NA	NA	NA	3975	NA	NA
04/13/82	6.60	7.2	690	320	52.0	34.4	14.0	10.0	2.860	13.000	0.670	0.490	NA	NA	NA	NA	NA	NA	NA	4728	NA	NA
04/15/82	6.60	9.0	750	390	63.8	41.6	16.0	12.0	2.120	13.000	0.580	0.420	NA	NA	NA	NA	NA	NA	NA	3774	NA	NA
04/19/82	6.60	7.7	1125	705	129.0	83.0	29.0	20.0	3.395	19.500	1.010	0.640	NA	NA	NA	NA	NA	NA	NA	1961	NA	NA
04/23/82	6.50	5.4	1500	950	188.0	110.0	80.0	24.0	4.450	26.000	1.340	0.860	NA	NA	NA	NA	NA	NA	NA	2438	NA	NA
04/28/82	6.50	7.7	1875	1000	160.0	128.0	68.0	32.0	4.520	25.000	1.380	0.950	NA	NA	NA	NA	NA	NA	NA	3721	NA	NA
05/03/82	6.40	5.4	2290	NA	240.0	164.0	104.0	42.0	6.490	40.000	2.030	1.360	NA	NA	NA	NA	NA	NA	NA	466	NA	NA
05/07/82	6.90	17.4	1960	975	133.0	125.0	109.0	25.0	3.630	20.300	1.105	0.740	NA	NA	NA	NA	NA	NA	NA	4527	NA	NA
05/11/82	6.90	21.2	1800	1040	156.0	122.0	64.0	28.0	4.410	24.000	1.400	0.890	NA	NA	NA	NA	NA	NA	NA	2782	NA	NA
05/14/82	6.80	16.2	2500	1465	202.0	160.0	110.0	40.0	7.250	44.000	2.410	1.650	NA	NA	NA	NA	NA	NA	NA	750	NA	NA
05/18/82	6.80	18.9	2450	1340	206.0	160.0	128.0	40.0	6.800	42.000	2.270	1.580	NA	NA	NA	NA	NA	NA	NA	681	NA	NA
05/21/82	6.25	NA	3200	2400	292.0	163.0	93.0	53.0	10.160	70.000	2.560	2.010	0.04	2.81	NA	NA	NA	NA	NA	3062	NA	NA
06/02/82	6.10	9.9	3600	2750	340.0	238.0	129.5	63.5	11.405	79.500	3.015	2.580	0.06	3.36	NA	NA	NA	NA	NA	1298	NA	NA
06/11/82	7.30	NA	3150	NA	306.0	183.0	139.0	52.0	5.230	45.000	1.920	1.480	0.05	1.96	NA	NA	NA	NA	NA	NA	NA	NA
06/12/82	7.70	NA	3450	NA	324.0	178.0	174.0	46.0	3.330	29.000	1.300	0.970	0.05	1.27	NA	NA	NA	NA	NA	NA	NA	NA
06/18/82	6.50	9.9	2500	1500	248.0	133.0	65.0	39.0	6.950	43.000	1.870	1.520	0.04	1.89	NA	NA	NA	NA	NA	148	NA	NA
07/02/82	7.40	81.6	2750	1300	297.0	136.0	178.0	34.0	2.290	21.090	0.770	0.560	0.03	0.78	NA	NA	NA	NA	NA	2824	NA	NA
07/08/82	6.35	11.7	2700	2000	354.0	150.0	108.0	48.0	8.850	67.000	2.300	1.560	0.02	2.56	NA	NA	NA	NA	NA	7291	NA	NA
07/26/82	6.37	10.6	2705	1675	298.0	149.0	77.0	51.0	7.570	79.500	3.240	1.610	0.39	3.03	NA	NA	NA	NA	NA	6261	NA	NA
08/30/82	6.91	50.3	2087	1175	298.0	126.0	70.0	40.0	6.240	45.000	1.710	1.150	0.13	1.98	NA	NA	NA	NA	NA	7779	NA	NA
10/04/82	6.65	14.6	2145	1357	223.0	126.0	59.0	44.0	9.520	80.000	3.220	1.620	0.13	2.94	NA	NA	NA	NA	NA	20301	NA	NA
11/01/82	6.20	3.6	2825	NA	248.0	160.0	72.0	54.0	9.490	91.000	3.650	1.870	0.02	3.52	NA	NA	NA	NA	NA	1257	NA	NA
11/22/82	6.10	4.5	2140	1600	270.0	116.0	64.0	NA	13.750	91.000	3.340	2.190	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
03/18/83	5.95	3.0	1775	1120	NA	NA	NA	NA	9.010	65.000	2.590	2.030	NA	NA	NA	NA	NA	NA	NA	8206	NA	NA
04/14/83	6.66	16.3	1070	510	NA	NA	NA	NA	5.010	41.000	1.410	1.060	NA	NA	NA	NA	NA	NA	NA	7414	NA	NA
05/15/83	6.41	6.9	1574	1000	NA	NA	NA	NA	6.440	58.000	2.140	1.690	NA	NA	NA	NA	NA	NA	NA	5109	NA	NA
06/16/83	6.19	6.9	1782	1300	NA	NA	NA	NA	5.360	43.000	1.720	1.280	NA	NA	NA	NA	NA	NA	NA	2717	NA	NA
07/14/83	5.90	5.0	2008	1680	NA	NA	NA	NA	12.030	54.000	2.830	2.320	NA	NA	NA	NA	NA	NA	NA	7748	NA	NA
08/14/83	5.41	3.4	2095	1750	NA	NA	NA	NA	16.750	71.000	3.300	2.880	NA	NA	NA	NA	NA	NA	NA	5553	NA	NA
09/15/83	5.64	2.5	2123	1940	NA	NA	NA	NA	16.500	75.000	3.590	3.080	NA	NA	NA	NA	NA	NA	NA	4565	NA	NA
10/13/83	5.70	3.8	2422	1880	NA	NA	NA	NA	19.000	78.000	3.770	3.140	NA	NA	NA	NA	NA	NA	NA	20303	NA	NA
11/09/83	5.90	3.6	2300	2600	NA	NA	NA	NA	20.400	99.000	4.440	3.320	NA	NA	NA	NA	NA	NA	NA	3925	NA	NA
11/30/83	5.50	2.7	2175	1960	NA	NA	NA	NA	20.200	89.600	4.050	3.040	NA	NA	NA	NA	NA	NA	NA	2589	NA	NA
02/24/84	4.80	0.9	1900	2040	236.3	189.9	50.7	NA	30.600	102.000	4.500	3.840	NA	NA	NA	NA	NA	NA	NA	2907	NA	NA
03/15/84	5.40	2.7	2375	2000	288.3	274.4	65.1	NA	35.000	144.000	7.000	4.890	NA	NA	NA	NA	NA	NA	NA	4375	NA	NA
04/15/84	6.40	10.4	878	580	104.0	70.6	19.9	NA	7.490	34.000	1.680	1.300	NA	NA	NA	NA	NA	NA	NA	15221	NA	NA
05/16/84	6.16	6.9	1190	980	165.6	109.4	31.2	NA	12.000	51.000	2.400	1.820	NA	NA	NA	NA	NA	NA	NA	8861	NA	NA
06/13/84	5.52	2.6	1519	1310	205.9	139.6	40.8	NA	17.500	69.400	3.150	2.290	NA	NA	NA	NA	NA	NA	NA	17886	NA	NA
07/14/84	5.05	2.5	2130	2480	NA	NA	57.4	NA	19.000	121.300	4.000	3.200	NA	NA	NA	NA	NA	NA	NA	6064	NA	NA
08/15/84	5.34	3.0	2050	1800	267.8	142.9	49.6	NA	16.100	73.760	4.880	2.360	NA	NA	NA	NA	NA	NA	NA	4478	NA	NA
09/12/84	4.96	2.3	2340	1760	317.1	220.0	70.5	NA	26.500	105.000	7.120	3.440	NA	NA	NA	NA	NA	NA	NA	1340	NA	NA
10/13/84	5.02	2.1	1860	1180	259.1	149.5	43.2	NA	34.000	92.000	6.830	3.010	NA	NA	NA	NA	NA	NA	NA	10579	NA	NA
11/14/84	4.96	1.6	2190	2700	381.0	450.0	84.7	NA	38.600	138.000	5.380	4.180	NA	NA	NA	NA	NA	NA	NA	2918	NA	NA
12/12/84	4.93	0.8	1585	1800	242.0	290.0	62.0	NA	26.500	108.800	4.840	3.490	NA	NA	NA	NA	NA	NA	NA	3459	NA	NA

Table A2.3. Drainage quality data from Duluth Complex test pile FL3, from 1978-1989 (0.63% S, 0.35% Cu, 0.08% Ni).

Date[1]	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/02/85	6.60	17.9	800	470	75.8	54.2	14.5	NA	5.800	27.000	1.070	0.860	NA	NA	NA	NA	NA	NA	NA	2394	NA	NA
04/16/85	5.94	9.0	1360	1010	144.0	121.6	28.6	NA	18.500	70.000	2.800	2.090	NA	NA	NA	NA	NA	NA	NA	16699	NA	NA
05/14/85	5.25	3.9	2400	2000	220.0	210.0	53.0	NA	24.000	100.000	4.710	2.830	NA	NA	NA	NA	NA	NA	NA	6828	NA	NA
06/14/85	5.57	4.0	2150	1736	209.0	169.0	46.3	NA	27.000	103.000	4.140	3.210	NA	NA	NA	NA	NA	NA	NA	26461	NA	NA
07/15/85	5.31	4.3	2300	1900	220.0	186.0	51.0	NA	23.000	99.000	4.310	3.490	NA	NA	NA	NA	NA	NA	NA	7630	NA	NA
08/13/85	5.47	4.0	2125	1750	226.0	153.4	48.0	NA	22.400	89.000	4.240	3.170	NA	NA	NA	NA	NA	NA	NA	5761	NA	NA
09/13/85	5.39	3.8	2325	1850	294.0	141.8	44.0	NA	39.000	100.600	4.820	3.560	NA	NA	NA	NA	NA	NA	NA	18187	NA	NA
10/15/85	5.44	4.7	2425	1650	239.0	160.0	43.0	NA	37.600	102.800	4.660	3.010	NA	NA	NA	NA	NA	NA	NA	17222	NA	NA
11/15/85	5.26	2.6	2675	1450	322.0	224.0	60.0	NA	41.800	133.000	6.020	4.020	NA	NA	NA	NA	NA	NA	NA	1949	NA	NA
03/20/86	5.25	2.2	3100	2900	366.0	334.0	78.0	NA	52.400	220.000	7.900	5.200	NA	NA	NA	NA	NA	NA	NA	401	NA	NA
03/23/86	6.30	9.5	975	700	90.0	67.2	13.0	NA	17.000	48.000	2.120	1.310	NA	NA	NA	NA	NA	NA	NA	17998	NA	NA
04/16/86	6.09	7.4	1175	700	126.0	84.0	18.0	NA	17.000	51.000	2.540	1.590	NA	NA	NA	NA	NA	NA	NA	13743	NA	NA
05/14/86	5.70	4.4	1650	1350	214.0	144.0	28.0	NA	29.000	120.000	5.220	2.330	NA	NA	NA	NA	NA	NA	NA	10995	NA	NA
06/11/86	5.29	3.6	2425	1650	234.0	173.0	49.0	NA	31.000	108.000	4.360	3.690	NA	NA	NA	NA	NA	NA	NA	3509	NA	NA
07/12/86	5.40	3.7	1775	1100	180.0	130.0	32.0	NA	27.000	80.000	3.350	2.690	NA	NA	NA	NA	NA	NA	NA	7104	NA	NA
08/13/86	5.17	3.0	2200	1550	228.0	155.6	40.0	NA	37.000	95.000	4.200	2.280	NA	NA	NA	NA	NA	NA	NA	4273	NA	NA
09/14/86	5.21	3.0	2000	1400	224.0	159.8	36.0	NA	41.000	98.000	5.100	2.360	NA	NA	NA	NA	NA	NA	NA	18682	NA	NA
10/16/86	5.11	2.1	2900	1825	258.0	227.0	48.0	NA	48.000	142.000	5.750	4.150	NA	NA	NA	NA	NA	NA	NA	4557	NA	NA
11/16/86	5.09	2.7	3150	1900	290.0	208.0	62.0	NA	72.000	161.000	9.100	6.100	NA	NA	NA	NA	NA	NA	NA	7899	NA	NA
03/22/87	5.30	7.1	1125	730	126.0	80.0	22.0	NA	20.200	60.000	2.780	1.930	NA	NA	NA	NA	NA	NA	NA	8656	NA	NA
04/15/87	5.73	4.6	1450	900	118.0	100.0	22.0	NA	28.000	68.000	3.070	2.240	NA	NA	NA	NA	NA	NA	NA	3861	NA	NA
05/16/87	5.13	4.5	2300	1770	319.0	256.0	56.0	NA	51.000	129.000	5.700	4.700	NA	NA	NA	NA	NA	NA	NA	24103	NA	NA
06/16/87	4.81	3.0	3550	2640	356.0	328.0	70.0	NA	63.000	197.000	6.000	4.100	NA	NA	NA	NA	NA	NA	NA	2048	NA	NA
07/15/87	5.05	5.7	2325	1780	226.0	187.0	45.0	NA	53.000	131.000	5.200	3.950	NA	NA	NA	NA	NA	NA	NA	18842	NA	NA
08/15/87	4.91	3.4	2425	1750	206.0	172.0	42.0	NA	43.000	121.000	4.800	3.720	NA	NA	NA	NA	NA	NA	NA	8501	NA	NA
09/16/87	5.01	2.8	2650	1615	200.0	158.0	36.0	NA	42.000	105.000	4.510	3.120	NA	NA	NA	NA	NA	NA	NA	12195	NA	NA
10/13/87	4.81	2.1	3825	2680	323.0	307.0	70.0	NA	71.000	192.000	7.730	5.430	NA	NA	NA	NA	NA	NA	NA	2142	NA	NA
11/10/87	4.71	1.8	3350	2700	288.0	278.0	64.0	NA	59.000	174.000	6.880	4.930	NA	NA	NA	NA	NA	NA	NA	1064	NA	NA
03/28/88	6.00	11.1	900	414	88.0	48.0	12.0	NA	11.000	32.000	1.600	1.600	NA	NA	NA	NA	NA	NA	NA	8653	NA	NA
04/17/88	5.69	6.3	1575	950	136.0	100.0	22.0	NA	24.000	90.000	3.800	4.100	NA	NA	NA	NA	NA	NA	NA	7297	NA	NA
05/12/88	5.17	3.4	1985	1270	172.0	160.0	38.0	NA	37.000	129.000	5.200	6.100	NA	NA	NA	NA	NA	NA	NA	10624	NA	NA
06/12/88	4.86	2.9	3500	2440	344.0	324.0	67.0	NA	70.000	178.500	10.200	11.500	NA	NA	NA	NA	NA	NA	NA	1654	NA	NA
07/15/88	4.91	3.5	3175	2230	278.0	286.0	58.0	NA	59.000	161.000	8.800	11.200	NA	NA	NA	NA	NA	NA	NA	995	NA	NA
08/16/88	5.26	2.5	1910	1240	176.0	130.0	34.0	NA	37.000	125.000	4.700	4.200	NA	NA	NA	NA	NA	NA	NA	45053	NA	NA
09/16/88	5.07	2.7	2300	1360	202.0	170.0	44.0	NA	41.000	145.000	5.700	5.300	NA	NA	NA	NA	NA	NA	NA	8134	NA	NA
10/18/88	5.12	3.0	2000	1330	158.0	144.0	32.0	NA	33.000	93.000	5.000	5.500	NA	NA	NA	NA	NA	NA	NA	3376	NA	NA
11/15/88	5.21	3.6	1625	1250	142.0	131.0	26.0	NA	32.500	87.000	4.700	5.000	NA	NA	NA	NA	NA	NA	NA	6654	NA	NA
12/05/88	4.80	3.2	3300	3150	298.0	350.0	62.0	NA	73.000	238.000	11.900	10.000	NA	NA	NA	NA	NA	NA	NA	1681	NA	NA
04/07/89	6.15	9.9	650	425	70.0	46.0	12.0	NA	12.700	32.000	1.900	1.700	NA	NA	NA	NA	NA	NA	NA	23758	NA	NA
04/24/89	5.91	6.5	1075	650	100.0	58.0	12.0	NA	16.000	46.000	2.500	2.500	NA	NA	NA	NA	NA	NA	NA	7638	NA	NA
05/16/89	5.18	3.9	1593	1090	128.0	108.0	20.0	NA	28.000	79.000	4.500	5.300	NA	NA	NA	NA	NA	NA	NA	9284	NA	NA
06/17/89	4.98	3.3	1821	1375	156.0	128.0	30.0	NA	41.000	119.000	6.800	5.000	NA	NA	NA	NA	NA	NA	NA	29326	NA	NA
07/17/89	4.81	2.2	2800	1950	306.0	210.0	46.0	NA	66.000	185.000	8.700	9.100	NA	NA	NA	NA	NA	NA	NA	2967	NA	NA
08/14/89	4.94	3.1	2900	1970	248.0	222.0	48.0	NA	61.000	240.000	8.200	8.200	NA	NA	NA	NA	NA	NA	NA	1328	NA	NA
09/14/89	4.90	2.7	2400	1600	252.0	166.0	36.0	NA	56.000	148.000	9.500	6.700	NA	NA	NA	NA	NA	NA	NA	17888	NA	NA
10/16/89	4.83	0.6	2775	1800	218.0	186.0	46.0	NA	47.000	161.000	12.100	8.400	NA	NA	NA	NA	NA	NA	NA	1835	NA	NA
11/10/89	5.00	2.8	2400	1820	240.0	182.0	40.0	NA	42.000	137.000	14.800	7.700	NA	NA	NA	NA	NA	NA	NA	900	NA	NA

NA: Not analyzed.

[1]Dates represent intermediate dates for measurement periods.

Table A2.4. Drainage quality data from Duluth Complex test pile FL4, from 1978-1982 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.20	NA	1100	NA	121.0	11.0	84.0	27.0	0.018	NA	0.004	0.048	NA	0.10	NA	NA	NA	NA	NA	NA	NA	4.0
05/25/78	7.40	NA	1750	NA	204.0	20.0	165.0	35.0	0.012	0.210	0.012	0.112	0.02	0.19	NA	NA	NA	NA	NA	NA	NA	NA
06/14/78	7.60	NA	2500	NA	228.0	29.0	262.0	37.0	0.024	0.220	0.016	0.128	0.04	0.21	NA	NA	NA	NA	NA	125	NA	9.0
06/16/78	7.70	NA	2600	NA	247.0	30.0	277.0	41.0	0.026	0.175	0.015	0.120	0.04	0.26	NA	NA	NA	NA	NA	94	NA	11.0
06/23/78	7.56	NA	2402	1150	205.0	27.8	271.4	39.5	0.022	0.308	0.023	0.134	0.03	0.21	25.5	NA	NA	NA	NA	307	NA	12.7
06/26/78	7.90	NA	2150	900	186.0	24.0	260.0	41.0	0.021	0.260	0.019	0.084	NA	0.18	28.0	NA	NA	NA	NA	5	NA	17.0
07/06/78	7.58	NA	1825	850	212.0	25.0	182.0	52.0	0.019	0.275	0.019	0.073	0.03	0.18	11.0	NA	NA	0.10	NA	88	NA	17.0
07/07/78	7.43	NA	1306	546	156.0	15.5	129.9	31.4	0.015	0.183	0.019	0.064	0.03	0.11	7.8	1.6	0.73	0.08	NA	4957	NA	11.0
07/08/78	7.46	NA	1406	830	220.2	16.8	127.0	41.1	0.014	0.206	0.020	0.058	0.03	0.14	6.8	NA	NA	0.08	NA	NA	NA	NA
07/09/78	7.55	NA	1550	NA	300.0	20.0	149.0	46.0	0.009	0.240	0.017	NA	0.02	0.16	NA	NA	NA	NA	NA	NA	NA	NA
07/13/78	7.65	NA	NA	850	273.0	18.0	153.0	44.0	0.012	NA	0.015	NA	0.02	0.14	7.0	2.2	0.90	0.08	NA	NA	NA	NA
08/04/78	8.00	NA	1100	660	191.0	13.0	115.0	43.0	0.020	0.200	0.010	0.052	0.02	0.14	3.0	1.2	0.06	0.08	NA	95	NA	22.0
08/15/78	7.80	39.0	1275	800	201.0	19.0	158.0	39.0	0.020	0.240	0.010	0.070	0.02	0.15	6.0	1.2	1.20	0.08	NA	NA	NA	21.0
08/16/78	7.50	28.0	825	100	118.0	11.0	60.0	26.0	0.020	0.130	0.020	0.030	0.03	0.09	1.0	1.2	NA	0.08	NA	NA	NA	NA
08/17/78	7.29	29.0	1175	1000	355.0	41.0	158.0	12.0	0.030	0.200	0.020	0.050	0.04	0.12	13.0	2.2	NA	0.08	NA	NA	NA	NA
08/18/78	7.60	36.0	1850	800	219.0	21.0	110.0	42.0	0.020	0.240	0.040	0.070	0.02	0.16	3.0	1.4	NA	0.08	NA	NA	NA	NA
08/21/78	7.75	NA	2050	NA	271.0	27.0	142.0	50.0	0.080	0.320	0.040	0.100	NA	0.20	NA	NA	NA	NA	NA	NA	NA	NA
08/23/78	7.47	30.0	1217	603	137.7	15.0	84.7	32.3	0.020	0.143	0.023	0.037	0.02	0.10	2.3	1.4	1.08	0.08	NA	NA	NA	23.0
08/25/78	7.55	33.0	1750	800	216.0	21.0	115.0	46.0	0.020	0.240	0.030	0.060	0.01	0.14	2.0	1.4	1.00	0.08	NA	NA	NA	23.0
08/28/78	7.55	41.8	2350	1250	280.0	29.0	187.0	60.0	0.030	0.340	0.040	0.100	0.02	0.19	2.0	2.4	NA	0.08	NA	NA	NA	NA
09/11/78	7.30	35.0	2300	1200	256.0	26.0	219.0	59.0	0.030	0.340	0.050	0.080	0.06	0.20	10.0	NA	NA	0.08	NA	NA	NA	NA
09/13/78	7.23	34.0	1650	852	191.7	17.7	107.3	44.3	0.023	0.210	0.033	0.053	0.06	0.14	2.5	1.4	0.06	0.08	NA	NA	NA	12.0
10/17/78	7.50	31.0	1800	1000	257.0	27.0	157.0	34.0	0.020	0.290	0.040	0.090	0.02	0.18	6.0	1.4	0.06	0.08	NA	NA	NA	NA
03/19/79	NA	NA	861	380	96.6	18.2	39.4	11.2	3.000	14.600	0.256	0.400	0.03	1.08	8.0	NA	NA	NA	4.1	2660	2428	NA
04/17/79	5.90	7.0	635	325	77.0	14.0	34.4	10.6	1.120	12.310	0.240	0.240	1.46	1.11	7.0	NA	NA	NA	3.5	6041	4676	NA
04/18/79	6.80	22.0	401	170	66.0	6.0	25.0	9.0	0.069	0.373	0.064	0.050	0.02	0.15	3.0	1.2	NA	NA	NA	4644	5152	NA
04/19/79	7.10	21.0	391	170	65.0	5.0	23.0	8.6	0.016	0.098	0.008	0.040	0.02	0.07	1.0	0.5	NA	NA	NA	5560	5332	NA
04/20/79	6.90	29.0	568	285	106.0	8.6	NA	14.2	0.064	0.210	0.008	0.040	0.02	0.10	3.0	0.6	0.12	NA	2.2	4364	3329	NA
04/20/79	6.40	22.0	364	145	67.0	4.8	24.0	17.8	0.014	0.089	0.008	0.030	0.02	0.07	2.0	0.4	0.05	NA	NA	3077	3329	NA
04/22/79	7.40	19.0	980	483	171.0	15.8	42.6	24.4	0.021	0.270	0.020	0.070	0.03	0.14	4.0	1.0	0.03	NA	NA	4599	1534	NA
05/09/79	7.50	NA	1039	NA	171.2	15.8	48.2	27.4	0.016	0.310	0.023	0.070	0.03	0.17	NA	NA	NA	NA	NA	647	1239	NA
05/10/79	7.50	NA	1010	532	162.4	16.6	50.4	25.6	0.021	0.280	0.020	0.060	0.01	0.15	4.0	NA	NA	NA	NA	8717	4417	NA
05/20/79	6.90	NA	1363	817	227.8	23.6	77.9	35.2	0.026	0.375	0.029	0.080	0.02	0.17	5.0	NA	NA	NA	5.7	3062	1080	NA
05/22/79	7.60	NA	984	583	178.4	18.8	68.8	30.0	0.025	0.320	0.023	0.080	0.01	0.15	3.0	NA	NA	NA	3.5	1904	1278	NA
05/23/79	7.20	NA	1264	700	193.4	21.6	74.0	32.8	0.030	0.360	0.027	0.070	0.02	0.16	4.0	NA	NA	NA	4.5	3357	699	NA
06/01/79	7.70	NA	1334	800	201.8	22.0	80.4	34.8	0.027	0.350	0.027	0.080	0.03	0.17	5.0	NA	NA	NA	4.7	2290	662	NA
06/06/79	7.60	NA	1213	700	180.6	19.4	48.4	48.2	0.023	0.300	0.021	0.080	0.01	0.14	5.0	NA	NA	NA	NA	2044	1227	NA
06/10/79	7.30	36.0	1116	560	159.4	18.0	45.0	41.6	0.023	0.240	0.013	0.060	0.03	0.11	4.0	NA	NA	NA	NA	2161	1411	22.0
06/12/79	7.30	32.0	1107	NA	157.0	17.2	43.6	40.8	0.013	0.240	0.009	0.060	0.01	0.11	NA	NA	NA	NA	NA	681	245	NA
06/15/79	7.30	33.0	1138	NA	160.4	18.4	45.8	37.6	0.016	0.280	0.009	0.060	0.01	0.12	NA	NA	NA	NA	NA	469	442	21.0
06/20/79	7.10	30.0	1347	570	171.0	21.0	72.0	38.0	0.020	0.260	0.020	0.070	0.03	0.97	5.0	NA	NA	NA	NA	2854	1339	17.0
06/30/79	NA	NA	NA	NA	176.0	21.0	73.0	35.0	0.010	0.270	0.020	0.050	0.01	0.10	NA	NA	NA	NA	NA	526	NA	NA
07/30/79	5.75	32.0	907	460	118.0	22.0	38.0	20.0	0.700	4.140	0.260	0.180	0.01	0.55	11.0	NA	NA	NA	4.9	2324	2847	23.0
09/01/79	7.00	10.4	1112	650	189.6	16.8	40.4	33.8	0.040	0.510	0.028	0.090	0.01	0.14	4.0	NA	NA	NA	4.0	4307	1988	20.5
09/05/79	7.10	14.4	1333	NA	246.4	22.4	46.8	40.0	0.040	0.670	0.034	0.120	0.01	0.19	NA	NA	NA	NA	5.1	337	270	20.0
09/12/79	7.00	12.8	1250	800	228.6	20.0	45.6	38.2	0.060	0.780	0.060	0.130	0.01	0.19	6.0	NA	NA	NA	3.7	1158	417	20.0
09/13/79	7.20	NA	1321	NA	237.8	21.4	55.6	43.4	0.030	0.710	0.035	0.110	0.02	0.18	NA	NA	NA	NA	NA	91	270	20.0
10/19/79	6.40	7.2	980	600	166.4	14.6	24.0	27.8	0.080	1.260	0.070	0.160	0.01	0.13	2.0	NA	NA	NA	1.9	541	307	NA
10/31/79	6.60	8.8	1025	550	152.0	15.0	38.0	28.8	0.080	1.040	0.060	0.140	0.01	0.12	3.0	NA	NA	NA	2.2	1764	908	NA
03/31/80	6.90	15.0	470	205	54.2	5.4	12.6	11.6	0.030	0.395	0.035	0.090	0.02	0.06	2.0	NA	NA	NA	1.8	1207	1154	NA
04/05/80	7.10	17.2	325	140	38.5	4.0	8.0	8.3	0.040	0.270	0.010	0.070	0.02	0.04	1.0	NA	NA	NA	1.0	5326	2260	NA

Table A2.4. Drainage quality data from Duluth Complex test pile FL4, from 1978-1982 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/06/80	6.80	16.1	575	250	76.0	7.0	11.2	13.0	0.050	0.410	0.030	0.100	0.03	0.12	2.0	NA	NA	NA	NA	1298	1349	NA
04/18/80	6.90	12.7	NA	258	83.9	7.8	13.3	14.5	0.035	0.695	0.035	0.150	0.02	0.14	2.0	NA	NA	NA	1.0	2215	450	NA
06/01/80	6.50	6.6	950	710	174.9	17.1	25.1	30.8	0.160	2.030	0.120	0.230	0.02	0.25	4.0	NA	NA	NA	NA	579	313	NA
06/06/80	6.80	6.6	1100	NA	164.6	16.6	24.4	28.6	0.180	2.250	0.160	0.270	0.03	0.27	NA	NA	NA	NA	NA	246	232	NA
07/03/80	6.70	7.6	1150	770	186.0	20.0	33.0	29.0	0.250	2.140	0.120	0.280	0.02	0.24	3.0	NA	NA	NA	NA	1344	615	NA
08/12/80	6.50	6.0	1400	860	224.8	28.8	34.4	35.6	1.140	8.380	0.440	0.510	0.01	0.46	3.0	NA	NA	NA	2.3	1090	683	NA
08/19/80	6.90	8.5	1500	975	263.8	31.6	51.9	43.5	0.690	5.990	0.300	0.460	0.01	0.53	4.0	NA	NA	NA	NA	1594	464	NA
08/22/80	NA	NA	1650	1100	275.0	37.0	57.0	36.0	0.490	4.080	0.250	0.350	0.03	0.38	6.0	NA	NA	NA	NA	1503	1084	NA
08/29/80	5.50	3.4	1360	1040	160.0	17.0	52.0	34.0	0.750	4.810	0.310	0.270	0.06	0.46	11.0	NA	NA	NA	3.4	11432	7250	NA
09/07/80	6.70	13.6	1695	1170	272.0	24.0	99.0	48.0	0.400	2.700	0.150	0.250	0.03	0.29	6.0	NA	NA	NA	NA	2142	1860	NA
09/11/80	6.80	11.5	1246	710	132.0	9.0	59.0	36.0	0.170	1.840	0.100	0.200	0.04	0.23	5.0	NA	NA	NA	NA	969	655	15.0
09/12/80	6.80	12.8	1275	780	142.0	11.0	60.0	35.0	0.350	2.180	0.110	0.190	0.04	0.23	4.0	NA	NA	NA	2.6	4895	3026	16.0
09/18/80	7.00	14.4	1450	910	191.0	16.0	67.0	38.0	0.310	1.850	0.090	0.180	0.04	0.18	5.0	NA	NA	NA	NA	10209	3354	14.0
09/23/80	6.40	17.8	1822	1150	313.2	48.4	96.0	12.8	0.320	2.610	0.140	0.270	0.03	0.22	5.0	NA	NA	NA	3.3	1579	878	14.0
10/23/80	6.50	12.7	1598	1050	270.0	28.0	73.0	45.0	1.240	5.800	0.280	0.420	0.01	0.40	6.0	NA	NA	NA	2.5	1571	250	10.0
10/26/80	6.50	8.5	1576	1050	238.0	32.0	60.0	46.0	0.480	4.160	0.230	0.310	0.02	0.36	5.0	NA	NA	NA	NA	568	148	13.0
02/20/81	5.20	2.6	635	325	63.7	19.2	27.3	NA	1.360	7.100	0.450	0.270	NA	NA	5.0	NA	NA	NA	NA	1601	264	7.0
03/23/81	7.00	6.4	598	305	69.5	10.5	19.5	NA	0.140	1.470	0.085	0.160	NA	NA	NA	NA	NA	NA	NA	1949	750	10.0
03/30/81	6.70	7.2	500	215	58.5	9.0	10.4	NA	0.260	1.720	0.110	0.150	NA	NA	NA	NA	NA	NA	1.7	9089	2006	NA
04/05/81	6.90	12.0	877	490	119.6	18.5	22.1	NA	0.520	4.210	0.220	0.310	NA	NA	NA	NA	NA	NA	NA	6367	1136	NA
04/14/81	NA	NA	NA	610	156.0	29.0	27.0	NA	0.460	3.230	0.170	0.260	NA	NA	NA	NA	NA	NA	NA	3660	1760	NA
04/17/81	7.20	13.6	1300	660	159.0	31.0	30.0	NA	0.340	2.900	0.150	0.230	NA	NA	NA	NA	NA	NA	1.7	2862	2063	NA
04/23/81	7.00	13.6	1400	660	179.0	35.0	31.0	NA	0.320	2.540	0.140	0.220	NA	NA	NA	NA	NA	NA	NA	3895	2177	NA
04/24/81	7.15	18.6	1400	700	168.0	35.5	32.0	NA	0.220	1.860	0.100	0.160	NA	NA	NA	NA	NA	NA	NA	7730	2460	NA
04/28/81	7.20	NA	1325	640	153.0	34.0	31.0	NA	0.180	1.580	0.100	0.140	NA	NA	NA	NA	NA	NA	1.7	4376	3237	NA
04/30/81	7.20	20.8	1430	845	220.0	36.0	38.0	NA	0.180	1.920	0.100	0.190	NA	NA	NA	NA	NA	NA	NA	4092	856	12.0
05/13/81	7.30	18.4	1574	700	276.0	40.0	48.0	NA	0.160	2.290	0.120	0.230	NA	NA	NA	NA	NA	NA	2.6	1749	42	18.5
06/01/81	7.00	16.1	1638	1000	262.0	40.0	64.0	NA	0.720	4.620	0.260	0.380	NA	NA	NA	NA	NA	NA	NA	3028	1563	16.0
06/07/81	7.10	9.1	1264	685	198.0	26.0	42.0	NA	0.360	3.000	0.170	0.270	NA	NA	NA	NA	NA	NA	3.6	2688	202	18.0
06/22/81	6.70	10.4	1240	675	194.0	28.0	42.0	NA	0.610	3.950	0.220	0.250	NA	NA	NA	NA	NA	NA	NA	5617	2156	16.0
06/24/81	6.85	8.0	1152	605	187.0	24.0	31.0	NA	0.450	2.980	0.160	0.190	NA	NA	NA	NA	NA	NA	NA	4747	1760	16.0
06/28/81	6.90	10.4	1197	665	192.0	26.0	36.0	NA	0.380	2.300	0.130	0.160	NA	NA	NA	NA	NA	NA	2.5	6488	2244	15.0
07/04/81	7.15	13.8	1528	1010	278.0	32.6	50.4	NA	0.240	1.690	0.090	0.180	NA	NA	NA	NA	NA	NA	NA	4247	167	22.0
07/17/81	6.95	12.0	1916	1300	332.0	46.6	81.0	NA	0.830	4.790	0.230	0.470	NA	NA	NA	NA	NA	NA	NA	787	481	20.0
07/25/81	6.85	8.4	1903	1200	328.0	46.6	75.8	NA	1.600	7.420	0.370	0.540	NA	NA	2.0	NA	NA	NA	1.9	1291	256	19.0
08/10/81	6.60	7.4	1711	1060	276.0	50.0	68.0	NA	2.000	8.000	0.470	0.590	NA	NA	NA	NA	NA	NA	NA	1548	560	18.0
08/13/81	6.80	10.6	1767	1050	282.0	44.0	64.0	48.0	0.690	6.000	0.390	0.460	NA	NA	NA	NA	NA	NA	NA	333	55	22.0
08/31/81	6.30	4.8	1800	1000	254.0	48.0	60.0	NA	3.840	11.290	0.590	0.740	NA	NA	NA	NA	NA	NA	1.9	496	337	NA
09/07/81	6.45	4.8	2250	1140	288.0	52.0	60.0	NA	3.030	12.000	0.640	0.730	NA	NA	NA	NA	NA	NA	NA	893	771	NA
09/26/81	6.00	3.2	2025	1190	286.0	56.0	58.0	NA	3.840	15.000	0.830	0.850	NA	NA	NA	NA	NA	NA	1.6	1688	1105	NA
10/01/81	6.40	4.8	1750	880	248.0	44.0	40.0	NA	1.680	9.610	0.510	0.600	NA	NA	NA	NA	NA	NA	NA	3838	2234	NA
10/04/81	6.50	4.8	1800	980	296.0	44.0	40.0	44.0	1.080	10.000	0.470	0.530	NA	NA	NA	NA	NA	NA	NA	7219	2033	NA
10/13/81	6.70	5.6	1850	1000	296.0	44.0	48.0	46.0	0.810	7.000	0.360	0.470	NA	NA	NA	NA	NA	NA	1.5	9596	2851	NA
10/17/81	6.80	7.2	1220	1000	272.0	42.0	52.0	48.0	0.650	5.320	0.300	0.360	NA	NA	NA	NA	NA	NA	NA	8725	2038	NA
11/06/81	6.85	6.4	2100	1300	340.0	62.0	68.0	58.0	0.710	5.100	0.250	0.270	NA	NA	NA	NA	NA	NA	2.5	3562	65	NA
04/11/82	6.80	4.8	750	360	90.2	13.6	14.0	16.0	0.590	3.930	0.210	0.300	NA	NA	NA	NA	NA	NA	NA	1832	NA	NA
04/15/82	6.50	4.5	750	370	87.6	16.4	14.0	14.0	0.700	5.380	0.310	0.330	NA	NA	NA	NA	NA	NA	NA	2074	NA	NA
04/19/82	6.70	6.3	800	430	126.0	20.0	16.0	18.0	0.620	6.210	0.340	0.350	NA	NA	NA	NA	NA	NA	NA	1754	NA	NA
04/23/82	6.60	6.3	1050	560	164.0	20.4	22.0	24.0	0.860	9.190	0.480	0.490	NA	NA	NA	NA	NA	NA	NA	1881	NA	NA
05/03/82	6.10	4.1	1300	750	181.0	42.0	30.0	30.0	1.660	11.355	0.620	0.620	NA	NA	NA	NA	NA	NA	NA	829	NA	NA
05/04/82	6.45	5.8	1250	690	164.0	36.0	22.0	28.0	0.900	8.715	0.460	0.460	NA	NA	NA	NA	NA	NA	NA	4281	NA	NA

Table A2.4. Drainage quality data from Duluth Complex test pile FL4, from 1978-1982 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
05/07/82	6.40	6.3	1290	720	152.0	40.0	22.0	26.0	0.810	8.520	0.460	0.420	NA	NA	NA	NA	NA	NA	NA	7730	NA	NA
05/11/82	6.75	9.9	1500	NA	196.0	50.0	30.0	32.0	0.660	8.150	0.430	0.400	NA	NA	NA	NA	NA	NA	NA	5129	NA	NA
05/14/82	6.60	9.9	1550	940	200.0	18.0	20.0	16.0	0.270	2.680	0.130	0.150	0.02	0.19	NA	NA	NA	NA	NA	5462	NA	NA
05/21/82	6.80	8.1	1850	1000	282.0	49.0	41.0	39.0	0.470	6.800	0.330	0.430	0.03	0.40	NA	NA	NA	NA	NA	1408	NA	NA
06/02/82	6.80	7.2	1980	790	288.0	52.6	48.8	42.6	0.950	8.310	0.400	0.560	0.06	0.53	NA	NA	NA	NA	NA	1594	NA	NA
06/18/82	6.40	9.9	1300	750	184.0	40.0	31.0	31.0	1.090	6.720	0.350	0.420	0.03	0.45	NA	NA	NA	NA	NA	984	NA	NA
07/02/82	NA	NA	NA	750	256.0	38.0	36.0	36.0	1.130	7.000	0.340	0.320	0.02	0.44	NA	NA	NA	NA	NA	25536	NA	NA
07/26/82	6.31	6.2	1520	785	232.0	36.0	40.0	40.0	1.150	8.860	0.460	0.330	0.10	0.47	NA	NA	NA	NA	NA	13832	NA	NA
08/30/82	6.28	4.4	1707	958	336.0	49.0	40.0	46.0	1.655	9.360	0.450	0.410	0.01	0.57	NA	NA	NA	NA	NA	10815	NA	NA

NA: Not analyzed.

'Dates represent intermediate dates for measurement periods.

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.25	NA	5000	NA	478.0	139.0	535.0	12.0	0.088	0.540	0.030	0.022	0.28	0.60	NA	NA	NA	NA	NA	NA	NA	4.0
05/25/78	7.25	NA	NA	NA	840.0	152.0	862.0	18.0	0.121	0.740	0.062	0.003	0.12	1.14	NA	NA	NA	NA	NA	NA	NA	NA
06/16/78	7.12	NA	8860	NA	1028.0	206.0	1106.0	22.0	0.258	0.665	0.059	0.033	0.12	1.64	NA	NA	NA	NA	NA	135	NA	15.0
07/05/78	7.40	NA	4610	2000	435.0	141.0	491.0	17.0	0.060	0.735	0.051	0.006	0.02	0.70	420.0	NA	NA	NA	NA	223	NA	12.0
07/13/78	7.49	NA	NA	1450	794.0	124.0	685.0	16.0	0.118	1.080	0.067	0.014	0.08	0.95	607.0	400.0	100.00	0.08	NA	NA	NA	NA
07/19/78	8.10	NA	3700	1300	540.0	93.0	508.0	12.0	0.092	0.860	0.052	0.006	0.05	0.75	312.0	160.0	100.00	1.90	NA	NA	NA	18.0
08/02/78	7.70	NA	2600	950	258.0	61.0	260.0	9.0	0.060	0.630	0.035	0.020	0.05	0.62	180.0	48.0	70.00	0.10	NA	NA	NA	23.0
08/04/78	7.75	NA	4150	1700	451.0	120.0	NA	12.0	0.120	0.930	0.140	0.030	0.04	1.04	320.0	210.0	100.00	0.10	NA	287	NA	22.0
08/10/78	7.55	47.0	6840	2400	748.0	178.0	700.0	22.0	0.170	1.430	0.280	0.050	0.15	1.68	600.0	NA	160.00	0.08	NA	77	NA	24.0
08/16/78	7.78	40.0	2225	1000	250.0	74.0	257.0	6.0	0.095	0.935	0.160	0.030	0.06	0.82	150.0	140.0	NA	0.08	NA	28390	NA	22.0
08/17/78	7.52	46.0	4600	1700	396.0	46.0	200.0	14.0	0.130	1.480	0.270	0.050	0.05	1.38	190.0	120.0	NA	0.08	NA	NA	NA	NA
08/23/78	7.55	30.0	4320	1600	368.0	112.0	396.0	12.0	0.130	1.540	0.240	0.050	0.05	1.40	280.0	170.0	NA	0.08	NA	NA	NA	NA
08/25/78	7.40	40.0	4750	1800	373.0	119.0	432.0	12.0	0.150	1.700	0.270	0.050	0.07	1.53	220.0	140.0	98.00	0.08	NA	NA	NA	23.0
08/28/78	7.85	57.0	5290	2150	481.0	135.0	681.0	15.0	0.180	2.150	0.390	0.070	0.07	1.82	390.0	230.0	NA	0.08	NA	NA	NA	NA
09/13/78	7.16	25.5	3400	2075	387.5	120.0	446.5	14.0	0.285	2.335	0.220	0.120	0.05	2.14	270.0	NA	NA	0.10	NA	667	NA	13.0
10/17/78	6.20	6.7	5320	NA	491.0	133.0	609.0	13.0	1.160	4.120	0.400	0.360	0.04	3.15	NA	NA	NA	NA	NA	NA	NA	NA
03/23/79	6.85	10.0	1562	600	200.0	54.0	97.2	4.0	0.200	4.590	0.360	0.150	0.02	1.74	10.0	NA	NA	NA	5.7	2290	46	NA
04/12/79	6.00	NA	2714	1400	327.0	93.2	201.0	6.2	0.740	10.250	0.710	0.290	0.03	3.40	95.0	NA	NA	NA	NA	1665	840	NA
04/16/79	6.60	NA	1358	560	152.6	74.2	91.6	3.4	0.120	2.320	0.210	0.080	1.02	1.17	45.0	NA	NA	NA	NA	3168	1308	NA
04/17/79	6.60	12.0	1146	440	124.6	31.6	76.2	3.0	0.120	2.350	0.210	0.090	0.02	1.18	28.0	NA	NA	NA	4.7	3641	3916	NA
04/18/79	6.80	16.0	1318	488	162.0	38.4	86.0	2.6	0.170	2.630	0.230	0.100	0.02	1.38	30.0	62.0	NA	NA	NA	6529	2997	NA
04/19/79	6.50	15.0	1054	210	141.0	34.0	72.0	2.0	0.130	2.140	0.200	0.190	0.03	1.12	29.0	50.0	11.00	NA	NA	3971	3862	NA
04/20/79	6.70	16.5	1498	559	185.0	56.1	95.0	3.0	0.130	3.145	0.290	0.100	0.02	1.58	51.0	67.5	13.50	NA	4.2	5257	3929	NA
04/25/79	6.65	14.0	2613	NA	292.4	103.4	205.6	4.6	0.280	5.250	0.420	0.200	0.03	2.74	110.0	120.0	24.00	NA	NA	5001	719	NA
05/02/79	6.40	4.0	3460	1650	394.4	149.0	270.0	7.0	0.510	9.000	0.590	0.300	0.05	4.07	171.0	190.0	23.00	NA	NA	3387	1902	NA
05/09/79	6.60	NA	3482	1030	312.0	122.0	238.0	4.0	0.490	7.000	0.480	0.270	0.04	3.23	15.0	NA	NA	NA	NA	1654	1286	NA
05/10/79	7.00	NA	2432	867	240.2	104.2	200.2	4.4	0.340	4.470	0.360	0.180	0.01	2.48	111.0	NA	NA	NA	NA	10250	4690	NA
05/20/79	NA	NA	NA	1750	341.0	130.0	292.0	5.0	0.470	7.440	0.470	0.290	0.03	3.23	152.0	NA	NA	NA	8.6	4644	1266	NA
05/22/79	7.20	NA	2201	883	211.0	86.0	170.0	3.0	0.330	3.900	0.290	0.170	0.02	2.02	94.0	NA	NA	NA	16.0	2188	1399	NA
05/23/79	7.00	NA	3377	1700	316.0	60.4	269.0	6.6	0.490	6.110	0.460	0.250	0.06	3.05	143.0	NA	NA	NA	7.9	2653	1007	NA
06/01/79	6.90	NA	2668	1050	231.2	56.6	212.6	5.2	0.435	4.160	0.350	0.215	0.05	2.28	111.5	NA	NA	NA	7.1	2567	729	NA
06/06/79	6.40	NA	2196	867	199.4	79.8	86.6	2.0	0.330	3.360	0.240	0.170	0.01	1.79	94.0	NA	NA	NA	NA	2570	1130	NA
06/10/79	6.90	14 0	1916	700	184.8	77.6	79.4	3.8	0.380	3.520	0.126	0.140	0.02	1.60	79.0	NA	NA	NA	NA	2574	1433	20.0
06/12/79	6.60	10.0	2673	1200	266.3	107.5	154.0	3.3	0.375	4.500	0.350	0.210	0.01	2.26	104.0	NA	NA	NA	NA	1506	494	NA
06/16/79	6.70	9.9	3450	1230	325.6	127.0	189.0	4.4	0.520	5.660	0.430	0.290	0.04	2.77	139.0	NA	NA	NA	NA	1612	442	23.0
06/21/79	6.70	11.0	2411	750	202.0	87.0	155.0	4.0	0.390	3.880	0.240	0.180	0.01	5.57	91.0	NA	NA	NA	NA	2233	1687	13.0
06/30/79	6.30	6.6	3074	1450	308.0	136.0	249.0	6.0	0.720	5.820	0.380	0.320	0.03	2.59	128.0	NA	NA	NA	8.9	1991	230	NA
07/10/79	6.20	4.8	4599	2200	467.0	193.5	409.5	9.5	1.755	9.960	0.620	0.665	0.03	4.10	220.0	NA	NA	NA	9.0	1628	98	19.0
07/30/79	5.50	3.0	2006	925	209.0	97.0	105.0	4.0	1.450	6.010	0.390	0.290	0.01	2.04	79.0	NA	NA	NA	6.3	5269	3374	22.0
08/04/79	5.40	4.0	3438	1670	444.0	169.0	242.0	9.0	2.720	10.150	0.660	0.580	0.04	3.72	118.0	NA	NA	NA	8.7	1957	284	23.0
08/13/79	5.00	2.4	4690	2420	540.0	126.2	438.0	13.8	4.810	15.000	1.130	1.010	0.07	6.13	181.0	NA	NA	NA	9.1	753	108	14.0
08/23/79	4.80	1.6	3739	1800	439.4	174.0	232.0	11.4	4.220	14.300	1.080	0.840	0.03	5.24	153.0	NA	NA	NA	8.3	1192	342	9.0
08/28/79	5.30	6.0	3154	1400	345.0	112.6	260.0	8.2	3.790	13.700	1.000	0.700	0.06	4.23	113.0	NA	NA	NA	6.6	848	709	20.0
09/01/79	5.00	2.0	2576	1200	327.2	112.4	136.0	6.8	3.380	11.300	0.820	0.510	0.05	3.35	86.0	NA	NA	NA	6.3	5553	2337	22.0
09/05/79	4.70	0.4	3819	1980	478.8	121.6	308.0	10.6	5.270	16.200	1.180	0.840	0.05	5.33	102.0	NA	NA	NA	7.8	988	611	18.0
09/11/79	4.90	2.0	3364	1650	398.8	160.8	229.0	9.8	4.730	14.000	1.040	0.720	0.06	4.70	107.0	NA	NA	NA	7.3	3842	846	17.0
09/13/79	4.90	2.8	3745	1800	444.0	175.0	238.0	10.2	5.020	15.400	1.130	0.760	0.08	5.00	29.0	NA	NA	NA	9.8	2112	699	19.0
09/19/79	4.70	2.4	5710	2680	692.0	272.0	478.0	16.2	6.800	23.600	1.700	1.310	0.16	8.10	201.0	NA	NA	NA	9.0	1567	176	13.5
10/18/79	4.60	0.8	5168	2600	646.0	254.0	450.0	14.6	5.960	26.600	1.910	1.340	0.30	8.45	174.0	NA	NA	NA	8.5	931	758	NA
10/19/79	4.70	2.4	2356	1050	251.0	106.0	142.0	5.6	4.400	15.500	1.090	0.630	0.04	3.70	64.0	NA	NA	NA	4.3	2161	1863	NA
10/31/79	4.60	0.8	3448	1650	408.0	177.0	198.0	7.8	7.200	24.000	2.090	1.020	0.11	5.78	99.0	NA	NA	NA	5.8	6423	2331	NA

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
11/01/79	4.50	NA	3702	1750	447.0	203.5	230.0	9.1	8.535	25.500	2.170	1.085	0.13	6.18	102.0	NA	NA	NA	6.7	4921	1170	NA
03/29/80	4.90	2.3	1050	440	94.2	35.9	48.5	2.3	1.880	7.360	0.445	0.220	0.05	1.23	18.0	NA	NA	NA	2.1	1363	390	13.0
04/01/80	5.00	3.4	800	360	74.0	26.4	27.6	1.4	1.180	4.630	0.300	0.150	0.02	1.00	12.0	NA	NA	NA	1.4	2536	1173	NA
04/06/80	5.50	4.6	1025	490	101.6	45.0	36.6	2.1	1.790	7.045	0.445	0.260	0.03	1.44	19.5	NA	NA	NA	NA	4543	1789	NA
04/14/80	4.50	NA	NA	875	226.8	98.2	92.8	4.0	5.290	18.000	1.290	0.750	0.04	4.11	44.0	NA	NA	NA	NA	3785	540	NA
04/18/80	4.60	1.2	NA	1100	248.0	106.0	106.0	4.4	5.670	21.300	1.490	0.820	0.13	4.71	61.0	NA	NA	NA	3.5	2044	568	NA
04/28/80	4.80	3.3	3590	1820	375.0	164.0	184.6	6.6	8.810	34.000	2.320	1.270	0.10	7.82	68.0	NA	NA	NA	NA	1817	200	11.0
05/10/80	4.70	2.2	3975	2020	420.4	186.0	210.0	8.0	9.250	38.000	2.440	1.360	0.10	8.78	75.0	NA	NA	NA	5.4	583	171	17.0
05/31/80	4.60	2.2	2400	1350	246.0	114.0	101.6	5.0	5.870	21.130	1.420	0.760	0.05	4.24	84.0	NA	NA	NA	1.3	3059	1946	NA
06/03/80	4.60	NA	3350	1750	381.0	177.0	149.0	6.6	7.620	27.700	1.890	1.010	0.05	5.99	110.0	NA	NA	NA	6.7	1018	207	NA
06/06/80	4.60	1.1	3100	1650	316.0	169.0	131.6	6.0	7.630	21.000	1.790	1.000	0.06	5.50	90.0	NA	NA	NA	NA	935	537	NA
06/13/80	4.60	1.1	3300	1650	317.0	156.0	142.8	6.6	7.940	27.270	1.810	0.950	0.10	5.41	86.0	NA	NA	NA	NA	689	464	NA
06/18/80	4.30	NA	3800	1800	359.2	184.0	171.0	7.4	7.710	27.700	1.840	0.970	0.13	5.80	110.0	NA	NA	NA	NA	348	49	NA
06/28/80	4.50	NA	3400	1650	311.0	153.0	148.0	6.4	7.350	25.000	2.100	1.040	0.10	5.55	83.0	NA	NA	NA	5.9	1321	476	NA
07/02/80	4.50	NA	2500	NA	274.7	134.7	91.5	6.0	9.500	24.500	2.000	1.020	0.07	4.57	NA	NA	NA	NA	NA	4334	1305	NA
07/07/80	4.00	NA	3800	1800	401.0	205.0	172.0	7.4	12.000	35.000	2.640	1.290	0.09	6.52	96.0	NA	NA	NA	10.0	1374	238	NA
07/17/80	4.30	NA	5000	2200	475.8	296.0	318.0	11.8	13.450	42.000	2.680	1.560	0.28	9.00	26.0	NA	NA	NA	NA	1253	458	NA
07/26/80	4.30	NA	4450	1880	421.2	219.0	244.0	9.8	11.180	32.000	2.250	1.280	0.11	7.18	126.0	NA	NA	NA	NA	1465	329	NA
08/03/80	4.20	NA	4800	1980	434.2	232.5	323.0	10.4	11.000	40.000	3.040	1.520	0.14	8.25	130.0	NA	NA	NA	6.8	958	528	NA
08/12/80	4.30	NA	3150	1550	303.0	161.0	152.0	7.4	13.000	33.000	2.460	1.190	0.14	6.17	81.0	NA	NA	NA	4.9	2271	900	NA
08/19/80	4.30	NA	3400	2150	375.6	211.6	166.6	9.0	20.000	44.000	3.510	1.740	0.17	7.90	93.0	NA	NA	NA	NA	2832	464	NA
08/22/80	NA	NA	3150	1900	280.5	244.0	126.5	3.5	19.000	38.000	2.670	1.530	0.19	6.88	82.0	NA	NA	NA	NA	1927	922	NA
08/29/80	4.30	NA	3625	2200	406.0	321.0	229.0	8.0	18.000	45.000	3.160	1.800	0.41	8.99	105.0	NA	NA	NA	8.5	1363	5237	NA
09/11/80	4.30	NA	2845	1700	316.0	231.0	172.0	7.0	16.000	35.000	2.640	1.520	0.15	6.98	78.0	NA	NA	NA	NA	950	589	13.0
09/12/80	4.40	NA	2383	1260	218.0	185.0	118.0	4.0	19.000	35.500	2.565	1.440	0.20	6.32	53.5	NA	NA	NA	4.9	5209	2795	21.0
09/18/80	4.40	NA	2625	1600	248.0	213.0	147.0	4.0	18.000	34.000	2.400	1.430	0.14	5.63	69.0	NA	NA	NA	NA	8108	2740	15.0
09/23/80	4.20	NA	3846	2120	400.0	309.0	226.0	31.0	20.900	54.600	3.150	1.990	0.24	8.82	114.0	NA	NA	NA	5.9	3592	798	14.0
10/22/80	4.10	NA	3160	2000	330.0	252.0	166.0	8.0	20.700	56.200	3.210	1.910	0.52	8.25	85.0	NA	NA	NA	4.8	2881	710	11.0
10/30/80	4.20	NA	2250	1700	314.0	158.0	84.0	5.0	17.000	40.000	2.470	1.380	0.12	5.79	64.0	NA	NA	NA	NA	1170	85	10.0
02/19/81	3.70	NA	2050	950	172.9	110.5	123.5	NA	13.300	50.000	2.540	1.270	NA	NA	36.0	NA	NA	NA	NA	1472	1423	5.0
02/21/81	4.50	NA	2550	1560	207.4	201.5	158.0	NA	20.600	82.500	4.190	2.320	NA	NA	NA	NA	NA	NA	NA	1484	325	24.0
03/28/81	4.70	0.8	1300	625	94.9	62.4	48.1	NA	14.400	41.000	2.110	1.100	NA	NA	NA	NA	NA	NA	2.3	8842	3797	NA
04/03/81	4.70	1.6	1900	1100	146.9	110.5	84.5	NA	20.300	64.000	3.300	1.830	NA	NA	NA	NA	NA	NA	NA	7692	2030	NA
04/13/81	4.50	NA	2500	1200	191.0	147.0	69.0	NA	20.200	60.000	3.480	1.870	NA	NA	NA	NA	NA	NA	NA	3376	1436	NA
04/17/81	4.70	1.6	2700	1380	148.0	156.0	58.0	NA	18.600	56.000	3.280	1.940	NA	NA	NA	NA	NA	NA	4.4	2858	1439	NA
04/23/81	4.70	1.6	3000	1415	219.0	161.0	97.0	NA	18.400	57.000	3.380	1.980	NA	NA	NA	NA	NA	NA	NA	3978	3580	NA
04/25/81	4.80	2.4	3100	1480	200.0	168.0	95.0	NA	18.400	57.000	3.420	2.090	NA	NA	NA	NA	NA	NA	NA	8260	2934	NA
04/28/81	4.90	4.0	2950	1350	205.0	179.0	99.0	NA	16.300	51.000	3.050	1.860	NA	NA	NA	NA	NA	NA	3.2	5534	2409	NA
04/29/81	4.70	NA	3515	1850	295.0	280.0	134.0	NA	17.700	43.000	3.600	2.170	NA	NA	NA	NA	NA	NA	NA	5307	1392	12.0
05/12/81	4.50	NA	3708	2250	358.0	370.0	184.0	NA	19.940	73.000	4.600	2.670	NA	NA	NA	NA	NA	NA	5.3	3441	272	19.0
06/01/81	4.50	NA	2660	1350	244.0	180.0	110.0	NA	16.620	38.000	2.870	1.790	NA	NA	NA	NA	NA	NA	NA	2960	1196	15.0
06/06/81	4.45	NA	1872	1160	188.0	112.0	52.0	NA	13.340	26.000	2.270	1.330	NA	NA	NA	NA	NA	NA	3.1	3274	289	18.0
06/22/81	4.20	NA	2650	1660	271.0	175.0	93.0	NA	17.920	43.000	3.360	1.730	NA	NA	NA	NA	NA	NA	NA	4766	2996	17.0
06/24/81	4.40	NA	2342	1620	246.0	154.0	70.0	NA	22.240	68.000	3.680	1.950	NA	NA	NA	NA	NA	NA	NA	4993	2335	16.0
06/28/81	4.50	NA	1657	1910	302.0	290.0	112.0	NA	20.570	42.000	3.550	1.760	NA	NA	NA	NA	NA	NA	3.4	6878	3559	13.0
07/04/81	4.40	NA	3568	2300	370.0	183.3	166.2	NA	29.500	68.000	3.820	2.530	NA	NA	NA	NA	NA	NA	NA	6197	295	22.0
07/15/81	4.25	NA	3526	2250	362.0	278.6	175.0	NA	29.000	72.000	4.100	2.660	NA	NA	NA	NA	NA	NA	NA	1628	102	19.0
07/24/81	4.40	NA	2830	1830	290.0	212.6	108.0	NA	26.500	56.000	3.310	2.050	NA	NA	49.0	NA	NA	NA	3.5	1938	437	19.0
08/04/81	4.00	NA	2951	2030	326.0	142.0	108.0	NA	27.500	71.500	3.920	1.980	NA	NA	NA	NA	NA	NA	NA	1783	106	18.0
08/17/81	4.05	NA	3238	2150	348.0	260.0	108.0	8.0	27.500	65.000	3.670	2.230	NA	NA	NA	NA	NA	NA	NA	1033	47	22.0
08/31/81	4.10	NA	3930	2560	360.0	270.0	128.0	NA	34.000	95.000	5.170	3.020	NA	NA	NA	NA	NA	NA	5.1	1136	1000	NA

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/07/81	4.10	NA	4200	2220	342.0	244.0	104.0	NA	34.000	84.000	4.420	2.660	NA	NA	NA	NA	NA	NA	NA	1109	588	NA
09/16/81	4.10	NA	4300	2640	378.0	284.0	114.0	NA	33.000	92.000	4.890	2.920	NA	NA	NA	NA	NA	NA	NA	431	20	NA
09/26/81	4.10	NA	3550	2220	328.0	240.0	74.0	NA	38.000	82.000	4.360	2.570	NA	NA	NA	NA	NA	NA	3.7	1949	1176	NA
10/01/81	4.20	NA	3225	1840	298.0	224.0	56.0	NA	42.000	80.000	4.460	2.280	NA	NA	NA	NA	NA	NA	NA	4675	2859	NA
10/04/81	4.20	NA	3700	2600	332.0	354.0	76.0	10.0	52.000	107.000	8.000	3.760	NA	NA	NA	NA	NA	NA	NA	3607	1859	NA
10/14/81	4.20	NA	3400	1960	276.0	222.0	90.0	8.0	39.000	78.000	5.000	3.080	NA	NA	NA	NA	NA	NA	3.5	5216	1771	NA
10/15/81	4.20	NA	4000	2760	252.0	304.0	100.0	22.0	123.000	306.000	21.000	10.000	NA	NA	NA	NA	NA	NA	NA	9906	2503	NA
11/01/81	4.10	NA	4000	2760	324.0	350.0	118.0	8.0	54.000	120.000	6.700	3.360	NA	NA	NA	NA	NA	NA	4.1	3516	193	NA
03/30/82	4.20	NA	1825	1000	138.0	102.0	34.0	4.0	19.000	49.000	2.800	1.500	NA	NA	NA	NA	NA	NA	NA	2336	NA	NA
04/11/82	4.30	NA	2350	1500	182.0	158.0	48.0	4.0	29.000	75.000	4.400	2.340	NA	NA	NA	NA	NA	NA	NA	1669	NA	NA
04/13/82	4.40	NA	1050	570	78.8	65.7	16.0	2.0	10.000	24.000	1.400	0.700	NA	NA	NA	NA	NA	NA	NA	4906	NA	NA
04/15/82	4.40	NA	1500	900	117.6	96.4	24.0	2.0	13.000	37.000	2.300	1.050	NA	NA	NA	NA	NA	NA	NA	4179	NA	NA
04/19/82	4.30	NA	1775	1100	176.0	144.0	34.0	4.0	22.000	52.000	3.000	1.490	NA	NA	NA	NA	NA	NA	NA	2245	NA	NA
04/23/82	4.20	NA	2425	1600	206.0	176.0	58.0	4.0	34.000	83.000	4.700	2.430	NA	NA	NA	NA	NA	NA	NA	2392	NA	NA
05/03/82	4.15	NA	3100	2300	204.0	190.0	78.0	6.0	55.000	128.000	8.100	4.200	NA	NA	NA	NA	NA	NA	NA	572	NA	NA
05/04/82	4.20	NA	2900	2000	192.0	208.0	62.0	6.0	46.000	110.000	6.100	3.380	NA	NA	NA	NA	NA	NA	NA	4902	NA	NA
05/10/82	4.30	NA	2875	1900	184.0	208.0	56.0	6.0	46.000	104.000	5.800	3.220	NA	NA	NA	NA	NA	NA	NA	4664	NA	NA
05/11/82	4.30	NA	3100	1700	202.0	223.0	84.0	6.0	49.500	123.500	6.900	3.890	NA	NA	NA	NA	NA	NA	NA	5337	NA	NA
05/14/82	4.30	NA	3400	2200	250.0	288.0	84.0	6.8	42.000	114.000	5.200	3.150	0.06	7.82	NA	NA	NA	NA	NA	5773	NA	NA
05/21/82	4.20	NA	5000	3600	340.0	382.0	95.0	8.0	57.000	169.000	7.600	4.360	0.13	11.44	NA	NA	NA	NA	NA	2052	NA	NA
06/02/82	4.10	NA	3700	2700	250.0	280.0	100.0	6.6	39.000	121.000	5.700	3.720	0.14	7.77	NA	NA	NA	NA	NA	1794	NA	NA
06/18/82	4.20	NA	3400	2400	264.0	264.0	79.0	6.8	40.000	106.000	5.100	3.300	0.11	6.89	NA	NA	NA	NA	NA	1185	NA	NA
07/02/82	NA	NA	NA	1965	332.0	208.0	78.0	8.0	47.000	111.000	5.300	2.560	0.10	7.44	NA	NA	NA	NA	NA	31423	NA	NA
07/26/82	3.94	NA	3531	2150	318.0	258.0	84.0	8.0	42.000	163.000	7.400	3.150	1.28	9.10	NA	NA	NA	NA	NA	14021	NA	NA
08/30/82	3.97	NA	3319	1920	366.0	234.0	76.0	8.0	58.000	125.000	5.900	2.790	0.30	8.03	NA	NA	NA	NA	NA	16239	NA	NA
10/04/82	4.27	NA	3073	1900	250.0	260.0	70.0	8.0	53.000	168.000	7.500	3.300	0.21	9.17	NA	NA	NA	NA	NA	20358	NA	NA
11/01/82	3.90	NA	2980	NA	176.0	222.0	62.0	6.0	41.000	141.000	6.300	2.800	0.25	7.57	NA	NA	NA	NA	NA	2491	NA	NA
11/22/82	4.10	NA	4100	3300	236.0	326.0	136.0	NA	41.600	213.000	8.900	3.820	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
03/18/83	4.35	NA	1425	780	NA	NA	NA	NA	24.000	37.000	18.000	1.380	NA	NA	NA	NA	NA	NA	NA	3286	NA	NA
04/14/83	4.33	NA	1387	700	NA	NA	NA	NA	22.000	40.000	2.450	1.400	NA	NA	NA	NA	NA	NA	NA	12694	NA	NA
05/15/83	4.21	NA	2532	2500	NA	NA	NA	NA	56.000	1.100	6.400	3.490	NA	NA	NA	NA	NA	NA	NA	6279	NA	NA
06/15/83	4.25	NA	2173	1940	NA	NA	NA	NA	40.000	76.000	3.400	1.880	NA	NA	NA	NA	NA	NA	NA	5022	NA	NA
07/14/83	4.14	NA	3960	5160	NA	NA	NA	NA	87.000	174.000	10.190	5.740	NA	NA	NA	NA	NA	NA	NA	9466	NA	NA
08/14/83	4.08	NA	2861	2300	NA	NA	NA	NA	68.000	120.000	6.450	3.720	NA	NA	NA	NA	NA	NA	NA	14103	NA	NA
09/15/83	4.16	NA	3486	2950	NA	NA	NA	NA	85.000	164.000	9.000	5.060	NA	NA	NA	NA	NA	NA	NA	8668	NA	NA
10/13/83	4.32	NA	3405	2400	NA	NA	NA	NA	77.000	136.000	8.000	4.340	NA	NA	NA	NA	NA	NA	NA	19970	NA	NA
11/09/83	4.10	NA	2750	2250	NA	NA	NA	NA	82.000	133.000	7.400	4.170	NA	NA	NA	NA	NA	NA	NA	4269	NA	NA
11/30/83	4.00	NA	2700	2820	NA	NA	NA	NA	73.400	125.000	6.950	3.970	NA	NA	NA	NA	NA	NA	NA	628	NA	NA
02/24/84	4.00	NA	2175	2300	276.8	414.3	56.6	NA	49.000	106.000	5.110	3.420	NA	NA	NA	NA	NA	NA	NA	75	NA	NA
03/15/84	4.25	NA	1100	830	110.4	105.7	17.3	NA	26.000	41.400	2.050	1.250	NA	NA	NA	NA	NA	NA	NA	4023	NA	NA
04/15/84	4.33	NA	1472	1540	188.2	170.3	27.2	NA	46.000	67.000	3.750	2.200	NA	NA	NA	NA	NA	NA	NA	20316	NA	NA
05/16/84	4.22	NA	1958	2500	236.3	279.5	42.0	NA	65.000	110.000	7.000	3.360	NA	NA	NA	NA	NA	NA	NA	11761	NA	NA
06/13/84	4.16	NA	2320	2300	256.1	328.6	52.1	NA	68.000	119.600	6.400	3.540	NA	NA	NA	NA	NA	NA	NA	23280	NA	NA
07/14/84	4.19	NA	2930	2700	NA	NA	NA	NA	95.600	186.000	6.600	4.000	NA	NA	NA	NA	NA	NA	NA	8771	NA	NA
08/15/84	4.10	NA	3385	3020	379.7	277.6	71.8	NA	90.100	160.420	11.600	5.080	NA	NA	NA	NA	NA	NA	NA	7737	NA	NA
09/12/84	3.96	NA	4095	3900	408.2	510.0	88.2	NA	107.500	225.000	15.700	6.630	NA	NA	NA	NA	NA	NA	NA	2612	NA	NA
10/13/84	3.94	NA	2720	3459	402.6	430.0	65.3	NA	109.500	169.000	12.700	5.240	NA	NA	NA	NA	NA	NA	NA	15261	NA	NA
11/14/84	4.11	NA	3160	4125	430.0	585.0	93.8	NA	135.000	255.000	12.400	6.800	NA	NA	NA	NA	NA	NA	NA	2615	NA	NA
12/01/84	3.95	NA	2150	3000	332.0	426.0	61.0	NA	90.800	152.000	7.900	4.160	NA	NA	NA	NA	NA	NA	NA	492	NA	NA
12/18/84	4.30	NA	2025	2200	252.0	302.0	52.0	NA	70.300	108.800	5.410	3.060	NA	NA	NA	NA	NA	NA	NA	2215	NA	NA
04/02/85	4.50	NA	1625	1130	158.0	138.8	21.0	NA	32.000	63.000	2.840	1.820	NA	NA	NA	NA	NA	NA	NA	2551	NA	NA

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/16/85	4.40	NA	2200	1500	238.0	256.0	37.6	NA	63.000	98.000	4.820	2.870	NA	NA	NA	NA	NA	NA	NA	16537	NA	NA
05/14/85	4.34	NA	3100	2750	240.0	364.0	48.5	NA	83.000	128.000	7.200	3.800	NA	NA	NA	NA	NA	NA	NA	9852	NA	NA
06/14/85	4.29	NA	3500	3200	281.0	453.0	63.0	NA	91.000	160.000	7.800	4.460	NA	NA	NA	NA	NA	NA	NA	34814	NA	NA
07/15/85	4.16	NA	3100	2773	276.0	377.0	55.0	NA	78.300	130.600	6.500	4.690	NA	NA	NA	NA	NA	NA	NA	14584	NA	NA
08/13/85	4.11	NA	2700	2400	268.0	252.0	46.0	NA	76.400	94.000	4.880	2.870	NA	NA	NA	NA	NA	NA	NA	15114	NA	NA
09/13/85	4.15	NA	3100	2550	309.0	270.0	48.0	NA	75.000	104.200	5.440	3.330	NA	NA	NA	NA	NA	NA	NA	31741	NA	NA
10/15/85	4.20	NA	3300	2430	244.0	319.0	43.1	NA	77.100	106.500	5.470	2.510	NA	NA	NA	NA	NA	NA	NA	20420	NA	NA
11/15/85	4.27	NA	3250	1750	284.0	440.0	56.0	NA	88.800	144.600	7.340	3.800	NA	NA	NA	NA	NA	NA	NA	2646	NA	NA
03/23/86	4.50	NA	1500	900	114.0	122.0	15.8	NA	34.000	55.000	2.910	1.580	NA	NA	NA	NA	NA	NA	NA	17289	NA	NA
04/16/86	4.36	NA	2025	1400	204.0	190.0	26.2	NA	59.000	86.000	4.840	2.280	NA	NA	NA	NA	NA	NA	NA	16332	NA	NA
05/14/86	4.35	NA	2325	1950	276.0	262.0	28.0	NA	72.000	133.000	8.000	2.920	NA	NA	NA	NA	NA	NA	NA	12740	NA	NA
06/11/86	4.30	NA	2750	2000	236.0	250.0	38.0	NA	63.000	100.000	4.800	3.100	NA	NA	NA	NA	NA	NA	NA	7631	NA	NA
07/12/86	4.27	NA	2650	1750	260.0	264.0	36.0	NA	65.000	100.000	4.420	2.850	NA	NA	NA	NA	NA	NA	NA	2365	NA	NA
08/13/86	4.13	NA	3200	2350	300.0	301.0	45.0	NA	80.000	99.000	6.200	2.550	NA	NA	NA	NA	NA	NA	NA	13346	NA	NA
09/14/86	4.12	NA	3000	1800	280.0	302.0	38.0	NA	76.000	94.000	6.600	2.400	NA	NA	NA	NA	NA	NA	NA	27335	NA	NA
10/16/86	4.06	NA	3425	2400	288.0	318.0	44.0	NA	93.000	122.000	6.000	3.510	NA	NA	NA	NA	NA	NA	NA	6442	NA	NA
11/16/86	3.99	NA	3475	3250	332.0	544.0	64.0	NA	156.000	169.000	8.800	5.300	NA	NA	NA	NA	NA	NA	NA	10954	NA	NA
03/28/87	4.40	NA	2300	1750	202.0	234.0	30.0	NA	83.000	110.000	5.390	3.140	NA	NA	NA	NA	NA	NA	NA	12608	NA	NA
04/18/87	4.31	NA	2150	1650	198.0	206.0	26.0	NA	68.000	95.000	4.520	2.580	NA	NA	NA	NA	NA	NA	NA	2525	NA	NA
05/16/87	3.86	NA	2750	2700	335.0	397.0	44.0	NA	97.000	115.000	5.600	3.400	NA	NA	NA	NA	NA	NA	NA	28020	NA	NA
06/16/87	4.17	NA	3950	3000	348.0	458.0	58.0	NA	106.000	163.000	5.800	3.420	NA	NA	NA	NA	NA	NA	NA	1419	NA	NA
07/15/87	4.00	NA	3325	2715	294.0	340.0	46.0	NA	92.000	117.000	5.200	3.430	NA	NA	NA	NA	NA	NA	NA	29811	NA	NA
08/15/87	3.72	NA	3050	2500	288.0	304.0	42.0	NA	93.000	112.000	5.000	3.130	NA	NA	NA	NA	NA	NA	NA	13350	NA	NA
09/16/87	3.84	NA	4400	2850	323.0	352.0	47.0	NA	101.000	113.000	5.090	3.050	NA	NA	NA	NA	NA	NA	NA	16885	NA	NA
10/13/87	3.92	NA	5000	4300	386.0	560.0	72.0	NA	138.000	173.000	7.560	4.550	NA	NA	NA	NA	NA	NA	NA	1915	NA	NA
11/10/87	4.03	NA	4375	3760	342.0	500.0	64.0	NA	128.000	165.000	7.330	4.510	NA	NA	NA	NA	NA	NA	NA	1601	NA	NA
03/28/88	4.25	NA	1500	880	142.0	106.0	14.0	NA	34.000	52.000	2.500	2.100	NA	NA	NA	NA	NA	NA	NA	7574	NA	NA
04/17/88	4.20	NA	2125	1860	170.0	186.0	18.0	NA	52.000	81.000	3.700	3.100	NA	NA	NA	NA	NA	NA	NA	6635	NA	NA
05/12/88	4.11	NA	3275	2300	298.0	322.0	50.0	NA	113.000	153.000	6.800	6.500	NA	NA	NA	NA	NA	NA	NA	12044	NA	NA
06/12/88	4.10	NA	3325	2600	346.0	342.0	40.0	NA	118.000	146.000	7.500	7.100	NA	NA	NA	NA	NA	NA	NA	2858	NA	NA
07/15/88	4.10	NA	3250	2440	312.0	314.0	36.0	NA	105.000	132.000	6.500	6.500	NA	NA	NA	NA	NA	NA	NA	4553	NA	NA
08/16/88	4.06	NA	3475	2400	302.0	352.0	52.0	NA	85.000	133.000	6.300	5.100	NA	NA	NA	NA	NA	NA	NA	60076	NA	NA
09/16/88	4.04	NA	3050	2000	246.0	290.0	44.0	NA	76.000	80.000	5.500	4.400	NA	NA	NA	NA	NA	NA	NA	12025	NA	NA
10/18/88	4.12	NA	2950	2160	212.0	274.0	30.0	NA	75.000	88.000	5.100	4.700	NA	NA	NA	NA	NA	NA	NA	3664	NA	NA
11/15/88	4.30	NA	2625	2080	210.0	265.0	26.0	NA	71.000	81.500	4.550	4.400	NA	NA	NA	NA	NA	NA	NA	7017	NA	NA
11/30/88	4.20	NA	3600	2500	232.0	320.0	32.0	NA	78.000	98.000	5.100	4.800	NA	NA	NA	NA	NA	NA	NA	681	NA	NA
04/07/89	4.65	1.1	900	1400	80.0	72.0	10.0	NA	18.600	23.000	1.400	1.100	NA	NA	NA	NA	NA	NA	NA	18641	NA	NA
04/24/89	4.40	NA	1975	1340	150.0	156.0	18.0	NA	45.000	57.000	3.400	3.000	NA	NA	NA	NA	NA	NA	NA	8804	NA	NA
05/16/89	4.27	NA	2550	1700	202.0	216.0	28.0	NA	66.000	79.000	4.700	4.300	NA	NA	NA	NA	NA	NA	NA	11620	NA	NA
06/17/89	4.28	NA	3500	2950	304.0	354.0	54.0	NA	91.000	159.000	10.200	6.300	NA	NA	NA	NA	NA	NA	NA	35500	NA	NA
07/17/89	4.13	NA	4625	3800	440.0	460.0	58.0	NA	131.000	204.000	11.300	9.700	NA	NA	NA	NA	NA	NA	NA	2952	NA	NA
08/14/89	4.17	NA	2950	2060	232.0	282.0	34.0	NA	82.000	138.000	5.700	5.500	NA	NA	NA	NA	NA	NA	NA	4466	NA	NA
09/14/89	4.01	NA	4325	3500	356.0	416.0	50.0	NA	123.000	176.000	9.900	8.200	NA	NA	NA	NA	NA	NA	NA	17161	NA	NA
10/16/89	4.05	NA	4800	3600	360.0	462.0	56.0	NA	137.000	149.000	16.400	9.400	NA	NA	NA	NA	NA	NA	NA	1832	NA	NA
11/10/89	4.20	NA	3050	2000	214.0	300.0	36.0	NA	94.000	96.000	15.300	7.000	NA	NA	NA	NA	NA	NA	NA	999	NA	NA
03/26/90	4.15	NA	5000	5440	468.0	744.0	93.0	NA	202.000	253.000	16.200	14.100	NA	NA	NA	NA	NA	NA	NA	34	NA	NA
04/12/90	4.39	NA	3125	2280	244.0	288.0	26.0	NA	96.000	108.000	5.610	5.440	NA	NA	NA	NA	NA	NA	NA	3883	NA	NA
05/17/90	4.32	NA	2675	2040	226.0	230.0	22.0	NA	84.000	78.000	4.630	4.260	NA	NA	NA	NA	NA	NA	NA	927	NA	NA
06/18/90	4.07	NA	3550	2440	266.0	286.0	28.0	NA	83.000	74.000	4.200	4.380	NA	NA	NA	NA	NA	NA	NA	2998	NA	NA
07/16/90	4.10	NA	3490	2460	270.0	286.0	26.0	NA	93.000	73.000	4.410	4.290	NA	NA	NA	NA	NA	NA	NA	1658	NA	NA
08/13/90	4.05	NA	3100	1970	190.0	286.0	28.0	NA	75.000	78.000	4.900	4.130	NA	NA	NA	NA	NA	NA	NA	700	NA	NA

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/10/90	4.10	NA	3300	2160	242.0	312.0	24.0	NA	63.000	71.000	4.120	3.150	NA	NA	NA	NA	NA	NA	NA	5583	NA	NA
10/15/90	3.98	NA	4295	3980	334.0	438.0	36.0	NA	88.000	112.000	6.410	4.330	NA	NA	NA	NA	NA	NA	NA	20246	NA	NA
11/16/90	4.35	NA	5000	4400	410.0	640.0	58.0	NA	144.000	165.000	9.990	9.180	NA	NA	NA	NA	NA	NA	NA	167	NA	NA
04/08/91	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	10795	NA	NA
04/22/91	4.22	NA	3625	2620	238.0	350.0	30.0	NA	106.000	116.000	5.200	3.300	NA	NA	NA	NA	NA	NA	NA	7960	NA	NA
05/20/91	4.18	NA	4675	3260	284.0	452.0	40.0	NA	133.000	163.000	6.900	4.400	NA	NA	NA	NA	NA	NA	NA	9122	NA	NA
06/17/91	4.18	NA	3350	2080	208.0	317.0	28.0	NA	91.000	103.000	5.000	2.800	NA	NA	NA	NA	NA	NA	NA	10000	NA	NA
07/15/91	4.07	NA	4800	2500	246.0	378.0	34.0	NA	107.000	117.000	5.900	4.600	NA	NA	NA	NA	NA	NA	NA	29610	NA	NA
08/12/91	4.10	NA	5000	3280	286.0	430.0	56.0	NA	137.000	150.000	7.600	6.300	NA	NA	NA	NA	NA	NA	NA	1185	NA	NA
09/09/91	4.09	NA	5000	3260	292.0	404.0	48.0	NA	119.000	130.000	6.600	4.900	NA	NA	NA	NA	NA	NA	NA	20079	NA	NA
10/07/91	4.17	NA	4450	2640	236.0	346.0	42.0	NA	105.000	114.000	5.600	4.600	NA	NA	NA	NA	NA	NA	NA	6344	NA	NA
11/07/91	4.11	NA	4775	2360	212.0	304.0	34.0	NA	96.000	98.000	5.000	4.000	NA	NA	NA	NA	NA	NA	NA	11692	NA	NA

NA: Not analyzed.
'Dates represent intermediate dates for measurement periods.

Table A2.5.1. Drainage quality data from Duluth Complex test pile FL6, 1992-1993 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH	Alk. mg/L	Lab Acy. mg/L	Calc. Acy. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/23/92	4.29	NA	140	122.5	1475	650	68.0	92.0	11.60	3.20	4	28.00	28	1.40	0.90	0.06	2.53	NA	NA	NA	NA	NA	5901	NA	NA
05/11/92	4.21	NA	325	315.4	3190	1760	186.0	234.0	28.40	5.30	10	74.00	71	3.60	2.40	0.11	6.77	NA	NA	NA	NA	NA	2529	NA	NA
05/26/92	4.32	NA	265	261.6	2450	1640	140.0	224.0	25.80	5.90	8	58.00	62	3.00	2.20	0.10	6.35	NA	NA	NA	NA	NA	3203	NA	NA
06/08/92	4.30	NA	785	644.7	5000	4060	300.0	600.0	58.00	10.30	16	128.00	178	7.80	5.70	0.24	15.70	NA	NA	NA	NA	NA	822	NA	NA
06/22/92	4.32	NA	290	270.2	2710	1740	134.0	248.0	28.00	6.80	7	61.00	66	3.70	2.30	0.12	7.02	NA	NA	NA	NA	NA	872	NA	NA
07/07/92	4.25	NA	260	259.4	2510	790	146.0	208.0	25.60	6.50	7	65.00	58	3.20	2.10	0.11	5.76	NA	NA	NA	NA	NA	3764	NA	NA
07/20/92	4.18	NA	560	491.7	4480	696	238.0	418.0	47.00	10.10	11	107.00	130	6.70	4.30	0.14	12.30	NA	NA	NA	NA	NA	894	NA	NA
08/03/92	4.21	NA	280	292.5	2640	1780	146.0	258.0	31.00	7.40	8	68.00	69	3.90	2.00	0.11	7.46	NA	NA	NA	NA	NA	1159	NA	NA
08/17/92	4.23	NA	370	354.3	3400	2360	210.0	324.0	33.60	8.70	9	84.00	84	4.70	3.10	0.18	8.69	NA	NA	NA	NA	NA	1565	NA	NA
08/31/92	4.24	NA	250	242.1	2500	1460	146.0	210.0	25.00	7.30	6	58.00	58	3.00	2.00	0.11	5.62	NA	NA	NA	NA	NA	1482	NA	NA
09/14/92	4.20	NA	415	340.0	3200	2180	202.0	312.0	34.00	8.90	8	86.00	78	4.50	3.00	0.11	8.13	NA	NA	NA	NA	NA	5218	NA	NA
09/28/92	4.21	NA	485	448.5	4850	3120	254.0	416.0	46.60	10.80	10	107.00	110	6.00	4.00	0.11	11.30	NA	NA	NA	NA	NA	2694	NA	NA
10/12/92	4.23	NA	525	536.1	4900	3280	272.0	460.0	51.60	11.70	11	127.00	137	6.70	4.50	0.14	12.70	NA	NA	NA	NA	NA	444	NA	NA
10/27/92	4.18	NA	425	395.4	3830	2440	204.0	366.0	41.00	10.10	9	96.00	95	5.30	3.30	0.10	9.98	NA	NA	NA	NA	NA	379	NA	NA
11/09/92	4.25	NA	NA	NA	4500	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	119	NA	NA
11/30/92	4.30	NA	300	244.5	2300	1570	142.0	230.0	24.40	7.20	6	60.00	57	3.10	2.00	0.07	6.12	NA	NA	NA	NA	NA	805	NA	NA
04/08/93	4.40	NA	NA	NA	1350	770	72.5	110.0	10.00	2.75	6	33.00	34	1.48	1.08	0.07	3.08	NA	NA	NA	NA	NA	7949	NA	NA
04/22/93	4.40	NA	NA	NA	1440	825	77.5	115.0	10.00	3.00	5	27.00	30	1.66	0.88	0.05	2.68	NA	NA	NA	NA	NA	5257	NA	NA
05/06/93	4.23	NA	NA	NA	2750	1950	197.5	247.5	21.25	5.00	13	71.00	71	3.85	1.99	0.14	5.80	NA	NA	NA	NA	NA	12820	NA	NA
05/20/93	4.31	NA	NA	NA	3200	2340	217.5	310.0	26.25	4.75	14	96.00	90	4.60	2.86	0.21	7.28	NA	NA	NA	NA	NA	2086	NA	NA
06/03/93	4.37	NA	NA	NA	1600	1255	120.0	172.5	16.00	3.50	7	54.00	51	2.61	1.52	0.10	4.01	NA	NA	NA	NA	NA	6945	NA	NA
06/17/93	4.48	NA	NA	NA	2350	1670	167.5	270.0	22.00	4.75	9	87.00	79	4.15	2.52	0.11	6.32	NA	NA	NA	NA	NA	7559	NA	NA
07/01/93	4.13	NA	NA	NA	4200	3180	270.0	490.0	39.00	7.75	18	139.00	147	7.40	4.52	0.33	13.60	NA	NA	NA	NA	NA	8928	NA	NA
07/14/93	4.22	NA	NA	NA	4220	3020	266.0	422.0	37.20	9.20	14	100.00	125	6.03	3.75	0.15	13.00	NA	NA	NA	NA	NA	15912	NA	NA
07/29/93	4.23	NA	NA	NA	1700	1130	118.0	138.0	15.60	5.40	7	49.00	44	2.20	1.40	0.10	3.82	NA	NA	NA	NA	NA	13240	NA	NA
08/12/93	4.15	NA	NA	NA	2425	1680	166.0	188.0	19.80	6.00	8	66.00	62	2.68	1.99	0.07	5.30	NA	NA	NA	NA	NA	7184	NA	NA
08/26/93	4.13	NA	NA	NA	2600	1520	158.0	240.0	23.00	6.80	8	66.00	70	3.14	2.28	0.10	6.66	NA	NA	NA	NA	NA	11336	NA	NA
09/09/93	4.30	NA	NA	NA	2700	1670	174.0	232.0	22.20	7.00	8	66.00	68	3.45	2.68	0.07	6.82	NA	NA	NA	NA	NA	11866	NA	NA
09/23/93	4.21	NA	NA	NA	2700	1720	174.0	250.0	23.40	6.80	9	64.00	74	3.42	2.37	0.10	6.97	NA	NA	NA	NA	NA	3955	NA	NA
10/07/93	4.28	NA	NA	NA	3450	2180	164.4	286.0	27.00	7.40	12	80.00	86	4.00	2.50	0.09	7.42	NA	NA	NA	NA	NA	2139	NA	NA
10/20/93	4.22	NA	NA	NA	3925	2520	252.0	366.0	33.80	8.40	15	105.00	113	5.06	3.34	0.09	9.73	NA	NA	NA	NA	NA	814	NA	NA
11/04/93	4.25	NA	NA	NA	3300	970	191.6	278.0	26.80	7.00	10	80.00	77	4.09	2.47	0.07	7.09	NA	NA	NA	NA	NA	2139	NA	NA
11/22/93	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	288	NA	NA

NA: Not analyzed.
'Dates represent intermediate dates for measurement periods.

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.20	NA	1750	NA	190.0	11.0	153.0	6.0	0.020	0.445	0.045	NA	NA	0.27	NA	NA	NA	NA	NA	NA	NA	6.0
05/25/78	7.60	NA	2500	NA	279.0	21.0	273.0	10.0	0.021	0.780	0.136	0.096	0.02	0.89	NA	NA	NA	NA	NA	NA	NA	NA
06/16/78	7.50	NA	3300	NA	366.0	26.0	336.0	15.0	0.044	0.835	0.160	0.174	0.03	1.40	NA	NA	NA	NA	NA	192	NA	13.0
07/05/78	7.05	NA	2950	1800	325.0	29.0	231.0	13.0	0.140	5.100	0.195	0.257	0.02	1.29	78.0	NA	NA	NA	NA	370	NA	13.0
07/13/78	7.22	NA	NA	1300	357.0	27.0	248.0	14.0	0.090	6.700	0.810	0.230	0.04	1.67	50.0	39.0	13.00	0.08	NA	NA	NA	NA
07/19/78	7.40	NA	2050	1300	580.0	29.0	245.0	16.0	0.141	21.500	2.160	0.396	0.02	2.40	59.0	54.0	12.00	0.08	NA	NA	NA	NA
08/02/78	7.00	NA	1725	775	276.0	20.0	144.5	13.0	0.265	10.280	1.240	0.400	0.05	2.08	25.0	66.5	5.70	0.24	NA	NA	NA	21.0
08/04/78	6.30	NA	2350	1400	483.0	32.0	238.0	17.0	0.420	3.230	1.860	0.774	0.05	3.32	44.0	83.0	7.50	0.08	NA	333	NA	22.0
08/10/78	5.50	7.0	4000	2000	636.0	47.0	338.0	20.0	1.280	3.310	2.200	1.220	0.12	4.29	52.0	NA	8.50	0.08	NA	147	NA	22.0
08/16/78	5.90	8.0	1492	850	245.3	26.0	104.7	9.7	1.157	22.453	3.790	0.623	7.96	3.06	19.5	41.0	4.00	0.08	NA	5948	NA	17.7
08/17/78	5.25	5.0	2100	800	400.0	40.0	178.0	33.0	2.530	40.380	6.730	1.240	11.54	5.14	27.0	55.0	NA	0.08	NA	1714	NA	NA
08/18/78	4.85	4.0	3100	800	350.0	44.0	354.0	8.0	2.980	42.210	7.220	1.450	5.28	5.87	34.0	67.0	NA	0.08	NA	498	NA	16.0
08/23/78	4.23	NA	1775	1050	203.5	24.5	107.0	10.0	1.450	20.240	3.400	0.625	4.01	2.99	16.5	34.5	4.00	0.08	NA	NA	NA	NA
08/25/78	3.90	NA	2800	1200	344.0	42.0	177.0	15.0	3.150	39.400	6.360	1.180	2.78	5.40	22.0	54.0	8.30	0.08	NA	NA	NA	21.0
08/28/78	4.25	NA	3250	1500	395.0	50.0	220.0	18.0	3.950	48.000	7.760	1.490	2.38	6.47	30.0	64.0	NA	0.08	NA	NA	NA	NA
09/11/78	4.80	NA	1475	700	140.0	20.0	79.0	9.0	1.810	18.000	1.510	0.560	0.75	2.90	8.0	NA	NA	0.20	NA	8801	NA	17.0
09/12/78	3.50	NA	2650	1550	260.0	50.0	133.0	17.0	7.280	67.000	4.880	1.750	24.00	10.00	9.0	NA	NA	0.08	NA	NA	NA	NA
09/13/78	3.62	NA	3250	1900	312.0	65.0	183.0	21.0	10.300	83.000	6.170	2.460	23.00	13.00	15.0	46.0	0.10	NA	NA	721	NA	13.0
10/17/78	4.10	NA	3300	NA	335.0	98.0	211.0	15.0	11.450	79.000	5.730	3.290	0.63	10.00	NA	NA	NA	NA	NA	NA	NA	NA
03/23/79	3.70	NA	2445	2100	271.6	108.0	121.8	14.8	44.000	169.000	14.000	5.000	4.01	11.80	11.0	NA	NA	NA	7.6	2801	283	NA
04/03/79	3.60	NA	4435	3000	395.0	231.0	286.0	28.0	91.000	335.000	25.000	11.000	11.00	28.00	18.0	NA	NA	NA	NA	2574	167	NA
04/11/79	3.60	NA	4766	4600	430.0	280.0	338.0	30.0	107.000	406.000	31.000	13.000	10.00	35.00	22.0	NA	NA	NA	NA	1983	129	NA
04/16/79	3.50	NA	2822	2350	269.0	124.0	146.8	15.2	38.000	173.000	14.000	5.000	8.73	15.70	14.0	NA	NA	NA	NA	4148	3192	NA
04/17/79	3.30	NA	1769	1050	197.0	70.6	70.0	8.2	15.000	87.000	6.000	2.080	7.70	8.50	8.0	9.8	NA	NA	NA	5098	3235	NA
04/18/79	3.40	NA	1769	1100	213.0	62.4	68.0	7.2	13.000	74.000	5.190	1.900	13.33	7.22	5.0	8.1	NA	NA	NA	6906	4242	NA
04/19/79	3.40	NA	1475	833	193.0	42.2	52.0	6.4	8.980	48.000	3.690	1.360	11.62	4.89	2.0	5.5	0.40	NA	NA	1843	2946	NA
04/20/79	3.50	NA	1380	867	178.0	49.0	65.0	6.6	14.000	53.000	4.300	1.890	3.78	5.35	4.0	8.5	0.64	NA	3.3	7210	2394	NA
04/30/79	3.50	NA	2562	2100	294.8	100.0	185.6	12.2	40.000	107.400	8.400	3.950	4.54	10.72	13.0	26.0	1.10	NA	NA	6431	364	NA
05/11/79	3.50	NA	2341	NA	254.0	89.0	133.0	9.0	33.000	94.000	7.400	3.600	1.80	9.60	NA	NA	NA	NA	NA	409	2115	NA
05/22/79	3.80	NA	1767	1150	201.0	64.0	79.0	7.0	23.000	71.000	4.860	2.370	3.61	6.65	9.0	NA	NA	NA	6.9	1813	1031	NA
05/23/79	3.20	NA	2713	2100	288.2	105.8	162.8	14.0	41.600	116.000	8.600	4.160	4.03	10.09	12.0	NA	NA	NA	5.4	3157	982	NA
05/27/79	3.38	NA	2578	2200	285.4	57.7	151.2	13.1	35.900	104.000	7.800	3.680	4.64	9.42	12.0	NA	NA	NA	3.7	3073	508	NA
06/01/79	3.60	NA	2227	1800	239.2	54.6	121.4	11.6	30.800	84.600	5.570	3.000	3.00	7.75	10.0	NA	NA	NA	3.9	2290	561	NA
06/06/79	3.90	NA	2154	1430	243.8	77.2	60.8	9.2	26.000	80.000	5.180	2.830	3.17	7.49	12.0	NA	NA	NA	NA	2358	788	NA
06/10/79	3.40	NA	2538	1750	287.6	93.0	68.2	11.0	30.000	94.000	6.260	3.310	7.32	8.70	13.0	NA	NA	NA	NA	3407	1308	NA
06/16/79	3.60	NA	2890	2100	324.0	110.0	115.0	14.8	40.000	115.000	7.570	4.130	5.41	10.10	11.0	NA	NA	NA	NA	935	244	23.0
06/20/79	3.50	NA	1891	1150	203.0	68.0	71.0	11.0	14.000	59.000	4.320	2.220	3.06	0.11	7.0	NA	NA	NA	NA	1798	1300	19.0
06/30/79	3.30	NA	2754	2225	328.0	117.0	132.0	16.5	25.500	96.500	7.205	3.890	6.78	9.38	19.0	NA	NA	NA	4.8	1904	296	14.0
07/10/79	3.30	NA	3830	3200	462.0	180.0	204.0	29.0	50.000	163.000	14.000	5.950	4.83	14.10	28.0	NA	NA	NA	5.4	1348	110	19.0
07/30/79	3.50	NA	1926	1400	229.0	82.0	63.0	12.0	7.000	46.000	4.420	2.320	2.10	6.32	6.0	NA	NA	NA	6.5	3179	3202	22.0
08/04/79	3.20	NA	3170	2800	416.0	144.0	120.0	21.0	25.000	132.000	8.190	4.470	8.49	13.32	13.0	NA	NA	NA	5.6	1442	322	23.0
08/13/79	3.40	NA	4636	4650	514.8	130.4	322.0	32.6	76.000	259.000	19.000	9.590	11.10	22.80	82.0	NA	NA	NA	8.7	712	87	14.0
08/23/79	3.50	NA	3708	3600	463.2	230.2	202.0	27.2	44.000	202.000	12.000	7.460	6.90	17.89	9.0	NA	NA	NA	8.7	855	243	12.0
08/28/79	3.40	NA	3332	3050	365.0	122.2	158.6	21.8	26.000	142.000	12.000	5.800	5.24	14.20	15.0	NA	NA	NA	8.3	424	330	20.0
09/01/79	3.20	NA	2923	1400	296.4	115.8	98.2	18.0	31.000	138.000	8.000	4.560	19.60	13.34	17.0	NA	NA	NA	6.2	5708	2338	21.0
09/05/79	3.20	NA	4197	3300	420.2	128.6	192.8	28.4	77.000	227.000	18.000	8.490	26.20	20.20	39.0	NA	NA	NA	8.6	11	523	19.0
09/13/79	3.30	NA	4120	3600	363.8	239.6	197.0	26.2	77.000	224.000	17.000	8.140	22.85	19.20	10.0	NA	NA	NA	7.7	2449	811	17.0
09/13/79	3.30	NA	3906	3580	372.8	234.0	192.6	26.6	78.000	219.000	16.000	8.290	22.00	19.00	14.0	NA	NA	NA	8.3	2615	811	17.0
09/19/79	3.30	NA	5852	5700	522.0	384.0	286.0	39.2	131.000	348.000	24.000	10.130	25.50	28.00	51.0	NA	NA	NA	11.0	1726	220	12.0
10/18/79	3.40	NA	5446	6330	518.0	398.0	272.0	36.4	91.000	347.000	25.000	10.130	15.76	28.00	9.0	NA	NA	NA	10.0	840	360	NA
10/19/79	3.60	NA	2356	1700	185.6	138.0	76.0	12.8	18.000	115.000	10.000	4.390	4.16	10.09	3.0	NA	NA	NA	5.4	1730	1591	NA

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
10/31/79	3.30	NA	3937	3310	320.0	260.0	150.0	22.0	49.000	268.000	25.100	8.110	10.60	19.45	6.0	NA	NA	NA	7.0	4046	1743	NA
11/01/79	3.40	NA	3567	3180	294.0	244.0	132.0	22.6	63.000	268.000	24.600	8.000	11.77	18.60	8.0	NA	NA	NA	6.2	4345	1671	NA
03/29/80	3.70	NA	3500	2100	222.7	190.9	90.8	15.0	41.110	206.500	15.110	6.530	2.26	15.00	7.0	NA	NA	NA	4.9	1401	994	16.0
03/31/80	3.60	NA	2450	1700	168.2	125.6	57.4	10.0	34.080	144.000	8.940	3.870	7.29	11.00	3.0	NA	NA	NA	2.5	3369	1070	NA
04/05/80	4.60	NA	2288	1055	169.4	112.6	49.0	9.0	21.855	124.000	7.725	3.300	8.13	9.25	2.5	NA	NA	NA	NA	4543	1984	NA
04/14/80	3.50	NA	NA	1220	190.4	135.6	63.6	10.4	30.600	131.000	9.000	4.140	5.90	11.00	4.0	NA	NA	NA	NA	2271	497	NA
04/18/80	3.50	NA	NA	2550	231.8	184.0	86.6	12.6	46.000	187.000	14.000	5.930	7.91	17.00	4.0	NA	NA	NA	3.5	1590	497	NA
04/27/80	3.50	NA	4825	3880	342.4	314.0	166.6	20.4	97.000	321.000	21.000	14.000	10.77	28.00	7.0	NA	NA	NA	NA	530	50	13.0
05/10/80	3.40	NA	6110	6700	376.8	442.0	210.6	26.4	133.000	423.000	30.000	18.000	15.00	37.00	10.0	NA	NA	NA	7.9	451	109	12.0
05/18/80	3.23	NA	6000	6125	386.4	485.0	223.6	29.6	140.000	464.500	35.500	20.000	12.19	40.00	13.0	NA	NA	NA	NA	640	21	19.0
05/31/80	3.40	NA	3450	3040	244.4	233.0	112.8	18.2	54.000	245.000	15.100	8.400	6.94	19.00	15.0	NA	NA	NA	4.2	791	134	NA
06/03/80	3.45	NA	4750	3960	292.2	274.0	139.8	22.0	58.000	273.000	17.600	9.760	6.08	20.00	15.0	NA	NA	NA	7.3	303	134	NA
06/07/80	3.40	NA	4550	3920	279.6	259.0	125.2	20.8	54.000	261.000	16.500	9.200	6.45	19.00	16.0	NA	NA	NA	NA	382	128	NA
06/13/80	3.30	NA	5000	4070	332.0	283.0	152.2	23.4	63.000	289.000	19.100	10.250	7.50	20.00	12.0	NA	NA	NA	NA	299	140	NA
06/19/80	3.30	NA	5000	6200	341.0	345.0	181.0	26.2	80.000	360.000	23.200	12.050	8.33	26.00	12.0	NA	NA	NA	NA	155	18	NA
06/28/80	3.30	NA	4674	4250	341.0	355.0	147.0	23.0	59.000	269.000	24.000	12.000	9.00	24.00	10.0	NA	NA	NA	9.2	507	158	NA
06/30/80	3.70	NA	5000	4600	358.0	363.0	155.0	23.0	67.000	330.000	28.000	13.000	8.00	26.00	14.0	NA	NA	NA	17.0	329	64	NA
07/02/80	3.50	NA	2600	1850	200.0	148.0	57.0	12.0	23.000	129.000	12.000	4.660	5.73	10.00	9.0	NA	NA	NA	NA	409	257	NA
07/13/80	3.20	NA	6360	5750	417.0	444.0	220.0	31.0	84.000	415.000	28.000	15.000	9.90	36.00	15.0	NA	NA	NA	NA	284	34	NA
07/23/80	3.20	NA	5000	5700	435.0	506.0	232.0	30.0	80.000	411.600	30.000	14.000	9.90	37.00	16.0	NA	NA	NA	NA	295	20	NA
08/03/80	3.20	NA	5000	NA	483.0	587.0	274.0	34.0	90.000	582.000	36.000	19.000	13.00	47.00	NA	NA	NA	NA	9.1	34	111	NA
08/12/80	3.35	NA	4750	4180	348.0	347.0	134.0	24.0	36.000	279.000	19.000	9.000	7.75	25.00	12.0	NA	NA	NA	9.3	492	316	NA
08/17/80	3.30	NA	5000	5560	393.0	446.0	162.0	28.0	46.000	357.000	25.000	11.000	7.45	30.00	13.0	NA	NA	NA	NA	261	26	NA
08/22/80	NA	NA	4900	4180	279.0	512.0	133.0	17.0	31.000	266.000	21.000	11.000	6.50	22.00	13.0	NA	NA	NA	NA	231	112	NA
08/29/80	3.50	NA	2540	2400	172.0	240.0	65.0	12.0	22.000	141.000	10.420	5.300	5.22	12.00	19.0	NA	NA	NA	8.2	1319	700	NA
09/08/80	3.40	NA	4078	4100	231.0	503.0	133.0	21.0	44.000	262.000	21.000	10.000	8.83	22.00	15.0	NA	NA	NA	NA	522	134	NA
09/09/80	3.30	NA	4350	4700	283.0	569.0	154.0	22.0	54.000	296.000	22.000	12.000	7.30	25.00	13.0	NA	NA	NA	NA	102	600	14.0
09/12/80	3.40	NA	2694	2580	103.0	296.0	75.0	13.0	28.000	174.000	11.000	8.000	5.16	14.00	11.0	NA	NA	NA	NA	916	596	18.0
09/18/80	3.40	NA	3310	3050	158.0	373.0	94.0	15.0	56.000	217.000	15.000	9.000	5.45	17.00	16.0	NA	NA	NA	NA	3691	2740	14.0
09/25/80	3.30	NA	5588	5200	380.0	608.0	189.0	38.0	60.000	443.000	26.500	15.500	8.71	35.00	22.0	NA	NA	NA	11.0	1109	288	14.0
10/23/80	3.30	NA	5742	7700	390.0	660.0	202.0	32.0	59.300	472.000	28.800	16.900	8.68	38.00	19.0	NA	NA	NA	9.4	757	593	10.0
10/31/80	3.40	NA	5148	4700	309.0	491.0	187.0	26.0	75.000	353.000	28.000	14.000	6.00	33.00	16.0	NA	NA	NA	NA	647	91	11.0
02/19/81	3.60	NA	3692	3450	198.9	279.0	148.0	NA	49.300	270.000	18.100	8.850	NA	NA	17.0	NA	NA	NA	NA	908	1419	8.0
02/21/81	3.60	NA	2500	2040	130.0	144.2	87.1	NA	26.800	183.000	11.000	4.920	NA	NA	14.0	NA	NA	NA	NA	1344	549	NA
03/29/81	3.50	NA	2000	1420	94.9	110.5	53.3	NA	20.200	123.000	7.600	3.680	NA	NA	NA	NA	NA	NA	4.9	3702	773	NA
04/03/81	3.90	NA	1825	1250	85.8	91.0	46.8	NA	23.600	115.000	6.900	3.590	NA	NA	NA	NA	NA	NA	NA	4364	927	NA
04/13/81	3.70	NA	4000	2800	155.0	214.0	69.0	NA	71.000	239.000	13.800	7.900	NA	NA	NA	NA	NA	NA	NA	1033	405	NA
04/17/81	3.80	NA	4500	3080	194.0	293.0	100.0	NA	76.000	264.000	14.800	8.510	NA	NA	NA	NA	NA	NA	7.3	806	413	NA
04/23/81	3.70	NA	4350	3850	192.0	338.0	108.0	NA	107.000	331.000	17.300	10.700	NA	NA	NA	NA	NA	NA	NA	1453	1594	NA
04/25/81	3.60	NA	4575	3000	170.0	279.0	81.0	NA	76.000	251.000	14.700	8.050	NA	NA	NA	NA	NA	NA	NA	6057	1715	NA
04/28/81	3.60	NA	4200	3000	174.0	270.0	76.0	NA	70.000	233.000	14.200	7.730	NA	NA	NA	NA	NA	NA	7.0	4168	2146	NA
04/29/81	3.50	NA	5004	4300	296.0	440.0	114.0	NA	105.000	312.000	20.300	8.150	NA	NA	NA	NA	NA	NA	NA	3736	1098	9.0
05/12/81	3.35	NA	6386	5250	374.0	590.0	200.0	NA	159.000	501.000	26.300	9.780	NA	NA	NA	NA	NA	NA	12.0	2165	119	18.0
05/28/81	3.35	NA	5952	7500	344.0	580.0	182.0	NA	137.000	500.000	25.200	9.630	NA	NA	NA	NA	NA	NA	NA	893	2146	16.0
06/08/81	3.25	NA	5709	6000	312.0	540.0	158.0	NA	102.000	441.500	25.910	9.580	NA	NA	NA	NA	NA	NA	9.6	621	51	19.0
06/22/81	3.40	NA	4570	4300	246.0	460.0	114.0	NA	63.000	354.000	21.750	7.990	NA	NA	NA	NA	NA	NA	NA	1143	607	16.0
06/24/81	3.40	NA	3150	2370	174.0	200.0	56.0	NA	34.000	199.000	13.200	5.700	NA	NA	NA	NA	NA	NA	NA	1189	659	17.0
06/28/81	3.35	NA	3032	2550	174.0	210.0	58.0	NA	45.000	204.500	13.800	5.600	NA	NA	NA	NA	NA	NA	6.6	2036	1352	15.0
07/04/81	3.30	NA	4298	4050	272.0	366.0	106.0	NA	117.000	307.500	22.000	11.000	NA	NA	NA	NA	NA	NA	NA	4891	173	22.0
07/24/81	3.30	NA	7122	8000	392.0	524.6	242.0	NA	185.000	550.000	43.000	21.500	NA	NA	33.0	NA	NA	NA	7.9	386	70	21.0
08/04/81	3.10	NA	8085	9350	426.0	830.0	256.0	NA	129.500	676.000	50.000	25.500	NA	NA	NA	NA	NA	NA	NA	291	22	18.0

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
08/17/81	3.10	NA	8037	8400	414.0	870.0	260.0	34.0	104.000	630.000	43.000	20.000	NA	NA	NA	NA	NA	NA	NA	182	4	22.0
08/31/81	3.30	NA	8900	9000	440.0	980.0	250.0	NA	108.000	741.000	47.000	25.000	NA	NA	NA	NA	NA	NA	9.8	106	36	17.0
09/07/81	3.30	NA	7500	7900	440.0	820.0	210.0	NA	80.000	660.000	42.000	22.000	NA	NA	NA	NA	NA	NA	NA	121	58	NA
09/16/81	3.30	NA	5000	9600	480.0	1030.0	260.0	NA	92.000	762.000	49.000	26.000	NA	NA	NA	NA	NA	NA	NA	72	4	NA
09/26/81	3.50	NA	4965	5000	330.0	520.0	120.0	NA	39.000	381.000	24.000	12.000	NA	NA	NA	NA	NA	NA	8.0	329	210	NA
10/01/81	3.50	NA	4375	2600	206.0	290.0	86.0	NA	23.000	240.000	16.000	8.000	NA	NA	NA	NA	NA	NA	NA	693	983	NA
10/04/81	3.45	NA	3900	2600	154.0	238.0	64.0	14.0	32.000	200.000	16.000	6.000	NA	NA	NA	NA	NA	NA	NA	1136	706	NA
10/14/81	3.40	NA	3950	2960	176.0	236.0	68.0	14.0	53.000	198.000	15.000	8.000	NA	NA	NA	NA	NA	NA	5.6	4554	1622	NA
10/18/81	3.30	NA	5000	4000	326.0	262.0	102.0	10.0	58.000	119.000	7.000	4.130	NA	NA	NA	NA	NA	NA	NA	6337	1288	NA
11/06/81	3.20	NA	5000	5400	292.0	530.0	140.0	26.0	140.000	412.000	26.000	11.000	NA	NA	NA	NA	NA	NA	6.6	3585	105	NA
03/30/82	3.45	NA	4200	3100	174.0	350.0	68.0	12.0	55.000	233.000	18.000	8.050	NA	NA	NA	NA	NA	NA	NA	3218	NA	NA
04/06/82	3.50	NA	4200	3200	169.0	355.0	70.0	11.0	65.000	253.000	17.500	8.180	NA	NA	NA	NA	NA	NA	NA	2657	NA	NA
04/13/82	3.50	NA	2600	1900	131.0	140.0	36.0	8.0	36.000	138.000	8.900	4.100	NA	NA	NA	NA	NA	NA	NA	5443	NA	NA
04/15/82	3.55	NA	1950	1015	111.1	94.0	23.0	6.0	28.000	88.500	6.200	2.430	NA	NA	NA	NA	NA	NA	NA	5182	NA	NA
04/19/82	3.50	NA	2250	1500	142.0	152.0	38.0	10.0	39.000	126.000	8.100	3.750	NA	NA	NA	NA	NA	NA	NA	2014	NA	NA
04/23/82	3.45	NA	3050	2300	188.0	192.0	70.0	12.0	62.000	197.000	12.200	5.470	NA	NA	NA	NA	NA	NA	NA	1344	NA	NA
05/03/82	3.40	NA	4010	5500	330.0	400.0	126.0	22.0	126.000	405.000	28.000	12.000	NA	NA	NA	NA	NA	NA	NA	114	NA	NA
05/04/82	3.50	NA	3450	2500	156.0	216.0	66.0	12.0	58.000	142.000	12.200	5.710	NA	NA	NA	NA	NA	NA	NA	867	NA	NA
05/07/82	3.40	NA	5000	4000	212.0	300.0	136.0	16.0	105.000	316.000	22.000	10.000	NA	NA	NA	NA	NA	NA	NA	1889	NA	NA
05/10/82	3.50	NA	3650	2700	172.0	226.0	74.0	12.0	72.000	204.000	12.700	5.740	NA	NA	NA	NA	NA	NA	NA	3751	NA	NA
05/11/82	3.40	NA	4250	4200	190.0	200.0	10.0	14.0	88.000	251.000	20.000	9.000	NA	NA	NA	NA	NA	NA	NA	4698	NA	NA
05/14/82	3.40	NA	4550	4700	248.0	348.0	86.0	16.4	88.000	254.000	16.300	7.780	3.63	17.04	NA	NA	NA	NA	NA	6787	NA	NA
05/21/82	3.30	NA	4725	8000	320.0	660.0	153.0	21.0	135.000	416.000	28.000	14.000	4.01	24.00	NA	NA	NA	NA	NA	2006	NA	NA
06/02/82	3.25	NA	6160	8500	342.0	790.0	190.0	26.0	156.000	549.000	32.000	20.000	4.51	33.00	NA	NA	NA	NA	NA	723	NA	NA
06/18/82	3.30	NA	5424	8000	298.0	680.0	155.0	22.0	113.000	440.000	30.000	16.000	3.05	28.00	NA	NA	NA	NA	NA	185	NA	NA
07/02/82	NA	NA	NA	5000	308.0	362.0	110.0	18.0	97.000	359.000	21.800	8.900	3.57	19.57	NA	NA	NA	NA	NA	7908	NA	NA
07/26/82	3.24	NA	5266	5500	314.0	408.0	120.0	20.0	102.000	443.000	25.000	10.500	5.52	23.29	NA	NA	NA	NA	NA	6908	NA	NA
08/30/82	3.26	NA	4421	3750	302.0	374.0	122.0	18.0	86.000	344.000	21.200	8.500	3.26	18.89	NA	NA	NA	NA	NA	4376	NA	NA
10/04/82	3.60	NA	4316	3750	230.0	336.0	92.0	16.0	92.000	389.000	20.000	8.300	5.44	20.50	NA	NA	NA	NA	NA	23454	NA	NA
11/01/82	3.40	NA	4200	NA	168.0	278.0	76.0	12.0	58.000	291.000	17.200	7.000	3.03	17.60	NA	NA	NA	NA	NA	1363	NA	NA
03/18/83	3.66	NA	3154	3360	NA	NA	NA	NA	38.000	161.000	13.000	5.380	NA	NA	NA	NA	NA	NA	NA	1537	NA	NA
04/14/83	3.70	NA	3209	3400	NA	NA	NA	NA	40.000	174.000	14.000	5.690	NA	NA	NA	NA	NA	NA	NA	3311	NA	NA
05/15/83	3.54	NA	3873	4720	NA	NA	NA	NA	90.000	240.000	18.000	8.540	NA	NA	NA	NA	NA	NA	NA	2361	NA	NA
06/16/83	3.67	NA	4471	5000	NA	NA	NA	NA	84.000	307.000	16.600	6.490	NA	NA	NA	NA	NA	NA	NA	507	NA	NA
07/04/83	3.50	NA	3825	3700	NA	NA	NA	NA	109.000	256.000	18.400	8.200	NA	NA	NA	NA	NA	NA	NA	5314	NA	NA
07/11/83	3.70	NA	5000	7200	NA	NA	NA	NA	123.000	480.000	32.400	14.800	NA	NA	NA	NA	NA	NA	NA	731	NA	NA
07/17/83	3.50	NA	6572	7700	NA	NA	NA	NA	127.000	418.000	28.300	15.200	NA	NA	NA	NA	NA	NA	NA	284	NA	NA
07/20/83	3.50	NA	6608	6200	NA	NA	NA	NA	140.000	392.000	28.000	13.400	NA	NA	NA	NA	NA	NA	NA	825	NA	NA
07/25/83	3.60	NA	6250	8200	NA	NA	NA	NA	154.000	510.000	36.200	16.200	NA	NA	NA	NA	NA	NA	NA	341	NA	NA
07/31/83	3.65	NA	6405	8900	NA	NA	NA	NA	142.000	535.000	38.000	16.800	NA	NA	NA	NA	NA	NA	NA	148	NA	NA
08/07/83	3.60	NA	4860	4700	NA	NA	NA	NA	70.600	314.000	23.800	10.600	NA	NA	NA	NA	NA	NA	NA	590	NA	NA
08/15/83	3.45	NA	6376	6400	NA	NA	NA	NA	81.000	384.000	29.600	12.600	NA	NA	NA	NA	NA	NA	NA	307	NA	NA
08/22/83	3.60	NA	3500	3150	NA	NA	NA	NA	71.000	201.000	14.640	5.920	NA	NA	NA	NA	NA	NA	NA	3653	NA	NA
08/28/83	3.60	NA	4300	4000	NA	NA	NA	NA	89.000	265.000	18.480	8.140	NA	NA	NA	NA	NA	NA	NA	1393	NA	NA
09/04/83	3.55	NA	5537	5500	NA	NA	NA	NA	110.000	349.000	23.500	10.830	NA	NA	NA	NA	NA	NA	NA	636	NA	NA
09/11/83	3.45	NA	3975	3150	NA	NA	NA	NA	75.000	237.000	16.790	7.370	NA	NA	NA	NA	NA	NA	NA	1298	NA	NA
09/18/83	3.50	NA	4500	4300	NA	NA	NA	NA	85.000	289.000	19.760	8.780	NA	NA	NA	NA	NA	NA	NA	560	NA	NA
09/25/83	3.80	NA	3725	3500	NA	NA	NA	NA	69.000	240.000	17.000	7.380	NA	NA	NA	NA	NA	NA	NA	874	NA	NA
10/02/83	3.50	NA	4475	3400	NA	NA	NA	NA	108.000	241.000	18.000	7.460	NA	NA	NA	NA	NA	NA	NA	10401	NA	NA
10/08/83	3.75	NA	4725	3650	NA	NA	NA	NA	106.000	258.000	15.500	7.820	NA	NA	NA	NA	NA	NA	NA	6018	NA	NA
10/15/83	3.70	NA	4624	4050	NA	NA	NA	NA	116.000	290.000	19.000	8.600	NA	NA	NA	NA	NA	NA	NA	6964	NA	NA

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
10/23/83	3.50	NA	5538	5300	NA	NA	NA	NA	130.000	357.000	23.000	10.350	NA	NA	NA	NA	NA	NA	NA	1480	NA	NA
10/30/83	3.65	NA	5576	6200	NA	NA	NA	NA	149.000	410.000	29.000	13.600	NA	NA	NA	NA	NA	NA	NA	1083	NA	NA
11/06/83	3.70	NA	6116	7000	NA	NA	NA	NA	131.800	473.000	32.800	15.200	NA	NA	NA	NA	NA	NA	NA	333	NA	NA
11/13/83	3.80	NA	6958	8100	NA	NA	NA	NA	136.000	520.000	36.800	17.400	NA	NA	NA	NA	NA	NA	NA	212	NA	NA
11/20/83	3.50	NA	3800	3250	NA	NA	NA	NA	82.000	236.000	16.200	7.800	NA	NA	NA	NA	NA	NA	NA	2120	NA	NA
12/01/83	3.55	NA	4875	4600	NA	NA	NA	NA	117.600	314.000	21.600	10.400	NA	NA	NA	NA	NA	NA	NA	1881	NA	NA
02/24/84	3.60	NA	6110	6800	356.5	991.3	165.9	NA	133.800	478.000	32.200	17.400	NA	NA	NA	NA	NA	NA	NA	889	NA	NA
03/15/84	3.55	NA	3225	3200	220.6	381.1	73.2	NA	77.000	220.000	15.000	7.200	NA	NA	NA	NA	NA	NA	NA	1211	NA	NA
04/15/84	3.81	NA	1260	1300	110.9	104.8	23.0	NA	31.000	62.000	4.400	2.300	NA	NA	NA	NA	NA	NA	NA	11004	NA	NA
05/16/84	3.65	NA	2392	2950	NA	NA	NA	NA	83.000	160.000	12.500	5.340	NA	NA	NA	NA	NA	NA	NA	8396	NA	NA
06/13/84	3.58	NA	2985	4500	242.8	342.5	69.3	NA	90.200	207.000	14.000	6.600	NA	NA	NA	NA	NA	NA	NA	11254	NA	NA
07/14/84	3.45	NA	5230	6700	NA	NA	122.0	NA	142.000	356.000	23.000	7.000	NA	NA	NA	NA	NA	NA	NA	2498	NA	NA
08/15/84	3.47	NA	4760	5000	323.7	348.7	115.6	NA	107.000	334.560	34.400	10.540	NA	NA	NA	NA	NA	NA	NA	2252	NA	NA
09/12/84	3.44	NA	5175	6000	374.0	580.0	146.5	NA	105.800	425.000	44.000	14.600	NA	NA	NA	NA	NA	NA	NA	886	NA	NA
10/13/84	3.41	NA	2870	3625	260.1	290.0	74.3	NA	88.700	218.000	21.600	7.700	NA	NA	NA	NA	NA	NA	NA	7066	NA	NA
11/14/84	3.51	NA	3150	2750	372.0	500.0	115.8	NA	134.700	338.000	19.400	10.700	NA	NA	NA	NA	NA	NA	NA	2074	NA	NA
12/05/84	3.42	NA	2470	2200	314.0	470.0	89.0	NA	100.100	260.000	15.500	8.500	NA	NA	NA	NA	NA	NA	NA	651	NA	NA
12/19/84	3.45	NA	2650	2900	254.0	285.0	88.0	NA	97.300	223.000	13.400	6.720	NA	NA	NA	NA	NA	NA	NA	1444	NA	NA
04/02/85	3.08	NA	5000	5650	330.0	562.0	135.6	NA	104.000	348.000	23.000	13.480	NA	NA	NA	NA	NA	NA	NA	489	NA	NA
04/16/85	3.94	NA	2745	2850	210.0	252.0	51.2	NA	86.000	157.000	13.000	5.400	NA	NA	NA	NA	NA	NA	NA	14152	NA	NA
05/14/85	3.74	NA	4640	4808	285.0	475.0	95.0	NA	150.000	279.000	10.500	18.800	NA	NA	NA	NA	NA	NA	NA	3970	NA	NA
06/14/85	3.58	NA	3850	4200	224.0	375.0	70.0	NA	110.000	211.000	14.600	7.100	NA	NA	NA	NA	NA	NA	NA	22384	NA	NA
07/15/85	3.48	NA	4900	5200	270.0	446.0	102.0	NA	110.000	269.000	17.000	7.800	NA	NA	NA	NA	NA	NA	NA	4534	NA	NA
08/13/85	3.47	NA	4450	5100	272.0	466.0	96.0	NA	96.000	266.000	17.800	6.380	NA	NA	NA	NA	NA	NA	NA	2203	NA	NA
09/13/85	3.50	NA	4325	4300	334.0	305.0	78.0	NA	103.000	221.000	14.800	6.500	NA	NA	NA	NA	NA	NA	NA	16207	NA	NA
10/15/85	3.50	NA	4075	3200	276.0	236.0	60.0	NA	94.000	174.000	12.500	5.320	NA	NA	NA	NA	NA	NA	NA	17676	NA	NA
11/15/85	3.54	NA	5200	2100	388.0	510.0	100.0	NA	119.000	296.000	20.700	8.570	NA	NA	NA	NA	NA	NA	NA	1499	NA	NA
12/10/85	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	106	NA	NA
03/23/86	3.80	NA	4800	4200	270.0	414.0	86.0	NA	67.000	250.000	22.000	7.340	NA	NA	NA	NA	NA	NA	NA	7244	NA	NA
04/16/86	3.90	NA	1950	1700	152.0	130.0	28.6	NA	47.000	97.000	11.600	3.010	NA	NA	NA	NA	NA	NA	NA	9909	NA	NA
05/14/86	3.69	NA	3000	3150	282.0	270.0	44.0	NA	92.000	251.000	11.000	5.000	NA	NA	NA	NA	NA	NA	NA	10125	NA	NA
06/11/86	3.63	NA	4750	4600	316.0	360.0	90.0	NA	124.000	260.000	14.000	8.200	NA	NA	NA	NA	NA	NA	NA	1711	NA	NA
07/12/86	3.59	NA	1725	3000	225.0	314.0	61.0	NA	77.000	180.000	12.700	6.300	NA	NA	NA	NA	NA	NA	NA	3085	NA	NA
08/13/86	3.55	NA	4100	3700	268.0	357.0	78.0	NA	79.000	206.000	22.400	7.400	NA	NA	NA	NA	NA	NA	NA	1597	NA	NA
09/14/86	3.40	NA	3550	2700	262.0	286.0	56.0	NA	98.000	159.000	19.400	5.980	NA	NA	NA	NA	NA	NA	NA	10795	NA	NA
10/16/86	3.53	NA	4400	3600	286.0	302.0	68.0	NA	110.000	203.000	13.700	7.150	NA	NA	NA	NA	NA	NA	NA	3982	NA	NA
11/16/86	3.70	NA	3550	2700	260.0	236.0	60.0	NA	110.000	158.000	10.600	7.200	NA	NA	NA	NA	NA	NA	NA	6075	NA	NA
03/28/87	3.65	NA	3075	2900	252.0	264.0	48.0	NA	75.000	160.000	10.800	5.160	NA	NA	NA	NA	NA	NA	NA	7835	NA	NA
04/18/87	3.67	NA	2625	2400	154.0	220.0	40.0	NA	63.000	137.000	10.000	4.440	NA	NA	NA	NA	NA	NA	NA	1620	NA	NA
05/16/87	3.61	NA	2700	2250	233.0	172.0	34.0	NA	73.000	97.000	6.700	3.730	NA	NA	NA	NA	NA	NA	NA	18573	NA	NA
06/16/87	3.44	NA	4900	4640	428.0	492.0	116.0	NA	153.000	331.000	18.600	25.200	NA	NA	NA	NA	NA	NA	NA	1400	NA	NA
07/15/87	3.36	NA	3000	2900	230.0	228.0	48.0	NA	82.000	140.000	9.000	4.930	NA	NA	NA	NA	NA	NA	NA	16908	NA	NA
08/15/87	3.24	NA	3950	3565	279.0	316.0	68.0	NA	97.000	193.000	12.200	6.590	NA	NA	NA	NA	NA	NA	NA	3948	NA	NA
09/16/87	3.25	NA	4100	1200	240.0	238.0	50.0	NA	79.000	135.000	7.980	4.660	NA	NA	NA	NA	NA	NA	NA	7612	NA	NA
10/13/87	3.32	NA	4950	5120	364.0	391.0	88.0	NA	132.000	220.000	13.010	7.250	NA	NA	NA	NA	NA	NA	NA	1457	NA	NA
11/10/87	3.43	NA	4325	3700	308.0	354.0	78.0	NA	9.300	193.000	11.240	6.360	NA	NA	NA	NA	NA	NA	NA	731	NA	NA
05/12/88	3.64	NA	3000	2300	224.0	198.0	54.0	NA	89.500	156.500	8.950	8.450	NA	NA	NA	NA	NA	NA	NA	7275	NA	NA
06/12/88	3.47	NA	4850	4500	412.0	376.0	72.0	NA	163.000	232.000	18.600	23.000	NA	NA	NA	NA	NA	NA	NA	1098	NA	NA
07/15/88	3.60	NA	4525	4360	368.0	382.0	74.0	NA	134.000	234.000	17.600	16.000	NA	NA	NA	NA	NA	NA	NA	757	NA	NA
08/16/88	3.39	NA	2900	2165	247.0	176.0	43.0	NA	72.000	152.000	8.800	6.250	NA	NA	NA	NA	NA	NA	NA	32744	NA	NA
09/16/88	3.37	NA	4275	3200	334.0	294.0	74.0	NA	93.000	117.000	14.300	10.300	NA	NA	NA	NA	NA	NA	NA	6310	NA	NA

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date[1]	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/24/88	3.70	NA	1875	1280	160.0	96.0	26.0	NA	36.000	71.000	4.300	3.700	NA	NA	NA	NA	NA	NA	NA	6934	NA	NA
10/18/88	3.67	NA	3700	3400	238.0	258.0	50.0	NA	64.000	143.000	10.900	8.700	NA	NA	NA	NA	NA	NA	NA	1790	NA	NA
10/24/88	3.77	NA	1980	1400	132.0	118.0	26.0	NA	43.000	88.000	4.900	4.400	NA	NA	NA	NA	NA	NA	NA	3819	NA	NA
11/15/88	3.71	NA	2975	2740	216.0	222.0	42.0	NA	68.000	107.000	8.200	7.200	NA	NA	NA	NA	NA	NA	NA	7494	NA	NA
11/30/88	3.44	NA	4200	3760	276.0	306.0	56.0	NA	102.000	169.000	11.500	9.300	NA	NA	NA	NA	NA	NA	NA	2665	NA	NA
04/07/89	3.95	NA	2700	2400	230.0	166.0	38.0	NA	55.000	98.000	9.800	7.000	NA	NA	NA	NA	NA	NA	NA	17581	NA	NA
04/24/89	4.00	NA	1050	650	78.0	38.0	10.0	NA	18.000	26.000	1.800	1.600	NA	NA	NA	NA	NA	NA	NA	7850	NA	NA
05/16/89	3.79	NA	2582	2100	182.0	152.0	34.0	NA	62.000	94.000	7.300	6.100	NA	NA	NA	NA	NA	NA	NA	5802	NA	NA
06/17/89	3.64	NA	3150	2800	258.0	184.0	52.0	NA	80.000	145.000	13.700	7.700	NA	NA	NA	NA	NA	NA	NA	25560	NA	NA
07/17/89	3.55	NA	4325	4100	420.0	304.0	62.0	NA	118.000	210.000	17.700	10.900	NA	NA	NA	NA	NA	NA	NA	2248	NA	NA
08/14/89	3.62	NA	3800	3540	286.0	284.0	58.0	NA	89.000	245.000	15.400	10.200	NA	NA	NA	NA	NA	NA	NA	1158	NA	NA
09/14/89	3.41	NA	2900	2460	272.0	168.0	36.0	NA	68.000	91.000	11.800	6.600	NA	NA	NA	NA	NA	NA	NA	16325	NA	NA
10/16/89	3.51	NA	3700	2960	276.0	226.0	52.0	NA	83.000	116.000	17.700	9.600	NA	NA	NA	NA	NA	NA	NA	2203	NA	NA
11/13/89	3.60	NA	3200	2480	298.0	204.0	44.0	NA	74.000	105.000	21.000	8.900	NA	NA	NA	NA	NA	NA	NA	3758	NA	NA
03/26/90	3.70	NA	2500	1900	226.0	146.0	31.2	NA	41.000	73.000	6.350	7.310	NA	NA	NA	NA	NA	NA	NA	3615	NA	NA
04/12/90	3.91	NA	1950	1660	182.0	108.0	20.0	NA	53.000	57.000	4.380	3.980	NA	NA	NA	NA	NA	NA	NA	20708	NA	NA
05/17/90	3.75	NA	2725	2300	222.0	156.0	26.0	NA	70.000	80.000	6.570	5.560	NA	NA	NA	NA	NA	NA	NA	10783	NA	NA
06/18/90	3.59	NA	3475	2660	236.0	180.0	38.0	NA	72.000	92.000	9.170	7.640	NA	NA	NA	NA	NA	NA	NA	9421	NA	NA
07/16/90	3.62	NA	4175	3460	306.0	226.0	40.0	NA	92.000	108.000	9.600	8.610	NA	NA	NA	NA	NA	NA	NA	6684	NA	NA
08/13/90	3.53	NA	4785	3920	370.0	326.0	62.0	NA	106.000	166.000	13.100	11.200	NA	NA	NA	NA	NA	NA	NA	757	NA	NA
09/10/90	3.64	NA	3260	2360	238.0	178.0	38.0	NA	54.000	99.000	7.280	5.380	NA	NA	NA	NA	NA	NA	NA	2434	NA	NA
10/15/90	3.55	NA	3415	2160	252.0	154.0	32.0	NA	74.000	88.000	6.510	5.000	NA	NA	NA	NA	NA	NA	NA	20643	NA	NA
11/16/90	3.60	NA	5000	3940	372.0	308.0	62.0	NA	120.000	142.000	12.170	12.440	NA	NA	NA	NA	NA	NA	NA	1397	NA	NA
04/08/91	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	9777	NA	NA
04/22/91	3.76	NA	1700	1010	120.0	80.0	14.0	NA	34.000	41.000	2.200	1.600	NA	NA	NA	NA	NA	NA	NA	5598	NA	NA
05/20/91	3.65	NA	4175	2500	262.0	205.0	32.0	NA	94.000	103.000	5.600	4.200	NA	NA	NA	NA	NA	NA	NA	8687	NA	NA
06/17/91	3.61	NA	2975	2000	188.0	168.0	26.0	NA	64.000	84.000	5.300	3.300	NA	NA	NA	NA	NA	NA	NA	3388	NA	NA
07/15/91	3.47	NA	3225	2020	198.0	140.0	24.0	NA	66.000	69.000	4.600	3.500	NA	NA	NA	NA	NA	NA	NA	16938	NA	NA
08/12/91	3.43	NA	5000	4460	394.0	334.0	72.0	NA	124.000	176.000	11.900	8.700	NA	NA	NA	NA	NA	NA	NA	1268	NA	NA
09/09/91	3.59	NA	3225	1940	200.0	126.0	32.0	NA	54.000	73.000	4.600	3.700	NA	NA	NA	NA	NA	NA	NA	6079	NA	NA
10/07/91	3.62	NA	3775	2380	238.0	158.0	38.0	NA	72.000	87.000	5.500	5.600	NA	NA	NA	NA	NA	NA	NA	3656	NA	NA
11/07/91	3.51	NA	3300	2280	236.0	156.0	34.0	NA	75.000	80.000	5.100	5.100	NA	NA	NA	NA	NA	NA	NA	14780	NA	NA

NA: Not analyzed.

[1]Dates represent intermediate dates for measurement periods.

Table A2.6.1. Drainage quality data from Duluth Complex test pile FL5, 1992-1993 (1.41% S, 0.30% Cu, 0.08% Ni).

Date[1]	pH	Alk. mg/L	Lab Acy. mg/L	Calc. Acy. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/23/92	3.86	NA	410	223.2	1525	720	76.0	56.0	14.00	3.30	24	19.90	28	1.70	1.20	0.34	3.04	NA	NA	NA	NA	NA	4634	NA	NA
05/11/92	3.76	NA	695	388.8	2420	725	148.0	96.0	22.00	5.30	37	53.00	47	2.90	2.10	0.87	5.52	NA	NA	NA	NA	NA	2162	NA	NA
05/26/92	3.71	NA	838	449.5	2480	1760	176.0	118.0	26.60	6.30	42	59.00	58	3.60	2.80	1.07	6.68	NA	NA	NA	NA	NA	2294	NA	NA
06/08/92	4.22	NA	940	541.4	3780	2560	320.0	216.0	40.00	11.00	44	79.00	82	5.10	4.00	0.47	9.48	NA	NA	NA	NA	NA	603	NA	NA
06/22/92	3.66	NA	1350	667.2	3850	2960	252.0	208.0	42.80	9.10	56	93.00	99	6.80	4.70	1.34	10.90	NA	NA	NA	NA	NA	238	NA	NA
07/07/92	3.68	NA	860	445.1	2640	1880	174.0	124.0	28.00	7.20	40	59.00	61	4.10	3.00	1.09	6.89	NA	NA	NA	NA	NA	1105	NA	NA
07/20/92	3.61	NA	1310	652.7	3880	5900	284.0	202.0	44.00	10.00	53	99.00	95	6.70	4.60	1.44	10.70	NA	NA	NA	NA	NA	366	NA	NA
08/03/92	3.64	NA	1050	486.2	2710	1990	192.0	144.0	34.00	8.60	43	63.00	69	4.90	3.60	1.13	8.08	NA	NA	NA	NA	NA	336	NA	NA
08/17/92	4.43	NA	550	337.1	3000	2060	214.0	228.0	39.00	17.10	23	51.00	60	4.30	3.00	0.27	7.93	NA	NA	NA	NA	NA	422	NA	NA
08/31/92	4.04	NA	410	271.4	2400	1400	170.0	144.0	28.40	18.20	17	39.00	56	3.20	2.30	0.44	5.70	NA	NA	NA	NA	NA	309	NA	NA
09/14/92	3.57	NA	1085	473.6	2800	2060	216.0	144.0	33.00	8.40	41	67.00	65	4.60	3.20	1.85	7.43	NA	NA	NA	NA	NA	2397	NA	NA
09/28/92	3.60	NA	1120	507.0	3300	2240	228.0	150.0	34.60	9.20	43	76.00	69	4.90	3.40	2.07	7.52	NA	NA	NA	NA	NA	1997	NA	NA
10/12/92	4.35	NA	1130	554.2	4200	2940	326.0	272.0	51.00	15.30	43	82.00	87	6.30	4.50	0.42	10.80	NA	NA	NA	NA	NA	285	NA	NA
10/27/92	3.57	NA	1700	720.1	4380	3400	288.0	260.0	53.40	11.40	57	97.00	119	7.90	5.50	1.46	12.90	NA	NA	NA	NA	NA	147	NA	NA
11/09/92	4.07	NA	NA	NA	5000	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	69	NA	NA
11/30/92	3.75	NA	1090	445.3	2700	2050	184.0	158.0	33.00	7.60	41	50.00	64	4.80	3.30	0.92	8.19	NA	NA	NA	NA	NA	411	NA	NA
04/08/93	3.82	NA	NA	NA	1825	1275	125.0	100.0	16.50	4.50	48	35.00	52	2.86	1.90	0.43	4.71	NA	NA	NA	NA	NA	15454	NA	NA
04/22/93	3.76	NA	NA	NA	2775	1930	180.0	137.5	21.50	5.50	58	57.00	64	4.21	2.27	1.05	6.15	NA	NA	NA	NA	NA	3653	NA	NA
05/06/93	3.68	NA	NA	NA	2100	1560	147.5	105.0	18.25	5.25	48	45.00	49	3.23	1.78	1.27	4.59	NA	NA	NA	NA	NA	8846	NA	NA
05/20/93	3.67	NA	NA	NA	3350	2880	245.0	200.0	32.00	6.50	77	95.00	98	6.30	4.10	1.56	8.97	NA	NA	NA	NA	NA	1915	NA	NA
06/03/93	3.76	NA	NA	NA	2550	1800	147.5	125.0	23.25	5.00	55	58.00	66	4.13	2.55	1.10	5.71	NA	NA	NA	NA	NA	2415	NA	NA
06/17/93	3.56	NA	NA	NA	2300	1990	180.0	152.5	25.50	5.25	59	76.00	80	5.16	3.17	1.33	7.15	NA	NA	NA	NA	NA	2933	NA	NA
07/01/93	3.58	NA	NA	NA	2550	1900	182.5	147.5	24.75	5.50	63	77.00	78	4.96	3.07	1.51	7.04	NA	NA	NA	NA	NA	3338	NA	NA
07/14/93	3.57	NA	NA	NA	3090	2820	228.0	154.0	28.40	7.00	58	77.00	84	5.04	3.22	1.72	7.58	NA	NA	NA	NA	NA	15715	NA	NA
07/29/93	3.58	NA	NA	NA	3100	2600	242.0	170.0	31.80	7.40	59	79.00	89	5.60	3.60	1.58	8.16	NA	NA	NA	NA	NA	2986	NA	NA
08/12/93	3.50	NA	NA	NA	3350	2450	238.0	170.0	31.60	7.20	66	75.00	94	5.10	3.83	1.55	8.83	NA	NA	NA	NA	NA	2245	NA	NA
08/26/93	3.47	NA	NA	NA	4030	3160	314.0	242.0	44.40	9.20	83	92.00	123	7.25	5.58	1.83	13.00	NA	NA	NA	NA	NA	3338	NA	NA
09/09/93	3.60	NA	NA	NA	2900	2080	216.0	144.0	28.00	7.20	54	62.00	76	4.74	3.37	1.27	7.85	NA	NA	NA	NA	NA	2865	NA	NA
09/23/93	3.61	NA	NA	NA	3300	2450	266.0	190.0	35.80	8.20	69	72.00	100	5.97	4.47	1.46	10.20	NA	NA	NA	NA	NA	1446	NA	NA
10/07/93	3.66	NA	NA	NA	3700	2560	276.0	200.0	38.80	8.80	76	77.00	105	6.20	4.21	1.68	9.71	NA	NA	NA	NA	NA	742	NA	NA
10/20/93	3.71	NA	NA	NA	3950	2730	280.0	208.0	40.60	8.80	76	83.00	114	6.20	4.28	1.38	10.00	NA	NA	NA	NA	NA	379	NA	NA
11/04/93	3.72	NA	NA	NA	3300	1250	159.4	138.0	26.40	6.00	52	50.00	69	4.42	2.78	0.75	6.35	NA	NA	NA	NA	NA	871	NA	NA
11/22/93	3.72	NA	NA	NA	3000	1170	270.0	210.0	39.00	8.00	63	72.00	101	6.70	4.61	1.00	10.00	NA	NA	NA	NA	NA	591	NA	NA

NA: Not analyzed.

[1]Dates represent intermediate dates for measurement periods.

Table A2.7. Annual flow-weighted mean concentration vs. annual drainage volume regression statistics[1].

Sulfur %	Pile	Flow-weighted Mean Concentration (mmol/L) = a • volume (L) + b											
		Sulfate			Calcium			Magnesium			Calcium plus Magnesium		
		$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n
0.63	FL1	-1.5	0.070	15	-0.6	0.049	14	-0.1	0.001	14	-0.7	0.026	14
0.63	FL2	1.2	0.047	15	0.8	0.141	14	0.2	0.003	14	1.0	0.065	14
0.63	FL3	0.4	0.000	11	-0.2	0.001	10	1.6	0.021	10	1.4	0.010	10
0.79	FL6	-3.3	0.034	15	0.6	0.037	14	-1.5	0.025	14	-0.9	0.008	14
1.41	FL5	3.7	0.013	15	1.8	0.389	14	-0.3	0.001	14	1.4	0.011	14

[1] Regressions excluded 1978 flow-weighted mean concentration data.

APPENDIX 3

MASS RELEASE

Appendix 4: Rates of Release

Figure A4.1.	Annual flow-weighted mean calcium plus magnesium concentrations for 1978 through 1993.
Figure A4.2.	Average rates of release of sulfate, calcium, magnesium, and calcium plus magnesium vs. sulfur content.
Figures A4.3.-A4.12.	Annual rates of release of sulfate, calcium, magnesium, and calcium plus magnesium vs. annual drainage volume.
Figures A4.13.-A4.17.	Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, FL3, FL6, and FL5 (individual plots).
Figure A4.18.	Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, and FL3 (composite plot).
Figure A4.19.	Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, FL3, FL6, and FL5 (composite plot).
Table A4.1.	Annual mass release rate vs. annual drainage volume regression statistics.

Figure A4.1. Annual calcium plus magnesium release rate over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

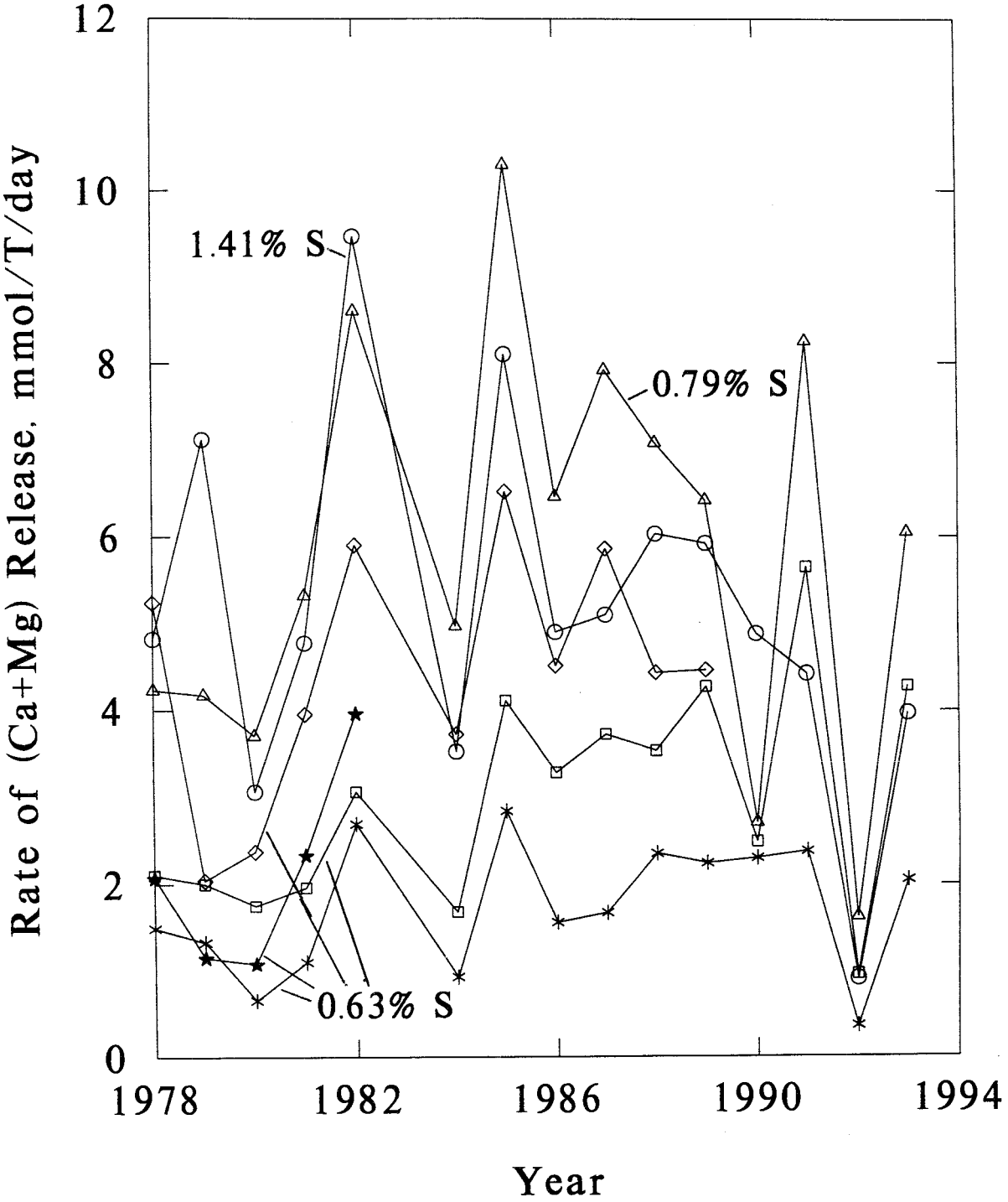

Figure A4.2. Annual rates of release of sulfate, calcium, magnesium, and calcium plus magnesium vs. sulfur content for Duluth Complex test pile FL1 (0.63% S).

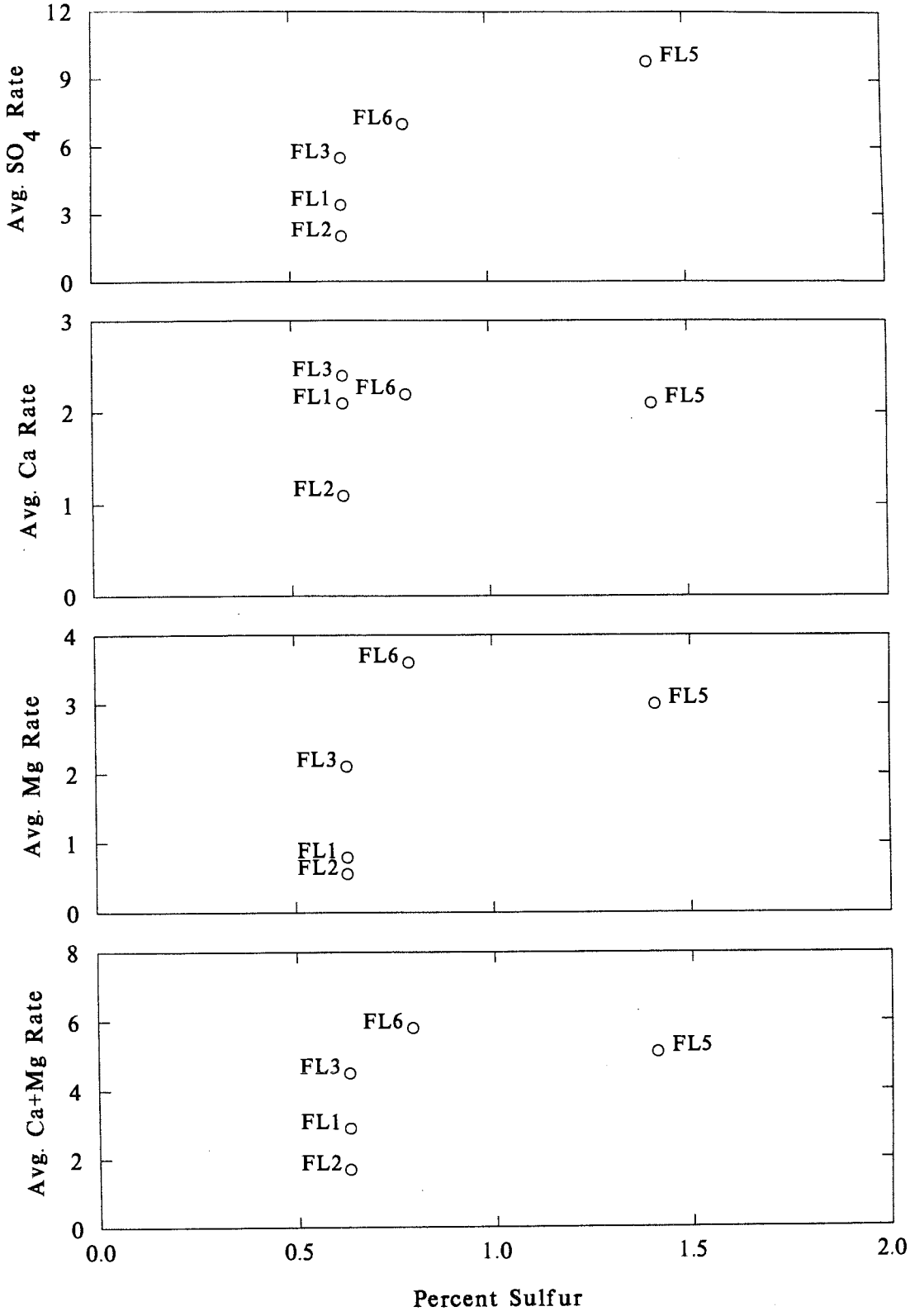

Figure A4.3. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL1 (0.63% S).

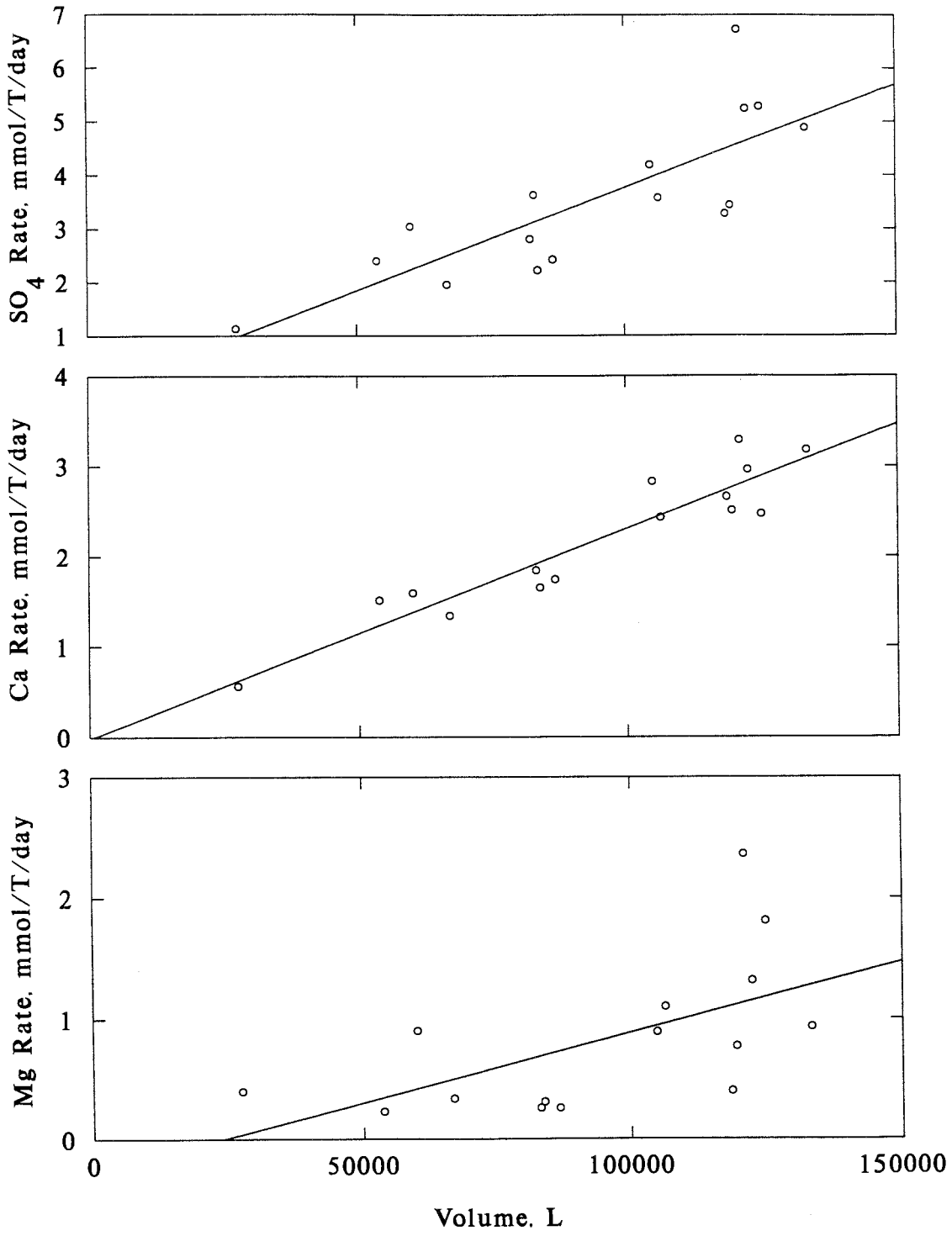

Figure A4.4. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL1 (0.63% S).

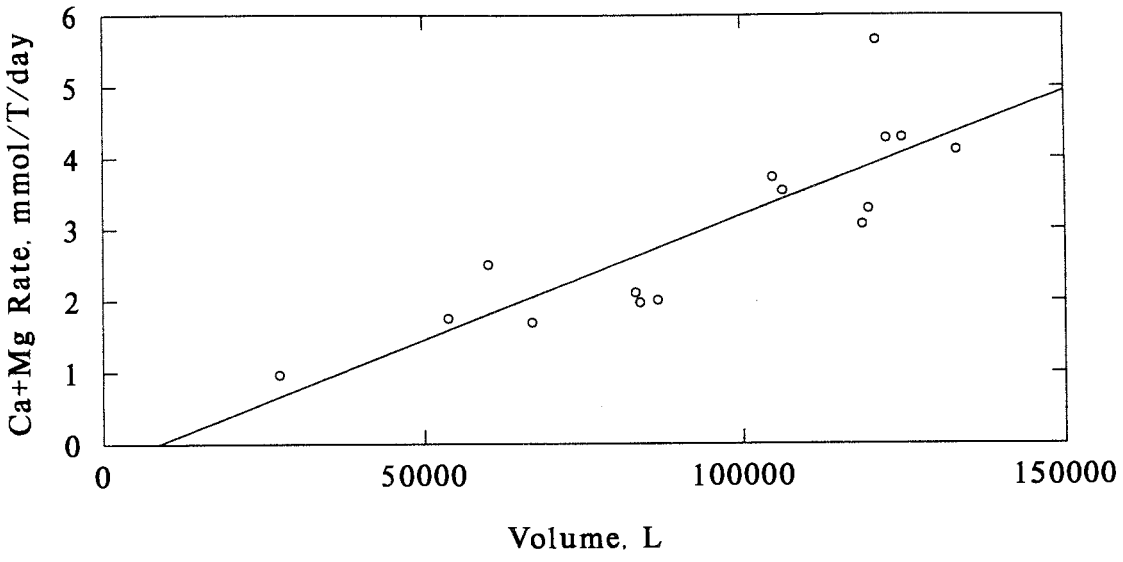

Figure A4.5. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL2 (0.63% S).

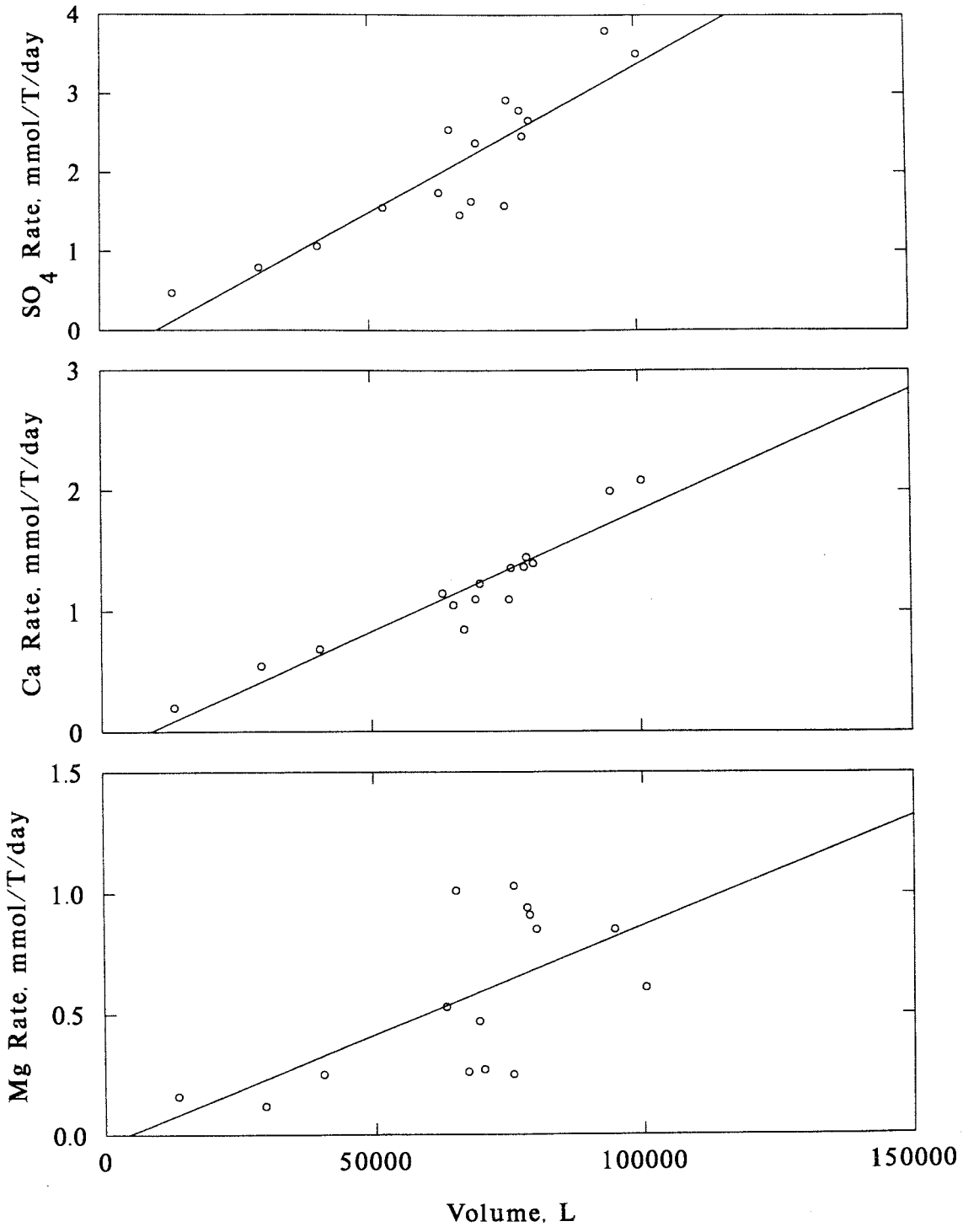

Figure A4.6. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL2 (0.63% S).

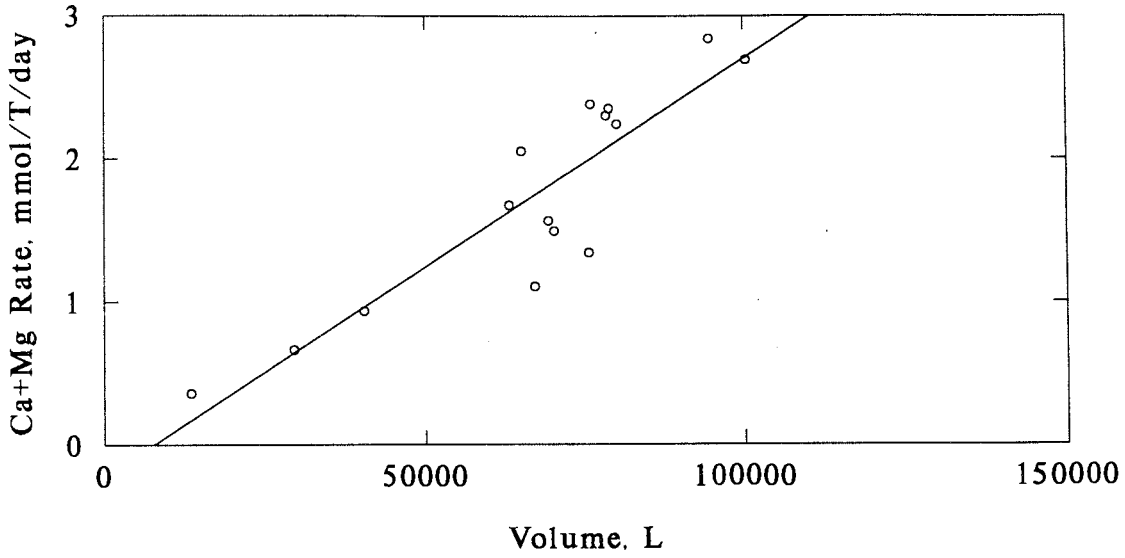

Figure A4.7. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL3 (0.63% S).

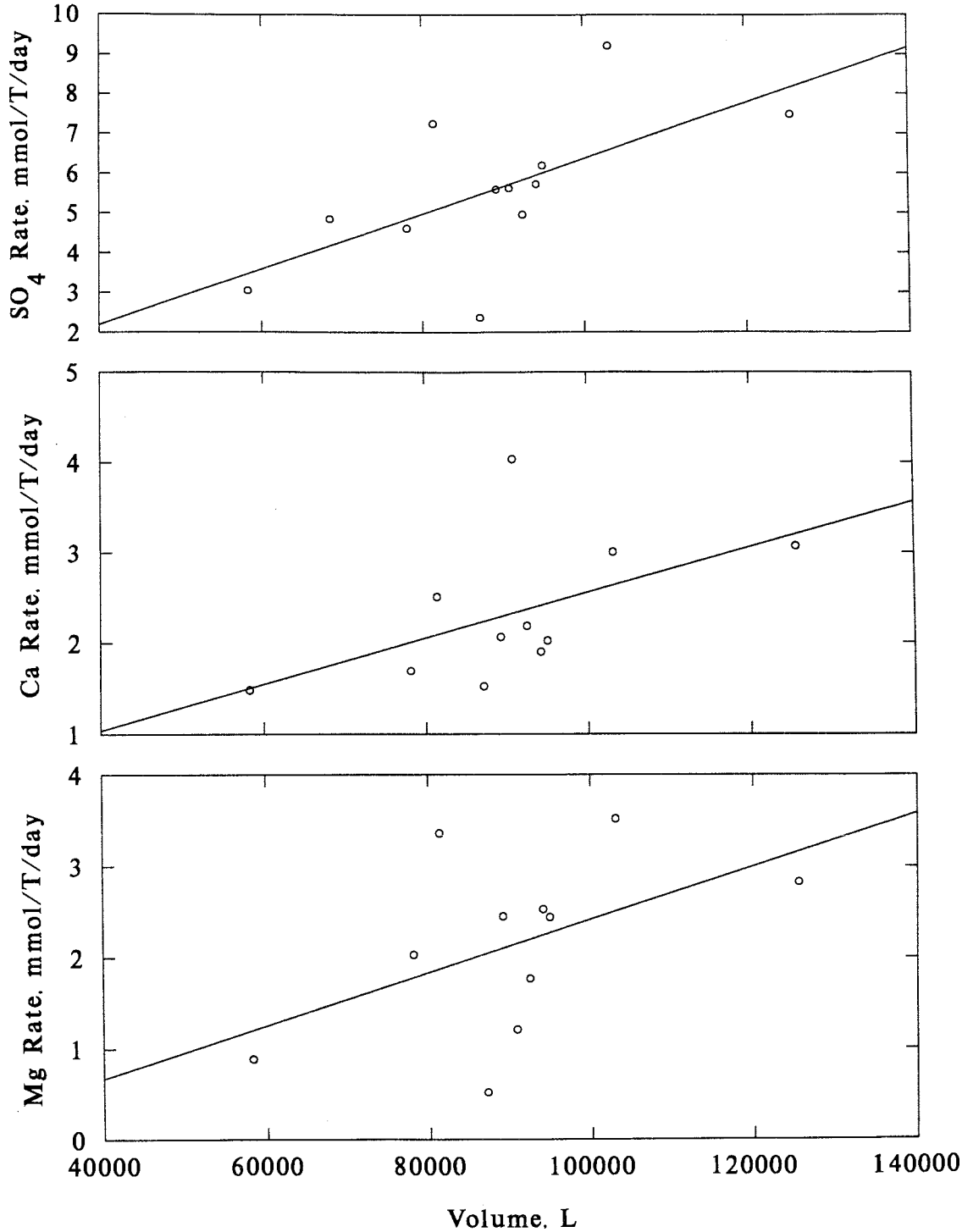

Figure A4.8. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL3 (0.63% S).

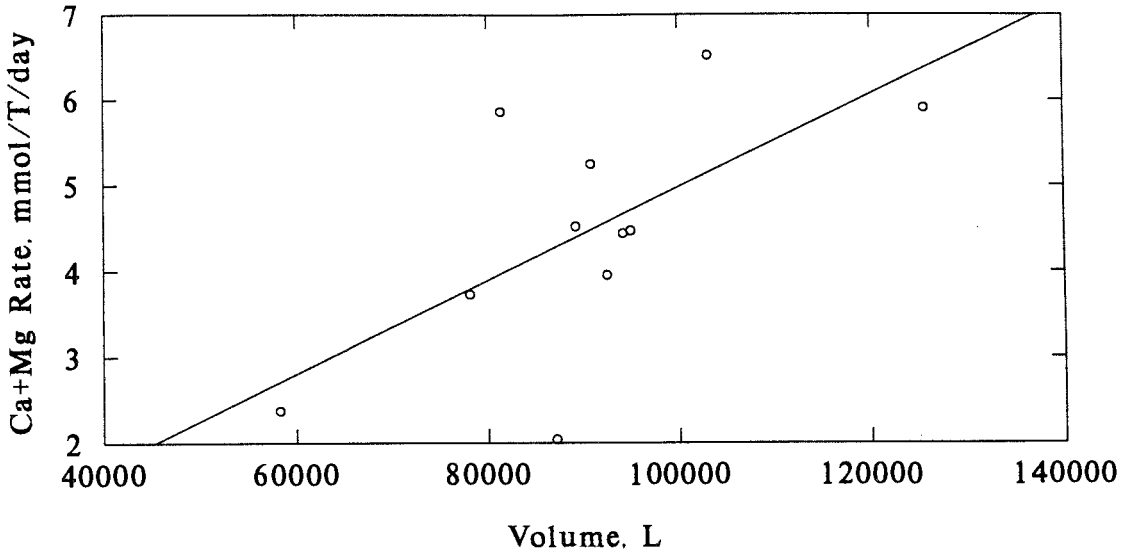

Figure A4.9. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL6 (0.79% S).

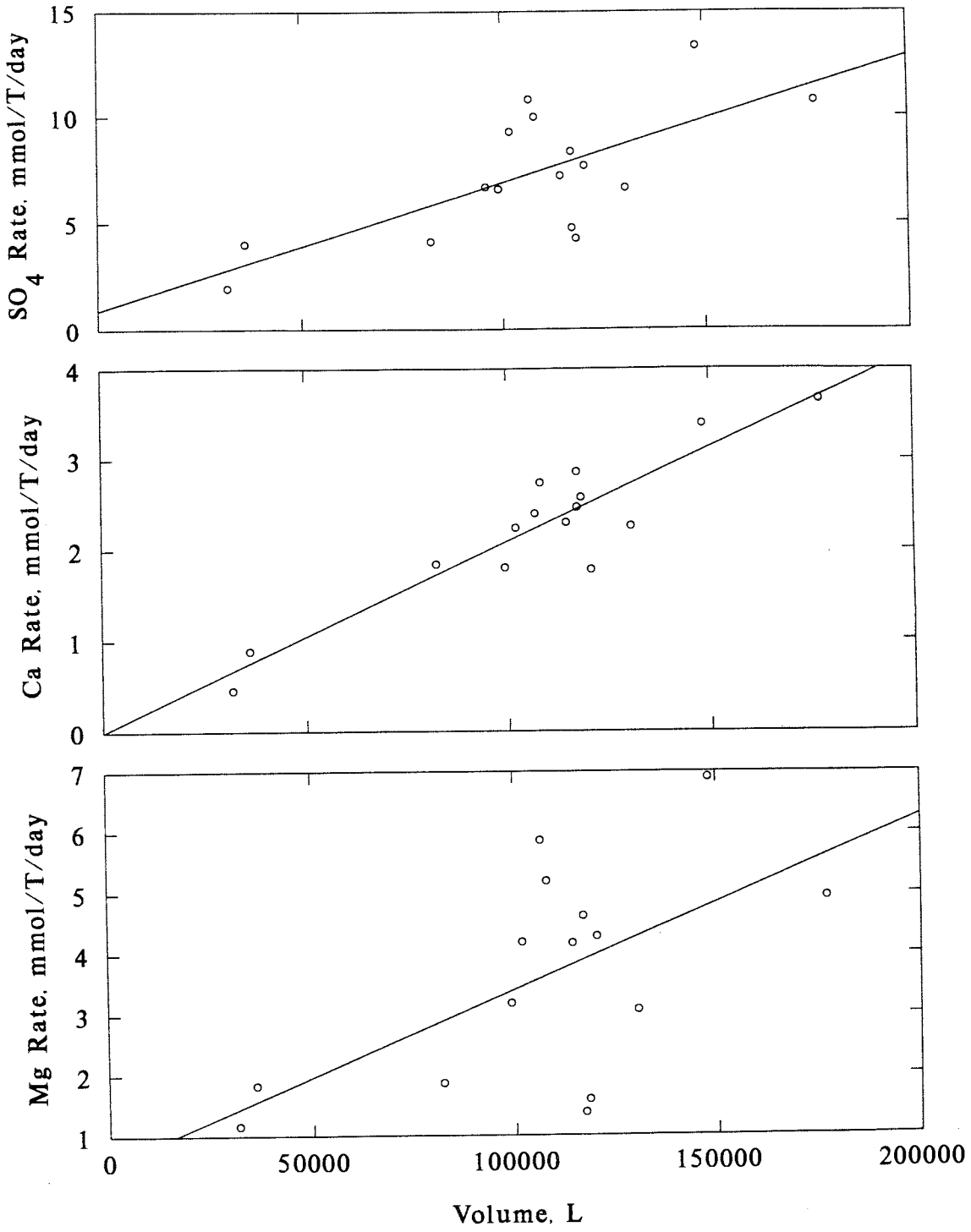

Figure A4.10. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL6 (0.79% S).

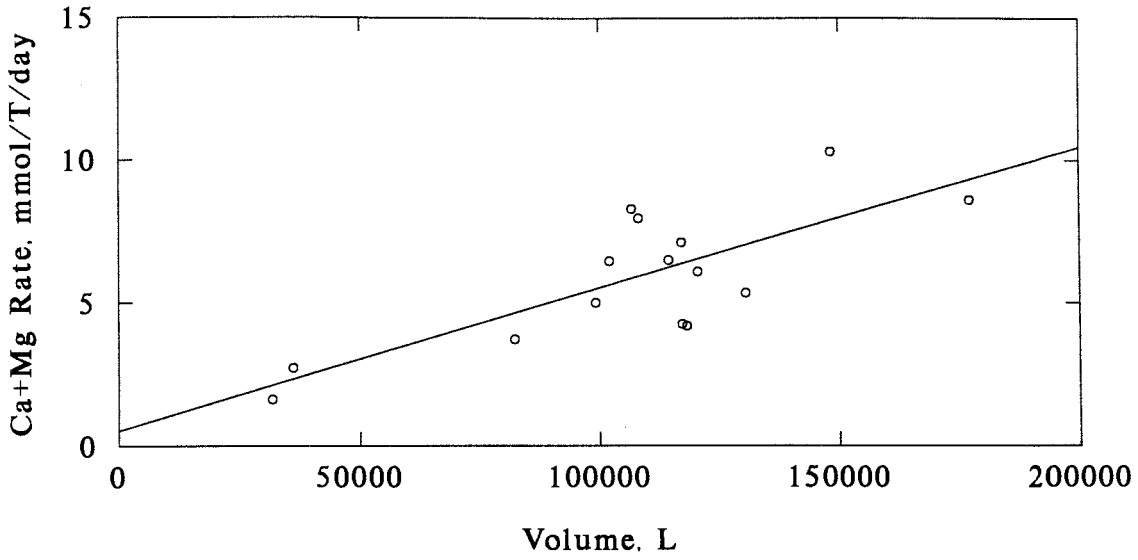

Figure A4.11. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL5 (1.41% S).

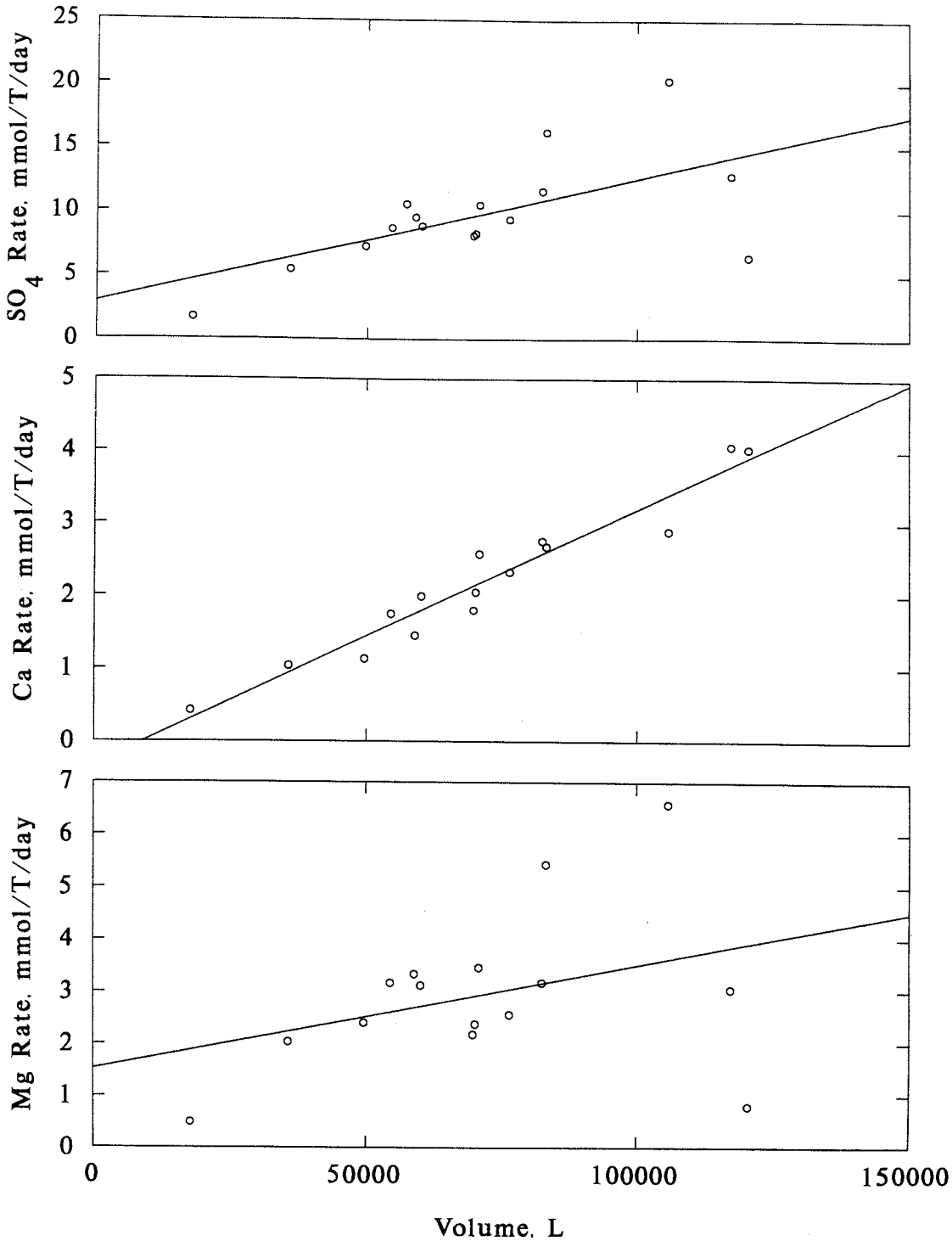

Figure A4.12. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL5 (1.41% S).

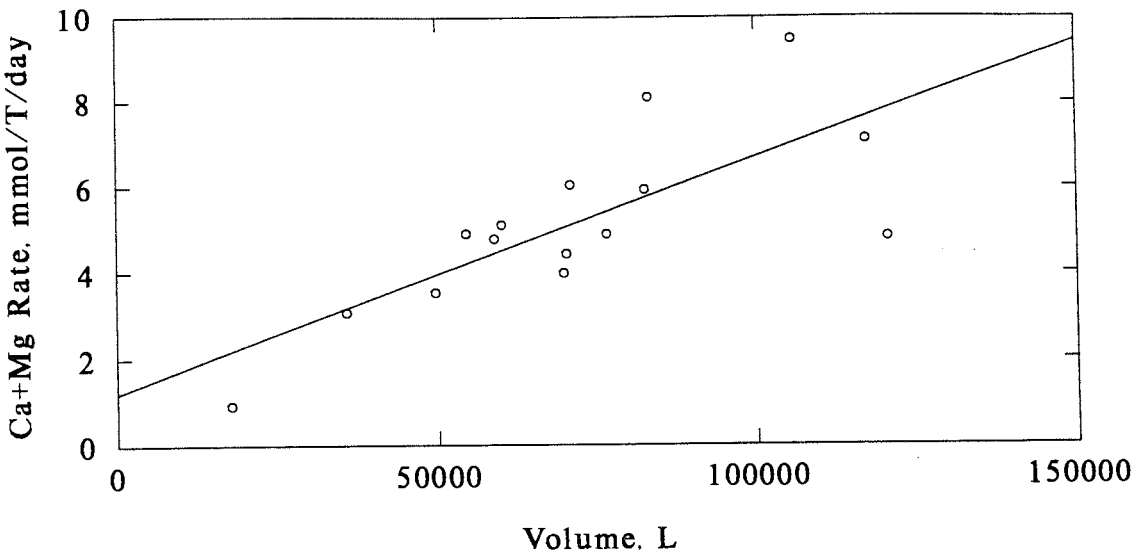

Figure A4.13. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL1 (0.63% S).

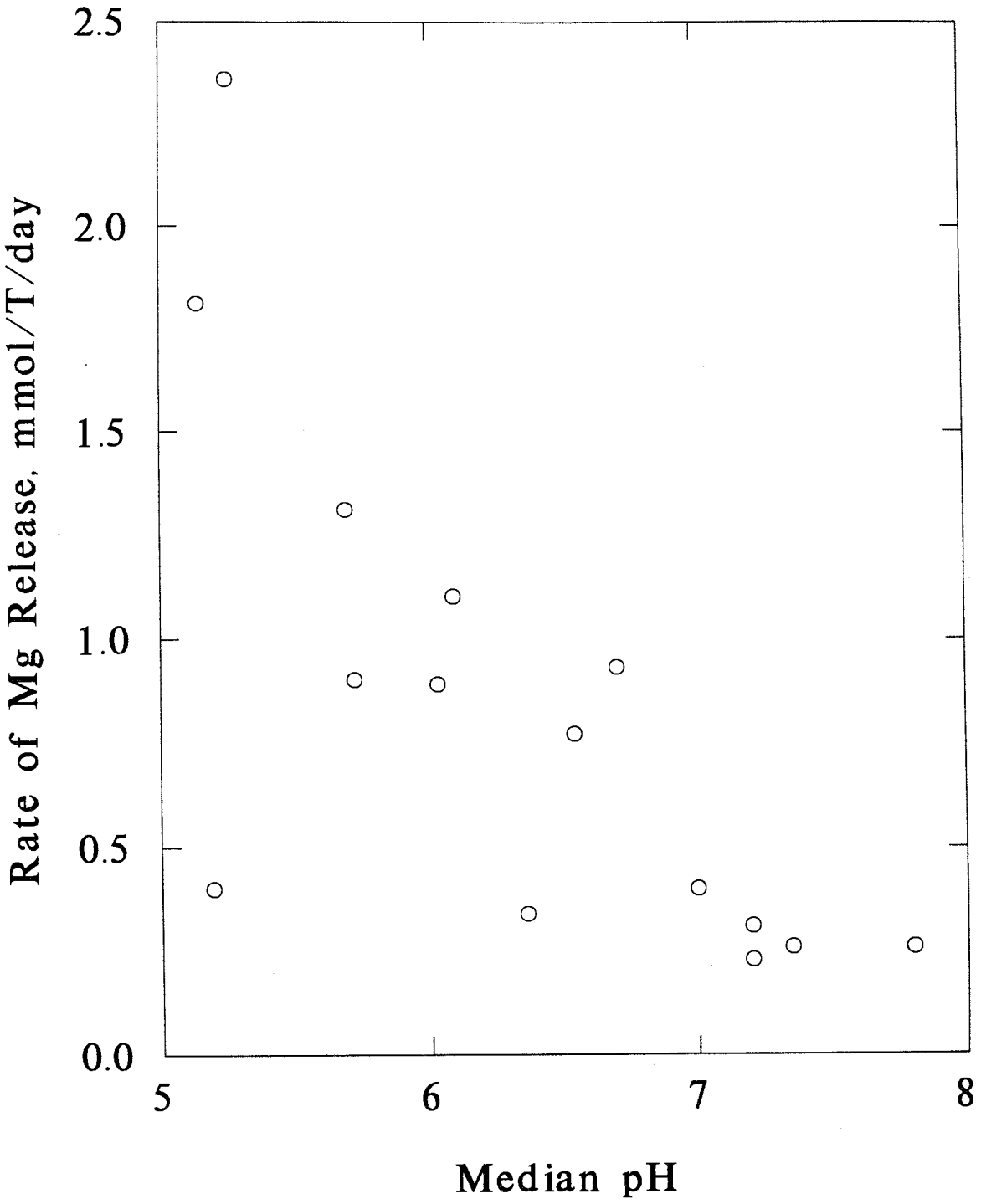

Figure A4.14. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL2 (0.63% S).

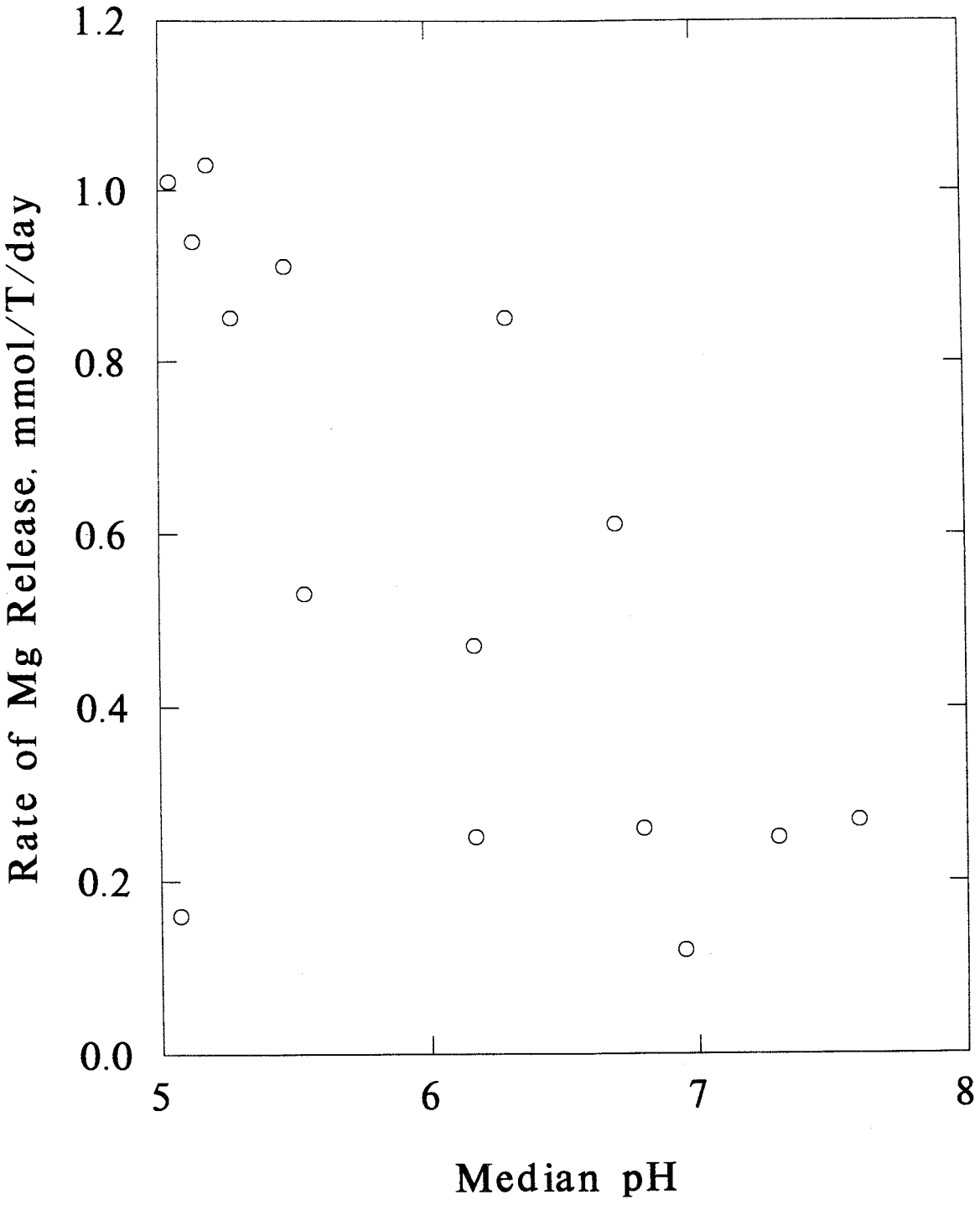

Figure A4.15. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL3 (0.63% S).

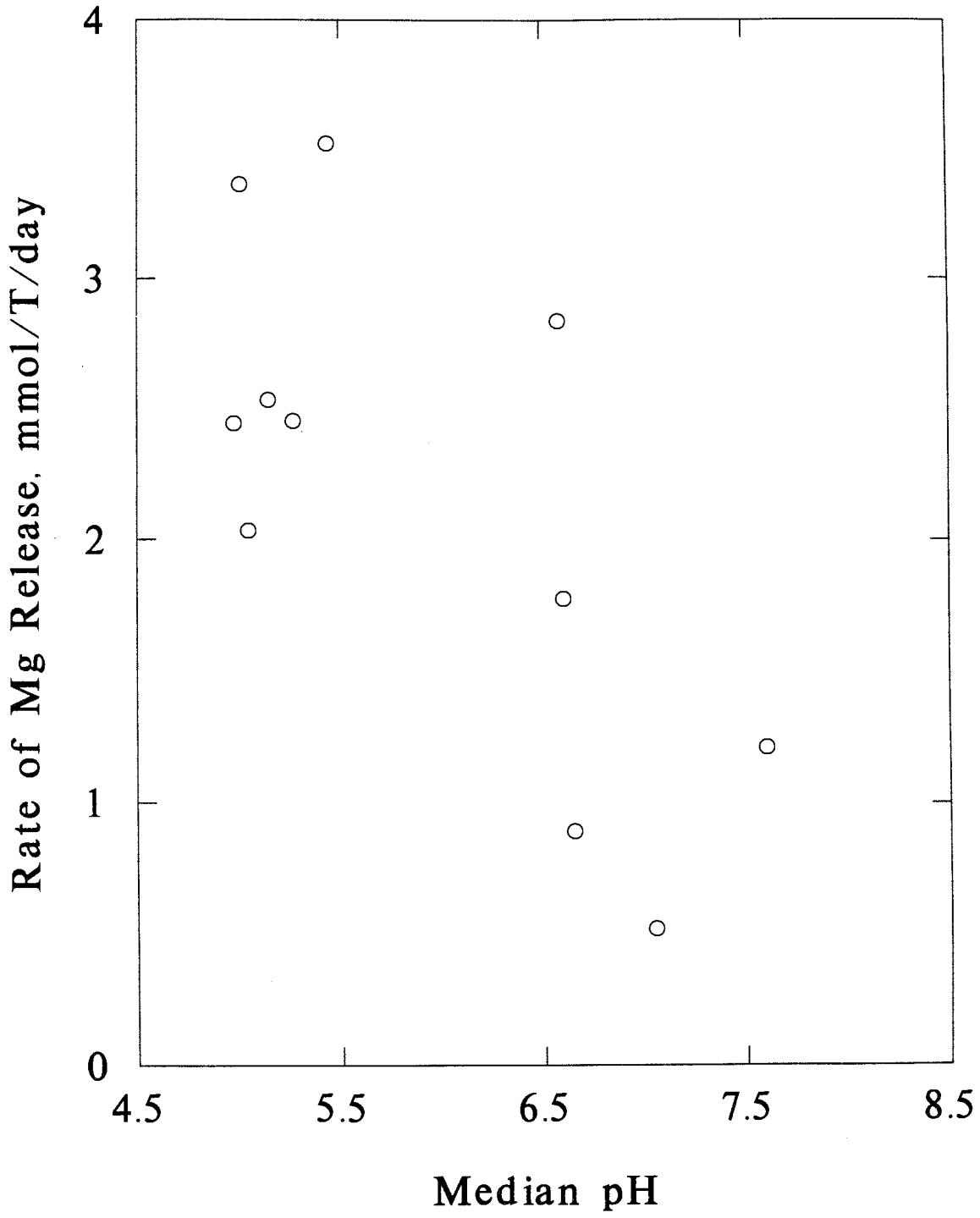

Figure A4.16. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL6 (0.79% S).

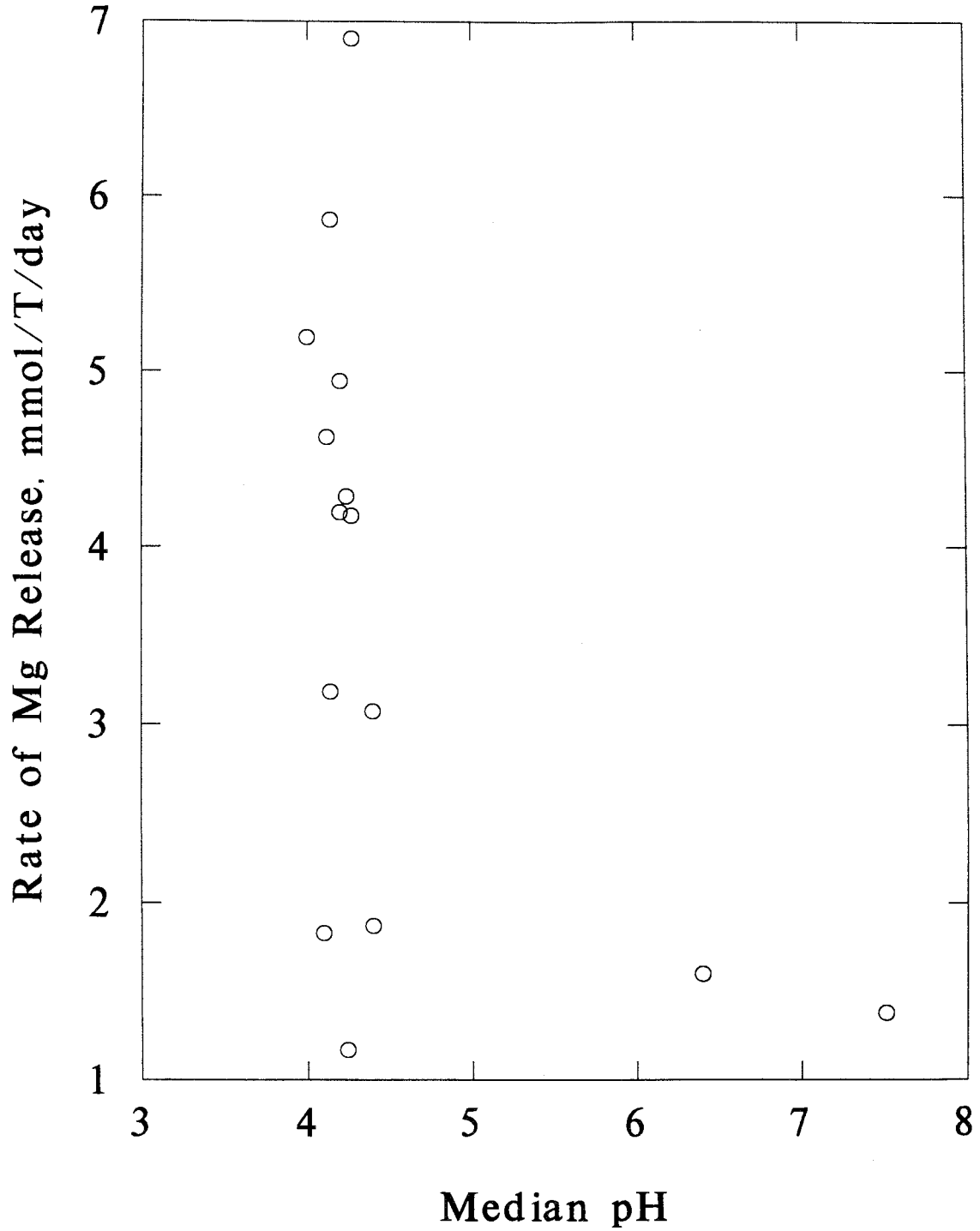

Figure A4.17. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL5 (1.41% S).

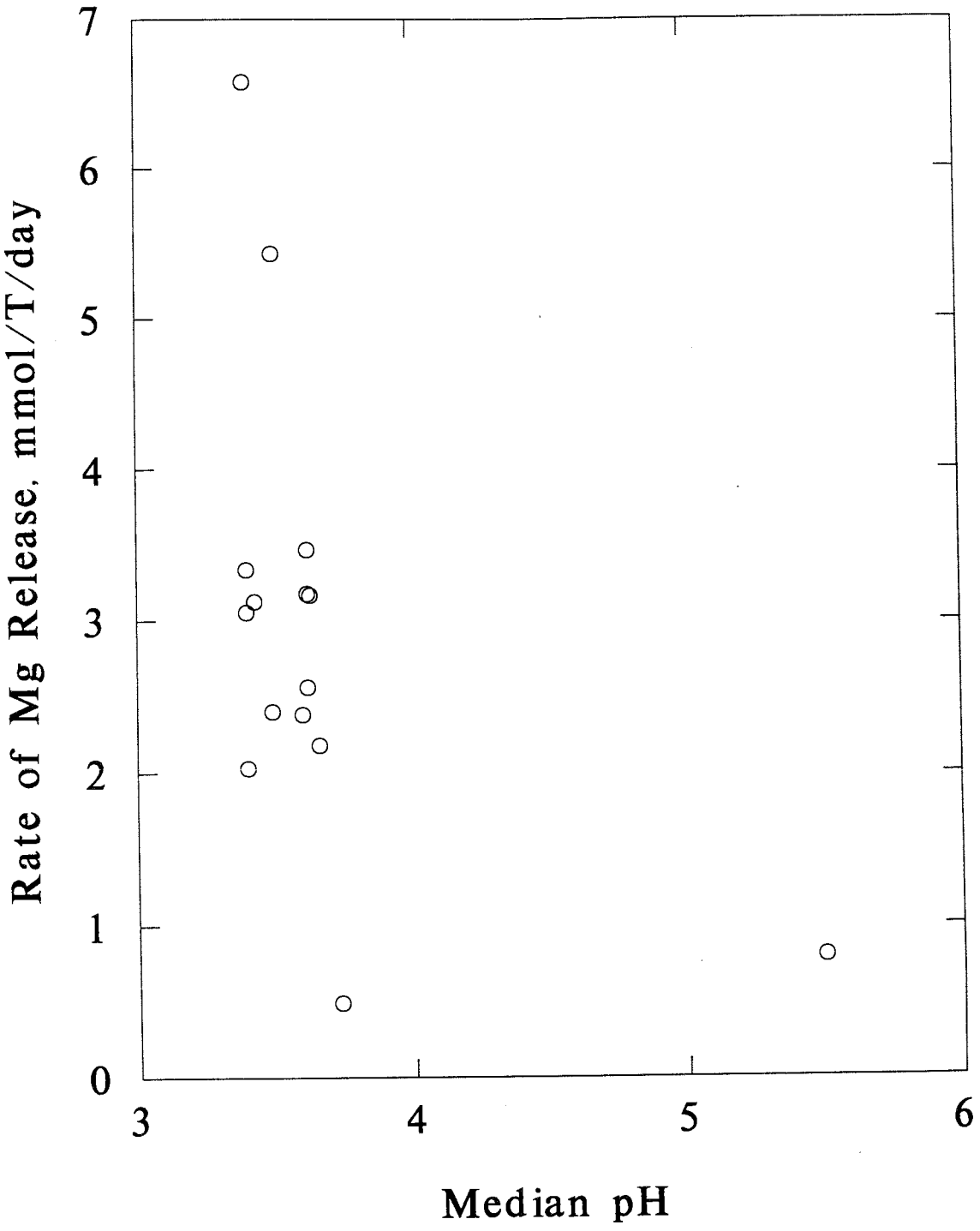

Figure A4.18. Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, and FL3.

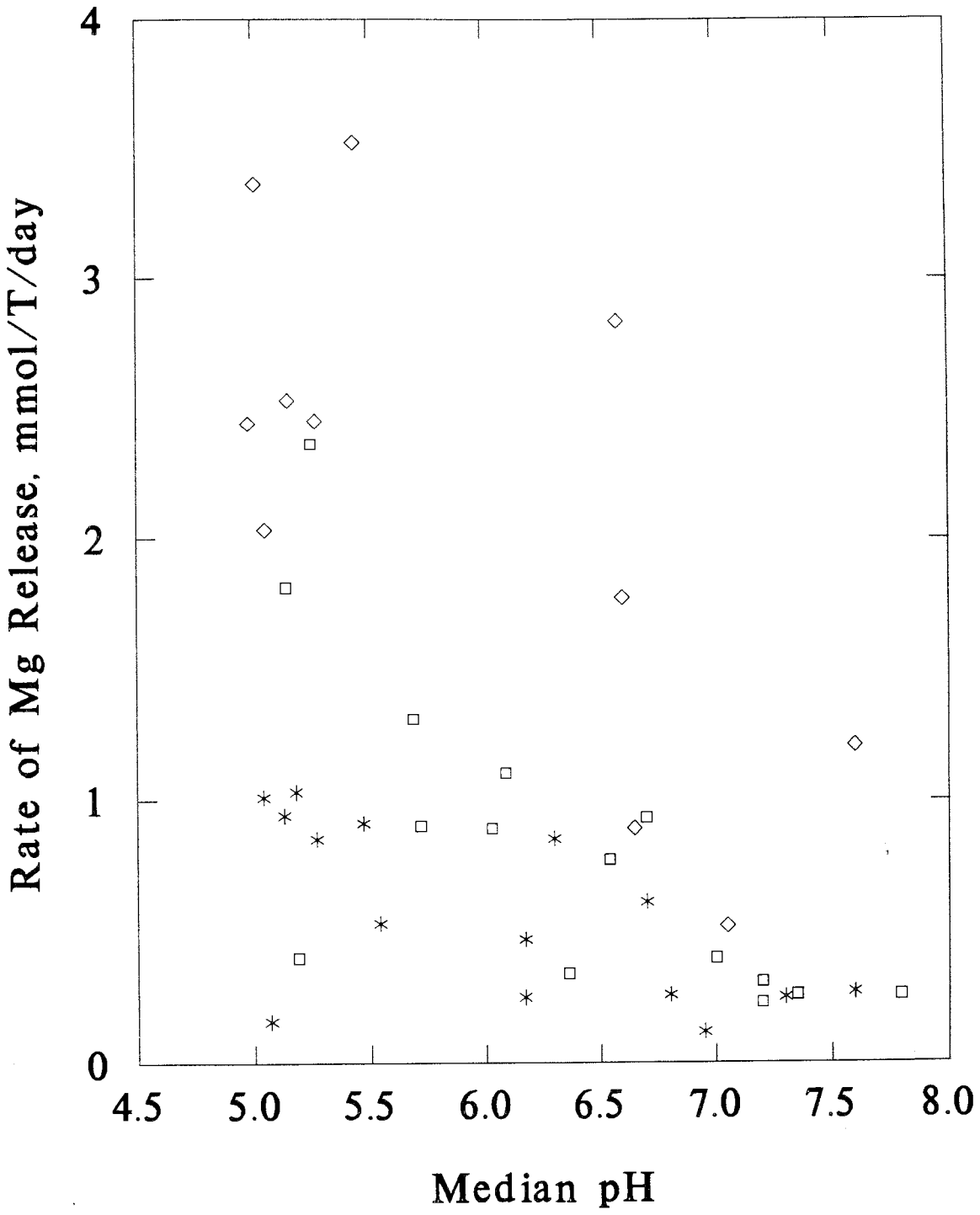

Figure A4.19. Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, FL3, FL6, and FL5.

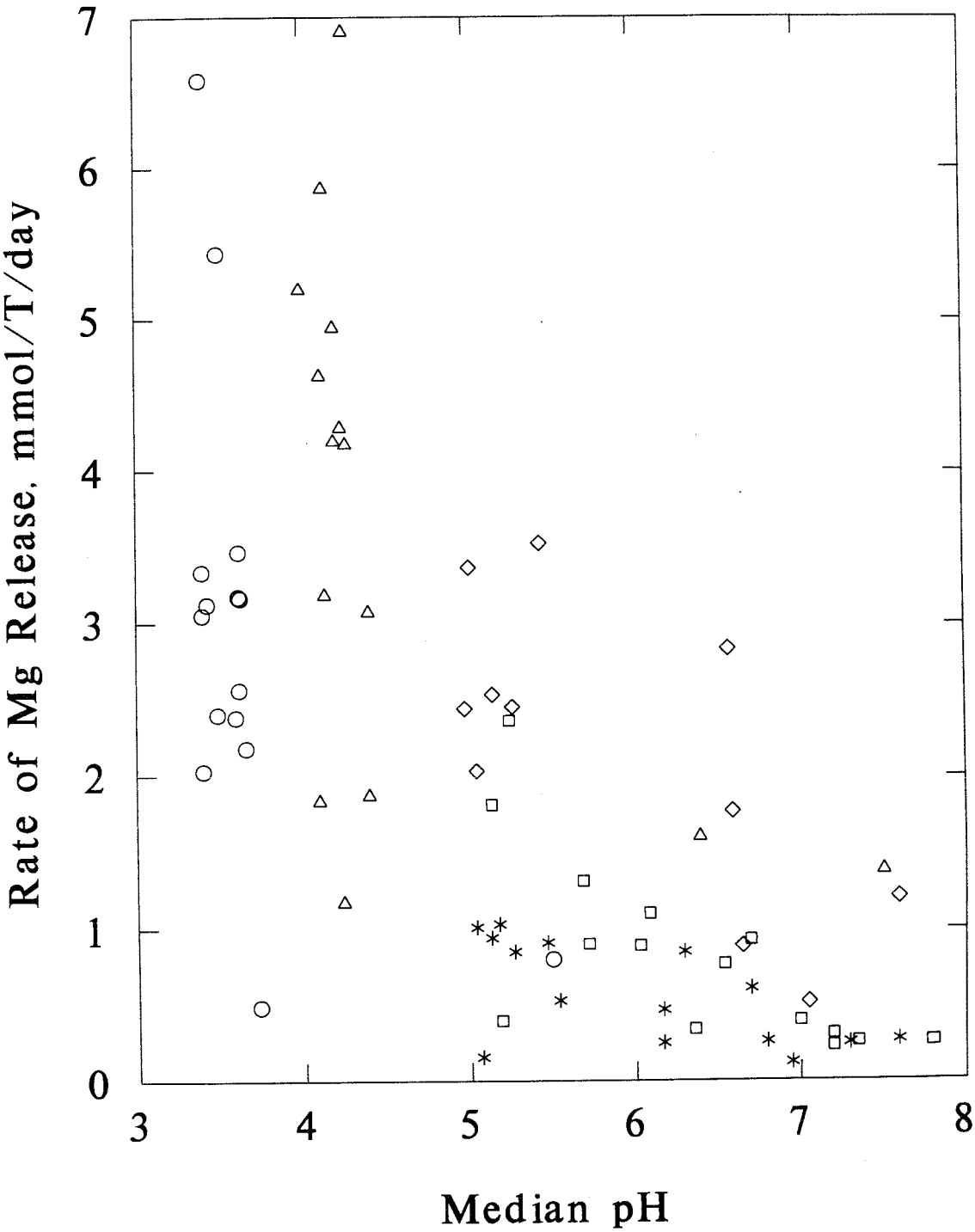

Table A4.1. Annual mass release rate vs. annual drainage volume regression statistics.

Sulfur %	Pile	Mass Release Rate (mmol/T/day) = a • volume (L) + b											
		Sulfate			Calcium			Magnesium			Calcium plus Magnesium		
		$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n
0.63	FL1	3.8	0.649	16	2.3	0.882	15	1.2	0.339	15	3.5	0.730	15
0.63	FL2	3.8	0.801	16	2.0	0.894	15	0.9	0.387	15	2.9	0.805	15
0.63	FL3	7.0	0.398	12	2.5	0.600	10[1]	2.9	0.245	11	5.4	0.403	11
0.79	FL6	6.0	0.492	16	2.1	0.851	15	2.9	0.359	15	5.0	0.599	15
1.41	FL5	9.7	0.386	16	3.5	0.937	15	2.0	0.134	15	5.5	0.564	15

[1] Regression excluded the 1978 calcium release rate, which was anomalously high.

APPENDIX 5

NEUTRALIZATION POTENTIAL

Appendix 5: Neutralization Potential

Table A5.1. Neutralization potentials as determined by drainage quality of Duluth Complex test piles.

Table A5.1. Neutralization potentials as determined by drainage quality of Duluth Complex test piles.

Sulfur %	Pile	Method	Neutralization Potential, mg $CaCO_3$/g rock					
			pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
0.63	FL1	Ca+Mg[1]	0.346	0.781	>1.059	>1.059	>1.059	>1.059
		SO_4[2]	0.422	0.923	>1.259	>1.259	>1.259	>1.259
		avg.	0.384	0.852	>1.159	>1.159	>1.159	>1.159
0.63	FL2	Ca+Mg	0.179	0.393	>0.642	>0.642	>0.642	>0.642
		SO_4	0.228	0.498	>0.783	>0.783	>0.783	>0.783
		avg.	0.204	0.446	>0.712	>0.712	>0.712	>0.712
0.63	FL3	Ca+Mg	0.442	1.254	>1.340	>1.340	>1.340	>1.340
		SO_4	0.537	1.599	>1.721	>1.721	>1.721	>1.721
		avg.	0.490	1.426	>1.530	>1.530	>1.530	>1.530
0.79	FL6	Ca+Mg	0.157	0.172	0.226	1.702	>2.074	>2.074
		SO_4	0.175	0.191	0.246	2.102	>2.607	>2.607
		avg.	0.166	0.182	0.236	1.902	>2.340	>2.340
1.41	FL5	Ca+Mg	0.044	0.044	0.096	0.130[3]	0.136[3]	>1.889
		SO_4	0.080	0.080	0.135	0.189[3]	0.201[3]	>3.627
		avg.	0.062	0.062	0.116	0.160[3]	0.169[3]	>2.758

1 Neutralization potential (NP) was calculated using the cumulative calcium and magnesium release (see methods section of report).

2 Neutralization potential (NP) was calculated using the cumulative sulfate release (see methods section of report).

3 Neutralization potential was calculated as of 1978, although higher pH values were observed later. Two pH values of 4.6 and 4.0 were observed in spring runoff in 1980 and 1989, respectively. Five pH values between 4.0 and 4.4 were observed in 1992.

Attachment 10

Lapakko K (1993) Field Dissolution of Test Piles of Duluth Complex Rock, Report to the US Bureau of Mines, Salt Lake City Research Center. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN.

FIELD DISSOLUTION OF TEST PILES OF DULUTH COMPLEX ROCK

Report to the US Bureau of Mines
Salt Lake City Research Center

Minnesota Department of Natural Resources
Division of Minerals

September 1993

FIELD DISSOLUTION OF

TEST PILES OF

DULUTH COMPLEX ROCK

Report to the US Bureau of Mines
Salt Lake City Research Center

Cooperative Agreement No. C0219003

Kim A. Lapakko

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Division of Minerals
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St. Paul, MN 55155-4045

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APPENDICES

1. Methods
2. Drainage Quality
3. Mass Release
4. Rates of Release
5. Neutralization Potential

INTRODUCTION

In 1974, AMAX Exploration Inc. reached an agreement with Bear Creek Mining Company and assumed their leases for properties located in T60N, R12W, about four miles southeast of Babbitt, Minnesota. The state owns the land and mineral rights on much of the property, and AMAX requested permission to construct an exploratory shaft. In 1974, very little information existed on the potential drainage problems associated with copper-nickel development in Minnesota. The state and AMAX agreed to commence field leaching tests on Duluth Complex rock.

In 1975 shaft development began, and in 1977 AMAX constructed six test stockpiles containing lean ore material. Water quality samples were collected periodically from each pile in 1977 and the data were compiled by AMAX (1978). The test pile studies were conducted from 1978 through the present by the Minnesota Department of Natural Resources (MNDNR), Division of Minerals. Kennecott Minerals Company assumed the leases in 1981 and terminated them in 1989.

This report presents data on the quantity and quality of drainage from the test piles. It is the third in a series of reports on dissolution of Duluth Complex rock. The two previous reports describe laboratory experiments on the dissolution of drill core and blast hole samples of Duluth Complex rock (Lapakko, 1993a; Lapakko, 1993b). The results from the laboratory and field studies will be used by the Salt Lake City Research Center to aid in development of a mathematical dissolution model and for comparison with laboratory data generated at the Center.

OBJECTIVES

As presented in Cooperative Agreement No. C0219003, the objectives of this report on the drainage quality from field test piles of Duluth Complex rock are to:

a) describe the variation of drainage quality with respect to time;
b) determine the chemical mass release of sulfate, calcium, and magnesium;
c) describe the variation of drainage quality and mass release with respect to flow;
d) describe the variation of drainage quality and mass release with respect to sulfur content;
e) determine the buffering capacities, or neutralization potentials, of the Duluth Complex rock samples examined; and
f) compare field results with those generated in the laboratory.

METHODS

Climate

Annual precipitation in the study area averages 72.1 cm (Hickok and Associates, 1977), and the average watershed runoff is 26.2 cm/yr (Siegel and Ericson, 1980). Temperatures are extreme, averaging -14°C in January and 19.1°C in July, with an annual mean of 3.6°C; the ground is covered with snow for an average of 140 days per year (Hickok and Associates, 1977).

Test Pile Construction

Materials

Six test stockpiles containing 820 to 1300 metric tons of low grade copper-nickel material were constructed in 1977 (figure 1). Stockpiles 1 to 4 were constructed of rock mined during the shaft sinking operation, while piles 5 and 6 contain material from underground drifts. The chemical compositional ranges for the piles are 0.30 to 0.35 percent Cu, 0.083 to 0.085 percent Ni, and 0.63 to 1.41 percent S (table 1). A grab sample was taken from test pile FL1 for more extensive chemical (table 2) and mineralogical analyses. The major sulfide minerals present in the gabbro are pyrrhotite, chalcopyrite, cubanite, and pentlandite, but over 90 percent of the rock is composed of silicate minerals. The major silicate minerals are plagioclase, pyroxenes, and olivine (table 3).

Approximately 40 percent of test pile FL4 was removed in 1982, and a representative sample, weighing over three tons, was collected for analysis of particle size distribution, specific surface area, chemistry, and mineralogy. Particle size distribution was determined by manual measurement, box screen, and a Sweco[1] separator. Boulders were measured with a tape measure and removed manually. The remaining material was raked through screens (0.9 by 1.2m) which were sloped about 20 percent, and had openings of 152, 76.2 and 25.4 mm. The material finer than 25.4 mm was then sieved with a Sweco Vibro-Energy Separator, with screen openings of 19.0, 12.7, 6.40, 2.00, and 1.41 mm. A representative sample of the less than 2.00 mm (10 mesh) material was then dry sieved on a Ro-Tap Sieve Shaker. Masses of 500 to 600 g were sieved for 20 minutes with screen openings of 0.50, 0.177, 0.149, 0.105, 0.074, and 0.053 mm. Finally, representative samples of each of the seven finest fractions were submitted to the Minnesota Department of Transportation (DOT) for a standard dry sieve/hydrometer analysis. Sieve openings of 0.84, 0.42, 0.250, 0.149, 0.105, and 0.074 mm were used. Hydrometer analysis was conducted on the less than 0.074 mm fraction to separate the less than 0.053 mm fraction. The particle size distribution data presented in table 4 are based on the unmodified coarse material separation plus the fine material dry-sieving as modified by the DOT results.

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[1] Reference to specific trade names does not imply endorsement by the MNDNR.

The sand, silt, and clay fractions were analyzed for specific surface area, chemistry, and mineralogy; the coarse fraction was excluded because its specific surface area is relatively small, and is therefore less influential in chemical reactions. The values determined represent both the Duluth Complex solids and precipitates which accumulated during leaching in the stockpile. The specific surface area of the fine fractions was roughly inversely proportional to the particle size diameter (table 5). The pH decreased and the sulfide and trace metal content increased with decreasing particle size of the gabbro material (table 5). The finer sand, silt, and clay fractions had S, Cu, and Ni contents above the 0.634 pct S, 0.348 pct Cu, and 0.083 pct Ni levels estimated for the FL4 stockpile as a whole.

X-ray diffraction (XRD) analyses were conducted on the 0.053 to 0.075 mm, the 0.075 to 0.105 mm, and the 0.105 to 0.149 mm size fractions of the rock. Approximately half of this material was plagioclase (with a Na:Ca ratio of about 2:3) and half a mixture of olivine (with a Fe:Mg ratio of 1:4 to 1:9), pyroxene, and biotite. Lesser amounts of magnetite, amphibole, chlorite, and possibly smectite and/or vermiculite were also identified. The phyllosilicate minerals (biotite, chlorite, smectite, vermiculite) were more common in the finer size fractions.

Reclamation Treatments

Three of the piles were left uncovered, as controls, and the remaining three piles were covered (May 1978) with 18 to 29 cm of soil obtained from a nearby borrow pit. Pile 2 was covered with fertile topsoil obtained from the top 30 cm of the borrow pit, while piles 3 and 5 were covered by infertile coarse sand. In 1980, an additional 30 cm of sandy till was added to pile 5.

In June of 1978, all covered piles were fertilized with 10-20-30 fertilizer (392 kg/ha), broadcast seeded, and mulched with straw (3.4 m/ha). Minnesota Highway #5, United States Forest Service wildlife mix, and native seed mixes were applied at rates of 34 and 67 kg/ha. In 1979, an additional plot containing jack pine, red pine, trembling aspen, and an upland willow was established on each of the covered piles. Additional seed was also applied to piles 3 and 5. When the sandy till was applied to pile 5 in 1980, the entire pile was reseeded with a mixture of alfalfa, U.S. Forest Service wildlife mix, and sweet clover at rates of 45, 67, and 113 kg/ha, respectively. A 13-12-12 fertilizer (730 kg/ha) was added to piles 2, 3, and 5 at this time. Additional details on vegetation, fertilization, as well as the application of lime-stabilized sludge to pile FL3 (1980, 1982) and a sodium-lauryl-based bactericide to FL5 (1982), are available in Eger and Lapakko (1985) and from the MNDNR.

In 1982 approximately 40 percent of test pile FL4 was removed for use in another study of stockpile reclamation techniques (Eger et al., 1984). Piles FL1, FL3, and FL6 were partially covered by plastic at the end of the 1989 field season, for the practical purpose of reducing the volume of drainage which the MNDNR was required to treat. The plastic covers were shredded during 1990 by exposure to the elements. A more robust Hypalon cover was placed onto FL3, thereby terminating flow from this pile at the end of the 1989 field season.

Water Sampling and Analysis

Volumes of input precipitation were measured with a standard rain gauge (Science Associates, Inc. NWS Spec. 450.230). Time and duration of rain events were obtained using a recording rain gauge (Science Associates, Inc., NWS Spec. 450.220) through 1984.

For drainage collection, each pile is underlain by an impervious Hypalon liner (30 mil) which is sloped toward a 15.2 cm perforated plastic pipe (figure 2). The drainage collected from the piles flows to a common sump. A limited number of water quality samples were collected in 1977, but no flow measurements were made. Since methods in this year deviated from those over the remainder of the study and the data were sparse, the results are not addressed in the text. During 1978, each pile was fitted with a cumulative flow meter, event recorder, and a flow-weighted-composite sampler (figure 3). For each pump discharge, a fixed volume of sample was placed into the compositing container. (The timing of flow was more precisely registered on a Rustrak event recorder (Model 292-4) from 1978 through 1984.)

The composite samples were analyzed weekly through 1990 and bi-weekly subsequently. Samples were filtered through 0.45-micron filters to remove suspended materials prior to analysis. Routine analyses included pH, alkalinity, specific conductance, sulfate, copper, nickel, cobalt, zinc, calcium, and magnesium, although calcium and magnesium were not analyzed in 1983. (Some samples were also analyzed for sodium, potassium, chloride, and dissolved organic carbon.) Specific conductance was analyzed using a Myron L conductivity meter. pH was determined using a Radiometer 29 pH meter from 1978 to 1988, and subsequently using an Orion SA 720 with a Ross combination electrode (8165). Sulfate was analyzed using either the barium sulfate turbidimetric technique (APHA et al., 1975) or occasionally, for low concentrations, a Technicon autoanalyzer. Metals were analyzed with a Perkin Elmer 603 atomic absorption spectrophotometer.

Calculations

From the weekly and bi-weekly analyses of composite samples, monthly-flow-weighted-mean concentrations were calculated. This concentration and the volume of drainage which occurred during the period were assigned to the date in the middle of the period. The mass of sulfate, calcium, and magnesium released during the period was calculated as the product of the flow-weighted-mean concentration and the drainage volume. Missing concentrations were estimated for the mass release calculation as the time-weighted mean of the previous and subsequent concentrations. Missing values occurring at the beginning (or end) of the year were estimated as the first (or last) measured concentration in that year.

Annual mass release was calculated as the sum of the mass release values for the individual periods for each year. Annual-flow-weighted-mean concentrations were calculated by dividing the annual mass release by the annual flow. The annual rates of release were calculated as the annual mass release divided by the product of the number of days of flow

and the mass of the pile (metric tons). Average rates of release for the entire period of record were calculated by dividing the total mass release by the number of days of flow.

Neutralization potentials (NP) were calculated to determine the amount of acid neutralized by the solids prior to the drainage pH decreasing and, for the duration of the period of record, remaining below pH values of 7, 6, 5, 4.5, 4, and 3.5. The acid neutralized was calculated by two methods: as the a) cumulative sulfate release (expressed as mg $CaCO_3$/g rock) and b) cumulative calcium and magnesium release (expressed as mg $CaCO_3$/g rock) prior to the drainage pH decreasing and remaining below the specified pH value. If the drainage pH from a given solid never decreased permanently below a specified value, the NP was reported as "greater than" the total sulfate release for the period of record. Additional detail on calculation of mass release and neutralization potential are presented in appendix 1.

RESULTS AND DISCUSSION

The generation of acidic drainage is the major water quality concern associated with mine wastes. Acid is produced as a result of the oxidation of iron sulfide minerals. The rate of this oxidation, and the attendant acid production, is proportional to the iron sulfide surface area available for reaction. The solids examined in this study were mined from a shaft and drifts in Duluth Complex rock. It is assumed the iron sulfide surface area is roughly proportional to the solid-phase-sulfur content. This assumption is predicated on the concept that the size distribution of rocks in the various piles is similar, and the grain size and degree of liberation of sulfide minerals does not vary greatly among the piles. The non-sulfide mineralogy of the rock was also assumed to be fairly uniform, although some exceptions most likely occurred. This suggests that the acid-neutralizing ability of the solids was somewhat uniform.

The test piles typically produced drainage between the middle of March and the middle of November, with an average flow season of 245 days. As was the case in laboratory studies, and consistent with the aforementioned assumptions, drainage pH decreased as the solid-phase sulfur content increased. The variation of drainage quality and chemical mass release with time, solid-phase sulfur content, and drainage volume is presented first. This presentation, in conjunction with the NP values determined for the test piles, provides the background necessary for comparison of the field results with those from the laboratory. The period of record for test pile 4 was only five years, as compared with 12 to 14 years for the remaining five piles. Although tabular and graphical data for this pile are presented, they are not discussed due to the short period of record.

Drainage Quality

Annual median pH and annual-flow-weighted-mean concentrations of sulfate, calcium, and magnesium are presented to describe the variation of drainage quality with respect to time,

solid-phase-sulfur content, and annual drainage volume. These values are presented in table 6. Detail on drainage quality variation over shorter periods is presented in appendix 2.

The pH of drainage from the test piles generally decreased over time, and decreased as the solid phase sulfur content increased (figure 4). At the beginning of 1978, the pH of drainage from all piles was between 7 and 8. For the three piles containing 0.63-percent sulfur, drainage pH decreased gradually from circumneutral to the range of 4.8 to 5.3. The pH of drainage from the piles containing 0.79 and 1.4 percent sulfur decreased and plateaued within a range of about 0.2 to 0.3 pH units. The pH of drainage from the 0.79-percent sulfur pile decreased until 1982 and subsequently oscillated in the typical range of 4.0 to 4.3. The pH of drainage from the 1.41-percent sulfur pile decreased to a lower level, from a median value of 5.5 in 1978 to a typical range of 3.4 to 3.6 from 1979 to 1991. The flow-weighted-median values actually increased from 1982 (3.40) to 1991 (3.60).

In general, annual-flow-weighted-mean sulfate concentrations for the individual piles increased during the first three to four years and then oscillated within a fairly constant range; increased as the solid-phase-sulfur content increased (figure 5); and were independent of flow (appendix 2). Some deviations from the general trends were observed. Most notably, sulfate concentrations in drainage from the 1.41-percent-sulfur pile increased to peak levels between 1981 and 1985, and then decreased. From 1987 to 1991, the flow-weighted-mean sulfate concentrations from this pile were comparable to, or lower than, those from the 0.79-percent-sulfur pile. Sulfate concentrations from the 0.79-percent sulfur pile increased over time (figure 5). However, these temporal variations from 1981 through 1991 were relatively small. In general the maximum annual-flow-weighted-mean sulfate concentrations were typically within a factor of two of the minimum value observed during this period.

Annual-flow-weighted-mean magnesium concentration variations over time were generally parallel to those observed for sulfate (figure 6), and were independent of flow volume (appendix 2). With respect to solid-phase-sulfur concentration, annual-flow-weighted-mean magnesium concentrations were lowest for the 0.63-percent-sulfur piles and in a similar range for the two piles of higher sulfur content. For the individual piles, annual-flow-weighted-mean magnesium concentrations tended to decrease as the annual median pH decreased.

Annual-flow-weighted-mean calcium concentrations were typically fairly constant, ranging from about 4 to 7 millimoles per liter (figure 7). As implied by the constant range, there was little dependence of annual-flow-weighted-mean calcium concentrations on time of dissolution, sulfur content, or flow. The values for the 1.41-percent-sulfur pile appeared to parallel the decreasing trend observed for sulfate concentrations from 1985 to 1991. As a result of the temporal parallelism of magnesium and sulfate concentrations, the sum of calcium and magnesium concentrations tended to parallel sulfate concentrations over time (figure 8).

Annual Rates of Release

The annual rate of sulfate release from each pile oscillated within a fairly constant range (tables 7-12). In general, annual sulfate release increased with the solid-phase-sulfur content of the pile, although the rates for the 0.79-percent-sulfur pile were comparable to or exceeded those for the 1.41-percent-sulfur pile from 1984 to 1991 (figure 9). Sulfate release also increased with annual flow (appendix 4), and temporal variations in the annual rate were largely due to variations in annual flow. A graphic example of the flow dependence is the low sulfate-release rate from the 0.79-percent-sulfur pile in 1990, during which the pile was partially covered with plastic. The drainage volume and sulfate mass release were consequently low.

Variations in magnesium release followed similar trends. Similar to the sulfate-release rates from 1984 through 1991, magnesium release from the 0.79-percent-sulfur pile was usually higher than that from the 1.41-percent-sulfur pile (figure 10). In contrast, the annual rates of calcium release typically fell into the range of one to three moles per ton per day (figure 11). They did not vary with time or solid-phase-sulfur content, but did increase linearly with annual flow (appendix 4). The average rates of sulfate, calcium, and magnesium release for the period of record are presented in table 13.

Neutralization Potential

The NP values calculated based on the sum of calcium and magnesium release ranged from 55 to 90 percent of those calculated based on sulfate release, which was deemed excessive. The combined copper and nickel sulfide content comprised 15 to 35 percent of the total sulfide mineral content in the test piles, a considerably higher fraction than for the laboratory samples. It was concluded that a significant amount of the sulfate release from the test piles was due to oxidation of trace metal sulfides (as opposed to iron sulfides). Since this oxidation does not necessarily lead to acid production, the sulfate release over-estimates acid production. Consequently, the neutralization potential data presented are based on the sum of calcium and magnesium release.

The pH of drainages from the 0.63-percent rock did not decrease permanently below pH 5.0, although the pH of drainage from pile 3 appeared to be near this point at the end of 1989. The approximate ranges of NP values determined for endpoint pH values of 7.0 (NP_{pH7}) and 6.0 (NP_{pH6})were 0.2 to 0.4 mg $CaCO_3$/g rock and 0.4 to 1.3 mg $CaCO_3$/g rock, respectively (table 14). The 1.3 mg $CaCO_3$/g rock value determined for pile 3 is assumed to be near the NP_{pH5} for this pile.

The lowest NP endpoint pH values for the piles with sulfur contents of 0.79 and 1.41 were pH 4.5 and pH 4.0, respectively. The NP_{pH7} and NP_{pH6} values for these piles were lower than those from the lower-sulfur rock (table 14). Furthermore, the NP values for the 1.41-percent rock were lower than those for the 0.79-percent rock. This trend of decreasing NP with increasing sulfur content was also observed with laboratory samples.

Comparison of Field and Laboratory Data

Certain similarities were apparent between the field data and laboratory data (table 15).

1) Drainage pH generally decreased, or decreased and plateaued, as the time of dissolution increased. A subtle variation on this trend was a possible slight increase in the pH of drainage from the 1.41-percent-sulfur pile from 1982 to 1991 (median annual pH values of 3.40 and 3.60, respectively).

2) In both laboratory and field drainage pH values decreased as solid-phase-sulfur content increased.

3) The rates of sulfate release were fairly constant over time. The ratio of maximum-sulfate-release rate to minimum-sulfate-release rate for a laboratory sample was typically less than 3:1. This ratio was observed to increase if drainage pH decreased below about 4. The corresponding ratios for the test piles ranged from 2.7:1 to 3.9:1.

4) In both laboratory and field tests the rate of sulfate release increased with solid-phase-sulfur content. Sulfate release rates from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

5) Rates of calcium release were relatively constant over time, although it must be noted that laboratory data on calcium release are limited.

6) Rates of calcium release were relatively constant with respect to sulfur content. The rates of calcium release from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

7) Rates of magnesium release increased as time increased, although it must be noted that laboratory data on magnesium release are limited.

8) Rates of magnesium release increased as solid-phase-sulfur content increased. The rates of magnesium release from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

9) Neutralization potential values were low and decreased with increasing sulfur content.

Field and laboratory data can also be quantitatively compared based on drainage pH, neutralization potentials, and rates of release. The minimum drainage pH values observed for the 0.63-percent-sulfur rock in the field ranged from 4.71 to 4.96, and were reasonably consistent with the laboratory relationship between drainage pH and solid-phase-sulfur content. However, the minimum pH of drainages from the test piles with sulfur contents of 0.79 and 1.41 percent sulfur were about one unit lower than expected based on laboratory

data (figure 12). (The minimum pH values observed were typically isolated excursions, and recurrent low pH values were typically 0.1 to 0.2 units higher.)

The discrepancy between laboratory and field drainage pH data suggests that the laboratory tests either inhibited the rate of acid production or enhanced the rate of acid neutralization relative to the field tests. Drainage pH in the field may have been lower due to a higher iron sulfide surface area than indicated by the sulfur content alone. Twelve percent of the rock in the piles was smaller than two millimeters in diameter (table 4). The specific surface areas of this material were in the range of 0.6 to 4.7 m^2/g (table 5). This was considerably higher than the 0.061 to 0.239 m^2/g and 0.020 to 0.039 m^2/g ranges estimated for the gravel and boulder/cobble fractions. The sulfur content of the smaller size fractions ranged from 0.67 to 1.94 percent (table 5), as compared to the bulk sulfur content of 0.63. Therefore, the fraction of the total available surface area contributed by the small particles is quite high relative to the larger particles. Due to their higher sulfur content, a greater fraction of the surface area is composed of iron sulfides. Thus, the fraction of the rock comprised of iron sulfide was most likely higher than indicated by the sulfur content of 0.63 percent. In contrast, the solids in the laboratory study were in the size range $0.053 <$ particle diameter ≤ 0.149 mm, and the potential of preferentially concentrating sulfides in a smaller size fraction was limited.

The acid neutralization may have been enhanced in the laboratory due to the consistently small particle size. As is the case with sulfide mineral oxidation, the dissolution of silicate minerals is a surface reaction. If the specific surface area of silicate minerals present in the laboratory samples exceeded that in the test piles, the laboratory solids would be expected to neutralize acid more efficiently.

The neutralization potentials of the test piles were lower than those for laboratory samples, which is consistent with the lower drainage pH levels observed in the field. The NP values for the 0.63-percent-sulfur rock were roughly 25 to 50 percent of those observed for laboratory samples of similar sulfur content (table 16). Although the NP_{pH6} and NP_{pH7} values from pile 3 were higher than this range, the NP_{pH5} value was 46 percent of the corresponding value for laboratory samples.

The field NP values for the 0.79-percent-sulfur rock were 13 to 54 percent of the corresponding values for laboratory samples of similar sulfur content. The ratios of field to laboratory NP were lowest for the 1.41-percent-sulfur rock, typically five percent or less than those determined in the laboratory.

The low field NP values may have been influenced by factors similar to those discussed regarding drainage pH. First, preferential flow in the field piles would reduce the amount of silicate minerals which could contribute to acid neutralization. For example, if flow through the pile contacted only half the silicate minerals, the observed neutralization potential of the rock would be half the actual value for the entire pile. Second, if the specific surface area of silicate minerals present in the laboratory samples exceeded that in

the test piles, the laboratory solids would be expected to neutralize acid more efficiently. Third, the ratio of available reactive surface area of sulfide minerals to that of silicate minerals present in the test pile rock may have been greater than that in the laboratory. This would be equivalent to increasing the sulfur content of rock in the laboratory. As was observed in the laboratory, neutralization potential tended to decrease as sulfur content increased.

For the entire period of record, average rates of sulfate release ranged from 2.0 to 10 millimoles per metric ton per day, and increased as sulfur content and drainage volume increased. The average rates of calcium release ranged from 2.1 to 2.4 millimoles per ton per day and were fairly constant with respect to sulfur content. Average rates of magnesium release ranged from 0.55 to 3.7 millimoles per ton per day and were lowest for the low-sulfur solids and highest for the solids containing 0.79-percent sulfur.

The rates of sulfate, calcium, and magnesium release observed for the test piles were compared to laboratory rates for the Duluth Complex blast hole samples of similar sulfur content. The laboratory rates were from the last 15 to 36 weeks of the period of record. The ratios of field rates to laboratory rates, or retardation factors, ranged from 0.053 to 0.46, with most values in the range of about 0.1 to 0.36 (table 17). This indicates that the laboratory rates were roughly three to ten times those in the field.

Several factors most likely contributed to the slower rates observed in the field. First, the rates of sulfide and silicate mineral dissolution (per unit mass rock) may be slower than those observed in the laboratory. As discussed previously with respect to drainage pH, the surface area of sulfide and silicate minerals per unit mass rock in the field most likely differs from that in the laboratory. This would affect the rates of sulfide oxidation and silicate mineral dissolution, since both are surface reactions. The reaction environment in the field, where temperatures are colder, may also retard reaction rates. Furthermore, potential lower levels of oxygen within the test piles may limit the rate of sulfide mineral oxidation.

Limited transport of reaction products in the field also may have reduced rates of release. Hydrologic variables which were not influential in the laboratory could have contributed to this transport limitation. The volume of rinse water per unit mass solids yielded approximately 300 L per ton rock per day of drainage in the laboratory, roughly three orders of magnitude greater than the yields of 0.27 to 0.38 L per ton rock per day in the field. The relatively low flow in the field may have limited transport. This is supported by the observation that release rates in the field increased with flow. Furthermore, the flow of the laboratory rinse water through the solids was fairly uniform, whereas preferential flow was probable in the field setting. Preferential flow would limit the transport of reaction products from some rock surfaces. This would essentially reduce the amount of rock in the test piles which was actually contributing to the observed chemical release.

The degree of transport in the field may also be limited by chemical precipitation of dissolution reaction products. In particular, the formation of gypsum might limit the rates

of sulfate and calcium release. Gypsum formation was not a problem in the laboratory since the high rinse water to solid ratio produced relatively low concentrations of sulfate and calcium in the drainages.

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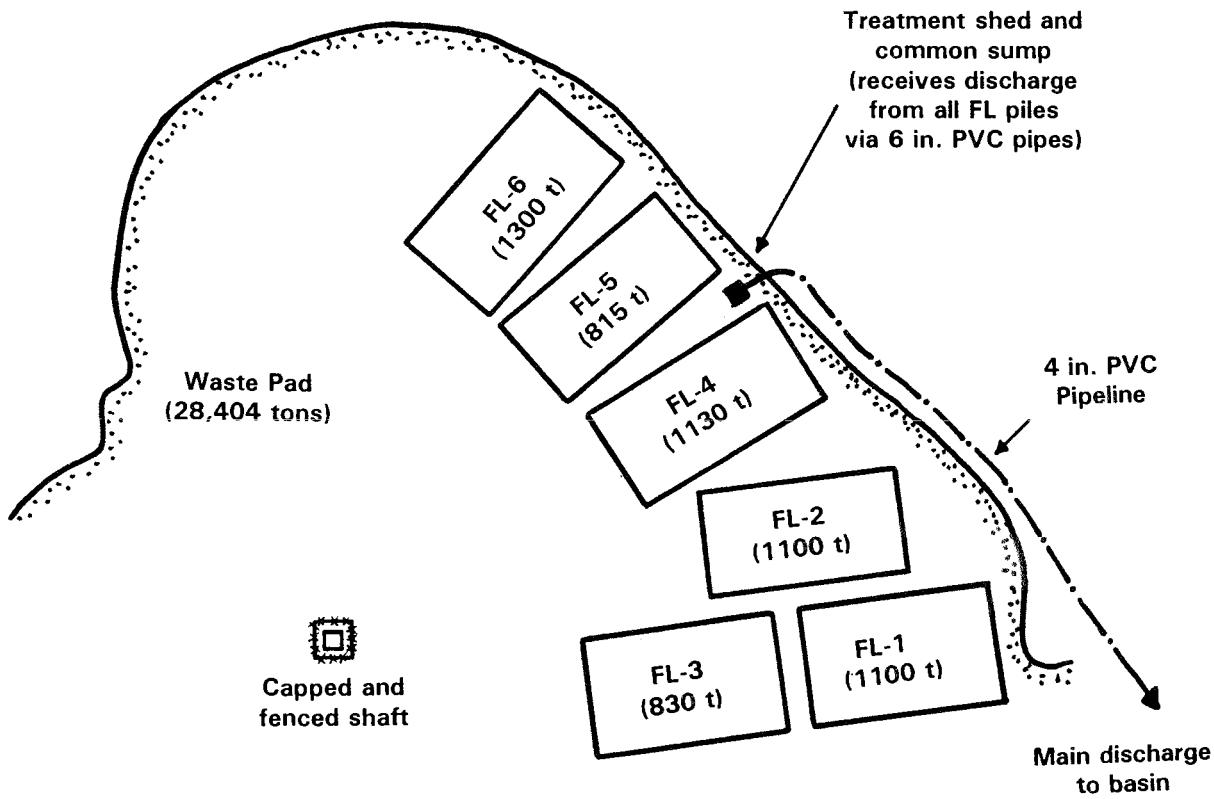

Figure 1. Test piles and drainage collection system layout.

Forty percent of FL-4 was removed in 1982 (leaving approximately 680 of the original 1130 t of material in the pile), thereby terminating the period of record. FL-3 was covered with a synthetic liner at the end of the 1989 field season, thereby terminating the periof of record. Additional information on the test piles is presented in table 1.

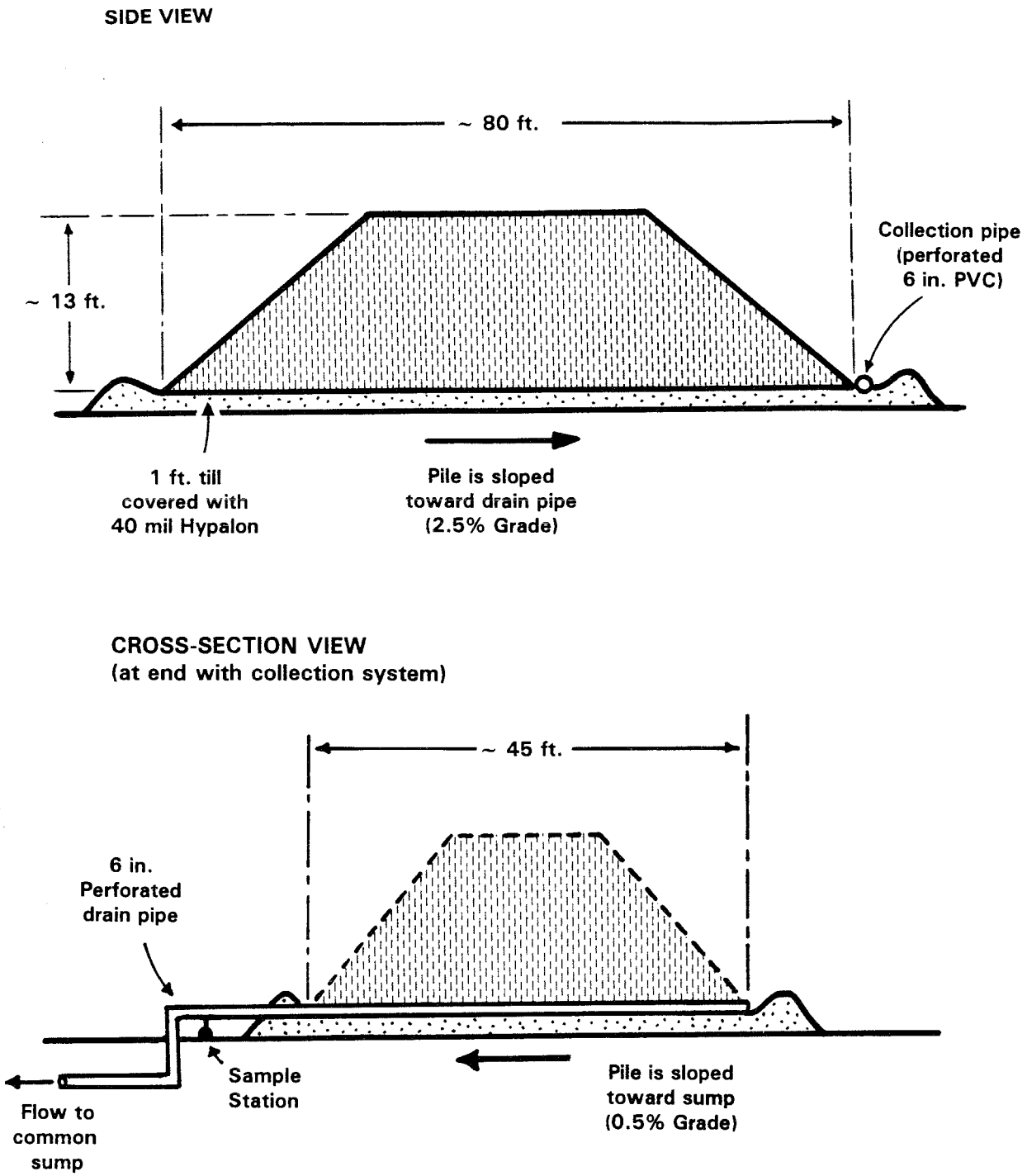

Figure 2. Leachate collection system beneath test piles (schematic; not to scale).

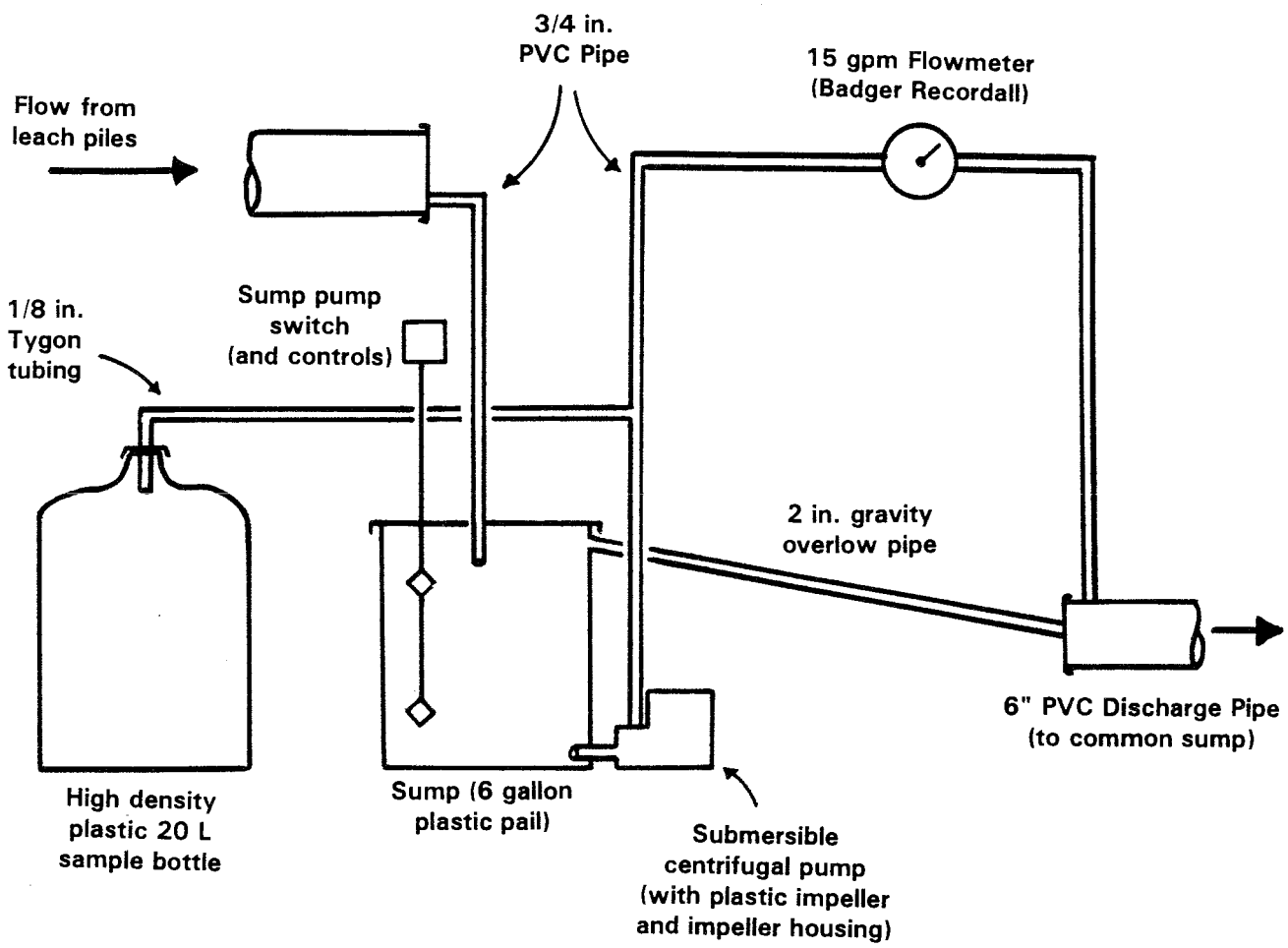

Figure 3. Flow-proportional sampler.

Figure 4. Annual median pH over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Drainage pH decreased over time and as solid-phase sulfur content increased.

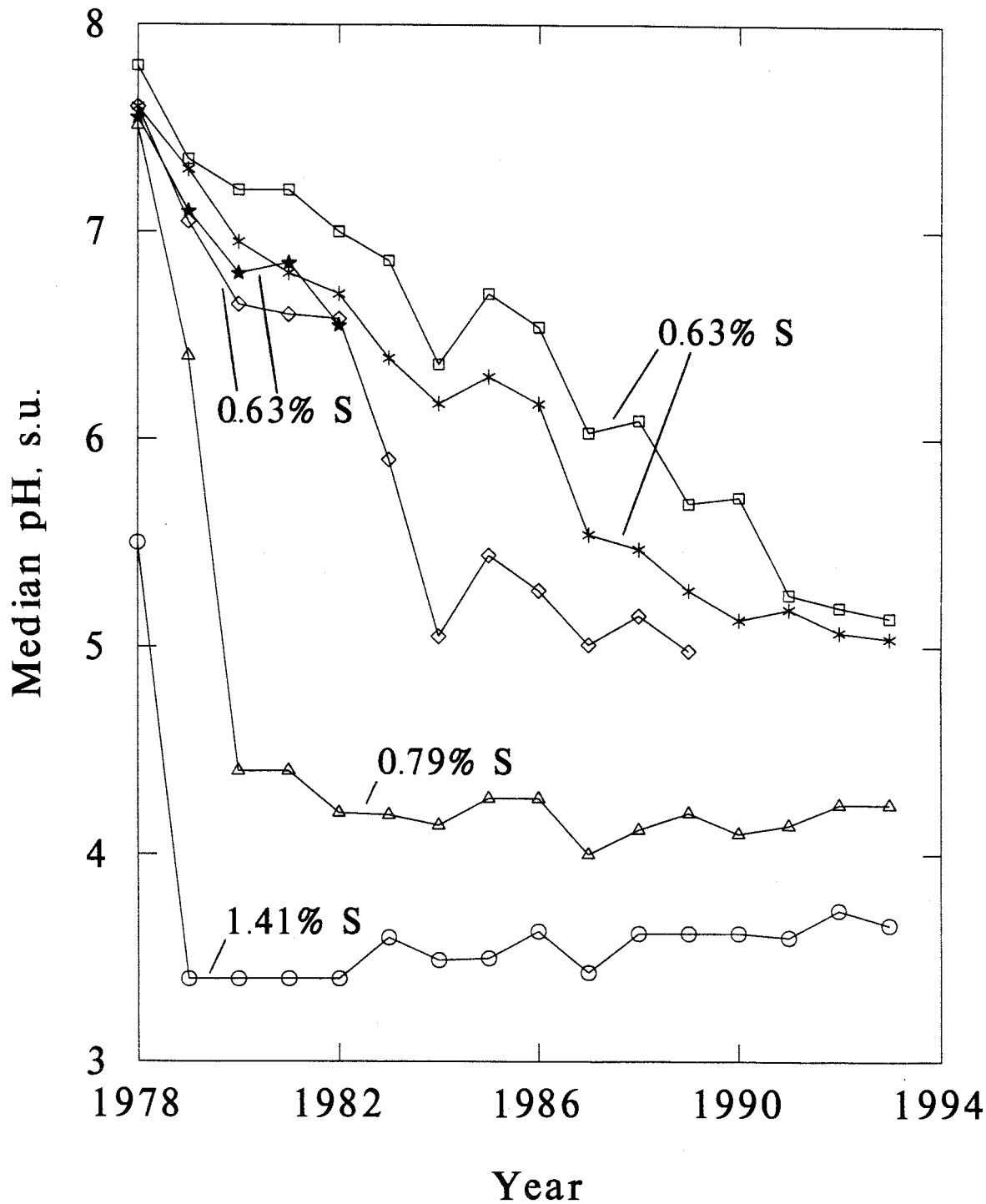

Figure 5. Annual-flow-weighted-mean sulfate concentration over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Sulfate concentrations generally increased with solid-phase sulfur content and were within a constant range over time.

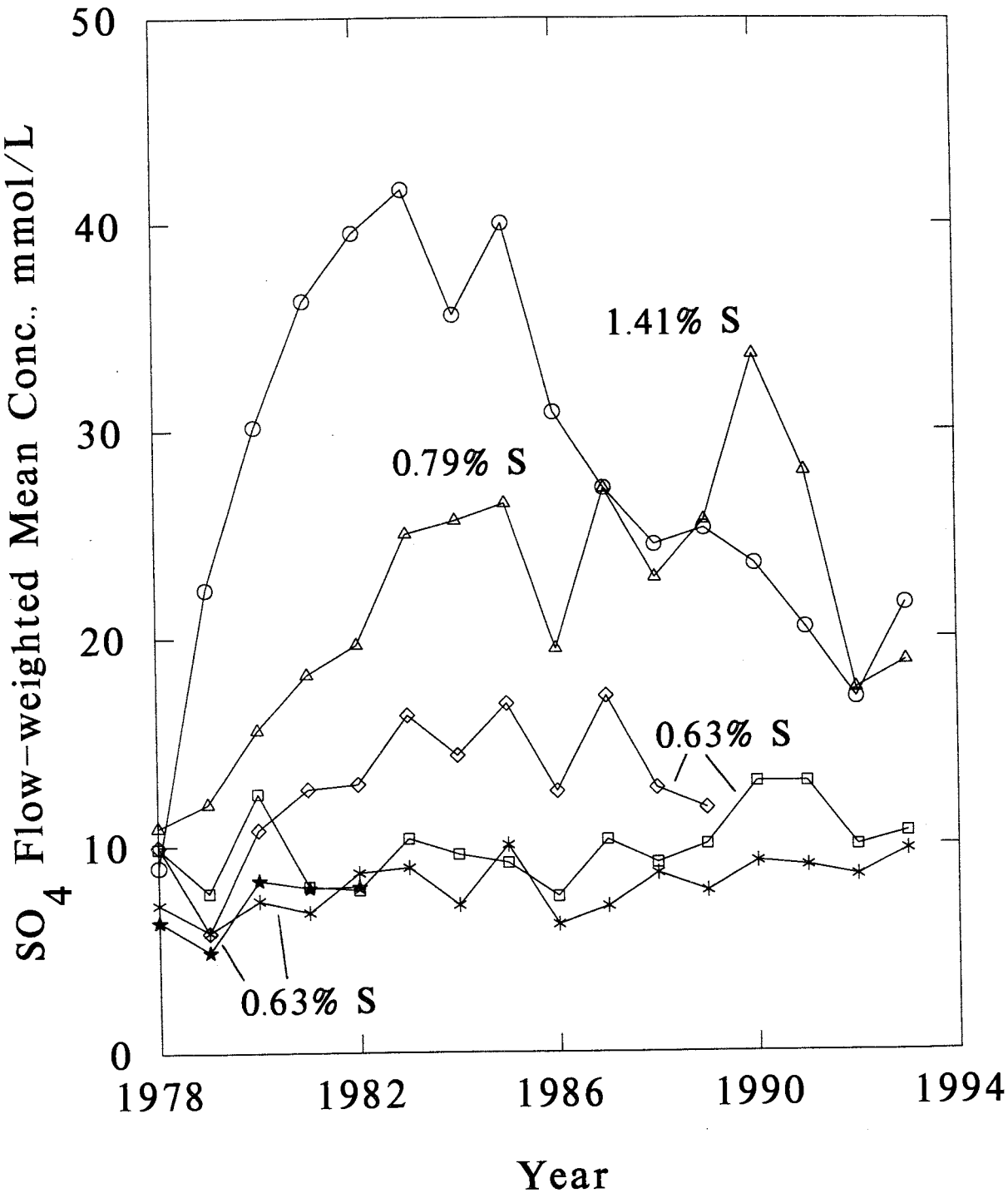

Figure 6. Annual-flow-weighted-mean magnesium concentration for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Magnesium concentrations generally increased with solid-phase sulfur content and time.

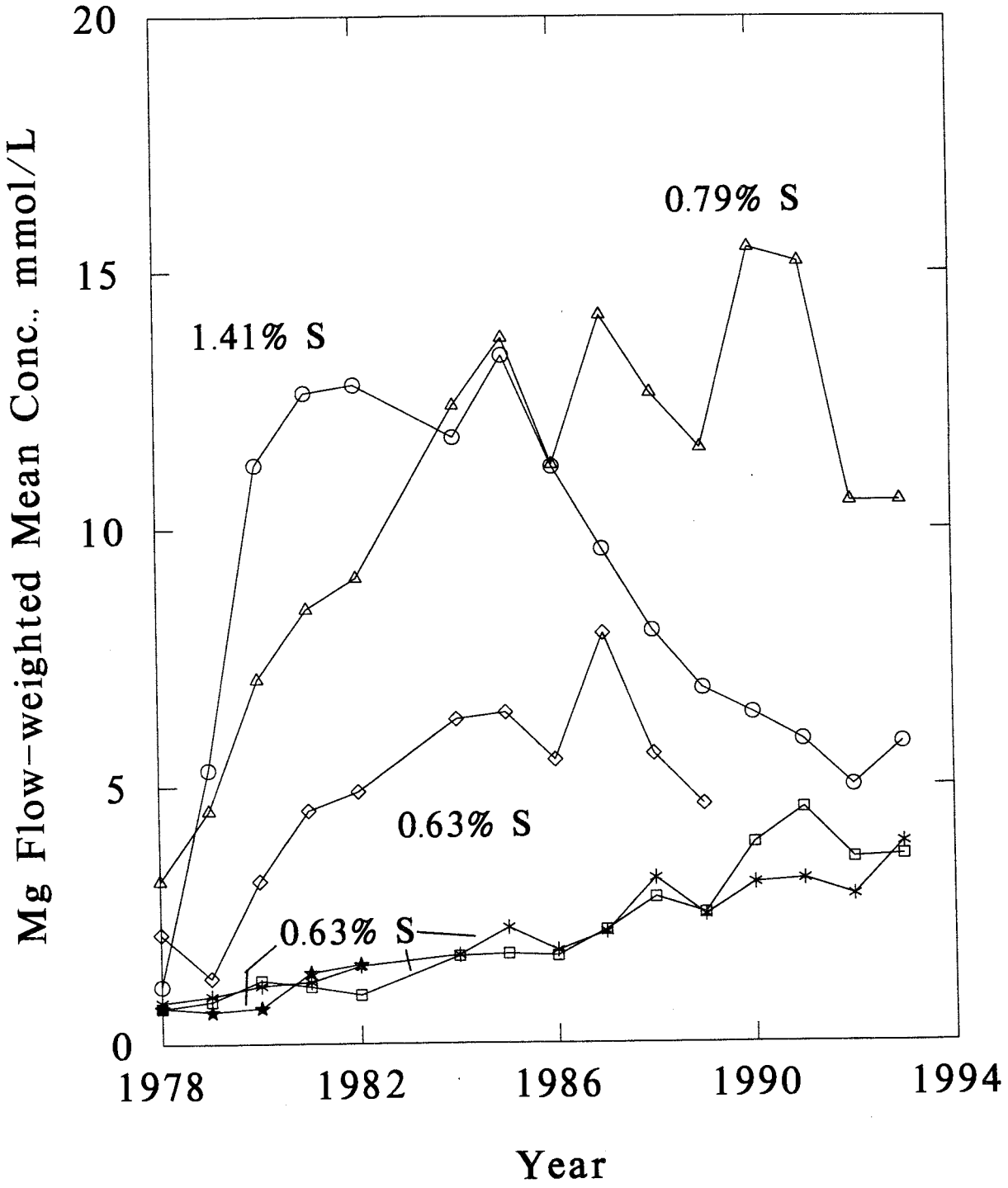

Figure 7. Annual-flow-weighted-mean calcium concentration for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (▵), and FL5 (○).

Calcium concentrations were fairly constant with respect to solid-phase sulfur content and time.

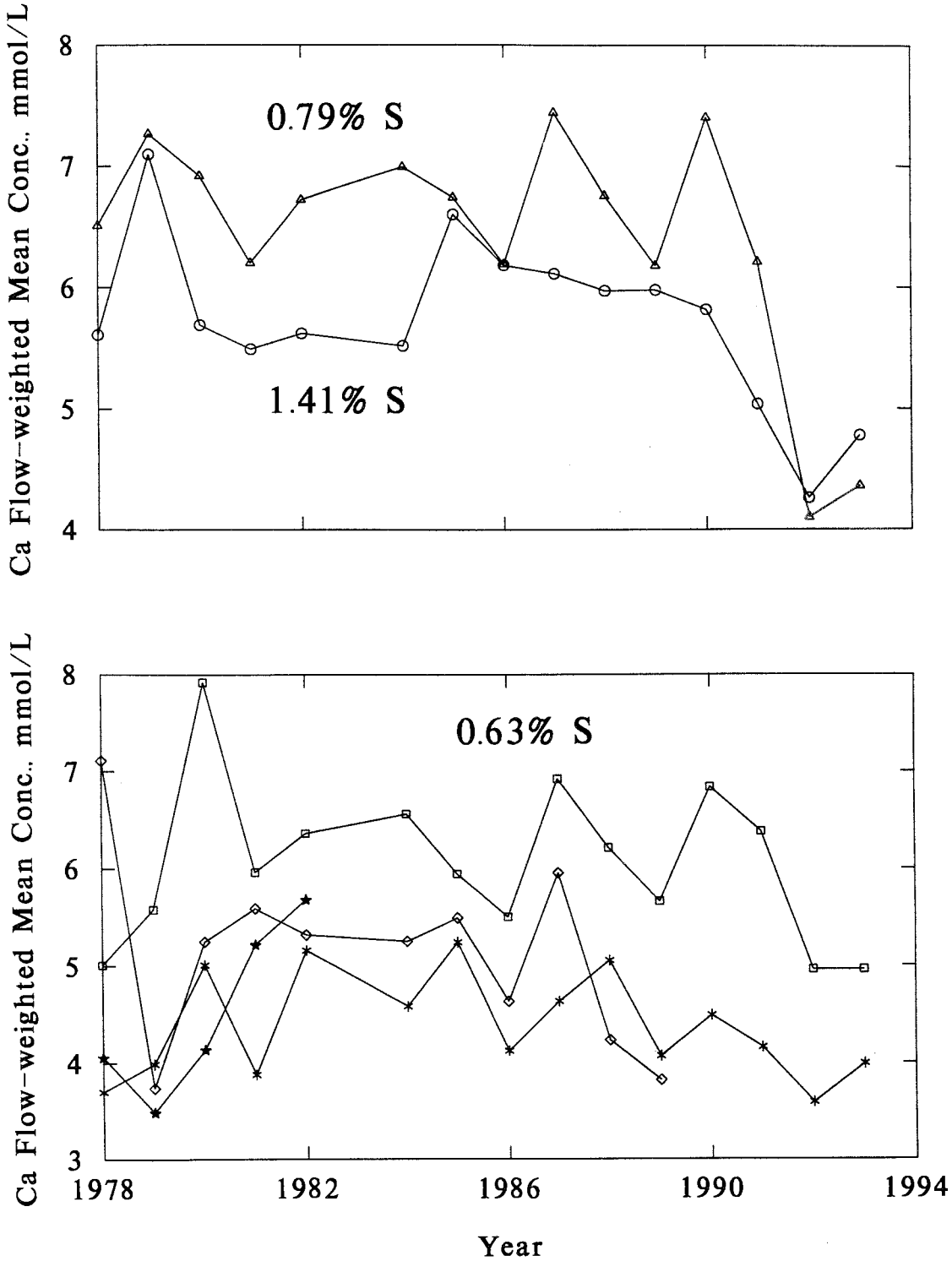

Figure 8. Annual-flow-weighted-mean calcium plus magnesium concentration for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

The sum of calcium and magnesium concentrations generally increased with solid-phase sulfur content and increased slightly over time.

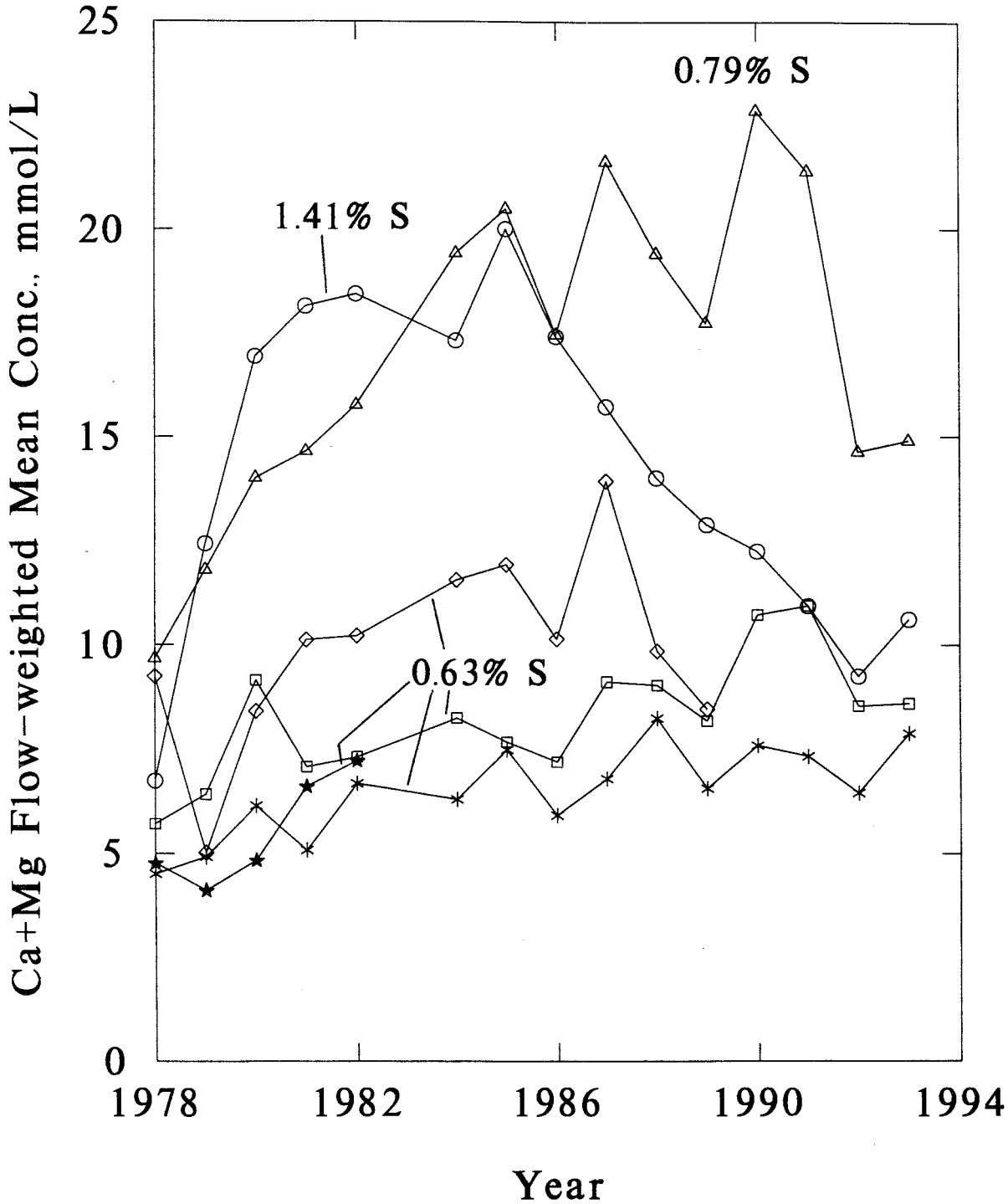

Figure 9. Annual sulfate release rate over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Sulfate release rates generally increased with solid-phase sulfur content and were within a constant range over time.

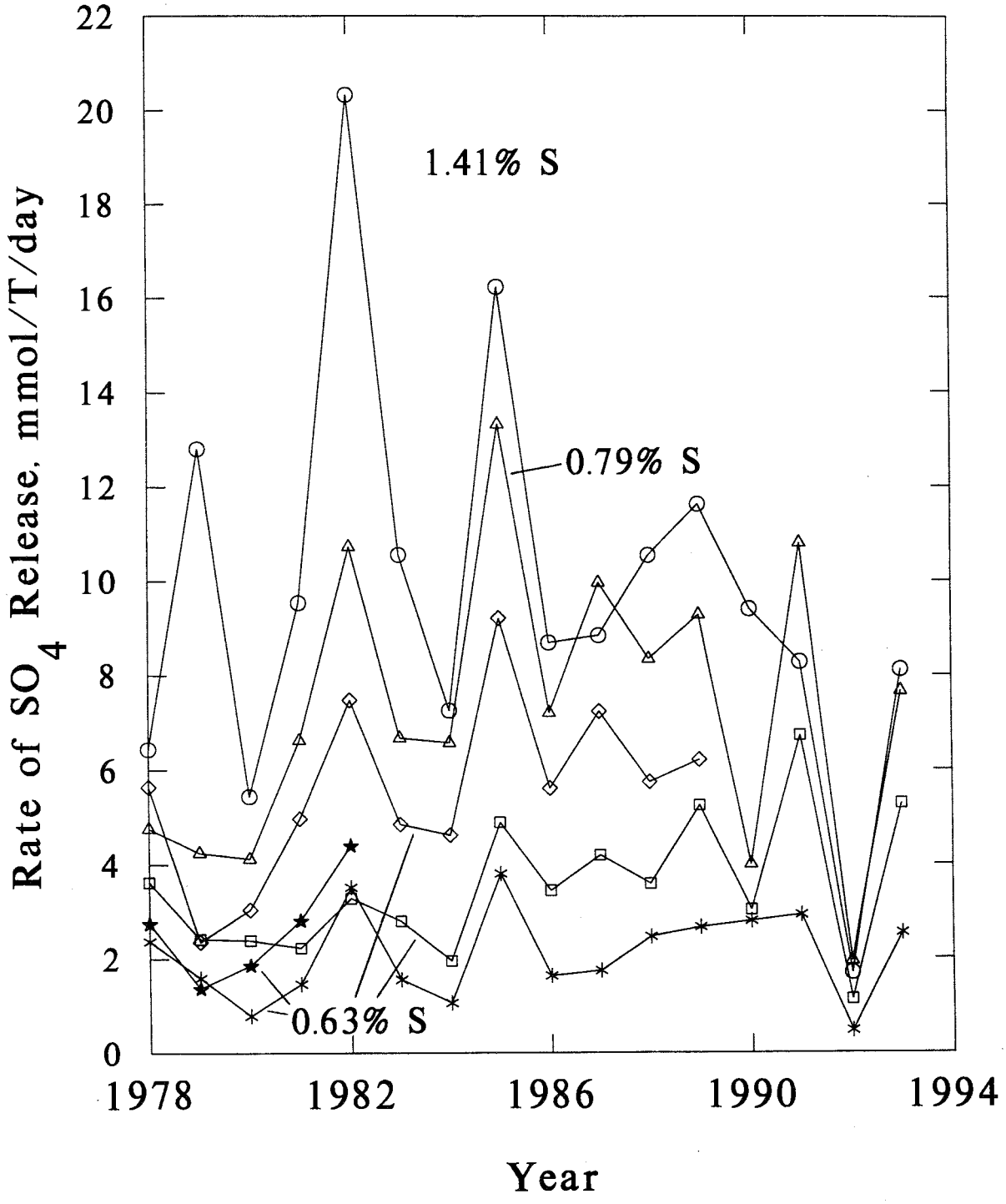

Figure 10. Annual magnesium release rate over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Magnesium release rates increase with solid-phase sulfur content and increased slightly over time.

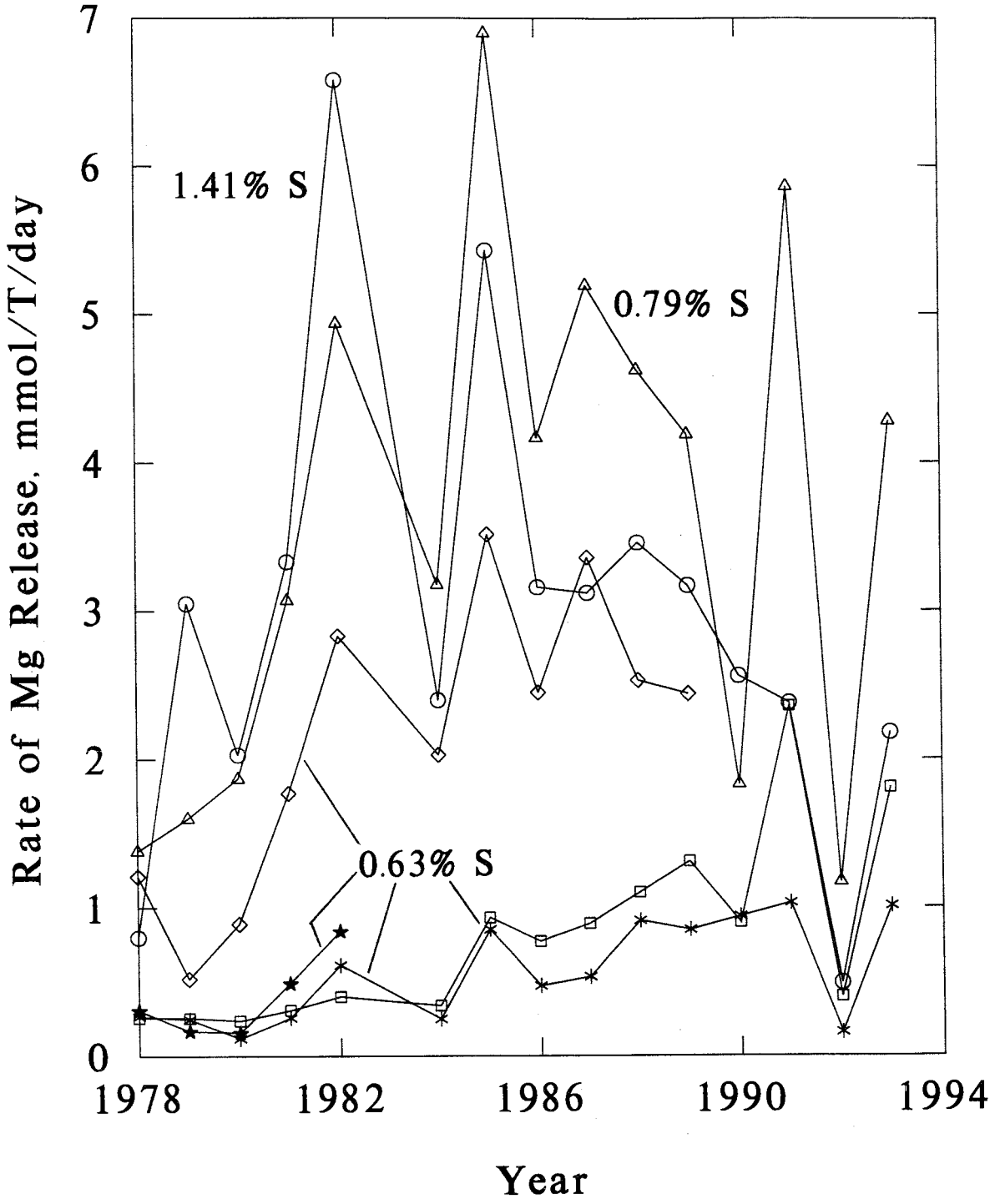

Figure 11. Annual calcium release rate over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

Calcium release rates were fairly constant with respect to solid-phase sulfur content and time.

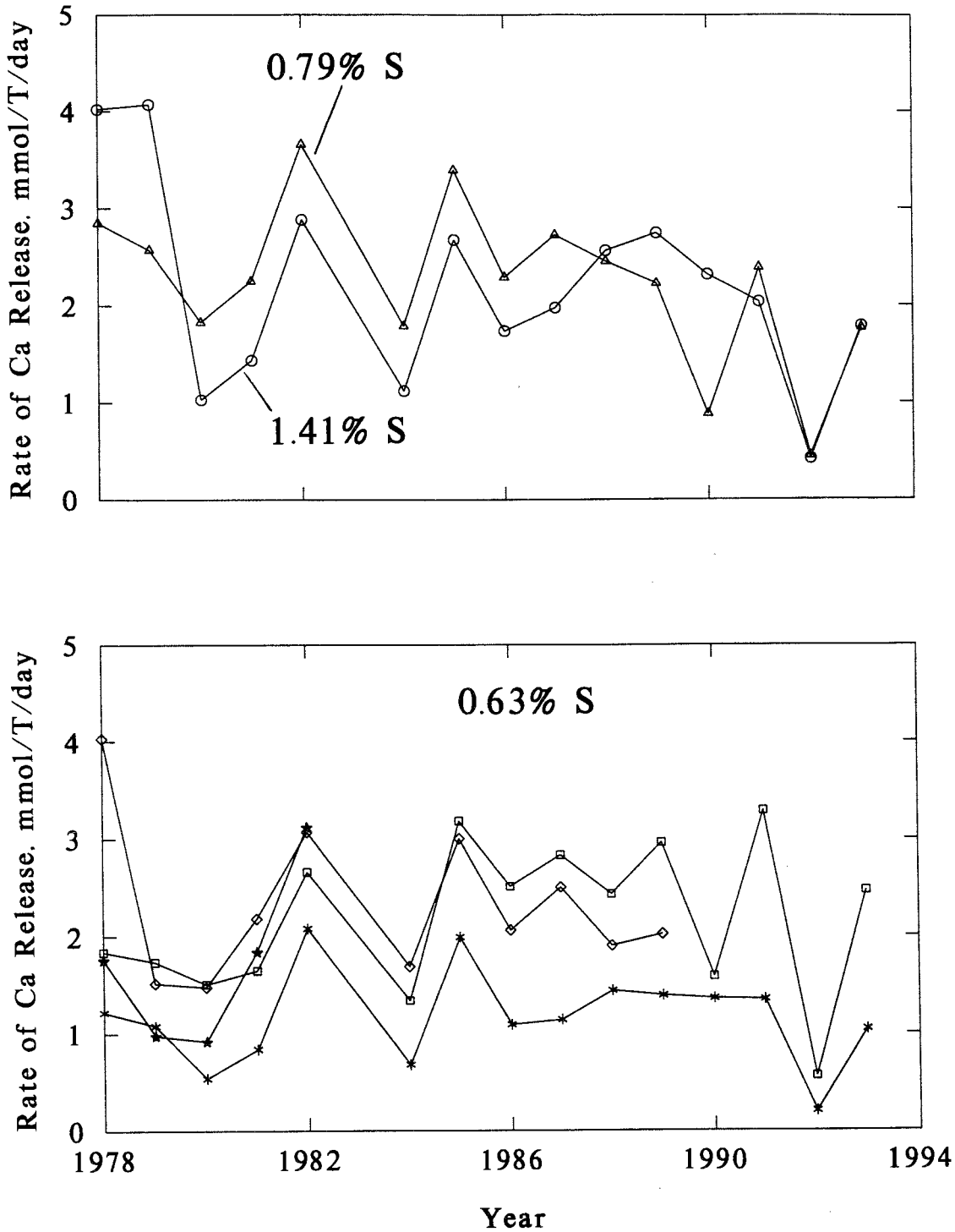

Figure 12. Comparison of minimum drainage pH in field (▲) and laboratory (○).

Field data represents overall recurrent minimum pH from the Duluth Complex test piles. Laboratory data represents drainage pH after 150 weeks of dissolution of Duluth Complex blast hole samples. The curve was generated with the log smoothing feature of the Systat statistical software package.

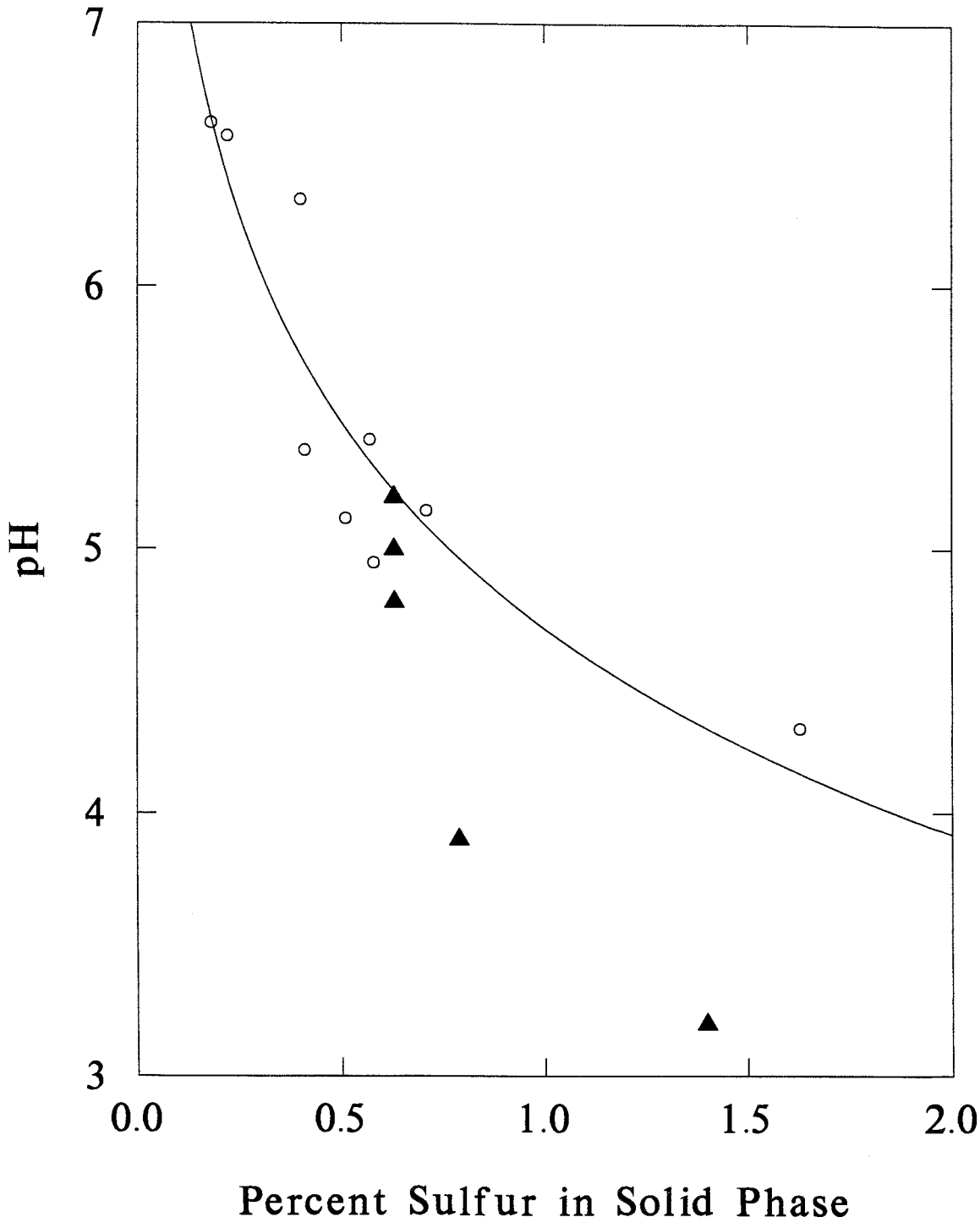

Table 1. Summary of chemical and physical characteristics of test piles.

	Pile 1	Pile 2	Pile 3	Pile 4	Pile 6	Pile 5
Date completed	4/20/77	4/20/77	4/20/77	4/20/77	9/30/77	9/10/77
% S	0.63	0.63	0.63	0.63	0.79	1.41
% Cu	0.35	0.35	0.35	0.35	0.34	0.30
% Ni	0.083	0.083	0.083	0.083	0.084	0.085
Mass, T	1100	1100	830	1130[1]	1300	815
Volume, m^3	540	530	400	830[1]	630	400
Collecting area, m^2	290	340	300	330[1]	320	340
Surface area, m^2	330	450	430	350[1]	390	370
Cover material	None	Topsoil	Glacial Till (coarse sand)	None	None	Sandy till over coarse sand
Avg. cover depth, cm	0	23	34	0	0	54
Vegetated	No	Yes	Yes	No	No	Yes

[1] Approximately 40 pct of FL4 was removed in September, 1982.

Table 2. Chemical composition of a grab sample from test pile 1.

Element	Concentration	Unit
S	0.655	Pct
Cu	0.306	Pct
Ni	0.073	Pct
Fe (S)[1]	0.696	Pct
Co	0.009	Pct
Zn	0.016	Pct
Si (SiO_2)	22.80 (48.81)	Pct
Al (Al_20_3)	8.98 (16.96)	Pct
Fe (FeO)	9.22 (11.86)	Pct
Mg (MgO)	4.31 (7.15)	Pct
Ca (CaO)	5.55 (7.76)	Pct
Na (Na_20)	2.00 (2.70)	Pct
K (K_20)	0.32 (0.39)	Pct
Ti ($Ti0_2$)	1.41 (2.35)	Pct
P (P_2O_5)	0.02 (0.05)	Pct
Mn (MnO)	0.12 (0.16)	Pct
Cr (Cr_20_3)	0.03 (0.15)	Pct
B	662.00	ppm
Ba	1173.00	ppm
Be	1.00	ppm
Sr	279.00	ppm
Pb	0	ppm
Ag	1.35	ppm
As	0	ppm
Hg	1.00	ppm
Mo	0.50	ppm
Cd	0	ppm
V	276.50	ppm
Th	6.40	ppm
Zr	80.00	ppm

[1] Iron associated with sulfur.

Table 3. Mineralogical composition of a grab sample from test pile 1 (volume percent).

Opaques[1]	5.098
Chalcopyrite-cubanite	0.769
Pentlandite	0.037
Pyrrhotite	0.844
Ilmenite-magnetite	3.447
Graphite	-
Plagioclase	59.112
Sericite	2.176
Olivine	10.510
Clinopyroxene	11.185
Orthopyroxene	3.716
Monocrystalline amphibole	3.567
Fibrous amphibole	0.288
Chlorite	1.136
Serpentine	0.257
Iddingsite	0.075
Talc	-
Biotite	1.738
Smectite	0.021
Caladonite	-
Spinel	-
Myrmekite	-
Apatite	0.085
Epidote	0.953
Allanite	-
Calcite	0.056
Quartz	-
Cordierite	0.027

[1] The value shown for opaques is the sum of the five following values.

Table 4. Particle size distribution rock from test pile 4.

	Particle diameter (mm)	Percent finer than
Boulders and cobbles	305	89.0
	152	82.2
Gravel	76.2	67.7
	25.4	38.3
	19.0	33.8
	12.7	27.1
	6.4	19.2
Sand	2.00	12.2
	1.41	10.8
	0.84	10.6
	.425	7.7
	.250	6.1
	.149	4.6
	.105	3.4
Silt and clay	.074	3.1
	.053	1.4 to 1.7[1]

Note: Boulders, cobbles, gravel, sand, and silt refer to U.S. Department of Agriculture (USDA) classification definitions.

[1] Estimated based on hydrometer analysis of the less than 0.074 mm fraction.

Table 5. Characterization of fine fractions, joint disposal bin gabbro material.

Size Fraction mm	pH	Total Sulfur %	Sulfide %	Cu %	Ni %	Co %	Zn %	Specific Surface Area m^2/g
Sand								
0.50-2.00	6.56	0.67	0.65	0.337	0.062	0.012	0.015	0.60
.177-.50	6.37	0.80	0.75	0.381	0.078	0.013	0.015	0.78
.149-.177	6.18	0.88	0.80	0.391	0.103	0.015	0.015	1.1
.105-.149	NA	1.12	1.05	0.495	0.147	0.019	0.017	1.6
.075-.105	5.95	1.37	1.30	0.585	0.221	0.022	0.019	1.7
Silt and clay								
.053-.075	5.78	1.65	1.57	0.647	0.253	0.024	0.019	2.6
<.053	5.77	1.94	1.86	0.814	0.295	0.027	0.023	4.7

NA = not available.

Table 6. Annual median pH, flow-weighted mean concentrations, and drainage volume for Duluth Complex test piles.

Pile	Sulfur %	Parameter	Flow-weighted Mean Concentration, mg/L, and Volume, 10^3 L															
			1978	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993
FL1	0.63	pH	7.80	7.35	7.20	7.20	7.00	6.86	6.36	6.70	6.54	6.03	6.09	5.69	5.72	5.25	5.19	5.14
		SO4	949	743	1202	772	755	993	920	880	723	986	879	965	1254	1253	958	1021
		Ca	201	224	318	239	255	NA	263	238	221	277	249	227	274	256	199	199
		Mg	17	20	30	27	23	NA	41	42	41	53	69	61	94	111	87	88
		Volume	83.1	86.6	53.8	83.8	118.6	82.4	66.9	133.4	119.6	104.8	106.3	122.5	60.1	121.0	27.5	125.0
FL2	0.63	pH	7.60	7.30	6.95	6.80	6.70	6.39	6.17	6.30	6.17	5.54	5.47	5.27	5.13	5.18	5.07	5.04
		SO4	689	561	707	652	834	858	685	963	593	676	829	748	884	863	818	938
		Ca	148	160	201	156	207	NA	183	210	165	186	203	163	180	167	144	160
		Mg	20	23	28	29	37	NA	41	54	43	52	78	60	75	77	69	94
		Volume	70.2	75.6	29.5	67.2	100.4	52.8	40.4	94.6	69.3	63.2	78.9	80.2	78.5	76.0	13.5	65.2
FL3	0.63	pH	7.60	7.05	6.65	6.60	6.58	5.90	5.05	5.44	5.27	5.01	5.15	4.98	NA	NA	NA	NA
		SO4	957	557	1035	1223	1244	1566	1378	1620	1210	1653	1225	1132	NA	NA	NA	NA
		Ca	285	150	211	224	213	NA	211	220	186	238	170	153	NA	NA	NA	NA
		Mg	52	31	77	110	119	NA	153	156	134	194	137	113	NA	NA	NA	NA
		Volume	90.8	87.1	58.3	92.4	125.6	68.5	78.1	103.1	89.2	81.4	94.1	94.9	NA	NA	NA	NA
FL4	0.63	pH	7.55	7.10	6.80	6.85	6.55	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
		SO4	610	470	801	765	771	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
		Ca	163	140	166	209	228	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
		Mg	17	15	17	34	37	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
		Volume	92.2	74.0	53.2	109.9	147.5	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
FL6	0.79	pH	7.51	6.40	4.40	4.40	4.20	4.19	4.14	4.27	4.27	4.00	4.12	4.20	4.10	4.14	4.24	4.24
		SO4	1046	1154	1496	1752	1891	2406	2469	2550	1873	2615	2201	2466	3234	2692	1683	1811
		Ca	261	292	277	249	270	NA	280	270	248	299	271	248	297	249	164	175
		Mg	77	110	172	205	220	NA	302	334	274	344	308	281	376	369	256	256
		Volume	117.2	118.2	82.3	130.5	117.2	96.0	99.2	148.3	114.4	108.1	117.1	102.0	36.2	106.8	31.9	120.4
FL5	1.41	pH	5.50	3.40	3.40	3.40	3.40	3.60	3.49	3.50	3.63	3.43	3.62	3.62	3.62	3.60	3.73	3.66
		SO4	862	2147	2900	3488	3805	4007	3424	3852	2973	2618	2355	2429	2264	1968	1642	2079
		Ca	225	285	228	220	225	NA	221	265	248	245	239	240	233	202	171	192
		Mg	27	129	273	308	311	NA	287	326	273	234	195	168	156	143	122	142
		Volume	120.5	117.3	35.7	58.9	105.8	57.2	49.6	83.2	54.5	60.1	70.9	82.5	76.4	70.2	17.8	69.7

Table 7. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL1 (0.63% S, 1100 T, no cover). Percent of total original sulfur released = 6.76.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	83100	206	9	821	416	58	3.62	1.84	0.26	7.80
1979	86600	252	33	670	483	71	2.42	1.74	0.26	7.35
1980	53800	256	28	673	426	66	2.39	1.51	0.23	7.20
1981	83800	275	28	673	499	93	2.22	1.65	0.31	7.20
1982	118600	258	28	932	755	113	3.28	2.66	0.40	7.00
1983	82400	277	10[2]	852	NA	NA	2.80	NA	NA	6.86
1984	66879	298	11	640	439	113	1.95	1.34	0.34	6.36
1985	133396	227	9	1221	793	231	4.89	3.18	0.93	6.70
1986	119551	238	9	900	658	202	3.44	2.51	0.77	6.54
1987	104768	233	9	1075	725	229	4.19	2.83	0.89	6.03
1988	106298	247	10	972	660	300	3.58	2.43	1.10	6.09
1989	122463	213	9	1230	693	308	5.25	2.96	1.31	5.69
1990	60101[3]	235	9	784	411	233	3.03	1.59	0.90	5.72
1991	120980	213	9	1577	772	552	6.73	3.29	2.36	5.25
1992	27465	221	16	274	136	98	1.13	0.56	0.40	5.19
1993	124968	228	17	1327	620	454	5.29	2.47	1.81	5.14

[1] Rates were calculated over the period of flow.
[2] Calcium and magnesium were not measured in 1983.
[3] FL1 was partially covered with plastic during 1990.

Table 8. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL2 (0.63% S, 1100 T, cover of topsoil). Percent of total original sulfur released = 3.94.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	70200	193	6	503	260	57	2.37	1.22	0.27	7.60
1979	75600	253	32	441	302	70	1.58	1.09	0.25	7.30
1980	29500	250	27	217	148	33	0.79	0.54	0.12	6.95
1981	67200	283	28	456	261	81	1.46	0.84	0.26	6.80
1982	100400	226	16	872	518	151	3.51	2.08	0.61	6.70
1983	52800	277	9[2]	472	NA	NA	1.55	NA	NA	6.39
1984	40431	247	10	288	185	69	1.06	0.68	0.25	6.17
1985	94649	227	9	948	496	212	3.80	1.99	0.85	6.30
1986	69334	238	9	428	286	124	1.63	1.09	0.47	6.17
1987	63221	233	9	445	293	136	1.74	1.14	0.53	5.54
1988	78948	252	10	681	399	252	2.46	1.44	0.91	5.47
1989	80188	213	9	624	326	199	2.66	1.39	0.85	5.27
1990	78458	235	9	722	352	243	2.79	1.36	0.94	5.13
1991	76029	213	9	683	316	241	2.92	1.35	1.03	5.18
1992	13481	221	16	115	48	39	0.47	0.20	0.16	5.07
1993	65210	228	17	636	260	253	2.54	1.04	1.01	5.04

[1] Rates were calculated over the period of flow.
[2] Calcium and magnesium were not measured in 1983.

Table 9. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL3 (0.63% S, 830 T, cover of glacial till (coarse sand)). Percent of total original sulfur released = 8.36.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	90800	193	5	904	646	194	5.64	4.03	1.21	7.60
1979	87100	258	34	505	326	111	2.36	1.52	0.52	7.05
1980	58300	249	30	628	306	184	3.04	1.48	0.89	6.65
1981	92400	285	31	1176	516	418	4.97	2.18	1.77	6.60
1982[2]	125600	262	21	1626	668	615	7.48	3.07	2.83	6.58
1983	68500	277	10[3]	1116	NA	NA	4.85	NA	NA	5.90
1984	78088	292	11	1120	410	493	4.62	1.69	2.03	5.05
1985	103131	227	9	1738	566	664	9.22	3.00	3.52	5.44
1986	89161	241	10	1123	413	491	5.61	2.06	2.45	5.27
1987	81412	233	9	1400	484	650	7.24	2.50	3.36	5.01
1988	94121	252	10	1200	398	529	5.74	1.90	2.53	5.15
1989[4]	94924	217	9	1118	363	440	6.21	2.02	2.44	4.98

[1] Rates were calculated over the period of flow.
[2] Fertilizer and 1500 gallons of lime-stabilized sludge were applied on May 19, 1982.
[3] Calcium and magnesium were not measured in 1983.
[4] FL3 was covered with plastic and subsequently hypalon, thereby terminating flow from the pile at the end of the 1989 field season.

Table 10. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL4 (0.63% S, 1130 T, no cover). Percent of total original sulfur released = 1.55.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	92200	188	7	585	374	65	2.75	1.76	0.31	7.55
1979	74000	233	26	362	258	46	1.37	0.98	0.17	7.10
1980	53200	211	18	444	220	37	1.86	0.92	0.16	6.80
1981	109900	276	29	875	574	152	2.81	1.84	0.49	6.85
1982	147500	238	15	1183	838	227	4.40	3.12	0.84	6.55

[1] Rates were calculated over the period of flow.

Table 11. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL6 (0.79% S, 1300 T, no cover). Percent of total original sulfur released = 11.30.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	117200	206	6	1275	764	370	4.76	2.85	1.38	7.51
1979	118200	257	36	1420	860	534	4.25	2.57	1.60	6.40
1980	82300	239	28	1281	570	582	4.12	1.83	1.87	4.40
1981	130500	276	30	2379	809	1103	6.63	2.25	3.07	4.40
1982	177200	250	19	3486	1191	1606	10.73	3.66	4.94	4.20
1983	96000	277	10[2]	2404	NA	NA	6.68	NA	NA	4.19
1984	99158	298	12	2548	693	1233	6.58	1.79	3.18	4.14
1985	148259	227	9	3934	1000	2037	13.33	3.39	6.90	4.27
1986	114434	238	9	2230	708	1291	7.21	2.29	4.17	4.27
1987	108134	227	9	2943	805	1533	9.97	2.73	5.19	4.00
1988	117127	247	10	2682	791	1483	8.35	2.46	4.62	4.12
1989	101975	217	9	2617	630	1181	9.28	2.23	4.19	4.20
1990	36196[3]	235	9	1218	268	560	3.99	0.88	1.83	4.10
1991	106787	213	9	2991	663	1622	10.80	2.39	5.86	4.14
1992	31850	221	16	558	130	335	1.94	0.45	1.17	4.24
1993	120417	228	17	2269	525	1270	7.66	1.77	4.28	4.24

[1] Rates were calculated over the period of flow.
[2] Calcium and magnesium were not measured in 1983.
[3] FL6 was partially covered with plastic during 1990.

Table 12. Annual total mass release and rate of release of sulfate, calcium, and magnesium for Duluth Complex test pile FL5 (1.41% S, 815 T, cover of sandy till over coarse sand). Percent of total original sulfur released = 8.58.

Year	Total Volume L	Period Days	Number of Cases	Total Mass Release mol			Rate of Release[1] mmol/T•day			Median pH
				SO_4	Ca	Mg	SO_4	Ca	Mg	
1978	120500	206	9	1081	676	135	6.44	4.03	0.80	5.50
1979	117300	251	34	2621	833	624	12.81	4.07	3.05	3.40
1980	35700	243	30	1077	203	402	5.44	1.03	2.03	3.40
1981	58900	275	29	2138	323	746	9.54	1.44	3.33	3.40
1982	105800	253	20	4189	595	1356	20.32	2.89	6.58	3.40
1983[2]	57200	277	27[3]	2385	NA	NA	10.56	NA	NA	3.60
1984	49625	299	12	1768	274	586	7.26	1.12	2.40	3.49
1985	83220	252	10	3336	549	1115	16.24	2.67	5.43	3.50
1986	54523	238	9	1687	337	613	8.70	1.74	3.16	3.63
1987	60084	227	9	1637	367	578	8.85	1.98	3.12	3.43
1988	70886	202	10	1737	423	569	10.55	2.57	3.46	3.62
1989	82485	220	9	2085	493	569	11.63	2.75	3.17	3.62
1990	76442	235	9	1801	445	491	9.40	2.32	2.56	3.62
1991	70171	213	9	1437	354	413	8.28	2.04	2.38	3.60
1992	17775	221	16	304	76	89	1.69	0.42	0.49	3.73
1993	69732	228	17	1508	333	406	8.12	1.79	2.18	3.66

[1] Rates were calculated over the period of flow.
[2] Bactericide was applied in July, 1983.
[3] Calcium and magnesium were not measured in 1983.

Table 13. Average rates of mass release and minimum drainage pH.

Percent Sulfur	Period of Record	Minimum pH[1]	Average Observed Rates of Release,[2] mmol/T•day			
			SO_4	Ca	Mg	Ca + Mg
0.63 (FL1)	1978-1993[3]	4.96	3.4	2.1	0.79	2.9
0.63 (FL2)	1978-1993	4.76	2.0	1.1	0.55	1.7
0.63 (FL3)	1978-1989	4.71	5.5	2.4	2.1	4.5
0.79 (FL6)	1978-1993[3]	3.70	7.0	2.2	3.6	5.8
1.41 (FL5)	1978-1993	3.08	9.8	2.1	3.0	5.1

[1] The minimum pH values usually occurred as isolated excursions. The recurrently observed minimum pH values were typically 0.2 to 0.3 units higher.

[2] This represents the release rate during the period of flow. 1983 was not included for the rates of Ca and Mg release, since these parameters were not analyzed in this year.

[3] Partially covered by plastic during 1990.

Table 14. Neutralization potentials as determined by drainage quality of Duluth Complex test piles.

Sulfur %	Pile	Neutralization Potential[1], mg $CaCO_3$/g rock					
		pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
0.63	FL1	0.346	0.781	>1.059	>1.059	>1.059	>1.059
0.63	FL2	0.179	0.393	>0.642	>0.642	>0.642	>0.642
0.63	FL3	0.442	1.254	>1.340	>1.340	>1.340	>1.340
0.79	FL6	0.157	0.172	0.226	1.702	>2.074	>2.074
1.41	FL5	0.044	0.044	0.096	0.130[2]	0.136[2]	>1.889

1 Neutralization potential (NP) was calculated using the cumulative calcium and magnesium release (see methods section of report).

2 Neutralization potential was calculated as of 1978, although higher pH values were observed later. Two pH values of 4.6 and 4.0 were observed in spring runoff in 1980 and 1989, respectively. Five pH values between 4.0 and 4.4 were observed in 1992.

Table 15. Qualitative comparison of test pile and laboratory data.

	Laboratory Drill Core[1]	Laboratory Blast Holes[2]	Field Test Piles
Drainage pH:			
as time increased	pH decreased	pH decreased	gradually decreased (0.63 pct S); decreased and plateaued (10.79 pct S); decreased and slightly increased (1.41 pct S)
as pct S increased	pH decreased	pH decreased	pH decreased
Sulfate Release Rate:			
as time increased max. rate: min. rate[3]	fairly constant, 1:1 to 3:1 for pH >4.5, 6:1 for some samples with pH~4	fairly constant, 1:1 to 3:1 for pH >4, 6:1 to 7:1 for pH <4	fairly constant, 2.7:1 to 3.9:1
as pct S increased	increased	increased	increased
as flow increased	no data	no data	increased
Calcium Release Rate:			
as time increased	fairly constant	limited data	fairly constant
as pct S increased	fairly constant	fairly constant	fairly constant
as flow increased	no data	no data	increased
Magnesium Release Rate:			
as time increased	increased	limited data	increased
as pct S increased	increased	increased	general increase although release from 0.79 pct S rock exceeded that from 1.41 pct S rock
as flow increased	no data	no data	increased

1 Lapakko, 1993a.
2 Lapakko, 1993b.
3 Ratio of maximum release rate to minimum release rate for a solid.

Table 16. Comparison of field and laboratory neutralization potentials.

Percent S	pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
Laboratory neutralization potential, mg $CaCO_3$/g rock[1]						
0.63	0.67	1.5	2.9	>2.9	>2.9	>2.9
0.79	0.29	0.75	1.7	>1.8	>1.8	>1.8
1.41	0.26	0.79	>1.7	>2.9	>3.1	>3.1
Ratio of field NP to laboratory NP						
0.63(FL1)	0.52	0.52	NAp	NAp	NAp	NAp
0.63(FL2)	0.27	0.26	NAp	NAp	NAp	NAp
0.63(FL3)	0.66	0.84	NAp	NAp	NAp	NAp
0.79(FL6)	0.54	0.23	0.13	NAp	NAp	NAp
1.41(FL5)	0.17	0.056	<0.056	<0.045	<0.044	NAp

NAp: not applicable

[1] See appendix 1 for sources of laboratory data.

Table 17. Comparison of field and laboratory blast hole release rates. Rates expressed in mole (g rock)$^{-1}$ s^{-1} x 10^{14}.

Percent Sulfur	Field Rate			Lab Rates			Retardation Factor			
	SO_4	Ca	Mg	SO_4	Ca	Mg	SO_4	Ca	Mg	Avg.
0.63[1] (FL1)	3.9	2.4	0.91	25[1]	6.7[1]	12[1]	0.16	0.36	0.076	0.20
0.63[1] (FL2)	2.3	1.3	0.64	25	6.7	12	0.093	0.19	0.053	0.11
0.63[1] (FL3)	6.4	2.8	2.4	25	6.7	12	0.26	0.42	0.20	0.29
0.79[2] (FL6)	8.1	2.6	4.2	26[2]	6.1[2]	12[2]	0.31	0.42	0.35	0.36
1.41[3] (FL5)	11	2.4	3.5	66[3]	19[3]	21[3]	0.17	0.13	0.17	0.16

1 The arithmetic mean of average rates from blast hole samples with sulfur contents of 0.57, 0.58, and 0.71 percent (150 week period of record) was used for comparison with 0.63-percent-sulfur rock. Minimum pH values for these samples ranged from 4.82 to 5.38 as compared to 4.7 to 5.0 in the field.

2 Rates were linearly extrapolated based on sulfur content using average rates from blast hole samples with sulfur contents of 0.71 and 1.16 percent (150- and 69-week periods of record, respectively). Minimum pH values for these samples ranged from 4.6 to 5.0, about one unit higher than the pH 3.7 minimum value for the field pile.

3 The arithmetic mean of average rates from blast hole samples with sulfur contents of 1.40 and 1.44 percent was used (69 week period of record) for comparison with the 1.41-percent-sulfur pile. Minimum drainage pH for these samples ranged from 4.4 to 4.9, about 1.5 units higher than the minimum drainage pH for the field pile and and one unit higher than its recurrent minimum pH.

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Appendix 3: Mass Release

Table A3.1. Percent of total original sulfur released.

Text A1.1. Neutralization potential calculation.

Individual drainage volume measurements were missing for given periods of record. Approximately half of the samples in the 1978 period of record for each test pile did not have an associated drainage volume value. One or two measurements were missing in 1982 and 1984 for each test pile. Calculation of the annual drainage volumes for this report omitted the missing values. When compared with the annual drainage volumes for 1978 through 1984 found in Eger and Lapakko (1985), discrepancies were revealed between the two sets of values. While most of the calculated values were generally five to twenty percent lower than the associated values in Eger and Lapakko (1985), the calculated 1978 values were several times lower.

To account for the discrepancies, the mass release values obtained using the calculated volumes were multiplied by adjusting factors. The factors were ratios of the annual volumes obtained in Eger and Lapakko (1985) over the calculated annual volumes for the respective piles. Adjusted mass release rates were calculated with the adjusted mass release values. The resulting rates compared favorably with those obtained in Eger and Lapakko (1985) (Table A1.2). The adjusted annual mass release and rates of release are used throughout this report. The annual drainage volumes for 1978 to 1984 obtained in Eger and Lapakko (1985) are used throughout the report as well.

The adjusted cumulative mass release values allowed more accurate determination of neutralization potentials (NP). These were calculated to determine the amount of acid neutralized by the solids prior to the drainage pH decreasing and, for the duration of the experiment, remaining below pH values of 7, 6, 5, 4.5, 4, and 3.5. The acid neutralized was calculated by two methods: as the a) cumulative sulfate release (expressed as mg $CaCO_3$/g rock) and b) cumulative calcium and magnesium release (expressed as mg $CaCO_3$/g rock) prior to the drainage pH decreasing and remaining below the specified pH value. If the drainage pH from a given solid never decreased permanently below a specified value, the NP was reported as "greater than" the total sulfate release for the period of record.

(a) Cumulative sulfate release to a given pH threshold:

$$NP = (\Sigma SO_4 \times 1.04) / m_p, \text{ where}$$

NP	=	neutralization potential, in mg $CaCO_3$/g rock
ΣSO_4	=	cumulative sulfate release to a given pH threshold, in milligrams
1.04	=	100 mg $CaCO_3$ / 96.1 mg SO_4
m_p	=	mass of pile, in grams

(b) Cumulative calcium plus magnesium release to a given pH threshold:

$$NP = [(\Sigma Ca \times 2.49) + (\Sigma Mg \times 4.12)] / m_p, \text{ where}$$

ΣCa	=	cumulative calcium release to a given pH threshold, in milligrams
2.49	=	100 mg $CaCO_3$ / 40.1 mg Ca
ΣMg	=	cumulative magnesium release to a given pH threshold, in milligrams
4.12	=	100 mg $CaCO_3$ / 24.3 mg Mg

Two steps were taken to increase the accuracy of the NP values by obtaining a more continuous record of cumulative release. First, mass release had to be estimated for 1977, when the piles were constructed. Mass release was estimated as the product of the mass of the pile, the number of days between the construction of the pile and freeze-up in November, 1977, and the mean rate of release from the pile for subsequent years. Rates of release from FL1 for 1978 to 1991 were used as representative rates for FL2 and FL3, because FL1 was uncovered for its entire period of record, while FL2 and FL3 were uncovered only in 1977.

The second step taken was to estimate mass release for 1983, when no calcium or magnesium concentrations were measured. Mass release for 1983 was estimated as the mean of the 1982 and 1984 flow-weighted-mean concentrations multiplied by the 1983 drainage volume. The estimated 1983 mass release was added only to those NP values corresponding to pH thresholds reached later than 1983. The pH 7 threshold for FL1 was reached in May, 1983, after one-fifth of the field season had passed. For that case, one-fifth of the estimated 1983 mass release was added to the NP_{pH7}.

Table A1.1. Monthly and annual precipitation at the Duluth Complex test piles site, 1978 - 1993.

Month	Precipitation, cm[1]															
	1978	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993
January	1.70	1.30	2.44	1.24	2.36	3.68[2]	0.43	1.14	1.80	1.17	3.07	3.23	2.36	0.00	0.00	0.00
February	0.81	4.85	1.68	3.40	1.07	1.68[2]	1.63	1.27	2.34	1.45	0.76	1.04	2.34	0.00	0.00	0.00
March	1.70	6.98	2.31	1.88	1.78	1.68[2]	0.61	1.88	1.12	0.89	6.10	3.33	5.46	3.07	0.00	0.00
April	3.25	2.50	1.90	11.2	2.41	1.40	3.00	6.81	6.76	0.56	0.23	1.30	10.4	1.83	1.55	5.49
May	9.35	9.73	1.32	2.46	11.1	4.24	3.48	10.7	4.80	14.5	7.65	8.15	2.18	8.13	9.60	6.65
June	4.90	10.1	9.09	15.5	6.65	5.13	15.2	12.1	11.0	3.12	6.58	18.3	13.3	12.1	7.04	9.98
July	19.1	7.04	6.27	4.29	16.4	8.51	4.80	8.41	10.2	16.8	6.50	2.84	6.43	13.4	8.59	19.0
August	14.1	5.89	13.4	4.39	12.5	12.4	6.43	9.73	9.12	9.45	30.5	11.2	3.58	6.25	13.5	9.78
September	8.23	5.08	10.8	5.97	9.70	8.48	4.83	16.2	13.3	8.05	6.27	10.7	7.77	13.3	10.9	6.07
October	2.59	7.59	3.66	9.55	13.3	11.0	7.16	4.75	2.39	2.06	4.83	4.93	8.99	4.67	2.39	0.64
November	2.46	1.40	2.84	1.60	5.21	8.86[2]	1.70	6.07	5.26	1.32	5.77	2.84	0.08	0.00	0.00	0.00
December	1.37	0.89	3.10	2.01	4.04	3.76[2]	4.27	1.04	0.94	0.97	2.84	1.40	1.57	0.00	0.00	0.00
TOTAL	69.6	63.3	58.8	63.4	86.5	70.9	53.5	80.1	69.0	60.3	81.1	69.3	64.5	62.8	60.0[3]	65.5[4]

[1] From Kennecott raingage unless otherwise noted.
[2] From Babbitt raingage.
[3] Includes 6.45 cm of spring runoff from snowpack.
[4] Includes 7.92 cm of spring runoff from snowpack.

Table A1.2. Comparison of adjusted rates and those obtained in the 1985 Heavy Metals Study. See Methods for more information.

Pile	Year	Adj. SO_4 Rate mmol/T/day	Adj. Ca Rate mmol/T/day	Adj. Mg Rate mmol/T/day	HMS SO_4 Rate mmol/T/day	HMS Ca Rate mmol/T/day	HMS Mg Rate mmol/T/day	Adj. SO_4/ HMS SO_4	Adj. Ca/ HMS Ca	Adj. Mg/ HMS Mg
FL1	1978	3.62	1.84	0.26	4.47	2.49	0.33	0.81	0.74	0.77
	1979	2.42	1.74	0.26	2.39	1.75	0.26	1.01	1.00	0.99
	1980	2.39	1.51	0.24	2.39	1.50	0.23	1.00	1.01	1.01
	1981	2.23	1.65	0.31	2.19	1.65	0.30	1.02	1.00	1.01
	1982	3.28	2.66	0.40	2.71	2.49	0.39	1.21	1.07	1.01
	1983	2.79	NA	NA	2.81	NA	NA	0.99	NA	NA
FL2	1978	2.37	1.22	0.27	3.23	1.87	0.37	0.73	0.65	0.71
	1979	1.59	1.08	0.25	1.46	1.10	0.26	1.09	0.99	0.98
	1980	0.79	0.54	0.12	0.80	0.55	0.12	0.98	0.98	0.99
	1981	1.46	0.84	0.26	1.46	0.85	0.26	1.00	0.99	0.99
	1982	3.51	2.08	0.61	3.02	1.95	0.58	1.16	1.07	1.06
	1983	1.55	NA	NA	1.56	NA	NA	0.99	NA	NA
FL3	1978	5.64	4.03	1.21	3.54	2.29	0.66	1.60	1.76	1.84
	1979	2.36	1.52	0.52	2.50	1.57	0.53	0.94	0.97	0.97
	1980	3.04	1.48	0.89	3.02	1.47	0.91	1.01	1.01	0.98
	1981	4.97	2.18	1.77	4.99	2.17	1.77	1.00	1.01	1.00
	1982	7.48	3.07	2.83	6.87	2.99	2.67	1.09	1.03	1.06
	1983	4.85	NA	NA	4.89	NA	NA	0.99	NA	NA
FL4	1978	2.75	1.76	0.31	3.75	2.37	0.40	0.73	0.74	0.78
	1979	1.37	0.98	0.17	1.35	1.00	0.18	1.01	0.98	0.94
	1980	1.86	0.92	0.16	1.87	0.92	0.16	0.99	1.00	1.00
	1981	2.81	1.84	0.49	2.81	1.85	0.49	1.00	0.99	1.00
	1982	4.40	3.12	0.84	3.95	2.99	0.78	1.11	1.04	1.08

Table A1.2. Comparison of adjusted rates and those obtained in the 1985 Heavy Metals Study. See Methods for more information.

Pile	Year	Adj. SO_4 Rate mmol/T/day	Adj. Ca Rate mmol/T/day	Adj. Mg Rate mmol/T/day	HMS SO_4 Rate mmol/T/day	HMS Ca Rate mmol/T/day	HMS Mg Rate mmol/T/day	Adj. SO_4/ HMS SO_4	Adj. Ca/ HMS Ca	Adj. Mg/ HMS Mg
FL6	1978	4.76	2.85	1.38	7.91	5.74	2.26	0.60	0.50	0.61
	1979	4.25	2.57	1.60	4.06	2.49	1.60	1.05	1.03	1.00
	1980	4.12	1.83	1.87	4.06	1.85	1.89	1.02	0.99	0.99
	1981	6.63	2.25	3.07	6.66	2.27	3.09	1.00	0.99	1.00
	1982	10.73	3.66	4.94	9.16	3.24	4.53	1.17	1.13	1.09
	1983	6.68	NA	NA	6.66	NA	NA	1.00	NA	NA
FL5	1978	6.44	4.02	0.80	9.47	5.99	1.11	0.68	0.67	0.72
	1979	12.81	4.07	3.05	12.49	3.99	2.96	1.03	1.02	1.03
	1980	5.44	1.03	2.03	5.41	1.02	2.02	1.01	1.00	1.01
	1981	9.54	1.44	3.33	9.57	1.45	3.33	1.00	1.00	1.00
	1982	20.32	2.88	6.58	19.77	2.74	6.58	1.03	1.05	1.00
	1983	10.56	NA	NA	10.41	NA	NA	1.02	NA	NA

Table A1.3. Laboratory neutralization potential data (for values listed as "greater than" averages represent "best guess").

Percent Sulfur	Sample Type[1]	Period of Record (weeks)	Neutralization Potential					
			pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
For comparison with 0.63 percent sulfur rock in field								
0.57	B	150	1.32	2.0	>3.1	>3.1	>3.1	>3.1
0.58	B	150	0.35	1.4	2.9	>3.0	>3.0	>3.0
0.71	B	150	0.35	1.1	>2.6	>2.6	>2.6	>2.6
Avg.			0.67	1.5	2.9	>2.9	>2.9	>2.9
For comparison with 0.79 percent sulfur rock in field								
0.71	B	150	0.35	1.1	>2.6	>2.6	>2.6	>2.6
0.80	D	17	0.32	>0.39	>0.39	>0.39	>0.39	>0.39
0.92	D	34	0.19	0.40	0.77	>0.93	>0.93	>0.93
Avg.			0.29	0.75[2]	1.7[3]	>1.8[4]	>1.8[4]	>1.8[4]
For comparison with 1.41 percent sulfur rock in field								
1.35	D	30	0	0	0	3.1	>3.7	>3.7
1.40	B	78	0.41	1.6	>3.0	>3.0	>3.0	>3.0
1.44	B	78	0.38	0.77	2.0	>2.7	>2.7	>2.7
Avg.			0.26	0.79	>1.7	>2.9	>3.1	>3.1

[1] D:drill core, B:blast hole
[2] (1.1 + 0.4)/2
[3] (2.6 + 0.77)/2
[4] (2.6 + 0.93)/2

Table A1.4. Release rates for drill core samples (Lapakko, 1993a).

Percent Sulfur	Period (weeks)	Rates, mole (g rock)$^{-1}$s^{-1}			Minimum pH
		SO_4	Ca	Mg	
0.47	2-13	13	19	3.3	6.85
0.59	7-17	22	26	6.4	7.30
0.80	6-17	36	34	7.9	6.75
0.80	9-17	35	30	9.3	6.90
0.92	23-34	82	28	28	4.60
0.92	27-34	42	17	19	4.90
1.17[1]	11-17	45	(78)[2]	12	8.30
1.17[1]	13-17	58	(83)[2]	13	8.20
1.24	2-21	49	26	14	5.70
1.24	2-21	45	25	14	5.70
1.26[3]	1-21	27	20	3.5	6.20
1.26[3]	1-21	28	19	3.7	5.95
1.35	6-25	93	18	20	4.20
1.47[4]	13-21	190	37	31	4.40
1.47[4]	14-21	310	39	38	4.25
1.74[4]	11-21	350	82	73	4.30
1.74[4]	14-21	170	59	50	4.55
1.87	17-34	150	59	59	5.05
1.87	17-34	160	53	72	4.90
2.01	17-30	380	24	15	3.70
2.17[5]	15-21	190	29	42	4.30
2.17[5]	9-21	76	19	17	4.65
2.57[5]	16-21	110	8.2	11	4.20
2.57[5]	16-21	170	14	13	4.10

1 Sample contained three percent $CaCO_3$ which is atypical of Duluth Complex rock.

2 Parentheses indicate anomalous value.

3 Sample underwent oxidation previously in a test pile.

4 Sample was amphibole-rich and thus atypical of Duluth Complex rock.

5 Contained a significant amount of quartz and was, therefore, atypical of Duluth Complex rock.

Table A1.5. Release rates for blast hole samples during period of Ca, Mg analysis (Lapakko, 1993b).

Percent Sulfur	Period, weeks		Rates, mole (g rock)$^{-1}$s^{-1}			Minimum pH
	SO_4	Ca, Mg	SO_4	Ca	Mg	
0.18	134-150	134-150	13	7.8	6.3	6.29
0.18	136-150	132-148	12	7.4	6.9	6.30
0.22	134-150	134-146	11	6.7	8.9	6.52
0.22	134-150	132-148	11	10	7.6	6.33
0.40	135-150	134-150	24	25	0.79	6.17
0.40	134-150	136-148	22	23	0.79	6.30
0.41	136-150	127-150	14	6.0	5.4	5.30
0.41	134-150	132-144	14	5.6	5.0	5.32
0.51	136-150	127-150	16	6.2	6.0	5.08
0.51	136-150	132-148	18	6.3	5.0	4.98
0.57	134-150	134-150	24	6.3	12	5.38
0.57	137-150	132-148	26	8.0	14	5.35
0.58	140-150	134-150	28	7.6	12	4.85
0.58	136-150	132-148	32	9.4	14	4.82
0.71	135-150	127-146	22	4.9	13	4.98
0.71	137-150	132-148	20	4.0	10	5.05
1.12	35-69	53-65	(100)[2]	(62)	(5.8)	4.62
1.12	37-69	51-65	(97)	(61)	(7.0)	4.75
1.16	53-69	47-69	48	11	17	4.69
1.16	50-69	51-67	51	17	16	4.60
1.40	35-57	46-65	79	29	21	4.78
1.40	33-69	51-67	75	25	24	4.92
1.44	37-69	53-65	56	9.8	19	4.59
1.44	36-69	51-65	56	11	20	4.41
1.63	117-145[1]	138-150	62	8.1	23	4.37
1.63	119-150	127-148	180	12	35	3.90
1.64	55-69	53-69	76	18	30	4.42
1.64	25-69	51-67	95	21	32	4.32

1 dSO_4/dt = 83 for wks 146-150

2 Parentheses indicate anomalous value.

Table A1.6. Comparison of field and laboratory drill core data.

Percent Sulfur	Field Rate			Min Field pH	Lab Rates			Min Lab pH	Retardation Factor				
	SO_4	Ca	Mg		SO_4	Ca	Mg		SO_4	Ca	Mg	Ca+Mg	Avg.
0.63 (FL1)[1]	3.9	2.4	0.91	4.96	25	27	6.7	6.8-7.3	0.16	0.089	0.14	0.098	0.13
0.63 (FL2)[1]	2.3	1.3	0.64	4.76	25	27	6.7	6.8-7.3	0.092	0.048	0.096	0.058	0.079
0.63 (FL3)[1]	6.4	2.8	2.4	4.71	25	27	6.7	6.8-7.3	0.26	0.10	0.36	0.15	0.24
0.79[2]	8.1	2.6	4.2	3.70	36	32	8.6	6.8-6.9	0.23	0.081	0.49	0.17	0.27
1.41[3]	11	2.4	3.5	3.08	170	28	27	4.2-4.4	0.065	0.086	0.13	0.11	0.094

1 Lab rates for 0.63-percent-sulfur determined by interpolation of rates from 0.59 and 0.80-percent-sulfur samples (17 week period of record) and assuming rates varied linearly with sulfur content.

2 Lab rates for 0.79-percent-sulfur from 0.80-percent-sulfur sample (17 week period of record).

3 Lab rates for 1.41-percent-sulfur sample determined by averaging rates for samples with sulfur contents of 1.35 (34 week period of record) and 1.47 (21 week period of record) percent.

Figure A2.1. pH over time for Duluth Complex test pile FL1 (0.63% S).

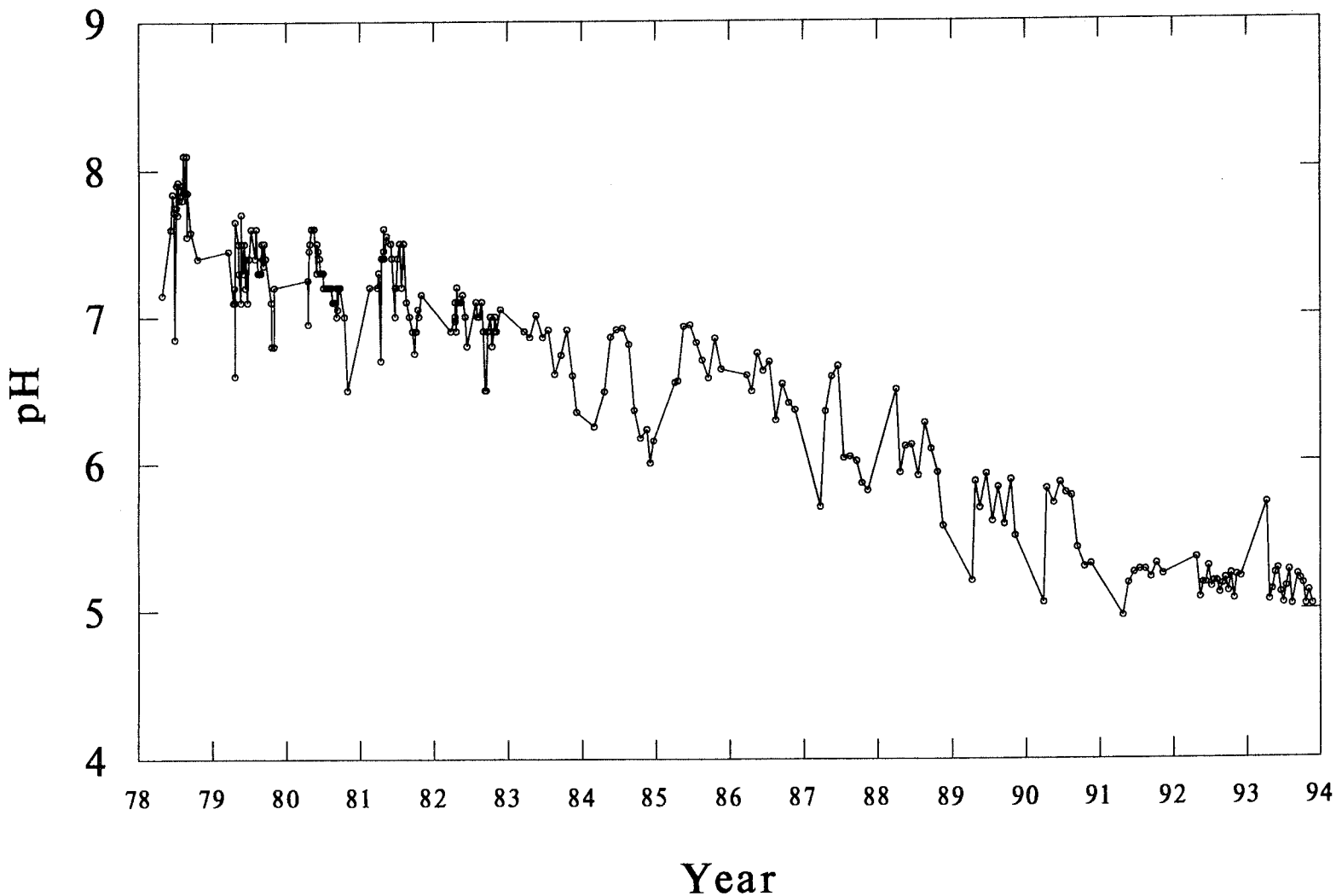

Figure A2.2. Sulfate concentrations over time for Duluth Complex test pile FL1 (0.63% S).

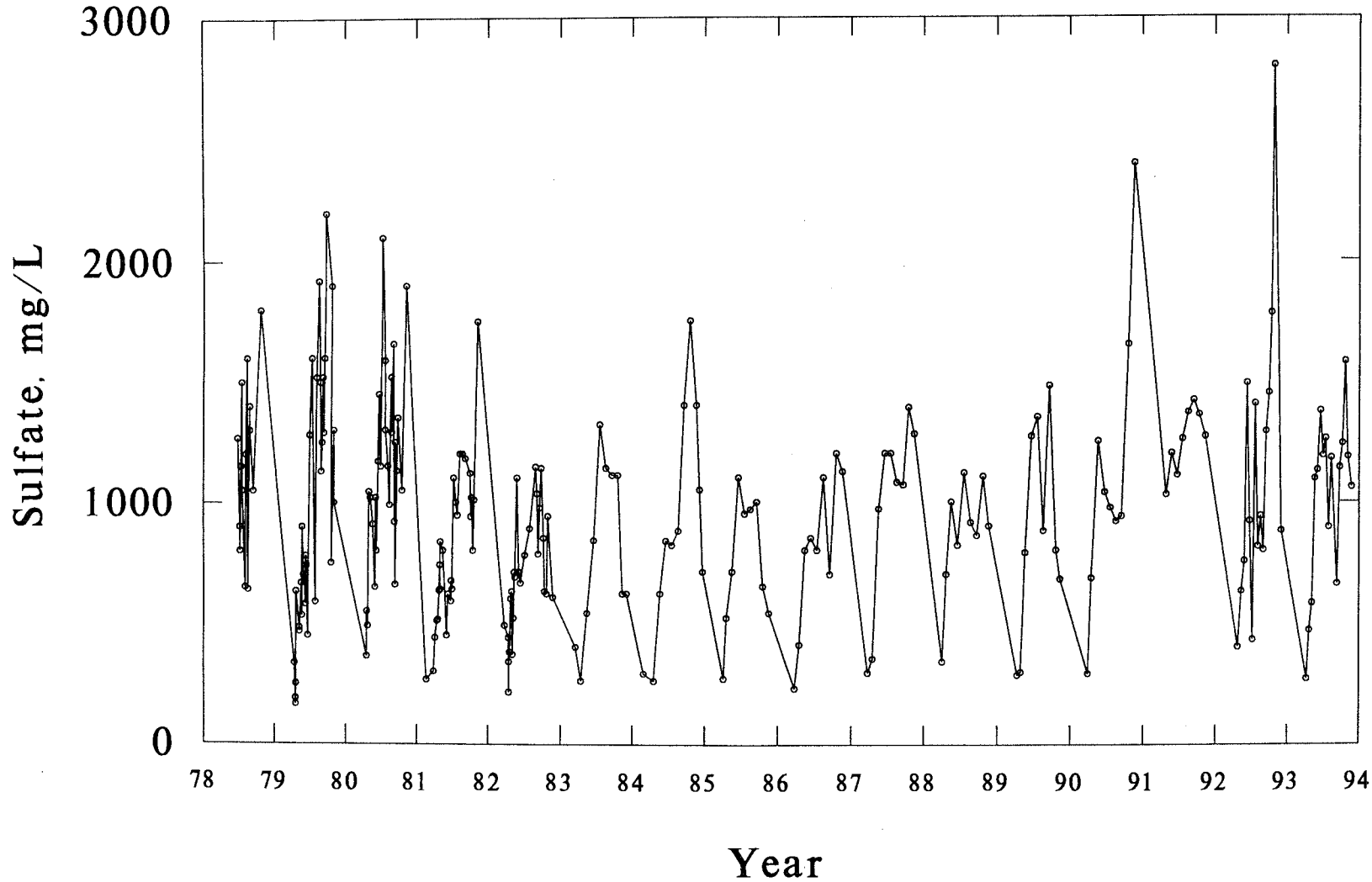

Figure A2.3. Calcium concentrations over time for Duluth Complex test pile FL1 (0.63% S).

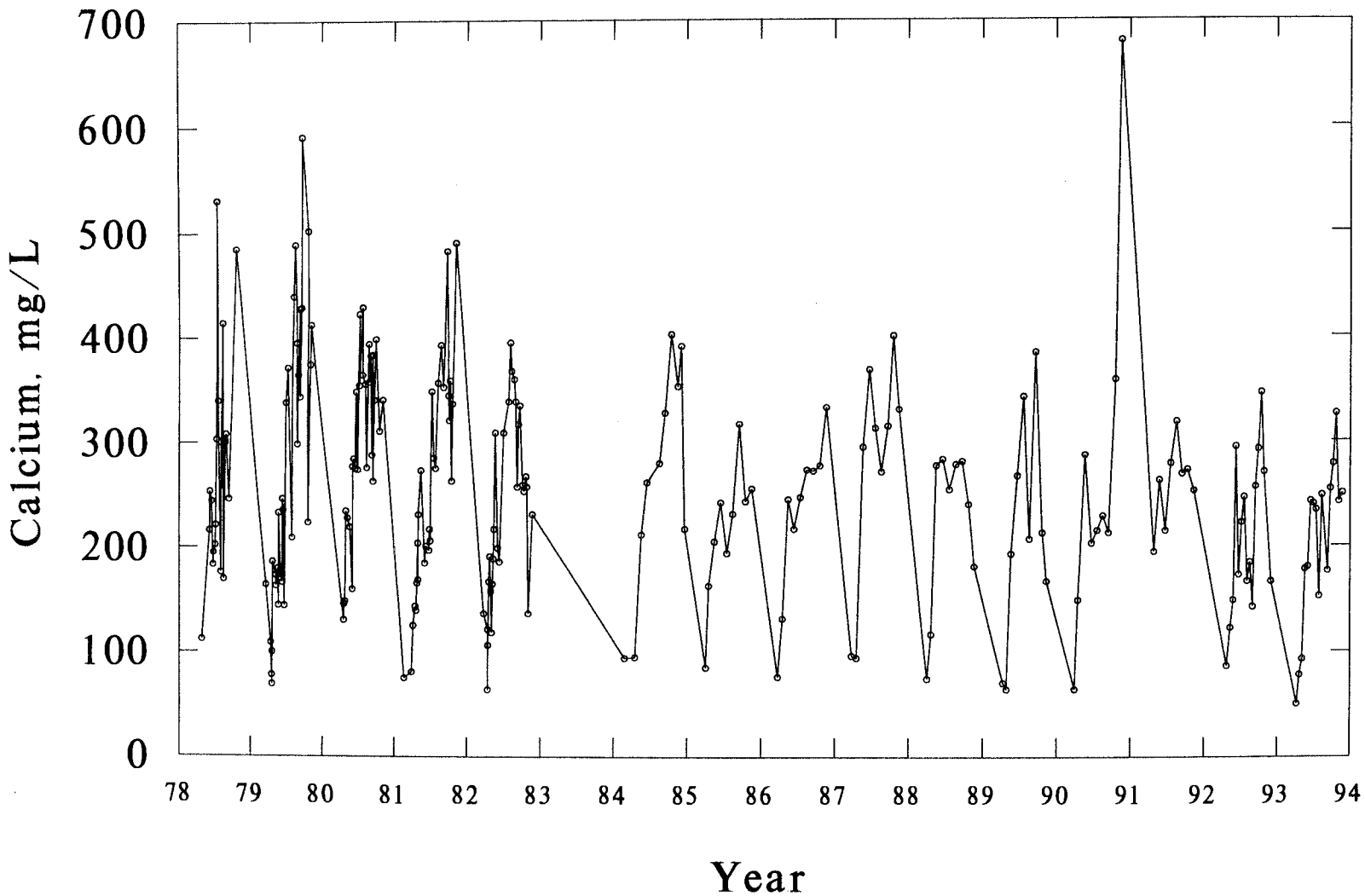

Figure A2.4. Magnesium concentrations over time for Duluth Complex test pile FL1 (0.63% S).

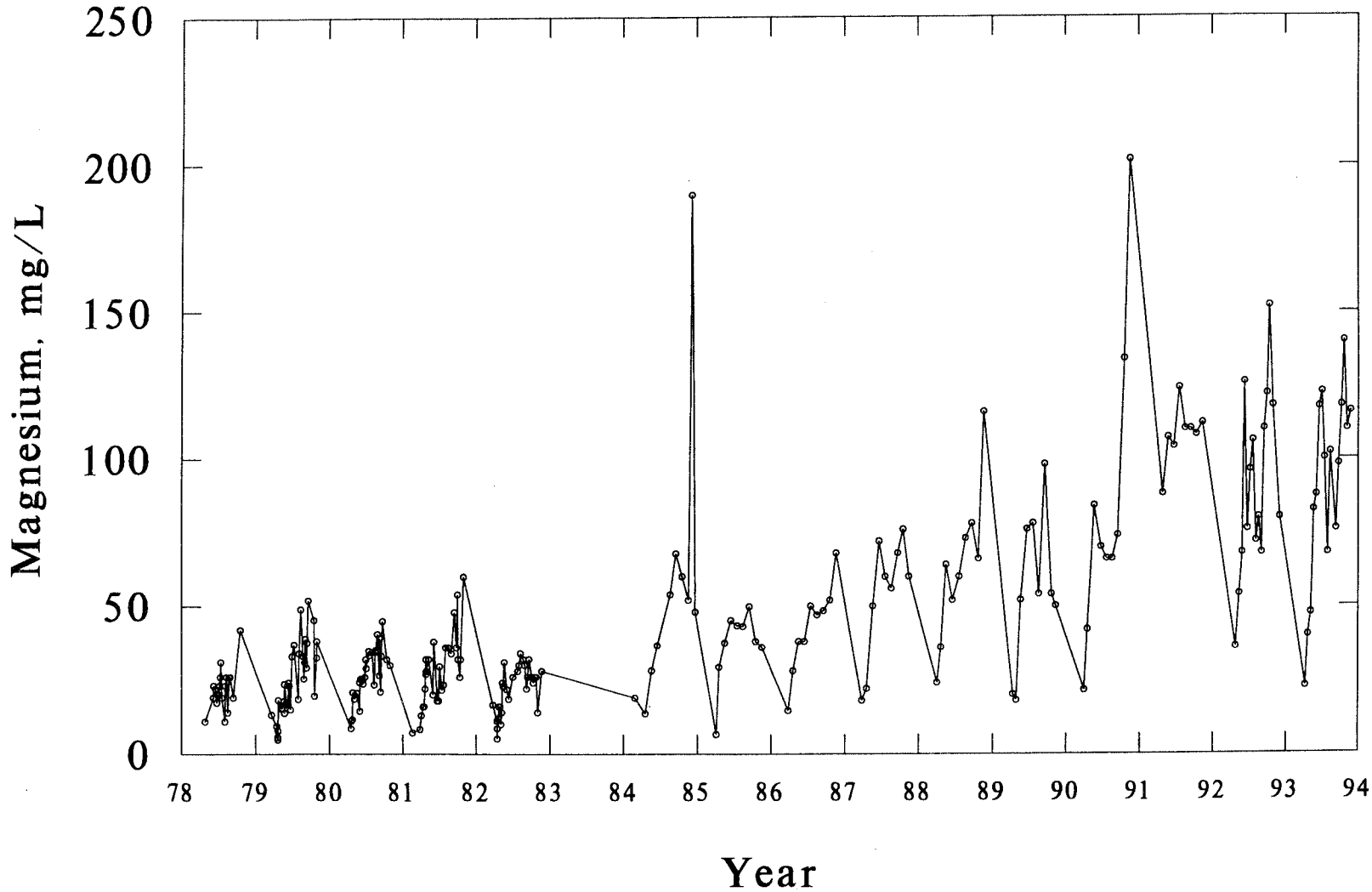

Figure A2.5. pH over time for Duluth Complex test pile FL2 (0.63% S).

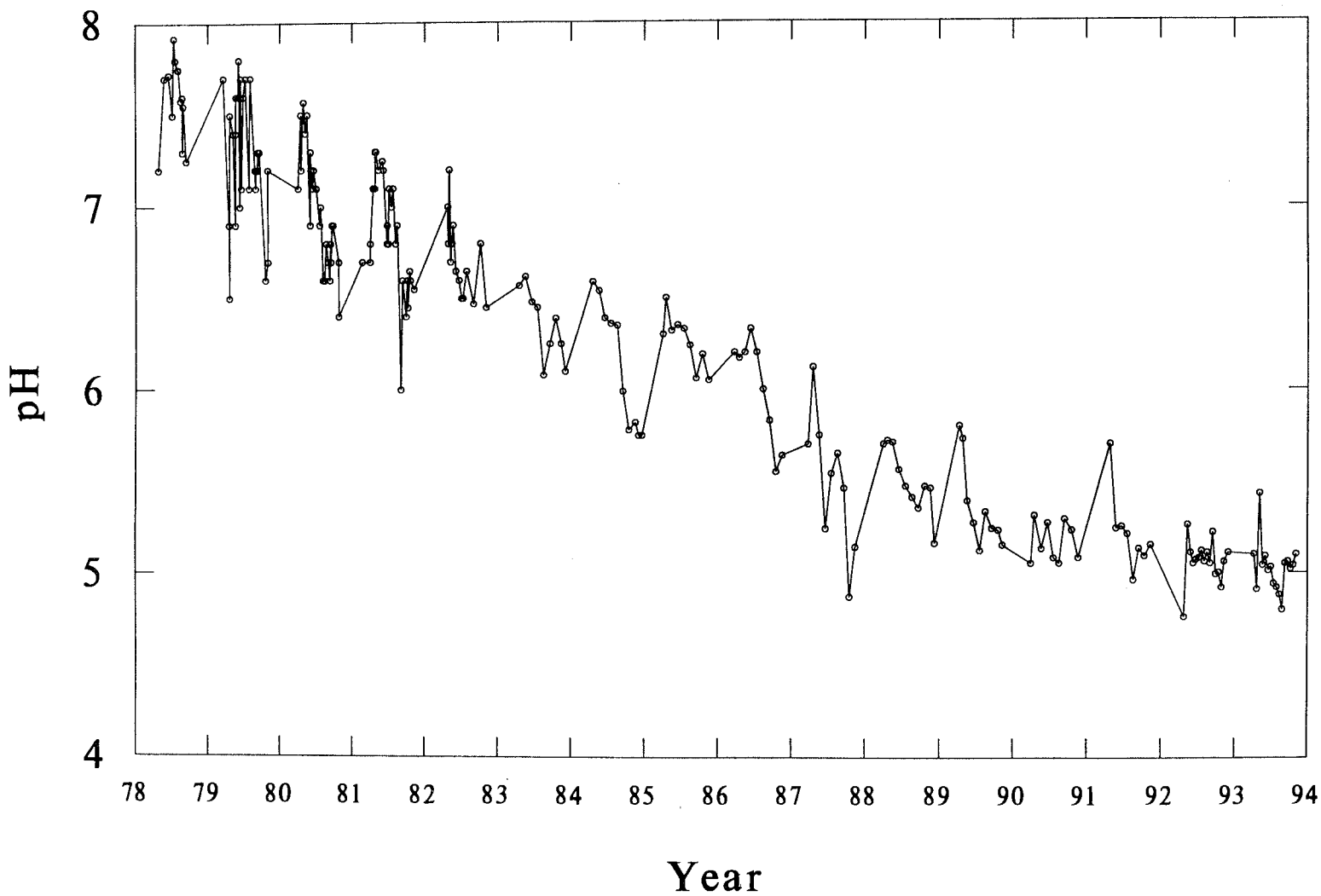

Figure A2.6. Sulfate concentrations over time for Duluth Complex test pile FL2 (0.63% S).

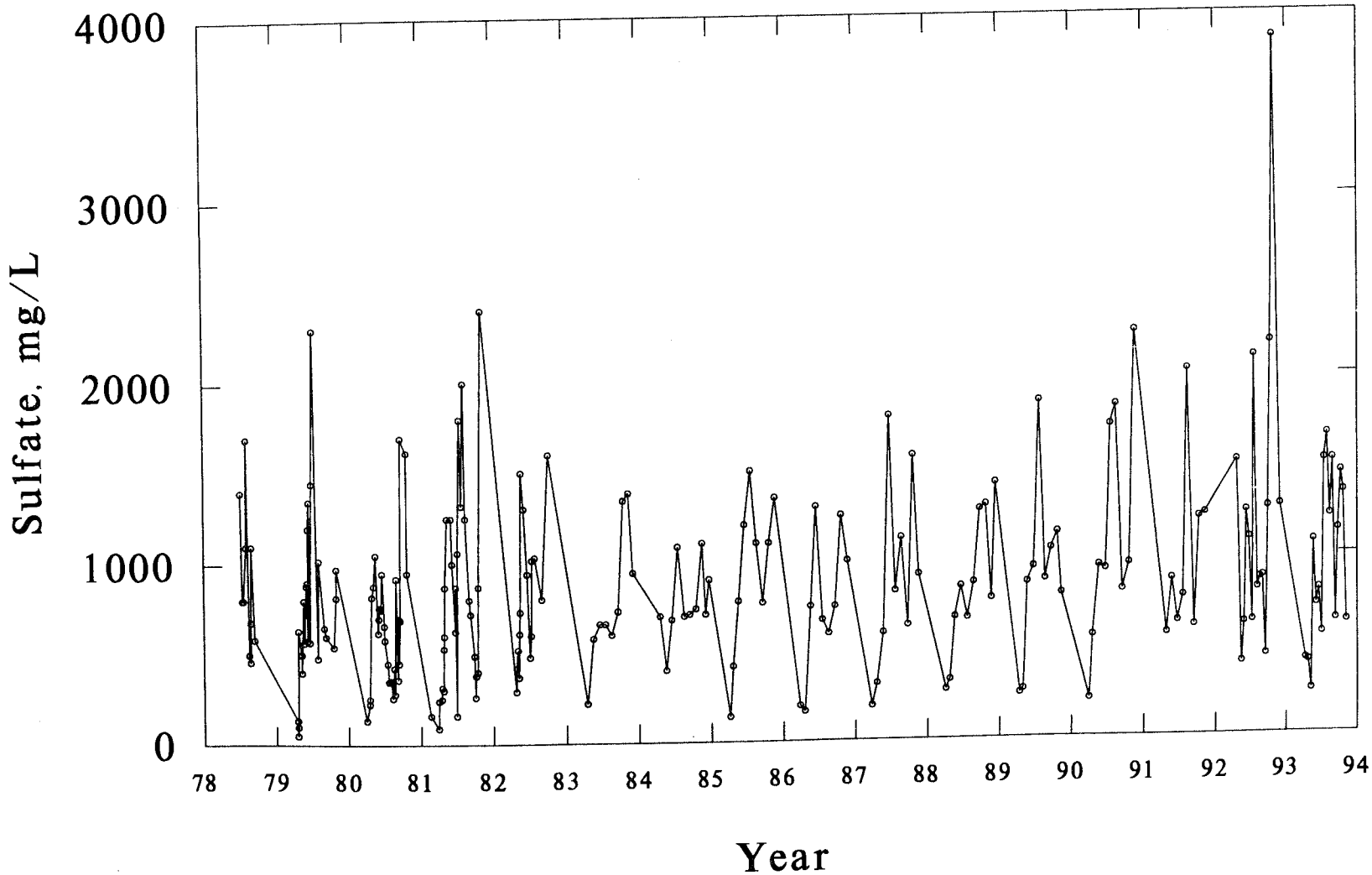

Figure A2.7. Calcium concentrations over time for Duluth Complex test pile FL2 (0.63% S).

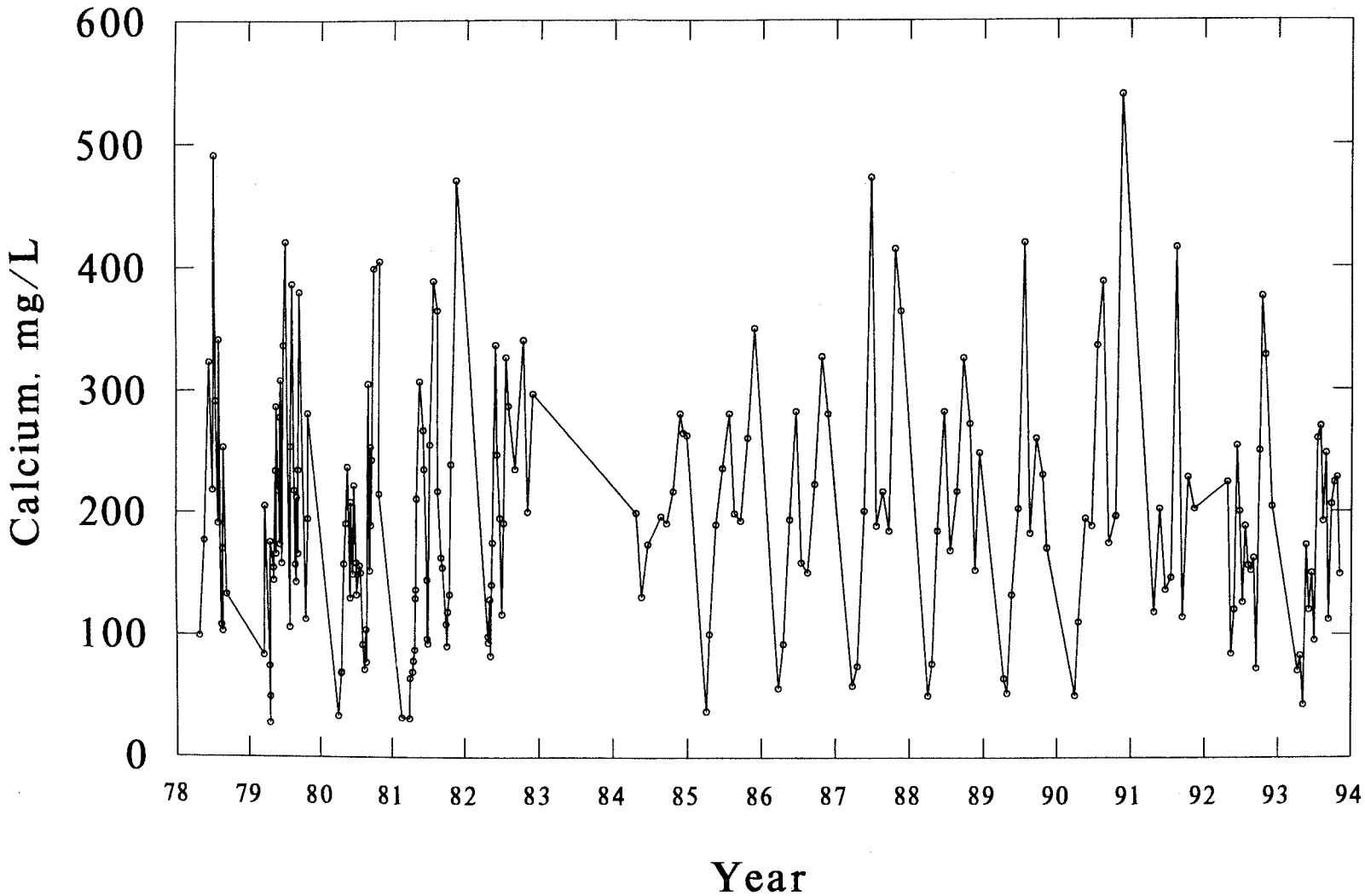

Figure A2.8. Magnesium concentrations over time for Duluth Complex test pile FL2 (0.63% S).

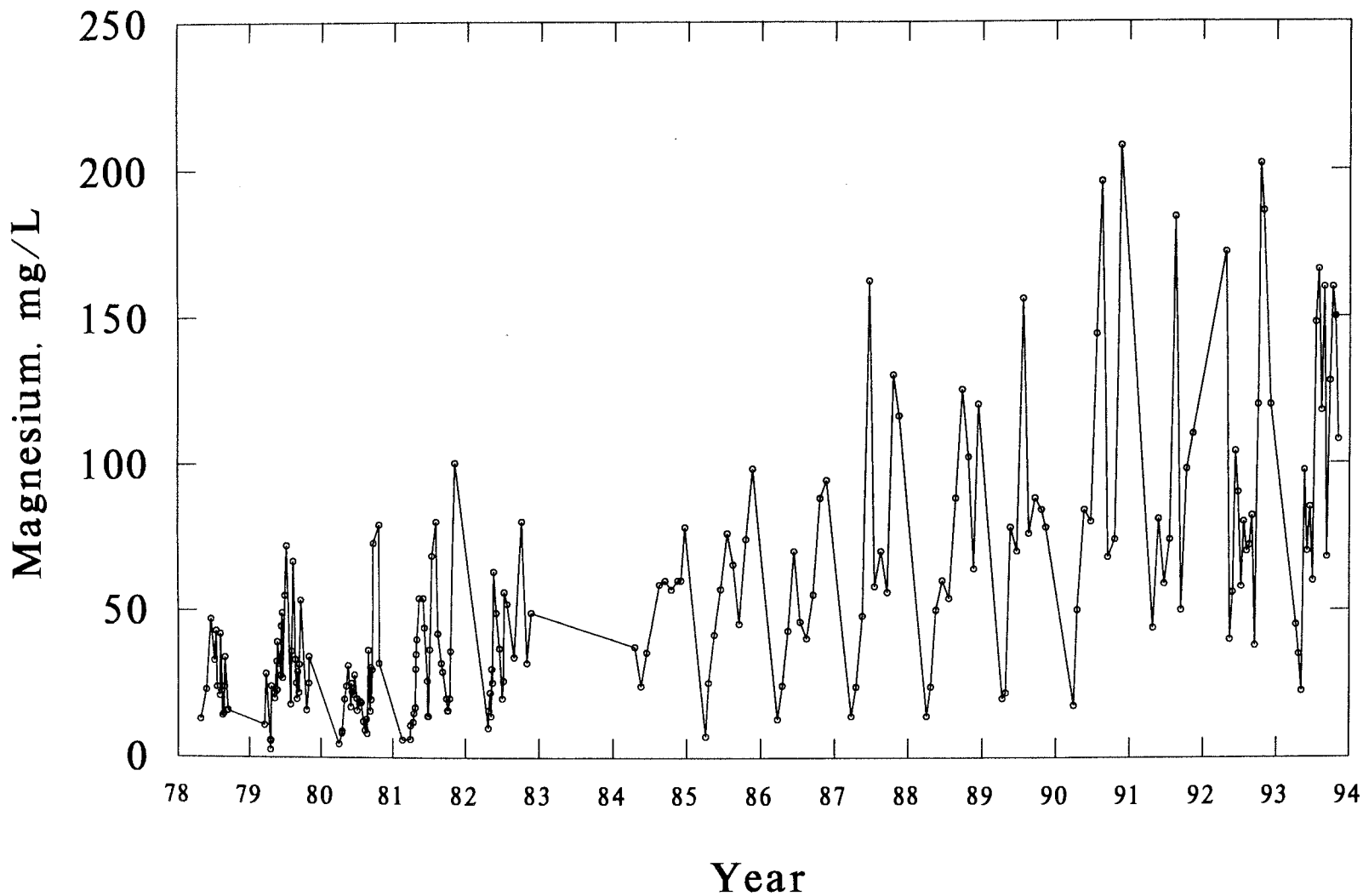

Figure A2.9. pH over time for Duluth Complex test pile FL3 (0.63% S).

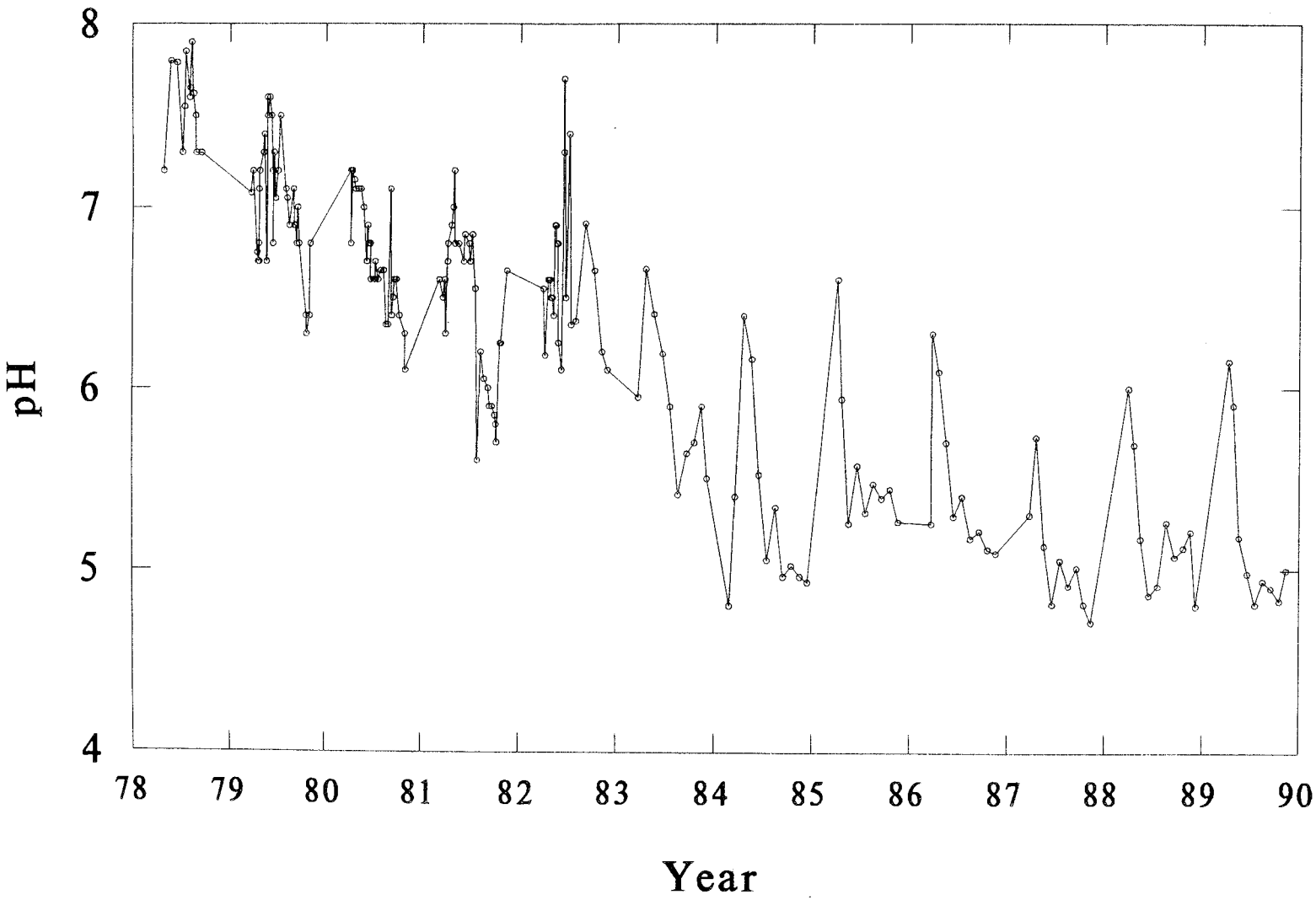

Figure A2.10. Sulfate concentrations over time for Duluth Complex test pile FL3 (0.63% S).

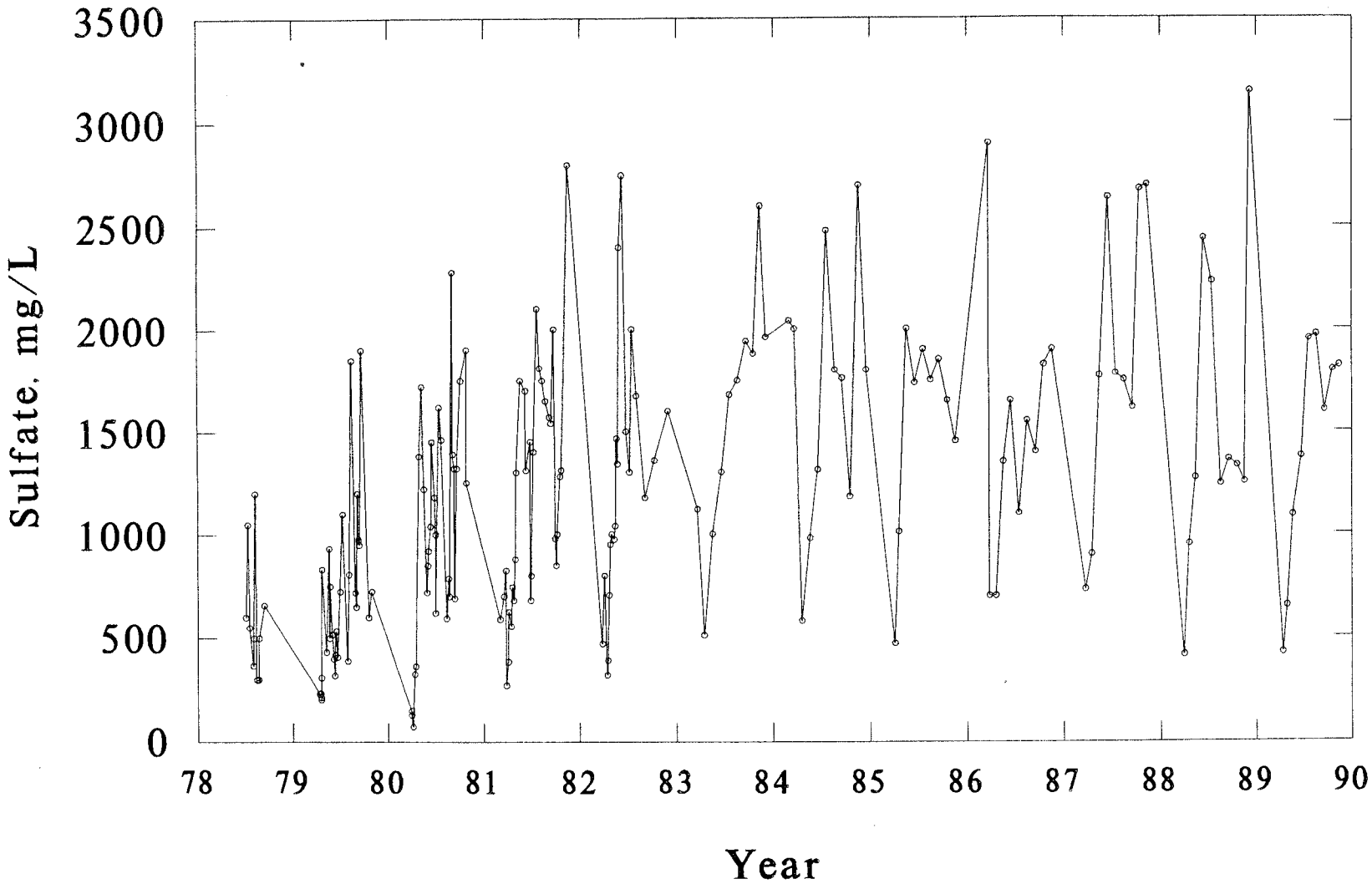

Figure A2.11. Calcium concentrations over time for Duluth Complex test pile FL3 (0.63% S).

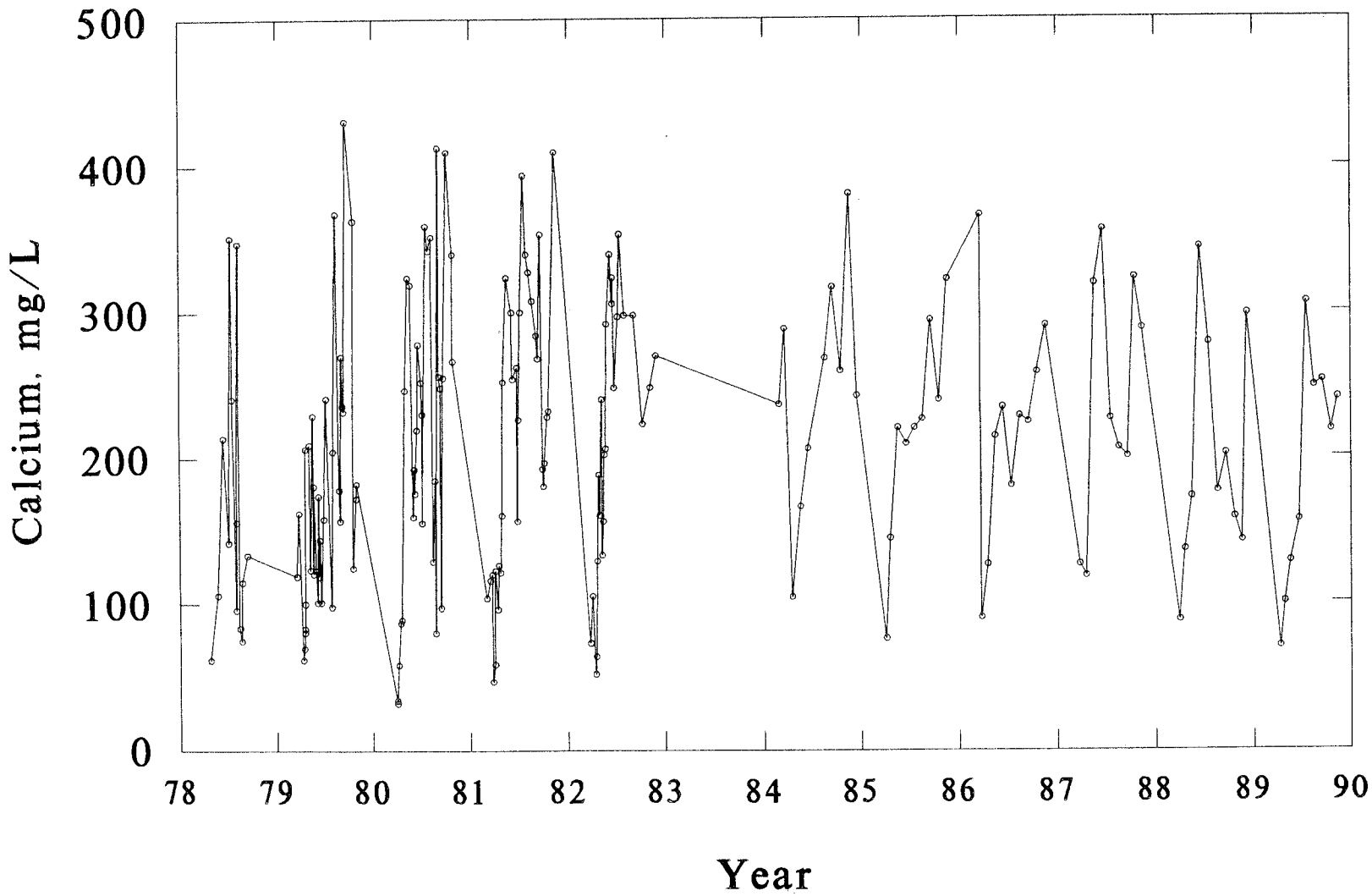

Figure A2.12. Magnesium concentrations over time for Duluth Complex test pile FL3 (0.63% S).

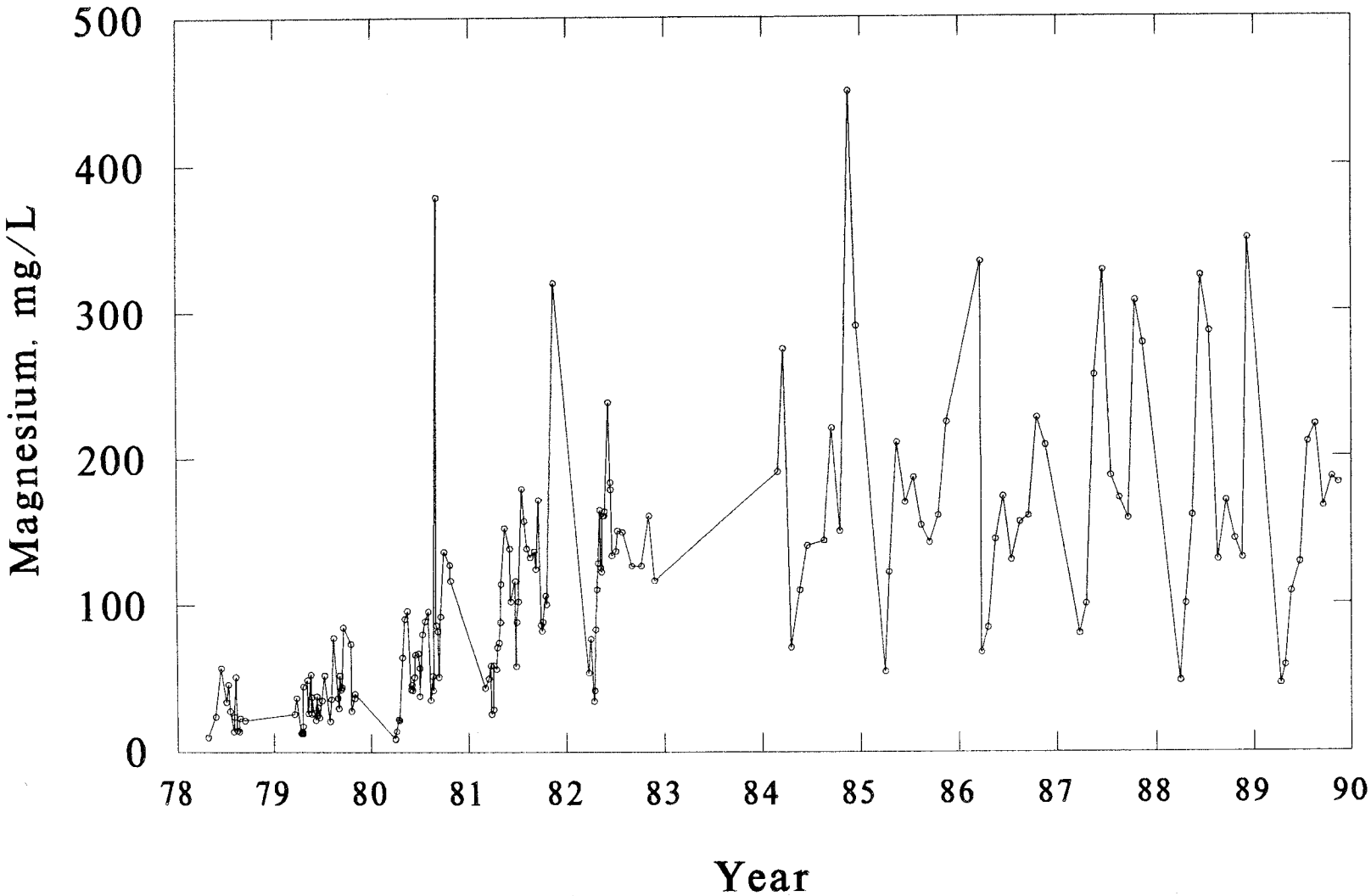

Figure A2.13. pH over time for Duluth Complex test pile FL4 (0.63% S).

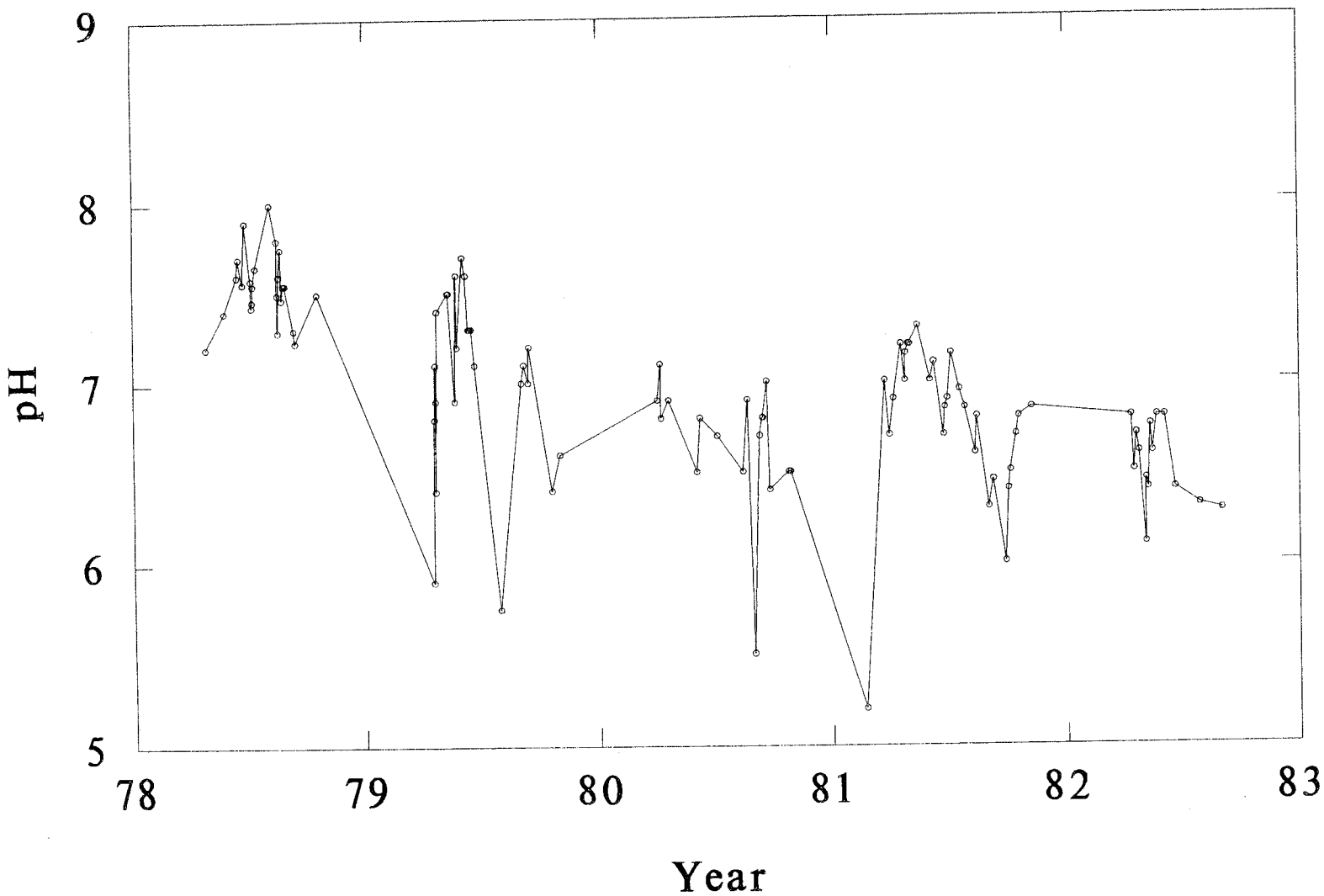

Figure A2.14. Sulfate concentrations over time for Duluth Complex test pile FL4 (0.63% S).

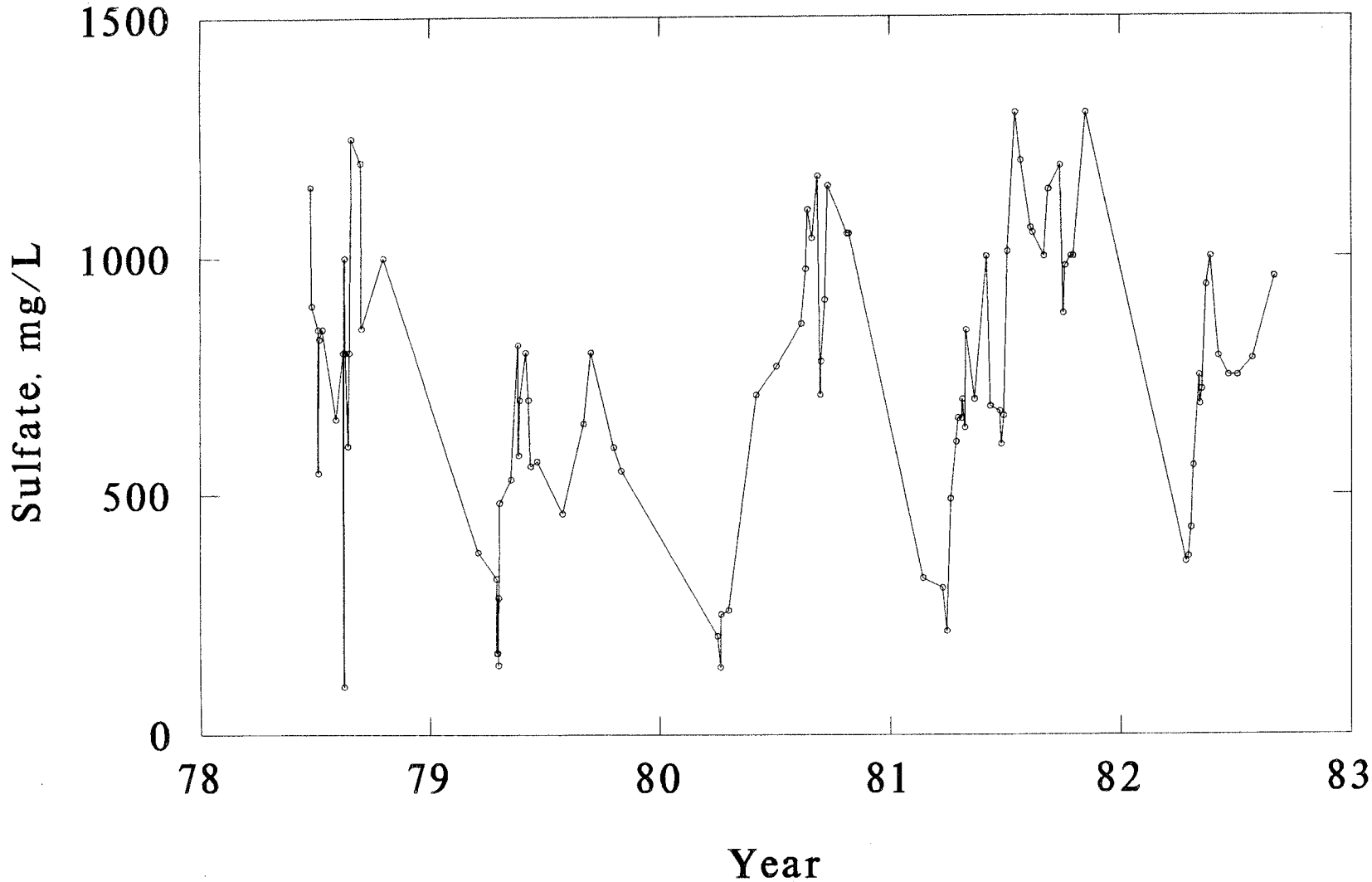

Figure A2.15. Calcium concentrations over time for Duluth Complex test pile FL4 (0.63% S).

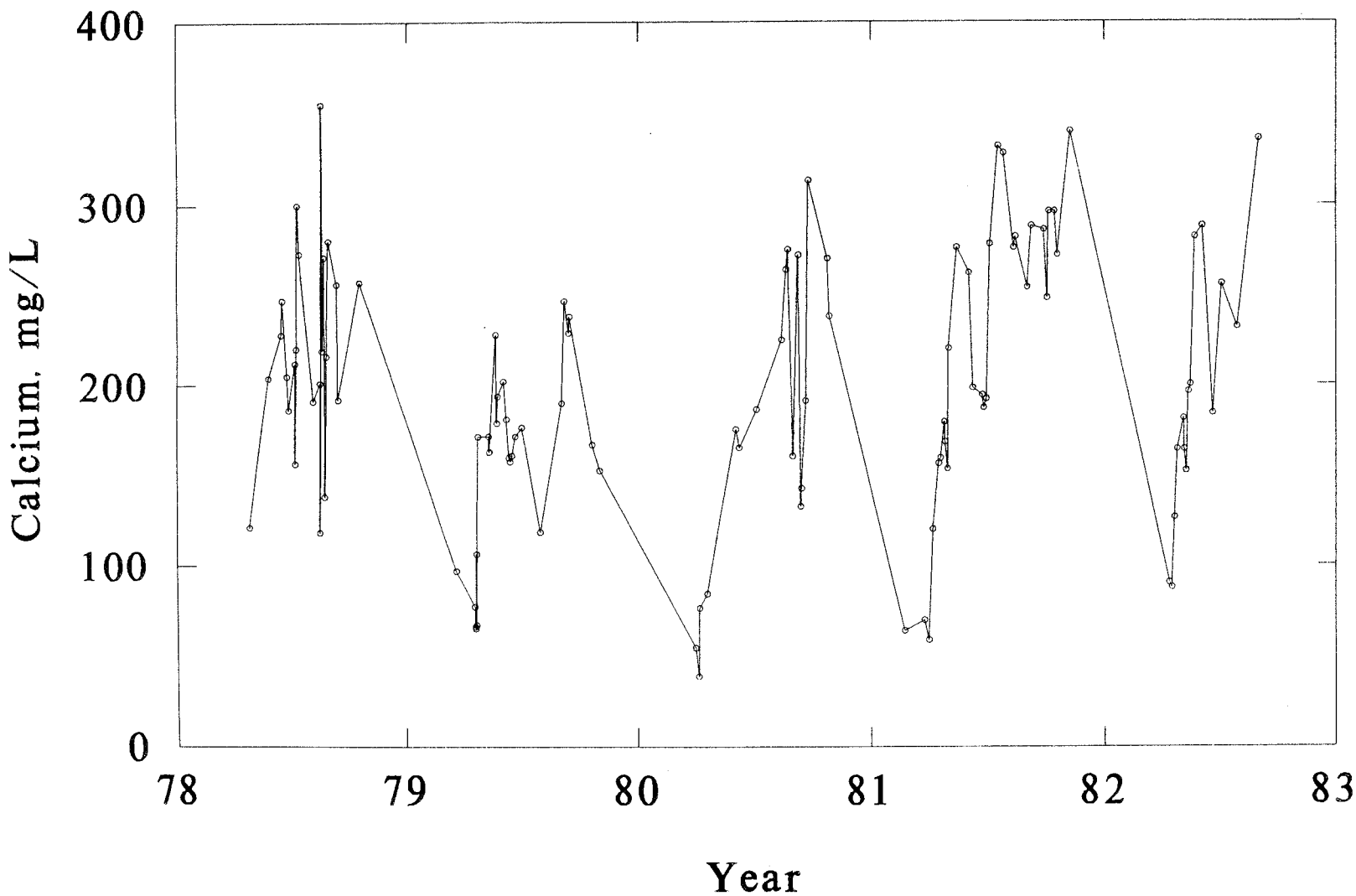

Figure A2.16. Magnesium concentrations over time for Duluth Complex test pile FL4 (0.63% S).

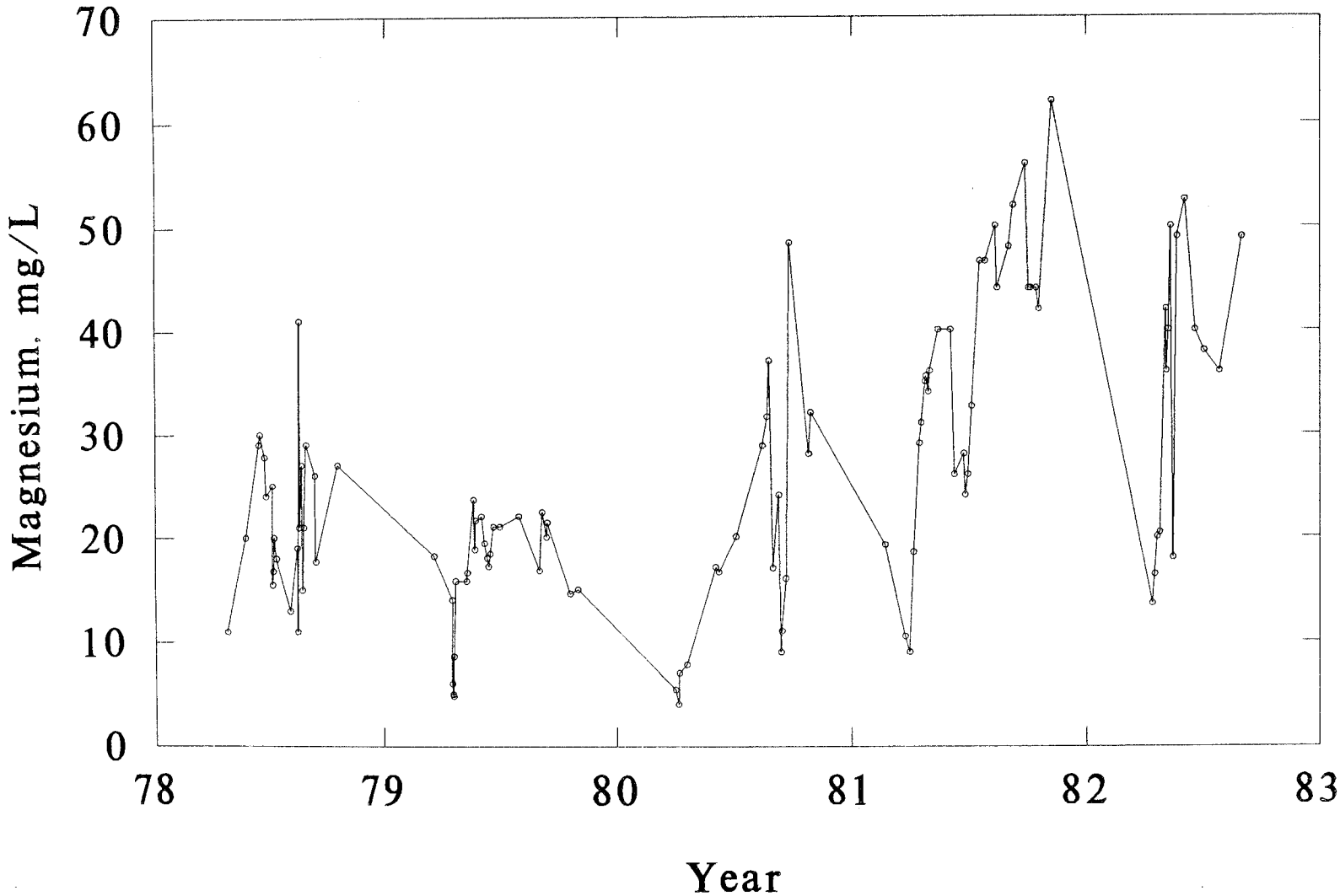

Figure A2.17. pH over time for Duluth Complex test pile FL6 (0.79% S).

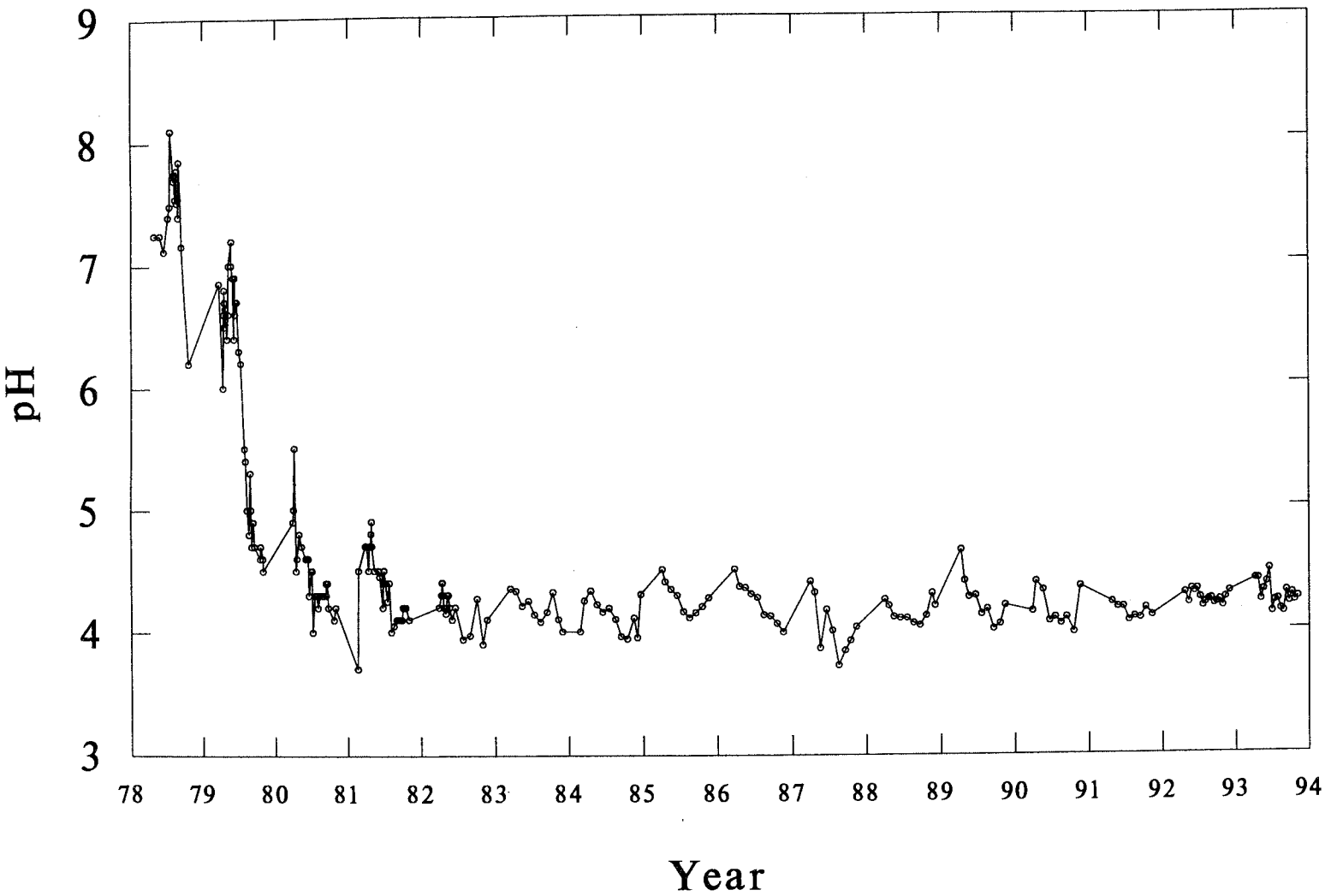

Figure A2.18. Sulfate concentrations over time for Duluth Complex test pile FL6 (0.79% S).

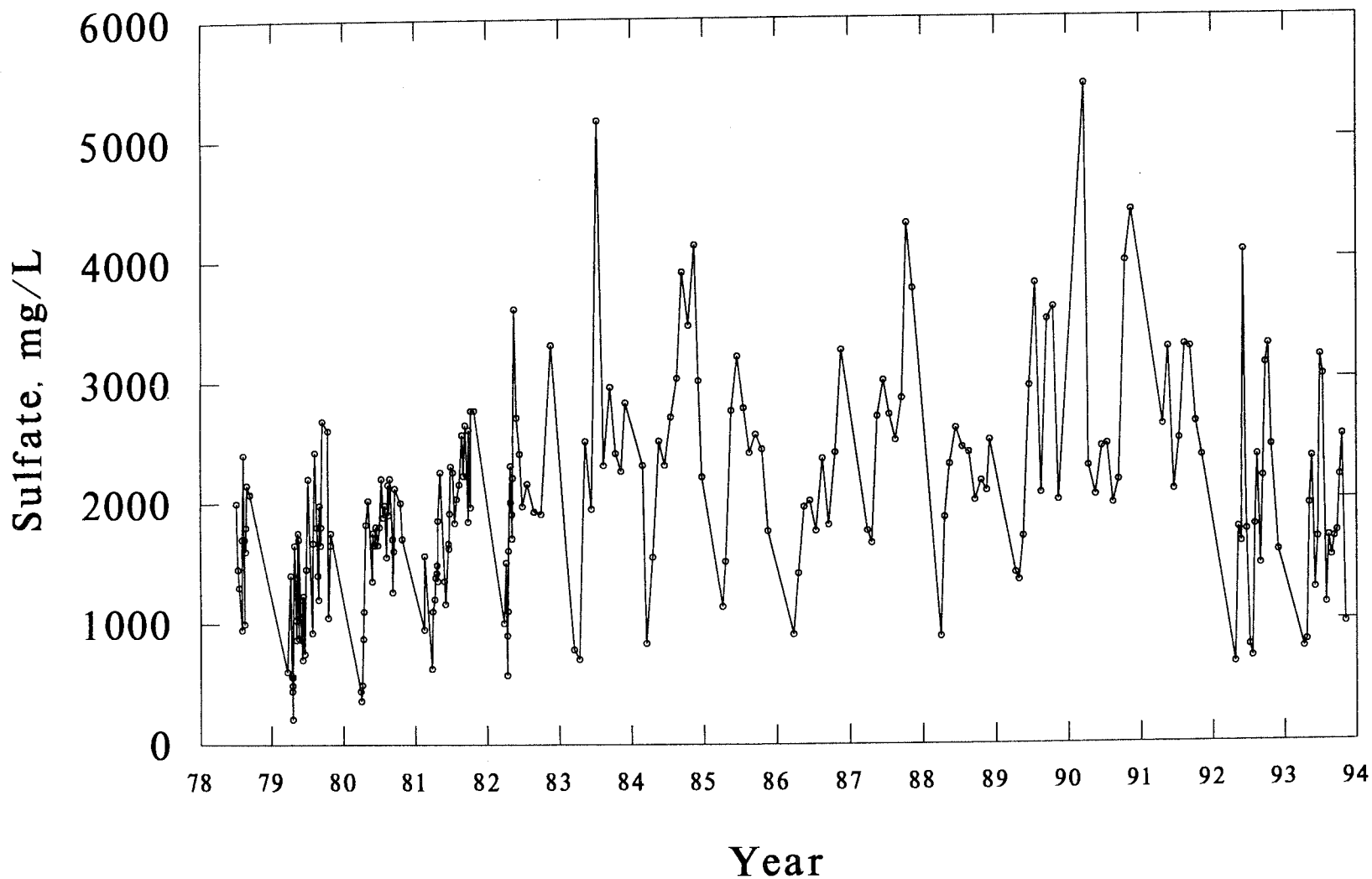

Figure A2.19. Calcium concentrations over time for Duluth Complex test pile FL6 (0.79% S).

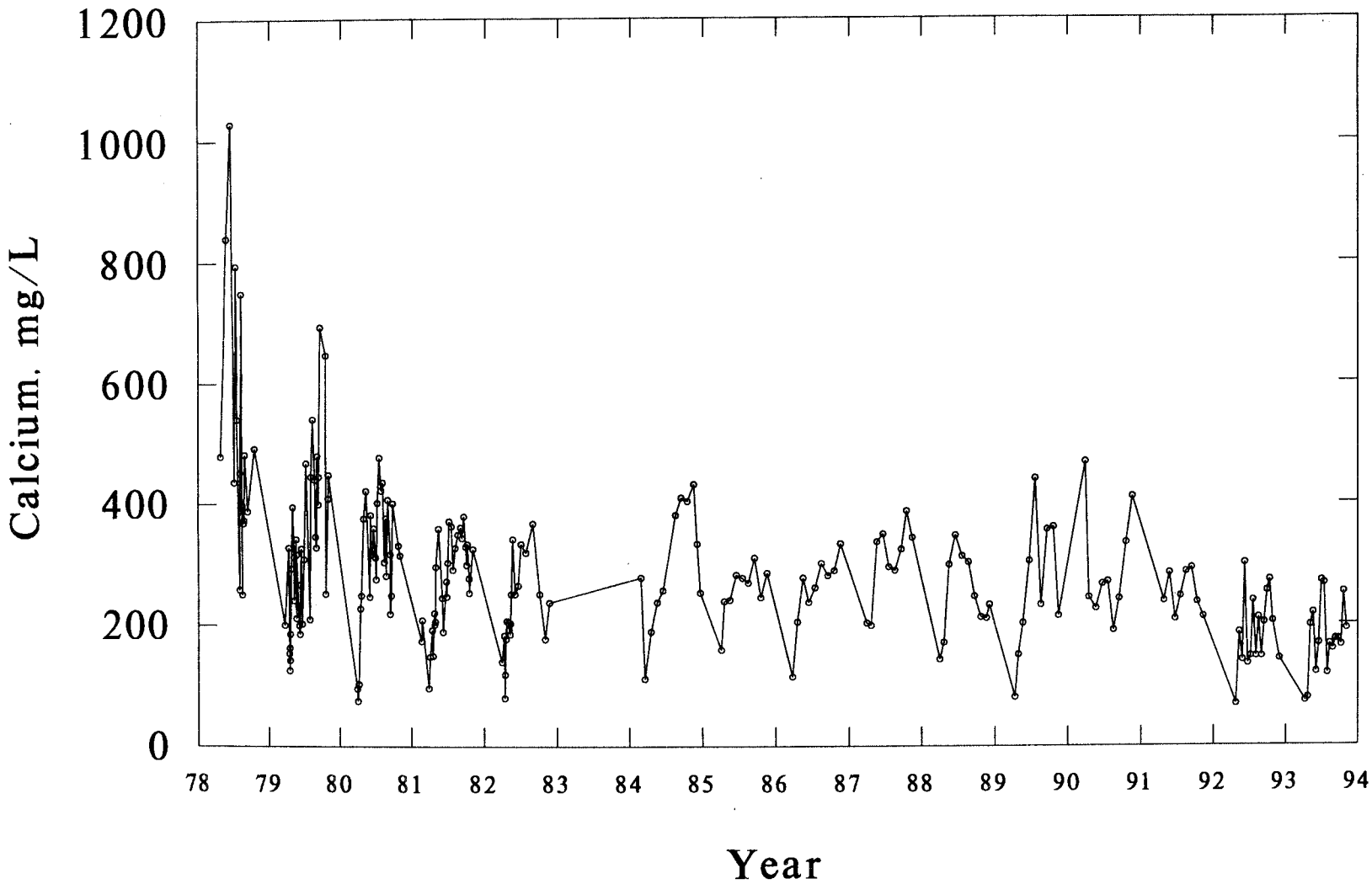

Figure A2.20. Magnesium concentrations over time for Duluth Complex test pile FL6 (0.79% S).

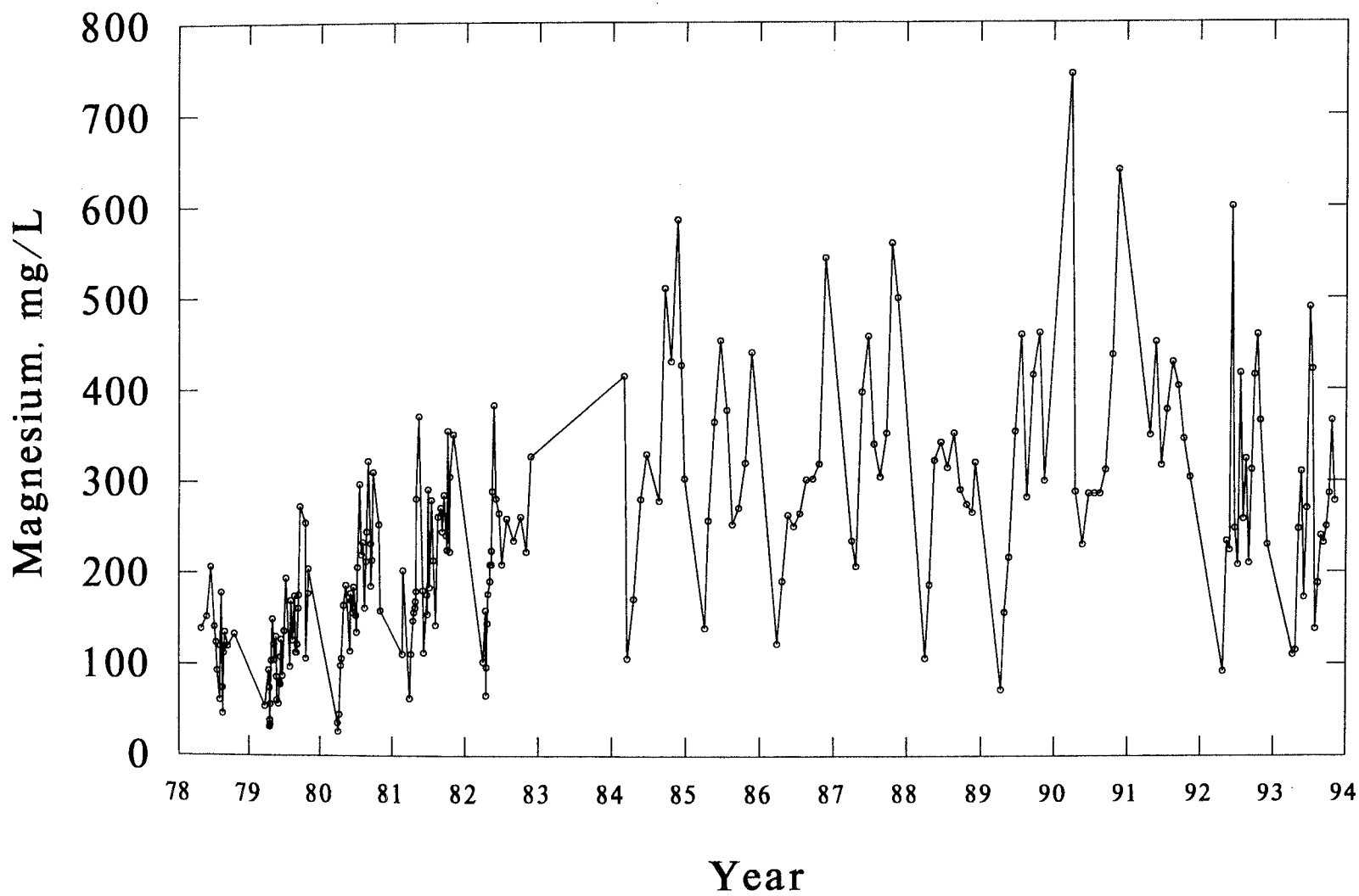

Figure A2.21. pH over time for Duluth Complex test pile FL5 (1.41% S).

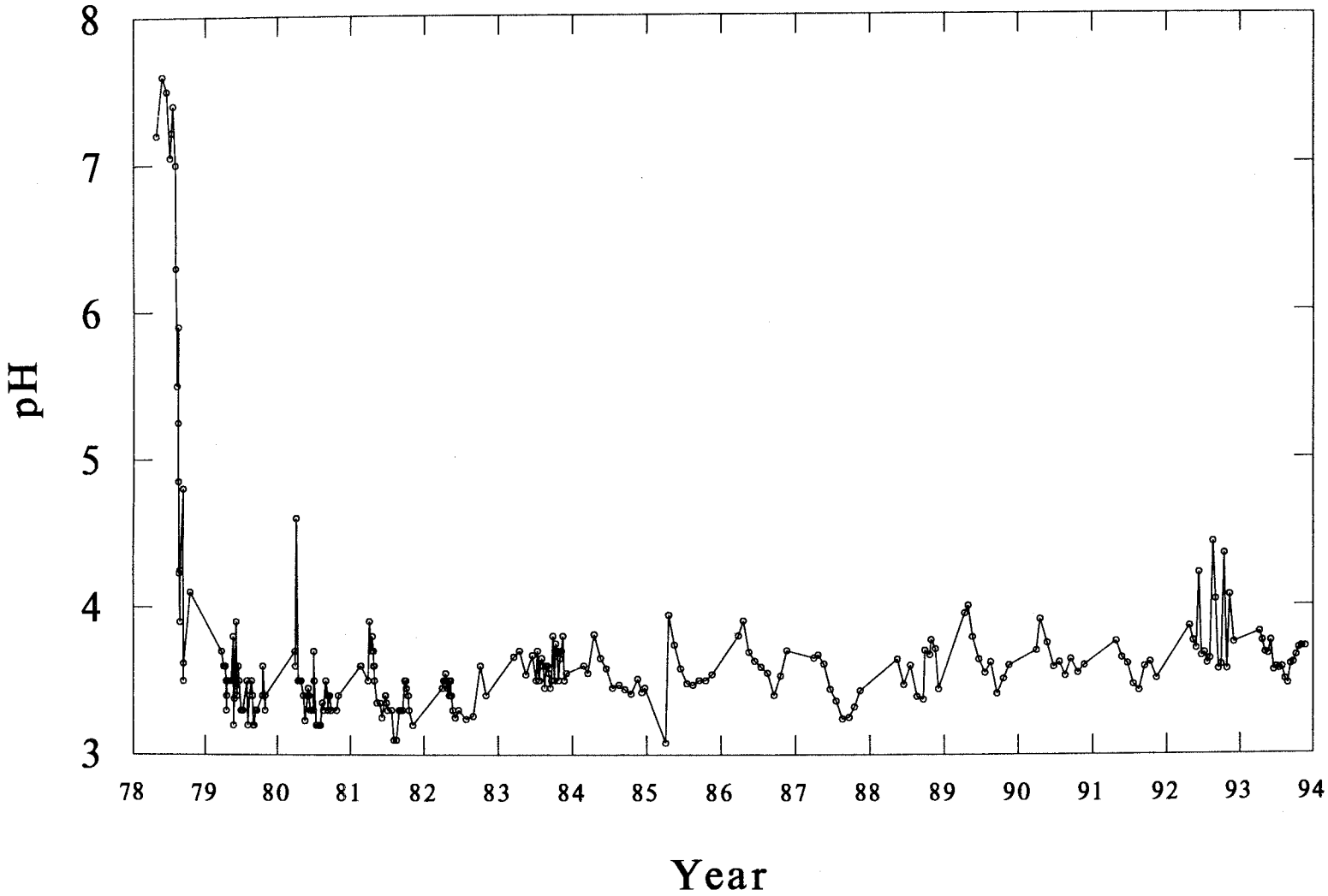

Figure A2.22. Sulfate concentrations over time for Duluth Complex test pile FL5 (1.41% S).

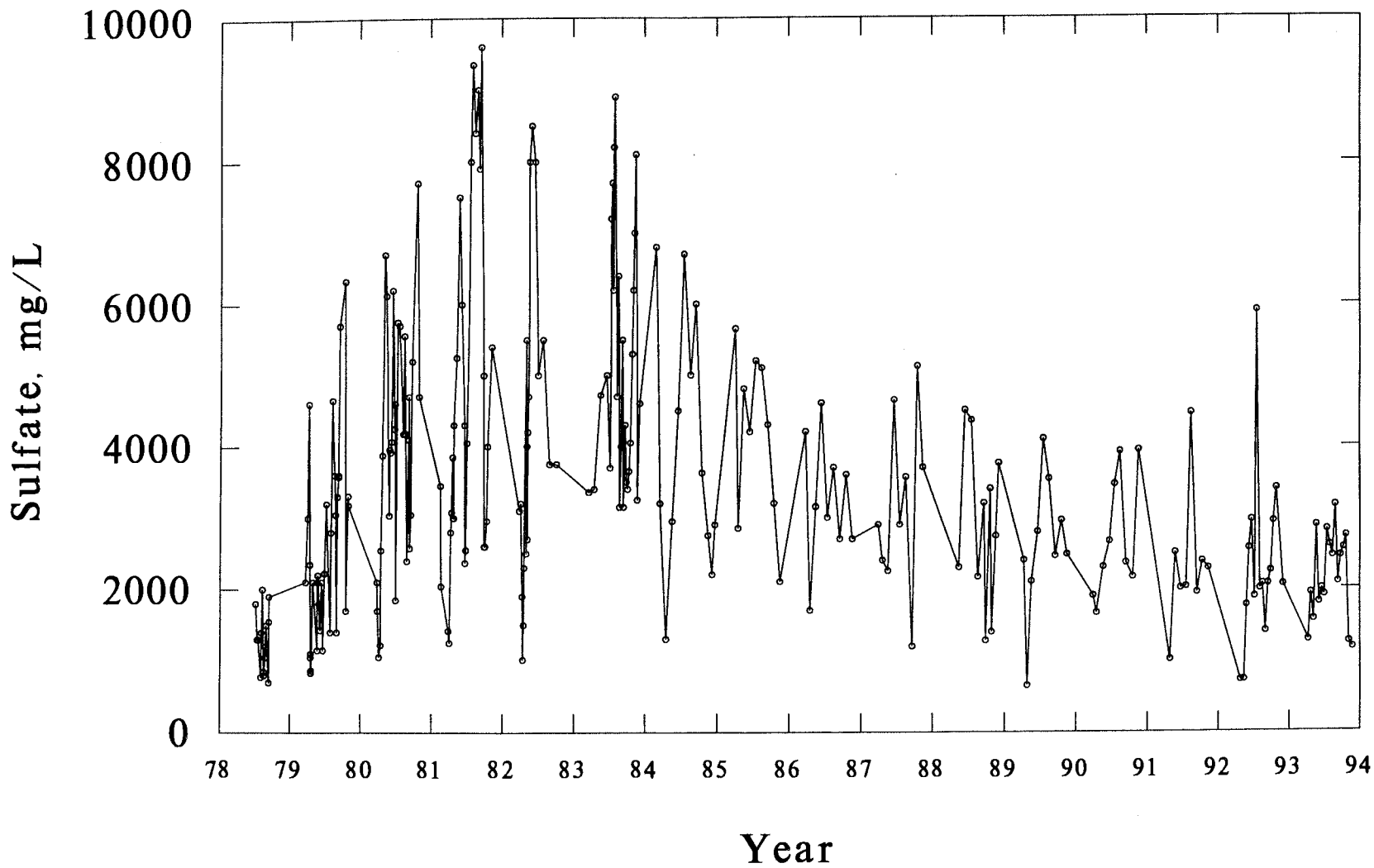

Figure A2.23. Calcium concentrations over time for Duluth Complex test pile FL5 (1.41% S).

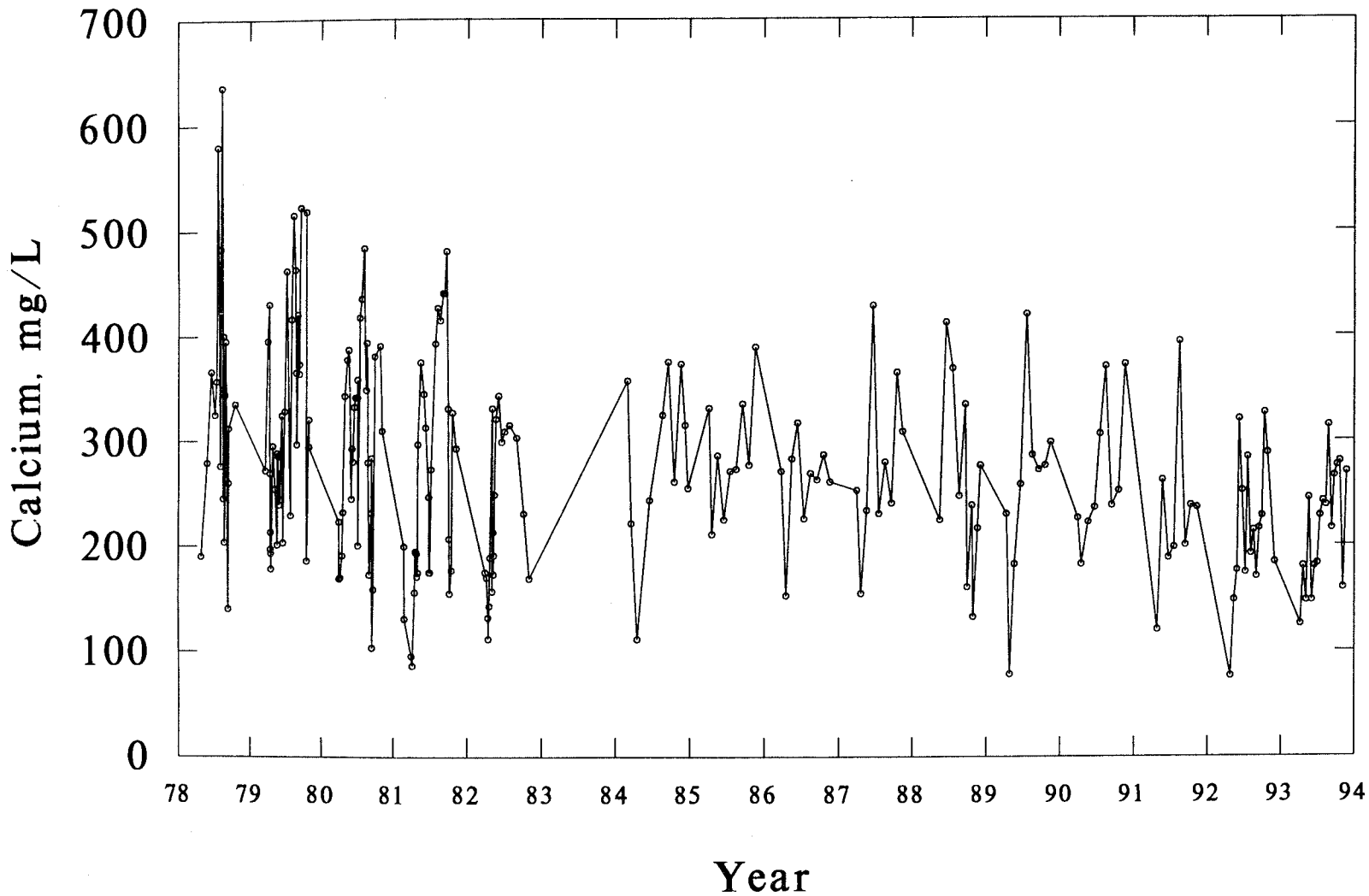

Figure A2.24. Magnesium concentrations over time for Duluth Complex test pile FL5 (1.41% S).

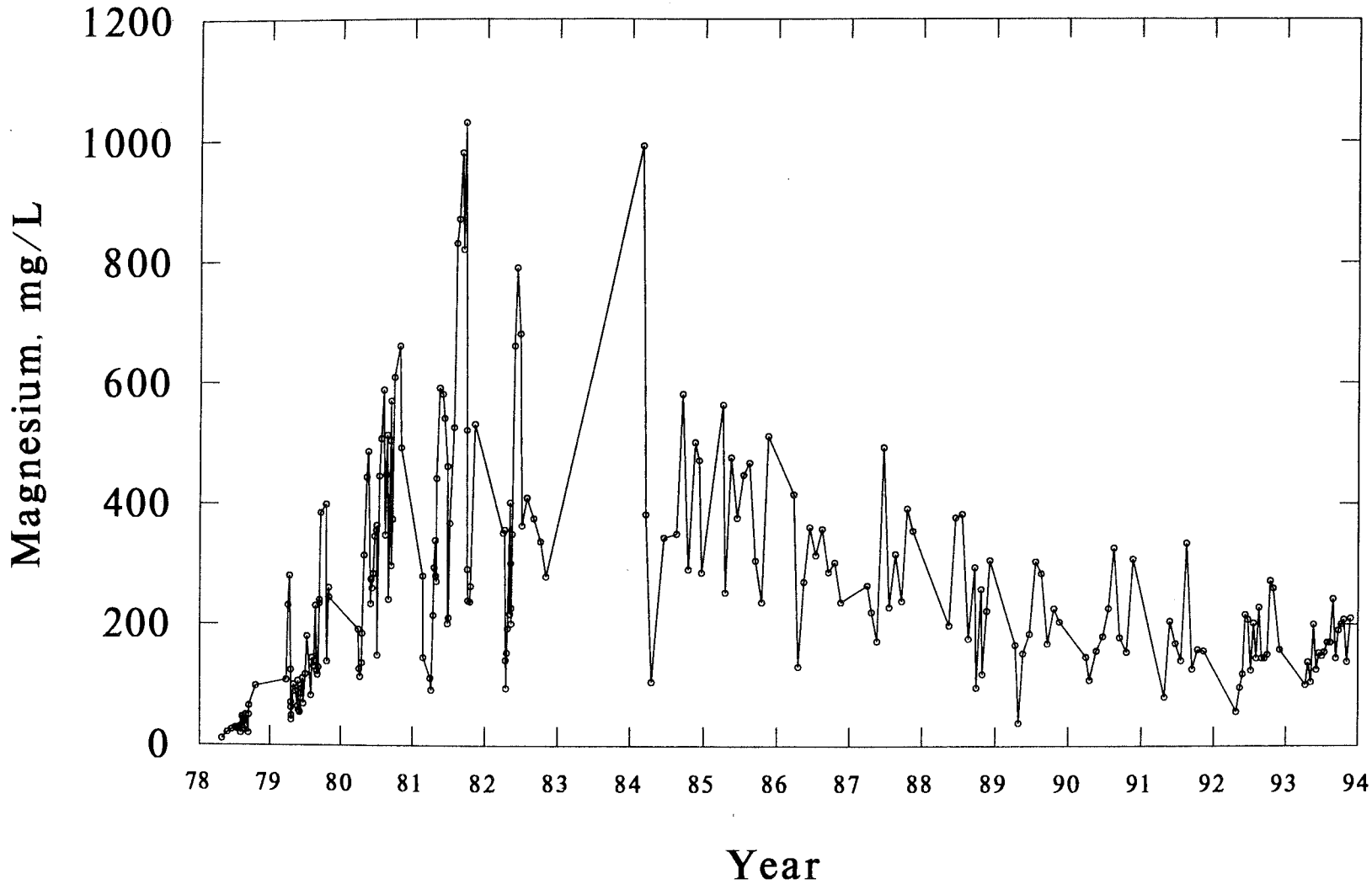

Figure A2.25. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual volume for Duluth Complex test pile FL1 (0.63% S).

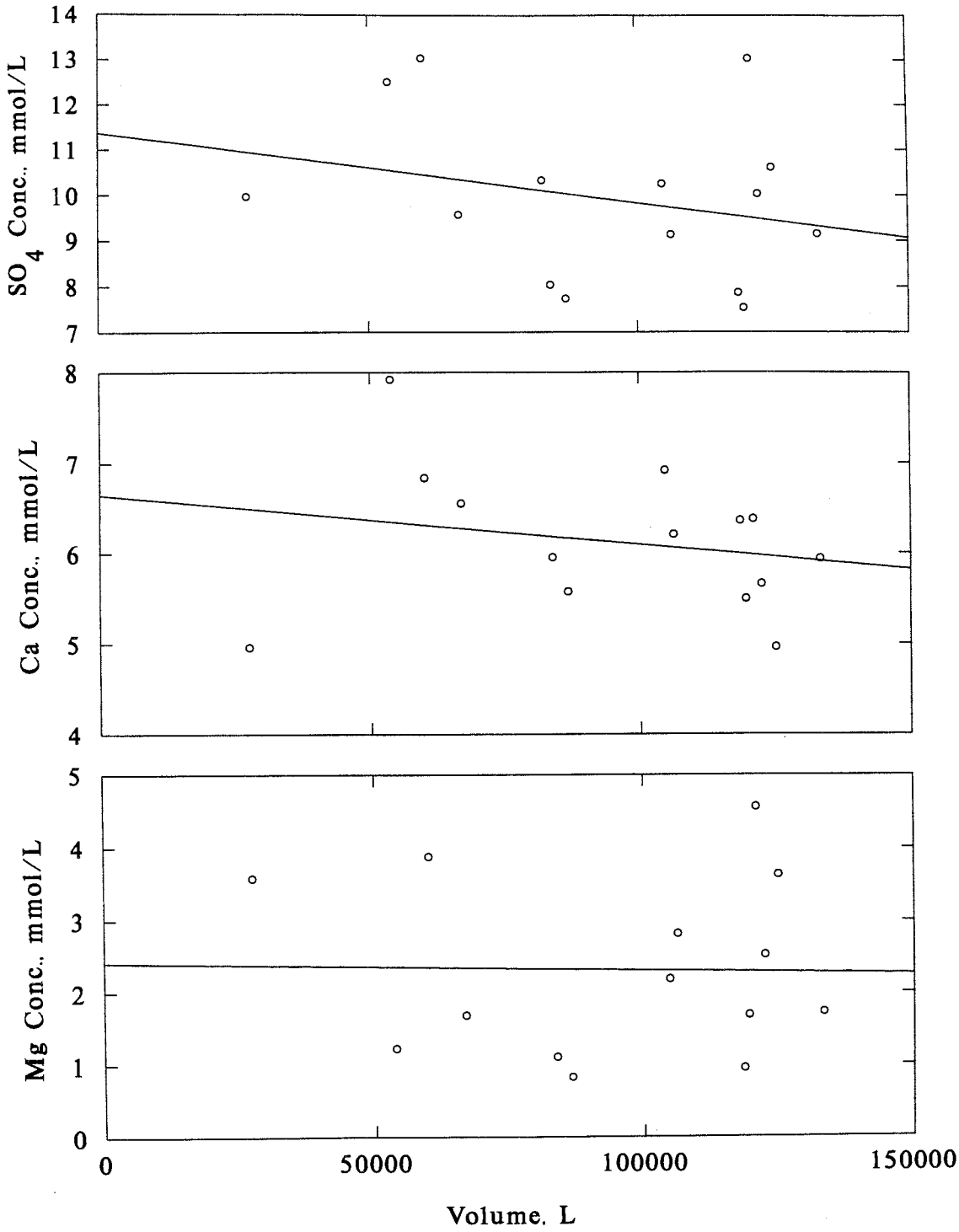

Figure A2.26. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual volume for Duluth Complex test pile FL1 (0.63% S).

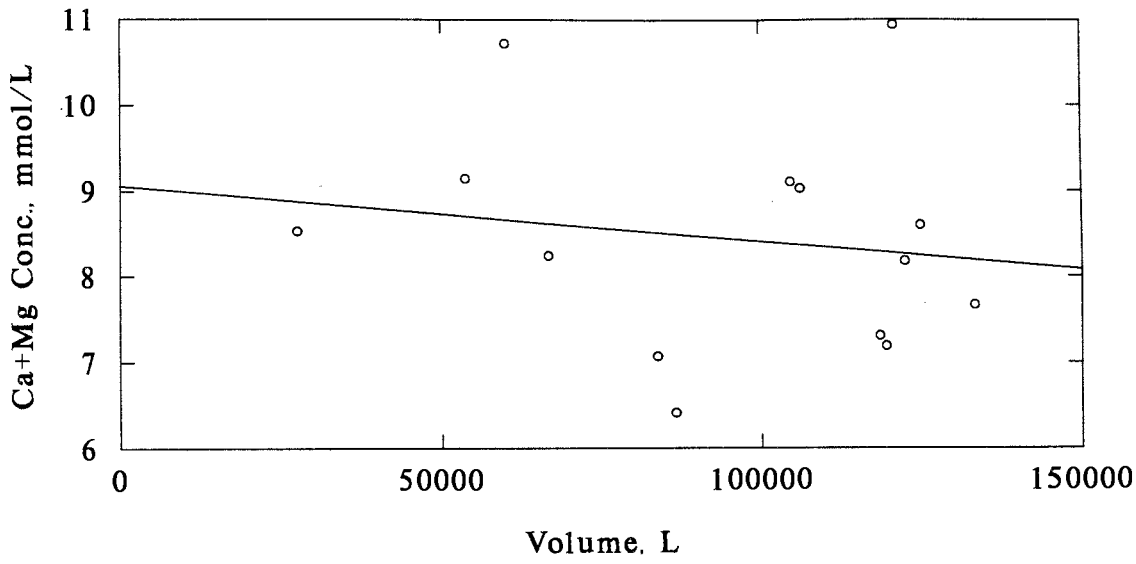

Figure A2.27. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual volume for Duluth Complex test pile FL2 (0.63% S).

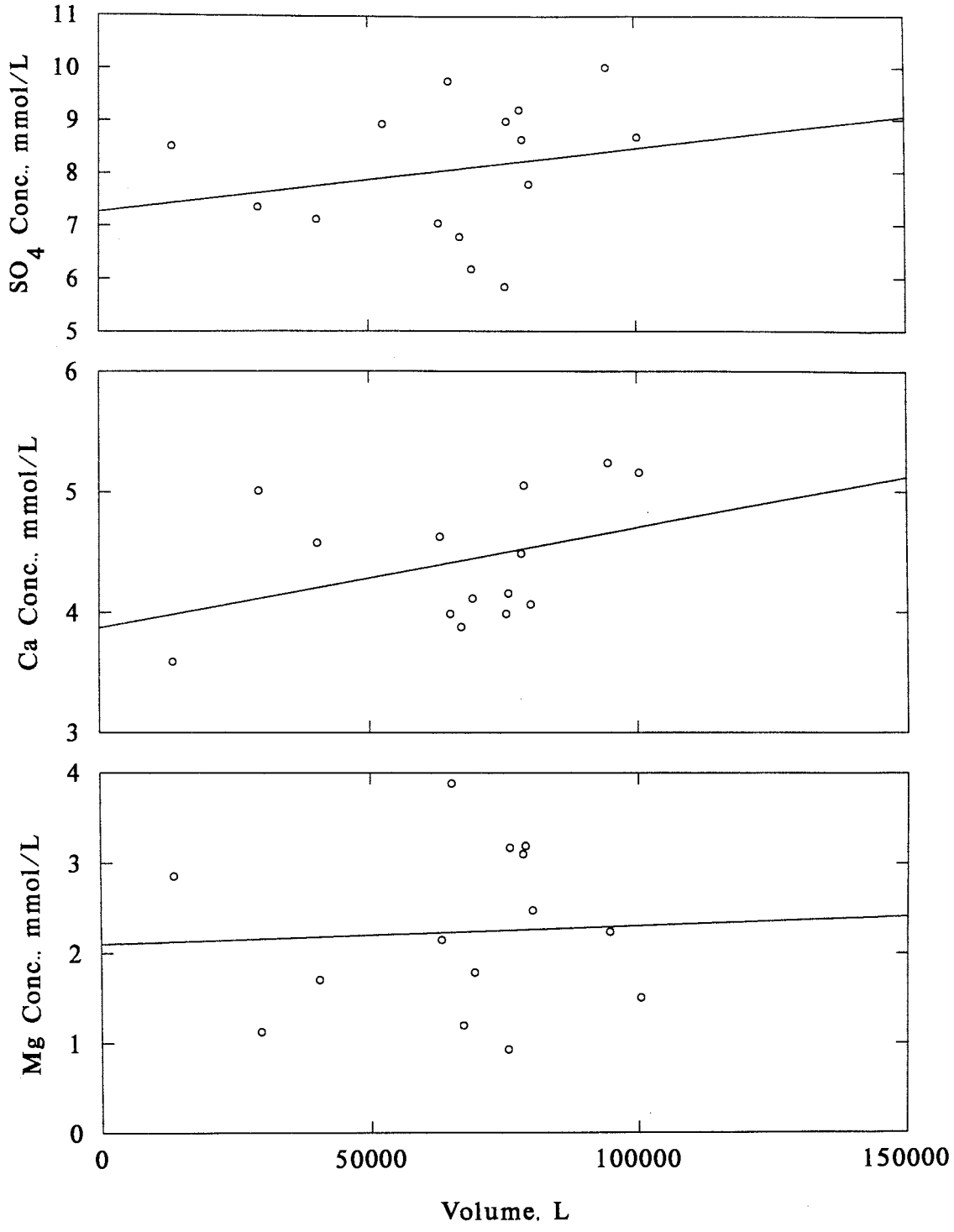

Figure A2.28. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual volume for Duluth Complex test pile FL2 (0.63% S).

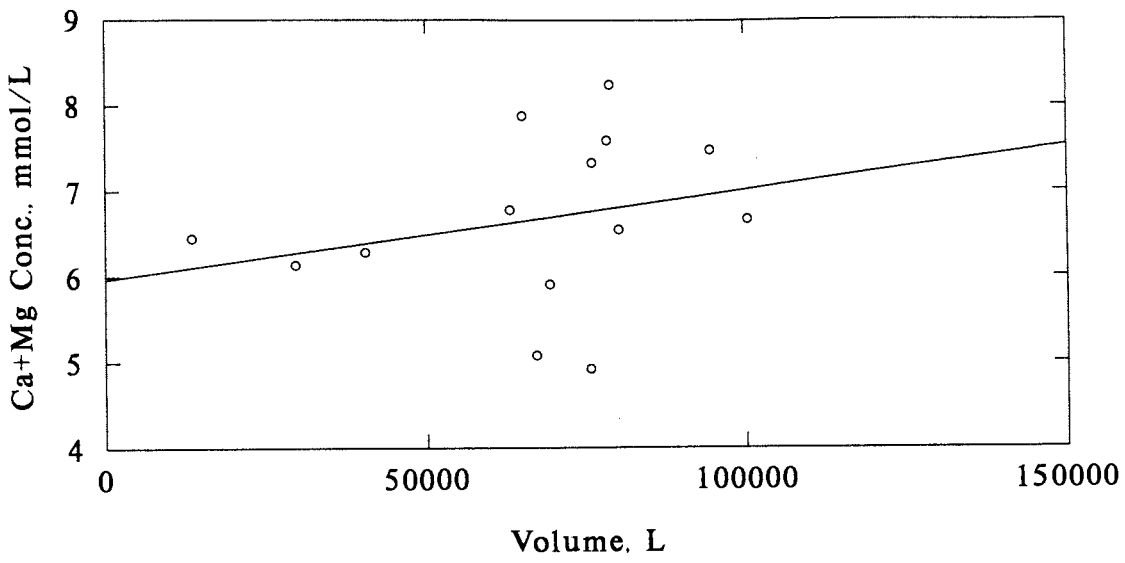

Figure A2.29. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL3 (0.63% S).

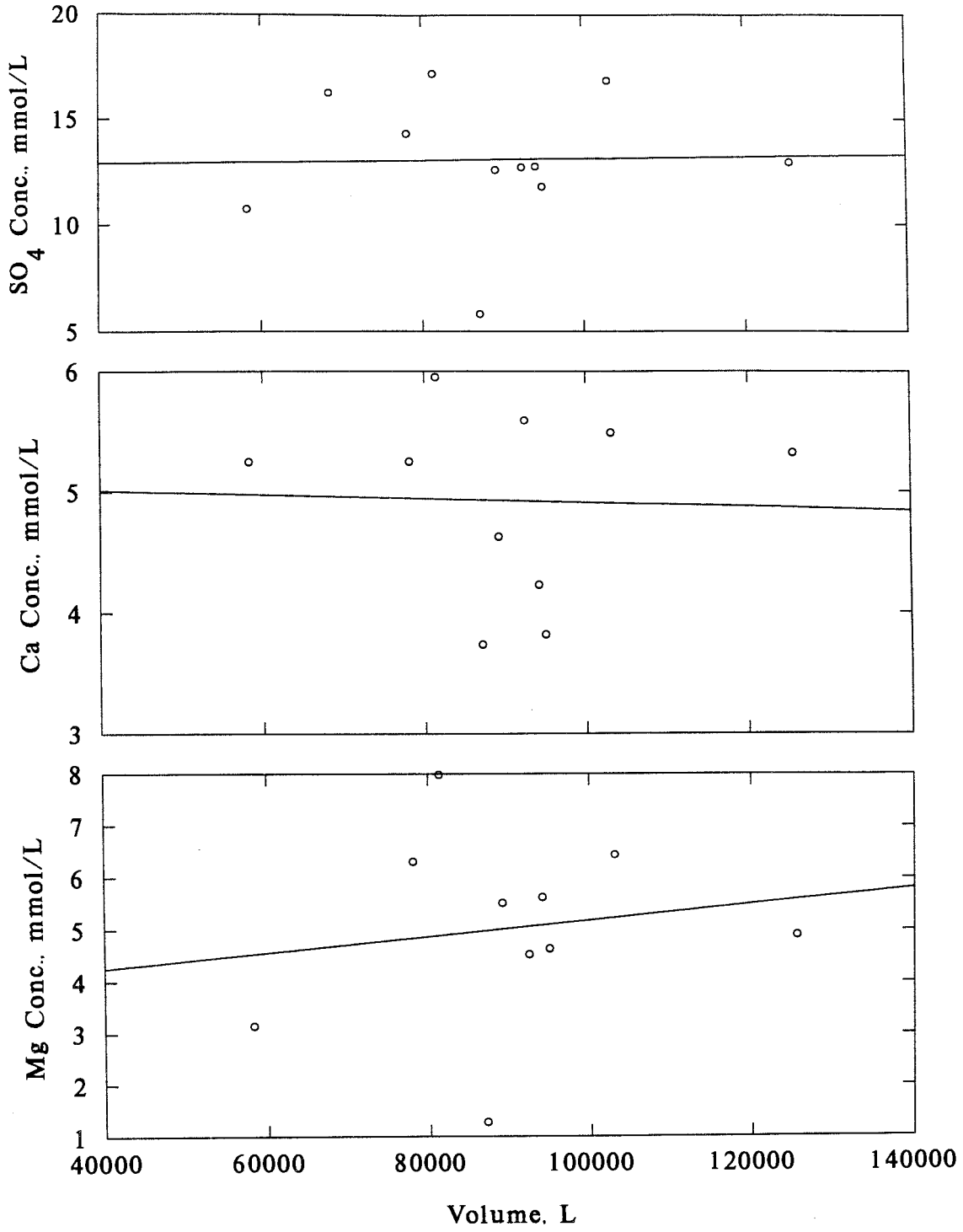

Figure A2.30. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL3 (0.63% S).

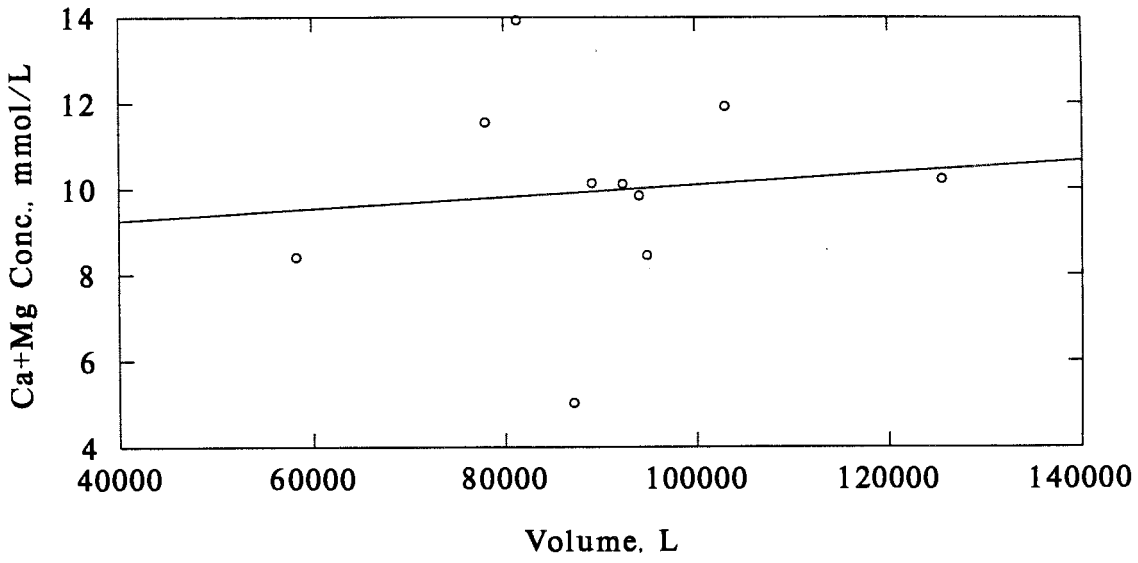

Figure A2.31. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual volume for Duluth Complex test pile FL6 (0.79% S).

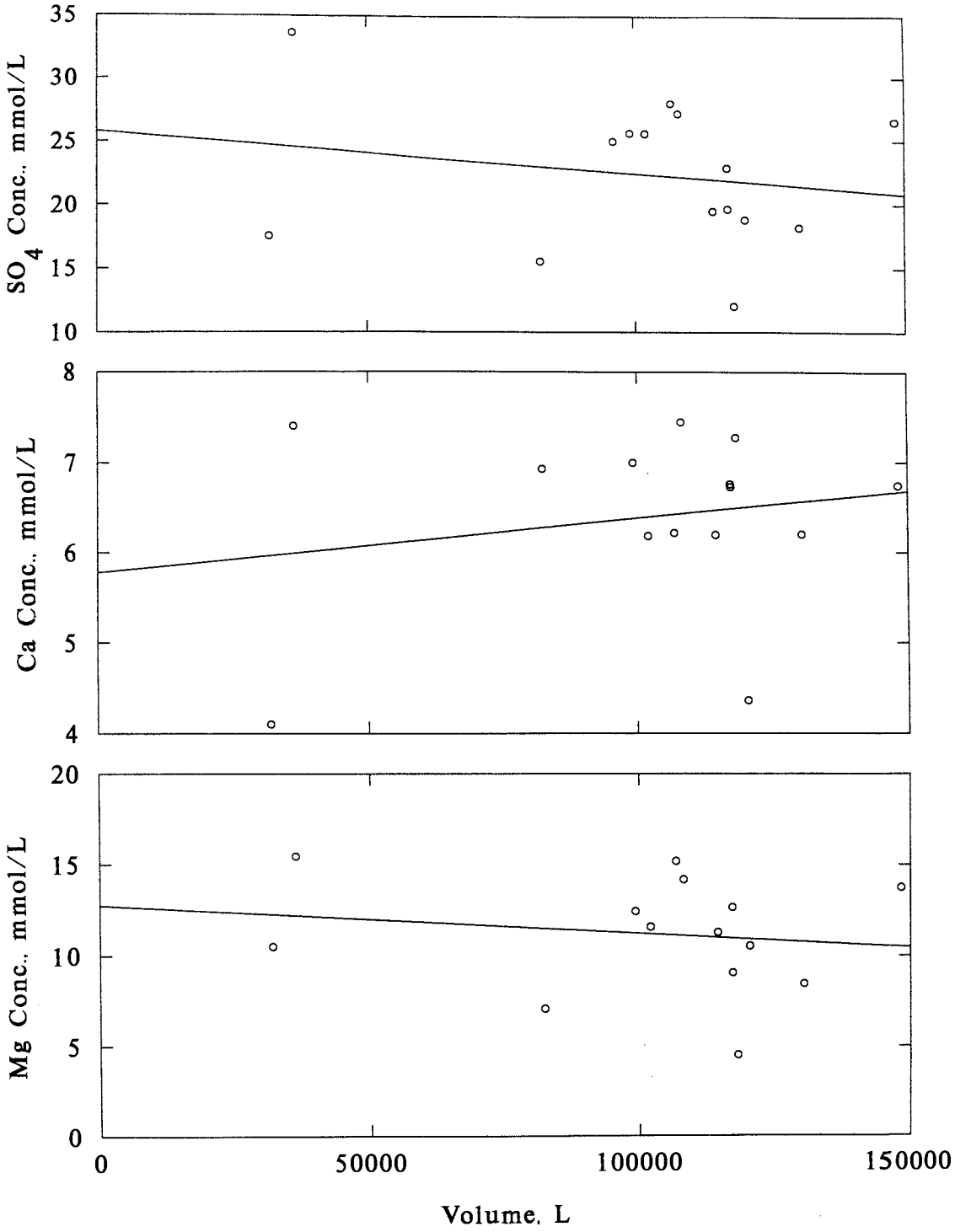

Figure A2.32. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual volume for Duluth Complex test pile FL6 (0.79% S).

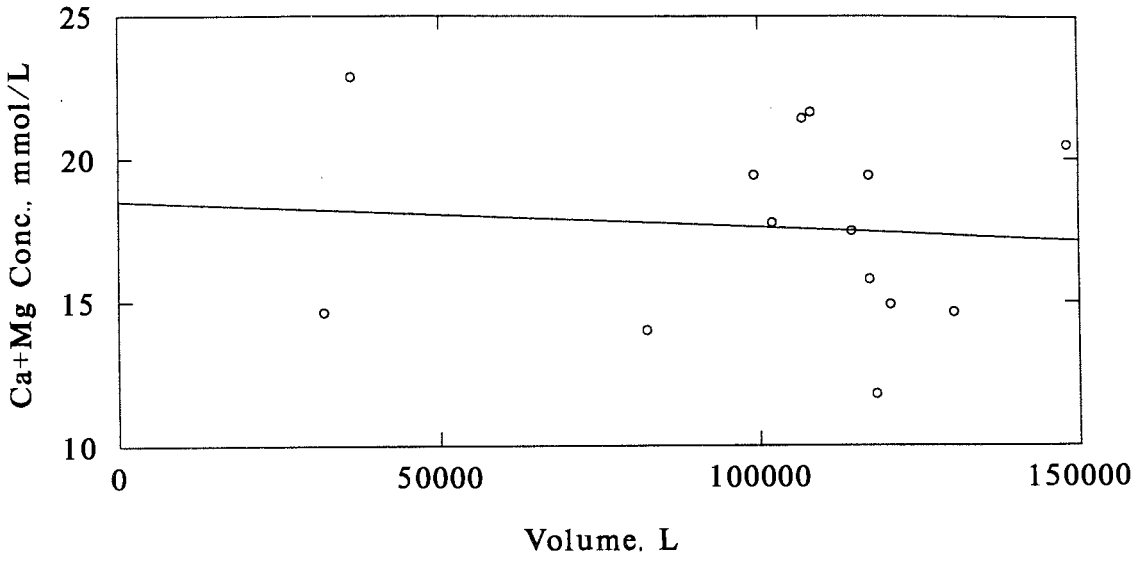

Figure A2.33. Annual flow-weighted mean concentrations of sulfate, calcium, and magnesium vs. annual volume for Duluth Complex test pile FL5 (1.41% S).

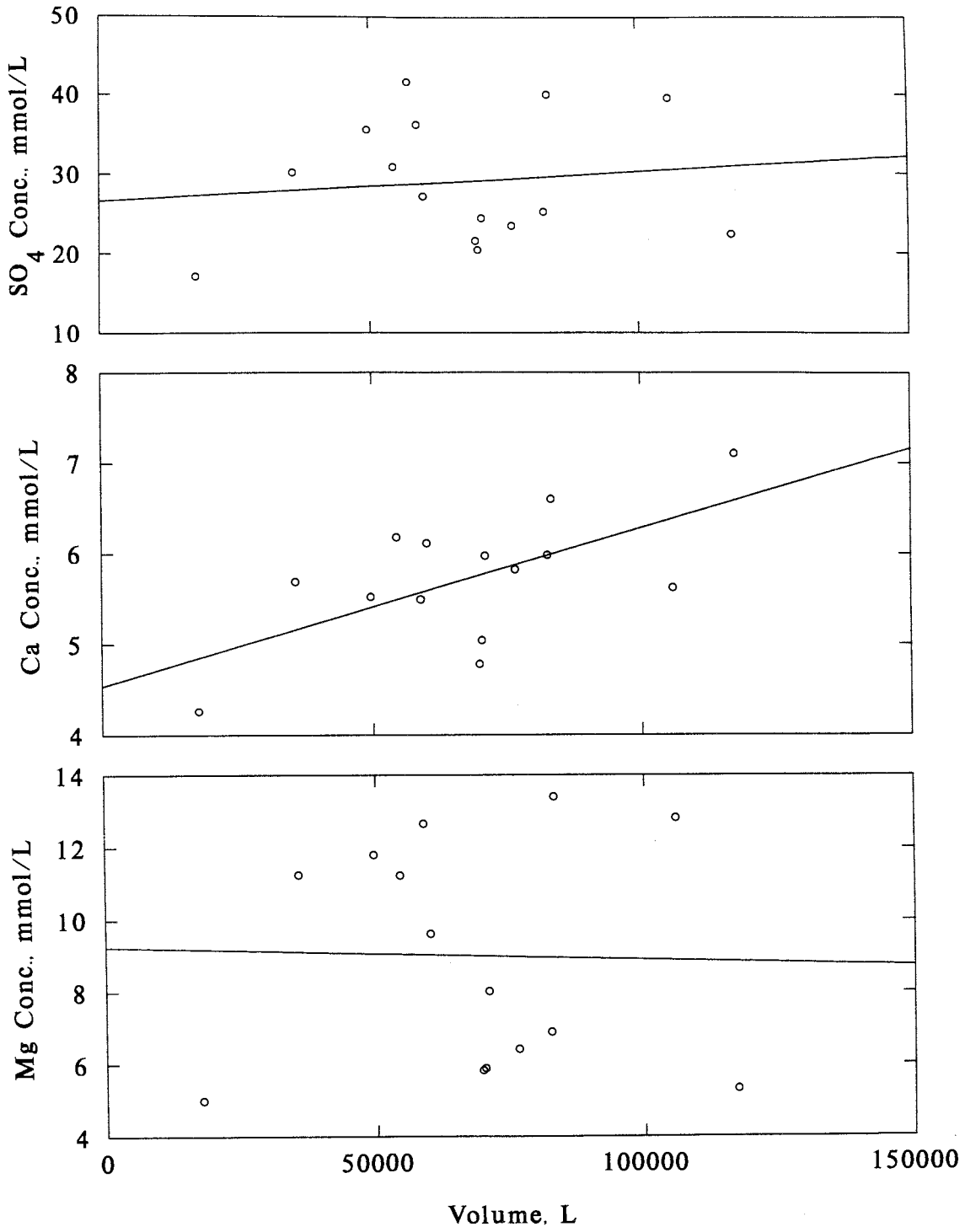

Figure A2.34. Annual flow-weighted mean concentrations of calcium plus magnesium vs. annual volume for Duluth Complex test pile FL5 (1.41% S).

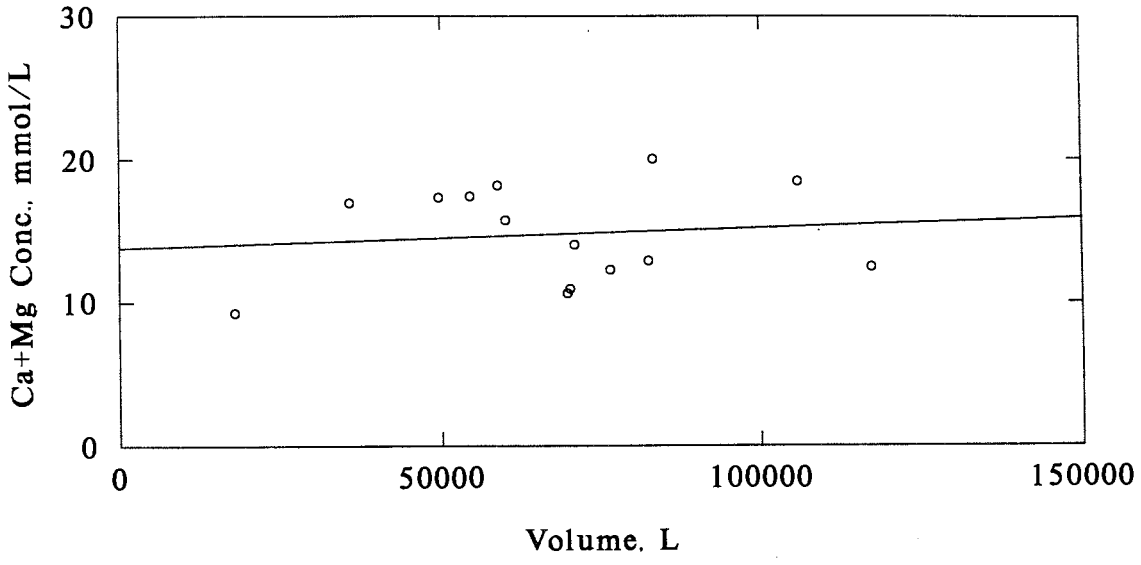

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.15	NA	1120	NA	112.0	11.0	113.0	13.0	0.032	NA	0.003	0.017	NA	0.06	NA	NA	NA	NA	NA	NA	0	9.0
06/06/78	NA	NA	1900	NA	216.0	19.0	268.0	20.0	0.030	0.250	0.009	0.002	0.02	0.13	NA	NA	NA	NA	NA	2558	0	7.0
06/07/78	7.60	NA	3325	NA	253.0	23.0	327.0	23.0	0.041	0.272	0.005	0.030	0.03	0.15	NA	NA	NA	NA	NA	NA	0	NA
06/16/78	7.84	NA	2700	NA	244.0	22.0	368.0	22.0	0.028	0.155	0.001	0.002	0.05	0.14	NA	NA	NA	NA	NA	120	0	11.0
06/23/78	7.72	NA	2042	1267	183.3	17.3	288.3	21.3	0.025	0.251	0.012	0.003	0.02	0.09	5.0	NA	NA	NA	NA	170	0	13.0
06/26/78	6.85	NA	2400	NA	195.0	20.0	313.0	21.0	0.024	0.294	0.014	0.010	0.04	0.09	NA	NA	NA	NA	NA	59	0	15.0
07/05/78	7.75	NA	2100	900	202.0	20.0	206.0	28.0	0.018	0.206	0.007	NA	0.01	0.09	4.0	NA	NA	NA	NA	177	0	14.0
07/07/78	7.90	NA	1900	800	221.0	23.0	222.0	34.0	0.024	0.285	0.019	0.005	0.04	0.09	4.0	5.7	0.44	0.08	NA	NA	0	19.0
07/11/78	7.70	NA	2450	1150	303.0	26.0	286.0	38.0	0.027	0.388	0.026	0.010	0.03	0.14	5.0	5.7	0.64	0.08	NA	NA	0	NA
07/13/78	7.92	NA	NA	1500	531.0	31.0	395.0	45.0	0.027	0.405	0.026	NA	0.01	0.16	4.0	6.2	0.55	0.08	NA	NA	0	NA
07/18/78	7.80	NA	1400	1050	340.0	19.0	220.0	34.0	0.023	0.255	0.019	0.003	0.01	0.11	1.0	3.9	0.50	0.08	NA	NA	0	NA
08/02/78	7.90	NA	1100	650	176.0	11.0	127.0	25.0	0.027	0.210	0.010	0.010	0.01	0.09	1.0	3.4	0.70	0.08	NA	NA	0	22.0
08/04/78	7.80	NA	1650	1200	300.0	23.0	209.0	37.0	0.034	0.330	0.016	0.013	0.03	0.13	3.0	5.0	0.06	0.10	NA	167	0	22.0
08/10/78	8.10	70.0	2400	1600	414.0	26.0	317.0	46.0	0.030	0.460	0.030	0.025	0.02	0.16	4.0	NA	0.10	0.08	NA	70	0	24.0
08/16/78	7.85	42.0	1538	640	169.5	14.0	102.0	25.0	0.020	0.185	0.020	0.015	0.02	0.07	2.0	2.5	NA	0.08	NA	3441	0	19.5
08/24/78	8.10	50.0	2600	1400	302.0	26.0	240.0	40.0	0.040	0.360	0.040	0.020	0.06	0.14	1.0	5.0	NA	0.08	NA	NA	0	NA
08/25/78	7.55	47.0	2500	1300	300.0	26.0	229.0	43.0	0.040	0.350	0.030	0.020	0.02	0.14	1.0	4.6	0.40	0.20	NA	NA	0	22.0
08/28/78	7.85	59.0	2700	NA	308.0	26.0	280.0	40.0	0.050	0.420	0.070	0.020	0.04	0.15	NA	NA	NA	NA	NA	NA	0	NA
09/13/78	7.58	40.5	2088	1050	246.0	19.0	160.0	39.5	0.045	0.275	0.020	0.010	0.03	0.12	1.5	NA	NA	0.08	NA	426	0	14.0
10/17/78	7.40	41.0	3207	1800	485.0	42.0	322.0	41.0	0.050	0.540	0.070	0.040	0.02	0.17	8.0	6.5	0.64	0.20	NA	NA	0	NA
03/19/79	7.45	21.0	807	NA	164.0	13.2	43.0	13.0	0.030	0.330	0.040	0.040	0.02	0.08	3.0	NA	NA	NA	2.9	2631	1986	NA
04/13/79	7.10	41.0	693	335	109.0	9.4	36.8	11.2	0.028	0.210	0.012	0.030	0.02	0.05	4.0	NA	NA	NA	3.6	2616	390	NA
04/18/79	7.20	33.0	436	190	78.0	5.6	27.0	6.0	0.023	0.106	0.008	0.020	0.02	0.04	1.0	1.3	NA	NA	NA	3665	3966	NA
04/19/79	7.10	35.0	360	165	69.0	4.8	27.0	5.0	0.025	0.102	0.008	0.010	0.02	0.03	2.0	0.8	0.12	NA	NA	4652	4029	NA
04/20/79	6.60	39.0	534	250	100.0	7.8	55.0	7.2	0.025	0.138	0.080	0.020	0.02	0.05	2.0	1.0	0.10	NA	3.1	3664	2274	NA
04/22/79	7.65	27.0	1194	633	185.9	18.1	69.8	15.5	0.028	0.315	0.020	0.040	0.02	0.09	4.0	2.2	0.23	NA	NA	4153	1200	NA
05/09/79	7.50	NA	983	483	173.0	15.2	60.6	14.8	0.025	0.250	0.015	0.020	0.02	0.08	4.0	NA	NA	NA	NA	685	815	NA
05/10/79	7.30	NA	1057	467	162.8	15.6	57.6	15.4	0.027	0.260	0.015	0.030	0.01	0.10	3.0	NA	NA	NA	NA	6572	2979	NA
05/20/79	7.10	NA	1144	667	179.6	17.8	75.6	16.2	0.031	0.320	0.020	0.030	0.03	0.10	3.0	NA	NA	NA	6.9	2991	784	NA
05/22/79	7.70	NA	897	533	144.4	13.8	60.4	14.0	0.027	0.220	0.015	0.020	0.01	0.09	1.0	NA	NA	NA	6.0	1386	848	NA
05/23/79	7.50	NA	1720	900	232.4	23.6	118.4	21.4	0.031	0.340	0.023	0.030	0.02	0.13	3.0	NA	NA	NA	5.0	2400	596	NA
06/01/79	7.30	NA	1185	700	170.2	17.2	88.8	18.4	0.033	0.290	0.020	0.020	0.03	0.12	2.0	NA	NA	NA	6.9	2468	737	NA
06/06/79	7.50	NA	1204	675	174.0	17.0	53.0	21.2	0.028	0.260	0.017	0.010	0.01	0.09	3.0	NA	NA	NA	NA	2230	941	NA
06/10/79	7.40	54.0	1168	580	166.0	16.4	49.9	20.4	0.031	0.225	0.011	0.010	0.01	0.09	3.0	NA	NA	NA	NA	2143	1098	20.0
06/12/79	7.20	45.0	1687	780	245.8	24.0	67.0	32.8	0.028	0.280	0.013	0.020	0.01	0.12	4.0	NA	NA	NA	NA	1181	416	NA
06/15/79	7.40	48.4	1488	740	234.8	23.0	68.4	28.6	0.028	0.270	0.009	0.010	0.01	0.11	5.0	NA	NA	NA	NA	545	494	20.0
06/21/79	7.10	30.0	1030	450	144.0	15.0	60.0	19.0	0.020	0.170	0.012	0.010	0.04	0.07	2.0	NA	NA	NA	NA	2343	1073	15.0
06/30/79	7.40	NA	1349	1280	338.0	33.0	160.0	32.0	0.030	0.370	0.025	0.020	0.02	0.11	4.0	NA	NA	NA	12.0	488	149	19.0
07/10/79	7.60	37.0	2120	1600	371.0	37.0	164.0	37.0	0.030	0.380	0.025	0.020	0.01	0.15	4.0	NA	NA	NA	11.0	1132	137	18.0
07/30/79	7.40	20.8	1221	589	208.5	18.5	50.0	22.0	0.020	0.210	0.014	0.010	0.01	0.07	3.0	NA	NA	NA	6.3	3449	2274	23.0
08/04/79	7.60	26.8	2094	1520	439.0	34.0	106.0	41.0	0.020	0.390	0.025	0.020	0.02	0.12	3.0	NA	NA	NA	6.1	1314	212	23.0
08/13/79	7.30	31.2	2562	1920	488.8	49.0	260.0	52.0	0.040	0.530	0.023	0.030	0.03	0.17	5.0	NA	NA	NA	11.0	345	47	13.0
08/23/79	7.30	24.8	1976	1500	394.8	33.2	174.0	38.6	0.030	0.430	0.023	0.030	0.02	0.13	5.0	NA	NA	NA	9.1	371	151	9.0
08/28/79	7.30	18.0	1642	1130	298.0	25.4	95.0	29.8	0.040	0.320	0.015	0.030	0.05	0.09	28.0	NA	NA	NA	8.3	548	458	20.0
09/01/79	7.50	23.2	2014	1250	364.0	31.0	80.2	33.0	0.040	0.450	0.020	0.030	0.03	0.10	5.0	NA	NA	NA	7.8	3710	1929	22.0
09/05/79	7.40	25.2	2145	1520	427.2	39.0	134.4	42.2	0.040	0.490	0.008	0.030	0.02	0.12	4.0	NA	NA	NA	10.0	742	550	18.0
09/10/79	7.35	24.0	1794	1290	343.4	29.2	135.0	35.9	0.035	0.415	0.019	0.030	0.02	0.10	5.0	NA	NA	NA	8.9	2608	516	17.5
09/13/79	7.50	24.0	2112	1600	428.2	37.6	165.6	41.0	0.050	0.520	0.023	0.030	0.01	0.12	5.0	NA	NA	NA	9.3	1408	500	16.0
09/19/79	7.40	25.2	3010	2200	590.8	52.0	232.0	53.2	0.050	0.660	0.060	0.040	0.03	0.12	8.0	NA	NA	NA	12.0	886	141	11.5
10/18/79	7.10	20.8	2603	1900	502.0	45.4	204.0	46.4	0.050	0.640	0.030	0.040	0.01	0.12	11.0	NA	NA	NA	11.0	352	733	NA
10/19/79	6.80	19.2	1318	750	223.0	19.6	56.0	20.8	0.040	0.460	0.021	0.030	0.01	0.05	3.0	NA	NA	NA	5.9	821	1222	NA

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
10/31/79	6.80	20.8	1959	1300	374.0	32.6	90.0	30.8	0.041	0.740	0.026	0.040	0.02	0.10	3.0	NA	NA	NA	7.2	4315	1513	NA
11/01/79	7.20	24.8	2364	1000	412.0	38.2	114.0	34.2	0.030	0.820	0.060	0.050	0.02	0.10	6.0	NA	NA	NA	8.5	2695	966	10.0
04/17/80	7.25	32.2	790	364	130.2	8.6	21.8	8.4	0.030	0.260	0.030	0.030	0.02	0.06	1.0	NA	NA	NA	NA	314	454	NA
04/18/80	6.95	38.0	NA	550	145.4	11.2	28.6	10.8	0.020	0.040	0.020	0.050	0.02	0.05	2.0	NA	NA	NA	2.4	927	907	NA
04/23/80	7.45	45.4	NA	490	148.0	11.6	22.6	11.0	0.010	0.390	0.040	0.050	0.02	0.06	2.0	NA	NA	NA	2.2	927	216	NA
04/27/80	7.50	52.7	1442	1045	233.7	20.8	58.5	17.0	0.020	0.505	0.040	0.040	0.03	0.12	3.0	NA	NA	NA	NA	2131	157	12.5
05/05/80	7.60	45.1	1330	1020	227.0	18.6	49.8	14.8	0.020	0.370	0.040	0.030	0.02	0.12	3.0	NA	NA	NA	4.5	420	201	11.0
05/18/80	7.60	46.2	1300	910	218.4	20.6	71.4	20.0	0.030	0.400	0.040	0.030	0.01	0.15	9.0	NA	NA	NA	NA	806	50	19.0
05/31/80	7.30	30.8	950	650	159.3	14.5	41.1	16.0	0.030	0.300	0.012	0.020	0.02	0.07	5.0	NA	NA	NA	6.3	1798	1190	NA
06/01/80	7.50	36.9	1575	1020	276.2	24.2	57.5	23.6	0.060	0.520	0.022	0.030	0.01	0.10	5.0	NA	NA	NA	6.2	893	441	NA
06/07/80	7.45	32.0	1475	800	283.6	25.4	67.2	25.6	0.020	0.450	0.014	0.020	0.03	0.02	5.0	NA	NA	NA	NA	799	189	NA
06/14/80	7.40	29.7	1700	1170	274.0	25.0	70.0	26.0	0.030	0.460	0.029	0.020	0.06	0.10	5.0	NA	NA	NA	NA	579	139	NA
06/18/80	7.30	31.9	2200	1450	348.0	23.7	101.0	32.8	0.040	0.530	0.012	0.030	0.03	0.12	6.0	NA	NA	NA	NA	367	46	NA
06/27/80	7.30	25.3	1700	1150	273.0	26.0	72.0	28.0	0.043	0.039	0.036	0.030	0.01	0.08	5.0	NA	NA	NA	8.2	996	245	NA
07/02/80	7.30	33.0	2300	2100	422.0	32.0	99.0	33.0	0.024	0.660	0.034	0.050	0.03	0.11	5.0	NA	NA	NA	8.8	1060	1272	NA
07/04/80	7.20	19.6	1850	NA	354.0	29.0	65.0	27.0	0.030	0.670	0.040	0.040	0.02	0.01	NA	NA	NA	NA	NA	727	1017	NA
07/17/80	7.20	28.6	2500	1590	428.4	34.8	190.3	42.4	0.040	0.630	0.040	0.040	0.03	0.12	5.5	NA	NA	NA	NA	1022	356	NA
07/20/80	7.20	30.8	2100	1300	363.8	33.4	169.0	37.8	0.040	0.550	0.040	0.040	0.02	0.10	6.0	NA	NA	NA	NA	1026	171	NA
08/03/80	7.20	24.2	2100	1150	355.2	34.4	131.6	37.4	0.060	0.540	0.026	0.060	0.04	0.13	6.0	NA	NA	NA	10.0	496	311	NA
08/12/80	7.20	9.8	1500	990	275.0	23.4	57.4	25.2	0.050	0.600	0.080	0.040	0.09	0.10	5.0	NA	NA	NA	6.1	1995	1239	NA
08/19/80	7.10	18.7	2000	1520	393.4	34.6	85.2	34.8	0.050	1.010	0.050	0.050	0.02	0.13	5.0	NA	NA	NA	NA	1643	350	NA
08/22/80	NA	NA	1850	1290	357.2	35.2	76.8	30.0	0.032	1.220	0.060	0.050	0.04	0.11	4.0	NA	NA	NA	NA	1344	657	NA
08/29/80	7.10	23.4	1850	1658	382.2	40.6	94.4	34.0	0.060	0.890	0.060	0.060	0.07	0.10	4.0	NA	NA	NA	7.5	8112	3456	NA
09/07/80	7.00	20.4	1500	920	287.0	26.6	80.6	30.0	0.035	0.660	0.040	0.050	0.04	0.08	4.0	NA	NA	NA	NA	2703	1050	NA
09/09/80	7.20	22.1	1760	1250	382.8	39.2	93.2	33.8	0.036	0.860	0.047	0.050	0.06	0.10	3.0	NA	NA	NA	NA	1147	361	14.0
09/12/80	7.05	20.0	1215	660	262.0	21.0	55.6	22.4	0.038	0.680	0.050	0.040	0.02	0.08	4.0	NA	NA	NA	4.9	3536	1890	16.0
09/18/80	7.20	21.0	1535	1130	340.0	33.2	84.0	29.2	0.046	0.740	0.033	0.050	0.06	0.08	4.0	NA	NA	NA	NA	7896	2008	12.0
09/23/80	7.20	21.3	2117	1350	398.0	45.0	122.0	10.8	0.050	0.840	0.070	0.060	0.04	0.08	4.0	NA	NA	NA	6.9	1938	588	14.0
10/15/80	7.00	20.4	1719	1050	310.0	32.0	87.8	30.0	0.050	0.700	0.040	0.060	0.03	0.07	5.0	NA	NA	NA	7.0	1741	101	8.0
10/31/80	6.50	17.0	1773	1900	340.0	30.0	65.0	29.0	0.070	1.590	0.070	0.090	0.02	0.10	4.0	NA	NA	NA	NA	1257	48	11.0
02/18/81	7.20	18.7	525	266	74.8	7.2	16.2	NA	0.020	0.630	0.030	0.040	NA	NA	1.0	NA	NA	NA	NA	832	722	8.0
03/27/81	7.20	19.2	600	300	80.6	8.3	15.8	NA	0.020	0.710	0.040	0.050	NA	NA	NA	NA	NA	NA	3.0	4742	2270	NA
04/03/81	7.30	27.2	800	440	124.8	13.0	20.8	NA	0.050	1.470	0.060	0.070	NA	NA	NA	NA	NA	NA	NA	4410	1459	NA
04/13/81	6.70	17.6	900	510	143.0	16.0	22.0	NA	0.090	1.290	0.060	0.060	NA	NA	NA	NA	NA	NA	NA	2441	1050	NA
04/17/81	7.40	30.4	1000	515	139.0	16.0	23.0	NA	0.040	1.280	0.050	0.050	NA	NA	NA	NA	NA	NA	3.6	1696	988	NA
04/23/81	7.40	28.8	1350	635	165.0	22.0	30.0	NA	0.040	1.260	0.060	0.060	NA	NA	NA	NA	NA	NA	NA	2350	2515	NA
04/27/81	7.45	29.2	1500	740	203.0	32.0	39.0	NA	0.040	1.350	0.070	0.060	NA	NA	NA	NA	NA	NA	NA	6162	2089	2.0
04/28/81	7.60	36.0	1250	640	168.0	27.0	36.0	NA	0.040	1.070	0.060	0.060	NA	NA	NA	NA	NA	NA	4.3	3785	1698	NA
04/29/81	7.40	37.0	1430	835	230.0	28.0	46.0	NA	0.040	1.050	0.080	0.070	NA	NA	NA	NA	NA	NA	NA	3323	1040	NA
05/12/81	7.55	38.4	1587	800	272.0	32.0	68.0	NA	0.030	0.830	0.040	0.070	NA	NA	NA	NA	NA	NA	6.2	3686	203	17.0
06/01/81	7.50	26.0	1064	450	184.0	20.0	46.0	NA	0.040	0.490	0.030	0.040	NA	NA	NA	NA	NA	NA	NA	2626	790	16.0
06/06/81	7.40	13.3	1202	620	200.0	38.0	56.0	22.0	0.040	0.880	0.050	0.050	NA	NA	NA	NA	NA	NA	3.6	1840	207	18.0
06/22/81	7.00	15.2	1112	590	196.0	18.0	32.0	NA	0.050	1.120	0.060	0.050	NA	NA	NA	NA	NA	NA	NA	4148	2105	NA
06/24/81	7.20	16.0	1216	675	216.0	18.0	30.0	NA	0.060	1.280	0.070	0.050	NA	NA	NA	NA	NA	NA	NA	3274	1541	16.0
06/28/81	7.20	15.2	1143	640	205.0	18.0	32.0	NA	0.040	1.040	0.050	0.040	NA	NA	NA	NA	NA	NA	3.0	4690	2922	15.0
07/04/81	7.40	21.4	1786	1100	348.0	29.6	65.6	NA	0.040	1.100	0.060	0.050	NA	NA	NA	NA	NA	NA	NA	2154	117	21.0
07/15/81	7.50	15.0	1586	1000	284.0	21.6	65.8	NA	0.060	0.670	0.050	0.040	NA	NA	NA	NA	NA	NA	NA	606	40	19.0
07/24/81	7.20	14.2	1464	945	274.0	23.4	48.2	NA	0.070	0.800	0.060	0.050	NA	NA	2.0	NA	NA	NA	3.7	1094	238	19.0
08/04/81	7.50	13.4	1798	1200	356.0	36.0	76.0	NA	0.050	0.980	0.050	0.090	NA	NA	NA	NA	NA	NA	NA	722	75	18.0
08/17/81	7.10	12.2	1936	1200	392.0	36.0	65.0	NA	0.060	1.220	0.080	0.080	NA	NA	NA	NA	NA	NA	NA	602	20	22.0
09/01/81	7.00	9.6	2275	1180	352.0	34.0	56.0	NA	0.140	1.840	0.060	0.140	NA	NA	NA	NA	NA	NA	NA	772	100	NA

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/16/81	6.90	4.8	2700	NA	482.0	48.0	75.0	NA	0.140	2.700	0.150	0.200	NA	NA	NA	NA	NA	NA	NA	148	5	NA
09/26/81	6.75	5.6	1850	1120	344.0	36.0	42.0	NA	0.270	4.300	0.190	0.230	NA	NA	NA	NA	NA	NA	6.7	1037	587	NA
10/01/81	6.90	6.4	1800	940	320.0	54.0	32.0	NA	0.390	6.500	0.290	0.300	NA	NA	NA	NA	NA	NA	NA	3043	1895	NA
10/04/81	6.90	8.8	1700	1020	358.0	32.0	40.0	30.0	0.280	5.750	0.230	0.260	NA	NA	NA	NA	NA	NA	NA	2642	1354	NA
10/13/81	7.05	10.8	1500	800	262.0	26.0	40.0	26.0	0.140	2.980	0.140	0.160	NA	NA	NA	NA	NA	NA	3.5	3622	1799	NA
10/17/81	7.00	11.2	1700	1010	336.0	32.0	50.0	32.0	0.170	3.820	0.180	0.180	NA	NA	NA	NA	NA	NA	NA	8856	1213	NA
10/30/81	7.15	6.4	2600	1750	490.0	60.0	108.0	46.0	0.120	2.040	0.090	0.110	NA	NA	NA	NA	NA	NA	6.2	3104	271	NA
03/24/82	6.90	10.8	950	490	136.0	16.6	14.0	12.0	0.390	7.700	0.300	0.250	NA	NA	NA	NA	NA	NA	NA	715	NA	NA
04/13/82	7.00	10.8	850	440	121.0	11.2	18.0	10.0	0.160	3.940	0.160	0.160	NA	NA	NA	NA	NA	NA	NA	1090	NA	NA
04/14/82	6.97	7.7	500	215	63.8	5.2	8.0	6.0	0.070	1.430	0.055	0.060	NA	NA	NA	NA	NA	NA	NA	4141	NA	NA
04/15/82	7.10	15.3	700	340	106.0	8.6	12.0	8.0	0.110	2.220	0.090	0.090	NA	NA	NA	NA	NA	NA	NA	2854	NA	NA
04/19/82	6.90	14.4	825	380	166.0	12.0	35.0	7.2	0.130	3.840	0.130	0.110	NA	NA	NA	NA	NA	NA	NA	912	NA	NA
04/23/82	7.20	18.0	1050	600	190.0	16.0	20.0	14.0	0.110	3.420	0.140	0.110	NA	NA	NA	NA	NA	NA	NA	1238	NA	NA
04/28/82	7.10	19.8	1175	630	157.0	16.0	22.0	16.0	0.080	2.840	0.110	0.080	NA	NA	NA	NA	NA	NA	NA	3274	NA	NA
05/03/82	7.10	12.6	750	370	118.0	10.0	16.0	10.0	0.100	1.610	0.090	0.070	NA	NA	NA	NA	NA	NA	NA	409	NA	NA
05/07/82	7.10	18.9	1030	520	164.0	14.0	22.0	12.0	0.080	3.050	0.140	0.090	NA	NA	NA	NA	NA	NA	NA	5167	NA	NA
05/11/82	7.10	18.0	1225	710	188.0	24.0	24.0	20.0	0.080	4.150	0.150	0.120	NA	NA	NA	NA	NA	NA	NA	3819	NA	NA
05/14/82	7.10	21.6	1200	690	216.0	22.8	26.6	20.4	0.060	3.000	0.110	0.080	0.04	0.12	NA	NA	NA	NA	NA	4038	NA	NA
05/21/82	7.15	NA	1800	1100	308.0	31.0	48.7	25.8	0.055	2.835	0.105	0.090	0.04	0.14	NA	NA	NA	NA	NA	1688	NA	NA
06/02/82	7.00	12.6	1275	710	198.0	21.8	39.0	14.0	0.050	1.660	0.070	0.070	0.04	0.11	NA	NA	NA	NA	NA	1253	NA	NA
06/11/82	6.80	NA	1175	665	185.0	18.5	23.0	18.0	0.050	3.245	0.125	0.090	0.03	0.14	NA	NA	NA	NA	NA	719	NA	NA
07/02/82	NA	NA	NA	780	308.0	26.0	36.0	26.0	0.060	3.540	0.160	0.080	0.01	0.13	NA	NA	NA	NA	NA	19809	NA	NA
07/26/82	7.10	12.6	1400	890	338.0	28.0	44.0	28.0	0.080	2.520	0.110	0.070	0.01	0.11	NA	NA	NA	NA	NA	1382	NA	NA
08/02/82	7.00	NA	1900	NA	394.0	30.0	42.0	32.0	0.090	3.400	0.150	0.070	0.02	0.13	NA	NA	NA	NA	NA	8025	NA	NA
08/09/82	7.00	17.1	2100	NA	367.0	34.0	62.0	36.0	0.100	5.170	0.220	0.090	0.02	0.13	NA	NA	NA	NA	NA	1817	NA	NA
08/23/82	7.10	8.1	1850	1145	359.0	32.0	48.0	34.0	0.130	7.020	0.310	0.120	0.02	0.17	NA	NA	NA	NA	NA	1790	NA	NA
08/30/82	6.90	9.9	1850	1035	338.0	30.0	44.0	30.0	0.120	5.940	0.270	0.120	0.30	0.17	NA	NA	NA	NA	NA	1147	NA	NA
09/08/82	6.50	10.8	1350	785	256.0	22.0	26.0	22.0	0.280	7.890	0.370	0.150	0.02	0.22	NA	NA	NA	NA	NA	4009	NA	NA
09/13/82	6.50	11.2	1300	975	316.0	26.0	32.0	28.0	0.190	7.350	0.280	0.140	0.02	0.19	NA	NA	NA	NA	NA	4190	NA	NA
09/20/82	6.90	NA	1900	1140	334.0	32.0	46.0	32.0	0.190	7.300	0.290	0.140	0.01	0.19	NA	NA	NA	NA	NA	810	NA	NA
10/04/82	7.00	16.2	1425	850	258.0	26.0	34.0	26.0	0.180	6.170	0.300	0.110	0.02	0.17	NA	NA	NA	NA	NA	15259	NA	NA
10/11/82	6.80	15.3	1400	630	252.0	24.0	38.0	26.0	0.110	4.570	0.150	0.080	0.01	0.12	NA	NA	NA	NA	NA	3782	NA	NA
10/21/82	6.90	13.0	1000	620	266.0	26.0	38.0	26.0	0.110	4.390	0.170	0.080	0.19	0.12	NA	NA	NA	NA	NA	1200	NA	NA
10/25/82	7.00	13.1	1830	940	256.0	26.0	34.0	28.0	0.150	5.640	0.240	0.100	0.03	0.14	NA	NA	NA	NA	NA	708	NA	NA
11/01/82	6.90	9.0	1000	NA	136.0	14.0	20.0	14.0	0.120	4.660	0.210	0.070	0.02	0.10	NA	NA	NA	NA	NA	1730	NA	NA
11/22/82	7.05	4.5	1400	605	230.0	28.0	32.0	NA	0.120	5.090	0.150	0.120	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
03/18/83	6.90	13.3	745	400	NA	NA	NA	NA	0.300	5.650	0.240	0.180	NA	NA	NA	NA	NA	NA	NA	2495	NA	NA
04/14/83	6.86	14.8	544	260	NA	NA	NA	NA	0.130	2.910	0.120	0.130	NA	NA	NA	NA	NA	NA	NA	4984	NA	NA
05/15/83	7.01	15.3	910	540	NA	NA	NA	NA	0.070	3.260	0.140	0.110	NA	NA	NA	NA	NA	NA	NA	6196	NA	NA
06/16/83	6.86	11.8	1348	840	NA	NA	NA	NA	0.070	4.870	0.190	0.100	NA	NA	NA	NA	NA	NA	NA	2168	NA	NA
07/14/83	6.91	12.6	1663	1320	NA	NA	NA	NA	0.260	7.470	0.280	0.230	NA	NA	NA	NA	NA	NA	NA	11710	NA	NA
08/14/83	6.61	7.5	1497	1140	NA	NA	NA	NA	0.730	9.950	0.400	0.340	NA	NA	NA	NA	NA	NA	NA	14114	NA	NA
09/15/83	6.74	7.1	1507	1110	NA	NA	NA	NA	0.780	10.220	0.460	0.350	NA	NA	NA	NA	NA	NA	NA	8997	NA	NA
10/13/83	6.91	9.8	1714	1110	NA	NA	NA	NA	0.520	8.350	0.370	0.240	NA	NA	NA	NA	NA	NA	NA	24777	NA	NA
11/09/83	6.60	5.4	900	620	NA	NA	NA	NA	0.570	6.470	0.290	0.220	NA	NA	NA	NA	NA	NA	NA	3459	NA	NA
11/30/83	6.35	5.4	900	620	NA	NA	NA	NA	0.830	13.350	0.540	0.360	NA	NA	NA	NA	NA	NA	NA	738	NA	NA
02/24/84	6.25	6.3	580	290	93.9	18.9	9.6	NA	0.730	12.000	0.500	0.300	NA	NA	NA	NA	NA	NA	NA	1222	NA	NA
03/15/84	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
04/15/84	6.49	4.9	429	260	94.7	13.6	8.2	NA	0.390	4.850	0.210	0.200	NA	NA	NA	NA	NA	NA	NA	6389	NA	NA
05/16/84	6.86	10.7	865	620	210.6	28.1	17.5	NA	0.360	9.000	0.290	0.200	NA	NA	NA	NA	NA	NA	NA	10099	NA	NA
06/13/84	6.91	7.9	1065	838	260.1	36.6	27.7	NA	0.590	12.400	0.430	0.290	NA	NA	NA	NA	NA	NA	NA	21645	NA	NA

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
07/14/84	6.92	9.0	1145	820	NA	NA	25.0	NA	0.300	9.600	0.350	0.190	NA	NA	NA	NA	NA	NA	NA	5190	NA	NA
08/15/84	6.81	6.8	1395	880	278.4	53.9	27.2	NA	0.870	24.000	0.880	0.380	NA	NA	NA	NA	NA	NA	NA	5678	NA	NA
09/12/84	6.36	3.3	1425	1400	326.6	67.8	33.5	NA	1.540	16.250	1.020	0.530	NA	NA	NA	NA	NA	NA	NA	1809	NA	NA
10/13/84	6.17	4.3	1425	1750	401.2	60.0	31.5	NA	3.300	27.900	1.790	0.790	NA	NA	NA	NA	NA	NA	NA	10556	NA	NA
11/14/84	6.23	4.9	1050	1400	352.0	52.0	32.4	NA	1.860	18.100	0.560	0.540	NA	NA	NA	NA	NA	NA	NA	2430	NA	NA
12/01/84	6.00	2.1	975	1050	390.0	190.0	25.0	NA	5.400	23.300	1.030	0.780	NA	NA	NA	NA	NA	NA	NA	409	NA	NA
12/18/84	6.15	4.2	875	710	216.0	48.0	20.0	NA	2.490	23.800	0.860	0.570	NA	NA	NA	NA	NA	NA	NA	1452	NA	NA
04/02/85	6.55	1.1	420	270	85.0	6.6	5.6	NA	0.470	3.090	0.120	0.120	NA	NA	NA	NA	NA	NA	NA	308	NA	NA
04/16/85	6.56	7.6	845	520	162.0	29.3	12.4	NA	1.770	14.500	0.550	0.440	NA	NA	NA	NA	NA	NA	NA	16654	NA	NA
05/14/85	6.93	6.9	1200	710	204.0	37.5	19.3	NA	0.670	12.300	0.450	0.270	NA	NA	NA	NA	NA	NA	NA	10821	NA	NA
06/14/85	6.94	2.6	1425	1100	241.0	45.1	25.4	NA	1.170	15.200	0.510	0.350	NA	NA	NA	NA	NA	NA	NA	33032	NA	NA
07/15/85	6.82	7.6	1350	950	193.0	43.4	28.5	NA	0.840	12.900	0.480	0.310	NA	NA	NA	NA	NA	NA	NA	12354	NA	NA
08/13/85	6.70	3.3	1350	968	230.2	43.1	25.0	NA	1.400	17.900	0.690	0.450	NA	NA	NA	NA	NA	NA	NA	12188	NA	NA
09/13/85	6.58	1.6	1475	1000	316.0	49.8	26.0	NA	1.750	16.100	0.630	0.450	NA	NA	NA	NA	NA	NA	NA	28330	NA	NA
10/15/85	6.85	5.2	1275	650	242.0	38.0	22.0	NA	1.300	10.600	0.320	0.300	NA	NA	NA	NA	NA	NA	NA	18471	NA	NA
11/15/85	6.64	1.1	1175	540	254.0	36.0	24.0	NA	0.800	7.250	0.220	0.220	NA	NA	NA	NA	NA	NA	NA	1238	NA	NA
03/23/86	6.60	1.1	570	230	76.0	14.6	5.8	NA	2.090	13.000	0.460	0.280	NA	NA	NA	NA	NA	NA	NA	18543	NA	NA
04/16/86	6.49	2.8	775	410	131.0	28.1	9.6	NA	2.400	17.800	0.580	0.380	NA	NA	NA	NA	NA	NA	NA	12687	NA	NA
05/14/86	6.75	5.2	1025	800	244.0	38.0	14.0	NA	1.540	15.900	0.600	0.360	NA	NA	NA	NA	NA	NA	NA	12884	NA	NA
06/11/86	6.63	5.6	1175	850	216.0	38.0	24.0	NA	1.270	14.100	0.520	0.330	NA	NA	NA	NA	NA	NA	NA	6703	NA	NA
07/12/86	6.69	5.9	1375	800	246.0	50.0	20.0	NA	2.740	20.400	0.800	0.520	NA	NA	NA	NA	NA	NA	NA	19489	NA	NA
08/13/86	6.29	4.2	1550	1100	272.0	47.0	26.0	NA	3.580	23.100	0.900	0.430	NA	NA	NA	NA	NA	NA	NA	11835	NA	NA
09/14/86	6.54	3.0	1500	700	271.0	48.4	22.0	NA	3.460	18.700	0.800	0.360	NA	NA	NA	NA	NA	NA	NA	23641	NA	NA
10/16/86	6.41	5.2	1925	1200	276.0	52.0	26.0	NA	2.720	19.800	0.700	0.490	NA	NA	NA	NA	NA	NA	NA	4621	NA	NA
11/16/86	6.36	3.4	1750	1125	332.0	68.0	26.0	NA	5.700	23.000	0.870	0.640	NA	NA	NA	NA	NA	NA	NA	9148	NA	NA
03/22/87	5.70	6.1	575	294	96.0	18.0	8.0	NA	3.600	19.800	0.540	0.390	NA	NA	NA	NA	NA	NA	NA	10068	NA	NA
04/15/87	6.35	4.6	725	352	94.0	22.0	8.0	NA	3.750	14.600	0.590	0.420	NA	NA	NA	NA	NA	NA	NA	3009	NA	NA
05/16/87	6.59	6.0	1500	970	294.0	50.0	20.0	NA	4.900	21.000	0.800	0.800	NA	NA	NA	NA	NA	NA	NA	32721	NA	NA
06/16/87	6.66	6.6	1900	1200	368.0	72.0	34.0	NA	1.970	16.000	0.500	0.290	NA	NA	NA	NA	NA	NA	NA	2328	NA	NA
07/15/87	6.03	7.4	1850	1200	312.0	60.0	28.0	NA	4.800	20.500	0.800	0.560	NA	NA	NA	NA	NA	NA	NA	25288	NA	NA
08/15/87	6.04	5.4	1600	1080	270.0	56.0	26.0	NA	5.400	27.000	0.900	0.680	NA	NA	NA	NA	NA	NA	NA	12176	NA	NA
09/16/87	6.01	4.3	2275	1070	314.0	68.0	26.0	NA	6.930	29.000	0.920	0.700	NA	NA	NA	NA	NA	NA	NA	16525	NA	NA
10/13/87	5.86	1.8	2150	1390	400.0	76.0	34.0	NA	4.700	25.000	0.780	0.650	NA	NA	NA	NA	NA	NA	NA	1764	NA	NA
11/10/87	5.81	2.9	1850	1280	330.0	60.0	30.0	NA	5.900	26.000	0.980	0.750	NA	NA	NA	NA	NA	NA	NA	889	NA	NA
03/28/88	6.50	5.1	675	340	74.0	24.0	6.0	NA	6.800	16.400	0.900	1.000	NA	NA	NA	NA	NA	NA	NA	7169	NA	NA
04/17/88	5.93	2.3	970	700	116.0	36.0	8.0	NA	9.500	33.000	1.400	1.500	NA	NA	NA	NA	NA	NA	NA	5008	NA	NA
05/12/88	6.11	6.7	1500	1000	276.0	64.0	18.0	NA	8.500	41.000	1.600	1.600	NA	NA	NA	NA	NA	NA	NA	11014	NA	NA
06/12/88	6.12	4.8	1320	820	282.0	52.0	18.0	NA	5.100	33.000	1.200	1.300	NA	NA	NA	NA	NA	NA	NA	3160	NA	NA
07/15/88	5.91	5.3	1450	1120	253.0	60.0	21.0	NA	9.000	49.000	1.800	1.900	NA	NA	NA	NA	NA	NA	NA	3565	NA	NA
08/16/88	6.27	7.1	1575	915	277.0	73.0	26.0	NA	5.600	38.500	1.100	0.800	NA	NA	NA	NA	NA	NA	NA	56169	NA	NA
09/16/88	6.09	3.0	1600	860	280.0	78.0	26.0	NA	5.200	42.000	1.200	0.900	NA	NA	NA	NA	NA	NA	NA	11276	NA	NA
10/18/88	5.93	3.6	1575	1105	239.0	66.0	20.0	NA	8.350	36.500	1.600	1.750	NA	NA	NA	NA	NA	NA	NA	3346	NA	NA
11/15/88	5.57	2.4	1200	900	180.0	116.0	18.0	NA	12.700	38.000	1.800	1.700	NA	NA	NA	NA	NA	NA	NA	5583	NA	NA
11/30/88	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	8	NA	NA
04/07/89	5.20	1.0	530	284	70.0	20.0	8.0	NA	5.200	20.500	0.800	0.700	NA	NA	NA	NA	NA	NA	NA	26374	NA	NA
04/24/89	5.87	3.7	575	296	64.0	18.0	6.0	NA	4.000	18.000	0.800	0.800	NA	NA	NA	NA	NA	NA	NA	7483	NA	NA
05/16/89	5.69	1.3	1325	790	192.0	52.0	14.0	NA	8.000	37.000	1.700	1.700	NA	NA	NA	NA	NA	NA	NA	13668	NA	NA
06/17/89	5.92	6.3	1725	1270	266.0	76.0	28.0	NA	8.000	46.000	1.800	1.200	NA	NA	NA	NA	NA	NA	NA	37191	NA	NA
07/17/89	5.60	4.4	1900	1350	342.0	78.0	30.0	NA	6.700	30.000	1.100	1.100	NA	NA	NA	NA	NA	NA	NA	3959	NA	NA
08/14/89	5.83	5.1	1400	880	206.0	54.0	20.0	NA	7.500	41.000	1.400	1.200	NA	NA	NA	NA	NA	NA	NA	4285	NA	NA
09/14/89	5.58	5.5	2100	1480	384.0	98.0	30.0	NA	13.100	39.000	2.300	1.600	NA	NA	NA	NA	NA	NA	NA	26192	NA	NA

Table A2.1. Drainage quality data from Duluth Complex test pile FL1, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
10/16/89	5.88	2.3	1400	800	212.0	54.0	22.0	NA	5.900	25.000	1.900	1.000	NA	NA	NA	NA	NA	NA	NA	2566	NA	NA
11/06/89	5.50	2.4	1175	680	166.0	50.0	16.0	NA	5.300	25.000	2.600	1.000	NA	NA	NA	NA	NA	NA	NA	745	NA	NA
03/26/90	5.05	3.8	600	290	64.0	21.4	7.4	NA	10.200	19.000	10.200	1.010	NA	NA	NA	NA	NA	NA	NA	742	NA	NA
04/12/90	5.82	1.1	975	685	148.0	42.0	10.0	NA	9.430	32.000	1.540	1.540	NA	NA	NA	NA	NA	NA	NA	5984	NA	NA
05/17/90	5.72	2.1	1800	1250	286.0	84.0	20.0	NA	10.000	39.000	1.910	1.780	NA	NA	NA	NA	NA	NA	NA	7127	NA	NA
06/18/90	5.86	2.0	1685	1040	202.0	70.0	22.0	NA	7.780	30.000	1.270	1.140	NA	NA	NA	NA	NA	NA	NA	10465	NA	NA
07/16/90	5.79	3.2	1650	975	214.0	66.0	20.0	NA	7.820	27.000	1.250	1.190	NA	NA	NA	NA	NA	NA	NA	6972	NA	NA
08/13/90	5.77	1.6	1575	920	228.0	66.0	22.0	NA	5.210	21.000	1.010	0.930	NA	NA	NA	NA	NA	NA	NA	1544	NA	NA
09/10/90	5.42	3.2	1715	940	212.0	74.0	18.0	NA	20.000	56.000	2.820	2.270	NA	NA	NA	NA	NA	NA	NA	3978	NA	NA
10/15/90	5.29	2.7	2600	1650	358.0	134.0	30.0	NA	35.000	93.000	4.350	3.100	NA	NA	NA	NA	NA	NA	NA	22191	NA	NA
11/16/90	5.31	1.9	3875	2400	680.0	202.0	54.0	NA	29.000	88.000	4.040	3.550	NA	NA	NA	NA	NA	NA	NA	1098	NA	NA
04/08/91	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	4179	NA	NA
04/22/91	4.96	2.5	1650	1030	194.0	88.0	16.0	NA	19.900	68.000	2.600	1.800	NA	NA	NA	NA	NA	NA	NA	8005	NA	NA
05/20/91	5.18	2.5	2375	1200	262.0	107.0	20.0	NA	20.300	68.000	2.400	1.500	NA	NA	NA	NA	NA	NA	NA	11983	NA	NA
06/17/91	5.25	2.0	2050	1110	214.0	104.0	20.0	NA	27.000	71.000	3.000	1.600	NA	NA	NA	NA	NA	NA	NA	11987	NA	NA
07/15/91	5.27	3.1	2725	1260	278.0	124.0	26.0	NA	21.000	59.000	2.400	1.500	NA	NA	NA	NA	NA	NA	NA	32415	NA	NA
08/12/91	5.27	2.4	2550	1370	318.0	110.0	34.0	NA	18.000	54.000	2.300	1.300	NA	NA	NA	NA	NA	NA	NA	3463	NA	NA
09/09/91	5.22	3.6	2675	1420	268.0	110.0	28.0	NA	30.000	84.000	3.400	2.000	NA	NA	NA	NA	NA	NA	NA	19792	NA	NA
10/07/91	5.31	3.6	2575	1360	272.0	108.0	32.0	NA	21.000	67.000	2.700	1.700	NA	NA	NA	NA	NA	NA	NA	7025	NA	NA
11/07/91	5.24	3.1	2475	1270	252.0	112.0	30.0	NA	27.000	80.000	3.200	2.100	NA	NA	NA	NA	NA	NA	NA	22131	NA	NA

NA: Not analyzed.

'Dates represent intermediate dates for measurement periods.

Table A2.1.1. Drainage quality data from Duluth Complex test pile FL1, 1992-1993 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH	Alk. mg/L	Lab Acy. mg/l.	Calc. Acy. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/23/92	5.35	3.8	39	54.0	875	400	86.0	36.0	10.60	10.50	<1	8.60	24	1.00	0.80	0.03	1.19	NA	NA	NA	NA	NA	1513	NA	NA
05/11/92	5.08	3.2	60	105.1	1340	630	122.0	54.0	13.40	12.00	<1	14.90	46	1.90	1.20	0.02	2.06	NA	NA	NA	NA	NA	2724	NA	NA
05/26/92	5.18	1.9	70	99.7	1380	755	148.0	68.0	18.80	18.20	<1	16.50	42	1.60	1.00	0.03	1.88	NA	NA	NA	NA	NA	4053	NA	NA
06/08/92	5.18	3.8	100	140.3	2470	1490	294.0	126.0	31.80	31.00	<1	20.00	61	2.30	1.40	0.04	2.38	NA	NA	NA	NA	NA	798	NA	NA
06/22/92	5.29	2.6	63	101.9	1600	920	172.0	76.0	21.80	23.00	<1	16.70	43	1.60	1.10	0.03	1.87	NA	NA	NA	NA	NA	914	NA	NA
07/07/92	5.15	2.1	85	145.5	2030	430	222.0	96.0	26.60	28.00	<1	24.00	60	2.50	1.40	0.04	2.53	NA	NA	NA	NA	NA	3350	NA	NA
07/20/92	5.19	3.1	90	119.6	2250	1405	246.0	106.0	29.80	30.00	<1	19.60	50	2.10	1.20	0.04	2.02	NA	NA	NA	NA	NA	1223	NA	NA
08/03/92	5.19	2.6	50	93.5	1180	815	166.0	72.0	23.00	23.00	<1	15.90	39	1.70	1.00	0.02	1.72	NA	NA	NA	NA	NA	1118	NA	NA
08/17/92	5.11	2.1	63	124.7	1900	940	184.0	80.0	22.00	25.00	<1	21.00	49	3.20	2.60	0.10	2.25	NA	NA	NA	NA	NA	1354	NA	NA
08/31/92	5.17	5.0	65	110.7	1400	800	142.0	68.0	17.80	23.00	<1	18.50	46	1.90	1.10	0.10	2.01	NA	NA	NA	NA	NA	1875	NA	NA
09/14/92	5.21	2.1	78	136.6	1920	1290	256.0	110.0	27.80	31.00	<1	24.00	55	2.40	1.40	0.01	2.41	NA	NA	NA	NA	NA	4141	NA	NA
09/28/92	5.12	3.0	80	127.3	2300	1450	292.0	122.0	33.00	34.00	<1	20.00	54	2.20	1.20	0.02	2.09	NA	NA	NA	NA	NA	2579	NA	NA
10/12/92	5.24	3.0	81	138.2	2800	1780	346.0	152.0	40.80	37.00	<1	21.00	59	2.40	1.40	0.01	2.16	NA	NA	NA	NA	NA	539	NA	NA
10/27/92	5.07	2.6	75	125.5	2300	2800	270.0	118.0	31.40	31.00	<1	17.70	55	2.20	1.20	0.01	2.13	NA	NA	NA	NA	NA	388	NA	NA
11/09/92	5.23	3.2	NA	NA	2850	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	129	NA	NA
11/30/92	5.22	2.7	95	112.4	1500	880	166.0	80.0	19.20	22.00	<1	18.30	47	2.00	1.20	0.02	2.06	NA	NA	NA	NA	NA	767	NA	NA
04/06/93	5.72	2.0	NA	NA	500	268	50.0	22.5	6.00	6.00	<1	5.87	14	0.52	0.39	0.01	0.63	NA	NA	NA	NA	NA	3698	NA	NA
04/22/93	5.06	2.5	NA	NA	880	468	77.5	40.0	7.50	8.00	<1	10.20	30	1.32	0.73	0.03	1.25	NA	NA	NA	NA	NA	3550	NA	NA
05/06/93	5.13	2.5	NA	NA	975	580	92.5	47.5	10.25	9.00	<1	13.00	39	1.70	0.87	0.05	1.58	NA	NA	NA	NA	NA	10848	NA	NA
05/20/93	5.24	2.0	NA	NA	1500	1095	177.5	82.5	14.75	16.00	<1	16.00	57	2.20	1.05	0.04	1.97	NA	NA	NA	NA	NA	3588	NA	NA
06/03/93	5.27	3.0	NA	NA	1720	1130	180.0	87.5	17.50	16.75	<1	25.00	66	2.50	1.35	0.05	2.24	NA	NA	NA	NA	NA	6639	NA	NA
06/17/93	5.11	2.5	NA	NA	1725	1375	242.5	117.5	21.75	21.00	<1	31.00	83	3.17	1.64	0.03	2.77	NA	NA	NA	NA	NA	7366	NA	NA
07/01/93	5.04	2.0	NA	NA	2100	1190	240.0	122.5	22.50	23.50	<1	33.00	80	3.07	1.58	0.04	2.64	NA	NA	NA	NA	NA	8691	NA	NA
07/14/93	5.15	0.0	NA	NA	2000	1260	234.0	100.0	22.40	26.40	<1	18.00	53	1.93	1.05	0.02	2.07	NA	NA	NA	NA	NA	20482	NA	NA
07/29/93	5.26	3.5	NA	NA	1350	895	152.0	68.0	15.00	19.00	<1	16.00	42	1.57	0.89	0.02	1.69	NA	NA	NA	NA	NA	13028	NA	NA
08/12/93	5.03	3.5	NA	NA	2050	1180	248.0	102.0	22.20	27.60	<1	21.00	57	1.98	1.22	0.02	2.30	NA	NA	NA	NA	NA	21862	NA	NA
08/26/93	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	3777	NA	NA
09/09/93	5.23	2.5	NA	NA	1600	660	176.0	76.0	17.20	21.20	<1	18.00	44	1.69	1.03	0.01	1.95	NA	NA	NA	NA	NA	10886	NA	NA
09/23/93	5.20	4.0	NA	NA	1900	1140	254.0	98.0	22.80	27.20	<1	20.00	52	1.94	1.20	0.03	2.28	NA	NA	NA	NA	NA	4315	NA	NA
10/07/93	5.17	2.5	NA	NA	2210	1240	278.0	118.0	26.00	29.60	<1	23.00	64	2.19	1.42	0.02	2.64	NA	NA	NA	NA	NA	2608	NA	NA
10/20/93	5.03	2.0	NA	NA	2800	1580	326.0	140.0	29.40	33.20	<1	25.00	72	2.60	1.41	0.02	2.67	NA	NA	NA	NA	NA	999	NA	NA
11/04/93	5.12	2.3	NA	NA	2200	1185	242.0	110.0	22.20	24.40	<1	25.00	68	2.67	1.33	0.53	2.62	NA	NA	NA	NA	NA	2112	NA	NA
11/22/93	5.03	2.0	NA	NA	2400	1060	250.0	116.0	24.60	24.20	<1	26.00	65	2.67	1.46	0.04	2.51	NA	NA	NA	NA	NA	519	NA	NA

NA: Not analyzed.

'Dates represent intermediate dates for measurement periods.

Table A2.2. Drainage quality data from Duluth Complex test pile FL2, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.20	NA	1000	NA	99.0	13.0	101.0	12.0	0.038	NA	0.003	0.020	NA	0.07	NA	NA	NA	NA	NA	NA	NA	4.0
05/25/78	7.70	NA	1900	NA	177.0	23.0	241.0	15.0	0.040	0.240	0.008	0.028	0.05	0.16	NA	NA	NA	NA	NA	NA	NA	NA
06/16/78	7.72	NA	4200	NA	323.0	47.0	555.0	23.0	0.056	0.200	0.015	0.075	0.03	0.31	NA	NA	NA	NA	NA	496	NA	11.0
07/05/78	7.50	NA	2700	1400	218.0	33.0	313.0	28.0	0.057	0.316	0.021	0.084	0.04	0.22	34.0	NA	NA	NA	NA	293	NA	12.0
07/13/78	7.92	NA	NA	800	491.0	43.0	376.0	37.0	0.049	0.410	0.033	0.090	0.03	0.29	30.0	12.0	1.00	0.08	NA	NA	NA	NA
07/19/78	7.80	NA	1300	800	291.0	24.0	189.0	28.0	0.062	0.240	0.014	0.033	0.01	0.16	41.0	10.0	0.75	0.10	NA	NA	NA	NA
08/02/78	7.75	NA	2000	1100	191.0	21.0	241.0	26.0	0.080	0.260	0.010	0.060	0.04	0.18	14.0	8.0	0.20	0.20	NA	NA	NA	19.0
08/04/78	7.75	NA	2500	1700	341.0	42.0	410.0	33.0	0.070	0.440	0.029	0.113	0.05	0.28	19.0	11.0	0.06	0.08	NA	1000	NA	22.0
08/16/78	7.58	39.5	1092	500	108.0	14.3	95.7	16.7	0.140	0.180	0.023	0.027	0.05	0.09	17.0	NA	0.10	0.08	NA	30	NA	23.0
08/23/78	7.60	31.0	1100	460	103.0	15.0	86.0	18.0	0.140	0.180	0.020	0.020	0.04	0.10	15.0	5.3	1.10	0.10	NA	6379	NA	NA
08/25/78	7.30	35.0	1600	680	170.0	24.0	125.0	28.0	0.110	0.240	0.030	0.050	0.01	0.15	25.0	3.6	NA	0.10	NA	NA	NA	23.0
08/28/78	7.55	48.0	2300	1100	253.0	34.0	196.0	32.0	0.100	0.380	0.045	0.090	0.01	0.23	34.0	8.5	1.30	0.08	NA	NA	NA	NA
09/13/78	7.25	29.0	1338	584	133.0	16.0	106.0	26.5	0.120	0.190	0.020	0.050	0.03	0.11	12.0	5.5	0.06	0.08	NA	451	NA	15.0
03/19/79	7.70	NA	1043	NA	83.2	11.0	43.8	15.8	0.119	0.170	0.016	0.050	0.06	0.08	NA	NA	NA	NA	12.0	2612	1891	NA
03/27/79	NA	NA	NA	NA	205.0	28.4	86.0	24.6	0.180	0.460	0.060	0.060	0.05	0.18	NA	NA	NA	NA	NA	53	NA	NA
04/17/79	6.90	22.0	388	136	74.2	5.8	25.6	11.8	0.108	0.134	0.008	0.030	0.06	0.05	10.0	136.0	NA	NA	14.0	5151	3480	NA
04/18/79	6.90	16.5	177	51	27.5	2.6	18.0	6.1	0.091	0.040	0.004	0.020	0.05	0.03	5.5	1.2	NA	NA	10.5	6056	3654	NA
04/20/79	6.50	21.0	296	100	49.0	5.6	22.0	8.0	0.113	0.080	0.008	0.020	0.06	0.05	12.0	2.2	0.45	NA	11.0	3263	3830	NA
04/22/79	7.50	18.0	1163	633	175.4	24.2	82.8	18.6	0.100	0.300	0.020	0.060	0.04	0.16	30.0	11.0	0.11	NA	NA	5020	1296	NA
05/09/79	7.40	NA	1217	500	154.6	22.0	83.0	18.1	0.084	0.295	0.020	0.040	0.04	0.14	26.0	NA	NA	NA	NA	825	1083	NA
05/10/79	7.40	NA	1009	400	144.4	20.0	59.6	16.6	0.100	0.250	0.015	0.040	0.03	0.13	22.0	NA	NA	NA	NA	6810	3129	NA
05/20/79	6.90	NA	1562	800	233.4	32.6	116.8	23.2	0.100	0.330	0.023	0.050	0.03	0.18	25.0	NA	NA	NA	14.0	3611	1034	NA
05/22/79	7.40	NA	1128	567	165.8	22.8	83.6	18.4	0.180	0.280	0.020	0.040	0.04	0.14	20.0	NA	NA	NA	16.0	1639	1041	NA
05/23/79	7.60	NA	1976	NA	286.0	39.2	174.4	28.6	0.100	0.440	0.027	0.070	0.21	0.23	NA	NA	NA	NA	4.7	2286	609	NA
06/02/79	7.60	NA	1683	883	217.2	31.8	164.2	24.0	0.160	0.340	0.023	0.050	0.07	0.18	23.0	NA	NA	NA	16.0	1518	474	NA
06/06/79	7.80	NA	1749	900	220.8	33.8	85.0	29.6	0.170	0.300	0.021	0.050	0.03	0.17	23.0	NA	NA	NA	NA	1999	531	NA
06/10/79	7.60	45.0	1414	580	173.4	27.8	71.8	24.6	0.190	0.270	0.013	0.030	0.06	0.13	19.0	NA	NA	NA	NA	2021	1211	22.0
06/12/79	7.00	43.0	2317	1200	277.0	44.6	107.2	40.0	0.132	0.370	0.017	0.070	0.02	0.29	27.0	NA	NA	NA	NA	1033	361	NA
06/16/79	7.70	51.0	2365	1350	307.4	49.1	179.0	39.8	0.131	0.415	0.029	0.080	0.03	0.23	31.0	NA	NA	NA	NA	1344	368	25.0
06/20/79	7.10	36.0	1363	570	158.0	27.0	139.0	22.0	0.130	0.220	0.017	0.040	0.04	0.12	15.0	NA	NA	NA	NA	2200	1011	18.0
06/30/79	7.60	59.0	2536	1450	336.0	55.0	339.0	31.0	0.070	0.430	0.036	0.090	0.03	0.13	31.0	NA	NA	NA	16.0	1132	170	12.0
07/10/79	7.70	45.0	3294	2300	420.0	72.0	430.0	38.0	0.060	0.590	0.052	0.110	0.02	0.34	28.0	NA	NA	NA	NA	1249	142	19.0
07/30/79	7.10	17.8	980	480	106.0	18.0	88.0	17.0	0.110	0.170	0.014	0.030	0.03	0.09	16.0	NA	NA	NA	14.0	2779	2096	22.0
08/04/79	7.70	30.8	1826	1020	253.0	36.0	183.0	32.0	0.070	0.360	0.030	0.070	0.07	0.19	15.0	NA	NA	NA	9.2	443	65	23.0
08/13/79	NA	NA	NA	NA	385.6	66.6	408.0	38.6	0.050	0.650	0.035	0.080	0.04	0.30	NA	NA	NA	NA	NA	91	11	NA
08/23/79	NA	NA	NA	NA	217.2	33.2	200.0	25.8	0.100	0.470	0.027	0.070	0.04	0.21	NA	NA	NA	NA	14.0	114	70	NA
08/28/79	7.20	13.6	1338	NA	157.0	25.2	132.6	20.0	0.150	0.360	0.021	0.060	0.07	0.15	NA	NA	NA	NA	15.0	326	293	23.0
09/01/79	7.10	18.4	1087	650	142.8	19.8	84.2	19.8	0.150	0.280	0.020	0.050	0.05	0.10	9.0	NA	NA	NA	13.0	1526	905	21.0
09/04/79	7.20	20.8	1380	NA	211.4	29.0	101.4	25.8	0.120	0.370	0.020	0.070	0.02	0.16	NA	NA	NA	NA	13.0	1775	179	20.0
09/10/79	7.20	15.2	1134	600	165.6	22.0	104.0	21.2	0.150	0.340	0.020	0.050	0.03	0.14	9.0	NA	NA	NA	14.0	1268	249	17.0
09/13/79	7.30	17.6	1587	NA	233.8	31.6	156.0	26.8	0.150	0.450	0.026	0.090	0.05	0.20	NA	NA	NA	NA	14.0	367	211	18.0
09/19/79	7.30	20.8	2592	NA	379.0	53.4	238.2	37.2	0.060	0.760	0.070	0.120	0.02	0.30	NA	NA	NA	NA	13.0	129	22	18.0
10/19/79	6.60	12.8	914	542	112.5	16.1	64.0	14.9	0.195	0.365	0.020	0.040	0.06	0.10	5.0	NA	NA	NA	14.0	1726	2456	NA
10/31/79	6.70	13.6	1319	817	194.0	25.2	84.0	20.8	0.160	0.490	0.050	0.090	0.05	0.16	4.0	NA	NA	NA	13.0	2835	1118	NA
11/01/79	7.20	17.6	1659	975	280.0	34.2	98.0	25.8	0.110	0.550	0.060	0.080	0.02	0.16	10.0	NA	NA	NA	13.0	1215	637	NA
04/01/80	7.10	17.3	350	135	33.2	4.6	17.8	7.8	0.135	0.095	0.010	0.030	0.08	0.04	3.5	NA	NA	NA	10.0	1582	382	10.5
04/16/80	7.50	35.0	600	225	68.2	8.2	33.8	13.2	0.130	0.130	0.030	0.060	0.03	0.06	5.0	NA	NA	NA	NA	856	831	NA
04/17/80	7.20	33.5	NA	253	69.0	9.0	30.2	10.3	0.150	0.210	0.020	0.050	0.07	0.07	5.5	NA	NA	NA	12.0	1893	774	NA
04/29/80	7.57	41.5	1105	820	157.1	19.8	46.0	16.0	0.100	0.360	0.025	0.060	0.02	0.12	9.0	NA	NA	NA	NA	1117	163	13.0
05/08/80	7.40	34.1	1230	880	190.0	24.2	58.6	18.8	0.070	0.450	0.040	0.060	0.02	0.14	10.0	NA	NA	NA	9.5	541	51	11.0
05/18/80	7.50	30.8	1500	1050	236.0	31.2	92.0	23.0	0.050	0.600	0.050	0.100	0.03	0.18	23.0	NA	NA	NA	NA	371	26	19.0

Table A2.2. Drainage quality data from Duluth Complex test pile FL2, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
05/31/80	6.90	16.5	875	620	129.4	17.2	45.5	16.2	0.030	0.300	0.022	0.020	0.02	0.10	14.0	NA	NA	NA	15.0	1703	1084	NA
06/03/80	7.30	25.3	1400	700	207.4	25.2	68.8	21.6	0.110	0.570	0.030	0.100	0.02	0.13	15.0	NA	NA	NA	9.8	174	50	NA
06/06/80	7.20	23.1	1300	760	174.6	22.2	63.2	20.0	0.150	0.490	0.025	0.090	0.03	0.12	14.0	NA	NA	NA	NA	371	235	NA
06/13/80	7.10	20.9	1200	750	148.8	21.4	65.6	19.6	0.150	0.450	0.022	0.080	0.03	0.11	12.0	NA	NA	NA	NA	265	197	NA
06/19/80	7.20	23.1	1650	950	221.0	28.0	98.8	23.4	0.110	0.580	0.021	0.110	0.06	0.15	16.0	NA	NA	NA	NA	23	12	NA
06/30/80	7.10	17.6	1150	660	158.0	20.0	58.0	17.0	0.130	0.480	0.070	0.100	0.05	0.13	10.0	NA	NA	NA	10.0	114	38	NA
07/02/80	7.10	15.3	950	580	132.0	16.0	45.0	16.0	0.170	0.380	0.022	0.090	0.04	0.10	9.0	NA	NA	NA	NA	246	96	NA
07/17/80	6.90	17.6	1150	450	155.6	19.0	58.4	19.6	0.140	0.500	0.030	0.090	0.03	0.14	10.0	NA	NA	NA	NA	95	124	NA
07/23/80	7.00	17.6	1100	350	149.8	18.6	66.4	19.2	0.110	0.470	0.060	0.090	0.02	0.15	9.0	NA	NA	NA	NA	95	525	NA
08/03/80	6.60	12.1	725	355	91.4	12.2	35.0	14.8	0.260	0.400	0.022	0.070	0.05	0.12	9.0	NA	NA	NA	16.0	106	95	NA
08/12/80	6.60	9.8	600	258	70.9	9.3	25.7	12.8	0.260	0.340	0.015	0.050	0.10	0.11	7.0	NA	NA	NA	14.0	1026	525	NA
08/19/80	6.80	10.2	700	424	103.6	13.0	28.8	16.0	0.190	0.450	0.024	0.080	0.05	0.16	6.0	NA	NA	NA	NA	379	106	NA
08/22/80	NA	NA	600	280	77.1	8.1	24.1	11.7	0.220	0.390	0.050	0.070	0.07	0.12	6.0	NA	NA	NA	NA	428	329	NA
08/29/80	6.70	15.3	1450	920	304.0	36.4	68.2	27.0	0.170	0.830	0.080	0.160	0.06	0.22	11.0	NA	NA	NA	10.0	2271	1473	NA
09/07/80	6.60	10.2	843	360	151.4	15.8	44.0	16.4	0.190	0.610	0.050	0.110	0.04	0.16	7.0	NA	NA	NA	NA	1450	1000	NA
09/10/80	6.80	11.5	1238	700	252.2	30.6	71.0	22.8	0.180	0.860	0.060	0.150	0.04	0.23	10.0	NA	NA	NA	NA	345	88	13.2
09/12/80	6.70	12.0	969	450	188.4	19.6	47.6	17.4	0.210	0.590	0.050	0.110	0.06	0.16	9.0	NA	NA	NA	13.0	2309	1578	17.5
09/18/80	6.90	12.3	1231	690	241.8	29.8	70.6	20.2	0.240	0.760	0.060	0.130	0.08	0.16	11.0	NA	NA	NA	NA	5330	1938	13.0
09/23/80	6.90	12.8	2527	1700	398.0	72.8	204.0	40.0	0.200	1.780	0.140	0.250	0.04	0.31	25.0	NA	NA	NA	13.0	1821	612	15.0
10/22/80	6.70	13.6	2664	1620	404.0	79.0	330.0	48.4	0.240	1.910	0.120	0.310	0.04	0.35	29.0	NA	NA	NA	13.0	1628	322	10.0
10/23/80	6.40	8.5	1763	950	214.0	32.0	170.0	23.0	0.190	1.260	0.090	0.200	0.04	0.25	31.0	NA	NA	NA	NA	795	268	12.0
02/18/81	6.70	8.5	408	160	31.8	6.0	39.0	NA	0.210	0.295	0.020	0.060	NA	NA	10.0	NA	NA	NA	NA	1760	1300	6.0
03/29/81	6.70	8.8	330	90	31.2	6.2	24.7	NA	0.210	0.220	0.020	0.050	NA	NA	NA	NA	NA	NA	15.0	6719	1726	NA
03/30/81	6.80	12.8	525	240	63.7	10.9	20.8	NA	0.280	0.490	0.030	0.080	NA	NA	NA	NA	NA	NA	NA	4236	1700	NA
04/13/81	NA	NA	800	250	69.0	12.0	21.0	NA	0.290	0.770	0.060	0.080	NA	NA	NA	NA	NA	NA	NA	2142	1281	NA
04/16/81	7.10	14.0	800	315	78.0	15.0	22.0	NA	0.250	0.920	0.060	0.090	NA	NA	NA	NA	NA	NA	15.0	1427	212	NA
04/23/81	7.10	13.6	900	300	87.0	17.0	27.0	NA	0.260	0.970	0.050	0.090	NA	NA	NA	NA	NA	NA	NA	2423	2478	NA
04/25/81	7.10	19.2	1155	530	129.0	30.0	36.0	NA	0.240	1.180	0.080	0.100	NA	NA	NA	NA	NA	NA	NA	4857	1589	NA
04/27/81	7.30	24.0	1250	600	136.0	35.0	48.0	NA	0.260	1.260	0.080	0.100	NA	NA	NA	NA	NA	NA	14.0	3869	2091	NA
04/30/81	7.30	24.0	1596	870	210.0	40.0	80.0	NA	0.200	1.700	0.100	0.140	NA	NA	NA	NA	NA	NA	NA	3206	994	NA
05/12/81	7.20	23.0	2179	1250	306.0	54.0	166.0	NA	0.130	2.100	0.140	0.170	NA	NA	NA	NA	NA	NA	14.0	3539	203	17.5
06/01/81	7.25	14.0	2116	1250	266.0	54.0	200.0	NA	0.160	1.160	0.070	0.150	NA	NA	NA	NA	NA	NA	NA	2370	705	17.0
06/06/81	7.20	18.2	1959	1000	234.0	44.0	174.0	NA	0.170	0.960	0.070	0.130	NA	NA	NA	NA	NA	NA	13.0	1522	150	18.0
06/22/81	6.80	11.2	1270	625	144.0	26.0	102.0	NA	0.200	0.710	0.060	0.090	NA	NA	NA	NA	NA	NA	NA	2411	1030	16.0
06/24/81	6.90	12.0	793	870	96.0	14.0	46.0	NA	0.250	0.550	0.040	0.060	NA	NA	NA	NA	NA	NA	NA	1874	993	17.0
06/28/81	6.80	11.2	718	160	92.0	14.0	36.0	NA	0.300	0.640	0.040	0.070	NA	NA	NA	NA	NA	NA	18.0	2612	1938	15.0
07/04/81	7.10	13.5	1630	1060	254.0	36.6	98.0	NA	0.270	1.370	0.090	0.140	NA	NA	NA	NA	NA	NA	NA	1037	53	22.0
07/15/81	7.00	11.7	2769	1800	388.0	68.4	259.6	NA	0.140	2.550	0.150	0.220	NA	NA	NA	NA	NA	NA	NA	575	26	19.0
07/24/81	7.10	10.5	2142	1320	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	21.0	NA	NA	NA	9.1	492	122	20.0
08/04/81	6.80	8.4	2864	2000	364.0	80.0	154.0	NA	0.190	2.440	0.160	0.310	NA	NA	NA	NA	NA	NA	NA	246	25	18.0
08/13/81	6.90	12.2	1926	1250	216.0	42.0	184.0	28.0	0.130	1.260	0.110	0.170	NA	NA	NA	NA	NA	NA	NA	64	6	22.0
08/31/81	6.00	8.0	1700	800	162.0	32.0	140.0	NA	0.420	1.230	0.090	0.160	NA	NA	NA	NA	NA	NA	15.0	454	382	NA
09/07/81	6.60	6.8	1800	720	154.0	29.0	124.0	NA	0.280	1.190	0.080	0.160	NA	NA	NA	NA	NA	NA	NA	284	255	NA
09/26/81	6.40	4.8	1093	490	108.0	20.0	70.0	NA	0.380	1.060	0.060	0.140	NA	NA	NA	NA	NA	NA	11.0	768	530	NA
10/01/81	6.60	4.8	750	260	90.0	16.0	32.0	NA	0.350	0.880	0.070	0.120	NA	NA	NA	NA	NA	NA	NA	1968	1257	NA
10/04/81	6.45	4.4	825	380	118.0	16.0	34.0	16.0	0.310	0.960	0.080	0.160	NA	NA	NA	NA	NA	NA	NA	852	1060	NA
10/13/81	6.65	5.0	950	400	132.0	20.0	34.0	18.0	0.310	1.370	0.100	0.200	NA	NA	NA	NA	NA	NA	9.7	3937	1603	NA
10/17/81	6.60	5.6	1600	870	238.0	36.0	58.0	28.0	0.510	4.200	0.240	0.360	NA	NA	NA	NA	NA	NA	NA	3896	810	NA
11/05/81	6.55	6.4	3800	2400	470.0	100.0	304.0	56.0	1.000	5.200	0.300	0.510	NA	NA	NA	NA	NA	NA	9.6	2839	99	NA
04/24/82	7.00	14.4	620	290	98.0	10.0	32.0	12.0	0.130	0.570	0.030	0.110	NA	NA	NA	NA	NA	NA	NA	3691	NA	NA
04/26/82	6.80	11.5	870	420	93.0	16.0	29.5	14.0	0.315	1.770	0.100	0.140	NA	NA	NA	NA	NA	NA	NA	4047	NA	NA

Table A2.2. Drainage quality data from Duluth Complex test pile FL2, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
05/03/82	7.20	10.8	1000	520	128.0	22.0	56.0	16.0	0.160	1.850	0.110	0.170	NA	NA	NA	NA	NA	NA	NA	969	NA	NA
05/07/82	6.70	9.0	780	370	82.0	14.0	20.0	12.0	0.320	1.800	0.110	0.130	NA	NA	NA	NA	NA	NA	NA	4789	NA	NA
05/11/82	6.80	12.2	1150	610	140.0	30.0	36.0	16.0	0.440	3.230	0.160	0.190	NA	NA	NA	NA	NA	NA	NA	3937	NA	NA
05/14/82	6.80	13.5	1450	730	174.0	25.4	40.4	14.6	0.320	2.100	0.100	0.140	0.03	0.15	NA	NA	NA	NA	NA	2623	NA	NA
05/21/82	6.90	15.3	2500	1500	336.0	63.0	143.0	32.0	0.410	4.030	0.210	0.270	0.04	0.24	NA	NA	NA	NA	NA	2248	NA	NA
06/02/82	6.65	12.6	2100	1300	246.0	49.0	134.0	26.0	0.350	2.500	0.130	0.200	0.04	0.20	NA	NA	NA	NA	NA	1787	NA	NA
06/18/82	6.60	11.7	1775	940	194.0	37.0	124.0	25.0	0.280	1.920	0.100	0.160	0.03	0.15	NA	NA	NA	NA	NA	390	NA	NA
07/02/82	6.50	8.1	950	480	116.0	20.0	56.0	16.0	0.350	1.300	0.060	0.100	0.03	0.11	NA	NA	NA	NA	NA	1098	NA	NA
07/08/82	6.50	9.0	1100	600	190.0	26.0	46.0	18.0	0.440	2.350	0.130	0.130	0.06	0.15	NA	NA	NA	NA	NA	3373	NA	NA
07/12/82	NA	NA	NA	1015	326.0	56.0	144.0	30.0	0.460	3.030	0.200	0.180	0.01	0.23	NA	NA	NA	NA	NA	4119	NA	NA
07/26/82	6.65	9.7	2171	1030	286.0	52.0	146.0	32.0	0.320	4.110	0.250	0.180	0.05	0.26	NA	NA	NA	NA	NA	6568	NA	NA
08/30/82	6.47	6.1	1356	800	234.0	34.0	76.0	24.0	0.460	2.040	0.130	0.160	0.06	0.17	NA	NA	NA	NA	NA	5765	NA	NA
10/04/82	6.80	8.8	2888	1600	340.0	80.0	216.0	42.0	0.800	5.890	0.380	0.250	0.07	0.38	NA	NA	NA	NA	NA	4096	NA	NA
11/01/82	6.45	5.2	1662	NA	199.0	32.0	76.0	16.0	0.360	2.205	0.125	0.100	0.03	0.13	NA	NA	NA	NA	NA	1643	NA	NA
11/22/82	NA	NA	NA	NA	296.0	49.0	192.0	NA	0.545	2.715	0.155	0.160	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
04/14/83	6.57	7.0	522	220	NA	NA	NA	NA	0.450	1.820	0.100	0.130	NA	NA	NA	NA	NA	NA	NA	9591	NA	NA
05/15/83	6.62	7.3	1115	580	NA	NA	NA	NA	0.590	3.470	0.200	0.270	NA	NA	NA	NA	NA	NA	NA	3826	NA	NA
06/16/83	6.48	6.2	1171	660	NA	NA	NA	NA	0.390	2.030	0.130	0.150	NA	NA	NA	NA	NA	NA	NA	1778	NA	NA
07/14/83	6.45	5.2	1164	660	NA	NA	NA	NA	0.620	3.580	0.190	0.240	NA	NA	NA	NA	NA	NA	NA	3993	NA	NA
08/14/83	6.08	3.9	1064	600	NA	NA	NA	NA	0.830	3.920	0.190	0.280	NA	NA	NA	NA	NA	NA	NA	6223	NA	NA
09/15/83	6.25	3.7	1200	730	NA	NA	NA	NA	1.140	4.710	0.280	0.380	NA	NA	NA	NA	NA	NA	NA	5095	NA	NA
10/13/83	6.39	5.0	1739	1340	NA	NA	NA	NA	2.120	6.830	0.400	0.460	NA	NA	NA	NA	NA	NA	NA	16559	NA	NA
11/09/83	6.25	2.7	1025	1380	NA	NA	NA	NA	1.580	4.770	0.270	0.360	NA	NA	NA	NA	NA	NA	NA	5053	NA	NA
11/30/83	6.10	2.7	1325	940	NA	NA	NA	NA	1.520	5.450	0.300	0.400	NA	NA	NA	NA	NA	NA	NA	276	NA	NA
04/15/84	6.59	9.9	936	700	198.3	37.5	45.8	NA	0.410	3.210	0.170	0.400	NA	NA	NA	NA	NA	NA	NA	7737	NA	NA
05/16/84	6.54	4.9	641	400	130.1	24.2	19.9	NA	0.670	4.120	0.140	0.190	NA	NA	NA	NA	NA	NA	NA	7835	NA	NA
06/13/84	6.39	4.9	950	680	172.7	35.6	32.7	NA	1.340	5.440	0.260	0.290	NA	NA	NA	NA	NA	NA	NA	11065	NA	NA
07/14/84	6.36	6.1	1460	1080	NA	NA	71.9	NA	1.300	6.330	0.240	0.300	NA	NA	NA	NA	NA	NA	NA	2710	NA	NA
08/15/84	6.35	4.8	1260	700	195.4	58.6	53.2	NA	0.950	4.820	0.270	0.270	NA	NA	NA	NA	NA	NA	NA	2172	NA	NA
09/12/84	5.99	3.4	1150	710	190.0	59.9	64.2	NA	1.250	5.310	0.310	0.330	NA	NA	NA	NA	NA	NA	NA	1086	NA	NA
10/13/84	5.78	2.7	1075	740	215.9	56.9	29.2	NA	1.920	8.500	0.600	0.540	NA	NA	NA	NA	NA	NA	NA	5443	NA	NA
11/14/84	5.82	3.2	1140	1100	280.0	60.0	51.0	NA	2.500	8.850	0.410	0.640	NA	NA	NA	NA	NA	NA	NA	1162	NA	NA
11/30/84	5.75	2.1	1125	710	264.0	60.0	54.0	NA	2.500	11.060	0.400	0.620	NA	NA	NA	NA	NA	NA	NA	98	NA	NA
12/18/84	5.75	2.1	1175	900	262.0	78.0	46.0	NA	3.900	14.400	0.830	0.810	NA	NA	NA	NA	NA	NA	NA	1123	NA	NA
04/02/85	6.30	4.5	260	140	37.2	7.2	8.4	NA	0.410	1.260	0.030	0.230	NA	NA	NA	NA	NA	NA	NA	867	NA	NA
04/16/85	6.50	7.6	650	420	100.0	25.4	17.8	NA	1.010	5.620	0.270	0.390	NA	NA	NA	NA	NA	NA	NA	15760	NA	NA
05/14/85	6.32	5.7	1215	780	189.0	41.6	27.5	NA	2.210	10.100	0.490	0.530	NA	NA	NA	NA	NA	NA	NA	7403	NA	NA
06/14/85	6.35	3.5	1525	1200	235.0	57.0	42.8	NA	3.170	11.600	0.490	0.520	NA	NA	NA	NA	NA	NA	NA	21991	NA	NA
07/15/85	6.33	6.2	1900	1500	280.0	76.0	97.0	NA	3.180	10.600	0.490	0.540	NA	NA	NA	NA	NA	NA	NA	8334	NA	NA
08/13/85	6.24	5.0	1425	1100	198.0	65.4	64.0	NA	1.960	6.990	0.360	0.390	NA	NA	NA	NA	NA	NA	NA	4129	NA	NA
09/13/85	6.06	3.2	1275	772	192.0	45.3	35.8	NA	1.890	8.670	0.420	0.450	NA	NA	NA	NA	NA	NA	NA	16188	NA	NA
10/15/85	6.19	3.5	1675	1100	260.0	74.0	58.0	NA	3.230	9.060	0.360	0.440	NA	NA	NA	NA	NA	NA	NA	17240	NA	NA
11/15/85	6.05	2.2	1925	1350	350.0	98.0	88.0	NA	3.340	9.200	0.390	0.510	NA	NA	NA	NA	NA	NA	NA	2737	NA	NA
03/23/86	6.20	5.0	440	195	56.0	13.0	8.2	NA	0.930	4.300	0.170	0.190	NA	NA	NA	NA	NA	NA	NA	17252	NA	NA
04/16/86	5.17	4.5	675	165	92.0	24.4	12.6	NA	1.620	7.300	0.310	0.290	NA	NA	NA	NA	NA	NA	NA	9550	NA	NA
05/14/86	6.20	3.8	1000	750	193.0	43.0	19.0	NA	2.940	13.300	0.480	0.390	NA	NA	NA	NA	NA	NA	NA	9357	NA	NA
06/11/86	6.33	5.1	1750	1300	282.0	70.0	64.0	NA	304.000	11.700	0.550	0.530	NA	NA	NA	NA	NA	NA	NA	2623	NA	NA
07/12/86	6.20	4.2	1125	675	158.0	46.0	36.0	NA	2.030	6.900	0.340	0.320	NA	NA	NA	NA	NA	NA	NA	4326	NA	NA
08/13/86	6.00	3.5	1150	600	150.0	40.4	36.0	NA	2.270	7.300	0.400	0.270	NA	NA	NA	NA	NA	NA	NA	3619	NA	NA
09/14/86	5.83	3.0	1475	750	222.0	55.2	32.0	NA	4.400	14.000	0.700	0.440	NA	NA	NA	NA	NA	NA	NA	11941	NA	NA
10/16/86	5.55	2.5	2150	1250	327.0	88.0	66.0	NA	6.200	19.000	0.800	0.770	NA	NA	NA	NA	NA	NA	NA	4372	NA	NA

Table A2.2. Drainage quality data from Duluth Complex test pile FL2, from 1978-1991 (0.63% S, 0.35% Cu, 0.08% Ni).

Date[1]	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
11/16/86	5.64	3.1	1950	1000	280.0	94.0	62.0	NA	8.000	19.400	0.930	0.870	NA	NA	NA	NA	NA	NA	NA	6294	NA	NA
03/22/87	5.70	6.1	350	190	58.0	14.0	10.0	NA	1.100	3.900	0.150	0.170	NA	NA	NA	NA	NA	NA	NA	9599	NA	NA
04/15/87	6.12	4.3	600	316	74.0	24.0	12.0	NA	1.670	5.900	0.280	0.320	NA	NA	NA	NA	NA	NA	NA	1957	NA	NA
05/16/87	5.75	4.3	1100	600	200.0	48.0	26.0	NA	3.400	10.900	0.500	0.310	NA	NA	NA	NA	NA	NA	NA	24103	NA	NA
06/16/87	5.24	3.2	2750	1800	473.0	162.0	103.0	NA	7.300	17.900	0.800	0.690	NA	NA	NA	NA	NA	NA	NA	2165	NA	NA
07/15/87	5.54	4.1	1475	830	188.0	58.0	34.0	NA	4.900	15.900	0.800	0.660	NA	NA	NA	NA	NA	NA	NA	12335	NA	NA
08/15/87	5.65	3.5	1600	1125	216.0	70.0	46.0	NA	5.200	15.100	0.700	0.670	NA	NA	NA	NA	NA	NA	NA	4500	NA	NA
09/16/87	5.46	3.2	1650	640	184.0	56.0	32.0	NA	3.870	12.370	0.550	0.560	NA	NA	NA	NA	NA	NA	NA	6953	NA	NA
10/13/87	4.87	1.9	2850	1580	415.0	130.0	82.0	NA	9.000	31.000	1.250	1.150	NA	NA	NA	NA	NA	NA	NA	1041	NA	NA
11/10/87	5.14	2.4	2525	920	364.0	116.0	78.0	NA	8.300	27.000	1.110	1.110	NA	NA	NA	NA	NA	NA	NA	568	NA	NA
03/28/88	5.70	3.0	400	275	50.0	14.0	6.0	NA	1.500	5.200	0.300	0.500	NA	NA	NA	NA	NA	NA	NA	6737	NA	NA
04/17/88	5.72	3.3	675	330	76.0	24.0	8.0	NA	2.400	7.800	0.500	0.800	NA	NA	NA	NA	NA	NA	NA	3384	NA	NA
05/12/88	5.71	4.0	1200	680	184.0	50.0	220.0	NA	4.700	14.600	0.900	1.400	NA	NA	NA	NA	NA	NA	NA	7127	NA	NA
06/12/88	5.56	3.6	1390	850	282.0	60.0	24.0	NA	6.200	22.000	1.100	1.600	NA	NA	NA	NA	NA	NA	NA	1605	NA	NA
07/15/88	5.47	3.5	1175	675	168.0	54.0	28.0	NA	5.000	13.000	1.000	1.300	NA	NA	NA	NA	NA	NA	NA	836	NA	NA
08/16/88	5.41	2.4	1550	870	216.0	88.0	40.0	NA	6.500	37.000	1.300	1.200	NA	NA	NA	NA	NA	NA	NA	43138	NA	NA
09/16/88	5.35	3.2	2025	1275	326.0	125.0	79.0	NA	6.650	35.500	1.250	1.250	NA	NA	NA	NA	NA	NA	NA	7926	NA	NA
10/18/88	5.47	3.2	1975	1300	272.0	102.0	51.0	NA	5.900	20.500	1.100	1.400	NA	NA	NA	NA	NA	NA	NA	1893	NA	NA
11/15/88	5.46	2.9	1275	780	152.0	64.0	28.0	NA	5.700	22.000	1.200	1.400	NA	NA	NA	NA	NA	NA	NA	5182	NA	NA
12/05/88	5.16	2.2	2125	1420	248.0	120.0	42.0	NA	13.400	53.000	2.600	2.900	NA	NA	NA	NA	NA	NA	NA	1120	NA	NA
04/07/89	5.80	4.4	500	248	64.0	20.0	10.0	NA	2.700	13.200	0.700	0.700	NA	NA	NA	NA	NA	NA	NA	20677	NA	NA
04/24/89	5.73	3.6	565	270	52.0	22.0	8.0	NA	2.900	15.000	0.700	0.800	NA	NA	NA	NA	NA	NA	NA	6249	NA	NA
05/16/89	5.39	3.6	1100	865	132.0	78.0	20.0	NA	20.000	61.000	3.800	4.100	NA	NA	NA	NA	NA	NA	NA	7907	NA	NA
06/17/89	5.27	3.3	1475	950	202.0	70.0	32.0	NA	7.600	44.000	2.000	1.600	NA	NA	NA	NA	NA	NA	NA	22827	NA	NA
07/17/89	5.12	3.2	2650	1870	420.0	156.0	76.0	NA	12.100	46.000	1.900	2.600	NA	NA	NA	NA	NA	NA	NA	1900	NA	NA
08/14/89	5.33	3.3	1450	880	182.0	76.0	38.0	NA	5.500	26.000	0.900	1.000	NA	NA	NA	NA	NA	NA	NA	1874	NA	NA
09/14/89	5.24	3.6	1750	1050	260.0	88.0	32.0	NA	10.000	39.000	2.700	2.100	NA	NA	NA	NA	NA	NA	NA	14311	NA	NA
10/16/89	5.23	2.0	1500	1140	230.0	84.0	44.0	NA	8.400	29.000	2.600	1.900	NA	NA	NA	NA	NA	NA	NA	2562	NA	NA
11/06/89	5.15	2.6	1340	800	170.0	78.0	34.0	NA	6.500	24.000	2.800	1.500	NA	NA	NA	NA	NA	NA	NA	1881	NA	NA
03/26/90	5.05	2.4	500	210	50.0	17.6	6.6	NA	2.700	9.120	0.480	0.600	NA	NA	NA	NA	NA	NA	NA	6601	NA	NA
04/12/90	5.31	3.0	982	565	110.0	50.0	12.0	NA	9.260	32.000	1.840	1.850	NA	NA	NA	NA	NA	NA	NA	19731	NA	NA
05/17/90	5.13	2.7	1600	950	194.0	84.0	24.0	NA	11.000	36.000	2.120	2.170	NA	NA	NA	NA	NA	NA	NA	11597	NA	NA
06/18/90	5.27	2.0	1750	930	188.0	80.0	32.0	NA	7.210	25.000	1.220	1.420	NA	NA	NA	NA	NA	NA	NA	11752	NA	NA
07/16/90	5.08	3.9	2850	1730	336.0	144.0	50.0	NA	14.000	42.000	2.430	2.680	NA	NA	NA	NA	NA	NA	NA	6283	NA	NA
08/13/90	5.05	3.0	3110	1840	388.0	196.0	70.0	NA	14.000	40.000	2.330	2.460	NA	NA	NA	NA	NA	NA	NA	1272	NA	NA
09/10/90	5.29	2.1	1545	815	174.0	68.0	30.0	NA	6.020	20.000	1.160	1.240	NA	NA	NA	NA	NA	NA	NA	3376	NA	NA
10/15/90	5.23	2.8	1700	960	196.0	74.0	28.0	NA	11.000	36.000	2.010	1.800	NA	NA	NA	NA	NA	NA	NA	16476	NA	NA
11/16/90	5.08	2.2	3750	2250	540.0	208.0	74.0	NA	22.000	69.000	3.740	3.860	NA	NA	NA	NA	NA	NA	NA	1370	NA	NA
04/08/91	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	712	NA	NA
04/22/91	5.70	4.0	1025	570	118.0	44.0	16.0	NA	4.400	17.000	0.700	0.800	NA	NA	NA	NA	NA	NA	NA	10579	NA	NA
05/20/91	5.24	3.2	1925	870	202.0	81.0	20.0	NA	9.900	36.000	1.500	1.200	NA	NA	NA	NA	NA	NA	NA	13471	NA	NA
06/17/91	5.25	2.0	1250	635	136.0	59.0	18.0	NA	6.900	23.000	1.100	0.800	NA	NA	NA	NA	NA	NA	NA	5185	NA	NA
07/15/91	5.21	2.3	1700	775	146.0	74.0	20.0	NA	9.000	30.000	1.400	1.000	NA	NA	NA	NA	NA	NA	NA	17070	NA	NA
08/12/91	4.96	2.0	3525	2030	416.0	184.0	72.0	NA	18.000	58.000	2.700	2.100	NA	NA	NA	NA	NA	NA	NA	1639	NA	NA
09/09/91	5.13	3.0	1350	610	114.0	50.0	20.0	NA	8.000	30.000	1.400	0.900	NA	NA	NA	NA	NA	NA	NA	9693	NA	NA
10/07/91	5.09	3.0	2375	1210	228.0	98.0	38.0	NA	13.000	43.000	2.100	1.700	NA	NA	NA	NA	NA	NA	NA	3051	NA	NA
11/07/91	5.15	3.0	1875	1230	202.0	110.0	30.0	NA	22.000	69.000	3.400	2.600	NA	NA	NA	NA	NA	NA	NA	14629	NA	NA

NA: Not analyzed.

[1]Dates represent intermediate dates for measurement periods.

Table A2.2.1. Drainage quality data from Duluth Complex test pile FL2, 1992-1993 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH	Alk. mg/L	Lab Acy. mg/L	Calc. Acy. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/23/92	4.76	3.8	80	177.4	2087	1520	224.0	172.0	49.20	31.00	1	17.80	75	3.20	2.30	0.10	3.79	NA	NA	NA	NA	NA	3	NA	NA
05/11/92	5.26	3.2	30	44.3	1030	400	84.0	40.0	13.20	12.70	<1	5.60	21	1.00	0.80	0.06	1.23	NA	NA	NA	NA	NA	3229	NA	NA
05/26/92	5.11	3.2	50	71.0	1120	620	120.0	56.0	16.00	15.70	<1	10.30	31	1.40	1.10	0.14	1.77	NA	NA	NA	NA	NA	2372	NA	NA
06/08/92	5.05	3.2	70	117.2	2200	1240	254.0	104.0	30.40	27.00	<1	16.50	50	2.20	1.90	0.07	2.67	NA	NA	NA	NA	NA	433	NA	NA
06/22/92	5.07	3.1	53	89.9	1860	1090	200.0	90.0	28.40	25.00	<1	12.40	39	2.00	1.40	0.04	2.12	NA	NA	NA	NA	NA	398	NA	NA
07/07/92	5.08	2.1	55	64.9	1290	630	126.0	58.0	20.20	19.20	<1	9.00	29	1.40	1.00	0.07	1.57	NA	NA	NA	NA	NA	1270	NA	NA
07/20/92	5.12	2.1	50	86.0	1780	2100	188.0	80.0	27.40	24.00	<1	11.30	38	1.90	1.40	0.07	1.95	NA	NA	NA	NA	NA	333	NA	NA
08/03/92	5.06	2.1	33	66.3	1270	810	156.0	70.0	25.40	22.00	<1	9.50	29	1.50	1.20	0.06	1.63	NA	NA	NA	NA	NA	489	NA	NA
08/17/92	5.11	2.1	65	68.8	1600	865	152.0	72.0	26.60	23.00	<1	9.90	30	1.50	1.20	0.05	1.73	NA	NA	NA	NA	NA	402	NA	NA
08/31/92	5.05	5.0	33	70.6	1600	875	162.0	82.0	27.40	23.00	<1	9.00	32	1.50	1.20	0.04	1.67	NA	NA	NA	NA	NA	551	NA	NA
09/14/92	5.22	3.2	18	41.2	800	440	72.0	38.0	13.40	15.20	<1	6.00	19	1.00	0.70	0.09	1.10	NA	NA	NA	NA	NA	1655	NA	NA
09/28/92	4.99	2.0	73	126.7	2200	1260	250.0	120.0	37.00	32.00	<1	17.50	54	2.60	1.90	0.07	2.90	NA	NA	NA	NA	NA	1409	NA	NA
10/12/92	5.00	2.6	100	168.9	3400	2180	376.0	202.0	64.00	44.00	<1	23.00	71	3.60	2.60	0.08	3.85	NA	NA	NA	NA	NA	324	NA	NA
10/27/92	4.92	2.6	85	153.4	3120	3860	328.0	186.0	60.60	40.00	<1	19.90	66	3.20	2.10	0.04	3.54	NA	NA	NA	NA	NA	178	NA	NA
11/09/92	5.06	3.0	NA	NA	3950	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	67	NA	NA
11/30/92	5.11	3.0	93	97.7	1950	1270	204.0	120.0	39.40	28.00	<1	11.50	44	2.10	1.50	0.04	2.35	NA	NA	NA	NA	NA	368	NA	NA
04/08/93	5.10	10.0	NA	NA	780	410	70.0	45.0	11.00	12.00	<1	4.84	20	0.72	0.57	0.06	0.91	NA	NA	NA	NA	NA	6707	NA	NA
04/22/93	4.91	5.0	NA	NA	875	400	82.5	35.0	9.00	9.00	<1	4.26	18	0.83	0.55	0.09	0.81	NA	NA	NA	NA	NA	3936	NA	NA
05/06/93	5.43	3.0	NA	NA	525	240	42.5	22.5	6.50	6.50	<1	3.19	12	0.57	0.39	0.13	0.54	NA	NA	NA	NA	NA	8660	NA	NA
05/20/93	5.04	2.0	NA	NA	1600	1070	172.5	97.5	20.25	17.50	<1	13.00	56	2.60	1.42	0.05	2.57	NA	NA	NA	NA	NA	3528	NA	NA
06/03/93	5.09	3.0	NA	NA	1300	715	120.0	70.0	17.25	13.25	<1	10.00	38	1.72	1.10	0.09	1.77	NA	NA	NA	NA	NA	3524	NA	NA
06/17/93	5.01	2.5	NA	NA	1300	800	150.0	85.0	21.00	16.25	<1	12.00	43	1.90	1.26	0.07	1.97	NA	NA	NA	NA	NA	3206	NA	NA
07/01/93	5.03	1.0	NA	NA	1050	555	95.0	60.0	15.25	13.75	<1	11.00	34	1.54	1.00	0.09	1.62	NA	NA	NA	NA	NA	3304	NA	NA
07/14/93	4.94	NA	NA	NA	2480	1520	260.0	148.0	33.40	32.40	2	24.00	80	3.44	2.30	0.09	4.14	NA	NA	NA	NA	NA	13693	NA	NA
07/29/93	4.92	3.5	NA	NA	2525	1660	270.0	166.0	40.80	34.60	2	21.00	75	3.33	2.18	0.12	3.89	NA	NA	NA	NA	NA	7142	NA	NA
08/12/93	4.88	5.0	NA	NA	2000	1210	192.0	118.0	33.40	27.40	<1	13.00	51	2.02	1.55	0.08	2.80	NA	NA	NA	NA	NA	2660	NA	NA
08/26/93	4.80	2.5	NA	NA	2410	1520	248.0	160.0	45.20	33.60	<1	16.00	63	2.48	1.91	0.07	3.44	NA	NA	NA	NA	NA	1457	NA	NA
09/09/93	5.05	2.0	NA	NA	1300	630	112.0	68.0	20.40	18.80	<1	8.98	30	1.31	0.98	0.07	1.80	NA	NA	NA	NA	NA	4171	NA	NA
09/23/93	5.06	2.0	NA	NA	2000	1130	206.0	128.0	38.20	30.00	<1	14.00	53	2.21	1.69	0.07	3.02	NA	NA	NA	NA	NA	1151	NA	NA
10/07/93	5.02	2.5	NA	NA	2550	1450	224.0	160.0	44.60	33.20	<1	18.00	62	2.69	1.73	0.08	3.13	NA	NA	NA	NA	NA	685	NA	NA
10/20/93	5.04	2.0	NA	NA	2500	1340	228.0	150.0	40.00	30.20	<1	16.00	61	2.45	1.66	0.05	2.98	NA	NA	NA	NA	NA	254	NA	NA
11/04/93	5.10	2.5	NA	NA	2100	620	149.0	108.0	27.60	22.00	<1	10.00	41	1.94	1.13	0.06	2.15	NA	NA	NA	NA	NA	753	NA	NA
11/22/93	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	379	NA	NA

NA: Not analyzed.
'Dates represent intermediate dates for measurement periods.

Table A2.3. Drainage quality data from Duluth Complex test pile FL3, from 1978-1989 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.20	NA	675	NA	62.0	10.0	52.0	12.0	0.023	0.060	NA	NA	0.70	0.05	NA	NA	NA	NA	NA	NA	NA	4.5
05/25/78	7.80	NA	1200	NA	106.0	24.0	122.0	15.0	0.013	0.042	0.050	0.020	0.04	0.09	NA	NA	NA	NA	NA	NA	NA	NA
06/16/78	7.79	NA	3000	NA	214.0	57.0	372.0	21.0	0.026	0.185	0.012	0.059	0.06	0.24	NA	NA	NA	NA	NA	192	NA	6.0
07/05/78	7.30	NA	1600	600	142.0	34.0	102.0	23.0	0.200	0.321	0.017	0.037	0.02	0.17	78.0	NA	NA	NA	NA	133	NA	9.0
07/13/78	7.55	NA	NA	1050	352.0	46.0	207.0	32.0	0.049	0.405	0.030	0.092	0.03	0.24	34.0	23.0	0.70	0.08	NA	NA	NA	NA
07/19/78	7.85	NA	1000	550	241.0	28.0	NA	27.0	0.092	0.235	0.020	0.060	0.01	0.15	35.0	24.0	0.55	0.08	NA	NA	NA	NA
08/02/78	7.60	NA	650	367	96.0	14.0	60.0	21.0	0.080	0.210	0.010	0.025	0.03	0.16	12.0	19.0	0.80	0.08	NA	NA	NA	22.0
08/04/78	7.65	NA	950	500	156.0	24.0	88.0	26.0	0.080	0.320	0.020	0.044	0.02	0.16	17.0	26.0	0.06	0.08	NA	100	NA	22.0
08/10/78	7.90	40.0	1900	1200	348.0	51.0	188.0	49.0	0.060	0.565	0.090	0.073	0.01	0.28	48.0	10.5	NA	0.45	17.0	851	NA	24.0
08/16/78	7.62	29.0	750	300	83.7	14.7	46.0	17.7	0.117	0.207	0.017	0.020	0.07	0.08	10.0	9.4	NA	0.08	NA	NA	NA	20.3
08/23/78	7.50	33.0	750	300	75.0	14.0	45.0	17.0	0.100	0.190	0.016	0.020	0.07	0.08	7.0	17.0	0.64	0.08	NA	NA	NA	NA
08/25/78	7.30	25.0	1150	500	115.0	23.0	67.0	23.0	0.080	0.250	0.010	0.030	0.04	0.10	11.0	24.0	NA	0.10	NA	NA	NA	24.0
09/13/78	7.30	36.0	690	659	133.5	21.5	74.5	27.5	0.085	0.300	0.020	0.040	0.04	0.12	7.0	13.0	0.08	0.08	NA	130	NA	14.0
03/19/79	7.08	NA	794	NA	118.8	25.9	50.1	15.9	0.080	0.790	0.050	0.110	0.08	0.10	NA	NA	NA	NA	9.3	3179	1765	NA
03/27/79	7.20	40.0	1233	NA	162.0	36.8	73.8	22.2	0.060	0.780	0.070	0.110	0.02	0.12	NA	NA	NA	NA	NA	1877	442	NA
04/13/79	6.75	NA	484	233	61.9	12.8	29.6	10.1	0.106	0.296	0.024	0.040	0.13	0.06	5.0	NA	NA	NA	NA	1877	442	NA
04/17/79	6.70	28.0	587	235	69.8	13.8	32.2	10.8	0.088	0.292	0.026	0.050	0.09	0.08	8.0	NA	NA	NA	5.3	3039	3576	NA
04/19/79	6.80	23.0	503	215	80.5	13.4	41.5	9.3	0.064	0.220	0.020	0.040	0.12	0.07	5.0	1.5	0.10	NA	NA	4341	4304	NA
04/19/79	7.10	27.0	532	205	83.0	12.6	31.0	8.6	0.072	0.238	0.020	0.050	0.05	0.07	4.0	1.5	NA	NA	11.0	3312	4304	NA
04/20/79	6.70	33.0	623	310	100.0	17.6	42.0	11.0	0.069	0.278	0.032	0.050	0.06	0.09	4.0	3.4	0.10	NA	4.7	4126	2592	NA
04/22/79	7.20	19.0	1356	833	206.4	44.6	121.6	24.8	0.056	0.920	0.051	0.120	0.03	0.17	11.0	6.9	0.04	NA	NA	8607	1630	NA
05/07/79	7.30	NA	1562	NA	209.2	49.0	131.6	27.6	0.046	0.890	0.047	0.110	0.03	0.18	NA	NA	NA	NA	NA	479	308	NA
05/10/79	7.40	NA	959	433	123.2	26.4	62.4	17.0	0.068	0.520	0.031	0.070	0.03	0.11	10.0	NA	NA	NA	NA	7070	3099	NA
05/20/79	6.70	NA	1711	933	229.2	52.6	130.8	29.6	0.060	0.750	0.047	0.120	0.02	0.16	13.0	NA	NA	NA	5.9	2619	900	NA
05/22/79	7.60	NA	1537	500	120.4	26.2	60.8	16.8	0.090	0.440	0.023	0.080	0.04	0.08	6.0	NA	NA	NA	7.6	1559	784	NA
05/23/79	7.50	NA	1331	750	180.4	37.4	92.2	24.4	0.058	0.620	0.040	0.080	0.04	0.12	7.0	NA	NA	NA	4.9	2449	838	NA
06/01/79	7.60	NA	998	517	122.8	26.6	59.8	17.4	0.100	0.460	0.023	0.060	0.08	0.10	6.0	NA	NA	NA	6.7	1938	369	NA
06/06/79	7.50	NA	824	400	101.0	21.8	33.4	15.6	0.081	0.370	0.021	0.050	0.03	0.08	4.0	NA	NA	NA	NA	1768	515	NA
06/10/79	7.20	35.0	725	320	173.8	38.0	50.8	38.8	0.100	0.310	0.010	0.040	0.02	0.05	4.0	NA	NA	NA	NA	1771	938	20.0
06/12/79	6.80	33.0	912	420	118.8	25.2	36.0	22.8	0.071	0.410	0.013	0.070	0.01	0.07	5.0	NA	NA	NA	NA	1011	361	NA
06/15/79	7.30	33.0	1081	533	143.8	30.4	42.4	23.4	0.061	0.470	0.013	0.050	0.01	0.08	7.0	NA	NA	NA	NA	329	325	20.0
06/21/79	7.05	30.5	823	409	101.0	23.5	42.0	19.0	0.070	0.340	0.021	0.040	0.02	0.06	4.0	NA	NA	NA	NA	2195	1070	16.5
06/30/79	7.20	29.0	1175	725	158.0	35.0	68.0	25.0	0.050	0.490	0.032	0.070	0.02	0.24	5.0	NA	NA	NA	5.6	1389	169	12.0
07/10/79	7.50	26.0	1649	1100	241.0	52.0	105.0	37.0	0.040	1.700	0.046	0.090	0.01	0.16	12.0	NA	NA	NA	5.3	734	74	18.0
07/30/79	7.10	12.0	772	390	98.0	21.0	31.0	19.0	0.090	0.580	0.040	0.060	0.01	0.10	1.0	NA	NA	NA	7.7	2597	1907	23.0
08/04/79	7.05	13.4	1277	809	204.5	36.0	53.0	33.0	0.080	0.980	0.045	0.105	0.03	0.15	4.0	NA	NA	NA	9.6	602	138	23.0
08/13/79	6.90	12.0	2064	1850	368.0	77.6	150.8	55.2	0.100	1.460	0.090	0.180	0.04	0.23	10.0	NA	NA	NA	5.2	295	46	14.0
08/28/79	7.10	12.4	1172	720	177.8	36.8	50.0	27.2	0.170	1.190	0.080	0.140	0.01	0.20	14.0	NA	NA	NA	8.2	208	161	21.0
09/01/79	6.90	12.0	1002	650	156.6	29.8	36.6	23.2	0.150	1.280	0.062	0.110	0.01	0.13	4.0	NA	NA	NA	6.1	3895	1538	20.0
09/05/79	6.90	12.0	1575	1200	269.6	52.0	69.0	36.6	0.140	1.640	0.090	0.150	0.01	0.17	5.0	NA	NA	NA	6.2	500	292	18.0
09/10/79	6.80	9.6	1401	975	235.6	42.6	77.4	33.0	0.140	1.440	0.110	0.130	0.02	0.17	6.0	NA	NA	NA	5.7	1983	408	18.0
09/13/79	7.00	11.2	1499	950	231.8	44.0	80.6	32.8	0.150	1.630	0.090	0.130	0.03	0.18	6.0	NA	NA	NA	5.3	1332	408	17.0
09/19/79	6.80	12.0	2598	1900	430.8	84.8	144.0	56.8	0.180	2.570	0.140	0.260	0.02	0.26	12.0	NA	NA	NA	5.0	863	109	15.0
10/17/79	6.40	7.2	2192	NA	363.2	73.4	122.8	48.0	0.190	2.560	0.140	0.270	0.01	0.29	NA	NA	NA	NA	4.6	337	200	NA
10/19/79	6.30	8.0	938	600	124.4	28.0	44.0	18.8	0.190	1.630	0.070	0.140	0.03	0.13	3.0	NA	NA	NA	5.9	1177	1297	NA
10/31/79	6.40	9.6	1233	725	172.0	36.5	54.2	23.9	0.220	2.840	0.140	0.195	0.02	0.19	3.0	NA	NA	NA	4.5	3963	1450	NA
11/01/79	6.80	12.8	1258	NA	182.0	39.6	60.0	25.6	0.190	2.540	0.120	0.180	0.02	0.15	NA	NA	NA	NA	4.5	3316	944	12.0
03/31/80	7.20	19.0	355	150	34.1	9.2	7.1	5.9	0.090	0.960	0.045	0.070	0.11	0.07	2.0	NA	NA	NA	4.0	1571	496	10.7
04/01/80	6.80	12.6	335	130	32.2	8.8	10.6	6.2	0.120	0.810	0.060	0.060	0.07	0.05	2.0	NA	NA	NA	3.1	1041	775	NA
04/06/80	7.20	21.3	500	73	58.2	14.2	14.0	8.8	0.110	1.000	0.040	0.090	0.04	0.09	2.0	NA	NA	NA	NA	2271	1300	NA
04/14/80	7.15	21.0	NA	326	86.7	22.3	23.6	12.0	0.105	1.490	0.080	0.100	0.03	0.12	7.0	NA	NA	NA	NA	1590	515	NA

Table A2.3. Drainage quality data from Duluth Complex test pile FL3, from 1978-1989 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/18/80	7.10	23.0	NA	364	88.8	21.6	24.8	11.8	0.120	2.100	0.100	0.130	0.02	0.18	4.0	NA	NA	NA	2.9	3449	703	NA
04/29/80	7.10	22.0	1860	1380	246.4	64.2	87.2	30.8	0.290	4.310	0.210	0.300	0.01	0.35	10.0	NA	NA	NA	NA	2082	180	11.0
05/08/80	7.10	20.9	2352	1720	324.0	90.4	136.0	42.6	0.260	4.290	0.240	0.370	0.02	0.34	13.0	NA	NA	NA	3.5	863	130	11.5
05/18/80	7.00	15.4	2300	1220	319.2	95.8	152.6	43.0	0.230	4.620	0.260	0.430	0.03	0.38	21.0	NA	NA	NA	NA	856	57	19.0
05/31/80	6.70	12.1	1300	720	159.2	42.7	57.9	24.3	0.270	3.140	0.180	0.250	0.01	0.23	36.0	NA	NA	NA	7.1	1673	775	NA
06/03/80	6.90	13.8	1500	850	191.8	45.8	57.2	27.4	0.250	3.790	0.210	0.280	0.01	0.26	46.0	NA	NA	NA	4.9	613	191	NA
06/06/80	6.80	11.0	1400	920	175.2	42.0	55.0	25.6	0.280	3.050	0.180	0.240	0.02	0.21	39.0	NA	NA	NA	NA	598	254	NA
06/13/80	6.80	11.0	1700	1040	219.2	50.9	73.9	30.3	0.315	3.370	0.195	0.290	0.03	0.24	28.0	NA	NA	NA	NA	413	207	NA
06/16/80	6.60	9.9	2200	1450	278.0	66.0	97.6	35.4	0.320	4.520	0.260	0.390	0.02	0.30	30.0	NA	NA	NA	NA	443	82	NA
06/27/80	6.60	11.0	2000	1180	252.0	67.0	89.0	35.0	0.230	3.490	0.230	0.360	0.02	0.28	18.0	NA	NA	NA	8.1	610	187	NA
07/02/80	6.60	11.0	1700	1000	230.0	57.0	66.0	29.0	0.460	3.760	0.270	0.370	0.04	0.30	27.0	NA	NA	NA	4.9	681	722	NA
07/02/80	6.70	8.5	1200	620	155.0	38.0	38.0	20.0	0.430	3.020	0.180	0.280	0.02	0.24	35.0	NA	NA	NA	NA	803	577	NA
07/13/80	6.60	12.1	2600	1620	359.6	80.0	133.6	51.0	0.560	5.350	0.280	0.480	0.02	0.41	30.0	NA	NA	NA	NA	598	108	NA
07/23/80	6.65	12.1	2350	1460	342.6	88.8	201.6	49.9	0.505	5.195	0.300	0.480	0.02	0.45	23.0	NA	NA	NA	NA	538	45	NA
08/03/80	6.65	11.0	NA	NA	352.4	95.4	162.0	50.4	0.530	6.240	0.310	0.540	0.02	0.51	NA	NA	NA	NA	5.9	189	80	NA
08/12/80	6.35	5.1	1100	595	128.8	35.3	44.6	21.4	0.670	3.570	0.200	0.300	0.06	0.34	12.0	NA	NA	NA	8.1	837	521	NA
08/20/80	6.35	5.1	1300	788	184.2	51.6	46.2	27.8	1.170	7.630	0.400	0.560	0.01	0.65	17.0	NA	NA	NA	NA	1030	166	NA
08/22/80	NA	NA	1400	700	80.0	42.0	42.0	20.0	1.360	7.040	0.410	0.590	0.04	0.62	21.0	NA	NA	NA	NA	655	269	NA
08/29/80	7.10	28.0	4850	2280	413.0	379.0	591.0	48.0	0.750	5.340	0.310	0.360	0.06	0.43	925.0	NA	NA	NA	9.5	1514	2500	NA
09/03/80	6.40	7.6	2020	1390	256.0	86.0	123.0	40.0	1.650	12.550	0.670	0.760	0.03	0.95	50.0	NA	NA	NA	NA	2105	512	NA
09/08/80	6.50	7.7	1912	1320	248.0	82.0	116.0	39.0	2.120	13.890	0.770	0.810	0.02	1.10	44.0	NA	NA	NA	NA	1336	758	15.0
09/12/80	6.60	11.0	1448	690	97.0	51.0	80.0	25.0	1.620	9.450	0.550	0.530	0.04	0.76	52.0	NA	NA	NA	4.0	3184	1512	18.0
09/19/80	6.60	11.0	2057	1320	255.0	92.0	128.0	39.0	2.660	12.780	0.670	0.720	0.05	0.90	77.0	NA	NA	NA	NA	7809	1846	13.0
10/01/80	6.40	10.2	2984	1750	410.0	136.0	181.0	NA	2.720	17.700	0.800	1.010	0.03	1.04	77.0	NA	NA	NA	3.4	2506	244	14.0
10/22/80	6.30	13.6	2575	1900	340.0	127.0	178.0	52.0	2.330	12.300	0.650	0.900	0.02	0.89	116.0	NA	NA	NA	4.3	1408	327	10.0
10/25/80	6.10	6.8	2205	1250	266.3	116.3	125.3	43.7	3.970	19.500	0.940	1.090	0.03	1.29	56.0	NA	NA	NA	NA	2044	321	10.0
03/02/81	6.60	6.8	900	590	103.4	43.6	42.2	NA	1.100	13.000	0.620	0.550	NA	NA	15.0	NA	NA	NA	NA	284	100	NA
03/16/81	6.50	6.4	1200	700	115.7	49.9	55.9	NA	1.300	23.000	0.950	0.800	NA	NA	NA	NA	NA	NA	NA	931	1393	NA
03/23/81	6.60	7.2	1224	825	119.6	58.5	60.7	NA	1.410	21.000	0.920	0.870	NA	NA	NA	NA	NA	NA	NA	821	526	9.0
03/26/81	6.30	5.6	600	272	46.8	25.9	19.5	NA	0.910	8.200	0.390	0.380	NA	NA	NA	NA	NA	NA	7.2	1158	723	NA
04/02/81	6.70	12.0	775	385	58.5	28.9	22.1	NA	1.590	15.000	0.690	0.550	NA	NA	NA	NA	NA	NA	NA	2055	506	NA
04/03/81	6.80	12.0	1350	625	122.2	58.8	66.3	NA	5.150	33.000	1.450	1.150	NA	NA	NA	NA	NA	NA	NA	3710	1229	NA
04/13/81	NA	NA	1100	555	96.0	56.0	33.0	NA	3.630	17.000	0.800	0.660	NA	NA	NA	NA	NA	NA	NA	2839	993	NA
04/17/81	6.90	10.4	1600	745	126.0	71.0	56.0	NA	3.730	19.000	0.860	0.710	NA	NA	NA	NA	NA	NA	NA	2680	1199	NA
04/23/81	7.00	14.4	1500	680	121.0	74.0	56.0	NA	2.300	13.000	0.620	0.550	NA	NA	NA	NA	NA	NA	NA	2063	2237	NA
04/28/81	7.20	16.0	1850	880	160.0	88.0	65.0	NA	2.760	17.000	0.830	0.730	NA	NA	NA	NA	NA	NA	3.2	4467	2436	NA
04/30/81	6.80	14.8	2176	1300	252.0	114.0	92.0	NA	3.680	25.800	1.230	1.070	NA	NA	NA	NA	NA	NA	4.0	3350	1200	10.0
05/13/81	6.80	8.8	2545	1750	324.0	152.0	120.0	NA	4.010	22.000	1.440	1.310	NA	NA	NA	NA	NA	NA	3.6	4115	100	17.0
06/02/81	6.70	9.1	2406	1700	300.0	138.0	118.0	NA	3.250	21.000	1.270	1.170	NA	NA	NA	NA	NA	NA	NA	1586	473	17.0
06/06/81	6.85	9.1	2157	1310	254.0	102.0	100.0	NA	3.290	18.000	1.190	1.060	NA	NA	NA	NA	NA	NA	4.1	2555	67	18.0
06/22/81	6.80	14.4	1250	1450	262.0	116.0	128.0	NA	2.840	19.000	1.110	0.880	NA	NA	NA	NA	NA	NA	NA	3482	1273	16.0
06/26/81	6.70	12.8	1330	680	156.0	58.0	42.0	NA	4.050	19.840	0.950	0.630	NA	NA	NA	NA	NA	NA	NA	3013	958	15.0
06/28/81	6.70	14.4	1862	800	226.0	88.0	70.0	NA	4.880	20.000	1.230	0.800	NA	NA	40.0	NA	NA	NA	3.5	4130	2493	15.0
07/04/81	6.85	13.5	2098	1400	300.0	102.0	88.8	NA	4.850	25.000	1.150	0.960	NA	NA	NA	NA	NA	NA	NA	2600	385	22.0
07/15/81	6.55	6.2	3046	2100	394.0	178.6	158.6	NA	6.200	37.000	1.650	1.530	NA	NA	NA	NA	NA	NA	NA	1283	120	20.0
07/25/81	5.60	NA	2564	1810	340.0	156.6	134.6	NA	5.400	30.000	1.430	1.360	NA	NA	23.0	NA	NA	NA	3.1	961	177	21.0
08/04/81	6.20	4.4	2574	1750	328.0	138.0	118.0	NA	5.500	30.000	1.470	1.320	NA	NA	NA	NA	NA	NA	NA	863	66	19.0
08/17/81	6.05	5.4	2485	1650	308.0	132.0	108.0	52.0	5.270	25.000	1.300	1.440	NA	NA	NA	NA	NA	NA	NA	723	46	22.0
09/01/81	6.00	4.8	2575	1570	284.0	136.0	98.0	NA	5.450	26.000	1.240	1.610	NA	NA	NA	NA	NA	NA	7.4	348	100	NA
09/07/81	5.90	3.2	2800	1540	268.0	124.0	88.0	NA	6.240	25.000	1.330	1.610	NA	NA	NA	NA	NA	NA	NA	390	129	17.0
09/16/81	5.90	3.8	3300	2000	354.0	171.0	115.0	NA	10.010	41.500	2.040	2.440	NA	NA	NA	NA	NA	NA	NA	265	22	NA

Table A2.3. Drainage quality data from Duluth Complex test pile FL3, from 1978-1989 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/27/81	5.85	2.4	1775	980	192.0	86.0	50.0	NA	6.300	22.000	1.140	1.330	NA	NA	NA	NA	NA	NA	4.7	598	251	NA
10/01/81	5.80	2.0	1700	850	180.0	82.0	32.0	NA	10.670	34.000	1.710	1.620	NA	NA	NA	NA	NA	NA	NA	2279	1459	NA
10/04/81	5.70	2.4	1700	1000	196.0	88.0	34.0	28.0	12.000	36.000	1.880	1.730	NA	NA	NA	NA	NA	NA	NA	1287	1013	NA
10/14/81	6.25	4.8	2150	1280	228.0	106.0	64.0	40.0	9.000	39.000	1.940	1.550	NA	NA	NA	NA	NA	NA	3.4	3165	1597	NA
10/18/81	6.25	4.0	2000	1310	232.0	100.0	56.0	38.0	9.000	36.000	1.880	1.430	NA	NA	NA	NA	NA	NA	NA	10569	1210	NA
11/09/81	6.65	11.2	4500	2800	410.0	320.0	240.0	84.0	10.980	62.000	2.480	2.250	NA	NA	NA	NA	NA	NA	3.5	4997	108	NA
03/25/82	6.55	7.2	900	470	73.0	53.4	20.0	14.0	4.140	19.000	0.940	0.690	NA	NA	NA	NA	NA	NA	NA	2839	NA	NA
04/01/82	6.18	5.2	1325	800	105.0	76.4	28.0	18.0	6.275	31.000	1.470	1.490	NA	NA	NA	NA	NA	NA	NA	3975	NA	NA
04/13/82	6.60	7.2	690	320	52.0	34.4	14.0	10.0	2.860	13.000	0.670	0.490	NA	NA	NA	NA	NA	NA	NA	4728	NA	NA
04/15/82	6.60	9.0	750	390	63.8	41.6	16.0	12.0	2.120	13.000	0.580	0.420	NA	NA	NA	NA	NA	NA	NA	3774	NA	NA
04/19/82	6.60	7.7	1125	705	129.0	83.0	29.0	20.0	3.395	19.500	1.010	0.640	NA	NA	NA	NA	NA	NA	NA	1961	NA	NA
04/23/82	6.50	5.4	1500	950	188.0	110.0	80.0	24.0	4.450	26.000	1.340	0.860	NA	NA	NA	NA	NA	NA	NA	2438	NA	NA
04/28/82	6.50	7.7	1875	1000	160.0	128.0	68.0	32.0	4.520	25.000	1.380	0.950	NA	NA	NA	NA	NA	NA	NA	3721	NA	NA
05/03/82	6.40	5.4	2290	NA	240.0	164.0	104.0	42.0	6.490	40.000	2.030	1.360	NA	NA	NA	NA	NA	NA	NA	466	NA	NA
05/07/82	6.90	17.4	1960	975	133.0	125.0	109.0	25.0	3.630	20.300	1.105	0.740	NA	NA	NA	NA	NA	NA	NA	4527	NA	NA
05/11/82	6.90	21.2	1800	1040	156.0	122.0	64.0	28.0	4.410	24.000	1.400	0.890	NA	NA	NA	NA	NA	NA	NA	2782	NA	NA
05/14/82	6.80	16.2	2500	1465	202.0	160.0	110.0	40.0	7.250	44.000	2.410	1.650	NA	NA	NA	NA	NA	NA	NA	750	NA	NA
05/18/82	6.80	18.9	2450	1340	206.0	160.0	128.0	40.0	6.800	42.000	2.270	1.580	NA	NA	NA	NA	NA	NA	NA	681	NA	NA
05/21/82	6.25	NA	3200	2400	292.0	163.0	93.0	53.0	10.160	70.000	2.560	2.010	0.04	2.81	NA	NA	NA	NA	NA	3062	NA	NA
06/02/82	6.10	9.9	3600	2750	340.0	238.0	129.5	63.5	11.405	79.500	3.015	2.580	0.06	3.36	NA	NA	NA	NA	NA	1298	NA	NA
06/11/82	7.30	NA	3150	NA	306.0	183.0	139.0	52.0	5.230	45.000	1.920	1.480	0.05	1.96	NA	NA	NA	NA	NA	NA	NA	NA
06/12/82	7.70	NA	3450	NA	324.0	178.0	174.0	46.0	3.330	29.000	1.300	0.970	0.05	1.27	NA	NA	NA	NA	NA	NA	NA	NA
06/18/82	6.50	9.9	2500	1500	248.0	133.0	65.0	39.0	6.950	43.000	1.870	1.520	0.04	1.89	NA	NA	NA	NA	NA	148	NA	NA
07/02/82	7.40	81.6	2750	1300	297.0	136.0	178.0	34.0	2.290	21.090	0.770	0.560	0.03	0.78	NA	NA	NA	NA	NA	2824	NA	NA
07/08/82	6.35	11.7	2700	2000	354.0	150.0	108.0	48.0	8.850	67.000	2.300	1.560	0.02	2.56	NA	NA	NA	NA	NA	7291	NA	NA
07/26/82	6.37	10.6	2705	1675	298.0	149.0	77.0	51.0	7.570	79.500	3.240	1.610	0.39	3.03	NA	NA	NA	NA	NA	6261	NA	NA
08/30/82	6.91	50.3	2087	1175	298.0	126.0	70.0	40.0	6.240	45.000	1.710	1.150	0.13	1.98	NA	NA	NA	NA	NA	7779	NA	NA
10/04/82	6.65	14.6	2145	1357	223.0	126.0	59.0	44.0	9.520	80.000	3.220	1.620	0.13	2.94	NA	NA	NA	NA	NA	20301	NA	NA
11/01/82	6.20	3.6	2825	NA	248.0	160.0	72.0	54.0	9.490	91.000	3.650	1.870	0.02	3.52	NA	NA	NA	NA	NA	1257	NA	NA
11/22/82	6.10	4.5	2140	1600	270.0	116.0	64.0	NA	13.750	91.000	3.340	2.190	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
03/18/83	5.95	3.0	1775	1120	NA	NA	NA	NA	9.010	65.000	2.590	2.030	NA	NA	NA	NA	NA	NA	NA	8206	NA	NA
04/14/83	6.66	16.3	1070	510	NA	NA	NA	NA	5.010	41.000	1.410	1.060	NA	NA	NA	NA	NA	NA	NA	7414	NA	NA
05/15/83	6.41	6.9	1574	1000	NA	NA	NA	NA	6.440	58.000	2.140	1.690	NA	NA	NA	NA	NA	NA	NA	5109	NA	NA
06/16/83	6.19	6.9	1782	1300	NA	NA	NA	NA	5.360	43.000	1.720	1.280	NA	NA	NA	NA	NA	NA	NA	2717	NA	NA
07/14/83	5.90	5.0	2008	1680	NA	NA	NA	NA	12.030	54.000	2.830	2.320	NA	NA	NA	NA	NA	NA	NA	7748	NA	NA
08/14/83	5.41	3.4	2095	1750	NA	NA	NA	NA	16.750	71.000	3.300	2.880	NA	NA	NA	NA	NA	NA	NA	5553	NA	NA
09/15/83	5.64	2.5	2123	1940	NA	NA	NA	NA	16.500	75.000	3.590	3.080	NA	NA	NA	NA	NA	NA	NA	4565	NA	NA
10/13/83	5.70	3.8	2422	1880	NA	NA	NA	NA	19.000	78.000	3.770	3.140	NA	NA	NA	NA	NA	NA	NA	20303	NA	NA
11/09/83	5.90	3.6	2300	2600	NA	NA	NA	NA	20.400	99.000	4.440	3.320	NA	NA	NA	NA	NA	NA	NA	3925	NA	NA
11/30/83	5.50	2.7	2175	1960	NA	NA	NA	NA	20.200	89.600	4.050	3.040	NA	NA	NA	NA	NA	NA	NA	2589	NA	NA
02/24/84	4.80	0.9	1900	2040	236.3	189.9	50.7	NA	30.600	102.000	4.500	3.840	NA	NA	NA	NA	NA	NA	NA	2907	NA	NA
03/15/84	5.40	2.7	2375	2000	288.3	274.4	65.1	NA	35.000	144.000	7.000	4.890	NA	NA	NA	NA	NA	NA	NA	4375	NA	NA
04/15/84	6.40	10.4	878	580	104.0	70.6	19.9	NA	7.490	34.000	1.680	1.300	NA	NA	NA	NA	NA	NA	NA	15221	NA	NA
05/16/84	6.16	6.9	1190	980	165.6	109.4	31.2	NA	12.000	51.000	2.400	1.820	NA	NA	NA	NA	NA	NA	NA	8861	NA	NA
06/13/84	5.52	2.6	1519	1310	205.9	139.6	40.8	NA	17.500	69.400	3.150	2.290	NA	NA	NA	NA	NA	NA	NA	17886	NA	NA
07/14/84	5.05	2.5	2130	2480	NA	NA	57.4	NA	19.000	121.300	4.000	3.200	NA	NA	NA	NA	NA	NA	NA	6064	NA	NA
08/15/84	5.34	3.0	2050	1800	267.8	142.9	49.6	NA	16.100	73.760	4.880	2.360	NA	NA	NA	NA	NA	NA	NA	4478	NA	NA
09/12/84	4.96	2.3	2340	1760	317.1	220.0	70.5	NA	26.500	105.000	7.120	3.440	NA	NA	NA	NA	NA	NA	NA	1340	NA	NA
10/13/84	5.02	2.1	1860	1180	259.1	149.5	43.2	NA	34.000	92.000	6.830	3.010	NA	NA	NA	NA	NA	NA	NA	10579	NA	NA
11/14/84	4.96	1.6	2190	2700	381.0	450.0	84.7	NA	38.600	138.000	5.380	4.180	NA	NA	NA	NA	NA	NA	NA	2918	NA	NA
12/12/84	4.93	0.8	1585	1800	242.0	290.0	62.0	NA	26.500	108.800	4.840	3.490	NA	NA	NA	NA	NA	NA	NA	3459	NA	NA

Table A2.3. Drainage quality data from Duluth Complex test pile FL3, from 1978-1989 (0.63% S, 0.35% Cu, 0.08% Ni).

Date[1]	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/02/85	6.60	17.9	800	470	75.8	54.2	14.5	NA	5.800	27.000	1.070	0.860	NA	NA	NA	NA	NA	NA	NA	2394	NA	NA
04/16/85	5.94	9.0	1360	1010	144.0	121.6	28.6	NA	18.500	70.000	2.800	2.090	NA	NA	NA	NA	NA	NA	NA	16699	NA	NA
05/14/85	5.25	3.9	2400	2000	220.0	210.0	53.0	NA	24.000	100.000	4.710	2.830	NA	NA	NA	NA	NA	NA	NA	6828	NA	NA
06/14/85	5.57	4.0	2150	1736	209.0	169.0	46.3	NA	27.000	103.000	4.140	3.210	NA	NA	NA	NA	NA	NA	NA	26461	NA	NA
07/15/85	5.31	4.3	2300	1900	220.0	186.0	51.0	NA	23.000	99.000	4.310	3.490	NA	NA	NA	NA	NA	NA	NA	7630	NA	NA
08/13/85	5.47	4.0	2125	1750	226.0	153.4	48.0	NA	22.400	89.000	4.240	3.170	NA	NA	NA	NA	NA	NA	NA	5761	NA	NA
09/13/85	5.39	3.8	2325	1850	294.0	141.8	44.0	NA	39.000	100.600	4.820	3.560	NA	NA	NA	NA	NA	NA	NA	18187	NA	NA
10/15/85	5.44	4.7	2425	1650	239.0	160.0	43.0	NA	37.600	102.800	4.660	3.010	NA	NA	NA	NA	NA	NA	NA	17222	NA	NA
11/15/85	5.26	2.6	2675	1450	322.0	224.0	60.0	NA	41.800	133.000	6.020	4.020	NA	NA	NA	NA	NA	NA	NA	1949	NA	NA
03/20/86	5.25	2.2	3100	2900	366.0	334.0	78.0	NA	52.400	220.000	7.900	5.200	NA	NA	NA	NA	NA	NA	NA	401	NA	NA
03/23/86	6.30	9.5	975	700	90.0	67.2	13.0	NA	17.000	48.000	2.120	1.310	NA	NA	NA	NA	NA	NA	NA	17998	NA	NA
04/16/86	6.09	7.4	1175	700	126.0	84.0	18.0	NA	17.000	51.000	2.540	1.590	NA	NA	NA	NA	NA	NA	NA	13743	NA	NA
05/14/86	5.70	4.4	1650	1350	214.0	144.0	28.0	NA	29.000	120.000	5.220	2.330	NA	NA	NA	NA	NA	NA	NA	10995	NA	NA
06/11/86	5.29	3.6	2425	1650	234.0	173.0	49.0	NA	31.000	108.000	4.360	3.690	NA	NA	NA	NA	NA	NA	NA	3509	NA	NA
07/12/86	5.40	3.7	1775	1100	180.0	130.0	32.0	NA	27.000	80.000	3.350	2.690	NA	NA	NA	NA	NA	NA	NA	7104	NA	NA
08/13/86	5.17	3.0	2200	1550	228.0	155.6	40.0	NA	37.000	95.000	4.200	2.280	NA	NA	NA	NA	NA	NA	NA	4273	NA	NA
09/14/86	5.21	3.0	2000	1400	224.0	159.8	36.0	NA	41.000	98.000	5.100	2.360	NA	NA	NA	NA	NA	NA	NA	18682	NA	NA
10/16/86	5.11	2.1	2900	1825	258.0	227.0	48.0	NA	48.000	142.000	5.750	4.150	NA	NA	NA	NA	NA	NA	NA	4557	NA	NA
11/16/86	5.09	2.7	3150	1900	290.0	208.0	62.0	NA	72.000	161.000	9.100	6.100	NA	NA	NA	NA	NA	NA	NA	7899	NA	NA
03/22/87	5.30	7.1	1125	730	126.0	80.0	22.0	NA	20.200	60.000	2.780	1.930	NA	NA	NA	NA	NA	NA	NA	8656	NA	NA
04/15/87	5.73	4.6	1450	900	118.0	100.0	22.0	NA	28.000	68.000	3.070	2.240	NA	NA	NA	NA	NA	NA	NA	3861	NA	NA
05/16/87	5.13	4.5	2300	1770	319.0	256.0	56.0	NA	51.000	129.000	5.700	4.700	NA	NA	NA	NA	NA	NA	NA	24103	NA	NA
06/16/87	4.81	3.0	3550	2640	356.0	328.0	70.0	NA	63.000	197.000	6.000	4.100	NA	NA	NA	NA	NA	NA	NA	2048	NA	NA
07/15/87	5.05	5.7	2325	1780	226.0	187.0	45.0	NA	53.000	131.000	5.200	3.950	NA	NA	NA	NA	NA	NA	NA	18842	NA	NA
08/15/87	4.91	3.4	2425	1750	206.0	172.0	42.0	NA	43.000	121.000	4.800	3.720	NA	NA	NA	NA	NA	NA	NA	8501	NA	NA
09/16/87	5.01	2.8	2650	1615	200.0	158.0	36.0	NA	42.000	105.000	4.510	3.120	NA	NA	NA	NA	NA	NA	NA	12195	NA	NA
10/13/87	4.81	2.1	3825	2680	323.0	307.0	70.0	NA	71.000	192.000	7.730	5.430	NA	NA	NA	NA	NA	NA	NA	2142	NA	NA
11/10/87	4.71	1.8	3350	2700	288.0	278.0	64.0	NA	59.000	174.000	6.880	4.930	NA	NA	NA	NA	NA	NA	NA	1064	NA	NA
03/28/88	6.00	11.1	900	414	88.0	48.0	12.0	NA	11.000	32.000	1.600	1.600	NA	NA	NA	NA	NA	NA	NA	8653	NA	NA
04/17/88	5.69	6.3	1575	950	136.0	100.0	22.0	NA	24.000	90.000	3.800	4.100	NA	NA	NA	NA	NA	NA	NA	7297	NA	NA
05/12/88	5.17	3.4	1985	1270	172.0	160.0	38.0	NA	37.000	129.000	5.200	6.100	NA	NA	NA	NA	NA	NA	NA	10624	NA	NA
06/12/88	4.86	2.9	3500	2440	344.0	324.0	67.0	NA	70.000	178.500	10.200	11.500	NA	NA	NA	NA	NA	NA	NA	1654	NA	NA
07/15/88	4.91	3.5	3175	2230	278.0	286.0	58.0	NA	59.000	161.000	8.800	11.200	NA	NA	NA	NA	NA	NA	NA	995	NA	NA
08/16/88	5.26	2.5	1910	1240	176.0	130.0	34.0	NA	37.000	125.000	4.700	4.200	NA	NA	NA	NA	NA	NA	NA	45053	NA	NA
09/16/88	5.07	2.7	2300	1360	202.0	170.0	44.0	NA	41.000	145.000	5.700	5.300	NA	NA	NA	NA	NA	NA	NA	8134	NA	NA
10/18/88	5.12	3.0	2000	1330	158.0	144.0	32.0	NA	33.000	93.000	5.000	5.500	NA	NA	NA	NA	NA	NA	NA	3376	NA	NA
11/15/88	5.21	3.6	1625	1250	142.0	131.0	26.0	NA	32.500	87.000	4.700	5.000	NA	NA	NA	NA	NA	NA	NA	6654	NA	NA
12/05/88	4.80	3.2	3300	3150	298.0	350.0	62.0	NA	73.000	238.000	11.900	10.000	NA	NA	NA	NA	NA	NA	NA	1681	NA	NA
04/07/89	6.15	9.9	650	425	70.0	46.0	12.0	NA	12.700	32.000	1.900	1.700	NA	NA	NA	NA	NA	NA	NA	23758	NA	NA
04/24/89	5.91	6.5	1075	650	100.0	58.0	12.0	NA	16.000	46.000	2.500	2.500	NA	NA	NA	NA	NA	NA	NA	7638	NA	NA
05/16/89	5.18	3.9	1593	1090	128.0	108.0	20.0	NA	28.000	79.000	4.500	5.300	NA	NA	NA	NA	NA	NA	NA	9284	NA	NA
06/17/89	4.98	3.3	1821	1375	156.0	128.0	30.0	NA	41.000	119.000	6.800	5.000	NA	NA	NA	NA	NA	NA	NA	29326	NA	NA
07/17/89	4.81	2.2	2800	1950	306.0	210.0	46.0	NA	66.000	185.000	8.700	9.100	NA	NA	NA	NA	NA	NA	NA	2967	NA	NA
08/14/89	4.94	3.1	2900	1970	248.0	222.0	48.0	NA	61.000	240.000	8.200	8.200	NA	NA	NA	NA	NA	NA	NA	1328	NA	NA
09/14/89	4.90	2.7	2400	1600	252.0	166.0	36.0	NA	56.000	148.000	9.500	6.700	NA	NA	NA	NA	NA	NA	NA	17888	NA	NA
10/16/89	4.83	0.6	2775	1800	218.0	186.0	46.0	NA	47.000	161.000	12.100	8.400	NA	NA	NA	NA	NA	NA	NA	1835	NA	NA
11/10/89	5.00	2.8	2400	1820	240.0	182.0	40.0	NA	42.000	137.000	14.800	7.700	NA	NA	NA	NA	NA	NA	NA	900	NA	NA

NA: Not analyzed.

[1]Dates represent intermediate dates for measurement periods.

Table A2.4. Drainage quality data from Duluth Complex test pile FL4, from 1978-1982 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.20	NA	1100	NA	121.0	11.0	84.0	27.0	0.018	NA	0.004	0.048	NA	0.10	NA	NA	NA	NA	NA	NA	NA	4.0
05/25/78	7.40	NA	1750	NA	204.0	20.0	165.0	35.0	0.012	0.210	0.012	0.112	0.02	0.19	NA	NA	NA	NA	NA	NA	NA	NA
06/14/78	7.60	NA	2500	NA	228.0	29.0	262.0	37.0	0.024	0.220	0.016	0.128	0.04	0.21	NA	NA	NA	NA	NA	125	NA	9.0
06/16/78	7.70	NA	2600	NA	247.0	30.0	277.0	41.0	0.026	0.175	0.015	0.120	0.04	0.26	NA	NA	NA	NA	NA	94	NA	11.0
06/23/78	7.56	NA	2402	1150	205.0	27.8	271.4	39.5	0.022	0.308	0.023	0.134	0.03	0.21	25.5	NA	NA	NA	NA	307	NA	12.7
06/26/78	7.90	NA	2150	900	186.0	24.0	260.0	41.0	0.021	0.260	0.019	0.084	NA	0.18	28.0	NA	NA	NA	NA	5	NA	17.0
07/06/78	7.58	NA	1825	850	212.0	25.0	182.0	52.0	0.019	0.275	0.019	0.073	0.03	0.18	11.0	NA	NA	0.10	NA	88	NA	17.0
07/07/78	7.43	NA	1306	546	156.0	15.5	129.9	31.4	0.015	0.183	0.019	0.064	0.03	0.11	7.8	1.6	0.73	0.08	NA	4957	NA	11.0
07/08/78	7.46	NA	1406	830	220.2	16.8	127.0	41.1	0.014	0.206	0.020	0.058	0.03	0.14	6.8	NA	NA	0.08	NA	NA	NA	NA
07/09/78	7.55	NA	1550	NA	300.0	20.0	149.0	46.0	0.009	0.240	0.017	NA	0.02	0.16	NA	NA	NA	NA	NA	NA	NA	NA
07/13/78	7.65	NA	NA	850	273.0	18.0	153.0	44.0	0.012	NA	0.015	NA	0.02	0.14	7.0	2.2	0.90	0.08	NA	NA	NA	NA
08/04/78	8.00	NA	1100	660	191.0	13.0	115.0	43.0	0.020	0.200	0.010	0.052	0.02	0.14	3.0	1.2	0.06	0.08	NA	95	NA	22.0
08/15/78	7.80	39.0	1275	800	201.0	19.0	158.0	39.0	0.020	0.240	0.010	0.070	0.02	0.15	6.0	1.2	1.20	0.08	NA	NA	NA	21.0
08/16/78	7.50	28.0	825	100	118.0	11.0	60.0	26.0	0.020	0.130	0.020	0.030	0.03	0.09	1.0	1.2	NA	0.08	NA	NA	NA	NA
08/17/78	7.29	29.0	1175	1000	355.0	41.0	158.0	12.0	0.030	0.200	0.020	0.050	0.04	0.12	13.0	2.2	NA	0.08	NA	NA	NA	NA
08/18/78	7.60	36.0	1850	800	219.0	21.0	110.0	42.0	0.020	0.240	0.040	0.070	0.02	0.16	3.0	1.4	NA	0.08	NA	NA	NA	NA
08/21/78	7.75	NA	2050	NA	271.0	27.0	142.0	50.0	0.080	0.320	0.040	0.100	NA	0.20	NA	NA	NA	NA	NA	NA	NA	NA
08/23/78	7.47	30.0	1217	603	137.7	15.0	84.7	32.3	0.020	0.143	0.023	0.037	0.02	0.10	2.3	1.4	1.08	0.08	NA	NA	NA	23.0
08/25/78	7.55	33.0	1750	800	216.0	21.0	115.0	46.0	0.020	0.240	0.030	0.060	0.01	0.14	2.0	1.4	1.00	0.08	NA	NA	NA	23.0
08/28/78	7.55	41.8	2350	1250	280.0	29.0	187.0	60.0	0.030	0.340	0.040	0.100	0.02	0.19	2.0	2.4	NA	0.08	NA	NA	NA	NA
09/11/78	7.30	35.0	2300	1200	256.0	26.0	219.0	59.0	0.030	0.340	0.050	0.080	0.06	0.20	10.0	NA	NA	0.08	NA	NA	NA	NA
09/13/78	7.23	34.0	1650	852	191.7	17.7	107.3	44.3	0.023	0.210	0.033	0.053	0.06	0.14	2.5	1.4	0.06	0.08	NA	NA	NA	12.0
10/17/78	7.50	31.0	1800	1000	257.0	27.0	157.0	34.0	0.020	0.290	0.040	0.090	0.02	0.18	6.0	1.4	0.06	0.08	NA	NA	NA	NA
03/19/79	NA	NA	861	380	96.6	18.2	39.4	11.2	3.000	14.600	0.256	0.400	0.03	1.08	8.0	NA	NA	NA	4.1	2660	2428	NA
04/17/79	5.90	7.0	635	325	77.0	14.0	34.4	10.6	1.120	12.310	0.240	0.240	1.46	1.11	7.0	NA	NA	NA	3.5	6041	4676	NA
04/18/79	6.80	22.0	401	170	66.0	6.0	25.0	9.0	0.069	0.373	0.064	0.050	0.02	0.15	3.0	1.2	NA	NA	NA	4644	5152	NA
04/19/79	7.10	21.0	391	170	65.0	5.0	23.0	8.6	0.016	0.098	0.008	0.040	0.02	0.07	1.0	0.5	NA	NA	NA	5560	5332	NA
04/20/79	6.90	29.0	568	285	106.0	8.6	NA	14.2	0.064	0.210	0.008	0.040	0.02	0.10	3.0	0.6	0.12	NA	2.2	4364	3329	NA
04/20/79	6.40	22.0	364	145	67.0	4.8	24.0	17.8	0.014	0.089	0.008	0.030	0.02	0.07	2.0	0.4	0.05	NA	NA	3077	3329	NA
04/22/79	7.40	19.0	980	483	171.0	15.8	42.6	24.4	0.021	0.270	0.020	0.070	0.03	0.14	4.0	1.0	0.03	NA	NA	4599	1534	NA
05/09/79	7.50	NA	1039	NA	171.2	15.8	48.2	27.4	0.016	0.310	0.023	0.070	0.03	0.17	NA	NA	NA	NA	NA	647	1239	NA
05/10/79	7.50	NA	1010	532	162.4	16.6	50.4	25.6	0.021	0.280	0.020	0.060	0.01	0.15	4.0	NA	NA	NA	NA	8717	4417	NA
05/20/79	6.90	NA	1363	817	227.8	23.6	77.9	35.2	0.026	0.375	0.029	0.080	0.02	0.17	5.0	NA	NA	NA	5.7	3062	1080	NA
05/22/79	7.60	NA	984	583	178.4	18.8	68.8	30.0	0.025	0.320	0.023	0.080	0.01	0.15	3.0	NA	NA	NA	3.5	1904	1278	NA
05/23/79	7.20	NA	1264	700	193.4	21.6	74.0	32.8	0.030	0.360	0.027	0.070	0.02	0.16	4.0	NA	NA	NA	4.5	3357	699	NA
06/01/79	7.70	NA	1334	800	201.8	22.0	80.4	34.8	0.027	0.350	0.027	0.080	0.03	0.17	5.0	NA	NA	NA	4.7	2290	662	NA
06/06/79	7.60	NA	1213	700	180.6	19.4	48.4	48.2	0.023	0.300	0.021	0.080	0.01	0.14	5.0	NA	NA	NA	NA	2044	1227	NA
06/10/79	7.30	36.0	1116	560	159.4	18.0	45.0	41.6	0.023	0.240	0.013	0.060	0.03	0.11	4.0	NA	NA	NA	NA	2161	1411	22.0
06/12/79	7.30	32.0	1107	NA	157.0	17.2	43.6	40.8	0.013	0.240	0.009	0.060	0.01	0.11	NA	NA	NA	NA	NA	681	245	NA
06/15/79	7.30	33.0	1138	NA	160.4	18.4	45.8	37.6	0.016	0.280	0.009	0.060	0.01	0.12	NA	NA	NA	NA	NA	469	442	21.0
06/20/79	7.10	30.0	1347	570	171.0	21.0	72.0	38.0	0.020	0.260	0.020	0.070	0.03	0.97	5.0	NA	NA	NA	NA	2854	1339	17.0
06/30/79	NA	NA	NA	NA	176.0	21.0	73.0	35.0	0.010	0.270	0.020	0.050	0.01	0.10	NA	NA	NA	NA	NA	526	NA	NA
07/30/79	5.75	32.0	907	460	118.0	22.0	38.0	20.0	0.700	4.140	0.260	0.180	0.01	0.55	11.0	NA	NA	NA	4.9	2324	2847	23.0
09/01/79	7.00	10.4	1112	650	189.6	16.8	40.4	33.8	0.040	0.510	0.028	0.090	0.01	0.14	4.0	NA	NA	NA	4.0	4307	1988	20.5
09/05/79	7.10	14.4	1333	NA	246.4	22.4	46.8	40.0	0.040	0.670	0.034	0.120	0.01	0.19	NA	NA	NA	NA	5.1	337	270	20.0
09/12/79	7.00	12.8	1250	800	228.6	20.0	45.6	38.2	0.060	0.780	0.060	0.130	0.01	0.19	6.0	NA	NA	NA	3.7	1158	417	20.0
09/13/79	7.20	NA	1321	NA	237.8	21.4	55.6	43.4	0.030	0.710	0.035	0.110	0.02	0.18	NA	NA	NA	NA	NA	91	270	20.0
10/19/79	6.40	7.2	980	600	166.4	14.6	24.0	27.8	0.080	1.260	0.070	0.160	0.01	0.13	2.0	NA	NA	NA	1.9	541	307	NA
10/31/79	6.60	8.8	1025	550	152.0	15.0	38.0	28.8	0.080	1.040	0.060	0.140	0.01	0.12	3.0	NA	NA	NA	2.2	1764	908	NA
03/31/80	6.90	15.0	470	205	54.2	5.4	12.6	11.6	0.030	0.395	0.035	0.090	0.02	0.06	2.0	NA	NA	NA	1.8	1207	1154	NA
04/05/80	7.10	17.2	325	140	38.5	4.0	8.0	8.3	0.040	0.270	0.010	0.070	0.02	0.04	1.0	NA	NA	NA	1.0	5326	2260	NA

Table A2.4. Drainage quality data from Duluth Complex test pile FL4, from 1978-1982 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/06/80	6.80	16.1	575	250	76.0	7.0	11.2	13.0	0.050	0.410	0.030	0.100	0.03	0.12	2.0	NA	NA	NA	NA	1298	1349	NA
04/18/80	6.90	12.7	NA	258	83.9	7.8	13.3	14.5	0.035	0.695	0.035	0.150	0.02	0.14	2.0	NA	NA	NA	1.0	2215	450	NA
06/01/80	6.50	6.6	950	710	174.9	17.1	25.1	30.8	0.160	2.030	0.120	0.230	0.02	0.25	4.0	NA	NA	NA	NA	579	313	NA
06/06/80	6.80	6.6	1100	NA	164.6	16.6	24.4	28.6	0.180	2.250	0.160	0.270	0.03	0.27	NA	NA	NA	NA	NA	246	232	NA
07/03/80	6.70	7.6	1150	770	186.0	20.0	33.0	29.0	0.250	2.140	0.120	0.280	0.02	0.24	3.0	NA	NA	NA	NA	1344	615	NA
08/12/80	6.50	6.0	1400	860	224.8	28.8	34.4	35.6	1.140	8.380	0.440	0.510	0.01	0.46	3.0	NA	NA	NA	2.3	1090	683	NA
08/19/80	6.90	8.5	1500	975	263.8	31.6	51.9	43.5	0.690	5.990	0.300	0.460	0.01	0.53	4.0	NA	NA	NA	NA	1594	464	NA
08/22/80	NA	NA	1650	1100	275.0	37.0	57.0	36.0	0.490	4.080	0.250	0.350	0.03	0.38	6.0	NA	NA	NA	NA	1503	1084	NA
08/29/80	5.50	3.4	1360	1040	160.0	17.0	52.0	34.0	0.750	4.810	0.310	0.270	0.06	0.46	11.0	NA	NA	NA	3.4	11432	7250	NA
09/07/80	6.70	13.6	1695	1170	272.0	24.0	99.0	48.0	0.400	2.700	0.150	0.250	0.03	0.29	6.0	NA	NA	NA	NA	2142	1860	NA
09/11/80	6.80	11.5	1246	710	132.0	9.0	59.0	36.0	0.170	1.840	0.100	0.200	0.04	0.23	5.0	NA	NA	NA	NA	969	655	15.0
09/12/80	6.80	12.8	1275	780	142.0	11.0	60.0	35.0	0.350	2.180	0.110	0.190	0.04	0.23	4.0	NA	NA	NA	2.6	4895	3026	16.0
09/18/80	7.00	14.4	1450	910	191.0	16.0	67.0	38.0	0.310	1.850	0.090	0.180	0.04	0.18	5.0	NA	NA	NA	NA	10209	3354	14.0
09/23/80	6.40	17.8	1822	1150	313.2	48.4	96.0	12.8	0.320	2.610	0.140	0.270	0.03	0.22	5.0	NA	NA	NA	3.3	1579	878	14.0
10/23/80	6.50	12.7	1598	1050	270.0	28.0	73.0	45.0	1.240	5.800	0.280	0.420	0.01	0.40	6.0	NA	NA	NA	2.5	1571	250	10.0
10/26/80	6.50	8.5	1576	1050	238.0	32.0	60.0	46.0	0.480	4.160	0.230	0.310	0.02	0.36	5.0	NA	NA	NA	NA	568	148	13.0
02/20/81	5.20	2.6	635	325	63.7	19.2	27.3	NA	1.360	7.100	0.450	0.270	NA	NA	5.0	NA	NA	NA	NA	1601	264	7.0
03/23/81	7.00	6.4	598	305	69.5	10.5	19.5	NA	0.140	1.470	0.085	0.160	NA	NA	NA	NA	NA	NA	NA	1949	750	10.0
03/30/81	6.70	7.2	500	215	58.5	9.0	10.4	NA	0.260	1.720	0.110	0.150	NA	NA	NA	NA	NA	NA	1.7	9089	2006	NA
04/05/81	6.90	12.0	877	490	119.6	18.5	22.1	NA	0.520	4.210	0.220	0.310	NA	NA	NA	NA	NA	NA	NA	6367	1136	NA
04/14/81	NA	NA	NA	610	156.0	29.0	27.0	NA	0.460	3.230	0.170	0.260	NA	NA	NA	NA	NA	NA	NA	3660	1760	NA
04/17/81	7.20	13.6	1300	660	159.0	31.0	30.0	NA	0.340	2.900	0.150	0.230	NA	NA	NA	NA	NA	NA	1.7	2862	2063	NA
04/23/81	7.00	13.6	1400	660	179.0	35.0	31.0	NA	0.320	2.540	0.140	0.220	NA	NA	NA	NA	NA	NA	NA	3895	2177	NA
04/24/81	7.15	18.6	1400	700	168.0	35.5	32.0	NA	0.220	1.860	0.100	0.160	NA	NA	NA	NA	NA	NA	NA	7730	2460	NA
04/28/81	7.20	NA	1325	640	153.0	34.0	31.0	NA	0.180	1.580	0.100	0.140	NA	NA	NA	NA	NA	NA	1.7	4376	3237	NA
04/30/81	7.20	20.8	1430	845	220.0	36.0	38.0	NA	0.180	1.920	0.100	0.190	NA	NA	NA	NA	NA	NA	NA	4092	856	12.0
05/13/81	7.30	18.4	1574	700	276.0	40.0	48.0	NA	0.160	2.290	0.120	0.230	NA	NA	NA	NA	NA	NA	2.6	1749	42	18.5
06/01/81	7.00	16.1	1638	1000	262.0	40.0	64.0	NA	0.720	4.620	0.260	0.380	NA	NA	NA	NA	NA	NA	NA	3028	1563	16.0
06/07/81	7.10	9.1	1264	685	198.0	26.0	42.0	NA	0.360	3.000	0.170	0.270	NA	NA	NA	NA	NA	NA	3.6	2688	202	18.0
06/22/81	6.70	10.4	1240	675	194.0	28.0	42.0	NA	0.610	3.950	0.220	0.250	NA	NA	NA	NA	NA	NA	NA	5617	2156	16.0
06/24/81	6.85	8.0	1152	605	187.0	24.0	31.0	NA	0.450	2.980	0.160	0.190	NA	NA	NA	NA	NA	NA	NA	4747	1760	16.0
06/28/81	6.90	10.4	1197	665	192.0	26.0	36.0	NA	0.380	2.300	0.130	0.160	NA	NA	NA	NA	NA	NA	2.5	6488	2244	15.0
07/04/81	7.15	13.8	1528	1010	278.0	32.6	50.4	NA	0.240	1.690	0.090	0.180	NA	NA	NA	NA	NA	NA	NA	4247	167	22.0
07/17/81	6.95	12.0	1916	1300	332.0	46.6	81.0	NA	0.830	4.790	0.230	0.470	NA	NA	NA	NA	NA	NA	NA	787	481	20.0
07/25/81	6.85	8.4	1903	1200	328.0	46.6	75.8	NA	1.600	7.420	0.370	0.540	NA	NA	2.0	NA	NA	NA	1.9	1291	256	19.0
08/10/81	6.60	7.4	1711	1060	276.0	50.0	68.0	NA	2.000	8.000	0.470	0.590	NA	NA	NA	NA	NA	NA	NA	1548	560	18.0
08/13/81	6.80	10.6	1767	1050	282.0	44.0	64.0	48.0	0.690	6.000	0.390	0.460	NA	NA	NA	NA	NA	NA	NA	333	55	22.0
08/31/81	6.30	4.8	1800	1000	254.0	48.0	60.0	NA	3.840	11.290	0.590	0.740	NA	NA	NA	NA	NA	NA	1.9	496	337	NA
09/07/81	6.45	4.8	2250	1140	288.0	52.0	60.0	NA	3.030	12.000	0.640	0.730	NA	NA	NA	NA	NA	NA	NA	893	771	NA
09/26/81	6.00	3.2	2025	1190	286.0	56.0	58.0	NA	3.840	15.000	0.830	0.850	NA	NA	NA	NA	NA	NA	1.6	1688	1105	NA
10/01/81	6.40	4.8	1750	880	248.0	44.0	40.0	NA	1.680	9.610	0.510	0.600	NA	NA	NA	NA	NA	NA	NA	3838	2234	NA
10/04/81	6.50	4.8	1800	980	296.0	44.0	40.0	44.0	1.080	10.000	0.470	0.530	NA	NA	NA	NA	NA	NA	NA	7219	2033	NA
10/13/81	6.70	5.6	1850	1000	296.0	44.0	48.0	46.0	0.810	7.000	0.360	0.470	NA	NA	NA	NA	NA	NA	1.5	9596	2851	NA
10/17/81	6.80	7.2	1220	1000	272.0	42.0	52.0	48.0	0.650	5.320	0.300	0.360	NA	NA	NA	NA	NA	NA	NA	8725	2038	NA
11/06/81	6.85	6.4	2100	1300	340.0	62.0	68.0	58.0	0.710	5.100	0.250	0.270	NA	NA	NA	NA	NA	NA	2.5	3562	65	NA
04/11/82	6.80	4.8	750	360	90.2	13.6	14.0	16.0	0.590	3.930	0.210	0.300	NA	NA	NA	NA	NA	NA	NA	1832	NA	NA
04/15/82	6.50	4.5	750	370	87.6	16.4	14.0	14.0	0.700	5.380	0.310	0.330	NA	NA	NA	NA	NA	NA	NA	2074	NA	NA
04/19/82	6.70	6.3	800	430	126.0	20.0	16.0	18.0	0.620	6.210	0.340	0.350	NA	NA	NA	NA	NA	NA	NA	1754	NA	NA
04/23/82	6.60	6.3	1050	560	164.0	20.4	22.0	24.0	0.860	9.190	0.480	0.490	NA	NA	NA	NA	NA	NA	NA	1881	NA	NA
05/03/82	6.10	4.1	1300	750	181.0	42.0	30.0	30.0	1.660	11.355	0.620	0.620	NA	NA	NA	NA	NA	NA	NA	829	NA	NA
05/04/82	6.45	5.8	1250	690	164.0	36.0	22.0	28.0	0.900	8.715	0.460	0.460	NA	NA	NA	NA	NA	NA	NA	4281	NA	NA

Table A2.4. Drainage quality data from Duluth Complex test pile FL4, from 1978-1982 (0.63% S, 0.35% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
05/07/82	6.40	6.3	1290	720	152.0	40.0	22.0	26.0	0.810	8.520	0.460	0.420	NA	NA	NA	NA	NA	NA	NA	7730	NA	NA
05/11/82	6.75	9.9	1500	NA	196.0	50.0	30.0	32.0	0.660	8.150	0.430	0.400	NA	NA	NA	NA	NA	NA	NA	5129	NA	NA
05/14/82	6.60	9.9	1550	940	200.0	18.0	20.0	16.0	0.270	2.680	0.130	0.150	0.02	0.19	NA	NA	NA	NA	NA	5462	NA	NA
05/21/82	6.80	8.1	1850	1000	282.0	49.0	41.0	39.0	0.470	6.800	0.330	0.430	0.03	0.40	NA	NA	NA	NA	NA	1408	NA	NA
06/02/82	6.80	7.2	1980	790	288.0	52.6	48.8	42.6	0.950	8.310	0.400	0.560	0.06	0.53	NA	NA	NA	NA	NA	1594	NA	NA
06/18/82	6.40	9.9	1300	750	184.0	40.0	31.0	31.0	1.090	6.720	0.350	0.420	0.03	0.45	NA	NA	NA	NA	NA	984	NA	NA
07/02/82	NA	NA	NA	750	256.0	38.0	36.0	36.0	1.130	7.000	0.340	0.320	0.02	0.44	NA	NA	NA	NA	NA	25536	NA	NA
07/26/82	6.31	6.2	1520	785	232.0	36.0	40.0	40.0	1.150	8.860	0.460	0.330	0.10	0.47	NA	NA	NA	NA	NA	13832	NA	NA
08/30/82	6.28	4.4	1707	958	336.0	49.0	40.0	46.0	1.655	9.360	0.450	0.410	0.01	0.57	NA	NA	NA	NA	NA	10815	NA	NA

NA: Not analyzed.

'Dates represent intermediate dates for measurement periods.

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.25	NA	5000	NA	478.0	139.0	535.0	12.0	0.088	0.540	0.030	0.022	0.28	0.60	NA	NA	NA	NA	NA	NA	NA	4.0
05/25/78	7.25	NA	NA	NA	840.0	152.0	862.0	18.0	0.121	0.740	0.062	0.003	0.12	1.14	NA	NA	NA	NA	NA	NA	NA	NA
06/16/78	7.12	NA	8860	NA	1028.0	206.0	1106.0	22.0	0.258	0.665	0.059	0.033	0.12	1.64	NA	NA	NA	NA	NA	135	NA	15.0
07/05/78	7.40	NA	4610	2000	435.0	141.0	491.0	17.0	0.060	0.735	0.051	0.006	0.02	0.70	420.0	NA	NA	NA	NA	223	NA	12.0
07/13/78	7.49	NA	NA	1450	794.0	124.0	685.0	16.0	0.118	1.080	0.067	0.014	0.08	0.95	607.0	400.0	100.00	0.08	NA	NA	NA	NA
07/19/78	8.10	NA	3700	1300	540.0	93.0	508.0	12.0	0.092	0.860	0.052	0.006	0.05	0.75	312.0	160.0	100.00	1.90	NA	NA	NA	18.0
08/02/78	7.70	NA	2600	950	258.0	61.0	260.0	9.0	0.060	0.630	0.035	0.020	0.05	0.62	180.0	48.0	70.00	0.10	NA	NA	NA	23.0
08/04/78	7.75	NA	4150	1700	451.0	120.0	NA	12.0	0.120	0.930	0.140	0.030	0.04	1.04	320.0	210.0	100.00	0.10	NA	287	NA	22.0
08/10/78	7.55	47.0	6840	2400	748.0	178.0	700.0	22.0	0.170	1.430	0.280	0.050	0.15	1.68	600.0	NA	160.00	0.08	NA	77	NA	24.0
08/16/78	7.78	40.0	2225	1000	250.0	74.0	257.0	6.0	0.095	0.935	0.160	0.030	0.06	0.82	150.0	140.0	NA	0.08	NA	28390	NA	22.0
08/17/78	7.52	46.0	4600	1700	396.0	46.0	200.0	14.0	0.130	1.480	0.270	0.050	0.05	1.38	190.0	120.0	NA	0.08	NA	NA	NA	NA
08/23/78	7.55	30.0	4320	1600	368.0	112.0	396.0	12.0	0.130	1.540	0.240	0.050	0.05	1.40	280.0	170.0	NA	0.08	NA	NA	NA	NA
08/25/78	7.40	40.0	4750	1800	373.0	119.0	432.0	12.0	0.150	1.700	0.270	0.050	0.07	1.53	220.0	140.0	98.00	0.08	NA	NA	NA	23.0
08/28/78	7.85	57.0	5290	2150	481.0	135.0	681.0	15.0	0.180	2.150	0.390	0.070	0.07	1.82	390.0	230.0	NA	0.08	NA	NA	NA	NA
09/13/78	7.16	25.5	3400	2075	387.5	120.0	446.5	14.0	0.285	2.335	0.220	0.120	0.05	2.14	270.0	NA	NA	0.10	NA	667	NA	13.0
10/17/78	6.20	6.7	5320	NA	491.0	133.0	609.0	13.0	1.160	4.120	0.400	0.360	0.04	3.15	NA	NA	NA	NA	NA	NA	NA	NA
03/23/79	6.85	10.0	1562	600	200.0	54.0	97.2	4.0	0.200	4.590	0.360	0.150	0.02	1.74	10.0	NA	NA	NA	5.7	2290	46	NA
04/12/79	6.00	NA	2714	1400	327.0	93.2	201.0	6.2	0.740	10.250	0.710	0.290	0.03	3.40	95.0	NA	NA	NA	NA	1665	840	NA
04/16/79	6.60	NA	1358	560	152.6	74.2	91.6	3.4	0.120	2.320	0.210	0.080	1.02	1.17	45.0	NA	NA	NA	NA	3168	1308	NA
04/17/79	6.60	12.0	1146	440	124.6	31.6	76.2	3.0	0.120	2.350	0.210	0.090	0.02	1.18	28.0	NA	NA	NA	4.7	3641	3916	NA
04/18/79	6.80	16.0	1318	488	162.0	38.4	86.0	2.6	0.170	2.630	0.230	0.100	0.02	1.38	30.0	62.0	NA	NA	NA	6529	2997	NA
04/19/79	6.50	15.0	1054	210	141.0	34.0	72.0	2.0	0.130	2.140	0.200	0.190	0.03	1.12	29.0	50.0	11.00	NA	NA	3971	3862	NA
04/20/79	6.70	16.5	1498	559	185.0	56.1	95.0	3.0	0.130	3.145	0.290	0.100	0.02	1.58	51.0	67.5	13.50	NA	4.2	5257	3929	NA
04/25/79	6.65	14.0	2613	NA	292.4	103.4	205.6	4.6	0.280	5.250	0.420	0.200	0.03	2.74	110.0	120.0	24.00	NA	NA	5001	719	NA
05/02/79	6.40	4.0	3460	1650	394.4	149.0	270.0	7.0	0.510	9.000	0.590	0.300	0.05	4.07	171.0	190.0	23.00	NA	NA	3387	1902	NA
05/09/79	6.60	NA	3482	1030	312.0	122.0	238.0	4.0	0.490	7.000	0.480	0.270	0.04	3.23	15.0	NA	NA	NA	NA	1654	1286	NA
05/10/79	7.00	NA	2432	867	240.2	104.2	200.2	4.4	0.340	4.470	0.360	0.180	0.01	2.48	111.0	NA	NA	NA	NA	10250	4690	NA
05/20/79	NA	NA	NA	1750	341.0	130.0	292.0	5.0	0.470	7.440	0.470	0.290	0.03	3.23	152.0	NA	NA	NA	8.6	4644	1266	NA
05/22/79	7.20	NA	2201	883	211.0	86.0	170.0	3.0	0.330	3.900	0.290	0.170	0.02	2.02	94.0	NA	NA	NA	16.0	2188	1399	NA
05/23/79	7.00	NA	3377	1700	316.0	60.4	269.0	6.6	0.490	6.110	0.460	0.250	0.06	3.05	143.0	NA	NA	NA	7.9	2653	1007	NA
06/01/79	6.90	NA	2668	1050	231.2	56.6	212.6	5.2	0.435	4.160	0.350	0.215	0.05	2.28	111.5	NA	NA	NA	7.1	2567	729	NA
06/06/79	6.40	NA	2196	867	199.4	79.8	86.6	2.0	0.330	3.360	0.240	0.170	0.01	1.79	94.0	NA	NA	NA	NA	2570	1130	NA
06/10/79	6.90	14 0	1916	700	184.8	77.6	79.4	3.8	0.380	3.520	0.126	0.140	0.02	1.60	79.0	NA	NA	NA	NA	2574	1433	20.0
06/12/79	6.60	10.0	2673	1200	266.3	107.5	154.0	3.3	0.375	4.500	0.350	0.210	0.01	2.26	104.0	NA	NA	NA	NA	1506	494	NA
06/16/79	6.70	9.9	3450	1230	325.6	127.0	189.0	4.4	0.520	5.660	0.430	0.290	0.04	2.77	139.0	NA	NA	NA	NA	1612	442	23.0
06/21/79	6.70	11.0	2411	750	202.0	87.0	155.0	4.0	0.390	3.880	0.240	0.180	0.01	5.57	91.0	NA	NA	NA	NA	2233	1687	13.0
06/30/79	6.30	6.6	3074	1450	308.0	136.0	249.0	6.0	0.720	5.820	0.380	0.320	0.03	2.59	128.0	NA	NA	NA	8.9	1991	230	NA
07/10/79	6.20	4.8	4599	2200	467.0	193.5	409.5	9.5	1.755	9.960	0.620	0.665	0.03	4.10	220.0	NA	NA	NA	9.0	1628	98	19.0
07/30/79	5.50	3.0	2006	925	209.0	97.0	105.0	4.0	1.450	6.010	0.390	0.290	0.01	2.04	79.0	NA	NA	NA	6.3	5269	3374	22.0
08/04/79	5.40	4.0	3438	1670	444.0	169.0	242.0	9.0	2.720	10.150	0.660	0.580	0.04	3.72	118.0	NA	NA	NA	8.7	1957	284	23.0
08/13/79	5.00	2.4	4690	2420	540.0	126.2	438.0	13.8	4.810	15.000	1.130	1.010	0.07	6.13	181.0	NA	NA	NA	9.1	753	108	14.0
08/23/79	4.80	1.6	3739	1800	439.4	174.0	232.0	11.4	4.220	14.300	1.080	0.840	0.03	5.24	153.0	NA	NA	NA	8.3	1192	342	9.0
08/28/79	5.30	6.0	3154	1400	345.0	112.6	260.0	8.2	3.790	13.700	1.000	0.700	0.06	4.23	113.0	NA	NA	NA	6.6	848	709	20.0
09/01/79	5.00	2.0	2576	1200	327.2	112.4	136.0	6.8	3.380	11.300	0.820	0.510	0.05	3.35	86.0	NA	NA	NA	6.3	5553	2337	22.0
09/05/79	4.70	0.4	3819	1980	478.8	121.6	308.0	10.6	5.270	16.200	1.180	0.840	0.05	5.33	102.0	NA	NA	NA	7.8	988	611	18.0
09/11/79	4.90	2.0	3364	1650	398.8	160.8	229.0	9.8	4.730	14.000	1.040	0.720	0.06	4.70	107.0	NA	NA	NA	7.3	3842	846	17.0
09/13/79	4.90	2.8	3745	1800	444.0	175.0	238.0	10.2	5.020	15.400	1.130	0.760	0.08	5.00	29.0	NA	NA	NA	9.8	2112	699	19.0
09/19/79	4.70	2.4	5710	2680	692.0	272.0	478.0	16.2	6.800	23.600	1.700	1.310	0.16	8.10	201.0	NA	NA	NA	9.0	1567	176	13.5
10/18/79	4.60	0.8	5168	2600	646.0	254.0	450.0	14.6	5.960	26.600	1.910	1.340	0.30	8.45	174.0	NA	NA	NA	8.5	931	758	NA
10/19/79	4.70	2.4	2356	1050	251.0	106.0	142.0	5.6	4.400	15.500	1.090	0.630	0.04	3.70	64.0	NA	NA	NA	4.3	2161	1863	NA
10/31/79	4.60	0.8	3448	1650	408.0	177.0	198.0	7.8	7.200	24.000	2.090	1.020	0.11	5.78	99.0	NA	NA	NA	5.8	6423	2331	NA

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
11/01/79	4.50	NA	3702	1750	447.0	203.5	230.0	9.1	8.535	25.500	2.170	1.085	0.13	6.18	102.0	NA	NA	NA	6.7	4921	1170	NA
03/29/80	4.90	2.3	1050	440	94.2	35.9	48.5	2.3	1.880	7.360	0.445	0.220	0.05	1.23	18.0	NA	NA	NA	2.1	1363	390	13.0
04/01/80	5.00	3.4	800	360	74.0	26.4	27.6	1.4	1.180	4.630	0.300	0.150	0.02	1.00	12.0	NA	NA	NA	1.4	2536	1173	NA
04/06/80	5.50	4.6	1025	490	101.6	45.0	36.6	2.1	1.790	7.045	0.445	0.260	0.03	1.44	19.5	NA	NA	NA	NA	4543	1789	NA
04/14/80	4.50	NA	NA	875	226.8	98.2	92.8	4.0	5.290	18.000	1.290	0.750	0.04	4.11	44.0	NA	NA	NA	NA	3785	540	NA
04/18/80	4.60	1.2	NA	1100	248.0	106.0	106.0	4.4	5.670	21.300	1.490	0.820	0.13	4.71	61.0	NA	NA	NA	3.5	2044	568	NA
04/28/80	4.80	3.3	3590	1820	375.0	164.0	184.6	6.6	8.810	34.000	2.320	1.270	0.10	7.82	68.0	NA	NA	NA	NA	1817	200	11.0
05/10/80	4.70	2.2	3975	2020	420.4	186.0	210.0	8.0	9.250	38.000	2.440	1.360	0.10	8.78	75.0	NA	NA	NA	5.4	583	171	17.0
05/31/80	4.60	2.2	2400	1350	246.0	114.0	101.6	5.0	5.870	21.130	1.420	0.760	0.05	4.24	84.0	NA	NA	NA	1.3	3059	1946	NA
06/03/80	4.60	NA	3350	1750	381.0	177.0	149.0	6.6	7.620	27.700	1.890	1.010	0.05	5.99	110.0	NA	NA	NA	6.7	1018	207	NA
06/06/80	4.60	1.1	3100	1650	316.0	169.0	131.6	6.0	7.630	21.000	1.790	1.000	0.06	5.50	90.0	NA	NA	NA	NA	935	537	NA
06/13/80	4.60	1.1	3300	1650	317.0	156.0	142.8	6.6	7.940	27.270	1.810	0.950	0.10	5.41	86.0	NA	NA	NA	NA	689	464	NA
06/18/80	4.30	NA	3800	1800	359.2	184.0	171.0	7.4	7.710	27.700	1.840	0.970	0.13	5.80	110.0	NA	NA	NA	NA	348	49	NA
06/28/80	4.50	NA	3400	1650	311.0	153.0	148.0	6.4	7.350	25.000	2.100	1.040	0.10	5.55	83.0	NA	NA	NA	5.9	1321	476	NA
07/02/80	4.50	NA	2500	NA	274.7	134.7	91.5	6.0	9.500	24.500	2.000	1.020	0.07	4.57	NA	NA	NA	NA	NA	4334	1305	NA
07/07/80	4.00	NA	3800	1800	401.0	205.0	172.0	7.4	12.000	35.000	2.640	1.290	0.09	6.52	96.0	NA	NA	NA	10.0	1374	238	NA
07/17/80	4.30	NA	5000	2200	475.8	296.0	318.0	11.8	13.450	42.000	2.680	1.560	0.28	9.00	26.0	NA	NA	NA	NA	1253	458	NA
07/26/80	4.30	NA	4450	1880	421.2	219.0	244.0	9.8	11.180	32.000	2.250	1.280	0.11	7.18	126.0	NA	NA	NA	NA	1465	329	NA
08/03/80	4.20	NA	4800	1980	434.2	232.5	323.0	10.4	11.000	40.000	3.040	1.520	0.14	8.25	130.0	NA	NA	NA	6.8	958	528	NA
08/12/80	4.30	NA	3150	1550	303.0	161.0	152.0	7.4	13.000	33.000	2.460	1.190	0.14	6.17	81.0	NA	NA	NA	4.9	2271	900	NA
08/19/80	4.30	NA	3400	2150	375.6	211.6	166.6	9.0	20.000	44.000	3.510	1.740	0.17	7.90	93.0	NA	NA	NA	NA	2832	464	NA
08/22/80	NA	NA	3150	1900	280.5	244.0	126.5	3.5	19.000	38.000	2.670	1.530	0.19	6.88	82.0	NA	NA	NA	NA	1927	922	NA
08/29/80	4.30	NA	3625	2200	406.0	321.0	229.0	8.0	18.000	45.000	3.160	1.800	0.41	8.99	105.0	NA	NA	NA	8.5	1363	5237	NA
09/11/80	4.30	NA	2845	1700	316.0	231.0	172.0	7.0	16.000	35.000	2.640	1.520	0.15	6.98	78.0	NA	NA	NA	NA	950	589	13.0
09/12/80	4.40	NA	2383	1260	218.0	185.0	118.0	4.0	19.000	35.500	2.565	1.440	0.20	6.32	53.5	NA	NA	NA	4.9	5209	2795	21.0
09/18/80	4.40	NA	2625	1600	248.0	213.0	147.0	4.0	18.000	34.000	2.400	1.430	0.14	5.63	69.0	NA	NA	NA	NA	8108	2740	15.0
09/23/80	4.20	NA	3846	2120	400.0	309.0	226.0	31.0	20.900	54.600	3.150	1.990	0.24	8.82	114.0	NA	NA	NA	5.9	3592	798	14.0
10/22/80	4.10	NA	3160	2000	330.0	252.0	166.0	8.0	20.700	56.200	3.210	1.910	0.52	8.25	85.0	NA	NA	NA	4.8	2881	710	11.0
10/30/80	4.20	NA	2250	1700	314.0	158.0	84.0	5.0	17.000	40.000	2.470	1.380	0.12	5.79	64.0	NA	NA	NA	NA	1170	85	10.0
02/19/81	3.70	NA	2050	950	172.9	110.5	123.5	NA	13.300	50.000	2.540	1.270	NA	NA	36.0	NA	NA	NA	NA	1472	1423	5.0
02/21/81	4.50	NA	2550	1560	207.4	201.5	158.0	NA	20.600	82.500	4.190	2.320	NA	NA	NA	NA	NA	NA	NA	1484	325	24.0
03/28/81	4.70	0.8	1300	625	94.9	62.4	48.1	NA	14.400	41.000	2.110	1.100	NA	NA	NA	NA	NA	NA	2.3	8842	3797	NA
04/03/81	4.70	1.6	1900	1100	146.9	110.5	84.5	NA	20.300	64.000	3.300	1.830	NA	NA	NA	NA	NA	NA	NA	7692	2030	NA
04/13/81	4.50	NA	2500	1200	191.0	147.0	69.0	NA	20.200	60.000	3.480	1.870	NA	NA	NA	NA	NA	NA	NA	3376	1436	NA
04/17/81	4.70	1.6	2700	1380	148.0	156.0	58.0	NA	18.600	56.000	3.280	1.940	NA	NA	NA	NA	NA	NA	4.4	2858	1439	NA
04/23/81	4.70	1.6	3000	1415	219.0	161.0	97.0	NA	18.400	57.000	3.380	1.980	NA	NA	NA	NA	NA	NA	NA	3978	3580	NA
04/25/81	4.80	2.4	3100	1480	200.0	168.0	95.0	NA	18.400	57.000	3.420	2.090	NA	NA	NA	NA	NA	NA	NA	8260	2934	NA
04/28/81	4.90	4.0	2950	1350	205.0	179.0	99.0	NA	16.300	51.000	3.050	1.860	NA	NA	NA	NA	NA	NA	3.2	5534	2409	NA
04/29/81	4.70	NA	3515	1850	295.0	280.0	134.0	NA	17.700	43.000	3.600	2.170	NA	NA	NA	NA	NA	NA	NA	5307	1392	12.0
05/12/81	4.50	NA	3708	2250	358.0	370.0	184.0	NA	19.940	73.000	4.600	2.670	NA	NA	NA	NA	NA	NA	5.3	3441	272	19.0
06/01/81	4.50	NA	2660	1350	244.0	180.0	110.0	NA	16.620	38.000	2.870	1.790	NA	NA	NA	NA	NA	NA	NA	2960	1196	15.0
06/06/81	4.45	NA	1872	1160	188.0	112.0	52.0	NA	13.340	26.000	2.270	1.330	NA	NA	NA	NA	NA	NA	3.1	3274	289	18.0
06/22/81	4.20	NA	2650	1660	271.0	175.0	93.0	NA	17.920	43.000	3.360	1.730	NA	NA	NA	NA	NA	NA	NA	4766	2996	17.0
06/24/81	4.40	NA	2342	1620	246.0	154.0	70.0	NA	22.240	68.000	3.680	1.950	NA	NA	NA	NA	NA	NA	NA	4993	2335	16.0
06/28/81	4.50	NA	1657	1910	302.0	290.0	112.0	NA	20.570	42.000	3.550	1.760	NA	NA	NA	NA	NA	NA	3.4	6878	3559	13.0
07/04/81	4.40	NA	3568	2300	370.0	183.3	166.2	NA	29.500	68.000	3.820	2.530	NA	NA	NA	NA	NA	NA	NA	6197	295	22.0
07/15/81	4.25	NA	3526	2250	362.0	278.6	175.0	NA	29.000	72.000	4.100	2.660	NA	NA	NA	NA	NA	NA	NA	1628	102	19.0
07/24/81	4.40	NA	2830	1830	290.0	212.6	108.0	NA	26.500	56.000	3.310	2.050	NA	NA	49.0	NA	NA	NA	3.5	1938	437	19.0
08/04/81	4.00	NA	2951	2030	326.0	142.0	108.0	NA	27.500	71.500	3.920	1.980	NA	NA	NA	NA	NA	NA	NA	1783	106	18.0
08/17/81	4.05	NA	3238	2150	348.0	260.0	108.0	8.0	27.500	65.000	3.670	2.230	NA	NA	NA	NA	NA	NA	NA	1033	47	22.0
08/31/81	4.10	NA	3930	2560	360.0	270.0	128.0	NA	34.000	95.000	5.170	3.020	NA	NA	NA	NA	NA	NA	5.1	1136	1000	NA

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/07/81	4.10	NA	4200	2220	342.0	244.0	104.0	NA	34.000	84.000	4.420	2.660	NA	NA	NA	NA	NA	NA	NA	1109	588	NA
09/16/81	4.10	NA	4300	2640	378.0	284.0	114.0	NA	33.000	92.000	4.890	2.920	NA	NA	NA	NA	NA	NA	NA	431	20	NA
09/26/81	4.10	NA	3550	2220	328.0	240.0	74.0	NA	38.000	82.000	4.360	2.570	NA	NA	NA	NA	NA	NA	3.7	1949	1176	NA
10/01/81	4.20	NA	3225	1840	298.0	224.0	56.0	NA	42.000	80.000	4.460	2.280	NA	NA	NA	NA	NA	NA	NA	4675	2859	NA
10/04/81	4.20	NA	3700	2600	332.0	354.0	76.0	10.0	52.000	107.000	8.000	3.760	NA	NA	NA	NA	NA	NA	NA	3607	1859	NA
10/14/81	4.20	NA	3400	1960	276.0	222.0	90.0	8.0	39.000	78.000	5.000	3.080	NA	NA	NA	NA	NA	NA	3.5	5216	1771	NA
10/15/81	4.20	NA	4000	2760	252.0	304.0	100.0	22.0	123.000	306.000	21.000	10.000	NA	NA	NA	NA	NA	NA	NA	9906	2503	NA
11/01/81	4.10	NA	4000	2760	324.0	350.0	118.0	8.0	54.000	120.000	6.700	3.360	NA	NA	NA	NA	NA	NA	4.1	3516	193	NA
03/30/82	4.20	NA	1825	1000	138.0	102.0	34.0	4.0	19.000	49.000	2.800	1.500	NA	NA	NA	NA	NA	NA	NA	2336	NA	NA
04/11/82	4.30	NA	2350	1500	182.0	158.0	48.0	4.0	29.000	75.000	4.400	2.340	NA	NA	NA	NA	NA	NA	NA	1669	NA	NA
04/13/82	4.40	NA	1050	570	78.8	65.7	16.0	2.0	10.000	24.000	1.400	0.700	NA	NA	NA	NA	NA	NA	NA	4906	NA	NA
04/15/82	4.40	NA	1500	900	117.6	96.4	24.0	2.0	13.000	37.000	2.300	1.050	NA	NA	NA	NA	NA	NA	NA	4179	NA	NA
04/19/82	4.30	NA	1775	1100	176.0	144.0	34.0	4.0	22.000	52.000	3.000	1.490	NA	NA	NA	NA	NA	NA	NA	2245	NA	NA
04/23/82	4.20	NA	2425	1600	206.0	176.0	58.0	4.0	34.000	83.000	4.700	2.430	NA	NA	NA	NA	NA	NA	NA	2392	NA	NA
05/03/82	4.15	NA	3100	2300	204.0	190.0	78.0	6.0	55.000	128.000	8.100	4.200	NA	NA	NA	NA	NA	NA	NA	572	NA	NA
05/04/82	4.20	NA	2900	2000	192.0	208.0	62.0	6.0	46.000	110.000	6.100	3.380	NA	NA	NA	NA	NA	NA	NA	4902	NA	NA
05/10/82	4.30	NA	2875	1900	184.0	208.0	56.0	6.0	46.000	104.000	5.800	3.220	NA	NA	NA	NA	NA	NA	NA	4664	NA	NA
05/11/82	4.30	NA	3100	1700	202.0	223.0	84.0	6.0	49.500	123.500	6.900	3.890	NA	NA	NA	NA	NA	NA	NA	5337	NA	NA
05/14/82	4.30	NA	3400	2200	250.0	288.0	84.0	6.8	42.000	114.000	5.200	3.150	0.06	7.82	NA	NA	NA	NA	NA	5773	NA	NA
05/21/82	4.20	NA	5000	3600	340.0	382.0	95.0	8.0	57.000	169.000	7.600	4.360	0.13	11.44	NA	NA	NA	NA	NA	2052	NA	NA
06/02/82	4.10	NA	3700	2700	250.0	280.0	100.0	6.6	39.000	121.000	5.700	3.720	0.14	7.77	NA	NA	NA	NA	NA	1794	NA	NA
06/18/82	4.20	NA	3400	2400	264.0	264.0	79.0	6.8	40.000	106.000	5.100	3.300	0.11	6.89	NA	NA	NA	NA	NA	1185	NA	NA
07/02/82	NA	NA	NA	1965	332.0	208.0	78.0	8.0	47.000	111.000	5.300	2.560	0.10	7.44	NA	NA	NA	NA	NA	31423	NA	NA
07/26/82	3.94	NA	3531	2150	318.0	258.0	84.0	8.0	42.000	163.000	7.400	3.150	1.28	9.10	NA	NA	NA	NA	NA	14021	NA	NA
08/30/82	3.97	NA	3319	1920	366.0	234.0	76.0	8.0	58.000	125.000	5.900	2.790	0.30	8.03	NA	NA	NA	NA	NA	16239	NA	NA
10/04/82	4.27	NA	3073	1900	250.0	260.0	70.0	8.0	53.000	168.000	7.500	3.300	0.21	9.17	NA	NA	NA	NA	NA	20358	NA	NA
11/01/82	3.90	NA	2980	NA	176.0	222.0	62.0	6.0	41.000	141.000	6.300	2.800	0.25	7.57	NA	NA	NA	NA	NA	2491	NA	NA
11/22/82	4.10	NA	4100	3300	236.0	326.0	136.0	NA	41.600	213.000	8.900	3.820	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
03/18/83	4.35	NA	1425	780	NA	NA	NA	NA	24.000	37.000	18.000	1.380	NA	NA	NA	NA	NA	NA	NA	3286	NA	NA
04/14/83	4.33	NA	1387	700	NA	NA	NA	NA	22.000	40.000	2.450	1.400	NA	NA	NA	NA	NA	NA	NA	12694	NA	NA
05/15/83	4.21	NA	2532	2500	NA	NA	NA	NA	56.000	1.100	6.400	3.490	NA	NA	NA	NA	NA	NA	NA	6279	NA	NA
06/15/83	4.25	NA	2173	1940	NA	NA	NA	NA	40.000	76.000	3.400	1.880	NA	NA	NA	NA	NA	NA	NA	5022	NA	NA
07/14/83	4.14	NA	3960	5160	NA	NA	NA	NA	87.000	174.000	10.190	5.740	NA	NA	NA	NA	NA	NA	NA	9466	NA	NA
08/14/83	4.08	NA	2861	2300	NA	NA	NA	NA	68.000	120.000	6.450	3.720	NA	NA	NA	NA	NA	NA	NA	14103	NA	NA
09/15/83	4.16	NA	3486	2950	NA	NA	NA	NA	85.000	164.000	9.000	5.060	NA	NA	NA	NA	NA	NA	NA	8668	NA	NA
10/13/83	4.32	NA	3405	2400	NA	NA	NA	NA	77.000	136.000	8.000	4.340	NA	NA	NA	NA	NA	NA	NA	19970	NA	NA
11/09/83	4.10	NA	2750	2250	NA	NA	NA	NA	82.000	133.000	7.400	4.170	NA	NA	NA	NA	NA	NA	NA	4269	NA	NA
11/30/83	4.00	NA	2700	2820	NA	NA	NA	NA	73.400	125.000	6.950	3.970	NA	NA	NA	NA	NA	NA	NA	628	NA	NA
02/24/84	4.00	NA	2175	2300	276.8	414.3	56.6	NA	49.000	106.000	5.110	3.420	NA	NA	NA	NA	NA	NA	NA	75	NA	NA
03/15/84	4.25	NA	1100	830	110.4	105.7	17.3	NA	26.000	41.400	2.050	1.250	NA	NA	NA	NA	NA	NA	NA	4023	NA	NA
04/15/84	4.33	NA	1472	1540	188.2	170.3	27.2	NA	46.000	67.000	3.750	2.200	NA	NA	NA	NA	NA	NA	NA	20316	NA	NA
05/16/84	4.22	NA	1958	2500	236.3	279.5	42.0	NA	65.000	110.000	7.000	3.360	NA	NA	NA	NA	NA	NA	NA	11761	NA	NA
06/13/84	4.16	NA	2320	2300	256.1	328.6	52.1	NA	68.000	119.600	6.400	3.540	NA	NA	NA	NA	NA	NA	NA	23280	NA	NA
07/14/84	4.19	NA	2930	2700	NA	NA	NA	NA	95.600	186.000	6.600	4.000	NA	NA	NA	NA	NA	NA	NA	8771	NA	NA
08/15/84	4.10	NA	3385	3020	379.7	277.6	71.8	NA	90.100	160.420	11.600	5.080	NA	NA	NA	NA	NA	NA	NA	7737	NA	NA
09/12/84	3.96	NA	4095	3900	408.2	510.0	88.2	NA	107.500	225.000	15.700	6.630	NA	NA	NA	NA	NA	NA	NA	2612	NA	NA
10/13/84	3.94	NA	2720	3459	402.6	430.0	65.3	NA	109.500	169.000	12.700	5.240	NA	NA	NA	NA	NA	NA	NA	15261	NA	NA
11/14/84	4.11	NA	3160	4125	430.0	585.0	93.8	NA	135.000	255.000	12.400	6.800	NA	NA	NA	NA	NA	NA	NA	2615	NA	NA
12/01/84	3.95	NA	2150	3000	332.0	426.0	61.0	NA	90.800	152.000	7.900	4.160	NA	NA	NA	NA	NA	NA	NA	492	NA	NA
12/18/84	4.30	NA	2025	2200	252.0	302.0	52.0	NA	70.300	108.800	5.410	3.060	NA	NA	NA	NA	NA	NA	NA	2215	NA	NA
04/02/85	4.50	NA	1625	1130	158.0	138.8	21.0	NA	32.000	63.000	2.840	1.820	NA	NA	NA	NA	NA	NA	NA	2551	NA	NA

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/16/85	4.40	NA	2200	1500	238.0	256.0	37.6	NA	63.000	98.000	4.820	2.870	NA	NA	NA	NA	NA	NA	NA	16537	NA	NA
05/14/85	4.34	NA	3100	2750	240.0	364.0	48.5	NA	83.000	128.000	7.200	3.800	NA	NA	NA	NA	NA	NA	NA	9852	NA	NA
06/14/85	4.29	NA	3500	3200	281.0	453.0	63.0	NA	91.000	160.000	7.800	4.460	NA	NA	NA	NA	NA	NA	NA	34814	NA	NA
07/15/85	4.16	NA	3100	2773	276.0	377.0	55.0	NA	78.300	130.600	6.500	4.690	NA	NA	NA	NA	NA	NA	NA	14584	NA	NA
08/13/85	4.11	NA	2700	2400	268.0	252.0	46.0	NA	76.400	94.000	4.880	2.870	NA	NA	NA	NA	NA	NA	NA	15114	NA	NA
09/13/85	4.15	NA	3100	2550	309.0	270.0	48.0	NA	75.000	104.200	5.440	3.330	NA	NA	NA	NA	NA	NA	NA	31741	NA	NA
10/15/85	4.20	NA	3300	2430	244.0	319.0	43.1	NA	77.100	106.500	5.470	2.510	NA	NA	NA	NA	NA	NA	NA	20420	NA	NA
11/15/85	4.27	NA	3250	1750	284.0	440.0	56.0	NA	88.800	144.600	7.340	3.800	NA	NA	NA	NA	NA	NA	NA	2646	NA	NA
03/23/86	4.50	NA	1500	900	114.0	122.0	15.8	NA	34.000	55.000	2.910	1.580	NA	NA	NA	NA	NA	NA	NA	17289	NA	NA
04/16/86	4.36	NA	2025	1400	204.0	190.0	26.2	NA	59.000	86.000	4.840	2.280	NA	NA	NA	NA	NA	NA	NA	16332	NA	NA
05/14/86	4.35	NA	2325	1950	276.0	262.0	28.0	NA	72.000	133.000	8.000	2.920	NA	NA	NA	NA	NA	NA	NA	12740	NA	NA
06/11/86	4.30	NA	2750	2000	236.0	250.0	38.0	NA	63.000	100.000	4.800	3.100	NA	NA	NA	NA	NA	NA	NA	7631	NA	NA
07/12/86	4.27	NA	2650	1750	260.0	264.0	36.0	NA	65.000	100.000	4.420	2.850	NA	NA	NA	NA	NA	NA	NA	2365	NA	NA
08/13/86	4.13	NA	3200	2350	300.0	301.0	45.0	NA	80.000	99.000	6.200	2.550	NA	NA	NA	NA	NA	NA	NA	13346	NA	NA
09/14/86	4.12	NA	3000	1800	280.0	302.0	38.0	NA	76.000	94.000	6.600	2.400	NA	NA	NA	NA	NA	NA	NA	27335	NA	NA
10/16/86	4.06	NA	3425	2400	288.0	318.0	44.0	NA	93.000	122.000	6.000	3.510	NA	NA	NA	NA	NA	NA	NA	6442	NA	NA
11/16/86	3.99	NA	3475	3250	332.0	544.0	64.0	NA	156.000	169.000	8.800	5.300	NA	NA	NA	NA	NA	NA	NA	10954	NA	NA
03/28/87	4.40	NA	2300	1750	202.0	234.0	30.0	NA	83.000	110.000	5.390	3.140	NA	NA	NA	NA	NA	NA	NA	12608	NA	NA
04/18/87	4.31	NA	2150	1650	198.0	206.0	26.0	NA	68.000	95.000	4.520	2.580	NA	NA	NA	NA	NA	NA	NA	2525	NA	NA
05/16/87	3.86	NA	2750	2700	335.0	397.0	44.0	NA	97.000	115.000	5.600	3.400	NA	NA	NA	NA	NA	NA	NA	28020	NA	NA
06/16/87	4.17	NA	3950	3000	348.0	458.0	58.0	NA	106.000	163.000	5.800	3.420	NA	NA	NA	NA	NA	NA	NA	1419	NA	NA
07/15/87	4.00	NA	3325	2715	294.0	340.0	46.0	NA	92.000	117.000	5.200	3.430	NA	NA	NA	NA	NA	NA	NA	29811	NA	NA
08/15/87	3.72	NA	3050	2500	288.0	304.0	42.0	NA	93.000	112.000	5.000	3.130	NA	NA	NA	NA	NA	NA	NA	13350	NA	NA
09/16/87	3.84	NA	4400	2850	323.0	352.0	47.0	NA	101.000	113.000	5.090	3.050	NA	NA	NA	NA	NA	NA	NA	16885	NA	NA
10/13/87	3.92	NA	5000	4300	386.0	560.0	72.0	NA	138.000	173.000	7.560	4.550	NA	NA	NA	NA	NA	NA	NA	1915	NA	NA
11/10/87	4.03	NA	4375	3760	342.0	500.0	64.0	NA	128.000	165.000	7.330	4.510	NA	NA	NA	NA	NA	NA	NA	1601	NA	NA
03/28/88	4.25	NA	1500	880	142.0	106.0	14.0	NA	34.000	52.000	2.500	2.100	NA	NA	NA	NA	NA	NA	NA	7574	NA	NA
04/17/88	4.20	NA	2125	1860	170.0	186.0	18.0	NA	52.000	81.000	3.700	3.100	NA	NA	NA	NA	NA	NA	NA	6635	NA	NA
05/12/88	4.11	NA	3275	2300	298.0	322.0	50.0	NA	113.000	153.000	6.800	6.500	NA	NA	NA	NA	NA	NA	NA	12044	NA	NA
06/12/88	4.10	NA	3325	2600	346.0	342.0	40.0	NA	118.000	146.000	7.500	7.100	NA	NA	NA	NA	NA	NA	NA	2858	NA	NA
07/15/88	4.10	NA	3250	2440	312.0	314.0	36.0	NA	105.000	132.000	6.500	6.500	NA	NA	NA	NA	NA	NA	NA	4553	NA	NA
08/16/88	4.06	NA	3475	2400	302.0	352.0	52.0	NA	85.000	133.000	6.300	5.100	NA	NA	NA	NA	NA	NA	NA	60076	NA	NA
09/16/88	4.04	NA	3050	2000	246.0	290.0	44.0	NA	76.000	80.000	5.500	4.400	NA	NA	NA	NA	NA	NA	NA	12025	NA	NA
10/18/88	4.12	NA	2950	2160	212.0	274.0	30.0	NA	75.000	88.000	5.100	4.700	NA	NA	NA	NA	NA	NA	NA	3664	NA	NA
11/15/88	4.30	NA	2625	2080	210.0	265.0	26.0	NA	71.000	81.500	4.550	4.400	NA	NA	NA	NA	NA	NA	NA	7017	NA	NA
11/30/88	4.20	NA	3600	2500	232.0	320.0	32.0	NA	78.000	98.000	5.100	4.800	NA	NA	NA	NA	NA	NA	NA	681	NA	NA
04/07/89	4.65	1.1	900	1400	80.0	72.0	10.0	NA	18.600	23.000	1.400	1.100	NA	NA	NA	NA	NA	NA	NA	18641	NA	NA
04/24/89	4.40	NA	1975	1340	150.0	156.0	18.0	NA	45.000	57.000	3.400	3.000	NA	NA	NA	NA	NA	NA	NA	8804	NA	NA
05/16/89	4.27	NA	2550	1700	202.0	216.0	28.0	NA	66.000	79.000	4.700	4.300	NA	NA	NA	NA	NA	NA	NA	11620	NA	NA
06/17/89	4.28	NA	3500	2950	304.0	354.0	54.0	NA	91.000	159.000	10.200	6.300	NA	NA	NA	NA	NA	NA	NA	35500	NA	NA
07/17/89	4.13	NA	4625	3800	440.0	460.0	58.0	NA	131.000	204.000	11.300	9.700	NA	NA	NA	NA	NA	NA	NA	2952	NA	NA
08/14/89	4.17	NA	2950	2060	232.0	282.0	34.0	NA	82.000	138.000	5.700	5.500	NA	NA	NA	NA	NA	NA	NA	4466	NA	NA
09/14/89	4.01	NA	4325	3500	356.0	416.0	50.0	NA	123.000	176.000	9.900	8.200	NA	NA	NA	NA	NA	NA	NA	17161	NA	NA
10/16/89	4.05	NA	4800	3600	360.0	462.0	56.0	NA	137.000	149.000	16.400	9.400	NA	NA	NA	NA	NA	NA	NA	1832	NA	NA
11/10/89	4.20	NA	3050	2000	214.0	300.0	36.0	NA	94.000	96.000	15.300	7.000	NA	NA	NA	NA	NA	NA	NA	999	NA	NA
03/26/90	4.15	NA	5000	5440	468.0	744.0	93.0	NA	202.000	253.000	16.200	14.100	NA	NA	NA	NA	NA	NA	NA	34	NA	NA
04/12/90	4.39	NA	3125	2280	244.0	288.0	26.0	NA	96.000	108.000	5.610	5.440	NA	NA	NA	NA	NA	NA	NA	3883	NA	NA
05/17/90	4.32	NA	2675	2040	226.0	230.0	22.0	NA	84.000	78.000	4.630	4.260	NA	NA	NA	NA	NA	NA	NA	927	NA	NA
06/18/90	4.07	NA	3550	2440	266.0	286.0	28.0	NA	83.000	74.000	4.200	4.380	NA	NA	NA	NA	NA	NA	NA	2998	NA	NA
07/16/90	4.10	NA	3490	2460	270.0	286.0	26.0	NA	93.000	73.000	4.410	4.290	NA	NA	NA	NA	NA	NA	NA	1658	NA	NA
08/13/90	4.05	NA	3100	1970	190.0	286.0	28.0	NA	75.000	78.000	4.900	4.130	NA	NA	NA	NA	NA	NA	NA	700	NA	NA

Table A2.5. Drainage quality data from Duluth Complex test pile FL6, from 1978-1991 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/10/90	4.10	NA	3300	2160	242.0	312.0	24.0	NA	63.000	71.000	4.120	3.150	NA	NA	NA	NA	NA	NA	NA	5583	NA	NA
10/15/90	3.98	NA	4295	3980	334.0	438.0	36.0	NA	88.000	112.000	6.410	4.330	NA	NA	NA	NA	NA	NA	NA	20246	NA	NA
11/16/90	4.35	NA	5000	4400	410.0	640.0	58.0	NA	144.000	165.000	9.990	9.180	NA	NA	NA	NA	NA	NA	NA	167	NA	NA
04/08/91	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	10795	NA	NA
04/22/91	4.22	NA	3625	2620	238.0	350.0	30.0	NA	106.000	116.000	5.200	3.300	NA	NA	NA	NA	NA	NA	NA	7960	NA	NA
05/20/91	4.18	NA	4675	3260	284.0	452.0	40.0	NA	133.000	163.000	6.900	4.400	NA	NA	NA	NA	NA	NA	NA	9122	NA	NA
06/17/91	4.18	NA	3350	2080	208.0	317.0	28.0	NA	91.000	103.000	5.000	2.800	NA	NA	NA	NA	NA	NA	NA	10000	NA	NA
07/15/91	4.07	NA	4800	2500	246.0	378.0	34.0	NA	107.000	117.000	5.900	4.600	NA	NA	NA	NA	NA	NA	NA	29610	NA	NA
08/12/91	4.10	NA	5000	3280	286.0	430.0	56.0	NA	137.000	150.000	7.600	6.300	NA	NA	NA	NA	NA	NA	NA	1185	NA	NA
09/09/91	4.09	NA	5000	3260	292.0	404.0	48.0	NA	119.000	130.000	6.600	4.900	NA	NA	NA	NA	NA	NA	NA	20079	NA	NA
10/07/91	4.17	NA	4450	2640	236.0	346.0	42.0	NA	105.000	114.000	5.600	4.600	NA	NA	NA	NA	NA	NA	NA	6344	NA	NA
11/07/91	4.11	NA	4775	2360	212.0	304.0	34.0	NA	96.000	98.000	5.000	4.000	NA	NA	NA	NA	NA	NA	NA	11692	NA	NA

NA: Not analyzed.
'Dates represent intermediate dates for measurement periods.

Table A2.5.1. Drainage quality data from Duluth Complex test pile FL6, 1992-1993 (0.79% S, 0.34% Cu, 0.08% Ni).

Date'	pH	Alk. mg/L	Lab Acy. mg/L	Calc. Acy. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/23/92	4.29	NA	140	122.5	1475	650	68.0	92.0	11.60	3.20	4	28.00	28	1.40	0.90	0.06	2.53	NA	NA	NA	NA	NA	5901	NA	NA
05/11/92	4.21	NA	325	315.4	3190	1760	186.0	234.0	28.40	5.30	10	74.00	71	3.60	2.40	0.11	6.77	NA	NA	NA	NA	NA	2529	NA	NA
05/26/92	4.32	NA	265	261.6	2450	1640	140.0	224.0	25.80	5.90	8	58.00	62	3.00	2.20	0.10	6.35	NA	NA	NA	NA	NA	3203	NA	NA
06/08/92	4.30	NA	785	644.7	5000	4060	300.0	600.0	58.00	10.30	16	128.00	178	7.80	5.70	0.24	15.70	NA	NA	NA	NA	NA	822	NA	NA
06/22/92	4.32	NA	290	270.2	2710	1740	134.0	248.0	28.00	6.80	7	61.00	66	3.70	2.30	0.12	7.02	NA	NA	NA	NA	NA	872	NA	NA
07/07/92	4.25	NA	260	259.4	2510	790	146.0	208.0	25.60	6.50	7	65.00	58	3.20	2.10	0.11	5.76	NA	NA	NA	NA	NA	3764	NA	NA
07/20/92	4.18	NA	560	491.7	4480	696	238.0	418.0	47.00	10.10	11	107.00	130	6.70	4.30	0.14	12.30	NA	NA	NA	NA	NA	894	NA	NA
08/03/92	4.21	NA	280	292.5	2640	1780	146.0	258.0	31.00	7.40	8	68.00	69	3.90	2.00	0.11	7.46	NA	NA	NA	NA	NA	1159	NA	NA
08/17/92	4.23	NA	370	354.3	3400	2360	210.0	324.0	33.60	8.70	9	84.00	84	4.70	3.10	0.18	8.69	NA	NA	NA	NA	NA	1565	NA	NA
08/31/92	4.24	NA	250	242.1	2500	1460	146.0	210.0	25.00	7.30	6	58.00	58	3.00	2.00	0.11	5.62	NA	NA	NA	NA	NA	1482	NA	NA
09/14/92	4.20	NA	415	340.0	3200	2180	202.0	312.0	34.00	8.90	8	86.00	78	4.50	3.00	0.11	8.13	NA	NA	NA	NA	NA	5218	NA	NA
09/28/92	4.21	NA	485	448.5	4850	3120	254.0	416.0	46.60	10.80	10	107.00	110	6.00	4.00	0.11	11.30	NA	NA	NA	NA	NA	2694	NA	NA
10/12/92	4.23	NA	525	536.1	4900	3280	272.0	460.0	51.60	11.70	11	127.00	137	6.70	4.50	0.14	12.70	NA	NA	NA	NA	NA	444	NA	NA
10/27/92	4.18	NA	425	395.4	3830	2440	204.0	366.0	41.00	10.10	9	96.00	95	5.30	3.30	0.10	9.98	NA	NA	NA	NA	NA	379	NA	NA
11/09/92	4.25	NA	NA	NA	4500	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	119	NA	NA
11/30/92	4.30	NA	300	244.5	2300	1570	142.0	230.0	24.40	7.20	6	60.00	57	3.10	2.00	0.07	6.12	NA	NA	NA	NA	NA	805	NA	NA
04/08/93	4.40	NA	NA	NA	1350	770	72.5	110.0	10.00	2.75	6	33.00	34	1.48	1.08	0.07	3.08	NA	NA	NA	NA	NA	7949	NA	NA
04/22/93	4.40	NA	NA	NA	1440	825	77.5	115.0	10.00	3.00	5	27.00	30	1.66	0.88	0.05	2.68	NA	NA	NA	NA	NA	5257	NA	NA
05/06/93	4.23	NA	NA	NA	2750	1950	197.5	247.5	21.25	5.00	13	71.00	71	3.85	1.99	0.14	5.80	NA	NA	NA	NA	NA	12820	NA	NA
05/20/93	4.31	NA	NA	NA	3200	2340	217.5	310.0	26.25	4.75	14	96.00	90	4.60	2.86	0.21	7.28	NA	NA	NA	NA	NA	2086	NA	NA
06/03/93	4.37	NA	NA	NA	1600	1255	120.0	172.5	16.00	3.50	7	54.00	51	2.61	1.52	0.10	4.01	NA	NA	NA	NA	NA	6945	NA	NA
06/17/93	4.48	NA	NA	NA	2350	1670	167.5	270.0	22.00	4.75	9	87.00	79	4.15	2.52	0.11	6.32	NA	NA	NA	NA	NA	7559	NA	NA
07/01/93	4.13	NA	NA	NA	4200	3180	270.0	490.0	39.00	7.75	18	139.00	147	7.40	4.52	0.33	13.60	NA	NA	NA	NA	NA	8928	NA	NA
07/14/93	4.22	NA	NA	NA	4220	3020	266.0	422.0	37.20	9.20	14	100.00	125	6.03	3.75	0.15	13.00	NA	NA	NA	NA	NA	15912	NA	NA
07/29/93	4.23	NA	NA	NA	1700	1130	118.0	138.0	15.60	5.40	7	49.00	44	2.20	1.40	0.10	3.82	NA	NA	NA	NA	NA	13240	NA	NA
08/12/93	4.15	NA	NA	NA	2425	1680	166.0	188.0	19.80	6.00	8	66.00	62	2.68	1.99	0.07	5.30	NA	NA	NA	NA	NA	7184	NA	NA
08/26/93	4.13	NA	NA	NA	2600	1520	158.0	240.0	23.00	6.80	8	66.00	70	3.14	2.28	0.10	6.66	NA	NA	NA	NA	NA	11336	NA	NA
09/09/93	4.30	NA	NA	NA	2700	1670	174.0	232.0	22.20	7.00	8	66.00	68	3.45	2.68	0.07	6.82	NA	NA	NA	NA	NA	11866	NA	NA
09/23/93	4.21	NA	NA	NA	2700	1720	174.0	250.0	23.40	6.80	9	64.00	74	3.42	2.37	0.10	6.97	NA	NA	NA	NA	NA	3955	NA	NA
10/07/93	4.28	NA	NA	NA	3450	2180	164.4	286.0	27.00	7.40	12	80.00	86	4.00	2.50	0.09	7.42	NA	NA	NA	NA	NA	2139	NA	NA
10/20/93	4.22	NA	NA	NA	3925	2520	252.0	366.0	33.80	8.40	15	105.00	113	5.06	3.34	0.09	9.73	NA	NA	NA	NA	NA	814	NA	NA
11/04/93	4.25	NA	NA	NA	3300	970	191.6	278.0	26.80	7.00	10	80.00	77	4.09	2.47	0.07	7.09	NA	NA	NA	NA	NA	2139	NA	NA
11/22/93	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	288	NA	NA

NA: Not analyzed.
'Dates represent intermediate dates for measurement periods.

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/26/78	7.20	NA	1750	NA	190.0	11.0	153.0	6.0	0.020	0.445	0.045	NA	NA	0.27	NA	NA	NA	NA	NA	NA	NA	6.0
05/25/78	7.60	NA	2500	NA	279.0	21.0	273.0	10.0	0.021	0.780	0.136	0.096	0.02	0.89	NA	NA	NA	NA	NA	NA	NA	NA
06/16/78	7.50	NA	3300	NA	366.0	26.0	336.0	15.0	0.044	0.835	0.160	0.174	0.03	1.40	NA	NA	NA	NA	NA	192	NA	13.0
07/05/78	7.05	NA	2950	1800	325.0	29.0	231.0	13.0	0.140	5.100	0.195	0.257	0.02	1.29	78.0	NA	NA	NA	NA	370	NA	13.0
07/13/78	7.22	NA	NA	1300	357.0	27.0	248.0	14.0	0.090	6.700	0.810	0.230	0.04	1.67	50.0	39.0	13.00	0.08	NA	NA	NA	NA
07/19/78	7.40	NA	2050	1300	580.0	29.0	245.0	16.0	0.141	21.500	2.160	0.396	0.02	2.40	59.0	54.0	12.00	0.08	NA	NA	NA	NA
08/02/78	7.00	NA	1725	775	276.0	20.0	144.5	13.0	0.265	10.280	1.240	0.400	0.05	2.08	25.0	66.5	5.70	0.24	NA	NA	NA	21.0
08/04/78	6.30	NA	2350	1400	483.0	32.0	238.0	17.0	0.420	3.230	1.860	0.774	0.05	3.32	44.0	83.0	7.50	0.08	NA	333	NA	22.0
08/10/78	5.50	7.0	4000	2000	636.0	47.0	338.0	20.0	1.280	3.310	2.200	1.220	0.12	4.29	52.0	NA	8.50	0.08	NA	147	NA	22.0
08/16/78	5.90	8.0	1492	850	245.3	26.0	104.7	9.7	1.157	22.453	3.790	0.623	7.96	3.06	19.5	41.0	4.00	0.08	NA	5948	NA	17.7
08/17/78	5.25	5.0	2100	800	400.0	40.0	178.0	33.0	2.530	40.380	6.730	1.240	11.54	5.14	27.0	55.0	NA	0.08	NA	1714	NA	NA
08/18/78	4.85	4.0	3100	800	350.0	44.0	354.0	8.0	2.980	42.210	7.220	1.450	5.28	5.87	34.0	67.0	NA	0.08	NA	498	NA	16.0
08/23/78	4.23	NA	1775	1050	203.5	24.5	107.0	10.0	1.450	20.240	3.400	0.625	4.01	2.99	16.5	34.5	4.00	0.08	NA	NA	NA	NA
08/25/78	3.90	NA	2800	1200	344.0	42.0	177.0	15.0	3.150	39.400	6.360	1.180	2.78	5.40	22.0	54.0	8.30	0.08	NA	NA	NA	21.0
08/28/78	4.25	NA	3250	1500	395.0	50.0	220.0	18.0	3.950	48.000	7.760	1.490	2.38	6.47	30.0	64.0	NA	0.08	NA	NA	NA	NA
09/11/78	4.80	NA	1475	700	140.0	20.0	79.0	9.0	1.810	18.000	1.510	0.560	0.75	2.90	8.0	NA	NA	0.20	NA	8801	NA	17.0
09/12/78	3.50	NA	2650	1550	260.0	50.0	133.0	17.0	7.280	67.000	4.880	1.750	24.00	10.00	9.0	NA	NA	0.08	NA	NA	NA	NA
09/13/78	3.62	NA	3250	1900	312.0	65.0	183.0	21.0	10.300	83.000	6.170	2.460	23.00	13.00	15.0	46.0	0.10	NA	NA	721	NA	13.0
10/17/78	4.10	NA	3300	NA	335.0	98.0	211.0	15.0	11.450	79.000	5.730	3.290	0.63	10.00	NA	NA	NA	NA	NA	NA	NA	NA
03/23/79	3.70	NA	2445	2100	271.6	108.0	121.8	14.8	44.000	169.000	14.000	5.000	4.01	11.80	11.0	NA	NA	NA	7.6	2801	283	NA
04/03/79	3.60	NA	4435	3000	395.0	231.0	286.0	28.0	91.000	335.000	25.000	11.000	11.00	28.00	18.0	NA	NA	NA	NA	2574	167	NA
04/11/79	3.60	NA	4766	4600	430.0	280.0	338.0	30.0	107.000	406.000	31.000	13.000	10.00	35.00	22.0	NA	NA	NA	NA	1983	129	NA
04/16/79	3.50	NA	2822	2350	269.0	124.0	146.8	15.2	38.000	173.000	14.000	5.000	8.73	15.70	14.0	NA	NA	NA	NA	4148	3192	NA
04/17/79	3.30	NA	1769	1050	197.0	70.6	70.0	8.2	15.000	87.000	6.000	2.080	7.70	8.50	8.0	9.8	NA	NA	NA	5098	3235	NA
04/18/79	3.40	NA	1769	1100	213.0	62.4	68.0	7.2	13.000	74.000	5.190	1.900	13.33	7.22	5.0	8.1	NA	NA	NA	6906	4242	NA
04/19/79	3.40	NA	1475	833	193.0	42.2	52.0	6.4	8.980	48.000	3.690	1.360	11.62	4.89	2.0	5.5	0.40	NA	NA	1843	2946	NA
04/20/79	3.50	NA	1380	867	178.0	49.0	65.0	6.6	14.000	53.000	4.300	1.890	3.78	5.35	4.0	8.5	0.64	NA	3.3	7210	2394	NA
04/30/79	3.50	NA	2562	2100	294.8	100.0	185.6	12.2	40.000	107.400	8.400	3.950	4.54	10.72	13.0	26.0	1.10	NA	NA	6431	364	NA
05/11/79	3.50	NA	2341	NA	254.0	89.0	133.0	9.0	33.000	94.000	7.400	3.600	1.80	9.60	NA	NA	NA	NA	NA	409	2115	NA
05/22/79	3.80	NA	1767	1150	201.0	64.0	79.0	7.0	23.000	71.000	4.860	2.370	3.61	6.65	9.0	NA	NA	NA	6.9	1813	1031	NA
05/23/79	3.20	NA	2713	2100	288.2	105.8	162.8	14.0	41.600	116.000	8.600	4.160	4.03	10.09	12.0	NA	NA	NA	5.4	3157	982	NA
05/27/79	3.38	NA	2578	2200	285.4	57.7	151.2	13.1	35.900	104.000	7.800	3.680	4.64	9.42	12.0	NA	NA	NA	3.7	3073	508	NA
06/01/79	3.60	NA	2227	1800	239.2	54.6	121.4	11.6	30.800	84.600	5.570	3.000	3.00	7.75	10.0	NA	NA	NA	3.9	2290	561	NA
06/06/79	3.90	NA	2154	1430	243.8	77.2	60.8	9.2	26.000	80.000	5.180	2.830	3.17	7.49	12.0	NA	NA	NA	NA	2358	788	NA
06/10/79	3.40	NA	2538	1750	287.6	93.0	68.2	11.0	30.000	94.000	6.260	3.310	7.32	8.70	13.0	NA	NA	NA	NA	3407	1308	NA
06/16/79	3.60	NA	2890	2100	324.0	110.0	115.0	14.8	40.000	115.000	7.570	4.130	5.41	10.10	11.0	NA	NA	NA	NA	935	244	23.0
06/20/79	3.50	NA	1891	1150	203.0	68.0	71.0	11.0	14.000	59.000	4.320	2.220	3.06	0.11	7.0	NA	NA	NA	NA	1798	1300	19.0
06/30/79	3.30	NA	2754	2225	328.0	117.0	132.0	16.5	25.500	96.500	7.205	3.890	6.78	9.38	19.0	NA	NA	NA	4.8	1904	296	14.0
07/10/79	3.30	NA	3830	3200	462.0	180.0	204.0	29.0	50.000	163.000	14.000	5.950	4.83	14.10	28.0	NA	NA	NA	5.4	1348	110	19.0
07/30/79	3.50	NA	1926	1400	229.0	82.0	63.0	12.0	7.000	46.000	4.420	2.320	2.10	6.32	6.0	NA	NA	NA	6.5	3179	3202	22.0
08/04/79	3.20	NA	3170	2800	416.0	144.0	120.0	21.0	25.000	132.000	8.190	4.470	8.49	13.32	13.0	NA	NA	NA	5.6	1442	322	23.0
08/13/79	3.40	NA	4636	4650	514.8	130.4	322.0	32.6	76.000	259.000	19.000	9.590	11.10	22.80	82.0	NA	NA	NA	8.7	712	87	14.0
08/23/79	3.50	NA	3708	3600	463.2	230.2	202.0	27.2	44.000	202.000	12.000	7.460	6.90	17.89	9.0	NA	NA	NA	8.7	855	243	12.0
08/28/79	3.40	NA	3332	3050	365.0	122.2	158.6	21.8	26.000	142.000	12.000	5.800	5.24	14.20	15.0	NA	NA	NA	8.3	424	330	20.0
09/01/79	3.20	NA	2923	1400	296.4	115.8	98.2	18.0	31.000	138.000	8.000	4.560	19.60	13.34	17.0	NA	NA	NA	6.2	5708	2338	21.0
09/05/79	3.20	NA	4197	3300	420.2	128.6	192.8	28.4	77.000	227.000	18.000	8.490	26.20	20.20	39.0	NA	NA	NA	8.6	11	523	19.0
09/13/79	3.30	NA	4120	3600	363.8	239.6	197.0	26.2	77.000	224.000	17.000	8.140	22.85	19.20	10.0	NA	NA	NA	7.7	2449	811	17.0
09/13/79	3.30	NA	3906	3580	372.8	234.0	192.6	26.6	78.000	219.000	16.000	8.290	22.00	19.00	14.0	NA	NA	NA	8.3	2615	811	17.0
09/19/79	3.30	NA	5852	5700	522.0	384.0	286.0	39.2	131.000	348.000	24.000	10.130	25.50	28.00	51.0	NA	NA	NA	11.0	1726	220	12.0
10/18/79	3.40	NA	5446	6330	518.0	398.0	272.0	36.4	91.000	347.000	25.000	10.130	15.76	28.00	9.0	NA	NA	NA	10.0	840	360	NA
10/19/79	3.60	NA	2356	1700	185.6	138.0	76.0	12.8	18.000	115.000	10.000	4.390	4.16	10.09	3.0	NA	NA	NA	5.4	1730	1591	NA

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
10/31/79	3.30	NA	3937	3310	320.0	260.0	150.0	22.0	49.000	268.000	25.100	8.110	10.60	19.45	6.0	NA	NA	NA	7.0	4046	1743	NA
11/01/79	3.40	NA	3567	3180	294.0	244.0	132.0	22.6	63.000	268.000	24.600	8.000	11.77	18.60	8.0	NA	NA	NA	6.2	4345	1671	NA
03/29/80	3.70	NA	3500	2100	222.7	190.9	90.8	15.0	41.110	206.500	15.110	6.530	2.26	15.00	7.0	NA	NA	NA	4.9	1401	994	16.0
03/31/80	3.60	NA	2450	1700	168.2	125.6	57.4	10.0	34.080	144.000	8.940	3.870	7.29	11.00	3.0	NA	NA	NA	2.5	3369	1070	NA
04/05/80	4.60	NA	2288	1055	169.4	112.6	49.0	9.0	21.855	124.000	7.725	3.300	8.13	9.25	2.5	NA	NA	NA	NA	4543	1984	NA
04/14/80	3.50	NA	NA	1220	190.4	135.6	63.6	10.4	30.600	131.000	9.000	4.140	5.90	11.00	4.0	NA	NA	NA	NA	2271	497	NA
04/18/80	3.50	NA	NA	2550	231.8	184.0	86.6	12.6	46.000	187.000	14.000	5.930	7.91	17.00	4.0	NA	NA	NA	3.5	1590	497	NA
04/27/80	3.50	NA	4825	3880	342.4	314.0	166.6	20.4	97.000	321.000	21.000	14.000	10.77	28.00	7.0	NA	NA	NA	NA	530	50	13.0
05/10/80	3.40	NA	6110	6700	376.8	442.0	210.6	26.4	133.000	423.000	30.000	18.000	15.00	37.00	10.0	NA	NA	NA	7.9	451	109	12.0
05/18/80	3.23	NA	6000	6125	386.4	485.0	223.6	29.6	140.000	464.500	35.500	20.000	12.19	40.00	13.0	NA	NA	NA	NA	640	21	19.0
05/31/80	3.40	NA	3450	3040	244.4	233.0	112.8	18.2	54.000	245.000	15.100	8.400	6.94	19.00	15.0	NA	NA	NA	4.2	791	134	NA
06/03/80	3.45	NA	4750	3960	292.2	274.0	139.8	22.0	58.000	273.000	17.600	9.760	6.08	20.00	15.0	NA	NA	NA	7.3	303	134	NA
06/07/80	3.40	NA	4550	3920	279.6	259.0	125.2	20.8	54.000	261.000	16.500	9.200	6.45	19.00	16.0	NA	NA	NA	NA	382	128	NA
06/13/80	3.30	NA	5000	4070	332.0	283.0	152.2	23.4	63.000	289.000	19.100	10.250	7.50	20.00	12.0	NA	NA	NA	NA	299	140	NA
06/19/80	3.30	NA	5000	6200	341.0	345.0	181.0	26.2	80.000	360.000	23.200	12.050	8.33	26.00	12.0	NA	NA	NA	NA	155	18	NA
06/28/80	3.30	NA	4674	4250	341.0	355.0	147.0	23.0	59.000	269.000	24.000	12.000	9.00	24.00	10.0	NA	NA	NA	9.2	507	158	NA
06/30/80	3.70	NA	5000	4600	358.0	363.0	155.0	23.0	67.000	330.000	28.000	13.000	8.00	26.00	14.0	NA	NA	NA	17.0	329	64	NA
07/02/80	3.50	NA	2600	1850	200.0	148.0	57.0	12.0	23.000	129.000	12.000	4.660	5.73	10.00	9.0	NA	NA	NA	NA	409	257	NA
07/13/80	3.20	NA	6360	5750	417.0	444.0	220.0	31.0	84.000	415.000	28.000	15.000	9.90	36.00	15.0	NA	NA	NA	NA	284	34	NA
07/23/80	3.20	NA	5000	5700	435.0	506.0	232.0	30.0	80.000	411.600	30.000	14.000	9.90	37.00	16.0	NA	NA	NA	NA	295	20	NA
08/03/80	3.20	NA	5000	NA	483.0	587.0	274.0	34.0	90.000	582.000	36.000	19.000	13.00	47.00	NA	NA	NA	NA	9.1	34	111	NA
08/12/80	3.35	NA	4750	4180	348.0	347.0	134.0	24.0	36.000	279.000	19.000	9.000	7.75	25.00	12.0	NA	NA	NA	9.3	492	316	NA
08/17/80	3.30	NA	5000	5560	393.0	446.0	162.0	28.0	46.000	357.000	25.000	11.000	7.45	30.00	13.0	NA	NA	NA	NA	261	26	NA
08/22/80	NA	NA	4900	4180	279.0	512.0	133.0	17.0	31.000	266.000	21.000	11.000	6.50	22.00	13.0	NA	NA	NA	NA	231	112	NA
08/29/80	3.50	NA	2540	2400	172.0	240.0	65.0	12.0	22.000	141.000	10.420	5.300	5.22	12.00	19.0	NA	NA	NA	8.2	1319	700	NA
09/08/80	3.40	NA	4078	4100	231.0	503.0	133.0	21.0	44.000	262.000	21.000	10.000	8.83	22.00	15.0	NA	NA	NA	NA	522	134	NA
09/09/80	3.30	NA	4350	4700	283.0	569.0	154.0	22.0	54.000	296.000	22.000	12.000	7.30	25.00	13.0	NA	NA	NA	NA	102	600	14.0
09/12/80	3.40	NA	2694	2580	103.0	296.0	75.0	13.0	28.000	174.000	11.000	8.000	5.16	14.00	11.0	NA	NA	NA	NA	916	596	18.0
09/18/80	3.40	NA	3310	3050	158.0	373.0	94.0	15.0	56.000	217.000	15.000	9.000	5.45	17.00	16.0	NA	NA	NA	NA	3691	2740	14.0
09/25/80	3.30	NA	5588	5200	380.0	608.0	189.0	38.0	60.000	443.000	26.500	15.500	8.71	35.00	22.0	NA	NA	NA	11.0	1109	288	14.0
10/23/80	3.30	NA	5742	7700	390.0	660.0	202.0	32.0	59.300	472.000	28.800	16.900	8.68	38.00	19.0	NA	NA	NA	9.4	757	593	10.0
10/31/80	3.40	NA	5148	4700	309.0	491.0	187.0	26.0	75.000	353.000	28.000	14.000	6.00	33.00	16.0	NA	NA	NA	NA	647	91	11.0
02/19/81	3.60	NA	3692	3450	198.9	279.0	148.0	NA	49.300	270.000	18.100	8.850	NA	NA	17.0	NA	NA	NA	NA	908	1419	8.0
02/21/81	3.60	NA	2500	2040	130.0	144.2	87.1	NA	26.800	183.000	11.000	4.920	NA	NA	14.0	NA	NA	NA	NA	1344	549	NA
03/29/81	3.50	NA	2000	1420	94.9	110.5	53.3	NA	20.200	123.000	7.600	3.680	NA	NA	NA	NA	NA	NA	4.9	3702	773	NA
04/03/81	3.90	NA	1825	1250	85.8	91.0	46.8	NA	23.600	115.000	6.900	3.590	NA	NA	NA	NA	NA	NA	NA	4364	927	NA
04/13/81	3.70	NA	4000	2800	155.0	214.0	69.0	NA	71.000	239.000	13.800	7.900	NA	NA	NA	NA	NA	NA	NA	1033	405	NA
04/17/81	3.80	NA	4500	3080	194.0	293.0	100.0	NA	76.000	264.000	14.800	8.510	NA	NA	NA	NA	NA	NA	7.3	806	413	NA
04/23/81	3.70	NA	4350	3850	192.0	338.0	108.0	NA	107.000	331.000	17.300	10.700	NA	NA	NA	NA	NA	NA	NA	1453	1594	NA
04/25/81	3.60	NA	4575	3000	170.0	279.0	81.0	NA	76.000	251.000	14.700	8.050	NA	NA	NA	NA	NA	NA	NA	6057	1715	NA
04/28/81	3.60	NA	4200	3000	174.0	270.0	76.0	NA	70.000	233.000	14.200	7.730	NA	NA	NA	NA	NA	NA	7.0	4168	2146	NA
04/29/81	3.50	NA	5004	4300	296.0	440.0	114.0	NA	105.000	312.000	20.300	8.150	NA	NA	NA	NA	NA	NA	NA	3736	1098	9.0
05/12/81	3.35	NA	6386	5250	374.0	590.0	200.0	NA	159.000	501.000	26.300	9.780	NA	NA	NA	NA	NA	NA	12.0	2165	119	18.0
05/28/81	3.35	NA	5952	7500	344.0	580.0	182.0	NA	137.000	500.000	25.200	9.630	NA	NA	NA	NA	NA	NA	NA	893	2146	16.0
06/08/81	3.25	NA	5709	6000	312.0	540.0	158.0	NA	102.000	441.500	25.910	9.580	NA	NA	NA	NA	NA	NA	9.6	621	51	19.0
06/22/81	3.40	NA	4570	4300	246.0	460.0	114.0	NA	63.000	354.000	21.750	7.990	NA	NA	NA	NA	NA	NA	NA	1143	607	16.0
06/24/81	3.40	NA	3150	2370	174.0	200.0	56.0	NA	34.000	199.000	13.200	5.700	NA	NA	NA	NA	NA	NA	NA	1189	659	17.0
06/28/81	3.35	NA	3032	2550	174.0	210.0	58.0	NA	45.000	204.500	13.800	5.600	NA	NA	NA	NA	NA	NA	6.6	2036	1352	15.0
07/04/81	3.30	NA	4298	4050	272.0	366.0	106.0	NA	117.000	307.500	22.000	11.000	NA	NA	NA	NA	NA	NA	NA	4891	173	22.0
07/24/81	3.30	NA	7122	8000	392.0	524.6	242.0	NA	185.000	550.000	43.000	21.500	NA	NA	33.0	NA	NA	NA	7.9	386	70	21.0
08/04/81	3.10	NA	8085	9350	426.0	830.0	256.0	NA	129.500	676.000	50.000	25.500	NA	NA	NA	NA	NA	NA	NA	291	22	18.0

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
08/17/81	3.10	NA	8037	8400	414.0	870.0	260.0	34.0	104.000	630.000	43.000	20.000	NA	NA	NA	NA	NA	NA	NA	182	4	22.0
08/31/81	3.30	NA	8900	9000	440.0	980.0	250.0	NA	108.000	741.000	47.000	25.000	NA	NA	NA	NA	NA	NA	9.8	106	36	17.0
09/07/81	3.30	NA	7500	7900	440.0	820.0	210.0	NA	80.000	660.000	42.000	22.000	NA	NA	NA	NA	NA	NA	NA	121	58	NA
09/16/81	3.30	NA	5000	9600	480.0	1030.0	260.0	NA	92.000	762.000	49.000	26.000	NA	NA	NA	NA	NA	NA	NA	72	4	NA
09/26/81	3.50	NA	4965	5000	330.0	520.0	120.0	NA	39.000	381.000	24.000	12.000	NA	NA	NA	NA	NA	NA	8.0	329	210	NA
10/01/81	3.50	NA	4375	2600	206.0	290.0	86.0	NA	23.000	240.000	16.000	8.000	NA	NA	NA	NA	NA	NA	NA	693	983	NA
10/04/81	3.45	NA	3900	2600	154.0	238.0	64.0	14.0	32.000	200.000	16.000	6.000	NA	NA	NA	NA	NA	NA	NA	1136	706	NA
10/14/81	3.40	NA	3950	2960	176.0	236.0	68.0	14.0	53.000	198.000	15.000	8.000	NA	NA	NA	NA	NA	NA	5.6	4554	1622	NA
10/18/81	3.30	NA	5000	4000	326.0	262.0	102.0	10.0	58.000	119.000	7.000	4.130	NA	NA	NA	NA	NA	NA	NA	6337	1288	NA
11/06/81	3.20	NA	5000	5400	292.0	530.0	140.0	26.0	140.000	412.000	26.000	11.000	NA	NA	NA	NA	NA	NA	6.6	3585	105	NA
03/30/82	3.45	NA	4200	3100	174.0	350.0	68.0	12.0	55.000	233.000	18.000	8.050	NA	NA	NA	NA	NA	NA	NA	3218	NA	NA
04/06/82	3.50	NA	4200	3200	169.0	355.0	70.0	11.0	65.000	253.000	17.500	8.180	NA	NA	NA	NA	NA	NA	NA	2657	NA	NA
04/13/82	3.50	NA	2600	1900	131.0	140.0	36.0	8.0	36.000	138.000	8.900	4.100	NA	NA	NA	NA	NA	NA	NA	5443	NA	NA
04/15/82	3.55	NA	1950	1015	111.1	94.0	23.0	6.0	28.000	88.500	6.200	2.430	NA	NA	NA	NA	NA	NA	NA	5182	NA	NA
04/19/82	3.50	NA	2250	1500	142.0	152.0	38.0	10.0	39.000	126.000	8.100	3.750	NA	NA	NA	NA	NA	NA	NA	2014	NA	NA
04/23/82	3.45	NA	3050	2300	188.0	192.0	70.0	12.0	62.000	197.000	12.200	5.470	NA	NA	NA	NA	NA	NA	NA	1344	NA	NA
05/03/82	3.40	NA	4010	5500	330.0	400.0	126.0	22.0	126.000	405.000	28.000	12.000	NA	NA	NA	NA	NA	NA	NA	114	NA	NA
05/04/82	3.50	NA	3450	2500	156.0	216.0	66.0	12.0	58.000	142.000	12.200	5.710	NA	NA	NA	NA	NA	NA	NA	867	NA	NA
05/07/82	3.40	NA	5000	4000	212.0	300.0	136.0	16.0	105.000	316.000	22.000	10.000	NA	NA	NA	NA	NA	NA	NA	1889	NA	NA
05/10/82	3.50	NA	3650	2700	172.0	226.0	74.0	12.0	72.000	204.000	12.700	5.740	NA	NA	NA	NA	NA	NA	NA	3751	NA	NA
05/11/82	3.40	NA	4250	4200	190.0	200.0	10.0	14.0	88.000	251.000	20.000	9.000	NA	NA	NA	NA	NA	NA	NA	4698	NA	NA
05/14/82	3.40	NA	4550	4700	248.0	348.0	86.0	16.4	88.000	254.000	16.300	7.780	3.63	17.04	NA	NA	NA	NA	NA	6787	NA	NA
05/21/82	3.30	NA	4725	8000	320.0	660.0	153.0	21.0	135.000	416.000	28.000	14.000	4.01	24.00	NA	NA	NA	NA	NA	2006	NA	NA
06/02/82	3.25	NA	6160	8500	342.0	790.0	190.0	26.0	156.000	549.000	32.000	20.000	4.51	33.00	NA	NA	NA	NA	NA	723	NA	NA
06/18/82	3.30	NA	5424	8000	298.0	680.0	155.0	22.0	113.000	440.000	30.000	16.000	3.05	28.00	NA	NA	NA	NA	NA	185	NA	NA
07/02/82	NA	NA	NA	5000	308.0	362.0	110.0	18.0	97.000	359.000	21.800	8.900	3.57	19.57	NA	NA	NA	NA	NA	7908	NA	NA
07/26/82	3.24	NA	5266	5500	314.0	408.0	120.0	20.0	102.000	443.000	25.000	10.500	5.52	23.29	NA	NA	NA	NA	NA	6908	NA	NA
08/30/82	3.26	NA	4421	3750	302.0	374.0	122.0	18.0	86.000	344.000	21.200	8.500	3.26	18.89	NA	NA	NA	NA	NA	4376	NA	NA
10/04/82	3.60	NA	4316	3750	230.0	336.0	92.0	16.0	92.000	389.000	20.000	8.300	5.44	20.50	NA	NA	NA	NA	NA	23454	NA	NA
11/01/82	3.40	NA	4200	NA	168.0	278.0	76.0	12.0	58.000	291.000	17.200	7.000	3.03	17.60	NA	NA	NA	NA	NA	1363	NA	NA
03/18/83	3.66	NA	3154	3360	NA	NA	NA	NA	38.000	161.000	13.000	5.380	NA	NA	NA	NA	NA	NA	NA	1537	NA	NA
04/14/83	3.70	NA	3209	3400	NA	NA	NA	NA	40.000	174.000	14.000	5.690	NA	NA	NA	NA	NA	NA	NA	3311	NA	NA
05/15/83	3.54	NA	3873	4720	NA	NA	NA	NA	90.000	240.000	18.000	8.540	NA	NA	NA	NA	NA	NA	NA	2361	NA	NA
06/16/83	3.67	NA	4471	5000	NA	NA	NA	NA	84.000	307.000	16.600	6.490	NA	NA	NA	NA	NA	NA	NA	507	NA	NA
07/04/83	3.50	NA	3825	3700	NA	NA	NA	NA	109.000	256.000	18.400	8.200	NA	NA	NA	NA	NA	NA	NA	5314	NA	NA
07/11/83	3.70	NA	5000	7200	NA	NA	NA	NA	123.000	480.000	32.400	14.800	NA	NA	NA	NA	NA	NA	NA	731	NA	NA
07/17/83	3.50	NA	6572	7700	NA	NA	NA	NA	127.000	418.000	28.300	15.200	NA	NA	NA	NA	NA	NA	NA	284	NA	NA
07/20/83	3.50	NA	6608	6200	NA	NA	NA	NA	140.000	392.000	28.000	13.400	NA	NA	NA	NA	NA	NA	NA	825	NA	NA
07/25/83	3.60	NA	6250	8200	NA	NA	NA	NA	154.000	510.000	36.200	16.200	NA	NA	NA	NA	NA	NA	NA	341	NA	NA
07/31/83	3.65	NA	6405	8900	NA	NA	NA	NA	142.000	535.000	38.000	16.800	NA	NA	NA	NA	NA	NA	NA	148	NA	NA
08/07/83	3.60	NA	4860	4700	NA	NA	NA	NA	70.600	314.000	23.800	10.600	NA	NA	NA	NA	NA	NA	NA	590	NA	NA
08/15/83	3.45	NA	6376	6400	NA	NA	NA	NA	81.000	384.000	29.600	12.600	NA	NA	NA	NA	NA	NA	NA	307	NA	NA
08/22/83	3.60	NA	3500	3150	NA	NA	NA	NA	71.000	201.000	14.640	5.920	NA	NA	NA	NA	NA	NA	NA	3653	NA	NA
08/28/83	3.60	NA	4300	4000	NA	NA	NA	NA	89.000	265.000	18.480	8.140	NA	NA	NA	NA	NA	NA	NA	1393	NA	NA
09/04/83	3.55	NA	5537	5500	NA	NA	NA	NA	110.000	349.000	23.500	10.830	NA	NA	NA	NA	NA	NA	NA	636	NA	NA
09/11/83	3.45	NA	3975	3150	NA	NA	NA	NA	75.000	237.000	16.790	7.370	NA	NA	NA	NA	NA	NA	NA	1298	NA	NA
09/18/83	3.50	NA	4500	4300	NA	NA	NA	NA	85.000	289.000	19.760	8.780	NA	NA	NA	NA	NA	NA	NA	560	NA	NA
09/25/83	3.80	NA	3725	3500	NA	NA	NA	NA	69.000	240.000	17.000	7.380	NA	NA	NA	NA	NA	NA	NA	874	NA	NA
10/02/83	3.50	NA	4475	3400	NA	NA	NA	NA	108.000	241.000	18.000	7.460	NA	NA	NA	NA	NA	NA	NA	10401	NA	NA
10/08/83	3.75	NA	4725	3650	NA	NA	NA	NA	106.000	258.000	15.500	7.820	NA	NA	NA	NA	NA	NA	NA	6018	NA	NA
10/15/83	3.70	NA	4624	4050	NA	NA	NA	NA	116.000	290.000	19.000	8.600	NA	NA	NA	NA	NA	NA	NA	6964	NA	NA

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date'	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
10/23/83	3.50	NA	5538	5300	NA	NA	NA	NA	130.000	357.000	23.000	10.350	NA	NA	NA	NA	NA	NA	NA	1480	NA	NA
10/30/83	3.65	NA	5576	6200	NA	NA	NA	NA	149.000	410.000	29.000	13.600	NA	NA	NA	NA	NA	NA	NA	1083	NA	NA
11/06/83	3.70	NA	6116	7000	NA	NA	NA	NA	131.800	473.000	32.800	15.200	NA	NA	NA	NA	NA	NA	NA	333	NA	NA
11/13/83	3.80	NA	6958	8100	NA	NA	NA	NA	136.000	520.000	36.800	17.400	NA	NA	NA	NA	NA	NA	NA	212	NA	NA
11/20/83	3.50	NA	3800	3250	NA	NA	NA	NA	82.000	236.000	16.200	7.800	NA	NA	NA	NA	NA	NA	NA	2120	NA	NA
12/01/83	3.55	NA	4875	4600	NA	NA	NA	NA	117.600	314.000	21.600	10.400	NA	NA	NA	NA	NA	NA	NA	1881	NA	NA
02/24/84	3.60	NA	6110	6800	356.5	991.3	165.9	NA	133.800	478.000	32.200	17.400	NA	NA	NA	NA	NA	NA	NA	889	NA	NA
03/15/84	3.55	NA	3225	3200	220.6	381.1	73.2	NA	77.000	220.000	15.000	7.200	NA	NA	NA	NA	NA	NA	NA	1211	NA	NA
04/15/84	3.81	NA	1260	1300	110.9	104.8	23.0	NA	31.000	62.000	4.400	2.300	NA	NA	NA	NA	NA	NA	NA	11004	NA	NA
05/16/84	3.65	NA	2392	2950	NA	NA	NA	NA	83.000	160.000	12.500	5.340	NA	NA	NA	NA	NA	NA	NA	8396	NA	NA
06/13/84	3.58	NA	2985	4500	242.8	342.5	69.3	NA	90.200	207.000	14.000	6.600	NA	NA	NA	NA	NA	NA	NA	11254	NA	NA
07/14/84	3.45	NA	5230	6700	NA	NA	122.0	NA	142.000	356.000	23.000	7.000	NA	NA	NA	NA	NA	NA	NA	2498	NA	NA
08/15/84	3.47	NA	4760	5000	323.7	348.7	115.6	NA	107.000	334.560	34.400	10.540	NA	NA	NA	NA	NA	NA	NA	2252	NA	NA
09/12/84	3.44	NA	5175	6000	374.0	580.0	146.5	NA	105.800	425.000	44.000	14.600	NA	NA	NA	NA	NA	NA	NA	886	NA	NA
10/13/84	3.41	NA	2870	3625	260.1	290.0	74.3	NA	88.700	218.000	21.600	7.700	NA	NA	NA	NA	NA	NA	NA	7066	NA	NA
11/14/84	3.51	NA	3150	2750	372.0	500.0	115.8	NA	134.700	338.000	19.400	10.700	NA	NA	NA	NA	NA	NA	NA	2074	NA	NA
12/05/84	3.42	NA	2470	2200	314.0	470.0	89.0	NA	100.100	260.000	15.500	8.500	NA	NA	NA	NA	NA	NA	NA	651	NA	NA
12/19/84	3.45	NA	2650	2900	254.0	285.0	88.0	NA	97.300	223.000	13.400	6.720	NA	NA	NA	NA	NA	NA	NA	1444	NA	NA
04/02/85	3.08	NA	5000	5650	330.0	562.0	135.6	NA	104.000	348.000	23.000	13.480	NA	NA	NA	NA	NA	NA	NA	489	NA	NA
04/16/85	3.94	NA	2745	2850	210.0	252.0	51.2	NA	86.000	157.000	13.000	5.400	NA	NA	NA	NA	NA	NA	NA	14152	NA	NA
05/14/85	3.74	NA	4640	4808	285.0	475.0	95.0	NA	150.000	279.000	10.500	18.800	NA	NA	NA	NA	NA	NA	NA	3970	NA	NA
06/14/85	3.58	NA	3850	4200	224.0	375.0	70.0	NA	110.000	211.000	14.600	7.100	NA	NA	NA	NA	NA	NA	NA	22384	NA	NA
07/15/85	3.48	NA	4900	5200	270.0	446.0	102.0	NA	110.000	269.000	17.000	7.800	NA	NA	NA	NA	NA	NA	NA	4534	NA	NA
08/13/85	3.47	NA	4450	5100	272.0	466.0	96.0	NA	96.000	266.000	17.800	6.380	NA	NA	NA	NA	NA	NA	NA	2203	NA	NA
09/13/85	3.50	NA	4325	4300	334.0	305.0	78.0	NA	103.000	221.000	14.800	6.500	NA	NA	NA	NA	NA	NA	NA	16207	NA	NA
10/15/85	3.50	NA	4075	3200	276.0	236.0	60.0	NA	94.000	174.000	12.500	5.320	NA	NA	NA	NA	NA	NA	NA	17676	NA	NA
11/15/85	3.54	NA	5200	2100	388.0	510.0	100.0	NA	119.000	296.000	20.700	8.570	NA	NA	NA	NA	NA	NA	NA	1499	NA	NA
12/10/85	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	106	NA	NA
03/23/86	3.80	NA	4800	4200	270.0	414.0	86.0	NA	67.000	250.000	22.000	7.340	NA	NA	NA	NA	NA	NA	NA	7244	NA	NA
04/16/86	3.90	NA	1950	1700	152.0	130.0	28.6	NA	47.000	97.000	11.600	3.010	NA	NA	NA	NA	NA	NA	NA	9909	NA	NA
05/14/86	3.69	NA	3000	3150	282.0	270.0	44.0	NA	92.000	251.000	11.000	5.000	NA	NA	NA	NA	NA	NA	NA	10125	NA	NA
06/11/86	3.63	NA	4750	4600	316.0	360.0	90.0	NA	124.000	260.000	14.000	8.200	NA	NA	NA	NA	NA	NA	NA	1711	NA	NA
07/12/86	3.59	NA	1725	3000	225.0	314.0	61.0	NA	77.000	180.000	12.700	6.300	NA	NA	NA	NA	NA	NA	NA	3085	NA	NA
08/13/86	3.55	NA	4100	3700	268.0	357.0	78.0	NA	79.000	206.000	22.400	7.400	NA	NA	NA	NA	NA	NA	NA	1597	NA	NA
09/14/86	3.40	NA	3550	2700	262.0	286.0	56.0	NA	98.000	159.000	19.400	5.980	NA	NA	NA	NA	NA	NA	NA	10795	NA	NA
10/16/86	3.53	NA	4400	3600	286.0	302.0	68.0	NA	110.000	203.000	13.700	7.150	NA	NA	NA	NA	NA	NA	NA	3982	NA	NA
11/16/86	3.70	NA	3550	2700	260.0	236.0	60.0	NA	110.000	158.000	10.600	7.200	NA	NA	NA	NA	NA	NA	NA	6075	NA	NA
03/28/87	3.65	NA	3075	2900	252.0	264.0	48.0	NA	75.000	160.000	10.800	5.160	NA	NA	NA	NA	NA	NA	NA	7835	NA	NA
04/18/87	3.67	NA	2625	2400	154.0	220.0	40.0	NA	63.000	137.000	10.000	4.440	NA	NA	NA	NA	NA	NA	NA	1620	NA	NA
05/16/87	3.61	NA	2700	2250	233.0	172.0	34.0	NA	73.000	97.000	6.700	3.730	NA	NA	NA	NA	NA	NA	NA	18573	NA	NA
06/16/87	3.44	NA	4900	4640	428.0	492.0	116.0	NA	153.000	331.000	18.600	25.200	NA	NA	NA	NA	NA	NA	NA	1400	NA	NA
07/15/87	3.36	NA	3000	2900	230.0	228.0	48.0	NA	82.000	140.000	9.000	4.930	NA	NA	NA	NA	NA	NA	NA	16908	NA	NA
08/15/87	3.24	NA	3950	3565	279.0	316.0	68.0	NA	97.000	193.000	12.200	6.590	NA	NA	NA	NA	NA	NA	NA	3948	NA	NA
09/16/87	3.25	NA	4100	1200	240.0	238.0	50.0	NA	79.000	135.000	7.980	4.660	NA	NA	NA	NA	NA	NA	NA	7612	NA	NA
10/13/87	3.32	NA	4950	5120	364.0	391.0	88.0	NA	132.000	220.000	13.010	7.250	NA	NA	NA	NA	NA	NA	NA	1457	NA	NA
11/10/87	3.43	NA	4325	3700	308.0	354.0	78.0	NA	9.300	193.000	11.240	6.360	NA	NA	NA	NA	NA	NA	NA	731	NA	NA
05/12/88	3.64	NA	3000	2300	224.0	198.0	54.0	NA	89.500	156.500	8.950	8.450	NA	NA	NA	NA	NA	NA	NA	7275	NA	NA
06/12/88	3.47	NA	4850	4500	412.0	376.0	72.0	NA	163.000	232.000	18.600	23.000	NA	NA	NA	NA	NA	NA	NA	1098	NA	NA
07/15/88	3.60	NA	4525	4360	368.0	382.0	74.0	NA	134.000	234.000	17.600	16.000	NA	NA	NA	NA	NA	NA	NA	757	NA	NA
08/16/88	3.39	NA	2900	2165	247.0	176.0	43.0	NA	72.000	152.000	8.800	6.250	NA	NA	NA	NA	NA	NA	NA	32744	NA	NA
09/16/88	3.37	NA	4275	3200	334.0	294.0	74.0	NA	93.000	117.000	14.300	10.300	NA	NA	NA	NA	NA	NA	NA	6310	NA	NA

Table A2.6. Drainage quality data from Duluth Complex test pile FL5, from 1978-1991 (1.41% S, 0.30% Cu, 0.08% Ni).

Date[1]	pH s.u.	Alk. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
09/24/88	3.70	NA	1875	1280	160.0	96.0	26.0	NA	36.000	71.000	4.300	3.700	NA	NA	NA	NA	NA	NA	NA	6934	NA	NA
10/18/88	3.67	NA	3700	3400	238.0	258.0	50.0	NA	64.000	143.000	10.900	8.700	NA	NA	NA	NA	NA	NA	NA	1790	NA	NA
10/24/88	3.77	NA	1980	1400	132.0	118.0	26.0	NA	43.000	88.000	4.900	4.400	NA	NA	NA	NA	NA	NA	NA	3819	NA	NA
11/15/88	3.71	NA	2975	2740	216.0	222.0	42.0	NA	68.000	107.000	8.200	7.200	NA	NA	NA	NA	NA	NA	NA	7494	NA	NA
11/30/88	3.44	NA	4200	3760	276.0	306.0	56.0	NA	102.000	169.000	11.500	9.300	NA	NA	NA	NA	NA	NA	NA	2665	NA	NA
04/07/89	3.95	NA	2700	2400	230.0	166.0	38.0	NA	55.000	98.000	9.800	7.000	NA	NA	NA	NA	NA	NA	NA	17581	NA	NA
04/24/89	4.00	NA	1050	650	78.0	38.0	10.0	NA	18.000	26.000	1.800	1.600	NA	NA	NA	NA	NA	NA	NA	7850	NA	NA
05/16/89	3.79	NA	2582	2100	182.0	152.0	34.0	NA	62.000	94.000	7.300	6.100	NA	NA	NA	NA	NA	NA	NA	5802	NA	NA
06/17/89	3.64	NA	3150	2800	258.0	184.0	52.0	NA	80.000	145.000	13.700	7.700	NA	NA	NA	NA	NA	NA	NA	25560	NA	NA
07/17/89	3.55	NA	4325	4100	420.0	304.0	62.0	NA	118.000	210.000	17.700	10.900	NA	NA	NA	NA	NA	NA	NA	2248	NA	NA
08/14/89	3.62	NA	3800	3540	286.0	284.0	58.0	NA	89.000	245.000	15.400	10.200	NA	NA	NA	NA	NA	NA	NA	1158	NA	NA
09/14/89	3.41	NA	2900	2460	272.0	168.0	36.0	NA	68.000	91.000	11.800	6.600	NA	NA	NA	NA	NA	NA	NA	16325	NA	NA
10/16/89	3.51	NA	3700	2960	276.0	226.0	52.0	NA	83.000	116.000	17.700	9.600	NA	NA	NA	NA	NA	NA	NA	2203	NA	NA
11/13/89	3.60	NA	3200	2480	298.0	204.0	44.0	NA	74.000	105.000	21.000	8.900	NA	NA	NA	NA	NA	NA	NA	3758	NA	NA
03/26/90	3.70	NA	2500	1900	226.0	146.0	31.2	NA	41.000	73.000	6.350	7.310	NA	NA	NA	NA	NA	NA	NA	3615	NA	NA
04/12/90	3.91	NA	1950	1660	182.0	108.0	20.0	NA	53.000	57.000	4.380	3.980	NA	NA	NA	NA	NA	NA	NA	20708	NA	NA
05/17/90	3.75	NA	2725	2300	222.0	156.0	26.0	NA	70.000	80.000	6.570	5.560	NA	NA	NA	NA	NA	NA	NA	10783	NA	NA
06/18/90	3.59	NA	3475	2660	236.0	180.0	38.0	NA	72.000	92.000	9.170	7.640	NA	NA	NA	NA	NA	NA	NA	9421	NA	NA
07/16/90	3.62	NA	4175	3460	306.0	226.0	40.0	NA	92.000	108.000	9.600	8.610	NA	NA	NA	NA	NA	NA	NA	6684	NA	NA
08/13/90	3.53	NA	4785	3920	370.0	326.0	62.0	NA	106.000	166.000	13.100	11.200	NA	NA	NA	NA	NA	NA	NA	757	NA	NA
09/10/90	3.64	NA	3260	2360	238.0	178.0	38.0	NA	54.000	99.000	7.280	5.380	NA	NA	NA	NA	NA	NA	NA	2434	NA	NA
10/15/90	3.55	NA	3415	2160	252.0	154.0	32.0	NA	74.000	88.000	6.510	5.000	NA	NA	NA	NA	NA	NA	NA	20643	NA	NA
11/16/90	3.60	NA	5000	3940	372.0	308.0	62.0	NA	120.000	142.000	12.170	12.440	NA	NA	NA	NA	NA	NA	NA	1397	NA	NA
04/08/91	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	9777	NA	NA
04/22/91	3.76	NA	1700	1010	120.0	80.0	14.0	NA	34.000	41.000	2.200	1.600	NA	NA	NA	NA	NA	NA	NA	5598	NA	NA
05/20/91	3.65	NA	4175	2500	262.0	205.0	32.0	NA	94.000	103.000	5.600	4.200	NA	NA	NA	NA	NA	NA	NA	8687	NA	NA
06/17/91	3.61	NA	2975	2000	188.0	168.0	26.0	NA	64.000	84.000	5.300	3.300	NA	NA	NA	NA	NA	NA	NA	3388	NA	NA
07/15/91	3.47	NA	3225	2020	198.0	140.0	24.0	NA	66.000	69.000	4.600	3.500	NA	NA	NA	NA	NA	NA	NA	16938	NA	NA
08/12/91	3.43	NA	5000	4460	394.0	334.0	72.0	NA	124.000	176.000	11.900	8.700	NA	NA	NA	NA	NA	NA	NA	1268	NA	NA
09/09/91	3.59	NA	3225	1940	200.0	126.0	32.0	NA	54.000	73.000	4.600	3.700	NA	NA	NA	NA	NA	NA	NA	6079	NA	NA
10/07/91	3.62	NA	3775	2380	238.0	158.0	38.0	NA	72.000	87.000	5.500	5.600	NA	NA	NA	NA	NA	NA	NA	3656	NA	NA
11/07/91	3.51	NA	3300	2280	236.0	156.0	34.0	NA	75.000	80.000	5.100	5.100	NA	NA	NA	NA	NA	NA	NA	14780	NA	NA

NA: Not analyzed.

[1]Dates represent intermediate dates for measurement periods.

Table A2.6.1. Drainage quality data from Duluth Complex test pile FL5, 1992-1993 (1.41% S, 0.30% Cu, 0.08% Ni).

Date[1]	pH	Alk. mg/L	Lab Acy. mg/L	Calc. Acy. mg/L	S.C. µmho/cm	SO4 mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Cl mg/L	NO3 mg/L	NH3 mg/L	P mg/L	DissOrgC mg/L	Vol. L	DailyFlow L	Temp. °C.
04/23/92	3.86	NA	410	223.2	1525	720	76.0	56.0	14.00	3.30	24	19.90	28	1.70	1.20	0.34	3.04	NA	NA	NA	NA	NA	4634	NA	NA
05/11/92	3.76	NA	695	388.8	2420	725	148.0	96.0	22.00	5.30	37	53.00	47	2.90	2.10	0.87	5.52	NA	NA	NA	NA	NA	2162	NA	NA
05/26/92	3.71	NA	838	449.5	2480	1760	176.0	118.0	26.60	6.30	42	59.00	58	3.60	2.80	1.07	6.68	NA	NA	NA	NA	NA	2294	NA	NA
06/08/92	4.22	NA	940	541.4	3780	2560	320.0	216.0	40.00	11.00	44	79.00	82	5.10	4.00	0.47	9.48	NA	NA	NA	NA	NA	603	NA	NA
06/22/92	3.66	NA	1350	667.2	3850	2960	252.0	208.0	42.80	9.10	56	93.00	99	6.80	4.70	1.34	10.90	NA	NA	NA	NA	NA	238	NA	NA
07/07/92	3.68	NA	860	445.1	2640	1880	174.0	124.0	28.00	7.20	40	59.00	61	4.10	3.00	1.09	6.89	NA	NA	NA	NA	NA	1105	NA	NA
07/20/92	3.61	NA	1310	652.7	3880	5900	284.0	202.0	44.00	10.00	53	99.00	95	6.70	4.60	1.44	10.70	NA	NA	NA	NA	NA	366	NA	NA
08/03/92	3.64	NA	1050	486.2	2710	1990	192.0	144.0	34.00	8.60	43	63.00	69	4.90	3.60	1.13	8.08	NA	NA	NA	NA	NA	336	NA	NA
08/17/92	4.43	NA	550	337.1	3000	2060	214.0	228.0	39.00	17.10	23	51.00	60	4.30	3.00	0.27	7.93	NA	NA	NA	NA	NA	422	NA	NA
08/31/92	4.04	NA	410	271.4	2400	1400	170.0	144.0	28.40	18.20	17	39.00	56	3.20	2.30	0.44	5.70	NA	NA	NA	NA	NA	309	NA	NA
09/14/92	3.57	NA	1085	473.6	2800	2060	216.0	144.0	33.00	8.40	41	67.00	65	4.60	3.20	1.85	7.43	NA	NA	NA	NA	NA	2397	NA	NA
09/28/92	3.60	NA	1120	507.0	3300	2240	228.0	150.0	34.60	9.20	43	76.00	69	4.90	3.40	2.07	7.52	NA	NA	NA	NA	NA	1997	NA	NA
10/12/92	4.35	NA	1130	554.2	4200	2940	326.0	272.0	51.00	15.30	43	82.00	87	6.30	4.50	0.42	10.80	NA	NA	NA	NA	NA	285	NA	NA
10/27/92	3.57	NA	1700	720.1	4380	3400	288.0	260.0	53.40	11.40	57	97.00	119	7.90	5.50	1.46	12.90	NA	NA	NA	NA	NA	147	NA	NA
11/09/92	4.07	NA	NA	NA	5000	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	69	NA	NA
11/30/92	3.75	NA	1090	445.3	2700	2050	184.0	158.0	33.00	7.60	41	50.00	64	4.80	3.30	0.92	8.19	NA	NA	NA	NA	NA	411	NA	NA
04/08/93	3.82	NA	NA	NA	1825	1275	125.0	100.0	16.50	4.50	48	35.00	52	2.86	1.90	0.43	4.71	NA	NA	NA	NA	NA	15454	NA	NA
04/22/93	3.76	NA	NA	NA	2775	1930	180.0	137.5	21.50	5.50	58	57.00	64	4.21	2.27	1.05	6.15	NA	NA	NA	NA	NA	3653	NA	NA
05/06/93	3.68	NA	NA	NA	2100	1560	147.5	105.0	18.25	5.25	48	45.00	49	3.23	1.78	1.27	4.59	NA	NA	NA	NA	NA	8846	NA	NA
05/20/93	3.67	NA	NA	NA	3350	2880	245.0	200.0	32.00	6.50	77	95.00	98	6.30	4.10	1.56	8.97	NA	NA	NA	NA	NA	1915	NA	NA
06/03/93	3.76	NA	NA	NA	2550	1800	147.5	125.0	23.25	5.00	55	58.00	66	4.13	2.55	1.10	5.71	NA	NA	NA	NA	NA	2415	NA	NA
06/17/93	3.56	NA	NA	NA	2300	1990	180.0	152.5	25.50	5.25	59	76.00	80	5.16	3.17	1.33	7.15	NA	NA	NA	NA	NA	2933	NA	NA
07/01/93	3.58	NA	NA	NA	2550	1900	182.5	147.5	24.75	5.50	63	77.00	78	4.96	3.07	1.51	7.04	NA	NA	NA	NA	NA	3338	NA	NA
07/14/93	3.57	NA	NA	NA	3090	2820	228.0	154.0	28.40	7.00	58	77.00	84	5.04	3.22	1.72	7.58	NA	NA	NA	NA	NA	15715	NA	NA
07/29/93	3.58	NA	NA	NA	3100	2600	242.0	170.0	31.80	7.40	59	79.00	89	5.60	3.60	1.58	8.16	NA	NA	NA	NA	NA	2986	NA	NA
08/12/93	3.50	NA	NA	NA	3350	2450	238.0	170.0	31.60	7.20	66	75.00	94	5.10	3.83	1.55	8.83	NA	NA	NA	NA	NA	2245	NA	NA
08/26/93	3.47	NA	NA	NA	4030	3160	314.0	242.0	44.40	9.20	83	92.00	123	7.25	5.58	1.83	13.00	NA	NA	NA	NA	NA	3338	NA	NA
09/09/93	3.60	NA	NA	NA	2900	2080	216.0	144.0	28.00	7.20	54	62.00	76	4.74	3.37	1.27	7.85	NA	NA	NA	NA	NA	2865	NA	NA
09/23/93	3.61	NA	NA	NA	3300	2450	266.0	190.0	35.80	8.20	69	72.00	100	5.97	4.47	1.46	10.20	NA	NA	NA	NA	NA	1446	NA	NA
10/07/93	3.66	NA	NA	NA	3700	2560	276.0	200.0	38.80	8.80	76	77.00	105	6.20	4.21	1.68	9.71	NA	NA	NA	NA	NA	742	NA	NA
10/20/93	3.71	NA	NA	NA	3950	2730	280.0	208.0	40.60	8.80	76	83.00	114	6.20	4.28	1.38	10.00	NA	NA	NA	NA	NA	379	NA	NA
11/04/93	3.72	NA	NA	NA	3300	1250	159.4	138.0	26.40	6.00	52	50.00	69	4.42	2.78	0.75	6.35	NA	NA	NA	NA	NA	871	NA	NA
11/22/93	3.72	NA	NA	NA	3000	1170	270.0	210.0	39.00	8.00	63	72.00	101	6.70	4.61	1.00	10.00	NA	NA	NA	NA	NA	591	NA	NA

NA: Not analyzed.

[1]Dates represent intermediate dates for measurement periods.

Table A2.7. Annual flow-weighted mean concentration vs. annual drainage volume regression statistics[1].

Sulfur %	Pile	Flow-weighted Mean Concentration (mmol/L) = a • volume (L) + b											
		Sulfate			Calcium			Magnesium			Calcium plus Magnesium		
		$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n
0.63	FL1	-1.5	0.070	15	-0.6	0.049	14	-0.1	0.001	14	-0.7	0.026	14
0.63	FL2	1.2	0.047	15	0.8	0.141	14	0.2	0.003	14	1.0	0.065	14
0.63	FL3	0.4	0.000	11	-0.2	0.001	10	1.6	0.021	10	1.4	0.010	10
0.79	FL6	-3.3	0.034	15	0.6	0.037	14	-1.5	0.025	14	-0.9	0.008	14
1.41	FL5	3.7	0.013	15	1.8	0.389	14	-0.3	0.001	14	1.4	0.011	14

[1] Regressions excluded 1978 flow-weighted mean concentration data.

APPENDIX 3

MASS RELEASE

Appendix 4: Rates of Release

Figure A4.1.	Annual flow-weighted mean calcium plus magnesium concentrations for 1978 through 1993.
Figure A4.2.	Average rates of release of sulfate, calcium, magnesium, and calcium plus magnesium vs. sulfur content.
Figures A4.3.-A4.12.	Annual rates of release of sulfate, calcium, magnesium, and calcium plus magnesium vs. annual drainage volume.
Figures A4.13.-A4.17.	Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, FL3, FL6, and FL5 (individual plots).
Figure A4.18.	Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, and FL3 (composite plot).
Figure A4.19.	Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, FL3, FL6, and FL5 (composite plot).
Table A4.1.	Annual mass release rate vs. annual drainage volume regression statistics.

Figure A4.1. Annual calcium plus magnesium release rate over time for 1978 through 1993 for FL1 (□), FL2 (*), FL3 (◊), FL4 (★), FL6 (△), and FL5 (○).

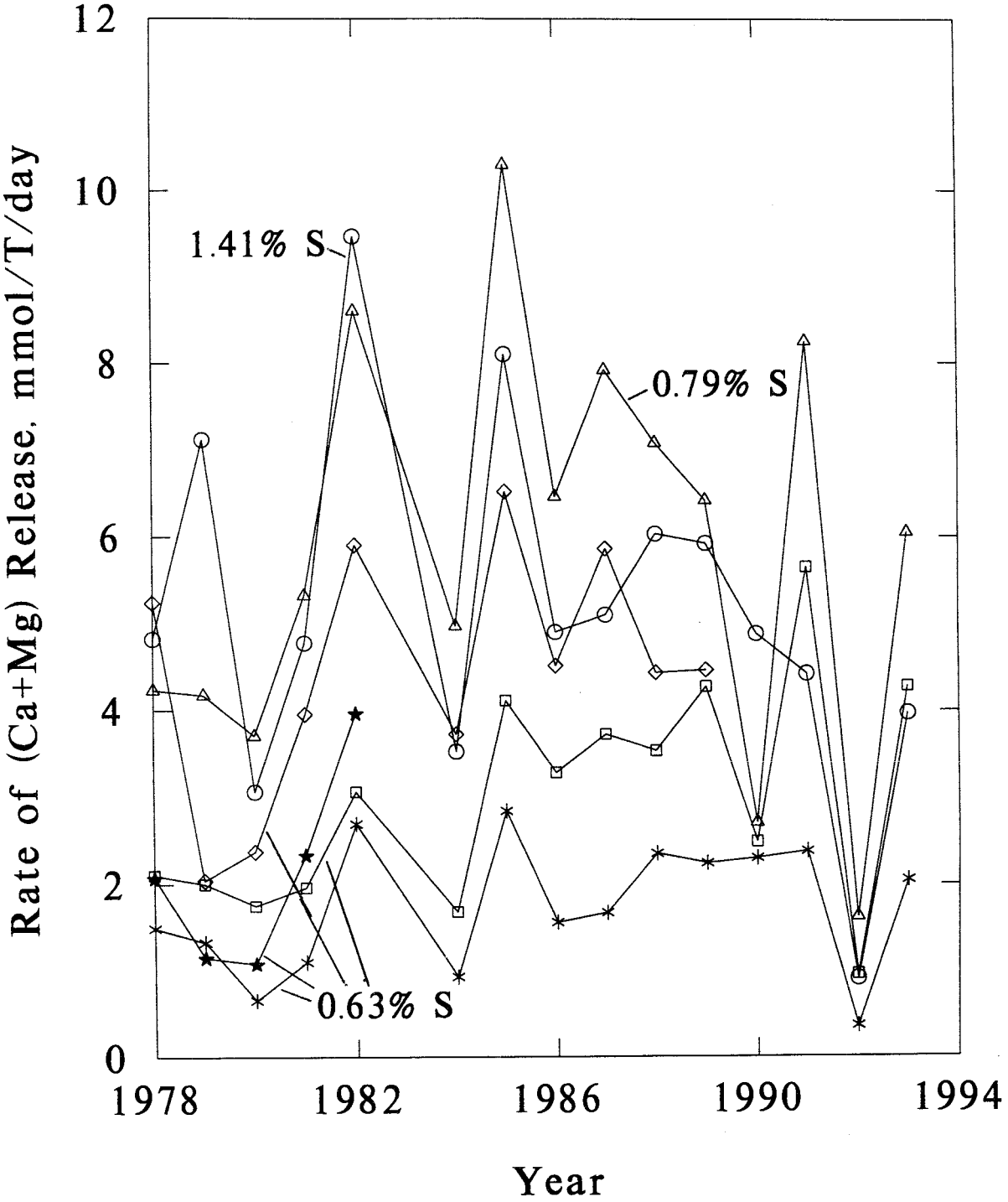

Figure A4.2. Annual rates of release of sulfate, calcium, magnesium, and calcium plus magnesium vs. sulfur content for Duluth Complex test pile FL1 (0.63% S).

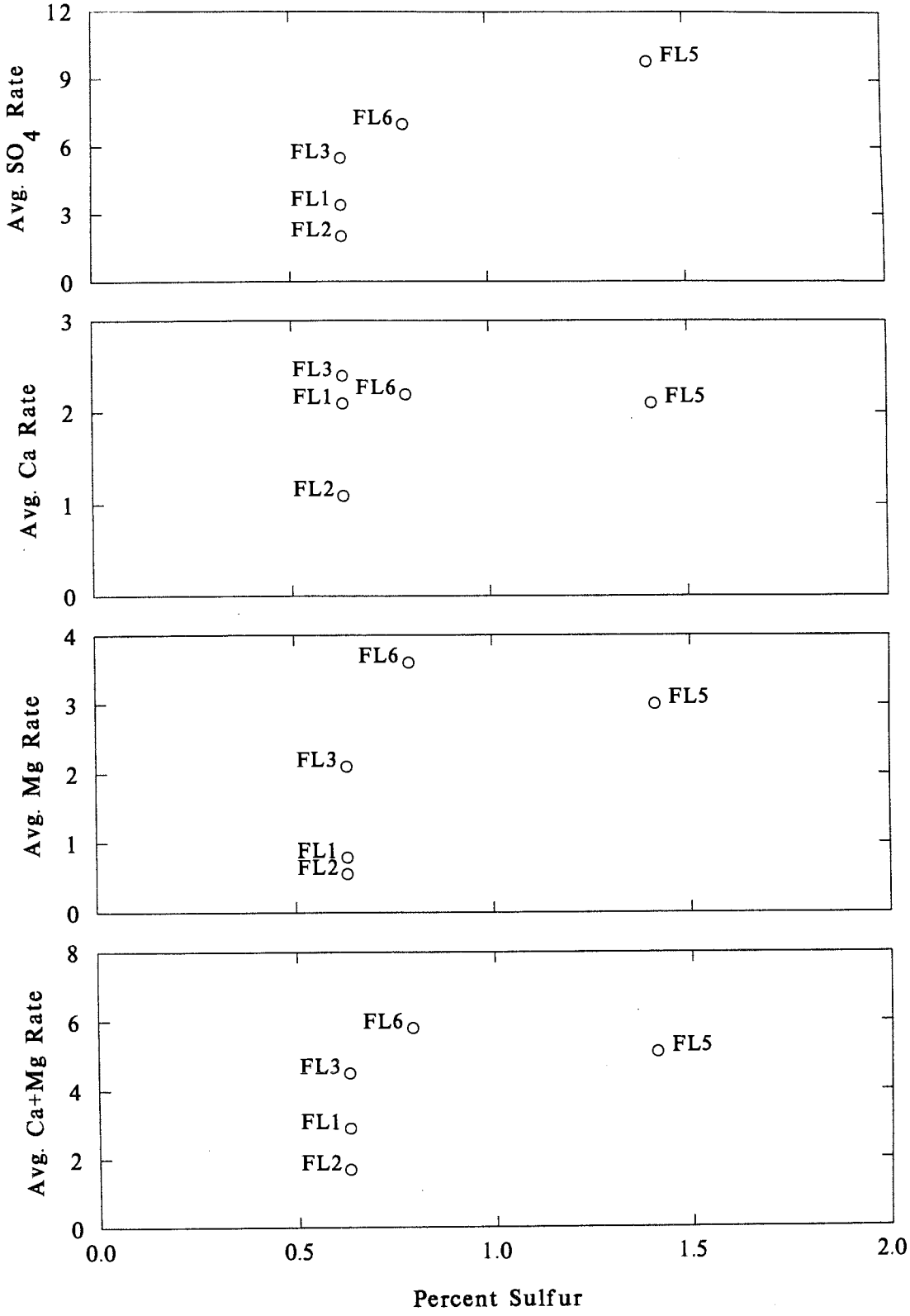

Figure A4.3. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL1 (0.63% S).

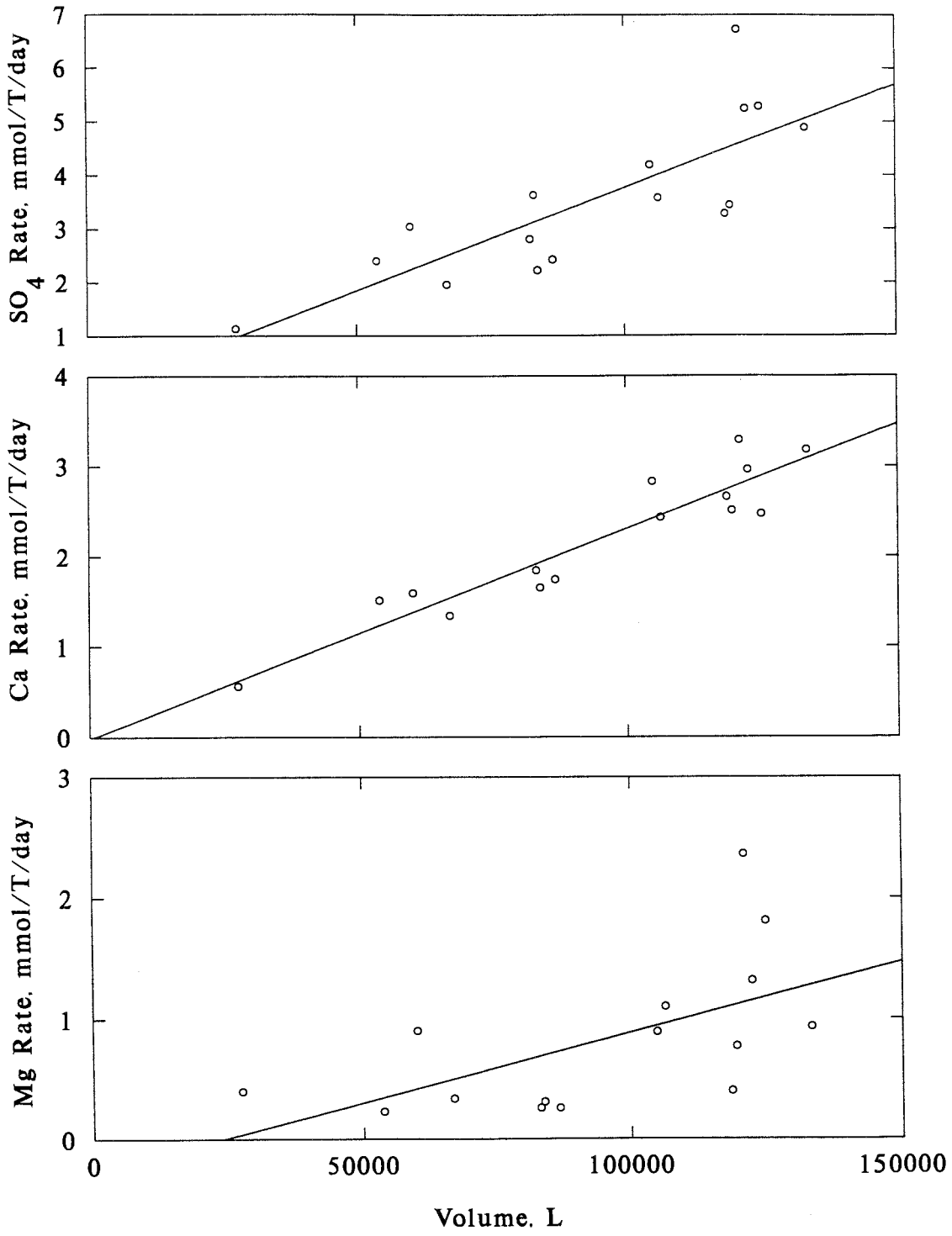

Figure A4.4. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL1 (0.63% S).

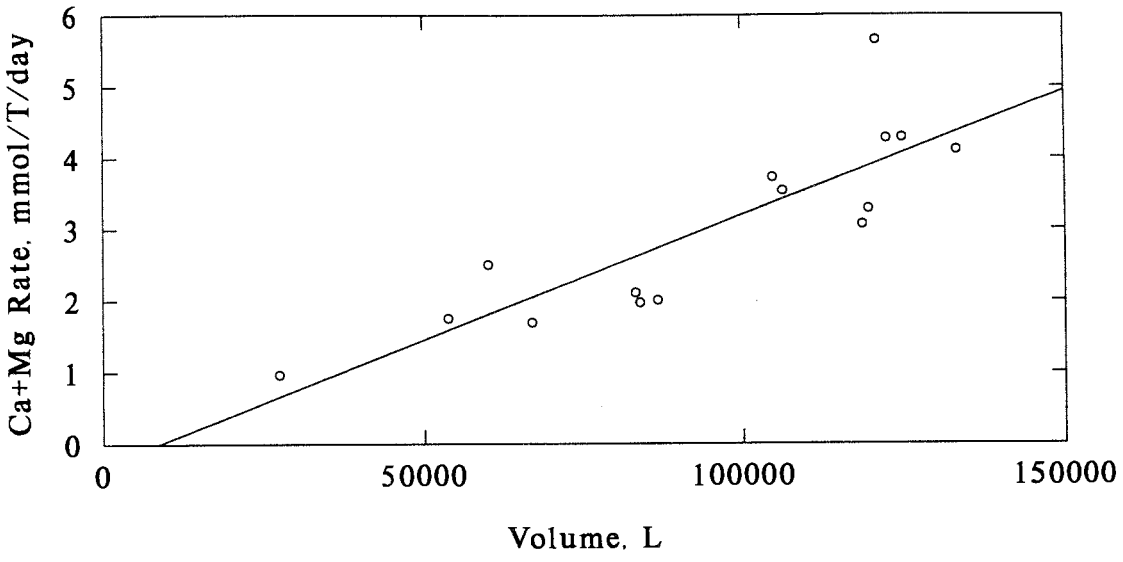

Figure A4.5. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL2 (0.63% S).

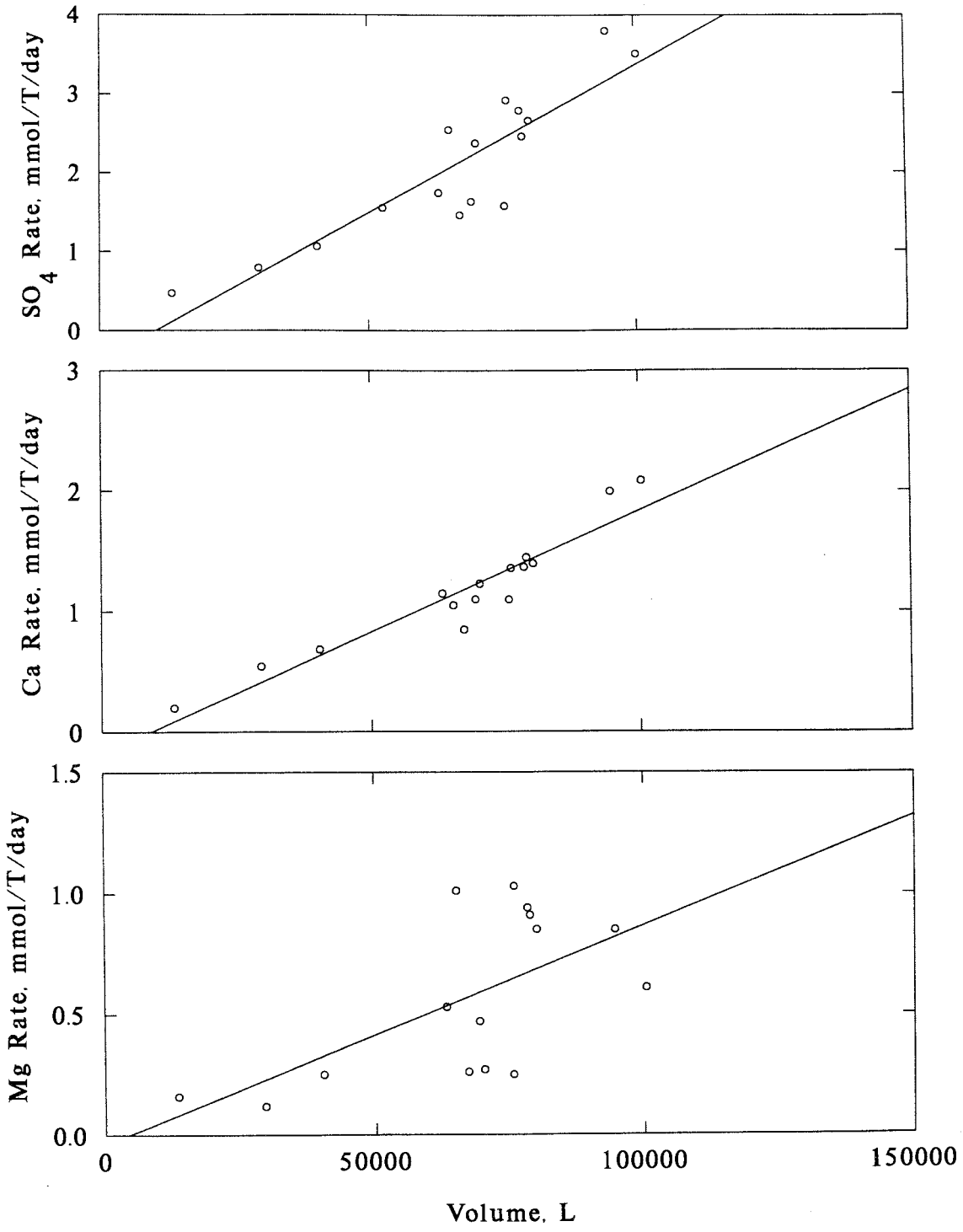

Figure A4.6. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL2 (0.63% S).

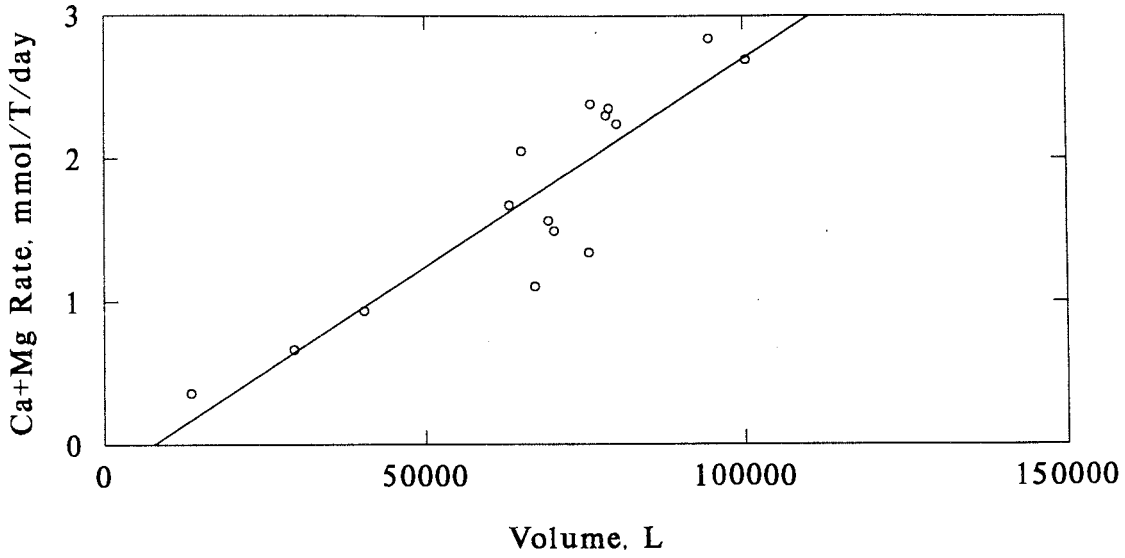

Figure A4.7. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL3 (0.63% S).

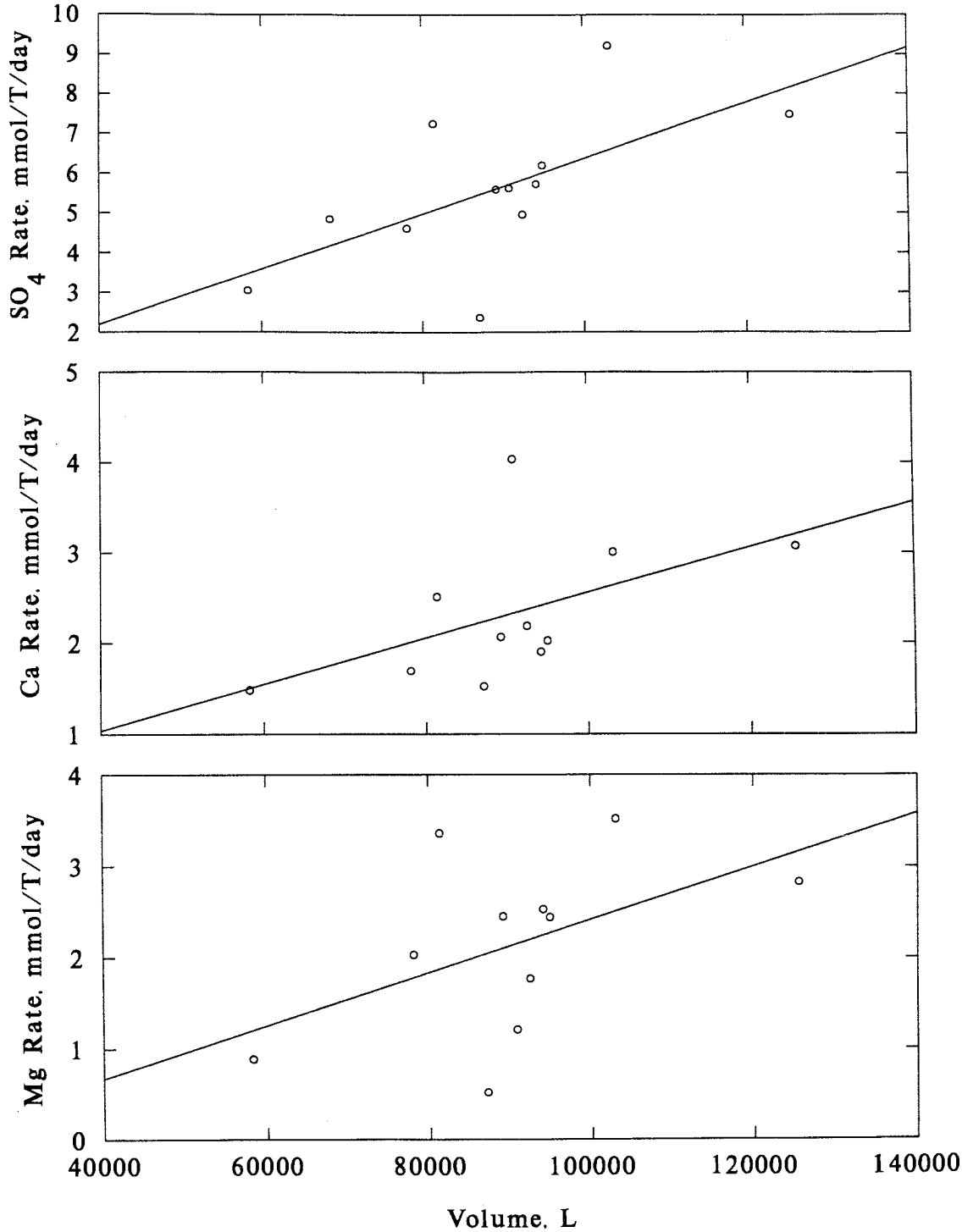

Figure A4.8. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL3 (0.63% S).

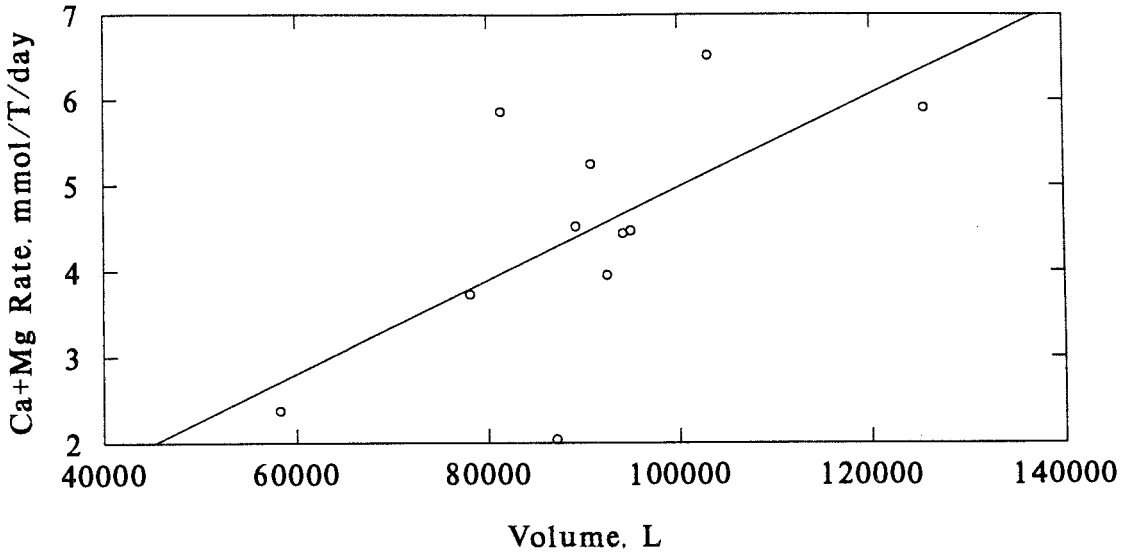

Figure A4.9. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL6 (0.79% S).

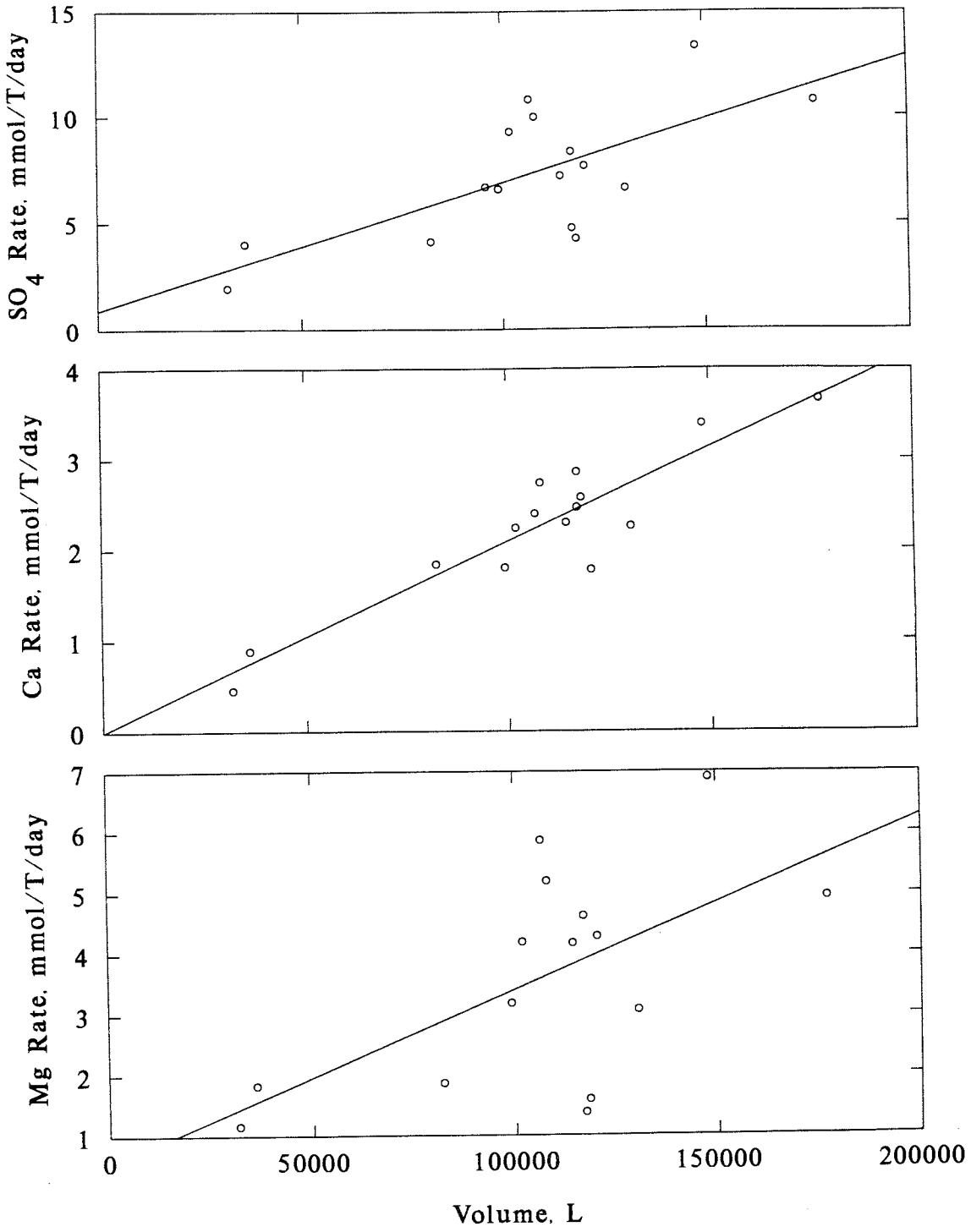

Figure A4.10. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL6 (0.79% S).

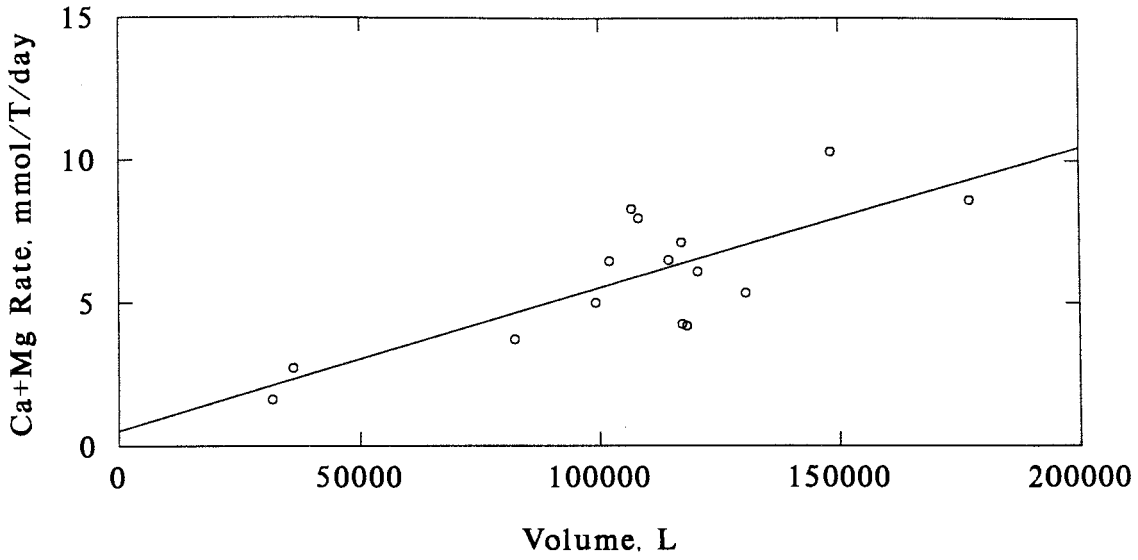

Figure A4.11. Annual rates of release of sulfate, calcium, and magnesium vs. annual drainage volume for Duluth Complex test pile FL5 (1.41% S).

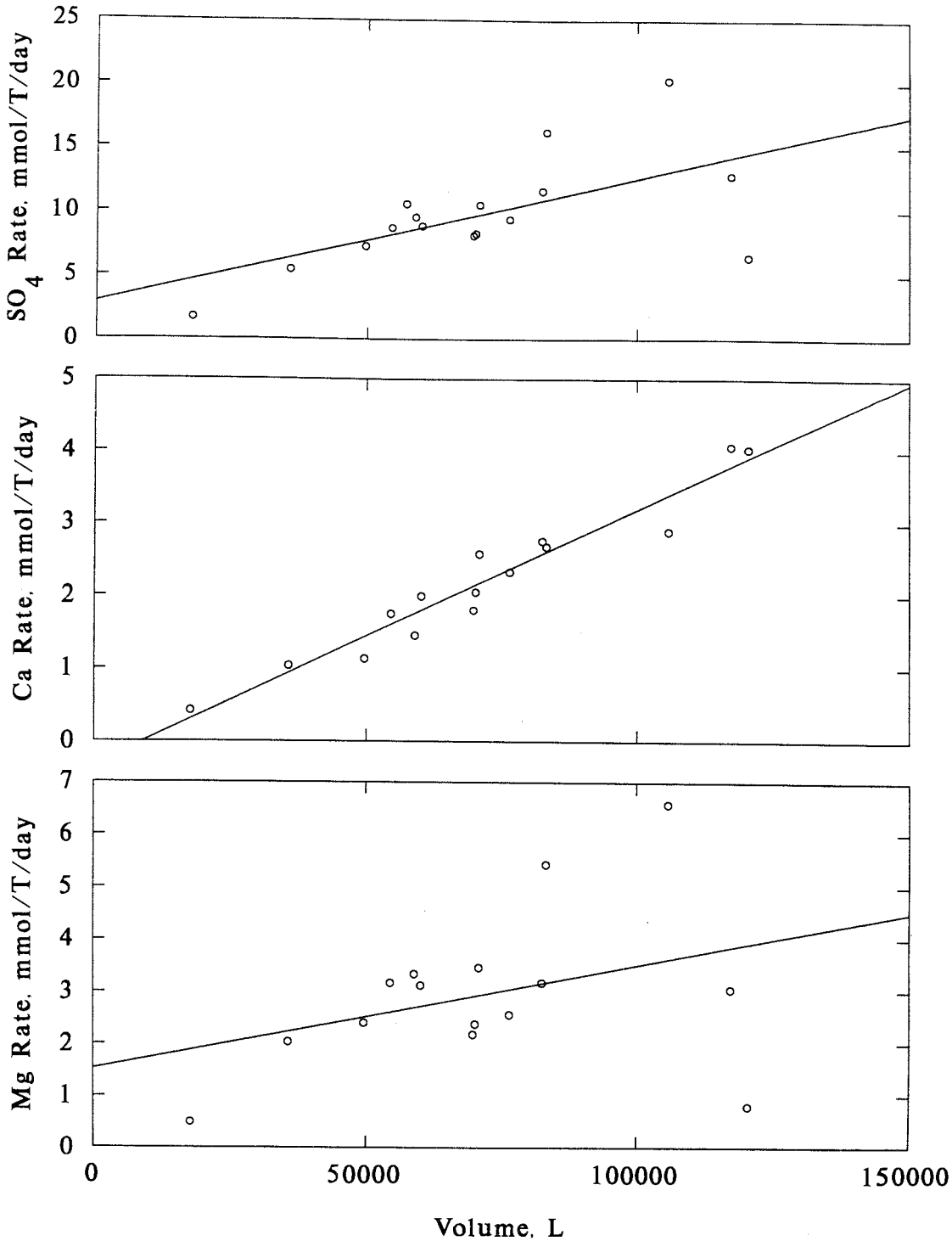

Figure A4.12. Annual rates of release of calcium plus magnesium vs. annual drainage volume for Duluth Complex test pile FL5 (1.41% S).

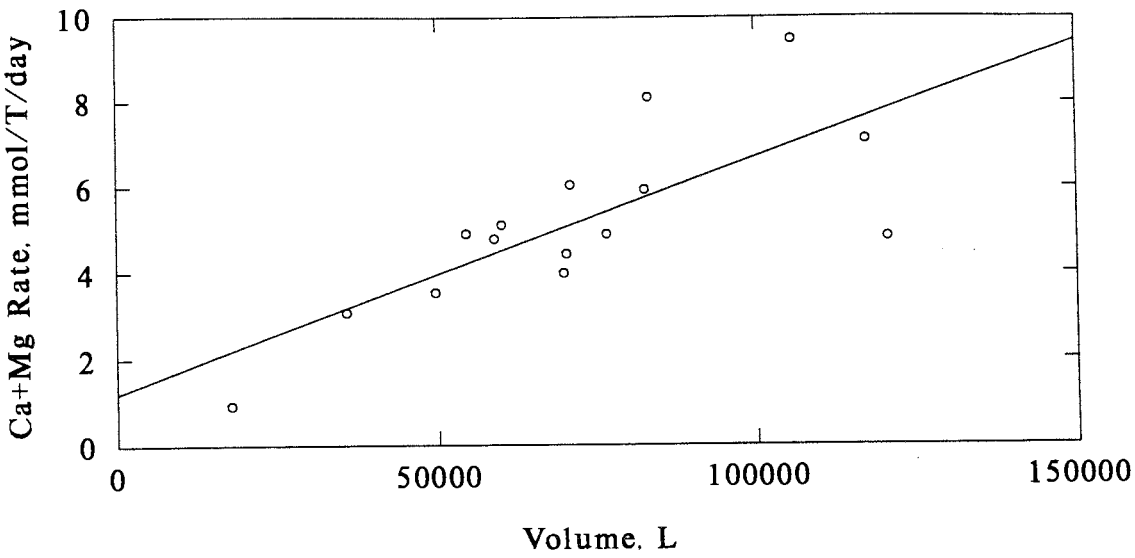

Figure A4.13. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL1 (0.63% S).

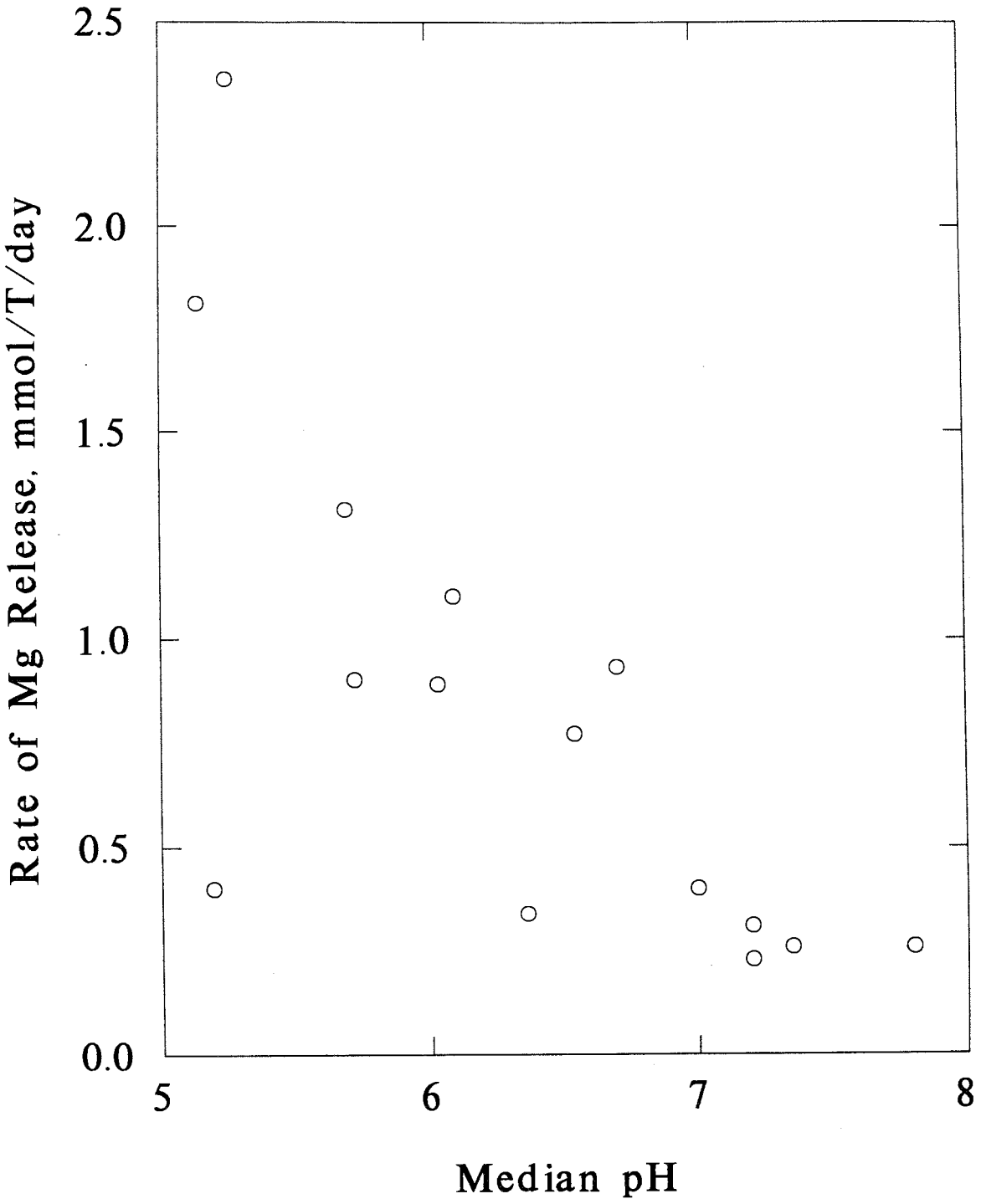

Figure A4.14. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL2 (0.63% S).

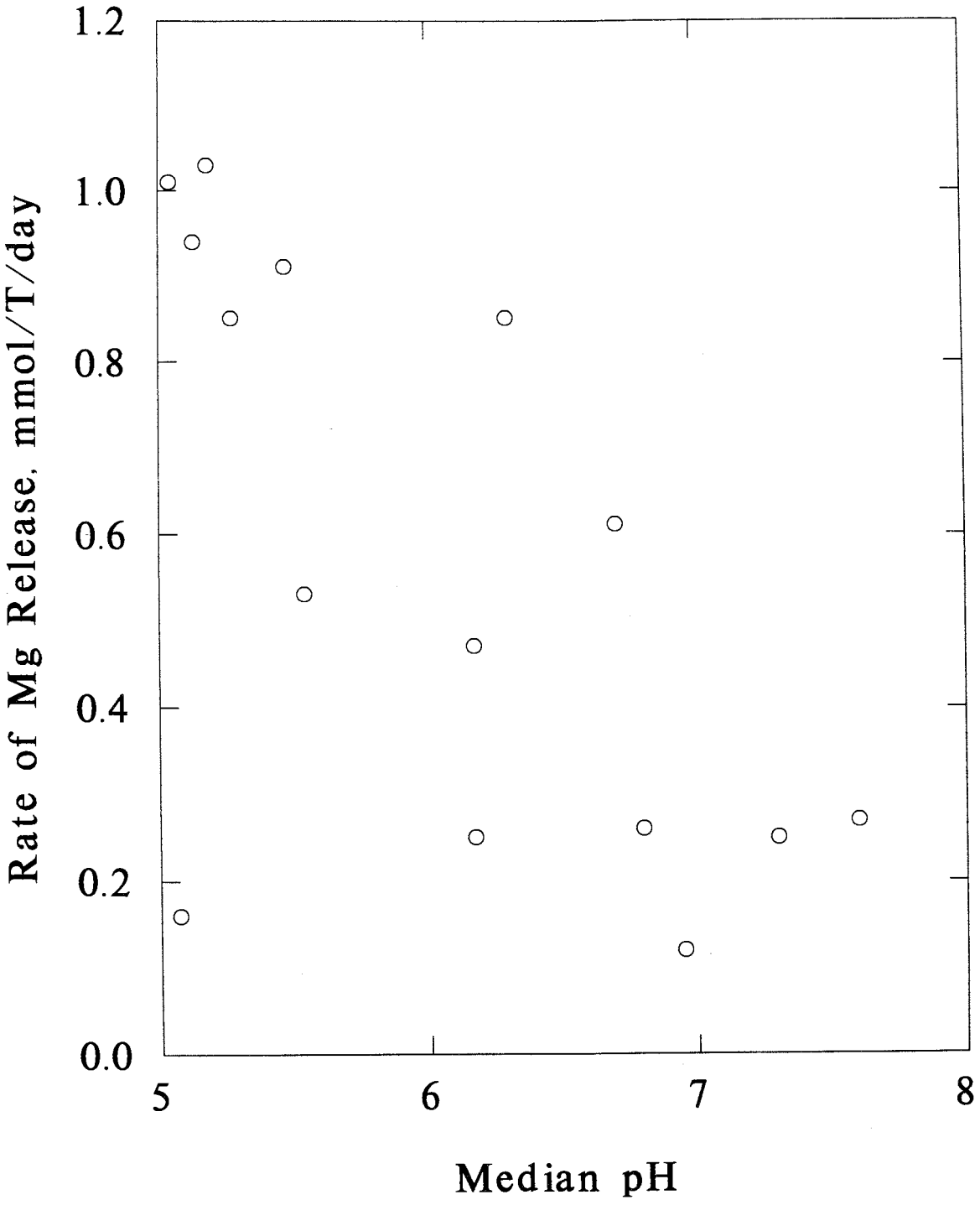

Figure A4.15. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL3 (0.63% S).

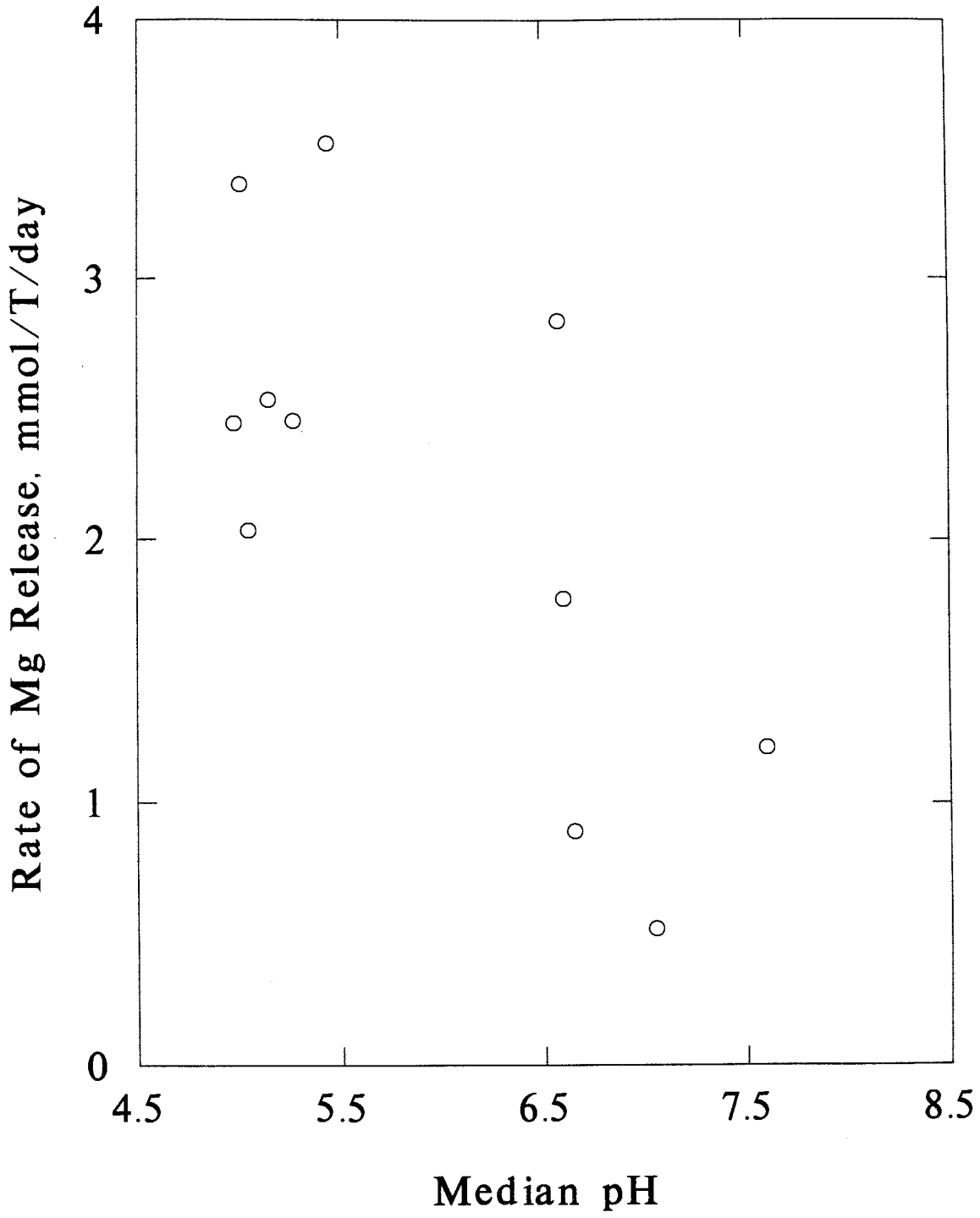

Figure A4.16. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL6 (0.79% S).

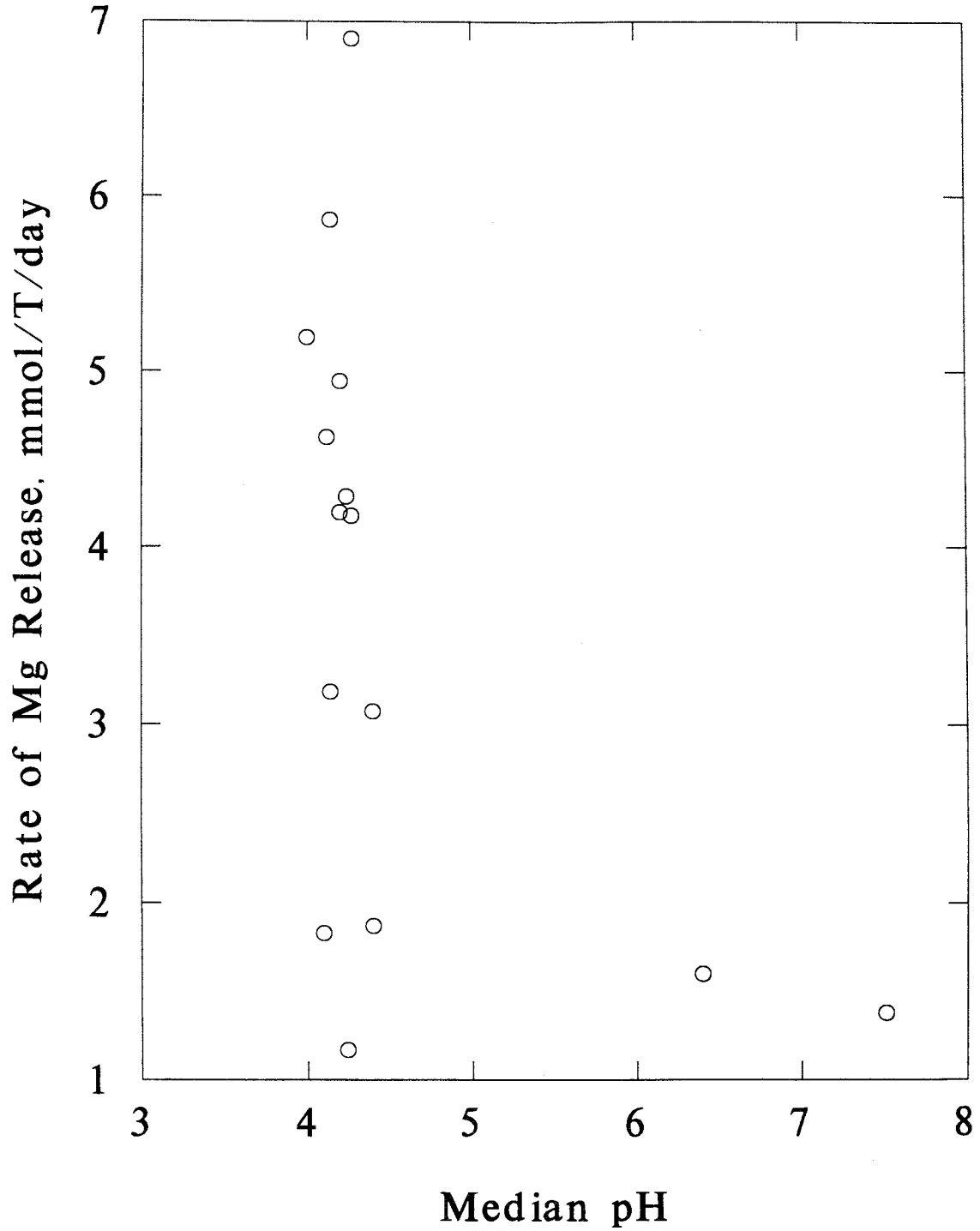

Figure A4.17. Annual magnesium release rate vs. annual median pH for Duluth Complex test pile FL5 (1.41% S).

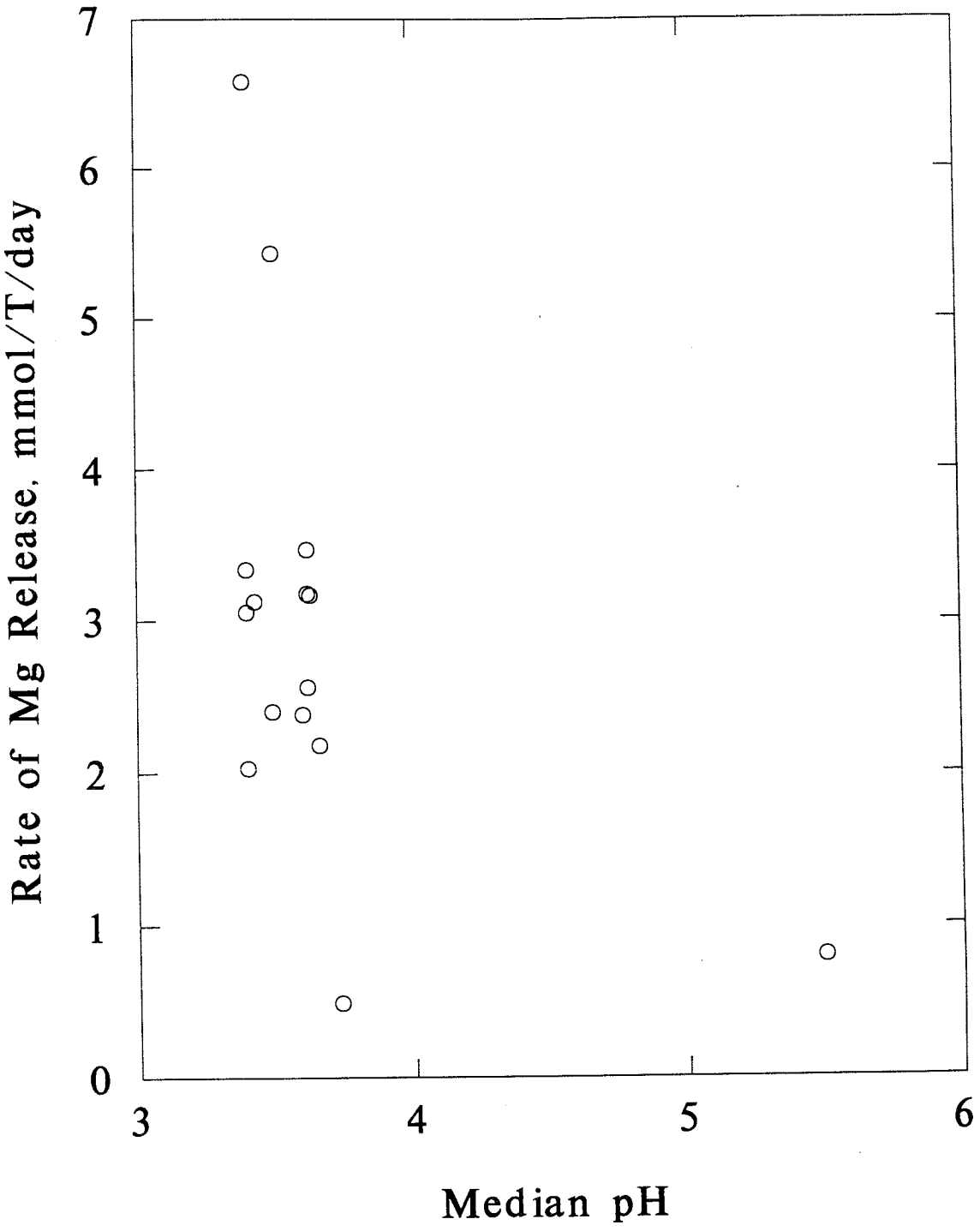

Figure A4.18. Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, and FL3.

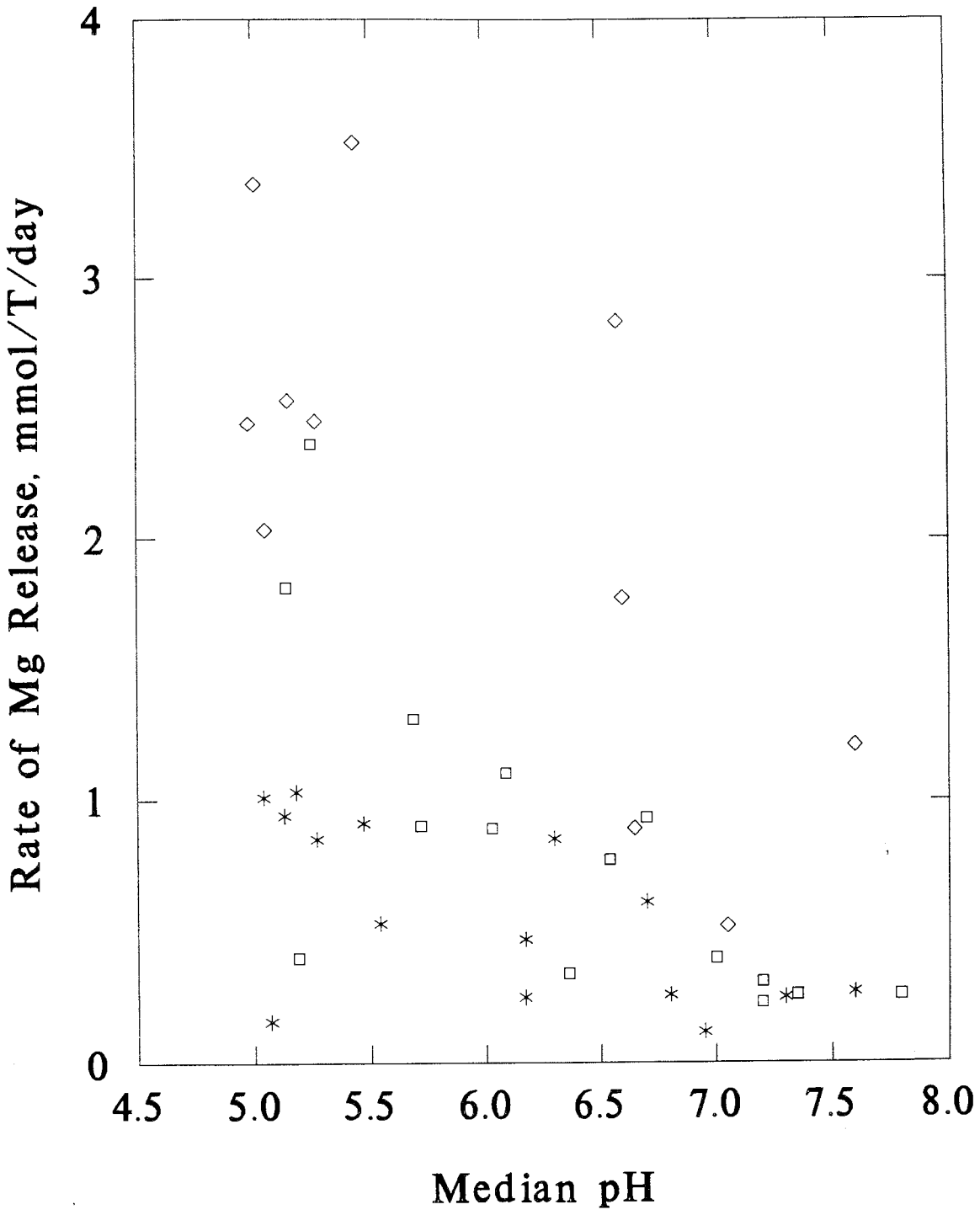

Figure A4.19. Annual magnesium release rate vs. annual median pH for Duluth Complex test piles FL1, FL2, FL3, FL6, and FL5.

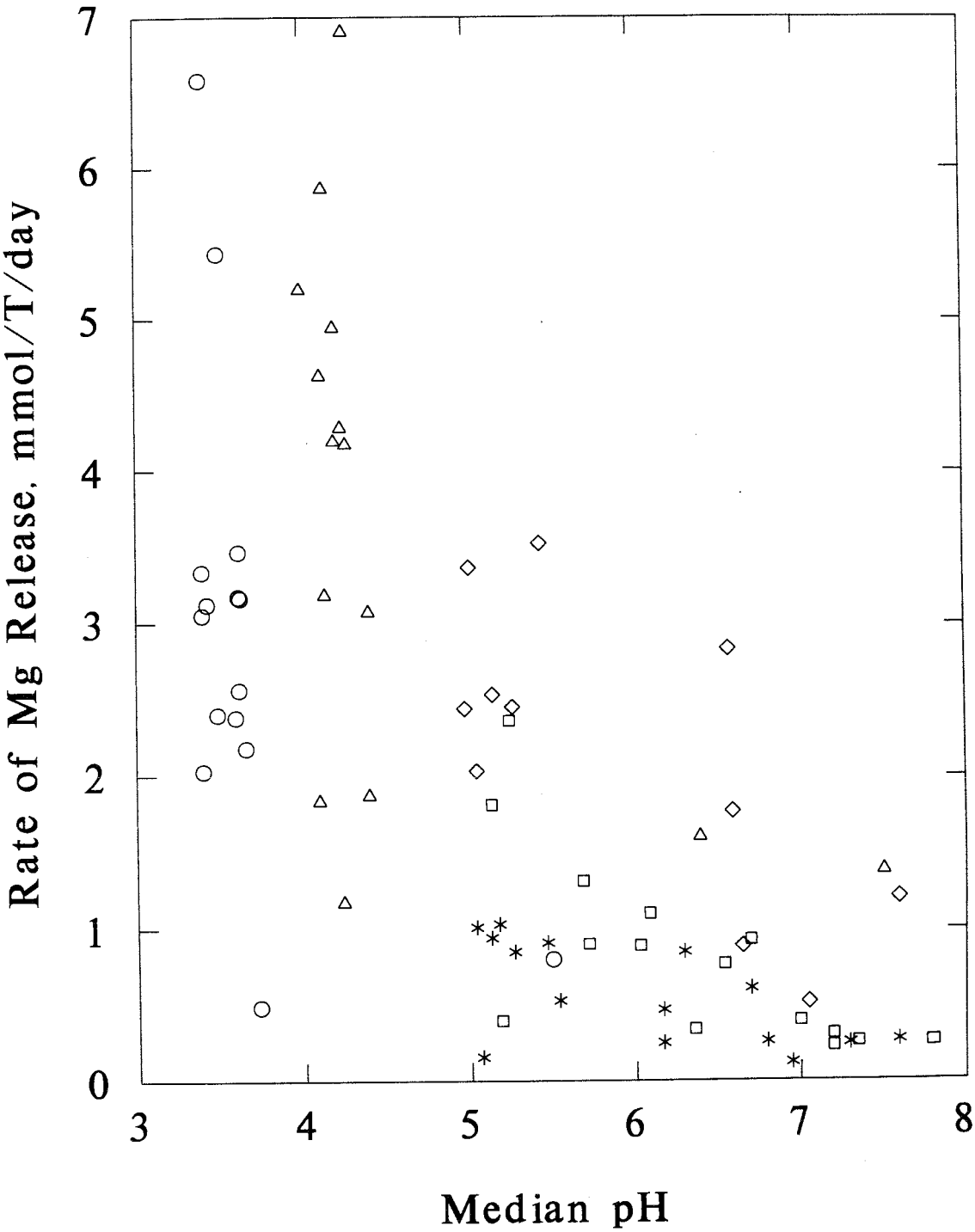

Table A4.1. Annual mass release rate vs. annual drainage volume regression statistics.

Sulfur %	Pile	Mass Release Rate (mmol/T/day) = a • volume (L) + b											
		Sulfate			Calcium			Magnesium			Calcium plus Magnesium		
		$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n	$a \times 10^5$	r^2	n
0.63	FL1	3.8	0.649	16	2.3	0.882	15	1.2	0.339	15	3.5	0.730	15
0.63	FL2	3.8	0.801	16	2.0	0.894	15	0.9	0.387	15	2.9	0.805	15
0.63	FL3	7.0	0.398	12	2.5	0.600	10[1]	2.9	0.245	11	5.4	0.403	11
0.79	FL6	6.0	0.492	16	2.1	0.851	15	2.9	0.359	15	5.0	0.599	15
1.41	FL5	9.7	0.386	16	3.5	0.937	15	2.0	0.134	15	5.5	0.564	15

[1] Regression excluded the 1978 calcium release rate, which was anomalously high.

APPENDIX 5

NEUTRALIZATION POTENTIAL

Appendix 5: Neutralization Potential

Table A5.1. Neutralization potentials as determined by drainage quality of Duluth Complex test piles.

Table A5.1. Neutralization potentials as determined by drainage quality of Duluth Complex test piles.

Sulfur %	Pile	Method	Neutralization Potential, mg $CaCO_3$/g rock					
			pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
0.63	FL1	Ca+Mg[1]	0.346	0.781	>1.059	>1.059	>1.059	>1.059
		SO_4[2]	0.422	0.923	>1.259	>1.259	>1.259	>1.259
		avg.	0.384	0.852	>1.159	>1.159	>1.159	>1.159
0.63	FL2	Ca+Mg	0.179	0.393	>0.642	>0.642	>0.642	>0.642
		SO_4	0.228	0.498	>0.783	>0.783	>0.783	>0.783
		avg.	0.204	0.446	>0.712	>0.712	>0.712	>0.712
0.63	FL3	Ca+Mg	0.442	1.254	>1.340	>1.340	>1.340	>1.340
		SO_4	0.537	1.599	>1.721	>1.721	>1.721	>1.721
		avg.	0.490	1.426	>1.530	>1.530	>1.530	>1.530
0.79	FL6	Ca+Mg	0.157	0.172	0.226	1.702	>2.074	>2.074
		SO_4	0.175	0.191	0.246	2.102	>2.607	>2.607
		avg.	0.166	0.182	0.236	1.902	>2.340	>2.340
1.41	FL5	Ca+Mg	0.044	0.044	0.096	0.130[3]	0.136[3]	>1.889
		SO_4	0.080	0.080	0.135	0.189[3]	0.201[3]	>3.627
		avg.	0.062	0.062	0.116	0.160[3]	0.169[3]	>2.758

1 Neutralization potential (NP) was calculated using the cumulative calcium and magnesium release (see methods section of report).

2 Neutralization potential (NP) was calculated using the cumulative sulfate release (see methods section of report).

3 Neutralization potential was calculated as of 1978, although higher pH values were observed later. Two pH values of 4.6 and 4.0 were observed in spring runoff in 1980 and 1989, respectively. Five pH values between 4.0 and 4.4 were observed in 1992.

Attachment 11

Lapakko K, Antonson D, Engstrom J (2004) Analytical Screening of Abandoned Waste Rock Piles, Report to the US Army Corps of Engineers Contract/order No. DACW45-02-P-0212. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN.

ANALYTICAL SCREENING OF ABANDONED WASTE ROCK PILES

Report to the US Army Corps of Engineers
Contract/Order No.DACW45-02-P-0212

June 2004

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1. INTRODUCTION

Six test piles of Duluth Complex rock (820-1300 metric tons, 0.6-1.4% S) were constructed near Babbitt, Minnesota in 1977. The volume of drainage and drainage quality were determined with the objectives of determining the effects of sulfur content and reclamation treatments on drainage quality and chemical mass release. One pile was partially disassembled in September 1982 (FL4, 0.6% S). A bulk sample and spatially distributed samples were collected, but only the bulk sample was analyzed. Piles FL1, FL3, and FL6 were partially covered by plastic at the end of the 1989 field season, for the practical purpose of reducing the volume of drainage that the MN DNR was required to treat. The plastic covers were shredded during 1990 by exposure to the elements. A more robust Hypalon® cover was placed onto pile FL3, thereby terminating the flow at the end of the 1989 season. Drainages from the remaining four piles were monitored through 1993.

The test piles were disassembled in October 1994. During this process the Minnesota Department of Natural Resources (MN DNR) the US Bureau of Mines (USBM) Salt Lake Research Center cooperated to collect a bulk sample and spatially distributed samples from each of three test piles (FL1, 0.6%S; FL6, 0.8% S; FL5, 1.4% S). Some of the samples were analyzed with the objective of gaining insight into the solid-phases controlling chemical release, the long term behavior of abandoned mine wastes, and the extent of water contact with rock in the pile. The project was terminated due to the closure of the USBM in early 1996. None of the solid-phase data from the test pile samples had been reported.

The present project began 13 August 2002 and is described in this report. Sixteen spatially distributed samples from the 0.8% S test pile were subjected to chemical and mineralogical analyses, and geochemical modeling was conducted. In addition, the aforementioned solid-phase data from the cooperative MN DNR-USBM project are presented. These results are presented in the context of information previously collected from the test piles. The information collected will provide a foundation for a more intensive analytical program.

2. OBJECTIVES

There are two major objectives of this study. First, determine the extent of water contact with rock in the pile. More generally this is related to the question of how much of the rock in the pile contributes to solute release. The second objective is to provide insight into the solid-phases controlling chemical release. These objectives will be addressed using existing drainage quality data, compiling solid-phase data generated in the MN DNR/USBM cooperative study, and solid-phase analyses and geochemical modeling conducted in the course of the present study.

3. BACKGROUND

The extent to which waste rock constituents are released with drainage is largely a function of rates of mineral dissolution and the extent of subsequent transport of dissolved components. The rate at which a given constituent is released from the rock matrix is related to the sum of the dissolution rates of minerals in which the constituent is present. Dissolution rates vary among minerals and

each rate is dependent on numerous factors. Factors that influence most mineral dissolution rates include the available mineral surface area, and this depends on the mode of mineral occurrence within the rock matrix (i.e. included within another mineral, interstitial to other minerals, liberated from the rock matrix), mineral grain size, and roughness of the surface. The reactivity of the surface is affected by the extent of mineral surface coating. Other factors influencing mineral dissolution rates include the availability of reactants (e.g. ferric iron or oxygen as an oxidant for sulfide mineral oxidation, hydrogen ion for silicate mineral dissolution), the presence of reaction inhibiting ions in solution, and reaction conditions (e.g. temperature).

The extent to which components, released during mineral dissolution, are transported from the pile is dependent on both chemical and physical factors. The solubility of the specific component with respect to the chemistry of the water will influence transport. One example, central to acid generation, is pyrite oxidation under circumneutral pH conditions. Both iron and sulfur are released during oxidation. Iron is oxidized from the ferrous to ferric state and precipitates from solution as ferric hydroxide. In contrast, sulfur is oxidized to sulfate, which is fairly soluble, and remains in solution. Consequently, the extent of iron transport from the pile is very low, while virtually all sulfate contacted by flow through a pile may be transported to the environment with drainage from the pile. (This ignores potential precipitation of sulfate as gypsum or jarosite.) Thus, although both are released from the same mineral, water that contacts the reaction site will transport the more soluble form of sulfur (sulfate) but not iron, due to differences in sulfate and iron solubilities. The availability of water as a transport medium also influences transport. If no flow contacts a reaction site, dissolved components will remain in place.

4. INFORMATION GENERATED PRIOR TO 1994

4.1. Analyses of Unleached Solids

4.1.1 AMAX Geological Rock Identification

The Duluth Complex rock analyzed in the present study was collected from three test piles and the back section of an experimental bin identified as 6B, all of which were located at the Minnesota Department of Natural Resources (MN DNR) research site near Babbitt in northeastern Minnesota. The results presented focus on the test piles. The experimental bin was loaded with about 30 metric tons of rock taken from test pile FL4 in 1982. The rock mass in this bin was much smaller than the roughly 1000 metric tons in the test piles. Due largely to the difference in experimental approaches the bin drainage quality is not discussed, and solid-phase results are incorporated only for purposes of comparison.

The rock for all of the plots was obtained during development of an exploration shaft and associated drifts identified as A, B, and C. The exploration shaft was excavated in the Partridge River Intrusion of the Duluth Complex near Babbitt, Minnesota. This deposit has been referred to as the Babbitt, Minnamax, and most recently as the Mesaba deposit. The exploration shaft was completed during 1976. In 1977, 3800 feet of drifts were driven at the 1700 foot level into the Tiger Boy (drift A) and Local Boy (drifts B, C) areas of the deposit. Bear Creek Mining (1969-70) and AMAX (1974)

exploration drill holes were used to determine the location of the exploration shaft. The exploration drill core was visually examined by geologists and observations were recorded.

Piles FL1 through FL4 were constructed of rock excavated from the exploration shaft, and piles FL5 and FL6 contain material from underground drifts A, B, and C (Table 1). During excavation of the shaft and the subsequent drifts, the rock excavated was described by blast round. For each round blasted, the rock type was described and presented in monthly reports from 1976 to 1977. The sulfur, copper, and nickel contents and rock types for the blast rounds are summarized in Tables 2, 3, 4). Piles FL1, FL6, and FL5 were constructed of rock with average total sulfur contents of 0.6%, 0.8%, and 1.4%, respectively (Table 5). Like FL1, three additional piles (FL2, FL3, FL4) were constructed of rock with an average total sulfur content of 0.6%. Each pile was underlain by an impermeable Hypalon® liner, which sloped toward a perforated plastic pipe for collection of drainage (Figure 1).

4.1.2. Exploration Phase Chemistry

The drill core was also analyzed for copper, nickel, and sulfur. Descriptions and chemistry of drill core are available at the MN DNR Hibbing core library.

4.1.3. FL1 Grab Sample

In the early 1980's, a grab sample was taken from FL1 and analyzed for chemistry and mineralogy (Tables 6, 7, respectively).

4.2. Sampling of FL4 Leached Rock in 1982

In September 1982, approximately 40% of pile FL4 was removed for use in six experimental bins designed to evaluate various treatment methods (Lapakko et al. 1986). A representative sample, weighing over three tons, was collected for analysis of particle size distribution, specific surface area, chemistry, and mineralogy. Particle size, surface area and chemistry of this sample are shown in Table 8 and Appendix 2, Table A2.3.

X-ray diffraction (XRD) analyses were conducted on the 0.053 to 0.075 mm, the 0.075 to 0.0105 mm, and the 0.0105 to 0.149 mm size factions of the gabbro material. Approximately half of this material was plagioclase (with a Na:Ca ratio of about 2:3), and half was a mixture of olivine (with a Fe:Mg ratio of 1:4 to 1:9), pyroxene, and biotite. Lesser amounts of magnetite, amphibole, chlorite, and possibly smectite and/or vermiculite were also identified. The phyllosilicate minerals (biotite, chlorite, smectite, and vermiculite) were more common in the finer size fractions. The FL4 rock samples analyzed in the present study were collected from the back segment of one of the experimental control bins (6B).

4.3. Drainage Quantity and Quality

Several reports were written on the early phases of the test pile study (AMAX Environmental Services 1978; Eger et al. 1979, 1980, 1981; Eger and Lapakko 1985). Lapakko (1993) presented test pile drainage quality data from 1978 through 1993. Lapakko (1994) compared data from the piles to that generated in laboratory experiments. Studies on passive mitigation were also conducted at the site by the mid-1980s (Eger et al. 1980b, 1984; Lapakko et al. 1983, 1986) and these results are not discussed in the present report.

5. METHODS OF MN DNR/USBM TEST PILE SAMPLING (1994)

5.1. Sample Collection

5.1.1. Bulk Samples

In 1994, all six test piles were removed and reclaimed. Piles FL1 (0.6% S), FL6 (0.8% S), and FL5 (1.4% S) were disassembled in horizontal layers or lifts (Figure 2). During the dismantling of piles FL1, FL6 and FL5, a bulk sample was collected by dragging the 3.5-yard backhoe bucket across each lift. The rock collected, approximately 2 to 2.5 tons per pile, was placed on a plastic liner and assumed to be representative of the pile. A bulk sample was also collected from experimental bin 6 using this technique.

5.1.2. Spatially Distributed Samples

The sampling objective was to collect from 16 spatially-distributed sites on the surface of each lift (Figure 3). Each of the three piles (FL1, FL5, and FL6) was staked at four corners so that the rectangular dimensions (approximately 80 x 40 feet) and orientation of each pile could be determined (Figure 4). Distances and bearings between each corner were determined using a Brunton pocket transit and a 100-foot tape measure. These readings were used to locate the pile and describe the sample locations in space. Dimensions were determined more accurately by measuring the perpendicular distance between the edge of the test-pile synthetic liner and the perimeter of the pile.

Sample grids, covering approximately half of the top of each lift, were established to identify locations for sample collection. A 4x4 grid was preferred but in some cases the shape of the exposed surface dictated a 3x5 or 3x4 sample grid. The first sample-location grid was located at the top of the pile. Additional grids were located on freshly-exposed surfaces created by removal of three successive lifts (Figure 5). Each lift was approximately three feet thick.

The individual grid-point samples were collected by shovel, and pick when necessary, from an area 30 to 40 centimeters in diameter and 20 to 30 centimeters deep. The sample typically filled a five-gallon plastic bucket and, even when the rock was strongly cemented, the sample volume was at least 2.5 gallons. The sample number was noted on the bucket lid, and a card was place inside each bucket identifying the sample number and any occurrences of note. Any unusual specimens were

marked, separated, and placed in either a plastic bag or an envelope. At the time of sampling, the appearances of the holes from which the spatially distributed samples were removed, the +1/4-inch fraction, and the −1/4-inch fraction of each spatially-distributed sample site were described by the sampler as to color, mineral occurrences, surface coatings, extent of oxidation, visible moisture, and the extent of compaction and cementation.

Sample numbers were assigned using three numbers that were separated by hyphens. The first number represented the pile number (1, 5, or 6), and the second number represented the lift number (1 through 4). Lift 1 was the original exposed surface at the top of the pile. Lifts 2 through 4 were the three successive surfaces exposed by the backhoe. The third number of the sample number represented the grid number of the sample collected on the lift surface (1 through XX, where XX was the number of samples collected on the lift). Grid-numbering sequence started with 1 at the southeast corner of the grid and proceeded to XX at the northeast corner.

Two additional spatially-distributed samples were collected at every other grid point, and numbered as described above. One sample was collected to determine the amount of acid present on the rock-particle surface (acid concentration present) and a second sample was collected for bacterial characterization.

5.2. Particle Size Distribution Analysis

5.2.1. Bulk Sample

Particle size distribution for the test piles was determined on a composite of samples collected by dragging the 3.5-yard backhoe bucket across each lift. The entire bulk sample from FL5 and FL6 and half of the sample from FL1 were used for determination of particle size. Particle size distribution for the joint disposal bin 6B was determined on a composite of samples collected by random backhoe buckets taken from the surface, center, and base of the bin. These samples, approximately one half to one ton in weight, were also placed on a plastic liner and assumed to be representative of the bin. The entire bulk sample from bin 6B was used to determine particle size.

The particle size analysis for the +½-inch fraction was performed on site by the Minnesota Department of Natural Resources, Division of Minerals (Hibbing, MN). Rock larger than twelve inches was presorted and weighed. The remaining rock (−12-inch) was screened through a four-foot by five-foot stationary grizzley with mesh sizes of six and three inches. The +6 and +3-inch rock was weighed and a five gallon representative sample was collected for further analysis. The −3-inch rock was then screened through a Nordbery Universal vibrating screener (type NU9) equipped with screen sizes of ½, ¾, and 1½ inches. Each size fraction was weighed and a representative five gallon sample was collected for further analysis.

A ten gallon representative sample of the −½-inch rock was delivered to Midland Research Center (Nashwauk, MN) for further particle size determination. Prior to particle size analysis of the −½-inch fraction, Midland Research determined the percent moisture by weight on the split portion of the −½-inch sample (see above). The sample was dried at 105°C. The percent moisture by weight

was calculated as in equation 1. This information was of marginal value because it was at times raining during sample collection and sample storage in the field prior to shipping for analysis.

$$\text{Percent moisture} = 100 \times (\text{wet weight} - \text{dry weight})/\text{dry weight} \quad (1)$$

Approximately one half of the −½-inch fraction, as received, was dried at 105 °C. The dried portion, about 10 kg, was screened on a ¼-inch (6.4 mm) sieve. A split of the resulting minus ¼-inch fraction, about 1 kg, was rotapped through the finer sieves (2.0, 0.5, 0.177, 0.149, 0.105, 0.075, 0.053 mm). The weight, in grams, of the +6.4 mm fraction was calculated as the difference in weights of the −½-inch fraction and the material finer than 6.4 mm. Assay pulps were split using a Sepor riffle splitter and submitted to Lerch Brothers (Hibbing, Minnesota) for sulfur and sulfate analyses, and to Chemex (Vancouver, Canada) for whole rock and trace metal analyses.

5.2.2. Spatially Distributed Samples

Sixteen samples were sent to the U.S. Bureau of Mines Salt Lake City Research Center (Salt Lake City, Utah; USBM). The corresponding sample numbers were: 6-1-02, 6-1-04, 6-1-05, 6-1-07, 6-2-02, 6-2-04, 6-2-07, 6-2-10, 6-3-01, 6-3-06, 6-3-08, 6-3-16, 6-4-02, 6-4-04, 6-4-05, and 6-4-07. All sixteen samples were initially screened into fourteen size fractions: +6", −6"/+3", −3"/ +1½", −1½"/+¾", −¾"/+½", −½"/+¼", −¼"/+10 mesh, −10/+30 mesh, −30/+70 mesh, −70/+100 mesh, −100/+150 mesh, −150/+200 mesh, −200/+270 mesh, and −270 mesh (Tables A5.6.1 - A5.6.16). Material coarser than +¼ inch was sieved by hand and that finer than +¼ inch was sieved using a RoTap.

5.3. Specific Surface Area of the Bulk Samples

Specific surface area of size fractions finer than 2.0 mm was determined by the Department of Soil, Water, and Climate at the University of Minnesota (Minneapolis, MN) under the direction of Paul Bloom. The method used at the University was BET nitrogen adsorption. Upon warming, nitrogen adsorbed onto the sample desorbed into the flow stream. A thermal conductivity detector sensed the change in concentration and gave a signal in proportion to the amount of nitrogen held on the surface and then desorbed. The results were given in square meters per gram.

5.4. Chemistry

5.4.1. Bulk Samples

5.4.1.1. Sulfur and Sulfate for the Bulk Samples

Lerch Brothers Inc. analyzed the −6" samples for sulfur and sulfate by particle size. They determined total sulfur with a carbon rod furnace (Dietert) using ASTM 395-70 and sulfate was determined following a sodium carbonate leach using ASTM D24-92.

5.4.1.2. Whole Rock Analysis

Whole rock analysis was performed by Chemex (Sparks, NV). A prepared 0.100 g sample was added to lithium metaborate flux (0.70 g), mixed well, and fused in a furnace at 1050°C. The resulting melt was then cooled and dissolved in 100 mL of 4% nitric acid. This solution was analyzed by inductively-coupled plasma-atomic emission spectroscopy (ICP-AES). The results were corrected for spectral inter-element interferences. The detection limit was 0.01% for each of the oxides (SiO_2, MgO, Al_2O_3, TiO_2, MnO, P_2O_5, Na_2O, Fe_2O_3, CaO, Cr_2O_3, and K_2O).

Loss on ignition was conducted by drying a porcelain crucible in an oven at 105°C, cooling the crucible and recoding the weight. A prepared 1.00 g sample was added to the crucible and then ashed at 1000°C for one hour. The sample was then cooled in a desiccator, weighed, and percent loss on ignition (L.O.I.) was calculated. The detection limit was 0.01%.

5.4.1.3. Trace Metals

Geochemical analysis of the −6" bulk samples for silver, cobalt, copper, iron, manganese, molybdenum, nickel, lead, and zinc was performed by Chemex. A prepared sample (1.0 g) was digested with concentrated nitric and aqua regia acids at medium heat for two hours. The acid solution was diluted to 25 mL with demineralized water, mixed, and analyzed using an inductively coupled plasma (ICP) spectrometer after calibration with proper standards. The analytical results were corrected for spectral inter-element interferences.

5.4.2. Spatially Distributed Samples: −6.4 mm fraction

5.4.2.1. Sample Preparation

One split of the −1/4-inch material from each of the 16 spatially distributed samples was analyzed for acid concentration present and paste pH. The remaining 1/4-inch fraction was then separated into eight size fractions for additional chemical analysis. Each of the eight particle size fractions from the sixteen spatially distributed samples was analyzed for sulfur and carbon species, whole rock components, and trace metal chemistry. These analyses were conducted at the U.S. Bureau of Mines Salt Lake City Research Center under the direction of Bill White.

Screen sizes from −¼"/+10 mesh to −100/+150 mesh were split into two parts. One half of each sample was split using a rotary splitter into twelve approximately equal parts. Thirty to fifty grams of this portion was pulverized to 80% passing 150 mesh screen using a ring and puck pulverizer. Five to ten grams of this material was then submitted to the USBM analytical lab for characterization and chemical analysis. The specific analyses performed were 1) acid concentration present and paste pH, 2) carbon and sulfur, 3) whole rock chemistry, and 4) trace metals (Cu, Ni, Co, Zn, Pb).

Size fractions −150/+200 mesh to −270 mesh were not halved and were directly split into twelve approximately equal parts on the rotary splitter. These samples did not require pulverization and

the amount of sample was relatively small. Five to ten gram samples of these splits were submitted for analysis as noted above.

5.4.2.2. Acid Concentration Present (ACP) and Paste pH

The samples collected for acid concentration present were analyzed at the U.S. Bureau of Mines Salt Lake City Research Center using a method designed by Newmont Metallurgical Services. From the −¼" fraction, a 20 gram sample mass was added to 50 mL deionized water. The resulting mixture was shaken for a one minute period, allowed to sit for 15 minutes, and this process repeated. The bottle was shaken for a third time and then allowed to sit for at least five minutes to settle the suspended solids. The pH of the solution was determined at this point and reported as the paste pH. Then an aliquot of the liquid was measured into a fixed volume titration vial and titrated to pH 8.3 using a standardized sodium hydroxide solution. The resulting sample acidity was recorded as ppm calcium carbonate equivalent per gram of mine waste.

5.4.2.3. Carbon and Sulfur

Total carbon and total sulfur analyses on the −0.25-inch fraction were conducted by the U.S. Bureau of Mines, Salt Lake City, Utah. These analyses were determined by a LECO combustion-infrared spectrophotometer (Model SC-444). Sample duplicates were subjected to partial-decomposition procedures, including pyrolysis (muffle roast at 550°C) and acid decomposition (either hydrochloric or nitric acid). The muffle-furnace roast drove off sulfide sulfur as sulfur dioxide, and organic carbon as carbon dioxide. The pretreated sample was analyzed for carbon and sulfur, and these results were subtracted from the total carbon and sulfur results of a non-treated sample split. The remaining sulfur determined in the pretreated split was assumed to be sulfate sulfur, and the remaining carbon was considered to be carbonate.

5.4.2.4. Whole Rock Chemistry

Whole rock chemistry was determined by several different methods. Silicon dioxide content of the particle size fractions of the 16 spatially distributed samples was determined by a gravimetric method at the U.S. Bureau of Mines, Salt Lake City, Utah. The sample was digested and the remaining solid was roasted at 900°C. The resulting sample weight was subtracted from the pre-roasted weight. The difference was the amount of silicon dioxide (in percent).

Aluminum, calcium, iron, magnesium, and manganese concentrations were determined using alkaline fusion and acid digestion, followed by inductively coupled plasma emission spectrometery (ICP-ES). A 0.5 g sample was pulverized to 80% passing 150 mesh. Sodium carbonate, sodium peroxide, and sodium hydroxide were added, the mixture fused at 650°C for one hour, and then cooled. Deionized water, hydrochloric acid, and nitric acid were added. The resulting solution was boiled, hydrofluoric acid added, and the solution decanted. Deionized water was added to obtain a solution volume of 250 mL. This solution was submitted for analysis by ICP-ES (Jarrel Ash fixed channel monochrometer). In addition to the aforementioned components, this approach also

determined concentrations of boron, barium, cadmium, cobalt, chromium, copper, molybdenum, nickel, lead, and zinc. These values are presented in the appendices but are not discussed in the text.

To determine sodium and potassium concentrations, rock samples were subjected to a four-acid digestion (nitric, hydrochloric, hydrofluoric, perchloric), similar to that described by Chou and Sanzalone (1992). The solution was analyzed by Perkin Elmer atomic absorption (AA) spectrophotometer with both flame and graphite furnace capacities.

5.4.2.5. Trace Metal Chemistry

The eight particle size fractions of the sixteen samples were analyzed for cobalt, copper, nickel, lead, and zinc using the same approach reported above for sodium and potassium.

X-ray fluorescence (XRF) analysis was also conducted to determine S, Ca, K, Al, Si, Fe, Mn, Ti, Ba, Cr, Ni, Cu, Co, Zn, Rb, Sr, Y, Zr, Pb, La, and Ce. The 80% percent −150 mesh samples were ground using a mortar and pestle to −325 mesh. The resulting powder was placed in the cylindrical mount and analyzed by X-ray fluorescence (Kevex Energy Dispersive XRF). These data were not presented in the text as it was believed they were less accurate than the determinations described previously. The data are available at the MN DNR.

5.4.3. Hand-Sized (−3/+1.5 inch) Spatially Distributed Samples

Hand-sized (−3/+1.5-inch) spatially distributed samples were analyzed for total sulfur, sulfate, sulfide, total carbon, evolved carbon dioxide, whole rock chemistry, and trace metal content by ACME Analytical Laboratories Ltd. (Vancouver, B.C.). The following method descriptions were summarized from ACME Analytical Laboratories Ltd. (2003).

5.4.3.1. Sulfur and Carbon Species

Total carbon and sulfur were determined using a Leco C244 Carbon-Sulfur analyzer. Samples were ignited at 1650ºC in an induction furnace, and the resultant gases were analyzed for sulfur and carbon by infrared absorption. For sulfide determination, the sample was crushed and pulverized to (−100 mesh, −150 microns). One sample split (0.1 g) was placed in a Leco C244 Carbon-Sulfur analyzer and analyzed for total sulfur. A second split from each sample was ignited at 800ºC for one hour to decompose sulfides and drive off the sulphur as SO_2. These splits, in which sulfur present was assumed present as sulfate, were then analyzed for total sulfur. Sulfide was determined by subtracting sulfate-sulfur from total sulfur.

For evolved CO_2 determination 0.2 g samples (−150 mesh, d < 100 microns) were placed in 500 mL polypropylene bottles to which 34 mL of water and 6 mL $HClO_4$ was added. The bottle was placed in a hot water bath (70º C) for 30 minutes, after which the gas evolved was analyzed for total carbon using a Leco 244 Carbon and Sulfur Analyzer.

5.4.3.2. Whole Rock Chemistry

For whole rock chemistry determination a 0.2-g aliquot, pulverized to 95% passing 150 mesh (100 microns), was mixed with 1.5 g of $LiBO_2$ flux, heated to 1050ºC for 15 minutes, and dissolved in 5% $HN0_3$. The resultant solution was analyzed by inductively coupled plasma emission spectrometry (ICP-ES; Jarrel Ash AtomComp Model 975) for the determination of 18 major oxides and elements. Loss on ignition (LOI) was determined by igniting a 1 g sample split at 950ºC for 90 minutes then measuring the weight loss. Ferrous iron was determined by digesting samples in 30% sulfuric- 48% hydrofluoric acid solution for 22 minutes at 95ºC. The resultant digestate was analyzed by potassium dichromate titration.

5.4.3.3. Trace Metal Chemistry

Trace metal determination was determined on one g samples pulverized to 95% passing 150 mesh (100 µm) in a mild-steel ring-and-puck mill. Samples were digested with 6 mL aqua regia (2:2:2 ACS grade HCL, ACS grade HNO_3, demineralized water) for one hour at 90 - 95ºC then diluted (20:1 mL/g final ratio). The digestate was analyzed using ICP-MS (Elan 6000 Mass Spectrometer).

5.5. Mineralogy

5.5.1. Bulk Samples

Five size fractions (−6" through −½ ") were delivered to and analyzed by Louis Mattson at Midland Research Center in Nashwauk, MN. The samples were dried and the −½ " fraction of each sample was dry screened into an additional nine size fractions. All size fractions greater than ¼" were crushed to minus ¼".

Visual and microscopic examination of the size fractions and chemical assays indicated the coarsest fraction (−6/+1/5 inch) was the least altered. The optical examinations and chemistry of this fraction were used to determine its mineral content. The composition of this fraction was used to approximate that of unweathered rock.

Mineralogical studies were conducted on the finer size fractions in an attempt to document alteration of primary minerals and identify new minerals formed while the samples were in the pile. Light and electron microscopy, x-ray diffraction, and thermal analysis were used at various stages. Each size fraction was examined by light microscopy and numerous grains were mounted for electron microscopy analysis. Due to the crushing of the +¼" fraction, many obvious alteration features were destroyed or reduced to very fine grained material.

5.5.2. Spatially Distributed Samples

5.5.2.1. X-ray Diffraction

X-ray diffraction (XRD) analysis was conducted by the U.S. Bureau of Mines, Salt Lake City, Utah. The 80% percent −150 mesh samples were ground using a mortar and pestle to −325 mesh. The resulting powder was pressed into a mount that was analyzed by a Phillips APD 3720 diffraction system. The results were presented as a list of dominant mineral species, with mineral formula and percent in the sample. Unfortunately these data were not interpreted by the time the USBM closed. The data sheets were retained and their interpretation will be pursued in the future.

5.5.2.2. Microprobe

The spatially distributed samples were examined in an electron microprobe study (Peter McSwiggen, McSwiggen & Associates, Minneapolis, MN). One objective of this study was to determine detailed characteristics, and the variability of these characteristics, of the rock when it was placed in the pile. A polished thin section was made from a rock selected from the 1.5 to 3 inch size fraction of each of the spatially distributed samples. The large pieces were used in an attempt to obtain a relatively unweathered sample that would approximate unleached rock. Each section was evaluated optically to determine the minerals present and their abundances. Electron microprobe analyses were then made on representative examples of each mineral to determine their elemental composition.

A second objective of the microprobe study was to evaluate the spatial variation of leaching in test pile FL6 (0.8% S). All 11 size fractions from three samples (6-2-4, 6-3-1, 6-4-2) were examined to determine features likely to reflect the extent of leaching and the appropriate size fraction for examination in the 13 remaining spatially distributed samples. The size fractions examined were −1.5/+0.75", −0.75/+0.5", −0.5/+0.25", −0.25"/+10 mesh, −10/+30 mesh, −30/+70 mesh, −70/+100 mesh, −100/+150 mesh, −150/+200 mesh, −200/+270 mesh, and −270 mesh.

For the three samples selected for extensive evaluation, polished grain mounts were made for each size fraction between 1.5 inches and −270. The loose material was poured into a plastic cup and impregnated with epoxy. The top surface of each mount was then polished and carbon coated. Each mount was evaluated using a JEOL 8600 Electron Microprobe. A backscattered electron image was collected at low magnification to give an overview of the sample's general appearance. Later backscattered electron images of representative sulfide minerals were collected to document the extent of alteration.

Sulfide mineral alteration was determined to most appropriately reflect the extent of leaching and a scale was established to quantify this extent. A leaching scale was established for each sulfide mineral based on the apparent extent of sulfide grain reaction. A 0-5 scale was established as follows:

0 - No alteration of any kind.
1 - Evidence of minor dissolution around the perimeter of the grain.
2 - The sulfide grains have developed a thin (< 10 microns) iron oxyhydroxide coating or have experienced significant dissolution.
3 - The sulfide grains have developed a thick (> 10 microns) iron oxyhydroxide coating.
4 - The sulfide grain has nearly reacted away.
5 - The sulfide grain has completely reacted away.

Once the individual sulfide minerals were evaluated, a second scheme was developed to provide a single leaching scale for the sample as a whole. For each sample an average value was recorded for each sulfide mineral present. These data were then used to determine a leaching factor for the sample as a whole (Table 9). Pyrite occurred in very low abundance and therefore was not included in the determination of the sample leaching factor.

Based on the examination of all size fraction in the three samples, the −30/+70 mesh fraction was selected as most appropriate for evaluating the extent of sulfide oxidation in the remaining spatially distributed samples. Most of the sulfide mineral grains were liberated or exposed in this size fraction. In larger fractions the sulfides were isolated within the silicate mineral matrix and therefore had not been exposed to leaching conditions. In smaller size fractions it was difficult to determine whether sulfide minerals had been leached or if they had never been present. The −30/+70 mesh fraction of the remaining 13 samples were examined using the same procedures described above.

5.6. Calculations

5.6.1. Mass Release and Rates of Release of Sulfate, Calcium, and Magnesium

The total volume, period of days of record, and number of cases were taken from Lapakko 1993, Tables 7 through 12. To each of these measurements, the total for the years of sampling were added at the bottom of each table by simply summing the volume, period, and number of cases in each year. Additionally, the annual mass release and rates of release of sulfate, calcium, and magnesium were also taken directly from these tables and thus the methods of calculation are as stated in Lapakko 1993. The total annual mass release of each of these parameters was calculated as the sum of the release for each year. All rates of release were calculated over the period of flow. The overall rates of release were calculated as the total annual mass release divided by the total number of days and by the tons of material in the pile.

5.6.2. Mass Release and Rates of Release of Copper, Nickel, Cobalt, and Zinc

Not all of the files from Lapakko 1993 were available to perform the calculations for copper, nickel, cobalt, and zinc. Adjustments were made as follows to calculate the mass release. The total volume was calculated as the sum of the volume measurements (as found in the Tables in section A2.5) made on each sampling date for each year. Number of days were tabulated starting with the date of the first sampling of a field season to the date of the last (as found in the Tables in section A5.2).

The annual mass releases of copper, nickel, cobalt, and zinc were calculated as the sum by year of the concentration in mg/L multiplied by the volume (L) reading for that same sampling date. Missing values were assumed to be zero for either the metal or the volume. The units were then converted to mmol. The rates of release were determined by dividing the mmol released per year of each metal by the number of tons in the pile and the number of days of the field season (number of days).

The total volumes and number of days in the field season did not directly correspond to the values in Lapakko 1993 (Tables 7-12) for the years 1978 to 1983. In all except one case, the newly calculated values were lower than Lapakko 1993. It was assumed that the values in Lapakko 1993 were accurate. Due to this underestimation of the mass release of these metals, the annual mass releases were adjusted for the higher flow. The calculated mass release was divided by the ratio of the 1993 total volume measurement and the recently calculated volume measurement. Rates of release for copper, nickel, cobalt, and zinc were calculated in the same manner as sulfate.

6. RESULTS

6.1. Introduction

Six Duluth Complex test piles were constructed from rock removed from an exploration shaft and the associated drifts. As rock was removed from the shaft and drifts, geologists visually determined the rock lithology, and rock samples from each round were analyzed for sulfur, copper, and nickel. The mass of rock placed into the piles was estimated based on the volume of rock removed from the shaft and by the surveyed volume of the piles.

The volume of drainage and the drainage quality were determined with the objectives of examining the effects of sulfur content and reclamation treatment on drainage quality and chemical mass release. The test piles typically produced drainage between the middle of March and the middle of November, with an average flow season of 245 days. One pile was partially disassembled in September 1982 (FL4, 0.6% S) for the purpose of constructing additional experimental field plots. At this time a bulk sample and spatially distributed samples were collected. Only the bulk sample was analyzed. The particle size distribution was determined, and the fractions finer than 2.0 mm were analyzed for paste pH, total sulfur, sulfate, copper, nickel, cobalt and zinc, as well as specific surface area. A second pile was covered with a geomembrane at the end of the 1989 field season (FL3, 0.6% S). Drainages from the remaining four piles were monitored through 1993 (Lapakko 1993) and all six piles were disassembled in 1994.

As the piles were being disassembled in October 1994, a bulk sample and spatially distributed samples were collected from three of the test piles (FL1, 0.6%S; FL6, 0.8% S; FL5, 1.4% S) in a cooperative project involving the MN DNR and US Bureau of Mines (USBM) Salt Lake Research Center. The particle size distribution of the bulk samples was determined. Chemical analysis was conducted on individual size fractions, and the specific surface area of fine fractions was determined. The mineral content of the coarse fraction was determined to estimate that of the unleached rock, and analyses were conducted to identify mineral reaction products present.

The spatially distributed samples were visually examined at the time of collection and observations were recorded. Particle size distribution was determined and the paste pH and acid concentration present were determined on the −0.25-inch fraction. Eight individual size fractions finer than 0.25 inches were analyzed for chemistry. The project was terminated due to the closure of the USBM in early 1996, and none of the solid-phase data were reported.

The present project began 13 August 2002, and this report summarizes information collected in the first stage of the study and conducted additional analyses (Table 10). Large particles from the spatially distributed samples were analyzed to determine their chemistry, mineral content, and individual mineral chemistries. Finer fractions were analyzed to determine the variability of sulfide mineral oxidation within the pile. The information summarized and generated is intended to provide insight into the solid-phases controlling chemical release, the long term behavior of abandoned mine wastes, and the extent of water contact with rock in the pile. Furthermore, it will provide the foundation for a more focused and intensive program for analysis of leached solids.

6.2. Unleached Waste Rock Characterization

The drilling of the test shaft and drifts from which rock was excavated for the test piles continued from December 1976 to September 1977. Piles FL1 - FL4 (0.6% S) were constructed in April 1977 from rock excavated from the test shaft. The two higher sulfur piles, FL6 (0.8% S) and FL5 (1.4% S) were constructed in September 1977 using rock from drifts off of the test shaft. The volume and mass of rock excavated were determined. The lithological units present were identified by visual examination, and samples of each round of blasted rock were analyzed for sulfur, copper, and nickel (Table 2, 3, 4).

Several samples were collected and analyzed to augment the limited chemical and mineralogical data. These samples had all weathered to some degree in the environment. The specimens selected were fairly large particles and it is assumed that the extent of weathering, at least of silicate and oxide minerals, was relatively small. A grab sample of the shaft rock was collected from pile FL1 in 1981(Appendix A2, Tables A2.1.1, A2.1.2), bulk samples were collected from FL1, FL6, and FL5 in 1994, and a hand-sized sample was collected with each of the spatially distributed samples collected from FL6 in 1994. The grab sample consisted of large fragments, as were the hand-size specimens collected from FL6. The −3/+1.5-inch fraction of the bulk sample was selected to represent unleached rock.

6.2.1. Test Pile Masses

During excavation, Minnamax estimated the mass of rock delivered to the surface using the dimensions of the shaft and drifts, the footage blasted, and a density of 2.88 g cm^{-3} (180 lb ft^{-3}) (Kulas 1980). Based on these calculations, the mass of 0.6% S rock was determined to be 6487 metric tons, which was assumed to be distributed equally among piles FL1 to FL4 (Table 11). The masses of the 0.8% S and 1.4% S piles were determined as 1566 and 2155 metric tons, respectively. However, it was clear from visual inspection that the 0.6% S piles varied in size and the 0.8% S pile was larger than the 1.4% S pile.

Due to these inconsistencies, the MN DNR surveyed the six piles in 1982 to determine their volumes and estimate the masses present using an assumed broken rock density of 128 lb ft^{-3} (Eger and Lapakko 1985, Appendix 1). This calculation assumed approximate values for rock density and percent voids of 2.9 g cm^{-3} and 30 percent, respectively. The individual pile masses determined ranged from 800 to 1700 metric tons. Although the mass determined for the FL4 (0.6% S) pile was in good agreement with the mass determined by Minnamax, the mass determined for the other five piles were roughly 40 to 80 percent of the values determined previously. The reason for the discrepancy is unclear, and the values determined based on the MN DNR survey are assumed to be correct. Data collected subsequently indicated that at a minimum the mass of FL5 rock (1.4% S) determined by Minnamax was excessive.

Approximately 680 tons of rock were removed from FL4 (0.6% S) in 1982. This rock was used to construct six experimental bins to examine the effect of mixing low-sulfur Duluth Complex tailings with rock on release of acid and metals from the rock (Eger et al. 1984, Lapakko et al. 1996). A bulk sample of the rock was collected at this time. Rock was also removed from FL1 (0.6% S, 197 yd^3), FL6 (0.8% S, 289 yd^3), and FL5 (1.4% S, 182 yd^3) as the piles were dismantled in 1994. These volumes represent masses of 290, 430, and 270 metric tons (Table 11). (This calculation assumes a rock density of 2.88 g cm^{-3} and bulk density of 1.92 g cm^{-3}.)

The mass of rock and other site materials was also determined as they were fed to a pug mill during final reclamation (Barr Engineering Co. and Mitchell 1995). The mass of FL5 rock (1.4% S) plus the cover material was determined as 1745 metric tons, of which the calculated rock masses represented 30 to 110 percent (Table 11). It is clear that the FL5 rock mass estimated by Minnamax was too high because it indicated the mass of rock reclaimed was 1880 metric tons. This exceeded the mass of rock plus the cover material, 120 metric tons of drill cuttings, and 90 metric tons of a sludge-sand mixture from the settling pond.

The mass of piles FL1 through FL4, FL6, and other site materials (sand from a settling basin, sludge, cover material from piles FL2 and FL3, sand berms from around the piles, and sand used around storage tanks) was determined as 10,230 metric tons (Table 11). Thus, rock masses calculated by Minnamax and the MN DNR represented roughly 50 to 70 percent of the material fed to the pug mill. Because the mass of the other site materials included in this measurement is unknown, this comparison sheds little light on the accuracy of the initial rock pile mass estimations.

6.2.2. Particle Size

No determinations of particle size distribution were made on the unleached rock during the construction of the piles. The best approximation of unleached rock particle size distribution was generated by the analysis of a bulk sample of rock from test pile FL4 (0.6% S) in 1982 (Appendix 2, Table A2.3). This rock had been exposed to environmental weathering for about five years at this time. About 38% of the rock was finer than 38 mm, about 19% was finer than 6.4 mm, and 4.6% was finer than 0.149 mm (100 mesh).

6.2.3. Chemistry

Chemical analyses of blast rounds placed into the test piles were conducted by MINNAMAX, and 27 to 67 samples were analyzed for each of the three compositions. Blast rounds from the test shaft were used to construct test piles FL1 - FL4. The 67 blast rounds analyzed averaged 0.64% S with a standard deviation of 0.27% S. Average copper and nickel values were 0.35% and 0.08%, respectively (Table 5). Blast rounds from the drifts that contributed to test pile FL6 averaged 0.79% S, 0.34% Cu, and 0.09% Ni (n = 27). The corresponding values for rock used to construct pile FL5 were 1.45% S, 0.31% Cu, and 0.09% Ni (n = 39). Based on the frequency of analyses for S, Cu, and Ni, these are believed to most accurately represent average concentrations of these elements in the test piles.

The ratio of the standard deviation to the mean concentrations indicates that rock composition was more variable in the 1.4% S pile (Table 5). The analyses indicate that some of the blast rounds for this pile were quite high in sulfur, copper, and nickel content. This is reflected by ratios of maximum to mean concentrations that ranged from roughly 3 to 5. The corresponding ratios for 0.6% S and 0.8% S rock were typically in the approximate range of 1.7 to 2 (Table 5).

The data indicate that, although the average sulfur content varied among the piles, average copper and nickel contents were fairly constant. This suggests that the variation in sulfur content was due to variation in iron sulfide content. The sulfur occurring with iron was estimated by assuming all sulfur was associated with copper, nickel or iron. The calculation indicated the sulfur associated with iron represented 0.42, 0.57 and 1.25 weight percent, respectively, of the 0.6%, 0.8% and 1.4% sulfur rock. It should be noted that this iron sulfide content includes iron occurring with mixed metal sulfides such as chalcopyrite, cubanite, and pentlandite ($CuFeS_2$, $CuFe_2S_3$, $(Fe,Ni)_9S_8$, respectively).

It is possible to estimate the with sulfur associated pyrrhotite, as well as with chalcopyrite-cubanite and pentlandite, by making several assumptions. First, it is assumed that all sulfur is associated with copper, nickel, and iron. This ignores the small amounts of sulfur associated with cobalt and zinc. Second, it is assumed that all copper is present in chalcopyrite and cubanite, and the chalcopyrite: cubanite ratio is 2:1. Third, it is assumed that all nickel is present in pentlandite with a formula of $Ni_{4.5}Fe_{4.5}S_8$. (It should be noted that nickel is known to occur in olivine and pyrrhotite.) These assumptions yield a S/Cu ratio of 1.18 for chalcopyrite and cubanite and a S/Ni ratio of 0.97 for pentlandite. Using the aforementioned ratios and the sulfur, copper, and nickel contents of the test piles, the amount of sulfur associated with pyrrhotite in the three piles was estimated. The values determined for the 0.6, 0.8, and 1.4% S piles were 0.15, 0.30, and 1.01 percent sulfur associated with pyrrhotite.

Additional chemical analyses of the 0.6% S rock were conducted on a grab sample of rock taken from pile FL1 in 1981 and the −6/+1.5-inch fraction of bulk samples from this pile and experimental pile 6B. The sulfur, copper, and nickel contents determined for the grab sample (Appendix 2, Table 2.1) were in good agreement with the average values of 0.64% S, 0.35% Cu, and 0.08% Ni from the 67 blast round samples (Table 5). In contrast, concentrations in the large particles from the FL1 bulk

sample were about 65% of those from the blast holes, and those from bin 6B sample were roughly 1 to 1.5 times those of the blast rounds. As an aside, chemical analyses were also conducted on the minus 2.0 mm size fractions of the bulk sample collected from FL4 (0.6% S) in 1982. The sulfur, copper, and nickel contents of the $0.5 < d < 2.0$ mm fraction were 0.65, 0.34, and 0.062 percent. These contents were in good agreement with those from the 67 blast round samples and the FL1 grab sample.

The whole rock component concentrations determined for the FL1 grab sample were in reasonable agreement with those from the FL1 bulk sample (Appendix 2, Table 2.1). Relative to these two samples, the bin 6B sample had higher concentrations of iron and magnesium and lower concentrations of silicon, aluminum, and calcium.

Additional chemical analyses of the 0.8% S rock were conducted on the −6/+1/5-inch fraction of the bulk sample from pile FL6 and on spatially distributed hand-sized specimens collected from this pile. All of these samples were collected in 1994. The copper content of the large particles from the bulk sample was essentially the same as that from the blast rounds. In contrast, the sulfur and nickel contents from the bulk sample were roughly 80 and 60 percent of those from the blast rounds. Average concentrations of sulfur and copper from the 16 spatially distributed hand-sized samples were lower yet, but nickel concentrations in these samples was in close agreement with that from the bulk sample (Table 6). The whole rock chemistries of the two FL6 samples were in reasonable agreement. Relative to the bulk sample, the average spatially distributed sample chemistry revealed higher concentrations of iron and magnesium and a lower calcium concentration.

Additional chemical analyses of the 1.4% S rock were conducted on the −6/+1/5-inch fraction of the bulk sample from pile FL5. Sulfur, copper, and nickel concentrations from the bulk sample were roughly 50% of the average determined for the 39 blast round samples. As indicated previously, the chemistry of this rock was more variable than the 0.6% S and 0.8% S rock. Apparently the bulk sample did not include a large mass of rock with elevated concentrations of these elements. The whole rock chemistry was largely in the range observed for the lower sulfur bulk samples. One notable difference was that the calcium and magnesium concentrations were lowest of all samples analyzed.

6.2.4. Mineralogy

6.2.4.1. Lithology Identification During Excavation

As rock was excavated it was visually examined by geologists to determine the lithologies. Rock from the test shaft ranged from troctolite (olivine rich gabbro) and hornfels from the deepest rounds, through various types of troctolites and minor hornfels, into anorthosite and gabbro at the shallowest portion of the test shaft sampled for the pile (Table 2). The description of drill hole B1-162 suggests that most of the rock used to construct test piles FL1-FL4 was troctolite (Appendix 2, Attachment A1.1). Rock from drifts A, B, and C and was primarily troctolite and hornfels (Tables 3, 4) and was used to construct test piles FL5 and FL6. Based on cursory examinations of these piles over time, including during sampling by the USBM and MN DNR, it is believed that most of this rock was also

troctolite. Pyrrhotite sulfide lenses and significant copper-nickel mineralization were noted in the 1.4% S pile rock.

6.2.4.2. Mineralogy of Test Piles

Mineralogical analyses were conducted on a grab sample from test pile FL1 (0.6% S) in 1981, the −6/+1.5- inch fraction of bulk samples from piles FL1 (0.6% S), FL6 (0.8% S), and FL5 (1.4% S) collected in 1994, and on hand-sized specimens collected with each of the spatially distributed samples from the 0.8% S pile. All samples were fairly large in size and it is assumed that they were not extensively weathered.

Feldspar (plagioclase) was the dominant mineral in all samples, with values for all but the FL1 grab sample in the range of 42 to 52 percent. Although the feldspar contents varied by about seven percent, mineral contents of the grab and bulk samples from pile FL1 (0.6% S) were in reasonable agreement. Olivine and clinopyroxene were the next most abundant minerals in the two 0.6% S samples, and were present in roughly equal amounts. The category of alteration minerals (primary amphiboles, serpentine, chlorite, minor secondary montmorillonite) was the next most abundant in these piles, contributing about six percent, followed by orthopyroxene, ilmenite, and biotite contents ranging from roughly 2% to 4%.

As was the case for the 0.6% S samples, feldspar, olivine, and clinopyroxene were the dominant minerals in the 0.8% S bulk sample. In addition, olivine and clinopyroxene were present in roughly equal amounts. Relative to the 0.6% S samples, the feldspar content was lower and the olivine and clinopyroxene contents were higher. Alteration minerals, biotite, orthopyroxene, and ilmenite contents ranged from roughly 3% to 5%.

Optical examination of thin sections of the hand-sized spatially distributed samples indicated considerable variability in mineral contents (Table 12). The most abundant minerals were plagioclase, hypersthene, olivine, and augite (Table 12). The plagioclase composition ranged from 15 to 55 weight percent and all of the samples had plagioclase (the only mineral the samples all had in common). The hypersthene mode (of the samples that had hypersthene) varied from 8 to 58 weight percent, while the augite mode varied from 2 to 35 weight percent. Serpentine was present in relatively high amounts in three samples and was most likely attributable to hydrothermal alteration of the olivine. The samples were classified generally as norites, gabbros, troctolites, pyroxenites, and peridotite (one sample), and hornfels (one sample). There was one picrite (an olivine-rich troctolite).

Sulfide mineral contents varied from 0 to 9 weight percent. Both the upper end of the range and the magnitude of the range were higher than anticipated. It was known that the abundances determined by examination of the thin sections did not yield accurate modes of the sulfide minerals. During the investigation of sulfide minerals grain reactions, sulfides were found in each of the spatially distributed samples in the 30/70 mesh size fraction. In these cases, sulfides grains were looked for and in the case of the thin sections, sulfides grains were examined if they were happened upon. Thus it should be noted that sulfide minerals were present in all of the spatially distributed grains.

The extent of sulfide minerals determined using the bulk chemistry may more accurately reflect sulfide mineral abundance.

The modal contents of the 16 hand-sized spatially distributed samples were averaged to determine a mean mineral content. The mean feldspar content of the spatially distributed samples was in good agreement with that of the bulk sample, as was the sum of olivine, clinopyroxene, and orthopyroxene contents. The sum of the three minerals in the two samples was similar (42.5 vs 39 percent, respectively), but the contents of the individual mineral were distinctly different. In particular, the mean orthopyroxene content of the spatially distributed samples was roughly four times that reported for the other four samples. It is assumed that the bulk sample provided a more reasonable estimation of the mineral contents. The bulk sample mass was in the range of 2 to 2.5 tons, and the spatially distributed mineral contents were averaged from 16 hand-sized rocks from spatially distributed sample locations.

The mineral contents of the 1.4% pile, which had the lowest feldspar content, was markedly different from that of the 0.6% and 0.8% piles. In decreasing order after feldspar, the major minerals were alteration minerals (14.5%), clinopyroxene (11.2%), biotite (9.6%), magnetite (5.2%), ilmenite (4.2%), and quartz (4.2%). The olivine content of this pile was only 1.6%, roughly and order of magnitude lower than that of the other two piles (Table 7).

It should be noted that rock chemistry (Table 6) was used to calculate mineral contents of the bulk and grab samples. This had a large bearing on the reported sulfide mineral contents. For example, the pyrrhotite content of the 0.8% S bulk sample is less than or equal to that of the two FL1 samples. The sulfide mineral contents of the unleached rock would be more accurately reflected by the chemical analyses conducted on blast rounds (Table 5). For example, the pyrrhotite content of the 0.8% S bulk sample should be 0.71, based on chemistry of the bulk sample. It should be noted that the mineral contents of the 0.6% S grab sample are in volume percent.

6.2.4.3. Mineral Chemistry of Spatially Distributed Samples from 0.8% S Pile (FL6)

As part of the present study, electron microprobe analysis was used to determine mineral chemistries of the 16 hand-sized specimens collected with the spatially distributed samples from the 0.8% S pile. The main objective of these analyses was to more accurately characterize minerals that dissolved to neutralize acid. Both the extent and rate of neutralization are dependent on chemistry. For example, if olivine ($(Fe,Mg)_2SiO_4$) is present as the iron end member (fayalite) it will provide no acid neutralization. In contrast, if olivine is present as the magnesium end member (forsterite), all of the olivine present may contribute to acid neutralization. Reported plagioclase dissolution kinetics provide an example of the effect of chemistry on dissolution kinetics. The rate of plagioclase dissolution decreases as the calcium content decreases. Consequently, the calcic end member (anorthite, $CaAl_2Si_2O_8$) dissolves more rapidly than intermediate members such as andesine ($Na_{0.5}Ca_{0.5}Al_{1.5}Si_{2.5}O_8$). In addition, mineral chemistries aid in the interpretation of drainage quality data because aqueous components can be used to indicate dissolution of specific minerals.

Data for all samples in which a given mineral occurred were averaged to determine mineral chemistries. Plagioclase was present in all 16 samples, with an average composition of $(Na_{0.40}Ca_{0.56})(Si_{2.41}Al_{1.60})O_8$. Olivine was present in 10 of the samples and was slightly enriched in magnesium. Clinopyroxene and orthopyroxene, identified as augite and ferrohypersthene, grains were each present in 11 samples, and biotite was present in six samples. The formulas of these and other minerals from the spatially distributed samples are presented in Table 13.

6.3. Test Pile Drainage

6.3.1. Drainage Quantity

Mass release from waste rock may be affected by the amount of water flowing through the pile. Yield coefficients were calculated for the test piles to express the annual drainage volume as a fraction of the precipitation falling on the pile. Two uncovered piles, FL1 (0.6% S) and FL6 (0.8% S) were uncovered and were monitored for 14 years. The annual yield coefficients for these piles typically ranged from 0.45 to 0.6, and averaged 0.51 and 0.54, respectively, over the period of record. The average yield coefficients for covered piles ranged from 0.30 to 0.43 (Table 14).

6.3.2. Drainage Quality

The pH of drainage from the test piles decreased as the solid phase sulfur content increased (Tables 15-20; Appendix 2, section A2.6). At the beginning of 1978, the pH of drainage from all piles was between 7 and 8. For the three piles containing 0.6-percent sulfur, drainage pH decreased gradually over time from circumneutral to the range of 4.8 to 5.3. The pH of drainage from the piles containing 0.8 and 1.4 percent sulfur decreased and plateaued within a range of about 0.2 to 0.3 pH units. The pH of drainage from the 0.8-percent sulfur pile decreased until 1982 and subsequently oscillated in the typical range of 4.0 to 4.3. The pH of drainage from the 1.4-percent sulfur pile decreased to a lower level, from a median value of 5.5 in 1978 to a typical range of 3.4 to 3.6 from 1979 to 1991. The flow-weighted-median values increased from 1982 (3.40) to 1992 (3.70).

Annual-flow-weighted-mean sulfate concentrations increased as the sulfur content of the piles increased and were independent of flow. For the three 0.6% S piles (ignoring the short period of record for FL4), annual flow-weighted mean sulfate concentrations typically ranged from 7 to 16 mmol L^{-1}. For these piles, concentrations were lowest from FL2 and highest from FL3.

For the individual piles, sulfate concentrations increased during the first three to four years and then oscillated within a fairly constant range (Appendix 2, section A2.6). Some deviations from the general trends were observed. Most notably, sulfate concentrations in drainage from the 1.4-percent-sulfur pile increased to peak levels between 1981 and 1985, and then decreased. From 1987 to 1991, the flow-weighted-mean sulfate concentrations from this pile were comparable to, or lower than, those from the 0.8-percent-sulfur pile. Sulfate concentrations from the 0.8-percent sulfur pile increased over time (Appendix 2, section A2.6). However, these temporal variations from 1981 through 1991 were relatively small, with the maximum annual-flow-weighted-mean sulfate concentration within a factor of two of the minimum value observed during this period.

Annual-flow-weighted-mean calcium concentrations were typically fairly constant, ranging from about 4 to 7 mmol L^{-1} for the five piles (Tables 15-20). As implied by the fairly small range, there was little dependence of annual-flow-weighted-mean calcium concentrations on time of dissolution, sulfur content, or flow. For the three 0.6% S piles, mean annual calcium concentrations typically ranged from 4 to 6.5 mmol L^{-1}. Concentrations were lowest from FL2, which also had the lowest sulfate concentrations. However, calcium concentrations from FL1 were slightly higher than those from FL3, which was the opposite of the order of sulfate concentrations from these piles. Typical annual flow-weighted mean calcium concentrations from the 0.8% S pile were only slightly higher (6.2 to 7.4 mmol L^{-1}), despite drainage pH values that were generally at least one unit lower than those from the low sulfur piles. Typical calcium concentrations from the 1.4% S pile ranged from 5.5 to 6.2 mmol L^{-1}, comparable to those from the low sulfur piles. Values for the 1.41-percent-sulfur pile appeared to parallel the decreasing trend observed for sulfate concentrations from 1985 to 1991.

Annual-flow-weighted-mean magnesium concentration variations over time were generally parallel to those observed for sulfate (Tables 15-20), and were independent of flow volume. With respect to solid-phase-sulfur concentration, annual-flow-weighted-mean magnesium concentrations were lowest for the 0.63-percent-sulfur piles (1-7 mmol L^{-1}) and in a similar range for the two piles of higher sulfur content (5-15 mmol L^{-1}). For the individual piles, annual-flow-weighted-mean magnesium concentrations tended to increase as the annual median pH decreased.

For all piles, concentrations of calcium initially exceeded those of magnesium. This was continuously the case for piles FL1 and FL2. For the remaining piles magnesium concentrations exceeded calcium concentrations for the final 7 to 14 years of operation. This is likely related to decreases in drainage pH with time. Magnesium concentrations increased as pH decreased over time, while calcium concentrations remained in a relatively constant range.

Copper, nickel, cobalt, and zinc concentrations tended to increase as pH decreased (Appendix 2). Reflecting this dependence, trace metal concentrations from the low sulfur piles increased over time, concurrent with a continual decrease in drainage pH. This trend was strongest for copper concentrations, which increased by two to three orders of magnitude as pH decreased from above 7.0 to near 5. During the same period, nickel, cobalt, and zinc concentrations increased by one to two orders of magnitude. Further reflecting this trend, trace metal concentrations were usually lowest in the low sulfur piles. However, as drainage pH values from the low sulfur piles neared 5.0, nickel, cobalt, and zinc metal concentrations in the drainages were similar to those observed for the 0.8% S pile. In contrast, copper concentrations from the low sulfur piles at this time were roughly one third of those from the 0.8% S pile. Maximum concentrations of all trace metals occurred in the low-pH drainage from the 1.4% S pile. After 1987, however, copper and nickel concentrations from the 0.8% S pile were comparable to or greater than those from the 1.4% S pile.

Trace metal concentrations generally increased in the order Zn < Co << Cu < Ni. The total concentrations of these metals was considerably lower than sulfate concentrations. This suggests that oxidation of pyrrhotite provided the majority of sulfate to the drainage.

6.3.3. Chemical Mass Release

Annual rates of mass release reflect flow-weighted mean concentrations and the volume of drainage from the pile. Consequently, release rates from pile with covers (FL2, FL3, FL5) might be expected to have lower rates of chemical mass release due to a reduction in drainage volume. Rates of sulfate and calcium release varied largely as a result of changes in drainage volume because the flow-weighted mean concentrations were fairly stable. In contrast, concentrations of magnesium and trace metals tended to increase as pH decreased. Therefore, their release rates were affected by both concentration and drainage volume.

The annual rates of sulfate release from each pile oscillated within a fairly constant range (Table 21, Appendix 2). In general, annual sulfate release increased with the solid-phase-sulfur content of the pile and flow. The rates for the 0.8% S pile were comparable to or exceeded those for the 1.4% S pile from 1984 to 1991 (Appendix 2, Section A2.8). The relatively low release from the 1.4% S pile was due, in part, to the fact that is was covered with sandy till and coarse sand (Table 22). Consequently, the yield coefficient for this pile was only about 60% of that for the 0.8% S pile, which was uncovered. Variations in drainage volume also influenced temporal variations in the annual rates of sulfate release. The total sulfate release represented roughly 4 to 11 percent of the sulfur originally present in the piles (Table 23).

Trends in calcium release rates tended to parallel those for sulfate. However, calcium release rates did not increase with the solid-phase-sulfur content of the piles. The average calcium release rates over the period of record for all piles were near 2.0 mmol T^{-1} d^{-1}, except for FL2 for which the average rate was roughly 1.1 mmol T^{-1} d^{-1}. The total calcium release over the period of record for the individual piles represented 0.3 to 0.7 percent of the total calcium originally present.

Magnesium release rates for a specific pile tended to increase as pH decreased. Average magnesium release rates from the 0.6% piles for the entire period of record ranged from 0.5 to 1.9 mmol T^{-1} d^{-1}. The corresponding overall rates for the 0.8% S and 1.4%S piles were 3.3 and 2.8 mmol T^{-1} d^{-1}, respectively. Similar to the sulfate-release rates from 1984 through 1991, magnesium release from the 0.8-percent-sulfur pile was usually higher than that from the 1.4-percent-sulfur pile (Appendix 2, Section A2.8). Once again, it should be noted that the comparatively low flow from the 1.4% S pile was a factor in the reduced magnesium release from the pile. The total magnesium release over the period of record for the individual piles represented 0.1 to 0.8 percent of the total magnesium originally present.

As was the case with magnesium, trace metal release trace metal release increased as pH decreased and as drainage volume increased. For the entire period of record, average trace metal release rates from the 0.6% S piles were lower than those from the high sulfur piles, with the exception of nickel release from FL3. The highest average release rates for all metals occurred at the 1.4% S pile, for which drainage pH was lowest. The ordering of average release rates for all piles was Ni > Cu >> Co > Zn. The total release over the period of record for the individual piles represented 2 to 26 percent of the nickel and 0.2 to 3.3 percent of the copper originally present. For the individual piles, the percent depletion of nickel ranged from roughly 6 to 16 times that of copper.

6.4. Leached Waste Rock Characterization

6.4.1. Physical Characteristics

6.4.1.1. Particle Size

Particle size distribution measurements were made on a bulk sample from FL4 (0.6% S) when it was partially disassembled in 1982. Additional determinations were made on bulk samples collected from the 0.6% S (FL1), 0.8% S (FL6), and 1.4% (FL5) S piles and the 0.6% S test bin (6B) in 1994. Particle size distributions were also determined for 16 spatially distributed samples collected from pile FL6 (0.8% S) in 1994.

The particle size distributions of the three 0.6% S rock samples were in reasonable agreement, as indicated by the approximate ranges for percent finer than 19.0 mm (32-33%), 6.4 mm (19-22%), 2.0 mm (11-14%), and 0.149 mm (3-5%) fractions. The size distributions for the 0.8% and 1.4% S samples were also in reasonable agreement. The respective percentages passing for given diameters were as follows:19.0 mm (34%, 27%), 6.4 mm (11%, 11%), 2.0 mm (5%, 6%), and 0.149 mm (1.5%, 1.6%). It should be noted that the distributions for the 0.8 and 1.4% S samples were coarser than that for the 0.6% rock. Although the percentages finer than 19.0 mm were similar for all samples, the fraction of 0.8% and 1.4% S rock in the minus 6.4 mm fraction was roughly half that determined for the 0.6% S samples (Table 24). The 1.4% S pile contained some agglomerated rock and was the only pile with material coarser than 12 inches. The influence of the agglomerates on the weight present in larger size fractions was recorded (Table 24).

The particle size distribution of the 16 spatially distributed samples from the 0.8% S pile were generally similar. About 40 percent of the rock was finer than 0.5 inches, 16 percent finer than 2.0 mm, and five percent finer than 0.149 mm (Table 25). This distribution differed from that of the bulk sample largely in the amount of material larger than 0.5 inches in diameter. The spatially distributed samples averaged about 60 percent in this coarse fraction as compared to 86 percent for the bulk sample. Considering only rock finer than 0.5 inches, the averaged spatially distributed samples compared favorably with the bulk sample. In both cases the −2.0 mm fraction was near 40 percent of the −0.5-inch material, and the −0.149 mm fractions for the two types of samples were 10 and 11 percent, respectively.

Percent moisture content was also determined for the −½" size fractions as part of the particle size distribution analysis of the bulk samples (Appendix 4, Table A4.3). These data are, however, of limited value for describing conditions in the piles. Samples were stored outdoors and subjected to rainfall prior to this determination.

6.4.1.2. Specific Surface Area of Bulk Sample Size Fractions

Specific surface area measurements were conducted on six size fractions finer than 2.0 mm of bulk samples of piles FL4 (0.6% S, collected in 1982), FL1 (0.6% S), FL6 (0.8% S), and FL5 (1.4% S). As would be expected, specific surface area generally increased as particle size decreased. The

specific surface area of the finest fraction (−0.053 mm) was 4 to 15 times that of the coarsest fraction (−2.0 /+0.50 mm; Table 26). The ratio of the maximum particle diameters in these fraction is close to 40 (2.0/0.053 = 37.7). Assuming particles were smooth, the ratio of the specific surface areas would be expected to be near 40 also, as opposed to the range of 4 to 15 observed.

Geometric surface areas for the coarsest and finest particle size fractions were calculated using respective arithmetic mean diameters of 0.125 and 0.027 mm and equation 2. This calculation yielded specific surface areas of 0.0016 and 0.0377 m^2g^{-1}, respectively for the coarsest and finest fractions. These values were more than two orders of magnitude lower than those reported. This suggests that surface irregularities may have contributed to the measured surface areas. Such irregularities might include etch pits, adhered reaction products and/or adhered fine particles.

$$A_S = 6\ (\rho d)\ (10^{-4}) \qquad [2]$$

where A_S = specific surface area in $cm^{-2}g^{-1}$,
ρ = density, assumed as 3.0 g cm^{-3},
d = diameter in cm,
10^{-4} = factor to convert area from cm^{-2} to m^{-2}.

The occurrence of the aforementioned surface irregularities would be expected to increase with the extent of reaction. This appears to be true for the three 0.6% S samples. The FL4 sample had been leached for about six years and the remaining two samples (FL1, 6B) were leached for 18 years. The specific surface areas for the FL1 samples were 1.2 to 2.3 times those of the corresponding FL4 size fractions. Specific surface areas for the samples from 6B were highest of the 0.6% S samples. This sample was leached in a relatively small bin, and leaching may have been more extensive due to more intimate contact of the rock and water flowing through it. For the three bulk samples collected from test piles in 1994, specific surface area almost always increased with the sulfur content of the rock. This further suggests that the specific surface area increased with the extent of reaction.

6.4.2. Chemistry

6.4.2.1. Bulk Samples

Sulfur, sulfate, trace metal (Cu, Ni, Co, Zn, Mo, Ag, Pb) and whole rock constituent concentrations were determined for particle size fractions of bulk sample collected from FL1 (0.6% S), FL6 (0.8% S), and FL5 (1.4% S) (Tables 27-34). Less extensive analyses were conducted on size fractions of the bulk sample collected from test pile FL4 (0.6% S) in 1982. For the 0.6% S samples collected from FL4 and FL1, total sulfur and sulfide contents tended to increase as particle size decreased. Sulfate contents of these piles increased as particle size decreased to the −0.177/+0.149 fraction and plateaued.

For the 0.8% S rock, sulfur contents were lowest in the two coarsest fractions and highest in the finest fraction, and sulfur contents of the intermediate sizes (−½-inch/+0.053 mm) were in a fairly constant range. Sulfate concentrations followed a similar trend. For the 1.4% S rock, sulfur

contents were lowest in the three coarsest fractions (0.7 - 1.0% S) and ranged from roughly 1.5 to 2.0 percent, with no obvious trend, for the finer fractions (−2.0 mm). Sulfate concentrations increased fairly consistently with decreasing particle size (Table 28).

Trace metal contents (copper, nickel, cobalt, zinc) were determined for each of the ten particle size fractions of bulk samples collected from the three piles and test bin 6B. Concentrations of copper, nickel, cobalt, and zinc were almost always lowest in the coarse fractions and highest in the fine fractions (Tables 27-30). Concentrations of these metals in the intermediate size particles often oscillated within a fairly stable range. Concentrations of silver, molybdenum, and lead were low relative to the aforementioned metals and displayed no consistent variation with respect to particle size. Due their low solid-phase concentrations these metals are not discussed; data on their concentrations are available in the appendices.

Concentrations of sulfur, copper, nickel, cobalt, and zinc in the FL4 rock increased strictly as particle size decreased in the fractions finer than 2.0 mm (Table 8). This trend was most closely approximated by the bulk sample from FL1, which had the same sulfur content as pile FL4 (0.6%). Furthermore, the ratio of sulfur and trace metal concentrations in the −0.053 mm fraction to corresponding concentrations in the −2.0/+0.5 mm fractions was similar for the two piles.

Whole rock chemistry (Al_2O_3, CaO, Cr_2O_3, Fe_2O_3, K_2O, MgO, MnO, Na_2O, P_2O_5, SiO_2, and TiO_2) was determined for ten particle size fractions of bulk samples from the three test piles and bin 6B. The mass weighted average chemistry was determined for each of the bulk samples. The components that comprised more than one percent of the masses of the three piles were SiO_2 (44.5-46.3%), Al_2O_3 (14.5-17.5%), Fe_2O_3 (14.8-16.6%), MgO (6.2-10%), CaO (6.5-8.6%), Na_2O (2.0-2.4%), TiO_2 (1.5-2.5%), and K_2O (0.5-1.0%) (Tables 31-34).

Loss on ignition (LOI) includes water within the rock structure and all forms of carbon. Reported values ranged from <0.01 to 8.16%, and were lowest for the 0.6% S rock and highest for the 1.4% rock. For all three test piles, LOI increased as particle size decreased (Tables 27-30).

Mattson (1996) calculated a chemical alteration index (CIA) for each size fraction by applying the method of Nesbitt and Young (1982) to the chemistry of the size fraction (equation 3; Appendix 4, Attachment 4.1). Increasing CIA values indicate an increased degree of feldspar weathering, during which calcium, sodium, and potassium are leached and aluminum remains in the solid phase. The CIA values were relatively constant with respect to particle size with the exception of values from the 0.8% S (FL6) bulk sample. FL6 size fractions finer than 0.149 mm tended to have higher CIA values than the coarser fractions. This suggests a greater degree of feldspar weathering in these fine fractions. This suggestion is subject to uncertainty due to the relatively small magnitude of the differences in CIA values and the lack of similar trends in the other bulk samples.

$$CIA = (Al_2O_3 \div (Al_2O_3 + CaO + Na_2O + K_2O)) \times 100 \quad (3)$$

The chemistry of individual size fractions indicated iron concentrations and loss on ignition (LOI) increased with decreasing particle size. In contrast, concentrations of SiO_2, Al_2O_3, CaO, and Na_2O

tended to decrease with decreasing particle size. Mattson (1996) calculated the following ratio to quantify these trends (equation 4). For all three piles, this ratio increased with decreasing particle size, and values for the two finest fractions were 1.3 to 2.2 times those in the corresponding coarsest fractions (Appendix 4, Attachment 4.1). The increases in the ratio were due largely to changes in measured values for iron and LOI. The sum of iron and LOI in the fine fractions were 1.2 to 1.8 time that in the corresponding coarse fraction.

$$\text{Ratio} = ((Fe_2O_3 + \text{LOI}) \div (SiO_2 + Al_2O_3 + CaO + Na_2O)) \times 100 \quad (4)$$

6.4.2.2. Spatially Distributed Samples from the 0.8% S Pile

Spatially distributed samples were collected from the 0.6, 0.8 and 1.4% S test piles. The USBM determined sulfur, sulfate, total carbon, evolved carbon dioxide, whole rock, and trace metal contents of the the eight size fractions finer than 1/4-inch of the 16 spatially distributed samples from the 0.8% S pile. The mass-weighted-average chemistry of the spatially distributed samples was determined for each of the 16 spatially distributed samples. These values were averaged to determine the overall average chemistry of the spatially distributed samples.

The average chemistry of the spatially distributed samples was generally in close agreement with that of the 0.8% S bulk sample. This comparison was based on the overall average chemistry of the spatially distributed samples and the mass-weighted average chemistry of the −0.5-inch fractions of the bulk sample. The −0.5/+2.0 mm fraction of the bulk samples was analyzed, therefore it was not possible to determine the chemistry of the −0.25-inch fraction. The −0.5-inch fraction was the closest approximation to the −0.25-inch spatially distributed samples.

The average sulfur content of the spatially distributed samples was calculated as 1.17 percent as compared to a sulfur content of 1.19 for the bulk sample from the 0.8% S pile (Table 35). Copper, nickel, and cobalt contents were also in good agreement. Although zinc concentrations were low in both spatially distributed and the bulk samples, values for the spatially distributed samples were about twice those in the bulk samples.

The average SiO_2 concentration for the spatially distributed samples was close to nine percent lower than that for the bulk sample (Table 35). Spatially distributed concentrations of CaO, MgO, and Al_2O_3 were 1.6 to 2.0 percent lower than corresponding concentrations in the bulk sample. Because of these discrepancies, and the fact that the sum of chemical components in the spatially distributed samples was less than 100%, eight pulps analyzed by the USBM were selected for reanalysis by Acme Analytical Laboratories Ltd. (Vancouver, B.C.).

Acme results indicated USBM analyses for SiO_2, MgO, and Al_2O_3 were low and, therefore, agreement between the spatially distributed and bulk sample chemistries was better than suggested by the USBM analyses (Table 36). The Acme results for SiO_2 exceeded those by the USBM by an average of 10 percent, and this would explain the low values reported for the spatially distributed samples. Similarly the the Acme results suggested Al_2O_3 concentrations reported by the USBM were low by roughly 1.5%, and this would account for the difference reported between the spatially

distributed and bulk samples. The error in MgO analyses implied by the Acme results would account for roughly 25% of the difference between values for spatially distributed and bulk samples. In contrast, CaO concentrations reported by the USBM were typically 0.3 to 0.6 percent higher than those reported by Acme. This supports the original notion that CaO contents of the spatially distributed samples were lower than those from the bulk samples. The two sets of analyses were typically in reasonable agreement for Na_2O, K_2O, and MnO. The reanalyses suggest that agreement between spatially distributed and bulk sample chemistries was excellent for S_T, Cu, Ni, and Co and reasonable for SiO_2, Al_2O_3, Na_2O, and Fe_2O_3. Compared to the bulk sample, the spatially distributed samples had lower CaO and Na_2O contents and higher K_2O content.

Concentrations of aluminum, calcium, potassium, and silica tended to be highest in the large particle size fractions, although the silica trend must be viewed with caution due to the large error associated with this analysis (see previous paragraph). In contrast, iron concentrations were highest in the fine fractions. Magnesium and sodium displayed no consistent variation with particle size.

6.4.3. Mineralogy

6.4.3.1. Bulk Samples

Mineralogical examination of the bulk samples was conducted by Louis Mattson (Appendix 4, Attachment A4.1). As discussed previously, the −6/+0.5-inch fractions of the bulk samples were used to approximate unweathered rock. The finer fractions were examined to identify evidence of mineral dissolution and reactions products generated while the rock was subjected to weathering in the environment.

The extent of sulfide mineral weathering was dependent on the rock particle size and the individual sulfide mineral. Most of the pyrrhotite in the fraction finer than 0.50 mm was highly weathered, and the only unweathered pyrrhotite occurred in the coarse fractions. A photomicrograph of a weathered pyrrhotite grain from the 0.6% S (FL1) bulk sample indicated the remnants consisted of iron oxides and a trace of sulfur (Appendix 4, Attachment A4.1, Photo 7). A single grain of pentlandite was observed and examination by SEM/EDS indicated it had altered to violarite (Ni_2FeS_4). Chalcopyrite grains were relatively unweathered in all size fractions. The major solid-phase reaction products of sulfide mineral oxidation were iron oxyhydroxides and gypsum.

Scanning electron microscope examination of the fine fractions of bulk samples provided evidence of ferromagnesian, clinopyroxene, and plagioclase mineral dissolution. Ferromagnesian minerals were the most altered of the silicate minerals present, although olivine grains examined from all three bulk samples were unweathered. Biotite grains were altered to various degrees, and appeared to alter to iron oxides devoid of potassium. Both single crystal and fibrous amphiboles were detected in all bulk samples, and both exhibited dissolution features.

Evidence of plagioclase leaching and clinopyroxene dissolution along cleavage planes was also discovered during examination of the 0.8% S bulk sample. Evidence of plagioclase leaching was

also discovered in the 1.4% S (FL6) bulk sample. In both cases plagioclase dissolution occurred along cleavage planes, and iron oxyhydroxides were present on the the plagioclase mineral surfaces.

XRD identified expandable montmorillonite group clays in the −270-mesh fractions of all three bulk samples, but no mixed layer clays were detected. Based on XRD peak intensities, the montmorillonite contents of the −270 mesh fractions were 10 to 20 times those of the −6/+1/5-inch fractions (Appendix 4, Attachment A4.1). (Attempts to quantify the clay content using thermal gravimetric analysis were unsuccessful.) Montmorillonite clay minerals consist of montmorillonite ($(Al,Mg)_8(Si_4O_{10})_4(OH)_8 \bullet 13H_2O$), beidellite ($(Ca,Na)_{0.3}Al_2(OH)_2(Al,Si)_4O_{10} \bullet (H_2O)_4$), nontronite ($Fe_2(Al,Si)_4O_{10}(OH)_2Na_{0.3}(H_2O)_4$), hectorite ($(Mg,Li)_3Si_4O_{10}(OH)_2Na_{0.3}(H_2O)_4$), and saponite ($(Mg,Fe)_3(Al,Si)_4O_{10}(OH)_2(Ca_{0.5}Na)_{0.3}(H_2O)_4$).

Scanning electron microscope photomicrographs provided visual evidence of apparent montmorillonite-group clay mineral formation on an altered fibrous amphibole from the 0.6% S pile (FL1), although it was not clear if the montmorillonite formed at this location. Montmorillonite formation on weathered orthopyroxene from the 0.8% S pile (FL6) was also documented. Energy dispersive spectrometry indicated the montmorillonite was of mixed composition. The iron-rich member, nontronite, was identified, and phases that were lower in iron and higher in calcium and magnesium were also probably present. The bulk elemental ratios in the clay were similar to those in the orthopyroxene.

6.4.3.2. Spatially Distributed Samples

Spatially distributed samples from the 0.8% S pile (FL6) were analyzed using XRD by the USBM, but the data were not interpreted, due to the closing of the USBM.

During the course of the present study, 16 spatially distributed samples from the 0.8% S pile (FL6) were examined to determine the extent of sulfide mineral weathering. A preliminary study was conducted on size fractions from three selected sample locations to determine an optimum size fraction for examining the remaining spatially distributed samples. The locations, 6-2-4, 6-3-1, and 6-4-1, were selected to represent different degrees of reaction, based on paste pH and acid concentration present, for the 16 spatially distributed samples subjected to chemical analysis. Respective paste pH values at these sites were 5.01, 4.51, and 3.49, as compared with the overall range of 3.28 to 5.44 (Table 37). The respective acid concentration present values were 0.38, 1.79, and 27.03 ppm acidity per gram, which spanned the observed range. Eleven size fractions were examined, ranging from −1.5/+0.75 inches to −270 mesh.

Examination of the 21 size fractions from three locations indicated that the extent of reaction of minerals present either decreased slightly or was relatively constant as particle size decreased (Appendix 5, Table A5.10.4). Copper sulfides (chalcopyrite, cubanite, and in one case bornite) were present in 20 of the 21 fractions. In contrast, pyrrhotite and pentlandite were present in 13 and 8 fractions, respectively. It was not possible to positively determine if they were absent in some fraction originally or if they had been present and reacted completely. Because pyrrhotite was originally present in approximately the same abundance as the total chalcopyrite plus cubanite

(approximately 1.3 weight percent pyrrhotite and 1.2 weight percent chalcopyrite plus cubanite), it is likely that the absence of pyrrhotite indicated its total oxidation. In contrast, the pentlandite content of the unleached rock was approximately 0.26 weight percent, or roughly 20 percent of the combined copper sulfide content. Since its initial concentration was relatively small, its absence in a given size fraction could be the result of either its initial absence or its complete reaction.

It is interesting to note that the number of cubanite grains reported was roughly twice the number of chalcopyrite grains reported. In contrast, cubanite grains in the hand sized specimens from the spatially distributed samples were reported roughly half as often as chalcopyrite grains. This change in relative abundances could be the result of preferential leaching of chalcopyrite or conversion of chalcopyrite to cubanite. However, there is no empirical or theoretical information that supports these explanations. This leaves the possibility that the change in relative abundances of chalcopyrite and cubanite was an artifact of sampling bias.

Based on examination of the size fractions from the three locations, the –30/+70 mesh fraction was selected as the target for examination based on the following reasons. First, most of the sulfide mineral grains in this size fraction were liberated or exposed in this size fraction, whereas the sulfides in larger fractions were isolated within the silicate mineral matrix and therefore had not been exposed to leaching conditions. Second, in smaller size fractions it was not possible to determine if the absence of sulfides was the result of their initial absence or the complete leaching of sulfides that had been present. Thus, the –30/+70 fraction was selected to assess the spatial variation of weathering.

The extent of reaction generally decreased in the order pyrrhotite > pentlandite > chalcopyrite/cubanite, as illustrated in Figure 6. Of the 16 samples examined, pyrrhotite was present in 13, pentlandite in nine, and copper sulfide minerals (chalcopyrite or cubanite) were present in all 16 (Table 37).

The reaction ratings for the 16 samples in the –30/+70 mesh fraction suggested that the extent of sulfide mineral reaction decreased with depth into the pile. Samples from the top of the pile had an average reaction rating of 4.5. Pyrrhotite and pentlandite were either nearly reacted away or absent in these sample. Chalcopyrite and cubanite had iron oxyhydroxide coatings in the neighborhood of 10 microns thick.

In contrast, samples from the fourth lift, at a depth of about 3 meters, had an average reaction rating of 2.5. The pyrrhotite observed in these samples indicated a range of reaction, from minor dissolution to nearly reacted away. Pentlandite was absent in three of the samples and had an iron oxyhydroxide coating less than 10 microns thick in the fourth sample. The extent of chalcopyrite and cubanite reaction in these samples was slight.

Photomicrographs of the spatially distributed samples revealed a consistent order of sulfide mineral oxidation, pyrrhotite > pentlandite > chalcopyrite-cubanite (Figure 6). This is consistent with previous observations on mine waste dissolution in general (Jambor 1994, 2003), as well as observations and theoretical electrochemical properties of individual sulfide minerals present in

Duluth Complex rock (Natarajan et al. 1982). The preferential oxidation of pyrrhotite indicates that the fraction of sulfide associated with pyrrhotite will decrease during the initial phase of weathering, and that associated with copper and nickel sulfide mineral will increase. At some stage of weathering evolution, all sulfur will be associated with nickel and copper sulfide minerals. After the pentlandite weathers entirely, all sulfur will be associated with chalcopyrite and cubanite.

This information can be used to estimate the amount of pyrrhotite that has oxidized during weathering based on the copper, nickel, and sulfide contents of a sample. The percent of pyrrhotite weathered from a sample can be expressed as follows.

$$\Delta \text{ po} = 100 \text{ x } (1 - [S^{2-}]_{po,\ leached}/[S^{2-}]_{po,\ unleached}), \text{ where} \quad (5)$$

$[S^{2-}]_{po,\ leached}$ and $[S^{2-}]_{po,\ unleached}$ are the sulfide contents associated with pyrrhotite in the leached and unleached samples respectively.

The sulfide content associated with pyrrhotite in the leached spatially distributed samples (the value $[S^{2-}]_{po,\ leached}$ in equation 5) was determined based on the sulfide, copper, and nickel contents of the leached rock. Sulfide in Duluth Complex rock is associated with iron, copper, nickel, cobalt, and zinc. To simplify calculations, it was assumed that the sulfide associated with cobalt and zinc was negligible. Analyses conducted on the FL6 (0.8% S) bulk sample indicate that sulfide associated with cobalt and zinc comprised about one percent of the total sulfide present. Thus, it was assumed that all sulfide was associated with iron, copper, and nickel.

Determination of pyrrhotite content based on chemical analysis of iron and sulfide of these samples was not possible because the vast majority of iron occurs in silicate and oxide minerals. Bulk chemical analyses provide no direct indication of the fraction of iron occurring with sulfide. In contrast, copper and nickel occur almost totally as sulfide minerals and, consequently, the amount of sulfide associated with these minerals can be estimated based on solid-phase chemistry. By determining total sulfide and the amount of sulfide occurring with copper and nickel sulfides, the amount of sulfide occurring with pyrrhotite can be determined by difference.

The copper and nickel present were assigned to specific mineral phases with specific chemistries, and no analyses were conducted on the leached samples to determine these chemistries. Copper was present largely as chalcopyrite and cubanite ($CuFeS_2$ and $CuFe_2S_3$, respectively), and it was assumed that all copper was present in these phases. Three cases were considered for the distribution of copper between chalcopyrite and cubanite. These three cases provided three different equations for expressing sulfide present in these minerals ($[S^{2-}]_{cp-cb}$) as a function of copper content of the sample ([Cu]). First, if all copper were present as chalcopyrite, then $[S^{2-}]_{cp-cb} = 1.01$ [Cu]. Second, if copper were divided equally between chalcopyrite and cubanite, then $[S^{2-}]_{cp-cb} = 1.26$ [Cu]. Third, if all copper were present as cubanite, then $[S^{2-}]_{cp-cb} = 1.51$ [Cu].

Three cases were also considered for the chemistry of pentlandite [$(Fe,Ni)_9S_8$], providing three equations for expressing sulfide present in pentlandite ($[S^{2-}]_{pn}$) as a function of the nickel content of the sample ([Ni]). First, Severson (2003) indicated pentlandite in the Duluth Complex contained

26.14 to 31.84 percent Ni, and this yields $[S^{2-}]_{pn} = 0.89$ [Ni]. Second, if pentlandite contained equal molar amounts of Ni and Fe, then $[S^{2-}]_{pn} = 0.97$ [Ni]. Third, microprobe analysis of one unleached pentlandite grain from sample 6-2-2 indicated 29.32 weight percent Ni and 34.01 weight percent S, respectively, and this yields the equation $[S^{2-}]_{pn} = 1.16$ [Ni].

To minimize the maximum error, the intermediate coefficients for both chalcopyrite-cubanite and pentlandite were used for calculations. Thus, the amount of sulfide associate with chalcopyrite-cubanite and pentlandite was calculated as follows (all values in percent).

$$[S^{2-}]_{cp-cb} + [S^{2-}]_{pn} = 1.26\ [Cu] + 0.97\ [Ni], \text{ where} \tag{6}$$

$[S^{2-}]_{cp-cb}$ = sulfide present with chalcopyrite and cubanite,
$[S^{2-}]_{pn}$ = sulfide present with pentlandite,
[Cu] = solid phase copper content and
[Ni] = solid phase nickel content.

If all sulfide is associated with pyrrhotite, chalcopyrite-cubanite, and pentlandite, then the amount of sulfide associated with pyrrhotite can be determined based on the sulfide ($[S^{2-}]$), copper, and nickel content of a sample. The sulfide associated with chalcopyrite-cubanite and pentlandite can be calculated, and the sulfide associated with pyrrhotite is the difference between this value and the sulfide content.

$$\begin{aligned}[S^{2-}]_{po,\ leached} &= [S^{2-}]_{leached} - ([S^{2-}]_{cp-cb,\ leached} + [S^{2-}]_{pn,\ leached}) \\ &= [S^{2-}]_{leached} - (1.26\ [Cu]_{leached} + 0.97\ [Ni]_{leached})\end{aligned} \tag{7}$$

Determination of the sulfide associated with pyrrhotite in the individual spatially distributed samples prior to weathering (the value $[S^{2-}]_{po,\ unleached}$ in equation 5) required additional assumptions. Chemical analyses of both bulk and spatially distributed samples indicated that sulfur content tended to increase with particle size. Data from the spatially distributed samples further indicated a spatial variation of sulfur contents. Because of these variations, there was no obvious single values to use for the sulfide content associated with pyrrhotite in the unleached spatially distributed samples.

The value of $[S^{2-}]_{po,\ unleached}$ was estimated assuming the ratio of sulfide associated with pyrrhotite to that associated with chalcopyrite-cubanite was constant for unleached rock. This ratio was calculated using the average bulk chemistry of the FL6 rock (0.79% S, 0.34% Cu, 0.09% Ni), and assuming all sulfur was initially present as sulfide. Based on this analyses, the amount of sulfide associated with copper and nickel sulfide was calculated using equation (7) as follows.

$$[S^{2-}]_{cp-cb} + [S^{2-}]_{pn} = 1.26\ [0.34] + 0.97\ [0.09] = 0.52\% \tag{8}$$

The sulfide present with pyrrhotite was calculated by difference.

$$[S^{2-}]_{po} = 0.79 - 0.52 = 0.27\% \tag{9}$$

Assuming the ratio of sulfide present with pyrrhotite to copper remains constant for the unleached solids (i.e. the ratio of pyrrhotite to chalcopyrite-cubanite is constant), this ratio can be quantified as follows.

$$([S^{2-}]_{po,\ unleached}/[Cu])_{unleached} = 0.27/0.34 = 0.79, \text{ or equivalently} \quad (10)$$

$$[S^{2-}]_{po,\ unleached} = 0.79\ [Cu]_{unleached} \quad (11)$$

Assuming copper is conserved in the solid phase (i.e. $[Cu]_{unleached} = [Cu]_{leached}$) , the initial sulfide content associated with pyrrhotite for a leached sample can be calculated as follows.

$$[S^{2-}]_{po,\ unleached} = 0.79\ [Cu]_{leached} \quad (12)$$

Substituted values for $[S^{2-}]_{po,\ leached}$ $[S^{2-}]_{po,\ unleached}$ (equations 7 and 11, respectively) into equation 12, the percent of pyrrhotite depleted can then be calculated as follows.

$$\Delta\ po = 100 \times (1 - S^{2-}{}_{po,\ leached}/S^{2-}{}_{po,\ unleached})$$

$$= 100 \times [1 - \{[S^{2-}]_{leached} - (1.26\ [Cu]_{leached} + 0.97\ [Ni]_{leached})\}/(0.79\ [Cu]_{leached})]$$

$$= 100 \times [0.79[Cu]_{leached} - \{[S^{2-}]_{leached} - (1.26\ [Cu]_{leached} + 0.97\ [Ni]_{leached})\}]/(0.79\ [Cu]_{leached})$$

$$= 100 \times (0.79[Cu]_{leached} - [S^{2-}]_{leached} + 1.26\ [Cu]_{leached} + 0.97\ [Ni]_{leached})/(0.79\ [Cu]_{leached})$$

$$= 100 \times (2.05\ [Cu]_{leached} - [S^{2-}]_{leached} + 0.97\ [Ni]_{leached})/(0.79\ [Cu]_{leached}) \quad (13)$$

These calculations indicated rock finer than 0.5 mm and 6.4 mm contributed 25 and 53 percent, respectively, of the pyrrhotite oxidation (Appendix 7). These values are likely on the lower end of the contribution of the fines. In particular, the calculated pyrrhotite oxidation in the fines neglected the higher percentage of sulfur associated with pyrrhotite in the fines relative to that in the bulk solids. This consideration would increase the pyrrhotite contribution of the fines by roughly 50 percent. The calculation also assumed no loss of copper from the fine fraction, and this reduced results by an undetermined amount.

This calculation was not expected to be accurate for each sample but to reflect the extent of oxidation over a large number of samples. A small amount of error was introduced by ignoring sulfide present with cobalt and zinc. Error was introduced due to assumptions regarding homogeneity in mineral content and mineral chemistry among samples. These values were known to vary, but this variation could not be quantified. It is assumed this variability would be accounted for by examining a large number of samples.

More specifically, the presence of copper as bornite (Cu_5FeS_4) and elemental copper was ignored, although single grains of each of these phases were observed in the 33 spatially distributed samples

examined using electron microprobe. Furthermore, nickel also occurs in pyrrhotite, olivine, ilmenite and magnetite, and may also be present in secondary iron oxyhydroxide phases. The nickel present in olivine, ilmenite, and magnetite could be accounted for using data from microprobe analyses of these samples. Because this analysis was intended to approximate the extent of pyrrhotite oxidation, this more rigorous approach was deemed unnecessary.

6.4.3.3. Visual Descriptions of Spatially Distributed Samples

Visual descriptions (Tables 38, 39, 40) were summarized from the field notes (Appendix 5, Attachments A5.1, A5.4.4).

6.4.3.4. Acid Concentration Present (ACP) and paste pH of the Spatially Distributed Samples

In the 0.6% S pile, ACP tended to decrease and the pH of the rock-water slurry (approximately 20 g solids per 50 mL deionized water) tended increase with increasing depth. Typical ranges for ACP and slurry pH for samples from the upper two lifts (1, 2) were 1 to 2 ppm g^{-1} and 4.0-4.3, respectively (Table 41). Typical ranges for lifts 3 and 4 were 0.4-0.6 and 0.3-0.6 ppm g^{-1} ACP, and 5.1-7.8 and 5.4-7.8 for slurry pH. The decreasing acidic signature with depth suggests that an acidic front was moving downward in the pile. Acid generation was apparently more rapid near the surface of the pile, perhaps due to a greater tendency to dry between precipitation events, higher oxygen levels, and/or higher sulfur contents in the rock near the surface.

The 0.8% S pile exhibited high ACP and lower slurry pH than the 0.6% S pile. There were no apparent consistent trends for ACP or slurry pH variation with depth. Lift 3 was the least acidic (relatively low ACP, high pH) and Lift 4 was the most acidic (Table 42). More detailed analysis of these data is warranted, particularly in the context of variations in sulfur content within the pile.

The 1.4% S pile had the highest ACP and lowest slurry pH values of the three test piles, and the extent of acidic conditions increased with depth (Table 43). Typical ACP ranges increased from 4-14 ppm g^{-1} at the surface of the pile to 30-50 ppm g^{-1} at the bottom of the pile. The variation of slurry pH with depth was less pronounced. It appeared that the range of values decreased with depth in the pile. The typical range for the surface of the pile (Lift 1) was 3.0 to 3.5 and all values from the bottom of the pile (Lift 4) were in the range of 3.1 to 3.3.

7. DISCUSSION

7.1. Introduction

Six test piles of Duluth Complex rock were constructed in 1977 to examine the effects of rock dissolution on water quality. As rock was excavated, its lithology was identified by visual examination, and samples of rock used in test pile construction were analyzed for sulfur, copper, and nickel. Some additional solid-phase analyses were conducted on a grab sample from one of the piles (chemistry, mineralogy) and a bulk sample collected from a pile that was partially dismantled in

1982 (particle size distribution, specific surface area and chemistry as a function of particle size, and mineralogical analysis of three fractions finer than 0.149 mm). The pile masses were determined based on a survey that determined pile dimensions. The quantity and quality of drainage from the piles was determined through 1993. In 1994, the piles were dismantled, and bulk sample were collected from three piles and analyzed for particle size distribution, chemistry as a function of particle size, and mineralogy. Specific surface areas were also determined for size fractions finer than 2.0 mm. From 59 to 63 spatially distributed samples were also collected from piles FL1 (0.6% S), FL6 (0.8% S), and FL5 (1.4% S). Sixteen of the spatially distributed samples from pile FL6 were analyzed for paste pH, residual acid content, particle size distribution, and chemistry as a function of particle size.

In the present study additional mineralogical analyses were conducted on size fractions from the spatially distributed samples from pile FL6. The chemistry and mineralogy of large (-3/+1.5 inch) fragments were determined with the intent of approximating the composition of the unleached rock. Microscope and microprobe analyses were conducted on fractions finer than 1.5 inches from three spatially distributed samples to determine the extent of sulfide mineral oxidation as a function of particle size. Microprobe analyses were conducted on the 30-70 mesh fractions of all 16 samples to determine the spatial variability of sulfide oxidation. The collection of previous and recent data were analyzed to determine what fraction of the rock contributed to chemical release and the solid-phases controlling chemical release.

The discussion will focus on test pile FL6 (0.8% S), on which solid-phase analyses were most extensive, and to a lesser extent on piles FL1 (0.6%S) and FL5 (1.4% S). These are the piles from which bulk and spatially distributed samples were collected in 1994 and, consequently, have the largest amount of solid-phase characterization data. Piles FL1 and FL6 were uncovered, and pile FL5 was covered with sandy till over coarse sand. Three additional 0.6% S piles were also studied. One uncovered pile, FL4, was dismantled in 1982, and the period of record for a pile covered with topsoil, FL3, ended in 1989. The final pile was covered with topsoil, and its period of record extended through 1993. These piles will be discussed only to elucidate the behaviors observed for FL1, FL6, and FL5.

7.2. Unleached Rock

Characterization of the rock prior to the beginning of the experiment was limited. The bulk sulfur, copper, and nickel contents of the piles were reasonably accurate, due to the large number of blast hole samples analyzed for these constituents (Table 5). These were average values and the compositional variability for these components was largest for the 1.4% S pile (FL5). The particle size distribution of the unleached rock was possibly similar to that determined for the bulk samples. These samples approximately 92 to 97 percent finer than 152 mm (six inches), 11 to 19 percent finer than 6.4 mm (0.25 inches), 5 to 11 percent finer than 2.0 mm, and 3 to 6 percent finer than 0.5 mm (Table 24).

The average mineral content of the unleached piles was estimated using the -6/+1.5-inch fraction from the three bulk samples and the -3/+1.5-inch fraction of the spatially distributed samples from

pile FL6. Analysis of the large particles from the bulk sample indicated the major host rock minerals were plagioclase (42-52%), augite (11-21%), olivine (1.6-18%), and hypersthene (3-5%). Particularly noteworthy was the low olivine content of pile FL5 relative to piles FL1 and FL6 (1.6 vs 13-18%). Relative to the bulk sample, analysis of the spatially distributed samples from FL6 yielded good agreement for the plagioclase content, lower olivine and augite contents, and higher hypersthene content. The discrepancies for the augite and hypersthene contents were 11 and 13 percent respectively (Table 7). This provides a measure of the quantitative variability of these results, which are dependent on the representativeness of the sample. It is believed that the bulk sample provided a more accurate assessment. It was roughly two tons in mass, as compared to the spatially distributed result, which was the average composition of 16 hand-sized particles. Analysis of the minerals present in the hand sized samples yielded average mineral compositions for plagioclase [$(Na_{0.4}\ Ca_{0.56})\ (Si_{2.41}\ Al_{1.6})\ O_8$], augite [$(Ca_{0.81}Na_{0.02})(Mg_{0.74}Fe_{0.38})(Si_{1.97}Al_{0.21})O_6$], olivine [$(Mg_{1.19}\ Fe_{0.8})\ SiO_4$], and hypersthene [$(Mg_{0.95}\ Fe_{0.96})\ Si_2O_6$] (Table 12).

Analysis of the bulk sample from pile FL1, which was dismantled in 1982, indicated that sulfur content increased as particle size decreased (Table 8). The sulfur content of particles finer than 0.50 mm in diameter was 1.24, or roughly twice the bulk sulfur content. This was apparently due to the fact that sulfide minerals are softer than other minerals in the rock. In addition, the fraction of sulfur associated with pyrrhotite in this fine fraction was calculate as 1.5 times that in the bulk rock. The sulfides in the finer fraction would also tend to be liberated from the host rock matrix to a far greater extent than those in the coarser particles. That is, a greater fraction of the sulfide mineral surface area would be available for reaction. The higher sulfur content, greater fraction of sulfur associated with pyrrhotite, and higher liberation all increase the potential of the fine fraction to produce acid.

7.3. Drainage Quantity

The yield coefficients for piles FL1 and FL6, which were uncovered, were 0.51 and 0.54, respectively. These values are in good agreement with the 0.55 value presented by Nichol et al. (2003) for uncovered test piles in northern Saskatchewan. Pile FL5 was covered with sandy till over coarse sand and, consequently, had a lower yield coefficient of 0.32.

In addition to the amount of water moving through a rock pile, solute transport is also affected by the flow path. Of particular interest are the fraction of the rock contacted by flow through the pile and the characteristics, particularly particle size, of the rock contacted. Morin and Hutt (1994) estimated that 5 to 20 percent of the rock surfaces more than one meter deep in the pile are regularly flushed by infiltration. Eriksson et al. (1997) indicated that 30-45% of the water content of a rock pile is stagnant and the remaining 55-70% flows preferentially. Tran et al. (2003) indicated that preferential flow may occur through particles size fractions with a median diameter in the range of 5 to 30 mm. That is, reaction products from this size fraction may be more readily transported than those in larger or smaller fractions. However, Strömberg and Banwart (1999) concluded rock finer than 0.25 mm in diameter constituted 30 percent of the mass of a rock pile in Sweden, but contributed 80 percent of the solute release. That is, the fines contributed a disproportionately high chemical mass release.

7.4. Drainage Quality

The pH of drainage from the three piles tended to decrease as sulfur content increased. The minimum annual flow-weighted pH for the 0.6%, 0.8% and 1.4% sulfur piles were 5.1, 4.0, and 3.4, respectively. These pH values were an indication of the balance between acid production and acid neutralization for the piles. Acid production rates were reflected by sulfate release rates, which increased with the sulfur content of the rock. Acid neutralization rates were reflected by calcium and magnesium release rates. The average rate of calcium release was relatively constant with respect to sulfur content, while that of magnesium of magnesium release was highest for the 0.8% S pile (Table 21). The low rate of magnesium release, and implied low acid neutralization, from the 1.4% S pile was apparently the result of the low olivine content of the rock. The difference between these rates represents excess acid production, and increased from 0.53 mmol/(T day) for the low sulfur pile to 4.7 mmol/(T day) for the high sulfur pile. This is consistent with the pH trend observed.

Chemical mass release from each of the piles was expressed as a fraction of the chemical constituent mass originally present. Sulfur release ranged from 6.8 to 11 percent of that originally present. Copper release was considerable lower, ranging from 0.26 to 3.3 percent of that originally present. So when normalized for the amounts present, copper release ranged from about 4 to 40 percent of sulfur release (Table 23). That is, sulfur was released preferentially from the pile relative to copper. In contrast, the percent release for nickel was close to that of sulfur for the 0.6% S pile and roughly three times that of sulfur for the 1.4% S pile. Nickel was also release preferential to sulfur in the 0.8% S pile.

Geochemical modeling indicated that copper and nickel were relatively mobile over the range of solution compositions likely to be present in the piles. This suggests that release of these metals was controlled by the rate of dissolution of the minerals in which they occurred. For pile FL6, the total respective sulfur and copper releases over the period of record were 12 and 2.6 percent of the associated mass originally present (Table 23). This indicates that sulfur release per unit mass originally present was 4.6 times that of copper. This further suggests that pyrrhotite oxidation must have contributed more of the sulfate than copper sulfide mineral oxidation. This is qualitatively consistent with microprobe analyses of the spatially distributed samples. Analyses indicated pyrrhotite was heavily oxidized or absent in the particles finer than 0.5 millimeters in diameter (Table 37). In contrast, chalcopyrite and cubanite grains were typically relatively fresh and pentlandite grains were oxidized to an intermediate extent (Figure 6).

The ratios of copper and nickel to sulfate increased in the order FL1 (0.6% S) < FL6 (0.8% S) < FL5 (1.4% S) (Tables 21-23). This appears somewhat unusual because the fraction of sulfur present as trace metal sulfides decreased in this order. This suggests that the relative oxidation rates of trace metal sulfides and pyrrhotite was different for the three piles. It is believed that this difference was due to pH conditions. Drainage pH decreased in the order FL1 > FL6 > FL5. Apparently bacterial catalysis enhanced trace metal sulfide oxidation to a greater degree than pyrrhotite oxidation as pH became more acidic.

7.5. Acid Production by Pyrrhotite Oxidation

Drainage quality data were used to determine both the total sulfide mineral oxidation and the fraction of this oxidation contributed by pyrrhotite. The observed sulfate release from pile FL6 was 36,000 moles, representing about 12 percent of the sulfur originally present (Tables 22, 23). Most of the release was from pyrrhotite but some was the result of oxidation of copper and nickel sulfides. (Some sulfate release occurred due to oxidation of cobalt and zinc sulfides, but this amount was relatively small. This release was ignored in calculations.) Sulfate release from copper sulfides was estimated assuming congruent dissolution of equal amounts of chalcopyrite and cubanite (with subsequent precipitation of iron oxyhydroxide). The copper released with drainage was 1700 moles, and this was multiplied by 2.5 to yield a release of 4200 moles of sulfate from copper sulfide oxidation. Similarly, sulfate release from nickel sulfides was estimated assuming all nickel release was from pentlandite ($Ni_{4.5}Fe_{4.5}S_8$). Nickel released with drainage was 2800 moles, and this was multiplied by 1.78 to yield a release of 4900 moles from oxidation of pentlandite. Thus, the sulfate release due to oxidation of pyrrhotite was calculated as 36,000 - (4200 + 4900) = 27,000 moles, or 75 percent of the total sulfate release.

The total sulfur release was also calculated based on analysis of the FL6 bulk sample particle size fractions, indicated a mass weighted mean sulfur content of 0.74 percent (Table 28). Furthermore, the average chemistry of the minus 6.4 mm of the16 spatially distributed samples from this pile, including sulfur content, was in good agreement with that of the bulk sample (Table 25). Subtracting the sulfur content of the bulk sample from that the unleached sulfur content of the pile (Table 5) yields a difference of 0.05% S, representing a sulfur loss from the pile of 650 kg or 20,000 moles. This value is roughly 56 percent of the sulfate release observed (Table 22). This agreement is better than might be expected, given the difference between mass weighted sulfur contents of piles FL1 and FL5 and their respective original sulfur contents (Tables 27, 29). In both cases the loss of sulfur calculated from the change in solid-phase sulfur content was considerably greater than that indicated by drainage quality analysis (Tables 22, 23). It is believed this discrepancy reflects the difficulty in collecting a representative sample from the piles.

7.6. Acid Production Contributed by Fines

Particle size and spatial variation are two aspects of the fraction of FL6 rock contributing to chemical mass release. Dissolution per unit mass tend to increases as particle size decreases, because a given mass of small particles has a larger surface area than large particles. Solid-phase analyses indicated that oxidation of pyrrhotite present in fine particles contributed a substantial amount of sulfate to solution. About 12 percent of the FL6 rock was finer than 6.4 mm and 3 to 8 percent was finer than 0.5 mm. Mineralogical examination of the bulk sample indicated most of the pyrrhotite in rock finer than 0.5 mm was heavily oxidized. Examination of the spatially distributed samples from the pile supported this observation.

The mass of sulfur release from pyrrhotite was calculated based on data from both the bulk sample and the spatially distributed samples. For the bulk sample it was assumed that all sulfur present as pyrrhotite in particles finer than 0.5 mm was oxidized. This calculation was made as follows.

$$M(SO_4)_{0.5} = (P_{0.5}/100)\ M_{pile}\ F_S\ (S_{bulk}/100)\ f_{Spo}/32.06 \quad (14)$$

where $M(SO_4)$ = moles of sulfate released from pyrrhotite present in particles finer than 0.5 mm

$P_{0.5}$ = for unleached rock, the percentage of rock finer than 0.5 mm (3 - 8%)

M_{pile} = original mass of pile in grams (1300 x 10^6)

F_S = for unleached rock, ratio of sulfur content in particles finer than 0.5 mm to bulk sulfur content for the unleached rock (2-2.5)

S_{bulk} = original bulk sulfur content of the pile, percent (0.79)

f_{Spo} = for unleached rock finer than 0.5 mm, the fraction of total sulfur initially present with pyrrhotite (0.34-0.51)

The range of percentages for rock finer than 0.5 mm was based on the range observed for the three test piles and experimental bin 6B (Table 24). The mass of the pile was based on the MN DNR survey of the pile, as described in section 6.2.1. The lower value for the ratio of sulfur content in particles finer than 0.5 mm to the bulk sulfur content was selected based on data from the FL4 bulk sample collected in 1992 (Table 8). The upper end was selected from solid phase data presented elsewhere for particle size fractions of Duluth Complex rock (Lapakko et al. 2004). The bulk sulfur content is that determined by analysis of blast hole samples for the piles (Table 5). The lower value for the fraction of the total sulfur present with pyrrhotite is based on the bulk chemistry of the unleached FL6 rock (Table 5). The upper value was determined using data from the FL4 bulk sample, which indicated the fraction of sulfur associated with pyrrhotite in the fractions finer than 0.5 mm was roughly 1.5 times that in the bulk rock. The lower end of the range for the fraction of total sulfur present with pyrrhotite was multiplied by this value to determine the upper end of the range. (For determinations of sulfur present with pyrrhotite it was assumed that copper was present in equal amounts of chalcopyrite and cubanite and nickel was present in pentlandite.)

The sulfur release from the pyrrhotite present in particles finer than 0.5 mm was calculated using the eight different permutations of values. The results of the calculations averaged 15,000 moles of sulfur and ranged from 6500 to 33,000 moles. These values correspond to 56, 24, and 122 percent, respectively, of the total sulfate contributed to drainage by pyrrhotite oxidation. Specifically, using the average release calculated based on the bulk sample analysis, oxidation of pyrrhotite in particles finer than 0.5 mm contributed 55 percent of the sulfate to drainage. This value corresponds to the sulfate release from about five to six percent of the pile mass.

Data from the spatially distributed samples were also used to calculate the extent of pyrrhotite oxidation in the fines. These calculations indicated rock finer than 0.5 mm and 6.4 mm contributed 25 and 53 percent, respectively, of the pyrrhotite oxidation (Appendix 7). These values are likely on the lower end of the contribution of the fines. In particular, the calculated pyrrhotite oxidation in the fines neglected the higher percentage of sulfur associated with pyrrhotite in the fines relative to that in the bulk solids. This consideration would increase the pyrrhotite contribution of the fines by roughly 50 percent. The calculation also assumed no loss of copper from the fine fraction, and this reduced results by an undetermined amount.

Analysis of data from the bulk sample and the spatially distributed sample suggest that roughly 50 percent of the pyrrhotite oxidation was contributed by the fines in pile FL6. This indicates that roughly five to ten percent of the test pile mass was responsible for half the acid production observed.

7.7. Spatial Variability of Acid Production

Examination of the 16 spatially distributed 30-70 mesh samples indicated that pyrrhotite was heavily oxidized in 70 percent of the samples (reaction ratings of 4 or 5 in Table 37), moderately oxidized in 18 percent of the samples (reaction rating 3), and relatively fresh (reaction rating 1) in 12 percent of the samples (Table 37). This suggests that 70 percent of the rock present contributed to reactions. Rock at the surface was most highly reacted, with nearly all the pyrrhotite highly reacted. The extent of reaction decreased with depth into the pile, and pyrrhotite oxidation in two of the four samples from the bottom lift was negligible and that for a third sample was limited. This indicates that at a depth of roughly three meters into the pile, oxidation of sulfide minerals decreased substantially.

The lesser oxidation may have been the result of decreased water contact or limited oxygen diffusion into the bottom of the pile. Morin and Hutt (1994) estimated that 5 to 20 percent of the rock surfaces more than one meter deep in the pile are regularly flushed by infiltration. The present data suggest oxidation was limited in 25 to 50 percent of the bottom lift, indicating that rock flushing was somewhat higher than the range suggested.

Calculations assuming homogeneous rock pile characteristics indicated that the pile was oxygen saturated (Appendix 7). However, it is possible that inhomogeneous conditions could exist. This possibility is supported by the observation of bornite (Cu_5FeS_4) and cubanite grains coated by bornite in the bottom lift. The presence of bornite suggests reducing rather than oxidizing conditions. Oxygen transport in these areas may have been limited by the cementation of particles. In the upper three lifts of the pile, the extent of agglomeration in the upper three lifts was typically noted as none to light, but that in the fourth lift was typically moderate to heavy (Table 39). These agglomerates consisted of fine and coarse particles and may have limited oxygen transport, especially if saturated with water. Furthermore, water in these agglomerates would likely remain trapped, due to differences in hydraulic conductivity with the surrounding materials.

8. CONCLUSIONS

Drainage quantity and quality from six test piles of Duluth Complex rock was determined for as long as 16 years. Solid-phase analyses were conducted on the unleached and leached rock. The objectives of the present study were to determine the extent of rock contributing to chemical release and the solid-phases controlling release of acid and trace metals.

Acid release was the balance of acid producing and acid neutralizing reactions. Oxidation of pyrrhotite was the dominant source of acid production. Acid neutralization was dominated by dissolution of calcium-bearing minerals above roughly pH 5.0 and by dissolution of magnesium

bearing-minerals below this value. The dominant calcium minerals were plagioclase and augite, and the dominant magnesium minerals were augite and olivine.

Copper release resulted from oxidation of chalcopyrite and cubanite, and pentlandite oxidation provided most of the nickel release. These metals were mobile in solution under most conditions observed in the drainages. The rates of trace metal sulfide oxidation, therefore, controlled trace metal release and were apparently accelerated under acidic conditions.

Particles finer than 0.5 mm constituted roughly five to six of the rock pile mass but were responsible for an estimated 25 to more than 50 percent of the pyrrhotite oxidation. An estimated 70 percent of the rock finer than 0.5 mm contributed to pyrrhotite oxidation. Rock at the surface of the piles oxidized to a greater degree than that at depth. Oxidation at depth may have been limited by availability of water or oxygen transport in specific areas. The oxygen transport may have been inhibited by the cementation of rock over time. Such cementation was observed during the dismantling of the rock piles.

9. ADDITIONAL RESEARCH NEEDS

9.1. Overview

The present study revealed that sulfides present in the 0.8% sulfur waste rock test pile (FL6) oxidized in the order pyrrhotite > pentlandite > chalcopyrite-cubanite. This preferential oxidation is consistent with previous reports on tailings (Jambor 2003). Drainage quality data suggest that this preferential oxidation also occurred in a 0.6% S pile (FL1) but not in a 0.8% S pile (FL5). This ordering, in conjunction with electron microprobe photomicrographs, provided a means of assessing the extent of sulfide mineral oxidation for 16 spatially distributed samples. The extent of oxidation was used to estimate 1) the amount of pyrrhotite oxidation that occurred over 17 years, 2) the variation of oxidation with particle size, and 3) the spatial variation of oxidation.

The results of this survey suggest that the majority of acid production was contributed by fine particle size fractions that constituted less than 10 percent of the pile mass. With regard to environmental mine waste management, this indicates that the mass of some types of waste rock requiring special handling and remediation may be much smaller than the mass of rock present.

The results may also have implications regarding recovery of economic values from abandoned mine rock. These values are often associated with sulfide minerals. For example gold may be included in pyrite, and results of the present study indicate sulfide minerals tend to be concentrated in the fine fraction. Furthermore, this fraction is more prone to oxidation and, consequently, gold may be liberated from sulfide minerals during the course of environmental weathering. This liberation may expedite recovery.

Chemical and electron microprobe analyses were essential to developing the estimations in the present study. In addition to the particle size and preferential mineral oxidation implications mentioned above, these analyses indicated that 1) the extent of chemical mass release varied

spatially and 2) although oxygen transport throughout the pile was predicted, microenvironments exist that are chemically reducing. These analyses were limited to 16 spatially distributed samples from a single pile, and roughly 60 of these samples were collected from each of three piles. Additional chemical and electron microprobe analyses would 1) allow verification of assumptions from the present study and 2) provide a more sound basis for quantifying the variation of sulfide mineral oxidation (and attendant acid production) as a function of particle size and the spatial variation of this oxidation.

Furthermore, the cumulative volume of data from previous and present studies is very large. Considerable additional effort could be applied to analysis of these data to better understand the behavior of abandoned waste rock piles (and heap leach piles) and efficient methods for remediating environmental consequences of these piles. Solid phase data generated in the present and previous studies (particularly mineral contents and chemistries), the existing long-term drainage record, and additional focused solid-phase analyses can be used to develop a kinetic model describing waste rock dissolution. Because the drainage quality and quantity data are so extensive, detailed solid-phase data and systematic data analysis would provide the information necessary for such a model. This would be particularly helpful in predicting the quality of drainage from waste rock piles and designing future waste rock facilities in an environmentally sound manner. Several attempts at such modeling have been made with limited success. The data presently available and additional analysis of available rock samples provide a valuable opportunity to advance this important endeavor.

9.2. Summary

Two major research needs were identified in the present survey of abandoned waste rock piles. First, additional solid-phase chemical and mineralogical analyses should be conducted on samples that were collected from the three rock piles. Of the 180 samples available from the three piles, only 16 spatially distributed samples from one pile (FL6, 0.8% S) have been subjected to detailed analyses. Existing solid-phase and drainage quality data, together with additional data analysis, provide a sound perspective for developing a focused analytical strategy. This strategy would include further analyses of samples from pile FL6 and systematic analysis of spatially distributed samples from piles FL1 (0.6% S) and FL5 (1.4% S).

Second, extensive analysis is required on existing data on both solid and aqueous phases and on additional solid phase data generated in future work. The present survey addressed the relationship among the initial solid-phase composition, drainage quality and quantity, and leached solid-phase composition. It would be highly valuable to analyze the data collected, in conjunction with additional solid-phase analyses, in the context of other recent related research. Recent research includes mineral dissolution kinetic studies, laboratory dissolution of Duluth Complex rock as a function of particle size (Lapakko et al. 2004), and extensive long-term laboratory experiments on Duluth Complex rock conducted at the MN DNR (unpublished data). One aspect of particular interest involves kinetic modeling using detailed solid phase data to interpret the long-term chemical release from the piles. Such a model would be of benefit in predicting the behavior of future rock piles, existing abandoned rock piles, and abandoned rock piles to which remedial measures have been applied.

10. ACKNOWLEDGMENTS

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Table 1. Summary of chemical and physical characteristics of the test piles.

	FL1	FL2	FL3	FL4	FL6	FL5
Date completed	4/20/77	4/20/77	4/20/77	4/20/77	9/30/77	9/30/77
S (%)	0.64	0.64	0.64	0.64	0.79	1.41
Cu (%)	0.35	0.35	0.35	0.35	0.34	0.31
Ni (%)	0.083	0.083	0.083	0.083	0.09	0.09
Mass (T)	1100	1100	830	1700[1]	1300	815
Volume (m^3)	540	530	400	930	630	400
Collecting area (m^2)	290	340	300	330	320	340
Surface area (m^2)	330	450	430	350	390	370
Cover material	none	topsoil	coarse sand (glacial till)	none	none	sandy till over coarse sand
Avg. cover depth (cm)	0	23	34	0	0	54
Vegetated	no	yes	yes	no	no	yes
Drainage period	1978-93[2]	1978-93	1978-89[3]	1978-89	1978-93	1978-93

[1] Approximately 40% of pile FL4 was removed in 1982.
[2] FL1 was partially covered with plastic during 1990
[3] FL3 was covered with plastic and Hypalon terminating flow at the end of the 1989 field season.

Table 2. Blast round chemistry and rock type for the 0.6% pile (from monthly reports).[1]

Depth (ft.)	Round	Footage used[2]	Average Concentrations (%)			Rock Type	Date
			Sulfur	Copper	Nickel		
716 ↕ 773	115 ↕ 122	44	0.44	0.27	0.07	gabbro	December 1976
893 ↕ 992	141 ↕ 156	99	0.62	0.38	0.09	anorthosite and troctolite anorthosite and gabbro troctolite augite and troctolite	January 1977
998 ↕ 1088	157 ↕ 172	36	0.37	0.26	0.07	coarse grained troctolite fine grained, inclusive hornfels	February 1977
1132 ↕ 1325	180 ↕ 209	132	0.68	0.37	0.08	medium-coarse grained troctolite, band of hornfels with minor hornfels scattered throughout	March 1977
1366 ↕ 1502	216 ↕ 238	105	0.77	0.36	0.08	medium-coarse grained troctolite coarse grained olivine gabbro	April 1977
1506 ↕ 1540	240 ↕ 245	12	0.50	0.27	0.06	augite troctolite	May 1977

[1] Monthly reports from AMAX geologists are on file at the MN Dept of Natural Resources Lands and Minerals office in Hibbing, MN.

[2] Some rounds (footage) within the intervals were not used to construct the leach pile.

Table 3. Blast round rock type for the 0.8% pile (from monthly reports).[1]

Drift	Round	Rock Type	Date
A	151 ↕ 221	hornfels troctolite	September 1977
A/B/C	152 ↕ 173	hornfels troctolite	September 1977
B/C	175 ↕ 222	hornfels	September 1977

[1] Monthly reports from AMAX geologists are on file at the MN Dept of Natural Resources Lands and Minerals office in Hibbing, MN.

Table 4. Blast round rock type for the 1.4% pile (from monthly reports).[1]

Drift	Round	Rock Type	Date
A	26 ↕ 128	variable intercalating mineralization interbanded troctolite some hornfels massive pyrrhotite sulfide lenses significant Cu/Ni mineralization	August 1977
B	28 ↕ 127	fine grained hornfels troctolite	August 1977
A/B/C	129 ↕ 149	hornfels troctolite	September 1977

[1] Monthly reports from AMAX geologists are on file at the MN Dept of Natural Resources Lands and Minerals office in Hibbing, MN.

Table 5. Sulfur, copper, and nickel concentration statistics in percent from the rounds placed into the test piles.

	Sulfur	Copper	Nickel
0.6% S Pile			
n	67	67	67
minimum	0.27	0.20	0.04
maximum	1.29	0.58	0.15
mean	0.64	0.35	0.08
standard deviation	0.27	0.11	0.03
median	0.58	0.32	0.08
0.8% S Pile			
n	27	27	27
minimum	0.25	0.10	0.03
maximum	1.37	0.56	0.20
mean	0.79	0.34	0.09
standard deviation	0.29	0.13	0.05
median	0.74	0.36	0.08
1.4% S Pile			
n	39	39	39
minimum	0.39	0.09	0.02
maximum	7.65	1.04	0.43
mean	1.45	0.31	0.09
standard deviation	1.33	0.19	0.08
median	1.03	0.26	0.06

Table 6. Selected chemistry for unweathered rock surrogates. Concentrations in percent unless otherwise noted. Analysis by Chemex (Sparks, NV). Additional data in Appendix 4.

Oxide/ Element	Grab Sample[1]	Bulk Samples, −6+1½"[2]				Spatially Distributed Samples[3]
	0.6%S	0.6% S	0.8% S	1.4% S	0.6% S	0.8% S
	Test Pile FL1	Test Pile FL1	Test Pile FL6	Test Pile FL5	Bin 6B	Test Pile FL6
S	0.66	0.45	0.60	0.74	0.95	0.39
SO_4 as S	NA	0.02	0.04	0.01	0.04	NA
Cu, ppm	3060	2390	3370	1455	4050	1964
Ni, ppm	730	507	570	315	767	607
Co, ppm	90	51	63	35	86	95
Zn, ppm	74	70	74	58	96	78
SiO_2	48.81	44.57	45.05	46.75	43.73	45.32
TiO_2	2.35	2.42	1.65	2.60	1.60	1.87
Al_2O_3	16.96	17.69	14.58	16.41	15.23	14.20
Cr_2O_3	0.15	0.03	0.04	0.06	0.04	0.02
Fe_2O_3[4]	13.18	14.73	15.38	16.20	18.05	16.60
MnO	0.16	0.15	0.18	0.16	0.18	0.17
MgO	7.15	7.55	9.74	6.38	9.43	10.40
CaO	7.76	8.24	8.86	6.45	7.10	7.16
Na_2O	2.70	2.38	2.00	2.11	2.26	1.81
K_2O	0.39	0.43	0.52	0.97	0.58	0.68
P_2O_5	0.05	0.20	0.25	0.42	0.24	0.23
LOI	NA	<0.01	0.26	0.34	0.18	1.34

[1] Collected in 1981.
[2] Collected in 1994.
[3] Average of 16 hand-sized spatially distributed samples collected in 1994.
[4] All Fe as Fe_2O_3.
NA: not analyzed

Table 7. Mineralogy for unweathered rock surrogates. Values in weight percent except FL1 grab sample, which is reported in volume percent.

Mineral	Grab Sample	Bulk Samples[1]			Spatially Distributed[2]
	0.6% S	0.6% S	0.8% S	1.4% S	0.8% S
	FL1[3]	FL1	FL6	FL5	FL6[7]
Feldspar	59.1	51.9	44.5	41.8	43
Olivine	10.5	13.5	18.4	1.6	13
Clinopyroxene	11.2	12.7	20.9	11.2	10
Orthopyroxene	3.7	4.4	3.2	4.8	16
Biotite	1.7	2.0	3.3	9.6	2
Alteration	5.2[4]	7.1[4]	4.7[4]	14.5[4]	6
Ilmenite	3.4[5]	4.4	2.8	4.2	3
Magnetite	[5]	2.2	<0.1	5.2	0.4
Apatite	0.085	0.5	0.6	1.0	trace
Quartz	<0.1	<0.1	<0.1	4.2	0.9
Cordierite	0.03	<0.1	<0.1	<0.1	3
Calcite	0.06	<0.1	<0.1	<0.1	trace
Pyrrhotite	0.84	0.38	0.38	1.38	0.5
Chalcopyrite	0.77[6]	0.66	0.93	0.39	0.2[6]
Cubanite	[6]	0.04	0.06	0.05	[6]
Pentlandite	0.04	0.15	0.17	0.09	NR
Cobaltite	NR	0.01	0.02	0.01	NR
Sphalerite	NR	0.01	0.01	0.01	NR
Galena	NR	<0.01	<0.01	<0.01	NR
Gypsum	NR	0.11	0.22	0.05	NR

1 Analysis by Midland Research of –6/+1½ inch fraction.
2 Analysis by McSwiggen and Associates of approximately three-inch-diameter samples. Calculated by determining average mode (sum of modes of mineral divided by number of samples with that mineral), multiplying by number of samples with that mineral and dividing by total number of samples (16).
3 Also reported 2.0% epidote and 2.2% sericite.
4 Alteration is mainly primary amphiboles with serpentine, chlorite, and minor secondary montmorillonite.
5 Includes magnetite.
6 Includes cubanite.
7 Also reported 0.6% sericite, 0.2% scapolite
NR: not reported.

Table 8. Characterization of the fine fractions of the 0.6% FL4 test pile material used in the joint disposal bins.

Size Fraction mm	pH	Total Sulfur %	Sulfide %	Cu %	Ni %	Co %	Zn %
Sand							
0.50–2.00	6.56	0.67	0.65	0.337	0.062	0.012	0.015
0.177–0.50	6.37	0.80	0.75	0.381	0.078	0.013	0.015
0.149–0.177	6.18	0.88	0.80	0.391	0.103	0.015	0.015
0.105–0.149	NA[1]	1.12	1.05	0.495	0.147	0.019	0.017
0.075–0.105	5.95	1.37	1.30	0.585	0.221	0.022	0.019
Silt and clay							
0.053–0.075	5.78	1.65	1.57	0.647	0.253	0.024	0.019
<0.053	5.77	1.94	1.86	0.814	0.295	0.027	0.023
mass weighted average concentration[2]		1.06	1.01	0.471	0.134	0.017	0.017

[1] NA = not available
[2] for –2.00 mm rock

Table 9. Sulfide mineral leaching scale where larger numbers indicate greater degrees of leaching. See text for further details and Appendix 5, Attachment A5.7.1.

Leaching Scale	Pyrrhotite	Cubanite/Chalcopyrite	Pentlandite
1	1	1	1
2	2	1-2	2
3	3	1-2	2-3
4	4	1-2	3-4
5	5	3	4-5
6	5	4	4-5
7	5	5	5

Table 10. Summary of analytical testing for the bulk and spatially distributed samples.

Parameter	Bulk Samples (%S content)						Spatially Distributed Samples[1]
	FL1 (0.6)	FL2 (0.6)	FL3 (0.6)	FL4 (0.6)	FL5 (1.4)	FL6 (0.8)	
unleached, original samples							
mass of piles	1976-1977						
particle size				1982[2]			
spec. surface area				1982			
S, Cu, Ni	1976-77				1976-77	1976-77	
chemistry (WR,TM)[3]	1981[4];1994			1982[5]; 1994	1994	1994	
lithology	1976-77				1976-77	1976-77	
mineralogy	1981; 1994			1982	1994	1994	1994; 2003 (FL6)
during pile existence							
drainage quality	1978-1993	1978-1993	1978-1989	1978-1989	1978-1993	1978-1993	NAp[6]
mass release + rates	SO_4, Ca, Mg, Cu, Ni, Co, Zn over the above periods of record						NAp
leached samples							
visual description							1994
particle size	1994				1994	1994	1994 (FL6)
spec. surface area	1994				1994	1994	
percent moisture	1994				1994	1994	
paste pH							1994
ACP[7]							1994
chemistry (WR, TM)[3]	1994				1994	1994	1994 (FL6)[8]
mineralogy	1994				1994	1994	1994 (FL6)[9]

1 Spatial samples exist for FL1, FL6, and FL5.
2 All samples collected after the construction of the piles were used to represent the characteristics of the original properties of the test pile rock.
3 WR = whole rock, including sulfur; TM = trace metals
4 Analyzed for sulfur, copper, nickel, iron with sulfur, cobalt, zinc, whole rock, and limited trace metals.
5 About a third of pile FL4 was removed and transferred to construct experimental bins. Analyzed for paste pH, total sulfur, sulfate, copper, nickel, cobalt, and zinc and a limited suite of trace metals.
6 NAp = not applicable
7 ACP = acid concentration present
8 Including total carbon and carbon dioxide
9 XRD and XRF analyses were conducted at the USBM but not interpreted.

Table 11. Determinations of test pile mass, rock removed, and rock reclaimed. Values in metric tons.

Pile	Initial Mass		Mass Removed[1]	Mass Reclaimed	
	Minnamax	MN DNR		Minnamax	MN DNR
FL1 (0.6% S)	1622	1100	290	1332	810
FL2 (0.6% S)	1622	1080	0	1622	1080
FL3 (0.6% S)	1622	830	0	1622	830
FL4 (0.6% S)	1622	1700	0	1622	1700
FL6 (0.8% S)	1566	1300	430	1136	870
∑ of 0.6 + 0.8 %S	8054	6010	720	7334	5290
FL5 (1.4% S)	2155	815	270	1880	545
Determined at reclamation					
FL1+FL2+FL3+FL4+FL6+site materials[2]	NAp	NAp	NAp	NAp	10,230[2]
FL5 + site materials[3]	NAp	NAp	NAp	NAp	1745[3]

[1] Rock moved to Hibbing in 1994.
[2] Included 500 t of basin sand went to the pugmill with rock from the other piles.
[3] Included 130 t of borehole cuttings and 100 t of a sludge-sand mixture from the settling basin.
NAp = not applicable

Table 12. Mode in weight percent of the sixteen spatially distributed sample from the 0.8 %S pile (FL6). Analysis by McSwiggen and Associates.

Sample	Plag	Hyp	Ol	Aug	Serp	Qtz	Kspar	Bio	Sericite	Sulfides	Ilm	Mag	Rock Name[1]
6-1-02	21	39	0[2]	14	0	8	0	10	0	0	8	0	mela-norite
6-1-04	48	0	26	12	0	0	0	0	6	9	0	0	troctolite
6-1-05	33	52	0	0	0	0	0	7	0	0	8	0	mela-norite
6-1-07	55	29	0	0	0	0	3	2	0	0	12	0	norite
6-2-02	40	8	32	8	0	0	0	5	0	4	3	0	picrite
6-2-04	26	40	21	0	0	0	0	5	0	0	8	0	olivine pyroxenite
6-2-07	53	0	27	2	18	0	0	0	0	0	0	0	troctolite
6-2-10	41	34	0	14	0	2	1	0	0	4	3	0	mela-norite
6-3-01	29	0	24	11	30	0	0	0	0	3	0	4	feldspathic peridotite
6-3-06	46	0	23	0	22	0	0	0	0	0	0	9	troctolite
6-3-08	37	0	11	35	0	0	0	0	0	0	16	0	mela-gabbro
6-3-16	47	13	29	11	0	0	0	0	0	0	0	0	gabbro
6-4-02	51	0	11	30	0	0	0	1	0	6	0	0	gabbro
6-4-04	15	58	0	0	0	0	7	0	8	8	5	0	hornfels
6-4-05	48	0	26	15	0	0	0	5	0	0	6	0	gabbro
6-4-07	31	37	0	14	0	2	8	0	0	0	8	0	pyroxenite

[1] Rock names determined by Rick Ruhanen of the Minnesota Department of Natural Resources, Division of Lands and Minerals, based on mode in volume percent by McSwiggen and Associates and using classification scheme DNR-modified from Phinney 1972.

[2] 0 = none reported

Table 13. Average mineral compositions for 0.8%S pile (FL6). Analysis by McSwiggen and Associates.

Mineral	Composition[1]
plagioclase	$(Na_{0.4}\ Ca_{0.56})\ (Si_{2.41}\ Al_{1.6})\ O_8$
hypersthene	$(Mg_{0.95}\ Fe_{0.96})\ Si_2O_6$
augite	$(Ca_{0.81}Na_{0.02})(Mg_{0.74}Fe_{0.38})(Si_{1.97}Al_{0.21})O_6$
olivine	$(Mg_{1.19}\ Fe_{0.8})\ SiO_4$
potassium feldspar	$(K_{0.58}Na_{0.22}Ca_{0.17})Al_{1.21}Si_{2.81}O_8$
serpentine	$(Mg_{2.25}Fe_{0.71})Si_2O_5(OH)_4$
biotite	$(K_{1.67}Na_{0.05}Ca_{0.01})(Mg_{2.77}Fe_{2.01}Ti_{0.54})[Al_{2.37}Si_{5.23}O_{20}(OH)_2]$
cordierite	$(Mg_{1.33}Fe_{0.66})Al_4Si_5O_{18}$

[1] Determined based on stoichiometric coefficients (S) for each element in a mineral presented by McSwiggen and Associates. (S × weight percent) to yield number of atoms for each of the 16 spatially distributed samples. The atom number for each element was averaged across the samples which contained the given mineral.

Table 14. Yield coefficients for the six test piles.

		FL1	FL2	FL3[1]	FL4[2]	FL6	FL5
	Collection Area (m^2)	290	340	300	330	320	340
	Cover Material	no cover	topsoil	coarse sand	no cover	no cover	sandy till over coarse sand
Year	Annual Precipitation (cm)	Yield Coefficients					
1978	69.6	0.41	0.30	0.43	0.40	0.53	0.51
1979	63.3	0.47	0.35	0.49	0.35	0.58	0.55
1980	58.8	0.32	0.15	0.33	0.27	0.44	0.18
1981	63.4	0.46	0.31	0.49	0.53	0.64	0.27
1982	86.5	0.47	0.34	0.48	0.33	0.64	0.36
1983	70.9	0.40	0.22	0.32	0.47	0.42	0.24
1984	53.5	0.43	0.22	0.49	0.52	0.58	0.27
1985	80.1	0.57	0.35	0.43	0.57	0.58	0.31
1986	69.0	0.60	0.30	0.43	0.55	0.52	0.23
1987	60.3	0.60	0.31	0.45	0.54	0.56	0.29
1988	81.1	0.45	0.29	0.39	0.45	0.45	0.26
1989	69.3	0.61	0.34	0.46	0.55	0.46	0.35
1990	64.5	0.32[3]	0.36			0.18[3]	0.35
1991	62.8	0.66	0.36			0.53	0.33
1992	60.0	0.60	0.25			0.63	0.33
1993	65.5	0.66	0.29			0.57	0.31
average		0.51	0.30	0.43	0.46	0.54	0.32

1 Pile FL3 was covered with a geomembrane at the end of the 1989 field season.

2 Pile FL4 was partially disassembled in September 1982 and then covered with a geomembrane at the end of the 1989 field season.

3 Piles FL1 and FL6 were covered with a plastic membrane at the end of the 1989 field season causing a decrease in flow for the 1990 season. These values were not included in the average for the experiment duration. The membrane was in effect for only the 1990 season.

Table 15. Flow weighted mean concentrations for the 0.6 %S pile, FL1.

Year	Median pH	Flow Weighted Mean						
		SO4	Ca	Mg	Cu	Ni	Co	Zn
	s.u.[1]	mmol/L						
1978	7.80	9.88	5.01	0.70	0.0004	0.0038	0.0003	0.0001
1979	7.35	7.74	5.58	0.82	0.0005	0.0056	0.0004	0.0004
1980	7.20	12.51	7.92	1.23	0.0007	0.0120	0.0008	0.0007
1981	7.20	8.03	5.95	1.11	0.0014	0.0329	0.0016	0.0015
1982	7.00	7.86	6.37	0.95	0.0018	0.0737	0.0032	0.0015
1983	6.86	10.34	7.73	1.54	0.0039	0.0689	0.0029	0.0020
1984	6.36	9.57	6.56	1.69	0.0175	0.2555	0.0114	0.0057
1985	6.70	9.15	5.94	1.73	0.0209	0.2447	0.0088	0.0057
1986	6.54	7.53	5.50	1.69	0.0447	0.3109	0.0118	0.0062
1987	6.03	10.26	6.92	2.19	0.0792	0.3840	0.0135	0.0099
1988	6.09	9.14	6.21	2.82	0.1051	0.6386	0.0211	0.0163
1989	5.69	10.04	5.66	2.52	0.1278	0.5970	0.0270	0.0184
1990	5.72	13.04	6.84	3.88	0.3056	0.9643	0.0469	0.0321
1991	5.25	13.04	6.38	4.56	0.3652	1.1568	0.0461	0.0256
1992	5.19	9.98	4.95	3.57	0.2980	0.8278	0.0349	0.0192
1993	5.14	10.62	4.96	3.63	0.3031	0.8902	0.0333	0.0167

[1] Standard units

Table 16. Flow weighted mean concentrations for the 0.6 %S pile, FL2.

Year	Median pH	Flow Weighted Mean						
		SO4	Ca	Mg	Cu	Ni	Co	Zn
	s.u.[1]	mmol/L						
1978	7.60	7.17	3.70	0.81	0.0019	0.0037	0.0004	0.0006
1979	7.30	5.83	3.99	0.93	0.0019	0.0046	0.0003	0.0007
1980	6.95	7.36	5.02	1.12	0.0028	0.0120	0.0009	0.0018
1981	6.80	6.79	3.88	1.21	0.0046	0.0234	0.0015	0.0022
1982	6.70	8.69	5.16	1.50	0.0062	0.0468	0.0027	0.0025
1983	6.39	8.94	5.53	1.84	0.0193	0.0767	0.0043	0.0049
1984	6.17	7.12	4.58	1.71	0.0187	0.0946	0.0049	0.0054
1985	6.30	10.02	5.24	2.24	0.0386	0.1545	0.0069	0.0071
1986	6.17	6.17	4.12	1.79	0.2277	0.1788	0.0079	0.0061
1987	5.54	7.04	4.63	2.15	0.0588	0.2016	0.0093	0.0068
1988	5.47	8.63	5.05	3.19	0.0906	0.4967	0.0193	0.0183
1989	5.26	7.78	4.07	2.48	0.1208	0.5667	0.0318	0.0254
1990	5.13	9.20	4.49	3.10	0.1505	0.5370	0.0300	0.0276
1991	5.18	8.98	4.16	3.17	0.1734	0.6326	0.0293	0.0204
1992	5.07	8.53	3.56	2.89	0.1555	0.5442	0.0261	0.0177
1993	5.04	9.75	3.99	3.88	0.2043	0.7902	0.0342	0.0206

[1] standard units

Table 17. Flow weighted mean concentrations for the 0.6 %S pile, FL3.

Year	Median pH	Flow Weighted Mean						
		SO4	Ca	Mg	Cu	Ni	Co	Zn
	s.u.[1]	mmol/L						
1978	7.60	9.96	7.11	2.14	0.0011	0.0076	0.0010	0.0010
1979	7.05	5.80	3.74	1.27	0.0015	0.0151	0.0009	0.0014
1980	6.65	10.77	5.25	3.16	0.0205	0.1346	0.0070	0.0076
1981	6.60	12.73	5.58	4.52	0.0880	0.4719	0.0233	0.0177
1982	6.58	12.95	5.32	4.90	0.1051	0.8594	0.0352	0.0193
1983	5.90	16.29	6.33	6.67	0.2167	1.1529	0.0519	0.0380
1984	5.05	14.34	5.25	6.31	0.3148	1.3302	0.0668	0.0392
1985	5.44	16.85	5.49	6.44	0.4492	1.6140	0.0704	0.0460
1986	5.27	12.60	4.63	5.51	0.5103	1.5432	0.0731	0.0388
1987	5.01	17.20	5.95	7.98	0.7214	2.0235	0.0841	0.0578
1988	5.15	12.75	4.23	5.62	0.5498	1.9666	0.0809	0.0714
1989	4.98	11.78	3.82	4.64	0.5452	1.6637	0.0979	0.0701

[1] standard units

Table 18. Flow weighted mean concentrations for the 0.6 %S pile, FL4.

Year	Median pH	Flow Weighted Mean						
		SO_4	Ca	Mg	Cu	Ni	Co	Zn
	s.u.[1]	mmol/L						
1978	7.55	6.34	4.06	0.70	0.0003	0.0033	0.0003	0.0011
1979	7.10	4.89	3.49	0.62	0.0041	0.0345	0.0010	0.0015
1980	6.80	8.35	4.14	0.70	0.0067	0.0483	0.0027	0.0034
1981	6.85	7.96	5.22	1.38	0.0103	0.0762	0.0041	0.0047
1982	6.55	7.73	5.67	1.56	0.0159	0.1318	0.0068	0.0054
1983		6.75	4.09	1.61	0.0423	0.2306	0.0130	0.0122
1984		8.09	3.87	2.24	0.0941	0.4337	0.0271	0.0182
1985		8.98	4.26	2.84	0.1354	0.5880	0.0298	0.0248
1986		8.88	3.93	3.29	0.2389	0.8741	0.0492	0.0294
1987		12.66	4.55	4.86	0.4193	1.4120	0.0685	0.0465
1988		11.07	3.39	4.93	0.4943	2.0142	0.0961	0.0871
1989		11.97	3.16	4.70	0.5441	1.7038	0.1385	0.0935

[1] standard units

Table 19. Flow weighted mean concentrations for the 0.8 %S pile, FL6.

Year	Median pH	Flow Weighted Mean						
		SO4	Ca	Mg	Cu	Ni	Co	Zn
	s.u.[1]	mmol/L						
1978	7.51	10.88	6.52	3.16	0.0016	0.0164	0.0027	0.0005
1979	6.40	12.01	7.28	4.52	0.0307	0.1484	0.0113	0.0060
1980	4.40	15.57	6.93	7.07	0.1928	0.5209	0.0360	0.0181
1981	4.40	18.23	6.20	8.45	0.5124	1.3955	0.0882	0.0424
1982	4.20	19.67	6.72	9.06	0.7012	2.0555	0.0981	0.0429
1983	4.19	25.04	7.68	11.83	1.0002	1.8457	0.1224	0.0564
1984	4.14	25.70	6.99	12.43	1.1793	2.1712	0.1278	0.0571
1985	4.27	26.53	6.74	13.74	1.2510	2.0460	0.1032	0.0540
1986	4.27	19.49	6.19	11.28	1.1766	1.7238	0.1003	0.0417
1987	4.00	27.22	7.44	14.18	1.4909	1.9802	0.0908	0.0506
1988	4.12	22.90	6.75	12.66	1.2931	1.9938	0.0978	0.0751
1989	4.20	25.66	6.18	11.58	1.2266	2.0219	0.1243	0.0808
1990	4.10	33.65	7.40	15.47	1.3348	1.6896	0.0958	0.0657
1991	4.14	28.01	6.21	15.19	1.5436	1.8448	0.0897	0.0590
1992	4.24	17.52	4.08	10.52	1.0924	1.1904	0.0629	0.0376
1993	4.24	18.84	4.36	10.55	1.1415	1.2964	0.0637	0.0366

[1] standard units

Table 20. Flow weighted mean concentrations for the 1.4 %S pile, FL5.

Year	Median pH	Flow Weighted Mean						
		SO4	Ca	Mg	Cu	Ni	Co	Zn
	s.u.[1]	mmol/L						
1978	5.50	8.97	5.61	1.12	0.0306	0.4055	0.0512	0.0113
1979	3.40	22.34	7.10	5.32	0.6040	2.4473	0.1872	0.0719
1980	3.40	30.17	5.69	11.26	0.7288	3.7781	0.2585	0.1210
1981	3.40	36.30	5.48	12.67	1.1689	4.3258	0.2713	0.1174
1982	3.40	39.59	5.62	12.82	1.2764	5.1831	0.3080	0.1205
1983	3.60	41.70	6.19	13.66	1.5563	4.5313	0.3153	0.1271
1984	3.49	35.63	5.52	11.81	1.2895	3.3643	0.2578	0.0969
1985	3.50	40.09	6.60	13.40	1.6158	3.5071	0.2390	0.1062
1986	3.63	30.94	6.18	11.24	1.3341	3.1648	0.2530	0.0885
1987	3.43	27.25	6.11	9.62	1.2648	2.3704	0.1524	0.0794
1988	3.62	24.50	5.97	8.03	1.1512	2.3129	0.1526	0.1090
1989	3.62	25.28	5.98	6.90	1.0401	1.8736	0.1948	0.1050
1990	3.62	23.56	5.82	6.42	1.0706	1.3991	0.1141	0.0881
1991	3.60	20.48	5.04	5.89	0.9416	1.1522	0.0714	0.0534
1992	3.73	17.10	4.28	5.01	0.8309	0.9275	0.0614	0.0397
1993	3.66	21.63	4.78	5.82	0.9764	1.2519	0.0755	0.0442

[1] standard units

Table 21. Summary of rates of release for the six test piles.

Pile	% S	Total Volume	Period	Period	Minimum pH	Overall Rates of Release						
						SO4	Ca	Mg	Cu	Ni	Co	Zn
		L	years	days		mmol/T day						
FL1	0.6	1,571,659	1978-93	3877	4.96	3.607	2.228	0.826	0.043	0.166	0.007	0.004
FL2	0.6	1,093,194	1978-93	3789	4.76	2.124	1.170	0.568	0.024	0.083	0.004	0.003
FL3	0.6	1,063,537	1978-89	2986	4.71	5.509	2.231	2.117	0.149	0.535	0.025	0.017
FL4	0.6	1,164,169	1978-89	2644	5.20	2.475	1.055	0.709	0.053	0.199	0.012	0.009
FL6	0.8	1,794,439	1978-93	3856	3.70	7.539	2.295	3.752	0.358	0.570	0.031	0.017
FL5	1.4	1,179,846	1978-93	3840	3.08	10.109	2.188	3.106	0.415	1.001	0.070	0.033

Table 22. Summary of total mass release for the six test piles.

Pile	% S	Total Volume	Period	Period	Minimum pH	Total Mass Release						
						SO_4	Ca	Mg	Cu	Ni	Co	Zn
		L	years	days		mol						
FL1	0.6	1,571,659	1978-93	3877	4.96	15384	9502	3521	183	708	28	17
FL2	0.6	1,093,194	1978-93	3789	4.76	8851	4875	2366	101	347	17	13
FL3	0.6	1,063,537	1978-89	2986	4.71	13654	5530	5246	370	1325	61	42
FL4	0.6	1,164,169	1978-89	2644	5.20	11124	4740	3189	239	894	52	39
FL6	0.8	1,794,439	1978-93	3856	3.70	37789	11507	18808	1792	2858	155	84
FL5	1.4	1,179,846	1978-93	3840	3.08	31638	6847	9721	1300	3131	220	104

Table 23. Sulfur, calcium, magnesium, copper, and nickel percent mass release from the six test piles for the entire period of record.

Pile	Sulfur	Calcium[1]	Magnesium[1]	Copper[2]	Nickel[2]
FL1	7.11	0.61	0.17	0.30	4.55
FL2	4.09	0.31	0.11	0.17	2.23
FL3	8.36	0.47	0.33	0.81	11.29
FL4	3.33	0.20	0.10	0.25	3.72
FL6	11.79	0.58	0.58	2.58	15.36
FL5	8.82	0.72	0.77	3.38	26.53

[1] Initial calcium and magnesium contents based on mass weighted averages for the −6" size fraction of the bulk samples.

[2] Initial copper and nickel contents based on blast rounds placed in the piles.

Table 24. Particle Size distribution (in weight percent passing) of the bulk samples. Analysis of the +½ inch by MN DNR in Hibbing, MN and the −½ inch (after drying) by Midland Research Center in Nashwauk, MN.

Particle Diameter				0.6%S[2]	0.6% S	0.8% S	1.4%S	Bin 6B
classification[1]	mesh	inches	mm					
boulders and cobbles		12	305	89.0	100	100	98.7[3]	100
		6	152	82.2	96.6	94.1	91.6[4]	100
gravel		3	76.2	67.7	80.2	80.6	73.2	83.8
		1.5	38.1	38.3	51.2	53.2	44.4	52.8
		0.75	19.1	33.8	32.3	34.0	26.9	32.8
		0.5	12.7	27.1	20.3	13.7	14.1	24.9
		0.25	6.4	19.2	18.7	11.6	11.0	22.1
sand	10	0.078	2.0	12.2	11.0	5.4	6.1	14.1
	14	0.056	1.41	10.8				
	20	0.033	0.84	10.6				
	35	0.020	0.50		6.0	3.2	3.6	8.2
	40	0.017	0.425	7.7				
	60	0.0098	0.250	6.1				
	80	0.0070	0.177		3.3	1.8	1.9	4.2
	100	0.0059	0.149	4.6	2.9	1.6	1.6	3.5
	140	0.0041	0.105	3.4	2.5	1.3	1.4	2.9
silt and clay	200	0.0030	0.074	3.1	1.9	1.0	1.0	2.1
	270	0.0021	0.053	1.4 to 1.7 [5]	1.4	0.7	0.7	1.5

1 Boulders, cobbles, gravel, sand and silt refer to US Department of Agriculture (USDA) classification definitions.
2 Sample collected in 1982 and analyzed by Minnesota Department of Natural Resources, Division of Minerals for particles >0.84 mm and by the Minnesota Department of Transportation for the particles <1.41 mm.
3 one cemented agglomerate
4 2.6% of sample retained on 152 mm screen associated with cemented agglomerates.
5 Estimated based on hydrometer analysis of the <0.074 mm fraction.

Table 25. Particle size distribution (in weight percent passing) of the spatially distributed and bulk samples from 0.8%S (FL6) test pile. Sizes are in mm unless otherwise indicated.

Sample	Percent Finer Than													
	12 in	6 in	3 in	1.5 in	0.75 in	0.5 in	6.4 mm	2.0 mm	0.5 mm	0.177 mm	0.149 mm	0.105 mm	0.075 mm	0.053 mm
6-1-02	100	100.0	90.8	50.2	19.7	12.7	6.8	4.5	3.25	2.29	1.85	1.37	0.94	0.32
6-1-04	100	75.8	70.1	41.1	23.8	18.1	11.9	8.0	5.06	3.18	2.49	1.81	0.98	0.35
6-1-05	100	100.0	86.3	66.2	47.8	37.6	24.7	14.5	9.06	5.83	4.78	3.43	2.02	0.65
6-1-07	100	100.0	100.0	92.6	70.9	59.7	30.7	16.0	8.43	5.10	4.18	2.89	1.86	1.10
6-2-02	100	100.0	90.3	62.7	41.8	33.8	23.3	14.7	8.19	3.99	3.39	2.07	0.85	0.17
6-2-04	100	100.0	92.6	74.6	57.6	48.5	35.7	23.3	14.57	7.02	6.03	3.85	1.48	0.31
6-2-07	100	100.0	100.0	76.2	57.1	46.6	32.4	21.0	12.70	6.93	4.91	3.32	1.26	0.21
6-2-10	100	100.0	95.5	79.7	59.5	49.2	32.6	19.3	10.95	5.33	3.86	2.24	0.77	0.26
6-3-01	100	100.0	100.0	78.0	55.6	46.2	32.7	19.7	11.51	5.27	4.02	2.39	0.90	0.19
6-3-06	100	100.0	70.8	59.8	42.4	34.2	23.6	14.5	8.07	4.19	3.31	2.08	0.79	0.18
6-3-08	100	53.6	51.0	27.1	14.2	10.2	6.4	4.4	2.90	1.85	1.49	1.09	0.48	0.14
6-3-16	100	100.0	91.6	73.3	49.2	39.7	27.2	17.3	10.72	5.60	4.31	3.05	0.93	0.22
6-4-02	100	100.0	93.4	74.2	57.7	48.5	34.5	21.1	13.28	7.40	5.22	2.68	0.92	0.34
6-4-04	100	82.9	82.9	66.7	46.6	37.5	26.5	16.3	9.07	5.20	3.67	2.46	0.92	0.30
6-4-05	100	100.0	100.0	72.0	53.4	44.0	29.3	17.3	9.14	4.71	3.61	2.45	1.09	0.31
6-4-07	100	100.0	100.0	80.0	61.4	51.2	36.4	22.5	13.61	6.66	4.67	2.28	0.64	0.26
Avg Sp. Dist.	100	94.5	88.4	67.1	47.4	38.6	25.9	15.9	9.40	5.03	3.86	2.46	1.05	0.33
0.8%S Bulk	100	94.1	80.5	53.2	33.9	13.5	11.4	5.3	3.14	1.72	1.53	1.34	0.98	0.70

Table 26. Specific surface area of particle size fractions of the bulk samples.

Particle Size (mm)	Specific Surface Area (30% N_2) m^2g^{-1}				
	FL4[1] 0.6% S	FL1[2] 0.6% S	FL6[2] 0.8% S	FL5[2] 1.4% S	Bin 6B[2] 0.6% S
−2.0/+0.50	0.60	0.73	2.52	2.28	2.54
−0.50/+0.177	0.78	1.53	3.23	3.07	3.44
−0.177/+0.149	1.1	1.99	3.90	4.11	5.08[2]
−0.149/+0.105	1.6	2.19	3.08	4.76	4.40[3]
−0.105/+0.075	1.7	2.35	4.47	7.92	6.32
−0.075/+0.053	2.6	4.34	5.59	8.04	10.8
−0.053	4.7	10.67	11.2	15.9	15.7

[1] Collected in 1982
[2] Collected in 1994

Table 27. Sulfur, sulfate, and trace metal geochemistry and mass weighted averages for particle size fractions of the 0.6% S bulk sample. Sulfur and sulfate analysis by Lerch Brothers. Trace metal analysis by Chemex Labs.

Size Fraction (mm)[1]	S_T pct	SO_4 pct[2]	S^{2-} pct[3]	Cu ppm	Ni ppm	Co ppm	Zn ppm
−6"/+1½"	0.45	0.02	0.43	2390	507	51	70
−1½"/+½"	0.58	0.02	0.56	3090	611	56	74
−½"/+2.0	0.44	0.01	0.43	2900	545	53	72
−2.0/+0.50	0.45	0.04	0.41	2920	488	47	72
−0.50/+0.177	0.58	0.10	0.48	3110	538	49	74
−0.177/+0.149	0.70	0.17	0.53	3770	678	57	86
−0.149/+0.105	0.64	0.14	0.50	3750	743	59	84
−0.105/+0.075	0.87	0.15	0.72	4320	989	68	92
−0.075/+0.053	0.99	0.09	0.90	5190	1380	83	104
−0.053	1.30	0.17	1.13	7140	1660	94	164
mass wtd avg of −6" fraction	0.51	0.03	0.48	2817	570	54	74

1 Unless otherwise indicated.
2 Sulfate as sulfur
3 Calculated by difference

Table 28. Sulfur, sulfate, and trace metal geochemistry and mass weighted averages for particle size fractions of the 0.8% S bulk sample. Sulfur and sulfate analysis by Lerch Brothers. Trace metal analysis by Chemex Labs.

Size Fraction (mm)[1]	S_T pct	SO_4 pct[2]	S^{2-} pct[3]	Cu ppm	Ni ppm	Co ppm	Zn ppm
−6"/+1½"	0.60	0.04	0.56	3370	570	63	74
−1½"/+½"	0.74	0.09	0.65	3860	695	69	76
−½"/+2.0	1.13	0.06	1.07	5060	1085	82	78
−2.0/+0.50	1.22	0.08	1.14	6320	1005	76	96
−0.50/+0.177	1.34	0.28	1.06	6620	942	74	80
−0.177/+0.149	1.20	0.18	1.02	6540	954	79	82
−0.149/+0.105	0.99	0.15	0.84	6150	963	81	84
−0.105/+0.075	1.11	0.14	0.97	6740	1100	87	86
−0.075/+0.053	1.26	0.12	1.14	8100	1405	101	94
−0.053	1.51	0.25	1.26	9260	1595	102	132
mass wtd avg of −6" fraction	0.74	0.07	0.67	3924	697	68	76

1 Unless otherwise indicated.
2 Sulfate as sulfur
3 Calculated by difference

Table 29. Sulfur, sulfate, and trace metal geochemistry and mass weighted averages for particle size fractions of the 1.4% S bulk sample. Sulfur and sulfate analysis by Lerch Brothers. Trace metal analysis by Chemex Labs.

Size Fraction (mm)[1]	S_T pct	SO_4 pct[2]	S^{2-} pct[3]	Cu ppm	Ni ppm	Co ppm	Zn ppm
−6"/+1½"	0.74	0.01	0.73	1455	315	35	58
−1½"/+½"	1.03	0.17	0.86	2200	422	45	56
−½"/+2.0	0.80	0.24	0.56	2470	288	32	48
−2.0/+0.50	2.10	0.40	1.70	3350	416	42	48
−0.50/+0.177	2.06	0.80	1.26	3260	385	36	50
−0.177/+0.149	1.82	0.98	0.84	2910	411	38	56
−0.149/+0.105	1.46	0.73	0.73	2810	463	41	54
−0.105/+0.075	1.59	0.80	0.79	2900	549	47	56
−0.075/+0.053	1.65	0.95	0.70	2930	554	48	56
−0.053	1.88	1.07	0.81	3870	506	44	66
mass wtd avg of −6" fraction	0.92	0.12	0.80	1913	356	38	56

1 Unless otherwise indicated.
2 Sulfate as sulfur
3 Calculated by difference

Table 30. Sulfur, sulfate, and trace metal geochemistry and mass weighted averages for particle size fractions of the 6B bulk sample. Sulfur and sulfate analysis by Lerch Brothers. Trace metal analysis by Chemex Labs.

Size Fraction (mm)[1]	S_t pct	SO_4 pct[2]	S^{2-} pct[3]	Cu (ppm)	Ni (ppm)	Co (ppm)	Zn (ppm)
−6"/+1½"	0.95	0.04	0.91	4050	767	86	96
−1½"/+½"	0.62	0.06	0.56	2940	538	62	80
−½"/+2.0	0.62	0.06	0.56	3030	415	54	78
−2.0/+0.50	0.56	0.07	0.49	3280	316	43	72
−0.50/+0.177	0.43	0.07	0.36	3180	294	45	76
−0.177/+0.149	0.43	0.09	0.34	3350	319	46	82
−0.149/+0.105	0.45	0.09	0.36	3650	337	47	86
−0.105/+0.075	0.54	0.12	0.42	3930	351	47	86
−0.075/+0.053	0.58	0.13	0.45	4550	393	48	118
−0.053	0.68	0.23	0.45	5310	527	51	160
mass wtd avg of −6" fraction	0.76	0.06	0.70	3564	605	70	88

1 Unless otherwise indicated.
2 Sulfate as sulfur
3 Calculated by difference

Table 31. Whole rock analysis and mass weighted totals of the 0.6% S bulk sample (in percent). Analysis by Chemex Labs.

Size Fraction (mm)[1]	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO_2	LOI[2]	mass %
−6"/+1½"	17.69	8.24	0.03	14.73	0.43	7.55	0.15	2.38	0.20	44.57	2.42	<0.01	45.3
−1½"/+½"	17.26	8.16	0.03	15.03	0.49	7.84	0.16	2.36	0.19	45.11	2.48	0.43	30.8
−½"/+2.0	17.59	8.46	0.03	14.35	0.48	7.73	0.15	2.37	0.20	45.44	2.14	0.48	9.2
−2.0/+0.50	18.30	8.48	0.03	13.38	0.52	7.22	0.14	2.61	0.16	46.19	1.93	0.68	5.0
−0.50/+0.177	17.46	8.03	0.04	14.80	0.48	7.48	0.15	2.46	0.16	44.86	2.80	0.96	2.7
−0.177/+0.149	16.37	7.67	0.04	15.90	0.44	7.82	0.16	2.28	0.22	43.27	3.19	1.14	0.4
−0.149/+0.105	16.44	7.70	0.05	15.79	0.45	7.83	0.16	2.27	0.25	43.54	3.12	1.21	0.4
−0.105/+0.075	16.35	7.75	0.06	15.83	0.47	7.79	0.16	2.25	0.33	43.21	2.99	1.43	0.6
−0.075/+0.053	16.02	7.63	0.06	16.19	0.49	7.66	0.15	2.16	0.36	42.25	2.89	1.82	0.5
−0.053	15.85	7.12	0.06	17.08	0.52	7.24	0.14	2.06	0.39	40.68	2.56	3.49	1.4
mass weighted avg of the −6" fraction[3]	17.51	8.21	0.03	14.78	0.46	7.64	0.15	2.38	0.20	44.83	2.41	0.33	96.3

[1] Unless otherwise indicated.
[2] LOI = Loss on ignition
[3] Assays not available for 3.4 wt. % +6" material.

Table 32. Whole rock analysis and mass weighted totals of the 0.8% S bulk sample (in percent). Analysis by Chemex Labs.

Size Fraction (mm)[1]	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO	LOI[2]	mass %
−6"/+1½"	14.58	8.86	0.04	15.38	0.52	9.74	0.16	2.00	0.25	45.05	1.65	0.26	40.8
−1½"/+½"	14.53	8.56	0.04	15.53	0.46	10.24	0.17	1.93	0.22	44.59	1.37	0.77	39.5
−½"/+2.0	14.63	8.41	0.04	15.82	0.47	10.13	0.17	1.92	0.24	44.23	1.18	1.19	8.2
−2.0/+0.50	14.30	7.68	0.04	15.64	0.59	9.54	0.16	2.05	0.21	45.16	1.21	2.07	2.2
−0.50/+0.177	13.73	7.61	0.04	16.31	0.53	9.69	0.16	1.92	0.19	44.07	1.37	2.45	1.4
−0.177/+0.149	13.74	7.31	0.04	17.32	0.52	10.31	0.17	1.89	0.23	42.77	1.51	2.02	0.2
−0.149/+0.105	14.24	7.30	0.04	16.61	0.54	10.12	0.16	1.99	0.25	43.37	1.38	1.80	0.2
−0.105/+0.075	14.31	7.21	0.03	17.02	0.57	10.04	0.16	1.95	0.26	43.30	1.38	2.20	0.3
−0.075/+0.053	13.44	6.89	0.03	18.62	0.53	10.37	0.17	1.82	0.32	41.96	1.86	2.38	0.3
−0.053	13.26	6.73	0.03	18.80	0.50	9.73	0.15	1.66	0.30	40.55	1.50	4.79	0.7
mass weighted avg of the −6" fraction[3]	14.53	8.62	0.04	15.56	0.49	9.98	0.17	1.96	0.24	44.71	1.47	0.89	93.8

[1] Unless otherwise indicated.
[2] LOI = Loss on ignition
[3] Assays not available for 4.6 wt. % +6" material.

Table 33. Whole rock analysis and mass weighted totals of the 1.4% S bulk sample (in percent). Analysis by Chemex Labs.

Size Fraction (mm)[1]	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO_2	LOI[2]	Total
−6"/+1½"	16.41	6.45	0.06	16.20	0.97	6.38	0.16	2.11	0.42	46.75	2.60	0.34	47.2
−1½"/+½"	16.27	6.82	0.07	16.30	0.92	6.31	0.16	1.98	0.36	46.19	2.37	1.14	30.3
−½"/+2.0	16.07	6.29	0.07	16.75	1.03	5.95	0.15	1.99	0.38	45.57	2.20	2.54	8.0
−2.0/+0.50	15.18	5.46	0.06	17.95	1.01	5.23	0.13	2.19	0.34	45.10	1.97	4.44	2.5
−0.50/+0.177	14.95	6.06	0.05	17.24	1.03	5.23	0.13	2.19	0.29	44.62	1.67	5.69	1.7
−0.177/+0.149	15.17	6.42	0.06	17.15	1.00	5.49	0.13	2.14	0.29	43.65	1.78	5.61	0.3
−0.149/+0.105	15.24	6.18	0.06	17.26	1.00	5.69	0.13	2.16	0.30	44.24	2.04	4.47	0.2
−0.105/+0.075	14.46	6.08	0.06	18.16	1.00	5.98	0.14	2.01	0.32	42.87	2.42	5.01	0.4
−0.075/+0.053	14.06	6.05	0.06	18.72	1.00	5.96	0.14	1.83	0.34	41.69	2.55	5.95	0.3
−0.053	13.06	5.39	0.06	21.76	0.95	5.27	0.12	1.60	0.51	38.16	2.40	8.16	0.7
mass weighted avg of the −6" fraction[3]	16.22	6.51	2.05	16.41	0.96	6.25	0.16	2.05	0.39	46.26	2.45	1.04	91.6

[1] Unless otherwise indicated.
[2] LOI = Loss on ignition
[3] Assays not available for 4.6 wt. % +6" material.

Table 34. Whole rock analysis and mass weighted totals of the 0.6% S 6B bulk sample (in percent). Analysis by Chemex Labs.

Size Fraction (mm)[1]	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO_2	LOI[2]	Total
−6"/+1½"	15.23	7.10	0.04	18.05	0.58	9.43	0.18	2.26	0.24	43.73	1.60	0.18	47.2
−1½"/+½"	16.89	7.99	0.04	15.20	0.54	8.12	0.16	2.50	0.22	45.00	1.65	0.28	27.9
−½"/+2.0	17.25	8.17	0.03	15.02	0.56	7.93	0.15	2.60	0.21	45.60	1.41	0.65	10.8
−2.0/+0.50	17.73	8.08	0.03	14.00	0.60	7.29	0.14	2.73	0.18	46.54	1.38	0.99	5.9
−0.50/+0.177	16.61	7.30	0.04	15.65	0.58	7.73	0.15	2.63	0.19	45.38	2.26	0.98	4.0
−0.177/+0.149	16.13	7.12	0.04	16.58	0.56	8.08	0.16	2.45	0.20	44.23	2.31	1.42	0.7
−0.149/+0.105	16.31	7.24	0.04	16.30	0.56	8.00	0.15	2.48	0.33	44.42	2.13	1.61	0.6
−0.105/+0.075	15.96	7.10	0.04	16.79	0.58	8.10	0.15	2.33	0.31	42.73	2.21	2.02	0.8
−0.075/+0.053	15.73	7.12	0.04	17.03	0.55	8.07	0.15	2.31	0.41	43.33	2.09	2.47	0.6
−0.053	15.29	6.48	0.03	18.46	0.55	7.70	0.14	2.08	0.41	40.87	1.91	4.99	1.5
mass weighted avg of the −6" fraction	16.14	7.52	0.04	16.56	0.57	8.65	0.17	2.41	0.23	44.47	1.63	0.46	100

[1] Unless otherwise indicated.
[2] LOI = Loss on ignition

Table 35. Summary of the mass weighted sulfur, sulfate, whole rock, and trace metal chemistry for sixteen spatially distributed samples and the bulk sample from the 0.8% test pile. Calculations are based on analyses of the −6.4 mm particle sizes of the spatially distributed samples and the −0.5 in. particle sizes for the bulk sample.

Sample	S	SO_4	C	CO_2	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	Na_2O	SiO_2	Cu	Ni	Co	Zn
06-01-02	1.15	0.093	0.39	0.799	12.7	5.84	18.1	1.01	6.025	1.58	35.8	0.67	0.090	0.009	0.012
06-01-04	1.53	0.120	0.13	0.017	10.3	4.84	20.1	0.32	12.08	1.50	34.7	0.74	0.299	0.013	0.014
06-01-05	0.79	0.084	0.25	0.025	13.2	6.16	18.2	0.97	5.689	1.64	38.0	0.48	0.044	0.006	0.019
06-01-07	1.87	0.232	0.18	0.005	14.3	5.60	16.8	1.43	5.99	1.56	35.6	0.75	0.051	0.005	0.038
06-02-02	1.12	0.091	0.13	0.008	11.6	5.55	16.8	0.64	9.78	2.10	35.7	0.64	0.127	0.011	0.013
06-02-04	1.01	0.018	0.07	0.005	13.2	8.04	16.3	1.20	10.09	1.92	34.9	0.63	0.143	0.011	0.012
06-02-07	1.08	0.089	0.13	0.058	11.8	6.07	16.9	0.68	9.49	1.38	37.3	0.62	0.121	0.010	0.015
06-02-10	1.27	0.193	0.20	0.005	12.9	5.92	17.0	0.87	8.04	1.12	34.5	0.61	0.100	0.010	0.015
06-03-01	1.49	0.116	0.17	0.161	10.5	4.85	16.8	0.72	9.89	1.84	35.2	0.70	0.175	0.009	0.014
06-03-06	0.69	0.106	0.33	0.009	11.4	5.30	14.7	1.85	8.91	2.07	37.0	0.57	0.112	0.009	0.011
06-03-08	0.92	0.084	0.14	0.095	12.1	8.08	14.2	0.32	9.24	1.10	32.9	0.76	0.114	0.010	0.011
06-03-16	1.23	0.159	0.15	0.015	12.1	7.64	14.9	0.51	8.96	1.74	33.6	0.72	0.112	0.009	0.012
06-04-02	1.06	0.331	0.41	0.061	14.7	6.12	15.8	0.97	6.02	1.92	35.3	0.48	0.048	0.006	0.031
06-04-04	0.84	0.185	0.27	0.189	14.8	5.76	13.1	1.07	6.00	2.10	39.7	0.43	0.089	0.006	0.015
06-04-05	1.35	0.257	0.17	0.108	13.3	6.07	16.3	0.58	8.40	1.86	31.1	0.76	0.136	0.011	0.014
06-04-07	1.34	0.263	0.37	0.056	15.1	5.16	14.3	0.99	6.10	1.07	37.4	0.73	0.054	0.010	0.020
Average of spatially distributed samples	1.17	0.151	0.22	0.101	12.8	6.06	16.2	0.88	8.17	1.66	35.5	0.64	0.115	0.009	0.017
Bulk 0.8%S	1.19	0.103	nd[1]	nd	14.4	8.04	16.2	0.50	9.998	1.93	44.2	0.58	0.107	0.008	0.008
Difference between spatial & bulk	-0.02	0.05			-1.6	-1.98	0.1	0.38	-1.83	-0.27	-8.6	0.06	0.01	0.00	0.01

[1] nd = not determined

Table 36. Comparison of USBM (1996) and ACME (2003) laboratories of whole rock analyses of randomly selected spatially distributed samples. Values are in percent, particle size in millimeters unless otherwise indicated.

ID	SiO_2		Al_2O_3		Fe_2O_3		MgO		CaO		Na_2O		K_2O		MnO	
	ACME	BLM	ACME	BLM	ACME	BLM	ACME	BLM	ACME	BLM	ACME	BLM	ACME	BLM	ACME	BLM
6-1-7 -1/4"/+2.0	45.6	36.9	17.0	15.5	16.4	15.4	6.69	6.4	5.56	6.14	1.49	1.40	1.34	1.44	0.11	0.13
6-1-7 -0.149/ +0.105	43.0	25.7	15.1	13.7	19.7	18.7	6.19	6.02	4.88	5.51	1.49	0.81	1.27	1.35	0.10	0.13
6-2-2 -2.0/+0.5	44.9	36.2	13.4	12.0	17.1	15.7	9.79	9.07	5.26	5.54	2.18	2.16	0.79	0.76	0.15	0.16
6-2-2 -0.177/ +0.149	41.4	33.9	11.5	10.4	20.7	19.3	11.8	10.9	4.74	5.08	1.58	2.45	0.59	0.51	0.18	0.20
6-4-4 -1/4"/+2.0	47.9	40.1	17.3	15.5	13.9	12.8	6.98	6.48	5.83	6.22	2.33	2.01	0.95	0.96	0.13	0.12
6-4-4 -0.177/ +0.149	46.1	39.3	15.6	14.5	15.8	15.2	6.68	6.55	5.33	5.95	2.20	1.77	0.88	0.94	0.14	0.13
6-4-5 -2.0/+0.5	43.3	28.9	15.1	13.7	16.9	15.9	8.67	8.22	6.20	6.07	1.97	1.91	0.64	0.61	0.14	0.14
6-4-5 -0.5/ +0.177	41.4	32.6	14.2	12.2	17.7	16.2	8.66	7.99	6.41	6.12	1.81	1.84	0.59	0.55	0.14	0.14
6-4-5 -0.5/ +0.177 duplicate	40.9		14.0		17.4		8.49		6.34		1.82		0.57		0.14	
avg Δ[1]	10.0		1.46		1.13		0.48		-0.30		0.09		-0.01		-0.01	

[1] Average of the differences between ACME and USBM analyses (ACME - USBM).

Table 37. Frequency of sulfide mineral photomicrographs for 30-70 mesh fraction.

Sample	Paste pH	Po	Pn	Cb	Cp	Py	Cu^0	Reaction Rating[2]	Weathering Index[3]
6-1-2	3.90	4	1	3	0	0	0	4	0.96
6-1-4	4.37	0	2	4	0	2	0	5	0.68
6-1-5	4.55	0	0	5	0	0	0	5	1.02
6-1-7	3.28	4	1	6	0	0	0	4	0.75
							AVE	4.5	0.85
6-2-2	4.37	4	3	3	0	0	0	3	0.92
6-2-4[1]	5.01	1	1	1	2	0	0	4	1.02
6-2-7	4.10	0	0	5	0	0	0	5	1.00
6-2-10	3.80	2	1	3	1	0	0	4	0.84
							AVE	4.0	0.94
6-3-1[1]	4.51	2	1	1	0	0	0	3	0.85
6-3-6	4.97	2	3	3	1	0	0	4	1.00[4]
6-3-8	4.88	2	1	2	1	0	0	4	1.11
6-3-16	5.44	2	0	3	0	0	0	4	0.92
							AVE	3.8	0.97
6-4-2[1]	3.49	3	0	0	1	0	0	1	0.76[5]
6-4-4	3.68	3	0	2	2	0	0	4	0.89
6-4-5	4.44	2	0	2	0	0	1	1	0.97
6-4-7	3.84	1	2	3	1	0	0	3	0.90
							AVE	2.5	0.88
Total		32	16	46	9	2	1		

1 All size fractions were examined. See Table A5.10.4.
2 Reaction rating based on microprobe analysis.
3 Mixed sulfide weathering index calculated using S, Cu, Ni content of sample from the 30-70 mesh size fraction, unless otherwise noted. MSWI = $[(1.26 \times Cu) + (0.97 \times Ni)]/ S^{2-}$. Cu with Cp and Cb = 1.26 × [Cu] and Ni with Pn = 0.97 × [Ni].
4 0.5–2.0 mm fraction used due to negative sulfide content of sample.
5 Reported as 1.24. Values reported for surrounding size fractions were 0.75 and 0.775.

Table 38. Summary of visual descriptions and acid concentration present field notes for the 0.6 %S (FL1) pile.

Sample	Color	Coat	Color[1]	Agglom[2]	Oxid[2]	Moist[3]	S^{0} [4]	Cu^{0} [4]	Gyp[5]	Sulfide[6] Notes	Other
FL1-1-01											
FL1-1-02	brown		gr bl	light			no				
FL1-1-03											
FL1-1-04	brown		gr	none			no				
FL1-1-05			gr								
FL1-1-06	brown		gr	none			no				
FL1-1-07			bl								
FL1-1-08	brown		none	none			no				
FL1-1-09											
FL1-1-10	brown	rust	ru	none			no				
FL1-1-11											
FL1-1-12	brown		ru	none			no				
FL1-1-13											
FL1-1-14	brown		ru	none			no				
FL1-1-15	gray		ru or								
FL1-2-01	brown	gray	pr or gr	none							
FL1-2-02											
FL1-2-03	brown		pr or gr	none							
FL1-2-04	brown		pr gr	none							
FL1-2-05	rust		gr bl	none							
FL1-2-06	brown	gray	ru pr or	none	mod					po	
FL1-2-07	gray		pr gr bl	none	light						
FL1-2-08	gray		ru pr or	none	light						
FL1-2-09	brown	rust	pr or	none							black rock
FL1-2-10	brown				light						
FL1-2-11	gray		ru pr or	none							
FL1-2-12	gray		or rd ye				yellow				
FL1-2-13	brown			none							
FL1-2-14	brown	gray	ru pr or	none	mod						
FL1-2-15	gray		ru or	none							
FL1-2-16	brown		ru pr or	none							
FL1-3-01	gray		unknown								
FL1-3-02	brown		pr gr bl	none	mod						
FL1-3-03	brown		unknown								
FL1-3-04	gray		ru pr or	none	light						
FL1-3-05	gray		pr or								
FL1-3-06	brown		gr bl		light						
FL1-3-07	gray		pr or	none			yellow				
FL1-3-08	gray		or rd		light						
FL1-3-09	gray		or								
FL1-3-10	gray		pr or	none	light						
FL1-3-11	gray				light						
FL1-3-12	gray		pr or gr	none	light						
FL1-3-13	gray		pr or	none	light						
FL1-3-14	gray				light						
FL1-3-15	gray		or	none	light						
FL1-3-16	gray		or								
FL1-4-01	gray		pr or								
FL1-4-02	gray		pr or	none			no	no	white	yes	
FL1-4-03	gray		pr br	none			yes	yes			
FL1-4-04	brown	gray	pr or gr	none			no	no			
FL1-4-05	gray		pr or gr	none		wet	yes	no	white	py	
FL1-4-06	gray		pr	none	mod						
FL1-4-07	gray		pr or	none	light		no	no	white		
FL1-4-08	gray		pr		light						
FL1-4-09	brown		ru	none							
FL1-4-10	gray		ru pr or	none		wet	yes	no			
FL1-4-11			pr br	none							
FL1-4-12	brown		pr or	none	mod		yes	no			

[1] color of surface coating noted: gr = green, bl = blue, ru = rust, or = orange, pr = purple, rd = red, ye = yellow; blank = no mention
[2] extent; none, light, mod = moderate, or heavy; ; blank = no mention
[3] amount of moisture in the sample in the hole; dry, moist, wet, or rain = raining during sampling
[4] native element noted (yellow assumed to be native sulfur)
[5] gyp = gypsum; white precipitate assumed to be gypsum
[6] specific sulfides noted; po = pyrrhotite, py = pyrite, ch = chalcopyrite, yes = presence noted but not while specific sulfide

Table 39. Summary of visual descriptions and acid concentration present field notes for the 0.8 %S (FL6) pile.

Sample	Color	Coat	Color[1]	Agglom[2]	Oxid[2]	Moist[3]	S^0 [4]	Cu^0 [4]	Gyp[5]	Sulfide[6]	Other Notes
FL6-1-01			gr	mod		moist					
FL6-1-02			gr			wet					
FL6-1-03				light		wet					
FL6-1-04	brown			light		moist					
FL6-1-05	brown		gr bl	light		moist					fibr organic
FL6-1-06	brown		gr	light		moist					
FL6-1-07	red		ru gr	mod		moist				yes	hot spot
FL6-1-08	brown		bl	none		moist					
FL6-1-09	brown		gr	light		moist					
FL6-1-10	brown		bl	light		moist					
FL6-1-11	red		gr bl	mod		moist				yes	
FL6-1-12	brown			light		moist					
FL6-2-01			pr bl	none	light	rain					
FL6-2-02	gray		pr or gr	none	light	rain				po	
FL6-2-03	gray		pr	none	light	rain					
FL6-2-04	gray		ru or	none	light	rain					paper plastc
FL6-2-05	gray orange		or gr	none	light	rain					
FL6-2-06			gr bl		mod	rain					
FL6-2-07	gray		or gr bl	light	mod	rain					
FL6-2-08	gray		ru		light	rain					
FL6-2-09			gr bl		mod	rain					
FL6-2-10	brown orange		or gr bl	none	mod	rain	yellow				
FL6-2-11	gray		pr		light	rain					
FL6-2-12	gray orange		pr or gr	none	light	rain				po	qtz
FL6-2-13	orange		gr bl	light	mod	rain					
FL6-2-14			gr bl	light	heavy	rain					
FL6-2-15	gray orange		ru or gr	light	mod	rain					
FL6-2-16						rain					
FL6-3-01	brown gray		pr bl	mod	light	rain				po	
FL6-3-02			gr bl		mod	rain					
FL6-3-03	gray		pr or gr	none	light	rain					
FL6-3-04			pr gr	none	mod	rain					
FL6-3-05			gr bl	none	light	rain					
FL6-3-06	brown		pr bl	light	mod	rain				po	clay balls
FL6-3-07	brown		gr bl	light	light	rain					
FL6-3-08	brown gray		pr gr bl	none	mod	rain		yes			clay balls
FL6-3-09	brown		pr gr bl	mod	heavy	rain	yes			ch	clay balls
FL6-3-10	gray		bl		light	rain					
FL6-3-11	brown orange		pr bl	none	light	rain	yes				clay balls
FL6-3-12			pr gr bl	none	mod	rain	yes	yes			
FL6-3-13			pr bl	mod	mod	rain					
FL6-3-14	brown		pr bl	mod	mod	rain					clay balls
FL6-3-15			pr	none	light	rain					
FL6-3-16	gray		pr or bl	none	light	rain					clay balls
FL6-4-01	brown orange			heavy							
FL6-4-02	brown orange		pr	heavy	mod		yes	yes			
FL6-4-03	brown		ru pr bl	heavy	mod		yellow	yes			
FL6-4-04	brown		ru pr gr	heavy	mod		yellow	yes			
FL6-4-05	brown rust		ru pr or	mod	mod		no	no		yes	
FL6-4-06	brown			heavy	mod						
FL6-4-07	brown		pr gr bl	mod	mod		no	no			
FL6-4-08			pr	mod	light		no	yes			
FL6-4-09	brown		pr gr bl	mod	light						
FL6-4-10	brown orange		pr	heavy	mod		no	no		yes	
FL6-4-11				heavy	mod		no				
FL6-4-12	brown orange		pr gr	light	mod					yes	

[1] color of surface coating noted: gr = green, bl = blue, ru = rust, or = orange, pr = purple, rd = red, ye = yellow; blank = no mention
[2] extent; none, light, mod = moderate, or heavy; ; blank = no mention
[3] amount of moisture in the sample in the hole; dry, moist, wet, or rain = raining during sampling
[4] native element noted (yellow assumed to be native sulfur)
[5] gyp = gypsum; white precipitate assumed to be gypsum
[6] specific sulfides noted; po = pyrrhotite, py = pyrite, ch = chalcopyrite, yes = presence noted but not while specific sulfide

Table 40. Summary of visual descriptions and acid concentration present field notes for the 1.4 %S (FL5) pile.

Sample	Color	Coat	Color[1]	Agglom[2]	Oxid[2]	Moist[3]	S^0 [4]	Cu^0 [4]	Gyp[5]	Sulfide[6]	Other Notes
FL5-1-01	brown		gr	light	none						
FL5-1-02	brown		gr bl	mod	heavy	wet	yes		white		pipe in hole
FL5-1-03	red		bl	heavy		moist	yes				
FL5-1-04				heavy							
FL5-1-05	brown			heavy	heavy	wet	yes	yes		po	pod pyrrho
FL5-1-06	brown	orange	gr bl	heavy		dry	yes			po	
FL5-1-07				heavy			yes				
FL5-1-08	brown	yellow	pr gr bl	mod		wet	yes			po	blgr 2nd min
FL5-1-09	orange			heavy			yes			po	
FL5-1-10				mod	mod		no			po	
FL5-1-11				heavy	heavy		yes	yes			
FL5-1-12			unknown	heavy	heavy		yes			po	
FL1-1-13			pr bl	none							
FL5-1-14			gr bl	none	heavy		no				blgr 2nd min
FL5-1-15			bl	mod	heavy	wet	yes			po	
FL5-2-01			pr	none		moist					
FL5-2-02	brown	red	pr	none		moist	yes			po	
FL5-2-03				heavy	heavy						
FL5-2-04	gray	red	ru pr or	heavy		moist	yes				
FL5-2-05	brown	gray	pr or	none		moist	yes				
FL5-2-06				none							
FL5-2-07			pr gr bl	light		wet	yes	yes			botryoidal
FL5-2-08				heavy	heavy		yes				
FL5-2-09				none			yes				
FL5-2-10	brown		pr or	none		wet	yes			po	
FL5-2-11			pr	light			yes				
FL5-2-12	orange	yello	pr or gr	mod		wet	yes				
FL5-2-13			pr bl	mod			yes			po	amber spher
FL5-2-14			gr bl		heavy		yes				
FL5-2-15			pr gr bl	mod	heavy	wet	yes	yes	yes	po	
FL5-2-16					heavy		yes				
FL5-3-01	brown		pr gr bl	none		wet	yes				
FL5-3-02			pr								
FL5-3-03			pr or	light		wet	yes				
FL5-3-04			pr				yes				
FL5-3-05				light			yes			po	
FL5-3-06			pr or bl	mod			yes			po	botryoidal
FL5-3-07			pr	mod		moist	yes				
FL5-3-08			pr or gr	mod		wet	yes			po	gr 2 min
FL5-3-09	brown		pr or	none		moist	yes				
FL5-3-10			pr			moist	yes				
FL5-3-11			pr bl	mod		wet	yes	yes		po	
FL5-3-12			pr				yes				
FL5-2-13			bl	light		wet	yes	yes			
FL5-3-14	brown		pr or	none		wet	yes				
FL5-3-15			bl	light			yes			po	
FL5-3-16			pr	none		moist	yes				
FL5-4-01	brown	red	pr	heavy	heavy		yes			yes	
FL5-4-02	brown			heavy	heavy		yes				
FL5-4-03	gray			heavy	heavy		yes				
FL5-4-04	gray	rust	or	heavy			yes	yes		yes	pyx rills
FL5-4-05	red	orange	pr	heavy	heavy		yes				
FL5-4-06			pr	heavy	heavy		yes				
FL5-4-07	orange		pr or	heavy	heavy		yes			yes	
FL5-4-08	brown			heavy	heavy		yes				
FL5-4-09			pr	heavy	heavy		yes				
FL5-4-10	orange		pr or	heavy	heavy		yes				
FL5-4-11				heavy	heavy		yes				
FL5-4-12	red	orange	ru	heavy	heavy		yes			yes	
FL5-4-13	brown	red	pr or gr	heavy	heavy		yes				
FL5-4-14	brown	rust		heavy			yes				
FL5-4-15	rust			heavy	heavy						
FL5-4-16	rust		pr or	light	heavy		yes				clay balls

[1] color of surface coating noted: gr = green, bl = blue, ru = rust, or = orange, pr = purple, rd = red, ye = yellow; blank = no mention
[2] extent; none, light, mod = moderate, or heavy; ; blank = no mention
[3] amount of moisture in the sample in the hole; dry, moist, wet, or rain = raining during sampling
[4] native element noted (yellow assumed to be native sulfur)
[5] gyp = gypsum; white precipitate assumed to be gypsum
[6] specific sulfides noted; po = pyrrhotite, py = pyrite, ch = chalcopyrite, yes = presence noted but not while specific sulfide

Table 41. ACP results for the 0.6% S pile.

Lift	Sample	$CaCO_3$[1] (ppm)	pH	Mass (g)	ACP (ppm/g)
1	FL1-1-02	24.00	4.23	20.62	1.16
1	FL1-1-04	16.00	4.22	19.84	0.81
1	FL1-1-06	20.00	4.17	19.10	1.05
1	FL1-1-08	12.00	4.19	17.12	0.70
1	FL1-1-10	12.00	4.11	20.42	0.59
1	FL1-1-12	20.00	4.04	17.09	1.17
1	FL1-1-14	16.00	4.20	16.93	0.95
2	FL1-2-01	20.00	4.39	17.20	1.16
2	FL1-2-03	20.00	4.33	15.37	1.30
2	FL1-2-06	36.00	4.13	15.76	2.28
2	FL1-2-08	24.00	4.32	16.13	1.49
2	FL1-2-09	44.00	4.11	20.54	2.14
2	FL1-2-11	16.00	4.92	17.05	0.94
2	FL1-2-14	60.00	3.82	20.78	2.89
2	FL1-2-16	28.00	4.00	17.64	1.59
3	FL1-3-02	68.00	3.84	16.76	4.06
3	FL1-3-04	44.00	4.20	16.97	2.59
3	FL1-3-05	8.00	7.77	21.60	0.37
3	FL1-3-07	12.00	5.07	18.77	0.64
3	FL1-3-10	8.00	5.23	20.75	0.39
3	FL1-3-12	8.00	5.24	18.46	0.43
3	FL1-3-13	8.00	7.34	19.84	0.40
3	FL1-3-15	8.00	6.21	18.59	0.43
4	FL1-4-02	8.00	7.79	22.87	0.35
4	FL1-4-04	16.00	5.53	22.97	0.70
4	FL1-4-05	8.00	7.33	22.30	0.36
4	FL1-4-07	8.00	5.39	22.96	0.35
4	FL1-4-10	8.00	6.37	25.08	0.32
4	FL1-4-12	12.00	5.64	20.06	0.60

[1] Acidity of 50 mL solution.

Table 42. ACP results for the 0.8% S pile.

Lift	Sample	$CaCO_3$[1] (ppm)	pH	Mass (g)	ACP (ppm/g)
1	FL6-1-01	28.00	3.90	20.18	1.39
1	FL6-1-02	16.00	3.94	17.31	0.92
1	FL6-1-03	12.00	3.62	18.81	0.64
1	FL6-1-04	28.00	4.37	18.16	1.54
1	FL6-1-05	36.00	4.55	20.24	1.78
1	FL6-1-06	24.00	4.13	20.50	1.17
1	FL6-1-07	112.00	3.28	21.56	5.19
1	FL6-1-08	16.00	4.04	19.10	0.84
1	FL6-1-09	44.00	4.15	22.44	1.96
1	FL6-1-10	24.00	3.98	19.33	1.24
1	FL6-1-11	48.00	3.53	20.55	2.34
1	FL6-1-12	76.00	3.76	16.37	4.64
2	FL6-2-02	84.00	4.37	18.27	4.60
2	FL6-2-04	8.00	5.01	21.24	0.38
2	FL6-2-05	136.00	3.67	17.62	7.72
2	FL6-2-07	60.00	4.10	18.35	3.27
2	FL6-2-10	184.00	3.80	16.18	11.37
2	FL6-2-12	100.00	4.54	18.25	5.48
2	FL6-2-13	108.00	3.91	22.91	4.71
2	FL6-2-15	68.00	4.19	16.83	4.04
3	FL6-3-01	36.00	4.51	20.09	1.79
3	FL6-3-03	16.00	4.96	22.29	0.72
3	FL6-3-06	40.00	4.97	21.76	1.84
3	FL6-3-08	32.00	4.88	19.47	1.64
3	FL6-3-09	68.00	4.32	20.59	3.30
3	FL6-3-11	48.00	4.47	20.64	2.33
3	FL6-3-14	28.00	4.71	20.32	1.38
3	FL6-3-16	24.00	5.44	24.55	0.98
4	FL6-4-02	572.00	3.49	21.16	27.03
4	FL6-4-04	312.00	3.68	21.80	14.31
4	FL6-4-05	56.00	4.44	21.89	2.56
4	FL6-4-07	420.00	3.84	24.03	17.48
4	FL6-4-10	68.00	4.14	22.93	2.97
4	FL6-4-12	100.00	4.21	21.25	4.71

[1] Acidity of 50 mL solution.

Table 43. ACP results for the 1.4% S pile.

Lift	Sample	$CaCO_3$[1] (ppm)	pH	Mass (g)	ACP (ppm/g)
1	FL5-1-01	40.00	3.90	18.91	2.12
1	FL5-1-02	100.00	3.39	21.61	4.63
1	FL5-1-03	316.00	3.17	18.67	16.93
1	FL5-1-05	1000.00	2.89	18.68	53.53
1	FL5-1-06	260.00	3.24	18.44	14.10
1	FL5-1-08	88.00	3.47	19.58	4.49
1	FL5-1-09	304.00	3.04	21.73	13.99
1	FL5-1-10	304.00	3.06	22.34	13.61
1	FL5-1-12	156.00	3.29	19.59	7.96
1	FL5-1-14	56.00	3.59	18.02	3.11
1	FL5-1-15	196.00	3.42	19.10	10.26
2	FL5-2-02	144.00	3.50	19.62	7.34
2	FL5-2-04	124.00	3.74	18.52	6.70
2	FL5-2-05	344.00	3.44	21.06	16.33
2	FL5-2-07	1388.00	3.15	17.65	78.64
2	FL5-2-10	540.00	3.15	18.55	29.11
2	FL5-2-12	240.00	3.56	15.18	15.81
2	FL5-2-13	600.00	3.37	18.15	33.06
2	FL5-2-15	1132.00	2.89	18.52	61.12
3	FL5-3-01	364.00	3.32	21.92	16.61
3	FL5-3-03	472.00	3.29	20.48	23.05
3	FL5-3-06	676.00	3.16	16.45	41.09
3	FL5-3-08	600.00	3.13	19.47	30.82
3	FL5-3-09	476.00	3.20	18.02	26.42
3	FL5-3-11	1000.00	3.08	16.99	58.86
3	FL5-3-14	532.00	3.19	18.02	29.52
3	FL5-3-16	644.00	3.18	19.24	33.47
4	FL5-4-02	956.00	3.20	19.73	48.45
4	FL5-4-04	1160.00	3.11	18.74	61.90
4	FL5-4-05	684.00	3.23	19.44	35.19
4	FL5-4-07	620.00	3.17	16.66	37.21
4	FL5-4-10	776.00	3.21	18.57	41.79
4	FL5-4-12	356.00	3.33	20.93	17.01
4	FL5-4-13	764.00	3.17	19.64	38.90
4	FL5-4-16	512.00	3.24	16.35	31.31

[1] Acidity of 50 mL solution.

Figure 1. Construction of test pile.

Figure 2. Lift removal from pile FL5.

Figure 3. Spatially distributed sampling sites (holes) and samplers on the surface of lift 3 of FL5.

Figure 4. Schematic diagram of test pile layout and measurements.

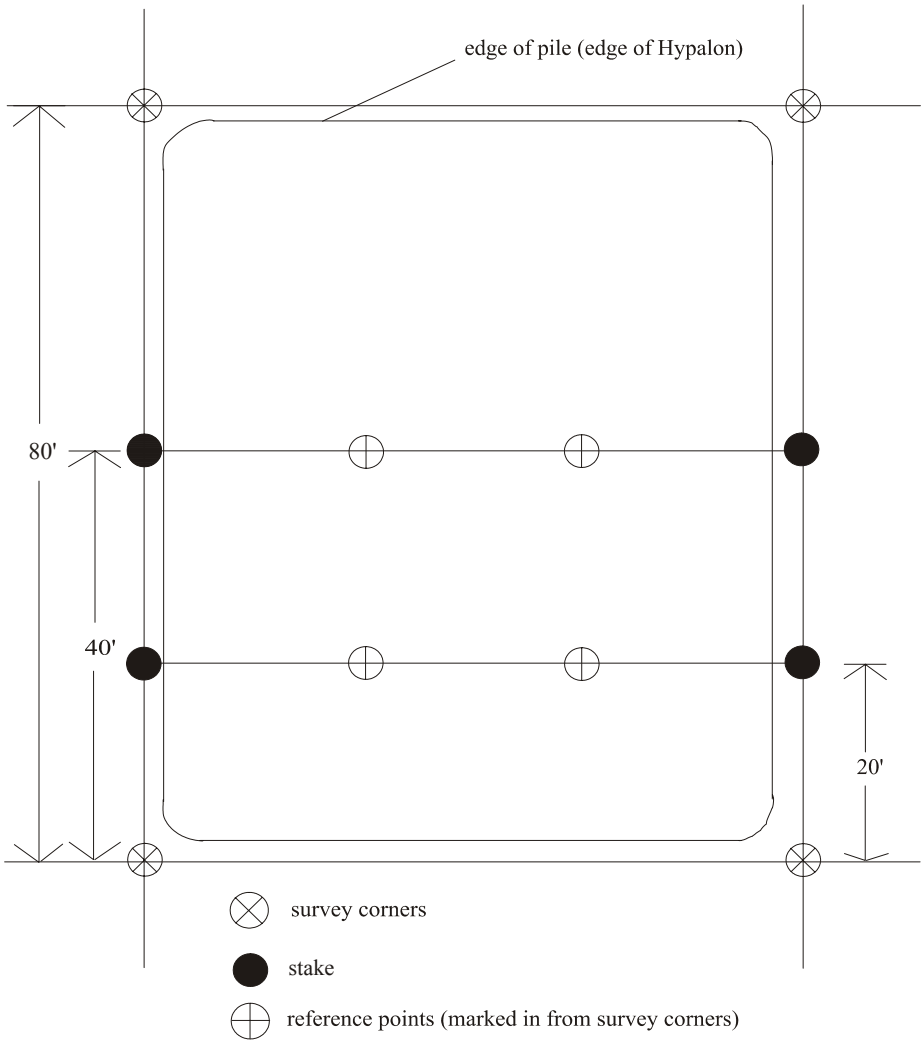


Figure 5. Side view of test pile.

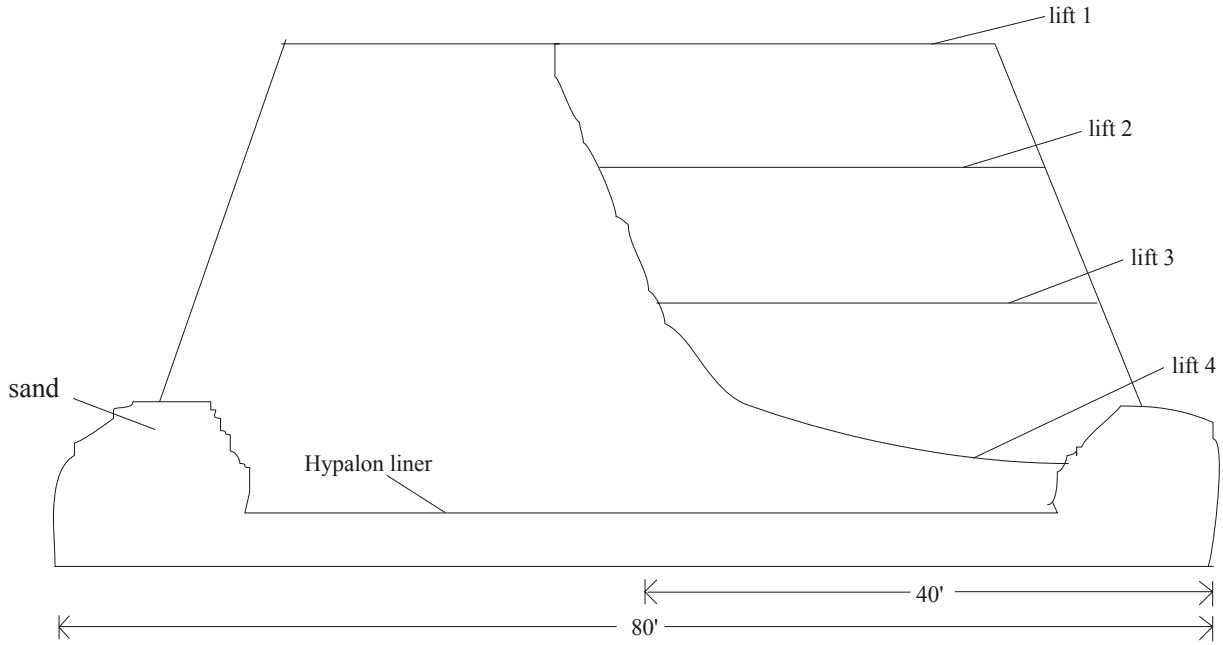
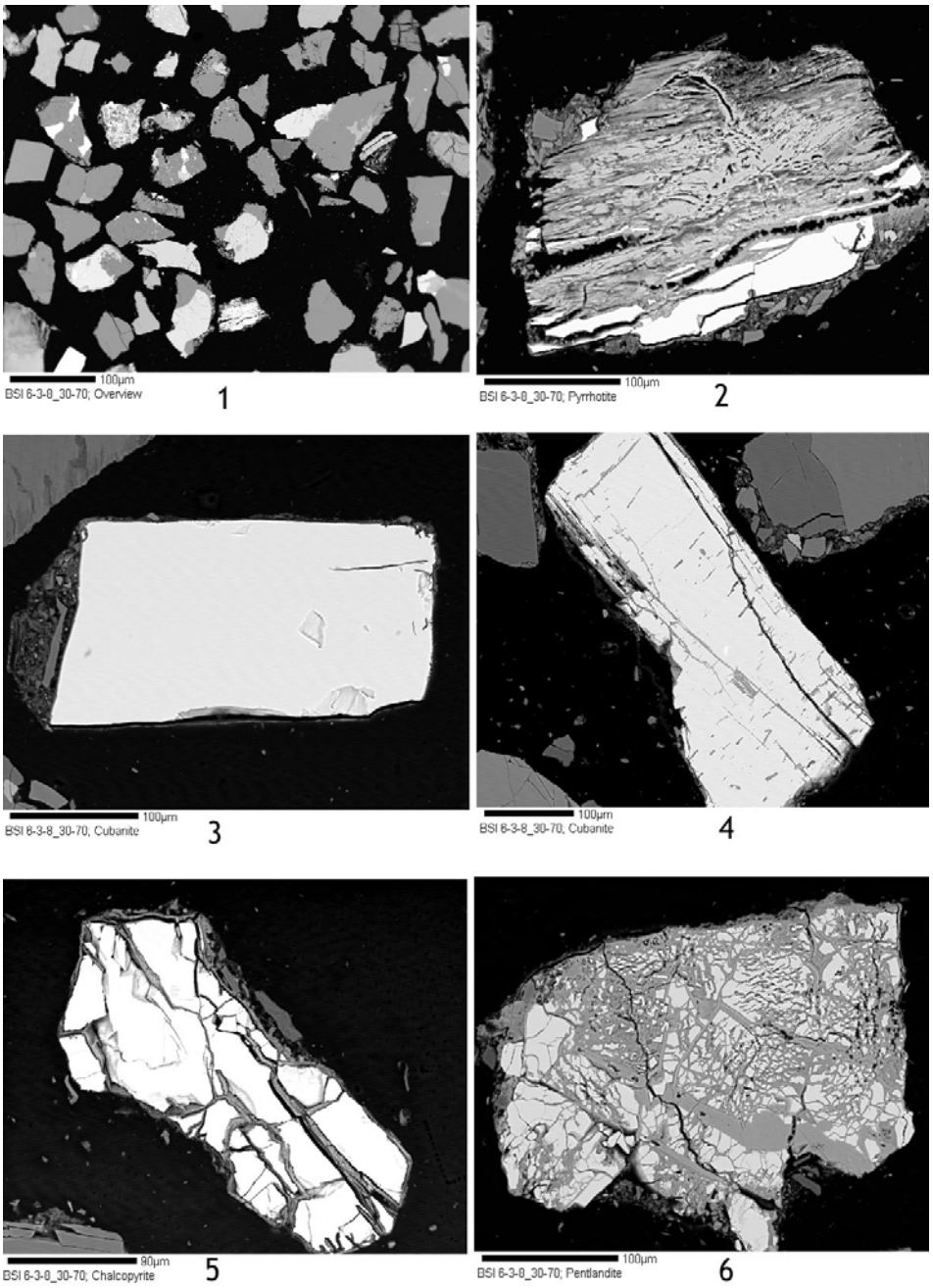


Figure 6. Backscatter microprobe images of iron sulfide grains in the 30 to 70 mesh size fraction of spatially distributed sample 6-3-8 from pile FL6 (0.6%S). The brightest gray in the large grain in photo 2 is the remaining pyrrhotite having been extensively altered. The pyrrhotite is the next least weathered in as seen in photo 6. The chalcopyrite (photo 5) and cubanite (photos 3, 4) are relatively unweathered.



Sample: 6-3-8; 30-70

APPENDIX 1

GEOLOGICAL DATA FROM SHAFT CONSTRUCTION

Table A1.1. Blast round chemistry and rock type for the 0.6% S pile (from monthly reports).

Table A1.2. Blast round rock type for the 0.8% S pile (from monthly reports).

Table A1.3. Blast round rock type for the 1.4% S pile (from monthly reports).

Table A1.4. Statistics for sulfur, copper and nickel concentrations from rounds placed in the test piles.

Table A1.5. Sulfur, copper, and nickel contents of the rounds placed in the 0.6% S pile.

Table A1.6. Sulfur, copper, and nickel contents of the rounds placed in the 0.8% S pile.

Table A1.7. Sulfur, copper, and nickel contents of the rounds placed in the 1.4% S pile.

Attachment A1.1. Description of AMAX exploration drill hole B1-162, guide hole for shaft expectations.

Table A1.1. Blast round chemistry and rock type for the 0.6% piles (from monthly reports).[1]

Depth (ft.)	Round	Footage used[2]	Average Concentrations (%)			Rock Type	Date
			Sulfur	Copper	Nickel		
716 ↕ 773	115 ↕ 122	44	0.44	0.27	0.07	gabbro	December 1976
893 ↕ 992	141 ↕ 156	99	0.62	0.38	0.09	anorthosite and troctolite anorthosite and gabbro troctolite augite troctolite	January 1977
998 ↕ 1088	157 ↕ 172	36	0.37	0.26	0.07	coarse grained troctolite fine grained, inclusive hornfels	February 1977
1132 ↕ 1325	180 ↕ 209	132	0.68	0.37	0.08	medium-coarse grained troctolite, band of hornfels with minor hornfels scattered throughout	March 1977
1366 ↕ 1502	216 ↕ 238	105	0.77	0.36	0.08	medium-coarse grained troctolite coarse grained olivine gabbro	April 1977
1506 ↕ 1540	240 ↕ 245	12	0.50	0.27	0.06	augite troctolite	May 1977

[1] Monthly reports from AMAX geologists are on file at the MN Dept of Natural Resources Lands and Minerals office in Hibbing, MN.

[2] Some rounds (footage) within the intervals were not used to construct the leach pile.

Table A1.2. Blast round rock type for the 0.8% pile (from monthly reports).[1]

Drift	Round	Rock Type	Date
A	151 ↕ 221	hornfels troctolite	September 1977
A/B/C	152 ↕ 173	hornfels troctolite	September 1977
B/C	175 ↕ 222	hornfels	September 1977

[1] Monthly reports from AMAX geologists are on file at the MN Dept of Natural Resources Lands and Minerals office in Hibbing, MN.

Table A.1.3. Blast round rock type for the 1.4% pile (from monthly reports).[1]

Drift	Round	Rock Type	Date
A	26 ↕ 128	variable intercalating mineralization interbanded troctolite some hornfels massive pyrrhotite sulfide lenses significant Cu/Ni mineralization	August 1977
B	28 ↕ 127	fine grained hornfels troctolite	August 1977
A/B/C	129 ↕ 149	hornfels troctolite	September 1977

[1] Monthly reports from AMAX geologists are on file at the MN Dept of Natural Resources Lands and Minerals office in Hibbing, MN.

Table A1.4. Statistics for sulfur, copper, and nickel concentrations in percent from the rounds placed into the test piles.

	Sulfur	Copper	Nickel
0.6% S Pile			
n	67	67	67
minimum	0.27	0.20	0.04
maximum	1.29	0.58	0.15
mean	0.64	0.35	0.08
standard deviation	0.27	0.11	0.03
median	0.58	0.32	0.08
0.8% S Pile			
n	27	27	27
minimum	0.25	0.10	0.03
maximum	1.37	0.56	0.20
mean	0.79	0.34	0.09
standard deviation	0.29	0.13	0.05
median	0.74	0.36	0.08
1.4% S Pile			
n	39	39	39
minimum	0.39	0.09	0.02
maximum	7.65	1.04	0.43
mean	1.45	0.31	0.09
standard deviation	1.33	0.19	0.08
median	1.03	0.26	0.06

Table A1.5. Sulfur, copper, and nickel contents of the rounds placed in the 0.6% S piles (piles FL1 - FL4).

Round	Sulfur (%)	Copper (%)	Nickel (%)	Thickness (feet)
115	0.45	0.20	0.06	7
117	0.69	0.40	0.10	6
118	0.54	0.30	0.08	6
119	0.40	0.26	0.07	5
120	0.46	0.32	0.08	7
121	0.31	0.22	0.06	6
122	0.27	0.21	0.05	7
141	0.33	0.25	0.05	7
143	0.48	0.46	0.12	7
144	0.29	0.25	0.05	6
145	0.62	0.48	0.11	6
146	0.60	0.43	0.10	7
147	0.58	0.39	0.11	6
148	0.50	0.36	0.09	6
149	0.50	0.36	0.09	7
150	0.63	0.42	0.10	6
151	0.88	0.42	0.14	7
152	1.05	0.55	0.12	7
153	0.81	0.34	0.09	7
154	0.66	0.32	0.08	6
155	0.72	0.35	0.08	7
156	0.53	0.28	0.07	7
157	0.40	0.22	0.07	6
158	0.39	0.23	0.07	7
159	0.38	0.20	0.06	5
166	0.29	0.23	0.06	6
171	0.44	0.38	0.08	6
172	0.32	0.28	0.06	6
180	0.39	0.30	0.05	7
181	0.47	0.37	0.09	7
184	0.37	0.27	0.05	5
185	0.56	0.30	0.06	7
186	0.47	0.35	0.06	7
187	0.62	0.32	0.06	6
188	0.64	0.20	0.04	7
190	0.69	0.23	0.06	8
191	0.79	0.29	0.07	6
192	0.50	0.24	0.06	7
193	0.53	0.25	0.06	6
194	0.41	0.22	0.05	6
195	0.66	0.31	0.07	5
196	0.90	0.44	0.10	8
197	0.84	0.47	0.11	7
199	1.12	0.56	0.12	7
200	0.72	0.56	0.12	6
201	0.87	0.56	0.13	7
208	1.07	0.58	0.15	7
209	0.94	0.55	0.12	6
216	1.11	0.56	0.13	6
219	1.29	0.54	0.10	4
220	1.18	0.53	0.10	6
221	0.91	0.41	0.08	7
222	0.81	0.48	0.11	6
223	0.57	0.33	0.10	7
224	0.62	0.32	0.08	6
225	0.68	0.31	0.08	7
226	0.82	0.37	0.10	5
227	1.28	0.44	0.13	6
228	1.21	0.42	0.12	6
229	0.93	0.42	0.09	7
230	0.54	0.27	0.05	6
232	0.54	0.26	0.05	6
233	0.29	0.20	0.05	9
235	0.36	0.22	0.04	6
238	0.31	0.23	0.04	5
243	0.61	0.33	0.07	6
245	0.40	0.22	0.06	6

Table A1.6. Sulfur, copper, and nickel contents of the rounds placed in the 0.8% S pile.

Round	Sulfur (%)	Copper (%)	Nickel (%)	Thickness (feet)
151	0.97	0.27	0.12	8
153	0.64	0.20	0.09	8
155	1.24	0.56	0.12	9
156	0.65	0.20	0.04	9
158	1.15	0.36	0.10	6
159	0.90	0.25	0.20	8
160	0.73	0.28	0.06	9
161	1.37	0.54	0.14	7
164	0.50	0.22	0.03	8
165	1.04	0.46	0.10	8
166	0.25	0.22	0.04	7
167	1.00	0.54	0.09	8
171	0.62	0.10	0.03	7
172	0.64	0.30	0.14	7
173	0.46	0.14	0.04	7
177	0.74	0.38	0.08	8
180	0.78	0.39	0.08	5
187	1.10	0.42	0.12	6
188	0.70	0.42	0.10	6
189	0.34	0.21	0.03	6
191	0.66	0.42	0.05	10
192	0.77	0.39	0.07	6
194	0.33	0.20	0.05	7
196	0.73	0.50	0.11	6
211	0.79	0.43	0.07	9
212	0.95	0.36	0.05	8
221	1.30	0.54	0.18	8

Table A1.7. Sulfur, copper, and nickel contents of the rounds placed in the 1.4% S pile.

Round	Sulfur (%)	Copper (%)	Nickel (%)	Thickness (feet)
26	1.77	0.50	0.09	4.60
28	2.09	0.68	0.10	8
29	1.40	0.57	0.08	6.40
31	0.41	0.18	0.02	8.80
34	0.72	0.27	0.04	6
36	0.58	0.12	0.02	6
37	1.42	0.40	0.06	4.50
41	2.13	0.41	0.10	4.60
47	0.54	0.17	0.04	7
49	1.00	0.27	0.05	5.70
51	0.81	0.27	0.06	6.30
55	4.39	0.61	0.24	4.50
57	7.65	1.04	0.43	9
58	2.20	0.27	0.12	4
66	2.27	0.27	0.13	4.50
68	1.86	0.27	0.12	5.50
70	1.24	0.20	0.08	7
76	1.14	0.24	0.07	7.60
78	1.01	0.45	0.06	5.10
80	1.69	0.50	0.09	6.60
82	1.39	0.34	0.09	6
84	0.95	0.20	0.06	5
88	0.84	0.15	0.05	10
90	0.85	0.22	0.06	7
92	0.90	0.20	0.04	5
94	0.71	0.22	0.04	9
96	0.53	0.18	0.02	7
102	3.39	0.45	0.22	6
104	2.89	0.40	0.20	7
116	0.54	0.16	0.03	9
127	1.24	0.26	0.08	7.70
129	1.14	0.26	0.06	8
133	0.64	0.09	0.02	7
135	0.92	0.10	0.04	8
139	1.03	0.22	0.05	7
142	1.08	0.29	0.09	8
144	0.48	0.16	0.06	8
146	0.48	0.14	0.08	8
148	0.39	0.22	0.08	9

Attachment A1.1. Description of Exploration Drill Hole B1-162, which was drilled by AMAX in 1974 and used as a guide for the proposed shaft expectations. Page 1 of 2.

Hole B1-162 met the requirements of a shaft pilot drill hole. Initially this hole recognized a relatively unmineralized portion of the basal troctolite which is generally the host rock for the copper-nickel bearing sulphides found in parts of the Duluth gabbro complex. Drill hole B1-162 therefore located the shaft site near the disseminated mineralization designated as the Local Boy zones from which a large underground sample was needed. This drill hole had two other features which served usefully as a pilot hole. Firstly, it showed relatively less deviation from the vertical than most drill holes. This would allow more confidence in the data to be predicted in the vertical shaft. Secondly, a relogging of the fracture characteristics indicated less than the average frequency of fracture, joints and possible fault zones. The anticipated rock condition could be predicted as competent.

Geologically the site is capped by about 35 feet of glacial outwash debris consisting of coarse boulders derived from the gabbro to the north and granites to the northwest. These are mixed by erratically narrow unsorted sand, gravel and silt beds.

The hole was drilled to a total of 1782 feet where it stopped in the top 16.5 feet of the Biwabik Iron Formation. The hole commenced drilling the Duluth gabbro at 35 feet and drilled through variations of this gabbro to 1,673 feet. At this point metamorphosed varieties of the Virginia slate referred to as hornfels were encountered until the Biwabik Iron Formation was intersected at 1765.5 feet.

A description of these rock units derived from logging is as follows:

<u>35-604 feet</u>

Typical olivine rich gabbro referred to as troctolite. This rock is coarse grained but variable with pegmatitic to fine grain sections of average composition occurring commonly.

The plagioclase content ranges from 60 to 70 percent. The mafics as noted are predominately olivine with subordinate variations in the content of the pyroxene, which is usually augite. Accessory minerals consist of a small amount of varying proportions of magnetite and ilmenite, and traces of biotite and alteration products such as serpentine. Sulphides are noted rarely in trace amount.

<u>604-755 feet</u>

This section contains troctolite similar to that described above, except for an inclusion of cordierite hornfels between 735 and 738. A trace of sulphides is noted throughout to range from a content of 0 to 0.7 percent. A moderately mineralized section between 740-755 assayed 0.52% copper, 0.11% nickel, and 0.72% sulphur.

755-893 feet

These sections consist of troctolite but the texture is more variable, consisting of numerous fine grained portions and rare pegmatitic sections. Some shearing and serpentinization is noted between 775 and 777 feet. No sulphides occur.

893-1446 feet

The troctolite is generally typical of the upper part of the hole except for serpentinized zones of vertical fracturing noted between 904-977. Cordierite hornfels occurs between 1171-1184. Sulphide mineralization occurs in traces throughout. The content ranges from 0 to 1 percent. A high content of sulphides ranging from 1 to 5 percent occur between 1185 and 1375. An assay breakdown for this zone follows:

From	To	Footage	%Cu	%Ni	%S
1185	1215	30	0.32	0.07	0.83
1215	1315	100	0.56	0.13	0.95
1315	1375	60	0.66	0.14	1.27

1446-1673 feet

Pyroxene forms as prominent large crystals. The pyroxene is subordinate to olivine but is increased to the extent that the rock my be called an augite troctolite. This section is magnetite rich containing 3-6 percent magnetite. Biotite is increased to 4-7 percent. A calc-silicate hornfels is included between 1580 to 1582 and a biotite hornfels between 1654 to 1656.

Traces of chalcopyrite and cubanite are noted.

1673-1765.6 feet

No sulphides occur in this section. Textural and compositional variations occur within this interval and are designated as follows.

1673-1678	hornfels breccia
1678-1695	gabbroic hornfels
1696-1732	mixed pyroxene, cordierite, biotite hornfels
1732-1765.5	biotite hornfels

1765.5-1782 feet

Biwabik Iron Formation.

This is composted of an upper marble band from 1765.5 to 1766.7 and a biotite hornfels from 1766.7 to 1770.5, followed by typical banded calc-diopside-wollastonite and quartzite laminae.

APPENDIX 2

PREVIOUS TEST PILE DATA

Table A2.1. Chemical composition of a grab sample from the 0.6%S test pile FL1.

Table A2.2. Mineralogical composition of a grab sample from the 0.6%S test pile FL1.

Table A2.3. Particle size distribution of rock from the 0.6%S test pile FL4.

Table A2.4. Characterization of fine fractions from the 0.6%S test pile FL4.

Section A2.5. Drainage quality from Duluth Complex test piles.

Table A2.5.1. Drainage quality data from Duluth Complex 0.6%S test pile (FL1), from 1978 to 1993.

Table A2.5.2. Drainage quality data for metals from Duluth Complex 0.6%S test pile (FL1), from 1978 to 1993.

Table A2.5.3. Drainage quality data for additional parameters from Duluth Complex 0.6%S test pile (FL1), from 1978 to 1993.

Table A2.5.4. Drainage quality data from Duluth Complex 0.6%S test pile (FL2), from 1978 to 1993.

Table A2.5.5. Drainage quality data for metals from Duluth Complex 0.6%S test pile (FL2), from 1978 to 1993.

Table A2.5.6. Drainage quality data for additional parameters from Duluth Complex 0.6%S test pile (FL2), from 1978 to 1993.

Table A2.5.7. Drainage quality data from Duluth Complex 0.6%S test pile (FL3), from 1978 to 1989.

Table A2.5.8. Drainage quality data for metals from Duluth Complex 0.6%S test pile (FL3), from 1978 to 1989.

Table A2.5.9. Drainage quality data for additional parameters from Duluth Complex 0.6%S test pile (FL3), from 1978 to 1989.

Table A2.5.10. Drainage quality data from Duluth Complex 0.6%S test pile (FL4), from 1978 to 1982.

Table A2.5.11. Drainage quality data for metals from 0.6%S Duluth Complex test pile (FL4), from 1978 to 1982.

Table A2.5.12. Drainage quality data for additional parameters from Duluth Complex 0.6%S test pile (FL4), from 1978 to 1982.
Table A2.5.13. Drainage quality data from Duluth Complex 0.8%S test pile (FL6), from 1978 to 1993.
Table A2.5.14. Drainage quality data for metals from Duluth Complex 0.8%S test pile (FL6), from 1978 to 1993.
Table A2.5.15. Drainage quality data for additional parameters from Duluth Complex 0.8%S test pile (FL6), from 1978 to 1993.
Table A2.5.16. Drainage quality data from Duluth Complex 1.4 %S test pile (FL5), from 1978 to 1993.
Table A2.5.17. Drainage quality data for metals from Duluth Complex 1.4 %S test pile (FL5), from 1978 to 1993.
Table A2.5.18. Drainage quality data for additional parameters from Duluth Complex 1.4 %S test pile (FL5), from 1978 to 1993.

Section A2.6. Drainage quality versus time figures.
Figure A2.6.1. pH and sulfate versus time for FL1 (0.6 %S): 1978-1993.
Figure A2.6.2. Calcium, magnesium, and sodium versus time for FL1 (0.6 %S): 1978-1993.
Figure A2.6.3. pH and sulfate versus time for FL2 (0.6 %S): 1978-1993.
Figure A2.6.4. Calcium, magnesium, and sodium versus time for FL2 (0.6 %S): 1978-1993.
Figure A2.6.5. pH and sulfate versus time for FL3 (0.6 %S): 1978-1989.
Figure A2.6.6. Calcium, magnesium, and sodium versus time for FL3 (0.6 %S): 1978-1989.
Figure A2.6.7. pH and sulfate versus time for FL4 (0.6 %S): 1978-1982.
Figure A2.6.8. Calcium, magnesium, and sodium versus time for FL4 (0.6 %S): 1978-1982.
Figure A2.6.9. pH and sulfate versus time for FL6 (0.8 %S): 1978-1993.
Figure A2.6.10. Calcium, magnesium, and sodium versus time for FL6 (0.8 %S): 1978-1993.
Figure A2.6.11. pH and sulfate versus time for FL5 (1.4 %S): 1978-1993.
Figure A2.6.12. Calcium, magnesium, and sodium versus time for FL5 (1.4 %S): 1978-1993.

Section A2.7. Annual total mass release from Duluth Complex test piles.
Table A2.7.1. Annual total mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.6%S test pile (FL1).
Table A2.7.2. Annual total mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.6 %S test pile (FL2).
Table A2.7.3. Annual total mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.6%S test pile (FL3).
Table A2.7.4. Annual total mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.6%S test pile (FL4).
Table A2.7.5. Annual total mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.8%S test pile (FL6).

Table A2.7.6. Annual total mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 1.4%S test pile (FL5).
Table A2.7.7. Sulfur, calcium, magnesium, copper, and nickel percent mass release from the six test piles.

Section A2.8. Annual rates of release from Duluth Complex test piles.
Table A2.8.1. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.6 %S test pile (FL1).
Table A2.8.2. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.6%S test pile (FL2).
Table A2.8.3. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.6%S test pile (FL3).
Table A2.8.4. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.6%S test pile (FL4).
Table A2.8.5. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 0.8%S test pile (FL6).
Table A2.8.6. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for 1.4%S test pile (FL5).

Section A2.9. Calcium, magnesium, and sodium versus sulfate figures.
Figure A2.9.1. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL1 (0.6%S): 1978-1993.
Figure A2.9.2. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL2 (0.6%S): 1978-1993.
Figure A2.9.3. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL3 (0.6%S): 1978-1989.
Figure A2.9.4. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL4 (0.6%S): 1978-1982.
Figure A2.9.5. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL6 (0.8%S): 1978-1993.
Figure A2.9.6. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL5 (1.4%S): 1978-1993.

Section A2.10. Ratios of concentrations of Mg:Ca and Na:Ca versus time figures.
Figure A2.10.1. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.6 %S pile (FL1): 1978-1993.
Figure A2.10.2. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.6 %S pile (FL2): 1978-1993.
Figure A2.10.3. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.6 %S pile (FL3): 1978-1989.
Figure A2.10.4. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.6 %S pile (FL4): 1978-1982.
Figure A2.10.5. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.8 %S pile (FL6): 1978-1993.
Figure A2.10.6. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 1.4 %S pile (FL5): 1978-1993.

Section A2.11. Calcium and magnesium in drainage from the test piles versus pH.
Figure A2.11.1. Calcium and magnesium in drainage from pile FL1 (0.6 %S) versus pH.
Figure A2.11.2. Calcium and magnesium in drainage from pile FL2 (0.6 %S) versus pH.
Figure A2.11.3. Calcium and magnesium in drainage from pile FL3 (0.6 %S) versus pH.
Figure A2.11.4. Calcium and magnesium in drainage from pile FL4 (0.6 %S) versus pH.
Figure A2.11.5. Calcium and magnesium in drainage from pile FL6 (0.8 %S) versus pH.
Figure A2.11.6. Calcium and magnesium in drainage from pile FL5 (1.4 %S) versus pH.

Section A2.12. Flow weighted mean concentrations for the test piles.
Table A2.12.1. Flow weighted mean concentrations for the 0.6 %S pile, FL1.
Table A2.12.2. Flow weighted mean concentrations for the 0.6 %S pile, FL2.
Table A2.12.3. Flow weighted mean concentrations for the 0.6 %S pile, FL3.
Table A2.12.4. Flow weighted mean concentrations for the 0.6 %S pile, FL4.
Table A2.12.5. Flow weighted mean concentrations for the 0.8 %S pile, FL6.
Table A2.12.6. Flow weighted mean concentrations for the 1.4 %S pile, FL5.

Table A2.1. Chemical composition of a grab sample from the 0.6%S test pile FL1 (1981).

Element	Concentration	Unit
S	0.655	pct
Cu	0.306	pct
Ni	0.073	pct
Fe (S)[1]	0.696	pct
Co	0.009	pct
Zn	0.016	pct
Si (SiO_2)	22.80 (48.81)	pct
Al (Al_20_3)	8.98 (16.96)	pct
Fe (FeO)	9.22 (11.86)	pct
Mg (MgO)	4.31 (7.15)	pct
Ca (CaO)	5.55 (7.76)	pct
Na (Na_20)	2.00 (2.70)	pct
K (K_20)	0.32 (0.39)	pct
Ti ($Ti0_2$)	1.41 (2.35)	pct
P (P_2O_5)	0.02 (0.05)	pct
Mn (MnO)	0.12 (0.16)	pct
Cr (Cr_20_3)	0.03 (0.15)	pct
B	662.00	ppm
Ba	1173.00	ppm
Be	1.00	ppm
Sr	279.00	ppm
Pb	0	ppm
Ag	1.35	ppm
As	0	ppm
Hg	1.00	ppm
Mo	0.50	ppm
Cd	0	ppm
V	276.50	ppm
Th	6.40	ppm
Zr	80.00	ppm

[1] Iron associated with sulfur

Table A2.2. Mineralogical composition of a grab sample from the 0.6%S test pile FL1.

Mineral	Volume percent
Opaques	5.098
Chalcopyrite-cubanite	0.769
Pentlandite	0.037
Pyrrhotite	0.844
Ilmenite-magnetite	3.447
Graphite	–[2]
Plagioclase	59.112
Sericite	2.176
Olivine	10.510
Clinopyroxene	11.185
Orthopyroxene	3.716
Monocrystalline amphibole	3.567
Fibrous amphibole	0.288
Chlorite	1.136
Serpentine	0.257
Iddingsite	0.075
Talc	–
Biotite	1.738
Smectite	0.021
Caladonite	–
Spinel	–
Myrmekite	–
Apatite	0.085
Epidote	1.953
Allanite	–
Calcite	0.056
Quartz	–
Cordierite	0.027

[1] The value shown for opaques is the sum of the five following values.
[2] None detected.

Table A2.3. Particle size distribution of rock from the 0.6%S test pile FL4. Sample collected in September 1982.

	Particle diameter (mm)	Percent finer than
Boulders & cobbles	305	89.0
	152	82.2
Gravel	76.2	67.7
	25.4	38.3
	19.1	33.8
	12.7	27.1
	6.4	19.2
Sand	2.00	12.2
	1.41	10.8
	0.84	10.6
	0.425	7.7
	0.250	6.1
	0.149	4.6
	0.105	3.4
Silt & clay	0.074	3.1
	0.053	1.4 to 1.7[1]

Note: Boulders, cobbles, gravel, sand and silt refer to US Department of Agriculture (USDA) classification definitions.

[1] Estimated based on hydrometer analysis of the <0.074 mm fraction.

Table A2.4. Characterization of the fine fractions of the 0.6% FL4 test pile material used in the joint disposal bins.

Size Fraction mm	pH	Total Sulfur %	Sulfide %	Cu %	Ni %	Co %	Zn %	Specific Surface Area m^2/g
Sand								
0.50–2.00	6.56	0.67	0.65	0.337	0.062	0.012	0.015	0.60
0.177–0.50	6.37	0.80	0.75	0.381	0.078	0.013	0.015	0.78
0.149–0.177	6.18	0.88	0.80	0.391	0.103	0.015	0.015	1.1
0.105–0.149	NA[1]	1.12	1.05	0.495	0.147	0.019	0.017	1.6
0.075–0.105	5.95	1.37	1.30	0.585	0.221	0.022	0.019	1.7
Silt and clay								
0.053–0.075	5.78	1.65	1.57	0.647	0.253	0.024	0.019	2.6
<0.053	5.77	1.94	1.86	0.814	0.295	0.027	0.023	4.7

[1] NA = not available.

Table A2.5.1. Drainage quality data from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
4	26	1978	.	0.0	7.15	1120.00	.	.	.	.	112.00	11.00	.
6	6	1978	2558.0	0.0	.	1900.00	.	.	.	.	216.00	19.00	.
6	7	1978	.	0.0	7.60	3325.00	.	.	.	.	253.00	23.00	.
6	16	1978	120.0	0.0	7.84	2700.00	.	.	.	.	244.00	22.00	.
6	23	1978	170.0	0.0	7.72	2041.67	.	.	.	1266.67	183.30	17.30	.
6	26	1978	59.0	0.0	6.85	2400.00	.	.	.	.	195.00	20.00	.
7	5	1978	177.0	0.0	7.75	2100.00	.	.	.	900.00	202.00	20.00	.
7	7	1978	.	0.0	7.90	1900.00	.	.	.	800.00	221.00	23.00	.
7	11	1978	.	0.0	7.70	2450.00	.	.	.	1150.00	303.00	26.00	.
7	13	1978	.	0.0	7.92	.	.	.	.	1500.00	531.00	31.00	.
7	18	1978	.	0.0	7.80	1400.00	.	.	.	1050.00	340.00	19.00	.
8	2	1978	.	0.0	7.90	1100.00	.	.	.	650.00	176.00	11.00	.
8	4	1978	167.0	0.0	7.80	1650.00	.	.	.	1200.00	300.00	23.00	.
8	10	1978	70.0	0.0	8.10	2400.00	70.0	.	.	1600.00	414.00	26.00	.
8	16	1978	3441.0	0.0	7.85	1538.00	42.0	.	.	640.00	169.50	14.00	.
8	24	1978	.	0.0	8.10	2600.00	50.0	.	.	1400.00	302.00	26.00	.
8	25	1978	.	0.0	7.55	2500.00	47.0	.	.	1300.00	300.00	26.00	.
8	28	1978	.	0.0	7.85	2700.00	59.0	.	.	.	308.00	26.00	.
9	13	1978	426.0	0.0	7.58	2087.50	40.5	.	.	1050.00	246.00	19.00	.
10	17	1978	.	0.0	7.40	3207.00	41.0	.	.	1800.00	485.00	42.00	.
3	19	1979	2630.6	1986.0	7.45	807.00	21.0	.	.	.	164.00	13.20	.
4	13	1979	2615.7	390.0	7.10	693.00	41.0	.	.	335.00	109.00	9.40	.
4	18	1979	3665.3	3966.0	7.20	436.00	33.0	.	.	190.00	78.00	5.60	.
4	19	1979	4652.3	4029.0	7.10	360.00	35.0	.	.	165.00	69.00	4.80	.
4	20	1979	3664.3	2274.0	6.60	534.00	39.0	.	.	250.00	100.00	7.80	.
4	22	1979	4152.6	1200.0	7.65	1193.50	27.0	.	.	633.00	185.90	18.10	.
5	9	1979	685.2	815.0	7.50	983.00	.	.	.	483.00	173.00	15.20	.
5	10	1979	6571.5	2979.0	7.30	1057.00	.	.	.	467.00	162.80	15.60	.
5	20	1979	2990.5	784.0	7.10	1144.00	.	.	.	667.00	179.60	17.80	.
5	22	1979	1385.5	848.0	7.70	897.00	.	.	.	533.00	144.40	13.80	.
5	23	1979	2400.0	596.0	7.50	1720.00	.	.	.	900.00	232.40	23.60	.
6	1	1979	2468.1	737.0	7.30	1185.00	.	.	.	700.00	170.20	17.20	.
6	6	1979	2229.6	941.0	7.50	1204.00	.	.	.	675.00	174.00	17.00	.
6	10	1979	2142.5	1098.0	7.40	1168.00	54.0	.	.	580.00	166.00	16.40	.
6	12	1979	1181.0	416.0	7.20	1687.00	45.0	.	.	780.00	245.80	24.00	.
6	15	1979	545.1	494.0	7.40	1488.00	48.4	.	.	740.00	234.80	23.00	.
6	21	1979	2343.2	1073.0	7.10	1030.00	30.0	.	.	450.00	144.00	15.00	.
6	30	1979	488.3	149.0	7.40	1349.00	.	.	.	1280.00	338.00	33.00	.
7	10	1979	1131.8	137.0	7.60	2120.00	37.0	.	.	1600.00	371.00	37.00	.
7	30	1979	3448.5	2274.0	7.40	1220.50	20.8	.	.	589.00	208.50	18.50	.
8	4	1979	1313.5	212.0	7.60	2094.00	26.8	.	.	1520.00	439.00	34.00	.
8	13	1979	344.5	47.0	7.30	2562.00	31.2	.	.	1920.00	488.80	49.00	.
8	23	1979	371.0	151.0	7.30	1976.00	24.8	.	.	1500.00	394.80	33.20	.
8	28	1979	548.0	458.0	7.30	1642.00	18.0	.	.	1130.00	298.00	25.40	.
9	1	1979	3709.7	1929.0	7.50	2014.00	23.2	.	.	1250.00	364.00	31.00	.
9	5	1979	741.9	550.0	7.40	2145.00	25.2	.	.	1520.00	427.20	39.00	.
9	10	1979	2608.0	516.0	7.35	1793.50	24.0	.	.	1290.00	343.40	29.20	.
9	13	1979	1408.2	500.0	7.50	2112.00	24.0	.	.	1600.00	428.20	37.60	.
9	19	1979	885.8	141.0	7.40	3010.00	25.2	.	.	2200.00	590.80	52.00	.
10	18	1979	352.0	733.0	7.10	2603.00	20.8	.	.	1900.00	502.00	45.40	.
10	19	1979	821.0	1222.0	6.80	1318.00	19.2	.	.	750.00	223.00	19.60	.
10	31	1979	4315.4	1513.0	6.80	1959.00	20.8	.	.	1300.00	374.00	32.60	.
11	1	1979	2695.2	966.0	7.20	2364.00	24.8	.	.	1000.00	412.00	38.20	.
4	17	1980	314.2	454.0	7.25	790.00	32.2	.	.	364.00	130.20	8.60	.
4	18	1980	927.4	907.0	6.95	.	38.0	.	.	550.00	145.40	11.20	.
4	23	1980	927.4	216.0	7.45	.	45.4	.	.	490.00	148.00	11.60	.
4	27	1980	2131.2	157.0	7.50	1442.00	52.7	.	.	1045.00	233.70	20.80	.
5	5	1980	420.2	201.0	7.60	1330.00	45.1	.	.	1020.00	227.00	18.60	.
5	18	1980	806.3	50.0	7.60	1300.00	46.2	.	.	910.00	218.40	20.60	.
5	31	1980	1798.1	1190.0	7.30	950.00	30.8	.	.	650.00	159.30	14.50	.
6	1	1980	893.4	441.0	7.50	1575.00	36.9	.	.	1020.00	276.20	24.20	.
6	7	1980	798.7	189.0	7.45	1475.00	32.0	.	.	800.00	283.60	25.40	.
6	14	1980	579.2	139.0	7.40	1700.00	29.7	.	.	1170.00	274.00	25.00	.
6	18	1980	367.2	46.0	7.30	2200.00	31.9	.	.	1450.00	348.00	23.70	.
6	27	1980	995.6	245.0	7.30	1700.00	25.3	.	.	1150.00	273.00	26.00	.

Table A2.5.1. Drainage quality data from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
7	2	1980	1059.9	1272.0	7.30	2300.00	33.0	.	.	2100.00	422.00	32.00	.
7	4	1980	726.8	1017.0	7.20	1850.00	19.6	.	.	.	354.00	29.00	.
7	17	1980	1022.1	356.0	7.20	2500.00	28.6	.	.	1590.00	428.40	34.80	.
7	20	1980	1025.8	171.0	7.20	2100.00	30.8	.	.	1300.00	363.80	33.40	.
8	3	1980	495.9	311.0	7.20	2100.00	24.2	.	.	1150.00	355.20	34.40	.
8	12	1980	1994.9	1239.0	7.20	1500.00	9.8	.	.	990.00	275.00	23.40	.
8	19	1980	1642.9	350.0	7.10	2000.00	18.7	.	.	1520.00	393.40	34.60	.
8	22	1980	1343.8	657.0	.	1850.00	.	.	.	1290.00	357.20	35.20	.
8	29	1980	8112.1	3456.0	7.10	1850.00	23.4	.	.	1658.00	382.20	40.60	.
9	7	1980	2702.8	1050.0	7.00	1500.00	20.4	.	.	920.00	287.00	26.60	.
9	9	1980	1147.0	361.0	7.20	1760.00	22.1	.	.	1250.00	382.80	39.20	.
9	12	1980	3535.6	1890.0	7.05	1215.00	20.0	.	.	660.00	262.00	21.00	.
9	18	1980	7896.4	2008.0	7.20	1535.00	21.0	.	.	1130.00	340.00	33.20	.
9	23	1980	1938.1	588.0	7.20	2117.00	21.3	.	.	1350.00	398.00	45.00	.
10	15	1980	1741.3	101.0	7.00	1719.00	20.4	.	.	1050.00	310.00	32.00	.
10	31	1980	1256.8	48.0	6.50	1773.00	17.0	.	.	1900.00	340.00	30.00	.
2	18	1981	832.0	722.0	7.20	525.00	18.7	.	.	266.00	74.80	7.20	.
3	27	1981	4742.0	2270.0	7.20	600.00	19.2	.	.	300.00	80.60	8.30	.
4	3	1981	4410.0	1459.0	7.30	800.00	27.2	.	.	440.00	124.80	13.00	.
4	13	1981	2441.0	1050.0	6.70	900.00	17.6	.	.	510.00	143.00	16.00	.
4	17	1981	1696.0	988.0	7.40	1000.00	30.4	.	.	515.00	139.00	16.00	.
4	23	1981	2350.0	2515.0	7.40	1350.00	28.8	.	.	635.00	165.00	22.00	.
4	27	1981	6162.0	2089.0	7.45	1500.00	29.2	.	.	740.00	203.00	32.00	.
4	28	1981	3785.0	1698.0	7.60	1250.00	36.0	.	.	640.00	168.00	27.00	.
4	29	1981	3323.0	1040.0	7.40	1430.00	37.0	.	.	835.00	230.00	28.00	.
5	12	1981	3686.0	203.0	7.55	1587.00	38.4	.	.	800.00	272.00	32.00	.
6	1	1981	2626.0	790.0	7.50	1064.00	26.0	.	.	450.00	184.00	20.00	.
6	6	1981	1840.0	207.0	7.40	1202.00	13.3	.	.	620.00	200.00	38.00	.
6	22	1981	4148.0	2105.0	7.00	1112.00	15.2	.	.	590.00	196.00	18.00	.
6	24	1981	3274.0	1541.0	7.20	1216.00	16.0	.	.	675.00	216.00	18.00	.
6	28	1981	4690.0	2922.0	7.20	1143.00	15.2	.	.	640.00	205.00	18.00	.
7	4	1981	2154.0	117.0	7.40	1786.00	21.4	.	.	1100.00	348.00	29.60	.
7	15	1981	606.0	40.0	7.50	1586.00	15.0	.	.	1000.00	284.00	21.60	.
7	24	1981	1094.0	238.0	7.20	1464.00	14.2	.	.	945.00	274.00	23.40	.
8	4	1981	722.0	75.0	7.50	1798.00	13.4	.	.	1200.00	356.00	36.00	.
8	17	1981	602.0	20.0	7.10	1936.00	12.2	.	.	1200.00	392.00	36.00	.
9	1	1981	772.0	100.0	7.00	2275.00	9.6	.	.	1180.00	352.00	34.00	.
9	16	1981	148.0	5.0	6.90	2700.00	4.8	.	.	.	482.00	48.00	.
9	26	1981	1037.0	587.0	6.75	1850.00	5.6	.	.	1120.00	344.00	36.00	.
10	1	1981	3043.0	1895.0	6.90	1800.00	6.4	.	.	940.00	320.00	54.00	.
10	4	1981	2642.0	1354.0	6.90	1700.00	8.8	.	.	1020.00	358.00	32.00	.
10	13	1981	3622.0	1799.0	7.05	1500.00	10.8	.	.	800.00	262.00	26.00	.
10	17	1981	8856.0	1213.0	7.00	1700.00	11.2	.	.	1010.00	336.00	32.00	.
10	30	1981	3104.0	271.0	7.15	2600.00	6.4	.	.	1750.00	490.00	60.00	.
3	24	1982	715.0	.	6.90	950.00	10.8	.	.	490.00	136.00	16.60	.
4	13	1982	1090.0	.	7.00	850.00	10.8	.	.	440.00	121.00	11.20	.
4	14	1982	4141.0	.	6.97	500.00	7.7	.	.	215.00	63.80	5.20	.
4	15	1982	2854.0	.	7.10	700.00	15.3	.	.	340.00	106.00	8.60	.
4	19	1982	912.0	.	6.90	825.00	14.4	.	.	380.00	166.00	12.00	.
4	23	1982	1238.0	.	7.20	1050.00	18.0	.	.	600.00	190.00	16.00	.
4	28	1982	3274.0	.	7.10	1175.00	19.8	.	.	630.00	157.00	16.00	.
5	3	1982	409.0	.	7.10	750.00	12.6	.	.	370.00	118.00	10.00	.
5	7	1982	5167.0	.	7.10	1030.00	18.9	.	.	520.00	164.00	14.00	.
5	11	1982	3819.0	.	7.10	1225.00	18.0	.	.	710.00	188.00	24.00	.
5	14	1982	4038.0	.	7.10	1200.00	21.6	.	.	690.00	216.00	22.80	.
5	21	1982	1688.0	.	7.15	1800.00	.	.	.	1100.00	308.00	31.00	.
6	2	1982	1253.0	.	7.00	1275.00	12.6	.	.	710.00	198.00	21.80	.
6	11	1982	719.0	.	6.80	1175.00	.	.	.	665.00	185.00	18.50	.
7	2	1982	19809.0	.	.	.	.	.	.	780.00	308.00	26.00	.
7	26	1982	1382.0	.	7.10	1400.00	12.6	.	.	890.00	338.00	28.00	.
8	2	1982	8025.0	.	7.00	1900.00	.	.	.	.	394.00	30.00	.
8	9	1982	1817.0	.	7.00	2100.00	17.1	.	.	.	367.00	34.00	.
8	23	1982	1790.0	.	7.10	1850.00	8.1	.	.	1145.00	359.00	32.00	.
8	30	1982	1147.0	.	6.90	1850.00	9.9	.	.	1035.00	338.00	30.00	.
9	8	1982	4009.0	.	6.50	1350.00	10.8	.	.	785.00	256.00	22.00	.

Table A2.5.1. Drainage quality data from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
9	13	1982	4190.0	.	6.50	1300.00	11.2	.	.	975.00	316.00	26.00	.
9	20	1982	810.0	.	6.90	1900.00	.	.	.	1140.00	334.00	32.00	.
10	4	1982	15259.0	.	7.00	1425.00	16.2	.	.	850.00	258.00	26.00	.
10	11	1982	3782.0	.	6.80	1400.00	15.3	.	.	630.00	252.00	24.00	.
10	21	1982	1200.0	.	6.90	1000.00	13.0	.	.	620.00	266.00	26.00	.
10	25	1982	708.0	.	7.00	1830.00	13.1	.	.	940.00	256.00	26.00	.
11	1	1982	1730.0	.	6.90	1000.00	9.0	.	.	.	136.00	14.00	.
11	22	1982	.	.	7.05	1400.00	4.5	.	.	605.00	230.00	28.00	.
3	18	1983	2495.0	.	6.90	745.00	13.3	.	.	400.00	.	.	.
4	14	1983	4984.0	.	6.86	544.00	14.8	.	.	260.00	.	.	.
5	15	1983	6196.0	.	7.01	910.00	15.3	.	.	540.00	.	.	.
6	16	1983	2168.0	.	6.86	1348.00	11.8	.	.	840.00	.	.	.
7	14	1983	11710.0	.	6.91	1663.00	12.6	.	.	1320.00	.	.	.
8	14	1983	14114.0	.	6.61	1497.00	7.5	.	.	1140.00	.	.	.
9	15	1983	8997.0	.	6.74	1507.00	7.1	.	.	1110.00	.	.	.
10	13	1983	24777.0	.	6.91	1714.00	9.8	.	.	1110.00	.	.	.
11	9	1983	3459.0	.	6.60	900.00	5.4	.	.	620.00	.	.	.
11	30	1983	738.0	.	6.35	900.00	5.4	.	.	620.00	.	.	.
2	24	1984	1222.0	.	6.25	580.00	6.3	.	.	290.00	93.90	18.90	100
3	15	1984	.	.	.	.	.	.	.	.	.	.	101
4	15	1984	6389.0	.	6.49	429.00	4.9	.	.	260.00	94.70	13.60	102
5	16	1984	10099.0	.	6.86	865.00	10.7	.	.	620.00	210.60	28.10	103
6	13	1984	21645.0	.	6.91	1065.00	7.9	.	.	838.00	260.10	36.60	104
7	14	1984	5190.0	.	6.92	1145.00	9.0	.	.	820.00	.	.	105
8	15	1984	5678.0	.	6.81	1395.00	6.8	.	.	880.00	278.40	53.90	106
9	12	1984	1809.0	.	6.36	1425.00	3.3	.	.	1400.00	326.60	67.80	107
10	13	1984	10556.0	.	6.17	1425.00	4.3	.	.	1750.00	401.20	60.00	108
11	14	1984	2430.0	.	6.23	1050.00	4.9	.	.	1400.00	352.00	52.00	109
12	1	1984	409.0	.	6.00	975.00	2.1	.	.	1050.00	390.00	190.00	110
12	18	1984	1452.0	.	6.15	875.00	4.2	.	.	710.00	216.00	48.00	111
4	2	1985	308.0	.	6.55	420.00	1.1	.	.	270.00	85.00	6.60	100
4	16	1985	16654.0	.	6.56	845.00	7.6	.	.	520.00	162.00	29.30	101
5	14	1985	10821.0	.	6.93	1200.00	6.9	.	.	710.00	204.00	37.50	102
6	14	1985	33032.0	.	6.94	1425.00	2.6	.	.	1100.00	241.00	45.10	103
7	15	1985	12354.0	.	6.82	1350.00	7.6	.	.	950.00	193.00	43.40	104
8	13	1985	12188.0	.	6.70	1350.00	3.3	.	.	968.00	230.20	43.10	105
9	13	1985	28330.0	.	6.58	1475.00	1.6	.	.	1000.00	316.00	49.80	106
10	15	1985	18471.0	.	6.85	1275.00	5.2	.	.	650.00	242.00	38.00	107
11	15	1985	1238.0	.	6.64	1175.00	1.1	.	.	540.00	254.00	36.00	108
3	23	1986	18543.0	.	6.60	570.00	1.1	.	.	230.00	76.00	14.60	100
4	16	1986	12687.0	.	6.49	775.00	2.8	.	.	410.00	131.00	28.10	101
5	14	1986	12884.0	.	6.75	1025.00	5.2	.	.	800.00	244.00	38.00	102
6	11	1986	6703.0	.	6.63	1175.00	5.6	.	.	850.00	216.00	38.00	103
7	12	1986	19489.0	.	6.69	1375.00	5.9	.	.	800.00	246.00	50.00	104
8	13	1986	11835.0	.	6.29	1550.00	4.2	.	.	1100.00	272.00	47.00	105
9	14	1986	23641.0	.	6.54	1500.00	3.0	.	.	700.00	271.00	48.40	106
10	16	1986	4621.0	.	6.41	1925.00	5.2	.	.	1200.00	276.00	52.00	107
11	16	1986	9148.0	.	6.36	1750.00	3.4	.	.	1125.00	332.00	68.00	108
3	22	1987	10068.0	.	5.70	575.00	6.1	.	.	294.00	96.00	18.00	100
4	15	1987	3009.0	.	6.35	725.00	4.6	.	.	352.00	94.00	22.00	101
5	16	1987	32721.0	.	6.59	1500.00	6.0	.	.	970.00	294.00	50.00	102
6	16	1987	2328.0	.	6.66	1900.00	6.6	.	.	1200.00	368.00	72.00	103
7	15	1987	25288.0	.	6.03	1850.00	7.4	.	.	1200.00	312.00	60.00	104
8	15	1987	12176.0	.	6.04	1600.00	5.4	.	.	1080.00	270.00	56.00	105
9	16	1987	16525.0	.	6.01	2275.00	4.3	.	.	1070.00	314.00	68.00	106
10	13	1987	1764.0	.	5.86	2150.00	1.8	.	.	1390.00	400.00	76.00	107
11	10	1987	889.0	.	5.81	1850.00	2.9	.	.	1280.00	330.00	60.00	108
3	28	1988	7169.0	.	6.50	675.00	5.1	.	.	340.00	74.00	24.00	100
4	17	1988	5008.0	.	5.93	970.00	2.3	.	.	700.00	116.00	36.00	101
5	12	1988	11014.0	.	6.11	1500.00	6.7	.	.	1000.00	276.00	64.00	102
6	12	1988	3160.0	.	6.12	1320.00	4.8	.	.	820.00	282.00	52.00	103
7	15	1988	3565.0	.	5.91	1450.00	5.3	.	.	1120.00	253.00	60.00	104
8	16	1988	56169.0	.	6.27	1575.00	7.1	.	.	915.00	277.00	73.00	105
9	16	1988	11276.0	.	6.09	1600.00	3.0	.	.	860.00	280.00	78.00	106
10	18	1988	3346.0	.	5.93	1575.00	3.6	.	.	1105.00	239.00	66.00	107

Table A2.5.1. Drainage quality data from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
11	15	1988	5583.0	.	5.57	1200.00	2.4	.	.	900.00	180.00	116.00	108
11	30	1988	8.0	.	.	.	.	.	.	.	.	.	109
4	7	1989	26374.0	.	5.20	530.00	1.0	.	.	284.00	70.00	20.00	100
4	24	1989	7483.0	.	5.87	575.00	3.7	.	.	296.00	64.00	18.00	101
5	16	1989	13668.0	.	5.69	1325.00	1.3	.	.	790.00	192.00	52.00	102
6	17	1989	37191.0	.	5.92	1725.00	6.3	.	.	1270.00	266.00	76.00	103
7	17	1989	3959.0	.	5.60	1900.00	4.4	.	.	1350.00	342.00	78.00	104
8	14	1989	4285.0	.	5.83	1400.00	5.1	.	.	880.00	206.00	54.00	105
9	14	1989	26192.0	.	5.58	2100.00	5.5	.	.	1480.00	384.00	98.00	106
10	16	1989	2566.0	.	5.88	1400.00	2.3	.	.	800.00	212.00	54.00	107
11	6	1989	745.0	.	5.50	1175.00	2.4	.	.	680.00	166.00	50.00	108
3	26	1990	742.0	.	5.05	600.00	3.8	.	.	290.00	64.00	21.40	100
4	12	1990	5984.0	.	5.82	975.00	1.1	.	.	685.00	148.00	42.00	101
5	17	1990	7127.0	.	5.72	1800.00	2.1	.	.	1250.00	286.00	84.00	102
6	18	1990	10465.0	.	5.86	1685.00	2.0	.	.	1040.00	202.00	70.00	103
7	16	1990	6972.0	.	5.79	1650.00	3.2	.	.	975.00	214.00	66.00	104
8	13	1990	1544.0	.	5.77	1575.00	1.6	.	.	920.00	228.00	66.00	105
9	10	1990	3978.0	.	5.42	1715.00	3.2	.	.	940.00	212.00	74.00	106
10	15	1990	22191.0	.	5.29	2600.00	2.7	.	.	1650.00	358.00	134.00	107
11	16	1990	1098.0	.	5.31	3875.00	1.9	.	.	2400.00	680.00	202.00	108
4	8	1991	4179.0	.	.	.	.	.	.	.	.	.	.
4	22	1991	8005.0	.	4.96	1650.00	2.5	.	.	1030.00	194.00	88.00	2019
5	20	1991	11983.0	.	5.18	2375.00	2.5	.	.	1200.00	262.00	107.00	2023
6	17	1991	11987.0	.	5.25	2050.00	2.0	.	.	1110.00	214.00	104.00	2039
7	15	1991	32415.0	.	5.27	2725.00	3.1	.	.	1260.00	278.00	124.00	2055
8	12	1991	3463.0	.	5.27	2550.00	2.4	.	.	1370.00	318.00	110.00	2071
9	9	1991	19792.0	.	5.22	2675.00	3.6	.	.	1420.00	268.00	110.00	2086
10	7	1991	7025.0	.	5.31	2575.00	3.6	.	.	1360.00	272.00	108.00	2102
11	7	1991	22131.0	.	5.24	2475.00	3.1	.	.	1270.00	252.00	112.00	2118
4	23	1992	1513.0	.	5.35	875.00	3.8	39.0	54.0	400.00	86.00	36.00	17000
5	11	1992	2724.0	.	5.08	1340.00	3.2	60.0	105.1	630.00	122.00	54.00	17013
5	26	1992	4053.0	.	5.18	1380.00	1.9	70.0	99.7	755.00	148.00	68.00	17041
6	8	1992	798.0	.	5.18	2470.00	3.8	100.0	140.3	1490.00	294.00	126.00	17045
6	22	1992	914.0	.	5.29	1600.00	2.6	63.0	101.9	920.00	172.00	76.00	17049
7	7	1992	3350.0	.	5.15	2030.00	2.1	85.0	145.5	430.00	222.00	96.00	17054
7	20	1992	1223.0	.	5.19	2250.00	3.1	90.0	119.6	1405.00	246.00	106.00	17058
8	3	1992	1118.0	.	5.19	1180.00	2.6	50.0	93.5	815.00	166.00	72.00	17074
8	17	1992	1354.0	.	5.11	1900.00	2.1	63.0	124.7	940.00	184.00	80.00	17078
8	31	1992	1875.0	.	5.17	1400.00	5.0	65.0	110.7	800.00	142.00	68.00	17094
9	14	1992	4141.0	.	5.21	1920.00	2.1	78.0	136.6	1290.00	256.00	110.00	17098
9	28	1992	2579.0	.	5.12	2300.00	3.0	80.0	127.3	1450.00	292.00	122.00	17115
10	12	1992	539.0	.	5.24	2800.00	3.0	81.0	138.2	1780.00	346.00	152.00	17119
10	27	1992	388.0	.	5.07	2300.00	2.6	75.0	125.5	2800.00	270.00	118.00	17135
11	9	1992	129.0	.	5.23	2850.00	3.2	.	.	.	.	.	.
11	30	1992	767.0	.	5.22	1500.00	2.7	95.0	112.4	880.00	166.00	80.00	17150
4	8	1993	3698.0	.	5.72	500.00	2.0	.	.	268.00	50.00	22.50	17162
4	22	1993	3550.0	.	5.06	880.00	2.5	.	.	468.00	77.50	40.00	17173
5	6	1993	10848.0	.	5.13	975.00	2.5	.	.	580.00	92.50	47.50	17178
5	20	1993	3588.0	.	5.24	1500.00	2.0	.	.	1095.00	177.50	82.50	17193
6	3	1993	6639.0	.	5.27	1720.00	3.0	.	.	1130.00	180.00	87.50	17197
6	17	1993	7366.0	.	5.11	1725.00	2.5	.	.	1375.00	242.50	117.50	17213
7	1	1993	8691.0	.	5.04	2100.00	2.0	.	.	1190.00	240.00	122.50	17217
7	14	1993	20482.0	.	5.15	2000.00	0.0	.	.	1260.00	234.00	100.00	17234
7	29	1993	13028.0	.	5.26	1350.00	3.5	.	.	895.00	152.00	68.00	17238
8	12	1993	21862.0	.	5.03	2050.00	3.5	.	.	1180.00	248.00	102.00	17254
8	26	1993	3777.0	.	.	.	.	.	.	.	.	.	.
9	9	1993	10886.0	.	5.23	1600.00	2.5	.	.	660.00	176.00	76.00	17274
9	23	1993	4315.0	.	5.20	1900.00	4.0	.	.	1140.00	254.00	98.00	17278
10	7	1993	2608.0	.	5.17	2210.00	2.5	.	.	1240.00	278.00	118.00	17294
10	20	1993	999.0	.	5.03	2800.00	2.0	.	.	1580.00	326.00	140.00	17298
11	4	1993	2112.0	.	5.12	2200.00	2.3	.	.	1185.00	242.00	110.00	17316
11	22	1993	519.0	.	5.03	2400.00	2.0	.	.	1060.00	250.00	116.00	17331

Table A2.5.2. Drainage quality data for metals from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
4	26	1978	113.00	13.00	.	0.032	.	0.003	0.017	.	0.060	.
6	6	1978	268.00	20.00	.	0.030	0.250	0.009	0.002	0.020	0.130	.
6	7	1978	327.00	23.00	.	0.041	0.272	0.005	0.030	0.030	0.150	.
6	16	1978	368.00	22.00	.	0.028	0.155	0.001	0.002	0.050	0.140	.
6	23	1978	288.30	21.30	.	0.025	0.251	0.012	0.003	0.020	0.090	.
6	26	1978	313.00	21.00	.	0.024	0.294	0.014	0.010	0.040	0.090	.
7	5	1978	206.00	28.00	.	0.018	0.206	0.007	.	0.010	0.090	.
7	7	1978	222.00	34.00	.	0.024	0.285	0.019	0.005	0.040	0.090	.
7	11	1978	286.00	38.00	.	0.027	0.388	0.026	0.010	0.030	0.140	.
7	13	1978	395.00	45.00	.	0.027	0.405	0.026	.	0.010	0.160	.
7	18	1978	220.00	34.00	.	0.023	0.255	0.019	0.003	0.010	0.110	.
8	2	1978	127.00	25.00	.	0.027	0.210	0.010	0.010	0.010	0.090	.
8	4	1978	209.00	37.00	.	0.034	0.330	0.016	0.013	0.030	0.130	.
8	10	1978	317.00	46.00	.	0.030	0.460	0.030	0.025	0.020	0.160	.
8	16	1978	102.00	25.00	.	0.020	0.185	0.020	0.015	0.020	0.070	.
8	24	1978	240.00	40.00	.	0.040	0.360	0.040	0.020	0.060	0.140	.
8	25	1978	229.00	43.00	.	0.040	0.350	0.030	0.020	0.020	0.140	.
8	28	1978	280.00	40.00	.	0.050	0.420	0.070	0.020	0.040	0.150	.
9	13	1978	160.00	39.50	.	0.045	0.275	0.020	0.010	0.030	0.120	.
10	17	1978	322.00	41.00	.	0.050	0.540	0.070	0.040	0.020	0.170	.
3	19	1979	43.00	13.00	.	0.030	0.330	0.040	0.040	0.020	0.080	.
4	13	1979	36.80	11.20	.	0.028	0.210	0.012	0.030	0.020	0.050	.
4	18	1979	27.00	6.00	.	0.023	0.106	0.008	0.020	0.020	0.040	.
4	19	1979	27.00	5.00	.	0.025	0.102	0.008	0.010	0.020	0.030	.
4	20	1979	55.00	7.20	.	0.025	0.138	0.080	0.020	0.020	0.050	.
4	22	1979	69.80	15.50	.	0.028	0.315	0.020	0.040	0.020	0.090	.
5	9	1979	60.60	14.80	.	0.025	0.250	0.015	0.020	0.020	0.080	.
5	10	1979	57.60	15.40	.	0.027	0.260	0.015	0.030	0.010	0.100	.
5	20	1979	75.60	16.20	.	0.031	0.320	0.020	0.030	0.030	0.100	.
5	22	1979	60.40	14.00	.	0.027	0.220	0.015	0.020	0.010	0.090	.
5	23	1979	118.40	21.40	.	0.031	0.340	0.023	0.030	0.020	0.130	.
6	1	1979	88.80	18.40	.	0.033	0.290	0.020	0.020	0.030	0.120	.
6	6	1979	53.00	21.20	.	0.028	0.260	0.017	0.010	0.010	0.090	.
6	10	1979	49.90	20.40	.	0.031	0.225	0.011	0.010	0.010	0.090	.
6	12	1979	67.00	32.80	.	0.028	0.280	0.013	0.020	0.010	0.120	.
6	15	1979	68.40	28.60	.	0.028	0.270	0.009	0.010	0.010	0.110	.
6	21	1979	60.00	19.00	.	0.020	0.170	0.012	0.010	0.040	0.070	.
6	30	1979	160.00	32.00	.	0.030	0.370	0.025	0.020	0.020	0.110	.
7	10	1979	164.00	37.00	.	0.030	0.380	0.025	0.020	0.010	0.150	.
7	30	1979	50.00	22.00	.	0.020	0.210	0.014	0.010	0.010	0.070	.
8	4	1979	106.00	41.00	.	0.020	0.390	0.025	0.020	0.020	0.120	.
8	13	1979	260.00	52.00	.	0.040	0.530	0.023	0.030	0.030	0.170	.
8	23	1979	174.00	38.60	.	0.030	0.430	0.023	0.030	0.020	0.130	.
8	28	1979	95.00	29.80	.	0.040	0.320	0.015	0.030	0.050	0.090	.
9	1	1979	80.20	33.00	.	0.040	0.450	0.020	0.030	0.030	0.100	.
9	5	1979	134.40	42.20	.	0.040	0.490	0.008	0.030	0.020	0.120	.
9	10	1979	135.00	35.90	.	0.035	0.415	0.019	0.030	0.020	0.100	.
9	13	1979	165.60	41.00	.	0.050	0.520	0.023	0.030	0.010	0.120	.
9	19	1979	232.00	53.20	.	0.050	0.660	0.060	0.040	0.030	0.120	.
10	18	1979	204.00	46.40	.	0.050	0.640	0.030	0.040	0.010	0.120	.
10	19	1979	56.00	20.80	.	0.040	0.460	0.021	0.030	0.010	0.050	.
10	31	1979	90.00	30.80	.	0.041	0.740	0.026	0.040	0.020	0.100	.
11	1	1979	114.00	34.20	.	0.030	0.820	0.060	0.050	0.020	0.100	.
4	17	1980	21.80	8.40	.	0.030	0.260	0.030	0.030	0.020	0.060	.
4	18	1980	28.60	10.80	.	0.020	0.040	0.020	0.050	0.020	0.050	.
4	23	1980	22.60	11.00	.	0.010	0.390	0.040	0.050	0.020	0.060	.
4	27	1980	58.50	17.00	.	0.020	0.505	0.040	0.040	0.030	0.120	.
5	5	1980	49.80	14.80	.	0.020	0.370	0.040	0.030	0.020	0.120	.
5	18	1980	71.40	20.00	.	0.030	0.400	0.040	0.030	0.010	0.150	.
5	31	1980	41.10	16.00	.	0.030	0.300	0.012	0.020	0.020	0.070	.
6	1	1980	57.50	23.60	.	0.060	0.520	0.022	0.030	0.010	0.100	.
6	7	1980	67.20	25.60	.	0.020	0.450	0.014	0.020	0.030	0.020	.
6	14	1980	70.00	26.00	.	0.030	0.460	0.029	0.020	0.060	0.100	.
6	18	1980	101.00	32.80	.	0.040	0.530	0.012	0.030	0.030	0.120	.
6	27	1980	72.00	28.00	.	0.043	0.039	0.036	0.030	0.010	0.080	.
7	2	1980	99.00	33.00	.	0.024	0.660	0.034	0.050	0.030	0.110	.

Table A2.5.2. Drainage quality data for metals from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
7	4	1980	65.00	27.00	.	0.030	0.670	0.040	0.040	0.020	0.010	.
7	17	1980	190.30	42.40	.	0.040	0.630	0.040	0.040	0.030	0.120	.
7	20	1980	169.00	37.80	.	0.040	0.550	0.040	0.040	0.020	0.100	.
8	3	1980	131.60	37.40	.	0.060	0.540	0.026	0.060	0.040	0.130	.
8	12	1980	57.40	25.20	.	0.050	0.600	0.080	0.040	0.090	0.100	.
8	19	1980	85.20	34.80	.	0.050	1.010	0.050	0.050	0.020	0.130	.
8	22	1980	76.80	30.00	.	0.032	1.220	0.060	0.050	0.040	0.110	.
8	29	1980	94.40	34.00	.	0.060	0.890	0.060	0.060	0.070	0.100	.
9	7	1980	80.60	30.00	.	0.035	0.660	0.040	0.050	0.040	0.080	.
9	9	1980	93.20	33.80	.	0.036	0.860	0.047	0.050	0.060	0.100	.
9	12	1980	55.60	22.40	.	0.038	0.680	0.050	0.040	0.020	0.080	.
9	18	1980	84.00	29.20	.	0.046	0.740	0.033	0.050	0.060	0.080	.
9	23	1980	122.00	10.80	.	0.050	0.840	0.070	0.060	0.040	0.080	.
10	15	1980	87.80	30.00	.	0.050	0.700	0.040	0.060	0.030	0.070	.
10	31	1980	65.00	29.00	.	0.070	1.590	0.070	0.090	0.020	0.100	.
2	18	1981	16.20	.	.	0.020	0.630	0.030	0.040	.	.	.
3	27	1981	15.80	.	.	0.020	0.710	0.040	0.050	.	.	.
4	3	1981	20.80	.	.	0.050	1.470	0.060	0.070	.	.	.
4	13	1981	22.00	.	.	0.090	1.290	0.060	0.060	.	.	.
4	17	1981	23.00	.	.	0.040	1.280	0.050	0.050	.	.	.
4	23	1981	30.00	.	.	0.040	1.260	0.060	0.060	.	.	.
4	27	1981	39.00	.	.	0.040	1.350	0.070	0.060	.	.	.
4	28	1981	36.00	.	.	0.040	1.070	0.060	0.060	.	.	.
4	29	1981	46.00	.	.	0.040	1.050	0.080	0.070	.	.	.
5	12	1981	68.00	.	.	0.030	0.830	0.040	0.070	.	.	.
6	1	1981	46.00	.	.	0.040	0.490	0.030	0.040	.	.	.
6	6	1981	56.00	22.00	.	0.040	0.880	0.050	0.050	.	.	.
6	22	1981	32.00	.	.	0.050	1.120	0.060	0.050	.	.	.
6	24	1981	30.00	.	.	0.060	1.280	0.070	0.050	.	.	.
6	28	1981	32.00	.	.	0.040	1.040	0.050	0.040	.	.	.
7	4	1981	65.60	.	.	0.040	1.100	0.060	0.050	.	.	.
7	15	1981	65.80	.	.	0.060	0.670	0.050	0.040	.	.	.
7	24	1981	48.20	.	.	0.070	0.800	0.060	0.050	.	.	.
8	4	1981	76.00	.	.	0.050	0.980	0.050	0.090	.	.	.
8	17	1981	65.00	.	.	0.060	1.220	0.080	0.080	.	.	.
9	1	1981	56.00	.	.	0.140	1.840	0.060	0.140	.	.	.
9	16	1981	75.00	.	.	0.140	2.700	0.150	0.200	.	.	.
9	26	1981	42.00	.	.	0.270	4.300	0.190	0.230	.	.	.
10	1	1981	32.00	.	.	0.390	6.500	0.290	0.300	.	.	.
10	4	1981	40.00	30.00	.	0.280	5.750	0.230	0.260	.	.	.
10	13	1981	40.00	26.00	.	0.140	2.980	0.140	0.160	.	.	.
10	17	1981	50.00	32.00	.	0.170	3.820	0.180	0.180	.	.	.
10	30	1981	108.00	46.00	.	0.120	2.040	0.090	0.110	.	.	.
3	24	1982	14.00	12.00	.	0.390	7.700	0.300	0.250	.	.	.
4	13	1982	18.00	10.00	.	0.160	3.940	0.160	0.160	.	.	.
4	14	1982	8.00	6.00	.	0.070	1.430	0.055	0.060	.	.	.
4	15	1982	12.00	8.00	.	0.110	2.220	0.090	0.090	.	.	.
4	19	1982	35.00	7.20	.	0.130	3.840	0.130	0.110	.	.	.
4	23	1982	20.00	14.00	.	0.110	3.420	0.140	0.110	.	.	.
4	28	1982	22.00	16.00	.	0.080	2.840	0.110	0.080	.	.	.
5	3	1982	16.00	10.00	.	0.100	1.610	0.090	0.070	.	.	.
5	7	1982	22.00	12.00	.	0.080	3.050	0.140	0.090	.	.	.
5	11	1982	24.00	20.00	.	0.080	4.150	0.150	0.120	.	.	.
5	14	1982	26.60	20.40	.	0.060	3.000	0.110	0.080	0.040	0.120	.
5	21	1982	48.70	25.80	.	0.055	2.835	0.105	0.090	0.040	0.140	.
6	2	1982	39.00	14.00	.	0.050	1.660	0.070	0.070	0.040	0.110	.
6	11	1982	23.00	18.00	.	0.050	3.245	0.125	0.090	0.030	0.140	.
7	2	1982	36.00	26.00	.	0.060	3.540	0.160	0.080	0.010	0.130	.
7	26	1982	44.00	28.00	.	0.080	2.520	0.110	0.070	0.010	0.110	.
8	2	1982	42.00	32.00	.	0.090	3.400	0.150	0.070	0.020	0.130	.
8	9	1982	62.00	36.00	.	0.100	5.170	0.220	0.090	0.020	0.130	.
8	23	1982	48.00	34.00	.	0.130	7.020	0.310	0.120	0.020	0.170	.
8	30	1982	44.00	30.00	.	0.120	5.940	0.270	0.120	0.300	0.170	.
9	8	1982	26.00	22.00	.	0.280	7.890	0.370	0.150	0.020	0.220	.
9	13	1982	32.00	28.00	.	0.190	7.350	0.280	0.140	0.020	0.190	.
9	20	1982	46.00	32.00	.	0.190	7.300	0.290	0.140	0.010	0.190	.

Table A2.5.2. Drainage quality data for metals from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
10	4	1982	34.00	26.00	.	0.180	6.170	0.300	0.110	0.020	0.170	.
10	11	1982	38.00	26.00	.	0.110	4.570	0.150	0.080	0.010	0.120	.
10	21	1982	38.00	26.00	.	0.110	4.390	0.170	0.080	0.190	0.120	.
10	25	1982	34.00	28.00	.	0.150	5.640	0.240	0.100	0.030	0.140	.
11	1	1982	20.00	14.00	.	0.120	4.660	0.210	0.070	0.020	0.100	.
11	22	1982	32.00	.	.	0.120	5.090	0.150	0.120	.	.	.
3	18	1983	.	.	.	0.300	5.650	0.240	0.180	.	.	.
4	14	1983	.	.	.	0.130	2.910	0.120	0.130	.	.	.
5	15	1983	.	.	.	0.070	3.260	0.140	0.110	.	.	.
6	16	1983	.	.	.	0.070	4.870	0.190	0.100	.	.	.
7	14	1983	.	.	.	0.260	7.470	0.280	0.230	.	.	.
8	14	1983	.	.	.	0.730	9.950	0.400	0.340	.	.	.
9	15	1983	.	.	.	0.780	10.220	0.460	0.350	.	.	.
10	13	1983	.	.	.	0.520	8.350	0.370	0.240	.	.	.
11	9	1983	.	.	.	0.570	6.470	0.290	0.220	.	.	.
11	30	1983	.	.	.	0.830	13.350	0.540	0.360	.	.	.
2	24	1984	9.60	.	.	0.730	12.000	0.500	0.300	.	.	100
3	15	1984	.	.	.	.	.	.	.	.	.	101
4	15	1984	8.20	.	.	0.390	4.850	0.210	0.200	.	.	102
5	16	1984	17.50	.	.	0.360	9.000	0.290	0.200	.	.	103
6	13	1984	27.70	.	.	0.590	12.400	0.430	0.290	.	.	104
7	14	1984	25.00	.	.	0.300	9.600	0.350	0.190	.	.	105
8	15	1984	27.20	.	.	0.870	24.000	0.880	0.380	.	.	106
9	12	1984	33.50	.	.	1.540	16.250	1.020	0.530	.	.	107
10	13	1984	31.50	.	.	3.300	27.900	1.790	0.790	.	.	108
11	14	1984	32.40	.	.	1.860	18.100	0.560	0.540	.	.	109
12	1	1984	25.00	.	.	5.400	23.300	1.030	0.780	.	.	110
12	18	1984	20.00	.	.	2.490	23.800	0.860	0.570	.	.	111
4	2	1985	5.60	.	.	0.470	3.090	0.120	0.120	.	.	100
4	16	1985	12.40	.	.	1.770	14.500	0.550	0.440	.	.	101
5	14	1985	19.30	.	.	0.670	12.300	0.450	0.270	.	.	102
6	14	1985	25.40	.	.	1.170	15.200	0.510	0.350	.	.	103
7	15	1985	28.50	.	.	0.840	12.900	0.480	0.310	.	.	104
8	13	1985	25.00	.	.	1.400	17.900	0.690	0.450	.	.	105
9	13	1985	26.00	.	.	1.750	16.100	0.630	0.450	.	.	106
10	15	1985	22.00	.	.	1.300	10.600	0.320	0.300	.	.	107
11	15	1985	24.00	.	.	0.800	7.250	0.220	0.220	.	.	108
3	23	1986	5.80	.	.	2.090	13.000	0.460	0.280	.	.	100
4	16	1986	9.60	.	.	2.400	17.800	0.580	0.380	.	.	101
5	14	1986	14.00	.	.	1.540	15.900	0.600	0.360	.	.	102
6	11	1986	24.00	.	.	1.270	14.100	0.520	0.330	.	.	103
7	12	1986	20.00	.	.	2.740	20.400	0.800	0.520	.	.	104
8	13	1986	26.00	.	.	3.580	23.100	0.900	0.430	.	.	105
9	14	1986	22.00	.	.	3.460	18.700	0.800	0.360	.	.	106
10	16	1986	26.00	.	.	2.720	19.800	0.700	0.490	.	.	107
11	16	1986	26.00	.	.	5.700	23.000	0.870	0.640	.	.	108
3	22	1987	8.00	.	.	3.600	19.800	0.540	0.390	.	.	100
4	15	1987	8.00	.	.	3.750	14.600	0.590	0.420	.	.	101
5	16	1987	20.00	.	.	4.900	21.000	0.800	0.800	.	.	102
6	16	1987	34.00	.	.	1.970	16.000	0.500	0.290	.	.	103
7	15	1987	28.00	.	.	4.800	20.500	0.800	0.560	.	.	104
8	15	1987	26.00	.	.	5.400	27.000	0.900	0.680	.	.	105
9	16	1987	26.00	.	.	6.930	29.000	0.920	0.700	.	.	106
10	13	1987	34.00	.	.	4.700	25.000	0.780	0.650	.	.	107
11	10	1987	30.00	.	.	5.900	26.000	0.980	0.750	.	.	108
3	28	1988	6.00	.	.	6.800	16.400	0.900	1.000	.	.	100
4	17	1988	8.00	.	.	9.500	33.000	1.400	1.500	.	.	101
5	12	1988	18.00	.	.	8.500	41.000	1.600	1.600	.	.	102
6	12	1988	18.00	.	.	5.100	33.000	1.200	1.300	.	.	103
7	15	1988	21.00	.	.	9.000	49.000	1.800	1.900	.	.	104
8	16	1988	26.00	.	.	5.600	38.500	1.100	0.800	.	.	105
9	16	1988	26.00	.	.	5.200	42.000	1.200	0.900	.	.	106
10	18	1988	20.00	.	.	8.350	36.500	1.600	1.750	.	.	107
11	15	1988	18.00	.	.	12.700	38.000	1.800	1.700	.	.	108
11	30	1988	.	.	.	.	.	.	.	.	.	109
4	7	1989	8.00	.	.	5.200	20.500	0.800	0.700	.	.	100

Table A2.5.2. Drainage quality data for metals from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
4	24	1989	6.00	.	.	4.000	18.000	0.800	0.800	.	.	101
5	16	1989	14.00	.	.	8.000	37.000	1.700	1.700	.	.	102
6	17	1989	28.00	.	.	8.000	46.000	1.800	1.200	.	.	103
7	17	1989	30.00	.	.	6.700	30.000	1.100	1.100	.	.	104
8	14	1989	20.00	.	.	7.500	41.000	1.400	1.200	.	.	105
9	14	1989	30.00	.	.	13.100	39.000	2.300	1.600	.	.	106
10	16	1989	22.00	.	.	5.900	25.000	1.900	1.000	.	.	107
11	6	1989	16.00	.	.	5.300	25.000	2.600	1.000	.	.	108
3	26	1990	7.40	.	.	10.200	19.000	10.200	1.010	.	.	100
4	12	1990	10.00	.	.	9.430	32.000	1.540	1.540	.	.	101
5	17	1990	20.00	.	.	10.000	39.000	1.910	1.780	.	.	102
6	18	1990	22.00	.	.	7.780	30.000	1.270	1.140	.	.	103
7	16	1990	20.00	.	.	7.820	27.000	1.250	1.190	.	.	104
8	13	1990	22.00	.	.	5.210	21.000	1.010	0.930	.	.	105
9	10	1990	18.00	.	.	20.000	56.000	2.820	2.270	.	.	106
10	15	1990	30.00	.	.	35.000	93.000	4.350	3.100	.	.	107
11	16	1990	54.00	.	.	29.000	88.000	4.040	3.550	.	.	108
4	8	1991	.	.	.	.	.	.	.	.	.	.
4	22	1991	16.00	.	.	19.900	68.000	2.600	1.800	.	.	2019
5	20	1991	20.00	.	.	20.300	68.000	2.400	1.500	.	.	2023
6	17	1991	20.00	.	.	27.000	71.000	3.000	1.600	.	.	2039
7	15	1991	26.00	.	.	21.000	59.000	2.400	1.500	.	.	2055
8	12	1991	34.00	.	.	18.000	54.000	2.300	1.300	.	.	2071
9	9	1991	28.00	.	.	30.000	84.000	3.400	2.000	.	.	2086
10	7	1991	32.00	.	.	21.000	67.000	2.700	1.700	.	.	2102
11	7	1991	30.00	.	.	27.000	80.000	3.200	2.100	.	.	2118
4	23	1992	10.60	10.50	<1.00	8.600	24.000	1.000	0.800	0.030	1.190	17000
5	11	1992	13.40	12.00	<1.00	14.900	46.000	1.900	1.200	0.020	2.060	17013
5	26	1992	18.80	18.20	<1.00	16.500	42.000	1.600	1.000	0.030	1.880	17041
6	8	1992	31.80	31.00	<1.00	20.000	61.000	2.300	1.400	0.040	2.380	17045
6	22	1992	21.80	23.00	<1.00	16.700	43.000	1.600	1.100	0.030	1.870	17049
7	7	1992	26.60	28.00	<1.00	24.000	60.000	2.500	1.400	0.040	2.530	17054
7	20	1992	29.80	30.00	<1.00	19.600	50.000	2.100	1.200	0.040	2.020	17058
8	3	1992	23.00	23.00	<1.00	15.900	39.000	1.700	1.000	0.020	1.720	17074
8	17	1992	22.00	25.00	<1.00	21.000	49.000	3.200	2.600	0.100	2.250	17078
8	31	1992	17.80	23.00	<1.00	18.500	46.000	1.900	1.100	0.100	2.010	17094
9	14	1992	27.80	31.00	<1.00	24.000	55.000	2.400	1.400	0.010	2.410	17098
9	28	1992	33.00	34.00	<1.00	20.000	54.000	2.200	1.200	0.020	2.090	17115
10	12	1992	40.80	37.00	<1.00	21.000	59.000	2.400	1.400	0.010	2.160	17119
10	27	1992	31.40	31.00	<1.00	17.700	55.000	2.200	1.200	0.010	2.130	17135
11	9	1992	.	.	.	.	.	.	.	.	.	.
11	30	1992	19.20	22.00	<1.00	18.300	47.000	2.000	1.200	0.020	2.060	17150
4	8	1993	6.00	6.00	<1.00	5.870	14.000	0.520	0.390	0.010	0.630	17162
4	22	1993	7.50	8.00	<1.00	10.200	30.000	1.320	0.730	0.030	1.250	17173
5	6	1993	10.30	9.00	<1.00	13.000	39.000	1.700	0.870	0.050	1.580	17178
5	20	1993	14.80	16.00	<1.00	16.000	57.000	2.200	1.050	0.040	1.970	17193
6	3	1993	17.50	16.80	<1.00	25.000	66.000	2.500	1.350	0.050	2.240	17197
6	17	1993	21.80	21.00	<1.00	31.000	83.000	3.170	1.640	0.030	2.770	17213
7	1	1993	22.50	23.50	<1.00	33.000	80.000	3.070	1.580	0.040	2.640	17217
7	14	1993	22.40	26.40	<1.00	18.000	53.000	1.930	1.050	0.020	2.070	17234
7	29	1993	15.00	19.00	<1.00	16.000	42.000	1.570	0.890	0.020	1.690	17238
8	12	1993	22.20	27.60	<1.00	21.000	57.000	1.980	1.220	0.020	2.300	17254
8	26	1993	.	.	.	.	.	.	.	.	.	.
9	9	1993	17.20	21.20	<1.00	18.000	44.000	1.690	1.030	0.010	1.950	17274
9	23	1993	22.80	27.20	<1.00	20.000	52.000	1.940	1.200	0.030	2.280	17278
10	7	1993	26.00	29.60	<1.00	23.000	64.000	2.190	1.420	0.020	2.640	17294
10	20	1993	29.40	33.20	<1.00	25.000	72.000	2.600	1.410	0.020	2.670	17298
11	4	1993	22.20	24.40	<1.00	25.000	68.000	2.670	1.330	0.530	2.620	17316
11	22	1993	24.60	24.20	<1.00	26.000	65.000	2.670	1.460	0.040	2.510	17331

Table A2.5.3. Drainage quality data for additional parameters from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
4	26	1978	.	.	.	.	.	9.0	.
6	6	1978	.	.	.	.	.	7.0	.
6	7	1978	.	.	.	.	.	.	.
6	16	1978	.	.	.	.	.	11.0	.
6	23	1978	5.00	.	.	.	.	13.0	.
6	26	1978	.	.	.	.	.	15.0	.
7	5	1978	4.00	.	.	.	.	14.0	.
7	7	1978	4.00	5.70	0.44	0.08	.	19.0	.
7	11	1978	5.00	5.70	0.64	0.08	.	.	.
7	13	1978	4.00	6.20	0.55	0.08	.	.	.
7	18	1978	1.00	3.90	0.50	0.08	.	.	.
8	2	1978	1.00	3.40	0.70	0.08	.	22.0	.
8	4	1978	3.00	5.00	0.06	0.10	.	22.0	.
8	10	1978	4.00	.	0.10	0.08	.	24.0	.
8	16	1978	2.00	2.50	.	0.08	.	19.5	.
8	24	1978	1.00	5.00	.	0.08	.	.	.
8	25	1978	1.00	4.60	0.40	0.20	.	22.0	.
8	28	1978	.	.	.	.	.	.	.
9	13	1978	1.50	.	.	0.08	.	14.0	.
10	17	1978	8.00	6.50	0.64	0.20	.	.	.
3	19	1979	3.00	.	.	.	2.90	.	.
4	13	1979	4.00	.	.	.	3.60	.	.
4	18	1979	1.00	1.30	.	.	.	.	.
4	19	1979	2.00	0.80	0.12	.	.	.	.
4	20	1979	2.00	1.00	0.10	.	3.10	.	.
4	22	1979	4.00	2.20	0.23	.	.	.	.
5	9	1979	4.00	.	.	.	.	.	.
5	10	1979	3.00	.	.	.	.	.	.
5	20	1979	3.00	.	.	.	6.90	.	.
5	22	1979	1.00	.	.	.	6.00	.	.
5	23	1979	3.00	.	.	.	5.00	.	.
6	1	1979	2.00	.	.	.	6.90	.	.
6	6	1979	3.00	.	.	.	.	.	.
6	10	1979	3.00	.	.	.	.	20.0	.
6	12	1979	4.00	.	.	.	.	.	.
6	15	1979	5.00	.	.	.	.	20.0	.
6	21	1979	2.00	.	.	.	.	15.0	.
6	30	1979	4.00	.	.	.	12.00	19.0	.
7	10	1979	4.00	.	.	.	11.00	18.0	.
7	30	1979	3.00	.	.	.	6.30	23.0	.
8	4	1979	3.00	.	.	.	6.10	23.0	.
8	13	1979	5.00	.	.	.	11.00	13.0	.
8	23	1979	5.00	.	.	.	9.10	9.0	.
8	28	1979	28.00	.	.	.	8.30	20.0	.
9	1	1979	5.00	.	.	.	7.80	22.0	.
9	5	1979	4.00	.	.	.	10.00	18.0	.
9	10	1979	5.00	.	.	.	8.90	17.5	.
9	13	1979	5.00	.	.	.	9.30	16.0	.
9	19	1979	8.00	.	.	.	12.00	11.5	.
10	18	1979	11.00	.	.	.	11.00	.	.
10	19	1979	3.00	.	.	.	5.90	.	.
10	31	1979	3.00	.	.	.	7.20	.	.
11	1	1979	6.00	.	.	.	8.50	10.0	.
4	17	1980	1.00	.	.	.	.	.	.
4	18	1980	2.00	.	.	.	2.40	.	.
4	23	1980	2.00	.	.	.	2.20	.	.
4	27	1980	3.00	.	.	.	.	12.5	.
5	5	1980	3.00	.	.	.	4.50	11.0	.
5	18	1980	9.00	.	.	.	.	19.0	.
5	31	1980	5.00	.	.	.	6.30	.	.
6	1	1980	5.00	.	.	.	6.20	.	.
6	7	1980	5.00	.	.	.	.	.	.
6	14	1980	5.00	.	.	.	.	.	.
6	18	1980	6.00	.	.	.	.	.	.
6	27	1980	5.00	.	.	.	8.20	.	.
7	2	1980	5.00	.	.	.	8.80	.	.

Table A2.5.3. Drainage quality data for additional parameters from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
7	4	1980	.	.	.	.	.	.	.
7	17	1980	5.50	.	.	.	.	.	.
7	20	1980	6.00	.	.	.	.	.	.
8	3	1980	6.00	.	.	.	10.00	.	.
8	12	1980	5.00	.	.	.	6.10	.	.
8	19	1980	5.00	.	.	.	.	.	.
8	22	1980	4.00	.	.	.	.	.	.
8	29	1980	4.00	.	.	.	7.50	.	.
9	7	1980	4.00	.	.	.	.	.	.
9	9	1980	3.00	.	.	.	.	14.0	.
9	12	1980	4.00	.	.	.	4.90	16.0	.
9	18	1980	4.00	.	.	.	.	12.0	.
9	23	1980	4.00	.	.	.	6.90	14.0	.
10	15	1980	5.00	.	.	.	7.00	8.0	.
10	31	1980	4.00	.	.	.	.	11.0	.
2	18	1981	1.00	.	.	.	.	8.0	.
3	27	1981	.	.	.	.	3.00	.	.
4	3	1981	.	.	.	.	.	.	.
4	13	1981	.	.	.	.	.	.	.
4	17	1981	.	.	.	.	3.60	.	.
4	23	1981	.	.	.	.	.	.	.
4	27	1981	.	.	.	.	.	2.0	.
4	28	1981	.	.	.	.	4.30	.	.
4	29	1981	.	.	.	.	.	.	.
5	12	1981	.	.	.	.	6.20	17.0	.
6	1	1981	.	.	.	.	.	16.0	.
6	6	1981	.	.	.	.	3.60	18.0	.
6	22	1981	.	.	.	.	.	.	.
6	24	1981	.	.	.	.	.	16.0	.
6	28	1981	.	.	.	.	3.00	15.0	.
7	4	1981	.	.	.	.	.	21.0	.
7	15	1981	.	.	.	.	.	19.0	.
7	24	1981	2.00	.	.	.	3.70	19.0	.
8	4	1981	.	.	.	.	.	18.0	.
8	17	1981	.	.	.	.	.	22.0	.
9	1	1981	.	.	.	.	.	.	.
9	16	1981	.	.	.	.	.	.	.
9	26	1981	.	.	.	.	6.70	.	.
10	1	1981	.	.	.	.	.	.	.
10	4	1981	.	.	.	.	.	.	.
10	13	1981	.	.	.	.	3.50	.	.
10	17	1981	.	.	.	.	.	.	.
10	30	1981	.	.	.	.	6.20	.	.
3	24	1982	.	.	.	.	.	.	.
4	13	1982	.	.	.	.	.	.	.
4	14	1982	.	.	.	.	.	.	.
4	15	1982	.	.	.	.	.	.	.
4	19	1982	.	.	.	.	.	.	.
4	23	1982	.	.	.	.	.	.	.
4	28	1982	.	.	.	.	.	.	.
5	3	1982	.	.	.	.	.	.	.
5	7	1982	.	.	.	.	.	.	.
5	11	1982	.	.	.	.	.	.	.
5	14	1982	.	.	.	.	.	.	.
5	21	1982	.	.	.	.	.	.	.
6	2	1982	.	.	.	.	.	.	.
6	11	1982	.	.	.	.	.	.	.
7	2	1982	.	.	.	.	.	.	.
7	26	1982	.	.	.	.	.	.	.
8	2	1982	.	.	.	.	.	.	.
8	9	1982	.	.	.	.	.	.	.
8	23	1982	.	.	.	.	.	.	.
8	30	1982	.	.	.	.	.	.	.
9	8	1982	.	.	.	.	.	.	.
9	13	1982	.	.	.	.	.	.	.
9	20	1982	.	.	.	.	.	.	.

Table A2.5.3. Drainage quality data for additional parameters from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
10	4	1982	.	.	.	.	.	.	.
10	11	1982	.	.	.	.	.	.	.
10	21	1982	.	.	.	.	.	.	.
10	25	1982	.	.	.	.	.	.	.
11	1	1982	.	.	.	.	.	.	.
11	22	1982	.	.	.	.	.	.	.
3	18	1983	.	.	.	.	.	.	.
4	14	1983	.	.	.	.	.	.	.
5	15	1983	.	.	.	.	.	.	.
6	16	1983	.	.	.	.	.	.	.
7	14	1983	.	.	.	.	.	.	.
8	14	1983	.	.	.	.	.	.	.
9	15	1983	.	.	.	.	.	.	.
10	13	1983	.	.	.	.	.	.	.
11	9	1983	.	.	.	.	.	.	.
11	30	1983	.	.	.	.	.	.	.
2	24	1984	.	.	.	.	.	.	100
3	15	1984	.	.	.	.	.	.	101
4	15	1984	.	.	.	.	.	.	102
5	16	1984	.	.	.	.	.	.	103
6	13	1984	.	.	.	.	.	.	104
7	14	1984	.	.	.	.	.	.	105
8	15	1984	.	.	.	.	.	.	106
9	12	1984	.	.	.	.	.	.	107
10	13	1984	.	.	.	.	.	.	108
11	14	1984	.	.	.	.	.	.	109
12	1	1984	.	.	.	.	.	.	110
12	18	1984	.	.	.	.	.	.	111
4	2	1985	.	.	.	.	.	.	100
4	16	1985	.	.	.	.	.	.	101
5	14	1985	.	.	.	.	.	.	102
6	14	1985	.	.	.	.	.	.	103
7	15	1985	.	.	.	.	.	.	104
8	13	1985	.	.	.	.	.	.	105
9	13	1985	.	.	.	.	.	.	106
10	15	1985	.	.	.	.	.	.	107
11	15	1985	.	.	.	.	.	.	108
3	23	1986	.	.	.	.	.	.	100
4	16	1986	.	.	.	.	.	.	101
5	14	1986	.	.	.	.	.	.	102
6	11	1986	.	.	.	.	.	.	103
7	12	1986	.	.	.	.	.	.	104
8	13	1986	.	.	.	.	.	.	105
9	14	1986	.	.	.	.	.	.	106
10	16	1986	.	.	.	.	.	.	107
11	16	1986	.	.	.	.	.	.	108
3	22	1987	.	.	.	.	.	.	100
4	15	1987	.	.	.	.	.	.	101
5	16	1987	.	.	.	.	.	.	102
6	16	1987	.	.	.	.	.	.	103
7	15	1987	.	.	.	.	.	.	104
8	15	1987	.	.	.	.	.	.	105
9	16	1987	.	.	.	.	.	.	106
10	13	1987	.	.	.	.	.	.	107
11	10	1987	.	.	.	.	.	.	108
3	28	1988	.	.	.	.	.	.	100
4	17	1988	.	.	.	.	.	.	101
5	12	1988	.	.	.	.	.	.	102
6	12	1988	.	.	.	.	.	.	103
7	15	1988	.	.	.	.	.	.	104
8	16	1988	.	.	.	.	.	.	105
9	16	1988	.	.	.	.	.	.	106
10	18	1988	.	.	.	.	.	.	107
11	15	1988	.	.	.	.	.	.	108
11	30	1988	.	.	.	.	.	.	109
4	7	1989	.	.	.	.	.	.	100

Table A2.5.3. Drainage quality data for additional parameters from Duluth Complex test pile FL1, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
4	24	1989	.	.	.	.	.	.	101
5	16	1989	.	.	.	.	.	.	102
6	17	1989	.	.	.	.	.	.	103
7	17	1989	.	.	.	.	.	.	104
8	14	1989	.	.	.	.	.	.	105
9	14	1989	.	.	.	.	.	.	106
10	16	1989	.	.	.	.	.	.	107
11	6	1989	.	.	.	.	.	.	108
3	26	1990	.	.	.	.	.	.	100
4	12	1990	.	.	.	.	.	.	101
5	17	1990	.	.	.	.	.	.	102
6	18	1990	.	.	.	.	.	.	103
7	16	1990	.	.	.	.	.	.	104
8	13	1990	.	.	.	.	.	.	105
9	10	1990	.	.	.	.	.	.	106
10	15	1990	.	.	.	.	.	.	107
11	16	1990	.	.	.	.	.	.	108
4	8	1991	.	.	.	.	.	.	.
4	22	1991	.	.	.	.	.	.	2019
5	20	1991	.	.	.	.	.	.	2023
6	17	1991	.	.	.	.	.	.	2039
7	15	1991	.	.	.	.	.	.	2055
8	12	1991	.	.	.	.	.	.	2071
9	9	1991	.	.	.	.	.	.	2086
10	7	1991	.	.	.	.	.	.	2102
11	7	1991	.	.	.	.	.	.	2118
4	23	1992	.	.	.	.	.	.	17000
5	11	1992	.	.	.	.	.	.	17013
5	26	1992	.	.	.	.	.	.	17041
6	8	1992	.	.	.	.	.	.	17045
6	22	1992	.	.	.	.	.	.	17049
7	7	1992	.	.	.	.	.	.	17054
7	20	1992	.	.	.	.	.	.	17058
8	3	1992	.	.	.	.	.	.	17074
8	17	1992	.	.	.	.	.	.	17078
8	31	1992	.	.	.	.	.	.	17094
9	14	1992	.	.	.	.	.	.	17098
9	28	1992	.	.	.	.	.	.	17115
10	12	1992	.	.	.	.	.	.	17119
10	27	1992	.	.	.	.	.	.	17135
11	9	1992	.	.	.	.	.	.	.
11	30	1992	.	.	.	.	.	.	17150
4	8	1993	.	.	.	.	.	.	17162
4	22	1993	.	.	.	.	.	.	17173
5	6	1993	.	.	.	.	.	.	17178
5	20	1993	.	.	.	.	.	.	17193
6	3	1993	.	.	.	.	.	.	17197
6	17	1993	.	.	.	.	.	.	17213
7	1	1993	.	.	.	.	.	.	17217
7	14	1993	.	.	.	.	.	.	17234
7	29	1993	.	.	.	.	.	.	17238
8	12	1993	.	.	.	.	.	.	17254
8	26	1993	.	.	.	.	.	.	.
9	9	1993	.	.	.	.	.	.	17274
9	23	1993	.	.	.	.	.	.	17278
10	7	1993	.	.	.	.	.	.	17294
10	20	1993	.	.	.	.	.	.	17298
11	4	1993	.	.	.	.	.	.	17316
11	22	1993	.	.	.	.	.	.	17331

Table A2.5.4. Drainage quality data from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
4	26	1978	.	.	7.20	1000.00	.	.	.	.	99.00	13.00	.
5	25	1978	.	.	7.70	1900.00	.	.	.	.	177.00	23.00	.
6	16	1978	496.0	.	7.72	4200.00	.	.	.	.	323.00	47.00	.
7	5	1978	293.0	.	7.50	2700.00	.	.	.	1400.00	218.00	33.00	.
7	13	1978	.	.	7.92	.	.	.	.	800.00	491.00	43.00	.
7	19	1978	.	.	7.80	1300.00	.	.	.	800.00	291.00	24.00	.
8	2	1978	.	.	7.75	2000.00	.	.	.	1100.00	191.00	21.00	.
8	4	1978	1000.0	.	7.75	2500.00	.	.	.	1700.00	341.00	42.00	.
8	16	1978	30.0	.	7.58	1091.67	39.5	.	.	500.00	108.00	14.30	.
8	23	1978	6379.0	.	7.60	1100.00	31.0	.	.	460.00	103.00	15.00	.
8	25	1978	.	.	7.30	1600.00	35.0	.	.	680.00	170.00	24.00	.
8	28	1978	.	.	7.55	2300.00	48.0	.	.	1100.00	253.00	34.00	.
9	13	1978	451.0	.	7.25	1337.50	29.0	.	.	583.50	133.00	16.00	.
3	19	1979	2611.7	1891.0	7.70	1043.00	.	.	.	.	83.20	11.00	.
3	27	1979	53.0	.	.	.	.	.	.	.	205.00	28.40	.
4	17	1979	5151.0	3480.0	6.90	388.00	22.0	.	.	136.00	74.20	5.80	.
4	18	1979	6056.0	3654.0	6.90	176.50	16.5	.	.	51.00	27.50	2.60	.
4	20	1979	3263.0	3830.0	6.50	296.00	21.0	.	.	100.00	49.00	5.60	.
4	22	1979	5019.5	1296.0	7.50	1163.00	18.0	.	.	633.00	175.40	24.20	.
5	9	1979	825.2	1083.0	7.40	1217.00	.	.	.	500.00	154.60	22.00	.
5	10	1979	6810.0	3129.0	7.40	1009.00	.	.	.	400.00	144.40	20.00	.
5	20	1979	3611.3	1034.0	6.90	1562.00	.	.	.	800.00	233.40	32.60	.
5	22	1979	1639.1	1041.0	7.40	1128.00	.	.	.	567.00	165.80	22.80	.
5	23	1979	2286.4	609.0	7.60	1976.00	.	.	.	.	286.00	39.20	.
6	2	1979	1518.0	474.0	7.60	1683.00	.	.	.	883.00	217.20	31.80	.
6	6	1979	1998.7	531.0	7.80	1749.00	.	.	.	900.00	220.80	33.80	.
6	10	1979	2021.4	1211.0	7.60	1414.00	45.0	.	.	580.00	173.40	27.80	.
6	12	1979	1033.4	361.0	7.00	2317.00	43.0	.	.	1200.00	277.00	44.60	.
6	16	1979	1343.8	368.0	7.70	2365.00	51.0	.	.	1350.00	307.40	49.10	.
6	20	1979	2199.5	1011.0	7.10	1363.00	36.0	.	.	570.00	158.00	27.00	.
6	30	1979	1131.8	170.0	7.60	2536.00	59.0	.	.	1450.00	336.00	55.00	.
7	10	1979	1249.2	142.0	7.70	3294.00	45.0	.	.	2300.00	420.00	72.00	.
7	30	1979	2778.5	2096.0	7.10	980.00	17.8	.	.	480.00	106.00	18.00	.
8	4	1979	442.9	65.0	7.70	1826.00	30.8	.	.	1020.00	253.00	36.00	.
8	13	1979	90.8	11.0	.	.	.	.	.	.	385.60	66.60	.
8	23	1979	113.6	70.0	.	.	.	.	.	.	217.20	33.20	.
8	28	1979	325.5	293.0	7.20	1338.00	13.6	.	.	.	157.00	25.20	.
9	1	1979	1525.5	905.0	7.10	1087.00	18.4	.	.	650.00	142.80	19.80	.
9	4	1979	1775.4	179.0	7.20	1380.00	20.8	.	.	.	211.40	29.00	.
9	10	1979	1268.1	249.0	7.20	1134.00	15.2	.	.	600.00	165.60	22.00	.
9	13	1979	367.2	211.0	7.30	1587.00	17.6	.	.	.	233.80	31.60	.
9	19	1979	128.7	22.0	7.30	2592.00	20.8	.	.	.	379.00	53.40	.
10	19	1979	1726.1	2456.0	6.60	914.00	12.8	.	.	541.50	112.50	16.10	.
10	31	1979	2835.3	1118.0	6.70	1319.00	13.6	.	.	817.00	194.00	25.20	.
11	1	1979	1215.0	637.0	7.20	1659.00	17.6	.	.	975.00	280.00	34.20	.
4	1	1980	1581.7	382.0	7.10	350.00	17.3	.	.	135.00	33.20	4.60	.
4	16	1980	855.5	831.0	7.50	600.00	35.0	.	.	225.00	68.20	8.20	.
4	17	1980	1892.7	774.0	7.20	.	33.5	.	.	253.00	69.00	9.00	.
4	29	1980	1116.7	163.0	7.57	1105.00	41.5	.	.	820.00	157.10	19.80	.
5	8	1980	541.3	51.0	7.40	1230.00	34.1	.	.	880.00	190.00	24.20	.
5	18	1980	371.0	26.0	7.50	1500.00	30.8	.	.	1050.00	236.00	31.20	.
5	31	1980	1703.4	1084.0	6.90	875.00	16.5	.	.	620.00	129.40	17.20	.
6	3	1980	174.1	50.0	7.30	1400.00	25.3	.	.	700.00	207.40	25.20	.
6	6	1980	371.0	235.0	7.20	1300.00	23.1	.	.	760.00	174.60	22.20	.
6	13	1980	265.0	197.0	7.10	1200.00	20.9	.	.	750.00	148.80	21.40	.
6	19	1980	22.7	12.0	7.20	1650.00	23.1	.	.	950.00	221.00	28.00	.
6	30	1980	113.6	38.0	7.10	1150.00	17.6	.	.	660.00	158.00	20.00	.
7	2	1980	246.1	96.0	7.10	950.00	15.3	.	.	580.00	132.00	16.00	.
7	17	1980	94.6	124.0	6.90	1150.00	17.6	.	.	450.00	155.60	19.00	.
7	23	1980	94.6	525.0	7.00	1100.00	17.6	.	.	350.00	149.80	18.60	.
8	3	1980	106.0	95.0	6.60	725.00	12.1	.	.	355.00	91.40	12.20	.
8	12	1980	1025.8	525.0	6.60	600.00	9.8	.	.	258.00	70.90	9.30	.
8	19	1980	378.5	106.0	6.80	700.00	10.2	.	.	424.00	103.60	13.00	.
8	22	1980	427.8	329.0	.	600.00	.	.	.	280.00	77.10	8.10	.
8	29	1980	2271.2	1473.0	6.70	1450.00	15.3	.	.	920.00	304.00	36.40	.

Table A2.5.4. Drainage quality data from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
9	7	1980	1449.8	1000.0	6.60	843.00	10.2	.	.	360.00	151.40	15.80	.
9	10	1980	344.5	88.0	6.80	1238.00	11.5	.	.	700.00	252.20	30.60	.
9	12	1980	2309.1	1578.0	6.70	969.00	12.0	.	.	450.00	188.40	19.60	.
9	18	1980	5329.9	1938.0	6.90	1231.00	12.3	.	.	690.00	241.80	29.80	.
9	23	1980	1820.8	612.0	6.90	2527.00	12.8	.	.	1700.00	398.00	72.80	.
10	22	1980	1627.7	322.0	6.70	2664.00	13.6	.	.	1620.00	404.00	79.00	.
10	23	1980	794.9	268.0	6.40	1763.00	8.5	.	.	950.00	214.00	32.00	.
2	18	1981	1760.0	1300.0	6.70	408.00	8.5	.	.	160.00	31.80	6.00	.
3	29	1981	6719.0	1726.0	6.70	330.00	8.8	.	.	90.00	31.20	6.20	.
3	30	1981	4236.0	1700.0	6.80	525.00	12.8	.	.	240.00	63.70	10.90	.
4	13	1981	2142.0	1281.0	.	800.00	.	.	.	250.00	69.00	12.00	.
4	16	1981	1427.0	212.0	7.10	800.00	14.0	.	.	315.00	78.00	15.00	.
4	23	1981	2423.0	2478.0	7.10	900.00	13.6	.	.	300.00	87.00	17.00	.
4	25	1981	4857.0	1589.0	7.10	1155.00	19.2	.	.	530.00	129.00	30.00	.
4	27	1981	3869.0	2091.0	7.30	1250.00	24.0	.	.	600.00	136.00	35.00	.
4	30	1981	3206.0	994.0	7.30	1596.00	24.0	.	.	870.00	210.00	40.00	.
5	12	1981	3539.0	203.0	7.20	2179.00	23.0	.	.	1250.00	306.00	54.00	.
6	1	1981	2370.0	705.0	7.25	2116.00	14.0	.	.	1250.00	266.00	54.00	.
6	6	1981	1522.0	150.0	7.20	1959.00	18.2	.	.	1000.00	234.00	44.00	.
6	22	1981	2411.0	1030.0	6.80	1270.00	11.2	.	.	625.00	144.00	26.00	.
6	24	1981	1874.0	993.0	6.90	793.00	12.0	.	.	870.00	96.00	14.00	.
6	28	1981	2612.0	1938.0	6.80	718.00	11.2	.	.	160.00	92.00	14.00	.
7	4	1981	1037.0	53.0	7.10	1630.00	13.5	.	.	1060.00	254.00	36.60	.
7	15	1981	575.0	26.0	7.00	2769.00	11.7	.	.	1800.00	388.00	68.40	.
7	24	1981	492.0	122.0	7.10	2142.00	10.5	.	.	1320.00	.	.	.
8	4	1981	246.0	25.0	6.80	2864.00	8.4	.	.	2000.00	364.00	80.00	.
8	13	1981	64.0	6.0	6.90	1926.00	12.2	.	.	1250.00	216.00	42.00	.
8	31	1981	454.0	382.0	6.00	1700.00	8.0	.	.	800.00	162.00	32.00	.
9	7	1981	284.0	255.0	6.60	1800.00	6.8	.	.	720.00	154.00	29.00	.
9	26	1981	768.0	530.0	6.40	1093.00	4.8	.	.	490.00	108.00	20.00	.
10	1	1981	1968.0	1257.0	6.60	750.00	4.8	.	.	260.00	90.00	16.00	.
10	4	1981	852.0	1060.0	6.45	825.00	4.4	.	.	380.00	118.00	16.00	.
10	13	1981	3937.0	1603.0	6.65	950.00	5.0	.	.	400.00	132.00	20.00	.
10	17	1981	3896.0	810.0	6.60	1600.00	5.6	.	.	870.00	238.00	36.00	.
11	5	1981	2839.0	99.0	6.55	3800.00	6.4	.	.	2400.00	470.00	100.00	.
4	24	1982	3691.0	.	7.00	620.00	14.4	.	.	290.00	98.00	10.00	.
4	26	1982	4047.0	.	6.80	870.00	11.5	.	.	420.00	93.00	16.00	.
5	3	1982	969.0	.	7.20	1000.00	10.8	.	.	520.00	128.00	22.00	.
5	7	1982	4789.0	.	6.70	780.00	9.0	.	.	370.00	82.00	14.00	.
5	11	1982	3937.0	.	6.80	1150.00	12.2	.	.	610.00	140.00	30.00	.
5	14	1982	2623.0	.	6.80	1450.00	13.5	.	.	730.00	174.00	25.40	.
5	21	1982	2248.0	.	6.90	2500.00	15.3	.	.	1500.00	336.00	63.00	.
6	2	1982	1787.0	.	6.65	2100.00	12.6	.	.	1300.00	246.00	49.00	.
6	18	1982	390.0	.	6.60	1775.00	11.7	.	.	940.00	194.00	37.00	.
7	2	1982	1098.0	.	6.50	950.00	8.1	.	.	480.00	116.00	20.00	.
7	8	1982	3373.0	.	6.50	1100.00	9.0	.	.	600.00	190.00	26.00	.
7	12	1982	4119.0	.	.	.	.	.	.	1015.00	326.00	56.00	.
7	26	1982	6568.0	.	6.65	2171.00	9.7	.	.	1030.00	286.00	52.00	.
8	30	1982	5765.0	.	6.47	1356.00	6.1	.	.	800.00	234.00	34.00	.
10	4	1982	4096.0	.	6.80	2888.00	8.8	.	.	1600.00	340.00	80.00	.
11	1	1982	1643.0	.	6.45	1662.00	5.2	.	.	.	199.00	32.00	.
11	22	1982	.	.	.	.	.	.	.	.	296.00	49.00	.
4	14	1983	9591.0	.	6.57	522.00	7.0	.	.	220.00	.	.	.
5	15	1983	3826.0	.	6.62	1115.00	7.3	.	.	580.00	.	.	.
6	16	1983	1778.0	.	6.48	1171.00	6.2	.	.	660.00	.	.	.
7	14	1983	3993.0	.	6.45	1164.00	5.2	.	.	660.00	.	.	.
8	14	1983	6223.0	.	6.08	1064.00	3.9	.	.	600.00	.	.	.
9	15	1983	5095.0	.	6.25	1200.00	3.7	.	.	730.00	.	.	.
10	13	1983	16559.0	.	6.39	1739.00	5.0	.	.	1340.00	.	.	.
11	9	1983	5053.0	.	6.25	1025.00	2.7	.	.	1380.00	.	.	.
11	30	1983	276.0	.	6.10	1325.00	2.7	.	.	940.00	.	.	.
4	15	1984	7737.0	.	6.59	936.00	9.9	.	.	700.00	198.30	37.50	202
5	16	1984	7835.0	.	6.54	641.00	4.9	.	.	400.00	130.10	24.20	203
6	13	1984	11065.0	.	6.39	950.00	4.9	.	.	680.00	172.70	35.60	204
7	14	1984	2710.0	.	6.36	1460.00	6.1	.	.	1080.00	.	.	205

Table A2.5.4. Drainage quality data from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
8	15	1984	2172.0	.	6.35	1260.00	4.8	.	.	700.00	195.40	58.60	206
9	12	1984	1086.0	.	5.99	1150.00	3.4	.	.	710.00	190.00	59.90	207
10	13	1984	5443.0	.	5.78	1075.00	2.7	.	.	740.00	215.90	56.90	208
11	14	1984	1162.0	.	5.82	1140.00	3.2	.	.	1100.00	280.00	60.00	209
11	30	1984	98.0	.	5.75	1125.00	2.1	.	.	710.00	264.00	60.00	210
12	18	1984	1123.0	.	5.75	1175.00	2.1	.	.	900.00	262.00	78.00	211
4	2	1985	867.0	.	6.30	260.00	4.5	.	.	140.00	37.20	7.20	200
4	16	1985	15760.0	.	6.50	650.00	7.6	.	.	420.00	100.00	25.40	201
5	14	1985	7403.0	.	6.32	1215.00	5.7	.	.	780.00	189.00	41.60	202
6	14	1985	21991.0	.	6.35	1525.00	3.5	.	.	1200.00	235.00	57.00	203
7	15	1985	8334.0	.	6.33	1900.00	6.2	.	.	1500.00	280.00	76.00	204
8	13	1985	4129.0	.	6.24	1425.00	5.0	.	.	1100.00	198.00	65.40	205
9	13	1985	16188.0	.	6.06	1275.00	3.2	.	.	772.00	192.00	45.30	206
10	15	1985	17240.0	.	6.19	1675.00	3.5	.	.	1100.00	260.00	74.00	207
11	15	1985	2737.0	.	6.05	1925.00	2.2	.	.	1350.00	350.00	98.00	208
3	23	1986	17252.0	.	6.20	440.00	5.0	.	.	195.00	56.00	13.00	200
4	16	1986	9550.0	.	6.17	675.00	4.5	.	.	165.00	92.00	24.40	201
5	14	1986	9357.0	.	6.20	1000.00	3.8	.	.	750.00	193.00	43.00	202
6	11	1986	2623.0	.	6.33	1750.00	5.1	.	.	1300.00	282.00	70.00	203
7	12	1986	4326.0	.	6.20	1125.00	4.2	.	.	675.00	158.00	46.00	204
8	13	1986	3619.0	.	6.00	1150.00	3.5	.	.	600.00	150.00	40.40	205
9	14	1986	11941.0	.	5.83	1475.00	3.0	.	.	750.00	222.00	55.20	206
10	16	1986	4372.0	.	5.55	2150.00	2.5	.	.	1250.00	327.00	88.00	207
11	16	1986	6294.0	.	5.64	1950.00	3.1	.	.	1000.00	280.00	94.00	208
3	22	1987	9599.0	.	5.70	350.00	6.1	.	.	190.00	58.00	14.00	200
4	15	1987	1957.0	.	6.12	600.00	4.3	.	.	316.00	74.00	24.00	201
5	16	1987	24103.0	.	5.75	1100.00	4.3	.	.	600.00	200.00	48.00	202
6	16	1987	2165.0	.	5.24	2750.00	3.2	.	.	1800.00	473.00	162.00	203
7	15	1987	12335.0	.	5.54	1475.00	4.1	.	.	830.00	188.00	58.00	204
8	15	1987	4500.0	.	5.65	1600.00	3.5	.	.	1125.00	216.00	70.00	205
9	16	1987	6953.0	.	5.46	1650.00	3.2	.	.	640.00	184.00	56.00	206
10	13	1987	1041.0	.	4.87	2850.00	1.9	.	.	1580.00	415.00	130.00	207
11	10	1987	568.0	.	5.14	2525.00	2.4	.	.	920.00	364.00	116.00	208
3	28	1988	6737.0	.	5.70	400.00	3.0	.	.	275.00	50.00	14.00	200
4	17	1988	3384.0	.	5.72	675.00	3.3	.	.	330.00	76.00	24.00	201
5	12	1988	7127.0	.	5.71	1200.00	4.0	.	.	680.00	184.00	50.00	202
6	12	1988	1605.0	.	5.56	1390.00	3.6	.	.	850.00	282.00	60.00	203
7	15	1988	836.0	.	5.47	1175.00	3.5	.	.	675.00	168.00	54.00	204
8	16	1988	43138.0	.	5.41	1550.00	2.4	.	.	870.00	216.00	88.00	205
9	16	1988	7926.0	.	5.35	2025.00	3.2	.	.	1275.00	326.00	125.00	206
10	18	1988	1893.0	.	5.47	1975.00	3.2	.	.	1300.00	272.00	102.00	207
11	15	1988	5182.0	.	5.46	1275.00	2.9	.	.	780.00	152.00	64.00	208
12	5	1988	1120.0	.	5.16	2125.00	2.2	.	.	1420.00	248.00	120.00	209
4	7	1989	20677.0	.	5.80	500.00	4.4	.	.	248.00	64.00	20.00	200
4	24	1989	6249.0	.	5.73	565.00	3.6	.	.	270.00	52.00	22.00	201
5	16	1989	7907.0	.	5.39	1100.00	3.6	.	.	865.00	132.00	78.00	202
6	17	1989	22827.0	.	5.27	1475.00	3.3	.	.	950.00	202.00	70.00	203
7	17	1989	1900.0	.	5.12	2650.00	3.2	.	.	1870.00	420.00	156.00	204
8	14	1989	1874.0	.	5.33	1450.00	3.3	.	.	880.00	182.00	76.00	205
9	14	1989	14311.0	.	5.24	1750.00	3.6	.	.	1050.00	260.00	88.00	206
10	16	1989	2562.0	.	5.23	1500.00	2.0	.	.	1140.00	230.00	84.00	207
11	6	1989	1881.0	.	5.15	1340.00	2.6	.	.	800.00	170.00	78.00	208
3	26	1990	6601.0	.	5.05	500.00	2.4	.	.	210.00	50.00	17.60	200
4	12	1990	19731.0	.	5.31	982.00	3.0	.	.	565.00	110.00	50.00	201
5	17	1990	11597.0	.	5.13	1600.00	2.7	.	.	950.00	194.00	84.00	202
6	18	1990	11752.0	.	5.27	1750.00	2.0	.	.	930.00	188.00	80.00	203
7	16	1990	6283.0	.	5.08	2850.00	3.9	.	.	1730.00	336.00	144.00	204
8	13	1990	1272.0	.	5.05	3110.00	3.0	.	.	1840.00	388.00	196.00	205
9	10	1990	3376.0	.	5.29	1545.00	2.1	.	.	815.00	174.00	68.00	206
10	15	1990	16476.0	.	5.23	1700.00	2.8	.	.	960.00	196.00	74.00	207
11	16	1990	1370.0	.	5.08	3750.00	2.2	.	.	2250.00	540.00	208.00	208
4	8	1991	712.0	.	.	.	.	.	.	.	.	.	.
4	22	1991	10579.0	.	5.70	1025.00	4.0	.	.	570.00	118.00	44.00	2020
5	20	1991	13471.0	.	5.24	1925.00	3.2	.	.	870.00	202.00	81.00	2024
6	17	1991	5185.0	.	5.25	1250.00	2.0	.	.	635.00	136.00	59.00	2040

Table A2.5.4. Drainage quality data from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
7	15	1991	17070.0	.	5.21	1700.00	2.3	.	.	775.00	146.00	74.00	2056
8	12	1991	1639.0	.	4.96	3525.00	2.0	.	.	2030.00	416.00	184.00	2072
9	9	1991	9693.0	.	5.13	1350.00	3.0	.	.	610.00	114.00	50.00	2087
10	7	1991	3051.0	.	5.09	2375.00	3.0	.	.	1210.00	228.00	98.00	2103
11	7	1991	14629.0	.	5.15	1875.00	3.0	.	.	1230.00	202.00	110.00	2119
4	23	1992	3.0	.	4.76	2087.00	3.8	80.0	177.4	1520.00	224.00	172.00	17001
5	11	1992	3229.0	.	5.26	1030.00	3.2	30.0	44.3	400.00	84.00	40.00	17014
5	26	1992	2372.0	.	5.11	1120.00	3.2	50.0	71.0	620.00	120.00	56.00	17042
6	8	1992	433.0	.	5.05	2200.00	3.2	70.0	117.2	1240.00	254.00	104.00	17046
6	22	1992	398.0	.	5.07	1860.00	3.1	53.0	89.9	1090.00	200.00	90.00	17050
7	7	1992	1270.0	.	5.08	1290.00	2.1	55.0	64.9	630.00	126.00	58.00	17055
7	20	1992	333.0	.	5.12	1780.00	2.1	50.0	86.0	2100.00	188.00	80.00	17059
8	3	1992	489.0	.	5.06	1270.00	2.1	33.0	66.3	810.00	156.00	70.00	17075
8	17	1992	402.0	.	5.11	1600.00	2.1	65.0	68.8	865.00	152.00	72.00	17079
8	31	1992	551.0	.	5.05	1600.00	5.0	33.0	70.6	875.00	162.00	82.00	17095
9	14	1992	1655.0	.	5.22	800.00	3.2	18.0	41.2	440.00	72.00	38.00	17099
9	28	1992	1409.0	.	4.99	2200.00	2.0	73.0	126.7	1260.00	250.00	120.00	17116
10	12	1992	324.0	.	5.00	3400.00	2.6	100.0	168.9	2180.00	376.00	202.00	17120
10	27	1992	178.0	.	4.92	3120.00	2.6	85.0	153.4	3860.00	328.00	186.00	17136
11	9	1992	67.0	.	5.06	3950.00	3.0	.	.	.	.	.	.
11	30	1992	368.0	.	5.11	1950.00	3.0	93.0	97.7	1270.00	204.00	120.00	17151
4	8	1993	6707.0	.	5.10	780.00	10.0	.	.	410.00	70.00	45.00	17163
4	22	1993	3936.0	.	4.91	875.00	5.0	.	.	400.00	82.50	35.00	17174
5	6	1993	8660.0	.	5.43	525.00	3.0	.	.	240.00	42.50	22.50	17179
5	20	1993	3528.0	.	5.04	1600.00	2.0	.	.	1070.00	172.50	97.50	17194
6	3	1993	3524.0	.	5.09	1300.00	3.0	.	.	715.00	120.00	70.00	17198
6	17	1993	3206.0	.	5.01	1300.00	2.5	.	.	800.00	150.00	85.00	17214
7	1	1993	3304.0	.	5.03	1050.00	1.0	.	.	555.00	95.00	60.00	17218
7	14	1993	13693.0	.	4.94	2480.00	.	.	.	1520.00	260.00	148.00	17235
7	29	1993	7142.0	.	4.92	2525.00	3.5	.	.	1660.00	270.00	166.00	17239
8	12	1993	2660.0	.	4.88	2000.00	5.0	.	.	1210.00	192.00	118.00	17255
8	26	1993	1457.0	.	4.80	2410.00	2.5	.	.	1520.00	248.00	160.00	17260
9	9	1993	4171.0	.	5.05	1300.00	2.0	.	.	630.00	112.00	68.00	17275
9	23	1993	1151.0	.	5.06	2000.00	2.0	.	.	1130.00	206.00	128.00	17279
10	7	1993	685.0	.	5.02	2550.00	2.5	.	.	1450.00	224.00	160.00	17295
10	20	1993	254.0	.	5.04	2500.00	2.0	.	.	1340.00	228.00	150.00	17299
11	4	1993	753.0	.	5.10	2100.00	2.5	.	.	620.00	149.00	108.00	17317
11	22	1993	379.0	.	.	.	.	.	.	.	.	.	.

Table A2.5.5. Drainage quality data for metals from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
4	26	1978	101.00	12.00	.	0.038	.	0.003	0.020	.	0.070	.
5	25	1978	241.00	15.00	.	0.040	0.240	0.008	0.028	0.050	0.160	.
6	16	1978	555.00	23.00	.	0.056	0.200	0.015	0.075	0.030	0.310	.
7	5	1978	313.00	28.00	.	0.057	0.316	0.021	0.084	0.040	0.220	.
7	13	1978	376.00	37.00	.	0.049	0.410	0.033	0.090	0.030	0.290	.
7	19	1978	189.00	28.00	.	0.062	0.240	0.014	0.033	0.010	0.160	.
8	2	1978	241.00	26.00	.	0.080	0.260	0.010	0.060	0.040	0.180	.
8	4	1978	410.00	33.00	.	0.070	0.440	0.029	0.113	0.050	0.280	.
8	16	1978	95.70	16.70	.	0.140	0.180	0.023	0.027	0.050	0.090	.
8	23	1978	86.00	18.00	.	0.140	0.180	0.020	0.020	0.040	0.100	.
8	25	1978	125.00	28.00	.	0.110	0.240	0.030	0.050	0.010	0.150	.
8	28	1978	196.00	32.00	.	0.100	0.380	0.045	0.090	0.010	0.230	.
9	13	1978	106.00	26.50	.	0.120	0.190	0.020	0.050	0.030	0.110	.
3	19	1979	43.80	15.80	.	0.119	0.170	0.016	0.050	0.060	0.080	.
3	27	1979	86.00	24.60	.	0.180	0.460	0.060	0.060	0.050	0.180	.
4	17	1979	25.60	11.80	.	0.108	0.134	0.008	0.030	0.060	0.050	.
4	18	1979	18.00	6.10	.	0.091	0.040	0.004	0.020	0.050	0.030	.
4	20	1979	22.00	8.00	.	0.113	0.080	0.008	0.020	0.060	0.050	.
4	22	1979	82.80	18.60	.	0.100	0.300	0.020	0.060	0.040	0.160	.
5	9	1979	83.00	18.10	.	0.084	0.295	0.020	0.040	0.040	0.140	.
5	10	1979	59.60	16.60	.	0.100	0.250	0.015	0.040	0.030	0.130	.
5	20	1979	116.80	23.20	.	0.100	0.330	0.023	0.050	0.030	0.180	.
5	22	1979	83.60	18.40	.	0.180	0.280	0.020	0.040	0.040	0.140	.
5	23	1979	174.40	28.60	.	0.100	0.440	0.027	0.070	0.210	0.230	.
6	2	1979	164.20	24.00	.	0.160	0.340	0.023	0.050	0.070	0.180	.
6	6	1979	85.00	29.60	.	0.170	0.300	0.021	0.050	0.030	0.170	.
6	10	1979	71.80	24.60	.	0.190	0.270	0.013	0.030	0.060	0.130	.
6	12	1979	107.20	40.00	.	0.132	0.370	0.017	0.070	0.020	0.290	.
6	16	1979	179.00	39.80	.	0.131	0.415	0.029	0.080	0.030	0.230	.
6	20	1979	139.00	22.00	.	0.130	0.220	0.017	0.040	0.040	0.120	.
6	30	1979	339.00	31.00	.	0.070	0.430	0.036	0.090	0.030	0.130	.
7	10	1979	430.00	38.00	.	0.060	0.590	0.052	0.110	0.020	0.340	.
7	30	1979	88.00	17.00	.	0.110	0.170	0.014	0.030	0.030	0.090	.
8	4	1979	183.00	32.00	.	0.070	0.360	0.030	0.070	0.070	0.190	.
8	13	1979	408.00	38.60	.	0.050	0.650	0.035	0.080	0.040	0.300	.
8	23	1979	200.00	25.80	.	0.100	0.470	0.027	0.070	0.040	0.210	.
8	28	1979	132.60	20.00	.	0.150	0.360	0.021	0.060	0.070	0.150	.
9	1	1979	84.20	19.80	.	0.150	0.280	0.020	0.050	0.050	0.100	.
9	4	1979	101.40	25.80	.	0.120	0.370	0.020	0.070	0.020	0.160	.
9	10	1979	104.00	21.20	.	0.150	0.340	0.020	0.050	0.030	0.140	.
9	13	1979	156.00	26.80	.	0.150	0.450	0.026	0.090	0.050	0.200	.
9	19	1979	238.20	37.20	.	0.060	0.760	0.070	0.120	0.020	0.300	.
10	19	1979	64.00	14.90	.	0.195	0.365	0.020	0.040	0.060	0.100	.
10	31	1979	84.00	20.80	.	0.160	0.490	0.050	0.090	0.050	0.160	.
11	1	1979	98.00	25.80	.	0.110	0.550	0.060	0.080	0.020	0.160	.
4	1	1980	17.80	7.80	.	0.135	0.095	0.010	0.030	0.080	0.040	.
4	16	1980	33.80	13.20	.	0.130	0.130	0.030	0.060	0.030	0.060	.
4	17	1980	30.20	10.30	.	0.150	0.210	0.020	0.050	0.070	0.070	.
4	29	1980	46.00	16.00	.	0.100	0.360	0.025	0.060	0.020	0.120	.
5	8	1980	58.60	18.80	.	0.070	0.450	0.040	0.060	0.020	0.140	.
5	18	1980	92.00	23.00	.	0.050	0.600	0.050	0.100	0.030	0.180	.
5	31	1980	45.50	16.20	.	0.030	0.300	0.022	0.020	0.020	0.100	.
6	3	1980	68.80	21.60	.	0.110	0.570	0.030	0.100	0.020	0.130	.
6	6	1980	63.20	20.00	.	0.150	0.490	0.025	0.090	0.030	0.120	.
6	13	1980	65.60	19.60	.	0.150	0.450	0.022	0.080	0.030	0.110	.
6	19	1980	98.80	23.40	.	0.110	0.580	0.021	0.110	0.060	0.150	.
6	30	1980	58.00	17.00	.	0.130	0.480	0.070	0.100	0.050	0.130	.
7	2	1980	45.00	16.00	.	0.170	0.380	0.022	0.090	0.040	0.100	.
7	17	1980	58.40	19.60	.	0.140	0.500	0.030	0.090	0.030	0.140	.
7	23	1980	66.40	19.20	.	0.110	0.470	0.060	0.090	0.020	0.150	.
8	3	1980	35.00	14.80	.	0.260	0.400	0.022	0.070	0.050	0.120	.
8	12	1980	25.70	12.80	.	0.260	0.340	0.015	0.050	0.100	0.110	.
8	19	1980	28.80	16.00	.	0.190	0.450	0.024	0.080	0.050	0.160	.
8	22	1980	24.10	11.70	.	0.220	0.390	0.050	0.070	0.070	0.120	.
8	29	1980	68.20	27.00	.	0.170	0.830	0.080	0.160	0.060	0.220	.
9	7	1980	44.00	16.40	.	0.190	0.610	0.050	0.110	0.040	0.160	.

Table A2.5.5. Drainage quality data for metals from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
9	10	1980	71.00	22.80	.	0.180	0.860	0.060	0.150	0.040	0.230	.
9	12	1980	47.60	17.40	.	0.210	0.590	0.050	0.110	0.060	0.160	.
9	18	1980	70.60	20.20	.	0.240	0.760	0.060	0.130	0.080	0.160	.
9	23	1980	204.00	40.00	.	0.200	1.780	0.140	0.250	0.040	0.310	.
10	22	1980	330.00	48.40	.	0.240	1.910	0.120	0.310	0.040	0.350	.
10	23	1980	170.00	23.00	.	0.190	1.260	0.090	0.200	0.040	0.250	.
2	18	1981	39.00	.	.	0.210	0.295	0.020	0.060	.	.	.
3	29	1981	24.70	.	.	0.210	0.220	0.020	0.050	.	.	.
3	30	1981	20.80	.	.	0.280	0.490	0.030	0.080	.	.	.
4	13	1981	21.00	.	.	0.290	0.770	0.060	0.080	.	.	.
4	16	1981	22.00	.	.	0.250	0.920	0.060	0.090	.	.	.
4	23	1981	27.00	.	.	0.260	0.970	0.050	0.090	.	.	.
4	25	1981	36.00	.	.	0.240	1.180	0.080	0.100	.	.	.
4	27	1981	48.00	.	.	0.260	1.260	0.080	0.100	.	.	.
4	30	1981	80.00	.	.	0.200	1.700	0.100	0.140	.	.	.
5	12	1981	166.00	.	.	0.130	2.100	0.140	0.170	.	.	.
6	1	1981	200.00	.	.	0.160	1.160	0.070	0.150	.	.	.
6	6	1981	174.00	.	.	0.170	0.960	0.070	0.130	.	.	.
6	22	1981	102.00	.	.	0.200	0.710	0.060	0.090	.	.	.
6	24	1981	46.00	.	.	0.250	0.550	0.040	0.060	.	.	.
6	28	1981	36.00	.	.	0.300	0.640	0.040	0.070	.	.	.
7	4	1981	98.00	.	.	0.270	1.370	0.090	0.140	.	.	.
7	15	1981	259.60	.	.	0.140	2.550	0.150	0.220	.	.	.
7	24	1981	.	.	.	.	.	.	.	.	.	.
8	4	1981	154.00	.	.	0.190	2.440	0.160	0.310	.	.	.
8	13	1981	184.00	28.00	.	0.130	1.260	0.110	0.170	.	.	.
8	31	1981	140.00	.	.	0.420	1.230	0.090	0.160	.	.	.
9	7	1981	124.00	.	.	0.280	1.190	0.080	0.160	.	.	.
9	26	1981	70.00	.	.	0.380	1.060	0.060	0.140	.	.	.
10	1	1981	32.00	.	.	0.350	0.880	0.070	0.120	.	.	.
10	4	1981	34.00	16.00	.	0.310	0.960	0.080	0.160	.	.	.
10	13	1981	34.00	18.00	.	0.310	1.370	0.100	0.200	.	.	.
10	17	1981	58.00	28.00	.	0.510	4.200	0.240	0.360	.	.	.
11	5	1981	304.00	56.00	.	1.000	5.200	0.300	0.510	.	.	.
4	24	1982	32.00	12.00	.	0.130	0.570	0.030	0.110	.	.	.
4	26	1982	29.50	14.00	.	0.315	1.770	0.100	0.140	.	.	.
5	3	1982	56.00	16.00	.	0.160	1.850	0.110	0.170	.	.	.
5	7	1982	20.00	12.00	.	0.320	1.800	0.110	0.130	.	.	.
5	11	1982	36.00	16.00	.	0.440	3.230	0.160	0.190	.	.	.
5	14	1982	40.40	14.60	.	0.320	2.100	0.100	0.140	0.030	0.150	.
5	21	1982	143.00	32.00	.	0.410	4.030	0.210	0.270	0.040	0.240	.
6	2	1982	134.00	26.00	.	0.350	2.500	0.130	0.200	0.040	0.200	.
6	18	1982	124.00	25.00	.	0.280	1.920	0.100	0.160	0.030	0.150	.
7	2	1982	56.00	16.00	.	0.350	1.300	0.060	0.100	0.030	0.110	.
7	8	1982	46.00	18.00	.	0.440	2.350	0.130	0.130	0.060	0.150	.
7	12	1982	144.00	30.00	.	0.460	3.030	0.200	0.180	0.010	0.230	.
7	26	1982	146.00	32.00	.	0.320	4.110	0.250	0.180	0.050	0.260	.
8	30	1982	76.00	24.00	.	0.460	2.040	0.130	0.160	0.060	0.170	.
10	4	1982	216.00	42.00	.	0.800	5.890	0.380	0.250	0.070	0.380	.
11	1	1982	76.00	16.00	.	0.360	2.205	0.125	0.100	0.030	0.130	.
11	22	1982	192.00	.	.	0.545	2.715	0.155	0.160	.	.	.
4	14	1983	.	.	.	0.450	1.820	0.100	0.130	.	.	.
5	15	1983	.	.	.	0.590	3.470	0.200	0.270	.	.	.
6	16	1983	.	.	.	0.390	2.030	0.130	0.150	.	.	.
7	14	1983	.	.	.	0.620	3.580	0.190	0.240	.	.	.
8	14	1983	.	.	.	0.830	3.920	0.190	0.280	.	.	.
9	15	1983	.	.	.	1.140	4.710	0.280	0.380	.	.	.
10	13	1983	.	.	.	2.120	6.830	0.400	0.460	.	.	.
11	9	1983	.	.	.	1.580	4.770	0.270	0.360	.	.	.
11	30	1983	.	.	.	1.520	5.450	0.300	0.400	.	.	.
4	15	1984	45.80	.	.	0.410	3.210	0.170	0.400	.	.	202
5	16	1984	19.90	.	.	0.670	4.120	0.140	0.190	.	.	203
6	13	1984	32.70	.	.	1.340	5.440	0.260	0.290	.	.	204
7	14	1984	71.90	.	.	1.300	6.330	0.240	0.300	.	.	205
8	15	1984	53.20	.	.	0.950	4.820	0.270	0.270	.	.	206
9	12	1984	64.20	.	.	1.250	5.310	0.310	0.330	.	.	207

Table A2.5.5. Drainage quality data for metals from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
10	13	1984	29.20	.	.	1.920	8.500	0.600	0.540	.	.	208
11	14	1984	51.00	.	.	2.500	8.850	0.410	0.640	.	.	209
11	30	1984	54.00	.	.	2.500	11.060	0.400	0.620	.	.	210
12	18	1984	46.00	.	.	3.900	14.400	0.830	0.810	.	.	211
4	2	1985	8.40	.	.	0.410	1.260	0.030	0.230	.	.	200
4	16	1985	17.80	.	.	1.010	5.620	0.270	0.390	.	.	201
5	14	1985	27.50	.	.	2.210	10.100	0.490	0.530	.	.	202
6	14	1985	42.80	.	.	3.170	11.600	0.490	0.520	.	.	203
7	15	1985	97.00	.	.	3.180	10.600	0.490	0.540	.	.	204
8	13	1985	64.00	.	.	1.960	6.990	0.360	0.390	.	.	205
9	13	1985	35.80	.	.	1.890	8.670	0.420	0.450	.	.	206
10	15	1985	58.00	.	.	3.230	9.060	0.360	0.440	.	.	207
11	15	1985	88.00	.	.	3.340	9.200	0.390	0.510	.	.	208
3	23	1986	8.20	.	.	0.930	4.300	0.170	0.190	.	.	200
4	16	1986	12.60	.	.	1.620	7.300	0.310	0.290	.	.	201
5	14	1986	19.00	.	.	2.940	13.300	0.480	0.390	.	.	202
6	11	1986	64.00	.	.	304.000	11.700	0.550	0.530	.	.	203
7	12	1986	36.00	.	.	2.030	6.900	0.340	0.320	.	.	204
8	13	1986	36.00	.	.	2.270	7.300	0.400	0.270	.	.	205
9	14	1986	32.00	.	.	4.400	14.000	0.700	0.440	.	.	206
10	16	1986	66.00	.	.	6.200	19.000	0.800	0.770	.	.	207
11	16	1986	62.00	.	.	8.000	19.400	0.930	0.870	.	.	208
3	22	1987	10.00	.	.	1.100	3.900	0.150	0.170	.	.	200
4	15	1987	12.00	.	.	1.670	5.900	0.280	0.320	.	.	201
5	16	1987	26.00	.	.	3.400	10.900	0.500	0.310	.	.	202
6	16	1987	103.00	.	.	7.300	17.900	0.800	0.690	.	.	203
7	15	1987	34.00	.	.	4.900	15.900	0.800	0.660	.	.	204
8	15	1987	46.00	.	.	5.200	15.100	0.700	0.670	.	.	205
9	16	1987	32.00	.	.	3.870	12.370	0.550	0.560	.	.	206
10	13	1987	82.00	.	.	9.000	31.000	1.250	1.150	.	.	207
11	10	1987	78.00	.	.	8.300	27.000	1.110	1.110	.	.	208
3	28	1988	6.00	.	.	1.500	5.200	0.300	0.500	.	.	200
4	17	1988	8.00	.	.	2.400	7.800	0.500	0.800	.	.	201
5	12	1988	220.00	.	.	4.700	14.600	0.900	1.400	.	.	202
6	12	1988	24.00	.	.	6.200	22.000	1.100	1.600	.	.	203
7	15	1988	28.00	.	.	5.000	13.000	1.000	1.300	.	.	204
8	16	1988	40.00	.	.	6.500	37.000	1.300	1.200	.	.	205
9	16	1988	79.00	.	.	6.650	35.500	1.250	1.250	.	.	206
10	18	1988	51.00	.	.	5.900	20.500	1.100	1.400	.	.	207
11	15	1988	28.00	.	.	5.700	22.000	1.200	1.400	.	.	208
12	5	1988	42.00	.	.	13.400	53.000	2.600	2.900	.	.	209
4	7	1989	10.00	.	.	2.700	13.200	0.700	0.700	.	.	200
4	24	1989	8.00	.	.	2.900	15.000	0.700	0.800	.	.	201
5	16	1989	20.00	.	.	20.000	61.000	3.800	4.100	.	.	202
6	17	1989	32.00	.	.	7.600	44.000	2.000	1.600	.	.	203
7	17	1989	76.00	.	.	12.100	46.000	1.900	2.600	.	.	204
8	14	1989	38.00	.	.	5.500	26.000	0.900	1.000	.	.	205
9	14	1989	32.00	.	.	10.000	39.000	2.700	2.100	.	.	206
10	16	1989	44.00	.	.	8.400	29.000	2.600	1.900	.	.	207
11	6	1989	34.00	.	.	6.500	24.000	2.800	1.500	.	.	208
3	26	1990	6.60	.	.	2.700	9.120	0.480	0.600	.	.	200
4	12	1990	12.00	.	.	9.260	32.000	1.840	1.850	.	.	201
5	17	1990	24.00	.	.	11.000	36.000	2.120	2.170	.	.	202
6	18	1990	32.00	.	.	7.210	25.000	1.220	1.420	.	.	203
7	16	1990	50.00	.	.	14.000	42.000	2.430	2.680	.	.	204
8	13	1990	70.00	.	.	14.000	40.000	2.330	2.460	.	.	205
9	10	1990	30.00	.	.	6.020	20.000	1.160	1.240	.	.	206
10	15	1990	28.00	.	.	11.000	36.000	2.010	1.800	.	.	207
11	16	1990	74.00	.	.	22.000	69.000	3.740	3.860	.	.	208
4	8	1991	.	.	.	.	.	.	.	.	.	.
4	22	1991	16.00	.	.	4.400	17.000	0.700	0.800	.	.	2020
5	20	1991	20.00	.	.	9.900	36.000	1.500	1.200	.	.	2024
6	17	1991	18.00	.	.	6.900	23.000	1.100	0.800	.	.	2040
7	15	1991	20.00	.	.	9.000	30.000	1.400	1.000	.	.	2056
8	12	1991	72.00	.	.	18.000	58.000	2.700	2.100	.	.	2072
9	9	1991	20.00	.	.	8.000	30.000	1.400	0.900	.	.	2087

Table A2.5.5. Drainage quality data for metals from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
10	7	1991	38.00	.	.	13.000	43.000	2.100	1.700	.	.	2103
11	7	1991	30.00	.	.	22.000	69.000	3.400	2.600	.	.	2119
4	23	1992	49.20	31.00	1.00	17.800	75.000	3.200	2.300	0.100	3.790	17001
5	11	1992	13.20	12.70	<1.00	5.600	21.000	1.000	0.800	0.060	1.230	17014
5	26	1992	16.00	15.70	<1.00	10.300	31.000	1.400	1.100	0.140	1.770	17042
6	8	1992	30.40	27.00	<1.00	16.500	50.000	2.200	1.900	0.070	2.670	17046
6	22	1992	28.40	25.00	<1.00	12.400	39.000	2.000	1.400	0.040	2.120	17050
7	7	1992	20.20	19.20	<1.00	9.000	29.000	1.400	1.000	0.070	1.570	17055
7	20	1992	27.40	24.00	<1.00	11.300	38.000	1.900	1.400	0.070	1.950	17059
8	3	1992	25.40	22.00	<1.00	9.500	29.000	1.500	1.200	0.060	1.630	17075
8	17	1992	26.60	23.00	<1.00	9.900	30.000	1.500	1.200	0.050	1.730	17079
8	31	1992	27.40	23.00	<1.00	9.000	32.000	1.500	1.200	0.040	1.670	17095
9	14	1992	13.40	15.20	<1.00	6.000	19.000	1.000	0.700	0.090	1.100	17099
9	28	1992	37.00	32.00	<1.00	17.500	54.000	2.600	1.900	0.070	2.900	17116
10	12	1992	64.00	44.00	<1.00	23.000	71.000	3.600	2.600	0.080	3.850	17120
10	27	1992	60.60	40.00	<1.00	19.900	66.000	3.200	2.100	0.040	3.540	17136
11	9	1992	.	.	.	.	.	.	.	.	.	.
11	30	1992	39.40	28.00	<1.00	11.500	44.000	2.100	1.500	0.040	2.350	17151
4	8	1993	11.00	12.00	<1.00	4.840	20.000	0.720	0.570	0.060	0.910	17163
4	22	1993	9.00	9.00	<1.00	4.260	18.000	0.830	0.550	0.090	0.810	17174
5	6	1993	6.50	6.50	<1.00	3.190	12.000	0.570	0.390	0.130	0.540	17179
5	20	1993	20.30	17.50	<1.00	13.000	56.000	2.600	1.420	0.050	2.570	17194
6	3	1993	17.30	13.30	<1.00	10.000	38.000	1.720	1.100	0.090	1.770	17198
6	17	1993	21.00	16.30	<1.00	12.000	43.000	1.900	1.260	0.070	1.970	17214
7	1	1993	15.30	13.80	<1.00	11.000	34.000	1.540	1.000	0.090	1.620	17218
7	14	1993	33.40	32.40	2.00	24.000	80.000	3.440	2.300	0.090	4.140	17235
7	29	1993	40.80	34.60	2.00	21.000	75.000	3.330	2.180	0.120	3.890	17239
8	12	1993	33.40	27.40	<1.00	13.000	51.000	2.020	1.550	0.080	2.800	17255
8	26	1993	45.20	33.60	<1.00	16.000	63.000	2.480	1.910	0.070	3.440	17260
9	9	1993	20.40	18.80	<1.00	8.980	30.000	1.310	0.980	0.070	1.800	17275
9	23	1993	38.20	30.00	<1.00	14.000	53.000	2.210	1.690	0.070	3.020	17279
10	7	1993	44.60	33.20	<1.00	18.000	62.000	2.690	1.730	0.080	3.130	17295
10	20	1993	40.00	30.20	<1.00	16.000	61.000	2.450	1.660	0.050	2.980	17299
11	4	1993	27.60	22.00	<1.00	10.000	41.000	1.940	1.130	0.060	2.150	17317
11	22	1993	.	.	.	.	.	.	.	.	.	.

Table A2.5.6. Drainage quality data for additional parameters from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
4	26	1978	.	.	.	.	.	4.0	.
5	25	1978	.	.	.	.	.	.	.
6	16	1978	.	.	.	.	.	11.0	.
7	5	1978	34.00	.	.	.	.	12.0	.
7	13	1978	30.00	12.00	1.00	0.08	.	.	.
7	19	1978	41.00	10.00	0.75	0.10	.	.	.
8	2	1978	14.00	8.00	0.20	0.20	.	19.0	.
8	4	1978	19.00	11.00	0.06	0.08	.	22.0	.
8	16	1978	17.00	.	0.10	0.08	.	23.0	.
8	23	1978	15.00	5.30	1.10	0.10	.	.	.
8	25	1978	25.00	3.60	.	0.10	.	23.0	.
8	28	1978	34.00	8.50	1.30	0.08	.	.	.
9	13	1978	12.00	5.50	0.06	0.08	.	15.0	.
3	19	1979	.	.	.	.	12.00	.	.
3	27	1979	.	.	.	.	.	.	.
4	17	1979	10.00	1.4E2	.	.	14.00	.	.
4	18	1979	5.50	1.20	.	.	10.50	.	.
4	20	1979	12.00	2.20	0.45	.	11.00	.	.
4	22	1979	30.00	11.00	0.11	.	.	.	.
5	9	1979	26.00	.	.	.	.	.	.
5	10	1979	22.00	.	.	.	.	.	.
5	20	1979	25.00	.	.	.	14.00	.	.
5	22	1979	20.00	.	.	.	16.00	.	.
5	23	1979	.	.	.	.	4.70	.	.
6	2	1979	23.00	.	.	.	16.00	.	.
6	6	1979	23.00	.	.	.	.	.	.
6	10	1979	19.00	.	.	.	.	22.0	.
6	12	1979	27.00	.	.	.	.	.	.
6	16	1979	31.00	.	.	.	.	25.0	.
6	20	1979	15.00	.	.	.	.	18.0	.
6	30	1979	31.00	.	.	.	16.00	12.0	.
7	10	1979	28.00	.	.	.	.	19.0	.
7	30	1979	16.00	.	.	.	14.00	22.0	.
8	4	1979	15.00	.	.	.	9.20	23.0	.
8	13	1979	.	.	.	.	.	.	.
8	23	1979	.	.	.	.	14.00	.	.
8	28	1979	.	.	.	.	15.00	23.0	.
9	1	1979	9.00	.	.	.	13.00	21.0	.
9	4	1979	.	.	.	.	13.00	20.0	.
9	10	1979	9.00	.	.	.	14.00	17.0	.
9	13	1979	.	.	.	.	14.00	18.0	.
9	19	1979	.	.	.	.	13.00	18.0	.
10	19	1979	5.00	.	.	.	14.00	.	.
10	31	1979	4.00	.	.	.	13.00	.	.
11	1	1979	10.00	.	.	.	13.00	.	.
4	1	1980	3.50	.	.	.	10.00	10.5	.
4	16	1980	5.00	.	.	.	.	.	.
4	17	1980	5.50	.	.	.	12.00	.	.
4	29	1980	9.00	.	.	.	.	13.0	.
5	8	1980	10.00	.	.	.	9.50	11.0	.
5	18	1980	23.00	.	.	.	.	19.0	.
5	31	1980	14.00	.	.	.	15.00	.	.
6	3	1980	15.00	.	.	.	9.80	.	.
6	6	1980	14.00	.	.	.	.	.	.
6	13	1980	12.00	.	.	.	.	.	.
6	19	1980	16.00	.	.	.	.	.	.
6	30	1980	10.00	.	.	.	10.00	.	.
7	2	1980	9.00	.	.	.	.	.	.
7	17	1980	10.00	.	.	.	.	.	.
7	23	1980	9.00	.	.	.	.	.	.
8	3	1980	9.00	.	.	.	16.00	.	.
8	12	1980	7.00	.	.	.	14.00	.	.
8	19	1980	6.00	.	.	.	.	.	.
8	22	1980	6.00	.	.	.	.	.	.
8	29	1980	11.00	.	.	.	10.00	.	.
9	7	1980	7.00	.	.	.	.	.	.

Table A2.5.6. Drainage quality data for additional parameters from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
9	10	1980	10.00	.	.	.	.	13.2	.
9	12	1980	9.00	.	.	.	13.00	17.5	.
9	18	1980	11.00	.	.	.	.	13.0	.
9	23	1980	25.00	.	.	.	13.00	15.0	.
10	22	1980	29.00	.	.	.	13.00	10.0	.
10	23	1980	31.00	.	.	.	.	12.0	.
2	18	1981	10.00	.	.	.	.	6.0	.
3	29	1981	.	.	.	.	15.00	.	.
3	30	1981	.	.	.	.	.	.	.
4	13	1981	.	.	.	.	.	.	.
4	16	1981	.	.	.	.	15.00	.	.
4	23	1981	.	.	.	.	.	.	.
4	25	1981	.	.	.	.	.	.	.
4	27	1981	.	.	.	.	14.00	.	.
4	30	1981	.	.	.	.	.	.	.
5	12	1981	.	.	.	.	14.00	17.5	.
6	1	1981	.	.	.	.	.	17.0	.
6	6	1981	.	.	.	.	13.00	18.0	.
6	22	1981	.	.	.	.	.	16.0	.
6	24	1981	.	.	.	.	.	17.0	.
6	28	1981	.	.	.	.	18.00	15.0	.
7	4	1981	.	.	.	.	.	22.0	.
7	15	1981	.	.	.	.	.	19.0	.
7	24	1981	21.00	.	.	.	9.10	20.0	.
8	4	1981	.	.	.	.	.	18.0	.
8	13	1981	.	.	.	.	.	22.0	.
8	31	1981	.	.	.	.	15.00	.	.
9	7	1981	.	.	.	.	.	.	.
9	26	1981	.	.	.	.	11.00	.	.
10	1	1981	.	.	.	.	.	.	.
10	4	1981	.	.	.	.	.	.	.
10	13	1981	.	.	.	.	9.70	.	.
10	17	1981	.	.	.	.	.	.	.
11	5	1981	.	.	.	.	9.60	.	.
4	24	1982	.	.	.	.	.	.	.
4	26	1982	.	.	.	.	.	.	.
5	3	1982	.	.	.	.	.	.	.
5	7	1982	.	.	.	.	.	.	.
5	11	1982	.	.	.	.	.	.	.
5	14	1982	.	.	.	.	.	.	.
5	21	1982	.	.	.	.	.	.	.
6	2	1982	.	.	.	.	.	.	.
6	18	1982	.	.	.	.	.	.	.
7	2	1982	.	.	.	.	.	.	.
7	8	1982	.	.	.	.	.	.	.
7	12	1982	.	.	.	.	.	.	.
7	26	1982	.	.	.	.	.	.	.
8	30	1982	.	.	.	.	.	.	.
10	4	1982	.	.	.	.	.	.	.
11	1	1982	.	.	.	.	.	.	.
11	22	1982	.	.	.	.	.	.	.
4	14	1983	.	.	.	.	.	.	.
5	15	1983	.	.	.	.	.	.	.
6	16	1983	.	.	.	.	.	.	.
7	14	1983	.	.	.	.	.	.	.
8	14	1983	.	.	.	.	.	.	.
9	15	1983	.	.	.	.	.	.	.
10	13	1983	.	.	.	.	.	.	.
11	9	1983	.	.	.	.	.	.	.
11	30	1983	.	.	.	.	.	.	.
4	15	1984	.	.	.	.	.	.	202
5	16	1984	.	.	.	.	.	.	203
6	13	1984	.	.	.	.	.	.	204
7	14	1984	.	.	.	.	.	.	205
8	15	1984	.	.	.	.	.	.	206
9	12	1984	.	.	.	.	.	.	207

Table A2.5.6. Drainage quality data for additional parameters from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
10	13	1984	.	.	.	.	.	.	208
11	14	1984	.	.	.	.	.	.	209
11	30	1984	.	.	.	.	.	.	210
12	18	1984	.	.	.	.	.	.	211
4	2	1985	.	.	.	.	.	.	200
4	16	1985	.	.	.	.	.	.	201
5	14	1985	.	.	.	.	.	.	202
6	14	1985	.	.	.	.	.	.	203
7	15	1985	.	.	.	.	.	.	204
8	13	1985	.	.	.	.	.	.	205
9	13	1985	.	.	.	.	.	.	206
10	15	1985	.	.	.	.	.	.	207
11	15	1985	.	.	.	.	.	.	208
3	23	1986	.	.	.	.	.	.	200
4	16	1986	.	.	.	.	.	.	201
5	14	1986	.	.	.	.	.	.	202
6	11	1986	.	.	.	.	.	.	203
7	12	1986	.	.	.	.	.	.	204
8	13	1986	.	.	.	.	.	.	205
9	14	1986	.	.	.	.	.	.	206
10	16	1986	.	.	.	.	.	.	207
11	16	1986	.	.	.	.	.	.	208
3	22	1987	.	.	.	.	.	.	200
4	15	1987	.	.	.	.	.	.	201
5	16	1987	.	.	.	.	.	.	202
6	16	1987	.	.	.	.	.	.	203
7	15	1987	.	.	.	.	.	.	204
8	15	1987	.	.	.	.	.	.	205
9	16	1987	.	.	.	.	.	.	206
10	13	1987	.	.	.	.	.	.	207
11	10	1987	.	.	.	.	.	.	208
3	28	1988	.	.	.	.	.	.	200
4	17	1988	.	.	.	.	.	.	201
5	12	1988	.	.	.	.	.	.	202
6	12	1988	.	.	.	.	.	.	203
7	15	1988	.	.	.	.	.	.	204
8	16	1988	.	.	.	.	.	.	205
9	16	1988	.	.	.	.	.	.	206
10	18	1988	.	.	.	.	.	.	207
11	15	1988	.	.	.	.	.	.	208
12	5	1988	.	.	.	.	.	.	209
4	7	1989	.	.	.	.	.	.	200
4	24	1989	.	.	.	.	.	.	201
5	16	1989	.	.	.	.	.	.	202
6	17	1989	.	.	.	.	.	.	203
7	17	1989	.	.	.	.	.	.	204
8	14	1989	.	.	.	.	.	.	205
9	14	1989	.	.	.	.	.	.	206
10	16	1989	.	.	.	.	.	.	207
11	6	1989	.	.	.	.	.	.	208
3	26	1990	.	.	.	.	.	.	200
4	12	1990	.	.	.	.	.	.	201
5	17	1990	.	.	.	.	.	.	202
6	18	1990	.	.	.	.	.	.	203
7	16	1990	.	.	.	.	.	.	204
8	13	1990	.	.	.	.	.	.	205
9	10	1990	.	.	.	.	.	.	206
10	15	1990	.	.	.	.	.	.	207
11	16	1990	.	.	.	.	.	.	208
4	8	1991	.	.	.	.	.	.	.
4	22	1991	.	.	.	.	.	.	2020
5	20	1991	.	.	.	.	.	.	2024
6	17	1991	.	.	.	.	.	.	2040
7	15	1991	.	.	.	.	.	.	2056
8	12	1991	.	.	.	.	.	.	2072
9	9	1991	.	.	.	.	.	.	2087

Table A2.5.6. Drainage quality data for additional parameters from Duluth Complex test pile FL2, from 1978 to 1993 (0.63% S, 0.35% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
10	7	1991	.	.	.	.	.	.	2103
11	7	1991	.	.	.	.	.	.	2119
4	23	1992	.	.	.	.	.	.	17001
5	11	1992	.	.	.	.	.	.	17014
5	26	1992	.	.	.	.	.	.	17042
6	8	1992	.	.	.	.	.	.	17046
6	22	1992	.	.	.	.	.	.	17050
7	7	1992	.	.	.	.	.	.	17055
7	20	1992	.	.	.	.	.	.	17059
8	3	1992	.	.	.	.	.	.	17075
8	17	1992	.	.	.	.	.	.	17079
8	31	1992	.	.	.	.	.	.	17095
9	14	1992	.	.	.	.	.	.	17099
9	28	1992	.	.	.	.	.	.	17116
10	12	1992	.	.	.	.	.	.	17120
10	27	1992	.	.	.	.	.	.	17136
11	9	1992	.	.	.	.	.	.	.
11	30	1992	.	.	.	.	.	.	17151
4	8	1993	.	.	.	.	.	.	17163
4	22	1993	.	.	.	.	.	.	17174
5	6	1993	.	.	.	.	.	.	17179
5	20	1993	.	.	.	.	.	.	17194
6	3	1993	.	.	.	.	.	.	17198
6	17	1993	.	.	.	.	.	.	17214
7	1	1993	.	.	.	.	.	.	17218
7	14	1993	.	.	.	.	.	.	17235
7	29	1993	.	.	.	.	.	.	17239
8	12	1993	.	.	.	.	.	.	17255
8	26	1993	.	.	.	.	.	.	17260
9	9	1993	.	.	.	.	.	.	17275
9	23	1993	.	.	.	.	.	.	17279
10	7	1993	.	.	.	.	.	.	17295
10	20	1993	.	.	.	.	.	.	17299
11	4	1993	.	.	.	.	.	.	17317
11	22	1993	.	.	.	.	.	.	.

Table A2.5.7. Drainage quality data from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
4	26	1978	.	.	7.20	675.00	.	.	.	.	62.00	10.00	.
5	25	1978	.	.	7.80	1200.00	.	.	.	.	106.00	24.00	.
6	16	1978	192.0	.	7.79	3000.00	.	.	.	.	214.00	57.00	.
7	5	1978	133.0	.	7.30	1600.00	.	.	.	600.00	142.00	34.00	.
7	13	1978	.	.	7.55	.	.	.	.	1050.00	352.00	46.00	.
7	19	1978	.	.	7.85	1000.00	.	.	.	550.00	241.00	28.00	.
8	2	1978	.	.	7.60	650.00	.	.	.	367.00	96.00	14.00	.
8	4	1978	100.0	.	7.65	950.00	.	.	.	500.00	156.00	24.00	.
8	10	1978	851.0	.	7.90	1900.00	40.0	.	.	1200.00	348.00	51.00	.
8	16	1978	.	.	7.62	750.00	29.0	.	.	300.00	83.70	14.70	.
8	23	1978	.	.	7.50	750.00	33.0	.	.	300.00	75.00	14.00	.
8	25	1978	.	.	7.30	1150.00	25.0	.	.	500.00	115.00	23.00	.
9	13	1978	130.0	.	7.30	689.50	36.0	.	.	658.50	133.50	21.50	.
3	19	1979	3179.4	1765.0	7.08	794.00	.	.	.	.	118.80	25.90	.
3	27	1979	1877.4	442.0	7.20	1233.00	40.0	.	.	.	162.00	36.80	.
4	13	1979	1877.4	442.0	6.75	484.00	.	.	.	232.50	61.90	12.80	.
4	17	1979	3039.4	3576.0	6.70	587.00	28.0	.	.	235.00	69.80	13.80	.
4	19	1979	3311.9	4304.0	7.10	532.00	27.0	.	.	205.00	83.00	12.60	.
4	19	1979	4341.4	4304.0	6.80	503.00	23.0	.	.	215.00	80.50	13.40	.
4	20	1979	4125.7	2592.0	6.70	623.00	33.0	.	.	310.00	100.00	17.60	.
4	22	1979	8607.1	1630.0	7.20	1356.00	19.0	.	.	833.00	206.40	44.60	.
5	7	1979	479.0	308.0	7.30	1562.00	.	.	.	.	209.20	49.00	.
5	10	1979	7070.4	3099.0	7.40	959.00	.	.	.	433.00	123.20	26.40	.
5	20	1979	2619.2	900.0	6.70	1711.00	.	.	.	933.00	229.20	52.60	.
5	22	1979	1559.4	784.0	7.60	1537.00	.	.	.	500.00	120.40	26.20	.
5	23	1979	2448.9	838.0	7.50	1331.00	.	.	.	750.00	180.40	37.40	.
6	1	1979	1937.9	369.0	7.60	998.00	.	.	.	517.00	122.80	26.60	.
6	6	1979	1767.6	515.0	7.50	824.00	.	.	.	400.00	101.00	21.80	.
6	10	1979	1771.4	938.0	7.20	725.00	35.0	.	.	320.00	173.80	38.00	.
6	12	1979	1010.6	361.0	6.80	912.00	33.0	.	.	420.00	118.80	25.20	.
6	15	1979	329.3	325.0	7.30	1081.00	33.0	.	.	533.00	143.80	30.40	.
6	21	1979	2195.3	1070.0	7.05	822.50	30.5	.	.	408.50	101.00	23.50	.
6	30	1979	1389.1	169.0	7.20	1175.00	29.0	.	.	725.00	158.00	35.00	.
7	10	1979	734.3	74.0	7.50	1649.00	26.0	.	.	1100.00	241.00	52.00	.
7	30	1979	2596.5	1907.0	7.10	772.00	12.0	.	.	390.00	98.00	21.00	.
8	4	1979	601.8	138.0	7.05	1277.00	13.4	.	.	808.50	204.50	36.00	.
8	13	1979	295.2	46.0	6.90	2064.00	12.0	.	.	1850.00	368.00	77.60	.
8	28	1979	208.2	161.0	7.10	1172.00	12.4	.	.	720.00	177.80	36.80	.
9	1	1979	3894.8	1538.0	6.90	1002.00	12.0	.	.	650.00	156.60	29.80	.
9	5	1979	499.6	292.0	6.90	1575.00	12.0	.	.	1200.00	269.60	52.00	.
9	10	1979	1983.3	408.0	6.80	1401.00	9.6	.	.	975.00	235.60	42.60	.
9	13	1979	1332.3	408.0	7.00	1499.00	11.2	.	.	950.00	231.80	44.00	.
9	19	1979	863.0	109.0	6.80	2598.00	12.0	.	.	1900.00	430.80	84.80	.
10	17	1979	336.9	200.0	6.40	2192.00	7.2	.	.	.	363.20	73.40	.
10	19	1979	1177.1	1297.0	6.30	938.00	8.0	.	.	600.00	124.40	28.00	.
10	31	1979	3962.9	1450.0	6.40	1233.00	9.6	.	.	725.00	172.00	36.50	.
11	1	1979	3315.7	944.0	6.80	1258.00	12.8	.	.	.	182.00	39.60	.
3	31	1980	1570.9	496.0	7.20	355.00	19.0	.	.	150.00	34.10	9.20	.
4	1	1980	1041.0	775.0	6.80	335.00	12.6	.	.	130.00	32.20	8.80	.
4	6	1980	2271.2	1300.0	7.20	500.00	21.3	.	.	73.00	58.20	14.20	.
4	14	1980	1589.9	515.0	7.15	.	21.0	.	.	326.00	86.70	22.30	.
4	18	1980	3448.5	703.0	7.10	.	23.0	.	.	364.00	88.80	21.60	.
4	29	1980	2082.0	180.0	7.10	1860.00	22.0	.	.	1380.00	246.40	64.20	.
5	8	1980	863.1	130.0	7.10	2352.00	20.9	.	.	1720.00	324.00	90.40	.
5	18	1980	855.5	57.0	7.00	2300.00	15.4	.	.	1220.00	319.20	95.80	.
5	31	1980	1673.2	775.0	6.70	1300.00	12.1	.	.	720.00	159.20	42.70	.
6	3	1980	613.2	191.0	6.90	1500.00	13.8	.	.	850.00	191.80	45.80	.
6	6	1980	598.1	254.0	6.80	1400.00	11.0	.	.	920.00	175.20	42.00	.
6	13	1980	412.6	207.0	6.80	1700.00	11.0	.	.	1040.00	219.20	50.90	.
6	16	1980	442.9	82.0	6.60	2200.00	9.9	.	.	1450.00	278.00	66.00	.
6	27	1980	609.5	187.0	6.60	2000.00	11.0	.	.	1180.00	252.00	67.00	.
7	2	1980	802.5	577.0	6.70	1200.00	8.5	.	.	620.00	155.00	38.00	.
7	2	1980	681.4	722.0	6.60	1700.00	11.0	.	.	1000.00	230.00	57.00	.
7	13	1980	598.1	108.0	6.60	2600.00	12.1	.	.	1620.00	359.60	80.00	.
7	23	1980	537.5	45.0	6.65	2350.00	12.1	.	.	1460.00	342.60	88.80	.

Table A2.5.7. Drainage quality data from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
8	3	1980	189.3	80.0	6.65	.	11.0	.	.	.	352.40	95.40	.
8	12	1980	836.6	521.0	6.35	1100.00	5.1	.	.	595.00	128.80	35.30	.
8	20	1980	1029.6	166.0	6.35	1300.00	5.1	.	.	788.00	184.20	51.60	.
8	22	1980	654.9	269.0	.	1400.00	.	.	.	700.00	80.00	42.00	.
8	29	1980	1514.2	2500.0	7.10	4850.00	28.0	.	.	2280.00	413.00	379.00	.
9	3	1980	2104.7	512.0	6.40	2020.00	7.6	.	.	1390.00	256.00	86.00	.
9	8	1980	1336.3	758.0	6.50	1912.00	7.7	.	.	1320.00	248.00	82.00	.
9	12	1980	3183.5	1512.0	6.60	1448.00	11.0	.	.	690.00	97.00	51.00	.
9	19	1980	7809.3	1846.0	6.60	2057.00	11.0	.	.	1320.00	255.00	92.00	.
10	1	1980	2505.9	244.0	6.40	2984.00	10.2	.	.	1750.00	410.00	136.00	.
10	22	1980	1408.2	327.0	6.30	2575.00	13.6	.	.	1900.00	340.00	127.00	.
10	25	1980	2044.1	321.0	6.10	2205.00	6.8	.	.	1250.00	266.30	116.30	.
3	2	1981	284.0	100.0	6.60	900.00	6.8	.	.	590.00	103.40	43.60	.
3	16	1981	931.0	1393.0	6.50	1200.00	6.4	.	.	700.00	115.70	49.90	.
3	23	1981	821.0	526.0	6.60	1224.00	7.2	.	.	825.00	119.60	58.50	.
3	26	1981	1158.0	723.0	6.30	600.00	5.6	.	.	272.00	46.80	25.90	.
4	2	1981	2055.0	506.0	6.70	775.00	12.0	.	.	385.00	58.50	28.90	.
4	3	1981	3710.0	1229.0	6.80	1350.00	12.0	.	.	625.00	122.20	58.80	.
4	13	1981	2839.0	993.0	.	1100.00	.	.	.	555.00	96.00	56.00	.
4	17	1981	2680.0	1199.0	6.90	1600.00	10.4	.	.	745.00	126.00	71.00	.
4	23	1981	2063.0	2237.0	7.00	1500.00	14.4	.	.	680.00	121.00	74.00	.
4	28	1981	4467.0	2436.0	7.20	1850.00	16.0	.	.	880.00	160.00	88.00	.
4	30	1981	3350.0	1200.0	6.80	2176.00	14.8	.	.	1300.00	252.00	114.00	.
5	13	1981	4115.0	100.0	6.80	2545.00	8.8	.	.	1750.00	324.00	152.00	.
6	2	1981	1586.0	473.0	6.70	2406.00	9.1	.	.	1700.00	300.00	138.00	.
6	6	1981	2555.0	67.0	6.85	2157.00	9.1	.	.	1310.00	254.00	102.00	.
6	22	1981	3482.0	1273.0	6.80	1250.00	14.4	.	.	1450.00	262.00	116.00	.
6	26	1981	3013.0	958.0	6.70	1330.00	12.8	.	.	680.00	156.00	58.00	.
6	28	1981	4130.0	2493.0	6.70	1862.00	14.4	.	.	800.00	226.00	88.00	.
7	4	1981	2600.0	385.0	6.85	2098.00	13.5	.	.	1400.00	300.00	102.00	.
7	15	1981	1283.0	120.0	6.55	3046.00	6.2	.	.	2100.00	394.00	178.60	.
7	25	1981	961.0	177.0	5.60	2564.00	.	.	.	1810.00	340.00	156.60	.
8	4	1981	863.0	66.0	6.20	2574.00	4.4	.	.	1750.00	328.00	138.00	.
8	17	1981	723.0	46.0	6.05	2485.00	5.4	.	.	1650.00	308.00	132.00	.
9	1	1981	348.0	100.0	6.00	2575.00	4.8	.	.	1570.00	284.00	136.00	.
9	7	1981	390.0	129.0	5.90	2800.00	3.2	.	.	1540.00	268.00	124.00	.
9	16	1981	265.0	22.0	5.90	3300.00	3.8	.	.	2000.00	354.00	171.00	.
9	27	1981	598.0	251.0	5.85	1775.00	2.4	.	.	980.00	192.00	86.00	.
10	1	1981	2279.0	1459.0	5.80	1700.00	2.0	.	.	850.00	180.00	82.00	.
10	4	1981	1287.0	1013.0	5.70	1700.00	2.4	.	.	1000.00	196.00	88.00	.
10	14	1981	3165.0	1597.0	6.25	2150.00	4.8	.	.	1280.00	228.00	106.00	.
10	18	1981	10569.0	1210.0	6.25	2000.00	4.0	.	.	1310.00	232.00	100.00	.
11	9	1981	4997.0	108.0	6.65	4500.00	11.2	.	.	2800.00	410.00	320.00	.
3	25	1982	2839.0	.	6.55	900.00	7.2	.	.	470.00	73.00	53.40	.
4	1	1982	3975.0	.	6.18	1325.00	5.2	.	.	800.00	105.00	76.40	.
4	13	1982	4728.0	.	6.60	690.00	7.2	.	.	320.00	52.00	34.40	.
4	15	1982	3774.0	.	6.60	750.00	9.0	.	.	390.00	63.80	41.60	.
4	19	1982	1961.0	.	6.60	1125.00	7.7	.	.	705.00	129.00	83.00	.
4	23	1982	2438.0	.	6.50	1500.00	5.4	.	.	950.00	188.00	110.00	.
4	28	1982	3721.0	.	6.50	1875.00	7.7	.	.	1000.00	160.00	128.00	.
5	3	1982	466.0	.	6.40	2290.00	5.4	.	.	.	240.00	164.00	.
5	7	1982	4527.0	.	6.90	1960.00	17.4	.	.	975.00	133.00	125.00	.
5	11	1982	2782.0	.	6.90	1800.00	21.2	.	.	1040.00	156.00	122.00	.
5	14	1982	750.0	.	6.80	2500.00	16.2	.	.	1465.00	202.00	160.00	.
5	18	1982	681.0	.	6.80	2450.00	18.9	.	.	1340.00	206.00	160.00	.
5	21	1982	3062.0	.	6.25	3200.00	.	.	.	2400.00	292.00	163.00	.
6	2	1982	1298.0	.	6.10	3600.00	9.9	.	.	2750.00	340.00	238.00	.
6	11	1982	.	.	7.30	3150.00	.	.	.	.	306.00	183.00	.
6	12	1982	.	.	7.70	3450.00	.	.	.	.	324.00	178.00	.
6	18	1982	148.0	.	6.50	2500.00	9.9	.	.	1500.00	248.00	133.00	.
7	2	1982	2824.0	.	7.40	2750.00	81.6	.	.	1300.00	297.00	136.00	.
7	8	1982	7291.0	.	6.35	2700.00	11.7	.	.	2000.00	354.00	150.00	.
7	26	1982	6261.0	.	6.37	2705.00	10.6	.	.	1675.00	298.00	149.00	.
8	30	1982	7779.0	.	6.91	2087.00	50.3	.	.	1175.00	298.00	126.00	.
10	4	1982	20301.0	.	6.65	2145.00	14.6	.	.	1357.00	223.00	126.00	.

Table A2.5.7. Drainage quality data from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
11	1	1982	1257.0	.	6.20	2825.00	3.6	.	.	.	248.00	160.00	.
11	22	1982	.	.	6.10	2140.00	4.5	.	.	1600.00	270.00	116.00	.
3	18	1983	8206.0	.	5.95	1775.00	3.0	.	.	1120.00	.	.	.
4	14	1983	7414.0	.	6.66	1070.00	16.3	.	.	510.00	.	.	.
5	15	1983	5109.0	.	6.41	1574.00	6.9	.	.	1000.00	.	.	.
6	16	1983	2717.0	.	6.19	1782.00	6.9	.	.	1300.00	.	.	.
7	14	1983	7748.0	.	5.90	2008.00	5.0	.	.	1680.00	.	.	.
8	14	1983	5553.0	.	5.41	2095.00	3.4	.	.	1750.00	.	.	.
9	15	1983	4565.0	.	5.64	2123.00	2.5	.	.	1940.00	.	.	.
10	13	1983	20303.0	.	5.70	2422.00	3.8	.	.	1880.00	.	.	.
11	9	1983	3925.0	.	5.90	2300.00	3.6	.	.	2600.00	.	.	.
11	30	1983	2589.0	.	5.50	2175.00	2.7	.	.	1960.00	.	.	.
2	24	1984	2907.0	.	4.80	1900.00	0.9	.	.	2040.00	236.30	189.90	300
3	15	1984	4375.0	.	5.40	2375.00	2.7	.	.	2000.00	288.30	274.40	301
4	15	1984	15221.0	.	6.40	878.00	10.4	.	.	580.00	104.00	70.60	302
5	16	1984	8861.0	.	6.16	1190.00	6.9	.	.	980.00	165.60	109.40	303
6	13	1984	17886.0	.	5.52	1519.00	2.6	.	.	1310.00	205.90	139.60	304
7	14	1984	6064.0	.	5.05	2130.00	2.5	.	.	2480.00	.	.	305
8	15	1984	4478.0	.	5.34	2050.00	3.0	.	.	1800.00	267.80	142.90	306
9	12	1984	1340.0	.	4.96	2340.00	2.3	.	.	1760.00	317.10	220.00	307
10	13	1984	10579.0	.	5.02	1860.00	2.1	.	.	1180.00	259.10	149.50	308
11	14	1984	2918.0	.	4.96	2190.00	1.6	.	.	2700.00	381.00	450.00	309
12	12	1984	3459.0	.	4.93	1585.00	0.8	.	.	1800.00	242.00	290.00	310
4	2	1985	2394.0	.	6.60	800.00	17.9	.	.	470.00	75.80	54.20	300
4	16	1985	16699.0	.	5.94	1360.00	9.0	.	.	1010.00	144.00	121.60	301
5	14	1985	6828.0	.	5.25	2400.00	3.9	.	.	2000.00	220.00	210.00	302
6	14	1985	26461.0	.	5.57	2150.00	4.0	.	.	1736.00	209.00	169.00	303
7	15	1985	7630.0	.	5.31	2300.00	4.3	.	.	1900.00	220.00	186.00	304
8	13	1985	5761.0	.	5.47	2125.00	4.0	.	.	1750.00	226.00	153.40	305
9	13	1985	18187.0	.	5.39	2325.00	3.8	.	.	1850.00	294.00	141.80	306
10	15	1985	17222.0	.	5.44	2425.00	4.7	.	.	1650.00	239.00	160.00	307
11	15	1985	1949.0	.	5.26	2675.00	2.6	.	.	1450.00	322.00	224.00	308
3	20	1986	401.0	.	5.25	3100.00	2.2	.	.	2900.00	366.00	334.00	309
3	23	1986	17998.0	.	6.30	975.00	9.5	.	.	700.00	90.00	67.20	300
4	16	1986	13743.0	.	6.09	1175.00	7.4	.	.	700.00	126.00	84.00	301
5	14	1986	10995.0	.	5.70	1650.00	4.4	.	.	1350.00	214.00	144.00	302
6	11	1986	3509.0	.	5.29	2425.00	3.6	.	.	1650.00	234.00	173.00	303
7	12	1986	7104.0	.	5.40	1775.00	3.7	.	.	1100.00	180.00	130.00	304
8	13	1986	4273.0	.	5.17	2200.00	3.0	.	.	1550.00	228.00	155.60	305
9	14	1986	18682.0	.	5.21	2000.00	3.0	.	.	1400.00	224.00	159.80	306
10	16	1986	4557.0	.	5.11	2900.00	2.1	.	.	1825.00	258.00	227.00	307
11	16	1986	7899.0	.	5.09	3150.00	2.7	.	.	1900.00	290.00	208.00	308
3	22	1987	8656.0	.	5.30	1125.00	7.1	.	.	730.00	126.00	80.00	300
4	15	1987	3861.0	.	5.73	1450.00	4.6	.	.	900.00	118.00	100.00	301
5	16	1987	24103.0	.	5.13	2300.00	4.5	.	.	1770.00	319.00	256.00	302
6	16	1987	2048.0	.	4.81	3550.00	3.0	.	.	2640.00	356.00	328.00	303
7	15	1987	18842.0	.	5.05	2325.00	5.7	.	.	1780.00	226.00	187.00	304
8	15	1987	8501.0	.	4.91	2425.00	3.4	.	.	1750.00	206.00	172.00	305
9	16	1987	12195.0	.	5.01	2650.00	2.8	.	.	1615.00	200.00	158.00	306
10	13	1987	2142.0	.	4.81	3825.00	2.1	.	.	2680.00	323.00	307.00	307
11	10	1987	1064.0	.	4.71	3350.00	1.8	.	.	2700.00	288.00	278.00	308
3	28	1988	8653.0	.	6.00	900.00	11.1	.	.	414.00	88.00	48.00	300
4	17	1988	7297.0	.	5.69	1575.00	6.3	.	.	950.00	136.00	100.00	301
5	12	1988	10624.0	.	5.17	1985.00	3.4	.	.	1270.00	172.00	160.00	302
6	12	1988	1654.0	.	4.86	3500.00	2.9	.	.	2440.00	344.00	324.00	303
7	15	1988	995.0	.	4.91	3175.00	3.5	.	.	2230.00	278.00	286.00	304
8	16	1988	45053.0	.	5.26	1910.00	2.5	.	.	1240.00	176.00	130.00	305
9	16	1988	8134.0	.	5.07	2300.00	2.7	.	.	1360.00	202.00	170.00	306
10	18	1988	3376.0	.	5.12	2000.00	3.0	.	.	1330.00	158.00	144.00	307
11	15	1988	6654.0	.	5.21	1625.00	3.6	.	.	1250.00	142.00	131.00	308
12	5	1988	1681.0	.	4.80	3300.00	3.2	.	.	3150.00	298.00	350.00	309
4	7	1989	23758.0	.	6.15	650.00	9.9	.	.	425.00	70.00	46.00	300
4	24	1989	7638.0	.	5.91	1075.00	6.5	.	.	650.00	100.00	58.00	301
5	16	1989	9284.0	.	5.18	1593.00	3.9	.	.	1090.00	128.00	108.00	302
6	17	1989	29326.0	.	4.98	1821.00	3.3	.	.	1375.00	156.00	128.00	303

Table A2.5.7. Drainage quality data from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
7	17	1989	2967.0	.	4.81	2800.00	2.2	.	.	1950.00	306.00	210.00	304
8	14	1989	1328.0	.	4.94	2900.00	3.1	.	.	1970.00	248.00	222.00	305
9	14	1989	17888.0	.	4.90	2400.00	2.7	.	.	1600.00	252.00	166.00	306
10	16	1989	1835.0	.	4.83	2775.00	0.6	.	.	1800.00	218.00	186.00	307
11	10	1989	900.0	.	5.00	2400.00	2.8	.	.	1820.00	240.00	182.00	308

Table A2.5.8. Drainage quality data for metals from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 3.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
4	26	1978	52.00	12.00	.	0.023	0.060	.	.	0.700	0.050	.
5	25	1978	122.00	15.00	.	0.013	0.042	0.050	0.020	0.040	0.090	.
6	16	1978	372.00	21.00	.	0.026	0.185	0.012	0.059	0.060	0.240	.
7	5	1978	102.00	23.00	.	0.200	0.321	0.017	0.037	0.020	0.170	.
7	13	1978	207.00	32.00	.	0.049	0.405	0.030	0.092	0.030	0.240	.
7	19	1978	.	27.00	.	0.092	0.235	0.020	0.060	0.010	0.150	.
8	2	1978	60.00	21.00	.	0.080	0.210	0.010	0.025	0.030	0.160	.
8	4	1978	88.00	26.00	.	0.080	0.320	0.020	0.044	0.020	0.160	.
8	10	1978	188.00	49.00	.	0.060	0.565	0.090	0.073	0.010	0.280	.
8	16	1978	46.00	17.70	.	0.117	0.207	0.017	0.020	0.070	0.080	.
8	23	1978	45.00	17.00	.	0.100	0.190	0.016	0.020	0.070	0.080	.
8	25	1978	67.00	23.00	.	0.080	0.250	0.010	0.030	0.040	0.100	.
9	13	1978	74.50	27.50	.	0.085	0.300	0.020	0.040	0.040	0.120	.
3	19	1979	50.10	15.90	.	0.080	0.790	0.050	0.110	0.080	0.100	.
3	27	1979	73.80	22.20	.	0.060	0.780	0.070	0.110	0.020	0.120	.
4	13	1979	29.60	10.10	.	0.106	0.296	0.024	0.040	0.130	0.060	.
4	17	1979	32.20	10.80	.	0.088	0.292	0.026	0.050	0.090	0.080	.
4	19	1979	31.00	8.60	.	0.072	0.238	0.020	0.050	0.050	0.070	.
4	19	1979	41.50	9.30	.	0.064	0.220	0.020	0.040	0.120	0.070	.
4	20	1979	42.00	11.00	.	0.069	0.278	0.032	0.050	0.060	0.090	.
4	22	1979	121.60	24.80	.	0.056	0.920	0.051	0.120	0.030	0.170	.
5	7	1979	131.60	27.60	.	0.046	0.890	0.047	0.110	0.030	0.180	.
5	10	1979	62.40	17.00	.	0.068	0.520	0.031	0.070	0.030	0.110	.
5	20	1979	130.80	29.60	.	0.060	0.750	0.047	0.120	0.020	0.160	.
5	22	1979	60.80	16.80	.	0.090	0.440	0.023	0.080	0.040	0.080	.
5	23	1979	92.20	24.40	.	0.058	0.620	0.040	0.080	0.040	0.120	.
6	1	1979	59.80	17.40	.	0.100	0.460	0.023	0.060	0.080	0.100	.
6	6	1979	33.40	15.60	.	0.081	0.370	0.021	0.050	0.030	0.080	.
6	10	1979	50.80	38.80	.	0.100	0.310	0.010	0.040	0.020	0.050	.
6	12	1979	36.00	22.80	.	0.071	0.410	0.013	0.070	0.010	0.070	.
6	15	1979	42.40	23.40	.	0.061	0.470	0.013	0.050	0.010	0.080	.
6	21	1979	42.00	19.00	.	0.070	0.340	0.021	0.040	0.020	0.060	.
6	30	1979	68.00	25.00	.	0.050	0.490	0.032	0.070	0.020	0.240	.
7	10	1979	105.00	37.00	.	0.040	1.700	0.046	0.090	0.010	0.160	.
7	30	1979	31.00	19.00	.	0.090	0.580	0.040	0.060	0.010	0.100	.
8	4	1979	53.00	33.00	.	0.080	0.980	0.045	0.105	0.030	0.150	.
8	13	1979	150.80	55.20	.	0.100	1.460	0.090	0.180	0.040	0.230	.
8	28	1979	50.00	27.20	.	0.170	1.190	0.080	0.140	0.010	0.200	.
9	1	1979	36.60	23.20	.	0.150	1.280	0.062	0.110	0.010	0.130	.
9	5	1979	69.00	36.60	.	0.140	1.640	0.090	0.150	0.010	0.170	.
9	10	1979	77.40	33.00	.	0.140	1.440	0.110	0.130	0.020	0.170	.
9	13	1979	80.60	32.80	.	0.150	1.630	0.090	0.130	0.030	0.180	.
9	19	1979	144.00	56.80	.	0.180	2.570	0.140	0.260	0.020	0.260	.
10	17	1979	122.80	48.00	.	0.190	2.560	0.140	0.270	0.010	0.290	.
10	19	1979	44.00	18.80	.	0.190	1.630	0.070	0.140	0.030	0.130	.
10	31	1979	54.20	23.90	.	0.220	2.840	0.140	0.195	0.020	0.190	.
11	1	1979	60.00	25.60	.	0.190	2.540	0.120	0.180	0.020	0.150	.
3	31	1980	7.10	5.90	.	0.090	0.960	0.045	0.070	0.110	0.070	.
4	1	1980	10.60	6.20	.	0.120	0.810	0.060	0.060	0.070	0.050	.
4	6	1980	14.00	8.80	.	0.110	1.000	0.040	0.090	0.040	0.090	.
4	14	1980	23.60	12.00	.	0.105	1.490	0.080	0.100	0.030	0.120	.
4	18	1980	24.80	11.80	.	0.120	2.100	0.100	0.130	0.020	0.180	.
4	29	1980	87.20	30.80	.	0.290	4.310	0.210	0.300	0.010	0.350	.
5	8	1980	136.00	42.60	.	0.260	4.290	0.240	0.370	0.020	0.340	.
5	18	1980	152.60	43.00	.	0.230	4.620	0.260	0.430	0.030	0.380	.
5	31	1980	57.90	24.30	.	0.270	3.140	0.180	0.250	0.010	0.230	.
6	3	1980	57.20	27.40	.	0.250	3.790	0.210	0.280	0.010	0.260	.
6	6	1980	55.00	25.60	.	0.280	3.050	0.180	0.240	0.020	0.210	.
6	13	1980	73.90	30.30	.	0.315	3.370	0.195	0.290	0.030	0.240	.
6	16	1980	97.60	35.40	.	0.320	4.520	0.260	0.390	0.020	0.300	.
6	27	1980	89.00	35.00	.	0.230	3.490	0.230	0.360	0.020	0.280	.
7	2	1980	38.00	20.00	.	0.430	3.020	0.180	0.280	0.020	0.240	.
7	2	1980	66.00	29.00	.	0.460	3.760	0.270	0.370	0.040	0.300	.
7	13	1980	133.60	51.00	.	0.560	5.350	0.280	0.480	0.020	0.410	.
7	23	1980	201.60	49.90	.	0.505	5.195	0.300	0.480	0.020	0.450	.
8	3	1980	162.00	50.40	.	0.530	6.240	0.310	0.540	0.020	0.510	.
8	12	1980	44.60	21.40	.	0.670	3.570	0.200	0.300	0.060	0.340	.

Table A2.5.8. Drainage quality data for metals from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 3.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
8	20	1980	46.20	27.80	.	1.170	7.630	0.400	0.560	0.010	0.650	.
8	22	1980	42.00	20.00	.	1.360	7.040	0.410	0.590	0.040	0.620	.
8	29	1980	591.00	48.00	.	0.750	5.340	0.310	0.360	0.060	0.430	.
9	3	1980	123.00	40.00	.	1.650	12.550	0.670	0.760	0.030	0.950	.
9	8	1980	116.00	39.00	.	2.120	13.890	0.770	0.810	0.020	1.100	.
9	12	1980	80.00	25.00	.	1.620	9.450	0.550	0.530	0.040	0.760	.
9	19	1980	128.00	39.00	.	2.660	12.780	0.670	0.720	0.050	0.900	.
10	1	1980	181.00	.	.	2.720	17.700	0.800	1.010	0.030	1.040	.
10	22	1980	178.00	52.00	.	2.330	12.300	0.650	0.900	0.020	0.890	.
10	25	1980	125.30	43.70	.	3.970	19.500	0.940	1.090	0.030	1.290	.
3	2	1981	42.20	.	.	1.100	13.000	0.620	0.550	.	.	.
3	16	1981	55.90	.	.	1.300	23.000	0.950	0.800	.	.	.
3	23	1981	60.70	.	.	1.410	21.000	0.920	0.870	.	.	.
3	26	1981	19.50	.	.	0.910	8.200	0.390	0.380	.	.	.
4	2	1981	22.10	.	.	1.590	15.000	0.690	0.550	.	.	.
4	3	1981	66.30	.	.	5.150	33.000	1.450	1.150	.	.	.
4	13	1981	33.00	.	.	3.630	17.000	0.800	0.660	.	.	.
4	17	1981	56.00	.	.	3.730	19.000	0.860	0.710	.	.	.
4	23	1981	56.00	.	.	2.300	13.000	0.620	0.550	.	.	.
4	28	1981	65.00	.	.	2.760	17.000	0.830	0.730	.	.	.
4	30	1981	92.00	.	.	3.680	25.800	1.230	1.070	.	.	.
5	13	1981	120.00	.	.	4.010	22.000	1.440	1.310	.	.	.
6	2	1981	118.00	.	.	3.250	21.000	1.270	1.170	.	.	.
6	6	1981	100.00	.	.	3.290	18.000	1.190	1.060	.	.	.
6	22	1981	128.00	.	.	2.840	19.000	1.110	0.880	.	.	.
6	26	1981	42.00	.	.	4.050	19.840	0.950	0.630	.	.	.
6	28	1981	70.00	.	.	4.880	20.000	1.230	0.800	.	.	.
7	4	1981	88.80	.	.	4.850	25.000	1.150	0.960	.	.	.
7	15	1981	158.60	.	.	6.200	37.000	1.650	1.530	.	.	.
7	25	1981	134.60	.	.	5.400	30.000	1.430	1.360	.	.	.
8	4	1981	118.00	.	.	5.500	30.000	1.470	1.320	.	.	.
8	17	1981	108.00	52.00	.	5.270	25.000	1.300	1.440	.	.	.
9	1	1981	98.00	.	.	5.450	26.000	1.240	1.610	.	.	.
9	7	1981	88.00	.	.	6.240	25.000	1.330	1.610	.	.	.
9	16	1981	115.00	.	.	10.010	41.500	2.040	2.440	.	.	.
9	27	1981	50.00	.	.	6.300	22.000	1.140	1.330	.	.	.
10	1	1981	32.00	.	.	10.670	34.000	1.710	1.620	.	.	.
10	4	1981	34.00	28.00	.	12.000	36.000	1.880	1.730	.	.	.
10	14	1981	64.00	40.00	.	9.000	39.000	1.940	1.550	.	.	.
10	18	1981	56.00	38.00	.	9.000	36.000	1.880	1.430	.	.	.
11	9	1981	240.00	84.00	.	10.980	62.000	2.480	2.250	.	.	.
3	25	1982	20.00	14.00	.	4.140	19.000	0.940	0.690	.	.	.
4	1	1982	28.00	18.00	.	6.275	31.000	1.470	1.490	.	.	.
4	13	1982	14.00	10.00	.	2.860	13.000	0.670	0.490	.	.	.
4	15	1982	16.00	12.00	.	2.120	13.000	0.580	0.420	.	.	.
4	19	1982	29.00	20.00	.	3.395	19.500	1.010	0.640	.	.	.
4	23	1982	80.00	24.00	.	4.450	26.000	1.340	0.860	.	.	.
4	28	1982	68.00	32.00	.	4.520	25.000	1.380	0.950	.	.	.
5	3	1982	104.00	42.00	.	6.490	40.000	2.030	1.360	.	.	.
5	7	1982	109.00	25.00	.	3.630	20.300	1.105	0.740	.	.	.
5	11	1982	64.00	28.00	.	4.410	24.000	1.400	0.890	.	.	.
5	14	1982	110.00	40.00	.	7.250	44.000	2.410	1.650	.	.	.
5	18	1982	128.00	40.00	.	6.800	42.000	2.270	1.580	.	.	.
5	21	1982	93.00	53.00	.	10.160	70.000	2.560	2.010	0.040	2.810	.
6	2	1982	129.50	63.50	.	11.405	79.500	3.015	2.580	0.060	3.360	.
6	11	1982	139.00	52.00	.	5.230	45.000	1.920	1.480	0.050	1.960	.
6	12	1982	174.00	46.00	.	3.330	29.000	1.300	0.970	0.050	1.270	.
6	18	1982	65.00	39.00	.	6.950	43.000	1.870	1.520	0.040	1.890	.
7	2	1982	178.00	34.00	.	2.290	21.090	0.770	0.560	0.030	0.780	.
7	8	1982	108.00	48.00	.	8.850	67.000	2.300	1.560	0.020	2.560	.
7	26	1982	77.00	51.00	.	7.570	79.500	3.240	1.610	0.390	3.030	.
8	30	1982	70.00	40.00	.	6.240	45.000	1.710	1.150	0.130	1.980	.
10	4	1982	59.00	44.00	.	9.520	80.000	3.220	1.620	0.130	2.940	.
11	1	1982	72.00	54.00	.	9.490	91.000	3.650	1.870	0.020	3.520	.
11	22	1982	64.00	.	.	13.750	91.000	3.340	2.190	.	.	.
3	18	1983	.	.	.	9.010	65.000	2.590	2.030	.	.	.
4	14	1983	.	.	.	5.010	41.000	1.410	1.060	.	.	.

Table A2.5.8. Drainage quality data for metals from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 3.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
5	15	1983	.	.	.	6.440	58.000	2.140	1.690	.	.	.
6	16	1983	.	.	.	5.360	43.000	1.720	1.280	.	.	.
7	14	1983	.	.	.	12.030	54.000	2.830	2.320	.	.	.
8	14	1983	.	.	.	16.750	71.000	3.300	2.880	.	.	.
9	15	1983	.	.	.	16.500	75.000	3.590	3.080	.	.	.
10	13	1983	.	.	.	19.000	78.000	3.770	3.140	.	.	.
11	9	1983	.	.	.	20.400	99.000	4.440	3.320	.	.	.
11	30	1983	.	.	.	20.200	89.600	4.050	3.040	.	.	.
2	24	1984	50.70	.	.	30.600	102.000	4.500	3.840	.	.	300
3	15	1984	65.10	.	.	35.000	144.000	7.000	4.890	.	.	301
4	15	1984	19.90	.	.	7.490	34.000	1.680	1.300	.	.	302
5	16	1984	31.20	.	.	12.000	51.000	2.400	1.820	.	.	303
6	13	1984	40.80	.	.	17.500	69.400	3.150	2.290	.	.	304
7	14	1984	57.40	.	.	19.000	121.300	4.000	3.200	.	.	305
8	15	1984	49.60	.	.	16.100	73.760	4.880	2.360	.	.	306
9	12	1984	70.50	.	.	26.500	105.000	7.120	3.440	.	.	307
10	13	1984	43.20	.	.	34.000	92.000	6.830	3.010	.	.	308
11	14	1984	84.70	.	.	38.600	138.000	5.380	4.180	.	.	309
12	12	1984	62.00	.	.	26.500	108.800	4.840	3.490	.	.	310
4	2	1985	14.50	.	.	5.800	27.000	1.070	0.860	.	.	300
4	16	1985	28.60	.	.	18.500	70.000	2.800	2.090	.	.	301
5	14	1985	53.00	.	.	24.000	100.000	4.710	2.830	.	.	302
6	14	1985	46.30	.	.	27.000	103.000	4.140	3.210	.	.	303
7	15	1985	51.00	.	.	23.000	99.000	4.310	3.490	.	.	304
8	13	1985	48.00	.	.	22.400	89.000	4.240	3.170	.	.	305
9	13	1985	44.00	.	.	39.000	100.600	4.820	3.560	.	.	306
10	15	1985	43.00	.	.	37.600	102.800	4.660	3.010	.	.	307
11	15	1985	60.00	.	.	41.800	133.000	6.020	4.020	.	.	308
3	20	1986	78.00	.	.	52.400	220.000	7.900	5.200	.	.	309
3	23	1986	13.00	.	.	17.000	48.000	2.120	1.310	.	.	300
4	16	1986	18.00	.	.	17.000	51.000	2.540	1.590	.	.	301
5	14	1986	28.00	.	.	29.000	120.000	5.220	2.330	.	.	302
6	11	1986	49.00	.	.	31.000	108.000	4.360	3.690	.	.	303
7	12	1986	32.00	.	.	27.000	80.000	3.350	2.690	.	.	304
8	13	1986	40.00	.	.	37.000	95.000	4.200	2.280	.	.	305
9	14	1986	36.00	.	.	41.000	98.000	5.100	2.360	.	.	306
10	16	1986	48.00	.	.	48.000	142.000	5.750	4.150	.	.	307
11	16	1986	62.00	.	.	72.000	161.000	9.100	6.100	.	.	308
3	22	1987	22.00	.	.	20.200	60.000	2.780	1.930	.	.	300
4	15	1987	22.00	.	.	28.000	68.000	3.070	2.240	.	.	301
5	16	1987	56.00	.	.	51.000	129.000	5.700	4.700	.	.	302
6	16	1987	70.00	.	.	63.000	197.000	6.000	4.100	.	.	303
7	15	1987	45.00	.	.	53.000	131.000	5.200	3.950	.	.	304
8	15	1987	42.00	.	.	43.000	121.000	4.800	3.720	.	.	305
9	16	1987	36.00	.	.	42.000	105.000	4.510	3.120	.	.	306
10	13	1987	70.00	.	.	71.000	192.000	7.730	5.430	.	.	307
11	10	1987	64.00	.	.	59.000	174.000	6.880	4.930	.	.	308
3	28	1988	12.00	.	.	11.000	32.000	1.600	1.600	.	.	300
4	17	1988	22.00	.	.	24.000	90.000	3.800	4.100	.	.	301
5	12	1988	38.00	.	.	37.000	129.000	5.200	6.100	.	.	302
6	12	1988	67.00	.	.	70.000	178.500	10.200	11.500	.	.	303
7	15	1988	58.00	.	.	59.000	161.000	8.800	11.200	.	.	304
8	16	1988	34.00	.	.	37.000	125.000	4.700	4.200	.	.	305
9	16	1988	44.00	.	.	41.000	145.000	5.700	5.300	.	.	306
10	18	1988	32.00	.	.	33.000	93.000	5.000	5.500	.	.	307
11	15	1988	26.00	.	.	32.500	87.000	4.700	5.000	.	.	308
12	5	1988	62.00	.	.	73.000	238.000	11.900	10.000	.	.	309
4	7	1989	12.00	.	.	12.700	32.000	1.900	1.700	.	.	300
4	24	1989	12.00	.	.	16.000	46.000	2.500	2.500	.	.	301
5	16	1989	20.00	.	.	28.000	79.000	4.500	5.300	.	.	302
6	17	1989	30.00	.	.	41.000	119.000	6.800	5.000	.	.	303
7	17	1989	46.00	.	.	66.000	185.000	8.700	9.100	.	.	304
8	14	1989	48.00	.	.	61.000	240.000	8.200	8.200	.	.	305
9	14	1989	36.00	.	.	56.000	148.000	9.500	6.700	.	.	306
10	16	1989	46.00	.	.	47.000	161.000	12.100	8.400	.	.	307
11	10	1989	40.00	.	.	42.000	137.000	14.800	7.700	.	.	308

Table A2.5.9. Drainage quality data for additional parameters from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 3.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
4	26	1978	.	.	.	.	.	4.5	.
5	25	1978	.	.	.	.	.	.	.
6	16	1978	.	.	.	.	.	6.0	.
7	5	1978	78.00	.	.	.	.	9.0	.
7	13	1978	34.00	23.00	0.70	0.08	.	.	.
7	19	1978	35.00	24.00	0.55	0.08	.	.	.
8	2	1978	12.00	19.00	0.80	0.08	.	22.0	.
8	4	1978	17.00	26.00	0.06	0.08	.	22.0	.
8	10	1978	48.00	10.50	.	0.45	17.00	24.0	.
8	16	1978	10.00	9.40	.	0.08	.	20.3	.
8	23	1978	7.00	17.00	0.64	0.08	.	.	.
8	25	1978	11.00	24.00	.	0.10	.	24.0	.
9	13	1978	7.00	13.00	0.08	0.08	.	14.0	.
3	19	1979	.	.	.	.	9.30	.	.
3	27	1979	.	.	.	.	.	.	.
4	13	1979	5.00	.	.	.	.	.	.
4	17	1979	8.00	.	.	.	5.30	.	.
4	19	1979	4.00	1.50	.	.	11.00	.	.
4	19	1979	5.00	1.50	0.10	.	.	.	.
4	20	1979	4.00	3.40	0.10	.	4.70	.	.
4	22	1979	11.00	6.90	0.04	.	.	.	.
5	7	1979	.	.	.	.	.	.	.
5	10	1979	10.00	.	.	.	.	.	.
5	20	1979	13.00	.	.	.	5.90	.	.
5	22	1979	6.00	.	.	.	7.60	.	.
5	23	1979	7.00	.	.	.	4.90	.	.
6	1	1979	6.00	.	.	.	6.70	.	.
6	6	1979	4.00	.	.	.	.	.	.
6	10	1979	4.00	.	.	.	.	20.0	.
6	12	1979	5.00	.	.	.	.	.	.
6	15	1979	7.00	.	.	.	.	20.0	.
6	21	1979	4.00	.	.	.	.	16.5	.
6	30	1979	5.00	.	.	.	5.60	12.0	.
7	10	1979	12.00	.	.	.	5.30	18.0	.
7	30	1979	1.00	.	.	.	7.70	23.0	.
8	4	1979	4.00	.	.	.	9.60	23.0	.
8	13	1979	10.00	.	.	.	5.20	14.0	.
8	28	1979	14.00	.	.	.	8.20	21.0	.
9	1	1979	4.00	.	.	.	6.10	20.0	.
9	5	1979	5.00	.	.	.	6.20	18.0	.
9	10	1979	6.00	.	.	.	5.70	18.0	.
9	13	1979	6.00	.	.	.	5.30	17.0	.
9	19	1979	12.00	.	.	.	5.00	15.0	.
10	17	1979	.	.	.	.	4.60	.	.
10	19	1979	3.00	.	.	.	5.90	.	.
10	31	1979	3.00	.	.	.	4.50	.	.
11	1	1979	.	.	.	.	4.50	12.0	.
3	31	1980	2.00	.	.	.	4.00	10.7	.
4	1	1980	2.00	.	.	.	3.10	.	.
4	6	1980	2.00	.	.	.	.	.	.
4	14	1980	7.00	.	.	.	.	.	.
4	18	1980	4.00	.	.	.	2.90	.	.
4	29	1980	10.00	.	.	.	.	11.0	.
5	8	1980	13.00	.	.	.	3.50	11.5	.
5	18	1980	21.00	.	.	.	.	19.0	.
5	31	1980	36.00	.	.	.	7.10	.	.
6	3	1980	46.00	.	.	.	4.90	.	.
6	6	1980	39.00	.	.	.	.	.	.
6	13	1980	28.00	.	.	.	.	.	.
6	16	1980	30.00	.	.	.	.	.	.
6	27	1980	18.00	.	.	.	8.10	.	.
7	2	1980	35.00	.	.	.	.	.	.
7	2	1980	27.00	.	.	.	4.90	.	.
7	13	1980	30.00	.	.	.	.	.	.
7	23	1980	23.00	.	.	.	.	.	.
8	3	1980	.	.	.	.	5.90	.	.

Table A2.5.9. Drainage quality data for additional parameters from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 3.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
8	12	1980	12.00	.	.	.	8.10	.	.
8	20	1980	17.00	.	.	.	.	.	.
8	22	1980	21.00	.	.	.	.	.	.
8	29	1980	9.3E2	.	.	.	9.50	.	.
9	3	1980	50.00	.	.	.	.	.	.
9	8	1980	44.00	.	.	.	.	15.0	.
9	12	1980	52.00	.	.	.	4.00	18.0	.
9	19	1980	77.00	.	.	.	.	13.0	.
10	1	1980	77.00	.	.	.	3.40	14.0	.
10	22	1980	1.2E2	.	.	.	4.30	10.0	.
10	25	1980	56.00	.	.	.	.	10.0	.
3	2	1981	15.00	.	.	.	.	.	.
3	16	1981	.	.	.	.	.	.	.
3	23	1981	.	.	.	.	.	9.0	.
3	26	1981	.	.	.	.	7.20	.	.
4	2	1981	.	.	.	.	.	.	.
4	3	1981	.	.	.	.	.	.	.
4	13	1981	.	.	.	.	.	.	.
4	17	1981	.	.	.	.	.	.	.
4	23	1981	.	.	.	.	.	.	.
4	28	1981	.	.	.	.	3.20	.	.
4	30	1981	.	.	.	.	4.00	10.0	.
5	13	1981	.	.	.	.	3.60	17.0	.
6	2	1981	.	.	.	.	.	17.0	.
6	6	1981	.	.	.	.	4.10	18.0	.
6	22	1981	.	.	.	.	.	16.0	.
6	26	1981	.	.	.	.	.	15.0	.
6	28	1981	40.00	.	.	.	3.50	15.0	.
7	4	1981	.	.	.	.	.	22.0	.
7	15	1981	.	.	.	.	.	20.0	.
7	25	1981	23.00	.	.	.	3.10	21.0	.
8	4	1981	.	.	.	.	.	19.0	.
8	17	1981	.	.	.	.	.	22.0	.
9	1	1981	.	.	.	.	7.40	.	.
9	7	1981	.	.	.	.	.	17.0	.
9	16	1981	.	.	.	.	.	.	.
9	27	1981	.	.	.	.	4.70	.	.
10	1	1981	.	.	.	.	.	.	.
10	4	1981	.	.	.	.	.	.	.
10	14	1981	.	.	.	.	3.40	.	.
10	18	1981	.	.	.	.	.	.	.
11	9	1981	.	.	.	.	3.50	.	.
3	25	1982	.	.	.	.	.	.	.
4	1	1982	.	.	.	.	.	.	.
4	13	1982	.	.	.	.	.	.	.
4	15	1982	.	.	.	.	.	.	.
4	19	1982	.	.	.	.	.	.	.
4	23	1982	.	.	.	.	.	.	.
4	28	1982	.	.	.	.	.	.	.
5	3	1982	.	.	.	.	.	.	.
5	7	1982	.	.	.	.	.	.	.
5	11	1982	.	.	.	.	.	.	.
5	14	1982	.	.	.	.	.	.	.
5	18	1982	.	.	.	.	.	.	.
5	21	1982	.	.	.	.	.	.	.
6	2	1982	.	.	.	.	.	.	.
6	11	1982	.	.	.	.	.	.	.
6	12	1982	.	.	.	.	.	.	.
6	18	1982	.	.	.	.	.	.	.
7	2	1982	.	.	.	.	.	.	.
7	8	1982	.	.	.	.	.	.	.
7	26	1982	.	.	.	.	.	.	.
8	30	1982	.	.	.	.	.	.	.
10	4	1982	.	.	.	.	.	.	.
11	1	1982	.	.	.	.	.	.	.
11	22	1982	.	.	.	.	.	.	.
3	18	1983	.	.	.	.	.	.	.

Table A2.5.9. Drainage quality data for additional parameters from Duluth Complex test pile FL3, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 3.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
4	14	1983	.	.	.	.	.	.	.
5	15	1983	.	.	.	.	.	.	.
6	16	1983	.	.	.	.	.	.	.
7	14	1983	.	.	.	.	.	.	.
8	14	1983	.	.	.	.	.	.	.
9	15	1983	.	.	.	.	.	.	.
10	13	1983	.	.	.	.	.	.	.
11	9	1983	.	.	.	.	.	.	.
11	30	1983	.	.	.	.	.	.	.
2	24	1984	.	.	.	.	.	.	300
3	15	1984	.	.	.	.	.	.	301
4	15	1984	.	.	.	.	.	.	302
5	16	1984	.	.	.	.	.	.	303
6	13	1984	.	.	.	.	.	.	304
7	14	1984	.	.	.	.	.	.	305
8	15	1984	.	.	.	.	.	.	306
9	12	1984	.	.	.	.	.	.	307
10	13	1984	.	.	.	.	.	.	308
11	14	1984	.	.	.	.	.	.	309
12	12	1984	.	.	.	.	.	.	310
4	2	1985	.	.	.	.	.	.	300
4	16	1985	.	.	.	.	.	.	301
5	14	1985	.	.	.	.	.	.	302
6	14	1985	.	.	.	.	.	.	303
7	15	1985	.	.	.	.	.	.	304
8	13	1985	.	.	.	.	.	.	305
9	13	1985	.	.	.	.	.	.	306
10	15	1985	.	.	.	.	.	.	307
11	15	1985	.	.	.	.	.	.	308
3	20	1986	.	.	.	.	.	.	309
3	23	1986	.	.	.	.	.	.	300
4	16	1986	.	.	.	.	.	.	301
5	14	1986	.	.	.	.	.	.	302
6	11	1986	.	.	.	.	.	.	303
7	12	1986	.	.	.	.	.	.	304
8	13	1986	.	.	.	.	.	.	305
9	14	1986	.	.	.	.	.	.	306
10	16	1986	.	.	.	.	.	.	307
11	16	1986	.	.	.	.	.	.	308
3	22	1987	.	.	.	.	.	.	300
4	15	1987	.	.	.	.	.	.	301
5	16	1987	.	.	.	.	.	.	302
6	16	1987	.	.	.	.	.	.	303
7	15	1987	.	.	.	.	.	.	304
8	15	1987	.	.	.	.	.	.	305
9	16	1987	.	.	.	.	.	.	306
10	13	1987	.	.	.	.	.	.	307
11	10	1987	.	.	.	.	.	.	308
3	28	1988	.	.	.	.	.	.	300
4	17	1988	.	.	.	.	.	.	301
5	12	1988	.	.	.	.	.	.	302
6	12	1988	.	.	.	.	.	.	303
7	15	1988	.	.	.	.	.	.	304
8	16	1988	.	.	.	.	.	.	305
9	16	1988	.	.	.	.	.	.	306
10	18	1988	.	.	.	.	.	.	307
11	15	1988	.	.	.	.	.	.	308
12	5	1988	.	.	.	.	.	.	309
4	7	1989	.	.	.	.	.	.	300
4	24	1989	.	.	.	.	.	.	301
5	16	1989	.	.	.	.	.	.	302
6	17	1989	.	.	.	.	.	.	303
7	17	1989	.	.	.	.	.	.	304
8	14	1989	.	.	.	.	.	.	305
9	14	1989	.	.	.	.	.	.	306
10	16	1989	.	.	.	.	.	.	307
11	10	1989	.	.	.	.	.	.	308

Table A2.5.10. Drainage quality data from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 3.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
4	26	1978	.	.	7.20	1100.00	.	.	.	.	121.00	11.00	.
5	25	1978	.	.	7.40	1750.00	.	.	.	.	204.00	20.00	.
6	14	1978	125.0	.	7.60	2500.00	.	.	.	.	228.00	29.00	.
6	16	1978	94.0	.	7.70	2600.00	.	.	.	.	247.00	30.00	.
6	23	1978	307.0	.	7.56	2402.27	.	.	.	1150.00	205.00	27.80	.
6	26	1978	5.0	.	7.90	2150.00	.	.	.	900.00	186.00	24.00	.
7	6	1978	88.0	.	7.58	1825.00	.	.	.	850.00	212.00	25.00	.
7	7	1978	4957.0	.	7.43	1305.95	.	.	.	546.47	156.00	15.50	.
7	8	1978	.	.	7.46	1406.25	.	.	.	830.00	220.20	16.80	.
7	9	1978	.	.	7.55	1550.00	.	.	.	.	300.00	20.00	.
7	13	1978	.	.	7.65	.	.	.	.	850.00	273.00	18.00	.
8	4	1978	95.0	.	8.00	1100.00	.	.	.	660.00	191.00	13.00	.
8	15	1978	.	.	7.80	1275.00	39.0	.	.	800.00	201.00	19.00	.
8	16	1978	.	.	7.50	825.00	28.0	.	.	100.00	118.00	11.00	.
8	17	1978	.	.	7.29	1175.00	29.0	.	.	1000.00	355.00	41.00	.
8	18	1978	.	.	7.60	1850.00	36.0	.	.	800.00	219.00	21.00	.
8	21	1978	.	.	7.75	2050.00	.	.	.	.	271.00	27.00	.
8	23	1978	.	.	7.47	1216.67	30.0	.	.	603.33	137.70	15.00	.
8	25	1978	.	.	7.55	1750.00	33.0	.	.	800.00	216.00	21.00	.
8	28	1978	.	.	7.55	2350.00	41.8	.	.	1250.00	280.00	29.00	.
9	11	1978	.	.	7.30	2300.00	35.0	.	.	1200.00	256.00	26.00	.
9	13	1978	.	.	7.23	1650.00	34.0	.	.	851.50	191.70	17.70	.
10	17	1978	.	.	7.50	1800.00	31.0	.	.	1000.00	257.00	27.00	.
3	19	1979	2660.0	2428.0	.	861.00	.	.	.	380.00	96.60	18.20	.
4	17	1979	6040.9	4676.0	5.90	635.00	7.0	.	.	325.00	77.00	14.00	.
4	18	1979	4644.2	5152.0	6.80	401.00	22.0	.	.	170.00	66.00	6.00	.
4	19	1979	5560.2	5332.0	7.10	391.00	21.0	.	.	170.00	65.00	5.00	.
4	20	1979	3077.2	3329.0	6.40	364.00	22.0	.	.	145.00	67.00	4.80	.
4	20	1979	4364.1	3329.0	6.90	568.00	29.0	.	.	285.00	106.00	8.60	.
4	22	1979	4598.8	1534.0	7.40	980.00	19.0	.	.	483.00	171.00	15.80	.
5	9	1979	647.2	1239.0	7.50	1039.00	.	.	.	.	171.20	15.80	.
5	10	1979	8716.9	4417.0	7.50	1010.00	.	.	.	532.00	162.40	16.60	.
5	20	1979	3062.1	1080.0	6.90	1363.00	.	.	.	816.50	227.80	23.60	.
5	22	1979	1903.9	1278.0	7.60	984.00	.	.	.	583.00	178.40	18.80	.
5	23	1979	3357.3	699.0	7.20	1264.00	.	.	.	700.00	193.40	21.60	.
6	1	1979	2289.9	662.0	7.70	1334.00	.	.	.	800.00	201.80	22.00	.
6	6	1979	2043.9	1227.0	7.60	1213.00	.	.	.	700.00	180.60	19.40	.
6	10	1979	2161.2	1411.0	7.30	1116.00	36.0	.	.	560.00	159.40	18.00	.
6	12	1979	681.3	245.0	7.30	1107.00	32.0	.	.	.	157.00	17.20	.
6	15	1979	469.3	442.0	7.30	1138.00	33.0	.	.	.	160.40	18.40	.
6	20	1979	2853.9	1339.0	7.10	1347.00	30.0	.	.	570.00	171.00	21.00	.
6	30	1979	526.1	.	.	.	.	.	.	.	176.00	21.00	.
7	30	1979	2324.0	2847.0	5.75	907.00	32.0	.	.	460.00	118.00	22.00	.
9	1	1979	4307.3	1988.0	7.00	1112.00	10.4	.	.	650.00	189.60	16.80	.
9	5	1979	336.9	270.0	7.10	1333.00	14.4	.	.	.	246.40	22.40	.
9	12	1979	1158.2	417.0	7.00	1250.00	12.8	.	.	800.00	228.60	20.00	.
9	13	1979	90.8	270.0	7.20	1321.00	.	.	.	.	237.80	21.40	.
10	19	1979	541.3	307.0	6.40	980.00	7.2	.	.	600.00	166.40	14.60	.
10	31	1979	1763.8	908.0	6.60	1025.00	8.8	.	.	550.00	152.00	15.00	.
3	31	1980	1206.5	1154.0	6.90	470.00	15.0	.	.	205.00	54.20	5.40	.
4	5	1980	5326.1	2260.0	7.10	325.00	17.2	.	.	140.00	38.50	4.00	.
4	6	1980	1298.4	1349.0	6.80	575.00	16.1	.	.	250.00	76.00	7.00	.
4	18	1980	2214.5	450.0	6.90	.	12.7	.	.	258.00	83.90	7.80	.
6	1	1980	579.2	313.0	6.50	950.00	6.6	.	.	710.00	174.90	17.10	.
6	6	1980	246.1	232.0	6.80	1100.00	6.6	.	.	.	164.60	16.60	.
7	3	1980	1343.8	615.0	6.70	1150.00	7.6	.	.	770.00	186.00	20.00	.
8	12	1980	1090.2	683.0	6.50	1400.00	6.0	.	.	860.00	224.80	28.80	.
8	19	1980	1593.7	464.0	6.90	1500.00	8.5	.	.	975.00	263.80	31.60	.
8	22	1980	1502.8	1084.0	.	1650.00	.	.	.	1100.00	275.00	37.00	.
8	29	1980	11432.0	7250.0	5.50	1360.00	3.4	.	.	1040.00	160.00	17.00	.
9	7	1980	2141.9	1860.0	6.70	1695.00	13.6	.	.	1170.00	272.00	24.00	.
9	11	1980	969.1	655.0	6.80	1246.00	11.5	.	.	710.00	132.00	9.00	.
9	12	1980	4894.5	3026.0	6.80	1275.00	12.8	.	.	780.00	142.00	11.00	.
9	18	1980	10209.3	3354.0	7.00	1450.00	14.4	.	.	910.00	191.00	16.00	.
9	23	1980	1578.5	878.0	6.40	1822.00	17.8	.	.	1150.00	313.20	48.40	.
10	23	1980	1570.9	250.0	6.50	1598.00	12.7	.	.	1050.00	270.00	28.00	.

Table A2.5.10. Drainage quality data from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 3.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
10	26	1980	567.8	148.0	6.50	1576.00	8.5	.	.	1050.00	238.00	32.00	.
2	20	1981	1601.0	264.0	5.20	635.00	2.6	.	.	325.00	63.70	19.20	.
3	23	1981	1949.0	750.0	7.00	598.00	6.4	.	.	305.00	69.50	10.50	.
3	30	1981	9089.0	2006.0	6.70	500.00	7.2	.	.	215.00	58.50	9.00	.
4	5	1981	6367.0	1136.0	6.90	877.00	12.0	.	.	490.00	119.60	18.50	.
4	14	1981	3660.0	1760.0	.	.	.	.	.	610.00	156.00	29.00	.
4	17	1981	2862.0	2063.0	7.20	1300.00	13.6	.	.	660.00	159.00	31.00	.
4	23	1981	3895.0	2177.0	7.00	1400.00	13.6	.	.	660.00	179.00	35.00	.
4	24	1981	7730.0	2460.0	7.15	1400.00	18.6	.	.	700.00	168.00	35.50	.
4	28	1981	4376.0	3237.0	7.20	1325.00	.	.	.	640.00	153.00	34.00	.
4	30	1981	4092.0	856.0	7.20	1430.00	20.8	.	.	845.00	220.00	36.00	.
5	13	1981	1749.0	42.0	7.30	1574.00	18.4	.	.	700.00	276.00	40.00	.
6	1	1981	3028.0	1563.0	7.00	1638.00	16.1	.	.	1000.00	262.00	40.00	.
6	7	1981	2688.0	202.0	7.10	1264.00	9.1	.	.	685.00	198.00	26.00	.
6	22	1981	5617.0	2156.0	6.70	1240.00	10.4	.	.	675.00	194.00	28.00	.
6	24	1981	4747.0	1760.0	6.85	1152.00	8.0	.	.	605.00	187.00	24.00	.
6	28	1981	6488.0	2244.0	6.90	1197.00	10.4	.	.	665.00	192.00	26.00	.
7	4	1981	4247.0	167.0	7.15	1528.00	13.8	.	.	1010.00	278.00	32.60	.
7	17	1981	787.0	481.0	6.95	1916.00	12.0	.	.	1300.00	332.00	46.60	.
7	25	1981	1291.0	256.0	6.85	1903.00	8.4	.	.	1200.00	328.00	46.60	.
8	10	1981	1548.0	560.0	6.60	1711.00	7.4	.	.	1060.00	276.00	50.00	.
8	13	1981	333.0	55.0	6.80	1767.00	10.6	.	.	1050.00	282.00	44.00	.
8	31	1981	496.0	337.0	6.30	1800.00	4.8	.	.	1000.00	254.00	48.00	.
9	7	1981	893.0	771.0	6.45	2250.00	4.8	.	.	1140.00	288.00	52.00	.
9	26	1981	1688.0	1105.0	6.00	2025.00	3.2	.	.	1190.00	286.00	56.00	.
10	1	1981	3838.0	2234.0	6.40	1750.00	4.8	.	.	880.00	248.00	44.00	.
10	4	1981	7219.0	2033.0	6.50	1800.00	4.8	.	.	980.00	296.00	44.00	.
10	13	1981	9596.0	2851.0	6.70	1850.00	5.6	.	.	1000.00	296.00	44.00	.
10	17	1981	8725.0	2038.0	6.80	1220.00	7.2	.	.	1000.00	272.00	42.00	.
11	6	1981	3562.0	65.0	6.85	2100.00	6.4	.	.	1300.00	340.00	62.00	.
4	11	1982	1832.0	.	6.80	750.00	4.8	.	.	360.00	90.20	13.60	.
4	15	1982	2074.0	.	6.50	750.00	4.5	.	.	370.00	87.60	16.40	.
4	19	1982	1754.0	.	6.70	800.00	6.3	.	.	430.00	126.00	20.00	.
4	23	1982	1881.0	.	6.60	1050.00	6.3	.	.	560.00	164.00	20.40	.
5	3	1982	829.0	.	6.10	1300.00	4.1	.	.	750.00	181.00	42.00	.
5	4	1982	4281.0	.	6.45	1250.00	5.8	.	.	690.00	164.00	36.00	.
5	7	1982	7730.0	.	6.40	1290.00	6.3	.	.	720.00	152.00	40.00	.
5	11	1982	5129.0	.	6.75	1500.00	9.9	.	.	.	196.00	50.00	.
5	14	1982	5462.0	.	6.60	1550.00	9.9	.	.	940.00	200.00	18.00	.
5	21	1982	1408.0	.	6.80	1850.00	8.1	.	.	1000.00	282.00	49.00	.
6	2	1982	1594.0	.	6.80	1980.00	7.2	.	.	790.00	288.00	52.60	.
6	18	1982	984.0	.	6.40	1300.00	9.9	.	.	750.00	184.00	40.00	.
7	2	1982	25536.0	.	.	.	.	.	.	750.00	256.00	38.00	.
7	26	1982	13832.0	.	6.31	1520.00	6.2	.	.	785.00	232.00	36.00	.
8	30	1982	10815.0	.	6.28	1707.00	4.4	.	.	958.00	336.00	49.00	.
10	4	1982	1.00E4	.	6.84	1497.00	8.7	.	.	940.00	220.00	42.00	.
11	22	1982	.	.	6.60	1220.00	7.2	.	.	1050.00	174.00	32.00	.
3	18	1983	4209.0	.	6.53	515.00	5.7	.	.	240.00	.	.	.
4	14	1983	7736.0	.	5.94	700.00	4.9	.	.	310.00	.	.	.
5	16	1983	4814.0	.	5.01	1198.00	1.9	.	.	640.00	.	.	.
6	16	1983	3705.0	.	5.23	1057.00	2.0	.	.	600.00	.	.	.
7	15	1983	1.53E4	.	5.79	1203.00	2.9	.	.	780.00	.	.	.
8	16	1983	1.92E4	.	5.17	1363.00	2.4	.	.	860.00	.	.	.
9	15	1983	1.20E4	.	5.21	1368.00	1.8	.	.	900.00	.	.	.
10	13	1983	2.70E4	.	5.59	1356.00	2.6	.	.	730.00	.	.	.
11	10	1983	5776.0	.	5.50	1025.00	1.8	.	.	620.00	.	.	.
12	1	1983	182.0	.	5.15	470.00	0.9	.	.	320.00	.	.	.
3	15	1984	4417.0	.	4.50	650.00	.	.	.	380.00	54.98	44.99	401
4	15	1984	1.37E4	.	4.73	560.00	1.0	.	.	320.00	75.91	29.94	402
5	16	1984	9160.0	.	5.70	824.00	3.1	.	.	620.00	153.71	37.84	403
6	13	1984	2.67E4	.	5.40	914.00	1.5	.	.	620.00	151.74	41.57	404
7	14	1984	7135.0	.	5.15	1080.00	1.9	.	.	680.00	.	.	405
8	15	1984	8748.0	.	4.86	1345.00	1.9	.	.	880.00	207.30	81.30	406
9	12	1984	3225.0	.	4.61	1700.00	1.1	.	.	1500.00	275.50	106.70	407
10	13	1984	1.61E4	.	4.53	1565.00	0.3	.	.	1500.00	273.70	109.80	408
11	14	1984	950.0	.	4.75	1350.00	1.1	.	.	1100.00	298.00	64.00	409
11	28	1984	19.0	.	.	.	.	.	.	.	.	.	410
12	17	1984	1840.0	.	4.65	600.00	0.5	.	.	368.00	74.00	28.00	411
4	2	1985	3353.0	.	5.55	700.00	2.2	.	.	400.00	102.20	21.40	400
4	16	1985	2.03E4	.	4.98	1150.00	2.6	.	.	770.00	170.00	64.60	401
5	14	1985	1.16E4	.	4.93	1440.00	1.7	.	.	1000.00	152.00	64.00	402
6	14	1985	3.71E4	.	5.12	1275.00	3.0	.	.	760.00	148.00	60.00	403
7	15	1985	1.32E4	.	4.92	1200.00	2.7	.	.	850.00	137.00	53.00	404
8	13	1985	1.40E4	.	4.76	1400.00	1.8	.	.	900.00	188.00	80.40	405
9	13	1985	3.11E4	.	4.77	1550.00	1.7	.	.	1050.00	220.00	82.60	406
10	15	1985	1.73E4	.	4.85	1500.00	2.2	.	.	850.00	168.00	80.00	407

Table A2.5.10. Drainage quality data from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 3.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
11	15	1985	2877.0	.	4.68	1400.00	1.1	.	.	750.00	180.00	92.00	408
3	23	1986	1.79E4	.	4.80	675.00	1.1	.	.	320.00	56.00	34.00	400
4	16	1986	1.39E4	.	4.91	950.00	3.0	.	.	540.00	120.00	60.60	401
5	14	1986	1.21E4	.	4.89	1125.00	2.0	.	.	950.00	168.00	66.00	402
6	11	1986	7634.0	.	4.82	1425.00	1.1	.	.	850.00	154.00	64.00	403
7	12	1986	2.16E4	.	4.59	1400.00	1.0	.	.	850.00	166.00	78.00	404
8	13	1986	1.44E4	.	4.42	1875.00	0.0	.	.	1350.00	224.00	104.20	405
9	14	1986	2.66E4	.	4.44	1750.00	0.0	.	.	950.00	194.00	108.00	406
10	16	1986	4637.0	.	4.29	2075.00	0.0	.	.	1375.00	190.00	122.00	407
11	16	1986	6552.0	.	4.50	1700.00	0.0	.	.	950.00	152.00	100.00	408
3	22	1987	1.20E4	.	4.75	600.00	2.0	.	.	286.00	66.00	28.00	400
4	15	1987	3380.0	.	4.64	875.00	1.5	.	.	478.00	77.00	48.00	401
5	16	1987	2.87E4	.	4.54	2000.00	0.5	.	.	1300.00	222.00	134.00	402
6	16	1987	447.0	.	4.21	2175.00	0.0	.	.	1430.00	230.00	172.00	403
7	15	1987	3.06E4	.	4.28	2100.00	0.0	.	.	1370.00	182.00	122.00	404
8	15	1987	1.25E4	.	4.15	2175.00	0.0	.	.	1300.00	186.00	122.00	405
9	16	1987	1.68E4	.	4.05	2525.00	0.0	.	.	1440.00	206.00	144.00	406
10	13	1987	1026.0	.	3.94	3025.00	0.0	.	.	1930.00	240.00	200.00	407
11	10	1987	1355.0	.	4.05	2525.00	0.0	.	.	1930.00	254.00	222.00	408
3	28	1988	1.26E4	.	4.60	480.00	1.0	.	.	226.00	44.00	20.00	400
4	17	1988	5715.0	.	4.50	890.00	0.0	.	.	446.00	64.00	46.00	401
5	12	1988	1.25E4	.	4.41	2025.00	0.0	.	.	1300.00	188.00	138.00	402
6	12	1988	3592.0	.	4.16	1950.00	0.0	.	.	1350.00	204.00	144.00	403
7	15	1988	4182.0	.	4.24	1900.00	0.0	.	.	1220.00	186.00	128.00	404
8	16	1988	5.83E4	.	4.18	1950.00	0.0	.	.	1200.00	148.00	140.00	405
9	16	1988	1.19E4	.	4.17	1650.00	0.0	.	.	900.00	116.00	114.00	406
10	18	1988	3637.0	.	4.37	2250.00	0.0	.	.	1460.00	142.00	150.00	407
11	15	1988	6900.0	.	4.44	1825.00	0.0	.	.	1350.00	134.00	137.00	408
4	7	1989	2.97E4	.	4.85	360.00	2.1	.	.	172.00	30.00	16.00	400
4	24	1989	5307.0	.	4.64	825.00	1.4	.	.	446.00	52.00	38.00	401
5	16	1989	1.72E4	.	4.56	1623.00	0.3	.	.	990.00	114.00	94.00	402
6	17	1989	3.96E4	.	4.47	2061.00	0.0	.	.	1550.00	136.00	148.00	403
7	17	1989	761.0	.	4.34	1975.00	0.0	.	.	1350.00	162.00	146.00	404
8	14	1989	5087.0	.	4.25	2325.00	0.0	.	.	1500.00	152.00	170.00	405
9	14	1989	2.55E4	.	4.14	2725.00	0.0	.	.	1840.00	244.00	192.00	406
10	16	1989	1783.0	.	4.08	1975.00	0.0	.	.	1200.00	112.00	130.00	407
11	4	1989	159.0	.	4.20	2550.00	0.0	.	.	1780.00	152.00	202.00	408

Table A2.5.11. Drainage quality data for metals from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 3.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
4	26	1978	84.00	27.00	.	0.018	.	0.004	0.048	.	0.100	.
5	25	1978	165.00	35.00	.	0.012	0.210	0.012	0.112	0.020	0.190	.
6	14	1978	262.00	37.00	.	0.024	0.220	0.016	0.128	0.040	0.210	.
6	16	1978	277.00	41.00	.	0.026	0.175	0.015	0.120	0.040	0.260	.
6	23	1978	271.40	39.50	.	0.022	0.308	0.023	0.134	0.030	0.210	.
6	26	1978	260.00	41.00	.	0.021	0.260	0.019	0.084	.	0.180	.
7	6	1978	182.00	52.00	.	0.019	0.275	0.019	0.073	0.030	0.180	.
7	7	1978	129.90	31.40	.	0.015	0.183	0.019	0.064	0.030	0.110	.
7	8	1978	127.00	41.10	.	0.014	0.206	0.020	0.058	0.030	0.140	.
7	9	1978	149.00	46.00	.	0.009	0.240	0.017	.	0.020	0.160	.
7	13	1978	153.00	44.00	.	0.012	.	0.015	.	0.020	0.140	.
8	4	1978	115.00	43.00	.	0.020	0.200	0.010	0.052	0.020	0.140	.
8	15	1978	158.00	39.00	.	0.020	0.240	0.010	0.070	0.020	0.150	.
8	16	1978	60.00	26.00	.	0.020	0.130	0.020	0.030	0.030	0.090	.
8	17	1978	158.00	12.00	.	0.030	0.200	0.020	0.050	0.040	0.120	.
8	18	1978	110.00	42.00	.	0.020	0.240	0.040	0.070	0.020	0.160	.
8	21	1978	142.00	50.00	.	0.080	0.320	0.040	0.100	.	0.200	.
8	23	1978	84.70	32.30	.	0.020	0.143	0.023	0.037	0.020	0.100	.
8	25	1978	115.00	46.00	.	0.020	0.240	0.030	0.060	0.010	0.140	.
8	28	1978	187.00	60.00	.	0.030	0.340	0.040	0.100	0.020	0.190	.
9	11	1978	219.00	59.00	.	0.030	0.340	0.050	0.080	0.060	0.200	.
9	13	1978	107.30	44.30	.	0.023	0.210	0.033	0.053	0.060	0.140	.
10	17	1978	157.00	34.00	.	0.020	0.290	0.040	0.090	0.020	0.180	.
3	19	1979	39.40	11.20	.	3.000	14.600	0.256	0.400	0.030	1.080	.
4	17	1979	34.40	10.60	.	1.120	12.310	0.240	0.240	1.460	1.110	.
4	18	1979	25.00	9.00	.	0.069	0.373	0.064	0.050	0.020	0.150	.
4	19	1979	23.00	8.60	.	0.016	0.098	0.008	0.040	0.020	0.070	.
4	20	1979	24.00	17.80	.	0.014	0.089	0.008	0.030	0.020	0.070	.
4	20	1979	.	14.20	.	0.064	0.210	0.008	0.040	0.020	0.100	.
4	22	1979	42.60	24.40	.	0.021	0.270	0.020	0.070	0.030	0.140	.
5	9	1979	48.20	27.40	.	0.016	0.310	0.023	0.070	0.030	0.170	.
5	10	1979	50.40	25.60	.	0.021	0.280	0.020	0.060	0.010	0.150	.
5	20	1979	77.90	35.20	.	0.026	0.375	0.029	0.080	0.020	0.170	.
5	22	1979	68.80	30.00	.	0.025	0.320	0.023	0.080	0.010	0.150	.
5	23	1979	74.00	32.80	.	0.030	0.360	0.027	0.070	0.020	0.160	.
6	1	1979	80.40	34.80	.	0.027	0.350	0.027	0.080	0.030	0.170	.
6	6	1979	48.40	48.20	.	0.023	0.300	0.021	0.080	0.010	0.140	.
6	10	1979	45.00	41.60	.	0.023	0.240	0.013	0.060	0.030	0.110	.
6	12	1979	43.60	40.80	.	0.013	0.240	0.009	0.060	0.010	0.110	.
6	15	1979	45.80	37.60	.	0.016	0.280	0.009	0.060	0.010	0.120	.
6	20	1979	72.00	38.00	.	0.020	0.260	0.020	0.070	0.030	0.970	.
6	30	1979	73.00	35.00	.	0.010	0.270	0.020	0.050	0.010	0.100	.
7	30	1979	38.00	20.00	.	0.700	4.140	0.260	0.180	0.010	0.550	.
9	1	1979	40.40	33.80	.	0.040	0.510	0.028	0.090	0.010	0.140	.
9	5	1979	46.80	40.00	.	0.040	0.670	0.034	0.120	0.010	0.190	.
9	12	1979	45.60	38.20	.	0.060	0.780	0.060	0.130	0.010	0.190	.
9	13	1979	55.60	43.40	.	0.030	0.710	0.035	0.110	0.020	0.180	.
10	19	1979	24.00	27.80	.	0.080	1.260	0.070	0.160	0.010	0.130	.
10	31	1979	38.00	28.80	.	0.080	1.040	0.060	0.140	0.010	0.120	.
3	31	1980	12.60	11.60	.	0.030	0.395	0.035	0.090	0.020	0.060	.
4	5	1980	8.00	8.30	.	0.040	0.270	0.010	0.070	0.020	0.040	.
4	6	1980	11.20	13.00	.	0.050	0.410	0.030	0.100	0.030	0.120	.
4	18	1980	13.30	14.50	.	0.035	0.695	0.035	0.150	0.020	0.140	.
6	1	1980	25.10	30.80	.	0.160	2.030	0.120	0.230	0.020	0.250	.
6	6	1980	24.40	28.60	.	0.180	2.250	0.160	0.270	0.030	0.270	.
7	3	1980	33.00	29.00	.	0.250	2.140	0.120	0.280	0.020	0.240	.
8	12	1980	34.40	35.60	.	1.140	8.380	0.440	0.510	0.010	0.460	.
8	19	1980	51.90	43.50	.	0.690	5.990	0.300	0.460	0.010	0.530	.
8	22	1980	57.00	36.00	.	0.490	4.080	0.250	0.350	0.030	0.380	.
8	29	1980	52.00	34.00	.	0.750	4.810	0.310	0.270	0.060	0.460	.
9	7	1980	99.00	48.00	.	0.400	2.700	0.150	0.250	0.030	0.290	.
9	11	1980	59.00	36.00	.	0.170	1.840	0.100	0.200	0.040	0.230	.
9	12	1980	60.00	35.00	.	0.350	2.180	0.110	0.190	0.040	0.230	.
9	18	1980	67.00	38.00	.	0.310	1.850	0.090	0.180	0.040	0.180	.
9	23	1980	96.00	12.80	.	0.320	2.610	0.140	0.270	0.030	0.220	.
10	23	1980	73.00	45.00	.	1.240	5.800	0.280	0.420	0.010	0.400	.

Table A2.5.11. Drainage quality data for metals from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 3.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
10	26	1980	60.00	46.00	.	0.480	4.160	0.230	0.310	0.020	0.360	.
2	20	1981	27.30	.	.	1.360	7.100	0.450	0.270	.	.	.
3	23	1981	19.50	.	.	0.140	1.470	0.085	0.160	.	.	.
3	30	1981	10.40	.	.	0.260	1.720	0.110	0.150	.	.	.
4	5	1981	22.10	.	.	0.520	4.210	0.220	0.310	.	.	.
4	14	1981	27.00	.	.	0.460	3.230	0.170	0.260	.	.	.
4	17	1981	30.00	.	.	0.340	2.900	0.150	0.230	.	.	.
4	23	1981	31.00	.	.	0.320	2.540	0.140	0.220	.	.	.
4	24	1981	32.00	.	.	0.220	1.860	0.100	0.160	.	.	.
4	28	1981	31.00	.	.	0.180	1.580	0.100	0.140	.	.	.
4	30	1981	38.00	.	.	0.180	1.920	0.100	0.190	.	.	.
5	13	1981	48.00	.	.	0.160	2.290	0.120	0.230	.	.	.
6	1	1981	64.00	.	.	0.720	4.620	0.260	0.380	.	.	.
6	7	1981	42.00	.	.	0.360	3.000	0.170	0.270	.	.	.
6	22	1981	42.00	.	.	0.610	3.950	0.220	0.250	.	.	.
6	24	1981	31.00	.	.	0.450	2.980	0.160	0.190	.	.	.
6	28	1981	36.00	.	.	0.380	2.300	0.130	0.160	.	.	.
7	4	1981	50.40	.	.	0.240	1.690	0.090	0.180	.	.	.
7	17	1981	81.00	.	.	0.830	4.790	0.230	0.470	.	.	.
7	25	1981	75.80	.	.	1.600	7.420	0.370	0.540	.	.	.
8	10	1981	68.00	.	.	2.000	8.000	0.470	0.590	.	.	.
8	13	1981	64.00	48.00	.	0.690	6.000	0.390	0.460	.	.	.
8	31	1981	60.00	.	.	3.840	11.290	0.590	0.740	.	.	.
9	7	1981	60.00	.	.	3.030	12.000	0.640	0.730	.	.	.
9	26	1981	58.00	.	.	3.840	15.000	0.830	0.850	.	.	.
10	1	1981	40.00	.	.	1.680	9.610	0.510	0.600	.	.	.
10	4	1981	40.00	44.00	.	1.080	10.000	0.470	0.530	.	.	.
10	13	1981	48.00	46.00	.	0.810	7.000	0.360	0.470	.	.	.
10	17	1981	52.00	48.00	.	0.650	5.320	0.300	0.360	.	.	.
11	6	1981	68.00	58.00	.	0.710	5.100	0.250	0.270	.	.	.
4	11	1982	14.00	16.00	.	0.590	3.930	0.210	0.300	.	.	.
4	15	1982	14.00	14.00	.	0.700	5.380	0.310	0.330	.	.	.
4	19	1982	16.00	18.00	.	0.620	6.210	0.340	0.350	.	.	.
4	23	1982	22.00	24.00	.	0.860	9.190	0.480	0.490	.	.	.
5	3	1982	30.00	30.00	.	1.660	11.355	0.620	0.620	.	.	.
5	4	1982	22.00	28.00	.	0.900	8.715	0.460	0.460	.	.	.
5	7	1982	22.00	26.00	.	0.810	8.520	0.460	0.420	.	.	.
5	11	1982	30.00	32.00	.	0.660	8.150	0.430	0.400	.	.	.
5	14	1982	20.00	16.00	.	0.270	2.680	0.130	0.150	0.020	0.190	.
5	21	1982	41.00	39.00	.	0.470	6.800	0.330	0.430	0.030	0.400	.
6	2	1982	48.80	42.60	.	0.950	8.310	0.400	0.560	0.060	0.530	.
6	18	1982	31.00	31.00	.	1.090	6.720	0.350	0.420	0.030	0.450	.
7	2	1982	36.00	36.00	.	1.130	7.000	0.340	0.320	0.020	0.440	.
7	26	1982	40.00	40.00	.	1.150	8.860	0.460	0.330	0.100	0.470	.
8	30	1982	40.00	46.00	.	1.655	9.360	0.450	0.410	0.010	0.570	.
10	4	1982	34.00	40.00	.	0.870	8.820	0.540	0.330	0.030	0.450	.
11	22	1982	20.00	.	.	1.100	7.800	0.320	0.370	.	.	.
3	18	1983	.	.	.	0.450	3.080	0.170	0.230	.	.	.
4	14	1983	.	.	.	2.120	7.250	0.450	0.510	.	.	.
5	16	1983	.	.	.	5.650	28.000	1.230	1.490	.	.	.
6	16	1983	.	.	.	2.530	9.910	0.600	0.660	.	.	.
7	15	1983	.	.	.	1.760	8.800	0.480	0.610	.	.	.
8	16	1983	.	.	.	3.730	16.590	0.860	1.000	.	.	.
9	15	1983	.	.	.	3.870	18.670	1.130	1.140	.	.	.
10	13	1983	.	.	.	2.600	14.390	0.890	0.840	.	.	.
11	10	1983	.	.	.	2.880	15.900	0.870	0.790	.	.	.
12	1	1983	.	.	.	1.420	8.360	0.460	0.410	.	.	.
3	15	1984	10.66	.	.	6.400	20.800	1.230	0.790	.	.	401
4	15	1984	8.96	.	.	5.950	16.000	0.840	0.600	.	.	402
5	16	1984	16.03	.	.	2.760	13.000	0.700	0.730	.	.	403
6	13	1984	18.39	.	.	3.150	16.000	0.800	0.760	.	.	404
7	14	1984	.	.	.	3.800	19.100	0.940	0.870	.	.	405
8	15	1984	23.54	.	.	6.730	39.400	2.160	1.440	.	.	406
9	12	1984	30.54	.	.	11.440	49.600	3.790	2.550	.	.	407
10	13	1984	28.50	.	.	12.050	48.000	3.770	2.460	.	.	408
11	14	1984	42.00	.	.	8.990	45.600	2.500	2.820	.	.	409
11	28	1984	.	.	.	.	.	.	.	.	.	410
12	17	1984	10.00	.	.	2.900	15.100	0.750	0.690	.	.	411
4	2	1985	8.00	.	.	2.300	10.100	0.460	0.710	.	.	400
4	16	1985	18.00	.	.	7.600	36.000	1.760	1.830	.	.	401
5	14	1985	19.10	.	.	7.000	29.000	1.750	1.600	.	.	402
6	14	1985	21.50	.	.	6.200	30.000	1.350	1.310	.	.	403
7	15	1985	20.20	.	.	6.800	28.000	1.440	1.350	.	.	404
8	13	1985	20.40	.	.	10.500	37.000	1.970	1.800	.	.	405
9	13	1985	26.00	.	.	11.800	41.600	2.240	1.980	.	.	406
10	15	1985	24.00	.	.	11.000	38.000	1.910	1.570	.	.	407
11	15	1985	26.00	.	.	8.600	43.200	2.160	1.780	.	.	408

Table A2.5.11. Drainage quality data for metals from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 3.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
3	23	1986	7.00	.	.	5.600	21.000	1.230	0.830	.	.	400
4	16	1986	12.00	.	.	11.100	39.000	2.100	1.520	.	.	401
5	14	1986	14.00	.	.	13.000	42.000	2.660	1.620	.	.	402
6	11	1986	24.00	.	.	11.000	44.000	1.910	1.740	.	.	403
7	12	1986	22.00	.	.	14.000	48.000	2.330	2.000	.	.	404
8	13	1986	28.00	.	.	20.000	67.000	3.700	2.260	.	.	405
9	14	1986	28.00	.	.	22.000	70.000	4.400	2.370	.	.	406
10	16	1986	32.00	.	.	23.000	84.000	4.300	3.140	.	.	407
11	16	1986	22.00	.	.	19.000	63.000	3.800	2.810	.	.	408
3	22	1987	8.00	.	.	7.600	34.000	1.150	0.930	.	.	400
4	15	1987	10.00	.	.	20.000	33.000	1.830	1.560	.	.	401
5	16	1987	24.00	.	.	24.000	83.000	4.300	3.330	.	.	402
6	16	1987	38.00	.	.	21.800	103.000	3.900	2.750	.	.	403
7	15	1987	30.00	.	.	29.000	90.000	4.300	3.180	.	.	404
8	15	1987	30.00	.	.	31.000	89.000	4.300	3.270	.	.	405
9	16	1987	32.00	.	.	37.000	100.000	4.980	3.600	.	.	406
10	13	1987	42.00	.	.	45.000	140.000	6.850	5.030	.	.	407
11	10	1987	48.00	.	.	34.000	157.000	7.340	5.600	.	.	408
3	28	1988	4.00	.	.	6.500	13.100	0.900	1.200	.	.	400
4	17	1988	8.00	.	.	16.000	43.000	2.200	2.700	.	.	401
5	12	1988	130.00	.	.	41.000	139.000	6.800	7.400	.	.	402
6	12	1988	30.00	.	.	34.000	130.000	6.900	9.100	.	.	403
7	15	1988	28.00	.	.	28.000	116.000	5.900	7.700	.	.	404
8	16	1988	34.00	.	.	37.000	149.000	6.700	6.200	.	.	405
9	16	1988	28.00	.	.	27.000	102.000	5.100	4.900	.	.	406
10	18	1988	28.00	.	.	33.000	109.000	6.600	7.100	.	.	407
11	15	1988	24.00	.	.	32.500	102.000	6.050	6.650	.	.	408
4	7	1989	6.00	.	.	5.700	14.400	0.800	0.800	.	.	400
4	24	1989	8.00	.	.	14.000	36.000	2.000	2.400	.	.	401
5	16	1989	20.00	.	.	30.000	83.000	5.200	5.700	.	.	402
6	17	1989	32.00	.	.	41.000	133.000	11.000	7.600	.	.	403
7	17	1989	30.00	.	.	30.000	125.000	7.300	8.200	.	.	404
8	14	1989	36.00	.	.	31.000	188.000	8.300	8.400	.	.	405
9	14	1989	36.00	.	.	67.000	153.000	15.300	10.400	.	.	406
10	16	1989	28.00	.	.	28.000	122.000	11.800	7.900	.	.	407
11	4	1989	40.00	.	.	33.000	180.000	20.900	9.900	.	.	408

Table A2.5.12. Drainage quality data for additional parameters from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 1 of 3.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
4	26	1978	.	.	.	.	.	4.0	.
5	25	1978	.	.	.	.	.	.	.
6	14	1978	.	.	.	.	.	9.0	.
6	16	1978	.	.	.	.	.	11.0	.
6	23	1978	25.50	.	.	.	.	12.7	.
6	26	1978	28.00	.	.	.	.	17.0	.
7	6	1978	11.00	.	.	0.10	.	17.0	.
7	7	1978	7.80	1.60	0.73	0.08	.	11.0	.
7	8	1978	6.80	.	.	0.08	.	.	.
7	9	1978	.	.	.	.	.	.	.
7	13	1978	7.00	2.20	0.90	0.08	.	.	.
8	4	1978	3.00	1.20	0.06	0.08	.	22.0	.
8	15	1978	6.00	1.20	1.20	0.08	.	21.0	.
8	16	1978	1.00	1.20	.	0.08	.	.	.
8	17	1978	13.00	2.20	.	0.08	.	.	.
8	18	1978	3.00	1.40	.	0.08	.	.	.
8	21	1978	.	.	.	.	.	.	.
8	23	1978	2.30	1.40	1.08	0.08	.	23.0	.
8	25	1978	2.00	1.40	1.00	0.08	.	23.0	.
8	28	1978	2.00	2.40	.	0.08	.	.	.
9	11	1978	10.00	.	.	0.08	.	.	.
9	13	1978	2.50	1.40	0.06	0.08	.	12.0	.
10	17	1978	6.00	1.40	0.06	0.08	.	.	.
3	19	1979	8.00	.	.	.	4.10	.	.
4	17	1979	7.00	.	.	.	3.50	.	.
4	18	1979	3.00	1.20	.	.	.	.	.
4	19	1979	1.00	0.50	.	.	.	.	.
4	20	1979	2.00	0.40	0.05	.	.	.	.
4	20	1979	3.00	0.60	0.12	.	2.20	.	.
4	22	1979	4.00	1.00	0.03	.	.	.	.
5	9	1979	.	.	.	.	.	.	.
5	10	1979	4.00	.	.	.	.	.	.
5	20	1979	5.00	.	.	.	5.70	.	.
5	22	1979	3.00	.	.	.	3.50	.	.
5	23	1979	4.00	.	.	.	4.50	.	.
6	1	1979	5.00	.	.	.	4.70	.	.
6	6	1979	5.00	.	.	.	.	.	.
6	10	1979	4.00	.	.	.	.	22.0	.
6	12	1979	.	.	.	.	.	.	.
6	15	1979	.	.	.	.	.	21.0	.
6	20	1979	5.00	.	.	.	.	17.0	.
6	30	1979	.	.	.	.	.	.	.
7	30	1979	11.00	.	.	.	4.90	23.0	.
9	1	1979	4.00	.	.	.	4.00	20.5	.
9	5	1979	.	.	.	.	5.10	20.0	.
9	12	1979	6.00	.	.	.	3.70	20.0	.
9	13	1979	.	.	.	.	.	20.0	.
10	19	1979	2.00	.	.	.	1.90	.	.
10	31	1979	3.00	.	.	.	2.20	.	.
3	31	1980	2.00	.	.	.	1.80	.	.
4	5	1980	1.00	.	.	.	1.00	.	.
4	6	1980	2.00	.	.	.	.	.	.
4	18	1980	2.00	.	.	.	1.00	.	.
6	1	1980	4.00	.	.	.	.	.	.
6	6	1980	.	.	.	.	.	.	.
7	3	1980	3.00	.	.	.	.	.	.
8	12	1980	3.00	.	.	.	2.30	.	.
8	19	1980	4.00	.	.	.	.	.	.
8	22	1980	6.00	.	.	.	.	.	.
8	29	1980	11.00	.	.	.	3.40	.	.
9	7	1980	6.00	.	.	.	.	.	.
9	11	1980	5.00	.	.	.	.	15.0	.
9	12	1980	4.00	.	.	.	2.60	16.0	.
9	18	1980	5.00	.	.	.	.	14.0	.
9	23	1980	5.00	.	.	.	3.30	14.0	.
10	23	1980	6.00	.	.	.	2.50	10.0	.

Table A2.5.12. Drainage quality data for additional parameters from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 2 of 3.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
10	26	1980	5.00	.	.	.	.	13.0	.
2	20	1981	5.00	.	.	.	.	7.0	.
3	23	1981	.	.	.	.	.	10.0	.
3	30	1981	.	.	.	.	1.70	.	.
4	5	1981	.	.	.	.	.	.	.
4	14	1981	.	.	.	.	.	.	.
4	17	1981	.	.	.	.	1.70	.	.
4	23	1981	.	.	.	.	.	.	.
4	24	1981	.	.	.	.	.	.	.
4	28	1981	.	.	.	.	1.70	.	.
4	30	1981	.	.	.	.	.	12.0	.
5	13	1981	.	.	.	.	2.60	18.5	.
6	1	1981	.	.	.	.	.	16.0	.
6	7	1981	.	.	.	.	3.60	18.0	.
6	22	1981	.	.	.	.	.	16.0	.
6	24	1981	.	.	.	.	.	16.0	.
6	28	1981	.	.	.	.	2.50	15.0	.
7	4	1981	.	.	.	.	.	22.0	.
7	17	1981	.	.	.	.	.	20.0	.
7	25	1981	2.00	.	.	.	1.90	19.0	.
8	10	1981	.	.	.	.	.	18.0	.
8	13	1981	.	.	.	.	.	22.0	.
8	31	1981	.	.	.	.	1.90	.	.
9	7	1981	.	.	.	.	.	.	.
9	26	1981	.	.	.	.	1.60	.	.
10	1	1981	.	.	.	.	.	.	.
10	4	1981	.	.	.	.	.	.	.
10	13	1981	.	.	.	.	1.50	.	.
10	17	1981	.	.	.	.	.	.	.
11	6	1981	.	.	.	.	2.50	.	.
4	11	1982	.	.	.	.	.	.	.
4	15	1982	.	.	.	.	.	.	.
4	19	1982	.	.	.	.	.	.	.
4	23	1982	.	.	.	.	.	.	.
5	3	1982	.	.	.	.	.	.	.
5	4	1982	.	.	.	.	.	.	.
5	7	1982	.	.	.	.	.	.	.
5	11	1982	.	.	.	.	.	.	.
5	14	1982	.	.	.	.	.	.	.
5	21	1982	.	.	.	.	.	.	.
6	2	1982	.	.	.	.	.	.	.
6	18	1982	.	.	.	.	.	.	.
7	2	1982	.	.	.	.	.	.	.
7	26	1982	.	.	.	.	.	.	.
8	30	1982	.	.	.	.	.	.	.
10	4	1982	.	.	.	.	.	.	.
11	22	1982	.	.	.	.	.	.	.
3	18	1983	.	.	.	.	.	.	.
4	14	1983	.	.	.	.	.	.	.
5	16	1983	.	.	.	.	.	.	.
6	16	1983	.	.	.	.	.	.	.
7	15	1983	.	.	.	.	.	.	.
8	16	1983	.	.	.	.	.	.	.
9	15	1983	.	.	.	.	.	.	.
10	13	1983	.	.	.	.	.	.	.
11	10	1983	.	.	.	.	.	.	.
12	1	1983	.	.	.	.	.	.	.
3	15	1984	.	.	.	.	.	.	401
4	15	1984	.	.	.	.	.	.	402
5	16	1984	.	.	.	.	.	.	403
6	13	1984	.	.	.	.	.	.	404
7	14	1984	.	.	.	.	.	.	405
8	15	1984	.	.	.	.	.	.	406
9	12	1984	.	.	.	.	.	.	407
10	13	1984	.	.	.	.	.	.	408
11	14	1984	.	.	.	.	.	.	409
11	28	1984	.	.	.	.	.	.	410
12	17	1984	.	.	.	.	.	.	411
4	2	1985	.	.	.	.	.	.	400
4	16	1985	.	.	.	.	.	.	401
5	14	1985	.	.	.	.	.	.	402
6	14	1985	.	.	.	.	.	.	403
7	15	1985	.	.	.	.	.	.	404
8	13	1985	.	.	.	.	.	.	405
9	13	1985	.	.	.	.	.	.	406
10	15	1985	.	.	.	.	.	.	407
11	15	1985	.	.	.	.	.	.	408

Table A2.5.12. Drainage quality data for additional parameters from Duluth Complex test pile FL4, from 1978 to 1989 (0.63% S, 0.35% Cu, 0.08% Ni), page 3 of 3.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
3	23	1986	.	.	.	.	.	.	400
4	16	1986	.	.	.	.	.	.	401
5	14	1986	.	.	.	.	.	.	402
6	11	1986	.	.	.	.	.	.	403
7	12	1986	.	.	.	.	.	.	404
8	13	1986	.	.	.	.	.	.	405
9	14	1986	.	.	.	.	.	.	406
10	16	1986	.	.	.	.	.	.	407
11	16	1986	.	.	.	.	.	.	408
3	22	1987	.	.	.	.	.	.	400
4	15	1987	.	.	.	.	.	.	401
5	16	1987	.	.	.	.	.	.	402
6	16	1987	.	.	.	.	.	.	403
7	15	1987	.	.	.	.	.	.	404
8	15	1987	.	.	.	.	.	.	405
9	16	1987	.	.	.	.	.	.	406
10	13	1987	.	.	.	.	.	.	407
11	10	1987	.	.	.	.	.	.	408
3	28	1988	.	.	.	.	.	.	400
4	17	1988	.	.	.	.	.	.	401
5	12	1988	.	.	.	.	.	.	402
6	12	1988	.	.	.	.	.	.	403
7	15	1988	.	.	.	.	.	.	404
8	16	1988	.	.	.	.	.	.	405
9	16	1988	.	.	.	.	.	.	406
10	18	1988	.	.	.	.	.	.	407
11	15	1988	.	.	.	.	.	.	408
4	7	1989	.	.	.	.	.	.	400
4	24	1989	.	.	.	.	.	.	401
5	16	1989	.	.	.	.	.	.	402
6	17	1989	.	.	.	.	.	.	403
7	17	1989	.	.	.	.	.	.	404
8	14	1989	.	.	.	.	.	.	405
9	14	1989	.	.	.	.	.	.	406
10	16	1989	.	.	.	.	.	.	407
11	4	1989	.	.	.	.	.	.	408

Table A2.5.13. Drainage quality data from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
4	26	1978	.	.	7.25	5000.00	.	.	.	.	478.00	139.00	.
5	25	1978	.	.	7.25	.	.	.	.	.	840.00	152.00	.
6	16	1978	135.0	.	7.12	8860.00	.	.	.	.	1028.00	206.00	.
7	5	1978	223.0	.	7.40	4610.00	.	.	.	2000.00	435.00	141.00	.
7	13	1978	.	.	7.49	.	.	.	.	1450.00	794.00	124.00	.
7	19	1978	.	.	8.10	3700.00	.	.	.	1300.00	540.00	93.00	.
8	2	1978	.	.	7.70	2600.00	.	.	.	950.00	258.00	61.00	.
8	4	1978	287.0	.	7.75	4150.00	.	.	.	1700.00	451.00	120.00	.
8	10	1978	77.0	.	7.55	6840.00	47.0	.	.	2400.00	748.00	178.00	.
8	16	1978	28390.0	.	7.78	2225.00	40.0	.	.	1000.00	250.00	74.00	.
8	17	1978	.	.	7.52	4600.00	46.0	.	.	1700.00	396.00	46.00	.
8	23	1978	.	.	7.55	4320.00	30.0	.	.	1600.00	368.00	112.00	.
8	25	1978	.	.	7.40	4750.00	40.0	.	.	1800.00	373.00	119.00	.
8	28	1978	.	.	7.85	5290.00	57.0	.	.	2150.00	481.00	135.00	.
9	13	1978	667.0	.	7.16	3400.00	25.5	.	.	2075.00	387.50	120.00	.
10	17	1978	.	.	6.20	5320.00	6.7	.	.	.	491.00	133.00	.
3	23	1979	2290.0	46.0	6.85	1562.00	10.0	.	.	600.00	200.00	54.00	.
4	12	1979	1665.4	840.0	6.00	2714.00	.	.	.	1400.00	327.00	93.20	.
4	16	1979	3168.0	1308.0	6.60	1358.00	.	.	.	560.00	152.60	74.20	.
4	17	1979	3641.2	3916.0	6.60	1146.00	12.0	.	.	440.00	124.60	31.60	.
4	18	1979	6529.0	2997.0	6.80	1318.00	16.0	.	.	488.00	162.00	38.40	.
4	19	1979	3970.5	3862.0	6.50	1054.00	15.0	.	.	210.00	141.00	34.00	.
4	20	1979	5257.4	3929.0	6.70	1498.00	16.5	.	.	558.50	185.00	56.10	.
4	25	1979	5000.6	719.0	6.65	2613.00	14.0	.	.	.	292.40	103.40	.
5	2	1979	3387.4	1902.0	6.40	3460.00	4.0	.	.	1650.00	394.40	149.00	.
5	9	1979	1654.0	1286.0	6.60	3482.00	.	.	.	1030.00	312.00	122.00	.
5	10	1979	10249.8	4690.0	7.00	2432.00	.	.	.	867.00	240.20	104.20	.
5	20	1979	4644.2	1266.0	.	.	.	.	.	1750.00	341.00	130.00	.
5	22	1979	2187.7	1399.0	7.20	2201.00	.	.	.	883.00	211.00	86.00	.
5	23	1979	2653.3	1007.0	7.00	3377.00	.	.	.	1700.00	316.00	60.40	.
6	1	1979	2567.3	729.0	6.90	2668.00	.	.	.	1050.00	231.20	56.60	.
6	6	1979	2570.0	1130.0	6.40	2196.00	.	.	.	867.00	199.40	79.80	.
6	10	1979	2573.8	1433.0	6.90	1916.00	14.0	.	.	700.00	184.80	77.60	.
6	12	1979	1506.4	494.0	6.60	2672.50	10.0	.	.	1200.00	266.30	107.50	.
6	16	1979	1612.4	442.0	6.70	3450.00	9.9	.	.	1230.00	325.60	127.00	.
6	21	1979	2233.2	1687.0	6.70	2411.00	11.0	.	.	750.00	202.00	87.00	.
6	30	1979	1990.9	230.0	6.30	3074.00	6.6	.	.	1450.00	308.00	136.00	.
7	10	1979	1627.6	98.0	6.20	4599.00	4.8	.	.	2200.00	467.00	193.50	.
7	30	1979	5268.7	3374.0	5.50	2006.00	3.0	.	.	925.00	209.00	97.00	.
8	4	1979	1956.8	284.0	5.40	3438.00	4.0	.	.	1670.00	444.00	169.00	.
8	13	1979	753.2	108.0	5.00	4690.00	2.4	.	.	2420.00	540.00	126.20	.
8	23	1979	1192.3	342.0	4.80	3739.00	1.6	.	.	1800.00	439.40	174.00	.
8	28	1979	847.8	709.0	5.30	3154.00	6.0	.	.	1400.00	345.00	112.60	.
9	1	1979	5552.6	2337.0	5.00	2576.00	2.0	.	.	1200.00	327.20	112.40	.
9	5	1979	987.9	611.0	4.70	3819.00	0.4	.	.	1980.00	478.80	121.60	.
9	11	1979	3841.8	846.0	4.90	3364.00	2.0	.	.	1650.00	398.80	160.80	.
9	13	1979	2112.0	699.0	4.90	3745.00	2.8	.	.	1800.00	444.00	175.00	.
9	19	1979	1567.0	176.0	4.70	5710.00	2.4	.	.	2680.00	692.00	272.00	.
10	18	1979	931.1	758.0	4.60	5168.00	0.8	.	.	2600.00	646.00	254.00	.
10	19	1979	2161.2	1863.0	4.70	2356.00	2.4	.	.	1050.00	251.00	106.00	.
10	31	1979	6423.1	2331.0	4.60	3448.00	0.8	.	.	1650.00	408.00	177.00	.
11	1	1979	4920.5	1170.0	4.50	3701.50	.	.	.	1750.00	447.00	203.50	.
3	29	1980	1362.7	390.0	4.90	1050.00	2.3	.	.	440.00	94.20	35.90	.
4	1	1980	2536.2	1173.0	5.00	800.00	3.4	.	.	360.00	74.00	26.40	.
4	6	1980	4542.5	1789.0	5.50	1025.00	4.6	.	.	490.00	101.60	45.00	.
4	14	1980	3785.4	540.0	4.50	.	.	.	.	875.00	226.80	98.20	.
4	18	1980	2044.1	568.0	4.60	.	1.2	.	.	1100.00	248.00	106.00	.
4	28	1980	1817.0	200.0	4.80	3590.00	3.3	.	.	1820.00	375.00	164.00	.
5	10	1980	583.0	171.0	4.70	3975.00	2.2	.	.	2020.00	420.40	186.00	.
5	31	1980	3058.6	1946.0	4.60	2400.00	2.2	.	.	1350.00	246.00	114.00	.
6	3	1980	1018.3	207.0	4.60	3350.00	.	.	.	1750.00	381.00	177.00	.
6	6	1980	935.0	537.0	4.60	3100.00	1.1	.	.	1650.00	316.00	169.00	.
6	13	1980	688.9	464.0	4.60	3300.00	1.1	.	.	1650.00	317.00	156.00	.
6	18	1980	348.3	49.0	4.30	3800.00	.	.	.	1800.00	359.20	184.00	.
6	28	1980	1321.1	476.0	4.50	3400.00	.	.	.	1650.00	311.00	153.00	.

Table A2.5.13. Drainage quality data from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
7	2	1980	4334.3	1305.0	4.50	2500.00	.	.	.	.	274.70	134.70	.
7	7	1980	1374.1	238.0	4.00	3800.00	.	.	.	1800.00	401.00	205.00	.
7	17	1980	1253.0	458.0	4.30	5000.00	.	.	.	2200.00	475.80	296.00	.
7	26	1980	1465.0	329.0	4.30	4450.00	.	.	.	1880.00	421.20	219.00	.
8	3	1980	957.7	528.0	4.20	4800.00	.	.	.	1980.00	434.20	232.50	.
8	12	1980	2271.2	900.0	4.30	3150.00	.	.	.	1550.00	303.00	161.00	.
8	19	1980	2831.5	464.0	4.30	3400.00	.	.	.	2150.00	375.60	211.60	.
8	22	1980	1926.8	922.0	.	3150.00	.	.	.	1900.00	280.50	244.00	.
8	29	1980	1362.7	5237.0	4.30	3625.00	.	.	.	2200.00	406.00	321.00	.
9	11	1980	950.1	589.0	4.30	2845.00	.	.	.	1700.00	316.00	231.00	.
9	12	1980	5208.7	2795.0	4.40	2383.00	.	.	.	1260.00	218.00	185.00	.
9	18	1980	8108.4	2740.0	4.40	2625.00	.	.	.	1600.00	248.00	213.00	.
9	23	1980	3592.4	798.0	4.20	3846.00	.	.	.	2120.00	400.00	309.00	.
10	22	1980	2880.7	710.0	4.10	3160.00	.	.	.	2000.00	330.00	252.00	.
10	30	1980	1169.7	85.0	4.20	2250.00	.	.	.	1700.00	314.00	158.00	.
2	19	1981	1472.0	1423.0	3.70	2050.00	.	.	.	950.00	172.90	110.50	.
2	21	1981	1484.0	325.0	4.50	2550.00	.	.	.	1560.00	207.40	201.50	.
3	28	1981	8842.0	3797.0	4.70	1300.00	0.8	.	.	625.00	94.90	62.40	.
4	3	1981	7692.0	2030.0	4.70	1900.00	1.6	.	.	1100.00	146.90	110.50	.
4	13	1981	3376.0	1436.0	4.50	2500.00	.	.	.	1200.00	191.00	147.00	.
4	17	1981	2858.0	1439.0	4.70	2700.00	1.6	.	.	1380.00	148.00	156.00	.
4	23	1981	3978.0	3580.0	4.70	3000.00	1.6	.	.	1415.00	219.00	161.00	.
4	25	1981	8260.0	2934.0	4.80	3100.00	2.4	.	.	1480.00	200.00	168.00	.
4	28	1981	5534.0	2409.0	4.90	2950.00	4.0	.	.	1350.00	205.00	179.00	.
4	29	1981	5307.0	1392.0	4.70	3515.00	.	.	.	1850.00	295.00	280.00	.
5	12	1981	3441.0	272.0	4.50	3708.00	.	.	.	2250.00	358.00	370.00	.
6	1	1981	2960.0	1196.0	4.50	2660.00	.	.	.	1350.00	244.00	180.00	.
6	6	1981	3274.0	289.0	4.45	1872.00	.	.	.	1160.00	188.00	112.00	.
6	22	1981	4766.0	2996.0	4.20	2650.00	.	.	.	1660.00	271.00	175.00	.
6	24	1981	4993.0	2335.0	4.40	2342.00	.	.	.	1620.00	246.00	154.00	.
6	28	1981	6878.0	3559.0	4.50	1657.00	.	.	.	1910.00	302.00	290.00	.
7	4	1981	6197.0	295.0	4.40	3568.00	.	.	.	2300.00	370.00	183.30	.
7	15	1981	1628.0	102.0	4.25	3526.00	.	.	.	2250.00	362.00	278.60	.
7	24	1981	1938.0	437.0	4.40	2830.00	.	.	.	1830.00	290.00	212.60	.
8	4	1981	1783.0	106.0	4.00	2951.00	.	.	.	2030.00	326.00	142.00	.
8	17	1981	1033.0	47.0	4.05	3238.00	.	.	.	2150.00	348.00	260.00	.
8	31	1981	1136.0	1000.0	4.10	3930.00	.	.	.	2560.00	360.00	270.00	.
9	7	1981	1109.0	588.0	4.10	4200.00	.	.	.	2220.00	342.00	244.00	.
9	16	1981	431.0	20.0	4.10	4300.00	.	.	.	2640.00	378.00	284.00	.
9	26	1981	1949.0	1176.0	4.10	3550.00	.	.	.	2220.00	328.00	240.00	.
10	1	1981	4675.0	2859.0	4.20	3225.00	.	.	.	1840.00	298.00	224.00	.
10	4	1981	3607.0	1859.0	4.20	3700.00	.	.	.	2600.00	332.00	354.00	.
10	14	1981	5216.0	1771.0	4.20	3400.00	.	.	.	1960.00	276.00	222.00	.
10	15	1981	9906.0	2503.0	4.20	4000.00	.	.	.	2760.00	252.00	304.00	.
11	1	1981	3516.0	193.0	4.10	4000.00	.	.	.	2760.00	324.00	350.00	.
3	30	1982	2336.0	.	4.20	1825.00	.	.	.	1000.00	138.00	102.00	.
4	11	1982	1669.0	.	4.30	2350.00	.	.	.	1500.00	182.00	158.00	.
4	13	1982	4906.0	.	4.40	1050.00	.	.	.	570.00	78.80	65.70	.
4	15	1982	4179.0	.	4.40	1500.00	.	.	.	900.00	117.60	96.40	.
4	19	1982	2245.0	.	4.30	1775.00	.	.	.	1100.00	176.00	144.00	.
4	23	1982	2392.0	.	4.20	2425.00	.	.	.	1600.00	206.00	176.00	.
5	3	1982	572.0	.	4.15	3100.00	.	.	.	2300.00	204.00	190.00	.
5	4	1982	4902.0	.	4.20	2900.00	.	.	.	2000.00	192.00	208.00	.
5	10	1982	4664.0	.	4.30	2875.00	.	.	.	1900.00	184.00	208.00	.
5	11	1982	5337.0	.	4.30	3100.00	.	.	.	1700.00	202.00	223.00	.
5	14	1982	5773.0	.	4.30	3400.00	.	.	.	2200.00	250.00	288.00	.
5	21	1982	2052.0	.	4.20	5000.00	.	.	.	3600.00	340.00	382.00	.
6	2	1982	1794.0	.	4.10	3700.00	.	.	.	2700.00	250.00	280.00	.
6	18	1982	1185.0	.	4.20	3400.00	.	.	.	2400.00	264.00	264.00	.
7	2	1982	31423.0	.	.	.	.	.	.	1965.00	332.00	208.00	.
7	26	1982	14021.0	.	3.94	3531.00	.	.	.	2150.00	318.00	258.00	.
8	30	1982	16239.0	.	3.97	3319.00	.	.	.	1920.00	366.00	234.00	.
10	4	1982	20358.0	.	4.27	3073.00	.	.	.	1900.00	250.00	260.00	.
11	1	1982	2491.0	.	3.90	2980.00	.	.	.	.	176.00	222.00	.
11	22	1982	.	.	4.10	4100.00	.	.	.	3300.00	236.00	326.00	.

Table A2.5.13. Drainage quality data from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
3	18	1983	3286.0	.	4.35	1425.00	.	.	.	780.00	.	.	.
4	14	1983	12694.0	.	4.33	1387.00	.	.	.	700.00	.	.	.
5	15	1983	6279.0	.	4.21	2532.00	.	.	.	2500.00	.	.	.
6	15	1983	5022.0	.	4.25	2173.00	.	.	.	1940.00	.	.	.
7	14	1983	9466.0	.	4.14	3960.00	.	.	.	5160.00	.	.	.
8	14	1983	14103.0	.	4.08	2861.00	.	.	.	2300.00	.	.	.
9	15	1983	8668.0	.	4.16	3486.00	.	.	.	2950.00	.	.	.
10	13	1983	19970.0	.	4.32	3405.00	.	.	.	2400.00	.	.	.
11	9	1983	4269.0	.	4.10	2750.00	.	.	.	2250.00	.	.	.
11	30	1983	628.0	.	4.00	2700.00	.	.	.	2820.00	.	.	.
2	24	1984	75.0	.	4.00	2175.00	.	.	.	2300.00	276.80	414.30	600
3	15	1984	4023.0	.	4.25	1100.00	.	.	.	830.00	110.40	105.70	601
4	15	1984	20316.0	.	4.33	1472.00	.	.	.	1540.00	188.20	170.30	602
5	16	1984	11761.0	.	4.22	1958.00	.	.	.	2500.00	236.30	279.50	603
6	13	1984	23280.0	.	4.16	2320.00	.	.	.	2300.00	256.10	328.60	604
7	14	1984	8771.0	.	4.19	2930.00	.	.	.	2700.00	.	.	605
8	15	1984	7737.0	.	4.10	3385.00	.	.	.	3020.00	379.70	277.60	606
9	12	1984	2612.0	.	3.96	4095.00	.	.	.	3900.00	408.20	510.00	607
10	13	1984	15261.0	.	3.94	2720.00	.	.	.	3459.00	402.60	430.00	608
11	14	1984	2615.0	.	4.11	3160.00	.	.	.	4125.00	430.00	585.00	609
12	1	1984	492.0	.	3.95	2150.00	.	.	.	3000.00	332.00	426.00	610
12	18	1984	2215.0	.	4.30	2025.00	.	.	.	2200.00	252.00	302.00	611
4	2	1985	2551.0	.	4.50	1625.00	.	.	.	1130.00	158.00	138.80	600
4	16	1985	16537.0	.	4.40	2200.00	.	.	.	1500.00	238.00	256.00	601
5	14	1985	9852.0	.	4.34	3100.00	.	.	.	2750.00	240.00	364.00	602
6	14	1985	34814.0	.	4.29	3500.00	.	.	.	3200.00	281.00	453.00	603
7	15	1985	14584.0	.	4.16	3100.00	.	.	.	2773.00	276.00	377.00	604
8	13	1985	15114.0	.	4.11	2700.00	.	.	.	2400.00	268.00	252.00	605
9	13	1985	31741.0	.	4.15	3100.00	.	.	.	2550.00	309.00	270.00	606
10	15	1985	20420.0	.	4.20	3300.00	.	.	.	2430.00	244.00	319.00	607
11	15	1985	2646.0	.	4.27	3250.00	.	.	.	1750.00	284.00	440.00	608
3	23	1986	17289.0	.	4.50	1500.00	.	.	.	900.00	114.00	122.00	600
4	16	1986	16332.0	.	4.36	2025.00	.	.	.	1400.00	204.00	190.00	601
5	14	1986	12740.0	.	4.35	2325.00	.	.	.	1950.00	276.00	262.00	602
6	11	1986	7631.0	.	4.30	2750.00	.	.	.	2000.00	236.00	250.00	603
7	12	1986	2365.0	.	4.27	2650.00	.	.	.	1750.00	260.00	264.00	604
8	13	1986	13346.0	.	4.13	3200.00	.	.	.	2350.00	300.00	301.00	605
9	14	1986	27335.0	.	4.12	3000.00	.	.	.	1800.00	280.00	302.00	606
10	16	1986	6442.0	.	4.06	3425.00	.	.	.	2400.00	288.00	318.00	607
11	16	1986	10954.0	.	3.99	3475.00	.	.	.	3250.00	332.00	544.00	608
3	28	1987	12608.0	.	4.40	2300.00	.	.	.	1750.00	202.00	234.00	600
4	18	1987	2525.0	.	4.31	2150.00	.	.	.	1650.00	198.00	206.00	601
5	16	1987	28020.0	.	3.86	2750.00	.	.	.	2700.00	335.00	397.00	602
6	16	1987	1419.0	.	4.17	3950.00	.	.	.	3000.00	348.00	458.00	603
7	15	1987	29811.0	.	4.00	3325.00	.	.	.	2715.00	294.00	340.00	604
8	15	1987	13350.0	.	3.72	3050.00	.	.	.	2500.00	288.00	304.00	605
9	16	1987	16885.0	.	3.84	4400.00	.	.	.	2850.00	323.00	352.00	606
10	13	1987	1915.0	.	3.92	5000.00	.	.	.	4300.00	386.00	560.00	607
11	10	1987	1601.0	.	4.03	4375.00	.	.	.	3760.00	342.00	500.00	608
3	28	1988	7574.0	.	4.25	1500.00	.	.	.	880.00	142.00	106.00	600
4	17	1988	6635.0	.	4.20	2125.00	.	.	.	1860.00	170.00	186.00	601
5	12	1988	12044.0	.	4.11	3275.00	.	.	.	2300.00	298.00	322.00	602
6	12	1988	2858.0	.	4.10	3325.00	.	.	.	2600.00	346.00	342.00	603
7	15	1988	4553.0	.	4.10	3250.00	.	.	.	2440.00	312.00	314.00	604
8	16	1988	60076.0	.	4.06	3475.00	.	.	.	2400.00	302.00	352.00	605
9	16	1988	12025.0	.	4.04	3050.00	.	.	.	2000.00	246.00	290.00	606
10	18	1988	3664.0	.	4.12	2950.00	.	.	.	2160.00	212.00	274.00	607
11	15	1988	7017.0	.	4.30	2625.00	.	.	.	2080.00	210.00	265.00	608
11	30	1988	681.0	.	4.20	3600.00	.	.	.	2500.00	232.00	320.00	609
4	7	1989	18641.0	.	4.65	900.00	1.1	.	.	1400.00	80.00	72.00	600
4	24	1989	8804.0	.	4.40	1975.00	.	.	.	1340.00	150.00	156.00	601
5	16	1989	11620.0	.	4.27	2550.00	.	.	.	1700.00	202.00	216.00	602
6	17	1989	35500.0	.	4.28	3500.00	.	.	.	2950.00	304.00	354.00	603
7	17	1989	2952.0	.	4.13	4625.00	.	.	.	3800.00	440.00	460.00	604
8	14	1989	4466.0	.	4.17	2950.00	.	.	.	2060.00	232.00	282.00	605

Table A2.5.13. Drainage quality data from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
9	14	1989	17161.0	.	4.01	4325.00	.	.	.	3500.00	356.00	416.00	606
10	16	1989	1832.0	.	4.05	4800.00	.	.	.	3600.00	360.00	462.00	607
11	10	1989	999.0	.	4.20	3050.00	.	.	.	2000.00	214.00	300.00	608
3	26	1990	34.0	.	4.15	5000.00	.	.	.	5440.00	468.00	744.00	600
4	12	1990	3883.0	.	4.39	3125.00	.	.	.	2280.00	244.00	288.00	601
5	17	1990	927.0	.	4.32	2675.00	.	.	.	2040.00	226.00	230.00	602
6	18	1990	2998.0	.	4.07	3550.00	.	.	.	2440.00	266.00	286.00	603
7	16	1990	1658.0	.	4.10	3490.00	.	.	.	2460.00	270.00	286.00	604
8	13	1990	700.0	.	4.05	3100.00	.	.	.	1970.00	190.00	286.00	605
9	10	1990	5583.0	.	4.10	3300.00	.	.	.	2160.00	242.00	312.00	606
10	15	1990	20246.0	.	3.98	4295.00	.	.	.	3980.00	334.00	438.00	607
11	16	1990	167.0	.	4.35	5000.00	.	.	.	4400.00	410.00	640.00	608
4	8	1991	10795.0	.	.	.	.	.	.	.	.	.	.
4	22	1991	7960.0	.	4.22	3625.00	.	.	.	2620.00	238.00	350.00	2022
5	20	1991	9122.0	.	4.18	4675.00	.	.	.	3260.00	284.00	452.00	2026
6	17	1991	10000.0	.	4.18	3350.00	.	.	.	2080.00	208.00	317.00	2042
7	15	1991	29610.0	.	4.07	4800.00	.	.	.	2500.00	246.00	378.00	2058
8	12	1991	1185.0	.	4.10	5000.00	.	.	.	3280.00	286.00	430.00	2074
9	9	1991	20079.0	.	4.09	5000.00	.	.	.	3260.00	292.00	404.00	2089
10	7	1991	6344.0	.	4.17	4450.00	.	.	.	2640.00	236.00	346.00	2105
11	7	1991	11692.0	.	4.11	4775.00	.	.	.	2360.00	212.00	304.00	2121
4	23	1992	5901.0	.	4.29	1475.00	.	140.0	122.5	650.00	68.00	92.00	17003
5	11	1992	2529.0	.	4.21	3190.00	.	325.0	315.4	1760.00	186.00	234.00	17016
5	26	1992	3203.0	.	4.32	2450.00	.	265.0	261.6	1640.00	140.00	224.00	17044
6	8	1992	822.0	.	4.30	5000.00	.	785.0	644.7	4060.00	300.00	600.00	17048
6	22	1992	872.0	.	4.32	2710.00	.	290.0	270.2	1740.00	134.00	248.00	17052
7	7	1992	3764.0	.	4.25	2510.00	.	260.0	259.4	790.00	146.00	208.00	17057
7	20	1992	894.0	.	4.18	4480.00	.	560.0	491.7	696.00	238.00	418.00	17061
8	3	1992	1159.0	.	4.21	2640.00	.	280.0	292.5	1780.00	146.00	258.00	17077
8	17	1992	1565.0	.	4.23	3400.00	.	370.0	354.3	2360.00	210.00	324.00	17081
8	31	1992	1482.0	.	4.24	2500.00	.	250.0	242.1	1460.00	146.00	210.00	17097
9	14	1992	5218.0	.	4.20	3200.00	.	415.0	340.0	2180.00	202.00	312.00	17101
9	28	1992	2694.0	.	4.21	4850.00	.	485.0	448.5	3120.00	254.00	416.00	17118
10	12	1992	444.0	.	4.23	4900.00	.	525.0	536.1	3280.00	272.00	460.00	17122
10	27	1992	379.0	.	4.18	3830.00	.	425.0	395.4	2440.00	204.00	366.00	17138
11	9	1992	119.0	.	4.25	4500.00	.	.	.	.	.	.	.
11	30	1992	805.0	.	4.30	2300.00	.	300.0	244.5	1570.00	142.00	230.00	17153
4	8	1993	7949.0	.	4.40	1350.00	.	.	.	770.00	72.50	110.00	17165
4	22	1993	5257.0	.	4.40	1440.00	.	.	.	825.00	77.50	115.00	17176
5	6	1993	12820.0	.	4.23	2750.00	.	.	.	1950.00	197.50	247.50	17181
5	20	1993	2086.0	.	4.31	3200.00	.	.	.	2340.00	217.50	310.00	17196
6	3	1993	6945.0	.	4.37	1600.00	.	.	.	1255.00	120.00	172.50	17200
6	17	1993	7559.0	.	4.48	2350.00	.	.	.	1670.00	167.50	270.00	17216
7	1	1993	8928.0	.	4.13	4200.00	.	.	.	3180.00	270.00	490.00	17220
7	14	1993	15912.0	.	4.22	4220.00	.	.	.	3020.00	266.00	422.00	17237
7	29	1993	13240.0	.	4.23	1700.00	.	.	.	1130.00	118.00	138.00	17241
8	12	1993	7184.0	.	4.15	2425.00	.	.	.	1680.00	166.00	188.00	17257
8	26	1993	11336.0	.	4.13	2600.00	.	.	.	1520.00	158.00	240.00	17262
9	9	1993	11866.0	.	4.30	2700.00	.	.	.	1670.00	174.00	232.00	17277
9	23	1993	3955.0	.	4.21	2700.00	.	.	.	1720.00	174.00	250.00	17281
10	7	1993	2139.0	.	4.28	3450.00	.	.	.	2180.00	164.40	286.00	17297
10	20	1993	814.0	.	4.22	3925.00	.	.	.	2520.00	252.00	366.00	17301
11	4	1993	2139.0	.	4.25	3300.00	.	.	.	970.00	191.60	278.00	17319
11	22	1993	288.0	.	.	.	.	.	.	.	.	.	.

Table A2.5.14. Drainage quality data for metals from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
4	26	1978	535.00	12.00	.	0.088	0.540	0.030	0.022	0.280	0.600	.
5	25	1978	862.00	18.00	.	0.121	0.740	0.062	0.003	0.120	1.140	.
6	16	1978	1.11E3	22.00	.	0.258	0.665	0.059	0.033	0.120	1.640	.
7	5	1978	491.00	17.00	.	0.060	0.735	0.051	0.006	0.020	0.700	.
7	13	1978	685.00	16.00	.	0.118	1.080	0.067	0.014	0.080	0.950	.
7	19	1978	508.00	12.00	.	0.092	0.860	0.052	0.006	0.050	0.750	.
8	2	1978	260.00	9.00	.	0.060	0.630	0.035	0.020	0.050	0.620	.
8	4	1978	.	12.00	.	0.120	0.930	0.140	0.030	0.040	1.040	.
8	10	1978	700.00	22.00	.	0.170	1.430	0.280	0.050	0.150	1.680	.
8	16	1978	257.00	6.00	.	0.095	0.935	0.160	0.030	0.060	0.820	.
8	17	1978	200.00	14.00	.	0.130	1.480	0.270	0.050	0.050	1.380	.
8	23	1978	396.00	12.00	.	0.130	1.540	0.240	0.050	0.050	1.400	.
8	25	1978	432.00	12.00	.	0.150	1.700	0.270	0.050	0.070	1.530	.
8	28	1978	681.00	15.00	.	0.180	2.150	0.390	0.070	0.070	1.820	.
9	13	1978	446.50	14.00	.	0.285	2.335	0.220	0.120	0.050	2.140	.
10	17	1978	609.00	13.00	.	1.160	4.120	0.400	0.360	0.040	3.150	.
3	23	1979	97.20	4.00	.	0.200	4.590	0.360	0.150	0.020	1.740	.
4	12	1979	201.00	6.20	.	0.740	10.250	0.710	0.290	0.030	3.400	.
4	16	1979	91.60	3.40	.	0.120	2.320	0.210	0.080	1.020	1.170	.
4	17	1979	76.20	3.00	.	0.120	2.350	0.210	0.090	0.020	1.180	.
4	18	1979	86.00	2.60	.	0.170	2.630	0.230	0.100	0.020	1.380	.
4	19	1979	72.00	2.00	.	0.130	2.140	0.200	0.190	0.030	1.120	.
4	20	1979	95.00	3.00	.	0.130	3.145	0.290	0.100	0.020	1.580	.
4	25	1979	205.60	4.60	.	0.280	5.250	0.420	0.200	0.030	2.740	.
5	2	1979	270.00	7.00	.	0.510	9.000	0.590	0.300	0.050	4.070	.
5	9	1979	238.00	4.00	.	0.490	7.000	0.480	0.270	0.040	3.230	.
5	10	1979	200.20	4.40	.	0.340	4.470	0.360	0.180	0.010	2.480	.
5	20	1979	292.00	5.00	.	0.470	7.440	0.470	0.290	0.030	3.230	.
5	22	1979	170.00	3.00	.	0.330	3.900	0.290	0.170	0.020	2.020	.
5	23	1979	269.00	6.60	.	0.490	6.110	0.460	0.250	0.060	3.050	.
6	1	1979	212.60	5.20	.	0.435	4.160	0.350	0.215	0.050	2.280	.
6	6	1979	86.60	2.00	.	0.330	3.360	0.240	0.170	0.010	1.790	.
6	10	1979	79.40	3.80	.	0.380	3.520	0.126	0.140	0.020	1.600	.
6	12	1979	154.00	3.30	.	0.375	4.500	0.350	0.210	0.010	2.260	.
6	16	1979	189.00	4.40	.	0.520	5.660	0.430	0.290	0.040	2.770	.
6	21	1979	155.00	4.00	.	0.390	3.880	0.240	0.180	0.010	5.570	.
6	30	1979	249.00	6.00	.	0.720	5.820	0.380	0.320	0.030	2.590	.
7	10	1979	409.50	9.50	.	1.755	9.960	0.620	0.665	0.030	4.100	.
7	30	1979	105.00	4.00	.	1.450	6.010	0.390	0.290	0.010	2.040	.
8	4	1979	242.00	9.00	.	2.720	10.150	0.660	0.580	0.040	3.720	.
8	13	1979	438.00	13.80	.	4.810	15.000	1.130	1.010	0.070	6.130	.
8	23	1979	232.00	11.40	.	4.220	14.300	1.080	0.840	0.030	5.240	.
8	28	1979	260.00	8.20	.	3.790	13.700	1.000	0.700	0.060	4.230	.
9	1	1979	136.00	6.80	.	3.380	11.300	0.820	0.510	0.050	3.350	.
9	5	1979	308.00	10.60	.	5.270	16.200	1.180	0.840	0.050	5.330	.
9	11	1979	229.00	9.80	.	4.730	14.000	1.040	0.720	0.060	4.700	.
9	13	1979	238.00	10.20	.	5.020	15.400	1.130	0.760	0.080	5.000	.
9	19	1979	478.00	16.20	.	6.800	23.600	1.700	1.310	0.160	8.100	.
10	18	1979	450.00	14.60	.	5.960	26.600	1.910	1.340	0.300	8.450	.
10	19	1979	142.00	5.60	.	4.400	15.500	1.090	0.630	0.040	3.700	.
10	31	1979	198.00	7.80	.	7.200	24.000	2.090	1.020	0.110	5.780	.
11	1	1979	230.00	9.10	.	8.535	25.500	2.170	1.085	0.130	6.180	.
3	29	1980	48.50	2.30	.	1.880	7.360	0.445	0.220	0.050	1.230	.
4	1	1980	27.60	1.40	.	1.180	4.630	0.300	0.150	0.020	1.000	.
4	6	1980	36.60	2.10	.	1.790	7.045	0.445	0.260	0.030	1.440	.
4	14	1980	92.80	4.00	.	5.290	18.000	1.290	0.750	0.040	4.110	.
4	18	1980	106.00	4.40	.	5.670	21.300	1.490	0.820	0.130	4.710	.
4	28	1980	184.60	6.60	.	8.810	34.000	2.320	1.270	0.100	7.820	.
5	10	1980	210.00	8.00	.	9.250	38.000	2.440	1.360	0.100	8.780	.
5	31	1980	101.60	5.00	.	5.870	21.130	1.420	0.760	0.050	4.240	.
6	3	1980	149.00	6.60	.	7.620	27.700	1.890	1.010	0.050	5.990	.
6	6	1980	131.60	6.00	.	7.630	21.000	1.790	1.000	0.060	5.500	.
6	13	1980	142.80	6.60	.	7.940	27.270	1.810	0.950	0.100	5.410	.
6	18	1980	171.00	7.40	.	7.710	27.700	1.840	0.970	0.130	5.800	.
6	28	1980	148.00	6.40	.	7.350	25.000	2.100	1.040	0.100	5.550	.
7	2	1980	91.50	6.00	.	9.500	24.500	2.000	1.020	0.070	4.570	.

Table A2.5.14. Drainage quality data for metals from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
7	7	1980	172.00	7.40	.	12.000	35.000	2.640	1.290	0.090	6.520	.
7	17	1980	318.00	11.80	.	13.450	42.000	2.680	1.560	0.280	9.000	.
7	26	1980	244.00	9.80	.	11.180	32.000	2.250	1.280	0.110	7.180	.
8	3	1980	323.00	10.40	.	11.000	40.000	3.040	1.520	0.140	8.250	.
8	12	1980	152.00	7.40	.	13.000	33.000	2.460	1.190	0.140	6.170	.
8	19	1980	166.60	9.00	.	20.000	44.000	3.510	1.740	0.170	7.900	.
8	22	1980	126.50	3.50	.	19.000	38.000	2.670	1.530	0.190	6.880	.
8	29	1980	229.00	8.00	.	18.000	45.000	3.160	1.800	0.410	8.990	.
9	11	1980	172.00	7.00	.	16.000	35.000	2.640	1.520	0.150	6.980	.
9	12	1980	118.00	4.00	.	19.000	35.500	2.565	1.440	0.200	6.320	.
9	18	1980	147.00	4.00	.	18.000	34.000	2.400	1.430	0.140	5.630	.
9	23	1980	226.00	31.00	.	20.900	54.600	3.150	1.990	0.240	8.820	.
10	22	1980	166.00	8.00	.	20.700	56.200	3.210	1.910	0.520	8.250	.
10	30	1980	84.00	5.00	.	17.000	40.000	2.470	1.380	0.120	5.790	.
2	19	1981	123.50	.	.	13.300	50.000	2.540	1.270	.	.	.
2	21	1981	158.00	.	.	20.600	82.500	4.190	2.320	.	.	.
3	28	1981	48.10	.	.	14.400	41.000	2.110	1.100	.	.	.
4	3	1981	84.50	.	.	20.300	64.000	3.300	1.830	.	.	.
4	13	1981	69.00	.	.	20.200	60.000	3.480	1.870	.	.	.
4	17	1981	58.00	.	.	18.600	56.000	3.280	1.940	.	.	.
4	23	1981	97.00	.	.	18.400	57.000	3.380	1.980	.	.	.
4	25	1981	95.00	.	.	18.400	57.000	3.420	2.090	.	.	.
4	28	1981	99.00	.	.	16.300	51.000	3.050	1.860	.	.	.
4	29	1981	134.00	.	.	17.700	43.000	3.600	2.170	.	.	.
5	12	1981	184.00	.	.	19.940	73.000	4.600	2.670	.	.	.
6	1	1981	110.00	.	.	16.620	38.000	2.870	1.790	.	.	.
6	6	1981	52.00	.	.	13.340	26.000	2.270	1.330	.	.	.
6	22	1981	93.00	.	.	17.920	43.000	3.360	1.730	.	.	.
6	24	1981	70.00	.	.	22.240	68.000	3.680	1.950	.	.	.
6	28	1981	112.00	.	.	20.570	42.000	3.550	1.760	.	.	.
7	4	1981	166.20	.	.	29.500	68.000	3.820	2.530	.	.	.
7	15	1981	175.00	.	.	29.000	72.000	4.100	2.660	.	.	.
7	24	1981	108.00	.	.	26.500	56.000	3.310	2.050	.	.	.
8	4	1981	108.00	.	.	27.500	71.500	3.920	1.980	.	.	.
8	17	1981	108.00	8.00	.	27.500	65.000	3.670	2.230	.	.	.
8	31	1981	128.00	.	.	34.000	95.000	5.170	3.020	.	.	.
9	7	1981	104.00	.	.	34.000	84.000	4.420	2.660	.	.	.
9	16	1981	114.00	.	.	33.000	92.000	4.890	2.920	.	.	.
9	26	1981	74.00	.	.	38.000	82.000	4.360	2.570	.	.	.
10	1	1981	56.00	.	.	42.000	80.000	4.460	2.280	.	.	.
10	4	1981	76.00	10.00	.	52.000	107.000	8.000	3.760	.	.	.
10	14	1981	90.00	8.00	.	39.000	78.000	5.000	3.080	.	.	.
10	15	1981	100.00	22.00	.	123.000	306.000	21.000	10.000	.	.	.
11	1	1981	118.00	8.00	.	54.000	120.000	6.700	3.360	.	.	.
3	30	1982	34.00	4.00	.	19.000	49.000	2.800	1.500	.	.	.
4	11	1982	48.00	4.00	.	29.000	75.000	4.400	2.340	.	.	.
4	13	1982	16.00	2.00	.	10.000	24.000	1.400	0.700	.	.	.
4	15	1982	24.00	2.00	.	13.000	37.000	2.300	1.050	.	.	.
4	19	1982	34.00	4.00	.	22.000	52.000	3.000	1.490	.	.	.
4	23	1982	58.00	4.00	.	34.000	83.000	4.700	2.430	.	.	.
5	3	1982	78.00	6.00	.	55.000	128.000	8.100	4.200	.	.	.
5	4	1982	62.00	6.00	.	46.000	110.000	6.100	3.380	.	.	.
5	10	1982	56.00	6.00	.	46.000	104.000	5.800	3.220	.	.	.
5	11	1982	84.00	6.00	.	49.500	123.500	6.900	3.890	.	.	.
5	14	1982	84.00	6.80	.	42.000	114.000	5.200	3.150	0.060	7.820	.
5	21	1982	95.00	8.00	.	57.000	169.000	7.600	4.360	0.130	11.440	.
6	2	1982	100.00	6.60	.	39.000	121.000	5.700	3.720	0.140	7.770	.
6	18	1982	79.00	6.80	.	40.000	106.000	5.100	3.300	0.110	6.890	.
7	2	1982	78.00	8.00	.	47.000	111.000	5.300	2.560	0.100	7.440	.
7	26	1982	84.00	8.00	.	42.000	163.000	7.400	3.150	1.280	9.100	.
8	30	1982	76.00	8.00	.	58.000	125.000	5.900	2.790	0.300	8.030	.
10	4	1982	70.00	8.00	.	53.000	168.000	7.500	3.300	0.210	9.170	.
11	1	1982	62.00	6.00	.	41.000	141.000	6.300	2.800	0.250	7.570	.
11	22	1982	136.00	.	.	41.600	213.000	8.900	3.820	.	.	.
3	18	1983	.	.	.	24.000	37.000	18.000	1.380	.	.	.
4	14	1983	.	.	.	22.000	40.000	2.450	1.400	.	.	.

Table A2.5.14. Drainage quality data for metals from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
5	15	1983	.	.	.	56.000	1.100	6.400	3.490	.	.	.
6	15	1983	.	.	.	40.000	76.000	3.400	1.880	.	.	.
7	14	1983	.	.	.	87.000	174.000	10.190	5.740	.	.	.
8	14	1983	.	.	.	68.000	120.000	6.450	3.720	.	.	.
9	15	1983	.	.	.	85.000	164.000	9.000	5.060	.	.	.
10	13	1983	.	.	.	77.000	136.000	8.000	4.340	.	.	.
11	9	1983	.	.	.	82.000	133.000	7.400	4.170	.	.	.
11	30	1983	.	.	.	73.400	125.000	6.950	3.970	.	.	.
2	24	1984	56.60	.	.	49.000	106.000	5.110	3.420	.	.	600
3	15	1984	17.30	.	.	26.000	41.400	2.050	1.250	.	.	601
4	15	1984	27.20	.	.	46.000	67.000	3.750	2.200	.	.	602
5	16	1984	42.00	.	.	65.000	110.000	7.000	3.360	.	.	603
6	13	1984	52.10	.	.	68.000	119.600	6.400	3.540	.	.	604
7	14	1984	.	.	.	95.600	186.000	6.600	4.000	.	.	605
8	15	1984	71.80	.	.	90.100	160.420	11.600	5.080	.	.	606
9	12	1984	88.20	.	.	107.500	225.000	15.700	6.630	.	.	607
10	13	1984	65.30	.	.	109.500	169.000	12.700	5.240	.	.	608
11	14	1984	93.80	.	.	135.000	255.000	12.400	6.800	.	.	609
12	1	1984	61.00	.	.	90.800	152.000	7.900	4.160	.	.	610
12	18	1984	52.00	.	.	70.300	108.800	5.410	3.060	.	.	611
4	2	1985	21.00	.	.	32.000	63.000	2.840	1.820	.	.	600
4	16	1985	37.60	.	.	63.000	98.000	4.820	2.870	.	.	601
5	14	1985	48.50	.	.	83.000	128.000	7.200	3.800	.	.	602
6	14	1985	63.00	.	.	91.000	160.000	7.800	4.460	.	.	603
7	15	1985	55.00	.	.	78.300	130.600	6.500	4.690	.	.	604
8	13	1985	46.00	.	.	76.400	94.000	4.880	2.870	.	.	605
9	13	1985	48.00	.	.	75.000	104.200	5.440	3.330	.	.	606
10	15	1985	43.10	.	.	77.100	106.500	5.470	2.510	.	.	607
11	15	1985	56.00	.	.	88.800	144.600	7.340	3.800	.	.	608
3	23	1986	15.80	.	.	34.000	55.000	2.910	1.580	.	.	600
4	16	1986	26.20	.	.	59.000	86.000	4.840	2.280	.	.	601
5	14	1986	28.00	.	.	72.000	133.000	8.000	2.920	.	.	602
6	11	1986	38.00	.	.	63.000	100.000	4.800	3.100	.	.	603
7	12	1986	36.00	.	.	65.000	100.000	4.420	2.850	.	.	604
8	13	1986	45.00	.	.	80.000	99.000	6.200	2.550	.	.	605
9	14	1986	38.00	.	.	76.000	94.000	6.600	2.400	.	.	606
10	16	1986	44.00	.	.	93.000	122.000	6.000	3.510	.	.	607
11	16	1986	64.00	.	.	156.000	169.000	8.800	5.300	.	.	608
3	28	1987	30.00	.	.	83.000	110.000	5.390	3.140	.	.	600
4	18	1987	26.00	.	.	68.000	95.000	4.520	2.580	.	.	601
5	16	1987	44.00	.	.	97.000	115.000	5.600	3.400	.	.	602
6	16	1987	58.00	.	.	106.000	163.000	5.800	3.420	.	.	603
7	15	1987	46.00	.	.	92.000	117.000	5.200	3.430	.	.	604
8	15	1987	42.00	.	.	93.000	112.000	5.000	3.130	.	.	605
9	16	1987	47.00	.	.	101.000	113.000	5.090	3.050	.	.	606
10	13	1987	72.00	.	.	138.000	173.000	7.560	4.550	.	.	607
11	10	1987	64.00	.	.	128.000	165.000	7.330	4.510	.	.	608
3	28	1988	14.00	.	.	34.000	52.000	2.500	2.100	.	.	600
4	17	1988	18.00	.	.	52.000	81.000	3.700	3.100	.	.	601
5	12	1988	50.00	.	.	113.000	153.000	6.800	6.500	.	.	602
6	12	1988	40.00	.	.	118.000	146.000	7.500	7.100	.	.	603
7	15	1988	36.00	.	.	105.000	132.000	6.500	6.500	.	.	604
8	16	1988	52.00	.	.	85.000	133.000	6.300	5.100	.	.	605
9	16	1988	44.00	.	.	76.000	80.000	5.500	4.400	.	.	606
10	18	1988	30.00	.	.	75.000	88.000	5.100	4.700	.	.	607
11	15	1988	26.00	.	.	71.000	81.500	4.550	4.400	.	.	608
11	30	1988	32.00	.	.	78.000	98.000	5.100	4.800	.	.	609
4	7	1989	10.00	.	.	18.600	23.000	1.400	1.100	.	.	600
4	24	1989	18.00	.	.	45.000	57.000	3.400	3.000	.	.	601
5	16	1989	28.00	.	.	66.000	79.000	4.700	4.300	.	.	602
6	17	1989	54.00	.	.	91.000	159.000	10.200	6.300	.	.	603
7	17	1989	58.00	.	.	131.000	204.000	11.300	9.700	.	.	604
8	14	1989	34.00	.	.	82.000	138.000	5.700	5.500	.	.	605
9	14	1989	50.00	.	.	123.000	176.000	9.900	8.200	.	.	606
10	16	1989	56.00	.	.	137.000	149.000	16.400	9.400	.	.	607
11	10	1989	36.00	.	.	94.000	96.000	15.300	7.000	.	.	608

Table A2.5.14. Drainage quality data for metals from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
3	26	1990	93.00	.	.	202.000	253.000	16.200	14.100	.	.	600
4	12	1990	26.00	.	.	96.000	108.000	5.610	5.440	.	.	601
5	17	1990	22.00	.	.	84.000	78.000	4.630	4.260	.	.	602
6	18	1990	28.00	.	.	83.000	74.000	4.200	4.380	.	.	603
7	16	1990	26.00	.	.	93.000	73.000	4.410	4.290	.	.	604
8	13	1990	28.00	.	.	75.000	78.000	4.900	4.130	.	.	605
9	10	1990	24.00	.	.	63.000	71.000	4.120	3.150	.	.	606
10	15	1990	36.00	.	.	88.000	112.000	6.410	4.330	.	.	607
11	16	1990	58.00	.	.	144.000	165.000	9.990	9.180	.	.	608
4	8	1991	.	.	.	.	.	.	.	.	.	.
4	22	1991	30.00	.	.	106.000	116.000	5.200	3.300	.	.	2022
5	20	1991	40.00	.	.	133.000	163.000	6.900	4.400	.	.	2026
6	17	1991	28.00	.	.	91.000	103.000	5.000	2.800	.	.	2042
7	15	1991	34.00	.	.	107.000	117.000	5.900	4.600	.	.	2058
8	12	1991	56.00	.	.	137.000	150.000	7.600	6.300	.	.	2074
9	9	1991	48.00	.	.	119.000	130.000	6.600	4.900	.	.	2089
10	7	1991	42.00	.	.	105.000	114.000	5.600	4.600	.	.	2105
11	7	1991	34.00	.	.	96.000	98.000	5.000	4.000	.	.	2121
4	23	1992	11.60	3.20	4.00	28.000	28.000	1.400	0.900	0.060	2.530	17003
5	11	1992	28.40	5.30	10.00	74.000	71.000	3.600	2.400	0.110	6.770	17016
5	26	1992	25.80	5.90	8.00	58.000	62.000	3.000	2.200	0.100	6.350	17044
6	8	1992	58.00	10.30	16.00	128.000	178.000	7.800	5.700	0.240	15.700	17048
6	22	1992	28.00	6.80	7.00	61.000	66.000	3.700	2.300	0.120	7.020	17052
7	7	1992	25.60	6.50	7.00	65.000	58.000	3.200	2.100	0.110	5.760	17057
7	20	1992	47.00	10.10	11.00	107.000	130.000	6.700	4.300	0.140	12.300	17061
8	3	1992	31.00	7.40	8.00	68.000	69.000	3.900	2.000	0.110	7.460	17077
8	17	1992	33.60	8.70	9.00	84.000	84.000	4.700	3.100	0.180	8.690	17081
8	31	1992	25.00	7.30	6.00	58.000	58.000	3.000	2.000	0.110	5.620	17097
9	14	1992	34.00	8.90	8.00	86.000	78.000	4.500	3.000	0.110	8.130	17101
9	28	1992	46.60	10.80	10.00	107.000	110.000	6.000	4.000	0.110	11.300	17118
10	12	1992	51.60	11.70	11.00	127.000	137.000	6.700	4.500	0.140	12.700	17122
10	27	1992	41.00	10.10	9.00	96.000	95.000	5.300	3.300	0.100	9.980	17138
11	9	1992	.	.	.	.	.	.	.	.	.	.
11	30	1992	24.40	7.20	6.00	60.000	57.000	3.100	2.000	0.070	6.120	17153
4	8	1993	10.00	2.80	6.00	33.000	34.000	1.480	1.080	0.070	3.080	17165
4	22	1993	10.00	3.00	5.00	27.000	30.000	1.660	0.880	0.050	2.680	17176
5	6	1993	21.30	5.00	13.00	71.000	71.000	3.850	1.990	0.140	5.800	17181
5	20	1993	26.30	4.80	14.00	96.000	90.000	4.600	2.860	0.210	7.280	17196
6	3	1993	16.00	3.50	7.00	54.000	51.000	2.610	1.520	0.100	4.010	17200
6	17	1993	22.00	4.80	9.00	87.000	79.000	4.150	2.520	0.110	6.320	17216
7	1	1993	39.00	7.80	18.00	139.000	147.000	7.400	4.520	0.330	13.600	17220
7	14	1993	37.20	9.20	14.00	100.000	125.000	6.030	3.750	0.150	13.000	17237
7	29	1993	15.60	5.40	7.00	49.000	44.000	2.200	1.400	0.100	3.820	17241
8	12	1993	19.80	6.00	8.00	66.000	62.000	2.680	1.990	0.070	5.300	17257
8	26	1993	23.00	6.80	8.00	66.000	70.000	3.140	2.280	0.100	6.660	17262
9	9	1993	22.20	7.00	8.00	66.000	68.000	3.450	2.680	0.070	6.820	17277
9	23	1993	23.40	6.80	9.00	64.000	74.000	3.420	2.370	0.100	6.970	17281
10	7	1993	27.00	7.40	12.00	80.000	86.000	4.000	2.500	0.090	7.420	17297
10	20	1993	33.80	8.40	15.00	105.000	113.000	5.060	3.340	0.090	9.730	17301
11	4	1993	26.80	7.00	10.00	80.000	77.000	4.090	2.470	0.070	7.090	17319
11	22	1993	.	.	.	.	.	.	.	.	.	.

Table A2.5.15. Drainage quality data for additional parameters from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
4	26	1978	.	.	.	.	.	4.0	.
5	25	1978	.	.	.	.	.	.	.
6	16	1978	.	.	.	.	.	15.0	.
7	5	1978	4.2E2	.	.	.	.	12.0	.
7	13	1978	6.1E2	4.0E2	1.0E2	0.08	.	.	.
7	19	1978	3.1E2	1.6E2	1.0E2	1.90	.	18.0	.
8	2	1978	1.8E2	48.00	70.00	0.10	.	23.0	.
8	4	1978	3.2E2	2.1E2	1.0E2	0.10	.	22.0	.
8	10	1978	6.0E2	.	1.6E2	0.08	.	24.0	.
8	16	1978	1.5E2	1.4E2	.	0.08	.	22.0	.
8	17	1978	1.9E2	1.2E2	.	0.08	.	.	.
8	23	1978	2.8E2	1.7E2	.	0.08	.	.	.
8	25	1978	2.2E2	1.4E2	98.00	0.08	.	23.0	.
8	28	1978	3.9E2	2.3E2	.	0.08	.	.	.
9	13	1978	2.7E2	.	.	0.10	.	13.0	.
10	17	1978	.	.	.	.	.	.	.
3	23	1979	10.00	.	.	.	5.70	.	.
4	12	1979	95.00	.	.	.	.	.	.
4	16	1979	45.00	.	.	.	.	.	.
4	17	1979	28.00	.	.	.	4.70	.	.
4	18	1979	30.00	62.00	.	.	.	.	.
4	19	1979	29.00	50.00	11.00	.	.	.	.
4	20	1979	51.00	67.50	13.50	.	4.20	.	.
4	25	1979	1.1E2	1.2E2	24.00	.	.	.	.
5	2	1979	1.7E2	1.9E2	23.00	.	.	.	.
5	9	1979	15.00	.	.	.	.	.	.
5	10	1979	1.1E2	.	.	.	.	.	.
5	20	1979	1.5E2	.	.	.	8.60	.	.
5	22	1979	94.00	.	.	.	16.00	.	.
5	23	1979	1.4E2	.	.	.	7.90	.	.
6	1	1979	1.1E2	.	.	.	7.10	.	.
6	6	1979	94.00	.	.	.	.	.	.
6	10	1979	79.00	.	.	.	.	20.0	.
6	12	1979	1.0E2	.	.	.	.	.	.
6	16	1979	1.4E2	.	.	.	.	23.0	.
6	21	1979	91.00	.	.	.	.	13.0	.
6	30	1979	1.3E2	.	.	.	8.90	.	.
7	10	1979	2.2E2	.	.	.	9.00	19.0	.
7	30	1979	79.00	.	.	.	6.30	22.0	.
8	4	1979	1.2E2	.	.	.	8.70	23.0	.
8	13	1979	1.8E2	.	.	.	9.10	14.0	.
8	23	1979	1.5E2	.	.	.	8.30	9.0	.
8	28	1979	1.1E2	.	.	.	6.60	20.0	.
9	1	1979	86.00	.	.	.	6.30	22.0	.
9	5	1979	1.0E2	.	.	.	7.80	18.0	.
9	11	1979	1.1E2	.	.	.	7.30	17.0	.
9	13	1979	29.00	.	.	.	9.80	19.0	.
9	19	1979	2.0E2	.	.	.	9.00	13.5	.
10	18	1979	1.7E2	.	.	.	8.50	.	.
10	19	1979	64.00	.	.	.	4.30	.	.
10	31	1979	99.00	.	.	.	5.80	.	.
11	1	1979	1.0E2	.	.	.	6.70	.	.
3	29	1980	18.00	.	.	.	2.10	13.0	.
4	1	1980	12.00	.	.	.	1.40	.	.
4	6	1980	19.50	.	.	.	.	.	.
4	14	1980	44.00	.	.	.	.	.	.
4	18	1980	61.00	.	.	.	3.50	.	.
4	28	1980	68.00	.	.	.	.	11.0	.
5	10	1980	75.00	.	.	.	5.40	17.0	.
5	31	1980	84.00	.	.	.	1.30	.	.
6	3	1980	1.1E2	.	.	.	6.70	.	.
6	6	1980	90.00	.	.	.	.	.	.
6	13	1980	86.00	.	.	.	.	.	.
6	18	1980	1.1E2	.	.	.	.	.	.
6	28	1980	83.00	.	.	.	5.90	.	.
7	2	1980	.	.	.	.	.	.	.

Table A2.5.15. Drainage quality data for additional parameters from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
7	7	1980	96.00	.	.	.	10.00	.	.
7	17	1980	26.00	.	.	.	.	.	.
7	26	1980	1.3E2	.	.	.	.	.	.
8	3	1980	1.3E2	.	.	.	6.80	.	.
8	12	1980	81.00	.	.	.	4.90	.	.
8	19	1980	93.00	.	.	.	.	.	.
8	22	1980	82.00	.	.	.	.	.	.
8	29	1980	1.1E2	.	.	.	8.50	.	.
9	11	1980	78.00	.	.	.	.	13.0	.
9	12	1980	53.50	.	.	.	4.90	21.0	.
9	18	1980	69.00	.	.	.	.	15.0	.
9	23	1980	1.1E2	.	.	.	5.90	14.0	.
10	22	1980	85.00	.	.	.	4.80	11.0	.
10	30	1980	64.00	.	.	.	.	10.0	.
2	19	1981	36.00	.	.	.	.	5.0	.
2	21	1981	.	.	.	.	.	24.0	.
3	28	1981	.	.	.	.	2.30	.	.
4	3	1981	.	.	.	.	.	.	.
4	13	1981	.	.	.	.	.	.	.
4	17	1981	.	.	.	.	4.40	.	.
4	23	1981	.	.	.	.	.	.	.
4	25	1981	.	.	.	.	.	.	.
4	28	1981	.	.	.	.	3.20	.	.
4	29	1981	.	.	.	.	.	12.0	.
5	12	1981	.	.	.	.	5.30	19.0	.
6	1	1981	.	.	.	.	.	15.0	.
6	6	1981	.	.	.	.	3.10	18.0	.
6	22	1981	.	.	.	.	.	17.0	.
6	24	1981	.	.	.	.	.	16.0	.
6	28	1981	.	.	.	.	3.40	13.0	.
7	4	1981	.	.	.	.	.	22.0	.
7	15	1981	.	.	.	.	.	19.0	.
7	24	1981	49.00	.	.	.	3.50	19.0	.
8	4	1981	.	.	.	.	.	18.0	.
8	17	1981	.	.	.	.	.	22.0	.
8	31	1981	.	.	.	.	5.10	.	.
9	7	1981	.	.	.	.	.	.	.
9	16	1981	.	.	.	.	.	.	.
9	26	1981	.	.	.	.	3.70	.	.
10	1	1981	.	.	.	.	.	.	.
10	4	1981	.	.	.	.	.	.	.
10	14	1981	.	.	.	.	3.50	.	.
10	15	1981	.	.	.	.	.	.	.
11	1	1981	.	.	.	.	4.10	.	.
3	30	1982	.	.	.	.	.	.	.
4	11	1982	.	.	.	.	.	.	.
4	13	1982	.	.	.	.	.	.	.
4	15	1982	.	.	.	.	.	.	.
4	19	1982	.	.	.	.	.	.	.
4	23	1982	.	.	.	.	.	.	.
5	3	1982	.	.	.	.	.	.	.
5	4	1982	.	.	.	.	.	.	.
5	10	1982	.	.	.	.	.	.	.
5	11	1982	.	.	.	.	.	.	.
5	14	1982	.	.	.	.	.	.	.
5	21	1982	.	.	.	.	.	.	.
6	2	1982	.	.	.	.	.	.	.
6	18	1982	.	.	.	.	.	.	.
7	2	1982	.	.	.	.	.	.	.
7	26	1982	.	.	.	.	.	.	.
8	30	1982	.	.	.	.	.	.	.
10	4	1982	.	.	.	.	.	.	.
11	1	1982	.	.	.	.	.	.	.
11	22	1982	.	.	.	.	.	.	.
3	18	1983	.	.	.	.	.	.	.
4	14	1983	.	.	.	.	.	.	.

Table A2.5.15. Drainage quality data for additional parameters from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
5	15	1983	.	.	.	.	.	.	.
6	15	1983	.	.	.	.	.	.	.
7	14	1983	.	.	.	.	.	.	.
8	14	1983	.	.	.	.	.	.	.
9	15	1983	.	.	.	.	.	.	.
10	13	1983	.	.	.	.	.	.	.
11	9	1983	.	.	.	.	.	.	.
11	30	1983	.	.	.	.	.	.	.
2	24	1984	.	.	.	.	.	.	600
3	15	1984	.	.	.	.	.	.	601
4	15	1984	.	.	.	.	.	.	602
5	16	1984	.	.	.	.	.	.	603
6	13	1984	.	.	.	.	.	.	604
7	14	1984	.	.	.	.	.	.	605
8	15	1984	.	.	.	.	.	.	606
9	12	1984	.	.	.	.	.	.	607
10	13	1984	.	.	.	.	.	.	608
11	14	1984	.	.	.	.	.	.	609
12	1	1984	.	.	.	.	.	.	610
12	18	1984	.	.	.	.	.	.	611
4	2	1985	.	.	.	.	.	.	600
4	16	1985	.	.	.	.	.	.	601
5	14	1985	.	.	.	.	.	.	602
6	14	1985	.	.	.	.	.	.	603
7	15	1985	.	.	.	.	.	.	604
8	13	1985	.	.	.	.	.	.	605
9	13	1985	.	.	.	.	.	.	606
10	15	1985	.	.	.	.	.	.	607
11	15	1985	.	.	.	.	.	.	608
3	23	1986	.	.	.	.	.	.	600
4	16	1986	.	.	.	.	.	.	601
5	14	1986	.	.	.	.	.	.	602
6	11	1986	.	.	.	.	.	.	603
7	12	1986	.	.	.	.	.	.	604
8	13	1986	.	.	.	.	.	.	605
9	14	1986	.	.	.	.	.	.	606
10	16	1986	.	.	.	.	.	.	607
11	16	1986	.	.	.	.	.	.	608
3	28	1987	.	.	.	.	.	.	600
4	18	1987	.	.	.	.	.	.	601
5	16	1987	.	.	.	.	.	.	602
6	16	1987	.	.	.	.	.	.	603
7	15	1987	.	.	.	.	.	.	604
8	15	1987	.	.	.	.	.	.	605
9	16	1987	.	.	.	.	.	.	606
10	13	1987	.	.	.	.	.	.	607
11	10	1987	.	.	.	.	.	.	608
3	28	1988	.	.	.	.	.	.	600
4	17	1988	.	.	.	.	.	.	601
5	12	1988	.	.	.	.	.	.	602
6	12	1988	.	.	.	.	.	.	603
7	15	1988	.	.	.	.	.	.	604
8	16	1988	.	.	.	.	.	.	605
9	16	1988	.	.	.	.	.	.	606
10	18	1988	.	.	.	.	.	.	607
11	15	1988	.	.	.	.	.	.	608
11	30	1988	.	.	.	.	.	.	609
4	7	1989	.	.	.	.	.	.	600
4	24	1989	.	.	.	.	.	.	601
5	16	1989	.	.	.	.	.	.	602
6	17	1989	.	.	.	.	.	.	603
7	17	1989	.	.	.	.	.	.	604
8	14	1989	.	.	.	.	.	.	605
9	14	1989	.	.	.	.	.	.	606
10	16	1989	.	.	.	.	.	.	607
11	10	1989	.	.	.	.	.	.	608

Table A2.5.15. Drainage quality data for additional parameters from Duluth Complex test pile FL6, from 1978 to 1993 (0.79% S, 0.34% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
3	26	1990	.	.	.	.	.	.	600
4	12	1990	.	.	.	.	.	.	601
5	17	1990	.	.	.	.	.	.	602
6	18	1990	.	.	.	.	.	.	603
7	16	1990	.	.	.	.	.	.	604
8	13	1990	.	.	.	.	.	.	605
9	10	1990	.	.	.	.	.	.	606
10	15	1990	.	.	.	.	.	.	607
11	16	1990	.	.	.	.	.	.	608
4	8	1991	.	.	.	.	.	.	.
4	22	1991	.	.	.	.	.	.	2022
5	20	1991	.	.	.	.	.	.	2026
6	17	1991	.	.	.	.	.	.	2042
7	15	1991	.	.	.	.	.	.	2058
8	12	1991	.	.	.	.	.	.	2074
9	9	1991	.	.	.	.	.	.	2089
10	7	1991	.	.	.	.	.	.	2105
11	7	1991	.	.	.	.	.	.	2121
4	23	1992	.	.	.	.	.	.	17003
5	11	1992	.	.	.	.	.	.	17016
5	26	1992	.	.	.	.	.	.	17044
6	8	1992	.	.	.	.	.	.	17048
6	22	1992	.	.	.	.	.	.	17052
7	7	1992	.	.	.	.	.	.	17057
7	20	1992	.	.	.	.	.	.	17061
8	3	1992	.	.	.	.	.	.	17077
8	17	1992	.	.	.	.	.	.	17081
8	31	1992	.	.	.	.	.	.	17097
9	14	1992	.	.	.	.	.	.	17101
9	28	1992	.	.	.	.	.	.	17118
10	12	1992	.	.	.	.	.	.	17122
10	27	1992	.	.	.	.	.	.	17138
11	9	1992	.	.	.	.	.	.	.
11	30	1992	.	.	.	.	.	.	17153
4	8	1993	.	.	.	.	.	.	17165
4	22	1993	.	.	.	.	.	.	17176
5	6	1993	.	.	.	.	.	.	17181
5	20	1993	.	.	.	.	.	.	17196
6	3	1993	.	.	.	.	.	.	17200
6	17	1993	.	.	.	.	.	.	17216
7	1	1993	.	.	.	.	.	.	17220
7	14	1993	.	.	.	.	.	.	17237
7	29	1993	.	.	.	.	.	.	17241
8	12	1993	.	.	.	.	.	.	17257
8	26	1993	.	.	.	.	.	.	17262
9	9	1993	.	.	.	.	.	.	17277
9	23	1993	.	.	.	.	.	.	17281
10	7	1993	.	.	.	.	.	.	17297
10	20	1993	.	.	.	.	.	.	17301
11	4	1993	.	.	.	.	.	.	17319
11	22	1993	.	.	.	.	.	.	.

Table A2.5.16. Drainage quality data from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 1 of 5.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
4	26	1978	.	.	7.20	1750.00	.	.	.	.	190.00	11.00	.
5	25	1978	.	.	7.60	2500.00	.	.	.	.	279.00	21.00	.
6	16	1978	192.0	.	7.50	3300.00	.	.	.	.	366.00	26.00	.
7	5	1978	370.0	.	7.05	2950.00	.	.	.	1800.00	325.00	29.00	.
7	13	1978	.	.	7.22	.	.	.	.	1300.00	357.00	27.00	.
7	19	1978	.	.	7.40	2050.00	.	.	.	1300.00	580.00	29.00	.
8	2	1978	.	.	7.00	1725.00	.	.	.	775.00	276.00	20.00	.
8	4	1978	333.0	.	6.30	2350.00	.	.	.	1400.00	483.00	32.00	.
8	10	1978	147.0	.	5.50	4000.00	7.0	.	.	2000.00	636.00	47.00	.
8	16	1978	5948.0	.	5.90	1491.67	8.0	.	.	850.00	245.30	26.00	.
8	17	1978	1714.0	.	5.25	2100.00	5.0	.	.	800.00	400.00	40.00	.
8	18	1978	498.0	.	4.85	3100.00	4.0	.	.	800.00	350.00	44.00	.
8	23	1978	.	.	4.23	1775.00	.	.	.	1050.00	203.50	24.50	.
8	25	1978	.	.	3.90	2800.00	.	.	.	1200.00	344.00	42.00	.
8	28	1978	.	.	4.25	3250.00	.	.	.	1500.00	395.00	50.00	.
9	11	1978	8801.0	.	4.80	1475.00	.	.	.	700.00	140.00	20.00	.
9	12	1978	.	.	3.50	2650.00	.	.	.	1550.00	260.00	50.00	.
9	13	1978	721.0	.	3.62	3250.00	.	.	.	1900.00	312.00	65.00	.
10	17	1978	.	.	4.10	3300.00	.	.	.	.	335.00	98.00	.
3	23	1979	2801.0	283.0	3.70	2445.00	.	.	.	2100.00	271.60	108.00	.
4	3	1979	2573.8	167.0	3.60	4435.00	.	.	.	3000.00	395.00	231.00	.
4	11	1979	1983.3	129.0	3.60	4766.00	.	.	.	4600.00	430.00	280.00	.
4	16	1979	4148.4	3192.0	3.50	2822.00	.	.	.	2350.00	269.00	124.00	.
4	17	1979	5098.4	3235.0	3.30	1769.00	.	.	.	1050.00	197.00	70.60	.
4	18	1979	6905.8	4242.0	3.40	1769.00	.	.	.	1100.00	213.00	62.40	.
4	19	1979	1843.3	2946.0	3.40	1475.00	.	.	.	833.00	193.00	42.20	.
4	20	1979	7210.4	2394.0	3.50	1380.00	.	.	.	867.00	178.00	49.00	.
4	30	1979	6430.7	364.0	3.50	2562.00	.	.	.	2100.00	294.80	100.00	.
5	11	1979	408.8	2115.0	3.50	2341.00	.	.	.	.	254.00	89.00	.
5	22	1979	1813.0	1031.0	3.80	1767.00	.	.	.	1150.00	201.00	64.00	.
5	23	1979	3156.7	982.0	3.20	2713.00	.	.	.	2100.00	288.20	105.80	.
5	27	1979	3073.4	508.0	3.38	2578.00	.	.	.	2200.00	285.40	57.70	.
6	1	1979	2289.9	561.0	3.60	2227.00	.	.	.	1800.00	239.20	54.60	.
6	6	1979	2358.0	788.0	3.90	2154.00	.	.	.	1430.00	243.80	77.20	.
6	10	1979	3406.5	1308.0	3.40	2538.00	.	.	.	1750.00	287.60	93.00	.
6	16	1979	934.9	244.0	3.60	2890.00	.	.	.	2100.00	324.00	110.00	.
6	20	1979	1798.0	1300.0	3.50	1891.00	.	.	.	1150.00	203.00	68.00	.
6	30	1979	1903.9	296.0	3.30	2753.50	.	.	.	2225.00	328.00	117.00	.
7	10	1979	1347.5	110.0	3.30	3830.00	.	.	.	3200.00	462.00	180.00	.
7	30	1979	3179.4	3202.0	3.50	1926.00	.	.	.	1400.00	229.00	82.00	.
8	4	1979	1442.1	322.0	3.20	3170.00	.	.	.	2800.00	416.00	144.00	.
8	13	1979	711.6	87.0	3.40	4636.00	.	.	.	4650.00	514.80	130.40	.
8	23	1979	855.4	243.0	3.50	3708.00	.	.	.	3600.00	463.20	230.20	.
8	28	1979	423.9	330.0	3.40	3332.00	.	.	.	3050.00	365.00	122.20	.
9	1	1979	5707.8	2338.0	3.20	2923.00	.	.	.	1400.00	296.40	115.80	.
9	5	1979	11.4	523.0	3.20	4197.00	.	.	.	3300.00	420.20	128.60	.
9	13	1979	2615.4	811.0	3.30	3906.00	.	.	.	3580.00	372.80	234.00	.
9	13	1979	2448.9	811.0	3.30	4120.00	.	.	.	3600.00	363.80	239.60	.
9	19	1979	1726.0	220.0	3.30	5852.00	.	.	.	5700.00	522.00	384.00	.
10	18	1979	840.3	360.0	3.40	5446.00	.	.	.	6330.00	518.00	398.00	.
10	19	1979	1729.7	1591.0	3.60	2356.00	.	.	.	1700.00	185.60	138.00	.
10	31	1979	4046.2	1743.0	3.30	3937.00	.	.	.	3310.00	320.00	260.00	.
11	1	1979	4345.2	1671.0	3.40	3567.00	.	.	.	3180.00	294.00	244.00	.
3	29	1980	1400.6	994.0	3.70	3500.00	.	.	.	2100.00	222.70	190.90	.
3	31	1980	3369.0	1070.0	3.60	2450.00	.	.	.	1700.00	168.20	125.60	.
4	5	1980	4542.5	1984.0	4.60	2288.00	.	.	.	1055.00	169.40	112.60	.
4	14	1980	2271.2	497.0	3.50	.	.	.	.	1220.00	190.40	135.60	.
4	18	1980	1589.9	497.0	3.50	.	.	.	.	2550.00	231.80	184.00	.
4	27	1980	530.0	50.0	3.50	4825.00	.	.	.	3880.00	342.40	314.00	.
5	10	1980	450.5	109.0	3.40	6110.00	.	.	.	6700.00	376.80	442.00	.
5	18	1980	639.7	21.0	3.23	6000.00	.	.	.	6125.00	386.40	485.00	.
5	31	1980	791.2	134.0	3.40	3450.00	.	.	.	3040.00	244.40	233.00	.
6	3	1980	302.8	134.0	3.45	4750.00	.	.	.	3960.00	292.20	274.00	.
6	7	1980	382.3	128.0	3.40	4550.00	.	.	.	3920.00	279.60	259.00	.
6	13	1980	299.0	140.0	3.30	5000.00	.	.	.	4070.00	332.00	283.00	.

Table A2.5.16. Drainage quality data from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 2 of 5.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
6	19	1980	155.2	18.0	3.30	5000.00	.	.	.	6200.00	341.00	345.00	.
6	28	1980	507.2	158.0	3.30	4674.00	.	.	.	4250.00	341.00	355.00	.
6	30	1980	329.3	64.0	3.70	5000.00	.	.	.	4600.00	358.00	363.00	.
7	2	1980	408.8	257.0	3.50	2600.00	.	.	.	1850.00	200.00	148.00	.
7	13	1980	283.9	34.0	3.20	6360.00	.	.	.	5750.00	417.00	444.00	.
7	23	1980	295.3	20.0	3.20	5000.00	.	.	.	5700.00	435.00	506.00	.
8	3	1980	34.4	111.0	3.20	5000.00	.	.	.	.	483.00	587.00	.
8	12	1980	492.1	316.0	3.35	4750.00	.	.	.	4180.00	348.00	347.00	.
8	17	1980	261.2	26.0	3.30	5000.00	.	.	.	5560.00	393.00	446.00	.
8	22	1980	230.9	112.0	.	4900.00	.	.	.	4180.00	279.00	512.00	.
8	29	1980	1319.3	700.0	3.50	2540.00	.	.	.	2400.00	172.00	240.00	.
9	8	1980	522.4	134.0	3.40	4078.00	.	.	.	4100.00	231.00	503.00	.
9	9	1980	102.2	600.0	3.30	4350.00	.	.	.	4700.00	283.00	569.00	.
9	12	1980	916.1	596.0	3.40	2694.00	.	.	.	2580.00	103.00	296.00	.
9	18	1980	3690.8	2740.0	3.40	3310.00	.	.	.	3050.00	158.00	373.00	.
9	25	1980	1109.1	288.0	3.30	5588.00	.	.	.	5200.00	380.00	608.00	.
10	23	1980	757.1	593.0	3.30	5742.00	.	.	.	7700.00	390.00	660.00	.
10	31	1980	647.3	91.0	3.40	5148.00	.	.	.	4700.00	309.00	491.00	.
2	19	1981	908.0	1419.0	3.60	3692.00	.	.	.	3450.00	198.90	279.00	.
2	21	1981	1344.0	549.0	3.60	2500.00	.	.	.	2040.00	130.00	144.20	.
3	29	1981	3702.0	773.0	3.50	2000.00	.	.	.	1420.00	94.90	110.50	.
4	3	1981	4364.0	927.0	3.90	1825.00	.	.	.	1250.00	85.80	91.00	.
4	13	1981	1033.0	405.0	3.70	4000.00	.	.	.	2800.00	155.00	214.00	.
4	17	1981	806.0	413.0	3.80	4500.00	.	.	.	3080.00	194.00	293.00	.
4	23	1981	1453.0	1594.0	3.70	4350.00	.	.	.	3850.00	192.00	338.00	.
4	25	1981	6057.0	1715.0	3.60	4575.00	.	.	.	3000.00	170.00	279.00	.
4	28	1981	4168.0	2146.0	3.60	4200.00	.	.	.	3000.00	174.00	270.00	.
4	29	1981	3736.0	1098.0	3.50	5004.00	.	.	.	4300.00	296.00	440.00	.
5	12	1981	2165.0	119.0	3.35	6386.00	.	.	.	5250.00	374.00	590.00	.
5	28	1981	893.0	2146.0	3.35	5952.00	.	.	.	7500.00	344.00	580.00	.
6	6	1981	621.0	51.0	3.25	5709.00	.	.	.	6000.00	312.00	540.00	.
6	22	1981	1143.0	607.0	3.40	4570.00	.	.	.	4300.00	246.00	460.00	.
6	24	1981	1189.0	659.0	3.40	3150.00	.	.	.	2370.00	174.00	200.00	.
6	28	1981	2036.0	1352.0	3.35	3032.00	.	.	.	2550.00	174.00	210.00	.
7	4	1981	4891.0	173.0	3.30	4298.00	.	.	.	4050.00	272.00	366.00	.
7	24	1981	386.0	70.0	3.30	7122.00	.	.	.	8000.00	392.00	524.60	.
8	4	1981	291.0	22.0	3.10	8085.00	.	.	.	9350.00	426.00	830.00	.
8	17	1981	182.0	4.0	3.10	8037.00	.	.	.	8400.00	414.00	870.00	.
8	31	1981	106.0	36.0	3.30	8900.00	.	.	.	9000.00	440.00	980.00	.
9	7	1981	121.0	58.0	3.30	7500.00	.	.	.	7900.00	440.00	820.00	.
9	16	1981	72.0	4.0	3.30	5000.00	.	.	.	9600.00	480.00	1030.00	.
9	26	1981	329.0	210.0	3.50	4965.00	.	.	.	5000.00	330.00	520.00	.
10	1	1981	693.0	983.0	3.50	4375.00	.	.	.	2600.00	206.00	290.00	.
10	4	1981	1136.0	706.0	3.45	3900.00	.	.	.	2600.00	154.00	238.00	.
10	14	1981	4554.0	1622.0	3.40	3950.00	.	.	.	2960.00	176.00	236.00	.
10	18	1981	6337.0	1288.0	3.30	5000.00	.	.	.	4000.00	326.00	262.00	.
11	6	1981	3585.0	105.0	3.20	5000.00	.	.	.	5400.00	292.00	530.00	.
3	30	1982	3218.0	.	3.45	4200.00	.	.	.	3100.00	174.00	350.00	.
4	6	1982	2657.0	.	3.50	4200.00	.	.	.	3200.00	169.00	355.00	.
4	13	1982	5443.0	.	3.50	2600.00	.	.	.	1900.00	131.00	140.00	.
4	15	1982	5182.0	.	3.55	1950.00	.	.	.	1015.00	111.10	94.00	.
4	19	1982	2014.0	.	3.50	2250.00	.	.	.	1500.00	142.00	152.00	.
4	23	1982	1344.0	.	3.45	3050.00	.	.	.	2300.00	188.00	192.00	.
5	3	1982	114.0	.	3.40	4010.00	.	.	.	5500.00	330.00	400.00	.
5	4	1982	867.0	.	3.50	3450.00	.	.	.	2500.00	156.00	216.00	.
5	7	1982	1889.0	.	3.40	5000.00	.	.	.	4000.00	212.00	300.00	.
5	10	1982	3751.0	.	3.50	3650.00	.	.	.	2700.00	172.00	226.00	.
5	11	1982	4698.0	.	3.40	4250.00	.	.	.	4200.00	190.00	200.00	.
5	14	1982	6787.0	.	3.40	4550.00	.	.	.	4700.00	248.00	348.00	.
5	21	1982	2006.0	.	3.30	4725.00	.	.	.	8000.00	320.00	660.00	.
6	2	1982	723.0	.	3.25	6160.00	.	.	.	8500.00	342.00	790.00	.
6	18	1982	185.0	.	3.30	5424.00	.	.	.	8000.00	298.00	680.00	.
7	2	1982	7908.0	.	.	.	.	.	.	5000.00	308.00	362.00	.
7	26	1982	6908.0	.	3.24	5266.00	.	.	.	5500.00	314.00	408.00	.
8	30	1982	4376.0	.	3.26	4421.00	.	.	.	3750.00	302.00	374.00	.

Table A2.5.16. Drainage quality data from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 3 of 5.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
10	4	1982	23454.0	.	3.60	4316.00	.	.	.	3750.00	230.00	336.00	.
11	1	1982	1363.0	.	3.40	4200.00	.	.	.	.	168.00	278.00	.
3	18	1983	1537.0	.	3.66	3154.00	.	.	.	3360.00	.	.	.
4	14	1983	3311.0	.	3.70	3209.00	.	.	.	3400.00	.	.	.
5	15	1983	2361.0	.	3.54	3873.00	.	.	.	4720.00	.	.	.
6	16	1983	507.0	.	3.67	4471.00	.	.	.	5000.00	.	.	.
7	4	1983	5314.0	.	3.50	3825.00	.	.	.	3700.00	.	.	.
7	11	1983	731.0	.	3.70	5000.00	.	.	.	7200.00	.	.	.
7	17	1983	284.0	.	3.50	6572.00	.	.	.	7700.00	.	.	.
7	20	1983	825.0	.	3.50	6608.00	.	.	.	6200.00	.	.	.
7	25	1983	341.0	.	3.60	6250.00	.	.	.	8200.00	.	.	.
7	31	1983	148.0	.	3.65	6405.00	.	.	.	8900.00	.	.	.
8	7	1983	590.0	.	3.60	4860.00	.	.	.	4700.00	.	.	.
8	15	1983	307.0	.	3.45	6376.00	.	.	.	6400.00	.	.	.
8	22	1983	3653.0	.	3.60	3500.00	.	.	.	3150.00	.	.	.
8	28	1983	1393.0	.	3.60	4300.00	.	.	.	4000.00	.	.	.
9	4	1983	636.0	.	3.55	5537.00	.	.	.	5500.00	.	.	.
9	11	1983	1298.0	.	3.45	3975.00	.	.	.	3150.00	.	.	.
9	18	1983	560.0	.	3.50	4500.00	.	.	.	4300.00	.	.	.
9	25	1983	874.0	.	3.80	3725.00	.	.	.	3500.00	.	.	.
10	2	1983	10401.0	.	3.50	4475.00	.	.	.	3400.00	.	.	.
10	8	1983	6018.0	.	3.75	4725.00	.	.	.	3650.00	.	.	.
10	15	1983	6964.0	.	3.70	4624.00	.	.	.	4050.00	.	.	.
10	23	1983	1480.0	.	3.50	5538.00	.	.	.	5300.00	.	.	.
10	30	1983	1083.0	.	3.65	5576.00	.	.	.	6200.00	.	.	.
11	6	1983	333.0	.	3.70	6116.00	.	.	.	7000.00	.	.	.
11	13	1983	212.0	.	3.80	6958.00	.	.	.	8100.00	.	.	.
11	20	1983	2120.0	.	3.50	3800.00	.	.	.	3250.00	.	.	.
12	1	1983	1881.0	.	3.55	4875.00	.	.	.	4600.00	.	.	.
2	24	1984	889.0	.	3.60	6110.00	.	.	.	6800.00	356.50	991.30	500
3	15	1984	1211.0	.	3.55	3225.00	.	.	.	3200.00	220.60	381.10	501
4	15	1984	11004.0	.	3.81	1260.00	.	.	.	1300.00	110.90	104.80	502
5	16	1984	8396.0	.	3.65	2392.00	.	.	.	2950.00	.	.	503
6	13	1984	11254.0	.	3.58	2985.00	.	.	.	4500.00	242.80	342.50	504
7	14	1984	2498.0	.	3.45	5230.00	.	.	.	6700.00	.	.	505
8	15	1984	2252.0	.	3.47	4760.00	.	.	.	5000.00	323.70	348.70	506
9	12	1984	886.0	.	3.44	5175.00	.	.	.	6000.00	374.00	580.00	507
10	13	1984	7066.0	.	3.41	2870.00	.	.	.	3625.00	260.10	290.00	508
11	14	1984	2074.0	.	3.51	3150.00	.	.	.	2750.00	372.00	500.00	509
12	5	1984	651.0	.	3.42	2470.00	.	.	.	2200.00	314.00	470.00	510
12	19	1984	1444.0	.	3.45	2650.00	.	.	.	2900.00	254.00	285.00	511
4	2	1985	489.0	.	3.08	5000.00	.	.	.	5650.00	330.00	562.00	500
4	16	1985	14152.0	.	3.94	2745.00	.	.	.	2850.00	210.00	252.00	501
5	14	1985	3970.0	.	3.74	4640.00	.	.	.	4808.00	285.00	475.00	502
6	14	1985	22384.0	.	3.58	3850.00	.	.	.	4200.00	224.00	375.00	503
7	15	1985	4534.0	.	3.48	4900.00	.	.	.	5200.00	270.00	446.00	504
8	13	1985	2203.0	.	3.47	4450.00	.	.	.	5100.00	272.00	466.00	505
9	13	1985	16207.0	.	3.50	4325.00	.	.	.	4300.00	334.00	305.00	506
10	15	1985	17676.0	.	3.50	4075.00	.	.	.	3200.00	276.00	236.00	507
11	15	1985	1499.0	.	3.54	5200.00	.	.	.	2100.00	388.00	510.00	508
12	10	1985	106.0	.	.	.	.	.	.	.	.	.	0
3	23	1986	7244.0	.	3.80	4800.00	.	.	.	4200.00	270.00	414.00	500
4	16	1986	9909.0	.	3.90	1950.00	.	.	.	1700.00	152.00	130.00	501
5	14	1986	10125.0	.	3.69	3000.00	.	.	.	3150.00	282.00	270.00	502
6	11	1986	1711.0	.	3.63	4750.00	.	.	.	4600.00	316.00	360.00	503
7	12	1986	3085.0	.	3.59	1725.00	.	.	.	3000.00	225.00	314.00	504
8	13	1986	1597.0	.	3.55	4100.00	.	.	.	3700.00	268.00	357.00	505
9	14	1986	10795.0	.	3.40	3550.00	.	.	.	2700.00	262.00	286.00	506
10	16	1986	3982.0	.	3.53	4400.00	.	.	.	3600.00	286.00	302.00	507
11	16	1986	6075.0	.	3.70	3550.00	.	.	.	2700.00	260.00	236.00	508
3	28	1987	7835.0	.	3.65	3075.00	.	.	.	2900.00	252.00	264.00	500
4	18	1987	1620.0	.	3.67	2625.00	.	.	.	2400.00	154.00	220.00	501
5	16	1987	18573.0	.	3.61	2700.00	.	.	.	2250.00	233.00	172.00	502
6	16	1987	1400.0	.	3.44	4900.00	.	.	.	4640.00	428.00	492.00	503
7	15	1987	16908.0	.	3.36	3000.00	.	.	.	2900.00	230.00	228.00	504

Table A2.5.16. Drainage quality data from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 4 of 5.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
8	15	1987	3948.0	.	3.24	3950.00	.	.	.	3565.00	279.00	316.00	505
9	16	1987	7612.0	.	3.25	4100.00	.	.	.	1200.00	240.00	238.00	506
10	13	1987	1457.0	.	3.32	4950.00	.	.	.	5120.00	364.00	391.00	507
11	10	1987	731.0	.	3.43	4325.00	.	.	.	3700.00	308.00	354.00	508
5	12	1988	7275.0	.	3.64	3000.00	.	.	.	2300.00	224.00	198.00	502
6	12	1988	1098.0	.	3.47	4850.00	.	.	.	4500.00	412.00	376.00	503
7	15	1988	757.0	.	3.60	4525.00	.	.	.	4360.00	368.00	382.00	504
8	16	1988	32744.0	.	3.39	2900.00	.	.	.	2165.00	247.00	176.00	505
9	16	1988	6310.0	.	3.37	4275.00	.	.	.	3200.00	334.00	294.00	506
9	24	1988	6934.0	.	3.70	1875.00	.	.	.	1280.00	160.00	96.00	500
10	18	1988	1790.0	.	3.67	3700.00	.	.	.	3400.00	238.00	258.00	507
10	24	1988	3819.0	.	3.77	1980.00	.	.	.	1400.00	132.00	118.00	501
11	15	1988	7494.0	.	3.71	2975.00	.	.	.	2740.00	216.00	222.00	508
11	30	1988	2665.0	.	3.44	4200.00	.	.	.	3760.00	276.00	306.00	509
4	7	1989	17581.0	.	3.95	2700.00	.	.	.	2400.00	230.00	166.00	500
4	24	1989	7850.0	.	4.00	1050.00	.	.	.	650.00	78.00	38.00	501
5	16	1989	5802.0	.	3.79	2582.00	.	.	.	2100.00	182.00	152.00	502
6	17	1989	25560.0	.	3.64	3150.00	.	.	.	2800.00	258.00	184.00	503
7	17	1989	2248.0	.	3.55	4325.00	.	.	.	4100.00	420.00	304.00	504
8	14	1989	1158.0	.	3.62	3800.00	.	.	.	3540.00	286.00	284.00	505
9	14	1989	16325.0	.	3.41	2900.00	.	.	.	2460.00	272.00	168.00	506
10	16	1989	2203.0	.	3.51	3700.00	.	.	.	2960.00	276.00	226.00	507
11	13	1989	3758.0	.	3.60	3200.00	.	.	.	2480.00	298.00	204.00	508
3	26	1990	3615.0	.	3.70	2500.00	.	.	.	1900.00	226.00	146.00	500
4	12	1990	20708.0	.	3.91	1950.00	.	.	.	1660.00	182.00	108.00	501
5	17	1990	10783.0	.	3.75	2725.00	.	.	.	2300.00	222.00	156.00	502
6	18	1990	9421.0	.	3.59	3475.00	.	.	.	2660.00	236.00	180.00	503
7	16	1990	6684.0	.	3.62	4175.00	.	.	.	3460.00	306.00	226.00	504
8	13	1990	757.0	.	3.53	4785.00	.	.	.	3920.00	370.00	326.00	505
9	10	1990	2434.0	.	3.64	3260.00	.	.	.	2360.00	238.00	178.00	506
10	15	1990	20643.0	.	3.55	3415.00	.	.	.	2160.00	252.00	154.00	507
11	16	1990	1397.0	.	3.60	5000.00	.	.	.	3940.00	372.00	308.00	508
4	8	1991	9777.0	.	.	.	.	.	.	.	.	.	.
4	22	1991	5598.0	.	3.76	1700.00	.	.	.	1010.00	120.00	80.00	2021
5	20	1991	8687.0	.	3.65	4175.00	.	.	.	2500.00	262.00	205.00	2025
6	17	1991	3388.0	.	3.61	2975.00	.	.	.	2000.00	188.00	168.00	2041
7	15	1991	16938.0	.	3.47	3225.00	.	.	.	2020.00	198.00	140.00	2057
8	12	1991	1268.0	.	3.43	5000.00	.	.	.	4460.00	394.00	334.00	2073
9	9	1991	6079.0	.	3.59	3225.00	.	.	.	1940.00	200.00	126.00	2088
10	7	1991	3656.0	.	3.62	3775.00	.	.	.	2380.00	238.00	158.00	2104
11	7	1991	14780.0	.	3.51	3300.00	.	.	.	2280.00	236.00	156.00	2120
4	23	1992	4634.0	.	3.86	1525.00	.	410.0	223.2	720.00	76.00	56.00	17002
5	11	1992	2162.0	.	3.76	2420.00	.	695.0	388.8	725.00	148.00	96.00	17015
5	26	1992	2294.0	.	3.71	2480.00	.	838.0	449.5	1760.00	176.00	118.00	17043
6	8	1992	603.0	.	4.22	3780.00	.	940.0	541.4	2560.00	320.00	216.00	17047
6	22	1992	238.0	.	3.66	3850.00	.	1350.0	667.2	2960.00	252.00	208.00	17051
7	7	1992	1105.0	.	3.68	2640.00	.	860.0	445.1	1880.00	174.00	124.00	17056
7	20	1992	366.0	.	3.61	3880.00	.	1310.0	652.7	5900.00	284.00	202.00	17060
8	3	1992	336.0	.	3.64	2710.00	.	1050.0	486.2	1990.00	192.00	144.00	17076
8	17	1992	422.0	.	4.43	3000.00	.	550.0	337.1	2060.00	214.00	228.00	17080
8	31	1992	309.0	.	4.04	2400.00	.	410.0	271.4	1400.00	170.00	144.00	17096
9	14	1992	2397.0	.	3.57	2800.00	.	1085.0	473.6	2060.00	216.00	144.00	17100
9	28	1992	1997.0	.	3.60	3300.00	.	1120.0	507.0	2240.00	228.00	150.00	17117
10	12	1992	285.0	.	4.35	4200.00	.	1130.0	554.2	2940.00	326.00	272.00	17121
10	27	1992	147.0	.	3.57	4380.00	.	1700.0	720.1	3400.00	288.00	260.00	17137
11	9	1992	69.0	.	4.07	5000.00	.	.	.	.	.	.	.
11	30	1992	411.0	.	3.75	2700.00	.	1090.0	445.3	2050.00	184.00	158.00	17152
4	8	1993	15454.0	.	3.82	1825.00	.	.	.	1275.00	125.00	100.00	17164
4	22	1993	3653.0	.	3.76	2775.00	.	.	.	1930.00	180.00	137.50	17175
5	6	1993	8846.0	.	3.68	2100.00	.	.	.	1560.00	147.50	105.00	17180
5	20	1993	1915.0	.	3.67	3350.00	.	.	.	2880.00	245.00	200.00	17195
6	3	1993	2415.0	.	3.76	2550.00	.	.	.	1800.00	147.50	125.00	17199
6	17	1993	2933.0	.	3.56	2300.00	.	.	.	1990.00	180.00	152.50	17215
7	1	1993	3338.0	.	3.58	2550.00	.	.	.	1900.00	182.50	147.50	17219
7	14	1993	15715.0	.	3.57	3090.00	.	.	.	2820.00	228.00	154.00	17236

Table A2.5.16. Drainage quality data from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 5 of 5.

Month	Day	Year	Volume L	Daily Flow L	pH s.u.	SC µMho/cm	Alk mg/L	Lab Acy mg/L	Calc Acy mg/L	Sulfate mg/L	Calcium mg/L	Magnesium mg/L	Sample Number
7	29	1993	2986.0	.	3.58	3100.00	.	.	.	2600.00	242.00	170.00	17240
8	12	1993	2245.0	.	3.50	3350.00	.	.	.	2450.00	238.00	170.00	17256
8	26	1993	3338.0	.	3.47	4030.00	.	.	.	3160.00	314.00	242.00	17261
9	9	1993	2865.0	.	3.60	2900.00	.	.	.	2080.00	216.00	144.00	17276
9	23	1993	1446.0	.	3.61	3300.00	.	.	.	2450.00	266.00	190.00	17280
10	7	1993	742.0	.	3.66	3700.00	.	.	.	2560.00	276.00	200.00	17296
10	20	1993	379.0	.	3.71	3950.00	.	.	.	2730.00	280.00	208.00	17300
11	4	1993	871.0	.	3.72	3300.00	.	.	.	1250.00	159.40	138.00	17318
11	22	1993	591.0	.	3.72	3000.00	.	.	.	1170.00	270.00	210.00	17332

Table A2.5.17. Drainage quality data for metals from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
4	26	1978	153.00	6.00	.	0.020	0.445	0.045	.	.	0.270	.
5	25	1978	273.00	10.00	.	0.021	0.780	0.136	0.096	0.020	0.890	.
6	16	1978	336.00	15.00	.	0.044	0.835	0.160	0.174	0.030	1.400	.
7	5	1978	231.00	13.00	.	0.140	5.100	0.195	0.257	0.020	1.290	.
7	13	1978	248.00	14.00	.	0.090	6.700	0.810	0.230	0.040	1.670	.
7	19	1978	245.00	16.00	.	0.141	21.500	2.160	0.396	0.020	2.400	.
8	2	1978	144.50	13.00	.	0.265	10.280	1.240	0.400	0.050	2.080	.
8	4	1978	238.00	17.00	.	0.420	3.230	1.860	0.774	0.050	3.320	.
8	10	1978	338.00	20.00	.	1.280	3.310	2.200	1.220	0.120	4.290	.
8	16	1978	104.70	9.70	.	1.157	22.453	3.790	0.623	7.960	3.060	.
8	17	1978	178.00	33.00	.	2.530	40.380	6.730	1.240	11.540	5.140	.
8	18	1978	354.00	8.00	.	2.980	42.210	7.220	1.450	5.280	5.870	.
8	23	1978	107.00	10.00	.	1.450	20.240	3.400	0.625	4.010	2.990	.
8	25	1978	177.00	15.00	.	3.150	39.400	6.360	1.180	2.780	5.400	.
8	28	1978	220.00	18.00	.	3.950	48.000	7.760	1.490	2.380	6.470	.
9	11	1978	79.00	9.00	.	1.810	18.000	1.510	0.560	0.750	2.900	.
9	12	1978	133.00	17.00	.	7.280	67.000	4.880	1.750	24.000	10.000	.
9	13	1978	183.00	21.00	.	10.300	83.000	6.170	2.460	23.000	13.000	.
10	17	1978	211.00	15.00	.	11.450	79.000	5.730	3.290	0.630	10.000	.
3	23	1979	121.80	14.80	.	44.000	169.000	14.000	5.000	4.010	11.800	.
4	3	1979	286.00	28.00	.	91.000	335.000	25.000	11.000	11.000	28.000	.
4	11	1979	338.00	30.00	.	107.000	406.000	31.000	13.000	10.000	35.000	.
4	16	1979	146.80	15.20	.	38.000	173.000	14.000	5.000	8.730	15.700	.
4	17	1979	70.00	8.20	.	15.000	87.000	6.000	2.080	7.700	8.500	.
4	18	1979	68.00	7.20	.	13.000	74.000	5.190	1.900	13.330	7.220	.
4	19	1979	52.00	6.40	.	8.980	48.000	3.690	1.360	11.620	4.890	.
4	20	1979	65.00	6.60	.	14.000	53.000	4.300	1.890	3.780	5.350	.
4	30	1979	185.60	12.20	.	40.000	107.400	8.400	3.950	4.540	10.720	.
5	11	1979	133.00	9.00	.	33.000	94.000	7.400	3.600	1.800	9.600	.
5	22	1979	79.00	7.00	.	23.000	71.000	4.860	2.370	3.610	6.650	.
5	23	1979	162.80	14.00	.	41.600	116.000	8.600	4.160	4.030	10.090	.
5	27	1979	151.20	13.10	.	35.900	104.000	7.800	3.680	4.640	9.420	.
6	1	1979	121.40	11.60	.	30.800	84.600	5.570	3.000	3.000	7.750	.
6	6	1979	60.80	9.20	.	26.000	80.000	5.180	2.830	3.170	7.490	.
6	10	1979	68.20	11.00	.	30.000	94.000	6.260	3.310	7.320	8.700	.
6	16	1979	115.00	14.80	.	40.000	115.000	7.570	4.130	5.410	10.100	.
6	20	1979	71.00	11.00	.	14.000	59.000	4.320	2.220	3.060	0.110	.
6	30	1979	132.00	16.50	.	25.500	96.500	7.205	3.890	6.780	9.380	.
7	10	1979	204.00	29.00	.	50.000	163.000	14.000	5.950	4.830	14.100	.
7	30	1979	63.00	12.00	.	7.000	46.000	4.420	2.320	2.100	6.320	.
8	4	1979	120.00	21.00	.	25.000	132.000	8.190	4.470	8.490	13.320	.
8	13	1979	322.00	32.60	.	76.000	259.000	19.000	9.590	11.100	22.800	.
8	23	1979	202.00	27.20	.	44.000	202.000	12.000	7.460	6.900	17.890	.
8	28	1979	158.60	21.80	.	26.000	142.000	12.000	5.800	5.240	14.200	.
9	1	1979	98.20	18.00	.	31.000	138.000	8.000	4.560	19.600	13.340	.
9	5	1979	192.80	28.40	.	77.000	227.000	18.000	8.490	26.200	20.200	.
9	13	1979	192.60	26.60	.	78.000	219.000	16.000	8.290	22.000	19.000	.
9	13	1979	197.00	26.20	.	77.000	224.000	17.000	8.140	22.850	19.200	.
9	19	1979	286.00	39.20	.	131.000	348.000	24.000	10.130	25.500	28.000	.
10	18	1979	272.00	36.40	.	91.000	347.000	25.000	10.130	15.760	28.000	.
10	19	1979	76.00	12.80	.	18.000	115.000	10.000	4.390	4.160	10.090	.
10	31	1979	150.00	22.00	.	49.000	268.000	25.100	8.110	10.600	19.450	.
11	1	1979	132.00	22.60	.	63.000	268.000	24.600	8.000	11.770	18.600	.
3	29	1980	90.80	15.00	.	41.110	206.500	15.110	6.530	2.260	15.000	.
3	31	1980	57.40	10.00	.	34.080	144.000	8.940	3.870	7.290	11.000	.
4	5	1980	49.00	9.00	.	21.855	124.000	7.725	3.300	8.130	9.250	.
4	14	1980	63.60	10.40	.	30.600	131.000	9.000	4.140	5.900	11.000	.
4	18	1980	86.60	12.60	.	46.000	187.000	14.000	5.930	7.910	17.000	.
4	27	1980	166.60	20.40	.	97.000	321.000	21.000	14.000	10.770	28.000	.
5	10	1980	210.60	26.40	.	133.000	423.000	30.000	18.000	15.000	37.000	.
5	18	1980	223.60	29.60	.	140.000	464.500	35.500	20.000	12.190	40.000	.
5	31	1980	112.80	18.20	.	54.000	245.000	15.100	8.400	6.940	19.000	.
6	3	1980	139.80	22.00	.	58.000	273.000	17.600	9.760	6.080	20.000	.
6	7	1980	125.20	20.80	.	54.000	261.000	16.500	9.200	6.450	19.000	.
6	13	1980	152.20	23.40	.	63.000	289.000	19.100	10.250	7.500	20.000	.
6	19	1980	181.00	26.20	.	80.000	360.000	23.200	12.050	8.330	26.000	.
6	28	1980	147.00	23.00	.	59.000	269.000	24.000	12.000	9.000	24.000	.

Table A2.5.17. Drainage quality data for metals from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
6	30	1980	155.00	23.00	.	67.000	330.000	28.000	13.000	8.000	26.000	.
7	2	1980	57.00	12.00	.	23.000	129.000	12.000	4.660	5.730	10.000	.
7	13	1980	220.00	31.00	.	84.000	415.000	28.000	15.000	9.900	36.000	.
7	23	1980	232.00	30.00	.	80.000	411.600	30.000	14.000	9.900	37.000	.
8	3	1980	274.00	34.00	.	90.000	582.000	36.000	19.000	13.000	47.000	.
8	12	1980	134.00	24.00	.	36.000	279.000	19.000	9.000	7.750	25.000	.
8	17	1980	162.00	28.00	.	46.000	357.000	25.000	11.000	7.450	30.000	.
8	22	1980	133.00	17.00	.	31.000	266.000	21.000	11.000	6.500	22.000	.
8	29	1980	65.00	12.00	.	22.000	141.000	10.420	5.300	5.220	12.000	.
9	8	1980	133.00	21.00	.	44.000	262.000	21.000	10.000	8.830	22.000	.
9	9	1980	154.00	22.00	.	54.000	296.000	22.000	12.000	7.300	25.000	.
9	12	1980	75.00	13.00	.	28.000	174.000	11.000	8.000	5.160	14.000	.
9	18	1980	94.00	15.00	.	56.000	217.000	15.000	9.000	5.450	17.000	.
9	25	1980	189.00	38.00	.	60.000	443.000	26.500	15.500	8.710	35.000	.
10	23	1980	202.00	32.00	.	59.300	472.000	28.800	16.900	8.680	38.000	.
10	31	1980	187.00	26.00	.	75.000	353.000	28.000	14.000	6.000	33.000	.
2	19	1981	148.00	.	.	49.300	270.000	18.100	8.850	.	.	.
2	21	1981	87.10	.	.	26.800	183.000	11.000	4.920	.	.	.
3	29	1981	53.30	.	.	20.200	123.000	7.600	3.680	.	.	.
4	3	1981	46.80	.	.	23.600	115.000	6.900	3.590	.	.	.
4	13	1981	69.00	.	.	71.000	239.000	13.800	7.900	.	.	.
4	17	1981	100.00	.	.	76.000	264.000	14.800	8.510	.	.	.
4	23	1981	108.00	.	.	107.000	331.000	17.300	10.700	.	.	.
4	25	1981	81.00	.	.	76.000	251.000	14.700	8.050	.	.	.
4	28	1981	76.00	.	.	70.000	233.000	14.200	7.730	.	.	.
4	29	1981	114.00	.	.	105.000	312.000	20.300	8.150	.	.	.
5	12	1981	200.00	.	.	159.000	501.000	26.300	9.780	.	.	.
5	28	1981	182.00	.	.	137.000	500.000	25.200	9.630	.	.	.
6	6	1981	158.00	.	.	102.000	441.500	25.910	9.580	.	.	.
6	22	1981	114.00	.	.	63.000	354.000	21.750	7.990	.	.	.
6	24	1981	56.00	.	.	34.000	199.000	13.200	5.700	.	.	.
6	28	1981	58.00	.	.	45.000	204.500	13.800	5.600	.	.	.
7	4	1981	106.00	.	.	117.000	307.500	22.000	11.000	.	.	.
7	24	1981	242.00	.	.	185.000	550.000	43.000	21.500	.	.	.
8	4	1981	256.00	.	.	129.500	676.000	50.000	25.500	.	.	.
8	17	1981	260.00	34.00	.	104.000	630.000	43.000	20.000	.	.	.
8	31	1981	250.00	.	.	108.000	741.000	47.000	25.000	.	.	.
9	7	1981	210.00	.	.	80.000	660.000	42.000	22.000	.	.	.
9	16	1981	260.00	.	.	92.000	762.000	49.000	26.000	.	.	.
9	26	1981	120.00	.	.	39.000	381.000	24.000	12.000	.	.	.
10	1	1981	86.00	.	.	23.000	240.000	16.000	8.000	.	.	.
10	4	1981	64.00	14.00	.	32.000	200.000	16.000	6.000	.	.	.
10	14	1981	68.00	14.00	.	53.000	198.000	15.000	8.000	.	.	.
10	18	1981	102.00	10.00	.	58.000	119.000	7.000	4.130	.	.	.
11	6	1981	140.00	26.00	.	140.000	412.000	26.000	11.000	.	.	.
3	30	1982	68.00	12.00	.	55.000	233.000	18.000	8.050	.	.	.
4	6	1982	70.00	11.00	.	65.000	253.000	17.500	8.180	.	.	.
4	13	1982	36.00	8.00	.	36.000	138.000	8.900	4.100	.	.	.
4	15	1982	23.00	6.00	.	28.000	88.500	6.200	2.430	.	.	.
4	19	1982	38.00	10.00	.	39.000	126.000	8.100	3.750	.	.	.
4	23	1982	70.00	12.00	.	62.000	197.000	12.200	5.470	.	.	.
5	3	1982	126.00	22.00	.	126.000	405.000	28.000	12.000	.	.	.
5	4	1982	66.00	12.00	.	58.000	142.000	12.200	5.710	.	.	.
5	7	1982	136.00	16.00	.	105.000	316.000	22.000	10.000	.	.	.
5	10	1982	74.00	12.00	.	72.000	204.000	12.700	5.740	.	.	.
5	11	1982	10.00	14.00	.	88.000	251.000	20.000	9.000	.	.	.
5	14	1982	86.00	16.40	.	88.000	254.000	16.300	7.780	3.630	17.040	.
5	21	1982	153.00	21.00	.	135.000	416.000	28.000	14.000	4.010	24.000	.
6	2	1982	190.00	26.00	.	156.000	549.000	32.000	20.000	4.510	33.000	.
6	18	1982	155.00	22.00	.	113.000	440.000	30.000	16.000	3.050	28.000	.
7	2	1982	110.00	18.00	.	97.000	359.000	21.800	8.900	3.570	19.570	.
7	26	1982	120.00	20.00	.	102.000	443.000	25.000	10.500	5.520	23.290	.
8	30	1982	122.00	18.00	.	86.000	344.000	21.200	8.500	3.260	18.890	.
10	4	1982	92.00	16.00	.	92.000	389.000	20.000	8.300	5.440	20.500	.
11	1	1982	76.00	12.00	.	58.000	291.000	17.200	7.000	3.030	17.600	.
3	18	1983	.	.	.	38.000	161.000	13.000	5.380	.	.	.
4	14	1983	.	.	.	40.000	174.000	14.000	5.690	.	.	.

Table A2.5.17. Drainage quality data for metals from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
5	15	1983	.	.	.	90.000	240.000	18.000	8.540	.	.	.
6	16	1983	.	.	.	84.000	307.000	16.600	6.490	.	.	.
7	4	1983	.	.	.	109.000	256.000	18.400	8.200	.	.	.
7	11	1983	.	.	.	123.000	480.000	32.400	14.800	.	.	.
7	17	1983	.	.	.	127.000	418.000	28.300	15.200	.	.	.
7	20	1983	.	.	.	140.000	392.000	28.000	13.400	.	.	.
7	25	1983	.	.	.	154.000	510.000	36.200	16.200	.	.	.
7	31	1983	.	.	.	142.000	535.000	38.000	16.800	.	.	.
8	7	1983	.	.	.	70.600	314.000	23.800	10.600	.	.	.
8	15	1983	.	.	.	81.000	384.000	29.600	12.600	.	.	.
8	22	1983	.	.	.	71.000	201.000	14.640	5.920	.	.	.
8	28	1983	.	.	.	89.000	265.000	18.480	8.140	.	.	.
9	4	1983	.	.	.	110.000	349.000	23.500	10.830	.	.	.
9	11	1983	.	.	.	75.000	237.000	16.790	7.370	.	.	.
9	18	1983	.	.	.	85.000	289.000	19.760	8.780	.	.	.
9	25	1983	.	.	.	69.000	240.000	17.000	7.380	.	.	.
10	2	1983	.	.	.	108.000	241.000	18.000	7.460	.	.	.
10	8	1983	.	.	.	106.000	258.000	15.500	7.820	.	.	.
10	15	1983	.	.	.	116.000	290.000	19.000	8.600	.	.	.
10	23	1983	.	.	.	130.000	357.000	23.000	10.350	.	.	.
10	30	1983	.	.	.	149.000	410.000	29.000	13.600	.	.	.
11	6	1983	.	.	.	131.800	473.000	32.800	15.200	.	.	.
11	13	1983	.	.	.	136.000	520.000	36.800	17.400	.	.	.
11	20	1983	.	.	.	82.000	236.000	16.200	7.800	.	.	.
12	1	1983	.	.	.	117.600	314.000	21.600	10.400	.	.	.
2	24	1984	165.90	.	.	133.800	478.000	32.200	17.400	.	.	500
3	15	1984	73.20	.	.	77.000	220.000	15.000	7.200	.	.	501
4	15	1984	23.00	.	.	31.000	62.000	4.400	2.300	.	.	502
5	16	1984	.	.	.	83.000	160.000	12.500	5.340	.	.	503
6	13	1984	69.30	.	.	90.200	207.000	14.000	6.600	.	.	504
7	14	1984	122.00	.	.	142.000	356.000	23.000	7.000	.	.	505
8	15	1984	115.60	.	.	107.000	334.560	34.400	10.540	.	.	506
9	12	1984	146.50	.	.	105.800	425.000	44.000	14.600	.	.	507
10	13	1984	74.30	.	.	88.700	218.000	21.600	7.700	.	.	508
11	14	1984	115.80	.	.	134.700	338.000	19.400	10.700	.	.	509
12	5	1984	89.00	.	.	100.100	260.000	15.500	8.500	.	.	510
12	19	1984	88.00	.	.	97.300	223.000	13.400	6.720	.	.	511
4	2	1985	135.60	.	.	104.000	348.000	23.000	13.480	.	.	500
4	16	1985	51.20	.	.	86.000	157.000	13.000	5.400	.	.	501
5	14	1985	95.00	.	.	150.000	279.000	10.500	18.800	.	.	502
6	14	1985	70.00	.	.	110.000	211.000	14.600	7.100	.	.	503
7	15	1985	102.00	.	.	110.000	269.000	17.000	7.800	.	.	504
8	13	1985	96.00	.	.	96.000	266.000	17.800	6.380	.	.	505
9	13	1985	78.00	.	.	103.000	221.000	14.800	6.500	.	.	506
10	15	1985	60.00	.	.	94.000	174.000	12.500	5.320	.	.	507
11	15	1985	100.00	.	.	119.000	296.000	20.700	8.570	.	.	508
12	10	1985	.	.	.	.	.	.	.	.	.	0
3	23	1986	86.00	.	.	67.000	250.000	22.000	7.340	.	.	500
4	16	1986	28.60	.	.	47.000	97.000	11.600	3.010	.	.	501
5	14	1986	44.00	.	.	92.000	251.000	11.000	5.000	.	.	502
6	11	1986	90.00	.	.	124.000	260.000	14.000	8.200	.	.	503
7	12	1986	61.00	.	.	77.000	180.000	12.700	6.300	.	.	504
8	13	1986	78.00	.	.	79.000	206.000	22.400	7.400	.	.	505
9	14	1986	56.00	.	.	98.000	159.000	19.400	5.980	.	.	506
10	16	1986	68.00	.	.	110.000	203.000	13.700	7.150	.	.	507
11	16	1986	60.00	.	.	110.000	158.000	10.600	7.200	.	.	508
3	28	1987	48.00	.	.	75.000	160.000	10.800	5.160	.	.	500
4	18	1987	40.00	.	.	63.000	137.000	10.000	4.440	.	.	501
5	16	1987	34.00	.	.	73.000	97.000	6.700	3.730	.	.	502
6	16	1987	116.00	.	.	153.000	331.000	18.600	25.200	.	.	503
7	15	1987	48.00	.	.	82.000	140.000	9.000	4.930	.	.	504
8	15	1987	68.00	.	.	97.000	193.000	12.200	6.590	.	.	505
9	16	1987	50.00	.	.	79.000	135.000	7.980	4.660	.	.	506
10	13	1987	88.00	.	.	132.000	220.000	13.010	7.250	.	.	507
11	10	1987	78.00	.	.	9.300	193.000	11.240	6.360	.	.	508
5	12	1988	54.00	.	.	89.500	156.500	8.950	8.450	.	.	502
6	12	1988	72.00	.	.	163.000	232.000	18.600	23.000	.	.	503

Table A2.5.17. Drainage quality data for metals from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Na mg/L	K mg/L	Al mg/L	Cu mg/L	Ni mg/L	Co mg/L	Zn mg/L	Fe mg/L	Mn mg/L	Sample Number
7	15	1988	74.00	.	.	134.000	234.000	17.600	16.000	.	.	504
8	16	1988	43.00	.	.	72.000	152.000	8.800	6.250	.	.	505
9	16	1988	74.00	.	.	93.000	117.000	14.300	10.300	.	.	506
9	24	1988	26.00	.	.	36.000	71.000	4.300	3.700	.	.	500
10	18	1988	50.00	.	.	64.000	143.000	10.900	8.700	.	.	507
10	24	1988	26.00	.	.	43.000	88.000	4.900	4.400	.	.	501
11	15	1988	42.00	.	.	68.000	107.000	8.200	7.200	.	.	508
11	30	1988	56.00	.	.	102.000	169.000	11.500	9.300	.	.	509
4	7	1989	38.00	.	.	55.000	98.000	9.800	7.000	.	.	500
4	24	1989	10.00	.	.	18.000	26.000	1.800	1.600	.	.	501
5	16	1989	34.00	.	.	62.000	94.000	7.300	6.100	.	.	502
6	17	1989	52.00	.	.	80.000	145.000	13.700	7.700	.	.	503
7	17	1989	62.00	.	.	118.000	210.000	17.700	10.900	.	.	504
8	14	1989	58.00	.	.	89.000	245.000	15.400	10.200	.	.	505
9	14	1989	36.00	.	.	68.000	91.000	11.800	6.600	.	.	506
10	16	1989	52.00	.	.	83.000	116.000	17.700	9.600	.	.	507
11	13	1989	44.00	.	.	74.000	105.000	21.000	8.900	.	.	508
3	26	1990	31.20	.	.	41.000	73.000	6.350	7.310	.	.	500
4	12	1990	20.00	.	.	53.000	57.000	4.380	3.980	.	.	501
5	17	1990	26.00	.	.	70.000	80.000	6.570	5.560	.	.	502
6	18	1990	38.00	.	.	72.000	92.000	9.170	7.640	.	.	503
7	16	1990	40.00	.	.	92.000	108.000	9.600	8.610	.	.	504
8	13	1990	62.00	.	.	106.000	166.000	13.100	11.200	.	.	505
9	10	1990	38.00	.	.	54.000	99.000	7.280	5.380	.	.	506
10	15	1990	32.00	.	.	74.000	88.000	6.510	5.000	.	.	507
11	16	1990	62.00	.	.	120.000	142.000	12.170	12.440	.	.	508
4	8	1991	.	.	.	.	.	.	.	.	.	.
4	22	1991	14.00	.	.	34.000	41.000	2.200	1.600	.	.	2021
5	20	1991	32.00	.	.	94.000	103.000	5.600	4.200	.	.	2025
6	17	1991	26.00	.	.	64.000	84.000	5.300	3.300	.	.	2041
7	15	1991	24.00	.	.	66.000	69.000	4.600	3.500	.	.	2057
8	12	1991	72.00	.	.	124.000	176.000	11.900	8.700	.	.	2073
9	9	1991	32.00	.	.	54.000	73.000	4.600	3.700	.	.	2088
10	7	1991	38.00	.	.	72.000	87.000	5.500	5.600	.	.	2104
11	7	1991	34.00	.	.	75.000	80.000	5.100	5.100	.	.	2120
4	23	1992	14.00	3.30	24.00	19.900	28.000	1.700	1.200	0.340	3.040	17002
5	11	1992	22.00	5.30	37.00	53.000	47.000	2.900	2.100	0.870	5.520	17015
5	26	1992	26.60	6.30	42.00	59.000	58.000	3.600	2.800	1.070	6.680	17043
6	8	1992	40.00	11.00	44.00	79.000	82.000	5.100	4.000	0.470	9.480	17047
6	22	1992	42.80	9.10	56.00	93.000	99.000	6.800	4.700	1.340	10.900	17051
7	7	1992	28.00	7.20	40.00	59.000	61.000	4.100	3.000	1.090	6.890	17056
7	20	1992	44.00	10.00	53.00	99.000	95.000	6.700	4.600	1.440	10.700	17060
8	3	1992	34.00	8.60	43.00	63.000	69.000	4.900	3.600	1.130	8.080	17076
8	17	1992	39.00	17.10	23.00	51.000	60.000	4.300	3.000	0.270	7.930	17080
8	31	1992	28.40	18.20	17.00	39.000	56.000	3.200	2.300	0.440	5.700	17096
9	14	1992	33.00	8.40	41.00	67.000	65.000	4.600	3.200	1.850	7.430	17100
9	28	1992	34.60	9.20	43.00	76.000	69.000	4.900	3.400	2.070	7.520	17117
10	12	1992	51.00	15.30	43.00	82.000	87.000	6.300	4.500	0.420	10.800	17121
10	27	1992	53.40	11.40	57.00	97.000	119.000	7.900	5.500	1.460	12.900	17137
11	9	1992	.	.	.	.	.	.	.	.	.	.
11	30	1992	33.00	7.60	41.00	50.000	64.000	4.800	3.300	0.920	8.190	17152
4	8	1993	16.50	4.50	48.00	35.000	52.000	2.860	1.900	0.430	4.710	17164
4	22	1993	21.50	5.50	58.00	57.000	64.000	4.210	2.270	1.050	6.150	17175
5	6	1993	18.30	5.30	48.00	45.000	49.000	3.230	1.780	1.270	4.590	17180
5	20	1993	32.00	6.50	77.00	95.000	98.000	6.300	4.100	1.560	8.970	17195
6	3	1993	23.30	5.00	55.00	58.000	66.000	4.130	2.550	1.100	5.710	17199
6	17	1993	25.50	5.30	59.00	76.000	80.000	5.160	3.170	1.330	7.150	17215
7	1	1993	24.80	5.50	63.00	77.000	78.000	4.960	3.070	1.510	7.040	17219
7	14	1993	28.40	7.00	58.00	77.000	84.000	5.040	3.220	1.720	7.580	17236
7	29	1993	31.80	7.40	59.00	79.000	89.000	5.600	3.600	1.580	8.160	17240
8	12	1993	31.60	7.20	66.00	75.000	94.000	5.100	3.830	1.550	8.830	17256
8	26	1993	44.40	9.20	83.00	92.000	123.000	7.250	5.580	1.830	13.000	17261
9	9	1993	28.00	7.20	54.00	62.000	76.000	4.740	3.370	1.270	7.850	17276
9	23	1993	35.80	8.20	69.00	72.000	100.000	5.970	4.470	1.460	10.200	17280
10	7	1993	38.80	8.80	76.00	77.000	105.000	6.200	4.210	1.680	9.710	17296
10	20	1993	40.60	8.80	76.00	83.000	114.000	6.200	4.280	1.380	10.000	17300
11	4	1993	26.40	6.00	52.00	50.000	69.000	4.420	2.780	0.750	6.350	17318
11	22	1993	39.00	8.00	63.00	72.000	101.000	6.700	4.610	1.000	10.000	17332

Table A2.5.18. Drainage quality data for additional parameters from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 1 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
4	26	1978	.	.	.	.	.	6.0	.
5	25	1978	.	.	.	.	.	.	.
6	16	1978	.	.	.	.	.	13.0	.
7	5	1978	78.00	.	.	.	.	13.0	.
7	13	1978	50.00	39.00	13.00	0.08	.	.	.
7	19	1978	59.00	54.00	12.00	0.08	.	.	.
8	2	1978	25.00	66.50	5.70	0.24	.	21.0	.
8	4	1978	44.00	83.00	7.50	0.08	.	22.0	.
8	10	1978	52.00	.	8.50	0.08	.	22.0	.
8	16	1978	19.50	41.00	4.00	0.08	.	17.7	.
8	17	1978	27.00	55.00	.	0.08	.	.	.
8	18	1978	34.00	67.00	.	0.08	.	16.0	.
8	23	1978	16.50	34.50	4.00	0.08	.	.	.
8	25	1978	22.00	54.00	8.30	0.08	.	21.0	.
8	28	1978	30.00	64.00	.	0.08	.	.	.
9	11	1978	8.00	.	.	0.20	.	17.0	.
9	12	1978	9.00	.	.	0.08	.	.	.
9	13	1978	15.00	46.00	0.10	.	.	13.0	.
10	17	1978	.	.	.	.	.	.	.
3	23	1979	11.00	.	.	.	7.60	.	.
4	3	1979	18.00	.	.	.	.	.	.
4	11	1979	22.00	.	.	.	.	.	.
4	16	1979	14.00	.	.	.	.	.	.
4	17	1979	8.00	9.80	.	.	.	.	.
4	18	1979	5.00	8.10	.	.	.	.	.
4	19	1979	2.00	5.50	0.40	.	.	.	.
4	20	1979	4.00	8.50	0.64	.	3.30	.	.
4	30	1979	13.00	26.00	1.10	.	.	.	.
5	11	1979	.	.	.	.	.	.	.
5	22	1979	9.00	.	.	.	6.90	.	.
5	23	1979	12.00	.	.	.	5.40	.	.
5	27	1979	12.00	.	.	.	3.70	.	.
6	1	1979	10.00	.	.	.	3.90	.	.
6	6	1979	12.00	.	.	.	.	.	.
6	10	1979	13.00	.	.	.	.	.	.
6	16	1979	11.00	.	.	.	.	23.0	.
6	20	1979	7.00	.	.	.	.	19.0	.
6	30	1979	19.00	.	.	.	4.80	14.0	.
7	10	1979	28.00	.	.	.	5.40	19.0	.
7	30	1979	6.00	.	.	.	6.50	22.0	.
8	4	1979	13.00	.	.	.	5.60	23.0	.
8	13	1979	82.00	.	.	.	8.70	14.0	.
8	23	1979	9.00	.	.	.	8.70	12.0	.
8	28	1979	15.00	.	.	.	8.30	20.0	.
9	1	1979	17.00	.	.	.	6.20	21.0	.
9	5	1979	39.00	.	.	.	8.60	19.0	.
9	13	1979	14.00	.	.	.	8.30	17.0	.
9	13	1979	10.00	.	.	.	7.70	17.0	.
9	19	1979	51.00	.	.	.	11.00	12.0	.
10	18	1979	9.00	.	.	.	10.00	.	.
10	19	1979	3.00	.	.	.	5.40	.	.
10	31	1979	6.00	.	.	.	7.00	.	.
11	1	1979	8.00	.	.	.	6.20	.	.
3	29	1980	7.00	.	.	.	4.90	16.0	.
3	31	1980	3.00	.	.	.	2.50	.	.
4	5	1980	2.50	.	.	.	.	.	.
4	14	1980	4.00	.	.	.	.	.	.
4	18	1980	4.00	.	.	.	3.50	.	.
4	27	1980	7.00	.	.	.	.	13.0	.
5	10	1980	10.00	.	.	.	7.90	12.0	.
5	18	1980	13.00	.	.	.	.	19.0	.
5	31	1980	15.00	.	.	.	4.20	.	.
6	3	1980	15.00	.	.	.	7.30	.	.
6	7	1980	16.00	.	.	.	.	.	.
6	13	1980	12.00	.	.	.	.	.	.
6	19	1980	12.00	.	.	.	.	.	.
6	28	1980	10.00	.	.	.	9.20	.	.

Table A2.5.18. Drainage quality data for additional parameters from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 2 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
6	30	1980	14.00	.	.	.	17.00	.	.
7	2	1980	9.00	.	.	.	.	.	.
7	13	1980	15.00	.	.	.	.	.	.
7	23	1980	16.00	.	.	.	.	.	.
8	3	1980	.	.	.	.	9.10	.	.
8	12	1980	12.00	.	.	.	9.30	.	.
8	17	1980	13.00	.	.	.	.	.	.
8	22	1980	13.00	.	.	.	.	.	.
8	29	1980	19.00	.	.	.	8.20	.	.
9	8	1980	15.00	.	.	.	.	.	.
9	9	1980	13.00	.	.	.	.	14.0	.
9	12	1980	11.00	.	.	.	.	18.0	.
9	18	1980	16.00	.	.	.	.	14.0	.
9	25	1980	22.00	.	.	.	11.00	14.0	.
10	23	1980	19.00	.	.	.	9.40	10.0	.
10	31	1980	16.00	.	.	.	.	11.0	.
2	19	1981	17.00	.	.	.	.	8.0	.
2	21	1981	14.00	.	.	.	.	.	.
3	29	1981	.	.	.	.	4.90	.	.
4	3	1981	.	.	.	.	.	.	.
4	13	1981	.	.	.	.	.	.	.
4	17	1981	.	.	.	.	7.30	.	.
4	23	1981	.	.	.	.	.	.	.
4	25	1981	.	.	.	.	.	.	.
4	28	1981	.	.	.	.	7.00	.	.
4	29	1981	.	.	.	.	.	9.0	.
5	12	1981	.	.	.	.	12.00	18.0	.
5	28	1981	.	.	.	.	.	16.0	.
6	6	1981	.	.	.	.	9.60	19.0	.
6	22	1981	.	.	.	.	.	16.0	.
6	24	1981	.	.	.	.	.	17.0	.
6	28	1981	.	.	.	.	6.60	15.0	.
7	4	1981	.	.	.	.	.	22.0	.
7	24	1981	33.00	.	.	.	7.90	21.0	.
8	4	1981	.	.	.	.	.	18.0	.
8	17	1981	.	.	.	.	.	22.0	.
8	31	1981	.	.	.	.	9.80	17.0	.
9	7	1981	.	.	.	.	.	.	.
9	16	1981	.	.	.	.	.	.	.
9	26	1981	.	.	.	.	8.00	.	.
10	1	1981	.	.	.	.	.	.	.
10	4	1981	.	.	.	.	.	.	.
10	14	1981	.	.	.	.	5.60	.	.
10	18	1981	.	.	.	.	.	.	.
11	6	1981	.	.	.	.	6.60	.	.
3	30	1982	.	.	.	.	.	.	.
4	6	1982	.	.	.	.	.	.	.
4	13	1982	.	.	.	.	.	.	.
4	15	1982	.	.	.	.	.	.	.
4	19	1982	.	.	.	.	.	.	.
4	23	1982	.	.	.	.	.	.	.
5	3	1982	.	.	.	.	.	.	.
5	4	1982	.	.	.	.	.	.	.
5	7	1982	.	.	.	.	.	.	.
5	10	1982	.	.	.	.	.	.	.
5	11	1982	.	.	.	.	.	.	.
5	14	1982	.	.	.	.	.	.	.
5	21	1982	.	.	.	.	.	.	.
6	2	1982	.	.	.	.	.	.	.
6	18	1982	.	.	.	.	.	.	.
7	2	1982	.	.	.	.	.	.	.
7	26	1982	.	.	.	.	.	.	.
8	30	1982	.	.	.	.	.	.	.
10	4	1982	.	.	.	.	.	.	.
11	1	1982	.	.	.	.	.	.	.
3	18	1983	.	.	.	.	.	.	.
4	14	1983	.	.	.	.	.	.	.

Table A2.5.18. Drainage quality data for additional parameters from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 3 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
5	15	1983	.	.	.	.	.	.	.
6	16	1983	.	.	.	.	.	.	.
7	4	1983	.	.	.	.	.	.	.
7	11	1983	.	.	.	.	.	.	.
7	17	1983	.	.	.	.	.	.	.
7	20	1983	.	.	.	.	.	.	.
7	25	1983	.	.	.	.	.	.	.
7	31	1983	.	.	.	.	.	.	.
8	7	1983	.	.	.	.	.	.	.
8	15	1983	.	.	.	.	.	.	.
8	22	1983	.	.	.	.	.	.	.
8	28	1983	.	.	.	.	.	.	.
9	4	1983	.	.	.	.	.	.	.
9	11	1983	.	.	.	.	.	.	.
9	18	1983	.	.	.	.	.	.	.
9	25	1983	.	.	.	.	.	.	.
10	2	1983	.	.	.	.	.	.	.
10	8	1983	.	.	.	.	.	.	.
10	15	1983	.	.	.	.	.	.	.
10	23	1983	.	.	.	.	.	.	.
10	30	1983	.	.	.	.	.	.	.
11	6	1983	.	.	.	.	.	.	.
11	13	1983	.	.	.	.	.	.	.
11	20	1983	.	.	.	.	.	.	.
12	1	1983	.	.	.	.	.	.	.
2	24	1984	.	.	.	.	.	.	500
3	15	1984	.	.	.	.	.	.	501
4	15	1984	.	.	.	.	.	.	502
5	16	1984	.	.	.	.	.	.	503
6	13	1984	.	.	.	.	.	.	504
7	14	1984	.	.	.	.	.	.	505
8	15	1984	.	.	.	.	.	.	506
9	12	1984	.	.	.	.	.	.	507
10	13	1984	.	.	.	.	.	.	508
11	14	1984	.	.	.	.	.	.	509
12	5	1984	.	.	.	.	.	.	510
12	19	1984	.	.	.	.	.	.	511
4	2	1985	.	.	.	.	.	.	500
4	16	1985	.	.	.	.	.	.	501
5	14	1985	.	.	.	.	.	.	502
6	14	1985	.	.	.	.	.	.	503
7	15	1985	.	.	.	.	.	.	504
8	13	1985	.	.	.	.	.	.	505
9	13	1985	.	.	.	.	.	.	506
10	15	1985	.	.	.	.	.	.	507
11	15	1985	.	.	.	.	.	.	508
12	10	1985	.	.	.	.	.	.	0
3	23	1986	.	.	.	.	.	.	500
4	16	1986	.	.	.	.	.	.	501
5	14	1986	.	.	.	.	.	.	502
6	11	1986	.	.	.	.	.	.	503
7	12	1986	.	.	.	.	.	.	504
8	13	1986	.	.	.	.	.	.	505
9	14	1986	.	.	.	.	.	.	506
10	16	1986	.	.	.	.	.	.	507
11	16	1986	.	.	.	.	.	.	508
3	28	1987	.	.	.	.	.	.	500
4	18	1987	.	.	.	.	.	.	501
5	16	1987	.	.	.	.	.	.	502
6	16	1987	.	.	.	.	.	.	503
7	15	1987	.	.	.	.	.	.	504
8	15	1987	.	.	.	.	.	.	505
9	16	1987	.	.	.	.	.	.	506
10	13	1987	.	.	.	.	.	.	507
11	10	1987	.	.	.	.	.	.	508
5	12	1988	.	.	.	.	.	.	502
6	12	1988	.	.	.	.	.	.	503

Table A2.5.18. Drainage quality data for additional parameters from Duluth Complex test pile FL5, from 1978 to 1993 (1.41% S, 0.30% Cu, 0.08% Ni), page 4 of 4.

Month	Day	Year	Cl mg/L	NO_3 mg/L	NH_3 mg/L	P mg/L	DOC mg/L	Temp °C	Sample Number
7	15	1988	.	.	.	.	.	.	504
8	16	1988	.	.	.	.	.	.	505
9	16	1988	.	.	.	.	.	.	506
9	24	1988	.	.	.	.	.	.	500
10	18	1988	.	.	.	.	.	.	507
10	24	1988	.	.	.	.	.	.	501
11	15	1988	.	.	.	.	.	.	508
11	30	1988	.	.	.	.	.	.	509
4	7	1989	.	.	.	.	.	.	500
4	24	1989	.	.	.	.	.	.	501
5	16	1989	.	.	.	.	.	.	502
6	17	1989	.	.	.	.	.	.	503
7	17	1989	.	.	.	.	.	.	504
8	14	1989	.	.	.	.	.	.	505
9	14	1989	.	.	.	.	.	.	506
10	16	1989	.	.	.	.	.	.	507
11	13	1989	.	.	.	.	.	.	508
3	26	1990	.	.	.	.	.	.	500
4	12	1990	.	.	.	.	.	.	501
5	17	1990	.	.	.	.	.	.	502
6	18	1990	.	.	.	.	.	.	503
7	16	1990	.	.	.	.	.	.	504
8	13	1990	.	.	.	.	.	.	505
9	10	1990	.	.	.	.	.	.	506
10	15	1990	.	.	.	.	.	.	507
11	16	1990	.	.	.	.	.	.	508
4	8	1991	.	.	.	.	.	.	.
4	22	1991	.	.	.	.	.	.	2021
5	20	1991	.	.	.	.	.	.	2025
6	17	1991	.	.	.	.	.	.	2041
7	15	1991	.	.	.	.	.	.	2057
8	12	1991	.	.	.	.	.	.	2073
9	9	1991	.	.	.	.	.	.	2088
10	7	1991	.	.	.	.	.	.	2104
11	7	1991	.	.	.	.	.	.	2120
4	23	1992	.	.	.	.	.	.	17002
5	11	1992	.	.	.	.	.	.	17015
5	26	1992	.	.	.	.	.	.	17043
6	8	1992	.	.	.	.	.	.	17047
6	22	1992	.	.	.	.	.	.	17051
7	7	1992	.	.	.	.	.	.	17056
7	20	1992	.	.	.	.	.	.	17060
8	3	1992	.	.	.	.	.	.	17076
8	17	1992	.	.	.	.	.	.	17080
8	31	1992	.	.	.	.	.	.	17096
9	14	1992	.	.	.	.	.	.	17100
9	28	1992	.	.	.	.	.	.	17117
10	12	1992	.	.	.	.	.	.	17121
10	27	1992	.	.	.	.	.	.	17137
11	9	1992	.	.	.	.	.	.	.
11	30	1992	.	.	.	.	.	.	17152
4	8	1993	.	.	.	.	.	.	17164
4	22	1993	.	.	.	.	.	.	17175
5	6	1993	.	.	.	.	.	.	17180
5	20	1993	.	.	.	.	.	.	17195
6	3	1993	.	.	.	.	.	.	17199
6	17	1993	.	.	.	.	.	.	17215
7	1	1993	.	.	.	.	.	.	17219
7	14	1993	.	.	.	.	.	.	17236
7	29	1993	.	.	.	.	.	.	17240
8	12	1993	.	.	.	.	.	.	17256
8	26	1993	.	.	.	.	.	.	17261
9	9	1993	.	.	.	.	.	.	17276
9	23	1993	.	.	.	.	.	.	17280
10	7	1993	.	.	.	.	.	.	17296
10	20	1993	.	.	.	.	.	.	17300
11	4	1993	.	.	.	.	.	.	17318
11	22	1993	.	.	.	.	.	.	17332

Figure A2.6.1. pH and sulfate versus time for FL1 (0.6 %S): 1978-1993.

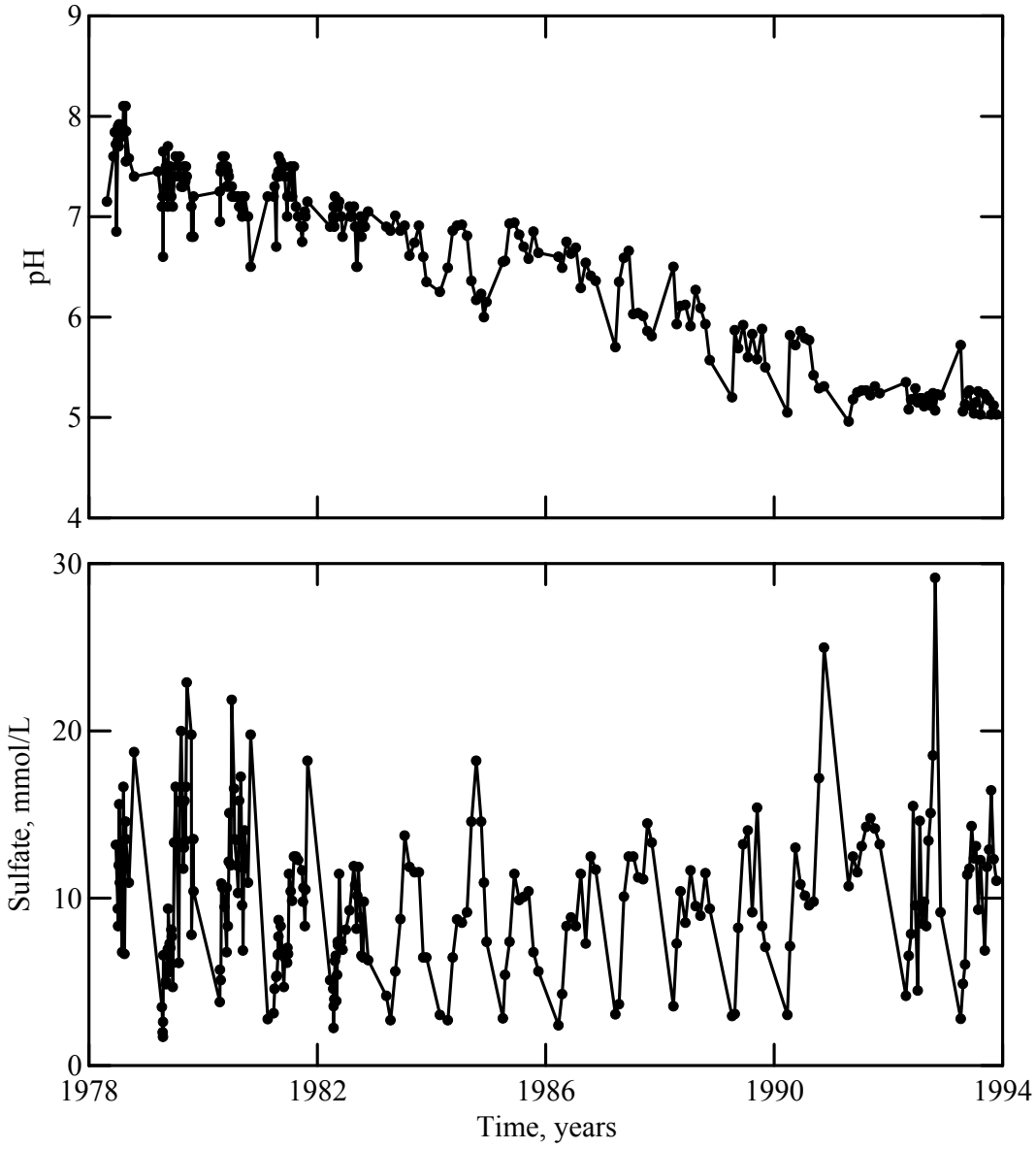


Figure A2.6.2. Calcium, magnesium, and sodium versus time for FL1 (0.6 %S): 1978-1993.

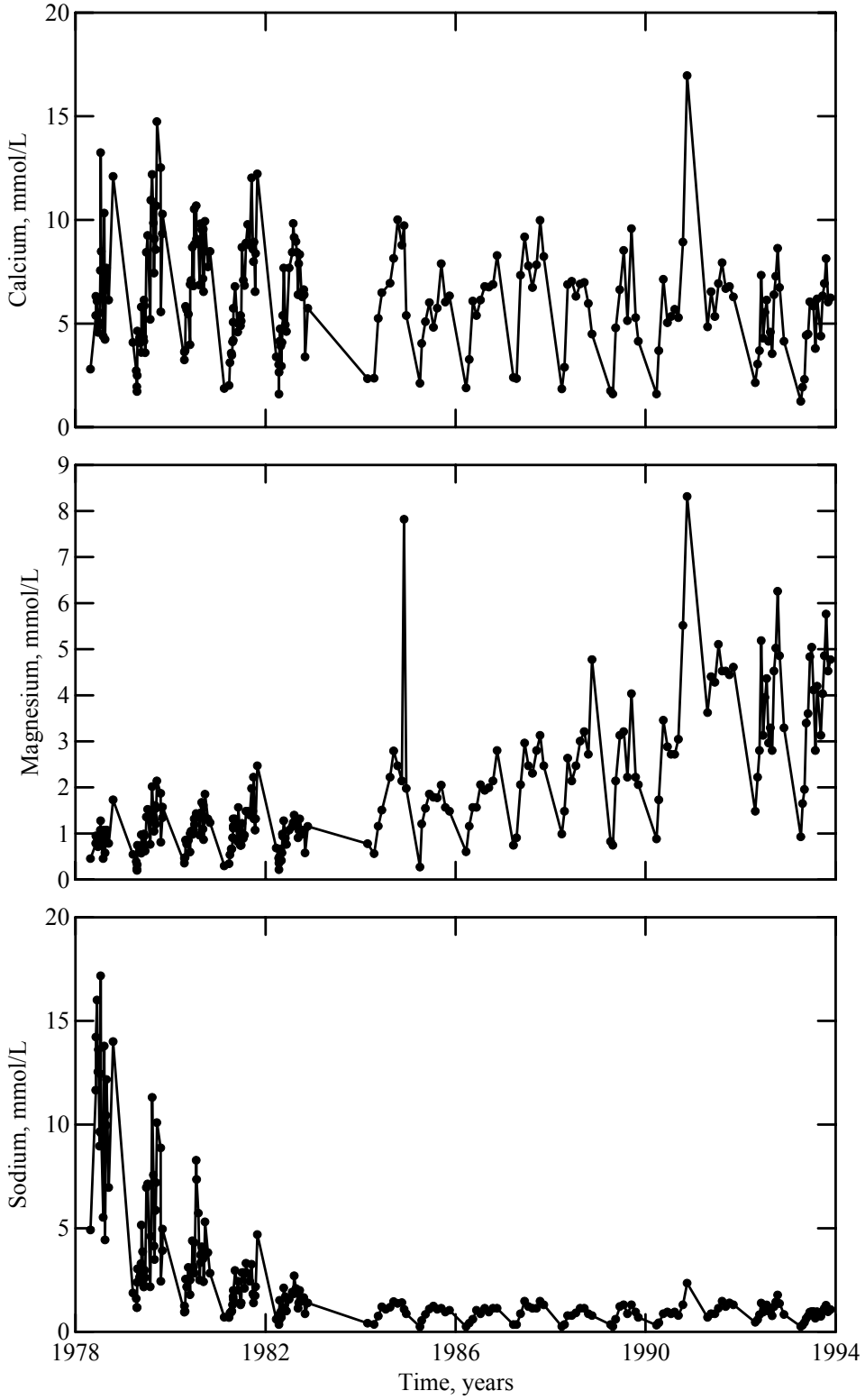


Figure A2.6.3. pH and sulfate versus time for FL2 (0.6 %S): 1978-1993.

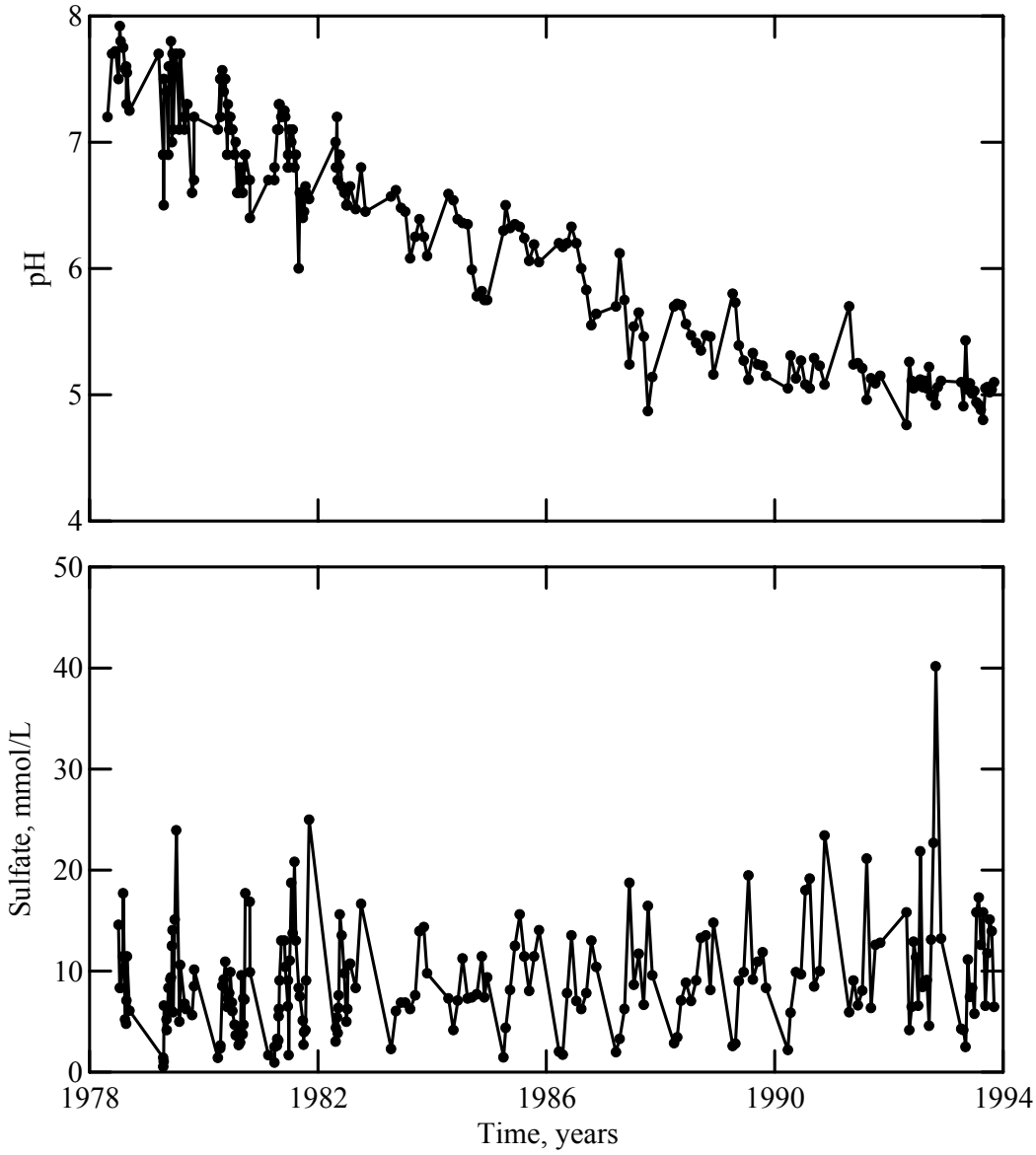


Figure A2.6.4. Calcium, magnesium, and sodium versus time for FL2 (0.6 %S): 1978-1993.

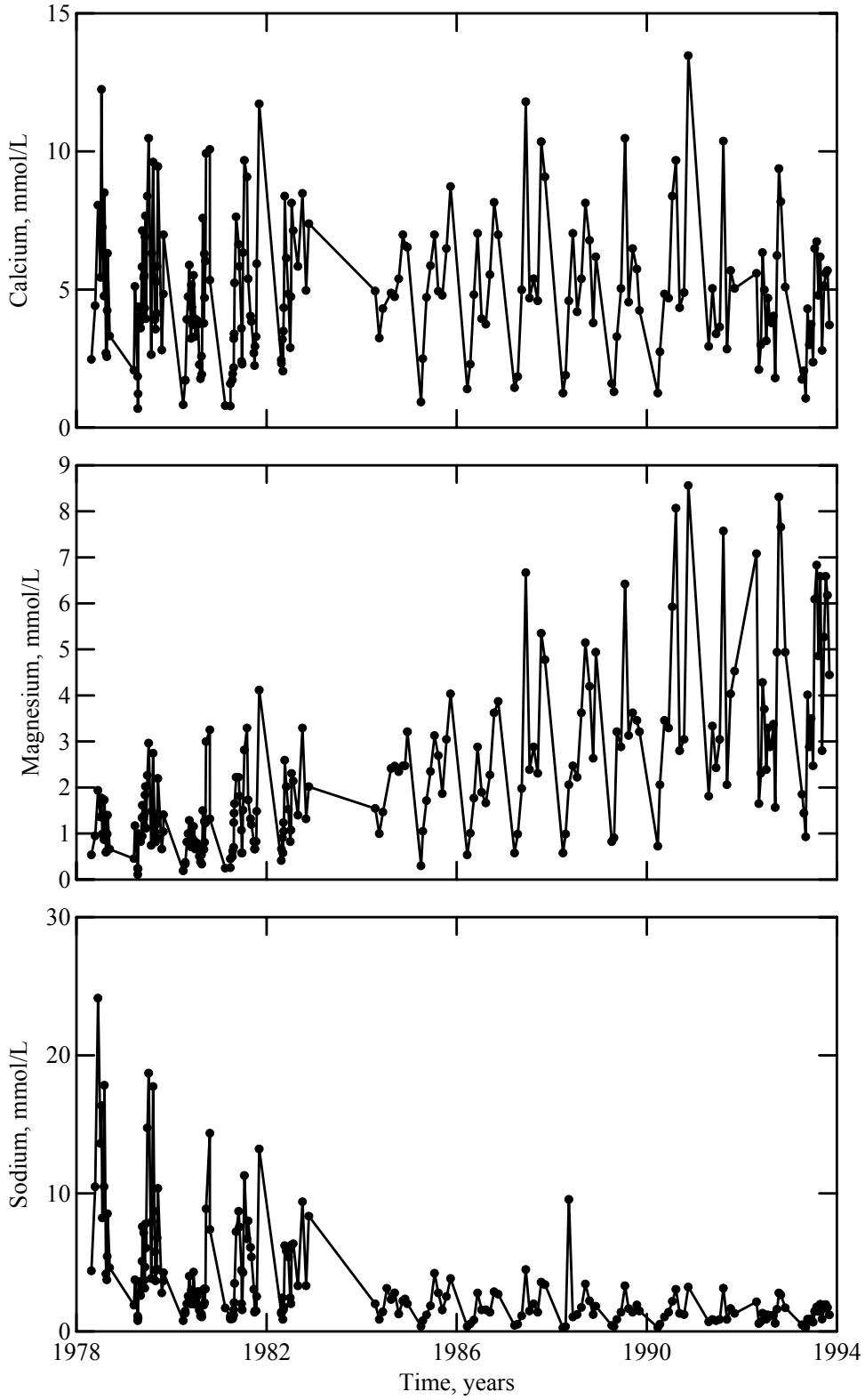


Figure A2.6.5. pH and sulfate versus time for FL3 (0.6 %S): 1978-1989.

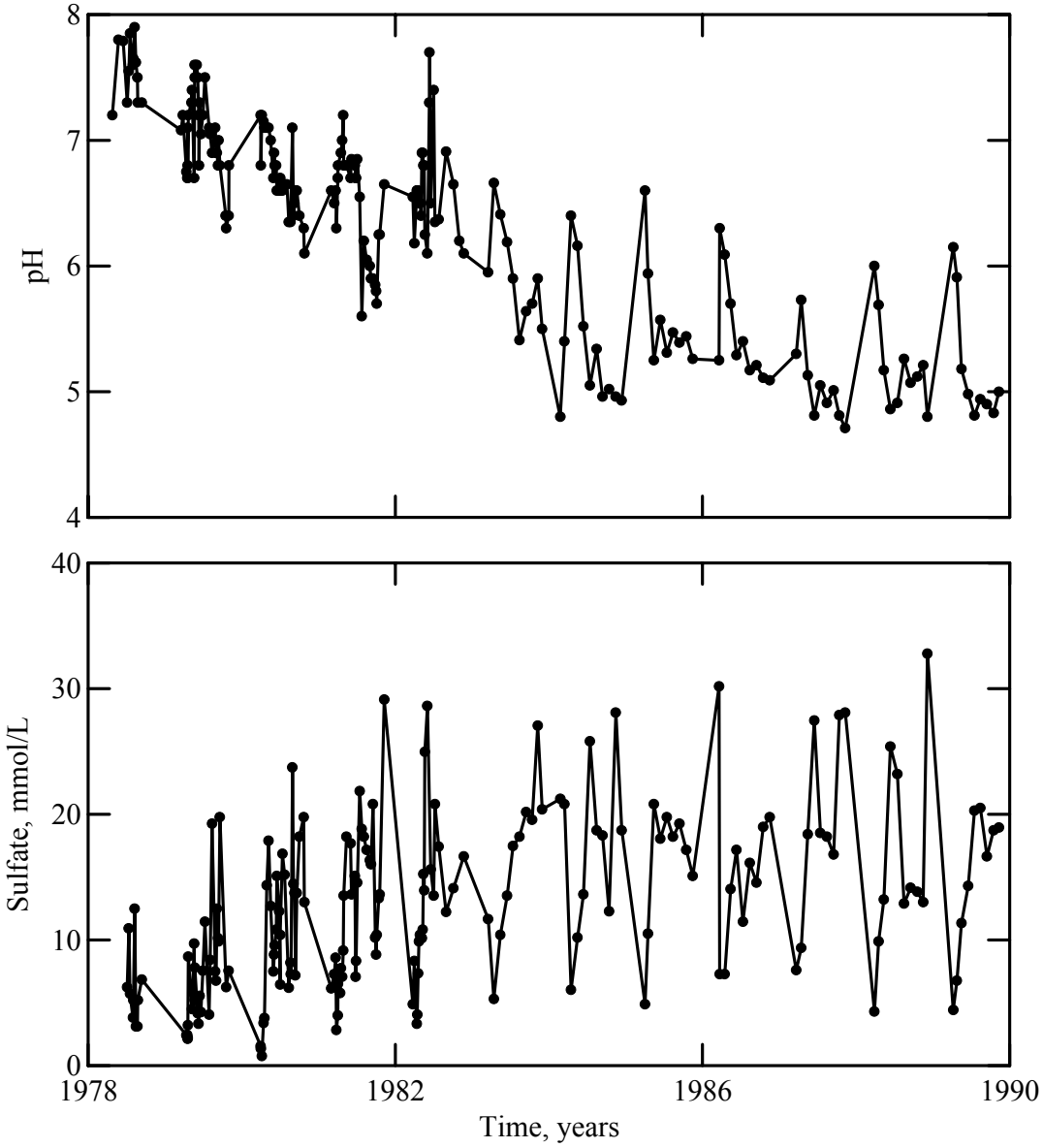


Figure A2.6.6. Calcium, magnesium, and sodium versus time for FL3 (0.6 %S): 1978-1989.

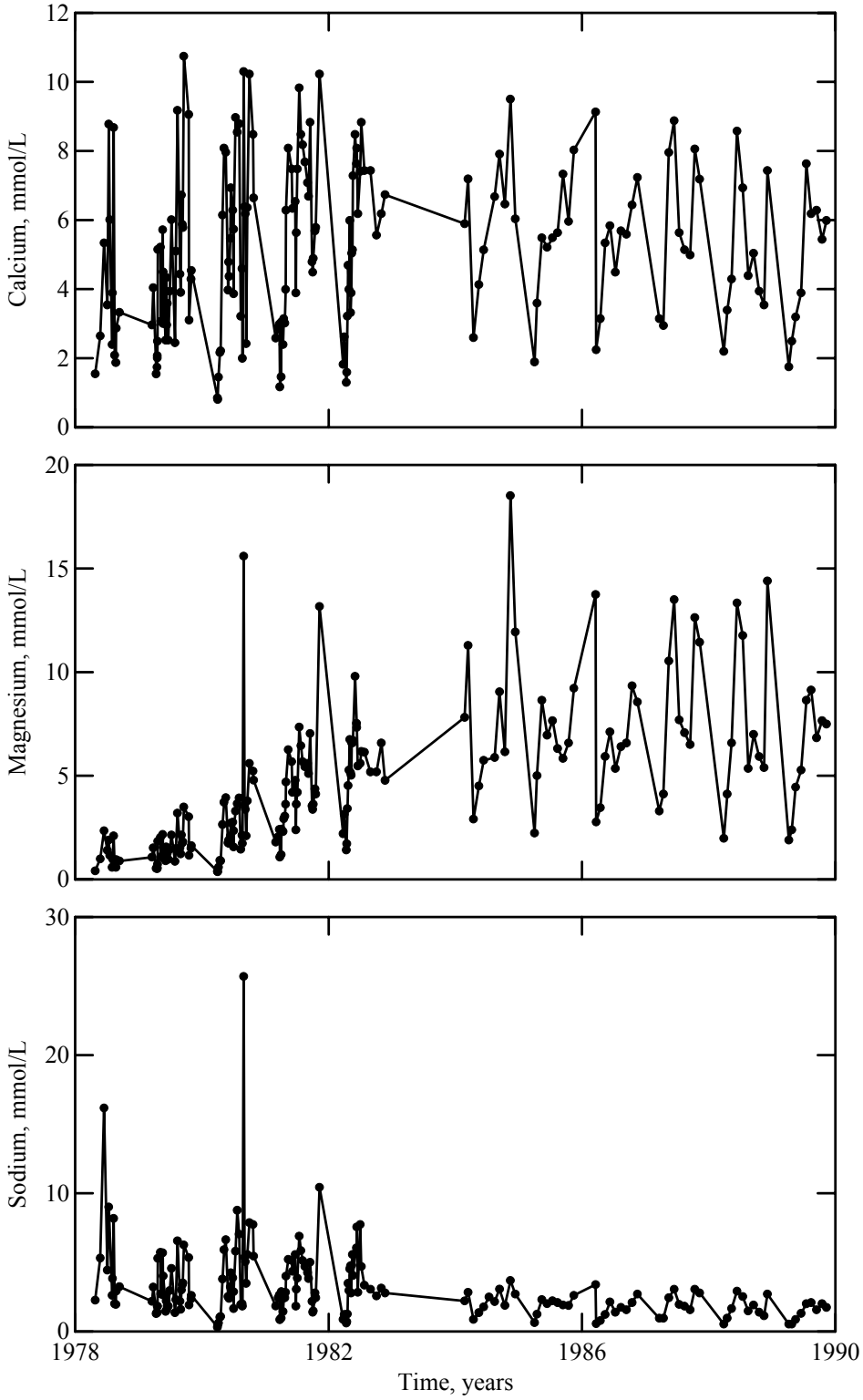


Figure A2.6.7. pH and sulfate versus time for FL4 (0.6 %S): 1978-1989.

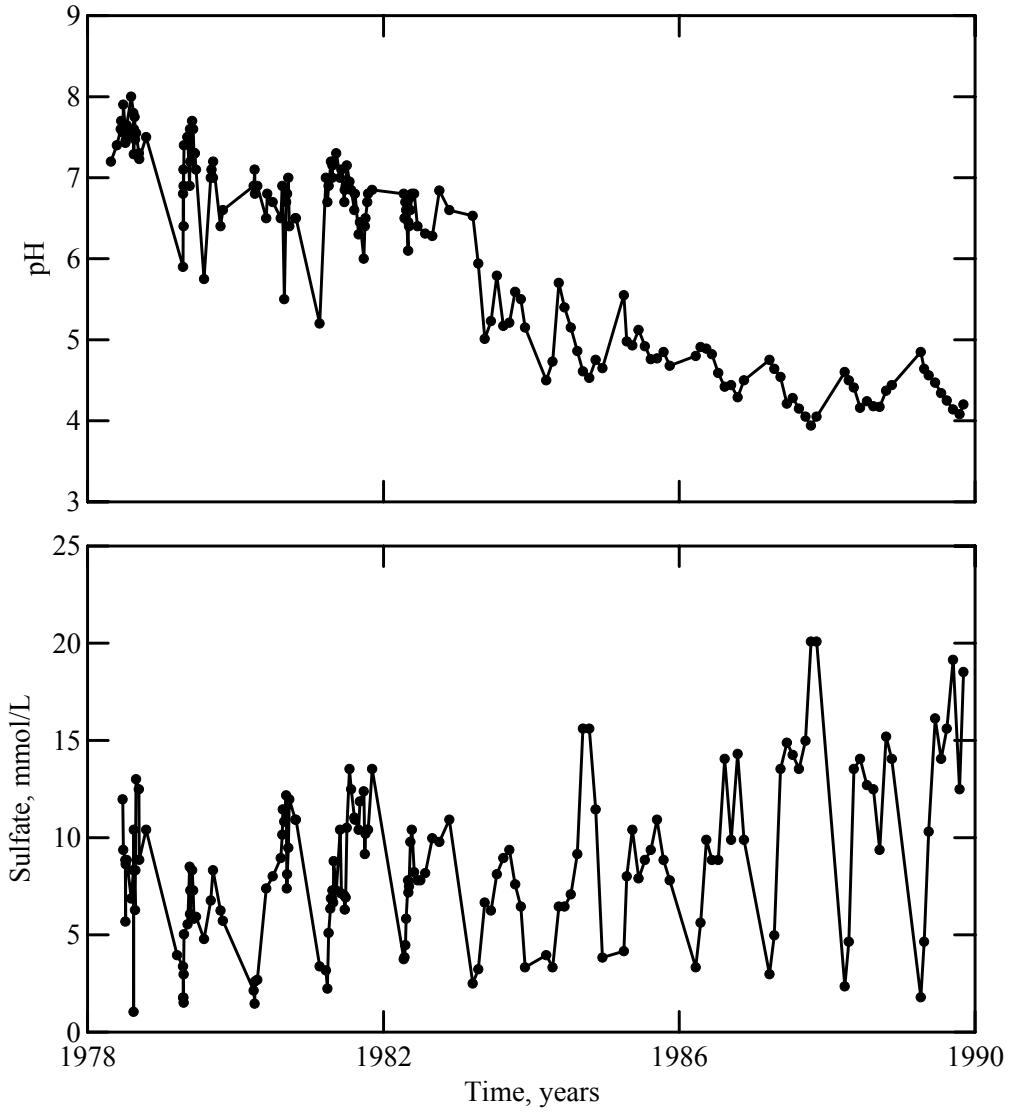


Figure A2.6.8. Calcium, magnesium, and sodium versus time for FL4 (0.6 %S): 1978-1989.

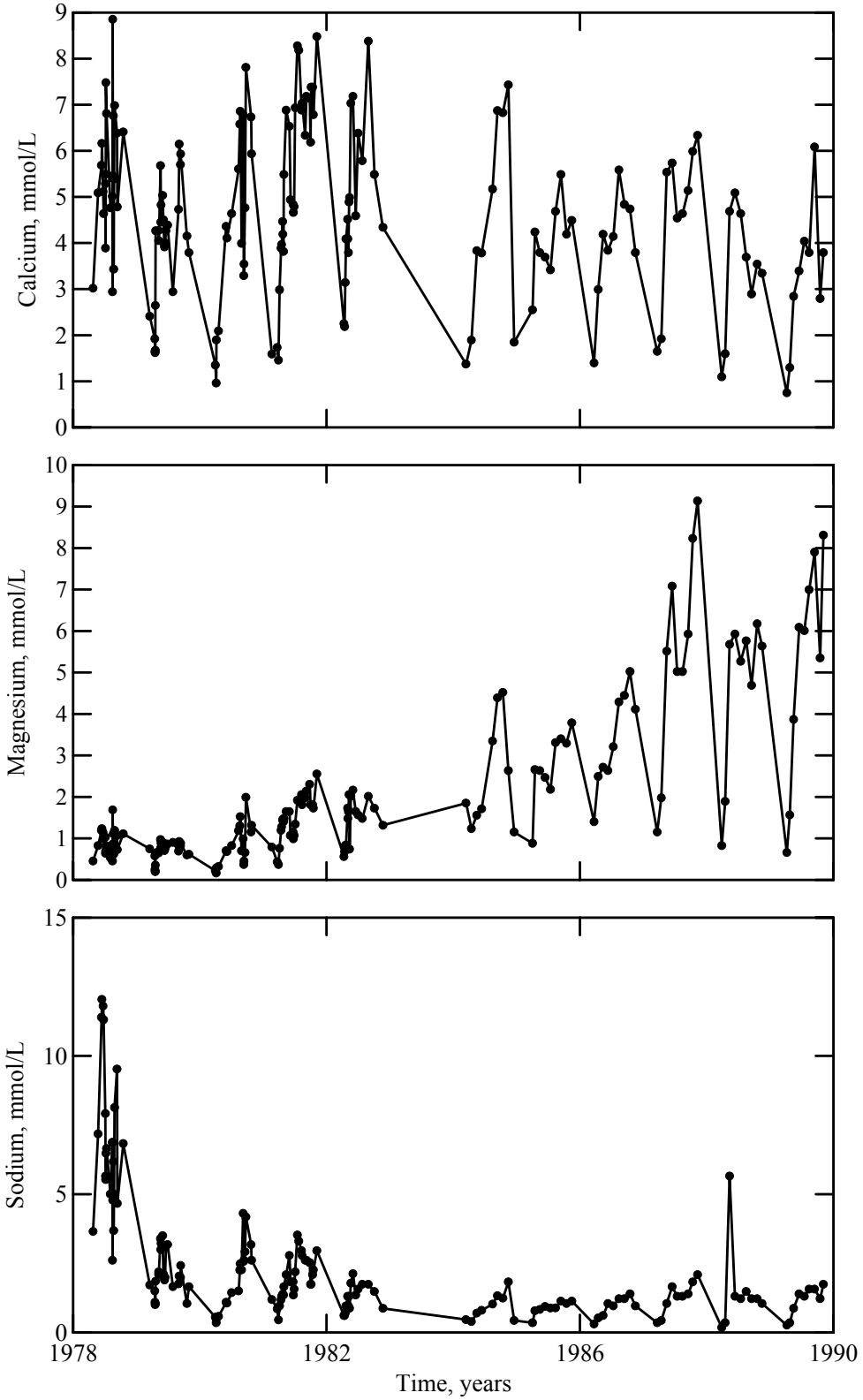


Figure A2.6.9. pH and sulfate versus time for FL6 (0.8 %S): 1978-1993.

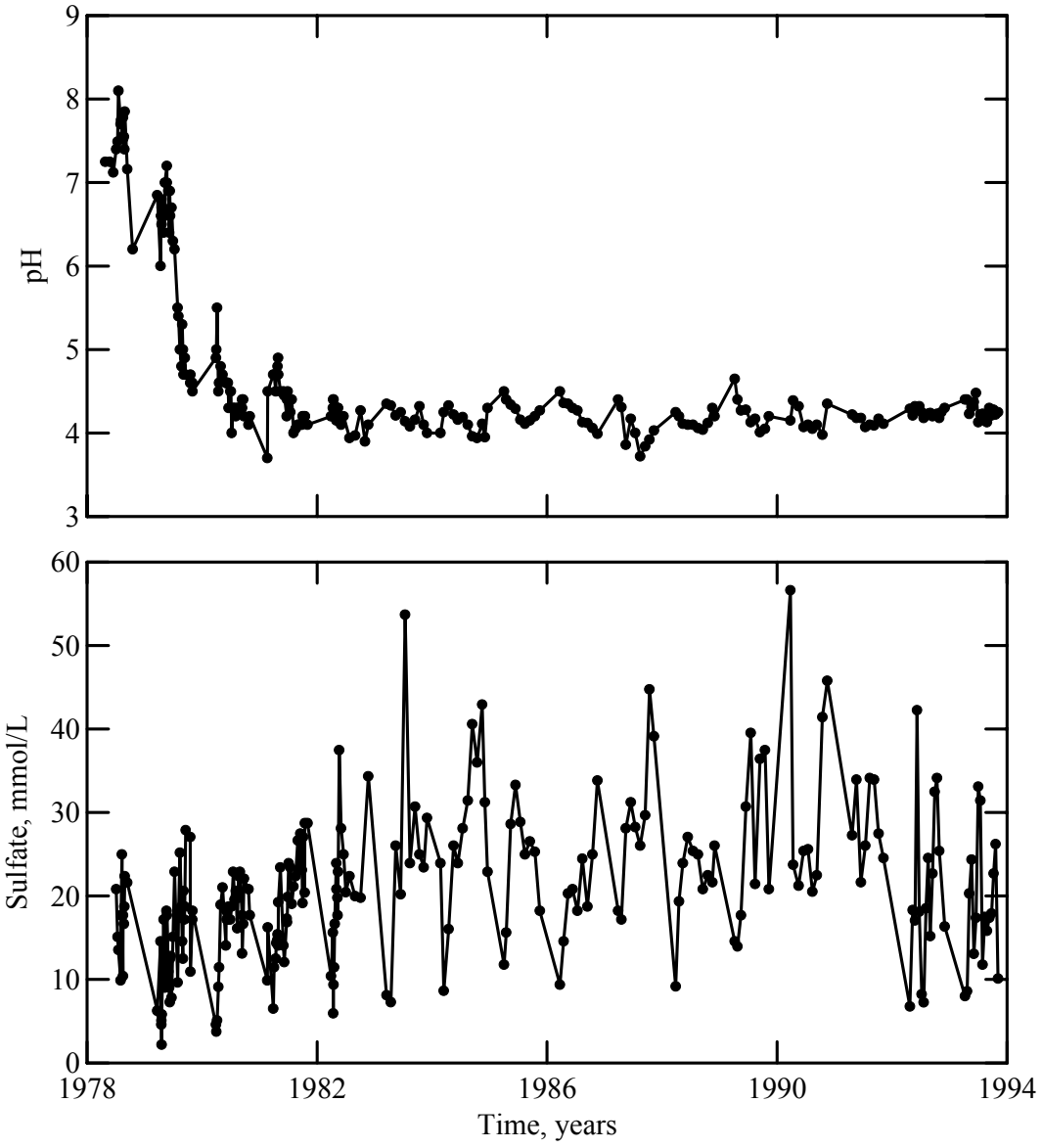


Figure A2.6.10. Calcium, magnesium, and sodium versus time for FL6 (0.8 %S): 1978-1993.

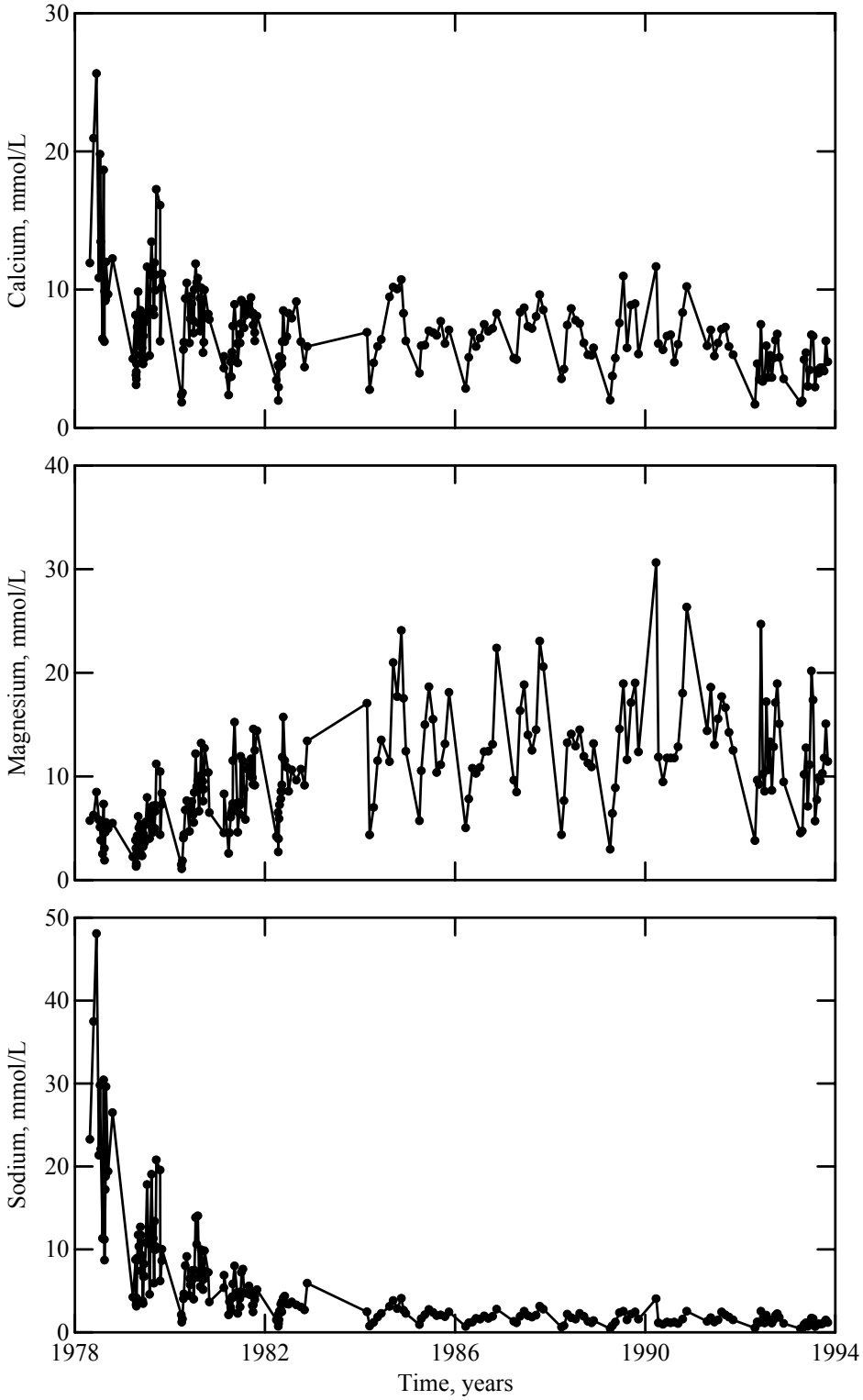


Figure A2.6.11. pH and sulfate versus time for FL5 (1.4 %S): 1978-1993.

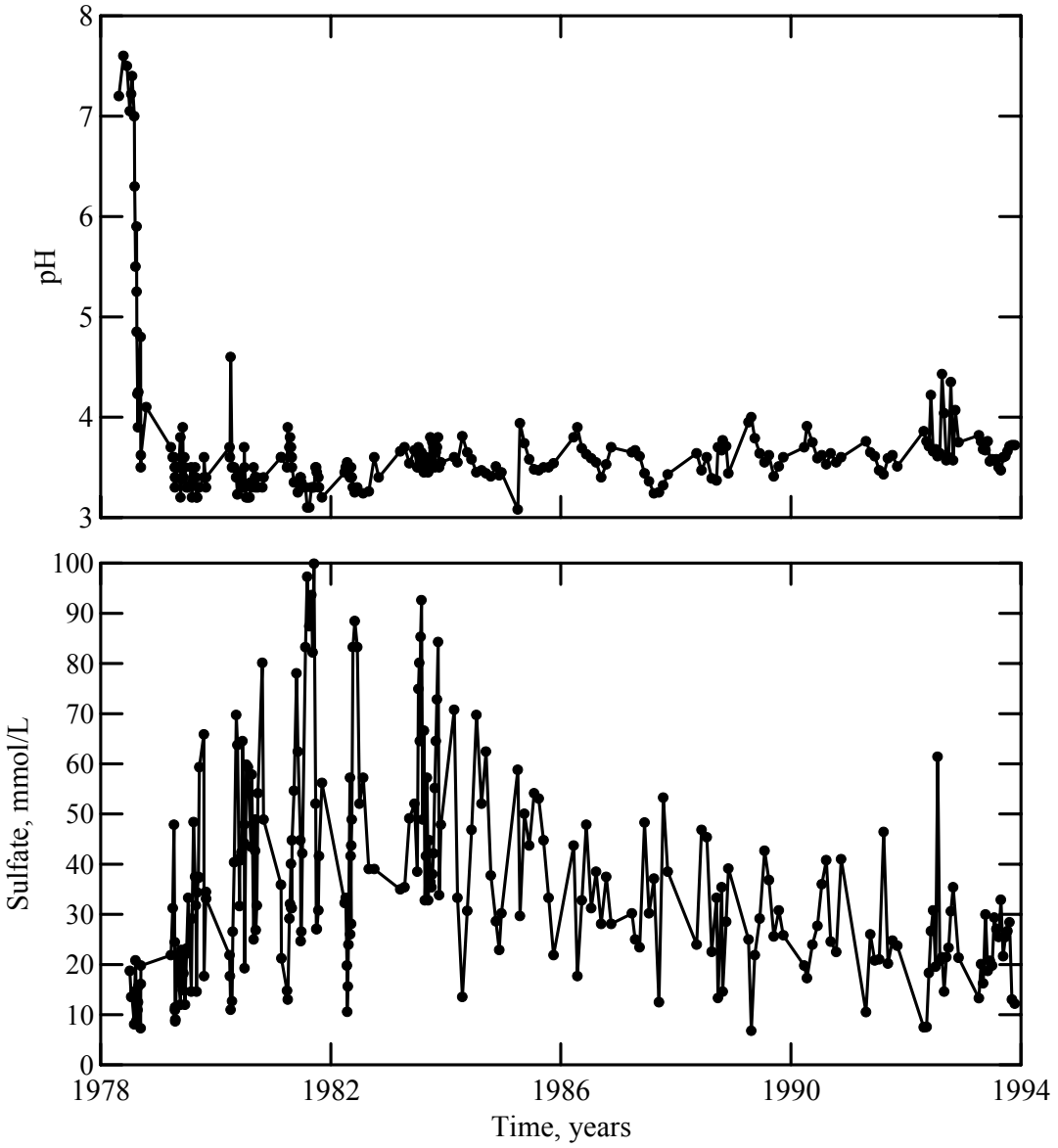


Figure A2.6.12. Calcium, magnesium, and sodium versus time for FL5 (1.4 %S): 1978-1993.

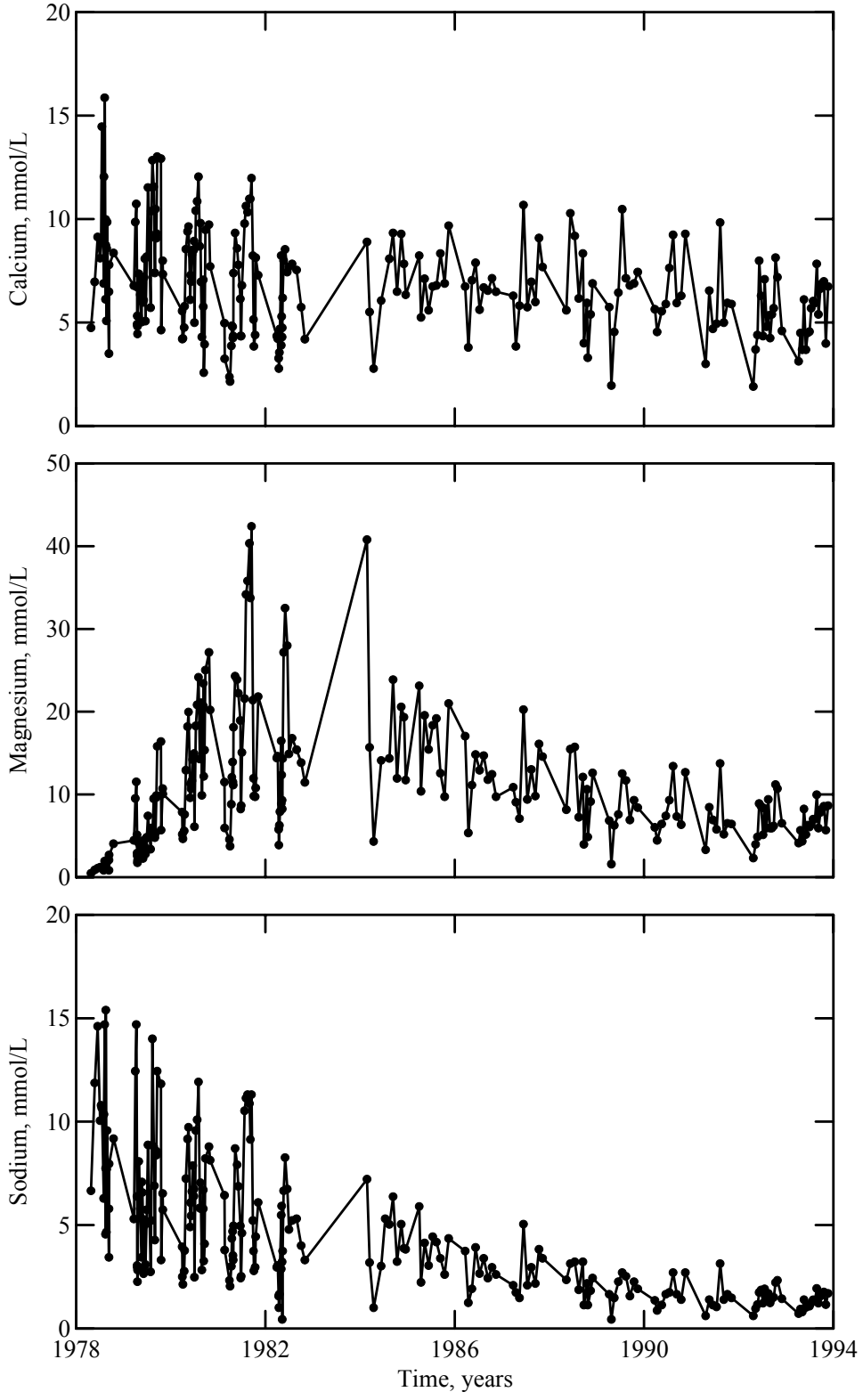


Table A2.7.1. Annual mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL1 (0.63% S, 1100 T, no cover[1]). Percent of total original sulfur released = 7.11.

Year	Total Volume	Period	Median pH	Annual Mass Release						
				SO_4	Ca[2]	Mg[2]	Cu	Ni	Co	Zn
	L	days					mol			
1978	83100	206	7.80	821	416	58	0.034	0.314	0.022	0.012
1979	86600	252	7.35	670	483	71	0.041	0.485	0.034	0.034
1980	53800	256	7.20	673	426	66	0.036	0.644	0.041	0.039
1981	83800	275	7.20	673	499	93	0.120	2.759	0.132	0.124
1982	118600	258	7.00	932	755	113	0.212	8.740	0.378	0.173
1983	82400	277	6.86	852	637	127	0.320	5.674	0.239	0.161
1984	66879	298	6.36	640	439	113	1.168	17.090	0.760	0.380
1985	133396	227	6.70	1221	793	231	2.794	32.647	1.176	0.760
1986	119551	238	6.54	900	658	202	5.348	37.165	1.416	0.745
1987	104768	233	6.03	1075	725	229	8.303	40.233	1.412	1.038
1988	106298	247	6.09	972	660	300	11.175	67.878	2.240	1.737
1989	122463	213	5.69	1230	693	308	15.654	73.106	3.303	2.248
1990	60101	235	5.72	784	411	233	18.369	57.954	2.819	1.927
1991	120980	213	5.25	1577	772	552	44.187	139.946	5.572	3.099
1992	103955	221	5.19	1037	515	371	30.980	86.056	3.629	1.999
1993	124968	228	5.14	1327	620	454	37.879	111.247	4.157	2.092
Total	1571659	3877		15384	9502	3521	182.932	708.313	28.376	17.223

[1] FL1 was partially covered with plastic during 1990.

[2] Calcium and magnesium were not measured in 1983. Calculated as mean of ratio of 1982 release to sulfate release and ratio of 1984 release to sulfate release (example $(Ca_{1982}/SO_{4\ 1982} + Ca_{1984}/SO_{4\ 1984})/2$). Sulfate for 1983 was then multiplied by the mean.

Table A2.7.2. Annual mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL2 (0.63% S, 1100 T, cover of topsoil). Percent of total original sulfur released = 4.09.

Year	Total Volume	Period	Median pH	Annual Mass Release						
				SO_4	Ca[1]	Mg[1]	Cu	Ni	Co	Zn
	L	days					mol			
1978	70200	193	7.60	503	260	57	0.136	0.259	0.025	0.040
1979	75600	253	7.30	441	302	70	0.142	0.348	0.025	0.056
1980	29500	250	6.95	217	148	33	0.083	0.354	0.028	0.054
1981	67200	283	6.80	456	261	81	0.308	1.576	0.100	0.146
1982	100400	226	6.70	872	518	151	0.620	4.700	0.275	0.255
1983	52800	277	6.39	472	292	97	1.018	4.048	0.229	0.258
1984	40431	247	6.17	288	185	69	0.758	3.825	0.196	0.217
1985	94649	227	6.30	948	496	212	3.656	14.627	0.650	0.674
1986	69334	238	6.17	428	286	124	15.789	12.398	0.551	0.421
1987	63221	233	5.54	445	293	136	3.720	12.748	0.586	0.430
1988	78948	252	5.47	681	399	252	7.155	39.211	1.526	1.445
1989	80188	213	5.27	624	326	199	9.689	45.444	2.552	2.033
1990	78458	235	5.13	722	352	243	11.806	42.134	2.355	2.162
1991	76029	213	5.18	683	316	241	13.184	48.093	2.229	1.548
1992	51026	221	5.07	435	182	148	7.936	27.768	1.330	0.906
1993	65210	228	5.04	636	260	253	13.325	51.527	2.227	1.346
Total	1093194	3789		8851	4875	2366	100.773	347.426	16.689	13.390

[1] Calcium and magnesium were not measured in 1983. Calculated as noted with FL1.

Table A2.7.3. Annual mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL3 (0.63% S, 830 T, cover of glacial till[1] (coarse sand)). Percent of total original sulfur released = 8.36.

Year	Total Volume	Period	Median pH	Annual Mass Release						
				SO_4	Ca^2	Mg^2	Cu	Ni	Co	Zn
	L	days					mol			
1978	90800	193	7.60	904	646	194	0.103	0.693	0.094	0.087
1979	87100	258	7.05	505	326	111	0.131	1.317	0.075	0.126
1980	58300	249	6.65	628	306	184	1.193	7.846	0.409	0.441
1981	92400	285	6.60	1176	516	418	8.131	43.606	2.155	1.636
1982	125600	262	6.58	1626	668	615	13.199	107.941	4.423	2.419
1983	68500	277	5.90	1116	434	457	14.844	78.971	3.556	2.603
1984	78088	292	5.05	1120	410	493	24.583	103.869	5.213	3.060
1985	103131	227	5.44	1738	566	664	46.325	166.454	7.263	4.747
1986	89161	241	5.27	1123	413	491	45.501	137.593	6.517	3.457
1987	81412	233	5.01	1400	484	650	58.733	164.738	6.843	4.709
1988	94121	252	5.15	1200	398	529	51.750	185.101	7.614	6.722
1989	94924	217	4.98	1118	363	440	51.752	157.925	9.292	6.657
Total	1063537	2986		13654	5530	5246	370.486	1325.067	61.380	42.230

[1] Fertilizer and 1500 gallons of lime-stabilized sludge were applied on May 19, 1982. FL3 was covered with plastic and subsequently hypalon, thereby terminating flow from the pile at the end of the 1989 field season.

[2] Calcium and magnesium were not measured in 1983. Calculated as noted with FL1.

Table A2.7.4. Annual mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL4 (0.63% S, 1700 T, no cover). Percent of total original sulfur released = 3.33.

Year	Total Volume	Period	Median pH	Annual Mass Release						
				SO_4	Ca^1	Mg^1	Cu	Ni	Co	Zn
	L	days					mol			
1978	92200	188	7.55	585	374	65	0.023	0.302	0.030	0.099
1979	74000	233	7.10	362	258	46	0.304	2.554	0.075	0.111
1980	53200	211	6.80	444	220	37	0.356	2.570	0.146	0.183
1981	109900	276	6.85	875	574	152	1.137	8.374	0.446	0.514
1982	95165	225	6.60	736	539	148	1.514	12.539	0.649	0.516
1983	110015	228	5.37	743	450	177	4.656	25.373	1.432	1.337
1984	91947	277	4.74	744	356	206	8.648	39.880	2.492	1.669
1985	150757	227	4.92	1353	642	428	20.405	88.641	4.487	3.741
1986	125263	238	4.59	1112	492	412	29.931	109.496	6.166	3.678
1987	106946	233	4.21	1354	486	519	44.846	151.003	7.325	4.970
1988	119294	232	4.37	1320	405	589	58.970	240.280	11.462	10.386
1989	125004	211	4.34	1496	395	587	68.020	212.986	17.319	11.688
Total	1164169	2644		11124	4740	3189	238.809	893.998	52.028	38.891

[1] Calcium and magnesium were not measured in 1983. Calculated as noted with FL1.

Table A2.7.5. Annual mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL6 (0.79% S, 1300 T, no cover[1]). Percent of total original sulfur released = 11.79.

Year	Total Volume	Period	Median pH	Annual Mass Release						
				SO_4	Ca^2	Mg^2	Cu	Ni	Co	Zn
	L	days					mol			
1978	117200	206	7.51	1275	764	370	0.185	1.927	0.319	0.057
1979	118200	257	6.40	1420	860	534	3.628	17.544	1.342	0.711
1980	82300	239	4.40	1281	570	582	15.868	42.868	2.962	1.490
1981	130500	276	4.40	2379	809	1103	66.868	182.112	11.508	5.529
1982	177200	250	4.20	3486	1191	1606	124.259	364.226	17.384	7.607
1983	96000	277	4.19	2404	738	1135	96.020	177.191	11.750	5.414
1984	99158	298	4.14	2548	693	1233	116.935	215.295	12.674	5.660
1985	148259	227	4.27	3934	1000	2037	185.469	303.336	15.303	8.005
1986	114434	238	4.27	2230	708	1291	134.638	197.263	11.481	4.777
1987	108134	227	4.00	2943	805	1533	161.216	214.123	9.816	5.469
1988	117127	247	4.12	2682	791	1483	151.460	233.523	11.456	8.795
1989	101975	217	4.20	2617	630	1181	125.082	206.181	12.672	8.235
1990	36196	235	4.10	1218	268	560	48.316	61.156	3.469	2.377
1991	106787	213	4.14	2991	663	1622	164.840	196.996	9.580	6.305
1992	120552	221	4.24	2112	492	1268	131.697	143.508	7.585	4.530
1993	120417	228	4.24	2269	525	1270	137.459	156.114	7.668	4.406
Total	1794439	3856		37789	11507	18808	1792.498	2857.551	154.686	84.395

[1] FL6 was partially covered with plastic during 1990.

[2] Calcium and magnesium were not measured in 1983. Calculated as noted with FL1.

Table A2.7.6. Annual mass release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL5 (1.41% S, 815 T, cover of sandy till over coarse sand[1]). Percent of total original sulfur released = 8.82.

Year	Total Volume	Period	Median pH	Annual Mass Release						
				SO_4	Ca^2	Mg^2	Cu	Ni	Co	Zn
	L	days					mol			
1978	120500	206	5.50	1081	676	135	3.691	48.868	6.166	1.360
1979	117300	251	3.40	2621	833	624	70.849	287.072	21.962	8.437
1980	35700	243	3.40	1077	203	402	26.018	134.880	9.228	4.319
1981	58900	275	3.40	2138	323	746	68.850	254.790	15.982	6.914
1982	105800	253	3.40	4189	595	1356	135.045	548.375	32.581	12.749
1983	57200	277	3.60	2385	354	781	89.023	259.192	18.032	7.272
1984	49625	299	3.49	1768	274	586	63.989	166.953	12.794	4.810
1985	83220	252	3.50	3336	549	1115	134.463	291.862	19.885	8.840
1986	54523	238	3.63	1687	337	613	72.740	172.556	13.795	4.826
1987	60084	227	3.43	1637	367	578	75.997	142.424	9.156	4.774
1988	70886	202	3.62	1737	423	569	81.601	163.952	10.816	7.724
1989	82485	220	3.62	2085	493	569	85.793	154.543	16.072	8.661
1990	76442	235	3.62	1801	445	491	81.836	106.953	8.724	6.734
1991	70171	213	3.60	1437	354	413	66.072	80.849	5.012	3.750
1992	67278	221	3.73	1151	288	337	55.905	62.402	4.128	2.670
1993	69732	228	3.66	1508	333	406	68.088	87.299	5.268	3.081
Total	1179846	3840		31638	6847	9721	1299.783	3131.484	219.986	104.179

[1] Bactericide was applied in July 1983.

[2] Calcium and magnesium were not measured in 1983. Calculated as noted with FL1.

Table A2.7.7. Sulfur, calcium, magnesium, copper, and nickel percent mass release from the six test piles.

	Initial Mass		Initial S_T Content		Total Mass Release			Release
					SO_4	SO_4	S	S
	ton[1]	kg	%	kg	mol	g	g	%
FL1	1100	1100000	0.63	6930	15384	1478411	492804	7.11
FL2	1100	1100000	0.63	6930	8851	850608	283536	4.09
FL3	830	830000	0.63	5229	13654	1312149	437383	8.36
FL4	1700	1700000	0.63	10710	11124	1068974	356325	3.33
FL6	1300	1300000	0.79	10270	37789	3631526	1210509	11.79
FL5	815	815000	1.41	11492	31638	3040377	1013459	8.82

	Initial Mass		Initial CaO Content[2]		Initial Ca Content	Total Mass Release		Release
						Ca	Ca	Ca
	ton	kg	%	kg	kg	mol	g	%
FL1	1100	1100000	7.91	87010	62194	9502	381033	0.61
FL2	1100	1100000	7.91	87010	62194	4875	195506	0.31
FL3	830	830000	7.91	65653	46928	5530	221733	0.47
FL4	1700	1700000	7.91	134470	96118	4740	190091	0.20
FL6	1300	1300000	8.62	112060	80100	11507	461416	0.58
FL5	815	815000	6.51	53057	37925	6847	274559	0.72

	Initial Mass		Initial MgO Content[2]		Initial Mg Content	Total Mass Release		Release
						Mg	Mg	Mg
	ton	kg	%	kg	kg	mol	g	%
FL1	1100	1100000	7.64	84040	50676	3521	85555	0.17
FL2	1100	1100000	7.64	84040	50676	2366	57494	0.11
FL3	830	830000	7.64	63412	38237	5246	127470	0.33
FL4	1700	1700000	7.64	129880	78317	3189	77487	0.10
FL6	1300	1300000	9.98	129740	78232	18808	457044	0.58
FL5	815	815000	6.25	50938	30715	9721	236224	0.77

	Initial Mass		Initial Cu Content[3]		Total Mass Release		Release
					Cu	Cu	Cu
	ton	kg	%	kg	mol	g	%
FL1	1100	1100000	0.35	3850	183	11616	0.30
FL2	1100	1100000	0.35	3850	101	6399	0.17
FL3	830	830000	0.35	2905	370	23526	0.81
FL4	1700	1700000	0.35	5950	239	15164	0.25
FL6	1300	1300000	0.34	4420	1792	113824	2.58
FL5	815	815000	0.30	2445	1300	82536	3.38

	Initial Mass		Initial Ni Content[3]		Total Mass Release		Release
					Ni	Ni	Ni
	ton	kg	%	kg	mol	g	%
FL1	1100	1100000	0.083	913	708	41578	4.55
FL2	1100	1100000	0.083	913	347	20394	2.23
FL3	830	830000	0.083	689	1325	77781	11.29
FL4	1700	1700000	0.083	1411	894	52478	3.72
FL6	1300	1300000	0.084	1092	2858	167738	15.36
FL5	815	815000	0.09	693	3131	183818	26.53

[1] all tons metric

[2] from mass weighted average of the bulk samples (appendix 4); leached samples, but very little change; best value availabl

[3] from average of analyses from rounds placed in the piles

Table A2.7.8. Calculation of yield coefficients for the six Amax test piles.

Annual Precipitation (cm)

from table A1.1. In Field Dissolution of Test Piles of Duluth Complex Rock, September 1993

	1978	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993
cm	69.6	63.3	58.8	63.4	86.5	70.9	53.5	80.1	69	60.3	81.1	69.3	64.5	62.8	60	65.5

Collecting Area (m^3)

from table 1. In Field Dissolution of Test Piles of Duluth Complex Rock, September 1993

	FL1	FL2	FL3	FL4	FL6	FL5
m^2	290	340	300	330	320	340

Yield Coefficients $Y_C = V/1000 /(P/100 * A)$

Pile	Year	Volume	Y_C	
FL1	1978	83100	0.41	
	1979	86600	0.47	
	1980	53800	0.32	
	1981	83800	0.46	
	1982	118600	0.47	
	1983	82400	0.40	
	1984	66879	0.43	
	1985	133396	0.57	
	1986	119551	0.60	
	1987	104768	0.60	
	1988	106298	0.45	
	1989	122463	0.61	
	1990	60101	0.32	
	1991	120980	0.66	
	1992	103955	0.60	
	1993	124968	0.66	
			0.50	average
			0.47	median

Pile	Year	Volume	Y_C	
FL2	1978	70200	0.30	
	1979	75600	0.35	
	1980	29500	0.15	
	1981	67200	0.31	
	1982	100400	0.34	
	1983	52800	0.22	
	1984	40431	0.22	
	1985	94649	0.35	
	1986	69334	0.30	
	1987	63221	0.31	
	1988	78948	0.29	
	1989	80188	0.34	
	1990	78458	0.36	
	1991	76029	0.36	
	1992	51026	0.25	
	1993	65210	0.29	
			0.30	average
			0.30	median

Pile	Year	Volume	Y_C	
FL3	1978	90800	0.43	
	1979	87100	0.46	
	1980	58300	0.33	
	1981	92400	0.49	
	1982	125600	0.48	
	1983	68500	0.32	
	1984	78088	0.49	
	1985	103131	0.43	
	1986	89161	0.43	
	1987	81412	0.45	
	1988	94121	0.39	
	1989	94924	0.46	
			0.43	average
			0.44	median

Pile	Year	Volume	Y_C	
FL4	1978	92200	0.40	
	1979	74000	0.35	
	1980	53200	0.27	
	1981	109900	0.53	
	1982	95165	0.33	
	1983	110015	0.47	
	1984	91947	0.52	
	1985	150757	0.57	
	1986	125263	0.55	
	1987	106946	0.54	
	1988	119294	0.45	
	1989	125004	0.55	
			0.46	average
			0.50	median

Pile	Year	Volume	Y_C	
FL6	1978	117200	0.53	
	1979	118200	0.58	
	1980	82300	0.44	
	1981	130500	0.64	
	1982	177200	0.64	
	1983	96000	0.42	
	1984	99158	0.58	
	1985	148259	0.58	
	1986	114434	0.52	
	1987	108134	0.56	
	1988	117127	0.45	
	1989	101975	0.46	
	1990	36196	0.18	
	1991	106787	0.53	
	1992	120552	0.63	
	1993	120417	0.57	
			0.52	average
			0.55	median

Pile	Year	Volume	Y_C	
FL5	1978	120500	0.51	
	1979	117300	0.55	
	1980	35700	0.18	
	1981	58900	0.27	
	1982	105800	0.36	
	1983	57200	0.24	
	1984	49625	0.27	
	1985	83220	0.31	
	1986	54523	0.23	
	1987	60084	0.29	
	1988	70886	0.26	
	1989	82485	0.35	
	1990	76442	0.35	
	1991	70171	0.33	
	1992	67278	0.33	
	1993	69732	0.31	
			0.32	average
			0.31	median

notes:

FL1 and FL6 were covered with plastic membrane at the end of the 1989 field season reducing flow for 1990.

This membrane was only in effect for the 1990 season.

Table A2.8.1. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL1 (0.63% S, 1100 T, no cover). Percent of total original sulfur released = 7.11.

Year	Total Volume	Period	Median pH	Annual Rates of Mass Release						
				SO_4	Ca	Mg	Cu	Ni	Co	Zn
	L	days		mmol/T day						
1978	83100	206	7.80	3.623	1.836	0.256	0.0001	0.0014	0.0001	0.0001
1979	86600	252	7.35	2.417	1.742	0.256	0.0001	0.0018	0.0001	0.0001
1980	53800	256	7.20	2.390	1.513	0.234	0.0001	0.0023	0.0001	0.0001
1981	83800	275	7.20	2.225	1.650	0.307	0.0004	0.0091	0.0004	0.0004
1982	118600	258	7.00	3.284	2.660	0.398	0.0007	0.0308	0.0013	0.0006
1983	82400	277	6.86	2.796	2.092	0.416	0.0011	0.0186	0.0008	0.0005
1984	66879	298	6.36	1.952	1.339	0.345	0.0036	0.0521	0.0023	0.0012
1985	133396	227	6.70	4.890	3.176	0.925	0.0112	0.1307	0.0047	0.0030
1986	119551	238	6.54	3.438	2.513	0.772	0.0204	0.1420	0.0054	0.0028
1987	104768	233	6.03	4.194	2.829	0.893	0.0324	0.1570	0.0055	0.0040
1988	106298	247	6.09	3.577	2.429	1.104	0.0411	0.2498	0.0082	0.0064
1989	122463	213	5.69	5.250	2.958	1.315	0.0668	0.3120	0.0141	0.0096
1990	60101	235	5.72	3.033	1.590	0.901	0.0711	0.2242	0.0109	0.0075
1991	120980	213	5.25	6.731	3.295	2.356	0.1886	0.5973	0.0238	0.0132
1992	103955	221	5.19	4.266	2.117	1.526	0.1274	0.3540	0.0149	0.0082
1993	124968	228	5.14	5.291	2.472	1.810	0.1510	0.4436	0.0166	0.0083
Overall[1]	1571659	3877		3.607	2.228	0.826	0.0429	0.1661	0.0067	0.0040

[1] Calculated as total release in (mol x 1000) / (mass of pile x number of days)

Table A2.8.2. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL2 (0.63% S, 1100 T, cover of topsoil). Percent of total original sulfur released = 4.09.

Year	Total Volume	Period	Median pH	Annual Rates of Mass Release						
				SO_4	Ca	Mg	Cu	Ni	Co	Zn
	L	days		mmol/T day						
1978	70200	193	7.60	2.369	1.225	0.268	0.0006	0.0012	0.0001	0.0002
1979	75600	253	7.30	1.585	1.085	0.252	0.0005	0.0013	0.0001	0.0002
1980	29500	250	6.95	0.789	0.538	0.120	0.0003	0.0013	0.0001	0.0002
1981	67200	283	6.80	1.465	0.838	0.260	0.0010	0.0051	0.0003	0.0005
1982	100400	226	6.70	3.508	2.084	0.607	0.0025	0.0189	0.0011	0.0010
1983	52800	277	6.39	1.549	0.958	0.320	0.0033	0.0133	0.0008	0.0008
1984	40431	247	6.17	1.060	0.681	0.254	0.0028	0.0141	0.0007	0.0008
1985	94649	227	6.30	3.797	1.986	0.849	0.0146	0.0586	0.0026	0.0027
1986	69334	238	6.17	1.635	1.092	0.474	0.0603	0.0474	0.0021	0.0016
1987	63221	233	5.54	1.736	1.143	0.531	0.0145	0.0497	0.0023	0.0017
1988	78948	252	5.47	2.457	1.439	0.909	0.0258	0.1415	0.0055	0.0052
1989	80188	213	5.27	2.663	1.391	0.849	0.0414	0.1940	0.0109	0.0087
1990	78458	235	5.13	2.793	1.362	0.940	0.0457	0.1630	0.0091	0.0084
1991	76029	213	5.18	2.915	1.349	1.029	0.0563	0.2053	0.0095	0.0066
1992	51026	221	5.07	1.791	0.747	0.607	0.0326	0.1142	0.0055	0.0037
1993	65210	228	5.04	2.536	1.037	1.009	0.0531	0.2054	0.0089	0.0054
Overall[1]	1093194	3789		2.124	1.170	0.568	0.0242	0.0834	0.0040	0.0032

[1] Calculated as (total release in mol x 1000) / (mass of pile x number of days)

Table A2.8.3. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL3 (0.63% S, 830 T, cover of glacial till (coarse sand)). Percent of total original sulfur released = 8.36.

Year	Total Volume	Period	Median pH	Annual Rates of Mass Release						
				SO_4	Ca	Mg	Cu	Ni	Co	Zn
	L	days		mmol/T day						
1978	90800	193	7.60	5.643	4.033	1.211	0.0006	0.0043	0.0006	0.0005
1979	87100	258	7.05	2.358	1.522	0.518	0.0006	0.0062	0.0003	0.0006
1980	58300	249	6.65	3.039	1.481	0.890	0.0058	0.0380	0.0020	0.0021
1981	92400	285	6.60	4.971	2.181	1.767	0.0344	0.1843	0.0091	0.0069
1982	125600	262	6.58	7.477	3.072	2.828	0.0607	0.4964	0.0203	0.0111
1983	68500	277	5.90	4.854	1.886	1.986	0.0646	0.3435	0.0155	0.0113
1984	78088	292	5.05	4.621	1.692	2.034	0.1014	0.4286	0.0215	0.0126
1985	103131	227	5.44	9.225	3.004	3.524	0.2459	0.8835	0.0385	0.0252
1986	89161	241	5.27	5.614	2.065	2.455	0.2275	0.6879	0.0326	0.0173
1987	81412	233	5.01	7.239	2.503	3.361	0.3037	0.8518	0.0354	0.0243
1988	94121	252	5.15	5.737	1.903	2.529	0.2474	0.8850	0.0364	0.0321
1989	94924	217	4.98	6.207	2.015	2.443	0.2873	0.8768	0.0516	0.0370
Overall[1]	1063537	2986		5.509	2.231	2.117	0.1495	0.5347	0.0248	0.0170

[1] Calculated as (total release in mol x 1000) / (mass of pile x number of days)

Table A2.8.4. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL4 (0.63% S, 1700 T, no cover). Percent of total original sulfur released = 3.33.

Year	Total Volume	Period	Median pH	Annual Rates of Mass Release						
				SO_4	Ca	Mg	Cu	Ni	Co	Zn
	L	days		mmol/T day						
1978	92200	188	7.55	1.830	1.170	0.203	0.0001	0.0009	0.0001	0.0003
1979	74000	233	7.10	0.914	0.651	0.116	0.0008	0.0064	0.0002	0.0003
1980	53200	211	6.80	1.238	0.613	0.103	0.0010	0.0072	0.0004	0.0005
1981	109900	276	6.85	1.865	1.223	0.324	0.0024	0.0178	0.0009	0.0011
1982	95165	225	6.60	1.924	1.410	0.387	0.0040	0.0328	0.0017	0.0013
1983	110015	228	5.37	1.916	0.000	0.000	0.0120	0.0655	0.0037	0.0034
1984	91947	277	4.74	1.580	0.755	0.437	0.0184	0.0847	0.0053	0.0035
1985	150757	227	4.92	3.506	1.663	1.108	0.0529	0.2297	0.0116	0.0097
1986	125263	238	4.59	2.748	1.215	1.019	0.0740	0.2706	0.0152	0.0091
1987	106946	233	4.21	3.419	1.228	1.311	0.1132	0.3812	0.0185	0.0125
1988	119294	232	4.37	3.348	1.026	1.492	0.1495	0.6092	0.0291	0.0263
1989	125004	211	4.34	4.170	1.101	1.638	0.1896	0.5938	0.0483	0.0326
Overall[1]	1164169	2644		2.475	1.055	0.709	0.0531	0.1988	0.0116	0.0086

[1] Calculated as (total release in mol x 1000) / (mass of pile x number of days)

Table A2.8.5. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL6 (0.79% S, 1300 T, no cover). Percent of total original sulfur released = 11.79.

Year	Total Volume	Period	Median pH	Annual Rates of Mass Release						
				SO_4	Ca	Mg	Cu	Ni	Co	Zn
	L	days		mmol/T day						
1978	117200	206	7.51	4.761	2.853	1.382	0.0007	0.0072	0.0012	0.0002
1979	118200	257	6.40	4.250	2.574	1.598	0.0109	0.0525	0.0040	0.0021
1980	82300	239	4.40	4.123	1.835	1.873	0.0511	0.1380	0.0095	0.0048
1981	130500	276	4.40	6.630	2.255	3.074	0.1864	0.5076	0.0321	0.0154
1982	177200	250	4.20	10.726	3.665	4.942	0.3823	1.1207	0.0535	0.0234
1983	96000	277	4.19	6.676	2.048	3.153	0.2666	0.4921	0.0326	0.0150
1984	99158	298	4.14	6.577	1.789	3.183	0.3018	0.5557	0.0327	0.0146
1985	148259	227	4.27	13.331	3.389	6.903	0.6285	1.0279	0.0519	0.0271
1986	114434	238	4.27	7.207	2.288	4.173	0.4352	0.6376	0.0371	0.0154
1987	108134	227	4.00	9.973	2.728	5.195	0.5463	0.7256	0.0333	0.0185
1988	117127	247	4.12	8.353	2.463	4.618	0.4717	0.7273	0.0357	0.0274
1989	101975	217	4.20	9.277	2.233	4.186	0.4434	0.7309	0.0449	0.0292
1990	36196	235	4.10	3.987	0.877	1.833	0.1582	0.2002	0.0114	0.0078
1991	106787	213	4.14	10.802	2.394	5.858	0.5953	0.7114	0.0346	0.0228
1992	120552	221	4.24	7.351	1.713	4.413	0.4584	0.4995	0.0264	0.0158
1993	120417	228	4.24	7.655	1.771	4.285	0.4638	0.5267	0.0259	0.0149
Overall[1]	1794439	3856		7.539	2.295	3.752	0.3576	0.5701	0.0309	0.0168

[1] Calculated as (total release in mol x 1000) / (mass of pile x number of days)

Table A2.8.6. Annual rates of release of sulfate, calcium, magnesium, copper, nickel, cobalt, and zinc for Duluth Complex test pile FL5 (1.41% S, 815 T, cover of sandy till over coarse sand). Percent of total original sulfur released = 8.82.

Year	Total Volume	Period	Median pH	Annual Rates of Mass Release						
				SO_4	Ca	Mg	Cu	Ni	Co	Zn
	L	days		mmol/T day						
1978	120500	206	5.50	6.439	4.026	0.804	0.0220	0.2911	0.0367	0.0081
1979	117300	251	3.40	12.813	4.072	3.050	0.3463	1.4033	0.1074	0.0412
1980	35700	243	3.40	5.438	1.025	2.030	0.1314	0.6811	0.0466	0.0218
1981	58900	275	3.40	9.539	1.441	3.328	0.3072	1.1368	0.0713	0.0308
1982	105800	253	3.40	20.316	2.886	6.576	0.6549	2.6595	0.1580	0.0618
1983	57200	277	3.60	10.565	1.569	3.461	0.3943	1.1481	0.0799	0.0322
1984	49625	299	3.49	7.255	1.124	2.405	0.2626	0.6851	0.0525	0.0197
1985	83220	252	3.50	16.243	2.673	5.429	0.6547	1.4211	0.0968	0.0430
1986	54523	238	3.63	8.697	1.737	3.160	0.3750	0.8896	0.0711	0.0249
1987	60084	227	3.43	8.848	1.984	3.124	0.4108	0.7698	0.0495	0.0258
1988	70886	202	3.62	10.551	2.569	3.456	0.4957	0.9959	0.0657	0.0469
1989	82485	220	3.62	11.629	2.750	3.173	0.4785	0.8619	0.0896	0.0483
1990	76442	235	3.62	9.403	2.323	2.564	0.4273	0.5584	0.0455	0.0352
1991	70171	213	3.60	8.278	2.039	2.379	0.3806	0.4657	0.0289	0.0216
1992	67278	221	3.73	6.388	1.597	1.870	0.3104	0.3465	0.0229	0.0148
1993	69732	228	3.66	8.115	1.792	2.185	0.3664	0.4698	0.0283	0.0166
Overall[1]	1179846	3840		10.109	2.188	3.106	0.4153	1.0006	0.0703	0.0333

[1] Calculated as (total release in mol x 1000) / (mass of pile x number of days)

Figure A2.9.1. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL1 (0.6%S): 1978-93.

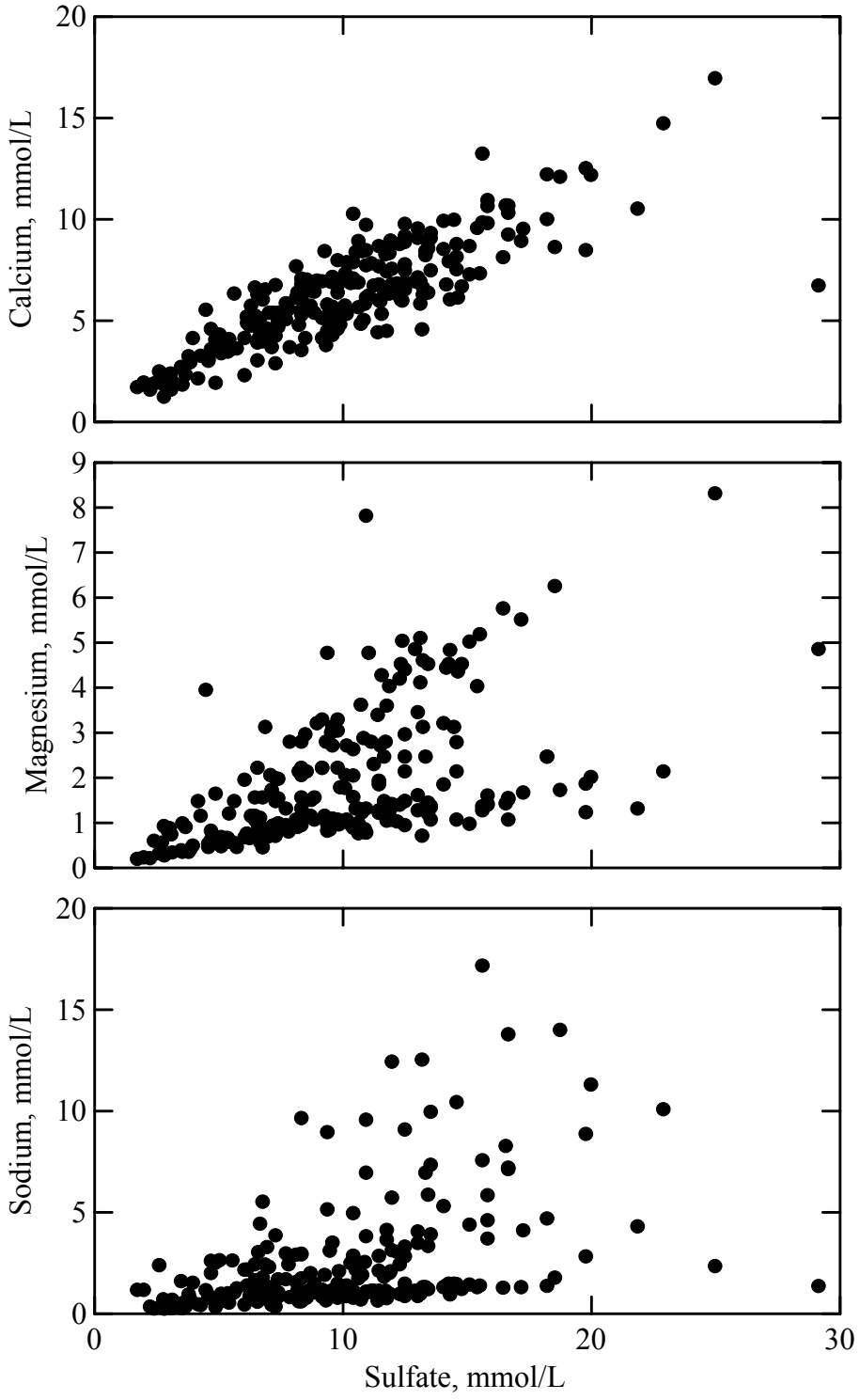


Figure A2.9.2. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL2 (0.6%S): 1978-93.

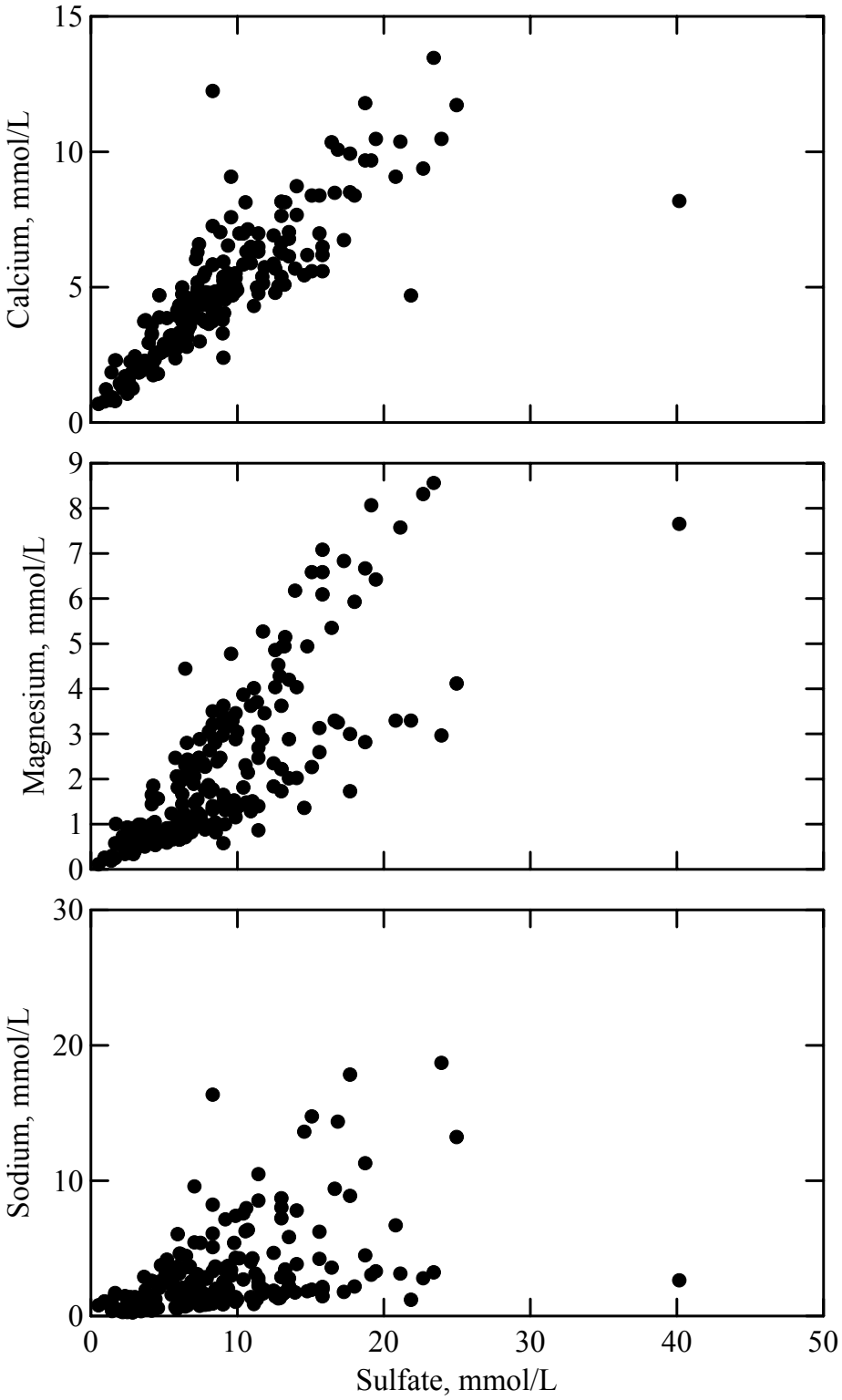


Figure A2.9.3. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL3 (0.6%S): 1978-89.

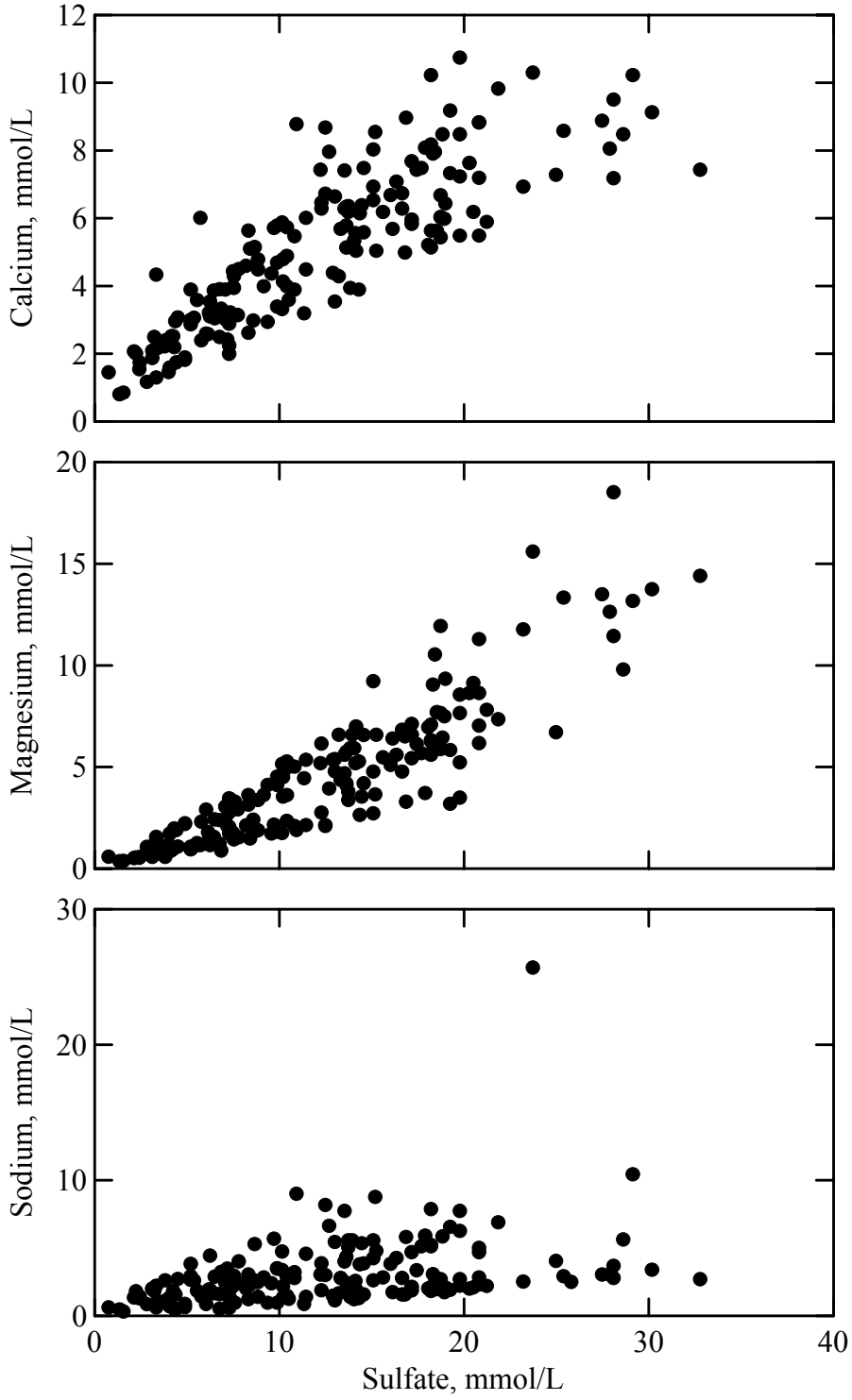


Figure A2.9.4. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL4 (0.6%S): 1978-89.

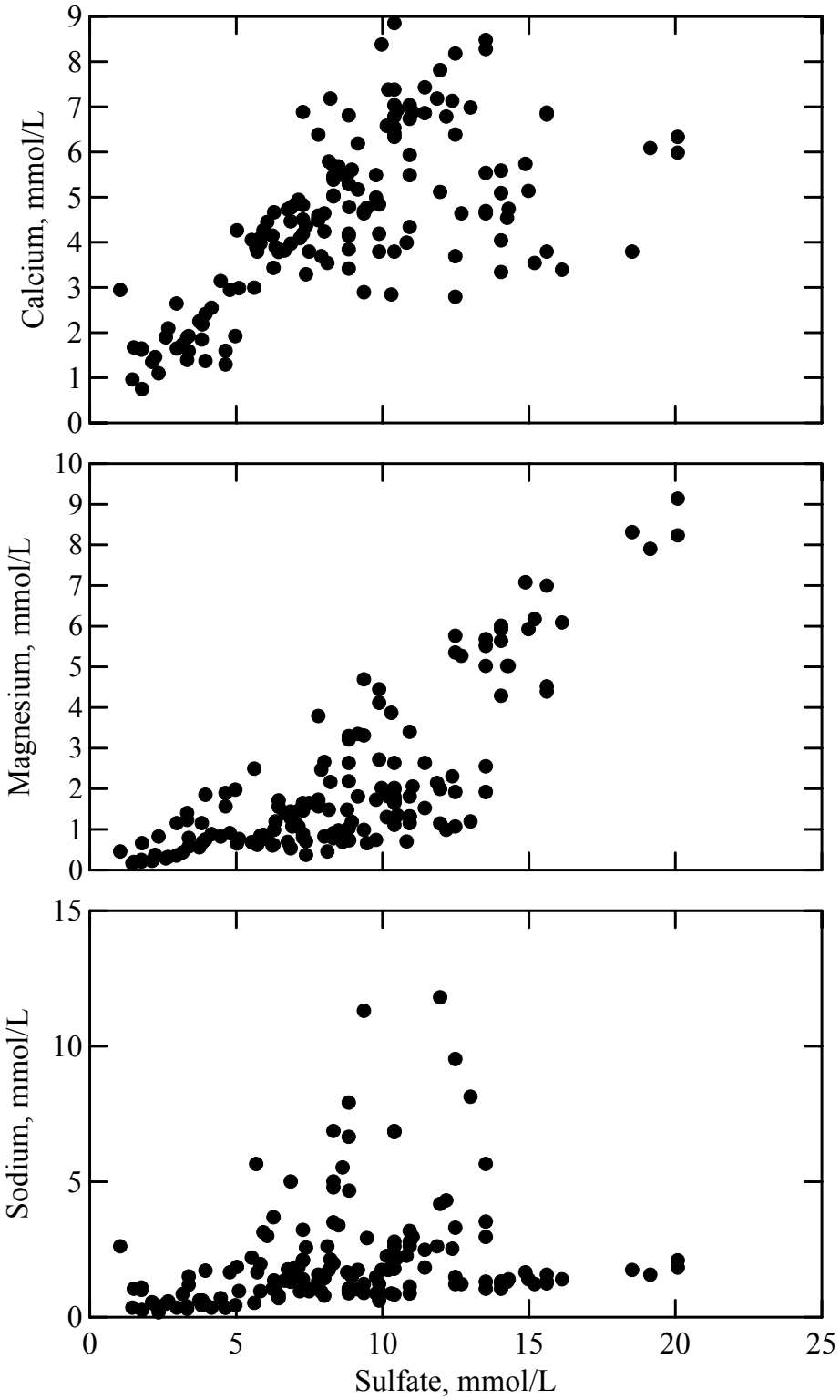


Figure A2.9.5. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL6 (0.8%S): 1978-93.

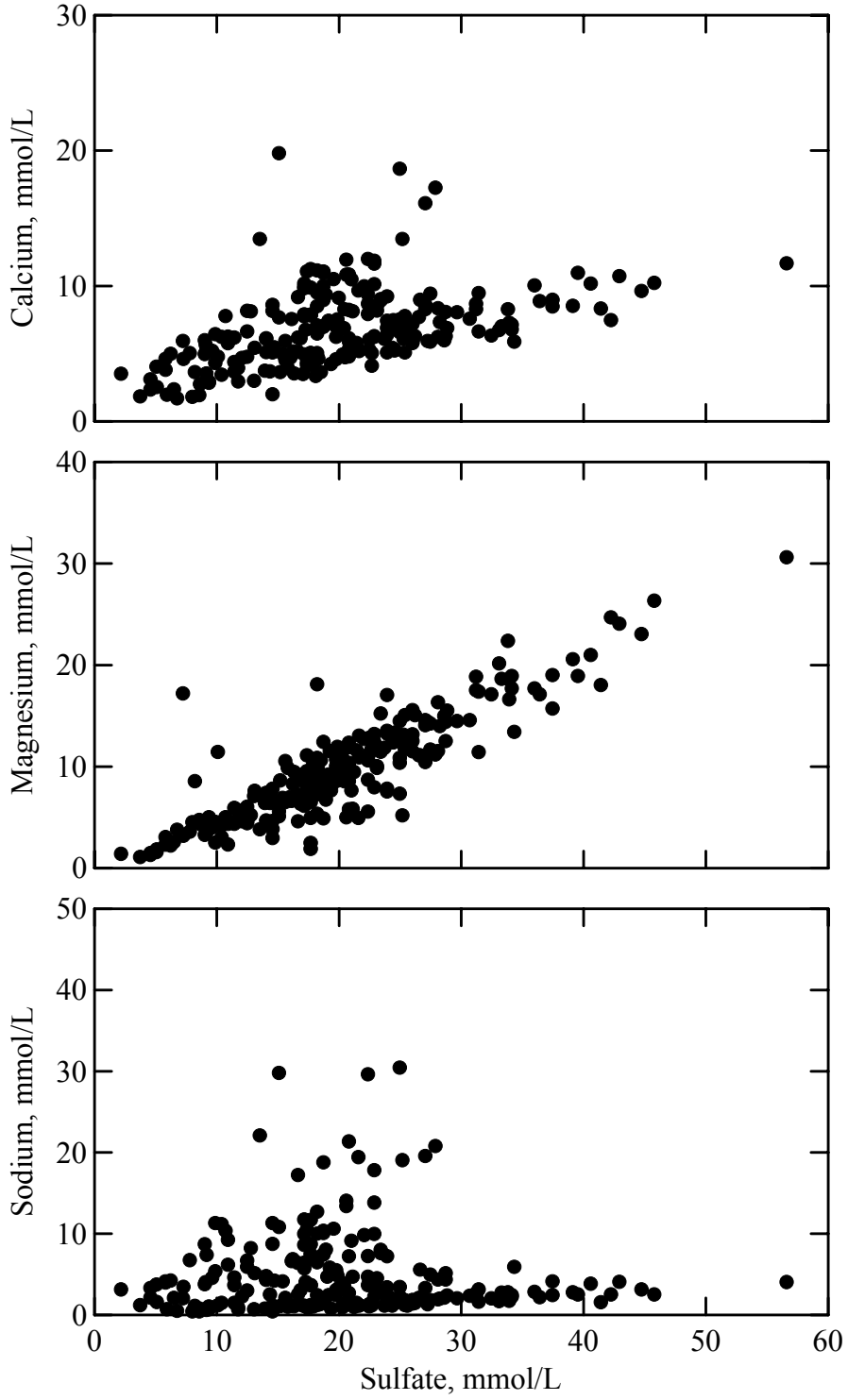


Figure A2.9.6. Calcium, magnesium, and sodium versus sulfate in drainage from pile FL5 (1.4%S): 1978-93.

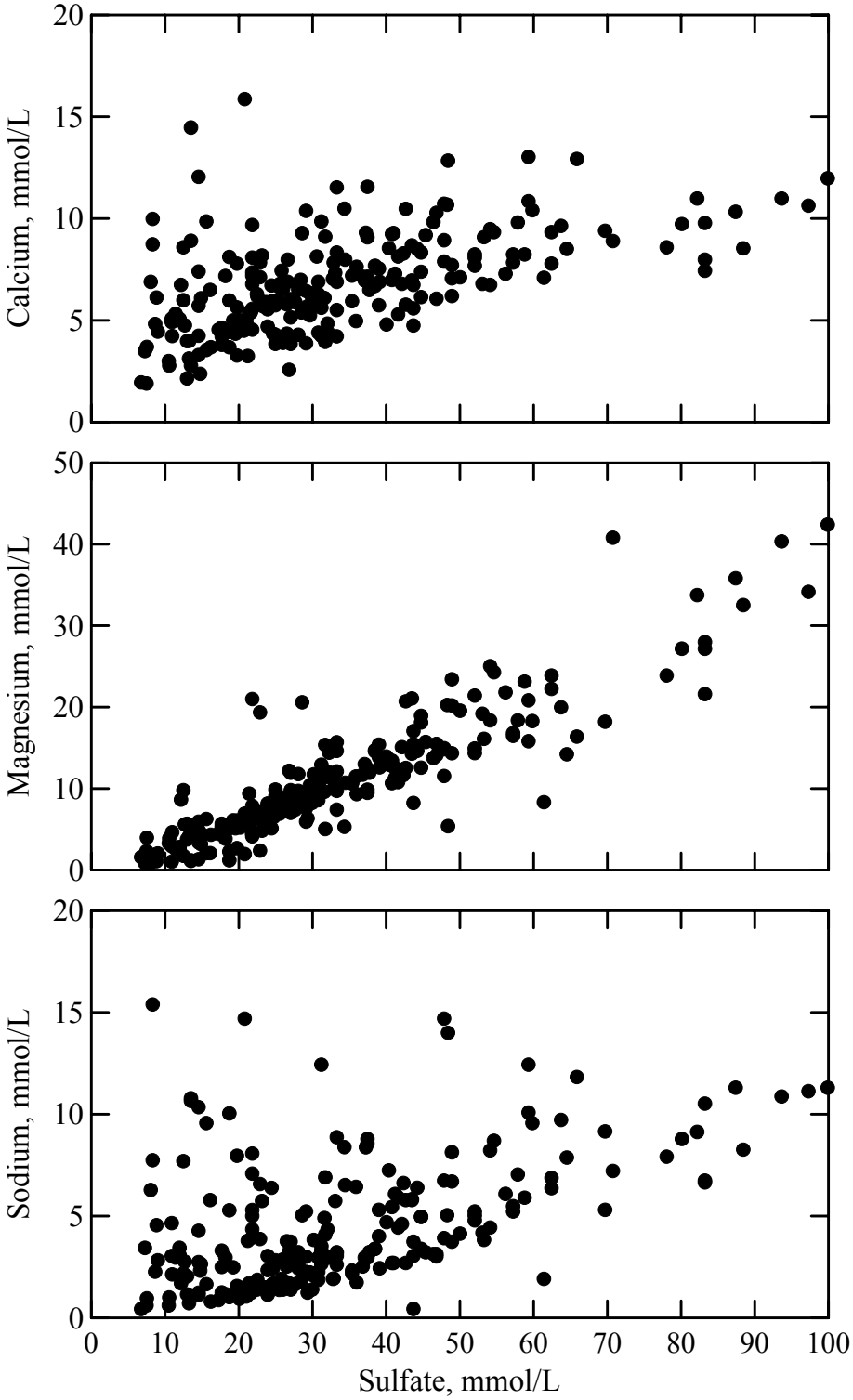


Figure A2.10.1. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.6 %S pile (FL1): 1978-1993.

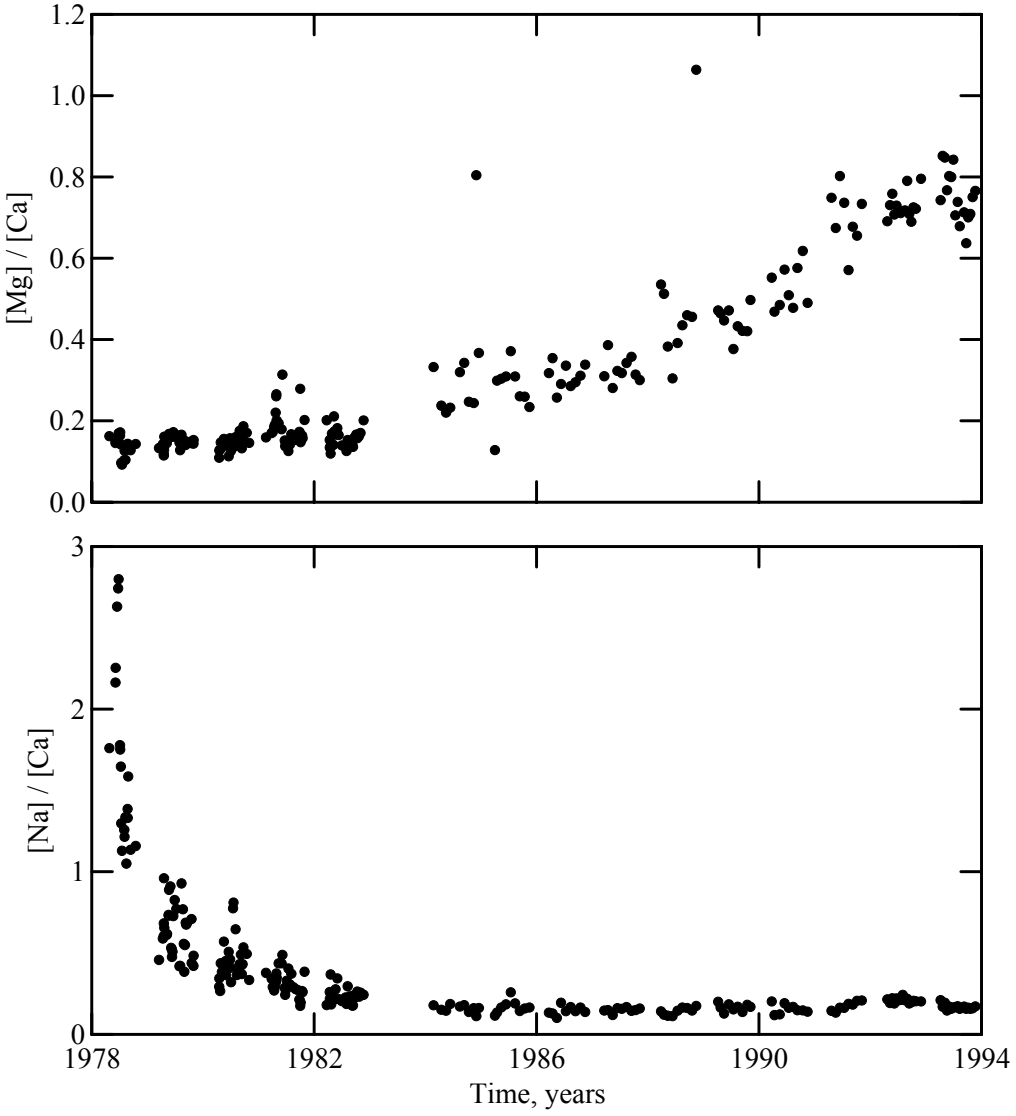


Figure A2.10.2. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.6 %S pile (FL2): 1978-1993.

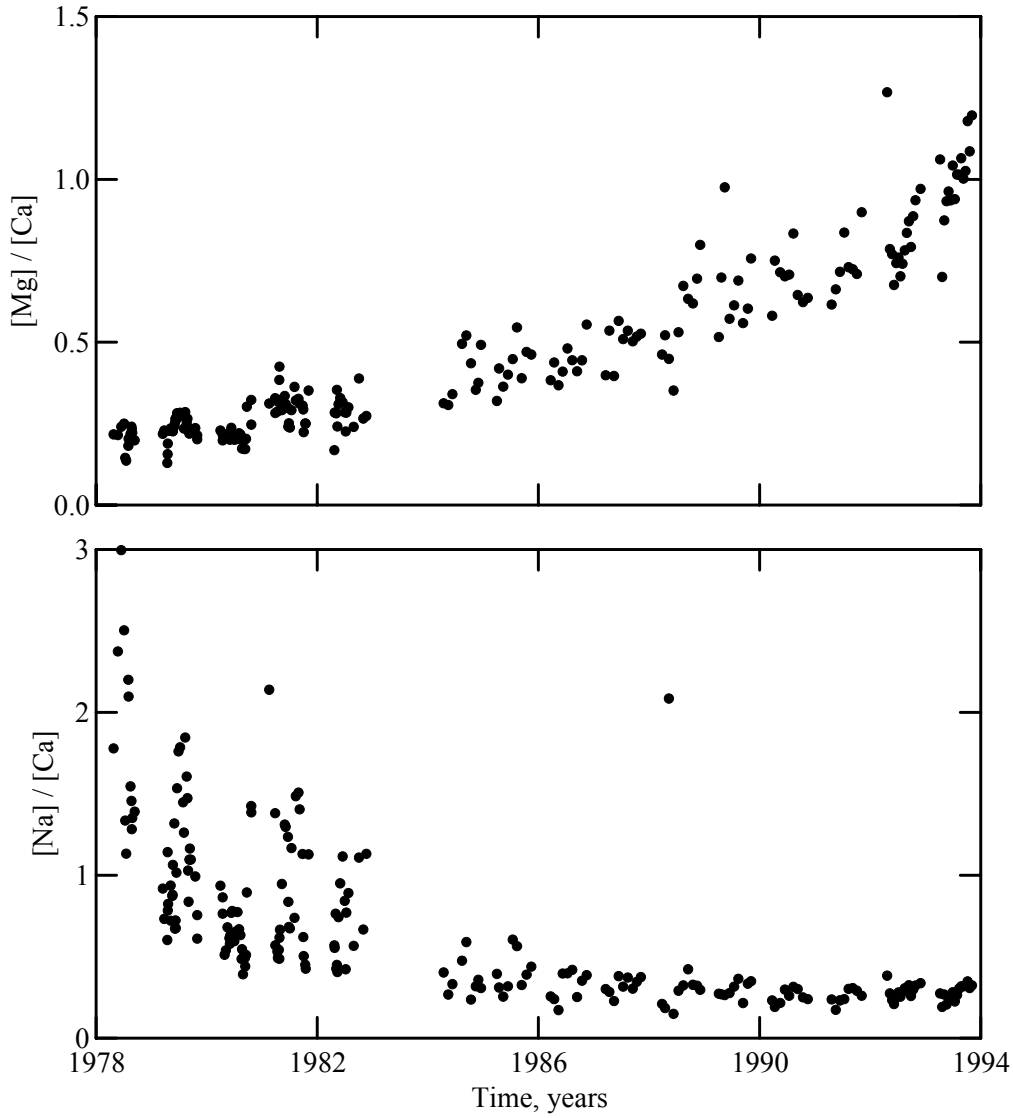


Figure A2.10.3. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.6 %S pile (FL3): 1978-1989.

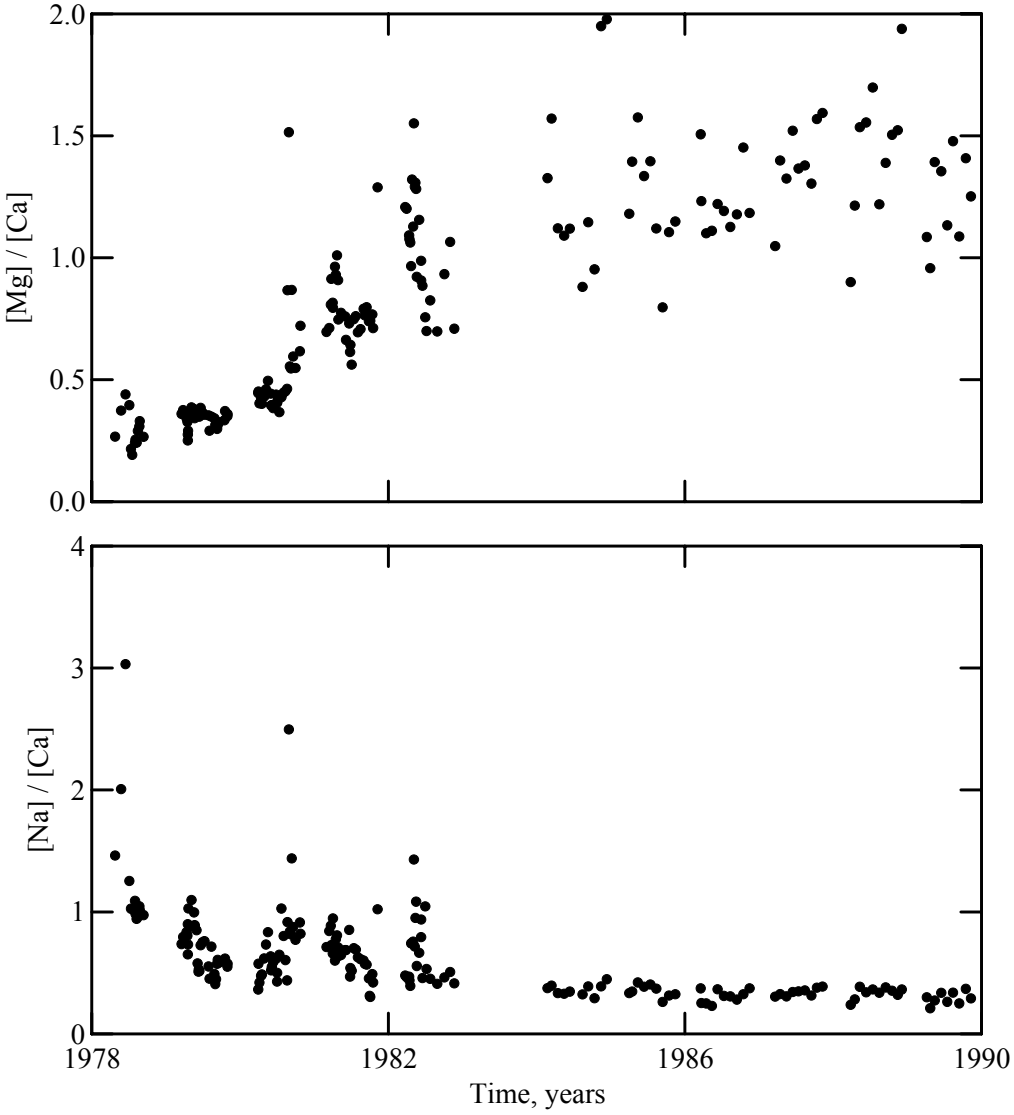


Figure A2.10.4. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.6 %S pile (FL4): 1978-1989.

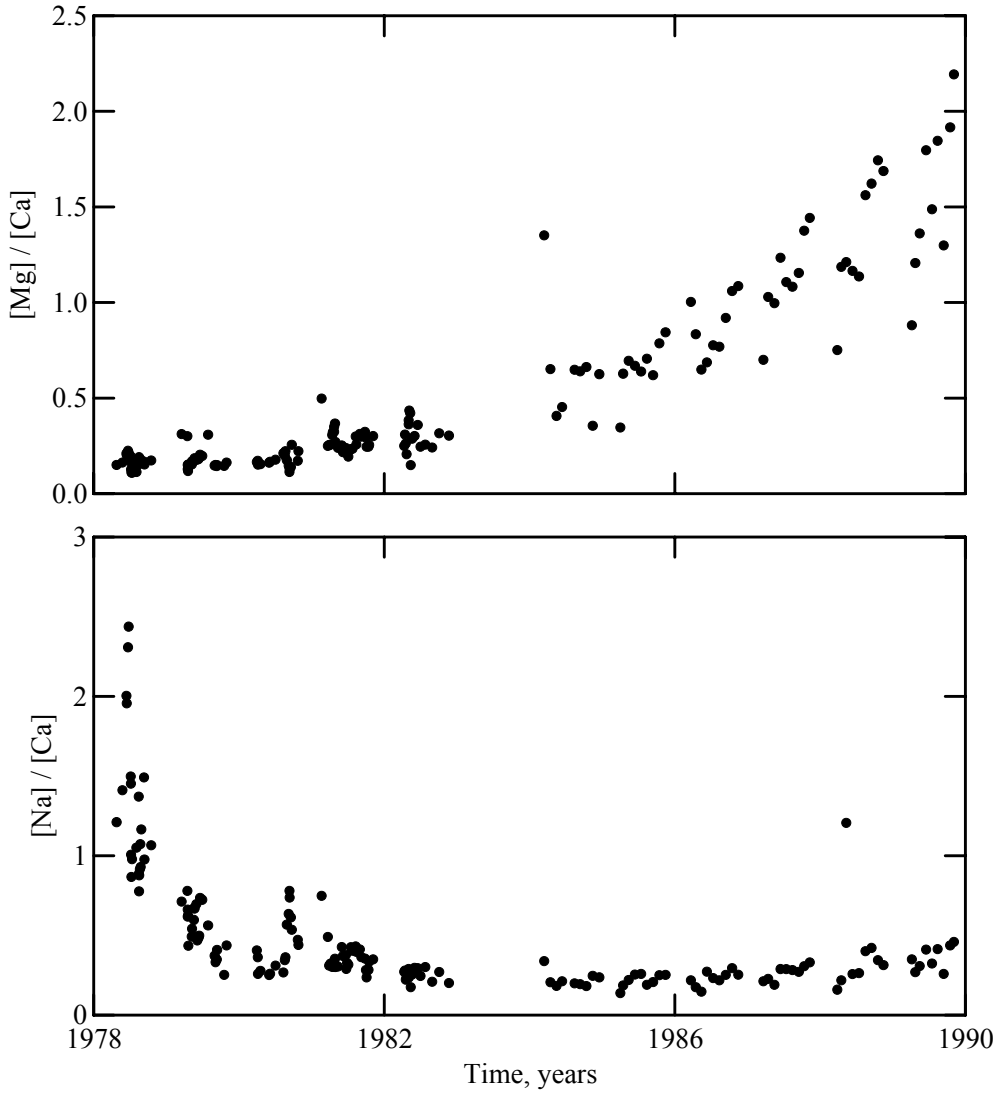


Figure A2.10.5. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 0.8 %S pile (FL6): 1978-1993.

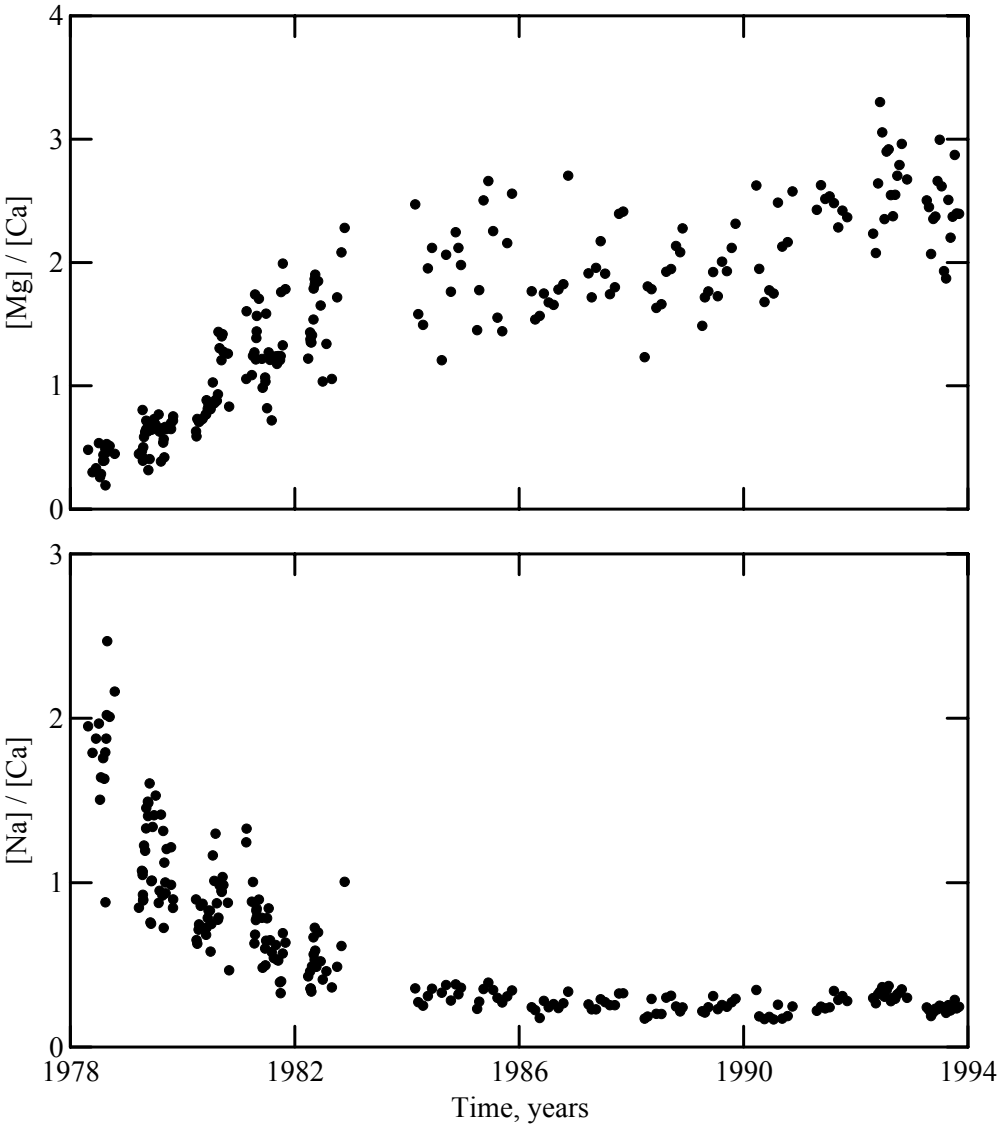


Figure A2.10.6. Ratios of concentrations of Mg:Ca and Na:Ca versus time for 1.4 %S pile (FL5): 1978-1993.

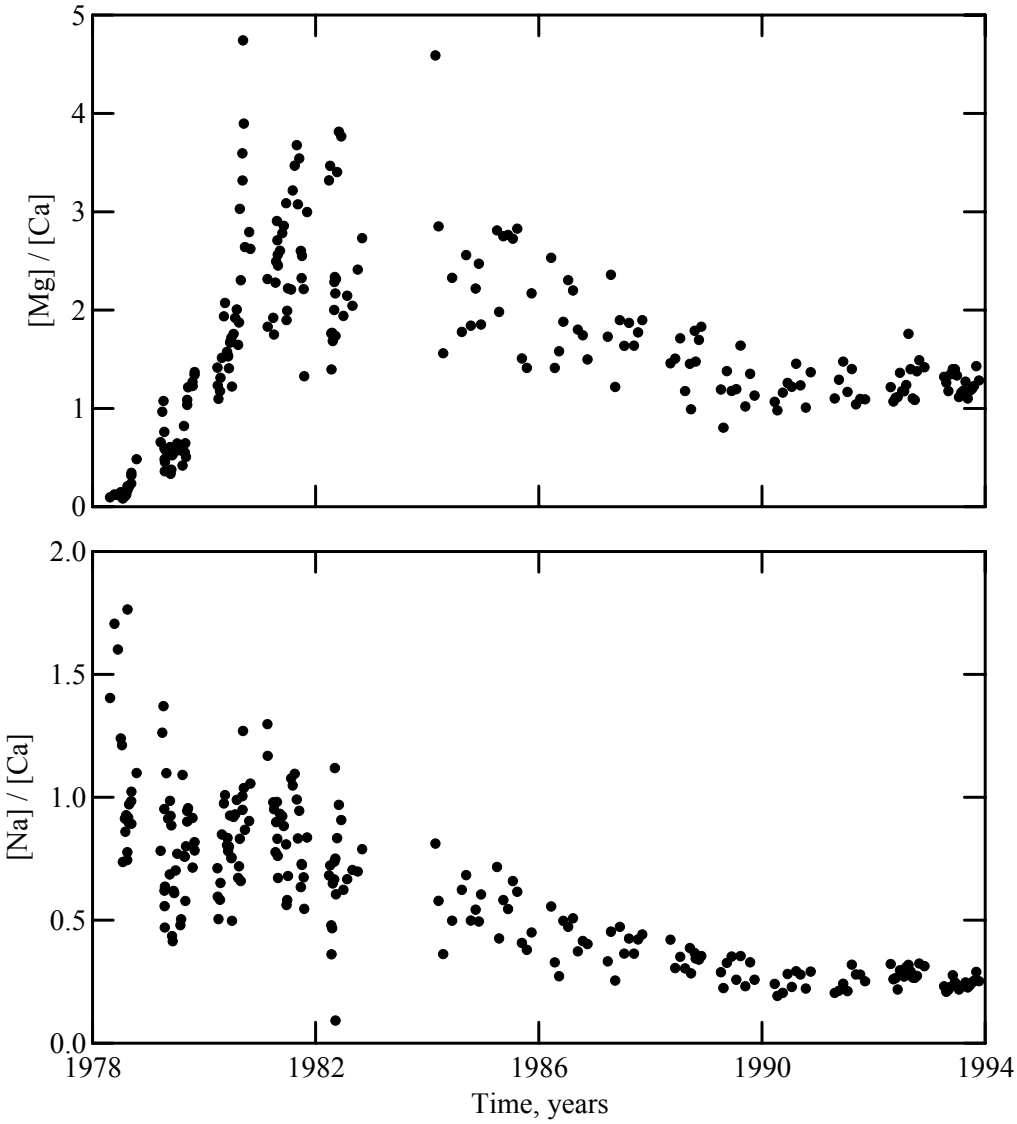


Figure A2.11.1. Calcium and magnesium in drainage from pile FL1 (0.6%S) versus pH.

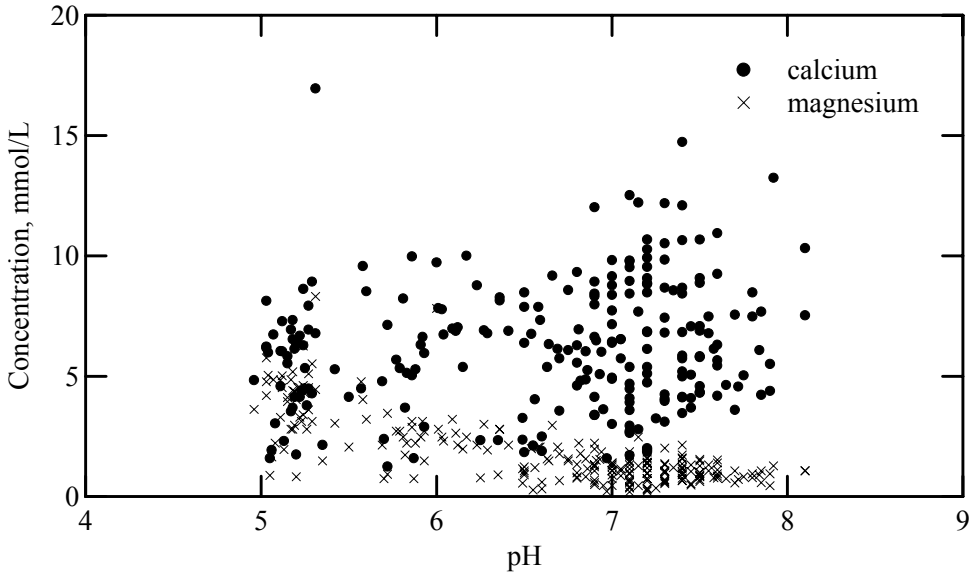


Figure A2.11.2. Calcium and magnesium in drainage from pile FL2 (0.6%S) versus pH.

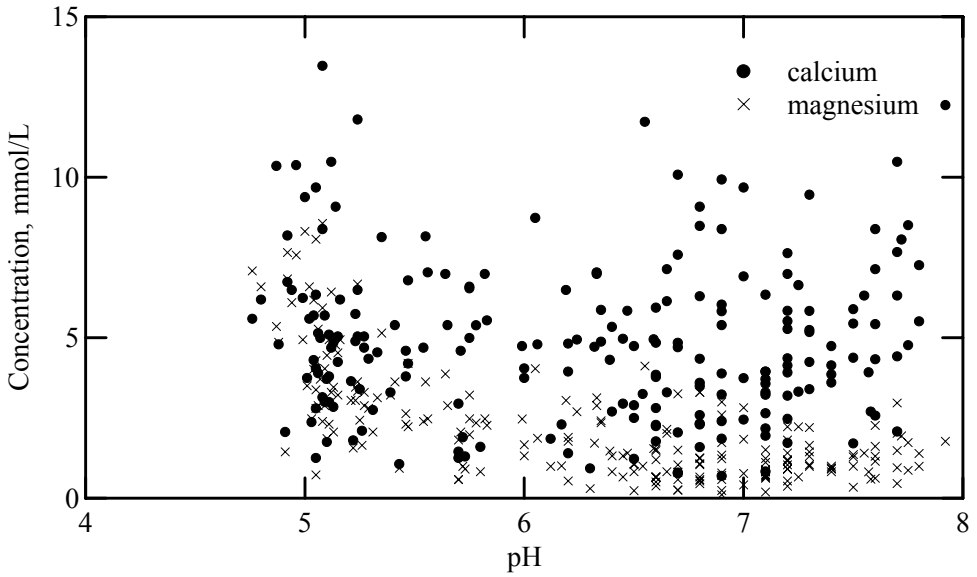


Figure A2.11.3. Calcium and magnesium in drainage from pile FL3 (0.6%S) versus pH.

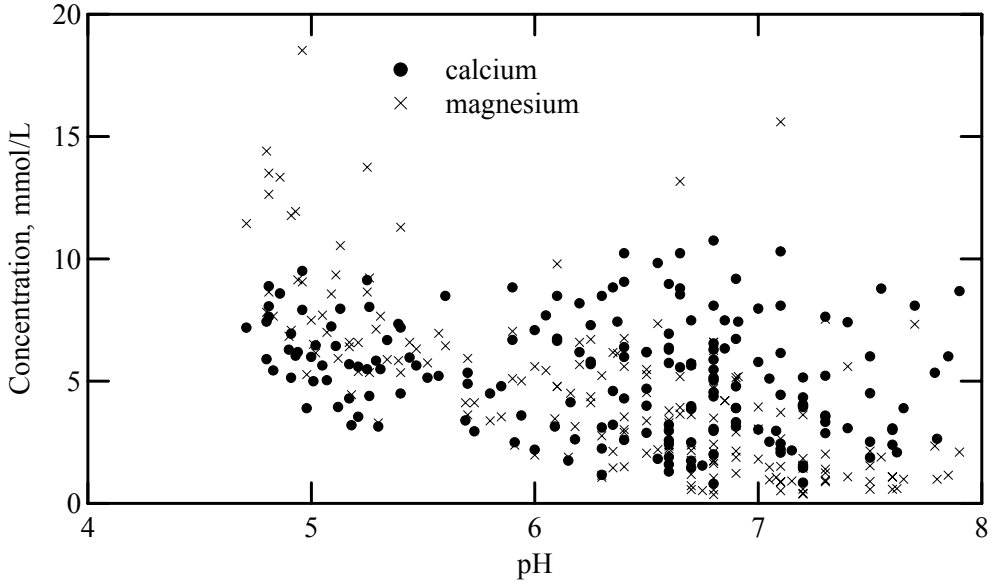


Figure A2.11.4. Calcium and magnesium in drainage from pile FL4 (0.6%S) versus pH.

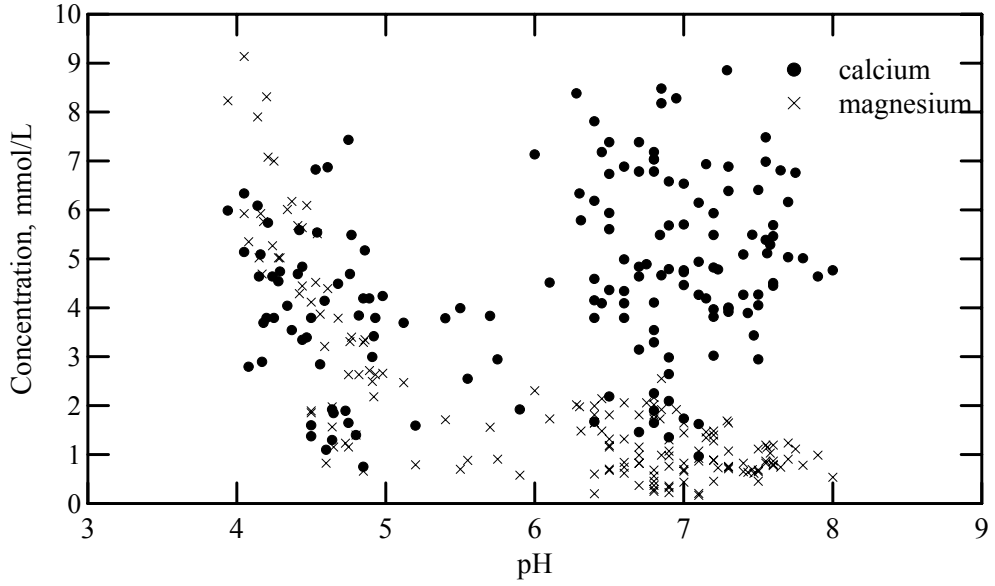


Figure A2.11.5. Calcium and magnesium in drainage from pile FL6 (0.8%S) versus pH.

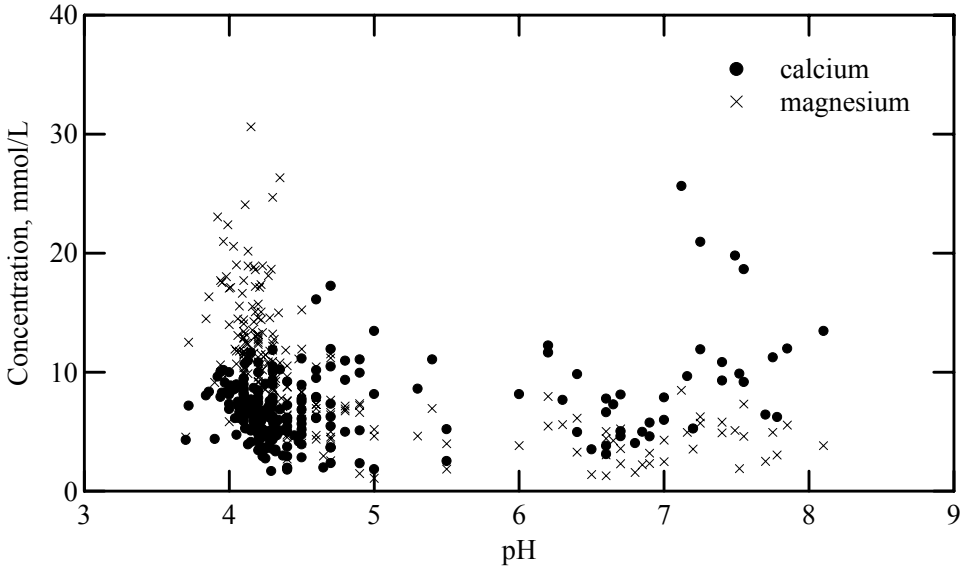


Figure A2.11.6. Calcium and magnesium in drainage from pile FL5 (1.4%S) versus pH.

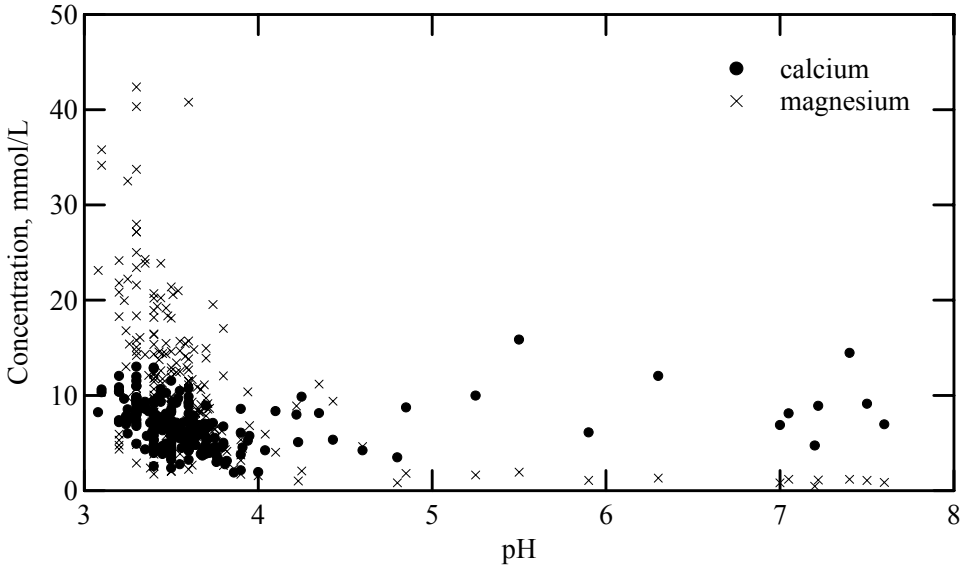


Table A2.12.1. Flow weighted mean concentrations for the 0.6 %S pile, FL1.

Year	Total Volume L	Flow Weighted Mean SO_4	Ca	Mg	Cu	Ni	Co	Zn
		mmol/L						
1978	83100	9.88	5.01	0.70	0.0004	0.0038	0.0003	0.0001
1979	86600	7.74	5.58	0.82	0.0005	0.0056	0.0004	0.0004
1980	53800	12.51	7.92	1.23	0.0007	0.0120	0.0008	0.0007
1981	83800	8.03	5.95	1.11	0.0014	0.0329	0.0016	0.0015
1982	118600	7.86	6.37	0.95	0.0018	0.0737	0.0032	0.0015
1983	82400	10.34	7.73	1.54	0.0039	0.0689	0.0029	0.0020
1984	66879	9.57	6.56	1.69	0.0175	0.2555	0.0114	0.0057
1985	133396	9.15	5.94	1.73	0.0209	0.2447	0.0088	0.0057
1986	119551	7.53	5.50	1.69	0.0447	0.3109	0.0118	0.0062
1987	104768	10.26	6.92	2.19	0.0792	0.3840	0.0135	0.0099
1988	106298	9.14	6.21	2.82	0.1051	0.6386	0.0211	0.0163
1989	122463	10.04	5.66	2.52	0.1278	0.5970	0.0270	0.0184
1990	60101	13.04	6.84	3.88	0.3056	0.9643	0.0469	0.0321
1991	120980	13.04	6.38	4.56	0.3652	1.1568	0.0461	0.0256
1992	103955	9.98	4.95	3.57	0.2980	0.8278	0.0349	0.0192
1993	124968	10.62	4.96	3.63	0.3031	0.8902	0.0333	0.0167

Table A2.12.2. Flow weighted mean concentrations for the 0.6 %S pile, FL2.

Year	Total Volume L	Flow Weighted Mean SO_4	Ca	Mg	Cu	Ni	Co	Zn
		mmol/L						
1978	70200	7.17	3.70	0.81	0.0019	0.0037	0.0004	0.0006
1979	75600	5.83	3.99	0.93	0.0019	0.0046	0.0003	0.0007
1980	29500	7.36	5.02	1.12	0.0028	0.0120	0.0009	0.0018
1981	67200	6.79	3.88	1.21	0.0046	0.0234	0.0015	0.0022
1982	100400	8.69	5.16	1.50	0.0062	0.0468	0.0027	0.0025
1983	52800	8.94	5.53	1.84	0.0193	0.0767	0.0043	0.0049
1984	40431	7.12	4.58	1.71	0.0187	0.0946	0.0049	0.0054
1985	94649	10.02	5.24	2.24	0.0386	0.1545	0.0069	0.0071
1986	69334	6.17	4.12	1.79	0.2277	0.1788	0.0079	0.0061
1987	63221	7.04	4.63	2.15	0.0588	0.2016	0.0093	0.0068
1988	78948	8.63	5.05	3.19	0.0906	0.4967	0.0193	0.0183
1989	80188	7.78	4.07	2.48	0.1208	0.5667	0.0318	0.0254
1990	78458	9.20	4.49	3.10	0.1505	0.5370	0.0300	0.0276
1991	76029	8.98	4.16	3.17	0.1734	0.6326	0.0293	0.0204
1992	51026	8.53	3.56	2.89	0.1555	0.5442	0.0261	0.0177
1993	65210	9.75	3.99	3.88	0.2043	0.7902	0.0342	0.0206

Table A2.12.3. Flow weighted mean concentrations for the 0.6 %S pile, FL3.

Year	Total Volume L	Flow Weighted Mean						
		SO_4	Ca	Mg	Cu	Ni	Co	Zn
		mmol/L						
1978	90800	9.96	7.11	2.14	0.0011	0.0076	0.0010	0.0010
1979	87100	5.80	3.74	1.27	0.0015	0.0151	0.0009	0.0014
1980	58300	10.77	5.25	3.16	0.0205	0.1346	0.0070	0.0076
1981	92400	12.73	5.58	4.52	0.0880	0.4719	0.0233	0.0177
1982	125600	12.95	5.32	4.90	0.1051	0.8594	0.0352	0.0193
1983	68500	16.29	6.33	6.67	0.2167	1.1529	0.0519	0.0380
1984	78088	14.34	5.25	6.31	0.3148	1.3302	0.0668	0.0392
1985	103131	16.85	5.49	6.44	0.4492	1.6140	0.0704	0.0460
1986	89161	12.60	4.63	5.51	0.5103	1.5432	0.0731	0.0388
1987	81412	17.20	5.95	7.98	0.7214	2.0235	0.0841	0.0578
1988	94121	12.75	4.23	5.62	0.5498	1.9666	0.0809	0.0714
1989	94924	11.78	3.82	4.64	0.5452	1.6637	0.0979	0.0701

Table A2.12.4. Flow weighted mean concentrations for the 0.6 %S pile, FL4.

Year	Total Volume L	Flow Weighted Mean						
		SO_4	Ca	Mg	Cu	Ni	Co	Zn
		mmol/L						
1978	92200	6.34	4.06	0.70	0.0003	0.0033	0.0003	0.0011
1979	74000	4.89	3.49	0.62	0.0041	0.0345	0.0010	0.0015
1980	53200	8.35	4.14	0.70	0.0067	0.0483	0.0027	0.0034
1981	109900	7.96	5.22	1.38	0.0103	0.0762	0.0041	0.0047
1982	95165	7.73	5.67	1.56	0.0159	0.1318	0.0068	0.0054
1983	110015	6.75	4.09	1.61	0.0423	0.2306	0.0130	0.0122
1984	91947	8.09	3.87	2.24	0.0941	0.4337	0.0271	0.0182
1985	150757	8.98	4.26	2.84	0.1354	0.5880	0.0298	0.0248
1986	125263	8.88	3.93	3.29	0.2389	0.8741	0.0492	0.0294
1987	106946	12.66	4.55	4.86	0.4193	1.4120	0.0685	0.0465
1988	119294	11.07	3.39	4.93	0.4943	2.0142	0.0961	0.0871
1989	125004	11.97	3.16	4.70	0.5441	1.7038	0.1385	0.0935

Table A2.12.5. Flow weighted mean concentrations for the 0.8 %S pile, FL6.

Year	Total Volume L	Flow Weighted Mean						
		SO_4	Ca	Mg	Cu	Ni	Co	Zn
		mmol/L						
1978	117200	10.88	6.52	3.16	0.0016	0.0164	0.0027	0.0005
1979	118200	12.01	7.28	4.52	0.0307	0.1484	0.0113	0.0060
1980	82300	15.57	6.93	7.07	0.1928	0.5209	0.0360	0.0181
1981	130500	18.23	6.20	8.45	0.5124	1.3955	0.0882	0.0424
1982	177200	19.67	6.72	9.06	0.7012	2.0555	0.0981	0.0429
1983	96000	25.04	7.68	11.83	1.0002	1.8457	0.1224	0.0564
1984	99158	25.70	6.99	12.43	1.1793	2.1712	0.1278	0.0571
1985	148259	26.53	6.74	13.74	1.2510	2.0460	0.1032	0.0540
1986	114434	19.49	6.19	11.28	1.1766	1.7238	0.1003	0.0417
1987	108134	27.22	7.44	14.18	1.4909	1.9802	0.0908	0.0506
1988	117127	22.90	6.75	12.66	1.2931	1.9938	0.0978	0.0751
1989	101975	25.66	6.18	11.58	1.2266	2.0219	0.1243	0.0808
1990	36196	33.65	7.40	15.47	1.3348	1.6896	0.0958	0.0657
1991	106787	28.01	6.21	15.19	1.5436	1.8448	0.0897	0.0590
1992	120552	17.52	4.08	10.52	1.0924	1.1904	0.0629	0.0376
1993	120417	18.84	4.36	10.55	1.1415	1.2964	0.0637	0.0366

Table A2.12.6. Flow weighted mean concentrations for the 1.4 %S pile, FL5.

Year	Total Volume L	Flow Weighted Mean						
		SO_4	Ca	Mg	Cu	Ni	Co	Zn
		mmol/L						
1978	120500	8.97	5.61	1.12	0.0306	0.4055	0.0512	0.0113
1979	117300	22.34	7.10	5.32	0.6040	2.4473	0.1872	0.0719
1980	35700	30.17	5.69	11.26	0.7288	3.7781	0.2585	0.1210
1981	58900	36.30	5.48	12.67	1.1689	4.3258	0.2713	0.1174
1982	105800	39.59	5.62	12.82	1.2764	5.1831	0.3080	0.1205
1983	57200	41.70	6.19	13.66	1.5563	4.5313	0.3153	0.1271
1984	49625	35.63	5.52	11.81	1.2895	3.3643	0.2578	0.0969
1985	83220	40.09	6.60	13.40	1.6158	3.5071	0.2390	0.1062
1986	54523	30.94	6.18	11.24	1.3341	3.1648	0.2530	0.0885
1987	60084	27.25	6.11	9.62	1.2648	2.3704	0.1524	0.0794
1988	70886	24.50	5.97	8.03	1.1512	2.3129	0.1526	0.1090
1989	82485	25.28	5.98	6.90	1.0401	1.8736	0.1948	0.1050
1990	76442	23.56	5.82	6.42	1.0706	1.3991	0.1141	0.0881
1991	70171	20.48	5.04	5.89	0.9416	1.1522	0.0714	0.0534
1992	67278	17.10	4.28	5.01	0.8310	0.9275	0.0614	0.0397
1993	69732	21.63	4.78	5.82	0.9764	1.2519	0.0755	0.0442

APPENDIX 3

MN DNR/BLM TEST PILE SAMPLING AND ANALYTICAL METHODS

Attachment A3.1. Test pile field sampling for the bulk samples.

Attachment A3.2. Details of test pile field sampling for the spatially distributed samples.

Attachment A3.3. Solid phase analytical methods used for the bulk samples.
- A3.3.1. Particle size analysis
- A3.3.2. Percent moisture
- A3.3.3. Specific surface area
- A3.3.4. Sulfur and sulfate
- A3.3.5. Whole rock analysis
- A3.3.6. Loss on ignition
- A3.3.7. Trace metal geochemistry
- A3.3.8. Mineralogy

Attachment A3.4. Solid phase analytical methods used for the spatially distributed samples.
- A3.4.1. Acid concentration present
- A3.4.2. Particle size distribution
- A3.4.3. Chemistry
  - A3.4.3.1. Sample preparation
  - A3.4.3.2. Carbonate and sulfate analysis
  - A3.4.3.3. X-ray fluorescence (XRF) analysis
  - A3.4.3.4. Atomic absorption analysis
  - A3.4.3.5. Inductively coupled plasma analysis (ICP)
  - A3.4.3.6. Silica dioxide
- A3.4.4. Mineralogy

Attachment A3.1. Test pile field sampling for the bulk samples.

The piles were disassembled in horizontal layers or lifts (see details in section A3.2.). During the dismantling of piles FL1, FL6 and FL5, a bulk sample was collected by dragging the 3.5-yard backhoe bucket across each lift. The rock collected, approximately 2 to 2.5 tons per pile, was placed on a plastic liner and assumed to be representative of the pile. A bulk sample was also collected from joint disposal experimental bin 6 using this technique.

Attachment A3.2. Details of test pile field sampling for the spatially distributed samples. Page 1 of 2.

Each of the three piles (FL1, FL6, and FL5) was staked off and four corners were marked. Readings were taken with a Brunton compass in each of the four directions around the pile. These readings were used to locate the pile and correlate the sample locations in space once the pile was no longer present in its original shape. The distances between the survey corners were measured with a 100 foot cloth tape measure. The actual dimensions of the pile were measured using the edges of the Hypalon as markers.

Two stakes were placed at approximately 20 and 40 feet, as measured from the survey corners, from the end that the sampling was performed. These stakes served as reference points for the sample locations on each lift. On each newly exposed surface, the four corners of the sample grid were marked by measuring in from the two stakes.

The first samples were taken on the top of each pile. The original exposed surface was used as the first sample level. Three lifts were removed and samples taken on each of these freshly exposed surfaces. A rope grid system was laid out to determine sample sites. The matrix of sites was determined by the shape of the exposed surface. Attempts were made to collect 16 samples on a surface, a 4×4 grid. This grid was adjusted according the shape of the exposed surfaces. When possible, four corners of the pile were used as four of the sample sites.

The samples were be collected by shovel, and pick when necessary, from an area approximately equal to the diameter of a five gallon bucket and a depth of 8" to 12". Each sample filled a five gallon bucket, unless difficult sampling prohibited this amount of sample. In those cases at least half a bucket was to be filled. Each bucket has a card inside identifying the sample number and any occurrences of note. Any unusual specimens were marked, separated, and placed in either a plastic bag or an envelope.

Sample numbers were assigned using four digits. The first digit was the pile number (1, 5, or 6). The second digit was the lift number (1-4). Lift #1 was the original exposed surface, lift #2 was the first lift removed by the back hoe, lift #3 the second lift removed, and lift #4 the third and final lift. The third and forth digits in the sample number were the number of the sample collected on the lift surface (01 through X), where X is the number of samples collected on that particular face.

Additional sampling occurred. From approximately every other sample site, a sample was collected to measure acid present (acid concentration present). In conjunction with this sampling, the sample hole, the +¼-inch, and the −¼-inch material were described (see following section). At approximately 100 randomly selected sites (every other sample site), among the three piles, a sample was collected for future bacteria testing. As the lifts were being removed, randomly selected portions were collected with the backhoe bucket and set aside. These samples were combined to create a bulk sample that will be assumed representative of the entire pile. The bulk sample was analyzed for particle size, chemistry, mineralogy, and specific surface area.

Attachment A3.2. Details of test pile field sampling for the spatially distributed samples. Page 2 of 2.

Personnel

Monday, October 3, 1994:	Kim Lapakko, Bill White, Dave Antonson, Jennifer Wessels, Cal Jokela
Tuesday, October 4, 1994:	Kim Lapakko, Bill White, Dave Antonson, Jennifer Wessels, Cal Jokela
Wednesday, October 5, 1994:	Kim Lapakko, Bill White, Dave Antonson, Jennifer Wessels
Thursday, October 6, 1994:	Kim Lapakko, Bill White, Dave Antonson, Jennifer Wessels, Cal Jokela, Al Klaysmat, Mike Ellet, Pat Geiselman
Friday, October 7, 1994:	Kim Lapakko, Bill White, Dave Antonson, Cal Jokela, Al Klaysmat, Mike Ellet, Anne Jagunich, Pat Geiselman, Ricco Riihilouma
Saturday, October 8, 1994:	Kim Lapakko, Bill White, Dave Antonson, Mike Ellet, Anne Jagunich, Ricco Riihilouma
Monday, October 10, 1994:	
Tuesday, October 11, 1994:	

Activities

Monday, October 3, 1994:	Bases of the piles were measured and the bearing determined. Stakes were established at several places near the bases of the piles. The relative elevations of the berm at the edge of the piles were determined. Some shallow cuts into the sides of piles FL1 and FL6 were made and examined cursorily.
Tuesday, October 4, 1994:	Sampled FL6, lift 1.
Wednesday, October 5, 1994:	Sampled FL5, lift 1; sampled FL1, lift 1.
Thursday, October 6, 1994:	Sampled FL5, lift 2 and lift 3.
Friday, October 7, 1994:	Sampled FL6, lift 2 and lift 3.
Saturday, October 8, 1994:	Sampled FL1, lift 2 and lift 3; sampled FL5 lift 4.
Monday, October 10, 1994:	Sampled FL6, lift 4.
Tuesday, October 11, 1994:	Sampled FL1, lift 4.

A3.3.1. Particle size analysis

Particle size distribution for the test piles was determined on a composite of samples collected by dragging the 3.5-yard backhoe bucket across each lift. The entire bulk sample from FL5 and FL6 and half of the sample from FL1 were used for determination of particle size. Particle size distribution for the joint disposal bins was determined on a composite of samples collected by random backhoe buckets taken from the surface, center, and base of the of the bin. These samples, approximately one half to one ton in weight, were also placed on a plastic liner and assumed to be representative of the bin. The entire bulk sample from bin 6 was used to determine particle size.

The particle size analysis for the +½-inch fraction was performed on site by the Minnesota Department of Natural Resources, Division of Minerals (Hibbing, MN). Rock larger than twelve inches was presorted and weighed. The remaining rock (−12-inch) was screened through a four-foot by five-foot stationary grizzley with mesh sizes of six and three inches. The +6 and +3-inch rock was weighed and a five gallon representative sample was collected for further analysis. The −3-inch rock was then screened through a Nordbery Universal vibrating screener (type NU9) equipped with screen sizes of ½, ¾, and 1½ inches. Each size fraction was weighed and a representative five gallon sample was collected for further analysis.

A ten gallon representative sample of the -½-inch rock was delivered to Midland Research Center (Nashwauk, MN) for further particle size determination. Approximately one half of the -½-inch fraction, as received, was dried at 105˚ C. The dried portion, about 10 kg, was screened on a ¼-inch (6.4 mm) sieve. A split of the resulting minus ?¼-inch fraction, about 1 kg, was rotapped through the finer sieves (2.0, 0.5, 0.177, 0.149, 0.105, 0.075, 0.053 mm). The weight, in grams, of the +6.4 mm fraction was calculated as the difference in weights of the -1/2-inch fraction and the material finer than 6.4 mm.

Assay pulps were split [method?] from the -6 in and smaller particle size fractions and submitted to Lerch Brothers (Hibbing, Minnesota) for sulfur and sulfate analyses, and to Chemex (Vancouver, Canada) for whole rock and trace metal analyses.

A3.3.2. Percent moisture

Prior to particle size analysis of the -½-inch fraction, Midland Research determined the percent moisture by weight on the split portion of the -½-inch sample (see above). The sample was dried at 105˚C. The percent moisture by weight was calculated as in equation 1.

$$\text{Percent moisture} = (\text{wet weight} - \text{dry weight})/\text{dry weight} \quad (1)$$

A3.3.3. Specific surface area

Specific surface area was analyzed by the Department of Soil, Water, and Climate at the University of Minnesota (Minneapolis, MN) under the direction of Paul Bloom. The method used at the

University was BET nitrogen adsorption. Upon warming, nitrogen adsorbed onto the sample desorbed into the flow stream. A thermal conductivity detector sensed the change in concentration and gave a signal in proportion to the amount of nitrogen held on the surface and then desorbed. The results were given in square meters per gram.

Step by Step Procedure

1. Fill the Dewar with liquid nitrogen to be about 1 inch from the cell holder. Clamp the Dewar to the bar.
2. Switch polarity to adsorb (rezero if necessary). Flow 10% nitrogen through the system until a constant reading is obtained.
3. Immerse the cell in liquid nitrogen in the Dewar. It takes several minutes for the counting to begin. Observe the adsorption signal and use the attenuator to place the signal peak at 50% of the full scale. Use P3 if necessary.
4. After the needle returns to zero when the adsorption is complete, reverse the polarity to desorb and rezero using the fine and coarse zero.
5. Lower the Dewar flask and immerse the cell in a beaker of room temperature water until the float M3 returns to its initial reading prior to the counts (~6.5). When the desorption signal is completed, record the intergrator number.
6. After counting stops and the needle returns to zero, remove the pure nitrogen from the out septum and inject it into the in septum. Record the volume injected and record the intergrator reading. When using large syringes (5 cc), do not inject too rapidly and cause the M3 to move off scale. Use 1 mL when attenuation is on the 32 attenuation. Start with 10% N_2 and work up to 30%, injecting pure nitrogen after each different N_2.

A3.3.4. Sulfur and sulfate

Lerch Brothers analyzed the −6" samples for sulfur and sulfate by particle size. Total sulfur was determined with a carbon rod furnace (Dietert) using ASTM 395-70 and sulfate was determined following a sodium carbonate leach using ASTM D24-92

A3.3.5. Whole rock analysis

Whole rock analysis was performed by Chemex. A prepared 0.100 g sample was added to lithium metaborate flux (0.70 g), mixed well, and fused in a furnace at 1050°C. The resulting melt was then cooled and dissolved in 100 mL of 4% nitric acid. This solution was analyzed by inductively-coupled plasma-atomic emission spectroscopy (ICP-AES). The results were corrected for spectral inter-element interferences. The detection limit was 0.01% for each of the oxides (SiO_2, MgO, Al_2O_3, TiO_2, MnO, P_2O_5, Na_2O, Fe_2O_3, CaO, Cr_2O_3, and K_2O).

A3.3.6. Loss on ignition

Loss on ignition was performed by Chemex. A porcelain crucible was dried in an oven at 105°C, cooled, and the weight recorded. A prepared 1.00 g sample was added to the crucible and then

ashed at 1000°C for one hour. The sample was then cooled in a desiccator, weighed, and percent loss on ignition (L.O.I.) was calculated. The detection limit was 0.01%.

A3.3.7. Trace metal geochemistry

Geochemical analysis of the –6" samples for silver, cobalt, copper, iron, manganese, molybdenum, nickel, lead, and zinc was performed by Chemex. A prepared sample (1.0 g) was digested with concentrated nitric and aqua regia acids at medium heat for two hours. The acid solution was diluted to 25 mL with demineralized water, mixed, and analyzed using an inductively coupled plasma (ICP) spectrometer after calibration with proper standards. The analytical results were corrected for spectral inter-element interferences.

Detection limits for chemical analysis (ICP) performed by Chemex.

Element	Detection limit	Upper limit
silver	0.2 ppm	0.02
cobalt	1 ppm	1
copper	1 ppm	1
iron	0.01%	15
manganese	5 ppm	1
molybdenum	1 ppm	1
nickel	1 ppm	1
lead	2 ppm	1
zinc	2 ppm	1

A3.3.8. Mineralogy

Five size fractions (–6" through -½ ") were delivered to and analyzed by Louis Mattson at Midland Research Center in Nashwauk, MN. The samples were dried and the -½ " fraction of each sample was dry screened into an additional nine size fractions. All size fractions greater than ¼" were crushed to minus ¼". Mineralogical studies were conducted on these size fractions in an attempt to document alteration of primary minerals and identify new minerals formed while the samples were
in the pile. Light and electron microscopy, x-ray diffraction, and thermal analysis were used at various stages. Each size fraction was examined by light microscopy and numerous grains were mounted for electron microscopy analysis. Due to the crushing of the +¼" fraction, many obvious alteration features were destroyed or reduced to very fine grained material.

Attachment A3.4. Solid phase analytical methods used for the spatially distributed samples. Page 1 of 7.

A3.4.1. Acid Concentration Present

Laboratory test version: The acid-concentration present (ACP) method was designed by Newmont Metallurgical Services (Bucknam, 1993, personal communication) to quantify the amount of acid produced by hydrated iron-sulfate salts (e.g. melanterite) present as coatings on the surfaces of mine waste materials (equations 1-4).

$$FeSO_4 \cdot 7H_2O(s) = Fe^{2+}(aq) + 7H_2O \quad (1)$$

$$Fe^{2+}(aq) + (¼)O_2(g) + H^+(aq) = Fe^{3+}(aq) + (½)H_2O \quad (2)$$

$$Fe^{3+}(aq) + 3H_2O = Fe(OH)_3(s) + 3H^+(aq) \quad (3)$$

The net result of equations 1 through 3 is summarized in equation 4 which shows a net production of two moles of acid produced for each mole of melanterite dissolved.

$$FeSO_4 \cdot 7H_2O(s) + (¼)O_2(g) = Fe(OH)_3(s) + SO_4^{2-}(aq) + (9/2)H_2O + 2H^+(aq) \quad (4)$$

Twenty grams of mine waste sample, pulverized to 80 percent 150 mesh, is mixed with 50 mL of deionized water and agitated on an orbital shaker for 20 minutes. The resulting pulp (comprised of a solid-to-water ratio of 1:2.5) is then filtered using a 0.45 micron filter, and the filtrate is titrated to a pH 8.3 endpoint using a standardized base. The resulting amount of base consumed in the titration is converted to parts per million (ppm) acid (expressed as calcium carbonate equivalent) per gram of mine waste sample.

Field test version: Recently, Newmont's laboratory method was modified for field use by USBM so that same day acid concentration present estimates could be made on-site rather than waiting several days for laboratory results. Use of a commercially available field titration kit (the LaMotte Acidity Titration Kit) has been combined with a modification of the laboratory method so that the contribution of hydrated iron-sulfide slats contained in the minus ¼-inch fraction of mine waste dump and tailings can be measured and mapped on site. Mine waste solids are screened through a ¼-inch (6.3 mm) sieve. The +¼-inch fraction is examined, and the contained primary and secondary minerals are visually described and quantified. Once examined, the +¼-inch material is either rejected or saved as a separate sample.

Because the USBM modification was designed to be used as a quick field estimate of ACP during time constrained activities as exploratory and blast-hole drilling projects, the selected sample mass was volumetrically estimated using a wide-mouthed 30 to 60 mL Nalgene bottle. The Nalgene bottle is marked at a level above its bottom that represents the average volume occupied by several 20-gram splits from the minus ¼-inch mine waste sample being tested. (Specific gravities of the selected samples ranged from 2.5 to 3.1, and the variability in sample mass per equivalent volume did not differ by more than 10 percent from the intended 20 grams.) A 250 mL Nalgene bottle

similarly marked at the level that represents the 50 mL volume of deionized water. This bottle is filled to the appropriate level with deionized water, and the volumetrically determined 20 gram mass of mine waste is carefully added. The resulting pulp contained in the 250 mL bottle is hand shaken for a one minute period (or approximately 100 shakes) and then allowed to sit for 15 minutes. This process is repeated again. The bottle is shaken for a third time and then allowed to sit for at least 5 minutes to settle to suspended solids. At this time an aliquot was withdrawn and the pH of the solution was determined.

After the 5 minute settling period, withdraw a second 5 mL aliquot of the liquid with a disposable pipet (careful to sample only from the top of the liquid so as not to stir the solids). Add the following reagents to the 5 mL aliquot: one drop of sodium thiosulfate, and one drop of phenolpthalein indicator (contained in the LaMotte Acidity Kit), and mix. Use the direct reading titrator and titer (also contained in the LaMotte Acidity Kit) and titrate to the phenolpthalein end point, which is reached when the solution turns red (the phenolpthalein end point is approximately pH 4.8). Record the value from the direct reading titrator as total acidity in ppm $CaCO_3$, once the end point is reached.

An approximate sample mass of 20 grams can be obtained from minus ¼-inch mine waste material by filling a 30 mL Nalgene bottle up to a mark located 5/8 of an inch from the bottom. Tap the bottle lightly and adjust the amount of the sample so that the sample is level with the mark. (This works within about 10 percent error for minus ¼-inch material with a specific gravity of around 2.8.)

Data obtained using this field test are as follows:

		Total acidity (ppm $CaCO_3$)			
Total test before settle time (minutes)	Number of times shaken	Settling time			
		two minutes	five minutes	ten minutes	overnight
0	1	92	64	64	96
30	3	120	80	80	96
60	5	96	80	76	–

The 30 minute test time was selected as the shortest and most accurate within reasonable limitations.

The laboratory version of the ACP test was performed on this same material using a shaker table and an automatic titrator. The results are given below for comparison.

Laboratory ACP Method	Cold water addition			Hot water addition	
	20 minutes	2 hours	weekend	20 minutes	2 hours
Total acidity (ppm $CaCO_3$)	65.1	78.3	120.1	101.3	112.0

The time given in the table above is the amount of time that the sample was on the shaker table at 150 rpm. The terms "hot" and "cold" represent the temperature of the water added to the sample, 85°C (boiling temperature at the altitude at USBM Salt Lake City) and room temperature, respectively.

The 30 minute test and 5 minute settle time of the field test procedure is off by only 2.1 percent in comparison with the laboratory 2 hour cold test.

Additional comments about this test: The normality of the base given in the LaMotte Acidity Test Kit is 0.02. If the ACP of the test material is about 1000 ppm $CaCO_3$ (0.10 percent $CaCO_3$) or higher, a solution of 0.1 N NaOH is recommended. If a stronger solution of NaOH is used, the method for calculating the ppm $CaCO_3$ is as follows:

$$\text{ppm } CaCO_3 = \frac{N_b \times (1000 \text{ mg/g}) \times (\text{equivalent wt } CaCO_3) \times V_b}{\text{Sample size (mL)}}$$

where

N_b = normality of the base used
V_b = volume of the base used
equivalent wt $CaCO_3$ = 50.04 grams per equivalent
sample size = volume (mL) of sample titrated

Simplified it becomes:

$$\text{ppm } CaCO_3 = \frac{50{,}040 \times N_b \times V_b}{\text{sample volume (mL)}}$$

The direct reading titrator holds 1 mL when filled to the 200 ppm level.

The LaMotte Acidity Test Kit also comes with a methyl orange indicator as well as the phenolpthalein indicator. The methyl orange acidity (MOA) gives the ppm $CaCO_3$ at a pH of 3.7. It may be difficult or impossible to use this indicator due to the cloudy solution that is used in the test. The methyl orange indicator is orange below a pH of 3.7 and is yellow above a pH of 3.7. For the sample used above, the solution was already yellow-orange, thus the MOA was masked and a change of color was not noticed.

Field Version of the ACP Test: Step by Step Procedure

1. Screen material to be tested through a ¼ inch screen.
2. With the 30 mL Nalgene bottle scoop up the minus ¼ inch material and tap it, so that it is level with the line drawn around the bottle.
3. Fill the 250 mL Nalgene bottle to the 50 mL line with deionized water.
4. Dump the material in the 30 mL bottle into the 250 mL with the water.
5. Cap the 250 mL bottle tightly and shake for 1 minute or 150 times.
6. Let the mixture set for 15 minutes.
7. Shake the mixture for 1 minute or 150 times.
8. Let the mixture set for 15 minutes.
9. Shake the mixture again for 1 minute or 150 times.
10. Let the mixture set for 5 minutes.
11. Using the disposable pipet, withdraw enough sample to fill the glass bottle to the 5 mL mark.
12. Add one drop each of sodium thiosulfate and phenolpthalein to the 5 mL.
13. Using the direct reading titrator, titrate the solution until it turns red.
14. Record the value from the direct reading titrator as the total acidity.

A3.4.2. Particle size distribution

Sixteen samples were sent to the U.S. Bureau of Mines Salt Lake City Research Center (Salt Lake City, Utah; USBM). The corresponding sample numbers were: 6-1-02, 6-1-04, 6-1-05, 6-1-07, 6-2-02, 6-2-04, 6-2-07, 6-2-10, 6-3-01, 6-3-06, 6-3-08, 6-3-16, 6-4-02, 6-4-04, 6-4-05, and 6-4-07. All sixteen samples were initially screened into fourteen size fractions: +6", −6"/+3", −3"/ +1½", −1½"/+¾", −¾"/+½", −½"/+¼", −¼"/+10 mesh, −10/+30 mesh, −30/+70 mesh, −70/+100 mesh, −100/+150 mesh, −150/+200 mesh, −200/+270 mesh, and −270 mesh.

The −1½"/+¾ to −½/ +¼" size fractions were split by hand, and an inventory of all of the sample splits was conducted.

A3.4.3. Chemistry

A3.4.3.1. Sample preparation

Screen sizes from −¼"/+10 mesh to −100/+150 mesh were split into two parts. One half of the each sample was split on the rotary splitter into twelve approximately equal parts. Thirty to fifty grams of this portion was pulverized to 80% −150 mesh screen using the puck and ring pulverizer. Five to ten grams of this material was then submitted to the USBM analytical lab for characterization and chemical analysis. The specific analyses requested were 1) trace metals (Co, Cu, K, Na, Ni, Pb, and Zn), and 2) sulfate, 3) carbon sulfur, 4)XRD, and 5)XRF.

Size fractions –150/+200 mesh to –270 mesh were not halved and were directly split into twelve approximately equal parts on the rotary splitter. These samples did not require pulverization and the amount of sample was relatively small. Five to ten gram samples of these splits were submitted for analysis as noted above.

A3.4.3.2. Carbonate and sulfate analysis

Total carbon and total sulfur analyses were conducted by the U.S. Bureau of Mines, Salt Lake City, Utah. These analyses were determined by a combustion-infrared spectrophotometer (LECO furnace). Sample duplicates were subjected to partial-decomposition procedures, including pyrolysis (muffle roast at 550°C) and acid decomposition (either hydrochloric or nitric acid). The muffle-furnace roast drove off sulfide sulfur as sulfur dioxide, and organic carbon as carbon dioxide. The pretreated sample was analyzed for carbon and sulfur, and these results were subtracted from the total carbon and sulfur results of a non-treated sample split. The remaining sulfur determined in the pretreated split was assumed to be sulfate sulfur, and the remaining carbon was considered to be carbonate.

A3.4.3.3. X-ray Fluorescence (XRF) analysis

X-ray fluorescence (XRF) analysis was conducted by the U.S. Bureau of Mines, Salt Lake City, Utah to determine S, Ca, K, Al, Si, Fe, Mn, Ti, Ba, Cr, Ni, Cu, Ca, Zn, Rb, Sr, Y, Zr, Pb, La, and Ce. The 80% percent –150 mesh samples were ground using a mortar and pestle to –325 mesh. The resulting powder was placed in the cylindrical mount. [XRF Instrument?] Elements below sodium on the periodic chart can not be determined by XRF and such may only be estimates.

A3.4.3.4. Atomic Absorption Analysis

The digestion of the samples prior to chemical analysis was performed by the U.S. Bureau of Mines Salt Lake City Research Center using EPA method 3051, Hazardous Waste Digestion, for multi-element, solid samples. A sample of 0.50 gram weight was placed in a CEM microwave vessel. Ten mL of trace metal grade HNO_3 was added and microwaved for 6 minutes. The temperature should rise to 75° C within 5.5 minutes, and remain between 170-180° C for 10 minutes. The pressure should peak at 88.2 psi (6 atm). The samples were cooled and poured into 100 mL flasks.

The particle size fractions of the sixteen samples were analyzed by Atomic Adsorption (AA) analysis by the U.S. Bureau of Mines, Salt Lake City, Utah under the direction of Bill White. The samples were analyzed for cobalt, copper, potassium, sodium, nickel, lead, and zinc using the following EPA procedure. A 0.50 gram sample was combined with deionized water and nitric acid. The mixture was warmed in a water bath and more nitric acid was added. After cooling the samples, deionized water and hydrogen peroxide were added and the sample was again warmed. Water and hydrochloric acid were added and the resulting solution was then diluted. The solution was analyzed by AA, with results reported in percent.

EPA Procedure for AA elements except Hg and Ag (solids)

1. Place a 5.0 g sample in an acid washed 100.0 mL volumetric flask.
2. Add 5 mL deionized water and 5 mL HNO_3 (trace metal grade).
3. Heat in 95° C water bath for 10 minutes then add 5 mL HNO_3 (trace metal grade) while in bath and then continue heating for 30 minutes longer.
4. Cool the samples in the hood.
5. Add 2 mL deionized water and 3 mL H_2O_2.
6. Carefully heat the samples again in the water bath for 10 minutes. Do not let effervescence foam over and out of the flask.
7. If digesting arsenic or selenium, the procedure is finished -- dilute to volume.
8. If not digesting arsenic, add 3 mL deionized water and 3 mL HCl.
9. Continue heating for 10 minutes and then cool and dilute to volume.

NOTE: This method requires a duplicate, triplicate, and two standard reference materials NBS2704 and ERA211.

A3.4.3.5. Inductively Coupled Plasma analysis

Solid-phase sample dissolution was accomplished using the following alkaline fusion method. A 0.5 g sample was pulverized to 80% passing 150 mesh. Sodium carbonate, sodium peroxide, and sodium hydroxide were added, the mixture fused at 650°C for one hour, and then cooled. Deionized water, hydrochloric acid, and nitric acid were added. The resulting solution was boiled, hydrofluoric acid added, and the solution decanted. Deionized water was added to obtain a solution volume of 250 mL. This solution was submitted for analysis by ICP.

The particle size fractions of the sixteen samples were analyzed by Inductively Coupled Plasma (ICP) analysis for an additional suite of metals by the U.S. Bureau of Mines, Salt Lake City, Utah under the direction of Bill White. The samples were analyzed for aluminum, boron, barium, calcium, cadmium, cobalt, chromium, copper, iron, magnesium, manganese, molybdenum, nickel, lead, and zinc.

USBM Solids Dissolution Method for ICP Analysis: Alkaline Fusion

1. This alkaline fusion methods is performed on a 0.5 g aliquot of the sample of interest that has been pulverized to 80% passing 150 mesh (100 micrometers).
2. The following reagents are added to the 0.5 g sample in a zirconium crucible: 1) sodium carbonate -- two 1-mL scoops, 2) sodium peroxide -- one 1-mL scoop, and 3) sodium hydroxide -- 2 to 4 pellets (to lower flux melting point to 650° C)
3. Mix sample with reagents by rolling crucible. Cover mix with two 1-mL scoops of sodium peroxide.
4. Place loaded zirconium crucible in a muffle furnace that has been pre-heated to 650° C and fuse for one hour.
5. At completion of one hour fusion, remove crucible and cool until fused mix solidifies.

6. Place crucible on its side in a Teflon beaker and add sufficient volume of deionized (DI) water to cover half of crucible mouth. To the deionized water add 20 mL concentrated hydrochloric acid and 10 mL concentrated nitric acid.
7. Place beaker containing crucible and acid-water mix on hot plate, bring solution to a boil, and remove from hot plate.
8. Add two drops of 48% hydrochloric acid.
9. Transfer solution to 250 mL Nalgene volumetric flask; rinse Teflon beaker with deionized water, into 250 mL volumetric flask. Bring up to 250 mL volume with deionized water addition.

A3.4.3.6. Silica dioxide

A gravimetric method was used on the particle size fractions of the sixteen samples. The analyses were conducted by U.S. Bureau of Mines, Salt Lake City, Utah. The sample was digested and the remaining solid was roasted at 900°C. The resulting sample weight was subtracted from the pre-roasted weight. The difference was the amount of silicon dioxide.

Gravimetric Method for Determination of SiO_2

1. Aqua-regia-perchloric digestion of solid sample is performed to solubilize everything except the SiO_2; the mix is then filtered and resulting filter-cake is retained.
2. Filter-cake is transferred to tarred platinum crucible and weighed to obtain pre-roasted filter-cake weight; crucible with sample is placed in furnace and gradually brought up to 900°C: sample is maintained at 900°C for 1 hour and subsequently removed and cooled.
3. Cooled sample in platinum crucible is weighed and resulting weight is subtracted from pre-roasted filter-cake weight to obtain SiO_2 weight by difference.
4. If roasted residue is discolored (should be white), then a hydrofluoric acid (HF) correction is performed as follows: approximately 25% HF and several drops of concentrated sulfuric acid are added to the crucible containing the roasted sample residue which is then heated on a hot plate; the HF reacts with SiO_2 to form fluorosilic acid and is subsequently fumed off; tarred crucible and any residue are then weighed and corrected SiO_2 weight is determined by difference.

A3.4.4. Mineralogy

X-ray diffraction (XRD) analysis was conducted by the U.S. Bureau of Mines, Salt Lake City, Utah. The 80% percent −150 mesh samples were ground using a mortar and pestle to −325 mesh. The resulting powder was pressed in the mount. The mount was analyzed by a Phillips diffraction system. The results were presented as a list of dominant mineral species, with mineral formula and percent in the sample.

APPENDIX 4

BULK SAMPLE ANALYSIS

Table A4.1. Particle size distribution of the bulk samples.

Table A4.2. Particle size distribution (in pounds) of the bulk samples.

Table A4.3. Percent moisture by weight of the −½" fraction of the bulk samples.

Table A4.4. Specific surface area of particle size fractions of bulk samples.

Table A4.5. Sulfur and sulfate content of bulk sample particle size fractions.

Table A4.6. Whole rock chemistry and mass weighted averages for particle size fractions of the 0.6% S bulk sample.

Table A4.7. Whole rock chemistry and mass weighted averages for particle size fractions of the 0.8% S bulk sample.

Table A4.8. Whole rock chemistry and mass weighted averages for particle size fractions of the 1.4% S bulk sample.

Table A4.9. Whole rock chemistry and mass weighted averages for particle size fractions of the 0.6% 6B bin sample.

Table A4.10. Trace metal geochemistry and mass weighted averages for particle size fractions of the 0.6% bulk sample.

Table A4.11. Trace metal geochemistry and mass weighted averages for particle size fractions of the 0.8% bulk sample.

Table A4.12. Trace metal geochemistry and mass weighted averages for particle size fractions of the 1.4% bulk sample.

Table A4.13. Trace metal geochemistry and mass weighted averages for particle size fractions of the 6B bulk sample.

Table A4.14. Selected chemistry and mineralogy for -6/+1½-inch fractions of bulk samples: 0.6 %S, sample FL1.

Table A4.15. Selected chemistry and mineralogy for -6/+1½-inch fractions of bulk samples: 0.6 %S, sample FL1.

Table A4.16. Selected chemistry and mineralogy for -6/+1½-inch fractions of bulk samples: 0.6 %S, sample FL1.

Attachment A4.1. Babbit Bulk Samples: Report from Midland Research Center regarding Amax Test Pile Bulk Samples, FL1, FL5, FL6, 6B, February 26, 1996

Table A4.1. Particle size distribution (in weight percent retained, normalized) of the bulk samples. Analysis of the +½ inch by MN DNR in Hibbing, MN and the −½ inch (after drying) by Midland Research Center in Nashwauk, MN.

Particle Size		0.6% S	0.8% S	1.4%S	Bin 6B
min	max				
12"	---	0	0	1.3[1]	0
6"	12"	3.4	5.9	7.1[2]	0
3"	6"	16.3	13.5	18.4	16.2
1½”	3"	29.0	27.3	28.8	31.0
¾”	1½”	18.8	19.2	17.5	20.0
½”	¾”	12.0	20.3	12.8	7.9
6.4 mm	½”	1.6	2.1	3.1	2.8
2.0 mm	6.4 mm	7.6	6.1	4.9	8.0
0.50 mm	2.0 mm	5.0	2.2	2.5	5.9
0.177 mm	0.50 mm	2.7	1.4	1.7	4.0
0.149 mm	0.177 mm	0.4	0.2	0.3	0.7
0.105 mm	0.149 mm	0.4	0.2	0.2	0.6
0.075 mm	0.105 mm	0.6	0.3	0.4	0.8
0.053 mm	0.075 mm	0.5	0.3	0.3	0.6
---	0.053 mm	1.4	0.7	0.7	1.5

[1] one cemented agglomerate
[2] 2.6% associated with cemented agglomerates

Table A4.2. Particle size distribution (in pounds) of the bulk samples. Analysis by Department of Natural Resources, Minerals Division (Hibbing, Minnesota).

Sample ID	Particle Size (inches)							
	+12	−12/+6	−6/+3	−3/+1½	−1½/+¾	−¾/+½	−½	Total
	Weight (pounds)							
0.6 %S[1]	0	63	299	531	345	219	374	1831
0.8 %S	0	288	661	1335	939	993	673	4889
1.4 %S[2]	61[4]	321[5]	833	1303	793	578	640	4529
1.4 %S[3]	0	201	833	1303	793	578	640	4348
Bunker 4F[1]	0	0	27	146	143	71	173	560
Bunker 4B[1]	0	0	45	146	134	82	163	570
Bunker 6F[1]	0	0	74	362	293	211	605	1545
Bunker 6B[1]	0	0	338	648	419	166	520	2091
	percent							
0.6 %S	0	3.4	16.3	29.0	18.8	12.0	20.4	99.9
0.8 %S	0	5.9	13.5	27.3	19.2	20.3	13.8	100
1.4 %S	1.3	7.1	18.4	28.8	17.5	12.8	14.1	100
1.4 %S	0	4.6	19.2	30.0	18.2	13.3	14.7	100
Bunker 4F	0	0	4.8	26.1	25.5	12.7	30.9	100
Bunker 4B	0	0	7.9	25.6	23.5	14.4	28.6	100
Bunker 6F	0	0	4.8	23.4	19.0	13.7	39.2	100.1
Bunker 6B	0	0	16.2	31.0	20.0	7.9	24.9	100
FL-4	11	6.8	14.5	33.9		6.7	27.1	100

[1] For FL-1 and the bunker samples, only half the sample collected was used.
[2] including agglomerates
[3] excluding agglomerates
[4] one cemented "particle"
[5] 120 lbs. associated with agglomerates

Table A4.3. Percent moisture by weight of the −½" fraction of the bulk samples. Analysis by Midland Research Center.

Sample	Moisture[1] (weight percent)
0.6% S	15.82
0.8% S	6.70
1.4% S	6.22
6B (FL4)	2.68

[1] Moisture (weight %) = (wet weight − dry weight)/dry weight

Table A4.4. Specific surface area of particle size fractions of the bulk samples.

Percent Sulfur	Particle Size (mm)	Specific Surface Area 30% N_2[1] (m^2/g)	Specific Surface Area 10, 20, 30% N_2 (m^2/g)	r^2
0.6	−2.0/+0.50	0.73	0.691	0.999
	−0.50/+0.177	1.53	1.55	0.999
	−0.177/+0.149	1.99	2.09	0.999
	−0.149/+0.105	2.19	2.20	1.0
	−0.105/+0.075	2.35	2.35	0.999
	−0.075/+0.053	4.34	4.28	0.997
	−0.053	10.67	12.0	0.997
0.8	−2.0/+0.50	2.52	2.56	0.997
	−0.50/+0.177	3.23	3.35	0.999
	−0.177/+0.149	3.90	−−	−−
	−0.149/+0.105	3.08	3.15	0.999
	−0.105/+0.075	4.47	−−	−−
	−0.075/+0.053	5.59	−−	−−
	−0.053	11.2	11.0	1.0
1.4	−2.0/+0.50	2.28	2.37	1.0
	−0.50/+0.177	3.07	3.09	0.999
	−0.177/+0.149	4.11	4.15	1.0
	−0.149/+0.105	4.76	4.90	0.999
	−0.105/+0.075	7.92	−−	−−
	−0.075/+0.053	8.04	−−	−−
	−0.053	15.9	15.69	0.996
6B FL4	−2.0/+0.50	2.54	2.49	0.999
	−0.50/+0.177	3.44	−−	−−
	−0.177/+0.149	5.08[2]	5.07[2]	0.998[2]
	−0.149/+0.105	4.40[3]	4.40[3]	1.0[3]
	−0.105/+0.075	6.32	−−	−−
	−0.075/+0.053	10.8	−−	−−
	−0.053	15.7	15.9	1.0

1 Use of 30% N_2 for single point can account for up to 96% of three point value if specific surface area is greater than 1 m^2/g.

2 One additional analysis for this sample. Results as follows: 30% N_2 = 5.25 m^2/g; 10, 20, 30% N_2 = 5.27; r^2 = 0.995.

3 Two additional analyses for this sample. Results as follows: second analysis 30% N_2 = 4.46; 10, 20, 30% N_2 = 4.39; r^2 = 1.0; third analysis 30% N_2 = 4.35; 10, 20, 30% N_2 = 4.39; r^2 = 0.992.

Table A4.5. Sulfur and sulfate content of bulk sample particle size fractions. Analysis by Lerch Brothers. Page 1 of 2.

Sample	Size Fraction (mm)[1]	$S_{(total)}$	SO_4 as S
0.6% S	−6"/+1½"	0.45	0.02
	−1½"/+½"	0.58	0.02
	−½"/+2.0	0.44	0.01
	−2.0/+0.50	0.45	0.04
	−0.50/+0.177	0.58	0.10
	−0.177/+0.149	0.70	0.17
	−0.149/+0.105	0.64	0.14
	−0.105/+0.075	0.87	0.15
	−0.075/+0.053	0.99	0.09
	−0.053	1.30	0.17
0.8% S	−6"/+1½"	0.60	0.04
	−1½"/+½"	0.74	0.09
	−½"/+2.0	1.13	0.06
	−2.0/+0.50	1.22	0.08
	−0.50/+0.177	1.34	0.28
	−0.177/+0.149	1.20	0.18
	−0.149/+0.105	0.99	0.15
	−0.105/+0.075	1.11	0.14
	−0.075/+0.053	1.26	0.12
	−0.053	1.51	0.25

[1] Unless otherwise indicated.

Table A4.5. Sulfur and sulfate content of bulk sample particle size fractions. Analysis by Lerch Brothers. Page 2 of 2.

Sample	Size Fraction (mm)[1]	$S_{(total)}$	SO_4 as S
1.4% S	−6"/+1½"	0.74	0.01
	−1½"/+½"	1.03	0.17
	−½"/+2.0	0.80	0.24
	−2.0/+0.50	2.10	0.40
	−0.50/+0.177	2.06	0.80
	−0.177/+0.149	1.82	0.98
	−0.149/+0.105	1.46	0.73
	−0.105/+0.075	1.59	0.80
	−0.075/+0.053	1.65	0.95
	−0.053	1.88	1.07
6B	−6"/+1½"	0.95	0.04
	−1½"/+½"	0.62	0.06
	−½"/+2.0	0.62	0.06
	−2.0/+0.50	0.56	0.07
	−0.50/+0.177	0.43	0.07
	−0.177/+0.149	0.43	0.09
	−0.149/+0.105	0.45	0.09
	−0.105/+0.075	0.54	0.12
	−0.075/+0.053	0.58	0.13
	−0.053	0.68	0.23

[1] Unless otherwise indicated.

Table A4.6. Whole rock chemistry and mass weighted averages for particle size fractions of the 0.6% S bulk sample (in percent). Sulfur and sulfate analysis by Lerch Brothers. Analysis by Chemex Labs.

Size Fraction (mm)[1]	S_t	SO_4-S	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO_2	LOI[2]	mass %
−6"/+1½"	0.45	0.02	17.69	8.24	0.03	14.73	0.43	7.55	0.15	2.38	0.20	44.57	2.42	<0.01	45.3
−1½"/+½"	0.58	0.02	17.26	8.16	0.03	15.03	0.49	7.84	0.16	2.36	0.19	45.11	2.48	0.43	30.8
−½"/+2.0	0.44	0.01	17.59	8.46	0.03	14.35	0.48	7.73	0.15	2.37	0.20	45.44	2.14	0.48	9.2
−2.0/+0.50	0.45	0.04	18.30	8.48	0.03	13.38	0.52	7.22	0.14	2.61	0.16	46.19	1.93	0.68	5.0
−0.50/+0.177	0.58	0.10	17.46	8.03	0.04	14.80	0.48	7.48	0.15	2.46	0.16	44.86	2.80	0.96	2.7
−0.177/+0.149	0.70	0.17	16.37	7.67	0.04	15.90	0.44	7.82	0.16	2.28	0.22	43.27	3.19	1.14	0.4
−0.149/+0.105	0.64	0.14	16.44	7.70	0.05	15.79	0.45	7.83	0.16	2.27	0.25	43.54	3.12	1.21	0.4
−0.105/+0.075	0.87	0.15	16.35	7.75	0.06	15.83	0.47	7.79	0.16	2.25	0.33	43.21	2.99	1.43	0.6
−0.075/+0.053	0.99	0.09	16.02	7.63	0.06	16.19	0.49	7.66	0.15	2.16	0.36	42.25	2.89	1.82	0.5
−0.053	1.30	0.17	15.85	7.12	0.06	17.08	0.52	7.24	0.14	2.06	0.39	40.68	2.56	3.49	1.4
mass weighted avg of the −6" fraction[3]	0.51	0.03	17.51	7.91	0.03	14.78	0.46	7.64	0.15	2.38	0.20	44.83	2.41	0.33	96.3

[1] Unless otherwise indicated.
[2] LOI = Loss on ignition
[3] Assays not available for 3.4 wt. % +6" material.

Table A4.7. Whole rock chemistry and mass weighted averages for particle size fractions of the 0.8% S bulk sample (in percent). Sulfur and sulfate analysis by Lerch Brothers. Analysis by Chemex Labs.

Size Fraction (mm)[1]	S_t	SO_4-S	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO_2	LOI[2]	Total
−6"/+1½"	0.60	0.04	14.58	8.86	0.04	15.38	0.52	9.74	0.16	2.00	0.25	45.05	1.65	0.26	40.8
−1½"/+½"	0.74	0.09	14.53	8.56	0.04	15.53	0.46	10.24	0.17	1.93	0.22	44.59	1.37	0.77	39.5
−½"/+2.0	1.13	0.06	14.63	8.41	0.04	15.82	0.47	10.13	0.17	1.92	0.24	44.23	1.18	1.19	8.2
−2.0/+0.50	1.22	0.08	14.30	7.68	0.04	15.64	0.59	9.54	0.16	2.05	0.21	45.16	1.21	2.07	2.2
−0.50/+0.177	1.34	0.28	13.73	7.61	0.04	16.31	0.53	9.69	0.16	1.92	0.19	44.07	1.37	2.45	1.4
−0.177/+0.149	1.20	0.18	13.74	7.31	0.04	17.32	0.52	10.31	0.17	1.89	0.23	42.77	1.51	2.02	0.2
−0.149/+0.105	0.99	0.15	14.24	7.30	0.04	16.61	0.54	10.12	0.16	1.99	0.25	43.37	1.38	1.80	0.2
−0.105/+0.075	1.11	0.14	14.31	7.21	0.03	17.02	0.57	10.04	0.16	1.95	0.26	43.30	1.38	2.20	0.3
−0.075/+0.053	1.26	0.12	13.44	6.89	0.03	18.62	0.53	10.37	0.17	1.82	0.32	41.96	1.86	2.38	0.3
−0.053	1.51	0.25	13.26	6.73	0.03	18.80	0.50	9.73	0.15	1.66	0.30	40.55	1.50	4.79	0.7
mass weighted avg of the −6" fraction[3]	0.74	0.07	14.53	8.62	0.04	15.56	0.49	9.98	0.17	1.96	0.24	44.71	1.47	0.69	93.8

[1] Unless otherwise indicated.
[2] LOI = Loss on ignition
[3] Assays not available for 5.9 wt. % +6" material.

Table A4.8. Whole rock chemistry and mass weighted averages for particle size fractions of the 1.4% S bulk sample (in percent). Sulfur and sulfate analysis by Lerch Brothers. Analysis by Chemex Labs.

Size Fraction (mm)[1]	S_t	SO_4-S	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO_2	LOI[2]	Total
−6"/+1½"	0.74	0.01	16.41	6.45	0.06	16.20	0.97	6.38	0.16	2.11	0.42	46.75	2.60	0.34	47.2
−1½"/+½"	1.03	0.17	16.27	6.82	0.07	16.30	0.92	6.31	0.16	1.98	0.36	46.19	2.37	1.14	30.3
−½"/+2.0	0.80	0.24	16.07	6.29	0.07	16.75	1.03	5.95	0.15	1.99	0.38	45.57	2.20	2.54	8.0
−2.0/+0.50	2.10	0.40	15.18	5.46	0.06	17.95	1.01	5.23	0.13	2.19	0.34	45.10	1.97	4.44	2.5
−0.50/+0.177	2.06	0.80	14.95	6.06	0.05	17.24	1.03	5.23	0.13	2.19	0.29	44.62	1.67	5.69	1.7
−0.177/+0.149	1.82	0.98	15.17	6.42	0.06	17.15	1.00	5.49	0.13	2.14	0.29	43.65	1.78	5.61	0.3
−0.149/+0.105	1.46	0.73	15.24	6.18	0.06	17.26	1.00	5.69	0.13	2.16	0.30	44.24	2.04	4.47	0.2
−0.105/+0.075	1.59	0.80	14.46	6.08	0.06	18.16	1.00	5.98	0.14	2.01	0.32	42.87	2.42	5.01	0.4
−0.075/+0.053	1.65	0.95	14.06	6.05	0.06	18.72	1.00	5.96	0.14	1.83	0.34	41.69	2.55	5.95	0.3
−0.053	1.88	1.07	13.06	5.39	0.06	21.76	0.95	5.27	0.12	1.60	0.51	38.16	2.40	8.16	0.7
mass weighted avg of the −6" fraction[3]	0.92	0.12	16.22	6.51	0.06	16.41	0.96	6.25	0.16	2.05	0.39	46.26	2.45	1.13	91.6

[1] Unless otherwise indicated.
[2] LOI = Loss on ignition
[3] Assays not available for 8.4 wt. % +6" material.

Table A4.9. Whole rock chemistry and mass weighted averages for particle size fractions of the 0.6% S 6B bin sample (in percent). Sulfur and sulfate analysis by Lerch Brothers. Analysis by Chemex Labs.

Size Fraction (mm)[1]	S_t	SO_4-S	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO_2	LOI[2]	Total
−6"/+1½"	0.95	0.04	15.23	7.10	0.04	18.05	0.58	9.43	0.18	2.26	0.24	43.73	1.60	0.18	47.2
−1½"/+½"	0.62	0.06	16.89	7.99	0.04	15.20	0.54	8.12	0.16	2.50	0.22	45.00	1.65	0.28	27.9
−½"/+2.0	0.62	0.06	17.25	8.17	0.03	15.02	0.56	7.93	0.15	2.60	0.21	45.60	1.41	0.65	10.8
−2.0/+0.50	0.56	0.07	17.73	8.08	0.03	14.00	0.60	7.29	0.14	2.73	0.18	46.54	1.38	0.99	5.9
−0.50/+0.177	0.43	0.07	16.61	7.30	0.04	15.65	0.58	7.73	0.15	2.63	0.19	45.38	2.26	0.98	4.0
−0.177/+0.149	0.43	0.09	16.13	7.12	0.04	16.58	0.56	8.08	0.16	2.45	0.20	44.23	2.31	1.42	0.7
−0.149/+0.105	0.45	0.09	16.31	7.24	0.04	16.30	0.56	8.00	0.15	2.48	0.33	44.42	2.13	1.61	0.6
−0.105/+0.075	0.54	0.12	15.96	7.10	0.04	16.79	0.58	8.10	0.15	2.33	0.31	42.73	2.21	2.02	0.8
−0.075/+0.053	0.58	0.13	15.73	7.12	0.04	17.03	0.55	8.07	0.15	2.31	0.41	43.33	2.09	2.47	0.6
−0.053	0.68	0.23	15.29	6.48	0.03	18.46	0.55	7.70	0.14	2.08	0.41	40.87	1.91	4.99	1.5
mass weighted avg of the −6" fraction	0.76	0.06	16.14	7.52	0.04	16.56	0.57	8.65	0.17	2.41	0.23	44.47	1.63	0.46	100

[1] Unless otherwise indicated.
[2] LOI = Loss on ignition

Table A4.10. Trace metal geochemistry and mass weighted averages for particle size fractions of the 0.6% S bulk sample. Analysis by Chemex Labs.

Size Fraction (mm)[1]	Ag (ppm)	Co (ppm)	Cu (ppm)	Fe (%)	Mn (ppm)	Mo (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
−6"/+1½"	1.0	51	2390	5.37	490	<1	507	4	70
−1½"/+½"	1.0	56	3090	5.42	480	<1	611	4	74
−½"/+2.0	1.5	53	2900	5.45	500	<1	545	2	72
−2.0/+0.50	1.0	47	2920	5.20	470	<1	488	36	72
−0.50/+0.177	1.0	49	3110	5.46	485	<1	538	6	74
−0.177/+0.149	1.0	57	3770	6.04	525	<1	678	10	86
−0.149/+0.105	1.0	59	3750	5.92	515	<1	743	8	84
−0.105/+0.075	1.0	68	4320	6.09	505	<1	989	6	92
−0.075/+0.053	1.5	83	5190	6.50	505	<1	1380	12	104
−0.053	3.5	94	7140	7.69	480	1	1660	20	164
mass weighted avg of the −6" fraction	1.1	54	2817	5.44	487	0.5[2]	570	6	74

[1] Unless otherwise indicated.
[2] Less than detection limit values assumed to be half of the detection limit for this calculation.

Table A4.11. Trace metal geochemistry and mass weighted averages for particle size fractions of the 0.8% S bulk sample. Analysis by Chemex Labs.

Size Fraction (mm)[1]	Ag (ppm)	Co (ppm)	Cu (ppm)	Fe (%)	Mn (ppm)	Mo (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
−6"/+1½"	1.5	63	3370	6.23	630	<1	570	18	74
−1½"/+½"	1.0	69	3860	6.57	640	<1	695	12	76
−½"/+2.0	1.5	82	5060	7.31	665	<1	1085	4	78
−2.0/+0.50	3.0	76	6320	7.40	625	<1	1005	4	96
−0.50/+0.177	1.5	74	6620	7.66	635	<1	942	4	80
−0.177/+0.149	1.5	79	6540	8.02	690	<1	954	6	82
−0.149/+0.105	1.5	81	6150	7.97	690	<1	963	4	84
−0.105/+0.075	1.5	87	6740	8.13	675	<1	1100	<2	86
−0.075/+0.053	2.5	101	8100	8.43	680	<1	1405	2	94
−0.053	4.5	102	9260	9.10	615	<1	1595	14	132
mass weighted avg of the −6" fraction	1.4	68	3924	6.56	638	0.5[2]	697	14[2]	76

[1] Unless otherwise indicated.
[2] Less than detection limit values assumed to be half of the detection limit for this calculation.

Table A4.12. Trace metal geochemistry and mass weighted averages for particle size fractions of the 1.4% S bulk sample. Analysis by Chemex Labs.

Size Fraction (mm)[1]	Ag (ppm)	Co (ppm)	Cu (ppm)	Fe (%)	Mn (ppm)	Mo (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
−6"/+1½"	0.5	35	1455	2.75	120	2	315	2	58
−1½"/+½"	0.5	45	2200	3.61	160	2	422	20	56
−½"/+2.0	0.5	32	2470	4.26	160	2	288	2	48
−2.0/+0.50	1.0	42	3350	6.00	160	2	416	22	48
−0.50/+0.177	1.0	36	3260	5.74	150	2	385	6	50
−0.177/+0.149	0.5	38	2910	5.53	140	1	411	4	56
−0.149/+0.105	1.0	41	2810	5.31	140	<1	463	4	54
−0.105/+0.075	1.0	47	2900	5.35	130	2	549	8	56
−0.075/+0.053	1.0	48	2930	5.83	130	3	554	8	56
−0.053	2.0	44	3870	8.84	135	4	506	14	66
mass weighted avg of the −6" fraction	0.5	38	1913	3.39	139	2[2]	356	9	56

[1] Unless otherwise indicated.
[2] Less than detection limit values assumed to be half of the detection limit for this calculation.

Table A4.13. Trace metal geochemistry and mass weighted averages for particle size fractions of the 6B bulk sample. Analysis by Chemex Labs.

Size Fraction (mm)[1]	Ag (ppm)	Co (ppm)	Cu (ppm)	Fe (%)	Mn (ppm)	Mo (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
−6"/+1½"	1.5	86	4050	7.89	675	<1	767	6	96
−1½"/+½"	1.0	62	2940	6.59	600	<1	538	<2	80
−½"/+2.0	1.0	54	3030	6.65	585	<1	415	<2	78
−2.0/+0.50	1.0	43	3280	6.35	550	<1	316	2	72
−0.50/+0.177	1.0	45	3180	6.98	600	<1	294	4	76
−0.177/+0.149	1.0	46	3350	7.36	615	<1	319	<2	82
−0.149/+0.105	1.5	47	3650	7.47	620	<1	337	4	86
−0.105/+0.075	1.5	47	3930	7.38	590	<1	351	4	86
−0.075/+0.053	2.0	48	4550	7.95	605	<1	393	12	118
−0.053	3.0	51	5310	9.35	545	<1	527	16	160
mass weighted avg of the −6" fraction	1.3	70	3564	7.38	630	0.5	605	4	88

[1] Unless otherwise indicated.
[2] Less than detection limit values assumed to be half of the detection limit for this calculation.

Attachment A4.1

Babbit Bulk Samples

Report from Midland Research Center
Regarding Amax Test Pile Bulk Samples
FL1, FL5, FL6, 6B

February 26, 1996

MIDLAND RESEARCH CENTER

A WHOLLY OWNED SUBSIDIARY OF MIDLAND STANDARD INCORPORATED
POST OFFICE BOX 67
NASHWAUK, MN 55769-0067
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February 28, 1996

MAR 4 1996 RECEIVED Div. of Minerals St. Paul

Mr. Kim Lapakko
Minnesota DNR-Division of Minerals
500 Lafayette Road
St. Paul, MN 55155-4045

RE: Babbitt Bulk Samples.

Dear Kim:

Five size fractions (-6" through -1/2") of Babbitt Bulk Samples FL-1, FL-5, FL-6, and 6B were delivered to Midland Research Center. The samples were dried and the -1/2" portions of each were dry screened into an additional 9 size fractions (6.4, 2.0, 0.5, 0.177, 0.149, 0.105, 0.075, 0.053 and -0.053 mm). All size fractions greater than 1/4" (6.4 mm) were crushed to minus 1/4". Assay pulps were split from and prepared for each size fraction and submitted to Chemex Labs for whole rock and trace metal assays and to Lerch Brothers for sulfur and sulfate assays. Screen structures and all assay data were reported previously. Portions from each of the minus 2.0 mm fractions from the 4 bulk samples were also submitted to Dr. Paul Bloom, University of Minnesota, for specific surface area measurements. That data was reported directly to you by Dr. Bloom.

Mineralogical studies were conducted on the size fractions of Bulk Samples FL-1, FL-5, and FL-6 in an attempt to document alteration of primary minerals and identify new minerals formed during alteration. Light and electron microscopy, X-ray diffraction, and thermal analysis were utilized at various stages of the study. Each size fraction was examined by light microscopy and numerous grains exhibiting potential alteration features were picked and mounted for electron microscopy-microanalysis. All of the coarser +1/4" (+6.4 mm) size fractions had been crushed to -1/4" for assay pulp splitting prior to examination and many of the obvious alteration features destroyed or reduced to very fine grained material.

In the following sections and photo descriptions minerals may be referred to as primary, secondary, or tertiary. Primary minerals are those that were present in the bulk samples when they were placed in the surface piles. Secondary minerals are those that formed by chemical reactions within the piles and tertiary minerals are those formed during drying of material removed

from the piles. The distinction between secondary and tertiary is obviously not always clearly evident. Gypsum, for example, occurs as relatively coarse secondary crystals (blades) intergrown with other secondary minerals and also as very fine sprays of tertiary crystals on the surface of particles. There are, however, numerous examples of intermediate sized gypsum crystals on the surface of secondary minerals that may or may not be tertiary.

MINERAL BALANCE

Review of the assay data and observations of the size fractions indicate that the coarsest sizes are the least altered and the finest sizes the most altered. To determine the mineralogical composition of the original unaltered rock in each bulk sample a mineral balance was determined for the slightly altered -6+11/2" fraction of each sample. The -6+11/2" chemistry and mineralogy, along with the total -6" chemistry for comparison, are shown in attached tables I, II, and III. The total -6" and -6+11/2" chemistry are generally very similar and reflect the relatively intense alteration in the finest fractions that constitute only a small percentage of the total bulk samples.

ALTERATION INDEX

Review of the whole rock assay data by size fraction for each of the samples indicates elemental variation related to particle size. The amount of SiO_2, Al_2O_3, CaO, and Na_2O all decrease with finer particle sizes and the Fe_2O_3 and LOI values increase with finer sizes. The other whole rock elements tend to remain relatively constant for all size fractions.

To provide a measure of alteration and perhaps determine which size particles are most susceptible to alteration a chemical index of alteration (CIA) and a simple "ratio" of increasing to decreasing elements were calculated for each size fraction.

$$CIA = (Al_2O_3/(Al_2O_3+CaO+Na_2O+K_2O))X100$$

$$\text{"Ratio"} -((Fe_2O_3+LOI)/(SiO_2+Al_2O_3+CaO+Na_2O))X100$$

ALTERATION INDEX

	FL-1		FL-5		FL-6	
Size	CIA	"ratio"	CIA	"ratio"	CIA	"ratio"
-6+11/2"	61.6	20.2	63.3	23.1	56.2	22.2
-11/2"+1/2"	61.1	21.2	62.6	24.5	57.0	23.4
-1/2+2.0 mm	60.9	20.1	63.3	27.6	57.5	24.6
-2.0+0.50	61.2	18.6	63.7	33.0	57.6	25.8
-0.50+0.177	61.4	21.6	61.7	33.8	57.2	27.8
-0.177+0.149	61.2	24.5	61.3	33.8	58.6	29.4
-0.149+0.105	61.2	24.3	62.0	32.0	59.2	27.8
-0.105+0.075	61.0	24.8	61.4	35.4	59.5	28.8
-0.075+0.053	60.9	26.5	61.3	38.8	59.3	32.8
-0.053 mm	62.0	31.3	62.2	51.4	59.9	37.9

The CIA index (Nesbitt & Young, Nature, Vol 299, Oct. 1982, P. 715) is essentially a measure of feldspar decomposition where increased alteration equals higher values, i.e., alkali element loss

and retention of Al_2O_3 as kaolinite. The nearly constant CIA values relative to particle size suggests either minimal feldspar alteration or nearly equal alumina and alkali loss from the bulk sample stock piles. Alumina was apparently not retained as secondary kaolinite and kaolinite was not detected in any of the XRD patterns. The CIA values for FL-6 exhibit a slight increase with finer particle size and may suggest some feldspar alteration in that sample. Assuming that no dissolved Na_2O is retained as secondary minerals, the less than 3% Na_2O overall loss can probably be accounted for by dissolution of the low Na ferromagnesium minerals.

Optically the ferromagnesium minerals are the most altered of the silicates and appear to be the source for much of the common hydrous iron oxides and montmorillonite group clay minerals. The "ratio" alteration index probably reflects this alteration and shows significant increases in values (greater alteration) toward the finer sizes. The "ratio" values for all 3 samples are lowest for the coarse sizes, exhibit higher but plateaued values for mid-sizes, and show a strong increase for the finest (-0.075 mm/-200 M) sizes.

MONTMORILLONITE-GROUP CLAYS

Montmorillonite group clays are one of the dominant secondary alteration products. XRD studies of the clays in the -0.053 mm (-270 mesh) fractions from all 3 samples indicate the clays are all expandable montmorillonite group clays. Mixed layer clays were not detected. Attempts were made to quantify the clay content in several size fractions from each sample using thermal gravimetric analysis (TGA), but were unsuccessful due to overlapping weight loss from other minerals. Relative montmorillonite contents, based on XRD peak intensity are listed below:

Montmorillonite XRD 14.5dA° Peak Intensity (CPS)			
Size	FL-1	FL-5	FL-6
-6+11/2" (-152.4+38.1 mm)	20	20	30
Total -1/2" (-12.7 mm)	130	55	120
-270 Mesh (-0.053 mm)	465	195	480

XRD peak intensity indicates that the finest fraction (-270m/-0.053 mm) contains from 10 to 20 times more montmorillonite than do the coarsest (-6+11/2") fractions.

Photos 1 and 2 show an orthopyroxene grain from FL-6 partly altered to montmorillinite-group clay minerals. Several EDS spectra suggest the clay is mixed composition; part Fe-rich member nontronite, along with other lower Fe/higher Ca and high Mg varieties are suggested. This was the only grain observed altering directly to montmorillonite and the bulk elemental ratio in the clay was similar to the host orthopyroxene. The CaO content of clinopyroxene is much too high for direct alteration to montmorillonite and most altered clinopyroxene observed exhibit leaching and solution along cleavage planes. See photo 3. None of the trace metals (Ni, Cu, Co, etc.) were detected in any of the montmorillonite examined.

OTHER SILICATES, ETC.

All of the observations in this study are on loose grains which limits detection of phases to surface features only. Thin sections and/or polished sections of altered rock fragments should be considered for future studies so that altered grains might be viewed in cross-section.

Olivine was observed only as unaltered grains from all 3 samples. Olivine should be the least stable of the silicate minerals and suspectable to early alteration. Biotite grains occur in various stages of alteration and should be the major source for any potassium (K) that might enter into jarosite formation. Jarosite was not detected either by XRD or SEM/EDS. Biotite alteration, where distinguishable as a discrete phase, appears to be iron oxides devoid of K.

Feldspars, as suggested by the CIA index, may not be highly altered and loose grain observations generally support that possibility. Photos 5 and 11 illustrate typical feldspar grains observed. Alteration, where present, appears to be some leaching along cleavage planes.

Amphiboles, both single crystal and fibrous, were observed in all 3 bulk samples. The amphiboles are a late stage primary alteration of the original rock and both types exhibit secondary alteration that occurred in the surface piles. Photo 6 shows a fibrous amphibole that appears to be simply "falling apart". No discrete alteration phases were detected except for a patch of montmorillonite that may or may not have formed in place.

Photo 4 shows an apatite crystal typical of those observed in samples FL-1 and FL-6. None of the apatite shows evidence of alteration or leaching.

SULFIDES

Primary pyrrhotite is readily susceptible to alteration and most of the pyrrhotite in the minus 0.50 mm fractions of these samples is highly altered. The only completely unaltered pyrrhotite observed was in the crushed portions of the coarser fractions. Photo 7 shows the remnants of a pyrrhotite grain, now iron oxides with a trace of sulfur, surrounded by hydrous iron oxide, probably ferrihydrite ($5Fe_2O_3 \cdot 9H_2O$) with its typical "mud-crack" texture.

Chalcopyrite is less susceptible to alteration than pyrrhotite and relatively unaltered grains were observed in all size fractions. The altered grains typically exhibit leaching along cleavage planes and apparent removal of leach products. See photo 8.

One single grain of pentlandite was observed and by SEM/EDS appeared to be partly altered to violarite, Ni_2FeS_4.

ALTERATION PRODUCTS

The alteration products observed, in addition to montmorillonite, are mainly a variety of hydrous iron oxides, various sulfates, some sulfides, and some carbonates. The only alteration products positively identified by XRD are montmorillonite and gypsum. A number of small samples containing concentrations of alteration products were selectively removed from several larger rock fragments, XRD patterns were run, and only gypsum peaks were clearly indicated. The identification of general mineral type in the photos is by light and electron microscopy. A number of common alteration products that might be expected are listed below:

Typical Alteration Minerals	
Goethite	$\propto FeO(OH)$
Lepidocrocite	$FeO(OH)$
Ferrihydrite	$5Fe_2O_3 \cdot 9H_2O$
Gypsum	$CaSO_4 \cdot 2H_2O$
Jarosite	$KFe_3(SO_4)_2(OH)_6$
Melanterite	$FeSO_4 \cdot 7H_2O$
Siderotil	$FeSO_4 \cdot 5H_2O$
Covellite	CuS
Chalcocite	Cu_2S
Malachite	$Cu_2(OH)_2(CO_5)$

Typical hydrous iron oxides observed are shown in photos 7, 9, 10 and 11. Gypsum, both secondary and tertiary, is shown in photos 1,5,12 and possibly 16. Iron sulfates appear in photos 10 and 14. Copper sulfides are shown in photos 13 and 14. Copper carbonate, malachite, is in photo 15 and possible copper sulfate is in photo 16.

When I looked at a number of your bulk sample hand specimens at the Hibbing DNR office on February 22, 1996, I picked and mounted a number of additional alteration grains for SEM examination. A number of these are good examples of products that were very difficult to pick from crushed and fine screen size fractions. The mounts will be examined and a short additional report issued.

Sincerely,

MIDLAND RESEARCH CENTER

Louis Mattson

Louis A. Mattson
Senior Technical Consultant

cc: File

Table II

BULK SAMPLE FL-5

SELECTED CHEMISTRY AND MINERALOGY

Chemical Assays		
Elements	Total -6"*	-6+11/2"
SiO_2	46.26	46.75
TiO	2.45	2.60
Al_2O_3	16.23	16.41
Cr_2O_3	0.06	0.06
Fe_2O_3**	16.41	16.20
MnO	0.16	0.16
MgO	6.25	6.38
CaO	6.51	6.45
Na_2O	2.05	2.11
K_2O	0.96	0.97
P_2O_5	0.39	0.42
LOI	1.13	0.34
S	0.92	0.74
SO_4 as S	0.13%	0.01%
Ag	0.5 ppm	0.5 ppm
Co	35	35
Cu	1912	1455
Fe	33900	27500
Mn	139	120
Mo	2	2
Ni	356	315
Pb	9	2
Zn	56 ppm	58 ppm

-6+11/2"Mineralogy	
Mineral	Wt. %
Feldspar	41.8
Olivine	1.6
Clinopyroxene	11.2
Orthopyroxene	4.8
Biotite	9.6
Alteration***	14.5
Ilmenite	4.2
Magnetite	5.2
Apatite	1.0
Quartz	4.2
Cordierite	<0.1
Calcite	<0.1
Pyrrhotite (+Pyrite)	1.38
Chalcopyrite	0.39
Cubanite	0.05
Pentlandite	0.09
Cobaltite	0.01
Sphalerite	0.01
Galena	<0.01
Gypsum	0.05

* Assays not available for 4.6 Wt. % +6" material.

** All Fe as Fe_2O_3.

*** Alteration is mainly primary amphiboles with serpentine, chlorite and minor secondary montmorillonite.

Table I

BULK SAMPLE FL-1

SELECTED CHEMISTRY AND MINERALOGY

Chemical Assays		
Elements	Total -6"*	-6+11/2"
SiO_2	44.83%	44.57%
TiO	2.41	2.42
Al_2O_3	17.51	17.69
Cr_2O_3	0.03	0.03
Fe_2O_3**	14.78	14.73
MnO	0.15	0.15
MgO	7.64	7.55
CaO	8.21	8.24
Na_2O	2.38	2.38
K_2O	0.46	0.43
P_2O_5	0.20	0.20
LOI	0.33	<0.01
S	0.51	0.45
SO_4 as S	0.03%	0.02%
Ag	1.1 ppm	1.0 ppm
Co	53	51
Cu	2820	2390
Fe	54400	53700
Mn	487	490
Mo	<1	<1
Ni	570	507
Pb	6	4
Zn	74 ppm	70 ppm

-6+11/2"Mineralogy	
Mineral	Wt. %
Feldspar	51.9
Olivine	13.5
Clinopyroxene	12.7
Orthopyroxene	4.4
Biotite	2.0
Alteration***	7.1
Ilmenite	4.4
Magnetite	2.2
Apatite	0.5
Quartz	<0.1
Cordierite	<0.1
Calcite	<0.1
Pyrrhotite (+Pyrite)	0.38
Chalcopyrite	0.66
Cubanite	0.04
Pentlandite	0.15
Cobaltite	0.01
Sphalerite	0.01
Galena	<0.01
Gypsum	0.11

* Assays not available for 3.5 Wt. % +6" material.

** All Fe as Fe_2O_3.

*** Alteration is mainly primary amphiboles with serpentine, chlorite and minor secondary montmorillonite.

Table III

BULK SAMPLE FL-6

SELECTED CHEMISTRY AND MINERALOGY

Chemical Assays		
Elements	Total -6"*	-6+11/2"
SiO_2	44.71	45.05
TiO	1.47	1.65
Al_2O_3	14.53	14.58
Cr_2O_3	0.04	0.04
Fe_2O_3**	15.56	15.38
MnO	0.17	0.18
MgO	9.98	9.74
CaO	8.62	8.86
Na_2O	1.96	2.00
K_2O	0.49	0.52
P_2O_5	0.24	0.25
LOI	0.89	0.26
S	0.74	0.60
SO_4 as S	0.07%	0.04%
Ag	1.3 ppm	1.5 ppm
Co	68	63
Cu	3931	3370
Fe	65600	62300
Mn	638	630
Mo	<1	<1
Ni	698	570
Pb	13	18
Zn	76 ppm	74 ppm

-6+11/2"Mineralogy	
Mineral	Wt. %
Feldspar	44.5
Olivine	18.4
Clinopyroxene	20.9
Orthopyroxene	3.2
Biotite	3.3
Alteration***	4.7
Ilmenite	2.8
Magnetite	<0.1
Apatite	0.6
Quartz	<0.1
Cordierite	<0.1
Calcite	<0.1
Pyrrhotite (+Pyrite)	0.38
Chalcopyrite	0.93
Cubanite	0.06
Pentlandite	0.17
Cobaltite	0.02
Sphalerite	0.01
Galena	<0.01
Gypsum	0.22

* Assays not available for 4.6 Wt. % +6" material.

** All Fe as Fe_2O_3.

*** Alteration is mainly primary amphiboles with serpentine, chlorite and minor secondary montmorillonite.

Photo 1: FL-6, Orthopyroxene altered (left end) to montmorillonite-group clay minerals. Note fine tertiary (?) gypsum needles, upper right. Magnification 120X, scale bar equals 100 microns.

Photo 2: FL-6, Montmorillonite-group clay minerals from orthopyroxene in photo 1 above. EDS spectra suggest mixed composition clay; much probably nontronite with other lower Fe- higher Ca varieties. Magnification 1000X, scale bar equals 10 microns.

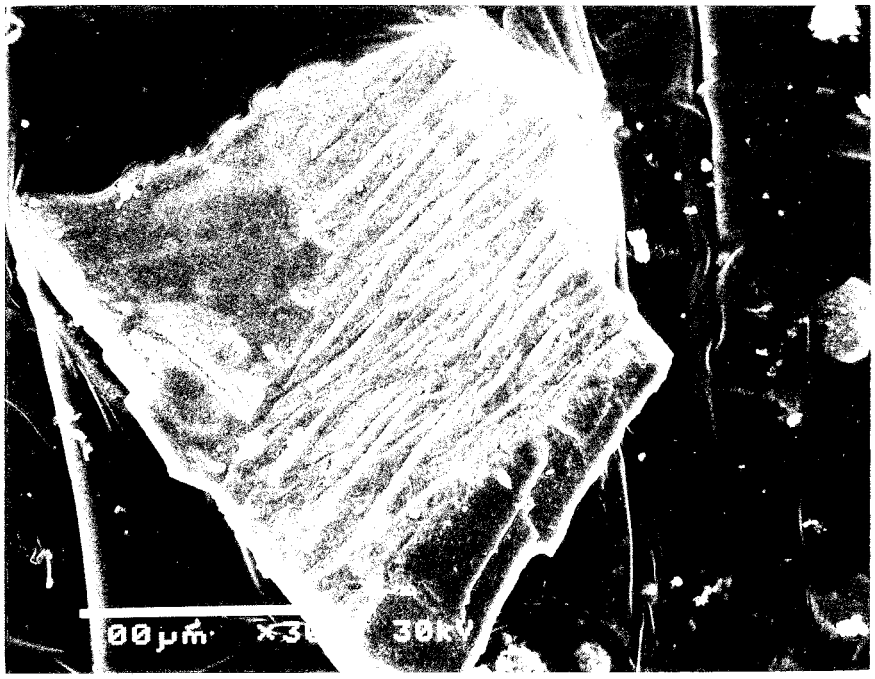


Photo 3: FL-6, Clinopyroxene showing leaching and solution along cleavage planes. Note unaltered surfaces to left and lower right. Magnification 300X, scale bar equals 100 microns.

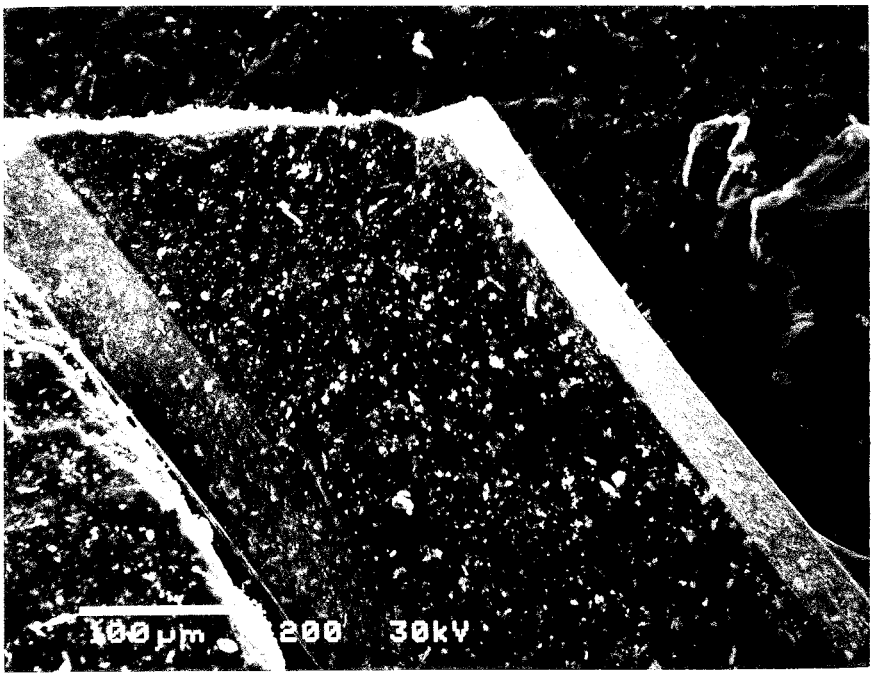


Photo 4: FL-1, Unaltered apatite crystal. These samples were all dry sieved and fines remain on the surface of the apatite. Altered apatite was not observed. Magnification 200X, scale bar equals 100 microns.

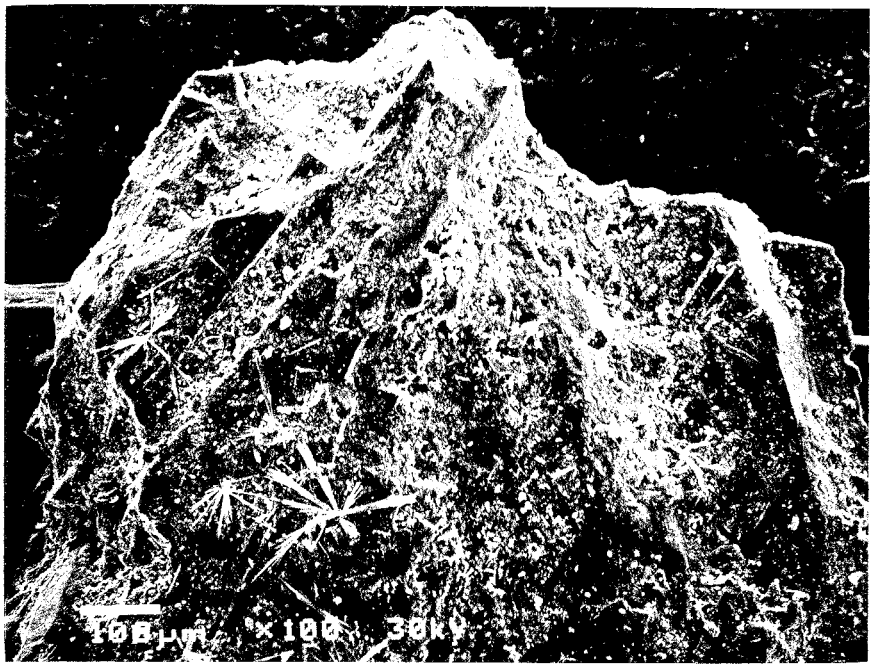


Photo 5: FL-5, Plagioclase feldspar coated with alteration products, mostly iron oxides. Note tertiary gypsum sprays on surface. The only altered area observed is the lath-like area on the left that appears selectively leached and filled with alteration rubble. Magnification 100X, scale bar equals 100 microns.

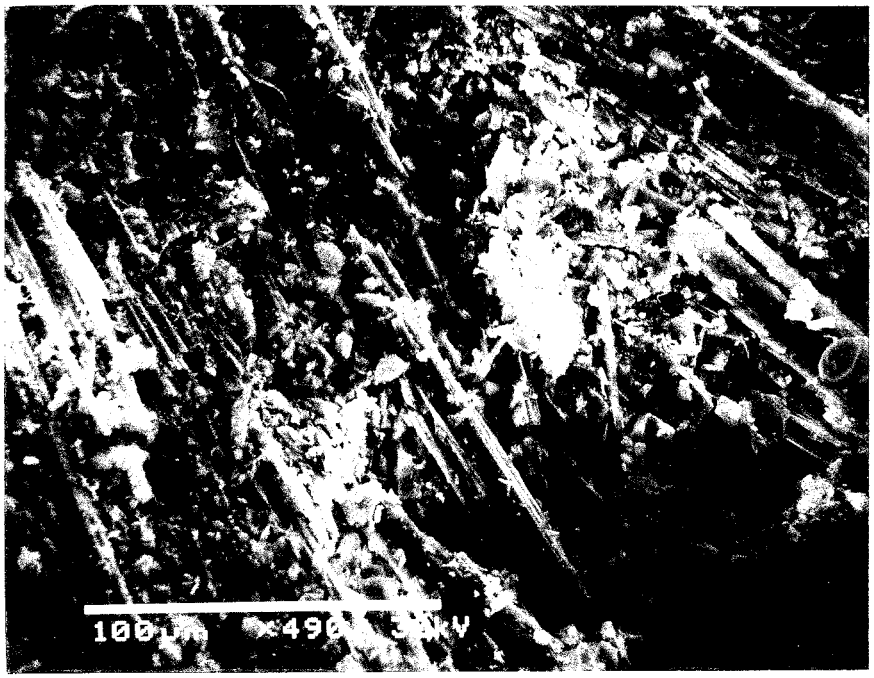


Photo 6: FL-1, Altered fibrous amphibole. The light colored patch in the center right appears to be montmorillonite. Other alteration phases were not defined. Magnification 490X, scale bar equals 100 microns.

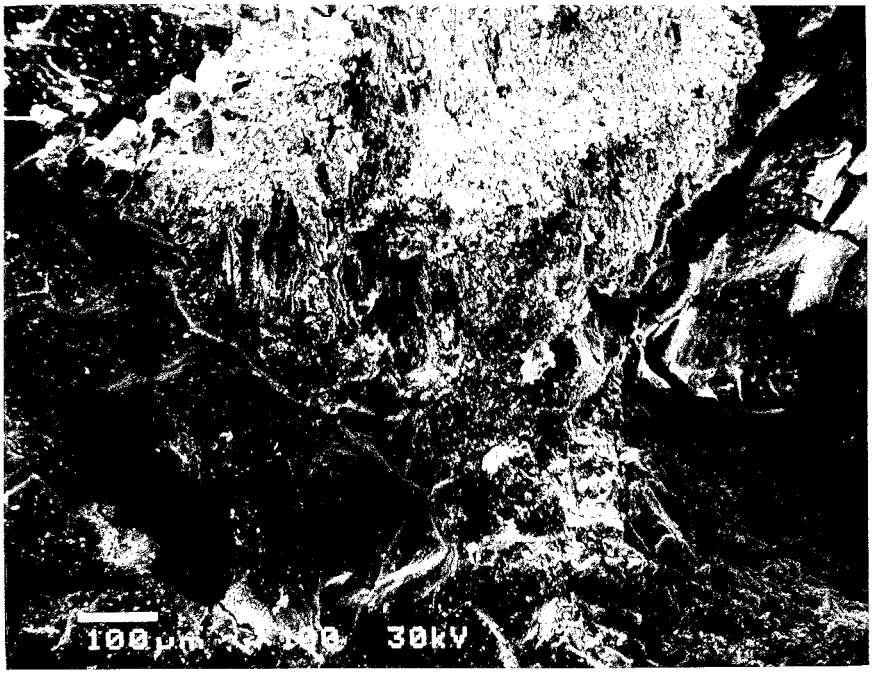


Photo 7: FL-1, Highly altered pyrrhotite (light colored, upper center) surrounded by hydrous iron oxide, possibly ferrihydrite ($5Fe_2O_3 \cdot 9H_2O$). Pyrrhotite grain is now iron oxides with a trace of sulfur. Magnification 100X, scale bar equals 100 microns.

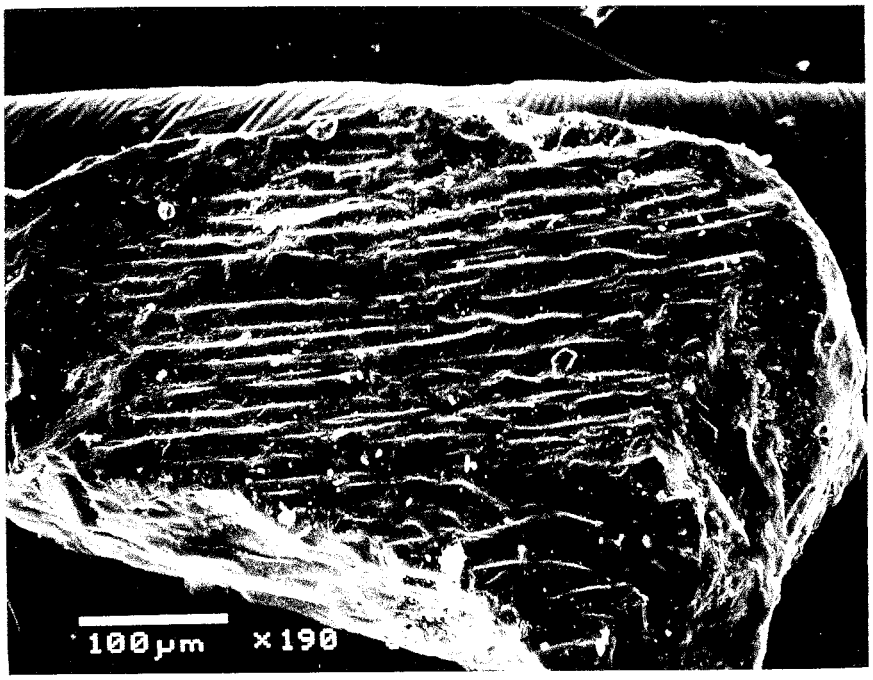


Photo 8: FL-6, Chalcopyrite, with some intergrown cubanite, showing significant leaching along crystal planes. Alteration products were not detected on the surface. Magnification 190X, scale bar equals 100 microns.

Photo 9: FL-6, Alteration products consisting mainly of hydrous iron oxides in a variety of forms. SEM/EDS failed to detect any chemical difference between forms. Magnification 200X, scale bar equals 100 microns.

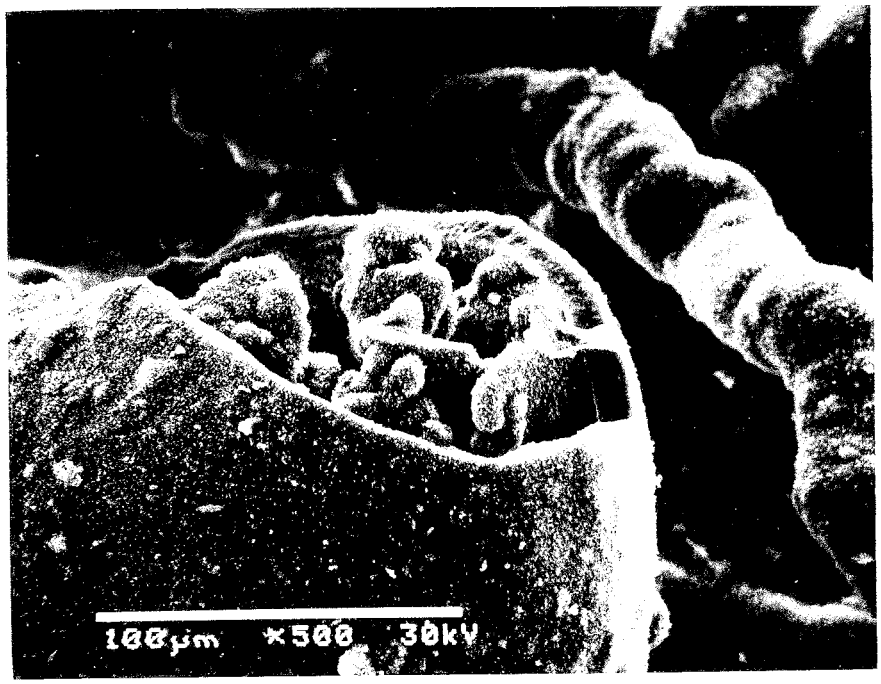


Photo 10: FL-6, Broken bubble with interior crystals and root-like forms from above photo 9. The very fine crystals coating all surfaces, except the broken surfaces of the bubble, are an iron sulfate. Magnification 500X, scale bar equals 100 microns.

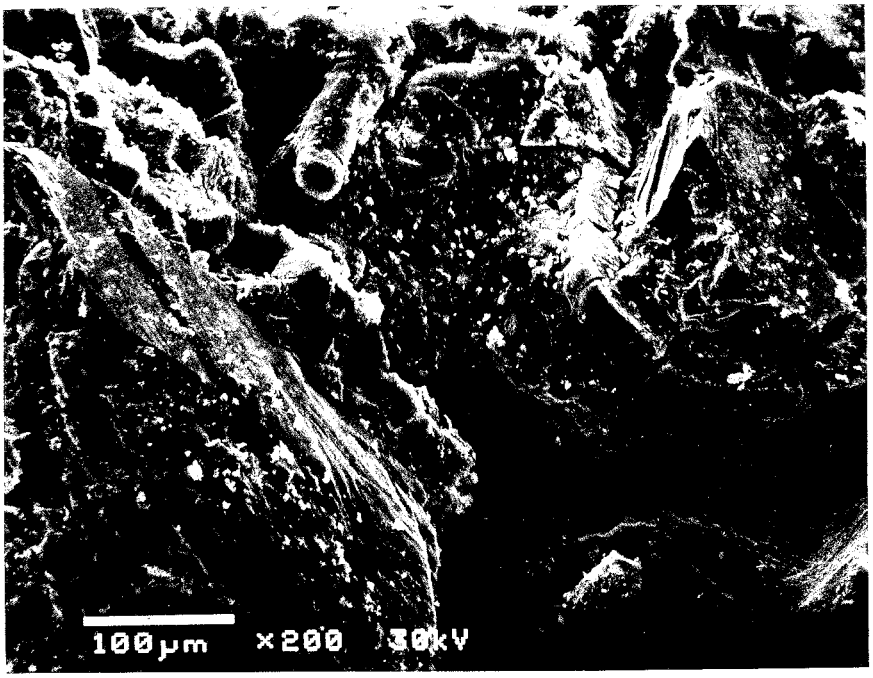


Photo 11: FL-6, Slightly leached plagioclase (lower left, upper right) partly covered with hydrous iron oxide, possibly ferrihydrite. Note the slightly leached surface (lower center) of left plagioclase. Magnification 200X, scale bar equals 100 microns.

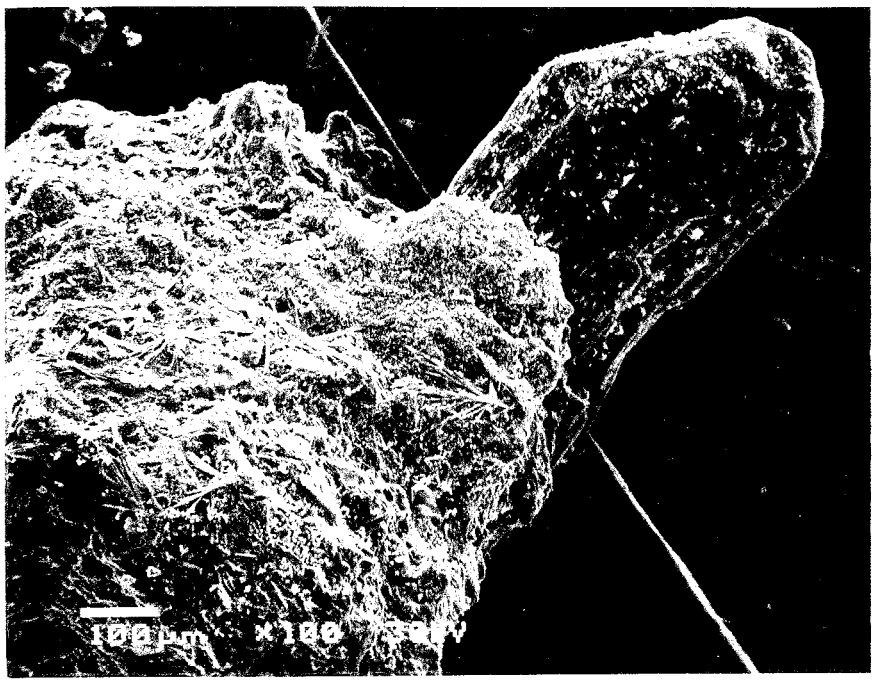


Photo 12: FL-5, Large blade of secondary gypsum (upper right) intergrown with mostly iron oxide (lower left). Note the abundant sprays of tertiary (?) gypsum covering the iron oxide. Magnification 100X, scale bar equals 100 microns.

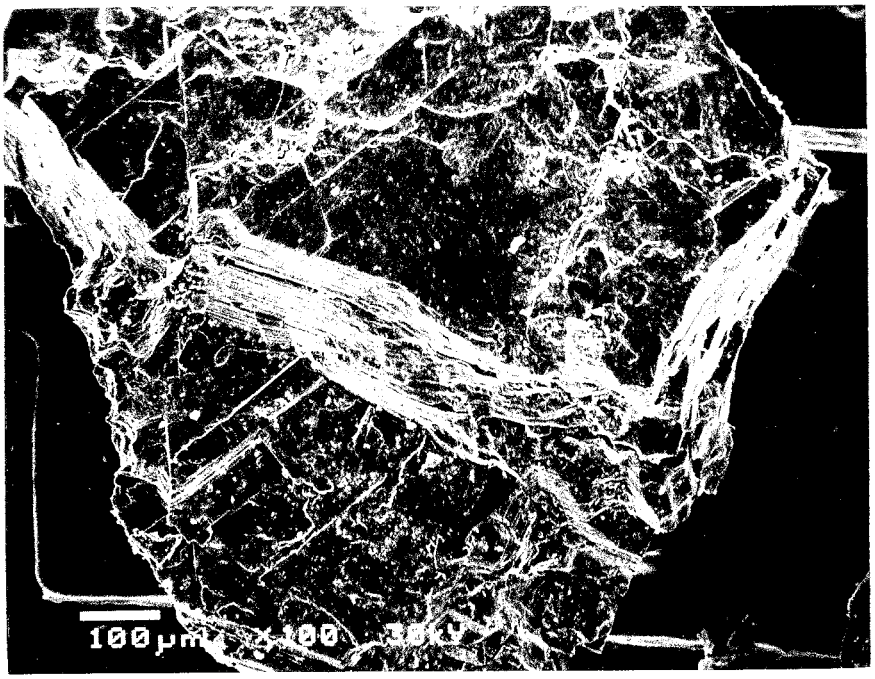


Photo 13: FL-5, Secondary Cl-bearing copper sulfide. Optically the sulfide looks like chalcocite but EDS Cu:S ratios are close to covellite (CuS) with a significant amount of chlorine. Cleavage is evident across the center of the grain. Magnification 100X, scale bar equals 100 microns.

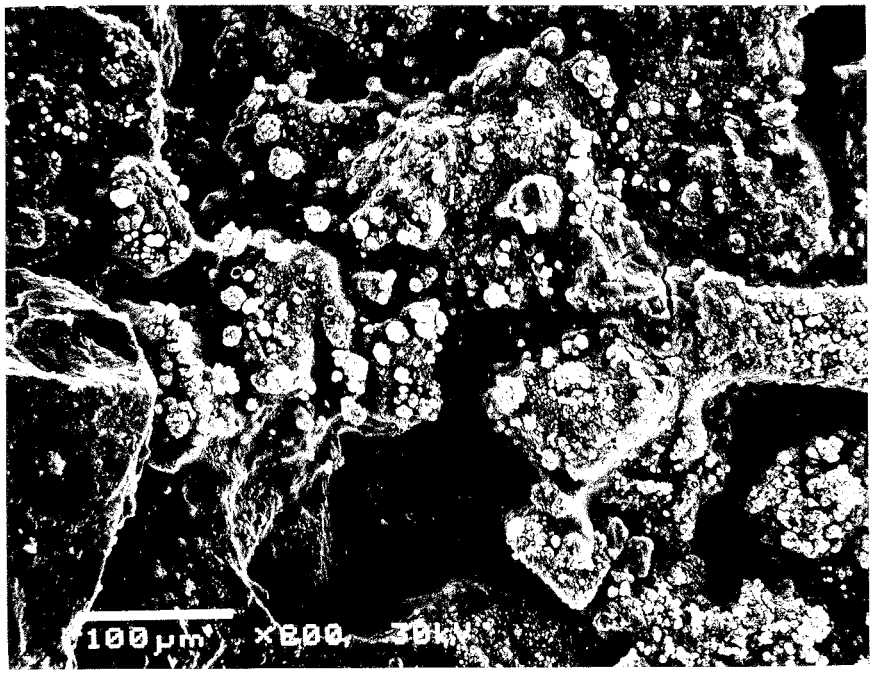


Photo 14: FL-6, Small black arborescent-like aggregates (light areas) of copper sulfide with EDS Cu:S ratios close to covellite (CuS). Much of the underlying material, with fewer CuS aggregates, is hydrous iron oxides. The raised areas, covered with the greater amounts of aggregates, appear to be iron sulfates. Magnification 200X, scale bar equals 100 microns.

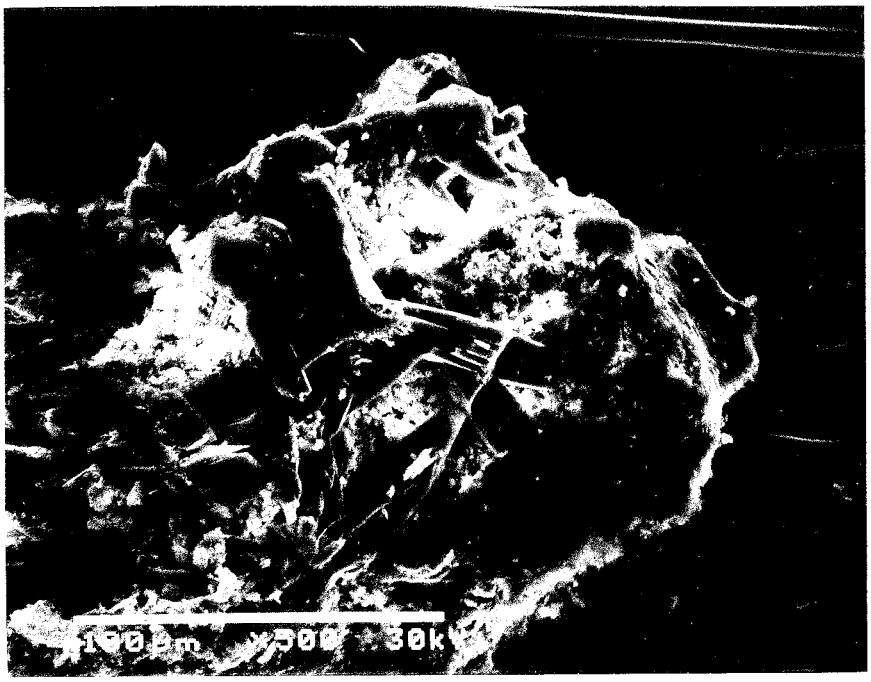


Photo 15: FL-6. Secondary emerald green malachite (smooth dark gray crystals). The light colored areas of "rubble" between malachite is Fe>NiCu>>Mn carbonates/oxides (?). Sulfur was not detected. Magnification 500X, scale bar equals 100 microns.

Photo 16: FL-6, Aggregate of delicate crystals found in ares of photo 15 above. Base (right) is pale blue in color and crystals are nearly white to faintly bluish. EDS yields variable Cu:Ca:S ratios suggesting mixed Ca-sulfate (gypsum/bassanite?) and Cu-sulfates/carbonates (?). Magnification 250X, scale bar equals 100 microns.

APPENDIX 5

SPATIALLY DISTRIBUTED SAMPLE ANALYSIS

Attachment A5.1. Visual descriptions and notes regarding sampling at test piles FL1, FL5, and FL6.

Table A5.1. Test pile spatially distributed sample weights.

Table A5.2.1. Particle size distribution (in weight percent retained) of the bulk and the spatially distributed samples from 0.8%S (FL6) test pile.

Table A5.2.2. Particle size distribution (in weight percent passing) of the bulk and the spatially distributed samples from 0.8%S (FL6) test pile.

Table A5.3. Sulfur and carbon analyses of hand sized rocks from the spatially distributed samples as representative of the unleached rocks.

Table A5.4. Whole rock analysis of hand sized rocks from the spatially distributed samples as representative of the unleached rocks.

Table A5.5. Metals analysis of hand sized rocks from the spatially distributed samples as representative of the unleached rocks.

Table A5.6. Trace metal and rare earth element analysis of hand sized rocks from the spatially distributed samples as representative of the unleached rocks.

Section A5.7. Acid concentration present (ACP) results.

- Table A5.7.1. ACP results for the 0.6% S pile.
- Table A5.7.2. ACP results for the 0.8% S pile.
- Table A5.7.3. ACP results for the 1.4% S pile.
- Attachment A5.7.4. Acid concentration present field notes.

Table A5.81. Summary of visual descriptions and acid concentration present field notes for the 0.6 %S (FL1) pile.

Table A5.8.2. Summary of visual descriptions and acid concentration present field notes for the 0.8 %S (FL6) pile.

Table A5.8.3. Summary of visual descriptions and acid concentration present field notes for the 1.4 %S (FL5) pile.

Table A5.8.4. Measured and calculated corner locations of the three test piles dismantled in October 1994 (FL1, FL6, and FL5).

Table A5.8.5. Sample locations for the spatially distributed samples for piles FL1, FL6, and FL5.

Section A5.9. Chemistry of the spatially distributed samples

Table A5.9.1. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-01-02 (0.8% S pile).

Table A5.9.2. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-01-04 (0.8% S pile).

Table A5.9.3. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-01-05 (0.8% S pile).

Table A5.9.4. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-01-07 (0.8% S pile).

Table A5.9.5. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-02-02 (0.8% S pile).

Table A5.9.6. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-02-04 (0.8% S pile).

Table A5.9.7. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-02-07 (0.8% S pile).

Table A5.9.8. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-02-10 (0.8% S pile).

Table A5.9.9. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-03-01 (0.8% S pile).

Table A5.9.10. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-03-06 (0.8% S pile).

Table A5.9.11. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-03-08 (0.8% S pile).

Table A5.9.12. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-03-16 (0.8% S pile).

Table A5.9.13. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-04-02 (0.8% S pile).

Table A5.9.14. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-04-04 (0.8% S pile).

Table A5.9.15. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-04-05 (0.8% S pile).

Table A5.9.16. Chemistry and particle size distribution for -6.4 mm (0.25 in) fraction of spatially distributed sample 06-04-07 (0.8% S pile).
Table A5.9.17. Chemistry and particle size distribution for the 0.8% S bulk sample.
Table A5.9.18. Summary of the mass weighted sulfur, sulfate, whole rock, and trace metal chemistry for sixteen spatially distributed samples and the bulk sample from the 0.8% test pile.
Table A5.9.19. Comparison of BLM (1996) and ACME (2003) laboratories of whole rock analyses of randomly selected spatially distributed samples.
Attachment A5.2. Analytical summary results from McSwiggen & Associates.
Table A5.10.1. Reaction rating for Amax Leach Pile samples.
Table A5.10.2. Summary of mineral dissolution/reaction rating.
Table A5.10.3. Frequency of sulfide mineral photomicrographs for 30-70 mesh fraction.
Table A5.10.4. Frequency of sulfide mineral occurrences in photomicrographs as a function of particle size.
Table A5.10.5. Average mineral compositions and sample modal analyses.
Table A5.10.6. Average mode and average mineral compositions for FL6 based on analysis of 16 spatially distributed samples.
Table A5.10.7. Average mode (with statistics) calculated by various methods for FL6 based on analysis of 16 spatially distributed samples.
Table A5.10.8. Determination of mode in weight percent for the spatially distributed samples.

General Observations

Overall color: The most common colors observed were gray, brown, orange, and red. The variation in color generally reflected the extent of oxidized iron present. Gray material, such as the original rock, represented little or no oxidation. Rock that was more heavily oxidized was marked by orange or red ("rust") staining.

Degree of agglomeration/cementation/ferricrete: The extent of agglomeration observed varied as follows:

- none
- fines cemented together and/or onto larger material
- large layers of strongly cemented material, up to at least three feet in diameter and one foot thick. Some of this material was difficult to break up, even with the aid of a jackhammer.

Reaction products present: Orange and red stains indicating the presence of iron oxyhydroxide were visible. Additionally, elemental sulfur, elemental copper, and blue and blue-green staining and grains were observed.

Primary minerals present: Unusually high levels of sulfide minerals were occasionally observed.

Moisture content: The material encountered was typically moist, however thin standing layers (a few millimeters deep) and droplets were occasionally observed on the surfaces of the rock within the pile.

Sample Notes

FL1, surface, lift #1

1-1-02: Some greenish rock; some ferricrete.
1-1-04: Minimal greenish rock.
1-1-05: Some greenish coating.
1-1-07: Apparent/patchy light blue coating.
1-1-08: Patchy, light blue coating.
1-1-10: Some rocks with green and rust color.
1-1-15: Some rocks with green and rust color.

NOTES: The original surface was reasonably symmetrical. The grid was 3×5, with 5 samples along the length of the pile.

Sample Grid

X1				X4
	1-1-15	1-1-10	1-1-05	
	1-1-14	1-1-09	1-1-04	
	1-1-13	1-1-08	1-1-03	
	1-1-12	1-1-07	1-1-02	
	1-1-11	1-1-06	1-1-01	
X2				X3

24' from X1 to X2; 8' from X2 to X3; 24' from X3 to X4; 9' from X4 to X1.

average of 4.25' between each vertical row, 6' between each horizontal row

Sample sites at the corners of the sample area were at the survey points (X1, X2, X3, X4).

FL1, lift #2

1-2-01: 100% brown color with some purple and green staining on some larger rocks; non-cemented.
1-2-03: Some light oxidation; some blue-green rock, with some brown rock.
1-2-04: 100% brown color with some purple and green staining on some of the larger rocks; non-cemented.
1-2-05: 100% rust color; slight green-blue rock; unconsolidated.
1-2-06: 80% rust; slight green-blue; a few rocks with heavy oxidation.
1-2-07: 90% gray color with some purple; some blue green rock with little oxidation; non-cemented.
1-2-08: 10% brown and 90% gray with orange rust spots on some of the larger rocks; non-cemented.
1-2-09: 80% rust color.
1-2-10: 90% brown and 10% gray; some light oxidation.; 1 rock 6"×5"×3" with brown color and some oxidation.
1-2-11: 80% gray; rust color on larger particle sizes.
1-2-12: Gray hole with obvious occurrence of reddish-yellow stains and orange spots.
1-2-13: Brown; unconsolidated.
1-2-14: 60% rust; 2 occurrences of green rocks; a few rocks with heavy oxidation.
1-2-15: Mostly gray with some brown, small fragments and some orange-rust staining on larger pieces; non-cemented.
1-2-16: 100% rust color; larger particles have purple staining.

Sample Grid

X1					X4
	1-2-16	1-2-12	1-2-08	1-2-04	
	1-2-15	1-2-11	1-2-07	1-2-03	
	1-1-14	1-2-10	1-2-06	1-2-02	
	1-2-13	1-2-09	1-2-05	1-2-01	
X2					X3

18'6" from X1 to X2; 16' from X2 to X3; 18'6" from X3 to X4; 17' from X4 to X1.

Average of 5'6" between each vertical row; 6'2" between each horizontal row.

Sample sites at the corners of the sample area were at the survey points (X1, X2, X3, X4).

FL1, lift #3

1-3-01: 90% gray; some staining on larger particle sizes.
1-3-02: Moderate to light oxidation; bluish green rock; 70% with oxidation.
1-3-03: 70% brown; some staining on larger particle sizes; an occurrence of green coloring.
1-3-04: 80% rust; very few other oxidation signs.
1-3-06: Brown hole; some bluish gray-green rock with light oxidation; some gray mixture.
1-3-07: Gray hole with occasional spots of yellow-orange coloring.
1-3-08: Mostly gray; small amount brown; occasional spots/pockets of orange/red weathering.
1-3-09: Mostly gray but some brown with occasional orange staining.
1-3-10: 90% gray; some oxidation on larger particle sizes.
1-3-11: 1 rock 8"×4"×2" gray rock with very little oxidation; 95% gray; some oxidation.
1-3-12: 90% gray; oxidation some larger particle sizes.
1-3-13: 90% gray: oxidation on larger particle sizes.
1-3-14: 95% or more gray: lightly oxidized rock surfaces.
1-3-15: 100% gray hole; gray rock and light oxidation.
1-3-16: Almost all gray: noticeable orange staining present.

Sample Grid

X1					X4
	1-3-16	1-3-12	1-3-08	1-3-04	
	1-3-15	1-3-11	1-3-07	1-3-03	
	1-3-14	1-3-10	1-3-06	1-3-02	
	1-3-13	1-3-09	1-3-05	1-3-01	
X2					X3

18'9" from X1 to X2; 16' from X2 to X3; 18'9" from X3 to X4; 17' from X4 to X1.

Average of 5'6" between each vertical row; 6'3" between each horizontal row.

Sample sites at the corners of the sample area were at the survey points (X1, X2, X3, X4).

FL1, lift #4

1-4-01: 80% gray, 20% brown; occurrences of orange and purple on the larger particle sizes.
1-4-02: 70% gray, 30% brown; occurrences of orange and purple; one occurrence of white precipitate.
1-4-03: One large rock 8"×4"×10"; brown to purple staining traces elementary sulfur; one occurrence of copper; mostly gray; non-cemented.
1-4-05: Occurrences of elementary sulfur; occurrences of brown to purple staining; possible elementary copper; wet and mostly gray; non-cemented; occurrence of pyrite.
1-4-06: 90% gray, with some brown; occurrences of purple; some rock has moderate to high oxidation; non-cemented.
1-4-07: 90% gray: with some brown occurrences of purple, some with light oxidation.
1-4-08: All gray hole, some light oxidation; one rock 7"×3"×2" slightly oxidized, gray in color; some purple color rock.
1-4-09: Brown; 0"–4" particles; +4" particles were gray; some rust colored spotting; non-cemented.
1-4-10: Spots of elementary sulfur; mostly gray and wet; rust staining; occurrences of brown, green, and purple staining; non-cemented.
1-4-11: One large rock 5"×5"×10"; light staining brown to purple; non-cemented.
1-4-12: Mostly brown, with some gray; some smaller pieces of rock with heavy oxidation, with some sulfur, and some purple; non-cemented; one rock 6"×6"×5" gray in color with some oxidation.

Sample Grid

X1			X4
1-4-12	1-4-08	1-4-04	
1-4-11	1-4-07	1-4-03	
1-4-10	1-4-06	1-4-02	
1-4-09	1-4-05	1-4-01	
X2			X3

17' from 1-4-09 to 1-4-12 and from 1-4-01 to 1-4-04; 12' from 1-4-09 to 1-4-01 and from 1-4-12 to 1-4-04.

6' between each vertical row; 5'8" between each horizontal row.

X1, X2, X3, and X4 are at survey points

1-4-09 through 1-4-12 row: located 4' outside of line from X1 to X2
1-4-09 through 1-4-01 row: located 3' in line from X2 to X3

FL5, surface, lift #1

5-1-01: Unoxidized at surface (cement?); partially cemented layer about 3" down that broke up when sampled.
5-1-02: Highly oxidized; cemented about 6" down; elemental sulfur present; white precipitate on rock in envelope in five gallon bucket; steel pipe in hole (6"×2.5"), pipe extends at least 10' into the pile.
5-1-03: Highly cemented 3" below surface; elemental sulfur present; rock in envelope in bucket has some blue coating.
5-1-04: Cemented, hard pan from surface.
5-1-05: Heavily oxidized; thick ferricrete, hard pan; elemental sulfur present.
5-1-06: Depth of the hole was 7"; from 3" down, some heavily cemented rock from 4"×3" in diameter; some sulfur from 4" to 7" deep.
5-1-07: Elemental sulfur present; cemented, hard pan.
5-1-08: Purplish staining on rock surfaces; not cemented; light brown soil color; card inside of bucket reads 5-1-09, but should read 5-1-08.
5-1-10: One rock 5"×6"; bottom of hole contained oxidized minerals; some cemented rock 3" down; some heavily oxidized rock 2"-3" down from surface.
5-1-11: Heavily cemented 6" below the surface; elemental sulfur present; heavily oxidized; one rock that appeared to have elemental copper (saved in plastic bag).
5-1-12: Highly cemented 3" below surface; elemental S present; heavily oxidized, various colors.

5-1-13: Not cemented; purplish color coating on 50% of rock; small amount of blue oxidized coating.
5-1-14: One rock 3"×6"; heavily oxidized mineral; not cemented.
5-1-15: Standing water observed on a rock surface about 10" below surface; heavily oxidized; cemented sand/rock below this level; one piece about ½" of solid pyrrhotite with blue coating; elemental sulfur.
NOTES: One fourth of this pile will be removed to Hibbing, therefore only one fourth of the pile will be sampled. A 3×5 grid was constructed. The most notable feature of this pile was a layer (as of yet undetermined thickness) of heavily oxidized and cemented rock particles. This layer was difficult to shovel. During the sample of this material, portions which should have been large sections (> 10") were broken apart. The particle size analysis of these samples will not be representative of the rock in place. 11 of the 15 sample sites had this hardened layer. The samples which did not have this layer tended to be on the northwestern side (the side with 5-1-15 along it). This side seemed to drop off at steeper angle than the opposite side.

Sample Grid

X1				X4
	5-1-15	5-1-10	5-1-05	
	5-1-14	5-1-09	5-1-04	
	5-1-13	5-1-08	5-1-03	
	5-1-12	5-1-07	5-1-02	
	5-1-11	5-1-06	5-1-01	
X2				X3

22' from X1 to X2 and from X3 to X4; 13' from X2 to X3 and from X4 to X1

6'6" between each vertical row, 5'6" between each horizontal row

Sample sites at the corners of the sample area were at the survey points (X1, X2, X3, X4).

FL5, lift #2

5-2-01: Purplish oxidation; slightly damp; unconsolidated; similar to gravel.
5-2-02: Purplish oxidation; some moisture; unconsolidated; elemental sulfur.
5-2-03: Highly cemented layer; highly oxidized.
5-2-04: Hard pan; some purplish oxidation; moisture.
5-2-05: Purplish oxidation; minimal elemental S; slightly damp; unconsolidated, gravel-like.

5-2-06: Unconsolidated, sand and gravel; no globbage.
5-2-07: Some sulfur present; cementing of some rock.
5-2-08: Highly cemented layer; elemental sulfur present; highly oxidized.
5-2-09: Unconsolidated sand and gravel; small amount of elemental sulfur.
5-2-10: Water on one rock; purplish oxidation; some elemental S; moisture in soil.
5-2-11: Purplish rock and gravel; some cementing; some sulfur on rock present.
5-2-12: Purplish oxidation; water on rock; minimal green precipitate; minimal elemental sulfur; semi-hard pan.
5-2-13: Some sulfur present and agglomeration of most areas; not cemented, except small pieces 1" to 2" in diameter; sample of blue pyrrhotite in envelope inside bucket.
5-2-14: Layer of digging heavy oxidized; some sulfur present; two rocks 6"×8"×3" and 3"×3"×1½" - not very oxidized; areas of heavy oxidation with blue-green coatings.
5-2-15: One rock 6"×3" was wet; lots of sulfur content showing; heavily oxidized rock; purple color; all samples wet; one sample 4"×1" had elemental copper present, not in bucket.
5-2-16: At layer of digging, heavily oxidized rock, with some sulfur present; two rocks 6"×3" found with heavy oxidation present.

NOTES: The top approximate 6" was scraped aside before the rock was collected from each sample site. The backhoe, during removal of the 3'–4' of material, reworked some of the pile material. In attempt to sample from undisturbed rock, this top 6" was avoided.

Sample Grid

X1					X4
	5-2-16	5-2-12	5-2-08	5-2-04	
	5-2-15	5-2-11	5-2-07	5-2-03	
	5-2-14	5-2-10	5-2-06	5-2-02	
	5-2-13	5-2-09	5-2-05	5-2-01	
X2					X3

20' from X1 to X2; 13' from X2 to X3; 20' from X3 to X4; 14' from X4 to X1

Average of 4'6" between the vertical rows; 6'8" between the horizontal rows

Sample sites at the corners of the survey site were at the survey points (X1, X2, X3, X4).

FL5, lift #3

5-3-01: Purplish rock; small amount of sulfur; one greenish-blue rock.
5-3-02: 8"×15" rock present; purplish rock.
5-3-03: Purplish rock; some samples wet; small amount of sulfur.
5-3-04: Purplish rock; small amount of sulfur present.
5-3-05: A few cemented chunks, mostly unconsolidated material; little elemental sulfur; one small rock of solid pyrrhotite in bag in bucket.
5-3-06: Moderate cementation; heavy elemental sulfur; some pyrrhotite chunks; blue color on a few of the cemented rocks.
5-3-07: Purplish oxidation; elemental sulfur; moisture on rock surfaces, moderate to heavy cementation.
5-3-08: Very wet; elemental sulfur; semi-hard pan; deep purple oxidation; mostly coarse.
5-3-09: Purplish oxidation; moisture on rock surfaces; not consolidated; minimal elemental sulfur.
5-3-10: Moderate to heavy purple; some sulfur content; some moisture content; one rock 5"×3" purple color.
5-3-11: Medium agglomeration; heavy moisture; purplish coatings; blue coatings and crystallization; sulfur present; sample of elemental copper in envelope; shiny glazing sample in envelope; some cementing; large rock 6"×9"×3".
5-3-12: Minimal purplish coatings; minimal elemental sulfur.
5-3-13: Slight agglomeration; sulfur present; possible elemental copper; non-cemented; heavy moisture; bluish coatings.
5-3-14: One rock 8"×5" purple in color; oxidation present with some water; one rock 4"×3" purple with sulfur; remainder of sample purplein color, with elemental sulfur.
5-3-15: Heavy elemental sulfur; slight cementation; pyrrhotite crystal on one rock face (blue color); other instances of pyrrhotite; slight blue coloring.
5-3-16: Purplish surfaces; elemental sulfur; damp; non-consolidated.

NOTES: The top approximate 6" was scraped aside before the rock was collected from each sample site. The backhoe, during removal of the 3'–4' of material, reworked some of the pile material. In attempt to sample from undisturbed rock, this top 6" was avoided. The lift was approximately 2.5 feet thick. There was visible pyrrhotite (1" to 2" cobbles) in the sample area. Three foot sections of agglomerate were relatively common.

Sample Grid

X1					X4
	5-3-16	5-3-12	5-3-08	5-3-04	
	5-3-15	5-3-11	5-3-07	5-3-03	
	5-3-14	5-3-10	5-3-06	5-3-02	
	5-3-13	5-3-09	5-3-05	5-3-01	
X2					X3

19' from X1 to X2; 13'6" from X2 to X3; 19' from X3 to X4; 14' from X4 to X1.

Average of 4'7" between the vertical rows; 6'4" between the horizontal rows

Sample sites at the corners of the survey site were at the survey points (X1, X2, X3, X4).

FL5, lift #4

5-4-01: Heavily oxidized; brown color; 100% heavily cemented; elemental sulfur.
5-4-02: Heavily cemented; highly oxidized; elemental sulfur; brown in color.
5-4-03: Very heavily cemented; heavily oxidized; 7" down -- gray material rock present; one rock 12"×6"×4" heavily oxidized; elemental sulfur.
5-4-04: Rusty and gray color; elemental sulfur and copper; highly cemented.
5-4-06: Cemented; elemental sulfur; very oxidized; purplish oxidation present; one rock 6"×6"×5" with heavy oxidation.
5-4-07: Very cemented; elemental sulfur; purplish oxidation present; very oxidized.
5-4-08: Heavily oxidized; cemented; brown in color; elemental sulfur.
5-4-09: Very cemented; one pocket of gray fines; elemental sulfur; purplish oxidation present; very oxidized.
5-4-10: 50% fine gray rock; 50% highly oxidized; very cemented; elemental sulfur present.
5-4-11: Cemented; elemental sulfur; very oxidized.
5-4-12: Heavily oxidized; heavily cemented; elemental sulfur; rust color.
5-4-13: Heavily cemented; elemental sulfur; purplish oxidation present; very oxidized; brown color throughout.
5-4-14: All brown/rusty color; cemented; elemental S.
5-4-15: Rusty color; cemented; highly oxidized.
5-4-16: Not very cemented; heavy oxidation; elemental sulfur; some gray color, mostly rust.

NOTES: To the left of sample sites 5-4-16 and 5-4-15 to the edge of the surface, there was a pocket of massive ferricrete that the excavator was unable to move. There was a moderate pocket of ferricrete to the right of sample sites 5-4-04 and 5-4-03, but the excavator was able to move that portion. All sample sites had to be broken up with a Pionjar hammer drill.

Sample Grid

X1					X4
	5-4-16	5-4-12	5-4-08	5-4-04	
	5-4-15	5-4-11	5-4-07	5-4-03	
	5-4-14	5-4-10	5-4-06	5-4-02	
	5-4-13	5-4-09	5-4-05	5-4-01	
X2					X3

18'6" from X1 to X2; 13'6" from X2 to X3; 18'6" from X3 to X4; 14' from X4 to X1.

Average of 4'7" between vertical rows; 6'2" between horizontal rows.

Sample sites at the corners of the survey site were at the survey points (X1, X2, X3, X4).

FL6, surface, lift #1

6-1-01: Some "globbage" (ferricrete); some greenish rock.
6-1-02: 13" block not included in five gallon sample; some greenish rock.
6-1-03: One rock with water on surface.
6-1-06: Greenish rock; some "globbage."
6-1-07: Some greenish rock.
6-1-09: Some greenish rock; 28"×16"×6" rock not included; 19"×14"×6" second rock not included.
6-1-11: Some greenish rock.

NOTES: The surface was mainly flat from samples 6-1-12, -08, -04, but crested midway between the rows of samples at 6-1-10 and the row at 6-1-09. After the crest, it began to slope downward. The area of the crest was composed of generally larger rock fragments than the area up to the crest. The grid for this surface was 3×4 (with four sample along the length of the pile).

Sample Grid

X1				X4
	6-1-12	6-1-08	6-1-04	
	6-1-11	6-1-07	6-1-03	
	6-1-10	6-1-06	6-1-02	
	6-1-09	6-1-05	6-1-01	
X2				X3

18'6" from X1 to X2 and from X3 to X4; 12' from X2 to X3 and from X4 to X1.

5' between each vertical row; 4' between each horizontal row.

X1, X2, X3, and X4 are at survey points

6-1-09 through 6-1-12 row: located 1' in from line between X1 and X2
6-1-01 through 6-1-04 row: located 1' in from line between X3 and X4

6-1-09 through 6-1-01 row: located 3'6" in from line between X2 and X3
6-1-12 through 6-1-04 row: located 4' in from line between X4 and X1

FL6, lift #2

6-2-01: Depth of 0–6" light oxidation; at 6" depth some blue and purple coatings; non-cemented; mostly fine material.
6-2-02: Slightly oxidized to depth of 6"; at depth of 6" and below purple and green staining with slight sulfur specks, non-cemented.
6-2-03: Depth of 0–6" lightly oxidized with mixture of gray unoxidized; non-cemented; at depth of 6" started seeing purple coating, moderately moist.
6-2-04: Grey material fine to 2"–3"; very small amount of oxidation; plastic and paper in hole.
6-2-05: Some lightly oxidized rock; light to moderate blue-gray rock; medium to heavy brown rock; one blue-gray rock 6"×2".
6-2-06: Moderate oxidation; blue-green coloring mostly on larger particles; lots of sand sized particles.
6-2-07: Medium oxidation; fine to pebble size particles; some green coating below 6" depth in hole.
6-2-08: Fines were grey and non-oxidized; larger particles had small amount of oxidation; rust colored.
6-2-09: Mostly oxidized; slightly blue-green coloring; mostly fine material.

6-2-10: Mostly oxidized; blue-green coloring and yellow coloring on larger rocks; primarily sand and gravel size particles.
6-2-11: Some visible purplish oxidation; some light to moderate oxidation; also some light grayish-blue rock.
6-2-12: Small white rock, apparently quartz; small amount of oxidized rock; some bluish-grey rock; some purplish oxidation visible.
6-2-13: Mostly oxidized; slight blue-green; slight cementation.
6-2-14: Some cementing; heavy oxidation; heavy blue-green coloring.
6-2-15: Moderate oxidation; light blue-green rock.

Sample Grid

X1					X4
	6-2-16	6-2-12	6-2-08	6-2-04	
	6-2-15	6-2-11	6-2-07	6-2-03	
	6-2-14	6-2-10	6-2-06	6-2-02	
	6-2-13	6-2-09	6-2-05	6-2-01	
X2					X3

19'6" from X1 to X2; 13'6" from X2 to X3; 19'6" from X3 to X4; 14' from X4 to X1.

Average 4'7" between vertical rows; 6'6" between horizontal rows.

Sample sites at the corners of the survey site were at the survey points (X1, X2, X3, X4).

FL6, lift #3

6-3-01: Some oxidation; 3"-4" material cementation down 12" (hard pan).
6-3-02: Mostly oxidized; light blue-green color.
6-3-03: Very slight blue-green color; light oxidation; mostly gray rock and fines.
6-3-04: Oxidation throughout; purple and green color; non-cemented.
6-3-05: Light blue-green color; light oxidation; unconsolidated.
6-3-06: Moderately oxidized; cementation at 4" depth; purple to blue color quite noticeable throughout.
6-3-07: 1 rock 6"× 6"× 4" with oxidation, blue-gray and some green color; some cemented area; mostly brown rock with blue-green color.
6-3-08: Mostly oxidized; heavy coloration; one large rock (10"× 7"× 4") stained; noticeable elemental copper; colors green to black; non-cemented.
6-3-09: Cementation about 8" down; moderate blue-green coloring; heavy oxidation.
6-3-10: Light oxidation; some blue-gray rock; one rock 3"× 5"× 2" oxidation visible.
6-3-11: Some light oxidation; blue-gray rock with some sulfur content; one rock 3"× 5" × 2" blue-gray with oxidation.
6-3-12: Non-cemented; particle size 0-4" moderately oxidized; particles larger than 4" blue-green purple color; possible elemental copper; minimal sulfur.
6-3-13: Mostly oxidized/some large pieces show blue-purple color cemented hard pan about 6-8" down to 12".
6-3-14: Mostly oxidized; some large pieces with blue-purple color; cemented hard-pan from about 6" deep to 12".
6-3-15: Slightly oxidized with purple color on some rocks; loose soil.
6-3-16: Slightly oxidized with blue and purple color on some rocks; loose soil.

Sample Grid

X1					X4
	6-3-16	6-3-12	6-3-08	6-3-04	
	6-3-13	6-3-11	6-3-07	6-3-03	
	6-3-14	6-3-10	6-3-06	6-3-02	
	6-3-13	6-3-09	6-3-05	6-3-01	
X2					X3

19'6" from X1 to X2; 13' from X2 to X3; 19'6" from X3 to X4; 12'6" from X4 to X1

6'6" between horizontal rows; average of 4'3" between vertical rows

Sample sites at the corners of the survey site are at survey points (X1, X2, X3, X4)

FL6, lift #4

6-4-01: Highly compacted; 40% gray and 60% brown/rust colored surfaces; gray occurred in fine particles; one large rock (> 12"× 12") that was not removed.
6-4-02: Brown hole; highly compacted/cemented; moderate oxidation; not a lot of coloration.
6-4-03: Mostly brown hole with yellow/rust highlights. Moderate oxidation purple blue colors. elemental copper and highly compact/cemented gray fines toward the top.
6-4-04: Mostly brown hole with yellow/rust highlights; moderate oxidation; purple-blue colors; elemental copper; highly compacted/cemented; gray fines toward top of hole.
6-4-05: Moderate to highly cemented; rusty and brown colored hole with a pocket of fine gray material; not a lot of purple oxidation colors; moderate oxidation.
6-4-06: Moderate to heavy oxidation; highly compacted material; mostly brown, with some gray rock.
6-4-07: Light to moderate oxidation; mostly brown to gray material; moderately cemented; some purple color rock.
6-4-08: Some purple coating; light oxidation; occurrence of copper; light agglomeration; heavy compaction; no sulfur.
6-4-09: Moderate cementation; some blue-green rock; some light oxidation; some purple, mostly brown with some gray.
6-4-10: Occurrences of purple stain; moderated oxidation; mostly brown; heavily cemented.
6-4-11: Occurrences of heavy oxidation; 50/50 moderate to light oxidation; heavily cemented; no sulfur.
6-4-12: Mostly oxidized material; highly compacted, but no cementation; some gray disintegrating rock.

NOTES: There were a few occurrences of cementations as the lift was being excavated. One 2'× 2' pocket of fine gray rock on the middle face about 1' up from the base of the lift. Blue-green stained rocks were quite common. A hammer drill was used to break up cementation before sampling all holes except 6-4-12.

Sample Grid

X1			X4
6-4-12	6-4-08	6-4-04	
6-4-11	6-4-07	6-4-03	
6-4-10	6-4-06	6-4-02	
6-4-09	6-4-05	6-4-01	
X2			X3

19' from X1 to X2; 12' from X2 to X3; 19' from X3 to X4; 12' from X4 to X1
6' between each vertical row; 6'4" between each horizontal row
6-4-09 through 6-4-12 row: located 4' outside a line from X1 to X2
6-4-01 through 6-4-04 row: located at the survey points X3 and X4

Notes During Removal of First (Top) Lift

October 6, 1994
FL-5: Started to excavate near middle of pile on north end of sample grid. Large cementation chunks on east half of grid. Chunks were mostly surface, approximately 12" thick and broke up in pieces approximately three feet in diameter. Beneath the cemented area quite a bit of elemental sulfur was observed in most areas. Pictures taken. Most areas of sample area was oxidized but to varying degrees. Particle size fractions 0" to 12", averaging 3". Approximately three large chunks were cemented hard enough to remain solid after excavation, the largest three foot diameter oval piece was set aside for further study. Two others were approximately 1.5" diameter and oval shaped, and approximately 10" thick.

October 7, 1994
FL-6: Rust colored fines about eight to 10 inches think at surface; then more grey unoxidized material. No cementation present. Appears to have larger particle size than FL-5. Highest oxidation is within one foot of surface.

Examination of the Hypalon

October 24, 1994
FL-1: The access hole was started at sample hole 1-03-12. The distance to the Hypalon from the sample surface of lift three was 35". The rock particles were slightly compacted, but non-cemented. At first, the rock particle size averaged +¼ to - 1½", then gradually increased in size and decreased in compaction nearer the Hypalon. Approximately 1½ to 2 feet down, rock size increased dramatically to +4 to -8". Large air spaces were between the rock particles and gravel size particles would cave into crevices and disappear beneath larger sizes during the excavation of the hole. The Hypalon was soft and pliable and no holes were present. The area visible was approximately 5"×10".

November 3, 1994
FL-6: The access hole was started at sample site 6-06-09. The distance to the Hypalon from the sample surface was 33". The upper 12" of rock was heavily cemented with +¼ to -2" particles. Below 12" depth, compaction decreased and particle size increased to an average of +1" to -6". Caving in of the sides of the hole was prevalent after a 20" depth. The bottom 15" of rock had purplish staining and clear glazing that appeared wet, but was not. Not much sulfur was noticed. Some +¼ to -¾" fragments were bound into larger agglomerates approximately -3" in size. The Hypalon was in good condition, pliable with no holes. There was ¼" of standing water on the Hypalon. Small fragments (bead size) of elemental copper were noticed three times at about 6" up from the Hypalon. The area uncovered was 5" square.

Table A5.1. Test pile spatially distributed sample weights (from USBM, Salt Lake City).

Sample Number	Gross Weight (kg)	Bucket Tare Weight (kg)	Net Dry Weight (kg)	Net Weight Loss (kg)	Difference (%)
6-1-2	26.6	1.3	25.0	0.3	1.13
6-1-4	32.6	1.3	31.0	0.3	0.92
6-1-5	25.0	1.3	23.3	0.4	1.60
6-1-7	23.7	1.3	21.9	0.5	2.11
6-2-2	28.1	1.3	26.3	0.5	1.78
6-2-4	31.3	1.3	29.3	0.7	2.24
6-2-7	28.3	1.3	26.4	0.6	2.12
6-2-10	28.8	1.3	27.0	0.5	1.74
6-3-1	30.9	1.3	29.0	0.6	1.94
6-3-6	31.5	1.3	29.4	0.8	2.54
6-3-8	29.8	1.3	28.1	0.4	1.34
6-3-16	28.0	1.3	26.0	0.7	2.50
6-4-2	21.3	1.3	19.5	0.5	2.35
6-4-4	20.0	1.3	18.2	0.5	2.50
6-4-5	20.6	1.3	18.9	0.4	1.94
6-4-7	24.2	1.3	22.3	0.6	2.48

Table A5.2.1. Particle size distribution (in weight percent retained) of the bulk and the spatially distributed samples from 0.8%S (FL6) test pile. Sizes are in mm unless otherwise indicated.

Sample	-12in/ +6in	-6in/ +3in	-3in/ +1.5in	-1.5in/ +0.75in	-0.75in/ +0.5in	-0.5in/ +6.4	-6.4/ +2.0	-2.0/ +0.5	-0.5/ +0.177	-0.177/ +0.149	-0.149/ +0.105	-0.105/ +0.075	-0.075/ +0.053	-0.053	Total
06-01-02		9.17	40.69	30.46	7.10	5.83	2.30	1.28	0.96	0.44	0.48	0.43	0.62	0.32	100.08
06-01-04	24.21	5.75	29.01	17.25	5.79	6.17	3.86	2.97	1.88	0.69	0.68	0.83	0.63	0.35	100.07
06-01-05		13.61	19.96	18.32	10.06	12.86	10.13	5.39	3.21	1.04	1.34	1.40	1.36	0.65	99.33
06-01-07			7.41	21.58	11.20	28.94	14.61	7.56	3.32	0.91	1.29	1.03	0.75	1.10	99.7
06-02-02		9.68	27.52	20.82	7.96	10.42	8.64	6.44	4.18	0.60	1.32	1.21	0.68	0.17	99.64
06-02-04		7.34	17.92	16.88	8.98	12.68	12.38	8.63	7.49	0.98	2.17	2.35	1.16	0.31	99.27
06-02-07			23.82	19.10	10.53	14.26	11.37	8.32	5.78	2.02	1.60	2.06	1.05	0.21	100.12
06-02-10		4.47	15.66	20.12	10.17	16.52	13.18	8.28	5.58	1.46	1.61	1.46	0.50	0.26	99.27
06-03-01			21.61	22.06	9.26	13.32	12.79	8.03	6.14	1.23	1.61	1.46	0.70	0.19	98.4
06-03-06		29.11	11.03	17.29	8.19	10.55	9.08	6.42	3.87	0.88	1.23	1.28	0.61	0.18	99.72
06-03-08	46.31	2.62	23.80	12.90	3.98	3.79	2.03	1.49	1.04	0.36	0.40	0.61	0.34	0.14	99.81
06-03-16		8.32	18.17	23.95	9.42	12.46	9.84	6.50	5.08	1.28	1.25	2.11	0.7	0.22	99.3
06-04-02		6.54	19.10	16.48	9.10	13.98	13.36	7.74	5.86	2.17	2.52	1.75	0.58	0.34	99.52
06-04-04	17.08		16.11	20.15	9.02	11.01	10.20	7.17	3.86	1.53	1.21	1.53	0.62	0.30	99.79
06-04-05			27.84	18.48	9.35	14.61	11.92	8.07	4.40	1.10	1.153	1.35	0.77	0.31	99.353
06-04-07			19.90	18.42	10.20	14.69	13.83	8.81	6.91	1.98	2.37	1.63	0.38	0.26	99.38
Average of spatially distributed samples	5.48	6.04	21.2	19.6	8.77	12.6	10.0	6.44	4.32	1.16	1.39	1.40	0.71	0.33	
0.8%S Bulk	5.9	13.5	27.3	19.2	20.3	2.1	6.1	2.2	1.4	0.26	0.24	0.35	0.28	0.7	99.83

Table A5.2.2. Particle size distribution (in weight percent passing) of the bulk and the spatially distributed samples from 0.8%S (FL6) test pile. Sizes are in mm unless otherwise indicated.

Sample	12"	6"	3"	1.5"	0.75"	0.5"	6.4	2.0	0.5	0.177	0.149	0.105	0.075
6-1-02	100.00	90.84	50.18	19.74	12.65	6.82	4.53	3.25	2.29	1.85	1.37	0.94	0.32
6-1-04	75.81	70.06	41.07	23.83	18.05	11.88	8.02	5.06	3.18	2.49	1.81	0.98	0.35
6-1-05	100.00	86.30	66.20	47.76	37.63	24.69	14.49	9.06	5.83	4.78	3.43	2.02	0.65
6-1-07	100.00	100.00	92.57	70.92	59.69	30.66	16.01	8.43	5.10	4.18	2.89	1.86	1.10
6-2-02	100.00	90.29	62.67	41.77	33.78	23.32	14.65	8.19	3.99	3.39	2.07	0.85	0.17
6-2-04	100.00	92.61	74.55	57.55	48.50	35.73	23.26	14.57	7.02	6.03	3.85	1.48	0.31
6-2-07	100.00	100.00	76.21	57.13	46.61	32.37	21.01	12.70	6.93	4.91	3.32	1.26	0.21
6-2-10	100.00	95.50	79.72	59.45	49.21	32.57	19.29	10.95	5.33	3.86	2.24	0.77	0.26
6-3-01	100.00	100.00	78.04	55.62	46.21	32.67	19.67	11.51	5.27	4.02	2.39	0.90	0.19
6-3-06	100.00	70.81	59.75	42.41	34.20	23.62	14.51	8.07	4.19	3.31	2.08	0.79	0.18
6-3-08	53.60	50.98	27.13	14.21	10.22	6.42	4.39	2.90	1.85	1.49	1.09	0.48	0.14
6-3-16	100.00	91.62	73.32	49.20	39.72	27.17	17.26	10.72	5.60	4.31	3.05	0.93	0.22
6-4-02	100.00	93.43	74.24	57.68	48.53	34.49	21.06	13.28	7.40	5.22	2.68	0.92	0.34
6-4-04	82.88	82.88	66.74	46.55	37.51	26.48	16.25	9.07	5.20	3.67	2.46	0.92	0.30
6-4-05	100.00	100.00	71.98	53.38	43.97	29.26	17.26	9.14	4.71	3.61	2.45	1.09	0.31
6-4-07	100.00	100.00	79.98	61.44	51.18	36.40	22.48	13.61	6.66	4.67	2.28	0.64	0.26
Avg of Spatially Distributed samples	94.50	88.43	67.12	47.39	38.58	25.89	15.87	9.40	5.03	3.86	2.46	1.05	0.33
0.8%S Bulk	94.10	80.6	53.2	34.0	13.7	11.6	5.44	3.24	1.83	1.57	1.33	0.98	0.70

Table A5.3. Sulfur and carbon analyses of hand sized rocks from the spatially distributed samples as representative of the unleached rocks. Analysis by Acme Analytical Laboratories.

Sample	S_T %	S^-/S %	SO_4 %	C_T %	CO_2 %
6-1-2	<0 .01	<0.03	0.06	0.01	0.04
6-1-4	1.33	0.72	1.84	0.06	0.08
6-1-5	0.24	0.19	0.14	0.01	0.02
6-1-7	0.35	0.19	0.47	0.14	< 0.01
6-2-2	1.26	0.87	1.17	0.01	0.03
6-2-4	0.1	0.05	0.14	0.02	0.01
6-2-7	0.03	<0.03	0.03	0.03	0.11
6-2-10	0.22	0.18	0.11	0.06	0.05
6-3-1	0.19	0.05	0.42	0.05	0.2
6-3-6	0.06	<0.03	0.17	0.04	0.17
6-3-6	0.06	<0.03	0.2	0.03	0.17
6-3-8	1.28	1.14	0.42	0.03	0.04
6-3-16	0.06	<0.03	0.14	0.03	0.07
6-4-2	0.24	0.21	0.08	0.25	0.05
6-4-4	0.66	0.63	0.08	0.69	<0.01
6-4-5	0.37	0.27	0.28	0.03	0.07
6-4-7	0.15	0.1	0.14	0.02	<0.01
average	0.39	0.28	0.35	0.09	0.07

Table A5.4. Whole rock analysis (by ICP-ES) of hand sized rocks from the spatially distributed samples as representative of the unleached rocks. Analysis by Acme Analytical Laboratories.

Sample	SiO_2 %	Al_2O_3 %	Fe_2O_3 %	MgO %	CaO %	Na_2O %	K_2O %	TiO_2 %	P_2O_5 %	MnO %	Cr_2O_3 %	LOI %
6-1-2	48.74	14.4	17.33	4.29	6.4	2.46	0.82	4.87	0.58	0.18	0	<0.1
6-1-4	45.13	19.9	12.76	7.96	8.9	2.51	0.22	0.42	0.08	0.11	0.01	1.4
6-1-5	46.25	13.6	20.36	4.69	5.96	2.19	0.68	5.12	0.66	0.2	0.01	<0.1
6-1-7	49.13	16.3	11.47	3.54	15.6	1.22	0.85	0.81	0.19	0.24	0.021	0.6
6-2-2	44.15	14.2	18.26	10.5	7.14	2.27	0.69	1.72	0.3	0.17	0.01	<0.1
6-2-4	45.22	14.9	17.52	11.6	7.31	2.11	0.5	1.31	0.21	0.19	0.012	<0.1
6-2-7	41.99	12.7	17.18	17.8	5.78	1.46	0.13	0.11	0.02	0.18	0.013	2.5
6-2-10	48.5	15.1	12.75	9.69	9.89	1.16	0.52	1.42	0.16	0.17	0.086	0.2
6-3-1	38.95	7.48	21.59	22.5	3.67	0.79	0.1	0.13	0.03	0.23	0.015	4.2
6-3-6	38.34	8.26	20.52	21.7	3.88	0.82	0.1	0.1	0.02	0.21	0.015	5.9
6-3-6	38.37	8.26	20.36	21.7	3.86	0.83	0.1	0.1	0.03	0.21	0.015	6
6-3-8	42.92	11.3	20.73	4.8	9.05	2.15	1.36	6.14	0.6	0.19	0.01	<0.1
6-3-16	48.25	15.4	11.19	11.1	11.7	1.75	0.1	0.23	<0.01	0.16	0.034	0.1
6-4-2	51.87	18.3	11.84	5.78	4.22	3.04	2.62	1.23	0.06	0.08	0.042	0.4
6-4-4	49.23	20.5	15.58	7.49	1.61	1.34	1.6	1.53	0.03	0.1	0.053	0.7
6-4-5	46.7	17.7	13.48	7.09	8.76	2.65	0.77	1.85	0.29	0.14	0.016	0.1
6-4-7	46.72	13.3	19.26	4.52	8.04	1.97	0.5	4.82	0.57	0.21	0.01	<0.1
average	45.32	14.2	16.6	10.4	7.16	1.81	0.68	1.87	0.23	0.17	0.02	1.34

Table A5.5. Metals analysis (by ICP-MS) of hand sized rocks from the spatially distributed samples as representative of the unleached rocks. Analysis by Acme Analytical Laboratories.

Sample	Mo ppm	Cu ppm	Pb ppm	Zn ppm	Ni ppm	As ppm	Cd ppm	Sb ppm	Bi ppm	Ag ppm	Au ppb	Hg ppm	Tl ppm	Se ppm	FeO %
6-1-2	0.3	84.4	2.6	48	46	1.8	0.2	0.4	0.1	<0.1	1.8	<0.1	0.1	<0.5	15.1
6-1-4	1.1	4140	4.6	57	1218	<0.5	0.6	<0.1	0.3	1.4	36.3	<0.1	<0.1	2.1	10.3
6-1-5	0.5	7073	2.2	58	73.1	1.2	0.4	0.4	0.1	0.9	4.4	<0.1	0.1	0.6	17.6
6-1-7	1.9	424	2.6	53	190.1	27.5	0.9	0.3	<0.1	0.2	5.6	<0.1	<0.1	1	10.3
6-2-2	0.7	6005	7.3	97	1862	5.5	0.5	0.3	2.6	3.1	486.3	<0.1	0.1	6	16.5
6-2-4	1	983.8	1.1	98	421	1.1	0.1	0.1	0.1	0.3	1	<0.1	<0.1	0.5	15.6
6-2-7	0.2	95	0.5	109	842.5	<0.5	0.1	<0.1	<0.1	0.1	0.8	<0.1	<0.1	<0.5	13.1
6-2-10	1.4	2190	5.1	37	260.6	4.9	0.4	0.3	0.1	0.2	3.4	0.01	<0.1	<0.5	11.6
6-3-1	0.2	1356	1.7	123	1263	<0.5	0.3	<0.1	0.1	0.5	7	<0.1	<0.1	0.6	15.2
6-3-6	0.6	372.6	1	131	990	<0.5	0.1	<0.1	0.1	0.1	1.1	<0.1	<0.1	<0.5	12.7
6-3-6	0.6	360.8	0.9	129	1054	<0.5	0.1	<0.1	0.1	0.1	1.6	<0.1	<0.1	<0.5	12.7
6-3-8	1	5683	4.6	99	970.2	1.4	0.7	0.1	0.4	2	11.2	<0.1	<0.1	2.1	17.8
6-3-16	0.2	329.9	0.6	37	286.2	<0.5	0.1	<0.1	<0.1	0.1	0.7	<0.1	<0.1	<0.5	9.7
6-4-2	4.5	552.5	1.5	85	137.6	2.9	0.2	0.1	<0.1	0.1	1.3	<0.1	0.3	0.7	10.4
6-4-4	7.8	490.5	0.7	54	243.7	8.6	0.3	0.1	0.1	0.1	4.5	<0.1	0.1	1.2	14
6-4-5	1.3	2544	4.5	78	336.3	2.7	0.3	0.2	0.9	0.8	4.5	<0.1	0.1	1.1	11.9
6-4-7	0.8	705.9	2.7	27	120.8	0.8	0.1	0.1	0.1	0.2	6.1	<0.1	<0.1	0.7	17.2
average	1.4	1964.2	2.6	78	606.8	3.6	0.3	0.2	0.3	0.6	34.0	0.1	0.1	1.2	13.6

Table A5.6. Trace metal and rare earth element analysis (by ICP-MS or ICP-ES) of hand sized rocks from the spatially distributed samples as representative of the unleached rocks. Analysis by Acme Analytical Laboratories.

Sample	Ba ppm	Ni ppm	Sc ppm	Co ppm	Cs ppm	Ga ppm	Hf ppm	Nb ppm	Rb ppm	Sn ppm	Sr ppm	Ta ppm	Th ppm	U ppm	V ppm	W ppm	Zr ppm	Y ppm
6-1-2	160	75	24	54.2	3.4	23	3.1	35	30.3	2	481	2.1	2.9	1.4	400	1.1	95.1	34.1
6-1-4	86	1201	5	121.4	0.4	20	0.6	2	3.8	<1	352	0.1	0.5	0.1	35	0.1	23	4.4
6-1-5	188	116	31	61.9	1.8	24	7.8	36	17.7	2	423	2.5	1.8	1.2	431	1	294	40.1
6-1-7	553	192	24	44.3	1.4	24.9	3.3	11	18.9	<1	317	0.6	5.6	2.1	142	1.9	122	75.5
6-2-2	190	1957	13	147.6	1.1	19	4.2	12	23.8	2	253	0.8	2.5	0.8	115	0.8	156	26.4
6-2-4	152	416	11	101.9	0.7	19	3.2	8.6	14.3	<1	257	0.5	1.7	0.6	94	0.2	123	17.5
6-2-7	49	716	6	121.2	1.1	11.9	<0.5	<0.5	1.1	<1	203	<0.1	0.2	<0.1	21	0.4	2.6	1.1
6-2-10	119	282	37	64.1	2.6	19	1.6	6.6	22.3	<1	259	0.5	0.7	0.2	288	0.1	58.3	22.2
6-3-1	23	1097	7	172.2	0.2	7.2	< .5	<0.5	1.4	<1	124	<0.1	<0.1	<0.1	30	0.3	4.7	1.2
6-3-6	34	937	6	148.6	0.2	7.2	<0.5	<0.5	0.9	<1	131	<0.1	<0.1	<0.1	18	<0.1	1.9	0.6
6-3-6	32	930	6	158.1	0.1	7.6	<0.5	<0.5	1	<1	138.4	<0.1	<0.1	<0.1	20	<0.1	2	0.6
6-3-8	606	1011	49	113.7	1	24	3.6	24	26.5	2	208.2	1.5	1.5	0.7	681	0.6	132	48.8
6-3-16	44	287	30	68.3	<0.1	16	<0.5	<0.5	0.8	<1	277	<0.1	0.1	<0.1	309	0.5	3.3	4.6
6-4-2	1642	146	41	41.6	1.5	28	3.5	9.8	66.8	<1	454.1	0.5	2.3	0.6	290	<0.1	117	14.4
6-4-4	1094	238	52	58.1	3.2	29	2.7	15	46.4	<1	208.3	0.9	3.3	1	358	0.7	79.5	13.2
6-4-5	212	353	15	70.3	1.6	22	4.3	12	27.8	2	315	0.7	3.1	0.9	127	0.2	145.7	27.1
6-4-7	189	147	29	65.4	1.2	23.2	7	33.6	13.3	<1	548	2.1	1.3	0.9	407	0.6	275	45
average	316	594	22.7	94.88	1.3	19	2.8	12	18.7	1.3	291	0.8	1.6	0.6	222	0.5	96.2	22.2

Sample	La ppm	Ce ppm	Pr ppm	Nd ppm	Sm ppm	Eu ppm	Gd ppm	Tb ppm	Dy ppm	Ho ppm	Er ppm	Tm ppm	Yb ppm	Lu ppm
6-1-2	26.4	62.5	8.51	38.6	8.4	3.07	7.82	1.14	6.53	1.19	3.44	0.46	2.9	0.43
6-1-4	3.7	8.2	0.97	3.5	0.9	0.85	0.89	0.16	0.69	0.17	0.44	0.06	0.37	0.07
6-1-5	28.5	67.6	8.9	39.2	9.5	2.95	8.51	1.39	7.33	1.45	3.96	0.56	3.83	0.55
6-1-7	45.5	93.1	11.32	48.1	10.6	2.52	11.08	1.74	11.2	2.22	6.24	0.82	5.54	0.8
6-2-2	18.2	42.2	5.39	23	5.5	2	5.24	0.81	4.79	0.94	2.58	0.36	2.18	0.36
6-2-4	12	27.4	3.41	13.9	3.2	1.48	3.64	0.52	2.98	0.58	1.72	0.22	1.57	0.24
6-2-7	1.1	2.1	0.24	1	0.2	0.44	0.24	0	0.18	<0.05	0.13	<0.05	0.11	0.03
6-2-10	10.4	26	3.45	15.9	4.2	1.35	4.33	0.68	4.18	0.82	2.35	0.3	1.84	0.27
6-3-1	1	2.1	0.25	1	0.2	0.23	0.34	0	0.22	<0.05	0.18	<0.05	0.13	0.03
6-3-6	0.7	1.2	0.13	0.5	0.1	0.27	0.08	0	0.1	<0.05	0.06	<0.05	0.1	0.01
6-3-6	0.6	1.2	0.14	0.4	0.1	0.27	0.2	0	0.12	<0.05	0.08	<0.05	0.11	0.03
6-3-8	22.9	61.6	8.42	39.6	9.9	2.32	10.22	1.56	9.05	1.74	4.47	0.59	3.99	0.53
6-3-16	0.9	2.4	0.29	1.6	0.7	0.69	0.72	0.14	0.76	0.16	0.45	0.06	0.41	0.07
6-4-2	47.4	84.4	8.8	33.1	4.6	3.52	2.9	0.43	2.35	0.55	1.5	0.28	2.16	0.35
6-4-4	22.8	41.5	4.51	17.3	2.5	1.83	2.11	0.33	2.07	0.52	1.48	0.26	2.12	0.34
6-4-5	17.7	42.1	5.33	23.4	5.4	1.73	5.24	0.85	5.02	0.97	2.57	0.36	2.3	0.35
6-4-7	25.5	66.8	9.32	45.5	10.5	3.32	9.93	1.52	8.69	1.54	4.29	0.55	3.84	0.55
average	16.8	37.2	4.67	20.3	4.5	1.70	4.32	0.67	3.90	0.77	2.11	0.30	1.97	0.29

Section A5.7. Acid concentration present (ACP) results (as determined in the laboratory).

Table A5.7.1. ACP results for the 0.6% S pile.

Lift	Sample	$CaCO_3$[1] (ppm)	pH	Mass (g)	ACP (ppm/g)
1	FL1-1-02	24.00	4.23	20.62	1.16
1	FL1-1-04	16.00	4.22	19.84	0.81
1	FL1-1-06	20.00	4.17	19.10	1.05
1	FL1-1-08	12.00	4.19	17.12	0.70
1	FL1-1-10	12.00	4.11	20.42	0.59
1	FL1-1-12	20.00	4.04	17.09	1.17
1	FL1-1-14	16.00	4.20	16.93	0.95
2	FL1-2-01	20.00	4.39	17.20	1.16
2	FL1-2-03	20.00	4.33	15.37	1.30
2	FL1-2-06	36.00	4.13	15.76	2.28
2	FL1-2-08	24.00	4.32	16.13	1.49
2	FL1-2-09	44.00	4.11	20.54	2.14
2	FL1-2-11	16.00	4.92	17.05	0.94
2	FL1-2-14	60.00	3.82	20.78	2.89
2	FL1-2-16	28.00	4.00	17.64	1.59
3	FL1-3-02	68.00	3.84	16.76	4.06
3	FL1-3-04	44.00	4.20	16.97	2.59
3	FL1-3-05	8.00	7.77	21.60	0.37
3	FL1-3-07	12.00	5.07	18.77	0.64
3	FL1-3-10	8.00	5.23	20.75	0.39
3	FL1-3-12	8.00	5.24	18.46	0.43
3	FL1-3-13	8.00	7.34	19.84	0.40
3	FL1-3-15	8.00	6.21	18.59	0.43
4	FL1-4-02	8.00	7.79	22.87	0.35
4	FL1-4-04	16.00	5.53	22.97	0.70
4	FL1-4-05	8.00	7.33	22.30	0.36
4	FL1-4-07	8.00	5.39	22.96	0.35
4	FL1-4-10	8.00	6.37	25.08	0.32
4	FL1-4-12	12.00	5.64	20.06	0.60

[1] Acidity of 50 mL solution.

Table A5.7.2. ACP results for the 0.8% S pile.

Lift	Sample	$CaCO_3$[1] (ppm)	pH	Mass (g)	ACP (ppm/g)
1	FL6-1-01	28.00	3.90	20.18	1.39
1	FL6-1-02	16.00	3.94	17.31	0.92
1	FL6-1-03	12.00	3.62	18.81	0.64
1	FL6-1-04	28.00	4.37	18.16	1.54
1	FL6-1-05	36.00	4.55	20.24	1.78
1	FL6-1-06	24.00	4.13	20.50	1.17
1	FL6-1-07	112.00	3.28	21.56	5.19
1	FL6-1-08	16.00	4.04	19.10	0.84
1	FL6-1-09	44.00	4.15	22.44	1.96
1	FL6-1-10	24.00	3.98	19.33	1.24
1	FL6-1-11	48.00	3.53	20.55	2.34
1	FL6-1-12	76.00	3.76	16.37	4.64
2	FL6-2-02	84.00	4.37	18.27	4.60
2	FL6-2-04	8.00	5.01	21.24	0.38
2	FL6-2-05	136.00	3.67	17.62	7.72
2	FL6-2-07	60.00	4.10	18.35	3.27
2	FL6-2-10	184.00	3.80	16.18	11.37
2	FL6-2-12	100.00	4.54	18.25	5.48
2	FL6-2-13	108.00	3.91	22.91	4.71
2	FL6-2-15	68.00	4.19	16.83	4.04
3	FL6-3-01	36.00	4.51	20.09	1.79
3	FL6-3-03	16.00	4.96	22.29	0.72
3	FL6-3-06	40.00	4.97	21.76	1.84
3	FL6-3-08	32.00	4.88	19.47	1.64
3	FL6-3-09	68.00	4.32	20.59	3.30
3	FL6-3-11	48.00	4.47	20.64	2.33
3	FL6-3-14	28.00	4.71	20.32	1.38
3	FL6-3-16	24.00	5.44	24.55	0.98
4	FL6-4-02	572.00	3.49	21.16	27.03
4	FL6-4-04	312.00	3.68	21.80	14.31
4	FL6-4-05	56.00	4.44	21.89	2.56
4	FL6-4-07	420.00	3.84	24.03	17.48
4	FL6-4-10	68.00	4.14	22.93	2.97
4	FL6-4-12	100.00	4.21	21.25	4.71

[1] Acidity of 50 mL solution.

Table A5.7.3. ACP results for the 1.4% S pile.

Lift	Sample	$CaCO_3$[1] (ppm)	pH	Mass (g)	ACP (ppm/g)
1	FL5-1-01	40.00	3.90	18.91	2.12
1	FL5-1-02	100.00	3.39	21.61	4.63
1	FL5-1-03	316.00	3.17	18.67	16.93
1	FL5-1-05	1000.00	2.89	18.68	53.53
1	FL5-1-06	260.00	3.24	18.44	14.10
1	FL5-1-08	88.00	3.47	19.58	4.49
1	FL5-1-09	304.00	3.04	21.73	13.99
1	FL5-1-10	304.00	3.06	22.34	13.61
1	FL5-1-12	156.00	3.29	19.59	7.96
1	FL5-1-14	56.00	3.59	18.02	3.11
1	FL5-1-15	196.00	3.42	19.10	10.26
2	FL5-2-02	144.00	3.50	19.62	7.34
2	FL5-2-04	124.00	3.74	18.52	6.70
2	FL5-2-05	344.00	3.44	21.06	16.33
2	FL5-2-07	1388.00	3.15	17.65	78.64
2	FL5-2-10	540.00	3.15	18.55	29.11
2	FL5-2-12	240.00	3.56	15.18	15.81
2	FL5-2-13	600.00	3.37	18.15	33.06
2	FL5-2-15	1132.00	2.89	18.52	61.12
3	FL5-3-01	364.00	3.32	21.92	16.61
3	FL5-3-03	472.00	3.29	20.48	23.05
3	FL5-3-06	676.00	3.16	16.45	41.09
3	FL5-3-08	600.00	3.13	19.47	30.82
3	FL5-3-09	476.00	3.20	18.02	26.42
3	FL5-3-11	1000.00	3.08	16.99	58.86
3	FL5-3-14	532.00	3.19	18.02	29.52
3	FL5-3-16	644.00	3.18	19.24	33.47
4	FL5-4-02	956.00	3.20	19.73	48.45
4	FL5-4-04	1160.00	3.11	18.74	61.90
4	FL5-4-05	684.00	3.23	19.44	35.19
4	FL5-4-07	620.00	3.17	16.66	37.21
4	FL5-4-10	776.00	3.21	18.57	41.79
4	FL5-4-12	356.00	3.33	20.93	17.01
4	FL5-4-13	764.00	3.17	19.64	38.90
4	FL5-4-16	512.00	3.24	16.35	31.31

[1] Acidity of 50 mL solution.

Pile FL1 October 4, 1994

Sample#	Remarks
1-1-02	Brown sand; no apparent ferricrete.
	+¼" - light ferricrete; no S^0; minimal blue coating.
	−¼" - one blue-green particle; minimal ferricrete; similar moisture to other holes.
1-1-04	Brown sand; no apparent ferricrete.
	+¼" - one green, secondary mineral; minimal ferricrete.
	−¼" - brown; minimal ferricrete; no greenies.
1-1-06	Brown sand; no apparent ferricrete.
	+¼" - some green surfaces, one particle; minimal ferricrete.
	−¼" - brown; no green surfaces; one piece ferricrete.
1-1-08	Brown sand; pieces, but no layers of ferricrete.
	+¼" - no green surfaces; minimal ferricrete.
	−¼" - brown; no ferricrete or green surfaces.
1-1-10	Brown; no obvious ferricrete zone.
	+¼" - rust-staining on rock; no agglomerated ferricrete; no green surfaces.
	−¼" - brown; no agglomeration; no green surfaces.
1-1-12	Brown; no obvious ferricrete zone.
	+¼" - rust-stained surfaces; no agglomeration or green surfaces.
	−¼" - no agglomeration or green surfaces; rust-stained surface.
1-1-14	Brown; no visible ferricrete.
	+¼" - rust-stained surfaces; one green particle.
	−¼" - one green particle; brown sand; no agglomeration.
NOTE:	FL1, surface: No S^0; −¼" fraction was apparently wetter than other piles as it stuck to plastic pan.

October 8, 1994
44°F, clouds, wind

1-2-01	Brown; unconsolidated; more gray at bottom.
	+¼" - mostly coated with fines; occurrence of purple and orange spots.
	−¼" - same as above.
1-2-03	Brown; no cement; some black, crumbly rock.
	+¼" - mostly coated with fines; occurrence of orange, purple, blue.
	−¼" - mostly coated with fines; one orange and one purple.
1-2-06	Mostly brown; unconsolidated; gray at bottom; brown down 9".
	+¼" - mostly coated with gray fines; occurrences of orange and purple stain; one occurrence of ferricrete texture; no green stain; no visible po.
	−¼" - occurrences of purple; almost all coated with fines.

Sample#	Remarks
1-2-08	Mostly gray; no cement; layer of brown on top.
	+¼" - occurrence of orange and purple stain; one occurrence of crumbly, black rock; most coated with fines.
	−¼" - light orange stain on −¼"/+⅛"; no ferricrete.
1-2-09	Mix of brown and gray; unconsolidated; gray at bottom; mostly brown.
	+¼" - occurrences of purple and orange stain; no visible ferricrete or blue/green stain; mostly coated with fines.
	−¼" - all coated with fines; brown-gray.
1-2-11	Mostly gray; +1" has orange and purple spotting; unconsolidated.
	+¼" - mostly coated with gray fines; purple and orange stains; one ferricrete associated with visible sulfides.
	−¼" - occurrences of orange and purple; remainder is g(reen?)
1-2-14	Brown top, gray on bottom; brown may be reworked; no cementing; gravel to cobble sized rocks.
	+¼" - light to moderate orange and purple stain spots; one occurrence of ferricrete; no green stain observed, but heavy coating by fines.
	−¼" - occurrences of orange and purple staining; grayer than previous sample; no green staining.
1-2-16	Brown; uncemented; orange spots on larger pieces.
	+¼" - light purple and orange staining/spots; one piece ferricrete; one green stained; most coated with brown-gray fines.
	−¼" - −¼"/+⅛" coated with fines; one purple stain visible; mostly brown.
1-3-02	Brown (may be reworked); unconsolidated; gray at 4" depth; many blue-green surfaces.
	+¼" - light to moderate purple stains; light green staining; no ferricrete texture.
	−¼" - some purple and green occurrences; all coated with fines.
1-3-04	Unconsolidated; gray.
	+¼" - light to moderate orange/purple stains; no ferricrete; one green coated surface; coated with gray fines.
	−¼" - light orange occurrence; coated with gray fines.
1-3-05	Mostly gray; few pieces with orange staining; clayey.
	+¼" - mostly coated with gray fines; two occurrences of orange/purple staining.
	−¼" - occurrences of orange stain; coated heavily with gray fines.
1-3-07	Unconsolidated; gray.
	+¼" - light orange and purple stain; rest coated with gray fines; no ferricrete, blue, or green.
	−¼" - occurrences of orange and purple; remainder gray.
1-3-10	Gray; unconsolidated.
	+¼" - gray silt coated; occurrence of orange and purple.
	−¼" - orange occurrence; remainder coated with gray fines.

Sample#	Remarks
1-3-12	Gray; unconsolidated; with orange and purple spots.
	+¼" - orange and purple spot occurrence; remainder heavily coated with gray fines; no ferricrete, green, or blue.
	−¼" - one green and one orange; remainder coated with gray.
1-3-13	Gray; unconsolidated.
	+¼" - occurrence of orange and purple stain; no ferricrete, blue, or green; all coated with gray fines.
	−¼" - purple and orange occurrence; rest coated with gray fines.
1-3-15	Unconsolidated; gray with occurrences of orange spotting.
	+¼" - mostly coated with gray fines; light orange spotting.
	−¼" - orange occurrence; remainder coated with gray fines.
NOTE:	FL1, lift #3: Brown material occurred on southeast side of pile and was reworked over holes 1-4, mostly; dug down below this layer before sampling.

October 11, 1994
45°F, sunny

1-4-02	70% gray with remainder brown; occurrences of orange on the larger particle sizes; occurrence of white precipitate; exposed sulfide on one rock (sample in bag); no ferricrete, S^0, Cu^0, or green color; occurrence of purple.
	+¼" - 90% gray with rest brown; occurrences of orange on larger particle sizes; no ferricrete, S^0, Cu^0, or green color, occurrence of purple.
	−¼" - 90+% gray; occurrences of orange and purple; no ferricrete, S^0, Cu^0, or green.
1-4-04	50% gray/50% brown; occurrences of orange and purple; one occurrence of green; no ferricrete, S^0, or Cu^0.
	+¼" - 90% gray with occurrences of purple; light orange; one occurrence of green and one occurrence of ferricrete; no S^0 or Cu^0.
	−¼" - 60% gray/40% brown; light orange; occurrences of purple; two occurrences of ferricrete; no S^0, Cu^0, or green.
1-4-05	80% gray/20% brown; occurrences of orange and purple; one occurrence of white precipitate; no ferricrete, S^0, Cu^0, or green.
	+¼ - 90% gray; occurrences of orange; one occurrence of purple; two occurrences of green; no S^0, Cu^0, or ferricrete.
	−¼" - 80% gray/20% brown; occurrences of orange and purple; one occurrence of green; no S^0, Cu^0, or ferricrete.
1-4-07	80% gray/20% brown; occurrences of orange and purple; one occurrence of white precipitate; no S^0, Cu^0, ferricrete, or green.
	+¼" - 90% gray/10% brown; occurrences of orange and purple; no S^0, Cu^0, ferricrete, or green.
	−¼" - 80% gray/20% brown; occurrences of orange; no purple, S^0, Cu^0, ferricrete, or green.
1-4-10	80% gray with rest being brown; occurrences of orange and purple on the larger particle sizes; no ferricrete; S^0, Cu^0, or green.

Sample#	Remarks
	+¼" - 90% gray with occurrences of orange on the larger rocks; one occurrence of green; no ferricrete, purple, S^0, or Cu^0.
	−¼" - 90% gray with occurrences of orange and purple; no ferricrete, S^0, Cu^0, or green.
1-4-12	60% gray/40% brown (brown may be wash); occurrences of orange; no purple, S^0, Cu^0, ferricrete, or green.
	+¼" - 90% gray/10% brown w/occurrences of orange and purple; no S^0, Cu^0, ferricrete, or green.
	−¼" - 70% gray/30% brown with occurrences of orange; no purple, S^0, Cu^0, ferricrete, or green.

Pile FL5

October 5, 1994

5-1-01	Brownish sand; moderate cementation.
	+¼" - moderate ferricrete; one occurrence of green coating.
	−¼" - light-moderate ferricrete; mostly brown; enough moisture to agglomerate fines to coarser particles; no blue coating.
5-1-02	Brownish sand; moderate cementation; water on larger rock; blue precipitate on rock.
	+¼" - occurrences of elemental sulfur; moderate ferricrete.
	−¼" - occurrences of S^0; moderate ferricrete; agglomerate of fines to coarse => high water content; one occurrence of green coating.
5-1-03	Yellow-red sand; strongly consolidated ferricrete.
	+¼" - all pieces were ferricrete; S^0.
	−¼" - reddish brown; heavy ferricrete; no blue or green coatings observed; moist.
5-1-05	Heavy ferricrete; S^0; brownish; wet.
	+¼" - all pieces are ferricrete or have S^0 on surface; visible pyrrhotite fragments (perhaps a "pod" of pyrrhotite); elemental Cu.
	−¼" - heavy ferricrete; some pyrrhotite; smell of sulfur; S^0 observed.
5-1-06	Orange and dry; color may shift brown -> orange with drying; moderate-heavy ferricrete; visible S^0.
	+¼" - dominated by ferricrete; visible pyrrhotite, more than one occurrence of pyrrhotite; S^0.
	−¼" - orangish-brown when dries; dominated by ferricrete; S^0 observed; more than one blue-green coating.

Sample#	Remarks
5-1-08	Yellow when dry; moderate ferricrete at 3-4" depth; green coating observed; drops of water on rock surface.
	+¼" - occurrence of green secondary mineralization; S^0; moderate ferricrete; finely disseminated pyrrhotite in rock.
	−¼" - moderate-heavy ferricrete; occurrence of blue-green; secondary mineral; yellowish-brown.
5-1-09	Orange; ferricrete at 4-5" depth; S^0 visible; hole is floored with ferricrete which is strongly agglomerated.
	+¼" - moderate-heavy ferricrete; visible pyrrhotite; no obvious secondary blue-green minerals; S^0 observed.
	−¼" - moderate-heavy ferricrete; occurrence of S^0 on −¼"/+⅛.
5-1-10	Heavy ferricrete; fairly solid ferricrete to bottom of hole; 1½" × 2" piece of pyrrhotite.
	+¼" - occurrences of pyrrhotite (freshly exposed by excavation); ferricrete and pieces of country rock with ferricrete on surfaces, as commonly seen in other holes; no visible S^0.
	−¼" - visible pyrrhotite; no visible S^0; no blue-green secondary mineralization.
5-1-12	5" unconsolidated above well consolidated ferricrete; S^0 observed; most sample collected from ferricrete layers.
	+¼" - S^0 present with ferricrete and country rock; pyrrhotite present with country rock.
	−¼" - heavy ferricrete; occurrences of S^0; no apparent secondary blue-green mineral.
5-1-14	Unconsolidated layer above ferricrete; secondary blue-green mineralization observed.
	+¼" - light ferricrete; secondary light blue coating observed; no S^0.
	−¼" - light-moderate ferricrete; no S^0.
5-1-15	Ferricrete at 6-8" below surface but not strongly consolidated; i.e. not cemented layer.
	+¼" - drops of water on some fragments; ferricrete dominant.
	−¼" - moderate ferricrete.
NOTE:	FL5, removal of first lift: The hardpan layer was 8-10" thick and the material below it was unconsolidated; one piece which broke loose was roughly circular; approximately 3 feet in diameter and 1 foot thick.

October 6, 1994

5-2-02	Red-brown; moist.
	+¼" - moderate agglomeration onto particles; visible pyrrhotite; S^0 present; no green coating.
	−¼" - moderate ferricrete; visible S^0; moist; no green surfaces.
5-2-04	Mostly gray with streaks of red.
	+¼" - slight agglomeration; all rust coated; 50-50 orange and purple; no green coatings; S^0 occurrences, some green specks in vicinity.
	−¼" - some green coatings; orange rust coatings; minimal aggregation; S^0 observed.

Sample#	Remarks
5-2-05	Gray and brown; no cement layers.
	+¼" - minimal agglomeration; minimal purple; no green surface or S^0; mostly brown-orange rust stain.
	−¼" - mostly brown with some orange and minimal purple coating; no green surfaces; some S^0.
5-2-07	Purple cement with water drops; visible S^0; some blue-green coating.
	+¼" - moderate agglomeration; orange and purple rust 50-50; some blue-green surfaces; S°; Cu^0, secondary.
	−¼" - two botryoidal Cu^0; some S^0 (minimal); moderate purple staining; no obvious agglomeration.
5-2-10	Brown; no concrete layers; some purple surfaces.
	+¼" - minimal purple; a few agglomerates; mostly orange iron stain; agglomerate have purple surfaces; one piece solid pyrrhotite; about half the surface is heavy orange stain/precipitate.
	−¼" - brown with some orange; little purple; little agglomeration; no green.
5-2-12	Mostly orange and yellow; slight cementing; visible S^0.
	+¼" - mostly orange coated with light to moderate purple; light agglomeration; one piece solid pyrrhotite; one S^0.
	−¼" - mix of orange with light to moderate purple; wet.
5-2-13	Purple; moderate cementation; S^0.
	+¼" - moderate to heavy agglomeration and purple and S^0; some visible pyrrhotite; amber spherule observed on agglomerate.
	−¼" - moderate-heavy purple; light agglomeration; light-moderate, visible S^0.
5-2-15	Substantial purple; light to moderate cementation; moderate to heavy S^0; very wet; blue specks on one piece.
	+¼" - one piece of pyrrhotite; moderate to heavy S^0 and purple and agglomeration; gypsum crystals on purple surfaces; one small blue-green spherule.
	−¼" - heavy purple; light to moderate agglomeration; moderate-heavy S^0; two green, small particles (−⅛); no Cu^0.
5-3-01	Brown; no cement.
	+¼" - brown with light purple; no S^0; slight ferricrete.
	−¼" - brown with light purple; wet sample.
5-3-03	Dominant purple with fine orange coatings; light to moderate cement.
	+¼" - moderate to heavy purple with fine orange particle coating; occurrence of S^0; light ferricrete.
	−¼" - color similar to +¼"; occurrence of S^0; light ferricrete; no green coating.

Sample#	Remarks
5-3-06	Dominant purple with fine orange particle coatings; S^0; light to moderate cementation of ferricrete.
	+¼" - mostly purple coated rocks coated with orange fines; moderate ferricrete; light S^0.
	−¼" - light ferricrete; color similar to +¼"; small, botryoidal, green particle occurrence of S^0; wet.
5-3-08	Light-moderate cement; S^0; wet; mostly purple.
	+¼" - heavy purple; light coating of orange fines; moderate ferricrete; three appearances of visible po in ferricrete; light-moderate S^0; one green botryoidal grain agglomerated to larger particle.
	−¼" - purple with coating of orange fines; occurrence of S^0; occurrence of ferricrete; wet.
5-3-09	Brown; unconsolidated.
	+¼" - moderate purple; most surfaces covered by orange fines; light ferricrete.
	−¼" - light, visible purple (−¼"/+⅛"); everything coated with orange fines; no S^0 or blue coating.
5-3-11	Heavy purple; occurrences of po visible; moderate to heavy cement; S^0.
	+¼" - heavy purple; moderate-heavy ferricrete; occurrences of pyrrhotite; moderate S^0.
	−¼" - heavy purple; light to moderate ferricrete; light S^0; no green; wet.
5-3-14	Brown; unconsolidated.
	+¼" - mostly orange-brown; heavy to moderate purple; much purple covered by orange; light ferricrete; occurrences of S^0; wet.
	−¼" - similar in color to +¼"; light ferricrete; occurrences of S^0.
5-3-16	Substantial purple (heavy); heavy cement; S^0; drops of water on larger pieces.
	+¼" - mostly purple with some orange (moderate); moderate S^0; heavy-moderate ferricrete; little green coating; minimal botryoidal Cu^0 and green precipitate; gypsum crystals everywhere.
	−¼" - occurrence of green; heavy purple; moderate orange; wet to degree; fines are clumping; light occurrences of S^0.
NOTE:	FL5, lift #3: Holes 5-3-1, 5-3-5, 5-3-9, 5-3-13 may be in reworked material.

October 8, 1994
cold, cloudy

5-4-02	Red with gray pods; moderate-heavy cement.
	+¼" - heavy orange staining; moderate ferricrete; some purple; no S^0.
	−¼" - mostly covered with fines; orange occurrence; occurrences of metal sulfides in −⅛ ; mucked up with clay.

Sample#	Remarks
5-4-04	Mostly orange with gray with visible sulfides; pyroxene rills around gray particles; moderate-heavy cement.
	+¼" - moderate to heavy orange stain; occurrence of visible sulfide and S^0; no ferricrete; fines coating.
	−¼" - light-moderate orange; remainder coated with fines; occurrences of metal sulfides.
5-4-05	Reddish orange with gray spots; moderate-heavy cement; occurrence of S^0.
	+¼" - most surface coated with orange fines; light-moderate purple; occurrence of S^0.
	−¼" - orange with purple occurrence; occurrence of ferricrete and S^0.
5-4-07	Orange; moderate-heavy cement; S^0 visible.
	+¼" - heavy orange stain; light to moderate purple; occurrence of S^0; light ferricrete texture; coated with orange sand and clay.
	−¼" - all coated with orange fines; occurrence of metallic sulfide.
5-4-10	Mostly orange with gray pod; heavy cement.
	+¼" - moderate orange stain; rest coated with gray fines; occurrence of ferricrete.
	−¼" - light orange; occurrence of purple; occurrence of black; rest coated with gray; 50-50 orange-gray.
5-4-12	Moderate cemented gray-brown; sandy texture; red zone to 3" depth; S^0 and ferricrete in this zone; brown below this.
	+¼" - half orange, half brownish-gray; occurrence of green stain; occurrence of S^0 and metallic sulfide.
	−¼" - coated with fines (gray); light to moderately orange.
5-4-13	Heavily cemented red and purple; S^0 present.
	+¼" - heavy orange; light-moderately purple and S^0; occurrences of ferricrete; some blue-green.
	−¼" - mostly orange; occurrence of green and S^0 coated with fines.
5-4-16	This hole extended to Hypalon approximately 28" depth; orange and purple ferricrete; lightly to moderately cemented; moderate to heavy S^0; light purple; moderate orange; remainder gray clay balls; remainder of hole is unconsolidated; one surface to 12" is orange then brown-gray; S^0 throughout; liner is flexible (pliable); exposed 6" diameter circle, appears to be in excellent condition.
	+¼" - moderate-heavy orange; moderate purple; light S^0; light ferricrete texture; one visible sulfide in ferricrete; hint of green in other ferricrete.

Pile FL6

October 4, 1994

Sample#	Remarks
6-1-01	
6-1-02	Standing water on 5"×8" cobble.
6-1-03	Coarse sand size material is redder; some ferricrete cement agglomerating fines to coarser material.
	+¼" - ranges from 2"×3" to 1"×¾" (dominant); ferricrete agglomerating fines up to ¼".
6-1-04	Occurrences of ferricrete but less than in 6-1-3.
	+¼" - some 1"×1½" but dominated by ⅜"×¾".
	−¼" - coarse sand more brownish than extremely red 6-1-3.
6-1-05	Blue-green agglomerated sand size particles observed.
	+¼" - mostly 1"×1¼"; minimal ferricrete present on two to three pieces.
	−¼" - brownish; some fibrous organic
6-1-06	
	+¼" - some isolated ferricrete, approximately six pieces; no green.
	−¼" - brownish; few isolated ferricrete.
6-1-07	Reddish material; moist; ferricrete visible in hole; more cemented than others.
	+¼" - dominated by ferricrete; visible sulfides; potential "hot spot."
6-1-08	Brownish; no visible ferricrete in hole.
	+¼" - little to moderate ferricrete; some blue coating.
	−¼" - few isolated pieces of ferricrete; scattered occurrences of blue coated rock.
6-1-09	Hole is brownish.
	+¼" - little to moderate ferricrete; some cemented aggregates ½"×1"; some small ½"×¼".
	−¼" - mostly brown sand; some isolated ferricrete.
6-1-10	
	+¼" - little ferricrete; some blue coating on rock surfaces.
	−¼" - brownish sand.
6-1-11	Reddish hole; blue-green coated rock encountered; cementation encountered.
	+¼" - light to moderate ferricrete; visible sulfide; similar to 6-1-7 and 6-1-3.
	−¼" - light ferricrete; minimal blue coatings.
6-1-12	Brownish with some cementation.

October 7, 1994
raining

Sample#	Remarks
	+¼" - some occurrences of ferricrete (light); no blue-green coatings.
	−¼" - light occurrences of ferricrete; no blue-green coatings.
NOTE:	FL6, lift #1: All holes have dampness with sand material; more green coating on −¼".
6-2-02	Gray with orange; no cement.
	+¼" - one cobble with green coating; one po mostly brown; light occurrence of orange; no cement; hint of green on three pieces; one purple.
	−¼" - one piece po; light-moderate orange stain; one green; no cement.
6-2-04	Gray; unconsolidated with rust spots (orange) on gravel/cobbles dominated by sand size; no cementing; moist.
	+¼" - mostly coated with gray sand to clay particles; light occurrences of rust spots (orange to purple).
	−¼" - light rust staining (−¼"/+⅛"); no green; no ferricrete; gray dominates.
6-2-05	Orange hole; no cement; one green.
	+¼" - slight orange coating; one definitive orange stain; mostly brown sand-clay coated.
	−¼" - similar to +¼" but more orange.
6-2-07	Orange mostly with gray pods.
	+¼" - one piece of ferricrete; two orange stained; rest coated with gray-orange sand-silt.
	−¼" - orangish; −¼"/+⅛" is light orange; rest coated with fines; no green.
6-2-10	Orange-brown hole; some green staining on gravel size piece; no cementation; no pods of heavy oxidation.
	+¼" - mostly coated with brown sand-clay; a few occurrences of orange rust stain.
	−¼" - similar to +¼" but a few occurrences of purple and one green particle.
6-2-12	Mostly gray with 6" pod of orange oxidation; no green stain; orange pod is unconsolidated.
	+¼" - one particle with purple and orange rust; blue-green stain and visible po; mostly gray; light orange staining one visible po; occurrences of purple and orange spotted pieces; no ferricrete.
	−¼" - mostly gray sand and clay coated fragments; light orange stain; no green or purple stain.
6-2-13	Entirely orange; two cobbles with secondary green staining.
	+¼" - all orange iron stained; one green; four to five occurrences of ferricrete particles.
	−¼" - mostly orange stained; light purple; a few green occurrences; sand size is more orange.
6-2-15	Half of hole is gray, while half is orange-stained and slightly cemented; discrete rust spots on larger particles; some gravel sized particles totally coated with orange.

Sample#	Remarks
	+¼" - one (particle) occurrence of ferricrete; occurrence of green stain on one surface; moderate orange iron stain; light to moderate gray due to coating with fines; 60-40 orange:gray.
	−¼" - moderate orange stain; light purple; one green surface.
NOTE:	FL6, lift #2: Due to rain, agglomeration occurs.
6-3-01	Brown-gray; cement on bottom; cobbles with blue coating.
	+¼" - occurrence of po; light purple stain occurrence of ferricrete; mostly coated with gray silt and clay.
	−¼" - occurrences of purple stain (−¼"/+⅛"); one ferricrete texture.
6-3-03	Gray; unconsolidated; occurrences of purple spots on rocks.
	+¼" - occurrences of purple and orange spots; mostly gray; all coated with fines.
	−¼" - same as +¼" except one green surface.
6-3-06	Brown; visible purple; po present.
	+¼" - one ferricrete particle; light purple spotting; most coated with gray fines.
	−¼" - purple spots occur; gray clay balls.
6-3-08	Uncemented; gray-brown; fleck of blue.
	+¼" - light purple spotted; one ferricrete particle; rest coated with fines; one blue stain.
	−¼" - Clay balls with some purple; one ferricrete.
6-3-09	Brown; moderate cement; purple at bottom; green spots on some cobbles.
	+¼" - $CuFeS_2$ in particle with green spots on exterior; S^0; occurrences of blue stain; light purple staining; occurrences of ferricrete; most pieces coated with fines.
	−¼" - occurrences (approximately three) of greens; light purple; clay balls common.
6-3-11	Brown-orange with purple stains in bottom; no cement.
	+¼" - mostly coated with gray fines; occurrence of purple.
	−¼" - occurrences of purple spotting; no ferricrete; gray clay balls.
6-3-14	Brown with moderate cementation; some purple staining.
	+¼" - particles coated with fines; light purple spotting; three occurrences of ferricrete.
	−¼" - occurrences of purple staining; clay balls.
6-3-16	Gray; unconsolidated; some cobbles with purple spots.
	+¼" - mostly covered with gray fines; one visible purple stain; some orange staining.
	−¼" - gray clay balls.

October 10, 1994
sunny, warm

Sample #	Remarks
6-4-02	Mostly brown with small pockets of fine gray; one pod of orange; heavily cemented.
	+¼" - most surfaces coated with brown fines; light to moderate purple; gray on freshly broken surfaces; occurrence of ferricrete; one S^0.
	−¼" - mostly orangish-brown; light purple; light ferricrete; no green; one Cu^0.
6-4-04	Mostly brown; some orange spots and gray rock and fines (gray); a couple occurrences of ferricrete.
	+¼" - mostly coated with orange-brown fines; light purple; gray-black on fresh surfaces; occurrences of ferricrete; occurrence of ¾" piece which disintegrated; no S^0 or Cu^0.
	−¼" - mostly orangish-brown; occurrence of ferricrete and purple; two blue-greens; no S^0 or Cu^0.
6-4-05	Orange-brown; pockets of gray and rust compaction but no cementing.
	+¼" - most particles coated with brownish-gray fines; fresh surfaces are gray; light to moderate orange rust; three occurrences of green; occurrences of purple; no ferricrete, S^0, Cu^0; occurrences of visible sulfide.
	−¼" - gray-brown; occurrences of purple; occurrences of orange; one occurrence of ferricrete; no greens, S^0, Cu^0.
6-4-07	Brown:gray = 60:40; gravel in compacted fines.
	+¼" - orange-brown:gray = 60:40; surfaces usually coated with orange-brown fines; occurrence - light purple; light blue-green; no S^0, Cu^0, ferricrete.
	−¼" - brown and gray; occurrences of purple; one occurrence of ferricrete; no S^0, Cu^0.
6-4-10	Brown with pocket of gray; moderate compaction; two ⅛" metal sulfide grains.
	+¼" - orangish-brown; light purple; no S^0, Cu^0, blue-green, or ferricrete.
	−¼" - brown; occurrence of purple; no ferricrete, S^0, Cu^0.
6-4-12	Orangish-brown; light compaction; no gray or ferricrete.
	+¼" - brownish-rust; occurrences of purple; no S^0 or ferricrete; fresh surfaces have some sulfides and are gray.
	−¼" - orangish-brown; occurrences of purple; two occurrences of green; no S^0 or ferricrete.
NOTE:	FL6, lift #4: Cleared; a cemented layer was encountered; this layer was not as strongly cemented as that in FL5.

Table A5.8.1. Summary of visual descriptions and acid concentration present field notes for the 0.6 %S (FL1) pile.

Sample	Color	Coat	Color[1]	Agglom[2]	Oxid[2]	Moist[3]	S^0 [4]	Cu^0 [4]	Gyp[5]	Sulfide[6]	Other Notes
FL1-1-01											
FL1-1-02	brown		gr bl	light			no				
FL1-1-03											
FL1-1-04	brown		gr	none			no				
FL1-1-05			gr								
FL1-1-06	brown		gr	none			no				
FL1-1-07			bl								
FL1-1-08	brown		none	none			no				
FL1-1-09											
FL1-1-10	brown	rust	ru	none			no				
FL1-1-11											
FL1-1-12	brown		ru	none			no				
FL1-1-13											
FL1-1-14	brown		ru	none			no				
FL1-1-15	gray		ru or								
FL1-2-01	brown	gray	pr or gr	none							
FL1-2-02											
FL1-2-03	brown		pr or gr	none							
FL1-2-04	brown		pr gr	none							
FL1-2-05	rust		gr bl	none							
FL1-2-06	brown	gray	ru pr or	none	mod					po	
FL1-2-07	gray		pr gr bl	none	light						
FL1-2-08	gray		ru pr or	none	light						
FL1-2-09	brown	rust	pr or	none							black rock
FL1-2-10	brown				light						
FL1-2-11	gray		ru pr or	none							
FL1-2-12	gray		or rd ye				yellow				
FL1-2-13	brown			none							
FL1-2-14	brown	gray	ru pr or	none	mod						
FL1-2-15	gray		ru or	none							
FL1-2-16	brown		ru pr or	none							
FL1-3-01	gray		unknown								
FL1-3-02	brown		pr gr bl	none	mod						
FL1-3-03	brown		unknown								
FL1-3-04	gray		ru pr or	none	light						
FL1-3-05	gray		pr or								
FL1-3-06	brown		gr bl		light						
FL1-3-07	gray		pr or	none			yellow				
FL1-3-08	gray		or rd		light						
FL1-3-09	gray		or								
FL1-3-10	gray		pr or	none	light						
FL1-3-11	gray				light						
FL1-3-12	gray		pr or gr	none	light						
FL1-3-13	gray		pr or	none	light						
FL1-3-14	gray				light						
FL1-3-15	gray		or	none	light						
FL1-3-16	gray		or								
FL1-4-01	gray		pr or								
FL1-4-02	gray		pr or	none			no	no	white	yes	
FL1-4-03	gray		pr br	none			yes	yes			
FL1-4-04	brown	gray	pr or gr	none			no	no			
FL1-4-05	gray		pr or gr	none		wet	yes	no	white	py	
FL1-4-06	gray		pr	none	mod						
FL1-4-07	gray		pr or	none	light		no	no	white		
FL1-4-08	gray		pr		light						
FL1-4-09	brown		ru	none							
FL1-4-10	gray		ru pr or	none		wet	yes	no			
FL1-4-11			pr br	none							
FL1-4-12	brown		pr or	none	mod		yes	no			

[1] color of surface coating noted: gr = green, bl = blue, ru = rust, or = orange, pr = purple, rd = red, ye = yellow; blank = no mention
[2] extent; none, light, mod = moderate, or heavy; ; blank = no mention
[3] amount of moisture in the sample in the hole; dry, moist, wet, or rain = raining during sampling
[4] native element noted (yellow assumed to be native sulfur)
[5] gyp = gypsum; white precipitate assumed to be gypsum
[6] specific sulfides noted; po = pyrrhotite, py = pyrite, ch = chalcopyrite, yes = presence noted but not while specific sulfide

Table A5.8.2. Summary of visual descriptions and acid concentration present field notes for the 0.8 %S (FL6) pile.

Sample	Color	Coat	Color[1]	Agglom[2]	Oxid[2]	Moist[3]	S^0 [4]	Cu^0 [4]	Gyp[5]	Sulfide[6]	Other Notes
FL6-1-01			gr	mod		moist					
FL6-1-02			gr			wet					
FL6-1-03				light		wet					
FL6-1-04	brown			light		moist					
FL6-1-05	brown		gr bl	light		moist					fibr organic
FL6-1-06	brown		gr	light		moist					
FL6-1-07	red		ru gr	mod		moist				yes	hot spot
FL6-1-08	brown		bl	none		moist					
FL6-1-09	brown		gr	light		moist					
FL6-1-10	brown		bl	light		moist					
FL6-1-11	red		gr bl	mod		moist				yes	
FL6-1-12	brown			light		moist					
FL6-2-01			pr bl	none	light	rain					
FL6-2-02	gray		pr or gr	none	light	rain				po	
FL6-2-03	gray		pr	none	light	rain					
FL6-2-04	gray		ru or	none	light	rain					paper plastc
FL6-2-05	gray orange		or gr	none	light	rain					
FL6-2-06			gr bl		mod	rain					
FL6-2-07	gray		or gr bl	light	mod	rain					
FL6-2-08	gray		ru		light	rain					
FL6-2-09			gr bl		mod	rain					
FL6-2-10	brown orange		or gr bl	none	mod	rain	yellow				
FL6-2-11	gray		pr		light	rain					
FL6-2-12	gray orange		pr or gr	none	light	rain				po	qtz
FL6-2-13	orange		gr bl	light	mod	rain					
FL6-2-14			gr bl	light	heavy	rain					
FL6-2-15	gray orange		ru or gr	light	mod	rain					
FL6-2-16						rain					
FL6-3-01	brown gray		pr bl	mod	light	rain				po	
FL6-3-02			gr bl		mod	rain					
FL6-3-03	gray		pr or gr	none	light	rain					
FL6-3-04			pr gr	none	mod	rain					
FL6-3-05			gr bl	none	light	rain					
FL6-3-06	brown		pr bl	light	mod	rain				po	clay balls
FL6-3-07	brown		gr bl	light	light	rain					
FL6-3-08	brown gray		pr gr bl	none	mod	rain		yes			clay balls
FL6-3-09	brown		pr gr bl	mod	heavy	rain	yes			ch	clay balls
FL6-3-10	gray		bl		light	rain					
FL6-3-11	brown orange		pr bl	none	light	rain	yes				clay balls
FL6-3-12			pr gr bl	none	mod	rain	yes	yes			
FL6-3-13			pr bl	mod	mod	rain					
FL6-3-14	brown		pr bl	mod	mod	rain					clay balls
FL6-3-15			pr	none	light	rain					
FL6-3-16	gray		pr or bl	none	light	rain					clay balls
FL6-4-01	brown orange			heavy							
FL6-4-02	brown orange		pr	heavy	mod		yes	yes			
FL6-4-03	brown		ru pr bl	heavy	mod		yellow	yes			
FL6-4-04	brown		ru pr gr	heavy	mod		yellow	yes			
FL6-4-05	brown rust		ru pr or	mod	mod		no	no		yes	
FL6-4-06	brown			heavy	mod						
FL6-4-07	brown		pr gr bl	mod	mod		no	no			
FL6-4-08			pr	mod	light		no	yes			
FL6-4-09	brown		pr gr bl	mod	light						
FL6-4-10	brown orange		pr	heavy	mod		no	no		yes	
FL6-4-11				heavy	mod		no				
FL6-4-12	brown orange		pr gr	light	mod					yes	

[1] color of surface coating noted: gr = green, bl = blue, ru = rust, or = orange, pr = purple, rd = red, ye = yellow; blank = no mention
[2] extent; none, light, mod = moderate, or heavy; ; blank = no mention
[3] amount of moisture in the sample in the hole; dry, moist, wet, or rain = raining during sampling
[4] native element noted (yellow assumed to be native sulfur)
[5] gyp = gypsum; white precipitate assumed to be gypsum
[6] specific sulfides noted; po = pyrrhotite, py = pyrite, ch = chalcopyrite, yes = presence noted but not while specific sulfide

Table A5.8.3. Summary of visual descriptions and acid concentration present field notes for the 1.4 %S (FL5) pile.

Sample	Color	Coat	Color[1]	Agglom[2]	Oxid[2]	Moist[3]	S^0 [4]	Cu^0 [4]	Gyp[5]	Sulfide[6]	Other Notes
FL5-1-01	brown		gr	light	none						
FL5-1-02	brown		gr bl	mod	heavy	wet	yes		white		pipe in hole
FL5-1-03	red		bl	heavy		moist	yes				
FL5-1-04				heavy							
FL5-1-05	brown			heavy	heavy	wet	yes	yes		po	pod pyrrho
FL5-1-06	brown	orange	gr bl	heavy		dry	yes			po	
FL5-1-07				heavy			yes				
FL5-1-08	brown	yellow	pr gr bl	mod		wet	yes			po	blgr 2nd min
FL5-1-09	orange			heavy			yes			po	
FL5-1-10				mod	mod		no			po	
FL5-1-11				heavy	heavy		yes	yes			
FL5-1-12			unknown	heavy	heavy		yes			po	
FL1-1-13			pr bl	none							
FL5-1-14			gr bl	none	heavy		no				blgr 2nd min
FL5-1-15			bl	mod	heavy	wet	yes			po	
FL5-2-01			pr	none		moist					
FL5-2-02	brown	red	pr	none		moist	yes			po	
FL5-2-03				heavy	heavy						
FL5-2-04	gray	red	ru pr or	heavy		moist	yes				
FL5-2-05	brown	gray	pr or	none		moist	yes				
FL5-2-06				none							
FL5-2-07			pr gr bl	light		wet	yes	yes			botryoidal
FL5-2-08				heavy	heavy		yes				
FL5-2-09				none			yes				
FL5-2-10	brown		pr or	none		wet	yes			po	
FL5-2-11			pr	light			yes				
FL5-2-12	orange	yello	pr or gr	mod		wet	yes				
FL5-2-13			pr bl	mod			yes			po	amber spher
FL5-2-14			gr bl		heavy		yes				
FL5-2-15			pr gr bl	mod	heavy	wet	yes	yes	yes	po	
FL5-2-16					heavy		yes				
FL5-3-01	brown		pr gr bl	none		wet	yes				
FL5-3-02			pr								
FL5-3-03			pr or	light		wet	yes				
FL5-3-04			pr				yes				
FL5-3-05				light			yes			po	
FL5-3-06			pr or bl	mod			yes			po	botryoidal
FL5-3-07			pr	mod		moist	yes				
FL5-3-08			pr or gr	mod		wet	yes			po	gr 2 min
FL5-3-09	brown		pr or	none		moist	yes				
FL5-3-10			pr			moist	yes				
FL5-3-11			pr bl	mod		wet	yes	yes		po	
FL5-3-12			pr				yes				
FL5-2-13			bl	light		wet	yes	yes			
FL5-3-14	brown		pr or	none		wet	yes				
FL5-3-15			bl	light			yes			po	
FL5-3-16			pr	none		moist	yes				
FL5-4-01	brown	red	pr	heavy	heavy		yes			yes	
FL5-4-02	brown			heavy	heavy		yes				
FL5-4-03	gray			heavy	heavy		yes				
FL5-4-04	gray	rust	or	heavy			yes	yes		yes	pyx rills
FL5-4-05	red	orange	pr	heavy	heavy		yes				
FL5-4-06			pr	heavy	heavy		yes				
FL5-4-07	orange		pr or	heavy	heavy		yes			yes	
FL5-4-08	brown			heavy	heavy		yes				
FL5-4-09			pr	heavy	heavy		yes				
FL5-4-10	orange		pr or	heavy	heavy		yes				
FL5-4-11				heavy	heavy		yes				
FL5-4-12	red	orange	ru	heavy	heavy		yes			yes	
FL5-4-13	brown	red	pr or gr	heavy	heavy		yes				
FL5-4-14	brown	rust		heavy			yes				
FL5-4-15	rust			heavy	heavy						
FL5-4-16	rust		pr or	light	heavy		yes				clay balls

[1] color of surface coating noted: gr = green, bl = blue, ru = rust, or = orange, pr = purple, rd = red, ye = yellow; blank = no mention
[2] extent; none, light, mod = moderate, or heavy; ; blank = no mention
[3] amount of moisture in the sample in the hole; dry, moist, wet, or rain = raining during sampling
[4] native element noted (yellow assumed to be native sulfur)
[5] gyp = gypsum; white precipitate assumed to be gypsum
[6] specific sulfides noted; po = pyrrhotite, py = pyrite, ch = chalcopyrite, yes = presence noted but not while specific sulfide

Table A5.8.4. Measured and calculated corner locations of the three test piles dismantled in October 1994 (FL1, FL6, and FL5).

Pile	S %	Base or Top	X[1] feet	Y[1] feet	Z[1] feet	Ground[2] feet	Height[3] feet	Calc Height[4] feet
FL1	0.6	base	2	5	1597	1594	0	3
FL1	0.6	base	48	0	1599	1594	0	5
FL1	0.6	base	6	80	1594	1594	0	0
FL1	0.6	base	45	74	1599	1594	0	5
FL1	0.6	top	23	40	1608	1594	11.5	14
FL1	0.6	top	30	39	1609	1594	10	15
FL1	0.6	top	23	59	1604	1594	11	10
FL1	0.6	top	30	58	1606	1594	9	12
FL6	0.8	base	5	0	1595	1594	0	1
FL6	0.8	base	50	4	1596	1594	0	2
FL6	0.8	base	45	72	1595	1594	0	1
FL6	0.8	base	6	76	1594	1594	0	0
FL6	0.8	top	20	20	1603	1594	10	9
FL6	0.8	top	34	21	1607	1594	10	13
FL6	0.8	top	34	41	1604	1594	13.5	10
FL6	0.8	top	20	40	1606	1594	12	12
FL5	1.4	base	4	4	1595	1594	0	1
FL5	1.4	base	45	0	1595	1594	0	1
FL5	1.4	base	11	80	1594	1594	0	0
FL5	1.4	base	48	76	1593	1594	0	-1
FL5	1.4	top	14	22	1603	1594	5	9
FL5	1.4	top	26	21	1603	1594	7	9
FL5	1.4	top	14	41	1604	1594	6.5	10
FL5	1.4	top	24	40	1605	1594	9	11

[1] X, Y, and Z represent locations of pile corners in three dimensional space.
[2] As recorded on Figure A1.1 of Eger and Lapakko 1985 (appendices).
[3] As indicated by Bill White on diagrams from October 1994.
[4] Calc Height = Z – Ground; height and calc height should be about equal.

Table A5.8.5. Sample locations for the spatially distributed samples for piles FL1, FL6, and FL5.

Sample	Pile	S %	Lift	X Space feet	Y Space feet	Height feet	Z Space feet
FL1-1-01	FL1	0.6	1	30.0	39.0	9.0	1603.0
FL1-1-02	FL1	0.6	1	30.0	44.0	9.5	1603.5
FL1-1-03	FL1	0.6	1	30.0	48.0	10.0	1604.0
FL1-1-04	FL1	0.6	1	30.0	53.0	10.5	1604.5
FL1-1-05	FL1	0.6	1	30.0	58.0	11.0	1605.0
FL1-1-06	FL1	0.6	1	26.5	39.5	9.5	1603.5
FL1-1-07	FL1	0.6	1	26.5	44.5	10.0	1604.0
FL1-1-08	FL1	0.6	1	26.5	48.5	10.5	1604.5
FL1-1-09	FL1	0.6	1	26.5	53.5	11.0	1605.0
FL1-1-10	FL1	0.6	1	26.5	58.5	11.5	1605.5
FL1-1-11	FL1	0.6	1	23.0	40.0	10.0	1604.0
FL1-1-12	FL1	0.6	1	23.0	45.0	10.5	1604.5
FL1-1-13	FL1	0.6	1	23.0	49.0	10.5	1604.5
FL1-1-14	FL1	0.6	1	23.0	54.0	11.0	1605.0
FL1-1-15	FL1	0.6	1	23.0	59.0	11.5	1605.5
FL1-2-01	FL1	0.6	2	30.0	39.0	7.0	1601.0
FL1-2-02	FL1	0.6	2	30.0	45.0	8.0	1602.0
FL1-2-03	FL1	0.6	2	30.0	51.0	8.0	1602.0
FL1-2-04	FL1	0.6	2	30.0	58.0	9.0	1603.0
FL1-2-05	FL1	0.6	2	27.3	39.5	7.0	1601.0
FL1-2-06	FL1	0.6	2	27.3	45.5	8.0	1602.0
FL1-2-07	FL1	0.6	2	27.3	51.5	8.0	1602.0
FL1-2-08	FL1	0.6	2	27.3	58.5	9.0	1603.0
FL1-2-09	FL1	0.6	2	25.3	39.5	7.5	1601.5
FL1-2-10	FL1	0.6	2	25.3	45.5	7.5	1601.5
FL1-2-11	FL1	0.6	2	25.3	51.5	8.0	1602.0
FL1-2-12	FL1	0.6	2	25.3	58.5	8.0	1602.0
FL1-2-13	FL1	0.6	2	23.0	40.0	7.5	1601.5
FL1-2-14	FL1	0.6	2	23.0	46.0	7.5	1601.5
FL1-2-15	FL1	0.6	2	23.0	52.0	8.0	1602.0
FL1-2-16	FL1	0.6	2	23.0	59.0	8.0	1602.0
FL1-3-01	FL1	0.6	3	30.0	39.0	3.5	1597.5
FL1-3-02	FL1	0.6	3	30.0	45.0	4.0	1598.0
FL1-3-03	FL1	0.6	3	30.0	51.0	5.0	1599.0
FL1-3-04	FL1	0.6	3	30.0	58.0	5.5	1599.5
FL1-3-05	FL1	0.6	3	27.3	39.5	4.0	1598.0
FL1-3-06	FL1	0.6	3	27.3	45.5	4.5	1598.5
FL1-3-07	FL1	0.6	3	27.3	51.5	5.0	1599.0
FL1-3-08	FL1	0.6	3	27.3	58.5	5.5	1599.5
FL1-3-09	FL1	0.6	3	25.3	39.5	4.5	1598.5
FL1-3-10	FL1	0.6	3	25.3	45.5	5.0	1599.0
FL1-3-11	FL1	0.6	3	25.3	51.5	5.0	1599.0
FL1-3-12	FL1	0.6	3	25.3	58.5	5.0	1599.0
FL1-3-13	FL1	0.6	3	23.0	40.0	5.0	1599.0
FL1-3-14	FL1	0.6	3	23.0	46.0	5.0	1599.0
FL1-3-15	FL1	0.6	3	23.0	52.0	5.0	1599.0

Table A5.8.5. Sample locations for the spatially distributed samples for piles FL1, FL6, and FL5.

Sample	Pile	S %	Lift	X Space feet	Y Space feet	Height feet	Z Space feet
FL1-3-16	FL1	0.6	3	23.0	59.0	5.0	1599.0
FL1-4-01	FL1	0.6	4	30.0	42.0	0.0	1594.0
FL1-4-02	FL1	0.6	4	30.0	47.5	1.0	1595.0
FL1-4-03	FL1	0.6	4	30.0	53.0	1.0	1595.0
FL1-4-04	FL1	0.6	4	30.0	58.0	2.0	1596.0
FL1-4-05	FL1	0.6	4	24.5	42.5	1.5	1595.5
FL1-4-06	FL1	0.6	4	24.5	48.0	1.5	1595.5
FL1-4-07	FL1	0.6	4	24.5	53.5	2.0	1596.0
FL1-4-08	FL1	0.6	4	24.5	58.5	2.0	1596.0
FL1-4-09	FL1	0.6	4	19.0	43.0	2.5	1596.5
FL1-4-10	FL1	0.6	4	19.0	48.5	2.5	1596.5
FL1-4-11	FL1	0.6	4	19.0	54.0	2.5	1596.5
FL1-4-12	FL1	0.6	4	19.0	59.0	2.5	1596.5
FL6-1-01	FL6	0.8	1	33.0	24.5	10.0	1604.0
FL6-1-02	FL6	0.8	1	33.0	28.5	11.0	1605.0
FL6-1-03	FL6	0.8	1	33.0	32.5	12.0	1606.0
FL6-1-04	FL6	0.8	1	33.0	36.5	13.5	1607.5
FL6-1-05	FL6	0.8	1	27.0	24.0	10.0	1604.0
FL6-1-06	FL6	0.8	1	27.0	28.0	11.0	1605.0
FL6-1-07	FL6	0.8	1	27.0	32.0	12.0	1606.0
FL6-1-08	FL6	0.8	1	27.0	36.0	13.0	1607.0
FL6-1-09	FL6	0.8	1	21.0	23.5	10.0	1604.0
FL6-1-10	FL6	0.8	1	21.0	27.5	11.0	1605.0
FL6-1-11	FL6	0.8	1	21.0	31.5	11.0	1605.0
FL6-1-12	FL6	0.8	1	21.0	35.5	12.0	1606.0
FL6-2-01	FL6	0.8	2	34.0	21.0	9.0	1603.0
FL6-2-02	FL6	0.8	2	34.0	28.0	9.0	1603.0
FL6-2-03	FL6	0.8	2	34.0	34.0	9.0	1603.0
FL6-2-04	FL6	0.8	2	34.0	41.0	9.0	1603.0
FL6-2-05	FL6	0.8	2	29.0	21.0	9.0	1603.0
FL6-2-06	FL6	0.8	2	29.0	28.0	9.0	1603.0
FL6-2-07	FL6	0.8	2	29.0	33.5	9.0	1603.0
FL6-2-08	FL6	0.8	2	29.0	40.5	9.0	1603.0
FL6-2-09	FL6	0.8	2	24.5	20.0	9.0	1603.0
FL6-2-10	FL6	0.8	2	24.5	27.0	9.0	1603.0
FL6-2-11	FL6	0.8	2	24.5	33.5	8.0	1602.0
FL6-2-12	FL6	0.8	2	24.5	40.5	8.0	1602.0
FL6-2-13	FL6	0.8	2	20.0	20.0	9.0	1603.0
FL6-2-14	FL6	0.8	2	20.0	27.0	9.0	1603.0
FL6-2-15	FL6	0.8	2	20.0	33.0	8.0	1602.0
FL6-2-16	FL6	0.8	2	20.0	40.0	8.0	1602.0
FL6-3-01	FL6	0.8	3	34.0	21.0	5.0	1599.0
FL6-3-02	FL6	0.8	3	34.0	28.0	5.0	1599.0
FL6-3-03	FL6	0.8	3	34.0	34.0	5.5	1599.5
FL6-3-04	FL6	0.8	3	34.0	41.0	5.5	1599.5
FL6-3-05	FL6	0.8	3	29.0	20.5	5.0	1599.0

Table A5.8.5. Sample locations for the spatially distributed samples for piles FL1, FL6, and FL5.

Sample	Pile	S %	Lift	X Space feet	Y Space feet	Height feet	Z Space feet
FL6-3-06	FL6	0.8	3	29.0	27.5	5.0	1599.0
FL6-3-07	FL6	0.8	3	29.0	34.5	5.5	1599.5
FL6-3-08	FL6	0.8	3	29.0	40.5	5.5	1599.5
FL6-3-09	FL6	0.8	3	24.5	20.5	6.0	1600.0
FL6-3-10	FL6	0.8	3	24.5	27.5	6.0	1600.0
FL6-3-11	FL6	0.8	3	24.5	34.5	5.0	1599.0
FL6-3-12	FL6	0.8	3	24.5	40.5	5.0	1599.0
FL6-3-13	FL6	0.8	3	20.0	20.0	6.0	1600.0
FL6-3-14	FL6	0.8	3	20.0	27.0	6.0	1600.0
FL6-3-15	FL6	0.8	3	20.0	33.0	5.0	1599.0
FL6-3-16	FL6	0.8	3	20.0	40.0	5.0	1599.0
FL6-4-01	FL6	0.8	4	34.0	21.0	0.0	1594.0
FL6-4-02	FL6	0.8	4	34.0	28.0	1.0	1595.0
FL6-4-03	FL6	0.8	4	34.0	34.0	1.0	1595.0
FL6-4-04	FL6	0.8	4	34.0	41.0	2.0	1596.0
FL6-4-05	FL6	0.8	4	25.0	20.5	0.0	1594.0
FL6-4-06	FL6	0.8	4	25.0	27.5	0.5	1594.5
FL6-4-07	FL6	0.8	4	25.0	33.5	0.5	1594.5
FL6-4-08	FL6	0.8	4	25.0	40.5	1.0	1595.0
FL6-4-09	FL6	0.8	4	16.0	20.0	0.0	1594.0
FL6-4-10	FL6	0.8	4	16.0	27.0	0.0	1594.0
FL6-4-11	FL6	0.8	4	16.0	33.0	0.0	1594.0
FL6-4-12	FL6	0.8	4	16.0	40.0	0.0	1594.0
FL5-1-01	FL5	1.4	1	26.0	21.0	7.0	1601.0
FL5-1-02	FL5	1.4	1	25.5	26.0	7.5	1601.5
FL5-1-03	FL5	1.4	1	25.0	31.0	8.0	1602.0
FL5-1-04	FL5	1.4	1	24.5	36.0	8.5	1602.5
FL5-1-05	FL5	1.4	1	24.0	40.0	9.0	1603.0
FL5-1-06	FL5	1.4	1	20.0	21.5	3.0	1597.0
FL5-1-07	FL5	1.4	1	20.0	26.5	6.5	1600.5
FL5-1-08	FL5	1.4	1	19.5	31.5	7.0	1601.0
FL5-1-09	FL5	1.4	1	19.5	36.5	7.5	1601.5
FL5-1-10	FL5	1.4	1	19.0	40.5	8.0	1602.0
FL5-1-11	FL5	1.4	1	14.0	22.0	5.0	1599.0
FL5-1-12	FL5	1.4	1	14.0	27.0	5.5	1599.5
FL1-1-13	FL5	1.4	1	14.0	32.0	6.0	1600.0
FL5-1-14	FL5	1.4	1	14.0	37.0	6.5	1600.5
FL5-1-15	FL5	1.4	1	14.0	41.0	6.5	1600.5
FL5-2-01	FL5	1.4	2	26.0	21.0	5.1	1599.1
FL5-2-02	FL5	1.4	2	25.0	27.0	5.1	1599.1
FL5-2-03	FL5	1.4	2	25.0	33.0	5.2	1599.2
FL5-2-04	FL5	1.4	2	24.0	40.0	5.2	1599.2
FL5-2-05	FL5	1.4	2	22.0	21.5	5.2	1599.2
FL5-2-06	FL5	1.4	2	21.0	27.5	5.0	1599.0
FL5-2-07	FL5	1.4	2	21.0	33.5	5.0	1599.0
FL5-2-08	FL5	1.4	2	20.0	40.5	4.5	1598.5

Table A5.8.5. Sample locations for the spatially distributed samples for piles FL1, FL6, and FL5.

Sample	Pile	S %	Lift	X Space feet	Y Space feet	Height feet	Z Space feet
FL5-2-09	FL5	1.4	2	18.0	21.5	5.0	1599.0
FL5-2-10	FL5	1.4	2	18.0	27.5	5.0	1599.0
FL5-2-11	FL5	1.4	2	17.0	33.5	4.5	1598.5
FL5-2-12	FL5	1.4	2	17.0	40.5	4.5	1598.5
FL5-2-13	FL5	1.4	2	14.0	22.0	5.0	1599.0
FL5-2-14	FL5	1.4	2	14.0	28.0	4.5	1598.5
FL5-2-15	FL5	1.4	2	14.0	34.0	4.5	1598.5
FL5-2-16	FL5	1.4	2	14.0	41.0	4.0	1598.0
FL5-3-01	FL5	1.4	3	26.0	21.0	3.3	1597.3
FL5-3-02	FL5	1.4	3	25.0	27.0	3.3	1597.3
FL5-3-03	FL5	1.4	3	25.0	33.0	3.2	1597.2
FL5-3-04	FL5	1.4	3	24.0	40.0	3.2	1597.2
FL5-3-05	FL5	1.4	3	22.0	21.5	2.7	1596.7
FL5-3-06	FL5	1.4	3	21.0	27.5	2.6	1596.6
FL5-3-07	FL5	1.4	3	21.0	33.5	2.6	1596.6
FL5-3-08	FL5	1.4	3	20.0	40.5	2.5	1596.5
FL5-3-09	FL5	1.4	3	18.0	21.5	2.0	1596.0
FL5-3-10	FL5	1.4	3	18.0	27.5	2.2	1596.2
FL5-3-11	FL5	1.4	3	17.0	33.5	2.2	1596.2
FL5-3-12	FL5	1.4	3	17.0	40.5	2.5	1596.5
FL5-2-13	FL5	1.4	3	14.0	22.0	1.3	1595.3
FL5-3-14	FL5	1.4	3	14.0	28.0	1.5	1595.5
FL5-3-15	FL5	1.4	3	14.0	34.0	1.8	1595.8
FL5-3-16	FL5	1.4	3	14.0	41.0	2.0	1596.0
FL5-4-01	FL5	1.4	4	26.0	21.0	0.0	1594.0
FL5-4-02	FL5	1.4	4	25.0	27.0	0.2	1594.2
FL5-4-03	FL5	1.4	4	25.0	33.0	0.4	1594.4
FL5-4-04	FL5	1.4	4	24.0	40.0	0.5	1594.5
FL5-4-05	FL5	1.4	4	22.0	21.5	0.0	1594.0
FL5-4-06	FL5	1.4	4	21.0	27.5	0.2	1594.2
FL5-4-07	FL5	1.4	4	21.0	33.5	0.4	1594.4
FL5-4-08	FL5	1.4	4	20.0	40.5	0.5	1594.5
FL5-4-09	FL5	1.4	4	18.0	21.5	0.0	1594.0
FL5-4-10	FL5	1.4	4	18.0	27.5	0.2	1594.2
FL5-4-11	FL5	1.4	4	17.0	33.5	0.4	1594.4
FL5-4-12	FL5	1.4	4	17.0	40.5	0.5	1594.5
FL5-4-13	FL5	1.4	4	14.0	22.0	0.0	1594.0
FL5-4-14	FL5	1.4	4	14.0	28.0	0.2	1594.2
FL5-4-15	FL5	1.4	4	14.0	34.0	0.4	1594.4
FL5-4-16	FL5	1.4	4	14.0	41.0	0.5	1594.5

As determined using table A1.1 from Eger and Lapakko 1985 and notes from 1994 dismantling of piles.

Section A5.9. Chemistry of the spatially distributed samples (see methods section in text for analytical techniques used).

Table A5.9.1. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-01-02 (in percent). Mass weighted average of −0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	2.30	1.41	0.053	0.26	0.80	12.92	6.23	18.16	0.928	6.52	0.177	1.577	36.1	0.631	0.010	0.140	0.019
-2.0mm/ +0.5mm	1.28	1.45	0.080	0.42	0.005	12.58	5.84	17.30	1.031	5.65	0.145	1.618	37.3	0.872	0.006	0.089	0.018
-0.5mm/ +0.177mm	0.96	1.21	0.114	0.37	3.46	12.74	5.51	17.87	1.107	5.85	0.134	1.645	37.3	0.784	0.005	0.061	0.017
-0.177mm/ +0.149 mm	0.44	0.89	0.109	0.41	0.18	13.47	5.79	17.87	1.168	6.00	0.134	1.671	35.6	0.657	0.005	0.055	0.018
-0.149mm/ +0.105mm	0.49	0.72	0.107	0.43	0.005	12.75	5.61	18.30	1.083	5.92	0.151	1.604	34.3	0.555	0.005	0.051	0.018
-0.105mm/ +0.075mm	0.43	0.66	0.107	0.41	0.05	12.43	5.54	18.87	1.001	5.89	0.179	1.576	31.2	0.564	0.005	0.049	0.018
-0.075mm / +0.053mm	0.62	0.63	0.156	0.63	0.23	12.06	5.49	18.87	0.955	5.77	0.163	1.446	34.3	0.519	0.005	0.050	0.015
-0.053mm	0.32	0.57	0.194	0.70	0.17	11.68	5.61	19.30	0.828	5.41	0.163	1.388	34.9	0.529	0.052	0.053	0.018
mass weighted average		1.149	0.093	0.391	0.799	12.68	5.84	18.11	1.005	6.03	0.159	1.581	35.79	0.674	0.009	0.090	0.012
total	6.84																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.2. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-01-04 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	3.86	2.000	0.055	0.06	0.005	11.17	5.26	18.87	0.372	11.92	0.21	1.972	36.1	0.773	0.015	0.160	0.014
-2.0mm/ +0.5mm	2.97	1.970	0.076	0.11	0.005	10.70	5.01	19.59	0.371	11.76	0.20	1.545	35.0	0.813	0.015	0.180	0.013
-0.5mm/ +0.177mm	1.88	1.600	0.109	0.18	0.005	9.66	4.56	21.16	0.390	12.37	0.19	1.411	34.1	0.682	0.013	0.160	0.014
-0.177mm/ +0.149 mm	0.69	1.260	0.104	0.19	0.005	9.33	4.42	21.73	0.343	13.23	0.20	2.469	33.2	0.661	0.013	0.160	0.015
-0.149mm/ +0.105mm	0.68	1.130	0.209	0.30	0.210	9.35	4.44	22.02	0.385	13.46	0.22	1.771	33.2	0.703	0.020	0.180	0.015
-0.105mm/ +0.075mm	0.83	0.015	0.347	0.31	0.005	8.99	4.30	20.73	0.004	12.32	0.20	0.019	33.3	0.637	0.001	1.209	0.016
-0.075mm / +0.053mm	0.63	0.014	0.300	0.04	0.005	8.96	4.25	20.88	0.004	11.33	0.20	0.019	33.2	0.673	0.001	0.895	0.015
-0.053mm	0.35	0.014	0.297	0.18	0.005	8.84	4.20	21.87	0.004	11.04	0.18	0.020	32.7	0.665	0.001	0.872	0.015
mass weighted average		1.533	0.120	0.133	0.017	10.25	4.84	20.07	0.317	12.08	0.20	1.495	34.70	0.739	0.013	0.299	0.014
total	11.89																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.3. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-01-05 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	10.13	0.90	.044	0.16	0.050	13.64	6.42	17.44	0.982	5.57	0.185	1.631	38.8	0.463	0.01	0.051	0.019
-2.0mm/ +0.5mm	5.39	1.17	.089	0.25	0.005	13.13	6.41	18.16	0.988	5.59	0.174	1.658	37.9	0.635	0.01	0.057	0.019
-0.5mm/ +0.177mm	3.21	0.73	.107	0.28	0.005	13.23	6.21	18.01	1.017	5.75	0.158	1.671	38.0	0.477	0.00	0.035	0.018
-0.177mm/ +0.149 mm	1.04	0.38	.108	0.30	0.005	13.47	6.18	18.30	1.043	5.95	0.139	1.725	37.8	0.301	0.00	0.026	0.018
-0.149mm/ +0.105mm	1.34	0.27	.109	0.35	0.005	12.94	5.51	18.44	0.975	5.87	0.163	1.631	38.2	0.250	0.00	0.023	0.017
-0.105mm/ +0.075mm	1.40	0.23	.122	0.34	0.005	12.32	5.18	19.73	0.905	6.05	0.185	1.487	35.5	0.189	0.00	0.023	0.020
-0.075mm / +0.053mm	1.36	0.23	.175	0.45	0.005	12.00	5.15	20.16	0.854	5.97	0.168	1.384	36.3	0.677	0.00	0.027	0.018
-0.053mm	0.65	0.26	.223	0.50	0.050	11.75	5.22	21.02	0.770	5.75	0.178	2.069	35.6	0.242	0.00	0.029	0.017
mass weighted average		0.789	0.084	0.247	0.025	13.22	6.16	18.15	0.974	5.69	0.175	1.637	38.03	0.475	0.006	0.044	0.019
total	24.52																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.4. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-01-07 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	14.61	2.04	0.152	0.07	0.005	15.48	6.14	15.44	1.438	6.37	0.133	1.403	36.9	0.74	0.01	0.061	0.056
-2.0mm/ +0.5mm	7.56	1.98	0.152	0.21	0.005	13.45	5.25	16.16	1.511	5.52	0.115	1.313	32.9	0.83	0.00	0.045	0.018
-0.5mm/ +0.177mm	3.32	1.95	0.377	0.25	0.005	13.89	5.32	19.02	1.393	5.69	0.130	0.806	32.5	0.90	0.00	0.043	0.023
-0.177mm/ +0.149 mm	0.91	1.61	0.400	0.30	0.005	13.89	5.44	18.73	1.295	5.64	0.134	0.496	32.6	0.78	0.01	0.045	0.024
-0.149mm/ +0.105mm	1.29	1.25	0.397	0.30	0.005	13.74	5.51	18.73	1.349	6.02	0.129	0.806	25.7	0.59	0.00	0.038	0.021
-0.105mm/ +0.075mm	1.03	1.03	0.430	0.35	0.005	11.13	3.79	22.73	1.325	5.16	0.081	2.518	60.6	0.60	0.00	0.032	0.030
-0.075mm / +0.053mm	0.75	0.90	0.563	0.42	0.005	12.22	4.37	21.16	1.291	6.10	0.094	3.549	37.9	0.49	0.00	0.032	0.026
-0.053mm	1.11	0.98	0.743	0.53	0.005	12.43	4.53	19.16	1.237	6.12	0.132	7.069	35.4	0.43	0.01	0.031	0.025
mass weighted average		1.87	0.232	0.175	0.005	14.34	5.60	16.76	1.428	5.99	0.125	1.558	35.58	0.752	0.005	0.051	0.038
total	30.58																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.5. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-02-02 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	8.64	0.90	0.045	0.07	0.005	12.17	6.06	15.87	0.625	9.98	0.168	1.933	35.9	0.475	0.010	0.093	0.012
-2.0mm/ +0.5mm	6.44	1.27	0.079	0.09	0.005	11.98	5.54	15.73	0.761	9.07	0.164	2.163	36.2	0.690	0.009	0.120	0.011
-0.5mm/ +0.177mm	4.18	1.26	0.123	0.15	0.005	10.96	5.21	18.01	0.583	10.02	0.169	2.201	34.5	0.721	0.012	0.140	0.014
-0.177mm/ +0.149 mm	0.60	1.20	0.126	0.16	0.005	10.37	5.08	19.30	0.513	10.93	0.200	2.449	33.9	0.751	0.013	0.150	0.017
-0.149mm/ +0.105mm	1.32	1.21	0.133	0.22	0.005	10.35	5.01	19.16	0.475	10.66	0.176	1.386	33.5	0.765	0.013	0.170	0.014
-0.105mm/ +0.075mm	1.21	1.16	0.202	0.32	0.005	9.98	4.50	18.87	0.610	9.85	0.187	3.087	37.7	0.810	0.015	0.220	0.017
-0.075mm / +0.053mm	0.68	1.10	0.257	0.40	0.005	9.92	4.41	19.16	0.581	8.28	0.186	1.775	36.0	0.830	0.015	0.250	0.017
-0.053mm	0.17	1.03	0.283	0.47	0.390	10.03	4.46	19.87	0.558	9.22	0.174	4.423	35.8	0.930	0.016	0.240	0.015
mass weighted average		1.115	0.091	0.127	0.008	11.57	5.55	16.80	0.642	9.78	0.170	2.104	35.70	0.635	0.011	0.127	0.013
total	23.24																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.6. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-02-04 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	12.38	0.62	0.008	0.06	0.005	14.13	8.40	15.30	1.184	9.85	0.201	1.790	36.2	0.354	0.008	0.081	0.011
-2.0mm/ +0.5mm	8.63	0.84	0.008	0.04	0.005	19.95	8.35	15.58	1.447	9.88	0.194	2.046	34.5	0.542	0.009	0.110	0.011
-0.5mm/ +0.177mm	7.49	1.08	0.026	0.07	0.005	12.90	7.70	17.30	1.299	10.76	0.209	1.946	34.7	0.746	0.011	0.140	0.012
-0.177mm/ +0.149 mm	0.98	1.29	0.020	0.10	0.005	10.71	5.88	16.73	1.293	10.21	0.167	1.824	34.6	0.905	0.012	0.180	0.013
-0.149mm/ +0.105mm	2.17	1.55	0.023	0.10	0.005	10.68	5.89	17.16	1.265	10.25	0.173	1.787	33.6	1.015	0.015	0.220	0.012
-0.105mm/ +0.075mm	2.35	2.06	0.044	0.13	0.005	11.22	6.49	18.30	1.465	10.16	0.203	1.738	32.5	1.190	0.020	0.350	0.013
-0.075mm / +0.053mm	1.16	2.36	0.061	0.17	0.005	11.30	6.45	18.30	1.378	9.63	0.169	2.913	32.9	1.322	0.023	0.420	0.016
-0.053mm	0.31	2.27	0.078	0.22	0.005	11.28	6.30	18.59	1.404	9.52	0.174	2.312	33.6	1.364	0.023	0.420	0.014
mass weighted average		1.012	0.018	0.07	0.005	13.20	8.04	16.25	1.198	10.09	0.197	1.922	34.86	0.633	0.011	0.143	0.012
total	35.47																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.7. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-02-07 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	11.37	1.00	0.044	0.07	0.005	12.60	6.67	16.16	0.670	9.65	0.181	1.657	36.2	0.45	0.010	0.097	0.016
-2.0mm/ +0.5mm	8.32	1.13	0.076	0.09	0.005	12.07	6.16	15.58	0.735	8.66	0.173	1.308	39.6	0.61	0.010	0.113	0.012
-0.5mm/ +0.177mm	5.78	1.13	0.113	0.13	0.005	10.96	5.70	17.59	0.684	9.70	0.165	1.050	32.8	0.72	0.010	0.118	0.014
-0.177mm/ +0.149 mm	2.02	1.03	0.126	0.20	0.005	10.68	5.53	18.44	0.660	10.28	0.195	1.126	33.5	0.81	0.011	0.130	0.016
-0.149mm/ +0.105mm	1.60	1.08	0.129	0.21	0.005	10.62	5.47	19.16	0.634	10.51	0.187	0.921	33.9	0.74	0.012	0.156	0.017
-0.105mm/ +0.075mm	2.06	1.14	0.177	0.30	0.560	10.64	5.06	19.30	0.588	9.82	0.190	1.518	32.5	0.95	0.013	0.200	0.014
-0.075mm / +0.053mm	1.05	1.12	0.224	0.37	0.500	10.35	4.98	19.44	0.524	8.34	0.167	1.538	30.8	0.84	0.013	0.220	0.015
-0.053mm	0.21	1.11	0.245	0.42	0.320	10.47	4.97	20.16	0.516	9.30	0.161	1.376	33.0	0.86	0.014	0.230	0.013
mass weighted average		1.076	0.089	0.128	0.058	11.75	6.07	16.89	0.676	9.49	0.177	1.376	37.25	0.623	0.010	0.121	0.015
total	32.41																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.8. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-02-10 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	13.19	1.13	0.103	0.19	0.005	13.72	6.31	16.16	0.909	8.12	0.165	0.845	36.7	0.51	0.01	0.089	0.018
-2.0mm/ +0.5mm	8.28	1.51	0.162	0.14	0.005	12.87	5.79	16.59	0.861	7.74	0.146	0.985	36.3	0.69	0.01	0.107	0.015
-0.5mm/ +0.177mm	5.59	1.41	0.295	0.19	0.005	11.98	5.63	17.73	0.817	8.16	0.154	1.161	25.6	0.67	0.01	0.101	0.016
-0.177mm/ +0.149 mm	1.46	1.20	0.328	0.25	0.005	12.24	5.75	18.44	0.822	8.49	0.161	0.813	34.6	0.65	0.01	0.105	0.018
-0.149mm/ +0.105mm	1.61	1.15	0.347	0.29	0.005	11.75	5.56	18.73	0.800	8.34	0.155	0.748	34.4	0.60	0.01	0.113	0.019
-0.105mm/ +0.075mm	1.46	1.04	0.380	0.37	0.005	11.00	4.91	18.73	0.844	7.79	0.163	4.140	37.0	0.72	0.01	0.120	0.018
-0.075mm / +0.053mm	0.50	1.01	0.427	0.41	0.005	10.87	4.95	18.87	0.797	7.58	0.170	1.313	36.8	0.83	0.01	0.130	0.019
-0.053mm	0.26	0.94	0.480	0.47	0.005	10.64	4.78	19.16	0.752	7.30	0.148	4.781	36.3	0.81	0.01	0.140	0.015
mass weighted average		1.274	0.193	0.199	0.005	12.86	5.92	16.97	0.867	8.04	0.158	1.118	34.53	0.612	0.01	0.100	0.017
total	32.35																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.9. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-03-01 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	12.79	1.32	0.051	0.14	0.005	10.86	5.04	16.16	0.679	10.08	0.189	1.833	36.2	0.585	0.001	0.150	0.014
-2.0mm/ +0.5mm	8.03	1.38	0.068	0.15	0.290	11.28	4.94	15.58	0.747	9.20	0.179	2.130	36.8	0.696	0.013	0.160	0.013
-0.5mm/ +0.177mm	6.14	1.56	0.173	0.18	0.360	9.75	4.49	17.16	0.605	9.82	0.185	1.766	32.8	0.792	0.014	0.180	0.015
-0.177mm/ +0.149 mm	1.23	1.91	0.203	0.24	0.350	9.11	4.44	19.16	0.548	11.09	0.191	1.442	33.5	0.882	0.017	0.230	0.016
-0.149mm/ +0.105mm	1.61	1.93	0.278	0.25	0.080	9.14	4.48	19.02	0.570	10.71	0.195	1.456	33.9	0.889	0.018	0.260	0.016
-0.105mm/ +0.075mm	1.46	2.14	0.333	0.26	0.005	9.30	4.73	19.16	1.241	9.93	0.173	1.357	32.5	0.992	0.019	0.180	0.017
-0.075mm / +0.053mm	0.79	1.92	0.367	0.34	0.005	10.15	5.06	20.87	1.313	10.28	0.187	1.601	30.8	0.585	0.022	0.380	0.016
-0.053mm	0.19	1.87	0.350	0.40	0.005	10.17	4.87	21.16	1.260	10.11	0.170	1.704	33.0	1.114	0.023	0.390	0.016
mass weighted average		1.49	0.116	0.171	0.161	10.52	4.85	16.75	0.715	9.89	0.185	1.835	35.22	0.701	0.009	0.175	0.014
total	32.24																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.10. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-03-06 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	9.08	0.82	0.050	0.11	0.005	11.79	5.65	14.73	1.835	9.52	0.146	1.960	36.0	0.550	0.009	0.091	0.011
-2.0mm/ +0.5mm	6.42	0.80	0.069	0.10	0.005	12.00	5.01	12.70	2.585	7.53	0.130	2.521	40.5	0.510	0.008	0.095	0.010
-0.5mm/ +0.177mm	3.86	0.16	0.204	1.06	0.005	10.75	5.28	15.01	1.758	8.67	0.146	2.000	36.2	0.612	0.009	0.120	0.012
-0.177mm/ +0.149 mm	0.88	0.98	0.188	0.21	0.100	10.47	5.20	17.30	1.427	10.28	0.165	1.786	35.9	0.277	0.012	0.093	0.013
-0.149mm/ +0.105mm	1.23	0.21	0.161	0.98	0.005	10.37	5.06	17.73	1.185	10.60	0.168	1.217	35.1	0.648	0.012	0.200	0.011
-0.105mm/ +0.075mm	1.29	0.95	0.215	0.29	0.005	10.75	5.00	17.73	0.460	9.78	0.151	2.091	34.6	0.772	0.013	0.190	0.015
-0.075mm / +0.053mm	0.61	0.89	0.280	0.35	0.005	10.56	4.94	17.87	0.484	9.28	0.151	1.496	34.8	0.743	0.013	0.200	0.014
-0.053mm	0.18	0.84	0.122	0.42	0.020	10.70	4.91	18.44	0.529	9.22	0.145	2.092	32.3	1.103	0.012	0.210	0.014
mass weighted average		0.688	0.106	0.330	0.009	11.44	5.30	14.72	1.854	8.91	0.144	2.067	37.01	0.565	0.009	0.112	0.011
total	23.55																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.11. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-03-08 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	2.03	0.59	0.021	0.08	0.005	12.38	8.61	13.38	0.219	9.48	0.172	0.706	32.1	0.674	0.010	0.059	0.010
-2.0mm/ +0.5mm	1.49	0.94	0.085	0.11	0.080	11.92	7.96	13.48	0.407	8.76	0.163	0.887	35.2	0.726	0.010	0.120	0.012
-0.5mm/ +0.177mm	1.04	1.12	0.117	0.14	0.260	11.70	8.52	13.80	0.388	9.09	0.155	1.324	31.3	0.792	0.010	0.120	0.011
-0.177mm/ +0.149 mm	0.36	1.09	0.108	0.17	0.340	12.09	7.84	15.01	0.379	9.72	0.204	1.105	30.8	0.740	0.010	0.120	0.012
-0.149mm/ +0.105mm	0.40	1.14	0.100	0.18	0.210	11.72	7.25	15.16	0.387	9.75	0.182	1.019	31.3	0.822	0.011	0.130	0.012
-0.105mm/ +0.075mm	0.61	1.20	0.120	0.21	0.005	12.17	7.26	16.16	0.248	9.50	0.176	2.076	34.1	0.887	0.012	0.170	0.012
-0.075mm / +0.053mm	0.34	1.16	0.184	0.36	0.005	12.00	7.08	15.87	0.296	8.82	0.163	1.878	33.3	0.952	0.012	0.220	0.013
-0.053mm	0.14	1.11	0.255	0.43	0.005	11.87	6.90	15.87	0.285	8.61	0.194	1.473	32.8	0.985	0.013	0.240	0.014
mass weighted average		0.921	0.084	0.143	0.095	12.08	8.08	14.15	0.318	9.24	0.170	1.102	32.91	0.762	0.010	0.114	0.011
total	6.41																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.12. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-03-16 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	9.84	1.16	0.034	0.15	0.005	12.43	7.61	14.58	0.614	8.97	0.148	1.901	34.5	0.743	0.009	0.089	0.013
-2.0mm/ +0.5mm	6.49	1.09	0.065	0.11	0.005	12.24	7.70	14.44	0.479	8.85	0.156	1.564	32.7	0.613	0.008	0.110	0.012
-0.5mm/ +0.177mm	5.08	1.45	0.357	0.13	0.005	11.81	8.20	14.58	0.436	8.92	0.164	1.874	33.3	0.715	0.009	0.110	0.011
-0.177mm/ +0.149 mm	1.28	1.42	0.304	0.16	0.005	11.70	7.35	15.87	0.451	9.48	0.150	1.618	34.0	0.716	0.010	0.130	0.012
-0.149mm/ +0.105mm	1.25	0.86	0.276	0.11	0.230	11.66	7.58	17.16	0.435	9.90	0.168	1.415	33.1	0.778	0.011	0.150	0.012
-0.105mm/ +0.075mm	2.11	1.48	0.034	0.23	0.005	11.66	7.05	16.01	0.400	8.89	0.181	1.622	34.2	0.818	0.011	0.170	0.011
-0.075mm / +0.053mm	0.70	1.42	0.350	0.27	0.005	11.34	6.74	15.58	0.369	8.47	0.156	1.526	33.0	0.894	0.011	0.180	0.011
-0.053mm	0.22	1.30	0.322	0.35	0.005	11.30	6.59	16.01	0.364	8.46	0.158	1.504	33.2	0.872	0.012	0.180	0.011
mass weighted average		1.229	0.159	0.146	0.015	12.08	7.64	14.86	0.506	8.96	0.157	1.742	33.63	0.717	0.009	0.112	0.012
total	26.97																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.13. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-04-02 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	13.36	0.72	0.110	0.38	0.005	15.46	6.21	15.58	0.994	6.33	0.164	1.593	35.3	0.34	0.005	0.031	0.039
-2.0mm/ +0.5mm	7.74	1.13	0.221	0.37	0.005	14.97	6.21	15.44	0.971	5.87	0.138	2.008	34.6	0.50	0.006	0.053	0.023
-0.5mm/ +0.177mm	5.86	1.75	0.793	0.36	0.005	13.96	6.55	14.87	0.962	5.51	0.133	1.941	36.0	0.90	0.005	0.055	0.030
-0.177mm/ +0.149 mm	2.17	1.19	0.520	0.45	0.060	13.87	5.89	16.30	0.900	6.04	0.134	1.491	35.0	0.37	0.006	0.055	0.023
-0.149mm/ +0.105mm	2.52	1.01	0.430	0.60	0.210	13.53	5.67	17.44	0.914	5.23	0.172	1.314	33.9	0.38	0.008	0.069	0.033
-0.105mm/ +0.075mm	1.75	0.95	0.493	0.62	0.005	13.28	4.98	17.30	0.934	5.80	0.161	5.508	35.9	0.41	0.008	0.078	0.026
-0.075mm / +0.053mm	0.58	1.05	0.557	0.62	1.820	13.43	5.11	17.73	0.841	5.65	0.145	1.612	40.4	0.46	0.009	0.086	0.024
-0.053mm	0.34	1.09	0.650	0.59	0.710	13.19	5.04	17.44	0.881	5.27	0.147	1.724	38.7	0.50	0.008	0.091	0.020
mass weighted average		1.061	0.331	0.414	0.061	14.70	6.12	15.77	0.966	6.02	0.151	1.923	35.33	0.484	0.006	0.048	0.031
total	34.32																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.14. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-04-04 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	10.20	0.68	0.074	0.20	0.10	15.46	6.23	12.84	0.955	6.48	0.120	2.013	40.1	0.35	0.006	0.096	0.014
-2.0mm/ +0.5mm	7.18	0.81	0.099	0.19	0.005	14.70	5.32	11.18	1.360	5.12	0.102	2.486	39.1	0.48	0.004	0.051	0.016
-0.5mm/ +0.177mm	3.86	1.14	0.400	0.30	0.08	14.30	5.89	13.31	1.065	5.74	0.118	2.257	41.1	0.48	0.005	0.055	0.014
-0.177mm/ +0.149 mm	1.53	1.07	0.373	0.38	0.04	14.49	5.95	15.16	0.935	6.55	0.130	1.770	39.3	0.45	0.007	0.063	0.016
-0.149mm/ +0.105mm	1.21	0.98	0.307	0.43	0.07	14.00	5.65	15.44	0.876	6.60	0.130	1.219	37.0	0.49	0.007	0.070	0.016
-0.105mm/ +0.075mm	1.53	0.93	0.363	0.62	1.59	13.83	4.88	16.30	0.899	6.42	0.116	1.825	39.4	0.53	0.007	0.079	0.018
-0.075mm / +0.053mm	0.62	0.87	0.460	0.67	1.11	13.87	4.91	16.87	0.897	6.27	0.134	1.574	40.1	0.46	0.008	0.078	0.016
-0.053mm	0.30	0.92	0.457	0.71	1.28	13.70	4.74	16.87	0.914	6.02	0.114	1.521	34.3	0.50	0.007	0.081	0.015
mass weighted average		0.840	0.185	0.274	0.189	14.81	5.76	13.05	1.072	6.00	0.116	2.101	39.69	0.431	0.006	0.089	0.015
total	26.43																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.15. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-04-05 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	11.93	0.94	0.080	0.13	0.03	14.15	6.23	15.87	0.577	8.64	0.155	1.936	29.9	0.58	0.010	0.11	0.013
-2.0mm/ +0.5mm	8.07	1.43	0.173	0.15	0.05	13.68	6.07	15.87	0.606	8.22	0.143	1.909	28.9	0.85	0.010	0.13	0.014
-0.5mm/ +0.177mm	4.40	1.95	0.597	0.20	0.39	12.21	6.12	16.16	0.553	7.99	0.143	1.840	32.6	0.92	0.012	0.16	0.016
-0.177mm/ +0.149 mm	1.10	2.00	0.730	0.24	0.16	11.83	6.09	17.16	0.530	8.46	0.143	1.756	33.7	0.90	0.012	0.16	0.017
-0.149mm/ +0.105mm	1.15	1.79	0.603	0.28	0.05	11.62	5.74	17.30	0.522	8.51	0.163	1.634	34.3	0.88	0.013	0.17	0.018
-0.105mm/ +0.075mm	1.35	1.59	0.443	0.32	0.005	11.87	5.34	18.44	0.573	8.66	0.172	1.580	39.2	0.87	0.013	0.20	0.014
-0.075mm / +0.053mm	0.77	1.48	0.443	0.34	0.52	11.85	5.36	19.16	0.522	8.49	0.136	1.500	38.0	0.97	0.015	0.22	0.021
-0.053mm	0.31	1.57	0.443	0.38	0.05	11.49	5.29	19.59	0.537	8.22	0.143	1.111	37.1	1.19	0.015	0.24	0.018
mass weighted average		1.353	0.257	0.173	0.108	13.34	6.07	16.26	0.575	8.40	0.150	1.858	31.06	0.760	0.011	0.136	0.014
total	29.1																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.16. Chemistry and particle size distribution for the -6.4 mm (0.25 in.) fraction of spatially distributed sample 06-04-07 (in percent). Mass weighted average of -0.5 in. size particles from the 0.8% S bulk sample is presented for comparison.

Size Fraction	mass %	S	SO_4-S	C	CO_3	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn
–6.4 mm / +2.0mm	13.83	0.87	0.082	0.30	0.005	16.00	5.15	14.02	1.094	6.38	0.141	0.930	38.7	0.43	0.010	0.080	0.021
-2.0mm/ +0.5mm	8.81	1.54	0.166	0.32	0.005	15.21	5.11	13.84	1.035	5.85	0.132	1.140	37.7	0.77	0.010	0.076	0.019
-0.5mm/ +0.177mm	6.91	1.91	0.623	0.37	0.005	14.47	5.54	14.44	0.914	5.84	0.147	1.138	36.8	0.85	0.010	0.095	0.021
-0.177mm/ +0.149 mm	1.97	1.57	0.467	0.48	0.005	14.13	5.15	15.30	0.764	6.20	0.133	1.091	36.9	0.72	0.010	0.099	0.021
-0.149mm/ +0.105mm	2.37	1.40	0.377	0.60	0.005	13.77	4.94	15.44	0.742	6.12	0.139	1.034	33.8	0.69	0.010	0.095	0.017
-0.105mm/ +0.075mm	1.63	1.37	0.390	0.71	0.880	14.13	4.58	16.30	0.817	6.20	0.145	1.519	34.4	0.95	0.010	0.093	0.016
-0.075mm / +0.053mm	0.38	1.44	0.403	0.70	0.280	14.32	4.56	16.59	0.790	6.14	0.115	1.126	35.6	0.89	0.010	0.087	0.020
-0.053mm	0.26	1.57	0.450	0.38	1.270	14.32	4.63	17.16	0.779	6.00	0.112	1.527	34.4	0.98	0.010	0.099	0.020
mass weighted average		1.337	0.263	0.370	0.056	15.14	5.16	14.29	0.985	6.10	0.139	1.068	37.37	0.730	0.010	0.054	0.020
total	36.16																
-1/2 in 0.8% bulk total		1.189	0.103			14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.008	0.109	0.008

Table A5.9.17. Chemistry and particle size distribution for the 0.8% S bulk sample (in percent).

Size Fraction	mass %	S	SO_4-S	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Co	Ni	Zn	LOI
-12 in / +6in	5.9															
-6in / +1.5in	40.8	0.60	0.04	14.58	8.86	15.38	0.52	9.74	0.16	2.00	45.05	0.337	0.006	0.057	0.007	0.26
-1.5in / +0.5in	39.5	0.74	0.09	14.53	8.56	15.53	0.46	10.24	0.17	1.93	44.59	0.386	0.007	0.070	0.008	0.77
–0.5 in / +2.0mm	8.3	1.13	0.06	14.63	8.41	15.82	0.47	10.13	0.17	1.92	44.23	0.506	0.008	0.108	0.008	1.19
-2.0mm/ +0.5mm	2.2	1.22	0.08	14.30	7.68	15.64	0.59	9.54	0.16	2.05	45.16	0.632	0.008	0.101	0.010	2.07
-0.5mm/ +0.177mm	1.4	1.34	0.28	13.73	7.61	16.31	0.53	9.69	0.16	1.92	44.07	0.662	0.007	0.090	0.008	2.45
-0.177mm/ +0.149 mm	0.26	1.20	0.18	13.74	7.31	17.32	0.52	10.31	0.17	1.89	42.77	0.654	0.008	0.100	0.008	2.02
-0.149mm/ +0.105mm	0.24	0.99	0.15	14.24	7.30	16.61	0.54	10.12	0.16	1.99	43.37	0.615	0.008	0.100	0.008	1.80
-0.105mm/ +0.075mm	0.35	1.11	0.14	14.31	7.21	17.02	0.57	10.04	0.16	1.95	43.30	0.674	0.009	0.110	0.009	2.20
-0.075mm / +0.053mm	0.28	1.26	0.12	13.44	6.89	18.62	0.53	10.37	0.17	1.82	41.96	0.810	0.010	0.140	0.009	2.38
-0.053mm	0.7	1.51	0.25	13.26	6.73	18.80	0.50	9.73	0.15	1.66	40.55	0.926	0.010	0.160	0.013	4.79
total	99.9															
analyzed mass weighted averages	94.0	0.74	0.069	14.6	8.62	15.6	0.49	9.99	0.164	1.96	44.7	0.39	0.006	0.07	0.007	0.688
mass weighted average of –0.5 in fraction	13.7	1.189	0.103	14.39	8.04	16.16	0.504	9.998	0.166	1.93	44.16	0.58	0.008	0.109	0.008	0.646

Table A5.9.18. Summary of the mass weighted sulfur, sulfate, whole rock, and trace metal chemistry for sixteen spatially distributed samples and the bulk sample from the 0.8% test pile. Calculations are based on analyses of the -6.4 mm particle sizes of the spatially distributed samples and the −0.5 in. particle sizes for the bulk sample.

Sample	S	SO_4	C	CO_2	Al_2O_3	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	SiO_2	Cu	Ni	Co	Zn
06-01-02	1.149	0.093	0.391	0.799	12.68	5.84	18.11	1.005	6.025	0.159	1.581	35.79	0.674	0.090	0.009	0.012
06-01-04	1.533	0.120	0.133	0.017	10.25	4.84	20.07	0.317	12.08	0.201	1.50	34.70	0.739	0.299	0.013	0.014
06-01-05	0.789	0.084	0.247	0.025	13.22	6.16	18.15	0.974	5.689	0.175	1.637	38.03	0.475	0.044	0.006	0.019
06-01-07	1.868	0.232	0.175	0.005	14.34	5.60	16.76	1.428	5.99	0.125	1.56	35.58	0.752	0.051	0.005	0.038
06-02-02	1.115	0.091	0.127	0.008	11.57	5.55	16.80	0.642	9.78	0.170	2.104	35.70	0.635	0.127	0.011	0.013
06-02-04	1.012	0.018	0.070	0.005	13.20	8.04	16.25	1.198	10.09	0.197	1.922	34.86	0.633	0.143	0.011	0.012
06-02-07	1.076	0.089	0.128	0.058	11.75	6.07	16.89	0.676	9.49	0.177	1.376	37.25	0.623	0.121	0.010	0.015
06-02-10	1.274	0.193	0.199	0.005	12.86	5.92	16.97	0.867	8.04	0.158	1.118	34.53	0.612	0.100	0.010	0.015
06-03-01	1.489	0.116	0.171	0.161	10.52	4.85	16.75	0.715	9.89	0.185	1.835	35.22	0.701	0.175	0.009	0.014
06-03-06	0.688	0.106	0.330	0.009	11.44	5.30	14.72	1.854	8.91	0.144	2.067	37.01	0.565	0.112	0.009	0.011
06-03-08	0.921	0.084	0.143	0.095	12.08	8.08	14.15	0.318	9.24	0.170	1.102	32.91	0.762	0.114	0.010	0.011
06-03-16	1.229	0.159	0.146	0.015	12.08	7.64	14.86	0.506	8.96	0.157	1.742	33.63	0.717	0.112	0.009	0.012
06-04-02	1.061	0.331	0.414	0.061	14.70	6.12	15.77	0.966	6.02	0.151	1.923	35.33	0.484	0.048	0.006	0.031
06-04-04	0.840	0.185	0.274	0.189	14.81	5.76	13.05	1.072	6.00	0.116	2.101	39.69	0.431	0.089	0.006	0.015
06-04-05	1.353	0.257	0.173	0.108	13.34	6.07	16.26	0.575	8.40	0.150	1.858	31.06	0.760	0.136	0.011	0.014
06-04-07	1.337	0.263	0.370	0.056	15.14	5.16	14.29	0.985	6.10	0.139	1.068	37.37	0.730	0.054	0.010	0.020
Average of spatially distributed samples	1.171	0.151	0.218	0.101	12.75	6.06	16.24	0.881	8.17	0.161	1.66	35.54	0.643	0.115	0.009	0.017
Bulk 0.8%S	1.189	0.103	nd[1]	nd	14.39	8.04	16.16	0.504	9.998	0.166	1.931	44.16	0.580	0.107	0.008	0.008
Difference between spatial & bulk	-0.02	0.05			-1.64	-1.98	0.08	0.38	-1.83	-0.01	-0.27	-8.62	0.06	0.01	0.00	0.01

[1] nd = not determined

Table A5.9.19. Comparison of BLM (1996) and ACME (2003) laboratories of whole rock analyses of randomly selected spatially distributed samples. Values are in percent, particle size in millimeters unless otherwise indicated.

ID	SiO_2 ACME	SiO_2 BLM	Al_2O_3 ACME	Al_2O_3 BLM	Fe_2O_3 ACME	Fe_2O_3 BLM	MgO ACME	MgO BLM	CaO ACME	CaO BLM	Na_2O ACME	Na_2O BLM	K_2O ACME	K_2O BLM	MnO ACME	MnO BLM
6-1-7 -1/4"/+2.0	45.6	36.9	17.0	15.5	16.4	15.4	6.69	6.4	5.56	6.14	1.49	1.40	1.34	1.44	0.11	0.13
6-1-7 -0.149/+0.105	43.0	25.7	15.1	13.7	19.7	18.7	6.19	6.02	4.88	5.51	1.49	0.81	1.27	1.35	0.10	0.13
6-2-2 -2.0/+0.5	44.9	36.2	13.4	12.0	17.1	15.7	9.79	9.07	5.26	5.54	2.18	2.16	0.79	0.76	0.15	0.16
6-2-2 -0.177/+0.149	41.4	33.9	11.5	10.4	20.7	19.3	11.8	10.9	4.74	5.08	1.58	2.45	0.59	0.51	0.18	0.20
6-4-4 -1/4"/+2.0	47.9	40.1	17.3	15.5	13.9	12.8	6.98	6.48	5.83	6.22	2.33	2.01	0.95	0.96	0.13	0.12
6-4-4 -0.177/+0.149	46.1	39.3	15.6	14.5	15.8	15.2	6.68	6.55	5.33	5.95	2.20	1.77	0.88	0.94	0.14	0.13
6-4-5 -2.0/+0.5	43.3	28.9	15.1	13.7	16.9	15.9	8.67	8.22	6.20	6.07	1.97	1.91	0.64	0.61	0.14	0.14
6-4-5 -0.5/+0.177	41.4	32.6	14.2	12.2	17.7	16.2	8.66	7.99	6.41	6.12	1.81	1.84	0.59	0.55	0.14	0.14
6-4-5 -0.5/+0.177 duplicate	40.9		14.0		17.4		8.49		6.34		1.82		0.57		0.14	
avg Δ[1]	10.0		1.46		1.13		0.48		-0.30		0.09		-0.01		-0.01	

[1] Average of the differences between ACME and BLM analyses.

Amax Leach Test - Analytical Results

McSwiggen & Associates
November 4, 2003

Introduction:

The ultimate objective of this work was to determine a way to quantify the amount leaching that took place in the various parts of one of the Amax leach test piles. To do this, a scale had to be established that could be reasonably applied to samples to rate how far the sulfide minerals had reacted. In order to establish this leaching scale, we first had to determine which size fraction would be the best indicator. All of the size fractions were looked at from three samples: 6-2-4, 6-3-1, and 6-4-2. A rating scale was then determined, and then all of the other samples were investigated using the one optimum size fraction (30-70).

Procedures:

For each size fraction between 1.5 inches and <270, polished grain mounts were made. The loose material was poured into a plastic cup and the impregnated with epoxy. The top surface of each mount was then polished and carbon coated. These polished mounts were used for all the analyses and imaging found in the accompanying documents.

Each mount was evaluated using a JEOL 8600 Electron Microprobe. A backscattered electron image was collected at low magnification to give an overview of the sample's general appearance. Later backscattered electron images of representative sulfide minerals were collected to document the extent of alteration. These images were later compiled into files found in the directory *Images_Fldr*.

A polished thin section was also made from each sample. The selected rock came from the −6 to +3 inch size fraction. The fraction was used in an attempt to get a fresh, unweathered sample. Each section was evaluated optically to determine the minerals present and the abundance of each mineral. Electron microprobe analyses were then made on representative examples of each mineral to get their elemental composition. Summaries of these data can be found in *SummaryQuant&Mode.xls.*

Determining the optimum size fraction:

Polished grain mounts were made of each of the following size fraction from three samples: 6-2-4, 6-3-1, and 6-4-2.

< 1.5 to >0.75
< 0.75 to >0.5
< 0.5 to >0.25
< 0.25 to >10
< 10 to >30
< 30 to >70
< 70 to >100
< 100 to >150
< 150 to >200
< 200 to >270
< 270

The size fractions <1.5 to >0.75, <0.75 to >0.5, <0.5 to >0.25, <0.25 to >10 were determined to be too large to be of much use. In those samples, the sulfide minerals were poorly exposed and therefore would not have experienced much in the way of leaching.

It was determined that the size fraction 30-70 was best for determining the degree of leaching for the following reasons:

1) Most of the sulfide mineral grains were liberated or exposed in this size fraction. Larger fractions were significantly included in the silicate minerals and therefore would not be exposed to leaching process.
2) Using smaller size fractions runs the risk that the pyrrhotite would be completely leached away. This would make it difficult to determine the extent of leaching. It would be difficult to know whether there had been pyrrhotite that had leached away, or whether there never was any there in the first place.

Leaching scale:

For each sulfide mineral a leach scale was established. It is based on the amount the sulfide grains appear to have been reacted. A 0-5 scale was established as follows:

0 - No alteration of any kind.
1 - Evidence of minor dissolution around the parameter of the grain.
2 - The sulfide grains have developed a thin (< 10 microns) iron oxyhydroxide coating or has experienced significant dissolution.
3 - The sulfide grains have developed a thick (> 10 microns) iron oxyhydroxide coating.
4 - The sulfide grain has nearly reacted away.
5 - The sulfide grain has completely reacted away.

Once the individual sulfide minerals have been evaluated, a second scheme was developed to provide a single leaching scale for the sample as a whole.

Sample Leach Scale	**Pyrrhotite Scale**	**Cubanite/Chalcopyrite Scale**	**Pentlandite Scale**
1	1	1	1
2	2	1-2	2
3	3	1-2	2-3
4	4	1-2	3-4
5	N	3	4-N
6	N	4	4-N
7	N	N	N

(where N denote "not present")

For each sample an average value was recorded for each sulfide mineral present. This data is tabulated in the file: *AmaxMineralReactionRating.xls*, and was then used to determine a leach factor for the sample as a whole using the table above. The results are listed in *SummaryReactionRating.xls.* Pyrite occurred in very low abundance and therefore was not included in the determination of the sample leaching factor.

Application of the leaching scale to the Amax samples:

The results indicate that there is a wide range in the degree of leaching in the samples from this particular leach pile. There is a range from 1 to 5 in the leaching factor. In general, the higher tiers (6-1 and 6-2) have experienced more leaching than samples from the lower tiers (6-3, and 6-4), and the higher tiers seem to be more uniform in results.

Another intriguing result of this study is that it appears that the degree of leaching is less in the small size fractions than in the large fraction (this applies only to the size fractions between >10 and <270 mesh). A good example is sample 6-3-1. In the size fractions between 10 and 100 mesh, the leaching factor is around 3, whereas in the 200 to >270 mesh range, the fraction is 1-2.

It should be noted that even though much of this work focused on the >10 mesh size fractions, and that these fractions indicated significant amount of leaching took place, the bulk of the material was in the much larger size fractions. So for example, sample 6-3-1 had a total mass of 29086 gms, of which 77% was in the size fractions between 6 inches and 10 mesh, where most of the sulfide minerals are enclosed in the silicates and therefore not exposed to leaching.

Future work:

There were a number of interesting results from this work, one of which was the fact that the different size fraction were effected differently by the leaching process. One might expect that the finer size fractions would be the first to react away, whereas it appears that the opposite was true. These smaller size fractions were less effected by the leaching process than the intermediate size fractions. However only three samples were investigated with this degree of detail, where each of the size fractions was looked at, and only two of these show this effect. It might be very useful for future work to investigate this matter more extensively with more samples to determine if this observation is universally true.

Table A5.10.1. Reaction Rating for test pile FL6 spatially distributed samples. Analysis by McSwiggen and Associates.

Sample	Sample #	Grain Size	Completed	Photo #	Mineral	Comment	Scale
1	.6-1-2	A:Tsect	X				
	.6-1-2	H:30-70	X	1	Overview		
				2	cubanite	Highly reacted	3
				3	pyrrhotite	almost reacted away	4
				4	pyrrhotite	highly reacted	3
				5	pentlandite	almost reacted away	4
				6	pyrrhotite	almost reacted away	4
				7	pyrrhotite	almost reacted away	4
				8	cubanite	thin coating	2
				9	cubanite	thin coating	2
2	.6-1-4	A:Tsect	X				
	.6-1-4	H:30-70	X	1	Overview		
				2	cubanite	thick oxide coatig	3
				3	pentlandite	almost reacted away	4
				4	pentlandite	almost reacted away	4
				5	cubanite	thick coating	3
				6	pyrite	almost reacted away	4
				7	cubanite	almost reacted away	4
				8	pyrite	almost reacted away	4
				9	cubanite	thin coating	2
3	.6-1-5	A:Tsect	X				
	.6-1-5	H:30-70	X	1	Overview		
				2	cubanite	almost reacted away	4
				3	cubanite	almost reacted away	4
				4	cubanite	highly reacted	3
				5	cubanite	highly reacted	3
				6	cubanite	highly reacted	3
4	.6-1-7	A:Tsect	X				
	.6-1-7	H:30-70	X	1	Overview		
				2	pyrrhotite	little reaction	2
				3	cubanite	minor reaction	2
				4	pyrrhotite	almost reacted away	4
				5	cubanite	little reaction	2
				6	cubanite	thick caoting	3
				7	pentlandite	highly reaction	4
				8	cubanite	highly reacted	4
				9	cubanite	little reaction	2
				10	cubanite	little reaction	2
				11	pyrrhotite	almost reacted away	4
				12	pyrrhotite	almost reacted away	4
5	.6-2-10	A:Tsect	X				
	.6-2-10	H:30-70	X	1	Overview		
				2	chalcopyrite	highly reacted	3
				3	cubanite	dissolution only	1
				4	cubanite	thin reactions	2
				5	cubanite	thin reactions	2
				6	pyrrhotite	almost reacted away	4
				7	pentlandite	almost reacted away	4
				8	pyrrhotite	thick reaction	3

Table A5.10.1. Reaction Rating for test pile FL6 spatially distributed samples. Analysis by McSwiggen and Associates.

Sample	Sample #	Grain Size	Completed	Photo #	Mineral	Comment	Scale
6	.6-2-2	A:Tsect	X				
	.6-2-2	H:30-70	X	1	Overview		
				2	pyrrhotite	thin reaction	2
				3	pyrrhotite	thick reaction	3
					cubanite	little to no reaction	1
				4	cubanite	no reaction	0
				5	pentlandite	thin coating	2
				6	pentlandite	thin coating	2
				7	cubanite	thin reaction	2
				8	pentlandite	thick coating	3
				9	pyrrhotite	almost reacted away	4
				10	pyrrhotite	almost reacted away	4
7	.6-2-4	A:Tsect	X				
	.6-2-4	C:1.5-0.75	X				
	.6-2-4	D:0.75-0.5	X				
	.6-2-4	E:0.5-0.25	X				
	.6-2-4	F:0.25-10	X	1	overview		
	.6-2-4	G:10-30	X	1	overview		
				2	pyrrhotite	almost reacted away	4
				3	pyrrhotite	almost reacted away	4
				4	chalcopyrite	thin coating	2
				5	chalcopyrite	thin coating	2
				6	pyrrhotite	almost reacted away	4
	.6-2-4	H:30-70	X	1	Overview		
				2	pentlandite	highly reacted	3
				3	cubanite	no reaction	0
				4	chalcopyrite	no reaction	0
				5	chalcopyrite	little reaction	1
				6	pyrrhotite	almost reacted away	4
	.6-2-4	I:70-100	X	1	Overview		
				2	pentlandite	highly reacted	4
				3	pentlandite	highly dissolved	3
				4	cubanite	litle to no reaction	1
				5	cubanite	litle to no reaction	0
	.6-2-4	J:100-150	X	1	Overview		
				2	cubanite	minor evidence of alteration	1
				3	pentlandite	highly reacted	3
				4	chalcopyrite	thin coating	2
				5	cubanite	minor alteration	2
	.6-2-4	K:150-200	X	1	Overview		
				2	pentlandite	minor alteration	2
				3	cubanite	thin coating	2
				4	cubanite	thin coating	2
				5	cubanite	no coating but dissolution	1
				6	cubanite	thin coating	2
	.6-2-4	L:200-270	X	1	overview		
				2	pentlandite	minor reaction	2
				3	cubanite	thin coating	2
				4	cubanite	thin coating	2
				5	cubanite	minor dissolution	1
				6	pentlandite	minor reaction	2

Table A5.10.1. Reaction Rating for test pile FL6 spatially distributed samples. Analysis by McSwiggen and Associates.

Sample	Sample #	Grain Size	Completed	Photo #	Mineral	Comment	Scale
	.6-2-4	M:>270	X	1	Overview		
				2	cubanite	no coating but dissolution	1
				3	cubanite	minor reaction	2
				4	cubanite	minor reaction	2
8	.6-2-7	A:Tsect	X				
	.6-2-7	H:30-70		1	overview		
				2	cubanite	mostly reacted away	4
				3	cubanite	highly reacted	3
				4	cubanite	mostly reacted away	4
				5	cubanite	minor rim coating	2
				6	cubanite	highly reacted	3
9	.6-3-16	A:Tsect	X				
	.6-3-16	H:30-70		1	overview		
				2	pyrrhotite	mostly racted away	4
				3	cubanite	minor disolution	1
				4	cubanite	minor coating	2
				5	cubanite	minor coating	2
				6	pyrrhotite	mostly racted away	4
10	.6-3-1	A:Tsect	X				
	.6-3-1	C:1.5-0.75	X				
	.6-3-1	D:0.75-0.5	X				
	.6-3-1	E:0.5-0.25	X				
	.6-3-1	F:0.25-10	X				
	.6-3-1	G:10-30	X	1	overview		
				2	chalcopyrite	minor dissolution	2
				3	chalcopyrite	minor dissolution	2
				4	chalcopyrite	minor coating	2
				5	chalcopyrite	minor coating	2
	.6-3-1	H:30-70	X	1	ovreview		
				2	pentlandite	high reacted	4
				3	cubanite	minor dissolution	1
				4	pyrrhotite	highly reacted	3
				5	pyrrhotite	mostly reacted away	4
	.6-3-1	I:70-100	X	1	overview		
				2	cubanite	thin coating	2
				3	pyrrhotite	thick coating	3
				4	cubanite	thin coating	2
				5	pentlandite	thin reaction	2
				6	pyrrhotite	highly reacted	4
	.6-3-1	J:100-150	X	1	overview		
				2	pyrrhotite	thin coating	2
				3	cubanite	thin coating	2
				4	pyrrhotite	thin coating	2
				5	pyrrhotite	thin coating	2
				6	cubanite	minor dissolution	1
	.6-3-1	K:150-200	X	1	overview		
				2	cubanite	thin coating	2
				3	cubanite	thin coating	2
				4	cubanite	thin coating	2
				5	pyrrhotite	minor reacted	2
				6	pyrrhotite	major reaction	3

Table A5.10.1. Reaction Rating for test pile FL6 spatially distributed samples. Analysis by McSwiggen and Associates.

Sample	Sample #	Grain Size	Completed	Photo #	Mineral	Comment	Scale
	.6-3-1	L:200-270	X	1	overview		
				2	pyrrhotite	minor dissolution	1
				3	cubanite	minor dissolution	1
				4	pyrrhotite	minor dissolution	1
				5	cubanite	minor dissolution	1
				6	pyrrhotite	thick coating	3
				6	cubanite	minor dissolution	1
	.6-3-1	M:>270	X	1	overview		
				2	pyrrhotite	minor dissolution	1
				3	cubanite	minor dissolution	1
				4	cubanite	thin coating	2
				5	cubanite	thin coating	2
				6	pyrrhotite	minor dissolution	1
11	.6-3-6	A:Tsect	X				
	.6-3-6	H:30-70	X	1	overview		
				2	pentlandite	significantly reacted away	4
				3	cubanite	minor dissolution	1
				4	chalcopyrite	minor dissolution	1
				5	pentlandite	highly reacted	3
				6	cubanite	highly reacted	4
				7	pentlandite	significantly reacted away	4
				8	cubanite	minor coating	2
				9	pyrrhotite	mostly reacted away	4
				10	pyrrhotite	mostly reacted away	4
12	.6-3-8	A:Tsect	X				
	.6-3-8	H:30-70	X	1	overview		
				2	pyrrhotite	mostly reacted away	4
				3	cubanite	minor dissolution	1
				4	cubanite	thin coating	2
				5	chalcopyrite	thin coating	2
				6	pentlandite	highly reacted	3
				7	pyrrhotite	mostly reacted away	4
13	.6-4-2	A:Tsect	X				
	.6-4-2	C:1.5-0.75	X				
	.6-4-2	D:0.75-0.5	X				
	.6-4-2	E:0.5-0.25	X				
	.6-4-2	F:0.25-10	X				
	.6-4-2	G:10-30	X	1	overview		
				2	cemented grains		
				3	pyrrhotite	minor dissolution	1
				4	pyrrhotite	minor dissolution	1
				5	pyrrhotite	minor dissolution	1
				6	pyrrhotite	major dissolution	2
	.6-4-2	H:30-70	X	1	overview		
				2	pyrrhotite	thin coating	2
				3	edge of pyrrhotite		
				4	pyrrhotite	minor dissolution	1
				5	chalcopyrite	minor dissolution	1
				6	pyrrhotite	thin coating	2

Table A5.10.1. Reaction Rating for test pile FL6 spatially distributed samples. Analysis by McSwiggen and Associates.

Sample	Sample #	Grain Size	Completed	Photo #	Mineral	Comment	Scale
	.6-4-2	I:70-100	X	1	overview		
				2	cubanite	minor dissolution	1
				3	pyrrhotite	minor dissolution	1
				4	cubanite	minor dissolution	1
				5	chalcopyrite	minor dissolution	1
				6	bornite	thin coating	2
				7	cubanite	major dissolution	2
				8	pyrrhotite	minor dissolution	1
				9	pyrrhotite	minor dissolution	1
	.6-4-2	J:100-150	X	1	overview		
				2	pyrrhotite	minor dissolution	1
				3	pyrrhotite	thin coating	2
				4	chalcopyrite	significant dissolution	2
				5	chalcopyrite	dissolution and coating	2
				6	pyrrhotite	with coating of bornite	2
	.6-4-2	K:150-200	X	1	overview		
				2	cubanite	minor dissolution	1
				3	pyrrhotite	minor reaction rim	2
				4	chalcopyrite	minor reaction rim	2
				5	cubanite	minor dissolution	1
				6	pyrrhotite	significant dissolution	1
	.6-4-2	L:200-270	X	1	overview		
				2	pentlandite	minor dissolution	1
				3	cubanite	thin coating	2
				4	cubanite	bornite coating	2
				5	cubanite	bornite coating	2
	.6-4-2	M:>270	X	1	overview		
				2	chalcopyrite	minor dissolution	1
				3	chalcopyrite	minor dissolution	1
				4	chalcopyrite	minor dissolution	1
14	.6-4-4	A:Tsect	X				
	.6-4-4	H:30-70	X	1	overview		
				2	pyrrhotite	almost completely reacted	4
				3	pyrrhotite	almost completely reacted	4
				4	pyrrhotite	almost completely reacted	4
				5	cubanite	minor dissolution	1
				6	cubanite	thin coating	2
				7	chalcopyrite	minor dissolution	1
				8	chalcopyrite	minor dissolution	1
15	.6-4-5	A:Tsect	X				
	.6-4-5	H:30-70	X	1	overview		
				2	cubanite	minor dissolution	1
				3	pyrrhotite	minor dissolution	1
				4	pyrrhotite	thin coating	2
				5	cubanite	thin coating	2
				6	native copper	coated with dentritic copper sulfate	

Table A5.10.1. Reaction Rating for test pile FL6 spatially distributed samples. Analysis by McSwiggen and Associates.

Sample	Sample #	Grain Size	Completed	Photo #	Mineral	Comment	Scale
16	.6-4-7	A:Tsect	X				
	.6-4-7	H:30-70		1	overview		
				2	cubanite	highly reacted	3
				3	pyrrhotite	significantly reacted	3
				4	cubanite	thin coating	2
				5	chalcopyrite	thin coating	2
				6	pentlandite	thin coating	2
				7	pentlandite	thin coating	2
				8	cubanite	significant reaction	3

Table A5.10.2. Summary of mineral dissolution/reaction rating. Analysis by McSwiggen and Associates.

				Mineral Dissolution/reaction rating				
Sample	Sample #	Grain size	Completed	Pyrrhotite	Pyrite	Chalco/Cuban	Pentlandite	Disolution scale
1	.6-1-2	A:Tsect	X					
	.6-1-2	H:30-70		4	N	2	4	4
2	.6-1-4	A:Tsect	X					
	.6-1-4	H:30-70		N	4	3	4	5
3	.6-1-5	A:Tsect	X					
	.6-1-5	H:30-70		N	N	3	N	5
4	.6-1-7	A:Tsect	X					
	.6-1-7	H:30-70		4	N	2	4	4
5	.6-2-10	A:Tsect	X					
	.6-2-10	H:30-70		4	N	2	4	4
6	.6-2-2	A:Tsect	X					
	.6-2-2	H:30-70		3	N	1	2	3
7	.6-2-4	A:Tsect	X					
	.6-2-4	C:1.5-0.75	X					
	.6-2-4	D:0.75-0.5	X					
	.6-2-4	E:0.5-0.25	X					
	.6-2-4	F:0.25-10	X					
	.6-2-4	G:10-30		4	N	2	N	4
	.6-2-4	H:30-70		4	N	1	3	4
	.6-2-4	I:70-100		N	N	1	3	4
	.6-2-4	J:100-150		N	N	2	3	4
	.6-2-4	K:150-200		N	N	2	2	3
	.6-2-4	L:200-270		N	N	2	2	3
	.6-2-4	M:>270		N	N	2	N	3
8	.6-2-7	A:Tsect	X					
	.6-2-7	H:30-70		N	N	3	N	5
9	.6-3-16	A:Tsect	X					
	.6-3-16	H:30-70		4	N	2	N	4
10	.6-3-1	A:Tsect	X					
	.6-3-1	C:1.5-0.75	X					
	.6-3-1	D:0.75-0.5	X					
	.6-3-1	E:0.5-0.25	X					
	.6-3-1	F:0.25-10	X					
	.6-3-1	G:10-30		N	N	2	N	3
	.6-3-1	H:30-70		3	N	1	4	3
	.6-3-1	I:70-100		3	N	2	2	3
	.6-3-1	J:100-150		2	N	2	N	2
	.6-3-1	K:150-200		2	N	2	N	2
	.6-3-1	L:200-270		1	N	1	N	1
	.6-3-1	M:>270		1	N	2	N	2
11	.6-3-6	A:Tsect	X					
	.6-3-6	H:30-70		4	N	2	N	4
12	.6-3-8	A:Tsect	X					
	.6-3-8	H:30-70		4	N	2	3	4

Table A5.10.2. Summary of mineral dissolution/reaction rating. Analysis by McSwiggen and Associates.

				Mineral Dissolution/reaction rating				
Sample	Sample #	Grain size	Completed	Pyrrhotite	Pyrite	Chalco/Cuban	Pentlandite	Disolution scale
13	.6-4-2	A:Tsect	X					
	.6-4-2	C:1.5-0.75	X					
	.6-4-2	D:0.75-0.5	X					
	.6-4-2	E:0.5-0.25	X					
	.6-4-2	F:0.25-10	X					
	.6-4-2	G:10-30		1	N	N	N	1
	.6-4-2	H:30-70		2	N	1	N	2
	.6-4-2	I:70-100		1	N	1	N	1
	.6-4-2	J:100-150		2	N	2	N	2
	.6-4-2	K:150-200		1	N	1	N	1
	.6-4-2	L:200-270		N	N	2	1	2
	.6-4-2	M:>270		N	N	1	N	1
14	.6-4-4	A:Tsect	X					
	.6-4-4	H:30-70		4	N	1	N	4
15	.6-4-5	A:Tsect	X					
	.6-4-5	H:30-70		1	N	1	N	1
16	.6-4-7	A:Tsect	X					
	.6-4-7	H:30-70		3	N	2	2	3

Table A5.10.3. Frequency of sulfide mineral photomicrographs for 30-70 mesh fraction.

Sample	Paste pH	Po	Pn	Cb	Cp	Py	Cu^0	Reaction Rating[2]	Weathering Index[3]
6-1-2	3.90	4	1	3	0	0	0	4	0.96
6-1-4	4.37	0	2	4	0	2	0	5	0.68
6-1-5	4.55	0	0	5	0	0	0	5	1.02
6-1-7	3.28	4	1	6	0	0	0	4	0.75
							AVE	4.5	0.85
6-2-2	4.37	4	3	3	0	0	0	3	0.92
6-2-4[1]	5.01	1	1	1	2	0	0	4	1.02
6-2-7	4.10	0	0	5	0	0	0	5	1.00
6-2-10	3.80	2	1	3	1	0	0	4	0.84
							AVE	4.0	0.94
6-3-1[1]	4.51	2	1	1	0	0	0	3	0.85
6-3-6	4.97	2	3	3	1	0	0	4	1.00[4]
6-3-8	4.88	2	1	2	1	0	0	4	1.11
6-3-16	5.44	2	0	3	0	0	0	4	0.92
							AVE	3.8	0.97
6-4-2[1]	3.49	3	0	0	1	0	0	1	0.76[5]
6-4-4	3.68	3	0	2	2	0	0	4	0.89
6-4-5	4.44	2	0	2	0	0	1	1	0.97
6-4-7	3.84	1	2	3	1	0	0	3	0.90
							AVE	2.5	0.88
Total		32	16	46	9	2	1		

[1] All size fractions were examined. See Table A5.10.4.
[2] Reaction rating based on microprobe analysis.
[3] Mixed sulfide weathering index calculated using S, Cu, Ni content of sample from the 30-70 mesh size fraction, unless otherwise noted. MSWI = $[(1.26 \times Cu) + (0.97 \times Ni)]/ S^{2-}$. Cu with Cp and Cb = 1.26 × [Cu] and Ni with Pn = 0.97 × [Ni].
[4] 0.5–2.0 mm fraction used due to negative sulfide content of sample.
[5] Reported as 1.24. Values reported for surrounding size fractions were 0.75 and 0.775.

NOTES:

RR=1: Index = 0.76, **0.97**
RR=3: Index = ***0.85***, *0.90*, ***0.92***
RR=4: Index = *0.75*, *0.84*, 0.89, **0.92**, *0.96*, **1.00**, ***1.02***, 1.11
RR=5: Index = ***0.68***, **1.00**, 1.02
Ave = 0.91, Median = 0.92

Ranges for Index: 0.68-0.76, 0.84-0.92, 0.96-1.02 and 1.11
Italics indicates pentlandite present; **Bold** indicates olivine content 20 to 30 percent. Index 2 average = 0.92
Consider normalizing values to their occurrence in the unleached sample. Their abundances in the unleached sample could be estimated using the S, Cu, and Ni contents of the unleached sample and assuming mineral chemistries.

Table A5.10.4. Frequency of sulfide mineral occurrences in photomicrographs as a function of particle size.

Mesh	Po	Pn	Cb	Cp	Py	Born	Cu^0	Reaction Rating[1]	Weathering Index[2]
6-2-4 (paste pH = 5.01)									
10-30	3	0	0	2	0	0	0	4	0.95
30-70	1	1	1	2	0	0	0	4	1.02
70-100	0	2	2	0	0	0	0	4	1.04
100-150	0	1	2	1	0	0	0	4	0.98
150-200	0	1	4	0	0	0	0	3	0.91
200-270	0	2	3	0	0	0	0	3	0.90
<270	0	0	3	0	0	0	0	3	0.97
							AVE	3.6	0.97
6-3-1 (paste pH = 4.51)									
10-30	0	0	0	4	0	0	0	3	0.79
30-70	2	1	1	0	0	0	0	3	0.85
70-100	2	1	2	0	0	0	0	3	0.78
100-150	3	0	2	0	0	0	0	2	0.83
150-200	2	0	3	0	0	0	0	2	0.79
200-270	3	0	3	0	0	0	0	1	0.71
<270	2	0	3	0	0	0	0	2	1.17
							AVE	2.3	0.85
6-4-2 (paste pH = 3.49)									
10-30	4	0	0	0	0	0	0	1	0.75
30-70	3	0	0	1	0	0	0	2	1.24
70-100	3	0	3	1	0	1	0	1	0.78
100-150[3]	3	0	0	2	0	0	0	2	0.94
150-200	2	0	2	1	0	0	0	1	1.30
200-270[4]	0	1	3	0	0	0	0	2	1.34
<270	0	0	0	3	0	0	0	1	1.63
							AVE	1.4	1.14
Total	33	10	37	17	0	1	0		

[1] Reaction rating based on microprobe analysis.

[2] Mixed sulfide weathering index calculated using S, Cu, Ni content of sample from the 30-70 mesh size fraction, unless otherwise noted. MSWI = $[(1.26 \times Cu) + (0.97 \times Ni)]/ S^{2-}$. Cu with Cp and Cb = 1.26 × [Cu] and Ni with Pn = 0.97 × [Ni].

[3] One Po grain had coating of bornite.

[4] Two grains had bornite (Cu_5FeS_4) coatings.

NOTES: It appears that the relative amounts of Cp and Cb may be a function of the contents in the unleached sample rather than relative susceptibility to oxidation.

Table A5.10.5. Average mineral compositions and sample modal analyses. Analysis by McSwiggen and Associates.

Modal analyses were determined from visual estimations from optical microscopes.
H_2O and excess O_2 was determined by difference.

Sample: 6-4-7

Mineral	**Abundance** (in percent)		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	5%	Average	0.00	0.12	51.73	46.26	0.16	0.46	0.00	0.01	0.00	0.04	0.01	1.13	99.92

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	35%	Average	54.34	29.05	0.00	0.15	0.00	0.00	0.09	5.19	10.78	99.60

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Hypersthene	33%	Average	50.00	0.39	0.16	36.78	11.16	0.54	0.02	0.02	1.07	0.08	100.22

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Augite	15%	Average	51.11	0.72	0.22	19.62	9.39	0.25	0.01	0.11	18.00	0.03	99.46

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Quartz	2%	Average	100.34	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	100.34

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
K-feldspar	10%	Average	58.74	25.18	0.00	0.07	0.00	0.00	5.48	3.75	6.77	99.99

Medium grained granulite. Grain size 0.5 - 2 mm.

Sample: 6-4-5

Mineral	**Abundance**		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	4%	Average	0.00	0.11	53.31	43.34	2.09	0.47	0.00	0.00	0.00	0.10	0.00	0.46	99.90

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	53%	Average	56.10	27.70	0.03	0.10	0.01	0.00	0.48	5.87	9.24	99.51

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Augite	15%	Average	52.67	1.48	0.55	9.02	14.37	0.23	0.00	0.20	20.98	0.07	99.56

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	23%	Average	35.44	0.00	0.03	38.01	25.77	0.49	0.00	0.00	0.08	0.07	0.09	99.98

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Biotite	5%	Average	36.66	14.28	5.47	10.67	16.81	0.04	9.41	0.24	0.10	6.10	99.76

Chlorite	trace

Coarse grade, unaltered gabbro. Grain size 3 - 7 mm.

Table A5.10.5. Average mineral compositions and sample modal analyses. Analysis by McSwiggen and Associates.

Sample: 6-4-4

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	2%	Average	0.00	0.08	52.38	46.02	0.56	0.34	0.00	0.02	0.00	0.21	0.03	0.21	99.85

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	10%	Average	58.64	26.81	0.01	0.07	0.00	0.01	0.21	6.64	8.05	100.44

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Hypersthene	30%	Average	50.38	2.71	0.22	30.19	15.68	0.30	0.01	0.00	0.17	0.19	99.85

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
K-feldspar	5%	Average	64.27	19.32	0.00	0.11	0.00	0.00	13.72	1.43	0.41	99.26

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Cordierite	45%	Average	49.13	33.63	0.01	7.82	8.77	0.05	0.01	0.06	0.01	0.47	99.96

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Biotite	trace	Average	36.70	15.27	4.65	16.07	12.83	0.03	9.50	0.04	0.00	4.18	99.28

		Fe wt	Ni wt	S wt	Cu wt	Total
Pyrrhotite	3%	61.84	0.42	38.92	0.00	101.17

Sericite 5%

Unaltered granulite. Grain size <1 mm. Some of the ilmenite grains are very elongated blades.

Sample: 6-4-2

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	55%	Average	54.69	29.68	0.02	0.09	0.02	0.02	0.25	4.91	11.23	100.89

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Augite	30%	Average	51.10	1.56	0.46	11.37	13.34	0.20	0.00	0.21	19.49	0.00	97.73

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	10%	Average	33.92	0.01	0.00	44.19	20.62	0.50	0.00	0.01	0.03	0.01	0.02	99.29

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Biotite	1%	Average	36.97	14.23	5.39	14.82	14.17	0.00	9.15	0.13	0.02	4.53	99.41

Sulfide	4%	Fe wt	Ni wt	S wt	Cu wt	Total
Cubanite		42.59	0.04	32.58	24.10	99.31
Pyrrhotite		57.52	0.59	42.29	0.00	100.40
Pyrrhotite		58.30	0.64	40.86	0.00	99.80

Unaltered gabbro. Grain size 3 - 10 mm.

Table A5.10.5. Average mineral compositions and sample modal analyses. Analysis by McSwiggen and Associates.

Sample: 6-3-16

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total		
Plagioclase	50%	Average	50.78	31.73	0.00	0.08	0.02	0.01	0.06	3.67	13.53	99.87		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total	
Hypersthene	11%	Average	53.86	0.76	0.26	18.30	24.61	0.42	0.01	0.01	0.91	0.04	99.19	
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total	
Augite	11%	Average	52.78	1.77	0.50	7.57	15.13	0.24	0.00	0.26	20.88	0.17	99.30	
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	25%	Average	36.41	0.00	0.05	32.19	30.48	0.45	0.01	0.00	0.04	0.05	0.15	99.83
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total	
Scapolite	3%	Average	47.22	24.67	0.00	0.02	0.07	0.02	0.00	0.04	13.36	14.47	99.86	
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total	
Serpentine	trace	Average	43.36	0.97	0.00	20.33	21.57	0.13	0.02	0.00	0.54	12.94	99.86	
Biotite	trace													
Ilmenite	trace													

Weakly altered gabbro. Grain size 1 - 2mm.

Sample: 6-3-8

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	10%	Average	0.00	0.13	51.40	45.70	0.68	0.41	0.00	0.01	0.01	0.23	0.10	1.06	99.73
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total			
Plagioclase	40%	Average	57.01	26.89	0.03	0.29	0.01	0.04	0.60	6.23	8.71	99.81			
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total			
K-feldspar	trace	Average	64.52	18.92	0.03	0.09	0.00	0.03	15.08	0.86	0.21	99.74			
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total		
Augite	35%	Average	51.31	1.10	0.23	15.55	10.89	0.23	0.00	0.20	19.34	0.00	98.85		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total	
Olivine	10%	Average	32.48	0.00	0.04	53.59	12.80	0.53	0.00	0.01	0.01	0.00	0.05	99.52	
Chlorite	trace														

Pyroxene rich gabbro, with very little alteration. Grain size 3 - 10 mm.

Table A5.10.5. Average mineral compositions and sample modal analyses. Analysis by McSwiggen and Associates.

Sample: 6-3-6

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	Cr2O3	NiO	ZnO	O2	Total
Magnetite	5%	Average	0.69	0.00	0.00	91.26	1.27	0.37	0.01	0.00	0.00	6.30	99.90

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	50%	Average	51.66	30.86	0.00	0.21	0.02	0.01	0.27	3.90	12.49	99.42

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	20%	Average	36.79	0.00	0.01	28.23	33.58	0.30	0.00	0.00	0.05	0.02	0.13	99.11

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Serpentine	25%	Average	38.59	0.68	0.02	12.63	31.27	0.18	0.00	0.00	0.55	15.89	99.81

Biotite trace

Highly altered gabbro. The olivine is highly altered to serpentine. Grain size is 3- 5 mm.

Sample: 6-3-1

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO T	MgO	MnO	Cr2O3	NiO	ZnO	O2	Total
Magnetite	2%	Average	0.00	0.00	0.01	92.79	0.12	0.43	0.02	0.04	0.10	6.49	100.00

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	30%	Average	48.97	32.97	0.01	0.11	0.01	0.01	0.14	2.73	14.60	99.54

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Augite	10%	Average	52.55	2.15	0.79	6.27	15.26	0.18	0.00	0.26	21.39	0.40	99.25

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	20%	Average	37.06	0.00	0.04	28.92	33.18	0.40	0.00	0.00	0.05	0.05	0.20	99.91

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Biotite	trace	Average	37.89	14.30	6.02	9.64	17.24	0.01	8.71	0.71	0.00	4.57	99.08

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Amphibole	3%	Average	44.62	10.63	1.01	9.98	14.85	0.10	0.23	2.44	11.37	4.36	99.60

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Serpentine	33%	Average	38.37	0.37	0.04	19.30	26.57	0.27	0.00	0.02	0.12	14.47	99.51

Sulfides	2%	Fe wt	Ni wt	S wt	Cu wt	Total
Bornite		15.77	0.01	25.17	60.57	101.51
Chalcopyrite		32.28	0.01	30.86	35.77	98.92
Chalcopyrite		32.58	0.00	31.13	35.99	99.70

Extremely altered gabbro. The olivine is has been largely retrograded to serpentine. The pyroxene is less altered. Grain size 5-15 mm.

Table A5.10.5. Average mineral compositions and sample modal analyses. Analysis by McSwiggen and Associates.

Sample: 6-2-10

Mineral	Abundance													
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	2%	Average	0.00	0.08	53.15	44.80	1.05	0.46	0.00	0.00	0.01	0.15	0.01	0.24	99.96
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total			
Plagioclase	47%	Average	49.11	33.43	0.00	0.24	0.01	0.02	0.58	2.48	14.16	100.03			
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total		
Hypersthene	30%	Average	53.80	0.43	0.15	22.85	21.67	0.38	0.00	0.00	0.98	0.11	100.38		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total			
K-spar	1%	Average	51.08	32.04	0.00	0.80	0.42	0.00	3.52	3.19	7.32	98.38			
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total			
Quartz	2%	Average	99.82	0.02	0.02	0.09	0.01	0.06	0.01	0.00	0.00	100.02			
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total		
Augite	15%	Average	54.13	0.89	0.29	8.86	14.52	0.18	0.00	0.15	20.33	0.20	99.54		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total		
Biotite	trace	Average	38.92	13.46	4.38	12.91	16.39	0.05	9.22	0.12	0.02	3.96	99.43		
			Fe wt	Ni wt	S wt	Cu wt	Total								
Chalcopyrite	3%		31.43	0.00	34.12	34.58	100.13								
Musc/Talc	trace														

Gabbro consisting of large poikiloblasts of pyroxene with plagioclase inclusion. Grain size 1-5 mm.

Sample: 6-2-7

Mineral	Abundance												
			SiO2	Al2O3	TiO2	FeO T	MgO	MnO	Cr2O3	NiO	ZnO	O2	Total
Magnetite	trace	Average	2.56	0.00	0.00	87.40	1.58	0.00	0.02	0.00	0.09	8.34	100.00
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total	
Plagioclase	55%	Average	51.59	30.80	0.03	0.38	0.10	0.02	0.17	3.90	12.73	99.72	
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Augite	2%	Average	52.74	1.60	0.42	6.76	16.10	0.23	0.06	0.21	21.14	0.31	99.57
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	23%	Average	37.34	0.00	0.03	28.56	33.62	0.35	0.00	0.00	0.06	0.05	0.17	100.17
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Serpentine	20%	Average	38.10	0.13	0.05	15.37	30.86	0.08	0.01	0.10	0.17	14.86	99.73

Highly altered gabbro. The olivine is highly altered to serpentine. Grain size 1-3 mm.

Table A5.10.5. Average mineral compositions and sample modal analyses. Analysis by McSwiggen and Associates.

Sample: 6-2-4

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	5%	Average	0.00	0.14	53.30	42.93	2.55	0.48	0.00	0.01	0.00	0.22	0.03	0.31	99.95

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	30%	Average	52.65	30.62	0.01	0.15	0.00	0.02	0.11	4.30	12.45	100.32

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Hypersthene	35%	Average	53.75	0.73	0.18	20.03	23.89	0.42	0.01	0.00	0.82	0.10	99.93

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	20%	Average	36.36	0.01	0.03	34.78	28.26	0.45	0.02	0.01	0.07	0.08	0.20	100.25

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Biotite	5%	Average	38.60	14.61	4.82	8.42	18.72	0.04	9.44	0.37	0.01	4.50	99.52

Calcite	trace
Chlorite	trace

Gabbro with weak alteration on one size of the sample. It looks like a vien. Grain size 1 - 4 mm.

Sample: 6-2-2

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	2%	Average	0.00	0.10	53.68	43.11	2.37	0.49	0.00	0.00	0.00	0.02	0.07	0.03	99.87

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	45%	Average	51.83	30.91	0.05	0.38	0.35	0.02	0.21	3.91	12.73	100.38

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Hypersthene	7%	Average	54.12	0.59	0.23	20.08	23.83	0.35	0.01	0.00	1.13	0.04	100.37

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Augite	8%	Average	51.92	1.74	1.17	10.65	15.36	0.20	0.00	0.22	18.55	0.08	99.90

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	30%	Average	35.98	0.01	0.08	35.18	28.50	0.41	0.01	0.00	0.05	0.12	0.26	100.60

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Biotite	5%	Average	37.97	14.29	5.90	10.09	17.72	0.02	9.57	0.25	0.03	3.84	99.66

Sulfides	3%	Fe wt	Ni wt	S wt	Cu wt	Total
Pyrrhotite		64.41	0.07	36.69	0.04	101.21
Pentlandite		36.92	29.32	34.01	0.08	100.33

Coarse grained gabbro, with no significant alteration. Commonly the olivine is coated with by pyroxene. Grain size 3-5 mm.

Table A5.10.5. Average mineral compositions and sample modal analyses. Analysis by McSwiggen and Associates.

Sample: 6-1-7

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	8%	Average	0.00	0.08	52.74	44.38	1.41	0.26	0.01	0.02	0.00	0.28	0.00	0.61	99.79

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	62%	Average	54.27	29.29	0.00	0.10	0.00	0.00	0.31	5.01	11.19	100.17

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Hypersthene	25%	Average	51.58	0.92	0.24	29.13	17.25	0.36	0.00	0.01	0.62	0.08	100.20

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
K-feldspar	3%	Average	63.91	19.13	0.09	0.02	0.00	0.01	14.99	0.70	0.07	98.92

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Chlorite	trace	Average	32.26	16.49	0.13	22.98	13.70	0.16	0.77	0.23	1.18	11.83	99.71

Biotite 2%

Fine grain. Little alteration. Grain size 0.1 to 1 mm.

Sample: 6-1-5

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	5%	Average	0.00	0.10	52.40	46.22	0.15	0.50	0.00	0.00	0.00	0.01	0.04	0.43	99.85

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total
Plagioclase	35%	Average	55.08	28.76	0.01	0.21	0.02	0.00	0.15	5.19	10.68	100.10

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total
Hypersthene	43%	Average	49.62	0.56	0.27	36.03	12.03	0.48	0.00	0.02	0.92	0.02	99.95

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Grunerite	5%	Average	52.72	0.61	0.16	29.71	12.61	0.37	0.01	0.03	1.30	2.32	99.83

			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total
Biotite	7%	Average	35.66	13.74	4.45	22.96	8.98	0.03	9.07	0.08	0.00	4.94	99.90

Unaltered gabbro. Grain size 1-2 mm.

Table A5.10.5. Average mineral compositions and sample modal analyses. Analysis by McSwiggen and Associates.

Sample: 6-1-4

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total		
Plagioclase	45%	Average	55.11	28.90	0.04	0.10	0.00	0.01	0.51	5.31	10.50	100.48		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total	
Greenalite	15%	Average	42.39	3.04	0.05	26.05	13.09	0.11	0.08	0.02	1.57	13.36	99.75	
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total	
Augite	10%	Average	53.09	1.79	0.55	7.66	15.07	0.22	0.01	0.28	21.67	0.23	100.55	
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	Total
Olivine	20%	Average	36.48	0.00	0.11	33.90	29.33	0.41	0.02	0.03	0.08	0.13	0.11	100.58
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total	
Biotite	trace	Average	38.65	14.80	4.16	9.79	18.62	0.02	8.62	0.86	0.05	3.85	99.40	
Sericite	5%													
Pyrrhotite	5%													

Very coarse grained gabbro. Olivine is moderately altered to greenalite; and plagioclase altered to sericite. Grain size 5 -10 mm.

Sample: 6-1-2

Mineral	Abundance		SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	O	Total
Ilmenite	5%	Average	0.00	0.08	52.50	45.71	0.65	0.47	0.00	0.03	0.00	0.03	0.00	0.28	99.76
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total			
Plagioclase	25%	Average	55.10	29.08	0.01	0.07	0.00	0.00	0.12	5.46	10.60	100.44			
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total		
Hypersthene	35%	Average	50.04	0.55	0.22	34.76	12.77	0.46	0.01	0.01	1.11	0.06	99.97		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	Total		
Augite	15%	Average	51.91	1.11	0.39	13.45	11.88	0.23	0.01	0.21	20.50	0.06	99.76		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	H2O	Total		
Biotite	10%	Average	35.77	13.84	4.93	22.67	9.12	0.02	8.91	0.11	0.03	4.44	99.85		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Total			
Quartz	10%	Average	99.82	0.02	0.00	0.04	0.00	0.00	0.00	0.01	0.00	99.89			
Sercite	trace														
Apatite	trace														

Table A5.10.6. Average mode and average mineral composition for FL6 based on analysis of 16 spatially distributed samples. Analyis by McSwiggen and Associates.

Mineral	Generic Formula	Average Mode[3]	Average Composition (weight percent)																		
			SiO2	Al2O3	TiO2	FeO	MgO	MnO	K2O	Na2O	CaO	Cr2O3	NiO	ZnO	O	H2O	Fe	Ni	S	Cu	Total
Plagioclase	$(NaCa)(Si,Al)_4O_8$	42%	53.56	29.84	0.02	0.17	0.04	0.01	0.27	4.67	11.48										100.04
Hypersthene	$(Mg,Fe)_2Si_2O_6$	28%	51.90	0.85	0.22	27.57	18.10	0.41	0.01	0.01	0.86	0.08									100.01
Olivine	$(Mg,Fe)_2SiO_4$	20%	35.82	0.00	0.04	35.75	27.61	0.43	0.01	0.01	0.05	0.06	0.14								99.92
Augite	$(Ca,Na)(Mg,Fe)(Si,Al)_2O_6$	15%	52.30	1.45	0.51	10.62	13.75	0.22	0.01	0.21	20.21	0.14									99.41
Serpentine	$Mg_3Si_2O_5(OH)_4$	26%	39.60	0.54	0.03	16.91	27.56	0.17	0.01	0.03	0.34					14.54					99.73
Quartz	SiO_2	5%	99.99	0.01	0.01	0.04	0.00	0.02	0.00	0.00	0.00										100.09
K-feldspar	$KAlSi_3O_8$	5%	60.50	22.92	0.02	0.22	0.08	0.01	10.56	1.99	2.96										99.26
Biotite	$K_2(Mg,Fe)_3[AlSi_3O_{10}(OH)_2]$	5%	37.43	14.28	5.10	13.47	15.30	0.03	9.20	0.29	0.03					4.43					99.54
Sericite	$KAl_2(Si_3Al)O_{10}(OH)_2$[1]	5%																			0.00
Sulfides		3%																			0.00
Pyrrhotite	$Fe_{1-x}S$	4%															60.52	0.43	39.69	0.01	100.64
Cubanite	$CuFe_2S_3$																42.59	0.04	32.58	24.10	99.31
Chalcopyrite	$CuFeS_2$	3%															33.31	7.33	32.53	26.60	99.77
Bornite	Cu_5FeS_4																15.77	0.01	25.17	60.57	101.51
Ilmenite	$FeTiO_3$	5%	0.00	0.10	52.66	44.85	1.17	0.44	0.00	0.01	0.00	0.13	0.03		0.48						99.86
Magnetite	Fe_2O_4	4%	1.08	0.00	0.00	90.48	0.99	0.27				0.02	0.01	0.06	7.05						99.97
Chlorite		trace	32.26	16.49	0.13	22.98	13.70	0.16	0.77	0.23	1.18					11.83					99.71
Muscovite/Talc	$Mg_3Si_4O_{10}(OH)_2$[2]	trace																			0.00
Cordierite	$(Mg,Fe)_2Al_4Si_5O_{18}$	45%	49.13	33.63	0.01	7.82	8.77	0.05	0.01	0.06	0.01					0.47					99.96
Scapolite	$(Na,Ca)_4[Al_3Si_9O_{24}]Cl$	3%	47.22	24.67	0.00	0.02	0.07	0.02	0.00	0.04	13.36					14.47					99.86
Calcite	$CaCO_3$	trace																			0.00
Amphibole		3%	44.62	10.63	1.01	9.98	14.85	0.10	0.23	2.44	11.37					4.36					
Grunerite	$Fe_7Si_8O_{22}(OH)_2$	5%	52.72	0.61	0.16	29.71	12.61	0.37	0.01	0.03	1.30					2.32					99.83
Greenalite	$Fe_{2-3}Si_2O_5(OH)_4$	15%	42.39	3.04	0.05	26.05	13.09	0.11	0.08	0.02	1.57					13.36					99.75
Apatite	$Ca_5(PO_4)_3(OH)$	trace																			0.00

[1] assumed white mica, used muscovite formula

[2] formua for talc, for muscovite see sericite

[3] calculated as sum of the mode of a given mineral divided by the number of samples that contained that mineral

Table A5.10.7. Average mode (with statistics) calculated by various methods FL6 based on analysis of 16 spatially distributed samples. Analyis by McSwiggen and Associates.

Mineral	Average Mode[1]	Standard Deviation	Minimum	Maximum	Number of Values	Normalized Average Mode	Altered Normalized Average Mode[2]	Adjusted Average Mode[3]	Altered Normalized Adjusted Avg Mode[4]
Plagioclase	42%	13%	10%	62%	16	17%	25%	42%	44%
Hypersthene	28%	12%	7%	43%	9	12%	17%	16%	16%
Olivine	20%	6%	10%	30%	10	8%	12%	13%	13%
Augite	15%	10%	2%	35%	11	6%	9%	10%	11%
Serpentine	26%	7%	20%	33%	3	11%	16%	5%	5%
Quartz	5%	5%	2%	10%	3	2%	3%	0.9%	1%
K-feldspar	5%	4%	1%	10%	4	2%	3%	1%	1%
Biotite	5%	3%	1%	10%	7	2%	3%	2%	2%
Sericite	5%	0%	5%	5%	2	2%	3%	0.6%	1%
Sulfides	3%	1%	2%	4%	3	1%	2%	0.6%	1%
Pyrrhotite	4%	1%	3%	5%	2	2%	2%	0.5%	1%
Cubanite									
Chalcopyrite	3%				1	1%		0.2%	
Bornite									
Ilmenite	5%	3%	2%	10%	10	2%	3%	3%	3%
Magnetite	4%	2%	2%	5%	2	1%	2%	0.4%	0%
Chlorite	trace								
Muscovite/Talc	trace								
Cordierite	45%				1	19%		3%	
Scapolite	3%				1	1%		0.2%	
Calcite	trace								
Amphibole	3%				1	1%		0.2%	
Grunerite	5%				1	2%		0.3%	
Greenalite	15%				1	6%		0.9%	
Apatite	trace								
total	239%					100%	100%	99%	100%
partial total[2]	165%							94%	

[1] calculated as sum of the modes of a given mineral divided by the number of samples that contained that mineral

[2] removed minerals that were only in one sample

[3] calculated as (average mode) * (number of samples with that mineral) / 16

[4] removed minerals that were only in one sample; (adjusted average mode)/(partial total)

Table A5.10.8. Determination of mode in weight percent for the spatially distributed samples.

Mode of Minerals in Each Sample [1]

Sample	Plag	Hyp	Ol	Aug	Serp	Qtz	Kspar	Bio	Ser	Sulf[2]	Ilm	Mag	Total
6-1-02	25%	35%		15%		10%		10%			5%		100%
6-1-04	45%		20%	10%					5%	5%			85%
6-1-05	35%	43%						7%			5%		90%
6-1-07	62%	25%					3%	2%			8%		100%
6-2-02	45%	7%	30%	8%				5%		3%	2%		100%
6-2-04	30%	35%	20%					5%			5%		95%
6-2-07	55%		23%	2%	20%								100%
6-2-10	47%	30%		15%		2%	1%			3%	2%		100%
6-3-01	30%		20%	10%	33%					2%		2%	97%
6-3-06	50%		20%		25%							5%	100%
6-3-08	40%		10%	35%							10%		95%
6-3-16	50%	11%	25%	11%									97%
6-4-02	55%		10%	30%				1%		4%			100%
6-4-04	10%	30%					5%		5%	3%	2%		55%
6-4-05	53%		23%	15%				5%			4%		100%
6-4-07	35%	33%		15%		2%	10%				5%		100%

Density	Plag[3]	Hyp[4]	Ol[5]	Aug[6]	Serp[7]	Qtz[8]	Kspar[9]	Bio[10]	Ser[11]	Sulf[12]	Ilm[13]	Mag[14]
g/cc^3	2.705	3.5	3.3	2.96	2.55	2.65	2.6	3	2.83	4.6	4.74	5.2

Weight Proportions (mode x density)

Sample	Plag	Hyp	Ol	Aug	Serp	Qtz	Kspar	Bio	Ser	Sulf	Ilm	Mag	Total
6-1-02	0.676	1.23	0	0.444	0	0.265	0	0.3	0	0	0.237	0	3.15
6-1-04	1.217	0	0.66	0.296	0	0	0	0	0.142	0.23	0	0	2.54
6-1-05	0.947	1.51	0	0	0	0	0	0.21	0	0	0.237	0	2.90
6-1-07	1.677	0.88	0	0	0	0	0.078	0.06	0	0	0.379	0	3.07
6-2-02	1.217	0.25	0.99	0.237	0	0	0	0.15	0	0.138	0.095	0	3.07
6-2-04	0.812	1.23	0.66	0	0	0	0	0.15	0	0	0.237	0	3.08
6-2-07	1.488	0	0.76	0.059	0.51	0	0	0	0	0	0	0	2.82
6-2-10	1.271	1.05	0	0.444	0	0.053	0.026	0	0	0.138	0.095	0	3.08
6-3-01	0.812	0	0.66	0.296	0.842	0	0	0	0	0.092	0	0.104	2.81
6-3-06	1.353	0	0.66	0	0.638	0	0	0	0	0	0	0.26	2.91
6-3-08	1.082	0	0.33	1.036	0	0	0	0	0	0	0.474	0	2.92
6-3-16	1.353	0.39	0.83	0.326	0	0	0	0	0	0	0	0	2.89
6-4-02	1.488	0	0.33	0.888	0	0	0	0.03	0	0.184	0	0	2.92
6-4-04	0.271	1.05	0	0	0	0	0.13	0	0.142	0.138	0.095	0	1.82
6-4-05	1.434	0	0.76	0.444	0	0	0	0.15	0	0	0.19	0	2.98
6-4-07	0.947	1.16	0	0.444	0	0.053	0.26	0	0	0	0.237	0	3.10

Table A5.10.8. Determination of mode in weight percent for the spatially distributed samples.

Weight Percent

Sample	Plag	Hyp	Ol	Aug	Serp	Qtz	Kspar	Bio	Ser	Sulf	Ilm	Mag	Total
6-1-02	21	39	0	14	0	8	0	10	0	0	8	0	100
6-1-04	48	0	26	12	0	0	0	0	6	9	0	0	100
6-1-05	33	52	0	0	0	0	0	7	0	0	8	0	100
6-1-07	55	29	0	0	0	0	3	2	0	0	12	0	100
6-2-02	40	8	32	8	0	0	0	5	0	4	3	0	100
6-2-04	26	40	21	0	0	0	0	5	0	0	8	0	100
6-2-07	53	0	27	2	18	0	0	0	0	0	0	0	100
6-2-10	41	34	0	14	0	2	1	0	0	4	3	0	100
6-3-01	29	0	24	11	30	0	0	0	0	3	0	4	100
6-3-06	46	0	23	0	22	0	0	0	0	0	0	9	100
6-3-08	37	0	11	35	0	0	0	0	0	0	16	0	100
6-3-16	47	13	29	11	0	0	0	0	0	0	0	0	100
6-4-02	51	0	11	30	0	0	0	1	0	6	0	0	100
6-4-04	15	58	0	0	0	0	7	0	8	8	5	0	100
6-4-05	48	0	26	15	0	0	0	5	0	0	6	0	100
6-4-07	31	37	0	14	0	2	8	0	0	0	8	0	100

[1] from McSwiggen and Assoiates data; only using "major" minerals, see mode listing; when mineral = trace, not included

[2] sulf = used mode for sulfides or pyrrhotite or chalcopyrite

[3] Deer, Howie, Zussman. 1966. Introduction to Rock Forming Minerals. Longman Scientific & Technical, Essex.; page 325, labradorite

[4] from DHZ, page 112, En_{60}

[5] from DHZ, page 6, Fo_{90}

[6] from DHZ, page 120, end member augite

[7] from DHZ, page 242, very similar across solid solution

[8] from DHZ, page 340

[9] from DHZ, page 300, for orthoclase

[10] from DHZ, page 211, average

[11] from DHZ, page 201, average

[12] from DHZ, page 450, pyrrhotite

[13] from DHZ, page 412, average

[14] from DHZ, page 424

APPENDIX 6

GEOCHEMICAL MODELING

Geochemical considerations:

Flow in an unsaturated waste-rock pile is heterogeneous by nature. Tracer tests have revealed that water draining from a pile actually represents a composite of waters having very different flow paths, residence times, and compositions (Smith and Beckie, 2003). As a result, a wide variety of mineral assemblages may be produced in different portions of the pile, and the composition of drainage water can vary considerably through time in response to seasonal and storm related events. In this section, we discuss thermodynamic constraints on metal transport processes and attempt to equate mineral assemblages found with drainage quality observed. We first provide mineral saturation calculations on drainage water to provide a reference for the dominant minerals involved in the chemical processes taking place in the rock piles, and then attempt to account for heterogeneity by providing reaction path simulations for changing proportions of sulfide and silicate minerals in rock piles. These calculations are provided as a means to asses the link between metal transport and mineralogy for Duluth Gabbro waste rock piles.

All calculations were performed using Geochemist's Workbench, Version 4.01 (Bethke, 2002). We employed the default dataset, "thermo.dat", which supports activity coefficients calculated according to an extended form of the Debye-Hückel equation (the "B-dot" equation), and which was compiled by T. Wolery, J. Delany, K. Jackson, J. Johnson, and other members of the geochemical modeling group at Lawrence Livermore National Laboratories.

Mineral Saturation

Methods: Fluid speciation calculations were conducted to determine mineral saturation for a representative set of drainage samples from each of the six AMAX waste-rock test piles described in the earlier sections of this paper. The calculations were performed on three samples from each pile, with individual samples selected on the basis of measured specific conductance: one each with high, intermediate, or low conductivity.

In each case that the measured values for pH, SO_4, Ca, Mg, Na, Ni, Cu, Zn, and Co were used in the speciation calculations, but that dissolved SiO2, Fe^{+++}, and Al^{+++} activities were controlled by saturation with respect to cristobalite (SiO_2), goethite (FeOOH), and kaolinite ($Al_2Si_2O_5(OH)_4$), respectively. Saturation assumptions often provide the best means to assess saturation when some elements have concentrations that are either very low or were not measured.

FeOOH and other Fe-oxyhydroxides are ubiquitous products of Fe-sulfide oxidation so it is reasonable to expect Fe^{+++} concentrations to be buffered at values close to or slightly above goethite saturation limits. Moreover, Fe can occur in solution as either Fe^{+++} or Fe^{++} and unless the pH is very low, the total iron concentration will be very low and difficult to measure accurately. Initial precipitation of Fe^{+++} is in the form of a more soluble and amorphous $Fe(OH)_3$ phase and progresses to less soluble forms such as goethite and eventually hematite. Goethite was chosen as a reasonable intermediate form.

In addition, the sporadic measurements for dissolved SiO_2 indicated that the solutions were characteristically supersaturated with respect to quartz, but under-saturated with respect to amorphous silica. Cristobalite saturation appears to provide a reasonable alternative means to predict the level of $SiO_{2(aq)}$ in solution, even though the concentration of $SiO_{2(aq)}$, itself, might have been controlled by a steady state process - such as coupled dissolution and precipitation, respectively, of primary and secondary silicate phases.

Finally, because Al concentrations can also be difficult to measure, selection of the mineral kaolinite to estimate Al^{+++} activity in solution has become a common practice used by geochemists to model groundwater evolution under mildly acidic conditions. Kaolinite is a clay mineral that commonly forms by the weathering of plagioclase (See for example, Drever, 1998).

To facilitate a full fluid speciation, it is common practice to adjust the concentration of the dominant anion or cation in solution to ensure the positive electrical charge from cations in solution is exactly balanced by the negative charge from anions. In this case, $SO_4^{=}$ values were adjusted from their measured values for this purpose.

It was also assumed that the water was saturated with respect to air (atmospheric O_2 and CO_2 levels). This assumption is in keeping with observations of general oxidation of pyrrhotite throughout the pile. Not all portions of the pile were likely open to the atmosphere, but this assumption is probably valid for waters penetrating the mostly open sections of the piles where the bulk of the solutions flowed. A possibility exists that CO_2 levels could be enhanced through minor dissolution of carbonate minerals and/or bacterial activity (oxidation of organic compounds) within the waste rock piles.

Results:

Within narrow limits, all of the speciation and mineral saturation calculations provided similar results so we can look at the results from one such calculation to illustrate results for the entire set of calculations (Table 1).

First, we note that the solution is dominated by dissolved $SO_4^{=}$, Ca^{++}, and Mg^{++}, but that the activities of these species are much lower than their bulk molal concentrations. In the sample speciation results shown in Table 1, the activities of $SO_4^{=}$, Mg^{++}, and Ca^{++} are only 0.0078, 0.0051, and 0.0018, respectively (molal and shown in log values), while their concentrations in solution are much higher: 0.031, 0.0071, and 0.0177 molal, respectively. This is because these ions combine with each other to form molecular species (complexes) such as $CaSO_4^o$ and $MgSO_4^o$ and also because activity coefficients for the remaining unbound ions are much less than unity. Similar effects affect the activity/concentration relations for the other dissolved metals, including, Cu^{++}, Ni^{++}, Zn^{++}, and Co^{++}, each of which complexes with $SO_4^{=}$ and has activity coefficients significantly less than unity.

In terms of minerals saturation, it appears that the generally acidic pH conditions that prevail in acidic mine drainage results in super-saturation with respect to relatively few minerals. However, virtually all of the solutions that were speciated as described

above were found to be supersaturated with respect to $Co(FeO_2)_2$, hematite, and nontronite, as well as the more stable polymorphs of SiO_2 (quartz, trydimite, chalcedony). A few samples, generally with pH above 5, were additionally supersaturated with respect to $Co_2(SiO_4)$. In one sample with pH above 6, the solution was found to be super-saturated with respect to $Co(FeO_2)_2$, $Co_2(SiO_4)$, Co_3O_4, tenorite (CuO), and $NiFe_2O_4$.

It is important to note, however, that super-saturation does not imply that a phase will precipitate. For example, although $Co(FeO_2)_2$ and the other Co-bearing phases might be thermodynamically favored under some conditions, we saw no evidence in the mineralogic or water chemistry data to support the notion that such phases actually precipitated. Without confirming data to indicate that precipitation was actually taking place, it should not be assumed that it did take place. $NiFe_2O_4$ is another phase that was predicted for one solution, but which based on other data probably did not precipitate in the waste-rock piles.

On the other hand, one super-saturated mineral that *was* observed in the waste rock pile is nontronite, a dioctahedral Fe^{+++}-rich, Al-poor expandable clay mineral (smectite). However, the nontronite found in the waste rock pile may or may not have a composition similar to that in the thermodynamic database. It is noteworthy, however, to observe that all of the ingredients needed for nontronite precipitation are provided in a coupled reaction involving oxidative dissolution of pyrrhotite and titration of the resulting acid by olivine and plagioclase. These processes release a steady supply of lattice elements (Fe^{+++}, Al^{+++}, SiO_2) and exchange cations (Ca^{++}, Na^{+}, and Mg^{++}) to solution. Thus, future study of the chemistry and distribution of nontronite in the waste-rock pile, a mineral containing an exchangeable cation site, may prove to have important consequences for understanding pH and metal mobility in waste-rock piles.

Some solutions were supersaturated with respect to tenorite (CuO), but only when pH approached neutrality. Tenorite or another Cu-oxide phase was observed in the waste-rock piles, consistent with the geochemical calculations. Other copper phases were found in the waste rock pile, even though the tested solutions were not found to be super-saturated with respect to them. These minerals include malachite, minor bornite, and native copper. The presence of these phases suggest, in the case of malachite, that CO_2 was sometimes more abundant than assumed, and in the case of bornite and native Cu, that O_2 was sometimes less abundant than assumed. This is, perhaps, illustrated best through use of phase diagrams that depict copper stability as a function of pH and the "fugacity" (or effective activity) of O_2 at given CO_2 levels within the waste rock pile (Figures 1 and 2).

For each of these diagrams it was assumed that a_{SO4} was10 mmolal (a_{SO4}=0.01), the fluid was in equilibrium with the mineral goethite, and that copper was considered "soluble" once the total copper in solution reached an activity of 0.001. These constraints are somewhat arbitrarily chosen to represent conditions in at least some portions of the waste-rock piles. Higher activities for copper and sulfate would result in expansion of the stability fields for copper and sulfur-bearing phases, respectively, while lower assumed activities would decrease the size of the stability fields.

Phase relations for well-aerated, acidic portions of the test piles are best represented by the low pH, high f_{O2} portion of the diagrams, where dissolved Cu^{++} dominates copper chemistry. Under oxidizing conditions and for pH greater than approximately 5.5, the most stable copper phase is tenorite (CuO) if the CO_2 fugacity is similar to that in the atmosphere (Fig. 1). However, the stability field for tenorite is replaced by malachite if the CO_2 fugacity is considerably higher than atmospheric values. Thus, observed malachite in some portions of the test piles indicates that those portions of the pile were dominated by near neutral pH and relatively high CO_2, compared to atmospheric. The fugacity of CO_2 needed to stabilize malachite relative to tenorite is more than ten times the atmospheric level.

Cuprite (Cu_2O) and elemental copper (Cu) were also observed in some portions of some piles, and these phases can be used to imply conditions where O_2 is less abundant than in air. Under reducing conditions, where f_{O2} is less than approximately 10^{-55} and in the presence of 10 mmolal total sulfur, copper- and copper-iron sulfide minerals (chalcocite, bornite, and chalcopyrite) become the most stable phases.

Finally, a few other minerals that were observed but not predicted to precipitate from the sampled waters, include gypsum and elemental sulfur. Phase relations also indicate that elemental sulfur that may be stable under condition of low f_{O2} and low to neutral pH (Fig. 3). Of the waters that were speciated, none were supersaturated with respect to gypsum, however, the abundant nature of both Ca^{++} and $SO_4^{=}$ in all of the drainage waters make this a likely phase to precipitate when drainage waters evaporate within the pile, or in portions of the pile where oxidative dissolution of pyrrhotite is enhanced by longer residence time and/or by relatively high surface area for pyrrhotite and plagioclase.

Reaction path modeling and metal transport processes:

The heterogeneous nature of air and water transport in rock piles makes it possible for a large range of conditions to exist simultaneously within a pile and to change with time. As discussed in the main body of the paper, alteration varies throughout the piles from nearly unaltered and fresh, to completely oxidized and altered, especially within the finest grain fractions. The chemical composition of water flowing from the waste rock piles, however, varies seasonally in response to changing temperatures and water availability.

Depending on the local permeability and flow constraints and on the prevailing temperatures, humidity, and precipitation rates, interstices may be locally filled with air, water, or a mixture of aqueous and gas phases. When water fills the voids, oxygen transport to reactive metal-sulfide surfaces may become limited by diffusion through water, and the local oxygen supply might become depleted and eventually consumed. If on the other hand, air fills the interstices, then sulfide oxidation may take place continuously, but transport of oxidation products from the mineral surfaces can become rate-limiting for a time (Jerz and Rimstidt, 2004), at least until the time that water penetrates that portion of the pile and rinses the soluble reaction products away.

Maximum oxidation of sulfide minerals occurs when conditions are humid, oxygen is readily available, and mineral products are periodically rinsed.

The large amount of oxidation observed throughout the waste rock piles we studied indicated that oxygenated conditions prevailed in all but a few smaller portions of the test piles. Under wet, oxygenated conditions the most important parameter is probably the relative ratio of surface areas for oxidizing sulfide minerals compared to that of pH buffering silicate minerals. If acid-generating reactions dominate, the pH of the solutions will be low and the metals released will largely remain in solution. If acid neutralizing reactions dominate, then pH will remain near neutral and some or most of the metals released to solution may precipitate and/or adsorb.

The following three computer simulations demonstrate reaction between sulfide minerals and iron-oxides under air-saturated conditions but under varying rates of acid generation and neutralization. In addition to evaluating chemical effects in sections of the piles where residence times might be long and reaction greater, the simulations allow evaluate absorption and precipitation reactions that affect element transport. In particular, Geochemists Workbench contains an extensive database for modeling the adsorption of metals to Fe-oxide minerals (Dzombak and Morrel, 1990) as well as solubility data for a range of sulfide and oxide minerals. By adjusting the relative rates of sulfide oxidation and mineral titration, we test the effect of pH on metal adsorption and precipitation under a very wide range of conditions that may exist in the test piles.

In each simulation, metal sulfides (pyrrhotite, sphalerite, chalcopyrite, NiS, and CoS) were oxidized at rates that simulated the relative release rates for SO_4, Cu, Ni, Co, and Zn observed in the field test piles. Although Fe^{++} is released when pyrrhotite oxidizes, it is quickly oxidized to Fe^{+++} and precipitated in a phase that can adsorb other trace metals (Fe-oxyhydroxides). The amount of metal adsorption is dependent on solution chemistry as well as on the surface area of the iron hydroxide phase. In this case, we adopted Dzombak and Morrel's default value for surface area of Fe-oxyhydroxides precipitating from solution (600 m^2/g). These values were derived from experimental measurement and likely represent maximum surface area, because it would be expected that the Fe-oxyhydroxide minerals would recrystallize to coarser grain sizes with time. Thus, these calculations probably provide a maximum estimate for metal adsorption by Fe-oxides during oxidation of the Duluth Complex.

Titration was accomplished, in this case, by addition of $Ca(OH)_2$ and $Mg(OH)_2$ at rates adjusted until steady state pH values were similar to those observed in the field. Because field observations indicated an increase in Mg^{++} relative to Ca^{++} in low pH compared to high pH solutions, the relative rates were adjusted by giving the Mg^{++} titration process a first order dependence on the activity of H^+ and the Ca^{++} titration process only a ½ order dependence. While actually buffering reactions likely involved plagioclase and olivine, rather than $Ca(OH)_2$ and $Mg(OH)_2$, including these minerals as reactants would introduce a series of alumino-silicate clay minerals (e.g., nontronite) which may alter the final Ca and Mg observed in the simulations.

Results:

Results of the calculations are provided in graphical form in Figures 4 through 6. The figures represent resulting phase assemblages (Fig. 4), water chemistry (Fig. 5), and metal adsorption (Fig. 6) as a function of total pyrrhotite reacted. In each figure, case A represents reaction progress where acid titration is relatively rapid compared to cases B and C. A steady-state pH was reached in each situation that was controlled by the relative rates of acid generation and consumption: 6.1 for case A, 4.1 in case B, and 3.8 in case C.

$Fe(OH)_3$ precipitated in all three cases (note that other ferric iron phases were suppressed), gypsum only precipitated in cases A and B, and Tenorite only precipitated under the relatively high pH conditions prevailing in case A (see Fig. 4).

Solution chemistry reflected the mineral changes, with Cu concentrations remaining low throughout case A, owing to tenorite precipitation, and with noticeable deflections occurring in Ca and SO_4 concentrations for both cases A and B when gypsum began to precipitate. Interestingly, once gypsum precipitation occurred, Ca concentration decreased, while SO_4 concentration continued increasing.

Finally, the calculations indicated that relatively little adsorption of the trace metals occurs except possibly for Cu, and even for copper the fraction adsorbed only become high, when the concentration of Cu in solution was low already due to tenorite precipitation. For the other trace metals (Zn, Co, Cr), these calculations suggest that precipitation and adsorption are minor processes and thus, once dissolved, the metals released remain soluble.

Table 1. Selected Output from fluid speciation for sample having high dissolved solids from pile FL6.

```
Temperature =  25.0 C    Pressure =  1.013 bars
pH =  4.100              log fO2 =   -0.699
Eh =   0.9761 volts      pe =  16.5010
Ionic strength      =    0.080956
Dissolved solids    =        4101 mg/kg sol'n
```

Aqueous species	molality	mg/kg sol'n	act. coef.	log act.
SO4--	0.02046	1957.	0.3814	-2.1077
Mg++	0.01093	264.6	0.4695	-2.2897
MgSO4	0.006762	810.6	1.0000	-2.1699
Ca++	0.004201	167.7	0.4285	-2.7447
CaSO4	0.002935	397.9	1.0000	-2.5325
Na+	0.002345	53.70	0.7904	-2.7319
Cu++	0.002153	136.2	0.4285	-3.0351
Ni++	0.001784	104.3	0.4285	-3.1166
NiSO4	0.0007705	118.8	1.0000	-3.1132
SiO2(aq)	0.0003485	20.85	1.0211	-3.4488
O2(aq)	0.0002472	7.876	1.0211	-3.5980
Co++	0.0001275	7.486	0.4285	-4.2624
CO2(aq)	0.0001116	4.891	1.0000	-3.9524
H+	9.449e-005	0.09485	0.8406	-4.1000
NaSO4-	9.044e-005	10.72	0.7904	-4.1458
HSO4-	7.718e-005	7.460	0.7904	-4.2147
AlSO4+	7.370e-005	9.031	0.7904	-4.2347
Zn++	5.402e-005	3.517	0.4285	-4.6355
Al(SO4)2-	4.499e-005	9.817	0.7904	-4.4490
ZnSO4	4.234e-005	6.808	1.0000	-4.3732
Al+++	3.702e-005	0.9948	0.1972	-5.1366
CuOH+	3.214e-006	0.2578	0.7904	-5.5951
AlOH++	2.634e-006	0.1154	0.4058	-5.9711
CoHSO4+	9.651e-007	0.1499	0.7904	-6.1176
HCO3-	7.591e-007	0.04613	0.7967	-6.2184
CoSO4	4.532e-007	0.06996	1.0000	-6.3437
Al(OH)2+	1.154e-007	0.007008	0.7904	-7.0401
MgHCO3+	4.019e-008	0.003415	0.7904	-7.4980
CuSO4	3.956e-008	0.006288	1.0000	-7.4028
CaHCO3+	2.255e-008	0.002270	0.8050	-7.7411

(only species > 1e-8 molal listed)

Mineral saturation states (log Q/K > 0 indicates supersaturation)

	log Q/K		log Q/K
Co(FeO2)2	4.1297s/sat	Amrph^silica	-0.7352
Nontronit-Mg	3.2421s/sat	Gibbsite	-0.7969
Nontronit-Ca	3.1487s/sat	Bassanite	-1.2161
Nontronit-Na	2.6409s/sat	Beidellit-Mg	-1.2910
Hematite	0.9565s/sat	Beidellit-H	-1.3242
Quartz	0.5505s/sat	Beidellit-Ca	-1.3844
Tridymite	0.3847s/sat	CaSO4^1/2H2O(bet	-1.3847
Chalcedony	0.2793s/sat	Diaspore	-1.5883
Cristobalite	0.0000 sat	Beidellit-Na	-1.8922
Kaolinite	0.0000 sat	Co2SiO4	-2.2540
Goethite	0.0000 sat	Boehmite	-2.4327
Gypsum	-0.4091	Tenorite	-2.5077
Pyrophyllite	-0.5283	Epsomite	-2.5848
Anhydrite	-0.5872	Hexahydrite	-2.8215

(only minerals with log Q/K > -3 listed)

Gases	fugacity	log fug.
O2(g)	0.2000	-0.699
Steam	0.03131	-1.504
CO2(g)	0.003162	-2.500
H2(g)	6.282e-042	-41.202
H2S(g)	3.335e-134	-133.477
CH4(g)	3.888e-145	-144.410
S2(g)	6.517e-211	-210.186

Elemental composition		In fluid		Sorbed	
	total moles	moles	mg/kg	moles	mg/kg
Aluminum	0.007906	0.0001585	4.258		
Calcium	0.007136	0.007136	284.8		
Carbon	0.0001124	0.0001124	1.345		
Cobalt	0.0001290	0.0001290	7.569		
Copper	0.002156	0.002156	136.4		
Iron	0.01125	1.246e-009	6.931e-005		
Magnesium	0.01769	0.01769	428.2		
Nickel	0.002555	0.002555	149.4		
Silicon	0.02474	0.0003485	9.746		
Sodium	0.002436	0.002436	55.77		
Sulfur	0.03130	0.03130	999.4		
Zinc	9.636e-005	9.636e-005	6.274		

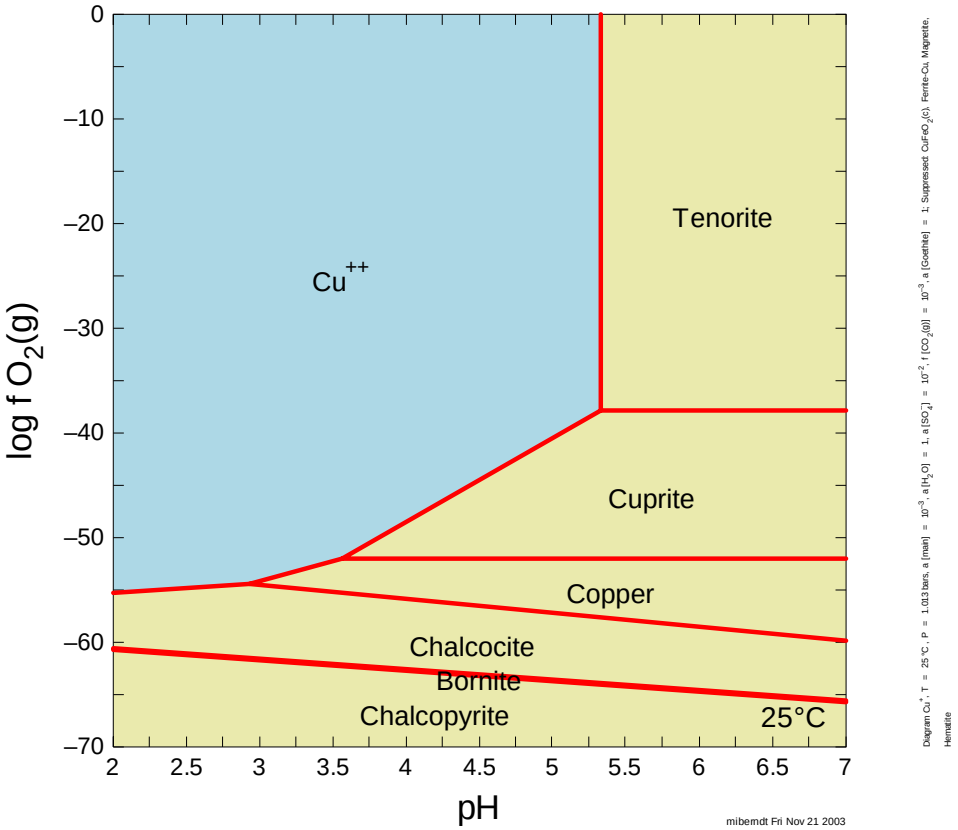


Figure 1: Phase relations for copper in oxidizing waste piles. Copper is very soluble at low pH and under oxidizing conditions, but may precipitate in a variety of forms if/when pH increases and or oxygen is consumed within the waste pile. Conditions in the waste pile were generally oxidizing and acidic and so copper was generally highly mobile, except in occasional microenvironments within the pile where other phases were sometimes found (copper oxides, copper, covellite, and malachite).

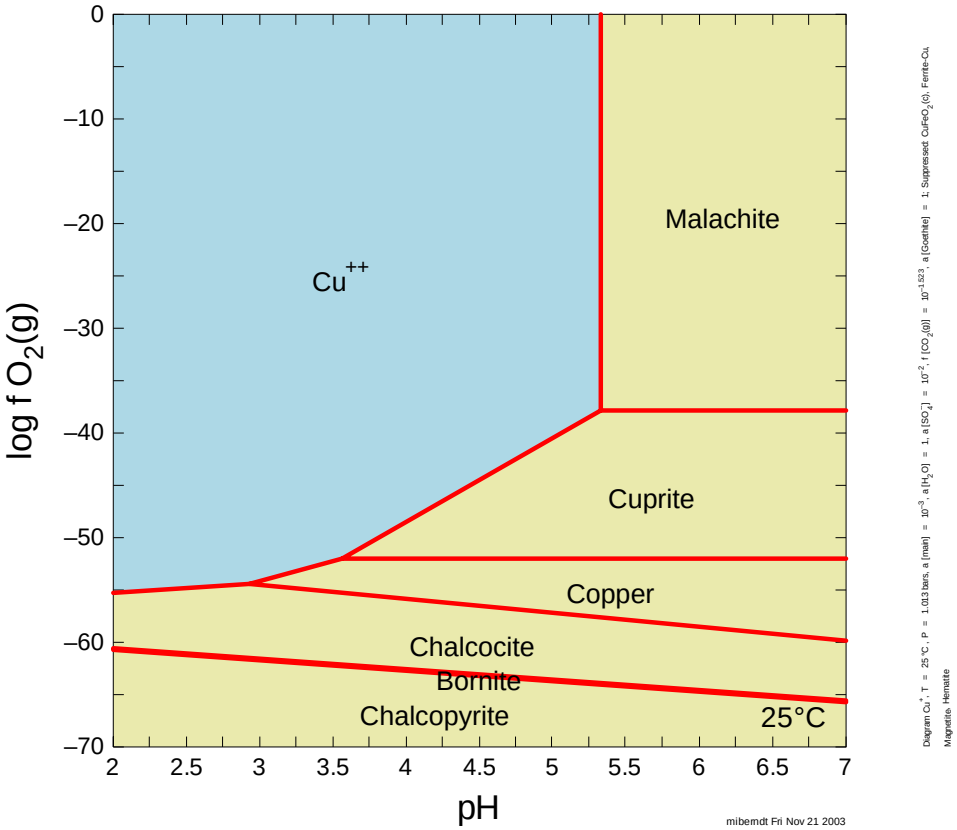


Fig. 2. Same as the previous plot except for higher f CO2(g) conditions where malachite replaces the tenorite stability field shown in the previous diagram.

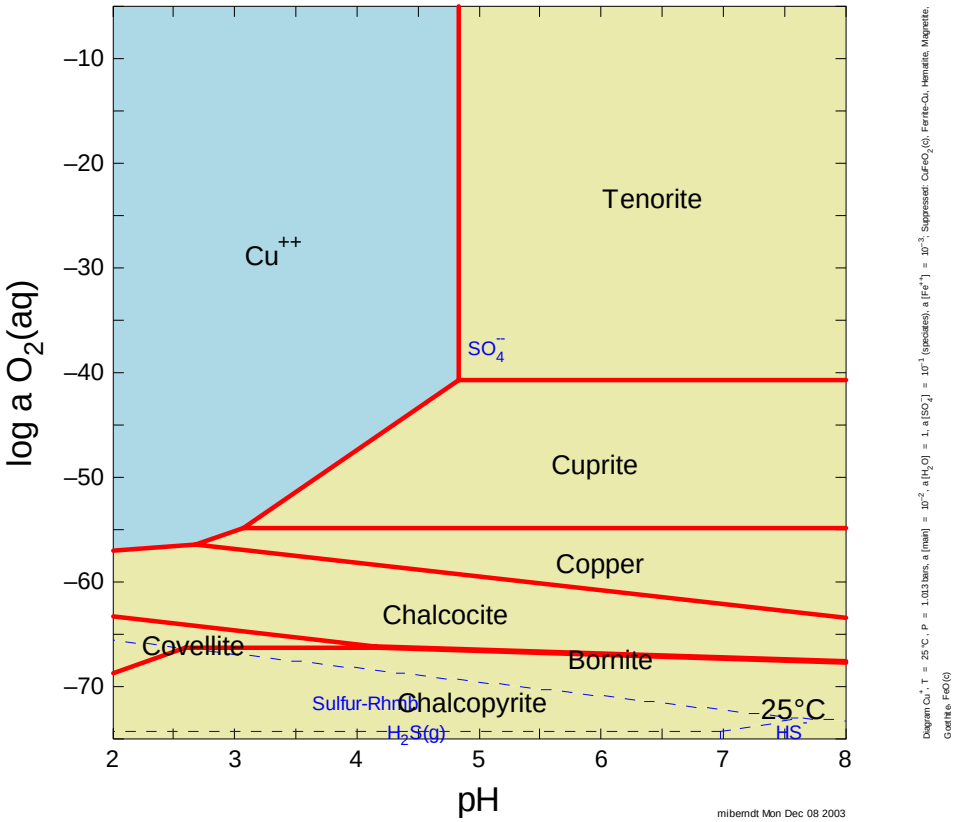
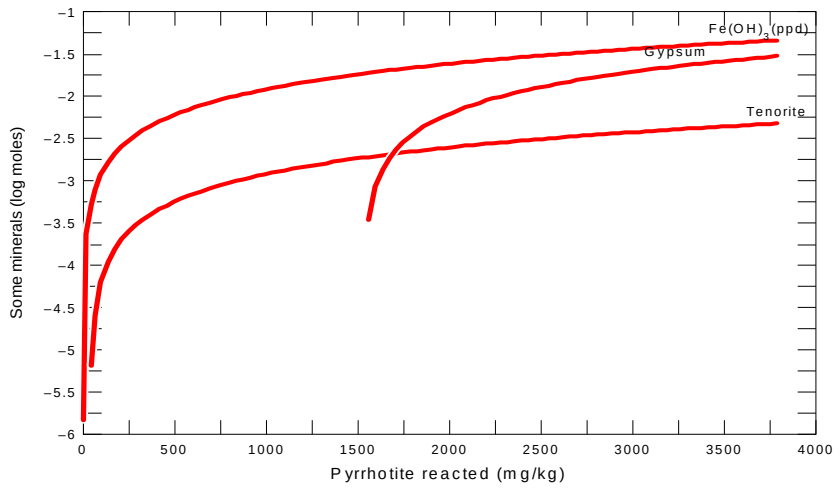


Fig. 3. This figure shows the copper phases that might be expected as a function of pH and O2(aq) activity. Saturation with respect to elemental sulfur is shown by the dashed line. The presence of elemental sulfur in some portions of the pile suggest the mineral is likely a metastable oxidation product of sulfide oxidation.



Fig, 4a
pH 6.1
Ca>Mg

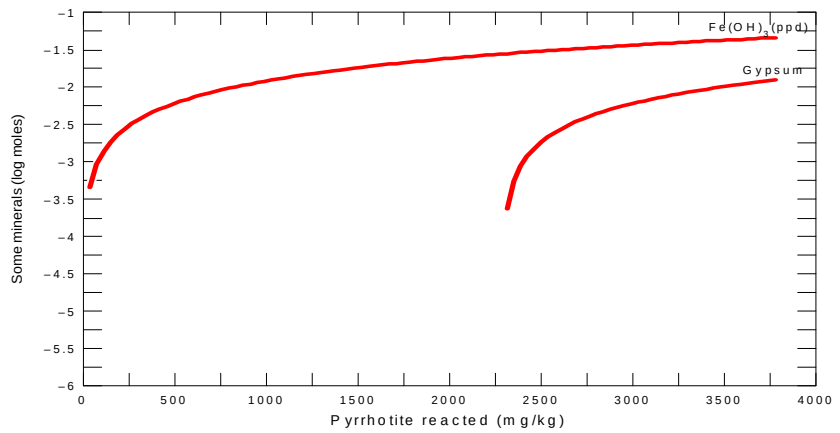


Fig. 4b
pH 4.6
Ca>Mg

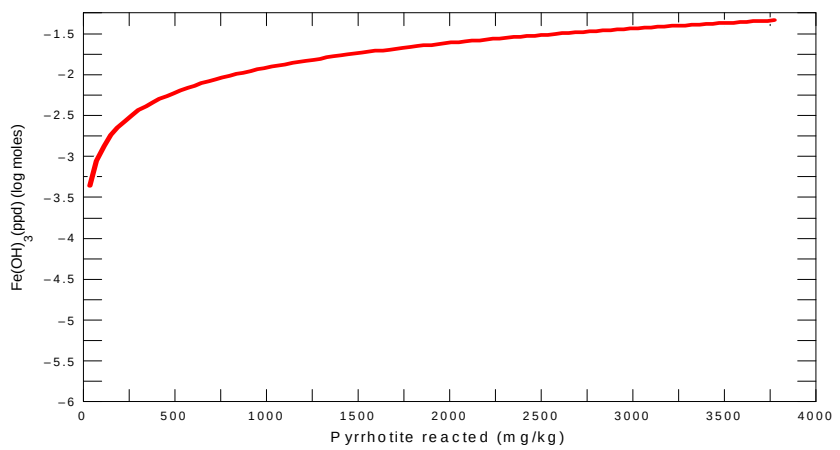
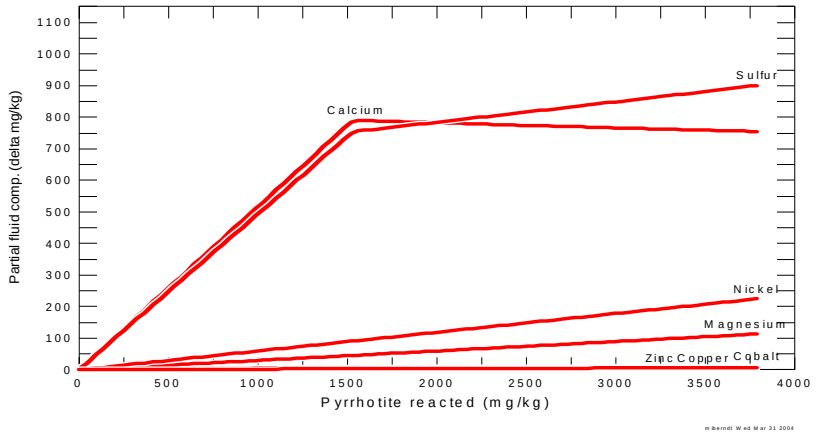


Fig. 4c
pH 3.8
Mg>Ca

Fig 4. Masses (per kg water) of minerals formed in the three simulations performed in this study. Fe(OH)3 formed throughout the process in each case, but gypsum forms only when Ca minerals buffer the solution pH. Tenorite (CuO) forms when buffering keeps pH above approximately 5.



Fig, 5a
pH 6.1
Ca>Mg

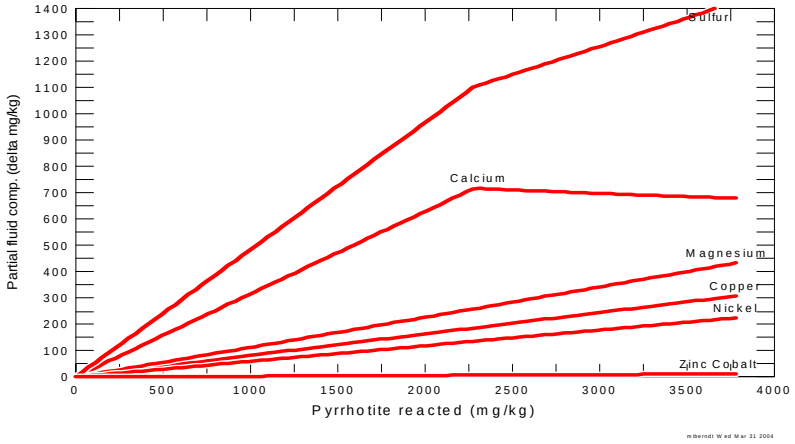


Fig. 5b
pH 4.6
Ca>Mg

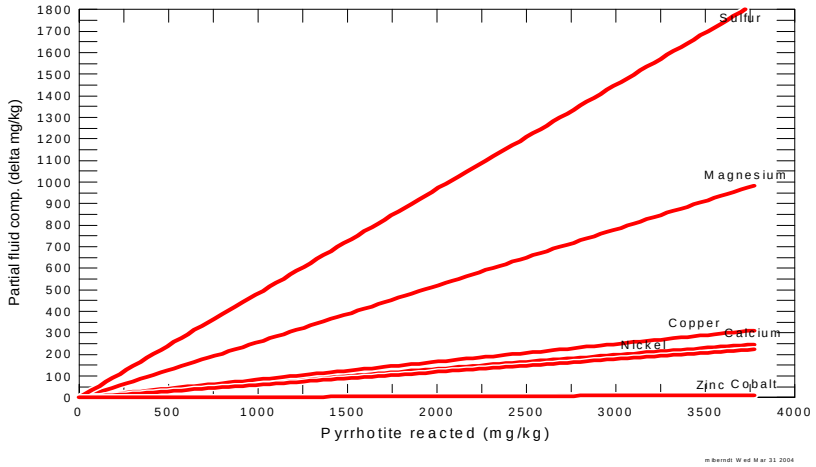
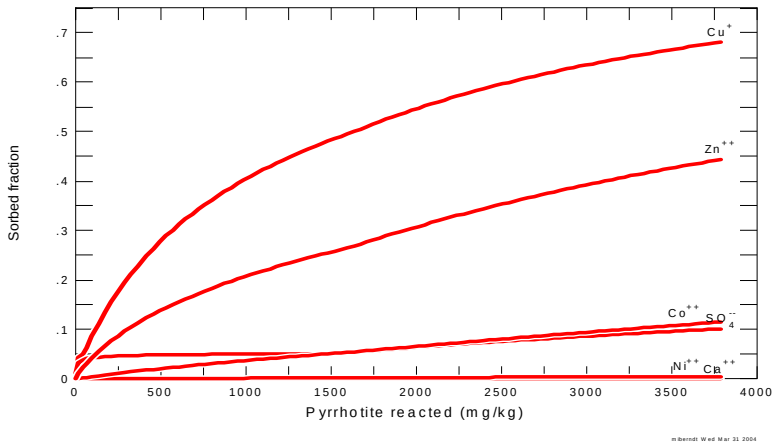


Fig. 5c
pH 3.8
Mg>Ca

Fig 5. Solution chemistry evolution during the three simulations.



Fig, 6a
pH 6.1
Ca>Mg

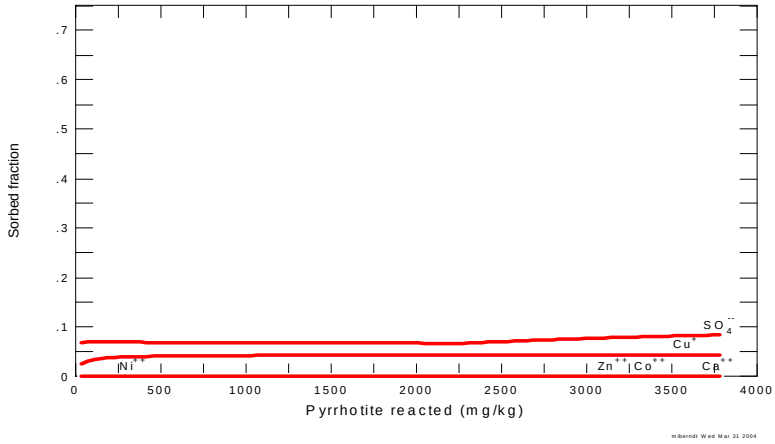


Fig. 6b
pH 4.6
Ca>Mg

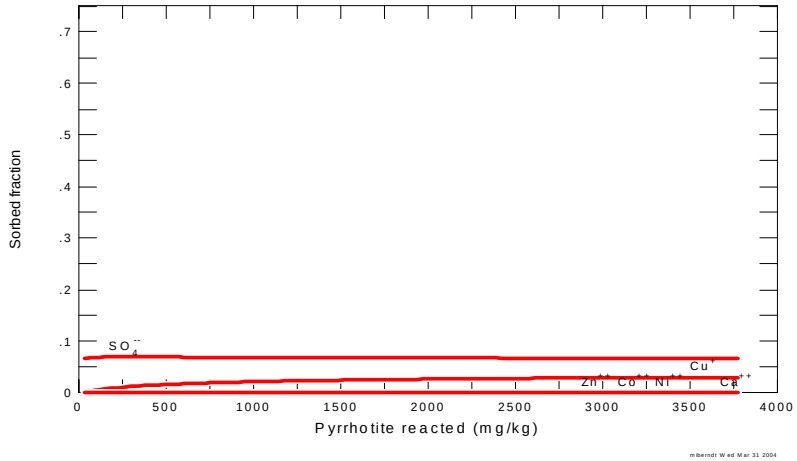


Fig. 6c
pH 3.8
Mg>Ca

Fig 6. Fraction of metals in solution that was adsorbed to $Fe(OH)_3$ in the simulations.

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APPENDIX 7

CALCULATIONS

Attachment A7.1.	Calculation of sulfate released to FL6 drainage from pyrrhotite oxidation
Attachment A7.2.	Calculation of sulfate release from pyrrhotite from particles finer than 0.5 mm in FL6 bulk sample
Attachment A7.3.	Calculate Length of Oxygen Diffusion into AMAX piles
Attachment A7.4.	Calculations of pyrrhotite depletion from fines

Attachment A7.1. Calculation of sulfate released to FL6 drainage from pyrrhotite oxidation

Observed release

Observed sulfate mass release from FL6 = 36,235 moles
Observed copper release = 1695 moles
Observed nickel release = 2756 moles

Estimated sulfate release from different minerals

Estimated sulfate release associated with copper sulfide mineral oxidation

Assuming copper was equally present in chalcopyrite and cubanite and these minerals oxidized at the same rate.

Sulfate released with copper = 2.5 x 1695 = 4238 moles

Estimated sulfate release associated with copper and nickel sulfide mineral oxidation

Assume nickel was released from pentlandite ($Ni_{4.5}$ $Fe_{4.5}S_8$]

Sulfate release with nickel = (8/4.5) x 2756 = 4900 moles

Estimated sulfate release from pyrrhotite = 36,235 - (4238+4900) = 27,097

Attachment A7.2. Calculation of sulfate release from pyrrhotite from particles finer than 0.5 mm in FL6 bulk sample

Assume all pyrrhotite in particles finer than 0.5 mm is oxidized

$M(SO_4) = (P_{0.5}/100)\ M_{pile}\ F_S\ (S_{bulk}/100)\ f_{Spo}/32.06$

where $M(SO_4)$ = moles of sulfate released from pyrrhotite present in particles finer than 0.5 mm

$P_{0.5}$ = percentage of rock finer than 0.5 mm (3 - 8%)

M_{pile} = mass of pile in grams (1300×10^6)

F_S = ratio of sulfur content in particles finer than 0.5 mm to bulk sulfur content (2-2.5)

S_{bulk} = bulk sulfur content of the pile, percent (0.79)

f_{Spo} = fraction of total sulfur present with pyrrhotite (0.34-0.51)

The sources of data used in calculations are described in the text.

Case 1: $P_{0.5} = 3$, $F_S = 2$, $f_{Spo} = 0.34$

$M(SO_4)$ = $[(3/100)\ 1300 \times 10^6\ 2\ (0.79/100)\ 0.34]/32.06$
= 6534 moles

Case 2: $P_{0.5} = 3$, $F_S = 2$, $f_{Spo} = 0.51$

$M(SO_4)$ = $[(3/100)\ 1300 \times 10^6\ 2\ (0.79/100)\ 0.51]/32.06$
= 9801 moles

Case 3: $P_{0.5} = 3$, $F_S = 2.5$, $f_{Spo} = 0.34$

$M(SO_4)$ = $[(3/100)\ 1300 \times 10^6\ 2.5\ (0.79/100)\ 0.34]/32.06$
= 8168 moles

Case 4: $P_{0.5} = 3$, $F_S = 2.5$, $f_{Spo} = 0.51$

$M(SO_4)$ = $[(3/100)\ 1300 \times 10^6\ 2.5\ (0.79/100)\ 0.51]/32.06$
= 12,252 moles

Case 5: $P_{0.5} = 8$, $F_S = 2$, $f_{Spo} = 0.34$

$M(SO_4)$ = $[(8/100)\ 1300 \times 10^6\ 2\ (0.79/100)\ 0.34]/32.06$
= 17,426 moles

Case 6: $P_{0.5} = 8$, $F_S = 2$, $f_{Spo} = 0.51$

$M(SO_4)$ = $[(8/100)\ 1300 \times 10^6\ 2\ (0.79/100)\ 0.51]/32.06$
= 26,139 moles

Case 7: $P_{0.5} = 8$, $F_S = 2.5$, $f_{Spo} = 0.34$

$M(SO_4)$ = $[(8/100)\ 1300 \times 10^6\ 2.5\ (0.79/100)\ 0.34]/32.06$
= 21,782 moles

Case 8: $P_{0.5} = 8$, $F_S = 2.5$, $f_{Spo} = 0.51$

$M(SO_4)$ = $[(8/100)\ 1300 \times 10^6\ 2.5\ (0.79/100)\ 0.51]/32.06$
= 32,674 moles

Case 9: Average of Cases 1-8: (6534 + 9801 + 8168 + 17,426 + 26,139 + 21,782 + 32,674)/8 = 15,316 moles

Fraction of pyrrhotite sulfate released to drainage from rock finer than 0.5 mm

6534/27,097 = 0.24
11,980/27,097 = 0.44
17,426/27,097 = 0.64

Attachment A7.3. Calculate Length of Oxygen Diffusion into AMAX piles (page 1 of 2).

From Sulfide Solutions short course notes (2001)

$X_1 = (2C_0D_{ox}/S_0)^{0.5}$

X_1 = depth of O_2 penetration, m

C_0 = 0.265 kg m^{-2} (from short course notes)

$D_{ox} = 1 \times 10^{-8}$ to 10^{-6} m^{-2} s^{-1} (short course notes), 4×10^{-6} used in short course

S_0 = kgO_2 m^{-3} s^{-1}, intrinsic oxidation rate (Sudbury 97?), use 5×10^{-8} to 5×10^{-6}

Sample calculations

If $D_{ox} = 1 \times 10^{-8}$ $m^{-2}s^{-1}$ and $S_0 = 5 \times 10^{-8}$ kg m^{-2} s^{-1}

$X_1 = (2C_0D/S_0)^{0.5} = [2(0.265 \times 1 \times 10^{-8})/5 \times 10^{-8}]^{0.5} = 3.3 \times 10^{-1}$ m

If $D_{ox} = 1 \times 10^{-8}$ $m^{-2}s^{-1}$ and $S_0 = 5 \times 10^{-6}$ kg m^{-2} s^{-1}

$X_1 = (2C_0D/S_0)^{0.5} = [2(0.265 \times 1 \times 10^{-8})/5 \times 10^{-6}]^{0.5} = 3.3 \times 10^{-2}$ m

If $D_{ox} = 4 \times 10^{-6}$ $m^{-2}s^{-1}$ and $S_0 = 5 \times 10^{-8}$ kg m^{-2} s^{-1}

$X_1 = (2C_0D/S_0)^{0.5} = [2(0.265 \times 5 \times 10^{-6})/5 \times 10^{-8}]^{0.5} = 7.3$ m

If $D_{ox} = 4 \times 10^{-6}$ $m^{-2}s^{-1}$ and $S_0 = 5 \times 10^{-6}$ kg m^{-2} s^{-1}

$X_1 = (2C_0D/S_0)^{0.5} = [2(0.265 \times 5 \times 10^{-6})/5 \times 10^{-6}]^{0.5} = 7.3 \times 10^{-1}$ m

DETERMINE PYRRHOTITE SURFACE AREA FOR TEST PILES

Specific surface area of rock from FL4, considering particle size distribution = 14 cm^2/g = 0.0014 m^2 g^{-1} = 1400 m^2/t (value is from p. size dist. in 1993 report & must be checked).

At 1.8 t m^{-3} (based on 40% voids), surface area per unit volume is 2500 m^2 m^{-3}

Assume pyrrhotite contributes 1 to 5 percent of the surface area or 25 to 125 m^2 t^{-1}

FROM LABORATORY EXPERIMENT: $S_0 = 1.4 \times 10^{-9}$ to 7.2×10^{-9} kg O_2 m^{-3} s^{-1}

Rates from laboratory experiments

Lab rate for pyrrhotite = 8×10^{-10} mol m^{-2} s^{-1}.

Oxygen consumption is 2.25 times this rate or 1.8×10^{-9} mol m^{-2} s^{-1}.

At 0.032 kg/mol O_2, this is equivalent to 5.8×10^{-11} kg O_2 m^{-2} pyrrhotite s^{-1}

Intrinsic oxidation rate for 25 and 125 m^2 m^{-3}

At 25 m^2 m^{-3}: $S_0 = 25 \times 1.4 \times 10^{-9}$ kg O_2 m^{-2} s^{-1} $= 1.4 \times 10^{-9}$ kg O_2 m^{-3} s^{-1}

At 125 m^2 m^{-3}: $S_0 = 125 \times 1.4 \times 10^{-9}$ kg O_2 m^{-2} s^{-1} $= 7.2 \times 10^{-9}$ kg O_2 m^{-3} s^{-1}

Attachment A7.3. Calculate Length of Oxygen Diffusion into AMAX piles (page 2 of 2).

FROM TEST PILES: $S_0 = 1.5 \times 10^{-8}$ kg O_2 m^{-3} s^{-1}

For FL6 the average annual SO_4 release was approximately 10 mmol t^{-1} d^{-1}.
This yields an intrinsic oxidation rate of
$S_0 = 10$ mmol t^{-1} d^{-1} x (2.25 mol O_2/mol SO_4) x (3.2×10^{-5} kg/mmol O_2)
x (1.8 t/m^3) x (d/[24 x 3600 s]) = 1.5×10^{-8} kg O_2 $m^{-3}s^{-1}$

NOTE: This doesn't seem correct. This suggests that the field rates were greater than the lab rates. There may be a problem with the estimated pyrrhotite surface area used to extrapolate lab rates to the field. The field rate seems to be strong because annual rates of sulfate release (mmol SO4 t^{-1} d^{-1}) were determined for all piles every year. These values are fairly consistent.

TIME TO DEPLETE INITIAL OXYGEN FROM TEST PILE

For 1 m^3 the oxygen content is, assuming 40% voids, 0.4 x 0.265 kg O_2 m^{-3} = 1.06×10^{-1} kg.
1.06×10^{-1} kg/(1.5×10^{-8} kg O_2 m^{-3} s^{-1}) = 7.07×10^6 s = 82 d

Attachment A7.4. Calculation of pyrrhotite depletion from fines.

CALCULATED PYRRHOTITE DEPLETION

	0.8% Bulk	6.1.2	6.1.4	6.1.5	6.1.7	6.2.2	6.2.4	6.2.7	6.2.10	6.3.1	6.3.3	6.3.8	6.3.16	6.4.2	6.4.4	6.4.5	6.4.7
1.5 - 6 in	69.92																
0.5 - 1.5 in	68.60																
2mm - 0.5"	18.27	14.45	-33.65	38.90	-53.397765	55.69	68.87	16.95	26.1016	16.32	84.64	122.76	82.37	43.71	74.00	95.16	50.27
0.5 - 2.0	50.79	73.15	-8.16	54.96	-12.584515	62.29	90.18	63.59	31.24	49.11	73.44	147.47	69.94	42.47	85.13	91.08	45.74
0.177- 0.5	74.24	92.14	11.56	103.09	44.0754805	83.72	103.75	100.76	67.35	65.77	106.21	174.44	84.82	132.44	78.42	94.64	81.61
0.149- 0.177	79.91	119.36	67.78	155.85	70.2125933	103.05	106.28	137.98	109.51	46.53	23.07	153.00	84.43	48.53	80.72	102.70	82.40
0.105-0.149	105.77	130.97	125.04	189.44	84.3223915	108.63	95.67	122.76	113.14	60.18	124.88	188.46	188.09	88.58	103.26	112.52	88.67
0.075-0.105	97.36	146.05	558.44	202.10	139.459916	143.18	81.20	157.07	163.92	51.19	172.66	222.68	108.60	141.86	142.46	120.88	140.93
0.053-0.075	102.71	155.80	476.57	254.17	180.540773	167.96	78.37	156.68	189.76	3.14	161.50	260.90	132.71	146.69	167.49	152.06	124.05
< .053mm	108.47	181.90	474.30	254.85	198.676283	189.55	93.85	165.01	208.83	129.76	344.14	280.79	142.82	170.45	162.09	164.41	127.23

PYRRHOTITE DEPLETION MODIFIED SUCH THAT VALUES <0 ARE SET =0 AND VALUES >100 ARE SET = 100

	0.8% Bulk	6.1.2	6.1.4	6.1.5	6.1.7	6.2.2	6.2.4	6.2.7	6.2.10	6.3.1	6.3.3	6.3.8	6.3.16	6.4.2	6.4.4	6.4.5	6.4.7	AVE
1.5 - 6 in	69.92																	
0.5 - 1.5 in	68.60																	
2mm - 0.5"	18.27	14.45	0.00	38.90	0	55.69	68.87	16.95	26.10	16.32	84.64	100.00	82.37	43.71	74.00	95.16	50.27	47.96
0.5 - 2.0	50.79	73.15	0.00	54.96	0	62.29	90.18	63.59	31.24	49.11	73.44	100.00	69.94	42.47	85.13	91.08	45.74	58.27
0.177- 0.5	74.24	92.14	11.56	100.00	44.08	83.72	100.00	100.00	67.35	65.77	100.00	100.00	84.82	100.00	78.42	94.64	81.61	81.51
0.149- 0.177	79.91	100.00	67.78	100.00	70.21	100.00	100.00	100.00	100.00	46.53	23.07	100.00	84.43	48.53	80.72	100.00	82.40	81.48
0.105-0.149	100.00	100.00	100.00	100.00	84.32	100.00	95.67	100.00	100.00	60.18	100.00	100.00	100.00	88.58	100.00	100.00	88.67	94.84
0.075-0.105	97.36	100.00	100.00	100.00	100	100.00	81.20	100.00	100.00	51.19	100.00	100.00	100.00	100.00	100.00	100.00	100.00	95.77
0.053-0.075	100.00	100.00	100.00	100.00	100	100.00	78.37	100.00	100.00	3.14	100.00	100.00	100.00	100.00	100.00	100.00	100.00	92.59
< .053mm	100.00	100.00	100.00	100.00	100	100.00	93.85	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	99.62

Attachment A7.4. Calculation of pyrrhotite depletion from fines.

	0.8% S Bulk Sample		
	% depl	S2-(po)f	S2-(po)init
1.5 - 6 in	69.92	0.08	0.27
0.5 - 1.5 in	68.60	0.10	0.30
2mm - 0.5"	18.27	0.33	0.40
0.5 - 2.0	50.79	0.25	0.50
0.177- 0.5	74.24	0.13	0.52
0.149- 0.177	79.91	0.10	0.52
0.105-0.149	105.77	-0.03	0.49
0.075-0.105	97.36	0.01	0.53
0.053-0.075	102.71	-0.02	0.64
< .053mm	108.47	-0.06	0.73

	% depl	S2-(po)f	S2-(po)init	% depl	S2-(po)f	S2-(po)init	% depl	S2-(po)f	S2-(po)init	% depl	S2-(po)f	S2-(po)init
	6.1.2			6.1.4			6.1.5			6.1.7		
2-6.4mm	14.45	0.43	0.50	-33.65	0.82	0.61	38.90	0.22	0.37	-53.40	0.90	0.58
0.5 - 2.0	73.15	0.18	0.69	-8.16	0.69	0.64	54.96	0.23	0.50	-12.58	0.74	0.66
0.177- 0.5	92.14	0.05	0.62	11.56	0.48	0.54	103.09	-0.01	0.38	44.08	0.40	0.71
0.149- 0.177	119.36	-0.10	0.52	67.78	0.17	0.52	155.85	-0.13	0.24	70.21	0.18	0.62
0.105-0.149	130.97	-0.14	0.44	125.04	-0.14	0.56	189.44	-0.18	0.20	84.32	0.07	0.47
0.075-0.105	146.05	-0.21	0.45	558.44	-2.31	0.50	202.10	-0.15	0.15	139.46	-0.19	0.47
0.053-0.075	155.80	-0.23	0.41	476.57	-2.00	0.53	254.17	-0.82	0.53	180.54	-0.31	0.39
< .053mm	181.90	-0.34	0.42	474.30	-1.97	0.53	254.85	-0.30	0.19	198.68	-0.34	0.34
	6.2.2			6.2.4			6.2.7			6.2.10		
2-6.4mm	55.69	0.17	0.38	68.87	0.09	0.28	16.95	0.30	0.36	26.10	0.30	0.40
0.5 - 2.0	62.29	0.21	0.55	90.18	0.04	0.43	63.59	0.18	0.48	31.24	0.37	0.55
0.177- 0.5	83.72	0.09	0.57	103.75	-0.02	0.59	100.76	0.00	0.57	67.35	0.17	0.53
0.149- 0.177	103.05	-0.02	0.59	106.28	-0.04	0.71	137.98	-0.24	0.64	109.51	-0.05	0.51
0.105-0.149	108.63	-0.05	0.60	95.67	0.03	0.80	122.76	-0.13	0.58	113.14	-0.06	0.47
0.075-0.105	143.18	-0.28	0.64	81.20	0.18	0.94	157.07	-0.43	0.75	163.92	-0.36	0.57
0.053-0.075	167.96	-0.45	0.66	78.37	0.23	1.04	156.68	-0.38	0.66	189.76	-0.59	0.66
< .053mm	189.55	-0.66	0.73	93.85	0.07	1.08	165.01	-0.44	0.68	208.83	-0.70	0.64
	6.3.1			6.3.3			6.3.8			6.3.16		
2-6.4mm	16.32	0.39	0.46	84.64	-0.01	0.43	122.76	-0.34	0.53	82.37	0.10	0.59
0.5 - 2.0	49.11	0.28	0.55	73.44	0.00	0.40	147.47	-0.18	0.57	69.94	0.15	0.48
0.177- 0.5	65.77	0.21	0.63	106.21	-0.93	0.48	174.44	-0.11	0.63	84.82	0.09	0.56
0.149- 0.177	46.53	0.37	0.70	23.07	0.35	0.22	153.00	-0.07	0.58	84.43	0.09	0.57
0.105-0.149	60.18	0.28	0.70	124.88	-0.96	0.51	188.46	-0.12	0.65	188.09	-0.54	0.61
0.075-0.105	51.19	0.38	0.78	172.66	-0.42	0.61	222.68	-0.20	0.70	108.60	-0.06	0.65
0.053-0.075	3.14	0.45	0.46	161.50	-0.52	0.59	260.90	-0.44	0.75	132.71	-0.23	0.71
< .053mm	129.76	-0.26	0.88	344.14	-0.88	0.87	280.79	-0.62	0.78	142.82	-0.29	0.69
	6.4.2			6.4.4			6.4.5			6.4.7		
2-6.4mm	43.71	0.15	0.27	74.00	0.07	0.28	95.16	0.02	0.46	50.27	0.17	0.34
0.5 - 2.0	42.47	0.23	0.40	85.13	0.06	0.38	91.08	0.06	0.67	45.74	0.33	0.61
0.177- 0.5	132.44	-0.23	0.71	78.42	0.08	0.38	94.64	0.04	0.73	81.61	0.12	0.67
0.149- 0.177	48.53	0.15	0.29	80.72	0.07	0.36	102.70	-0.02	0.71	82.40	0.10	0.57
0.105-0.149	88.58	0.03	0.30	103.26	-0.01	0.39	112.52	-0.09	0.70	88.67	0.06	0.55
0.075-0.105	141.86	-0.14	0.32	142.46	-0.18	0.42	120.88	-0.14	0.69	140.93	-0.31	0.75
0.053-0.075	146.69	-0.17	0.36	167.49	-0.25	0.36	152.06	-0.40	0.77	124.05	-0.17	0.70
< .053mm	170.45	-0.28	0.40	162.09	-0.25	0.40	164.41	-0.61	0.94	127.23	-0.21	0.77

Attachment A7.4. Calculation of pyrrhotite depletion from fines.

CALCULATED INITIAL SULFIDE WITH PYRRHOTITE

	0.8% Bulk	6.1.2	6.1.4	6.1.5	6.1.7	6.2.2	6.2.4	6.2.7	6.2.10	6.3.1	6.3.3	6.3.8	6.3.16	6.4.2	6.4.4	6.4.5	6.4.7	AVE
1.5 - 6 in	0.27																	
0.5 - 1.5 in	0.30																	
2mm - 0.5"	0.40	0.50	0.61	0.37	0.58	0.38	0.28	0.36	0.40	0.46	0.43	0.53	0.59	0.27	0.28	0.46	0.34	0.43
0.5 - 2.0	0.50	0.69	0.64	0.50	0.66	0.55	0.43	0.48	0.55	0.55	0.40	0.57	0.48	0.40	0.38	0.67	0.61	0.53
0.177- 0.5	0.52	0.62	0.54	0.38	0.71	0.57	0.59	0.57	0.53	0.63	0.48	0.63	0.56	0.71	0.38	0.73	0.67	0.58
0.149- 0.177	0.52	0.52	0.52	0.24	0.62	0.59	0.71	0.64	0.51	0.70	0.22	0.58	0.57	0.29	0.36	0.71	0.57	0.52
0.105-0.149	0.49	0.44	0.56	0.20	0.47	0.60	0.80	0.58	0.47	0.70	0.51	0.65	0.61	0.30	0.39	0.70	0.55	0.53
0.075-0.105	0.53	0.45	0.50	0.15	0.47	0.64	0.94	0.75	0.57	0.78	0.61	0.70	0.65	0.32	0.42	0.69	0.75	0.59
0.053-0.075	0.64	0.41	0.53	0.53	0.39	0.66	1.04	0.66	0.66	0.46	0.59	0.75	0.71	0.36	0.36	0.77	0.70	0.60
< .053mm	0.73	0.42	0.53	0.19	0.34	0.73	1.08	0.68	0.64	0.88	0.87	0.78	0.69	0.40	0.40	0.94	0.77	0.65

	AVE VALUES FOR FL6 (1300 t)			S release
	S(po)init	% depl	%mass	Moles
1.5 - 6 in	0.27	69.92	45.4	34752
0.5 - 1.5 in	0.3	68.60	32.5	27123
2 - 6.4mm	0.43	47.96	6.1	5066
0.5 - 2.0	0.53	58.27	2.2	2779
0.177- 0.5	0.58	81.51	1.4	2687
0.149- 0.177	0.52	81.48	0.2	345
0.105-0.149	0.53	94.84	0.2	410
0.075-0.105	0.59	95.77	0.3	684
0.053-0.075	0.60	92.59	0.3	675
< .053mm	0.65	99.62	0.7	1825
				14470

S release for d < 6.4 mm is 14,470 moles, drainage quality indicates 36,235 moles; 14,470/36,235 = 0.40
11.4% of mass in this size fraction

Drainage quality data indicate Cu and Ni releases of 1695 and 2756.
Assuming 2.5 S release for each Cu and 1.78 S release for each Ni yields 4240 and 4910 moles S, a total of 9140 moles S
This would provide a total of 14,470 + 9140 = 23,610 moles S

Attachment 12

Lapakko K (1994) Comparison of Duluth Complex Rock Dissolution in the Laboratory and Field. In: International Land Reclamation and Mine Drainage Conference and the 3rd International Conference on the Abatement of Acidic Drainage. Pittsburgh, PA, pp 419–428

COMPARISON OF DULUTH COMPLEX ROCK DISSOLUTION IN THE LABORATORY AND FIELD[1]

Kim A. Lapakko[2]

Abstract: The quality of drainage and the release of sulfate, calcium, and magnesium from five field test piles of Duluth Complex rock with sulfur contents of 0.63% (three piles), 0.79%, and 1.41% were determined for periods of 12 to 14 yr. The pH of drainage from the piles decreased over time and with increasing solid-phase sulfur content. The ultimate drainage pH values for the 0.63% S piles ranged from 4.8 to 5, similar to values observed in the laboratory. Ultimate drainage pH values for sulfur contents of 0.79% (pH 4) and 1.41% (pH 3.5) were about one unit lower than the corresponding laboratory values for these sulfur contents. Over the entire period of record, average rates of sulfate release (2.1 to 10.5 mmol/mt/d) and magnesium release (0.55 to 3.7 mmol/mt/d) increased as the solid-phase sulfur content of the pile increased, while those of calcium release (1.2 to 2.4 mmol/mt/d) were relatively constant with respect to solid-phase sulfur content. Annual release rates for sulfate and calcium were relatively constant over time, while those for magnesium tended to increase over time. Annual release rates for all three parameters increased with increased annual drainage volume, a variable not examined in the laboratory. Release rates in the field were typically 10% to 30% (and in one case, nearly 40%) of those observed in the laboratory. Empirical neutralization potentials in the field decreased as solid-phase sulfur content decreased and were typically 10% to 50% of the corresponding average laboratory values. The empirical neutralization potentials to maintain drainage pH above 6.0 were approximately 0.5% to 8% of ABA NP values for Duluth Complex rock.

Additional key words: acid mine drainage, sulfide mineral oxidation, pyrrhotite, field test, rates.

Introduction

In the mid-1970's AMAX Exploration Inc. was evaluating the potential for developing copper-nickel resources of the Duluth Complex in northeastern Minnesota. At this time little information existed on the potential drainage problems associated with copper-nickel development in Minnesota. The State and AMAX agreed to commence field leaching tests on Duluth Complex rock. AMAX began excavation of a shaft in 1975 and in 1977 constructed six test stockpiles containing lean ore material. Water quality samples were collected periodically from each pile in 1977, and the data were compiled by AMAX Environmental Services Inc. (1978). The test pile studies were conducted from 1978 through the present by the Minnesota Department of Natural Resources (MNDNR), Division of Minerals (Eger and Lapakko 1981, 1985). As part of a cooperative research project with the U. S. Bureau of Mines, the compilation and analysis of data from the test piles were updated through 1991 to aid in development of a dissolution model by the Salt Lake City Research Center. This paper presents data on the quality and quantity of drainage from five of the test piles and the rates of mineral dissolution observed from 1978 through 1991. The period of record for pile 4 was only 5 yr, and these data are not presented. The data are compared with those previously presented on laboratory studies on the dissolution of Duluth Complex rock.

Objectives

The objectives of this paper on the quality of drainage from Duluth Complex rock field test piles are to

1. Describe the variation of drainage quality with respect to solid-phase sulfur content, time, and flow.
2. Determine the rates of sulfate, calcium, and magnesium release.
3. Describe the variation of drainage quality and mass release with respect to sulfur content, time, and flow.

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4. Determine the empirical acid neutralization potentials of the Duluth Complex rock samples examined.
5. Compare field results with those generated in the laboratory.

Methods

Climate

Annual precipitation in the study area averages 72.1 cm (Hickok and Associates 1977), and the average watershed runoff is 26.2 cm/yr (Siegel and Ericson 1980). Temperatures are extreme, averaging -14° C in January and 19.1° C in July, with an annual mean of 3.6° C; the ground is covered with snow for an average of 140 days per year (Hickok and Associates 1977).

Test Pile Construction

Materials. Six test stockpiles containing 820 to 1300 metric tons (mt) of low-grade copper-nickel material were constructed in 1977. The piles were approximately 4 m high and 15 m by 25 m at the base. Stockpiles 1 to 4 were constructed of rock mined during the shaft-sinking operation, while piles 5 and 6 contain material from underground drifts. The chemical compositional ranges for the piles are 0.63% to 1.41% S, 0.30% to 0.35% Cu, and 0.083% to 0.085% Ni (table 1). A grab sample was taken from test pile 1 for more extensive chemical and mineralogical analyses (Eger and Lapakko 1981). The major sulfide minerals identified and their volume percentages were pyrrhotite (0.844%), chalcopyrite-cubanite (0.769%), and pentlandite (0.037%). The major silicate minerals and their approximate volume percentages were plagioclase (59%), clinopyroxene (11%), olivine (11%), orthopyroxene (3.7%), and monocrystalline amphibole (3.6%).

Approximately 40% of test pile 4 was removed in 1982 for use in another study of stockpile reclamation techniques (Eger et al. 1984, Lapakko et al. 1986). At this time a representative sample weighing over 3 mt was collected for analysis of particle size distribution, specific surface area, chemistry, and mineralogy. The methods and detailed results of this characterization are presented in Lapakko et al. (1986). The particle size analysis revealed that 11% of the grains were larger than boulder/cobble size classification ($d > 305$mm), 6.8% were boulder/cobble ($76.2 < d \leq 305$ mm), 63% were gravel ($2 < d \leq 76.2$ mm), 15.8% were sand ($0.074 < d \leq 2$ mm), and 4.7% were silt/clay ($0.053 < d \leq 0.105$ mm). The sand, silt, and clay fractions were analyzed for specific surface area, chemistry, and mineralogy. The specific surface area of the fine fractions was roughly inversely proportional to the particle size diameter (table 2). The pH decreased and the sulfide and trace metal content increased with decreasing particle size of the rock. The finer sand, silt, and clay fractions had S, Cu, and Ni contents above the 0.634% S, 0.348% Cu, and 0.083% Ni levels estimated for pile 4 as a whole.

Table 1. Summary of chemical and physical characteristics of test piles.

	Pile 1	Pile 2	Pile 3	Pile 6	Pile 5
Date completed	4/20/77	4/20/77	4/20/77	9/30/77	9/10/77
S . . . . . %	0.63	0.63	0.63	0.79	1.41
Cu . . . . . %	.35	.35	.35	.34	.30
Ni . . . . . %	.083	.083	.083	.084	.085
Mass . . . . . mt	1,100	1,100	830	1,300	815
Volume . . . . . m^3	540	530	400	630	400
Collecting area . . . . m^2	290	340	300	320	340
Surface area . . . . . m^2	330	450	430	390	370
Cover material	None	Topsoil	Glacial till (coarse sand)	None	Sandy till over coarse sand
Ave. cover depth . . cm	0	23	34	0	54
Vegetated	No	Yes	Yes	No	Yes

Table 2. Characterization of fine fractions from test pile 4.

Size fraction, mm	pH	Total sulfur, %	Sulfide, %	Cu, %	Ni, %	Co, %	Zn, %	Specific surface area, m^2/g
Sand								
0.50 -2.00	6.56	0.67	0.65	0.337	0.062	0.012	0.015	0.60
.177- .50	6.37	.80	.75	.381	.078	.013	.015	.78
.149- .177	6.18	.88	.80	.391	.103	.015	.015	1.1
.105- .149	NA	1.12	1.05	.495	.147	.019	.017	1.6
.075- .105	5.95	1.37	1.30	.585	.221	.022	.019	1.7
Silt and clay								
.053- .075	5.78	1.65	1.57	.647	.253	.024	.019	2.6
<.053	5.77	1.94	1.86	.814	.295	.027	.023	4.7

NA Not available.

X-ray diffraction (XRD) analyses were conducted on the 0.053- to 0.075-mm, the 0.075- to 0.105-mm, and the 0.105- to 0.149-mm size fractions of the rock. Approximately half of this material was plagioclase (with a Na:Ca ratio of about 2:3), and half was a mixture of olivine (with a Fe:Mg ratio of 1:4 to 1:9), pyroxene, and biotite. Lesser amounts of magnetite, amphibole, chlorite, and possibly smectite and/or vermiculite were also identified. The phyllosilicate minerals (biotite, chlorite, smectite, vermiculite) were more common in the finer size fractions.

Piles 1, 3, and 6 were partly covered by plastic at the end of the 1989 field season, for the practical purpose of reducing the volume of drainage the MNDNR is required to treat. The plastic covers were shredded during 1990 by exposure to the elements. A more robust Hypalon cover was placed onto pile 3, thereby terminating flow from this pile at the end of the 1989 field season.

Reclamation Treatments. Three of the piles were left uncovered, as controls, and the remaining three piles were covered (May 1978) with 18 to 29 cm of soil obtained from a nearby borrow pit. Pile 2 was covered with fertile topsoil obtained from the top 30 cm of the borrow pit, while piles 3 and 5 were covered by infertile coarse sand. In 1980 an additional 30 cm of sandy till was added to pile 5. Additional details on vegetation, fertilization, and the application of lime-stabilized sludge to pile 3 (1980, 1982) and a sodium-lauryl-based bactericide to pile 5 (1982) are available in Eger and Lapakko (1981, 1985).

Water Sampling and Analysis. For drainage collection, each pile is underlain by an impervious Hypalon liner (30 mil) which is sloped toward a 15.2-cm perforated plastic pipe. The drainage collected from the piles flows to a common sump. A limited number of water quality samples were collected in 1977, but no flow measurements were made. Since methods in this year deviated from those over the remainder of the study and the data were sparse, the results are not addressed in the text. During 1978, each pile was fitted with a cumulative flow meter and a flow-weighted composite sampler (Eger and Lapakko 1981). For each pump discharge, a fixed volume of sample was placed into the compositing container.

The composite samples were analyzed weekly through 1990 and biweekly subsequently. Samples were filtered through 0.45-μm filters to remove suspended materials prior to analysis. Routine analyses included pH, alkalinity, specific conductance, sulfate, copper, nickel, cobalt, zinc, calcium, and magnesium, although calcium and magnesium were not analyzed in 1983. (Some samples were also analyzed for sodium, potassium, chloride, and dissolved organic carbon.) Specific conductance was analyzed using a Myron L conductivity meter. pH was determined using a Radiometer 29 pH meter from 1978 to 1988, and subsequently using an Orion SA 720 with a

Ross combination electrode (8165). Sulfate was analyzed using the barium sulfate turbidimetric technique (APHA et al. 1992). Metals were analyzed with a Perkin Elmer 603 atomic absorption spectrophotometer.

Calculations

For each period over which composite samples were collected, the mass release was calculated as the product of the concentration in the composite sample and the volume of flow during the period over which the composite was collected. Annual mass release was calculated as the sum of the mass release values for the individual periods for each year. Annual flow-weighted mean concentrations were calculated by dividing the annual mass release by the annual flow. The annual rates of release were calculated as the annual mass release divided by the product of the number of days of flow and the mass of the pile (mt). Average rates of release for the entire period of record were calculated by dividing the total mass release by the number of days of flow.

Empirical neutralization potentials (ENP) were calculated to determine the amount of acid neutralized by the solids prior to the drainage pH decreasing and, for the duration of the period of record, remaining below pH values of 7, 6, 5, 4.5, and 4. The acid neutralized was calculated as the cumulative calcium and magnesium release (expressed as kilograms of $CaCO_3$ per metric ton of rock, or kg/mt $CaCO_3$) prior to the drainage pH decreasing and remaining below the specified pH value. If the drainage pH from a given solid never decreased permanently below a specified value, the ENP was reported as "greater than" the total calcium plus magnesium release for the period of record. Sulfate release was not used for the ENP calculation since some of the sulfate was the product of trace metal sulfide oxidation. The sulfate released from oxidation of copper and nickel sulfides, which comprised 15% to 35% of the sulfide mineral content of the test piles, does not necessarily result in acid production.

Results

Drainage Quality

The test piles typically produced drainage between the middle of March and the middle of November, with an average flow season of 245 days. The pH of drainage from the test piles generally decreased over time, and decreased as the solid-phase sulfur content increased (fig. 1). At the beginning of 1978, the pH of drainage from all piles was between 7 and 8. For the three piles containing 0.63% sulfur, drainage pH decreased gradually from circumneutral to the range of 4.8 to 5.3. The pH of drainage from the piles containing 0.79% and 1.4% sulfur decreased and plateaued within a range of about 0.2 to 0.3 pH unit. The pH of drainage from the 0.79%-sulfur pile decreased until 1982 and subsequently oscillated in the typical range of 4.0 to 4.3. The pH of drainage from the 1.41%-sulfur pile decreased to a lower level, from a median value of 5.5 in 1978 to a typical range of 3.4 to 3.6 from 1979 to 1991. The annual median values actually increased from 1982 (3.40) to 1991 (3.60).

Annual flow-weighted mean sulfate concentrations for the individual piles tended to increase during the first 3 to 4 yr and then oscillated within a fairly constant range, increased as the solid-phase sulfur content increased, and were independent of the annual flow volume. Some deviations from the general trends were observed. Most notably, sulfate concentrations in drainage from the 1.41%-sulfur pile increased to peak levels between 1981 and 1985 and then decreased. From 1987 to 1991 the flow-weighted mean sulfate concentrations from this pile were comparable to, or lower than, those from the 0.79%-sulfur pile. Sulfate concentrations from the 0.79%-sulfur pile increased over time. However, these temporal variations from 1981 through 1991 were relatively small. In general the maximum annual flow-weighted mean sulfate concentrations were typically within a factor of 2 of the minimum value observed during this period.

Annual flow-weighted mean magnesium concentration variations over time were generally parallel to those observed for sulfate and were independent of annual flow volume. Magnesium concentrations were lowest in drainage from the 0.63%-sulfur piles, typically ranging from 2 to 6 mmol/L. The corresponding range for the two piles of higher sulfur content was 7 to 15 mmol/L. For the individual piles, annual flow-weighted mean magnesium

concentrations tended to increase as the annual median pH decreased.

Annual flow-weighted mean calcium concentrations were typically fairly constant, ranging from about 4 to 7 mmol/L. As implied by the constant range, there was little dependence of annual flow-weighted mean calcium concentrations on time of dissolution, sulfur content, or annual flow. The values for the 1.41%-sulfur pile appeared to parallel the decreasing trend observed for sulfate concentrations from 1985 to 1991.

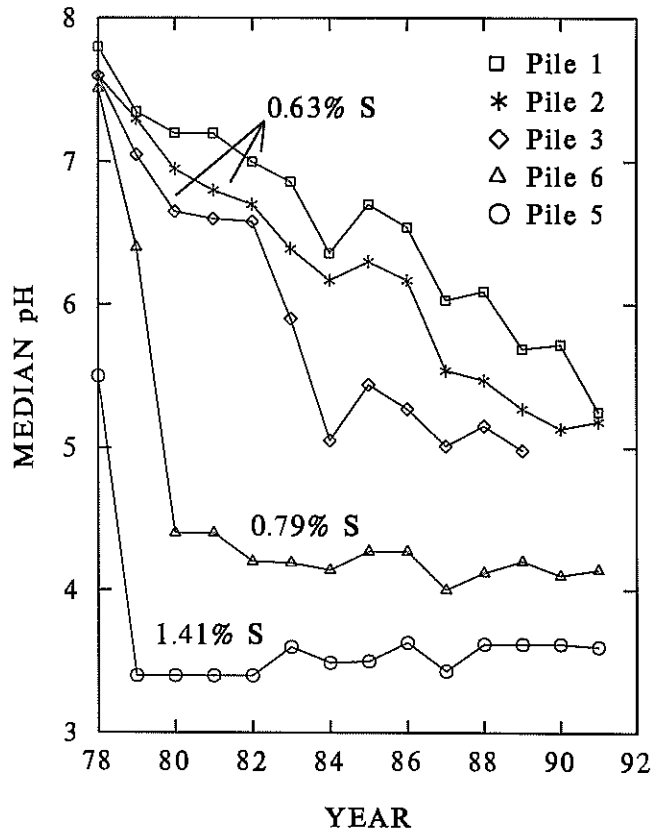

Figure 1. Annual median pH from 1978 to 1991. Drainage pH decreased over time and as solid-phase sulfur content increased.

Annual Rates of Release

The annual rate of sulfate release from each pile oscillated within a fairly constant range (fig. 2). In general, annual sulfate release increased with the solid-phase sulfur content of the pile, although the rates for the 0.79%-sulfur pile were comparable to or exceeded those for the 1.41%-sulfur pile from 1984 to 1991. Sulfate release also increased with annual flow, and temporal variations in the annual rate were largely due to variations in annual flow volume. A graphic example of the flow dependence is the low sulfate-release rate from the 0.79%-sulfur pile in 1990, while the pile was partly covered with plastic. The drainage volume and sulfate mass release were consequently low.

Variations in magnesium release followed similar trends. Similar to the sulfate-release rates from 1984 through 1991, magnesium release from the 0.79%-sulfur pile was usually higher than that from the 1.41%-sulfur pile. In contrast, the annual rates of calcium release typically fell into the range of 1 to 3 mol/mt/d. They did not vary with time or solid-phase sulfur content, but did increase linearly with annual flow.

Empirical Neutralization Potential (ENP)

The pH of drainages from the 0.63%-sulfur rock did not decrease permanently below 5.0. The approximate ranges of ENP values determined for endpoint pH values of 7.0 and 6.0 (ENP_{pH7} and ENP_{pH6}, respectively) were 0.2 to 0.4 and 0.4 to 1.3 kg/mt $CaCO_3$, respectively. The pH of drainage from pile 3 approached 5 at the end of 1989. The 1.3 kg/mt $CaCO_3$ ENP value determined for this pile is assumed to be near the ENP_{pH5} for this pile. The lowest ENP endpoint pH values for the piles with sulfur contents of 0.79% and 1.41% were pH 4.5 and pH 4.0, respectively. The ENP values for any pH value tended to decrease as sulfur content increased.

Comparison of Field and Laboratory Data

Certain qualitative similarities were apparent between the field data and laboratory data (Lapakko 1988, Lapakko and Antonson 1993).

1. Drainage pH generally decreased, or decreased and plateaued, as the time of dissolution increased. A subtle variation on this trend was a possible slight increase in the pH of drainage from the 1.41%-sulfur pile from 1982 to 1991 (median annual pH values of 3.40 and 3.60, respectively).

2. In both laboratory and field, drainage pH values decreased as solid-phase sulfur content increased.

3. The rates of sulfate release were fairly constant over time. The ratio of maximum-sulfate-release rate to minimum-sulfate-release rate for a laboratory sample was typically less than 3:1. This ratio was observed to increase if drainage pH decreased below about 4. The corresponding ratios for the test piles ranged from 2.7:1 to 3.9:1.

4. In both laboratory and field tests the rate of sulfate release increased with solid-phase sulfur content. Sulfate release rates from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

5. Rates of calcium release were relatively constant over time, although it must be noted that laboratory data on calcium release are limited.

6. Rates of calcium release were relatively constant with respect to sulfur content. The rates of calcium release from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

7. Rates of magnesium release increased as time increased, although it must be noted that laboratory data on magnesium release are limited.

8. Rates of magnesium release increased as solid-phase sulfur content increased. The rates of magnesium release from the test piles also increased with drainage volume, a variable not examined in laboratory tests.

9. Neutralization potential values were low and decreased with increasing sulfur content.

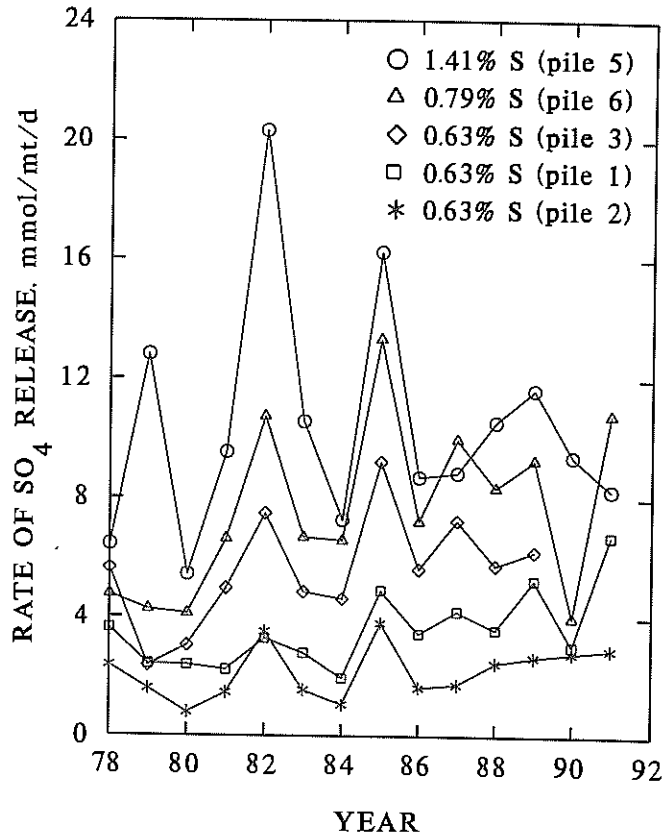

Figure 2. Annual sulfate release rate from 1978 to 1991. Sulfate concentrations generally increased with solid-phase sulfur content and were within a constant range over time.

Field and laboratory data were also quantitatively compared based on drainage pH, neutralization potentials, and rates of release. The minimum drainage pH values observed for the 0.63%-sulfur rock in the field ranged from 4.71 to 4.96 and were reasonably consistent with the laboratory relationship between drainage pH and solid-phase sulfur content. However, the minimum pH values for drainages from the test piles with sulfur contents of 0.79% and 1.41% were about one unit lower than expected based on laboratory data (fig. 3). The minimum pH values observed were typically occasional excursions, and recurrent low pH values were typically 0.1 to 0.2 unit higher.

For the entire period of record, average rates of sulfate release ranged from 2.0 to 10 mmol/mt/d, and increased as sulfur content and drainage volume increased. The average rates of calcium release ranged from 2.1 to 2.4 mmol/mt/d and were fairly constant with respect to sulfur content. Average rates of magnesium release ranged from 0.55 to 3.7 mmol/mt/d and were lowest for the low-sulfur solids and highest for the solids containing 0.79% sulfur.

The rates of sulfate, calcium, and magnesium release observed for the test piles were compared to laboratory rates for the Duluth Complex samples of similar sulfur content (Lapakko and Antonson 1993). The laboratory rates were from the last 15 to 36 weeks of the period of record. The ratios of field rates to laboratory rates, or retardation factors, ranged from 0.053 to 0.46, with most values in the range of about 0.1 to 0.36 (table 3). This indicates that the laboratory rates were roughly 3 to 10 times those in the field.

The neutralization potentials of the test piles were lower than those for laboratory samples, which is consistent with the lower drainage pH levels observed in the field. The ENP values for the 0.63%-sulfur rock were roughly 25% to 50% of those observed for laboratory samples of similar sulfur content (table 4). Although the ENP_{pH6} and ENP_{pH7} values from pile 3 were higher than this range, the ENP_{pH5} value was 46% of the corresponding value for laboratory samples.

The field ENP values for the 0.79%-sulfur rock were 13% to 54% of the corresponding values for laboratory samples of similar sulfur content. The ratios of field to laboratory ENP were lowest for the 1.41%-sulfur rock, typically 5% or less than those determined in the laboratory.

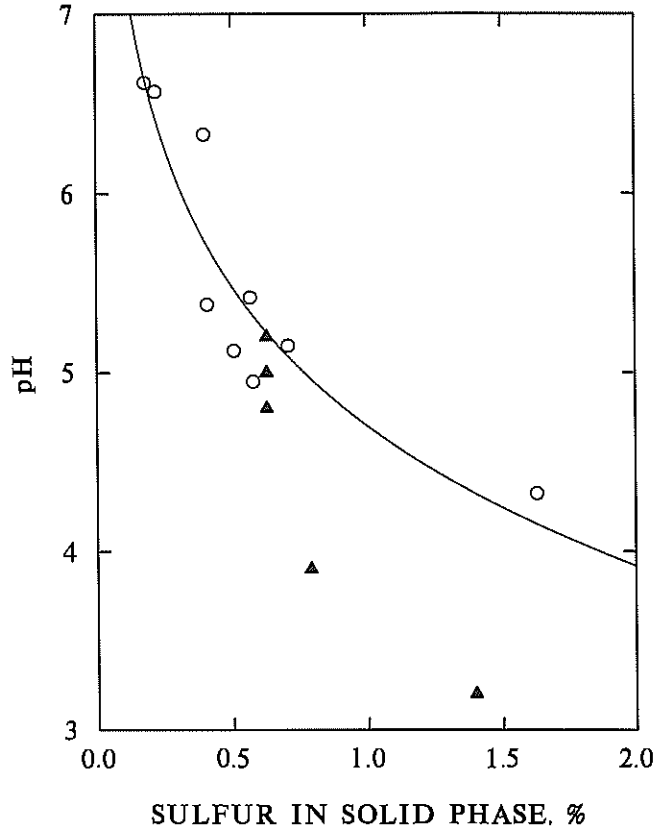

Figure 3. Comparison of recurrent minimum drainage pH in field (▲) and laboratory (○). Laboratory data represents drainage pH after 150 weeks of dissolution of Duluth Complex samples (Lapakko and Antonson 1993).

Discussion

There were several differences between the conditions in the laboratory dissolution experiment and those in the field study. Laboratory particles were smaller than the average particle in the field, ranging from 0.053 mm to 0.149 mm in diameter, and therefore had a higher specific surface area. Due to the relatively uniform particle size, the available mineral surface area was roughly proportional to the amount of the mineral present. In contrast, the fraction of the rock surface area comprised of iron sulfide minerals in the field was most likely higher than that indicated by the sulfur content of 0.63%. Sulfide minerals comprised a disproportionately large fraction of the fine material the field, as indicated by the elevated sulfur content of this fraction (table 2). This material also had the highest specific surface area of the field rock. Therefore, iron sulfide minerals comprised a larger fraction of the field rock surface area than indicated by the sulfur content alone.

Hydrologic differences also existed. The volume of rinse water per unit mass rock yielded approximately 300 L per ton of rock per day of drainage in the laboratory, roughly three orders of magnitude greater than the yields of 0.27 to 0.38 L per ton of rock per day in the field. Furthermore, the flow of the laboratory rinse water through the solids was fairly uniform, whereas preferential flow was probable in the field setting.

Despite these differences there were numerous similarities between the field and laboratory results. As cited earlier, the variations of drainage pH and chemical mass release were qualitatively comparable in the two settings.

Table 3. Comparison of field and laboratory release rates. Rates in moles per gram rock per second x 10^{-14}.

Sulfur, %	Field rate			Lab rates			Retardation factor			
	SO_4	Ca	Mg	SO_4	Ca	Mg	SO_4	Ca	Mg	Ave.
0.63[1] (pile 1)	3.9	2.5	0.86	[1]25	[1]6.7	[1]12	0.16	0.37	0.072	0.20
.63[1] (pile 2)	2.4	1.4	0.64	25	6.7	12	0.096	0.21	0.053	0.12
.63[1] (pile 3)	6.4	2.8	2.4	25	6.7	12	0.26	0.42	0.20	0.29
.79[2] (pile 6)	8.5	2.8	4.3	[2]26	[2]6.1	[2]12	0.33	0.46	0.36	0.38
1.41[3] (pile 5)	12	2.7	3.7	[3]66	[3]19	[3]21	0.18	0.14	0.18	0.17

[1] The arithmetic mean of average rates from samples with sulfur contents of 0.57%, 0.58%, and 0.71% (150 week period of record) was used for comparison with 0.63%-sulfur rock. Minimum pH values for these samples ranged from 4.82 to 5.38, as compared with 4.7 to 5.0 in the field.

[2] Rates were linearly extrapolated based on sulfur content using average rates from samples with sulfur contents of 0.71% and 1.16% (150- and 69-week periods of record, respectively). Minimum pH values for these samples ranged from 4.6 to 5.0, about 1 unit higher than the pH 3.7 minimum value for the field pile.

[3] The arithmetic mean of average rates from samples with sulfur contents of 1.40% and 1.44% was used (69-week period of record) for comparison with the 1.41%-sulfur pile. Minimum drainage pH for these samples ranged from 4.4 to 4.9, about 1.5 units higher than the minimum drainage pH for the field pile and and 1 unit higher than its recurrent minimum pH.

Table 4. Comparison of field and laboratory neutralization potentials.

S, %	pH 7	pH 6	pH 5	pH 4.5	pH 4	pH 3.5
FIELD NEUTRALIZATION POTENTIAL[1]						
0.63 (pile 1)	0.346	0.781	>1.059	>1.059	>1.059	>1.059
.63 (pile 2)	.179	.393	>.642	>.642	>.642	>.642
.63 (pile 3)	.442	1.254	>1.340	>1.340	>1.340	>1.340
.79 (pile 6)	.157	.172	.226	1.702	>2.074	>2.074
1.41 (pile 5)	.044	.044	.096	.130	.136	>1.889
RATIO OF FIELD ENP TO LABORATORY ENP						
0.63 (pile 1)	0.52	0.52	NAp	NAp	NAp	NAp
.63 (pile 2)	.27	.26	NAp	NAp	NAp	NAp
.63 (pile 3)	.66	.84	NAp	NAp	NAp	NAp
.79 (pile 6)	.54	.23	0.13	NAp	NAp	NAp
1.41 (pile 5)	.17	.056	<.056	<0.045	<0.044	NAp

[1] kg/mt $CaCO_3$
NAp Not applicable.

Quantitatively, the drainage from 0.63-percent sulfur rock in the laboratory approximated that in the field. However, pH of field drainages from the 0.79%- and 1.4%-sulfur rock were about 1 and 1.5 units, respectively, lower than the corresponding laboratory values. These field drainage pH values may have been lower due to a higher iron sulfide surface area than indicated by the sulfur content alone. The low rinse water to rock ratio in the field would also lend to increased H^+ concentrations.

Both the rates of mineral dissolution and subsequent reaction product transport probably contributed to the lower sulfate, calcium, and magnesium release rates (per unit mass) observed in the field. First, the larger field particles have a lower specific surface area (surface area per unit mass) than the smaller particles used in the laboratory. The lower surface area in the field would retard the sulfate, calcium, and magnesium release per unit mass rock, since both sulfide mineral oxidation and silicate mineral dissolution are surface reactions. The reaction environment in the field, where temperatures are colder, may also retard reaction rates. Limitation of sulfide oxidation by oxygen transport seems unlikely due to the small test pile size.

Limited transport of reaction products in the field also may have reduced rates of release. Hydrologic variables, which were not influential in the laboratory, could have contributed to this transport limitation. The relatively low flow in the field may have limited transport. This is supported by the observation that release rates in the field increased with flow. Preferential flow would limit the transport of reaction products from some rock surfaces. This would essentially reduce the amount of rock in the test piles that was actually contributing to the observed chemical release. The degree of transport in the field may also be limited by chemical precipitation of dissolution reaction products. In particular, the formation of gypsum might limit the rates of sulfate and calcium release. Gypsum formation was not a problem in the laboratory since the high rinse-water-to-solid ratio produced relatively low concentrations of sulfate and calcium in the drainages.

In addition to the larger particle size in the field, the low field ENP values may have been influenced by factors similar to those discussed regarding drainage pH. First, preferential flow in the field piles would reduce the amount of silicate minerals, which could contribute to acid neutralization. For example, if flow through the pile contacted only half the silicate minerals, the observed neutralization potential of the rock would be half the actual value for the entire pile. Second, if the fraction of rock surface area comprised of silicate minerals in the laboratory samples exceeded that in the test piles, the laboratory solids would be expected to neutralize acid more efficiently. Third, the ratio of available reactive surface area of sulfide minerals to that of silicate minerals present in the test pile rock may have been greater than that in the laboratory. This would be equivalent to increasing the sulfur content of rock in the laboratory. As was observed in the laboratory, neutralization potential tended to decrease as sulfur content increased.

More rigorous analysis of the recently compiled laboratory and field data is presently in progress at the Minnesota Department of Natural Resources. The data are also being used to aid development of a geochemical predictive model for acid mine drainage at the US Bureau of Mines Salt Lake City Research Center.

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The test piles were constructed by AMAX Exploration Inc. Sulfate was analyzed by Jean Matthew, while calcium and magnesium were analyzed by Albert Klaysmat at the Hibbing office of the Minnesota Department of Natural Resources. Since 1978 sampling and sampling maintenance have been the responsibility of several people: Bruce Johnson, Anne Jagunich, and, most recently, Cal Jokela and David Antonson. Paul Eger has supervised these activities. Jon Wagner provided much of the data input and Jim Porter was responsible for the most recent data management, as well as tabular and graphical output. The U. S. Bureau of Mines provided funding to support data compilation and analysis as part of Cooperative Agreement C0219003.

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Attachment 13

Wenz Z (2016) Geochemistry of Leachate from a Naturally Weathering Duluth Complex Rock Pile: Sulfur Removal Models for a 38 Year Record of Leachate Composition. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN.

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EXECUTIVE SUMMARY

The Minnesota Department of Natural Resources (MN DNR) has been performing environmental research to assess the reactivity of sulfide bearing rock for decades. One of the longest running experiments is the Duluth Complex Leach Pile (DCLP) located at the MN DNR Hibbing field research site. The DCLP is a composite of three sulfide bearing Duluth Complex rock piles originally constructed under a 1978 cooperative research agreement between the MN DNR and Amax Environmental Services Inc. Upon termination of the cooperative research program in 1994, rock from three of the piles was moved to Hibbing to continue the experimental work. At the Hibbing field research site, the rock was placed on a double lined leachate collection system and has been monitored since 1996. The continuation of this rock weathering experiment has documented a 38 year record of a changing rock leachate composition.

The DCLP leachate composition and flow volume data was used to calculate annual sulfate release rates from 1978 to 2015. These release rates are incorporated into an empirical model that numerically describes the rate of sulfur removal from the rock pile over time. This empirical model shows that over the 38 year period of record, the rate of sulfur release has decreased by about a factor of six, and approximately 1,000 kg of sulfur has been removed from the 812 tonne rock pile.

A theoretical model was developed using a shrinking particle model that incorporated rock characterization data (e.g., sulfur concentration, particle size distribution, and sulfide surface area exposed) and an experimentally derived sulfide oxidation rate law. The sensitivity of the theoretical model to the changing mass fractions of different rock particle sizes was investigated. The sensitivity analysis found that the theoretical model is very sensitive to the mass of particles less than about 0.149 mm highlighting that a well-defined particle size distribution is paramount for accurate theoretical predictions.

The theoretical model results are within a factor of two of the empirical modeling results demonstrating theoretical prediction can be a relatively accurate approach when the chemical and physical parameters of the waste rock are well defined. The relatively close agreement between the empirical and theoretical sulfur removal models demonstrates that theoretical approaches for predicting the rate of sulfur removal from a waste rock can be of value for developing mine waste management strategies.

1.0 INTRODUCTION

The Minnesota Department of Natural Resources (MN DNR) has conducted numerous field and laboratory rock weathering experiments. Some of these experiments have operated for long periods of time providing the opportunity to characterize how the governing hydrogeophysical properties change as the reactant mass, namely sulfide minerals, decreases. This report focuses specifically on Duluth Complex waste rock extracted from the Mesaba prospect in the late 1970's. Historically, the Mesaba prospect has been referred to as the Babbitt or Minnamax prospect. This report combines previously reported rock leachate data (Lapakko et al., 2004) with new data to evaluate the long term changes of leachate geochemistry for sulfide-bearing Duluth Complex waste rock. The rock leachate data is used to develop a data based empirical model and compare it to a theoretical model to identify the governing physical and hydrogeochemical parameters.

In 1977, Amax Exploration Inc. constructed six rock piles consisting of lean ore from the Duluth Complex Mesaba prospect (fig. 1.1). The rock piles ranged in mass from 830 to 1700 tonnes and had average sulfur concentrations between 0.6 and 1.4 wt%. In 1978, the MN DNR and Amax Environmental Services Inc. initiated a cooperative environmental research program to evaluate the composition of leachate emanating from the rock piles. Leachate from these rock piles was collected and analyzed until 1994, when the MN DNR and United States Bureau of Mines (USBM) performed a comprehensive characterization study of the rock piles during reclamation of the site. The leachate monitoring and subsequent MN DNR and USBM characterization study are collectively referred to as the 'Amax field study' (reported in Eger et al., 1979; Eger and Lapakko, 1985; and Lapakko et al., 2004). During the reclamation process, some rock from three of the piles was extracted, temporarily stored under cover, and ultimately placed on a leachate collection system at the MN DNR Hibbing field research site in 1996. The composite of the material from the three Amax field study rock piles is referred to as the Duluth Complex Leach Pile (DCLP) and has a volume weighted average sulfur concentration of 0.91 wt%. The DCLP was originally constructed to generate leachate for assessment and developing treatment methods for metal-rich acidic leachate. Beginning in 1996, leachate from the DCLP has been monitored and treated at the MN DNR Hibbing field research site (fig. 1.1). Collectively, the Amax field study and monitoring of the DCLP represent a 38 year record of the leachate composition from naturally weathering sulfide bearing Duluth Complex waste rock.

The objective of this report is to use the 38 year DCLP dataset to evaluate the changing composition of the rock leachate and develop an empirical model to describe the rate at which sulfur is removed from the rock pile. This empirical model is then compared to a theoretical model to identify the governing physical and hydrogeochemical factors and how they have changed over time. Substantiating a theoretical model for predicting long term solute release would provide a valuable predictive tool that can be used by the MN DNR to more accurately execute environmental review and mine permitting responsibilities. Of particular importance is identifying an accurate method for determining long term mining impacts which can be used to inform mine waste management strategies for existing and proposed mining operations.

2.0 ROCK WEATHERING GEOCHEMICAL PRINCIPLES SUMMARY

Mineral reaction rates, such as sulfide oxidation, are fundamentally dependent on the amount of surface area available for reaction (eq. 2.1). As mineral surface area increases, the rate of reaction increases resulting in a greater production of reactants per unit time.

$$r = A_{SA} \times k_{+} \qquad \text{eq. 2.1}$$

r= rate of reaction (mol·sec^{-1})
A_{SA}= surface area (m^2)
k_+= dissolution flux or 'rate constant' (mol·m^{-2}·s^{-1})

Numerous researchers have determined that sulfide oxidation rates are also dependent on fluid pH and the concentration of the oxidant (McKibben and Barnes, 1986; Williamson and Rimstidt, 1994; Holmes and Crundwell, 2000; Kimball et al., 2010; and Bilenker et al., 2015). Silicate mineral reaction rates are also commonly dependent on fluid pH, but generally have much slower reaction rates. Carbonate minerals reaction rates are strongly dependent on fluid pH and have reaction rates similar in magnitude or greater than many sulfide oxidation rates. Figure 2.1 illustrates the relative rates of reaction for some minerals common in Duluth Complex rock types.

For a naturally weathering waste rock pile it is the complex interplay of the various physical and hydrogeochemical processes that ultimately lead to the leachate concentration. For Duluth Complex rock, water and oxygen contacting the sulfide minerals pyrrhotite ($Fe_{1-x}S$), cubanite ($CuFe_2S_3$), and chalcopyrite ($CuFeS_2$) will generate acid (H^+) through oxidation reactions (rxns 2.1-2.3). The acid generated by these reactions can be neutralized by alkalinity generated from reaction of carbonate and silicate minerals such as calcite ($CaCO_3$) and plagioclase ($CaAl_2Si_2O_8$, anorthite end member) via reactions 2.4 and 2.5, respectively. The final pH of the leachate depends on the relative amounts of surface area reacted from each of the minerals and the dissolution flux (k_+) of each reaction. As the amount of sulfide surface area increases relative to the neutralizing minerals the leachate pH will decrease. Olivine and plagioclase are much more reactive in weakly acidic fluids (fig 2.1) than many other silicates. The Duluth Complex typically contains low concentrations of carbonate minerals with wt% $CaCO_3$ concentrations commonly less than 0.2 (Lapakko et al., 2013; Wenz et al., 2013; and Kellogg et al., 2014). This small amount of carbonate is important for neutralizing acid early in the lifetime of a weathering rock pile, particularly for rock with a low sulfide concentration. After the neutralization provided by carbonate minerals is depleted, the abundant silicate mineral assemblage becomes the most important acid neutralizer. The Duluth Complex is generally troctolitic to gabbroic in composition, thus the mineralogy is predominantly olivine ($(Fe,Mg)_2SiO_4$) and plagioclase ($(Ca,Na)(Al,Si)_4O_8$). Therefore, the silicate mineral assemblage of Duluth Complex rock types can provide a substantial amount of neutralization despite containing a small amount of carbonate.

$$8Fe_{1-x}S + [18-6x]\,O_2 + [20-12x]H_2O = [8-8x]Fe(OH)_3 + 8SO_4^{2-} + 16H^+ \qquad \text{rxn. 2.1}$$

$$\tfrac{8}{3}CuFe_2S_3 + \tfrac{52}{3}O_2 + \tfrac{40}{3}H_2O = \tfrac{16}{3}Fe(OH)_3 + 8SO_4^{2-} + \tfrac{8}{3}Cu^{2+} + \tfrac{32}{3}H^+ \qquad \text{rxn. 2.2}$$

$$4CuFe_2S_3 + 17O_2 + 10H_2O = 4Fe(OH)_3 + 8SO_4^{2-} + 4Cu^{2+} + 8H^+ \qquad \text{rxn. 2.3}$$

$$CaCO_3 + 2H^+ = H_2CO_3 + Ca^{2+} \qquad \text{rxn. 2.4}$$

$$CaAl_2Si_2O_8 + 2H^+ + H_2O = Ca^{2+} + Al_2Si_2O_5(OH)_4 \qquad \text{rxn. 2.5}$$

3.0 METHODS

3.1 Duluth Complex Leach Pile Construction and Monitoring

In 1994, the Amax field study located near Babbit Minnesota was deconstructed and reclaimed. Of the six original Amax field study rock piles, rock from three of the piles was extracted and transported to Hibbing Minnesota. At the MN DNR Hibbing field research site the rock was temporarily stored in a 30

mil chlorosulfonated polyethylene (CSPE) synthetic rubber cover and liner system until a leachate collection system and water treatment plant was built. The leachate collection system consists of a double-lined containment system with a drain for collection of the leachate with an underlying leak detection layer. The double-lined system consists of an upper 45 mil CSPE liner underlain by a bed of sand and a 30 mil poly(vinyl chloride) bottom liner. In 1996, construction of the leachate collection system and treatment plant was completed and the temporarily stored rock was placed on the leachate collection system. The leachate emanating from the rock pile flows into a top drain and down through a drain pipe into the treatment plant where the water collects in a bin for a composite sample. When the collection bin is close to filling, a float trigger activates a pump and discharges the water from the bin. The volume of the discharge is recorded by a flow meter.

The rock moved to the Hibbing field research site included the FL1, FL5 and FL6 Amax field study rock piles with estimated volumes of 197, 182, and 289 y^3, respectively (Lapakko et al., 2004). The FL1 and FL6 rock piles were uncovered and naturally weathering whereas the FL5 rock pile was covered with a sandy till (30 cm thick in 1980) over coarse sand (about 24 cm thick in May of 1978). The FL5 rock pile was also vegetated and fertilized in 1978 (see Lapakko, 1993 for more details). It is not clear if the cover material on the FL5 pile resulted in a decrease in sulfur removal. Data from the Amax field study indicates that the amount of sulfate released from the rock piles generally increased in the order FL1 > FL6 > FL5, which is consistent with increasing rock pile sulfur concentration. Thus, the cover materials did not result in depressing the sulfate release rate from FL5 to less than that of the FL6 pile. However, it is not clear if the amount of sulfur released from the FL5 pile would have been greater if cover materials had not been applied. In 2015, the DCLP was surveyed using modern GPS survey equipment (fig. 1.1). The 2015 survey found the rock pile volume was 488 y^3, which is 180 y^3 less than the original estimate. Calculations for this study used the 488 y^3 volume for the DCLP, an estimated rock pile porosity of 25%, and a rock density of 3.0 g/cm^3, which corresponds to a rock pile mass of 812,146 kg.

From 1996 to 2002, leachate from the DCLP was collected and analyzed intermittently during the spring to fall months. Beginning in the fall of 2013, leachate collection and analysis resumed. Leachate volume was recorded from 1996 to 1998, 2002, and for some periods in 2013 and 2015. The leachate samples were vacuum filtered through a 0.45 micrometer filter and metal samples were treated with 0.1 mL of analytical grade nitric acid per 50 mL of leachate. Appendix 1 contains all of the DCLP monitoring data.

3.2 Composite Leachate Composition Calculation

An estimate of the pre-1996 DCLP leachate composition was calculated from the FL1, FL5, and FL6 rock pile leachate data set from the Amax field study. This composite leachate composition was based on the volume weighted proportion of each of the three rock piles for which there was a coincident leachate sampling date. This approach allowed for an approximation of what the leachate composition would have been for a rock pile with the same composition as the DCLP (0.91 wt% sulfur). Although this approach is an approximation, the calculated composite leachate solute concentrations are consistent with the anticipated solute concentrations based on the DCLP and original Amax field study rock piles sulfur concentration (fig. 3.1). This estimation approach works well for conservative species (sulfate, calcium, magnesium), but is not accurate for non-conservative species such as bicarbonate (i.e., alkalinity) and metals that have low pH sorption edges (e.g., copper and zinc). The leachate composite calculation spans the period from 1978 to 1993 and is tabulated in Appendix 2.

3.3 Solid Phase Analysis

A 13.7 kg composite sample was collected from the DCLP pile in 2014. The composite sample consisted of large rock samples generally greater than 10×10×10 cm. The weathered rind was sawed from each of the samples to ensure a fresh rock sample for solid phase analysis. The composite sample was crushed

following the ASTM D5744-e1 procedure. A split from the total crushed mass was sieved into five separate particle size intervals from which the mass fractions were determined.

The composite sample was analyzed as a bulk sample and as individual size intervals. The bulk sample was analyzed for mineral content, major oxides, sulfur and carbon, and trace metal and non-metals by Activation Laboratories Ltd (Actlabs). Mineral content was determined by x-ray diffraction (XRD) and Scanning Electron Microscopy (SEM) Mineral Liberation Analysis (MLA). The MLA mineral content for the bulk sample was determined from a split that was crushed down to less than 300 um. Major oxide concentrations were determined by lithium metaborate-tetraborate fusion followed by nitric acid digestion and inductively coupled plasma-mass spectrometry (ICP-MS) and inductively coupled plasma-optical emission spectrometry (ICP-OES) analysis. Sulfur and carbon were analyzed by combustion in an induction furnace. Trace metals were analyzed by ICP-MS after a sodium peroxide fusion. The sodium peroxide fusion technique dissolves all minerals. Mercury was determined by the cold vapor flow injection mercury system (FIMS) method. Bromide and chloride were measured by instrumental neutron activation analysis (INAA). Fluoride was analyzed by an ion-selective electrode (ISE) method after a lithium metaborate-tetraborate fusion followed by nitric acid digestion with an ammonium citrate buffer added.

MLA analysis was also used to determine the individual sulfide minerals grain size distribution and surface area exposure for each of the five particle size sieve intervals. The surface area exposed was determined in 10% gradations by wt% free surface. The 100 wt% exposed sulfide grains represent the so-called 'liberated' fraction.

3.4 Leachate Analysis

Leachate samples were collected from a holding tank. The holding tank has a capacity of about 12 liters. Once the capacity is near full, an electric float is triggered that turns on a pump which removes approximately 80% of the holding tank capacity. Therefore, at any one time the leachate collected from the holding tank represents a leachate composite for each sampling interval. The leachate pH, acidity or alkalinity, and specific conductivity were measured at the MN DNR Hibbing laboratory. Leachate was analyzed for anion and metal concentrations by the Minnesota Department of Agriculture from 1996 to 2012. In 2013, leachate was analyzed by Pace Analytical Services Inc. Starting in 2014, Legend Technical Services Inc. analyzed the leachate samples using EPA methods 6010B and 6020A for cations and EPA method 9056 for anions.

3.5 Empirical and Theoretical Modeling Approach

Sulfate leachate concentration was used to develop an empirical model for the DCLP that numerically describes the rate of sulfur removal with time. The empirical model was constructed with Microsoft Excel software using linear and nonlinear regression analysis of the annual sulfate release rates measured from the DCLP leachate. For years 1978 to 1995, the rock volume weighted average of the annual sulfate release rates from the AMAX field study (Lapakko et al., 2004) was used. For the years after 1995, the annual volume weighted average of the DCLP leachate composition was used. The theoretical model used the volume weighted particle size distribution of the original Amax field study rock piles, MLA determined sulfide surface exposure values, and a laboratory derived surface area normalized pyrrhotite oxidation rate.

4.0 RESULTS

4.1 DCLP Composite Sample Solid Phase Analysis

The DCLP composite sample was analyzed as a bulk sample and as five different particle size intervals. The five particle size intervals were 2.0-6.35 mm, 0.5-2.0 mm, 0.149-0.5 mm, 0.053-0.149 mm, and <0.053 mm.

4.1.1 Bulk Sample

The bulk sample was analyzed for mineralogy, major oxides, sulfur, carbon, and trace metal and nonmetal analysis. Mineralogy is presented on table 4.1, major oxides on table 4.2, carbon, sulfur and halogen concentrations on table 4.3, and trace metals and nonmetal concentrations on table 4.4.

The mineralogy from XRD and MLA analysis was generally in good agreement. The major discrepancies were the apparent detection of amphibole (14.24 wt%) and non-detection of olivine for the MLA, whereas the XRD analysis identified olivine (8.9 wt%), an amorphous phase (12.9 wt%), and no amphibole. It is possible that the amorphous material was compositionally classified as amphibole by the MLA, but did not have a crystalline structure which would explain why no amphibole was detected from the XRD analysis. The occurrence of the amorphous material indicates substantial post crystallization alteration of some primary mineral phase(s). A petrographic analysis is required to accurately determine the original mineral type of the amorphous material. The XRD analysis also identified low concentrations of cordierite, whereas MLA did not detect cordierite. Mineralogical analyses of the same rock presented in Lapakko et al. (2004) indicated 1.6 to 18.4 wt% olivine, <0.1 to 3 wt% cordierite, and no detection of a pure amphibole phase. The presence of cordierite suggests some amount of assimilation of the adjacent pelitic rocks during crystallization. Based on the classification scheme of Miller et al. (2002) the rock type is either an olivine gabbronorite or norite based on the XRD and MLA analyses, respectively.

Major oxide concentrations were consistent with the mafic mineralogy. Total sulfur and carbon concentrations were determined by combustion of two separate splits at two different labs. Total sulfur was additionally analyzed by sodium peroxide fusion followed by ICP-MS. All carbon and sulfur values were in good agreement. The average total carbon and sulfur concentrations were 0.03 wt% and 1.24 wt%, respectively. Both bromide and fluoride were less than the detection limit and chloride had a concentration of 0.04 wt%. Copper and nickel are the most common trace metals with respective concentrations of 6,630 and 1,420 parts per million (ppm).

4.1.1 Particle Size Interval Samples

The particle size interval samples were analyzed individually for mass fraction, mineralogy, major oxides, sulfur, carbon, and trace metal and nonmetal analysis. The particle size interval data is presented on the same tables as that for the bulk analyses. Dry and wet mass weighted particle size data is presented on table 4.5. The particle size intervals were also analyzed by SEM MLA to determine the sulfide grain size distribution and amount of sulfide mineral exposure. The MLA grain size distribution and sulfide mineral exposure data is presented in tables 4.6 and 4.7, respectively.

Particle size interval mass fractions were determined for both dry and wet sieving. For the less than 0.053 mm interval the wet sieve was only 0.1 wt% greater indicating a very small amount of fine material adheres to the larger particles. The two largest size intervals accounted for 85% of the total mass of the sample. Importantly, the relatively large portion of the sample mass consisting of large particles will limit the overall reactivity of mass due to decreasing the samples specific surface area (SSA; cm^2/g).

MLA particle size interval mineralogy generally agreed with the bulk mineralogy. The primary exception was the carbonate and sulfide mineral contents which generally increased with decreasing particle size and plagioclase mineral content which generally decreased with decreasing particle size. The particle size dependent mineralogy trends were also apparent in the major oxide and carbon and sulfur compositions. The SiO_2 and CaO wt% values decreased with decreasing particle size, whereas the sulfur and carbon wt% values increased with decreasing particle size. In addition, the copper and nickel concentrations increased with decreasing particle size consistent with an increase of Cu- and Ni-bearing sulfide mineral content with decreasing particle size.

MLA was used to determine the sulfide mineral grain size distribution and the amount of each sulfide mineral that was exposed for each of the five particle size intervals. The sulfide grain size distributions did not vary substantially among the different sulfide types except for the 0.5-2.0 mm size interval for chalcopyrite. The chalcopyrite grain size distribution for the 0.5-2.0 mm size interval ended at about 400 micrometers, whereas the other sulfides were between about 800 and 1,000 micrometers. No one size interval exhibited 100% exposure for any sulfide mineral. Excluding the 0.053-0.149 mm size interval for pentlandite, the two smallest particle size intervals had about 70-80% of the sulfide minerals exposed. Excluding the 0.5-2.0 mm size interval for cubanite, the two largest particle size intervals had less than about 2 wt% exposure. The 0.149-0.5 mm size interval exhibited the greatest variation among the sulfide mineral types, ranging between about 20 to 50 wt% exposure.

4.2 DCLP Leachate Composition Trend Analysis

The 38 year monitoring record for the DCLP illustrates substantial changes of leachate solute concentrations, indicating changing geochemical conditions. In general, there are four principal leachate solute concentration trends easily identifiable from the DCLP leachate sulfate concentration time series data (fig. 4.1). First, leachate concentration increased for about the first five years. Second, after about year five solute concentrations were relatively stable through about year 13. Third, after about year 13 solute concentrations decreased. Fourth, by year 27 leachate concentrations had apparently stabilized at values about ten times lower than the highest concentrations recorded. The gap in data from year 27 to 34 obscures the exact change of the decreasing solute concentrations. However, it does appear that between years 13 and 27, the decrease in concentration was much greater than the interval from years 27 to 38. The single leachate measurement from year 27 (i.e., 2005) suggest the rapid decrease in leachate solute concentrations may have ended around that time. This generalized solute concentration trend indicates a relatively short lived large solute release period followed by a decreasing solute release rate. The following text describes the time series solute concentrations and other measured leachate parameters in greater detail, though the general trend indicated above remains true for most analytes.

4.2.1 DCLP Leachate pH and Specific Conductivity

The DCLP leachate pH trend is complicated by the history of the placement of the rock comprising the DCLP. Initially, leachate pH for the DCLP composite (calculated from the Amax field study data) was circumneutral, though steadily decreasing, for about one year (fig. 4.2). By year two, leachate pH was about five and steadily decreased to about 4.3 by year 15. Leachate pH was again circumneutral for about two years after the DCLP was constructed (fig. 4.2). Whereas the original pH decrease was due to the development of acidic drainage from sulfide oxidation, the initial circumneutral leachate pH for the DCLP was likely resultant from disturbance of the original flow paths and incorporation of new mineral surfaces for neutralization of acid and/or development of more fine neutralizing material from breaking up reaction products during transport and construction of the DCLP. The initial large leachate pH for the DCLP could also be from neutralization of the leachate as it infiltrated through the fresh sand layer of the leachate collection system. The Amax field study FL6 and FL5 rock piles had sulfur concentrations of 0.80 and 1.41 wt%, respectively, and the DCLP has a rock volume weighted sulfur concentration of 0.91

wt%. Based on rock sulfur concentration the leachate pH of the composite should be similar to the FL6 rock pile leachate pH values (fig. 4.3).

Leachate specific conductivity was generally between 2,000 and 4,000 $\mu S \cdot cm^{-1}$ for the first 15 years (fig. 4.4). After construction of the DCLP the maximum specific conductivity decreased from about 4,000 to 1,500 $\mu S \cdot cm^{-1}$. From years 34 to 38 specific conductivity has varied from 350 to 900 $\mu S \cdot cm^{-1}$.

4.2.2 DCLP Leachate Sulfate and Major Cation Concentration Trends

The DCLP leachate sulfate concentrations have ranged from about 200 to 4,000 mg/L (fig. 4.5). Leachate calcium and magnesium concentrations are roughly similar, although for the first two years and since year15, the leachate calcium was slightly greater than magnesium. Collectively, calcium and magnesium leachate concentrations have ranged from about 20 to 600 mg/L over the period of record. Leachate sodium concentrations were initially as large as calcium, but substantially decreased over the first few years in comparison to calcium and magnesium. Leachate sodium concentrations have exhibited the greatest magnitude of change decreasing from about 500 to 5. Potassium was analyzed much less frequently and has shown the most stable decreasing trend starting at about 30 mg/L and decreasing to about 5 mg/L. For the 38 years of record, the leachate concentration of sulfate, calcium, magnesium, sodium and potassium have decreased by factors of 20, 20, 26, 120, and 10, respectively.

4.2.3 DCLP Leachate Nickel, Copper, Cobalt and Zinc Concentration Trends

For the first 25 years of record, leachate metal concentrations have decreased in order from nickel > copper > cobalt > zinc (fig. 4.6). For this period, nickel and copper concentrations ranged from about 10 to 200 mg/L and 10 to 100 mg/L, respectively, and cobalt and nickel ranged from about 0.5 to 10 mg/L. For years 27 to 38, metal concentrations decreased in the order copper > nickel > zinc > cobalt. For this later period copper concentrations ranged between 10 to 30 mg/L, nickel between 4 to 12 mg/L, and zinc and cobalt between about 0.2 to 1 mg/L.

4.2.4 DCLP Leachate Mineral Saturation Indices

The 2012 and 2014 to 2015 leachate data was evaluated by geochemical modeling to assess the leachate charge imbalance and mineral saturation indices. Geochemical modeling was performed using the Geochemists Workbench software with the packaged MINTEQ database. Charge imbalance error ranged from −6% to 5% with an absolute average of 1.9%. Super saturation of many manganese phases indicates that the manganese in solution is metastable and in time would precipitate. The leachate was also supersaturated with respect to silicon oxide phases (such as chalcedony and quartz), aluminum oxide-hydroxide phases (such as gibbsite and diaspore), iron oxide-hydroxide phases (such as ferrihydrite and goethite), and barite. The leachate was under saturated with respect to gypsum exhibiting a log(Q/K) between −1.0 and −1.7.

4.3 Comparison Between Field and Laboratory Scale Rock Weathering Leachate Trends

Both field and laboratory scale rock weathering experiments exhibit the same overall leachate concentration trend of initially large solute concentrations that decrease over time. However, the period of time over which the concentration changes occur is drastically different, as is the magnitude of the release rate. For example, humidity cell sulfate release rates decrease by a factor of about two in one year, whereas the DCLP sulfate release rate was relatively constant for about 15 years (fig. 4.7). Some of the possible mechanisms for the disparity are likely related to the available reactant load (e.g., sulfate) and accessibility of the reactants to infiltrating water. In the laboratory setting there is a much greater water to rock (W:R) mass ratio, whereas in a field setting the W:R ratio is typically much smaller and is dependent

on precipitation rates. Moreover, the standard humidity cell is completely saturated each week, efficiently removing the reactant load, whereas in a natural setting infiltrating water likely follows preferred channels except for large precipitation events. This flow channelization phenomenon has been supported by modeling exercises which show infiltration in a field setting develops channelized flow paths that slowly expand over time (Fala et al., 2005). Therefore, field rock weathering experiments exhibit a slower depletion of the available reactant load albeit at a greater leachate concentration (due mainly to the W:R ratio) than laboratory experiments.

4.4 Scanning Electron Microscopy Sulfide Grain Size Distribution

All sulfide grains had sizes in the range of 2.4 to 1,200 micrometers. In general, the size distributions of each sulfide mineral were similar. The one exception was chalcopyrite which had fewer large sized particles, particularly in the 0.5-2.0 mm interval, than the rest of sulfides (table 4.6).

The 2.0-6.35 size fraction did not contain the largest sulfide minerals. Instead, the 0.5-2.0 mm size interval contained the largest sulfide minerals. This data is in contrast to visual inspection of the crushed sample where numerous sulfide grains were observed in sizes greater than 1,200 microns. It is likely that the limited sample size for the coarsest particle size fraction resulted in an inaccurate determination of the actual sulfide grain size distribution. Because of this apparent sample size limitation, the true sulfide grain size distribution for the 2.0-6.35 mm size interval may not have been determined. However, it is possible the larger sulfide grains may have been preferentially concentrated in the 0.5-2.0 mm size interval from the crushing process. Enrichment of the sulfide in the 0.5-2.0 mm fraction is supported by the greater sulfur concentration in that fraction (1.22 wt%) compared to the 2.0-6.35 mm fraction (1.05 wt%) which was less than the bulk sulfur concentration (1.24 wt%). It is not clear how much of an effect the sample size limitation for the coarsest fraction impacted the measured sulfide grain size distribution.

4.5 Sulfur Release Models

Two separate models, one empirical and the other theoretical, were developed to evaluate the rate at which sulfur was removed from the DCLP. The empirical model was constructed from leachate composition and volume data collected over a 38 year period. This model numerically describes the average annual amount of sulfur removed from a naturally weathering system and provides a basis for evaluating theoretical models.

The application of a theoretically based sulfide oxidation model has an extensive history of use in the fields of metallurgy (e.g., Levenspiel, 1972) and environmental science (Davis and Ritchie, 1986; Davis and Ritchie, 1987; Scharer et al., 1995; Wunderly et al., 1996; and Huminicki and Rimstidt, 2009). The most common theoretical models, employed to predict sulfide oxidation of natural systems, include the shrinking core and the shrinking particle models. In the environmental sciences, the shrinking core model is more commonly used because it accounts for the development of a reaction product coating on the surface of sulfide grains. For the shrinking core model, the rate of the oxidation reaction is limited by oxygen diffusion through the coating (e.g., an iron oxide coating or hydrated iron sulfate compounds, etc.). The theoretical model constructed in this report is based on the shrinking particle model as described by Rimstidt (2014). The shrinking particle model was selected because the low pH environment of the DCLP may not consistently result in substantial development of an iron oxide coating (e.g., ferrihydrite) on the sulfide surfaces. Previous SEM analyses and reflected light petrography studies have variably observed the presence of iron oxide coatings on sulfide mineral surfaces (see appendices in Lapakko et al., 2004). Geochemical modeling using the Geochemists Workbench software applying the Minteq thermodynamic database also indicates that ferrihydrite is variably saturated in the DCLP leachate (assuming one half the reporting limit for iron (0.005 mg/kg)). Regardless of whether or not iron oxide coatings are precipitating on the sulfide mineral surfaces, this study applies the shrinking particle model

as a simplified approach to assess how well a theoretical model, with thoroughly characterized rock material, compares to the empirically determined rate of sulfur removal.

4.5.1 Empirical Model

The empirical model incorporated the annual sulfate release rates reported in the Amax field study (Lapakko et al., 2004) and the more recent DCLP data set (Appendix 1). The empirical model was constructed in two parts due to the change in sulfate release rate over time. The first part is characterized by a linear regression model and the second as a power function regression model using Microsoft Excel software. The coefficients of determination (i.e., R^2 values) for the linear and power function regressions are 0.13 and 0.84, respectively. The p-value for the linear regression is 0.22, indicating a 78 percent confidence interval. For a single year, the average annual release rate was weighted by volume from each of the three Amax field study rock piles that the DCLP was constructed from. Annual release rates from years 1980, 1982, and 1985 were not included in the model due to either very low annual rainfall volume (1980) or an extremely elevated rainfall volume (1982 and 1985). The remaining 13 years of data were incorporated into a linear regression model to develop a mathematical expression for sulfur removal with time. Beginning in 1996, annual average sulfate release rates were calculated from the DCLP leachate composition and recorded leaching volume. Leachate composition and flow volume data is only available for 1996-1998, 2002, 2012, and 2015. In 2002, there was a substantially greater amount of rain fall during the late summer months (July through August) during which only one volume (80,345 L) and leachate sample was collected. Because such a large volume of leachate is represented by only one sulfate concentration measurement it is quite possible that the single value is not representative of the entire volume and only the last amount of leachate that passed through the system is represented. Therefore, this single sample event in 2002 was excluded to avoid misrepresentation of the data. The exclusion of data from 1980, 1982, 1985, and 2002 offers an empirical model that reflects more normal conditions and provides a consistent baseline model for comparison to the theoretical model which, likewise, does not include extreme events. The empirical model with regression line formulas is shown in figure 4.8. The data points used in the regression models are listed on table 4.8.

4.5.2 Theoretical Model

The theoretical model was constructed using physical and chemical data from a 14.7 kg composite rock sample collected from the DCLP that is described in section 3.3. The sole exception is that the rock pile volume weighted average sulfur content from the three original Amax field study rock piles was used (0.91 wt% sulfur) instead of the composite sample sulfur concentration (1.24 wt%). The volume weighted average incorporated 133 sulfur analyses from 'blast rounds' during the original excavation of the rock (Eger et al., 1980). This rock data was then incorporated into a shrinking particle model using a pyrrhotite oxidation rate determined from a laboratory batch flow reactor experiment. Pyrrhotite oxidation rates, like all mineral reactions, are strongly surface area dependent (eq. 2.1). Therefore, the total amount of sulfide surface area for the DCLP needed to be determined. For a mass of rock of different size particles both the sulfide grain size distribution and number of sulfide grains must be known to apply the shrinking particle model. For the theoretical model the sulfide grain size measurements were assumed to represent the diameter of a sphere. The number of sulfide spheres available for oxidation within a specific rock particle size interval (i) can be calculated using equation 4.1. The SA_{exp} and V_{sphere_i} parameters of eq. 4.1 were determined by SEM MLA methods described in section 4.1. The decision to only use the sulfide grains that are greater than 80% exposed was based on inspection of the MLA data on table 4.7. This table shows that there is a relatively small percentage of sulfide grains in the 30% to 80 % surface exposure ranges and only the two largest particle size intervals had a large amount of sulfides in the less than 30% exposed surface area range. Therefore, using only the fraction of grains that were greater than 80% surface area exposed incorporates the vast majority of the total sulfide surface area. Parameters and calculated values for eq. 4.1 are listed in table 4.9.

$$\#\ sulfide\ spheres_i = M_i \times C_i \times \frac{Po_{mm}}{S_{mm}} \div\ Po_{MW} \times V_{m_{Po}} \times\ SA_{exp} \div V_{sphere_i} \qquad \text{eq. 4.1}$$

M_i= mass of particle size interval (kg)
C_i= rock sulfur concentration as g_{sulfur} /10 kg_{rock}
$\frac{Po_{mm}}{S_{mm}}$= pyrrhotite / sulfur molecular mass ratio
Po_{MW}= pyrrhotite molecular weight (82.3255 g/mole)
$V_{m_{Po}}$= pyrrhotite molecular volume (1.82×10^{-5} m^3/mole)
SA_{exp}= MLA determined sulfur wt% of >80 % surface area exposed
V_{sphere_i}= volume of average sulfide grain; P50 value from pyrrhotite MLA data

Once the number of sulfide spheres for each rock particle size fraction is determined, the shrinking particle model (in Rimstidt (2014); eq. 4.2) can be applied to calculate the amount of sulfur oxidized over time. The surface area normalized pyrrhotite oxidation rate of Bilenker et al. (2015; eq. 4.3) was used for the theoretical predictions assuming an atmosphere equilibrated fluid (oxygen saturated) with a pH of 4. This pyrrhotite oxidation rate was determined for sulfide oxidation in seawater, but is similar to other measured rates determined for specific pH values (fig. 4.9). The rate law of Bilenker et al. (2015) was selected because it was measured at a relatively constant temperature (about 22° Celsius) over a range of different fluid pH (2 to 6) and oxygen partial pressures (0.01 to 0.995 atm) values to fully characterize the pyrrhotite oxidation rate dependence on fluid pH and oxygen concentration (Romano, 2012). Other published pyrrhotite oxidation rates were measured for individual fluid pH values, used chelating agents, and/or did not specify the dissolved oxygen concentration (see references on fig. 4.9). The amount of sulfur oxidized at any given time can be calculated by the sphere volume calculated from the difference between D_t and D_o for all of the sulfide grains.

$$D_t = D_o - 2 \times k_{Po} \times V_{m_{Po}} \times t \qquad \text{eq. 4.2}$$

$$k_{Po} = -10^{-7.27} \times m_{O_2}^{0.30\pm0.07} \times m_{H+}^{0.08\pm0.03} \qquad \text{eq. 4.3}$$

D_t= grain diameter at time t
D_o=initial grain diameter
k_{Po}= surface area normalized pyrrhotite oxidation rate ($mol\cdot m^{-2}\cdot s^{-1}$)
t= time (s)
m_{O_2}= molality aqueous oxygen
m_{H+}= molality of hydronium ion

Two separate theoretical model calculations were performed. One used the DCLP volume weighted average of particle size interval masses from the FL1, FL5, and FL6 Amax field study rock piles and the other used the particle size interval masses for an individual sample collected from the FL4 rock pile in 1982, which had a greater amount of fine particles (see table 4.9). Figure 4.10 shows the theoretical model results as the cumulative amount of sulfur removed and the calculated values are listed in table 4.10. The early large slope is due to the greater specific surface area ($m^2\cdot kg^{-1}$) of the fine fractions resulting in large amounts of sulfide being oxidized early in the system. Table 4.10 lists the percentages of sulfur removed at each one year interval relative to the total amount removed at year 38. Figure 4.11 shows the time at which sulfide grains from each of the rock particle size intervals have been entirely reacted. The early large sulfur removal slope, relative high percentages of sulfur removal, and theoretical indication of the fine sulfides being exhausted by year 15 clearly indicate that the initial removal of

sulfide is predominantly from the fine sulfide grains. After the fine sulfide grains have been exhausted, the rate at which sulfur is removed decreases (fig. 4.10).

4.5.3 Model Comparison

Figure 4.12 shows the sulfur removal trends for both the empirical and theoretical models. The theoretical models both under predict the removal of sulfur by year 12. The theoretical model incorporating a greater amount of fines over estimates the empirical model for the first 11 years whereas the other theoretical model very closely agrees with the empirical model for about the first five years. Figure 4.12 shows the relative percent difference of the theoretical models from the empirical model illustrating that both models are within a factor of 2 from the empirical model and that the greater fines example under predicts by about 30% after all of the finest grained sulfides have been removed.

5.0 DISCUSSION

Developing and comparing the two different model types has afforded the opportunity to elucidate the governing physical and hydrogeochemical parameters and determine how these parameters influence model results. Although the large copper and nickel concentrations of the rock modeled in this study may be considered ore by current standards, there are existing naturally weathering Duluth Complex waste rock piles that have similar copper, nickel and sulfur concentration (e.g., Dunka Mine and historic INCO shaft site). The results of this study are important for understanding weathering processes at those locations as well as assessing the general reactivity of potential future mine wastes with sulfur concentrations similar to the values modeled in this report. The following discussion describes the sensitivity of the empirical and theoretical models to some parameters, provides a general assessment of the modeling approach, and evaluates how the theoretical model could be improved.

5.1 Model Sensitivity

A number of studies have highlighted the importance of the smaller sized particles (namely sulfides) governing the leachate composition for rock weathering at the laboratory and field scales (Wenz et al., 2013; Lapakko et al., 2006; Stromberg and Banwart, 1999 and Doepker, 1994). Theoretically, the sulfide reaction rate is strongly dependent on the amount of surface area available for reaction (eq. 2.1). Applying basic geometry and a specified grain diameter and mineral density the total surface area of a mass can be calculated. An example of this calculation for 1 gram of pyrrhotite for different grain diameters is shown in table 4.11. This calculation shows the orders of magnitude increase in surface area for decreasing sulfide grain size. Relating the surface area data from table 4.11 to eq. 2.1 shows that the rate of pyrrhotite reaction increases by orders of magnitude from sub-millimeter to millimeter sized grains. This mathematical exercise clearly demonstrates that accurately constraining the amount and size distribution of the smallest sulfide grains is very important for modeling the release of sulfur from a mass of weathering rock.

To assess the sensitivity of the theoretical model used in this study the amount of the smaller sized rock particle intervals was increased using the greatest amount of fines measured from sieve analyses of the FL4 rock pile from the Amax field study (Lapakko et al., 2004). This resulted in substantially increasing the number of pyrrhotite spheres in the theoretical model (table 4.9). Comparison between the two different theoretical models showed that the model is very sensitive to small changes in the amount of small sized sulfide grains. Changing the mass fraction of the four smallest particle size intervals (by a combined amount of 0.037) increased the total amount of sulfur removed by about 50%. Therefore, a very accurate representation of the rock particle size distribution is necessary for applying the shrinking particle model.

For this data set, the MLA sulfide mineral exposure data was only valid for the fractions less than 2 mm because the required MLA sample size is too small to incorporate enough grains to accurately represent large size fraction intervals. To assess whether or not the 2.0-6.35 mm size fraction could substantially impact the overall model results, the theoretical model was extended to include the 2.0-6.35 mm particle size interval. This model extension assumed 50% of the sulfide grains were available and had an average diameter of 3.765 mm. For the extended model the total mass of sulfur removed at year 38 would only increase by 21 kg or about 4%. This demonstrated that even the 2.0-6.35 mm size fraction has little impact to the overall model results.

It is not clear how much of an impact rock particles greater than 6.35 mm would have on the overall model. Although the greater than 6.35 mm particles would have a relatively small sulfide surf ace area the 6.35-38.1 mm size fraction accounts for 37% of the total rock pile mass. Thus, despite the relatively small SSA of the larger particles the large percentage of mass for the 6.35-38.1 mm size range could influence the model results. The larger size fractions would not contribute much of the total sulfur removed early, but more likely would be responsible for the long term release of sulfate from the system. Including the greater than 2 mm sized particles into the theoretical model was not possible for this modeling exercise because of the limitation of the sulfide grain size measurements at about 1 mm. The sulfide grain size limitation is inherent to MLA analysis due to sample size requirements. A more complete sulfide grain size distribution would likely improve the theoretical model results.

The empirical model represents the average sulfate release rate from 1978 to 1993, excluding anomalous low and high annual precipitation years. Comparison among the actual annual sulfate release rates show up to a factor of three variation has occurred (table 4.8; years 1980 and 1982). Thus, in applying the type of modeling presented in this report for regulatory purposes, the effect of variable precipitation rates must be considered.

5.2 Model Interpretation

The difference in sulfur removal trends between the empirical and theoretical models for about the first 20 years indicates that the theoretical model is not accounting for some of the physical and/or hydrogeochemical mechanisms early in the weathering history (fig. 4.12). However, the general agreement in slope between the empirical and theoretical models after about 20 years indicates that the theoretical model may be accurately characterizing the various hydrogeochemical mechanisms. Some of the more obvious factors responsible for the disagreement include the 1) accuracy of the exposed sulfide surface area calculations, 2) efficiency of sulfide oxidation products removal, and 3) representativeness of the shrinking particle model.

The accuracy of the exposed sulfide surface area calculations is dependent on the mass of the particle size intervals and representativeness of the MLA determined sulfide surface area measurements. Figure 4.11 shows that at about year 15 the smallest sized sulfide particles (less than 0.053 micron particle fraction) are entirely reacted.), Based on comparison to the empirical model, this may indicate that the amount of smaller sized particles used in the theoretical model is wrong because a greater amount of fine material would have raised the magnitude of the sulfide removal trend up to the level of the empirical model. Alternatively, not accounting for the partially exposed sulfide grains may have resulted in a greater amount of sulfide removal than was accounted for in the theoretical model.

The difference in sulfur removal trends may also be due to differences in the efficiency of sulfide oxidation product removal (e.g., sulfate). The theoretical model calculation sums the mass of reacted material at each time step, whereas in a natural setting the sulfate that is transported in the leachate may not represent the entire mass of reacted material with time because it could be stored in the waste rock pile. In effect, the overall removal of sulfur via sulfide oxidation is slowed because infiltrating water does

not collect all of the reactant products. Considering the likely possibility of sulfur storage and limited water contact, the theoretical model which incorporated a greater amount of fines is more likely because the overall greater mass and rate of sulfur removal should exceed that of the empirical model that inherently contains reaction product release limitations.

The application of the shrinking particle model is a reasonable first approximation. However, it is likely that some reaction products have precipitated on some of the mineral surfaces. The extent to which the coating may limit the rate of sulfide oxidation depends on the specific type (composition and morphology) of coating that develops. Numerous investigators have reported accumulation of sulfide compounds on the pyrrhotite mineral surface during oxidation (e.g., Buckley and Woods, 1985 and Steger, 1982). However, it is unclear if these reaction products are precipitates or if they represent specific half reactions for the overall oxidation reaction (i.e., rxn. 2.1). Furthermore, applying the shrinking core model requires knowledge of the diffusion rate of oxygen through the developed coating. Although some studies have calculated the diffusion rate for coatings on pyrite (e.g., Huminicki and Rimstidt, 2009 and Jerz and Rimstidt, 2004), there are no available oxygen diffusion rates for reaction product coatings on pyrrhotite.

Ultimately, the differences between the empirical and theoretical models are most likely a combination of factors described in the preceding paragraphs. Determining the proportion of which is responsible for the model differences is not possible without a more robust characterization of the system. For example, using the shrinking particle model instead of the shrinking core model may have resulted in a greater amount of sulfur removal for the theoretical model whereas the actual greater release in the empirical model was due to oxidation of the partially exposed sulfide grains that were not accounted for in the theoretical model. Furthermore, storage of sulfate in the rock pile may also account for a lower measured sulfur removal compared to the theoretical model which accounted for the total mass of reacted sulfur.

After about year 20, all three of the models generally exhibit the same sulfur removal trend (fig. 4.12). This implies that the remaining sulfide amount and size distributions are similar among the models. Thus, the overall lower magnitude of sulfur removed in the theoretical models is likely due to either inaccurate measurements of rock particle and sulfide mineral grain abundance and size distribution, or some other unidentified parameter that accelerated the rate of sulfur removal in the field.

5.3 Model Improvement

The close agreement between the empirical and theoretical model results demonstrates that relatively accurate theoretical prediction models can be developed with minimal data input. Despite the simplifying assumptions (e.g., constant fluid pH, all sulfur in the form of pyrrhotite, etc.) incorporated into the theoretical model, there was less than a factor of two difference between the empirical and theoretical models for the 38 year record. Although a factor of two difference is generally not considered very accurate, in terms of evaluating mine waste management options that level of accuracy can guide mine waste management decisions. Decreasing the variability and uncertainty of the input parameters and incorporation of additional hydrogeochemical parameters will likely improve the predictive capability of the theoretical approach.

This modeling exercise has indicated that the shrinking particle model can be a feasible approach for predicting sulfur release from waste rock with a small leachate pH if the physical parameters (e.g., particle size distribution, sulfide grain size, and amount of sulfide grain surface exposure) of the system are accurately characterized. For circumneutral pH environments the shrinking particle model is less likely to be applicable because ferrihydrite is not soluble and iron-oxide rims would most likely develop on the sulfide surfaces. Accurately quantifying the amount of the small sized rock particles is difficult due to the small sample size typically employed for sieve analysis. This theoretical model incorporated three

particle size interval wt% measurements from the Amax field study. Increasing the number of weighted particle size interval measurements would likely improve the accuracy of the model. Another important parameter is the amount of sulfide surface area exposed. In this exercise only the material less than 2 mm was used to calculate the total mass of sulfur removed. The 2 mm cut off was based on the MLA sulfide exposure values of the less than 6.35 mm material. However, both stereo microscope inspection of the greater than 2 mm sized material used for MLA and field observation of the DCLP qualitatively revealed that millimeter sized sulfide grains are more common than that determined by the MLA analysis. Further refinement of this modeling approach should include accounting for the additional sulfide surface area on the greater than 2 mm sized material. Of particular importance is accurately accounting for all of the sulfide that is readily available for oxidation during the first few years of weathering.

The theoretical model used in this report applied a stepped size interval approach to apply the shrinking particle model in which the average (P50) sulfide size was applied for each size interval. Further refinement could be incorporated by applying the actual MLA determined sulfide grain size distribution for each of the particle size intervals. This was not possible with the existing data because the wt% of exposed sulfide surface area was determined for each of the rock particle size intervals and not the sulfide grain size distribution.

Petrographic analysis of the fine materials would provide visual evidence of whether or not reaction products are coating the different sulfide minerals. In addition, petrography would also provide information on the competency of the surface coatings. Assessing the competency of the coatings could indicate if there is a diffusion limitation or if there are fractures in the coating and parting along the coating and sulfide interface that may be more accurately represented by a shrinking particle model.

Determining the relationship among infiltration volume, leachate concentration, and time between precipitation events could inform the type and value of hydrologic parameters that could be incorporated into the theoretical model. Evaluating this relationship could be accomplished by more intensive monitoring of the leachate volume and greater frequency of leachate chemical analysis.

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Table 4.1 Bulk and particle size interval XRD and MLA mineralogy. Values are in as weight percent.

	XRD	Mineral Liberation Analysis					
Mineral (Wt%)	Bulk	Bulk	2.0-6.35	0.5-2.0	0.149-0.5	0.053-0.149	< 0.053
Amorphous	12.9	--	--	--	--	--	--
Amphibole	--	14.24	10.64	9.86	10.15	13.12	16.97
Apatite Cl	--	0.25	0.51	0.32	0.53	0.93	1.1
Augite (Cpx)	7.5	8.46	6.18	15.18	8.87	7.05	7.42
Baddeleyite	--	--	--	--	--	--	0.04
Biotite	2.7	3.28	1.83	1.54	2.57	1.96	1.04
Chlorite	--	1.24	0.53	1.6	1.04	1.11	1.36
Cordierite	2.2	--	--	--	--	--	--
Enstatite (Opx)	6.2	20.52	15.44	12.08	16.25	15.72	13.68
Epidote	--	0.43	0.14	0.38	0.32	0.19	0.4
Fe Silicate clay	--	--	--	0.01	0.05	0.21	0.2
K-feldspar	--	0.28	1.65	0.48	0.47	0.43	0.46
Muscovite	--	0.09	0.07	0.05	0.08	0.05	0.07
Olivine (Forsterite)	8.9	--	--	--	--	--	--
Plagioclase	55.5	40.79	54.18	44.36	38.2	37.42	32.81
Quartz	--	--	--	0.07	0.21	0.22	0.38
Smectite	--	0.04	0.01	0.06	0.04	0.04	0.06
Talc	--	0.01	--	--	0.08	0.05	0.02
Zircon	--	0.01	0.03	--	--	--	0.03
Fe (O,OH)	--	0.05	--	0.07	0.66	0.63	1.49
Ilmenite	2.6	7.25	7.59	6.54	12.33	9.3	8.3
Calcite	--	0.02	--	0.01	0.04	0.05	0.06
Siderite	--	0.06	0.02	0.05	0.28	0.34	0.62
Bornite	--	0.01	0.03	--	--	0.02	0.04
Chalcopyrite	0.5	0.52	0.39	1.66	0.77	1.06	2.17
Cubanite	1	1.05	0.42	3.57	3.36	5.8	6.81
Pentlandite	--	0.17	0.02	1.62	1.36	1.12	1.61
Pyrrhotite	--	0.74	0.06	0.13	1.88	2.5	1.7
FeCuNi Sulphate	--	0.06	0.01	0.07	0.09	0.18	0.62
Others	--	0.44	0.23	0.29	0.39	0.51	0.54
Total	100	100	100	100	100	100	100

Table 4.2 Bulk sample major oxide concentrations.

Size Interval (mm)	SiO_2 wt%	Al_2O_3 wt%	Fe_2O_3 wt%	MnO wt%	MgO wt%	CaO wt%	Na_2O wt%	K_2O wt%	TiO_2 wt%	P_2O_5 wt%	LOI	Total
2.0-6.35 mm	44.66	15.94	16.97	0.169	6.83	8.76	2.26	0.51	2.268	0.25	-0.27	98.35
0.5-2.0 mm	45.06	15.62	17.17	0.171	6.75	8.8	2.29	0.5	2.317	0.15	-0.24	98.57
0.149-0.5	43.85	16.15	17.33	0.167	6.2	8.46	2.36	0.56	2.82	0.2	-0.19	97.89
0.053-0.149	43.22	15.38	17.99	0.178	6.33	8.18	2.26	0.53	2.886	0.37	0.19	97.51
< 0.053	42.83	15.68	18.03	0.185	6.35	8.22	2.17	0.41	3.086	0.5	0.86	98.31
Bulk	45.01	16.02	17.25	0.171	6.82	8.66	2.24	0.51	2.37	0.19	-0.18	99.06

Size interval (mm)	SiO_2 wt%*	Al_2O_3 wt%*	Fe_2O_3 wt%*	MnO wt%*	MgO wt%*	CaO wt%*	Na_2O wt%*	K_2O wt%*	TiO_2 wt%*	P_2O_5 wt%*
2.0-6.35 mm	44.93	16.72	17.01	0.17	7.06	8.95	--	0.48	2.30	0.18
0.5-2.0 mm	44.50	16.67	17.01	0.17	6.92	8.88	--	0.48	2.32	0.15
0.149-0.5 mm	43.64	16.72	17.59	0.18	6.38	8.72	--	0.48	2.84	0.20
0.053-0.149 mm	43.21	16.17	18.73	0.18	6.58	8.63	--	0.48	2.94	0.36
< 0.053 mm	42.14	15.44	20.16	0.20	6.67	8.55	--	0.36	3.00	0.50
Bulk	45.35	16.63	17.16	0.18	7.08	8.97	--	0.60	2.42	0.19

* Indicates oxide values calculated from the elemental wt% values determined from sodium peroxide fusion and ICP-MS analysis.

Table 4.3 Bulk sample and particle size interval total sulfur, carbon, chloride, fluoride, and bromide concentrations.

Size Interval (mm)	C_T wt%[1]	S_T wt%[1]	S_T wt%*	Cl wt%	F wt%	Br (ppm)	C_T wt%[2]	S_T wt%[2]	C_T wt%[3]	S_T wt%[3]
2.0-6.35	0.03	1.07	1.04	0.04	< 0.01	< 0.5	0.04	1.02	0.04	1.05
0.5-2.0	0.03	1.28	1.23	0.06	< 0.01	< 0.5	0.05	1.15	0.04	1.22
0.149-0.5	0.04	1.68	1.63	0.03	< 0.01	< 0.5	0.04	1.83	0.04	1.76
0.053-0.149	0.07	1.92	1.89	0.08	< 0.01	< 0.5	0.06	1.92	0.06	1.92
< 0.053	0.12	2.31	2.39	0.04	< 0.01	< 0.5	0.08	2.33	0.10	2.32
Bulk	0.03	1.22	1.22	0.04	< 0.01	< 0.5	0.03	1.25	0.03	1.24

* Values determined from sodium peroxide fusion and ICP-MS analysis.
[1] Analyses performed by Actlabs.
[2] Analyses performed by Lerch Brothers Inc. Hibbing, MN
[3] Average values of the sulfur and carbon combustion analyses.

Table 4.4 Bulk and particle size interval trace metal and nonmetal analyses. Concentrations are in parts per million.

Size Interval (mm)	Cu	Ni	V	V*	Sr	Sr*	Ba	Ba*	Zn	Cr	Co	Zr*	B	Li	Se	Ga	Pb	Rb
2.0-6.35	6320	1100	349	254	295	264	220	199	170	160	106	116	60	12	19.9	22.6	14	13.5
0.5-2.0	6780	1390	325	247	291	265	298	279	220	190	121	95	80	12	11.9	21.8	18.7	13.4
0.149-0.5	8100	2110	356	259	300	266	208	200	200	170	154	131	100	12	27.3	23.3	24.8	15.3
0.053-0.149	9610	2220	350	273	286	255	568	493	410	170	159	182	90	19	18	21.6	205	13.7
< 0.053	>10000	2950	365	271	277	254	163	153	230	180	199	224	110	11	17	21.2	33.6	9.9
Bulk	6630	1420	346	258	283	264	187	183	170	150	122	107	90	30	25.6	21.6	16.1	14

Size Interval (mm)	Nb	Ge	Mo	Cs	Sn	Ta	Hf	Te	As	Bi	Cd	Sb	Be	Be*	W	In	Tl
2.0-6.35	11.6	5.4	5	0.9	< 0.5	0.7	< 10	< 6	7	< 2	< 2	< 2	< 4	1	< 0.7	< 0.2	< 0.1
0.5-2.0	9.9	4.2	1	0.9	1.2	0.5	< 10	< 6	82	< 2	< 2	< 2	< 4	< 1	< 0.7	< 0.2	< 0.1
0.149-0.5	15.5	4.2	3	0.8	5.1	1.2	< 10	< 6	< 5	< 2	< 2	< 2	< 4	< 1	< 0.7	< 0.2	< 0.1
0.053-0.149	12.7	5.5	2	1	19.2	0.6	< 10	< 6	16	< 2	< 2	< 2	< 4	< 1	< 0.7	< 0.2	< 0.1
< 0.053	13.9	5.2	3	0.7	91.8	0.7	< 10	< 6	11	< 2	< 2	< 2	< 4	< 1	1	< 0.2	< 0.1
Bulk	10.5	5.2	3	1	0.5	0.3	< 10	< 6	< 5	< 2	< 2	< 2	< 4	< 1	< 0.7	< 0.2	< 0.1

Size Interval (mm)	Sc*	Y	Y*	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Th	U
2.0-6.35	20	20.8	16	15.1	38.8	4.7	19.5	4.5	1.8	4.6	0.6	4	0.8	2.3	0.3	2	2.9	0.7
0.5-2.0	21	18.5	15	11.9	25.6	3.3	14.7	3.6	1.7	4	0.6	3.6	0.7	2	0.3	2.1	1.3	0.2
0.149-0.5	20	20.3	17	127	344	24.7	111	9.8	1.8	7	0.7	4	0.8	2.1	0.3	1.9	216	4.6
0.053-0.149	19	26.2	21	18.4	42.4	5.6	24.4	5.7	1.8	5.9	0.9	5	1	2.7	0.4	2.4	2.2	0.6
< 0.053	19	32.6	22	22.1	52	6.8	31.7	7.2	1.7	7.7	1.1	6.1	1.1	3.1	0.4	2.5	3.2	1.1
Bulk	21	20.2	16	14.6	33.1	4.1	18.8	4.3	1.8	4.4	0.6	3.9	0.8	2.2	0.3	2	2.2	0.4

* Lithium metaborate-tetraborate fusion and ICP-MS analysis.

Table 4.5 Mass weighted particle size sieve analysis.

Size Interval (mm)	Lerch Bros.				Actlabs		Average
	mass (g) dry	mass (g) wet	wt% dry	wt% wet	mass (g) dry	wt% dry	wt% dry
2.0-6.35	120	120.4	60	60.2	95	58.64	59.32
0.5-2.0	49	49.6	24.5	24.8	39	24.07	24.29
0.149-0.5	17.7	17.3	8.85	8.65	17	10.49	9.67
0.053-0.149	8.8	8	4.4	4	8	4.94	4.67
< 0.053	4.5	4.7	2.25	2.35	3	1.85	2.05
total=	200	200	100	100	162	100	100

Table 4.6 MLA determined sulfide grain size distribution. Values are in wt% passing.

	Cubanite					Pentlandite				
	Particle Size Interval (mm)					Particle Size Interval (mm)				
Grain Size (µm)	2.0-6.35	0.5-2.0	0.149-0.5	0.053-0.149	< 0.053	2.0-6.35	0.5-2.0	0.149-0.5	0.053-0.149	< 0.053
1.75	0	0	0	0	0	0	0	0	0	0
2.4	0	0	0	0	0.01	0	0	0	0	0.01
3.4	0	0	0.01	0	0.02	0	0	0	0	0.03
4.8	0.25	0.04	0.07	0.05	0.1	0.63	0.22	0.11	0.05	0.08
6.8	0.87	0.12	0.16	0.15	0.33	1.57	0.59	0.31	0.2	0.29
9.6	2.25	0.28	0.36	0.33	0.8	3.12	1.17	0.58	0.35	0.71
13.5	4.23	0.55	0.76	0.74	2	9.5	1.91	1	0.69	2.06
19	6.96	0.93	1.35	1.38	5.76	14.75	2.93	1.44	1.06	5.78
27	9.72	1.41	2.41	3.05	20.69	21.12	4.11	1.89	2.66	23.09
38	15.48	2.3	3.62	6.68	50.29	47.67	4.81	2.35	5.72	60.79
53	21.91	3.15	5.24	15.63	87.55	67.53	6.21	2.74	14.17	91.93
75	31.86	4.82	7.81	35.33	100	100	6.8	4.25	29.6	100
106	43.37	6.54	11.55	60.83	0	0	9.05	7.78	53.72	0
150	57.97	10.12	21.43	89.41	0	0	12.66	13.42	80.81	0
212	88.41	15.36	47.12	100	0	0	12.66	28.78	100	0
300	100	20.65	69.24	0	0	0	12.66	51.83	0	0
425	0	32.27	93.57	0	0	0	12.66	70.48	0	0
600	0	44.83	100	0	0	0	12.66	100	0	0
850	0	65.1	0	0	0	0	12.66	0	0	0
1200	0	100	0	0	0	0	100	0	0	0

	Pyrrhotite					Chalcopyrite				
	Particle Size Interval (mm)					Particle Size Interval (mm)				
Grain Size (µm)	2.0-6.35	0.5-2.0	0.149-0.5	0.053-0.149	< 0.053	2.0-6.35	0.5-2.0	0.149-0.5	0.053-0.149	< 0.053
1.75	0	0	0	0	0	0	0	0	0	0
2.4	0	0	0	0	0.01	0	0	0.01	0.01	0.02
3.4	0	0	0	0	0.03	0	0	0.02	0.04	0.04
4.8	0.4	0.04	0.01	0.03	0.09	0.3	0.21	0.14	0.24	0.19
6.8	1.32	0.16	0.03	0.07	0.35	1.92	1.11	0.52	0.75	0.54
9.6	3.9	0.3	0.11	0.16	0.96	5.32	2.77	1.27	1.77	1.46
13.5	9.12	0.68	0.24	0.36	2	10.88	6.25	2.72	3.48	2.95
19	15.74	1.29	0.41	0.75	5.55	18.38	10.04	4.59	6.06	7.07
27	25.52	2.26	0.67	2.07	19.34	26.68	16.41	7.26	10.03	23.76
38	37.83	3.32	0.86	4.38	51.38	38.89	23.1	9.71	14.36	55.1
53	47.83	5.74	1.39	11.6	86.61	45.88	30.94	12.58	27.25	86.21
75	62.44	7.31	2.51	28.61	100	61.88	44.28	16.69	51.28	100
106	100	10.53	4.47	56.16	0	71.5	49.89	23.22	73.14	0
150	0	12.58	10.37	83.46	0	82.08	61.33	35.04	96.74	0
212	0	20.09	19.66	100	0	100	69.07	48.95	100	0
300	0	20.09	51.82	0	0	0	83.02	78.75	0	0
425	0	33.44	79.41	0	0	0	100	100	0	0
600	0	33.44	100	0	0	0	0	0	0	0
850	0	100	0	0	0	0	0	0	0	0

Table 4.7 MLA determined sulfide surface area exposed in wt%.

Pyrrhotite												
Size Interval (mm)	0%	0%-10%	10%-20%	20%-30%	30%-40%	40%-50%	50%-60%	60%-70%	70%-80%	80%-90%	90%-100%	100%
2.0-6.35	4.7	92.97	0	0	0	0	0	0	0	0	0	2.33
0.5-2.0	2.29	11.05	6.63	13.45	0	0	0	0	0	66.57	0	0
0.149-0.5	0.22	1.06	0.72	0.37	3.09	0.72	2.06	9.28	0.2	1.74	41.51	39.01
0.053-0.149	0.11	0.47	0.19	0.8	0.96	1.6	2.02	2.75	3.1	3.36	11.41	73.22
< 0.053	0	0.28	0.42	1.21	1.14	1.53	4.92	4.44	3	5.47	4.02	73.57
Cubanite												
Size Interval (mm)	0%	0%-10%	10%-20%	20%-30%	30%-40%	40%-50%	50%-60%	60%-70%	70%-80%	80%-90%	90%-100%	100%
2.0-6.35	6.02	93.59	0.4	0	0	0	0	0	0	0	0	0
0.5-2.0	0.69	12.26	9.25	5.44	0.79	13.59	2.65	0.1	0	9.7	19.61	25.92
0.149-0.5	0.69	3.12	1.81	2.08	4.05	1.97	3.13	1.97	1.5	6.35	19.7	53.63
0.053-0.149	0.12	0.75	0.47	0.98	1.08	0.95	1.13	1.59	0.73	4.28	5.71	82.21
< 0.053	0.01	0.32	0.61	0.93	1.29	2.36	2.44	3.71	2.97	4.9	5.13	75.34
Chalcopyrite												
Size Interval (mm)	0%	0%-10%	10%-20%	20%-30%	30%-40%	40%-50%	50%-60%	60%-70%	70%-80%	80%-90%	90%-100%	100%
2.0-6.35	16.57	82.94	0	0	0	0	0	0	0	0	0	0.5
0.5-2.0	6.46	59.13	8.34	25.57	0	0	0	0	0	0	0	0.5
0.149-0.5	3.03	11.77	3.41	3.01	1.33	0	0.68	1.52	8.75	6.99	26.97	32.55
0.053-0.149	1.26	5.32	2.7	2.07	3.46	1.6	2.6	1.21	0.51	7.42	2.01	69.86
< 0.053	0.26	0.72	1.63	2.67	1.66	1.87	4.26	3.33	5.9	6.24	2.78	68.66
Pentlandite												
Size Interval (mm)	0%	0%-10%	10%-20%	20%-30%	30%-40%	40%-50%	50%-60%	60%-70%	70%-80%	80%-90%	90%-100%	100%
2.0-6.35	43.28	56.72	0	0	0	0	0	0	0	0	0	0
0.5-2.0	3.25	9.37	0	0	0	0	0	0	0	0	87.39	0
0.149-0.5	0.5	1.53	0.06	3.05	0.44	0.03	0.83	0	2.66	17.63	53.53	19.74
0.053-0.149	0.16	0.7	0.29	0.66	0.2	0.92	0.58	0.99	2.95	13.75	19.49	59.3
< 0.053	0.06	0.37	1.03	1.12	1.21	2.76	1.42	2.22	4.76	6.58	4.79	73.67

Table 4.8 Measured sulfate release rates from the Amax field study composite and the DCLP.

Calendar Year	Model Year	Annual sulfate release rate ($mmol \cdot kg^{-1} \cdot wk^{-1}$)
1978	1	3.33E-02
1979	2	4.74E-02
1980	3	2.75E-02
1981	4	4.41E-02
1982	5	8.39E-02
1983	6	4.77E-02
1984	7	3.75E-02
1985	8	8.14E-02
1986	9	4.49E-02
1987	10	5.28E-02
1988	11	5.27E-02
1989	12	6.00E-02
1990	13	3.85E-02
1991	14	5.67E-02
1992	15	4.02E-02
1993	16	4.68E-02
1996	19	2.62E-02
1997	20	1.41E-02
1998	21	2.14E-02
2002	25	2.28E-02
2012	35	4.72E-03
2015	38	8.61E-03

Table 4.9 Theoretical model sulfide sphere calculations and parameter values for each of five particle size intervals.

Particle Size Interval (mm)	Average sulfide volume (m^3)	DCLP Mass (kg)	DCLP size interval mass fraction	DCLP greater fines mass fraction	sulfur (g/ kg_{rock})	Pyrrhotite / Sulfur molecular mass ratio	Pyrrhotite (m^3/mole)	>80% sulfide surface area available	# Pyrrhotite spheres (from volume weighted size interval mass fraction)	# Pyrrhotite spheres (from greater fines mass fraction)
2.0-6.35	9.33E-14	812,146	0.063	0.077	10.7	2.5675	1.82E-05	0	0	0
0.5-2.0	1.52E-10	812,146	0.031	0.05	12.8	2.5675	1.82E-05	0.50	601,492,692	970,149,503
0.149-0.5	1.34E-11	812,146	0.022	0.031	16.8	2.5675	1.82E-05	0.81	10,265,560,336	14,465,107,747
0.053-0.149	5.09E-13	812,146	0.011	0.015	19.2	2.5675	1.82E-05	0.90	172,101,963,643	234,684,495,877
<0.053	2.77E-14	812,146	0.009	0.014	23.1	2.5675	1.82E-05	0.84	2,910,859,056,830	4,528,002,977,291

Table 4.10 Theoretical calculation of cumulative mass of sulfur removed.

Model year	Mass of sulfur removed (kg)		Percentage of sulfur removed	
	Volume weighted composite	Greater fines	Volume weighted composite	Greater fines
1	46	68	8.8%	8.9%
2	87	130	7.8%	8.1%
3	125	186	7.3%	7.3%
4	159	236	6.5%	6.6%
5	190	281	5.9%	5.9%
6	217	322	5.2%	5.4%
7	242	357	4.8%	4.6%
8	263	389	4.0%	4.2%
9	283	418	3.8%	3.8%
10	301	443	3.4%	3.3%
11	316	466	2.9%	3.0%
12	330	486	2.7%	2.6%
13	343	504	2.5%	2.4%
14	355	521	2.3%	2.2%
15	366	537	2.1%	2.1%
16	377	552	2.1%	2.0%
17	387	566	1.9%	1.8%
18	397	580	1.9%	1.8%
19	406	593	1.7%	1.7%
20	415	606	1.7%	1.7%
21	423	618	1.5%	1.6%
22	432	630	1.7%	1.6%
23	439	641	1.3%	1.4%
24	447	652	1.5%	1.4%
25	454	662	1.3%	1.3%
26	461	672	1.3%	1.3%
27	467	681	1.1%	1.2%
28	473	690	1.1%	1.2%
29	479	699	1.1%	1.2%
30	485	707	1.1%	1.0%
31	490	715	1.0%	1.0%
32	496	723	1.1%	1.0%
33	501	730	1.0%	0.9%
34	505	737	0.8%	0.9%
35	510	744	1.0%	0.9%
36	515	751	1.0%	0.9%
37	519	757	0.8%	0.8%
38	523	763	0.8%	0.8%

Table 4.11 Tabulated surface area for 1 gram of pyrrhotite spheres over a range of diameters.

pyrrhotite diameter (mm)	SA_{sphere} (mm^2)	V_{sphere} (mm^3)	mass of particle (g)	# spheres in 1 gram	total surface area for 1 gram (mm^2)	Relative amount of surface area	SSA_{sphere}* (mm^2/g)
0.001	3.14E-06	5.24E-10	2.41E-12	4.15E+11	1.30E+06	10,000	217,391
0.01	3.14E-04	5.24E-07	2.41E-09	4.15E+08	1.30E+05	1,000	21,739
0.1	3.14E-02	5.24E-04	2.41E-06	4.15E+05	1.30E+04	100	2,174
1	3.14E+00	5.24E-01	2.41E-03	4.15E+02	1.30E+03	10	217
10	3.14E+02	5.24E+02	2.41E+00	4.15E-01	1.30E+02	1	22

* Used a pyrrhotite density of 4.6 $g \cdot cm^{-3}$ (0.0046 $g \cdot mm^{-3}$).

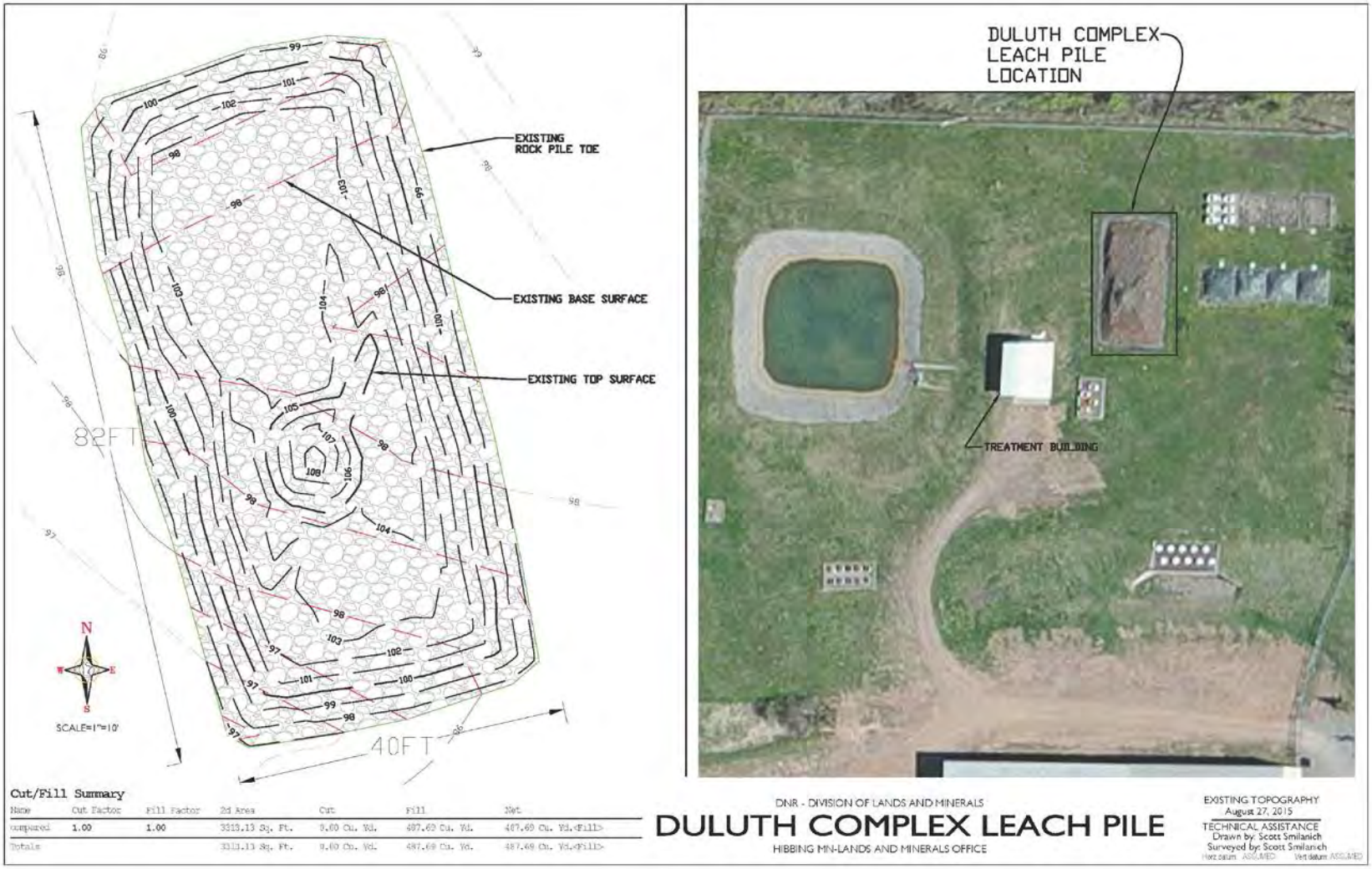

Figure 1.1 Surveyed topography of the DCLP and satellite image of the Hibbing field research site indicating the location of the DCLP.

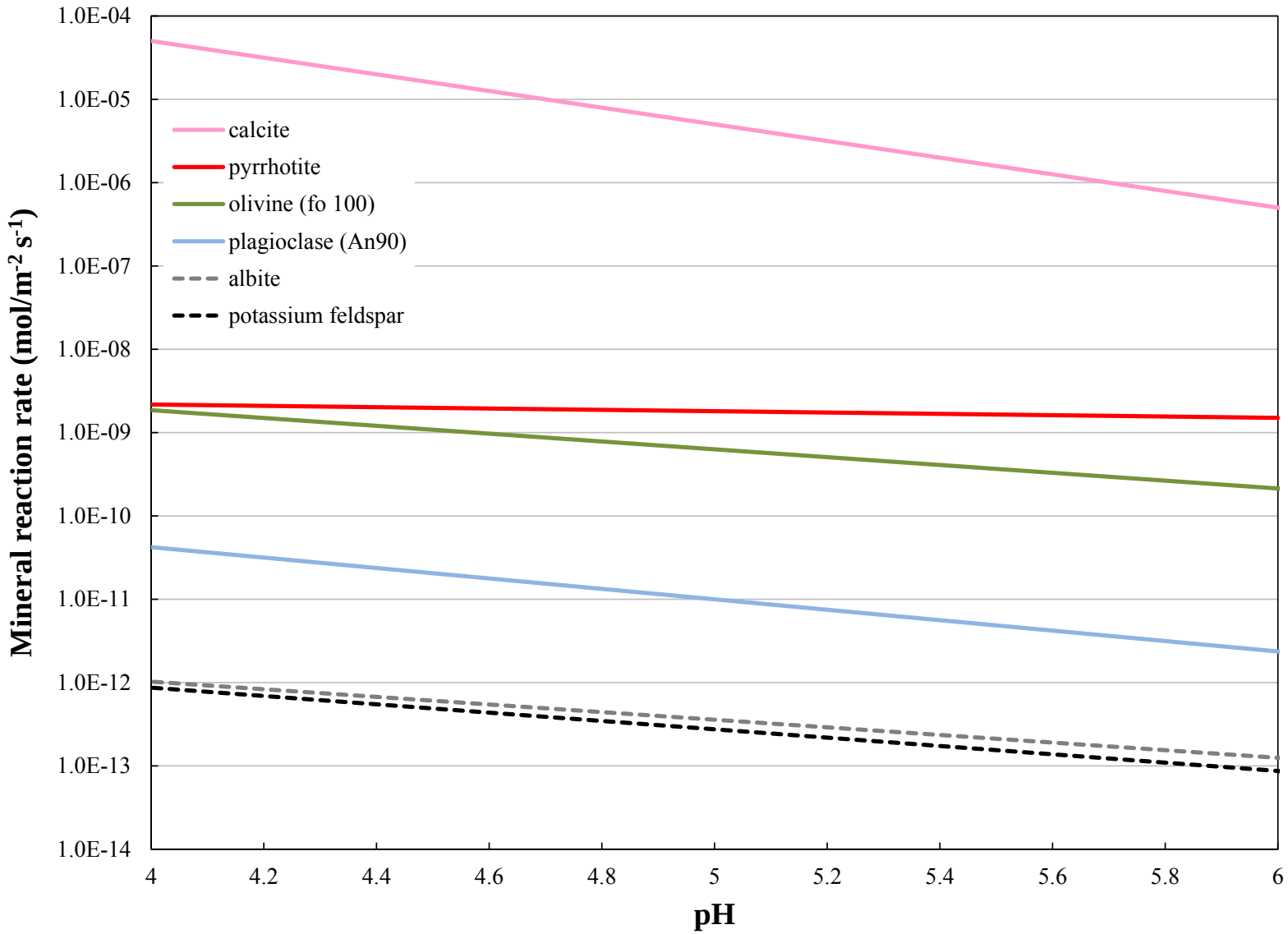

Figure 2.1 Mineral reaction rates as a function of pH (Palandri and Kharaka, 2004). For pyrrhotite an atmospheric equilibrated aqueous oxygen concentration was used (Bilenker et al., 2015).

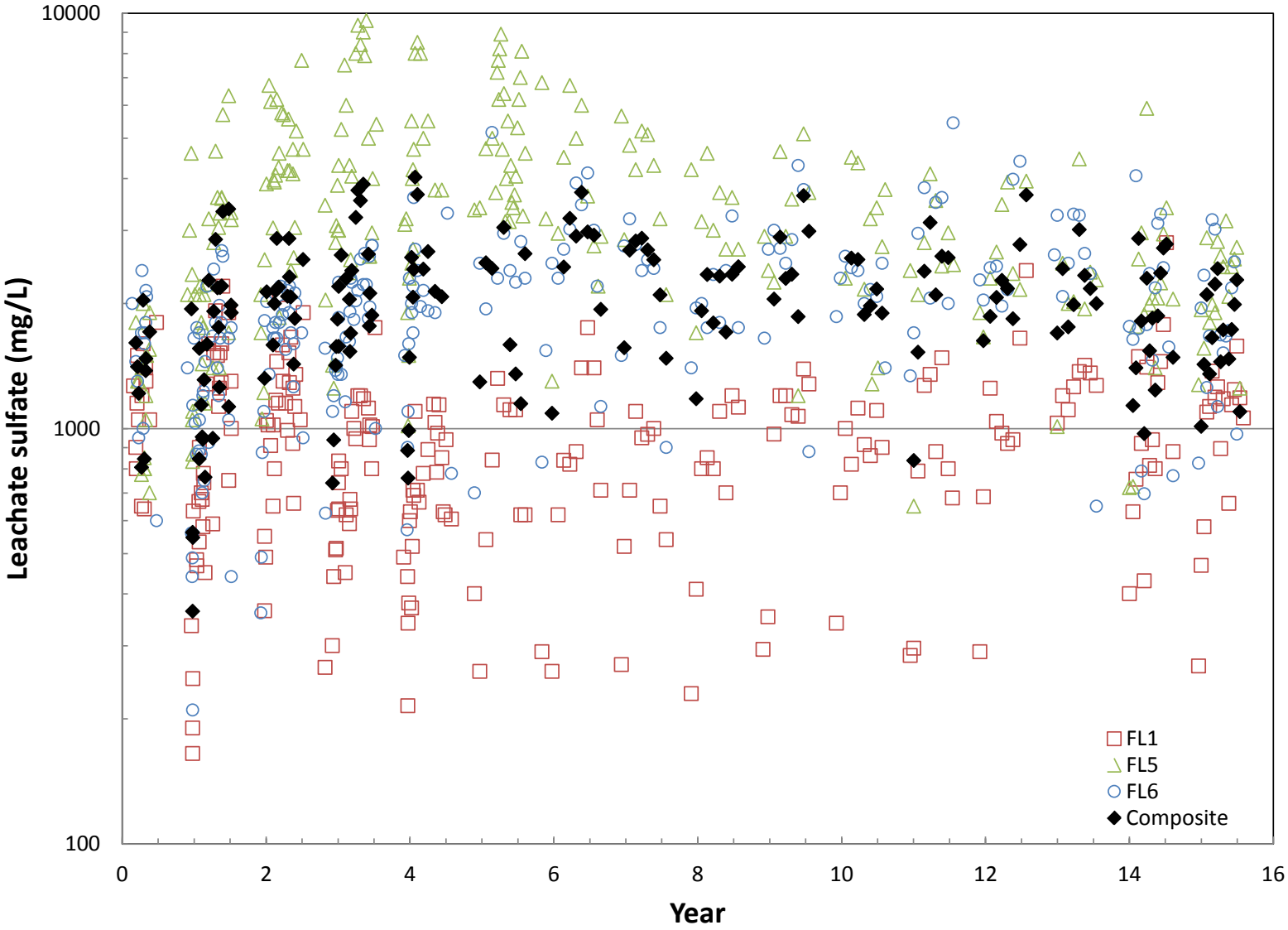

Figure 3.1 Leachate sulfate concentrations for the three former Amax field study rock piles and the calculated volume weighted composite values used in this study. The calculated values show good agreement based on what would be expected from bulk sulfur concentration of the DCLP.

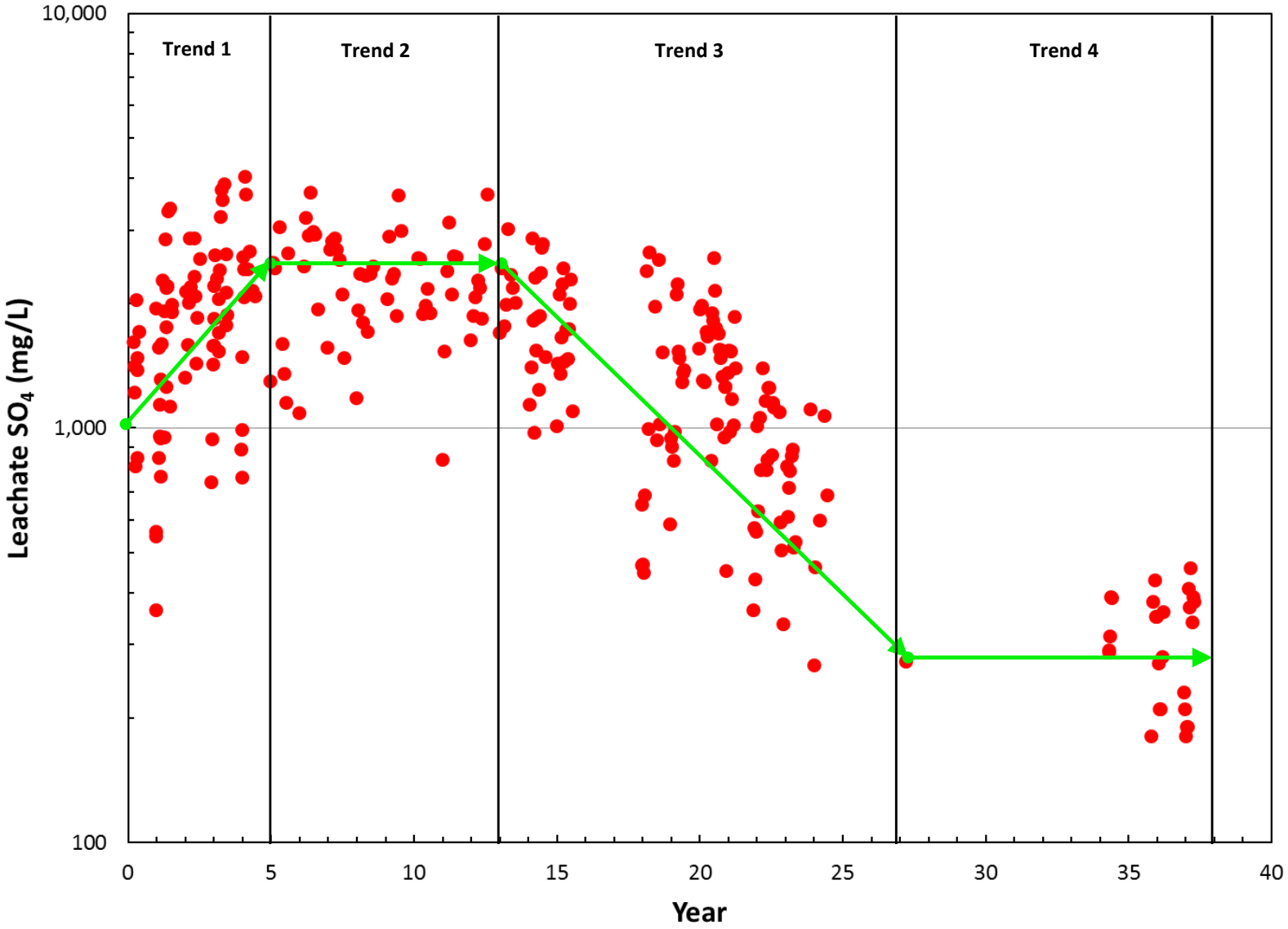

Figure 4.1 DCLP 38 year record of leachate sulfate concentration illustrating four principal concentration trends.

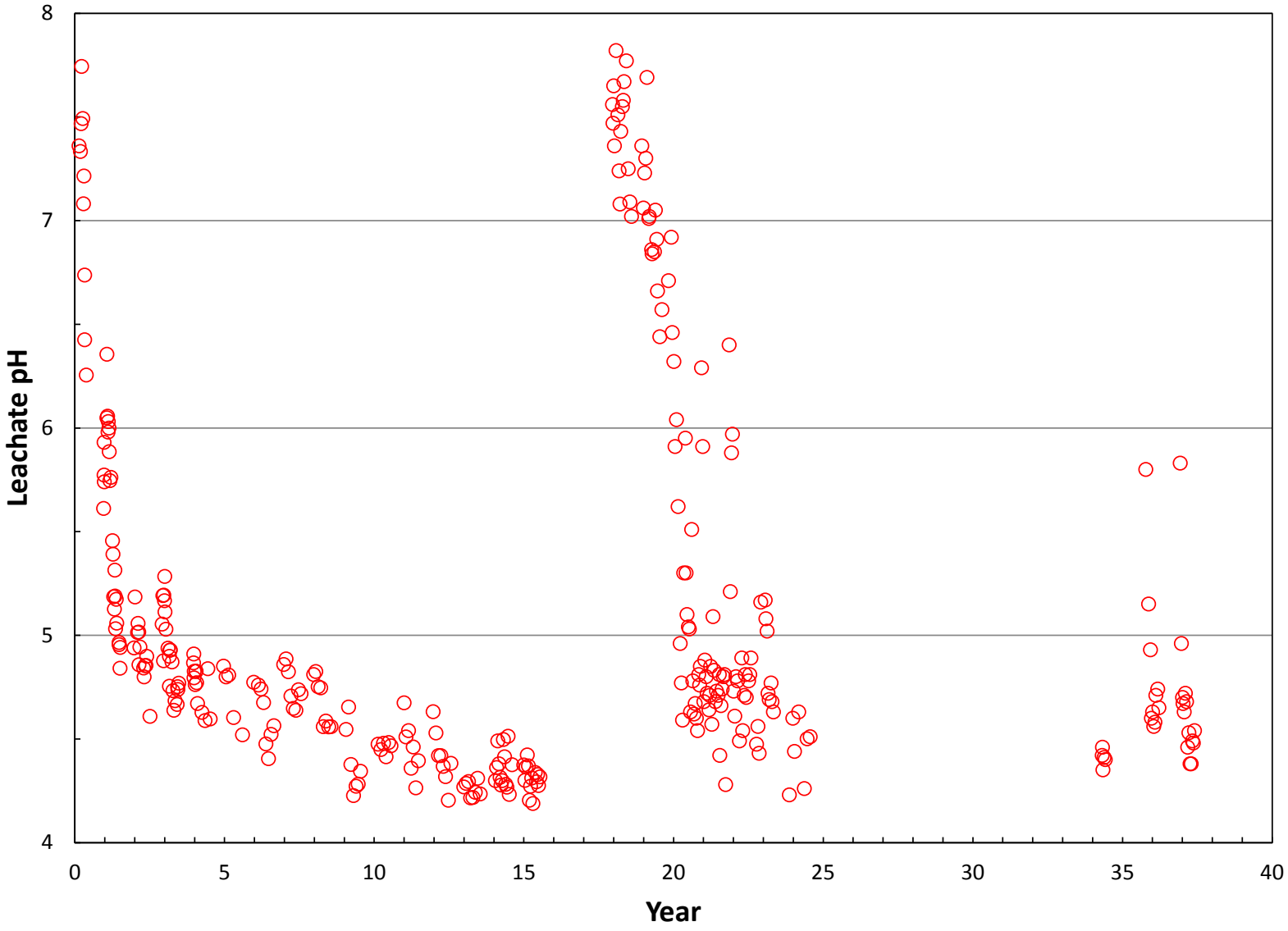

Figure 4.2 Leachate pH time series plot for the DCLP. Leachate pH at year 18 reflects the time at which the DCLP was placed at the Hibbing field research site.

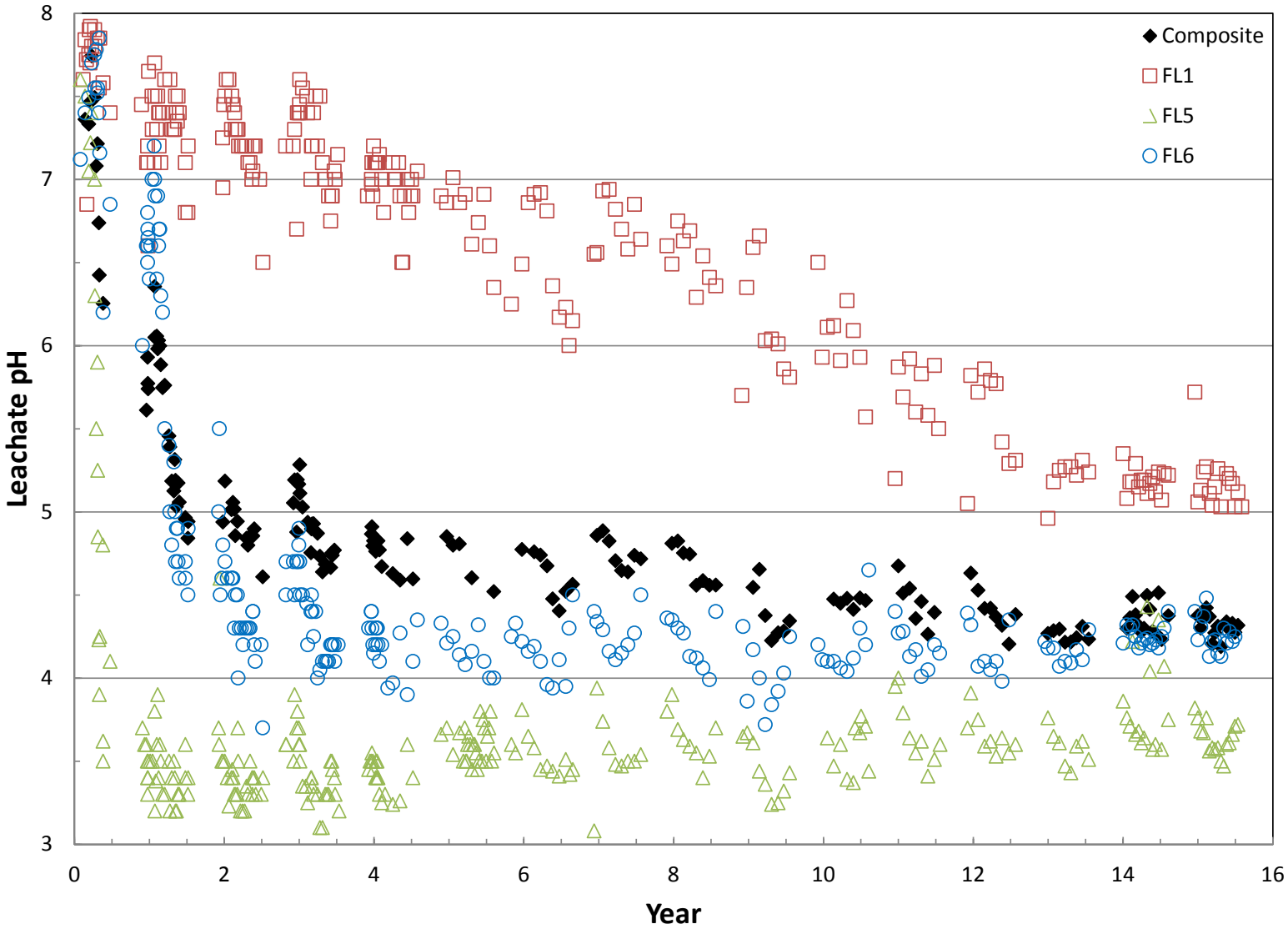

Figure 4.3 Leachate pH time series plot for the DCLP composite calculation and the three Amax field study rock piles the DCLP was calculated from.

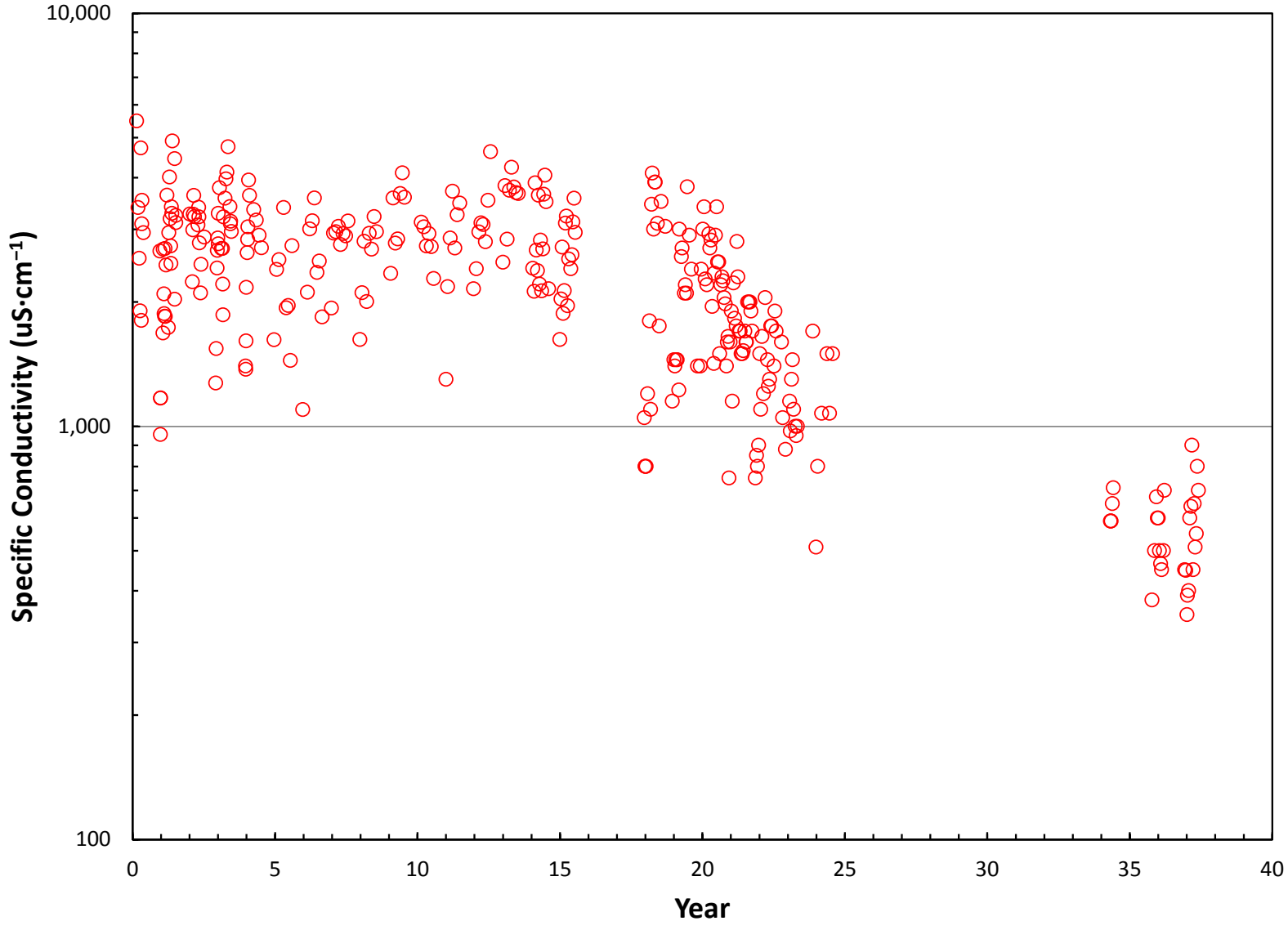

Figure 4.4 DCLP leachate specific conductivity time series plot.

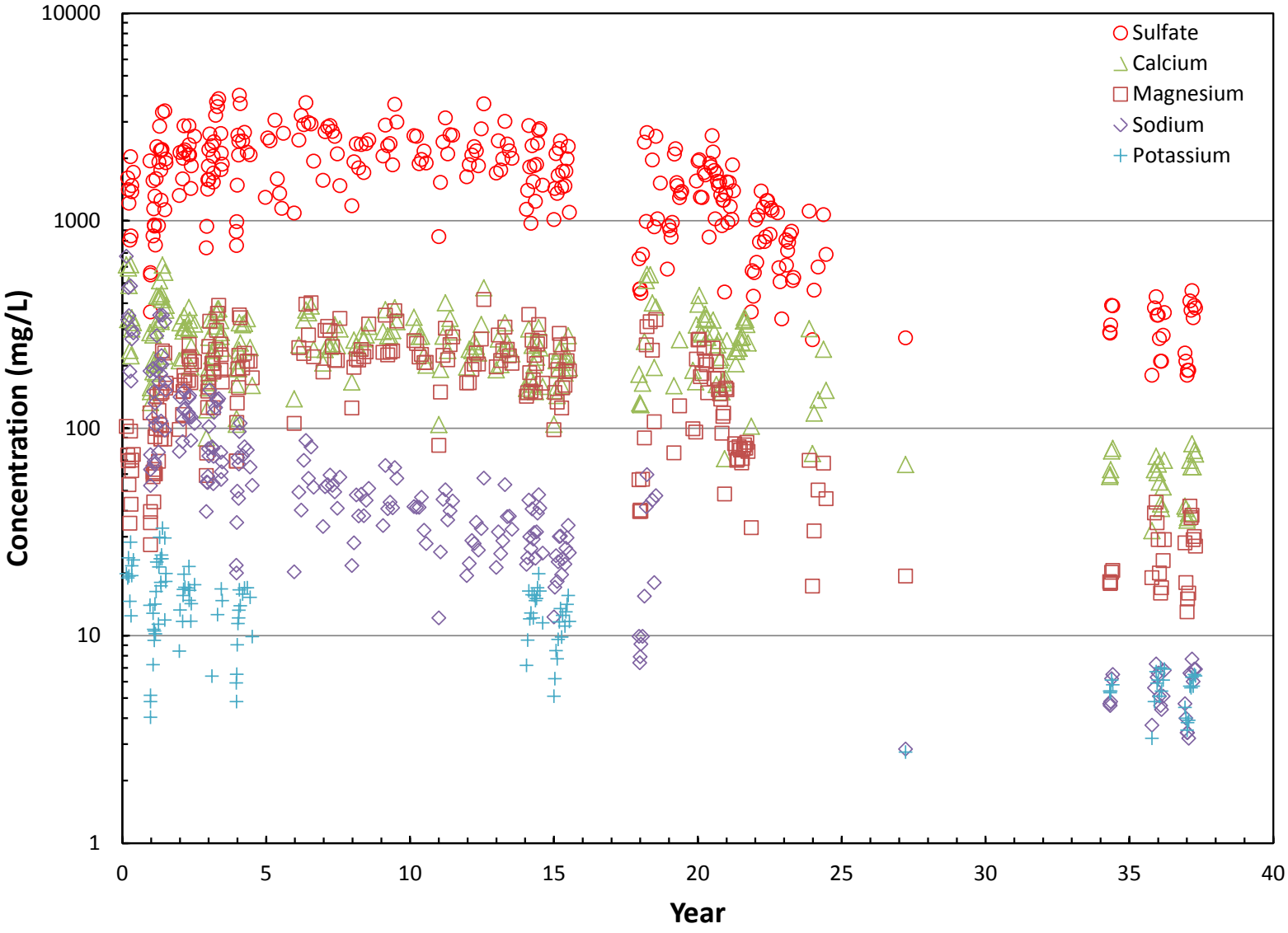

Figure 4.5 DCLP leachate sulfate and major cations concentrations time series plot.

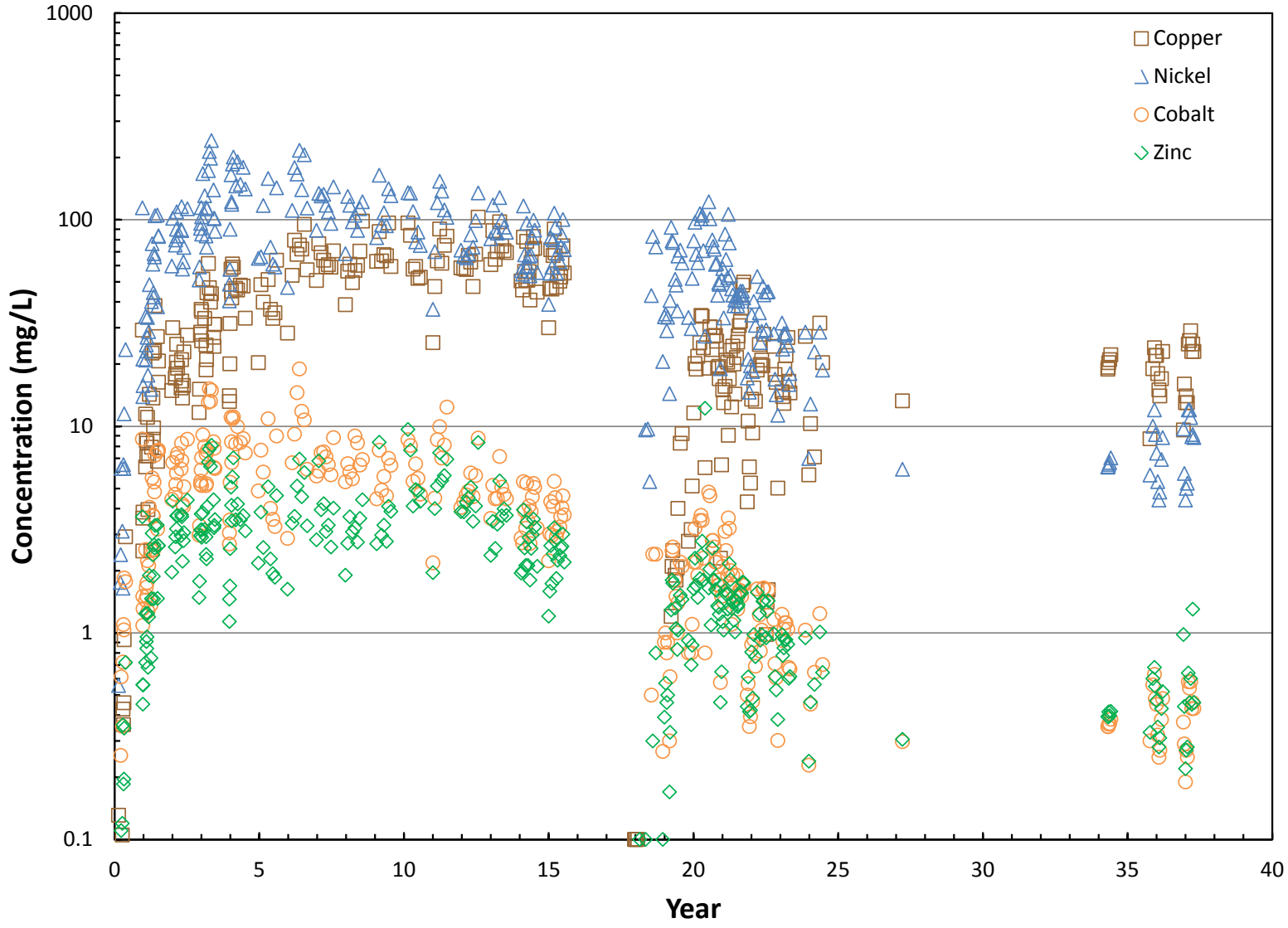

Figure 4.6 Select DCLP leachate metal concentrations time series plot.

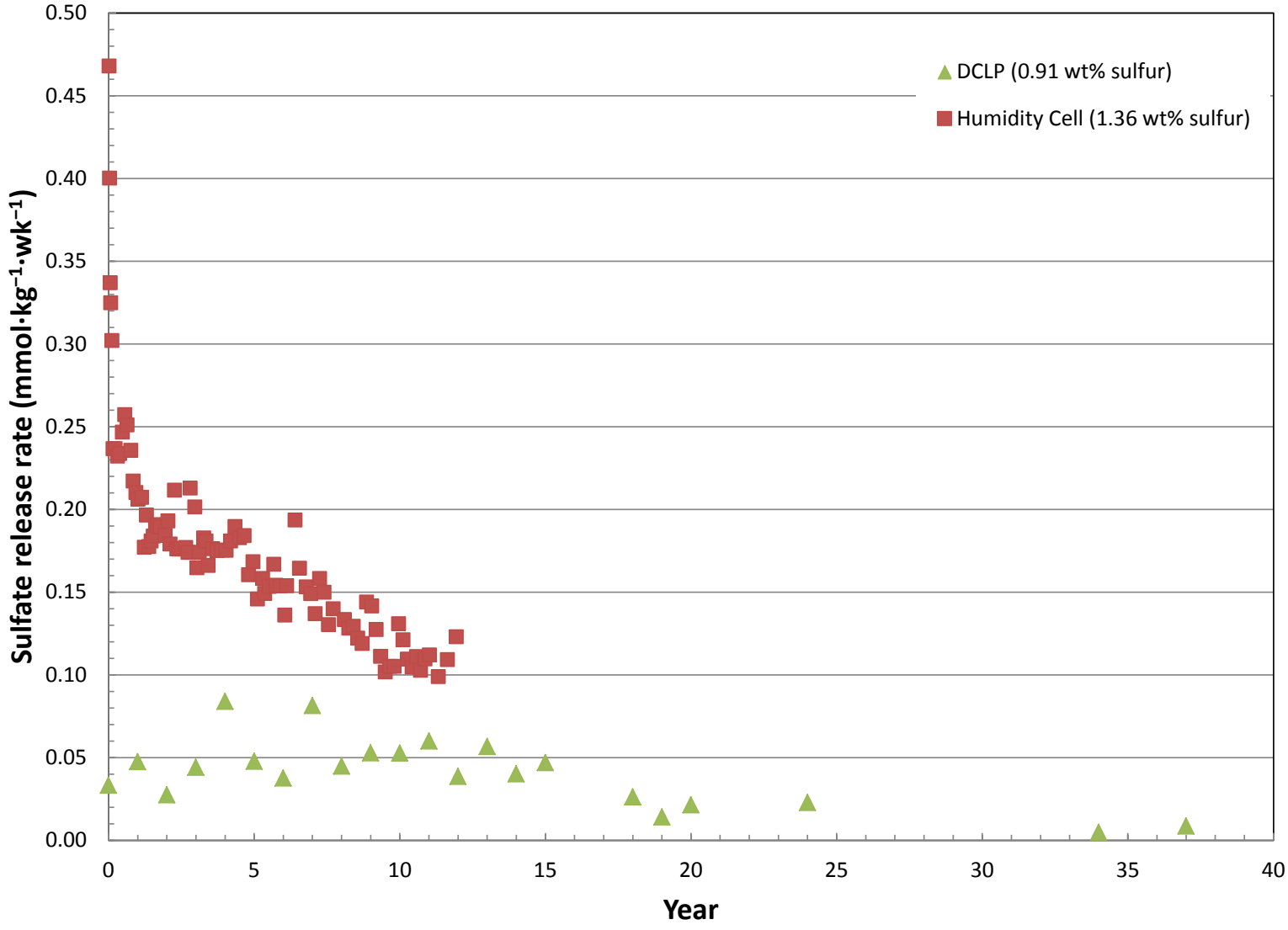

Figure 4.7 Sulfate release rate time series plot for leachate from the DCLP and a MN DNR Duluth Complex humidity cell. The humidity cell is number 16 from Lapakko et al. (2012).

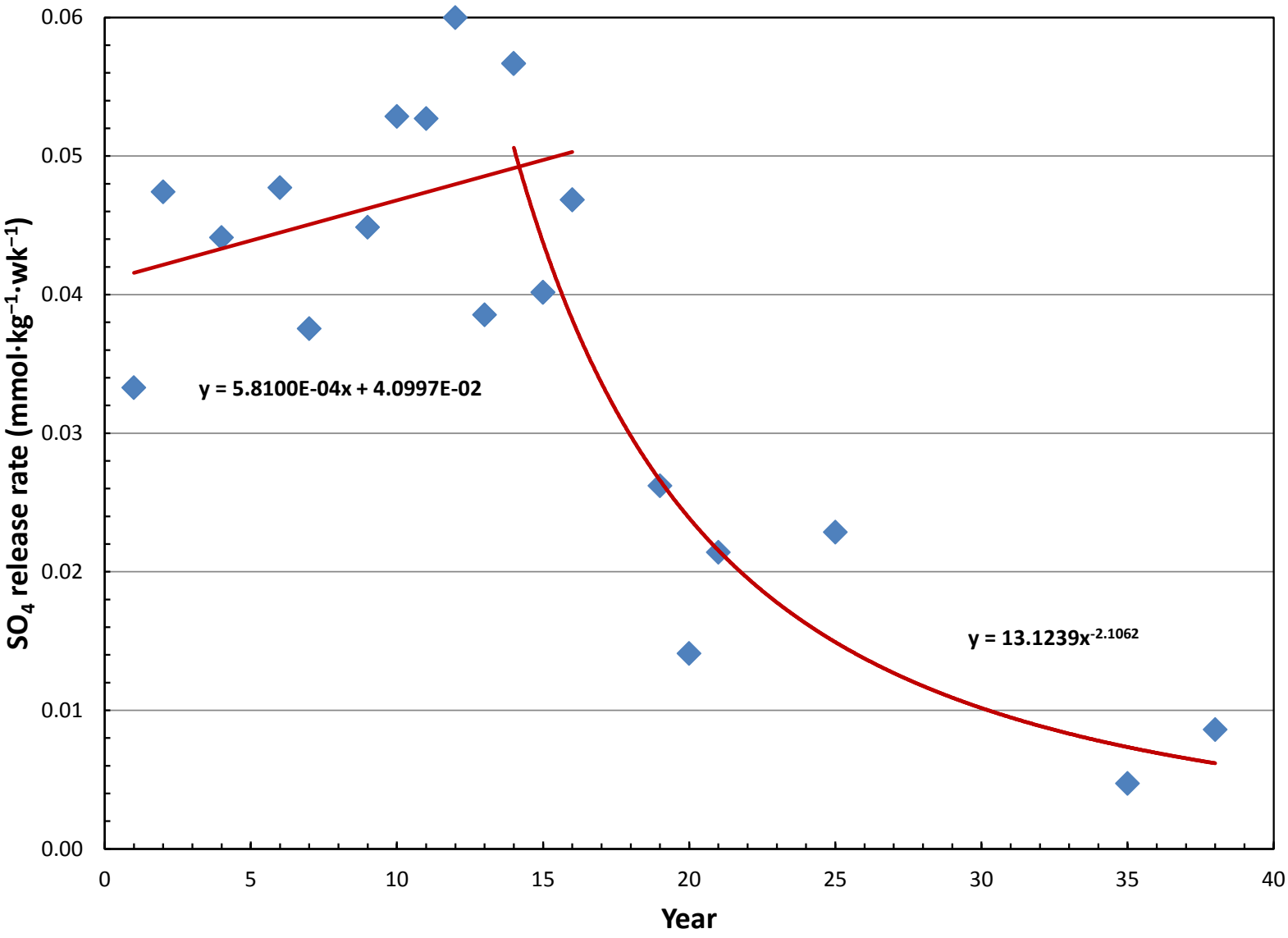

Figure 4.8 Empirical model of the linear and power function fit to the measured annual sulfate release rates. Sulfur removal was calculated using the equations of the lines for each regression.

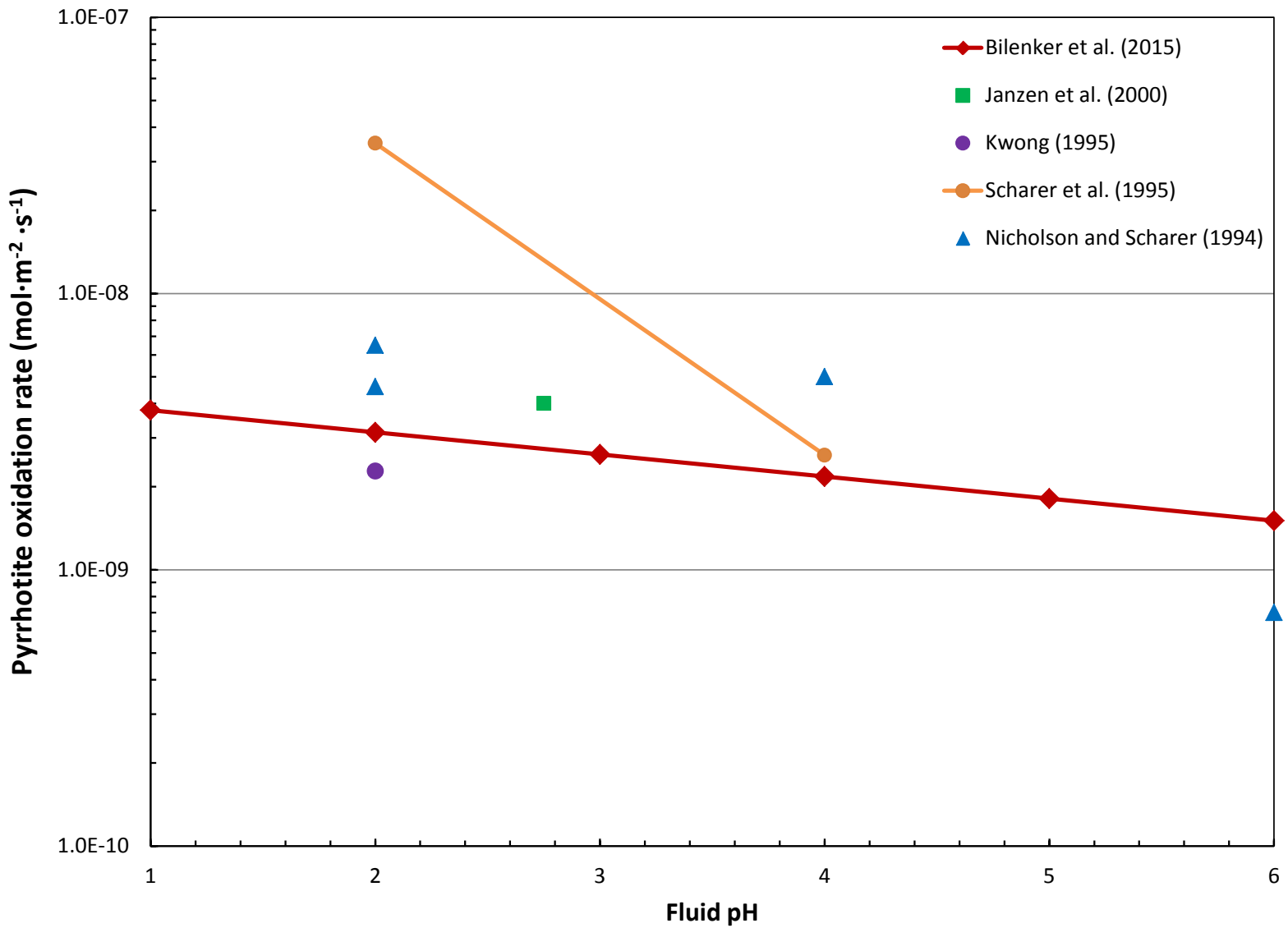

Figure 4.9 Surface area normalized pyrrhotite oxidation rates as a function of pH. The Bilenker et al. (2015) rate value was calculated using an atmospheric equilibrated oxygen concentration (2.648×10^{-4} molal). The Scharer et al. (1995) range spans a temperature range of 10 to 30° C.

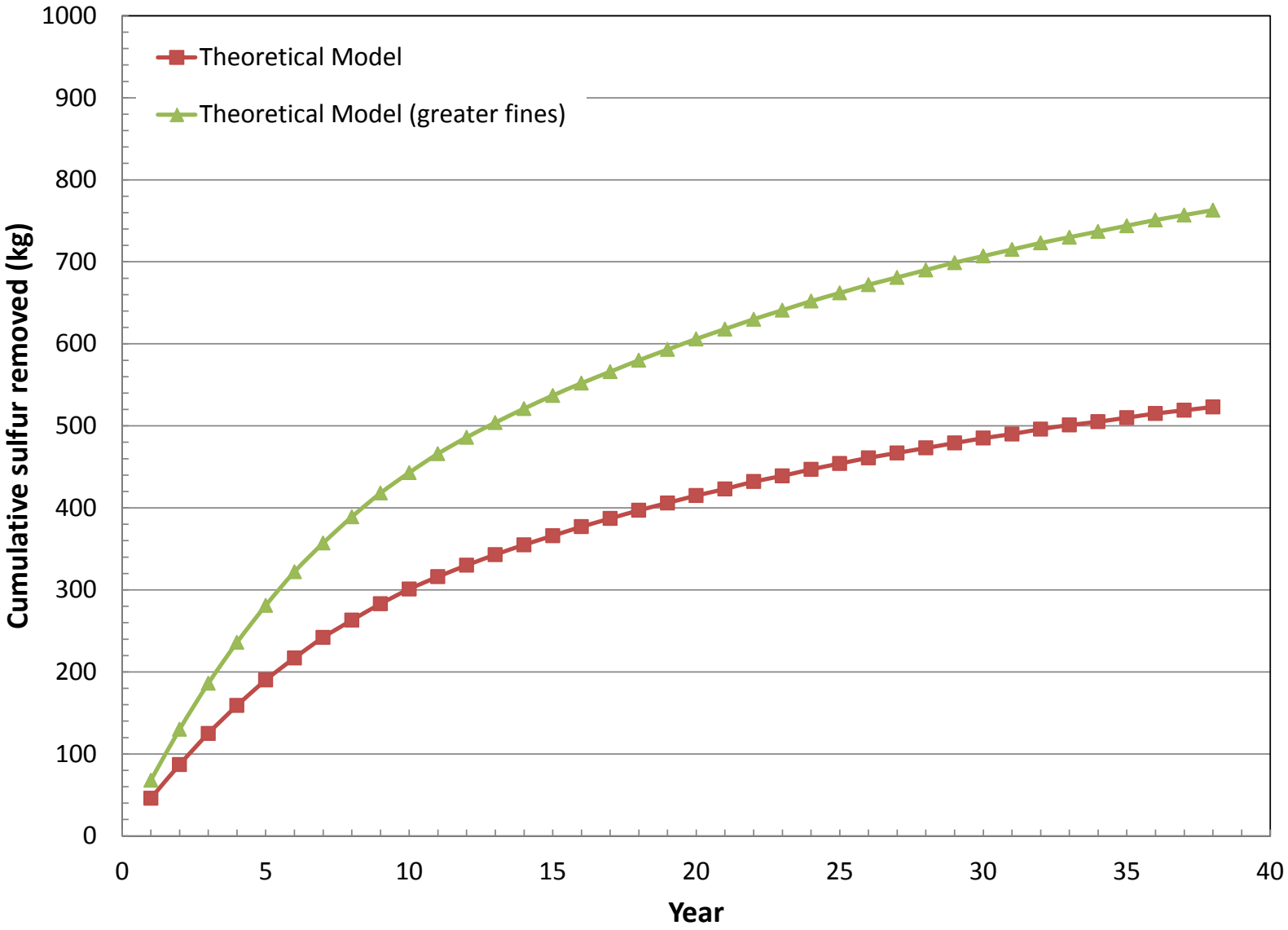

Figure 4.10 Theoretical model cumulative sulfur removal predictions for the normal and greater fines examples.

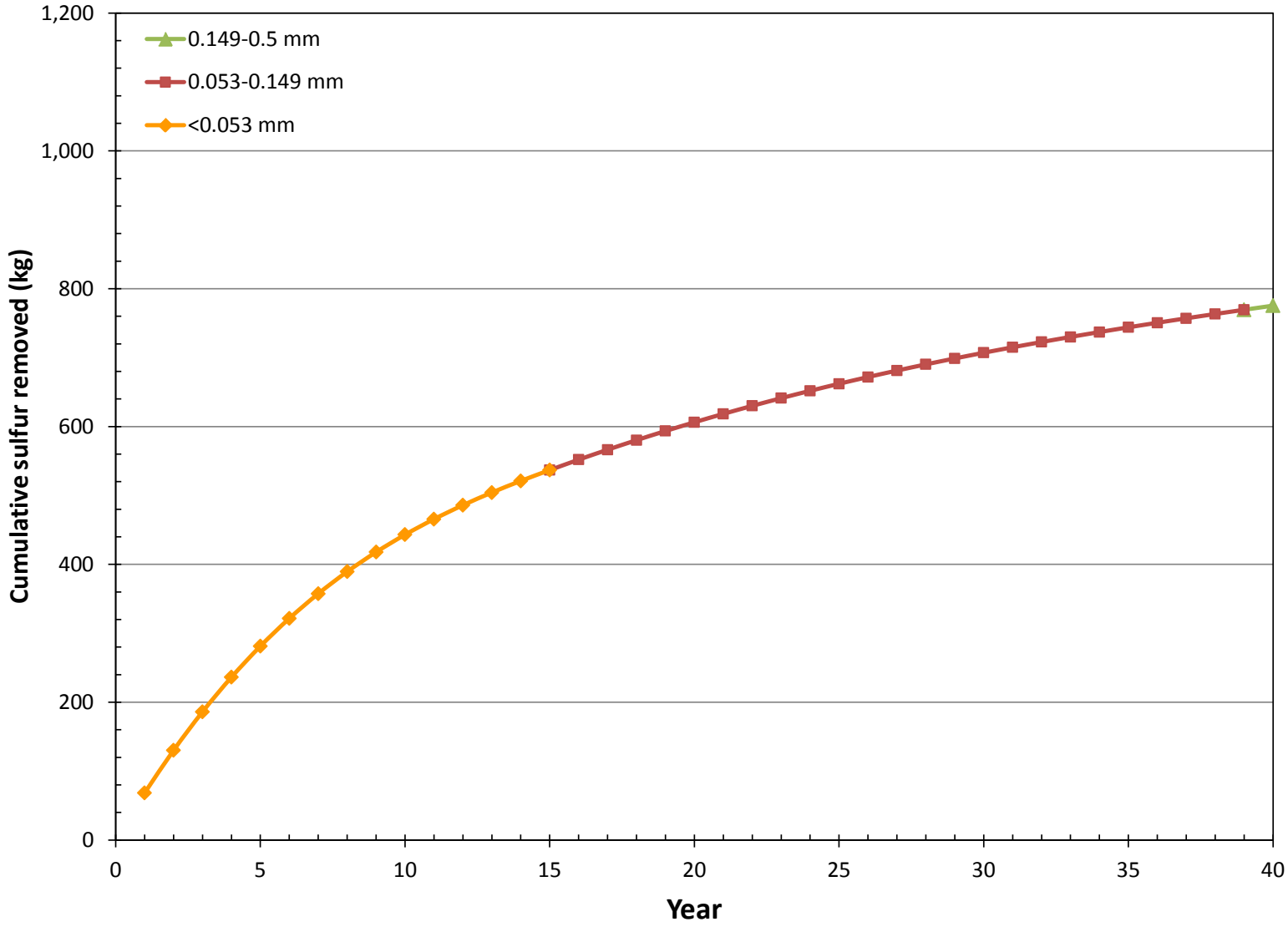

Figure 4.11 Cumulative sulfide removal of the greater fines example indicating the particle lifetime of the sulfide grains from specific rock particle size intervals.

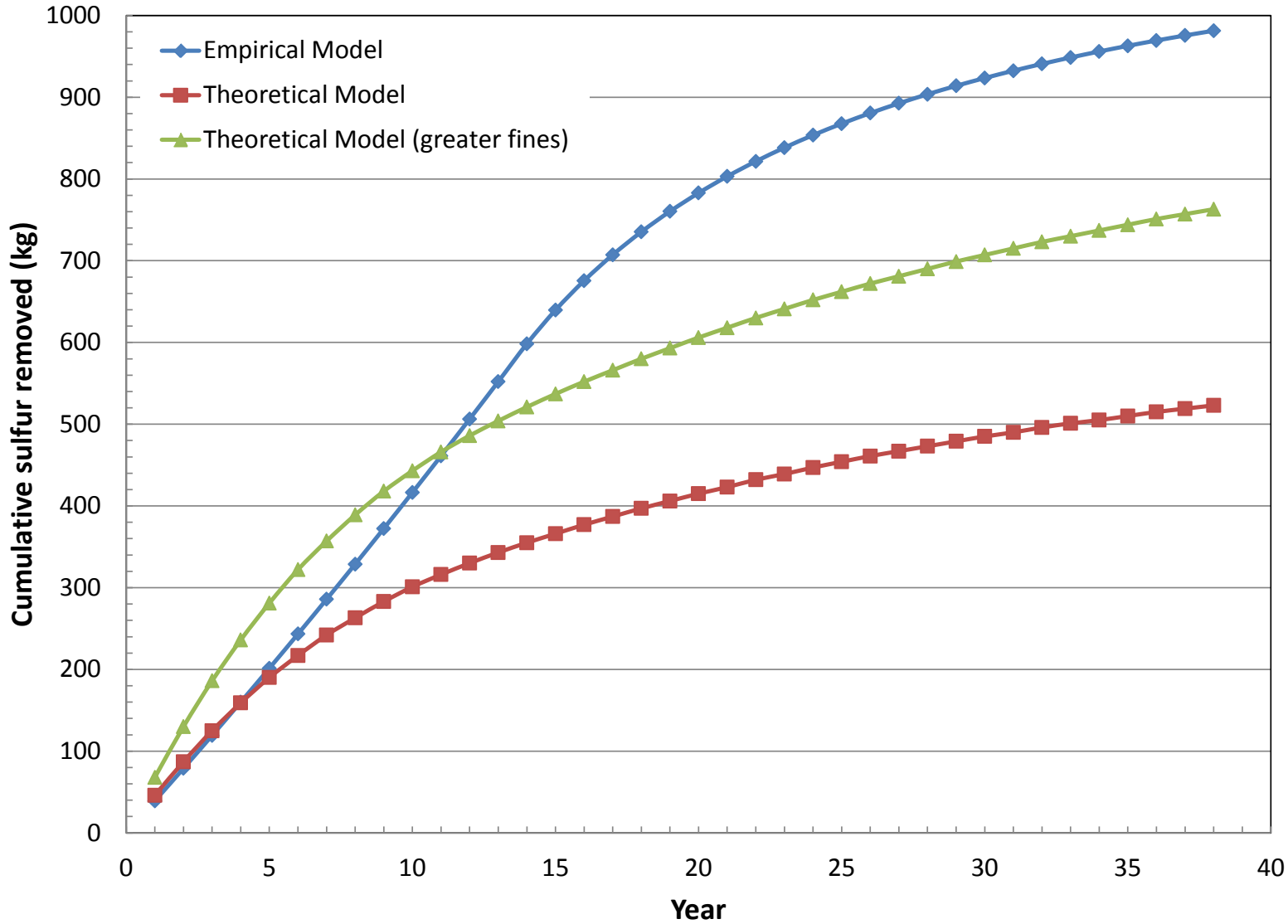

Figure 4.12 Cumulative sulfur removal plot for the empirical and theoretical models.

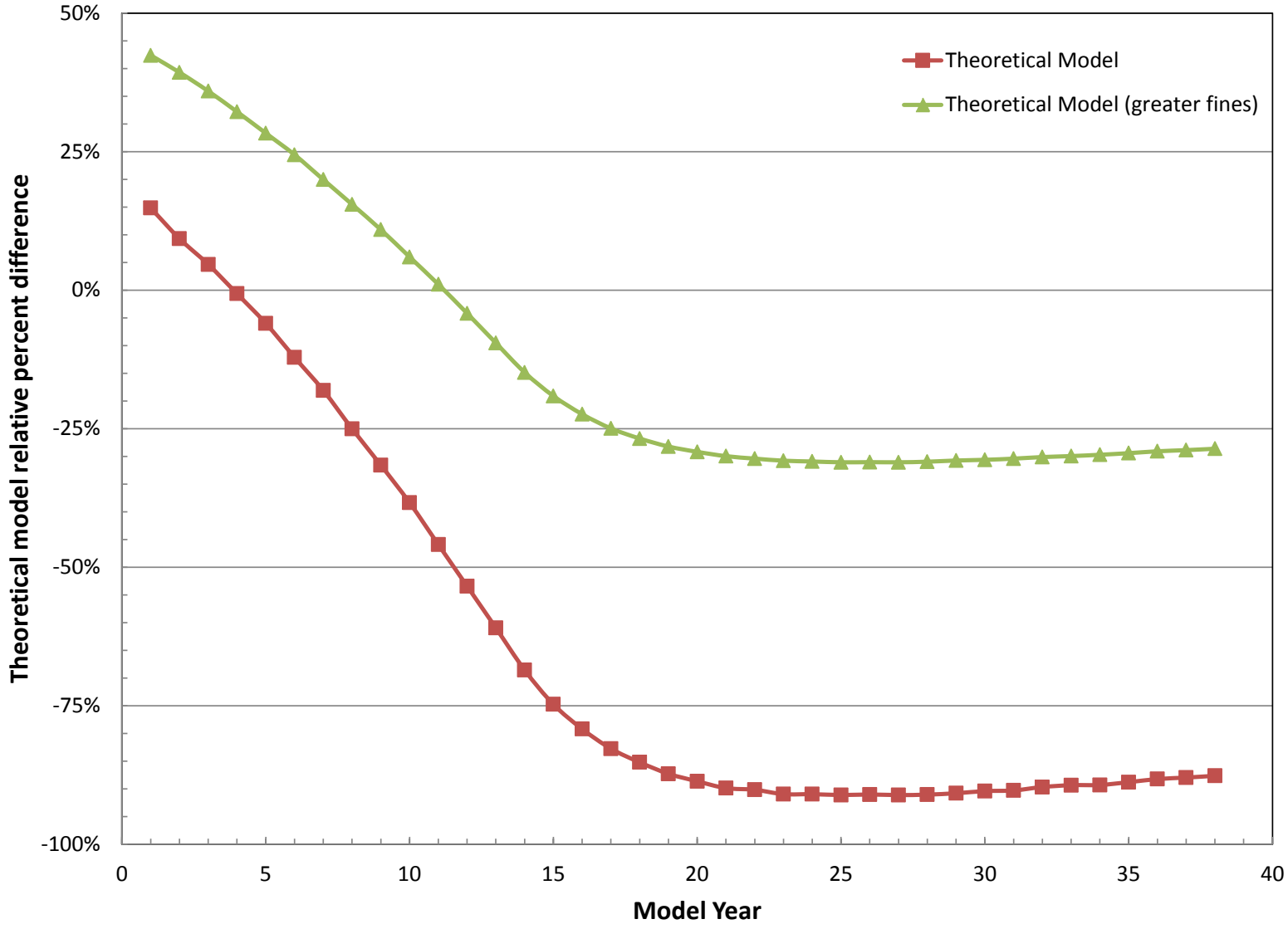

Figure 4.13 Relative percent difference of cumulative sulfur removal between the empirical model and the theoretical models.

Attachment 14

Kellogg C, Lapakko K, Olson M, Jenzen E, Antonson D (2014) Laboratory Dissolution of Blast Hole Samples of Duluth Complex Rock from the South Kawishiwi Intrusion : Twenty-four year laboratory experiment. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN.

Laboratory Dissolution of Blast Hole Samples of Duluth Complex Rock from the South Kawishiwi Intrusion: Twenty-four year laboratory experiment

February 2014

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Saint Paul, MN 55155-4045

Laboratory Dissolution of Blast Hole Samples of Duluth Complex Rock from the South Kawishiwi Intrusion: Twenty-four year laboratory experiment

February 2014

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0. EXECUTIVE SUMMARY

The *research reported will be applied* in the environmental review and permitting of future nonferrous mines in Minnesota, particularly those in the Duluth Complex. These regulatory activities require prediction of solute release from waste rock during mine operation and in the decades and centuries following mine closure. As a foundation for such predictions, the reactions occurring in waste rock stockpiles and variables affecting these reactions must be understood. This knowledge and a description of the proposed method of waste rock disposal are necessary to 1) design appropriate tests for predicting the release of potential pollutants from waste rock, 2) extrapolate results of such tests to field conditions, and 3) if necessary, implement mitigation measures that will ensure that water resources are not adversely impacted. In the present project, laboratory studies were conducted to examine solute release and variables affecting this release from waste rock from the South Kawishiwi intrusion (SKI) of the Duluth Complex. This study provides scientific and technical information necessary to address questions inherent to environmental review and permitting of nonferrous metal mines. The project also provides empirical data describing up to 24 years of rock dissolution for comparison with and extrapolation of data submitted for environmental review and permitting by companies proposing mining in the Duluth Complex.

The *objectives* of the study were as follows.

1) Determine the variation of laboratory drainage pH with solid-phase sulfur content of Duluth Complex samples.
2) Based on the aforementioned relationship, categorize solids based on sulfur content and associated drainage pH.
3) Determine the variation of major ion release rates with solid-phase sulfur content and dissolution time.
4) Determine the empirical neutralization potentials of solids producing acidic drainage and compare these values to those determined by solid-phase analysis.
5) Determine rates of sulfide and silicate mineral dissolution as a function of dissolution time and drainage pH.

To *summarize the methods*, dissolution tests were conducted on fairly fine ($0.053 < d < 0.149$ mm; $0.18 \leq \%S \leq 1.71$) rock samples that were characterized with respect to chemical composition, mineral content, and mineral chemistry. Tests on ten SKI rock samples (and four Virginia formation samples) from Dunka Mine blast holes were initiated in February 1989 and on four more of these samples in September 1990. These samples were blasted 9 to 64 week prior to experimentation. Dissolution tests on six additional SKI rock samples that had been stored for at least 20 years were initiated subsequently. Some of the tests remain in progress and the following report presents data generated through week 1252. Leached rock samples from terminated tests were examined for dissolution features. This report summarizes drainage quality results, categorizes samples based on sulfur content and minimum drainage pH, and estimates dissolution rates for pyrrhotite, plagioclase, augite, olivine, hypersthene, and biotite.

Dependence of drainage quality on sulfur content of the rock and time was assessed based on dissolution test results. These results indicated that, in general, drainage pH tended to decrease and release rates of heavy metals (copper, nickel, cobalt, zinc) and major cations (calcium, magnesium, sodium, potassium) tended to increase with increasing sulfur content. Based on this observation, samples were divided into three groups based on sulfur content and corresponding drainage quality: Group I, 0.18, 0.22 %S; Group II, $0.40 \leq \%S \leq 0.70$; and Group III, $0.70 < \%S \leq 1.64$. Drainage pH for the Group I samples was generally above 6.0. The minimum drainage pH values for Groups II and III were 3.8 and 3.0, respectively. Minimum pH values and corresponding maximum sulfate release values were typically

attained after 300 to 400 weeks (6 to 8 years) of dissolution. The maximum time to minimum pH was roughly 800 weeks, emphasizing the fact that short term drainage quality results do not necessarily reflect long-term trends. The decreasing sulfate release following attainment of maximum values was attributed to development of coatings on the sulfide mineral surface, and this was supported by photographs of leached sulfide grains. Diminishing sulfide mineral content over time contributed, to a lesser extent, to the decrease in sulfate release.

Neutralization potentials (NP) are a measure of the capacity of a sample to neutralize acid, and this capacity was determined empirically based on the drainage quality observed in dissolution tests. The empirical neutralization potential (ENP) was used for comparison with results from three methods used to determine NP based on solid-phase analysis. ENP values typically fell between 0.5 and 2 kg $CaCO_3$/t rock. NP calculated using carbon dioxide analysis to estimate total calcium plus magnesium carbonate content were closest to the ENP values (<0.7 to 2.5 kg $CaCO_3$/t rock), therefore the best estimate of the capacity of these samples to neutralize acid. Values determined on a subset of samples by a direct titration technique (Lapakko 1994a) yielded values roughly 1 to 3 kg $CaCO_3$/t rock higher than the observed values. Values determined by one common method (Sobek et al. 1978) almost always fell between 12 and 21 kg $CaCO_3$/t rock, substantially higher than the observed values. These results indicate that caution must be exercised when selecting solid-phase methods for determining NP and interpreting the results generated.

Mineral dissolution rates were calculated for an iron sulfide mineral and five silicate minerals. The iron sulfide (pyrrhotite) oxidation rates were based on sulfate release and, at circumneutral pH, were in reasonable agreement with rates reported for laboratory tests on pyrrhotite alone (i.e. in the absence of other minerals). These rates were higher under conditions of low pH and tended to increase after 300 to 400 weeks of dissolution, following a period of low pH and high sulfate release. Silicate dissolution rates calculated using four different models were generally in fair agreement. There did not appear to be strong dependence of rates on pH, with the exception of augite dissolution rates, and generally little dependence on time of dissolution. These estimates are intended to facilitate extrapolation of results from the rock samples examined to rock of varying mineral content within the Duluth Complex.

In summary these results demonstrate the 1) dependence of sulfide mineral oxidation (as reflected by sulfate release) and the attendant acid production on the amount of sulfide minerals present; 2) importance of sulfide mineral oxidation as a driving force for release of sulfate, acid, heavy metals and major cations; 3) importance of long-term dissolution tests in obtaining a technically defensible empirical foundation for mine waste drainage quality prediction; 4) caution that must be exercised when assessing the capacity of rock to neutralize acid based on solid-phase analyses; and 5) potential for determining mineral dissolution rates from laboratory test results and thereby extrapolating results to variable compositions of Duluth Complex rock.

Practical implications of these results for environmentally sound management of waste rock generated by mining in Minnesota include the potential for managing waste rock based on sulfur content, a relatively simple analytical assessment. Based on drainage pH alone, disposal of waste rock in Group I (%S $\leq$ 0.22 %) would probably require no rigorous reclamation. Potential impacts of sulfate, copper, nickel, cobalt, and zinc release from rock in this range of sulfur content would require consideration. In order to meet water quality standards, waste rock producing drainage pH values similar to Group III samples (0.70 < %S $\leq$1.64) would require the most rigorous control measures of the samples tested. Elevated release of sulfate and heavy metals from these groups would likely require mitigation as well. The results have implications of substance to mine waste drainage quality prediction. First, caution that must be exercised when assessing the capacity of rock to neutralize acid based on solid-phase analyses designed to quantify NP. Second, results generated from short-term dissolution tests do not necessarily

simulate long-term drainage quality. Emphasizing this point, drainage pH from one sample was circumneutral for 800 weeks and then acidified, reaching a minimum pH of 3.8. Third, tests conducted on samples that had been stored for 8 to 20 years yielded lower rates of sulfate release and higher drainage than samples within about one year of excavation. Therefore, samples used for mine waste drainage quality prediction should be tested as close to the time of their excavation as possible.

Additional work should be conducted to further the benefits generated to date. This includes continued dissolution testing to confidently define long-term trends, analysis of leached solids to increase understanding of chemical reactions controlling solute release (e.g. chemical precipitation and adsorption), and more detailed analysis of the present data, including calculation of mineral dissolution rates. Furthermore, the methods for calculating mineral dissolution rates and approaches for other aspects of data analysis developed for this project should be applied to experiments examining dissolution of Partridge River Intrusion rocks.

1. INTRODUCTION

Tailings and waste rock, as well as the mine itself, are components of metal mining operations that remain long after ore removal has ceased. These mine wastes can be contacted by precipitation, ground water, or surface water. The quality of water exiting mine wastes spans a wide range, with some drainage producing little adverse impact on natural waters. In contrast, some drainage can adversely affect water quality long after mining operations have ceased. Acidic drainage, for example, was observed in 1977 at a Norwegian mine abandoned in 1833, nearly 150 years after termination of mining activities (Iversen and Johannessen, 1987). Dependent upon size and severity of the problem, remediation of mining impacts can cost hundreds of millions of dollars (Biggs, 1989). This potential for long-term impacts has led to regulations requiring post-closure care of mine wastes (Ciccarelli et al., 2009; Jerz and Rimstidt, 2000; Lapakko, 1990).

Regulations governing the mitigation of potentially problematic mine waste drainage and any associated financial liability have been developed. Plans and costs for closure and post-closure care of mine wastes are an important part of these policies and must be submitted prior to mine development (e.g. Lapakko, 1990). Any mine waste reclamation costs can then be associated with other mining expenses during assessment of mineral recovery costs.

Prediction of the quality and quantity of drainage generated by flow through waste rock lithologies excavated to access ore is integral to developing effective and economically efficient waste rock management plans. Based on these predictions and their associated impacts, mitigation techniques for individual mine waste units can be tailored to the potential for adverse impact. The best indicators of drainage quality for specific waste lithologies are existing data from similar geologic units, subjected to similar mining and mineral processing methods, and exposed to similar environmental conditions. Because data of this nature are rarely available, it is often necessary to use other means to predict the quality of drainage data, such as compositional characterization and dissolution testing.

Minnesota's Precambrian rocks, specifically the Duluth Complex and Archean metavolcanics and metasedimentary formations, have potential for base and precious metal development. The Duluth Complex is a composite series of massive gabbroic intrusions in northeastern Minnesota, some of which contain low-grade copper and nickel sulfides, titanium oxides, and platinum group elements (PGE's). In the present study 25 rock samples from the Duluth Complex and underlying Virginia Formation were characterized (physical, chemical, mineralogical) and subjected to laboratory dissolution testing. The intent of this testing was to generate data that would assist in the environmentally sound management of Duluth Complex waste rock generated during mining. The general approach was to determine the dependence of drainage quality on solid-phase composition of troctolitic/gabbroic Duluth Complex rocks and dissolution time. By establishing such a relationship, solid-phase composition of these rocks would provide beneficial insight on their dissolution behavior in the field. Dissolution rates of major minerals present were also calculated to aid in extrapolation of results on the solids examined to rocks with different amounts of the major minerals. Mineral dissolution rates might also be helpful in extrapolation of laboratory rates to field settings.

2. OBJECTIVES

The general objective of this study is to correlate drainage quality with rock composition. More specifically, the following objectives are addressed in the present report.

1) Determine the variation of laboratory drainage pH with solid-phase sulfur content of Duluth Complex samples.
2) Based on the aforementioned relationship, categorize solids based on sulfur content and associated drainage pH.
3) Determine the variation of major ion release rates with solid-phase sulfur content and dissolution time.
4) Determine the empirical neutralization potentials of solids producing acidic drainage and compare these values to those determined by solid-phase analysis.
5) Determine rates of sulfide and silicate mineral dissolution as a function of dissolution time and drainage pH.

The data generated by the study will be used to assist in environmentally sound management of Duluth Complex waste rock. The present report describes the laboratory dissolution of 25 rock samples, 21 Duluth Complex and 4 with a large Virginia Formation component, containing 0.18% to 5.44% sulfur for periods varying from 78 to 1252 weeks

3. BACKGROUND

3.1. Mine Waste Dissolution

The following discussion focuses on mineral dissolution reactions that influence mine waste drainage quality. The reactions presented result in acid production, acid neutralization, and trace metal release. Within the discussion, several solid phases of interest in mine waste characterization are identified. Detailed discussion of aqueous geochemistry fundamentals and geochemistry of acid mine drainage is presented by Nordstrom (1999), Nordstrom and Alpers (1999), Smith (1999) and Smith and Huyck (1999).

3.1.1. Acid Production

There are three general types of acid generation from mine wastes: iron sulfide oxidation, dissolution of soluble iron sulfate minerals, and dissolution of less soluble sulfate minerals of the alunite/jarosite series. Oxidation of iron sulfide minerals common in intrusive mafic complexes (e.g., Duluth Complex), such as pyrite (FeS_2, reaction 1) and pyrrhotite ($Fe_{1-x}S$, $0 \leq x \leq 0.2$, reaction 2), are responsible for most acid production by mine wastes (Stumm and Morgan 1981).

$$FeS_2(s) + (15/4)O_2(aq) + (7/2)H_2O = Fe(OH)_3(s) + 2SO_4^{2-}(aq) + 4H^+(aq) \quad \textbf{(1)}$$

$$Fe_{1-x}S(s) + [(9-3x)/4]O_2(aq) + [(5-3x)/2]H_2O = (1-x)Fe(OH)_3(s) + SO_4^{2-}(aq) + 2H^+(aq) \quad \textbf{(2)}$$

In addition to acid production, sulfate is also released to solution, and its presence in mine waste drainage is typically the first indicator of sulfide mineral oxidation. The iron product presented is ferric hydroxide ($Fe^{3+}(OH^-)_3$, although formation of intermediate aqueous ferric hydroxyl species (e.g. $Fe(OH)_2^+(aq)$, $Fe(OH)_3(aq)$) will also result in acid production.

Ferric iron can also oxidize sulfide minerals (reactions 3, 4). The amount of acid generated as a result of iron sulfide oxidation is the same as that for oxidation by oxygen. The additional acid generated is due to precipitation of the aqueous ferric iron (left side of equation) as ferric hydroxide (equation 5).

$$FeS_2(s) + 14Fe^{3+}(aq) + (19/2)H_2O + 15/4O_2(aq) = 15Fe(OH)_3(s) + 2SO_4^{2-}(aq) + 46H^+(aq) \quad \textbf{(3)}$$

$$Fe_{1-x}S(s) + (8-2x)Fe^{3+}(aq) + [(9-3x)/4]O_2(aq) + [(53-15x)/2]H_2O = (9-3x)Fe(OH)_3(s) + SO_4^{2-}(aq) + (26-6x)H^+(aq) \quad \textbf{(4)}$$

$$Fe^{3+}(aq) + 3H_2O = Fe(OH)_3(s) + 3H^+(aq) \quad \textbf{(5)}$$

The rate of iron sulfide oxidation and attendant acid production is dependent on solid-phase composition and microbial activity, as well as the availability of oxygen and water. Oxidation rates vary among sulfide minerals, with reactivity decreasing as follows: marcasite > pyrrhotite > pyrite (e.g. Kwong and Ferguson 1990). However, variations in the reactivity of iron sulfide minerals have been reported. Reactivity may be a function of reaction conditions, trace element concentrations, and crystal-

morphology characteristics, among other factors (Jambor 1994; Plumlee 1999).

Mineral surface area also plays a role in rates of acid production. For any sulfide mineral, oxidation rates increase with available surface area. Consequently, acid production by a given mass of framboidal pyrite (high associated surface area) is much more than that resulting from oxidation of a comparable mass of euhedral pyrite (low associated surface area) (Pugh et al. 1984; White and Jeffers 1994). Mineral grain coatings (Janzen et al. 2000) and sulfur-depleted outer grain layers can also reduce oxidation rates (Belzile et al. 2004).

Abiotic oxidation rate is also pH dependant. Whereas the abiotic rate of pyrite oxidation by oxygen declines slightly as pH decreases, the overall abiotic rate increases as pH declines below about 4.5. This occurs due to ferric iron becoming the dominant oxidant (Williamson and Rimstidt 1994). Nordstrom (1982) reported, "as pH decreases to 4.5, ferric iron becomes more soluble and begins to act as an oxidizing agent." As pH further decreases, bacterial oxidation of ferrous iron becomes the rate limiting step in the oxidation of pyrite by ferric iron (Singer and Stumm 1970), which is the only significant oxidizing agent in this pH range (Nordstrom 1982; Singer and Stumm 1970; Kleinmann et al. 1981). In laboratory tests, pyrrhotite oxidation at pH 3.5 - 4.05 was roughly six to seven times that at pH 5.35 - 6.1. Rate increases were attributed to bacterially mediated oxidation (Lapakko and Antonson, 1994). It has also been indicated that bacterially mediated pyrite oxidation by ferric iron at pH 2 is roughly two to three orders of magnitude faster than abiotic oxidation (Nordstrom and Alpers 1999).

The combination of sulfide mineral characteristics and weathering reactions produce acidic, iron- and sulfate-rich waters which can 1) further react with sulfide minerals to accelerate oxidation, 2) evaporate partially or totally to precipitate hydrated iron-sulfates and other minerals, and/or 3) contact host rock minerals, which react to neutralize some or all of the acid. Acidic flow that is not neutralized within the mine waste will exit as acid rock drainage (ARD).

In some cases, flow events (rain or snow-melt) can result in ARD. Hydrated iron-sulfate and trace-metal sulfate minerals will precipitate out during the evaporation of acidic, metal- and sulfate-rich water within mine-waste materials, as will any heavy metals released during sulfide mineral oxidation. The precipitated materials can be reoxidized by additional flow through the precipitated waste (e.g. rain events, snow melt). The more common hydrated iron-sulfate minerals that occur as efflorescent salts on the surfaces of weathering pyrite include melanterite, rozenite, szomolnokite, romerite and copiapite ($FeSO_4 \bullet 7H_2O$, $FeSO_4 \bullet 4H_2O$, $FeSO_4 \bullet H_2O$, $[Fe^{2+}(Fe^{3+})_2(SO_4)_4 \bullet 14H_2O]$, and $[Fe^{2+}(Fe^{3+})_4(SO_4)_6(OH)_2 \bullet 20H_2O]$, respectively; Alpers et al., 1994). These efflorescent salts are highly soluble and provide an instantaneous source of acidic water upon dissolution and hydrolysis (Nordstrom, 1982; Cravotta, 1994). They may be partially responsible for increased acidity and metals loadings in the receiving environment during rain events. The storage and incremental release of these salts may help explain lag from mine-waste placement to AMD-formation, particularly in arid climates.

As an example, reactions 6, 7 and 8 summarize the step-wise dissolution of melanterite, a hydrated iron-sulfide mineral that can be deposited through evaporation and precipitation. The net result of equations 6 – 8 is summarized in equation 9, which shows net production of two moles of acid for each mole of melanterite dissolved. (Cravotta 1994) showed that a similar aqueous dissolution of romerite produced six moles of acid for each mole of romerite dissolved.

$$FeSO_4 \bullet 7H_2O(s) = Fe^{2+}(aq) + SO_4^{2-}(aq) + 7H_2O(aq) \quad \textbf{(6)}$$

$$Fe^{2+}(aq) + (1/4)O_2(g) + H^+(aq) = Fe^{3+}(aq) + (½)H_2O(aq) \quad \textbf{(7)}$$

$$Fe^{3+}(aq) + 3H_2O(aq) = Fe(OH)_3(s) + 3H^+(aq) \quad \textbf{(8)}$$

$$FeSO_4 \bullet 7H_2O(s) + (1/4)O_2(g) = Fe(OH)_3(s) + SO_4^{2-}(aq) + (9/2)H_2O + 2H^+(aq) \quad \textbf{(9)}$$

The alunite-jarosite mineral group consists of sulfate minerals that are less soluble than the efflorescent sulfate salts. The evaporative concentration of efflorescent iron sulfates leads to the precipitation of the more common iron minerals such as goethite and jarosite (Nordstrom 1982). Similar reaction of efflorescent aluminum sulfates will produce alunite. Alpers et al. (1994) reported that jarosite is slightly soluble and can, therefore, contribute acid according to equation 10. Lapakko and Berndt (2003) demonstrated that the rate of acid production from potassium jarosite is markedly slower than that from samples of similar sulfur content in which the sulfur occurs as pyrite. Consistent with data presented in the aforementioned paper, preliminary leach studies on natural and synthetic jarosites conducted by the U.S. Bureau of Mines showed a drop in pH from 6 to the range of 3 to 4 after contact with the jarosites (White 2000). It should be noted, however, that there are a wide variety of alunite and jarosite minerals, as well as mineral properties, and consequently, reactivities can also vary. For example, Alpers (2000) speculated that a pure jarosite or hydronium jarosite might buffer pH in the range of 1.5-3, and this is supported by data presented by Lapakko and Berndt (2003).

$$KFe_3(SO_4)_2(OH)_6(s) = K^+ + 3FeOOH(s) + 2SO_4^{2-}(aq) + 3H^+(aq) \quad \textbf{(10)}$$

3.1.2. Acid Neutralization

The balance between the rates of acid production by iron-sulfide mineral oxidation and host-rock mineral neutralization will determine the acidity of mine-waste drainage. The most effective minerals for neutralizing acid are those containing calcium carbonate and magnesium carbonate, including calcite ($CaCO_3$), magnesite ($MgCO_3$), dolomite ($CaMg(CO_3)_2$), and ankerite (Ca (Fe, Mg, Mn)$(CO_3)_2$). Reaction 11 represents the dominant acid-neutralizing reaction of calcite ($CaCO_3$) above pH 6.4, whereas reaction 12 is the dominant reaction below pH 6.4 (Drever, 1988):

$$CaCO_3(s) + H^+(aq) = HCO_3^-(aq) + Ca^{2+}(aq) \quad \textbf{(11)}$$

$$CaCO_3(s) + 2H^+(aq) = H_2CO_3(aq) + Ca^{2+}(aq) \quad \textbf{(12)}$$

Of the carbonate minerals, calcite ($CaCO_3$) dissolves most rapidly (Busenberg and Plummer 1986). Relative to calcite, the rate of dolomite [$(Ca,Mg)CO_3$] dissolution is about an order of magnitude slower (Busenberg and Plummer 1982) and the rate of magnesite ($MgCO_3$) dissolution is about four orders of magnitude slower (Chou et al. 1989). The rate of siderite ($FeCO_3$) dissolution under anoxic conditions is reported to be three orders of magnitude slower than that of calcite (Greenberg and Tomson 1992). Iron and manganese carbonates do not provide net acid neutralization under oxidizing conditions, due to acid production from hydrolysis and precipitation of oxidized iron or manganese (e.g. reactions 7, 8).

Dissolution of silicates such as plagioclase-feldspars (e.g. anorthite in equation 13, from Busenberg and Clemency 1976) and olivine (e.g. forsterite in equation 14, from Hem 1970) can also neutralize acid. However, their rates of dissolution and consequent acid neutralization are slow relative to the carbonate minerals (Nesbitt and Jambor 1998). For example, at near neutral pH, the dissolution rate of calcite is approximately 7 orders of magnitude faster than the dissolution of plagioclase feldspar (White et al., 1999). Nonetheless, silicate mineral dissolution can maintain neutral conditions if the rate of acid production is quite slow (Lapakko and Antonson 1993; Lapakko et al. 1997; Lapakko and Antonson 2002). The effectiveness of silicate minerals in neutralizing acid rises with increasing mineral surface area, which increases with elevated mineral cation content and decreasing grain size.

$$CaAl_2Si_2O_8(s) + 2H^+(aq) + H_2O(aq) = Ca^{2+}(aq) + Al_2Si_2O_5(OH)_4(s) \quad \textbf{(13)}$$

$$Mg_2SiO_4(s) + 4H^+(aq) = 2Mg^{2+}(aq) + H_4SiO_4(aq) \quad \textbf{(14)}$$

The dissolution of plagioclase in the neutral range (above approximately pH 4.3) produces kaolinite ($Al_2Si_2O_5(OH)_4(s)$, equation 13), which produces gibbsite (equation 15, from Stumm and Morgan 1981).

$$Al_2Si_2O_5(OH)_4(s) + 5H_2O = 2H_4SiO_4 + Al_2O_3 \bullet 3H_2O(s) \quad \textbf{(15)}$$

As aluminum becomes increasingly soluble near pH 4, gibbsite dissolution increases considerably, and is capable of neutralizing 6 moles of acid. So, with every mole of Ca released from plagioclase, there is a potential additional neutralization from gibbsite dissolution. This assumes that the alteration rate of kaolinite to gibbsite is faster than neutral range plagioclase dissolution. This situation presents the possibility that gibbsite could limit pH minima to approximately 4, if sulfur oxidation was slowed before gibbsite was depleted. Sulfide mineral oxidation could either be slowed by depletion of minerals in low-sulfur samples, or by coating of sulfide grains in higher-sulfur samples. This mechanism of acid neutralization would ultimately be limited by the rate of plagioclase dissolution. After the gibbsite is depleted, neutralization would be dependent on silicate mineral dissolution exclusively.

3.1.3. Trace Metal Release by Trace Metal Sulfide Oxidation

Trace metals are metals that occur at low average concentrations in the earth's crust but can be present in higher concentrations in mineralized areas. Trace metals can occur in sulfide minerals, from which they can be released by oxidation (e.g. reaction 16). Once released to solution, the migration of these metals can be influenced by a number of factors. At a regional scale, generalizations frequently can be used to estimate broad trends in metal mobility. However, as the scale becomes increasingly finer, estimating metal behavior generally becomes increasingly difficult (Smith and Huyck 1999).

$$ZnS(s) + 2O_2(aq) = Zn^{2+}(aq) + SO_4^{2-}(aq) \quad \textbf{(16)}$$

In general, metals may remain in solution or be removed in secondary phases. For removal from solution, trace metals may precipitate as oxides, hydroxides, or carbonates, be adsorbed by surfaces such as iron oxyhydroxides (Smith 1999), or co-precipitate with other solid phases. In acidic solutions trace metal removal is limited, and elevated trace metal concentrations are often associated with these solutions. However, circumneutral drainages can also contain elevated concentrations of trace metals such as nickel, copper, cobalt (Lapakko 1993c), zinc, manganese (Smith and Huyck 1999), molybdenum (Brown 1989), arsenic, and antimony. Concentrations of molybdenum, arsenic, and antimony in particular can be elevated even as pH increases above 7.

Oxidation of arsenic and antimony sulfides can produce acid, as can oxidation of the iron sulfide fraction of mixed sulfide minerals such as chalcopyrite (Plumlee 1999). Other trace metal sulfide oxidation will produce acid if and only if the metal released hydrolyzes (reaction 17) or precipitates as a hydroxide, oxide, or carbonate (reaction 18). For most trace metals this will occur only at pH levels above 6, and as pH decreases below this level the secondary phases will dissolve. Consequently, they do not generally contribute to acid production observed at lower pH levels.

$$Zn^{2+}(aq) + H_2O(aq) = Zn(OH)^+(aq) + H^+(aq) \quad \textbf{(17)}$$

$$Zn^{2+}(aq) + 2H_2O(aq) = Zn(OH)_2(s) + 2H^+(aq) \quad \textbf{(18)}$$

3.1.4. Key Solid-Phase Factors Controlling Mineral Oxidation and Dissolution

Whereas the balance of the acid producing and acid-neutralizing minerals influence mine waste drainage quality, there are several mineralogical factors that substantially influence drainage quality. Individual minerals may be entirely liberated from the rock matrix, occur interstitial to other minerals (partially liberated), or as inclusions within other minerals. The extent of liberation affects availability for reaction. In addition, acid-producing or acid-neutralizing minerals present as inclusions within unreactive minerals (e.g. quartz) will be essentially unavailable for reaction.

Oxidation of sulfide minerals and dissolution of some carbonate minerals are surface reactions and, therefore, the rates of these reactions are dependent on the reactive surface area. Reactivity decreases as mineral surfaces are covered by non-reactive reaction byproducts, such as iron oxyhydroxides (precipitants from iron sulfide oxidation), whereas the concentration of lattice defects increases mineral surface area, and therefore tends to increase reactivity. Mineral surface area is also dependent on the extent to which the mineral is liberated from the rock matrix, mineral grain size, and the "roughness" of the mineral surface.

3.1.5. Common Minerals and Specific Factors Controlling Mineral Oxidation and Dissolution

3.1.5.1. Sulfide Minerals

The most common sulfide mineral in these samples was pyrrhotite. Pyrrhotite has a chemical formula of $Fe_{1-x}S$, where $0 \leq x \leq 0.2$. Thus the pyrrhotite formula may range from FeS to Fe_7S_8. Pyrrhotite has a specific gravity of 4.58-4.65 and a hardness of 4 (Klein and Hurlbut 1985). The FeS end member has an orthorhombic structure. $Fe_{1-x}S$ such as Fe_8S_9, Fe_9S_{10} and $Fe_{10}S_{11}$, have hexagonal structures and Fe_7S_8 is monoclinic. Pyrrhotite is typically found as mixtures of hexagonal and monoclinic phases. These different crystal structures are thought by some researchers to have effects on oxidation rates (see equation 2 for oxidation reaction). However, Janzen (1996) found that the literature on this topic gave conflicting reports of the effect of crystal structure on oxidation rate. Furthermore, in his own experiment, Janzen found no correlation between pyrrhotite oxidation rates and crystal structure in the presence of dissolved oxygen or ferric iron.

3.1.5.2. Silicate Minerals

Typical silicate minerals in these samples were labradorite from the plagioclase feldspar group, forsterite as a reasonable representative of the olivine group, biotite from the mica group, as well as augite and bronzite (as a substitute for hypersthene) from the pyroxene group. Each of these minerals have published research on dissolution rates available, as well as information on physical factors that affect dissolution rate, such as specific gravity, hardness and surface roughness. Research available on these minerals in 2001, as well as six other silicate minerals present in the Duluth Complex, was compiled in a literature survey (Phillips et al. 2001). The rest of the material in this section is taken from that report, which has a more detailed treatment of this topic available.

Plagioclase feldspar forms a solid solution series from albite ($NaAlSi_3O_8$), the sodium-rich end member, to anorthite ($CaAl_2Si_2O_8$), the calcium-rich end member. Labradorite ($An_{50-70}Ab_{30-50}$; notation indicates 50-70% anorthite and 30-50% albite) was the plagioclase feldspar selected as representative for these samples. It has a specific gravity of 2.71 and a hardness of 6 (Klein and Hurlbut 1999). No data for surface roughness were available, but bytownite (An_{70-90}) had a range of 6.6-27.1 and a mean of 12.6 (Blum 1994). Blum and Stillings (1995) express the overall feldspar weathering process as the

alteration of feldspar to common clay minerals (reaction 19).

$$2NaAlSi_3O_8(s) + 2H^+(aq) + 9H_2O(aq) = 2Na^+(aq) + Al_2Si_2O_5(OH)_4(s) + 4H_4SiO_4(aq) \quad (\mathbf{19})$$

Olivine forms a solid solution series from forsterite (Mg_2SiO_4) to fayalite (Fe_2SiO_4). Olivine has a specific gravity of 3.27-4.37 and a hardness of 6.5-7.0 (Klein and Hurlbut 1999). A surface roughness factor of 85 is reported for olivine (Sverdrup 1990). The weathering of forsteritic olivine is expressed by reaction 14 (Hem 1970).

Biotite ($K(Mg,Fe)_3(AlSi_3O_{10})(OH)_2$) is a phyllosilicate with a specific gravity of 2.8-3.2 and a hardness of 2.5-3.0 (Klein and Hurlbut 1999). Sverdrup (1990) reported a surface roughness factor of 17. The first stage of biotite weathering in which biotite is converted to vermiculite is expressed by reaction 20 (Acker and Bricker 1992).

$$3K_2(Mg_3Fe_3)\ (Al_2Si_6O_{20})(OH)_4(s) + 8H_2O + 12H_2CO_3 + 6H^+(aq) + 3/2O_2 = 2[(Mg_3Fe_3)(Al_3Si_5O_{20})(OH)_4 \bullet 8H_2O] + 6K^+ + 3Mg^{2+} + 3Fe^{2+} + 12H_2CO_3^- + 8\ SiO_2 + 3H_2O \quad (\mathbf{20})$$

Pyroxenes are inosilicates, or silicates that form chain structures by sharing oxygen atoms. The pyroxenes form single chain structures that can be divided into two groups, orthopyroxene and clinopyroxene, based on their crystal systems. Orthopyroxenes form in the orthorhombic crystal system and include hypersthene. Clinopyroxenes form in the monoclinic crystal system and include augite.

Hypersthene has only limited research into its dissolution, so bronzite was used as a substitute. Both are orthopyroxenes, with hypersthene having 50-70% Mg and bronzite having 70-88% (Klein and Hurlbut 1985). Hypersthene and bronzite ($(Mg,Fe)SiO_3$) have specific gravities of 3.3-3.5 and hardnesses of 5-6 (Klein and Hurlbut 1999). Sverdrup (1990) reported a surface roughness factor of 17.

Augite ($(Ca,Na)(Mg,Fe,Al)(Si,Al)_2O_6$) is a clinopyroxene with a specific gravity of 3.2-3.4 and a hardness of 5-6 (Klein and Hurlbut 1999). Sverdrup (1990) reported a surface roughness factor of 50.

3.2. Duluth Complex Geology

3.2.1. General Description

The Duluth Complex is one of the largest mafic intrusive complexes in the world. It covers roughly 6500 km^2, ranging from the Duluth, MN area northward 240 km and is over 50 km wide at some points (Figure 1, Miller et al, 2001). Geologic contacts on the north and west are easily recognized, where the mafic intrusives and volcanics of the Duluth Complex form the headwall of the contact, and metagreywackes and slates of the Animike Group form most of the footwall of the contact. The Animike Group is composed primarily of the Virginia Formation and Biwabik Iron Formation that dominate the Animike metasediments near the contact. The Virginia Formation is a well-bedded sequence of argillaceous siltstone, carbonaceous shale, mudstone, fine-grained feldspathic greywacke, and minor limey-cherty interbeds variably contact metamorphosed to hornfels facies by the Duluth Complex. The Biwabik Iron Formation is a well bedded, iron-bearing strata of alternating cherty intervals also variably contact metamorphosed by the Duluth Complex. The Giants Range Batholith, predominantly late Archean quartz monzonite, granite and monzodiorite, also forms a portion of the western footwall (Figure 1).

The Duluth Complex has been classified into four major series of intrusions on the basis of age, lithology, internal structure, and structural position (Weiblen & Morey, 1980). These series (in order of

emplacement) are the Felsic series, the Early Gabbro series, the Anorthositic series, and the Layered Series. The Layered Series hosts all known sulfide mineralization, including the South Kawishiwi and Partridge River Intrusions, and has the highest potential for undiscovered mineralized zones (Severson, 1994).

The South Kawishiwi is a southeast-dipping, layered igneous intrusion forming part of the northwestern margin of the Duluth Complex. It is predominantly composed of troctolitic (plagioclase-olivine) cumulates with local concentrations of anorthositic rock and volcanic hornfels inclusions.

3.2.2. Mineralization

Sulfide and platinum group element (PGE) mineralization occurs near the basal contact between the country rock and the intrusive complex. Four distinct types of magmatic mineral deposits are recognized. The first two deposit types include (1) large-low grade disseminated Ni-Cu concentrations, and (2) localized high-grade zones of massive Ni-Cu sulfides, both with some local PGE enrichment. These two types occur only at or very near the basal contact zone, and are the types found near the Dunka Mine (Figure 2, Miller et al, 2005). Other types include (3) strata-bound reefs with PGE enrichment, and (4) oxide-rich ultramafic plugs, possible sources of titanium and vanadium. These types can be found both in the basal contact zone and stratigraphically higher in the complex.

Several sulfide deposits are associated with the basal contact of the Duluth Complex with the Animike Group and the Giants Range Batholith. Identified deposits include the Dunka Mine and the Babbitt/Mesaba deposit, among other sulfide mineral deposits (Figure 3). The Dunka Mine lies on the contact of the Giants Range Batholith, the Animike Group, and the Duluth Complex. South Kawishiwi Intrusion rock was excavated to mine the underlying Biwabik Iron Formation, and comprises much of the waste rock at the mine. The dominant Duluth Complex lithology at the mine is olivine gabbro with occurrences of norite and olivine norite. Sulfide mineralization is concentrated within these rocks, especially those near the contact with the underlying rocks. The footwall of the contact with the South Kawishiwi intrusion is composed of Virginia Formation along most of its length, but borders the Biwabik Iron Formation and the Giants Range batholith along strike to the northeast and down dip to the southeast (Figure 2).

The Babbitt/Mesaba deposit is located five miles southwest of the Dunka Mine and is situated within very similar lithologies. Sulfide bearing rocks are found in a very similar setting in the Partridge River Intrusion, a layered cogenetic mafic layered intrusion of the Duluth Complex south of the South Kawishiwi Intrusion (Figure 1). The Partridge River intrusion is composed of a complex assemblage of gabbroic to troctolitic rocks, basaltic hornfels, and olivine gabbro, all of which are cross cut by heterogeneous troctolitic rocks ascribed to the upper Partridge River intrusion. Extensive exploratory drilling has defined the Babbitt/Mesaba copper-nickel-PGE sulfide deposit along the basal contact zone of the Partridge River intrusion. Additional information on prospects and deposits within the Duluth Complex is available in publications by the Natural Resources Research Institute (2013).

3.3. Previous Work

Experimental results after 150 weeks of laboratory dissolution were described previously (Lapakko 1993b; Lapakko and Antonson 1993), and included results for four Dunka mine blast hole samples (2.06%S ˜S_{TOT}˜ 5.44%S) for which testing was terminated after 78 weeks. The results from this laboratory testing displayed similarities to those from field test piles of Duluth Complex rock (Lapakko 1994b, 1993c). An earlier dissolution experiment of shorter duration was conducted on 14 drill-core samples and a test-shaft sample from the Duluth Complex. Observations related to the rock chemistry,

mineralogy, and surface area were made and initial results pertaining to the oxidation of sulfide minerals, the dissolution of silicate minerals, and associated effects on drainage chemistry were reported (Lapakko 1993a, 1988). In addition, experimental results after 8-10 years of laboratory dissolution on Partridge River Intrusion waste rock and tailing samples were recently reported and provide a comparison with the South Kawishiwi Intrusion samples discussed in this report (Lapakko et al. 2013a, 2013b).

4. METHODS

4.1. Materials

This laboratory experiment examined the effects of Duluth Complex and Virginia Formation rock dissolution on water quality. Twenty-five crushed rock samples ($0.053 \leq d \leq 0.149$ mm), ranging from 0.18-5.44% sulfur content (Table 1), were subjected to weekly rinses of deionized water. Nineteen samples, each replicated twice, were initiated on February 1989 (14 samples, reactors 1-28) and September 1990 (5 samples, reactors 29-38). These samples originated from blasts between December 1988 and October 1989 and were stored for 9 to 64 weeks prior to initiation of dissolution tests. Even numbered experimental reactors from 1-38 contained the same rock sample as the preceding odd numbered reactor. Fifteen of these samples were from the Duluth Complex and four contained a substantial fraction of metamorphosed rock from the underlying Virginia Formation. Six additional samples in the middle range of sulfur content were added in August 1997. These samples originated mostly from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation of dissolution tests. These final six samples were analyzed more frequently, thoroughly based on what was learned from prior samples, and were not replicated. All samples were chosen for low metal concentrations, so metal concentrations were in a narrow range for all samples.

The Duluth Complex samples came from three locations. All seventeen Dunka blast hole samples were from the South Kawishiwi intrusion near Babbitt, Minnesota, and the one ARIMETCO and the three AMAX samples came from the Babbitt/Mesaba deposit of the Partridge River intrusion.

The four Virginia Formation samples (Reactors 21-28, 2.06 ~~%S ~~5.44) were terminated after 78 weeks of operation (August 1990). The data at this time indicated that waste rock with sulfur content in this range would produce drainage in need of rigorous environmental controls. It was concluded that testing of rock with lower sulfur contents would provide information more beneficial to environmental mine waste management planning than continuation of the higher sulfur samples already known to produce strongly acidic drainage. Drainage quality from first 15 Duluth Complex samples replicated well (see section 5.2.1), so even numbered reactors in the ranges 2-20 and 30-38 were discontinued after 247-328 weeks to save time and resources. The 1.71% S sample introduced in 1997 was terminated after 144 weeks. All other reactor data presented in this report were for periods ranging from 360-1252 weeks (Table 1).

4.2. Reactor Design and Operation

All rock samples begun in 1989 and 1990 (reactors 1-38) were reduced by hand, through the use of a bucking maul, or mechanically, with a pulverizor in 1989. The solids were then crushed to particles with diameters from 0.053 to 0.149 mm (-100/+270 mesh) and analyzed for sulfur content. The maximum diameter was selected to ensure sufficient liberation of the sulfide minerals present (Lapakko, 1988). These solids were then sieved and placed in reactors. Samples with 1.12, 1.16, 1.40, 1.44, and 1.64%S were wet-sieved, while the remaining samples were dry-sieved (Lapakko, 1988).

Samples begun in 1997 (reactors 39-44) were stage crushed with a jaw crusher. After each crushing step the sample was wet-sieved to obtain the –100/+270 fraction. This was continued until an adequate sample mass was obtained. These samples received initial crushing prior to storage. Reactor 39 was crushed to –1.9 cm and stored in 1996. All other samples crushed to –0.64 cm prior to storage. Reactors 40 and 43 were crushed in 1989, reactors 42 and 44 in 1977, and reactor 41 in 1976.

Seventy-five grams of crushed rock were placed into the upper segment of a two-stage filter unit (Figure

4). The solids were placed on a glass fiber filter that rested on a perforated plastic plate near the bottom of the reactor. At the inception of the experiment, each reactor was rinsed three times to remove any oxidation products that accumulated between sample crushing and onset of the experiment. Each week thereafter, two hundred milliliters of distilled-deionized water was added, and remained in contact with the solids for four to seven minutes, and then vacuum-pumped from the upper stage through a 0.45-micron filter on top of the lower stage of the filter unit. This procedure changed slightly beginning in July 2002. During weeks where no sample was collected, the reactors were filled with water as before, but were then gravity drained. The procedure remained the same as before for weeks during which samples were collected.

Between rinses, the solids remained within the reactors, allowing evaporation of retained rinse water and continuous oxidation. The reactors were stored in individual cubicles that formed a rectangular matrix within a topless housing with a perforated base. The cells were stored in a temperature and humidity controlled room (8.5 x 10.5 x 9.5 ft).

4.3. Analyses

4.3.1. Replication, Environmental Conditions, and Water Retention

4.3.1.1. Analysis of Humidity and Temperature Control

Between sampling, reactors were stored in a room with a thermostatically controlled heater (Lakewood 750). A Holmes HM 3500 humidifier and Sears Kenmore dehumidifier, both controlled by Honeywell humidistat, maintained humidity. From February 14, 1989 to February 11, 2013 temperature and relative humidity were measured three to four times weekly with a Taylor wet-bulb, dry-bulb hygrometer, and mean weekly values were determined. Range, mean and standard deviation of temperature and humidity of the average weekly values were determined to assess effectiveness of environmental controls in the testing chamber.

4.3.1.2. Testing for Differences between Replicates

All samples initiated in 1989 or 1990 were replicated. Average releases of calcium (Ca), magnesium (Mg) and sulfate (SO_4), as well as minimum pH measurements were compared between the two replicates of these 19 samples. These measurements were determined only for the period of record in which both replicate reactors were running (78-328 weeks). Differences between replicates were tested using the paired t-test in Systat 10.2. Details of drainage analysis are presented in section 5.2.1.

4.3.1.3. Analysis of Water Retention and Evaporation of Samples

In a six-week trial during 2003, eight samples were weighed daily, with the exception of weekends, to measure water retention in the sample and evaporation rate of the retained water. Mean amount of water retained and mean rate of evaporation were determined.

4.3.1.4. Analysis of Effect of Rinse Volume on Drainage pH

The effect of rinse water volume on drainage pH of samples was testing using two volumes: 10 and 200 ml. This was done in sixteen reactors by separately collecting the first approximately 10 ml of drainage water to leave the reactor. After the first 10 ml was collected, the vacuum pump was briefly turned off to allow another sample vessel to be positioned to collect the remaining leachate. The pH for each 10 ml sample was determined, before adding this volume to the rest of the drainage and determining pH for the

entire volume. All reactors chosen had a relatively stable pH for eight weeks prior to this experiment (Table 20). pH from different volume drainages was compared using a paired Student's t test.

4.3.2. Characterization of Solid Samples

4.3.2.1. Particle Size

Effects of particle size were assessed in two ways using an electron microprobe. First, the perimeter to area ratio for a large number of grains in each sample was determined as an overall indicator of particle size. Second, the extent of small particles adhering to the surface of larger particles was assessed by comparing washed and unwashed grains.

Selected samples were analyzed to determine the extent of grains finer than that intended by sieving (-100/+270 mesh or 49-153 microns). As an indicator of particle size, three images of multiple grains (two for 0.40%S unleached and 0.58%S leached, four for 1.16%S leached) were analyzed from leached and unleached samples with 0.22-1.71%S (Table 17). These images were processed using *NIH ImageJ* software to analyze the total number of particles, their area and circumferences. From these data, a single value was calculated to give the cumulative perimeter to area ratio (McSwiggen and Associates, P.A., Saint Anthony MN). This ratio was used as a surrogate for particle size to compare among samples and to assess the effects of leaching. These data were also used to determine average particle diameter by assuming spherical particles and applying the formula specified in Table 17.

Cleaned and uncleaned grains of unleached and leached samples containing 0.22 and 0.40%S were imaged. To clean the samples, mineral grains were placed in distilled alcohol for five minutes, and then placed in an ultrasonic cleaner for an additional five minutes before analysis. Images of these grains allowed a qualitative comparison of the amount of fine particulates on the surface of leached and unleached samples.

4.3.2.2. Chemistry

4.3.2.2.1. Whole Rock Chemical Composition

Initial sulfur content was determined with a LECO furnace (Lerch Brothers, Inc., Hibbing, MN) using ASTM 395 (ASTM 2000). ACTLABS, Inc. (Tucson, AZ) determined SO_4 and evolved carbon dioxide. SO_4 was determined for leached samples only using ASTM E-1915-97 (ASTM 2000). Evolved carbon dioxide was analyzed using ASTM E350-89C (ASTM 2000). A 10% hydrochloric acid solution was used to dissolve the carbonate minerals, and the carbonate present was quantified as the difference between total carbon in the initial sample and that in the residue. Whole rock oxide percentages were determined using a lithium tetraborate fusion modified from ASTM E886-94 (ASTM 2000) and analysis by inductively coupled plasma-atomic emission spectroscopy (ICP-AES) using a Thermo Jarrell-Ash ENVIRO II ICP (ACTLABS, Inc., Ancaster, ON). Chemex Labs (Sparks NV) also conducted similar whole rock chemistry on a subset of these samples. These results are not discussed in the text, but are presented in Appendix 1, Attachment A1.4.

Concentrations of Ag, Cd, Cu, Ni, Pb, Zn, and Bi were determined using a total digestion modified from Crock et al. (1983), with analysis by ICP-AES (ACTLABS, Inc., Ancaster, ON). Other trace elements were determined using instrumental neutron activation analysis (Hoffman 1992) (ACTLABS, Inc., Ancaster, ON).

4.3.2.2.2. Solid-Phase Acid Production and Neutralization Potential

Acid production potential was determined mathematically as %S x 31.18. This calculation was based on production of 2 moles of acid per mole of S, and expressing this value as kg $CaCO_3$ equivalents/ton rock.

Two methods were used to determine neutralization potential for the initial nineteen samples (those started in 1989 and 1990). The first was the Standard ABA Sobek method, in which the sample was crushed to pass a 60-mesh screen (diameter <0.25 mm). Approximately 0.5 g of crushed sample was placed on aluminum foil with one or two drops of one part hydrochloric acid (HCl) to three parts deionized water. Based on extent of "fizzing" in this test, the volume and concentration of HCl were selected and 2 g of crushed rock was combined with the HCl solution in a 250-mL Erlenmeyer flask. The rock-acid mixture was boiled until the reaction ceased, as indicated by the termination of the production of bubbles. The solution was diluted to 125 ml using distilled water, boiled for 1 min and cooled to room temperature. The mixture was then titrated to pH 7.0 with sodium hydroxide (NaOH) to determine the amount of acid neutralized by the sample (as $CaCO_3$). The NP (acid neutralization per unit mass rock) was expressed in units of kilograms of $CaCO_3$ per metric ton of mine waste (kg $CaCO_3$/MT)

The second method used was the Modified ABA determination of neutralization potential (Lawrence 1990). The acid/rock mixture was prepared as above and digested at room temperature for 24 hours, at which time pH was measured. If pH was not in the range of 1.5-2.0, the digestion was rerun with an adjusted acid addition, based on the observed pH, until the pH range of 1.5-2.0 was attained. After the pH criterion was met, the amount of acid neutralized by the rock was determined by titrating the acid/rock mixture NaOH until a pH endpoint of 8.3 was maintained for at least 30 seconds. The NP of the lithology is then recorded in kg $CaCO_3$/MT.

A third method, a slight modification of the NP(pH6) method (Lapakko 1994a), was used in the determination of NP, but only for the final six samples of this study (reactors 39-44). NP was determined by titrating a stirred mixture of 2 g sample (-100/+270 mesh) in 100 ml distilled water with 1.0N sulfuric acid. The titration was conducted using a Mettler DL21 automatic titrator equipped with a Ross combination pH electrode. The instrument was programmed for a pH stat titration with an endpoint of pH 6.00 and the minimum increment of acid addition was 0.005 ml. A 2 g sample was placed in a 150 ml glass beaker with 100 ml of distilled deionized water. The beaker was placed in a Thermolyne model 7200 auto stirrer and a Teflon magnetic stirring bar provided constant mixing of the solution. The pH probe was suspended in the beaker at a depth of approximately 2 cm above the stirring bar. pH, incremental and cumulative acid additions were recorded every thirty minutes. The titration continued until pH 6.0 was reached and less than 0.1 ml of acid was added over a period of four hours.

4.3.2.3. Mineralogy

4.3.2.3.1. Mineral Content and Chemistry

The mineral content and mineral chemistry of leached and unleached samples were determined by microprobe analysis of 100 to 125 mineral grains from each sample. Every attempt was made to pick the grains in a random manner, so that each mineral would be analyzed in proportion to their occurrence in the sample. Grains were either loose mounted on carbon tape or mounted in epoxy. Mineral chemistry analyses were conducted using a JEOL 8600 electron microprobe. Mineral standards were used for all calibrations. See Appendix 1, Attachment A1.2 for detailed methods.

In addition to microprobe determination, sulfide mineral contents were determined through a calculation based on the bulk chemistry of each sample for pyrrhotite (Po), chalcopyrite (Cp), and pentlandite (Pn). Solid-phase sulfur within each Duluth Complex sample was attributed to chalcopyrite (approximately 20% of copper associated with cubanite), pentlandite and pyrrhotite based on the presence of copper, nickel and iron along with solid-phase sulfur. This is a more accurate method of assessing sulfide speciation. This method assumes the lithology formed under equilibrium conditions, and the results are vulnerable to error if this assumption is incorrect.

Additionally, eight of the unleached samples analyzed above (0.40-1.44%S) had mineral and chemical composition determined by an independent lab (Midland Research Center, Nashwauk, MN). The McSwiggen data are presented in the text, while the Midland data are presented in Appendix 1, Attachment A1.3.

4.3.2.3.2. Mineral Evidence of Dissolution

Microprobe analysis (back scatter imaging of polished epoxy mounts) was used to assess visual evidence of dissolution in unleached and leached mineral grains. Evidence of leaching occurred in the form of pitting of grains, as well as the formation of coatings indicating oxidation. The thickness of coatings on mineral grains was assessed by examination of polished the epoxy-mounted samples.

4.3.3. Aqueous

4.3.3.1. pH, Conductivity and Alkalinity

The rinse water from each reactor was analyzed for specific conductance, pH, alkalinity and acidity at the Minnesota Department of Natural Resources (MDNR) laboratory in Hibbing, MN (see Table 6 for sampling frequency). Specific conductance was analyzed using a Myron L conductivity meter, while either a Radiometer 29 or an Orion SA720 meter was used for pH analyses. Alkalinity (for pH≥6.3) and acidity were determined using standard techniques for endpoints of 4.5 and 8.3, respectively (APHA et al. 1992).

4.3.3.2. Sulfate, Calcium, Magnesium and Metals

The volume of filtered effluent was determined by weighing the drainage. The rinse water was analyzed for SO_4, Ca, Mg, Na, K, Co, Cu, Ni, Zn, Fe, Al and Si. The timing of measurements for all reactors is listed in Table 6. Samples taken for metal analyses were acidified with 0.2 ml ultra-pure nitric acid per 50 ml sample.

Sulfate and metal concentrations were determined at the MNDNR lab in Hibbing, MN until 17 May 1994 (representing weeks 273 and 192 for the first two sets of samples). SO_4 was analyzed using an HF Scientific DRT-100 nephelometer for the barium sulfate turbidimetric method (APHA et al., 1992). Ca and Mg were analyzed with a Perkin Elmer 603 atomic absorption spectrophotometer (AAS). Subsequent metal and SO_4 determinations were conducted at the Minnesota Department of Agriculture (MDA) laboratory. From 17 May 1994 to 23 August 1999, metals were determined by AAS (Varian 400 SPECTRAA) with a Zeeman GFAA furnace was attached for low concentrations. Ca, Mg, sodium (Na), and potassium (K) were analyzed by AAS throughout the experiment. Subsequent analyses for metals other than Ca, Mg, Na, and K were conducted using inductively coupled plasma mass spectrometry (ICP-MS, Hewlett Packard HP4500 Series, model #G1820A). For $[SO_4] > 5$ mgL^{-1}, SO_4 was determined using a Technicon AA2 automated colorimeter. Lower concentrations were determined using a Dionex ion chromatograph and, after 10 November 1998, a Lachat QuickChem 8000.

4.4. Calculations

4.4.1. Problems with Ca and Mg Determinations Prior to Week 150

Ca and Mg were the predominant cations in drainage samples and SO_4 was the predominant anion. Calculation of the molar ratio of the sum of Ca and Mg concentrations to the SO_4 concentration for the initial 150 weeks indicated an excess of cations. The mean, median, and standard deviation for the ratios were 1.79, 1.39, and 1.41, respectively. It was recognized that this charge imbalance was unrealistic and it was suspected that the Ca and Mg analyses were in error. The secondary cations for these reactions are K and Na. The data for these cations also showed several similar inconsistencies and remains suspect throughout the dataset.

Unfortunately, only four to six values from the last 13 to 24 weeks were available for most of the reactors and no values were available for the reactors containing 2.06 and 3.12 percent sulfur (reactors 21 to 24). Initially the samples were analyzed using a 1:20 dilution factor. This may have been the source of the problem, allowing for erroneous calculation of the concentration of the cations present. The remaining available samples were re-analyzed using a 1:5 dilution to provide more accurate calculations of the cation concentrations. Lanthanum chloride was added to the blank used for the standardization, a procedure that was omitted in the initial analysis.

Reanalyzed Ca and Mg values yielded more reasonable charge balances based on molar ratios of the sum of Ca plus Mg concentrations to SO_4 concentration. The mean, median, and standard deviation of the ratios for the reanalyzed samples were 0.72, 0.65, and 0.29, much lower than the previously calculated results, as would be expected with decreasing pH. Average ratios for the three solids with drainage pH above 6.0 ranged from 0.92 to 1.27. For lower drainage pH, ratios ranged from 0.49 to 0.81. Lower ratios (indicating a cation deficit) at lower pH were assumed to result from significant concentrations of cations other than Ca and Mg in the more acidic drainages.

Regression analysis was used to define a relationship between the initial (1:20 dilution) analyses and the secondary (1:5 dilution) analyses. The initial analyses for Ca and Mg were then adjusted by linear transformation using the relationship defined. The molar ratios of the sum of the transformed Ca and Mg concentration to SO_4 concentration were then determined. The mean, median, and standard deviation of the molar ratios using the transformed concentrations were 1.51, 0.79, and 3.29. The large mean and standard deviation were the result of ratios that were unusually high for 41 of the 593 values transformed. These ratios ranged from 2.5 to 29, averaging 10.6. Of the 41 elevated values, 38 occurred between weeks 12 and 60, suggesting that the initial analyses during these weeks were particularly prone to error.

The more reasonable charge balance data indicates that transformed Ca and Mg concentrations are more accurate than the original analyses. Nonetheless, these values were determined by an unorthodox approach and, consequently, conclusions based on these data must be drawn with caution. Additional detail on the Ca and Mg data is presented by Lapakko (1993b). The problem was corrected after week 150 for the original reactors.

4.4.2. Chemical Mass Release

Cumulative released masses of sulfate, major cations, and trace metals were calculated as the product of observed concentration in the drainage and drainage volume. Missing concentrations were estimated as the average of previous and subsequent analyzed values. Initial values were averaged between the final pre-experiment rinse and the first observed value taken, except for samples started in 1997, which did

not have pre-experiment rinses analyzed. It was assumed that experimental values after the last observed value (and prior to observed values for 1997 samples) were identical to the nearest observed value. Values reported as less than the detection limit were recorded as equal to one-half the detection limit.

4.4.3. Empirical Neutralization Potential for Drainage

Empirical neutralization potentials (ENP) were calculated to estimate the amount of acid neutralized by silicate dissolution prior to drainage pH decreasing and remaining below pH values of 7, 6, 5, 4.5, 4 and 3.5. The acid neutralized was calculated as the cumulative SO_4 release (in mg $CaCO_3$/g rock, see Table 21 for exact formula) prior to the point at which drainage pH decreased and remained below the specified pH value. If drainage pH from a given solid never decreased permanently below a specified value, the ENP was reported as "greater than" the total SO_4 release for the period of record. If drainage pH was continuously below the specified pH, ENP for that pH was reported as zero.

4.4.4. Rates of Release

Solute release rates of SO_4, Ca, Mg, Na, K, Co, Cu, Ni and Zn were determined for each sample from observed concentrations of each respective ion within the leachate. Iron, Al and Si release rates were calculated separately for Duluth Complex and Virginia Formation samples, and determined as the average of all samples in each group. Release was calculated as the product of observed concentration (mol) and volume of drainage (l), divided by number of seconds in a week. Rates of solute release were determined based on weekly concentration measurements.

SO_4 release (log transformed) and pH were plotted as a function of sulfur content (log transformed) using linear regression. This was done for the four sample periods detailed in section 4.4.5.1. Calcium, Mg, and Ca+Mg release (all log transformed) were plotted as a function of SO_4 release (log transformed) using linear regression during the four time periods.

During a three week period (July 1 to July 21, 2011) the ongoing samples to date (0.18, 0.22, 0.67 (39, 40), 0.72, 0.82, 0.92%S) where not rinsed due the State of Minnesota shutdown. Because of this change to the rinsing frequency, adjustments were made to the associated rate calculations. Observed sulfate, calcium, and magnesium release after the shutdown (week 728 for R39-R43, week 1171 for R1-R3) where divided by four in order to account for accumulated mass in the sample. For cumulative mass release calculations, zero mass release was entered for the weeks during the shutdown and the higher observed release measured after the shutdown was retained. For concentration vs. time figures (Appendix 3, Figures A3.1-A3.44) no adjustment was made. It should be noted that this change in rinsing frequency only affected 0.3-0.5% of the measured values during the entire period of record and had minimal effect on the overall release rate calculations.

4.4.5. Mineral Dissolution Rates

Five models were developed to determine ion release rates from major silicate and sulfide minerals in each sample. Silicate mineral dissolution and sulfide mineral oxidation rates were based on observed solute release rates, assumed mineral sources of solutes, determined mineral chemistry, and mineral surface areas. For each sample, mineral-specific surface areas were based on mineral content determined, calculated geometric surface areas, and surface roughness factors reported in the literature (Appendix 5, Table A5.1). Additionally, maximal gibbsite production was determined based on stoichiometric calculations.

4.4.5.1. Silicate Mineral Dissolution Rates

Four modeling approaches were used, all of which assumed stoichiometric release of major cations (Ca, Mg, Na, K) from the most common minerals in these samples (plagioclase, augite, hypersthene, olivine, biotite). That is, the ratio of major cations released to solution during the dissolution of each mineral was equal to that which existed prior to dissolution in each sample. The models are presented in order of most to least appropriate for these data. The first model was chosen as most appropriate because it is based on plagioclase, by far the most common mineral in these samples, and because the model is primarily based on Ca, which was thoroughly sampled throughout the experiment for all samples. Sodium, the basis for model 3, would normally be the preferred element to model for these samples because it is almost entirely exclusive to plagioclase, as well as not having any dramatic changes in solubility over the pH range found in this experiment. The reason it is not the basis for the preferred model in this experiment is the lack of Na analysis during the first eight years for the majority of samples. Magnesium was used in model 2 to determine augite dissolution, which was the basis for that model. Potassium was used as the basis for model 4, with plagioclase dissolution based on sodium as in model 3. Aluminum was not used in any models due to its infrequent sampling and low solubility until pH was lower than 5.

These models were applied to four periods of drainage conditions: (A) the initial period of highly variable ion release, (B) a more stable period prior to pH consistently falling below 4.4, (C) a period with pH consistently below 4.4 and concurrent increases of Ca and Mg release, and (D) a period when pH rises above 4.4 until the end of the experiment. The concurrent release of Ca and Mg in period C was crucial to model 1, because it indicated dissolution of augite, which was the only common sample mineral with approximately equal proportions of Ca and Mg.

In addition to the models mentioned above, maximal release of plagioclase and olivine were determined for the entire period of record. This was done for plagioclase by dividing cumulative Ca release (mmol, Table 25) by the average molar Ca content of each sample (Table 11). Olivine was computed similarly with cumulative Mg release (Table 25) and Mg content in olivine (Table 11).

4.4.5.1.1. Model 1

This model calculated a neutral-pH mechanism plagioclase dissolution rate for periods during which Ca release from a given sample was fairly constant, accompanied by augite dissolution during acidic periods (period C).

Calcium release was attributed solely to plagioclase dissolution. The periods used for plagioclase calculation excludes periods during which concurrent increases in rates of Ca and Mg release were observed. These periods of concurrent Ca and Mg release were likely due to augite dissolution, which was the only common mineral present in our samples with approximately equal amounts of the two elements. This presumed increase in augite dissolution typically began at pH 4.1-4.4. Relatively constant Ca release, both prior to reaching this pH and subsequent to rising above this pH, was attributed solely to plagioclase dissolution. The plagioclase dissolution rate was calculated as follows.

$$dplag/dt = (dCa/dt)_{obs}/[(A_{plag})_{geom}(SR_{plag})(m_{Ca/plag})]$$

where, $dplag/dt$ = rate of plagioclase dissolution (mol m^{-2} s^{-1}),
$(dCa/dt)_{obs}$ = observed rate of Ca release (mol s^{-1}),
$(A_{plag})_{geom}$ = geometrically calculated plagioclase surface area (m^2),
SR_{plag} = surface roughness factor for plagioclase (dimensionless), and

$m_{Ca/plag}$ = moles of Ca per mole plagioclase (dimensionless).

Surface roughness factor is a value accounting for the deviation of actual surface area from the smooth surface assumed in the geometric calculation assuming smooth spheres (see Table 48 for specific factors for each silicate mineral).

Geometric mineral surface area for all models was determined by the equation (see Table 48 for citations of mineral-specific information and Table 49 for specific silicate mineral values):

$$(A_m)_{geom} = (6*M_m)/(\rho*d)$$

where, $(A_m)_{geom}$ = surface area of mineral m,
M_m = mass of mineral m (g),
ρ = density (g/cm^3), and
d = diameter (cm).

All samples in the experiment were sieved to 0.0053 cm < d < 0.0149 cm. Mean particle diameter was determined using a log-normal particle size distribution (Brantley and Mellott, 2000).

$$d_{geo\ mean} = (0.0053 \times 0.0149)^{0.5} = 0.0089 \text{ cm}$$

This model overestimated plagioclase dissolution rates. It assigned all observed Ca release to plagioclase dissolution. Furthermore, it provided a higher plagioclase dissolution rate than if Ca were preferentially leached.

During periods of concurrent increases in Ca and Mg (Period C), which typically began with pH<4.4, Ca release above the average rate of plagioclase dissolution from the prior period (Period B) was attributed to augite. This assumption could overestimate augite dissolution because the rate of plagioclase dissolution might have increased during these periods of decreasing pH.

$$daug/dt = (dCa/dt)_{obs}/[(A_{aug})_{geom}(SR_{aug})(m_{Ca/aug})]$$

This equation was used to calculate the rate of Mg release from augite by assuming stoichiometric release of Ca and Mg. Residual Mg (that not attributed to augite dissolution) was calculated by subtracting this value from the observed rate of Mg release. The residual Mg release could occur from olivine, hypersthene or biotite, and data analysis could not distribute the release among these minerals. Consequently, maximum dissolution rates were determined for each mineral, assuming all residual Mg release was from a single mineral. As mentioned above, it was assumed that minerals dissolved stoichiometrically.

$$dol/dt = (dMg/dt)_{res}/[(A_{ol})_{geom}(SR_{ol})(m_{Mg/ol})]$$

$$dhyp/dt = (dMg/dt)_{res}/[(A_{hyp})_{geom}(SR_{hyp})(m_{Mg/hyp})]$$

$$dbio/dt = (dMg/dt)_{res}/[(A_{bio})_{geom}(SR_{bio})(m_{Mg/bio})]$$

If calculated Mg release from augite dissolution exceeded observed Mg release, the calculated rate was rejected and observed Mg release was used to calculate augite dissolution.

$$daug/dt = (dMg/dt)_{obs}/[(A_{aug})_{geom}(SR_{aug})(m_{Mg/aug})]$$

In these cases, all Mg was attributed to augite dissolution, with no residual to attribute to other Mg-bearing minerals. These cases also resulted in orphan Ca, which was not attributed to the dissolution of any mineral.

The allotment of all available Ca to plagioclase dissolution during neutral periods (pH > ~4.4) prevents consideration of augite dissolution as a possible source of Ca and Mg release. Therefore, any Mg observed in the neutral period was attributed to dissolution of other ferromagnesian minerals. Maximum dissolution rates for each mineral were determined by assuming all Mg came from one mineral, and applying the previous dissolution equations.

4.4.5.1.2. Model 2

This model assumed all Mg release was due to stoichiometric augite dissolution. Because some Mg release might have come from other minerals, this assumption overestimated augite dissolution.

$$daug/dt = (dMg/dt)_{obs}/[(A_{aug})_{geom}(SR_{aug})(m_{Mg/aug})]$$

The Ca release from augite was calculated using the augite dissolution rate above and assuming it dissolved stoichiometrically, releasing Ca and Mg proportionally. Therefore, Ca release from augite with respect to Mg content can be calculated using the following equation

$$(dCa/dt)_{aug} = (m_{Ca/aug}/m_{Mg/aug})(dMg/dt)_{obs}$$

If the observed Ca release rate exceeded that associated with augite dissolution, residual Ca was attributed to plagioclase dissolution. If observed Mg exceeded stoichiometric amounts of Ca, then Ca was used for determination of augite dissolution and there was no plagioclase dissolution. In this case, excess Mg was attributed to dissolution of hypersthene, olivine and biotite, by determining a maximum possible dissolution rate for each of these minerals based on excess Mg.

4.4.5.1.3. Model 3

In this model, plagioclase dissolution was calculated based on observed Na release rates for both neutral and acid conditions. It was assumed that Na was released stoichiometrically from plagioclase. Substantial Na release is only possible from plagioclase, as it comprises >86% of the Na present in the reactor. Potassium feldspar is the next most abundant source of Na, but comprises <13% of total Na. Cation speciation and mineral surface area information used in the previous calculations and results are presented in Table 48. Because Na data were not available for the entire period of record in most samples, this model was applied to samples begun in 1989 and 1990 beginning in weeks 402-422, the period after Na analysis began. For samples begun in 1997, the model was applied from week 12, when Na was first sampled.

$$dplag/dt = (dNa/dt)_{obs}/[(A_{plag})_{geom}(SR_{plag})(m_{Na/plag})]$$

The rate of augite dissolution was then calculated based on the assumption that Na and Ca release from plagioclase was stoichiometric. Differences between observed rates of Ca release and calculated rates from plagioclase dissolution were attributed to augite dissolution. Calcium release from augite was assumed to be stoichiometric. In the event that observed Ca was less than the calculated Ca from plagioclase, there was assumed to be no augite dissolution.

$$daug/dt = [(dCa/dt)_{obs} - (m_{Ca/plag}/ m_{Na/plag})(dNa/dt)]/ [(A_{aug})_{geom}(SR_{aug})(m_{Ca/aug})]$$

However, observed rates of Mg release must also be taken into account based on assumptions that Ca and Mg release from augite dissolution was stoichiometric. If observed Mg concentrations were less than the stoichiometric amount indicated by the difference between the observed rate of Ca release and that attributed to dissolution of plagioclase, then the rate calculated from residual Ca release was rejected and observed Mg release was used to calculate augite dissolution. In these cases, there was no residual Mg to attribute to dissolution of other ferromagnesian minerals. These cases also created orphan Ca, which was not attributed to the dissolution of any mineral.

Residual Mg was calculated as follows.

$$dMg_{res}/dt = (dMg/dt)_{obs} - (daug/dt \times [(A_{aug})_{geom} (SR_{aug})(m_{Mg/aug})])$$

The source of residual Mg release could not be determined by data analysis. To address this, it was assumed that the residual Mg was from olivine, hypersthene, or biotite. A maximal stoichiometric rate of dissolution using all residual Mg was determined for each mineral.

4.4.5.1.4. Model 4

This model began with the initial step in Model 3 (section 4.4.5.1.3), where plagioclase dissolution was calculated based on observed Na release rates for both neutral and acid conditions. Potassium release was measured over the same times as Na, and collection days for both elements were generally identical. The same limitations on interpretation of model 3 at early stages of sample dissolution also apply to this model.

$$dplag/dt = (dNa/dt)_{obs} / [(A_{plag})_{geom}(SR_{plag})(m_{Na/plag})]$$

The rates of biotite and potassium feldspar dissolution were then calculated based on the assumption that Na and K release from plagioclase was stoichiometric. Differences between observed rates of K release and calculated rates from plagioclase dissolution were attributed to each mineral individually to give a maximum dissolution rate for these conditions. The equation for the dissolution rate calculation, with biotite as an example, was:

$$dbio/dt = [(dK/dt)_{obs} - (m_{K/plag}/ m_{Na/plag})(dNa/dt)] / [(A_{bio})_{geom}(SR_{bio})(m_{K/bio})]$$

The rate of augite dissolution was calculated in the same manner as in Model 3. Residual Mg was then determined by subtracting Mg release attributed to biotite and augite dissolution from observed Mg release.

$$dMg_{res}/dt = (dMg/dt)_{obs} - (daug/dt \times [(A_{aug})_{geom} (SR_{aug})(m_{Mg/aug})]) - (dbio/dt \times [(A_{bio})_{geom}(SR_{bio})(m_{K/bio})])$$

Maximum stoichiometric rates of hypersthene and olivine were then determined based on residual Mg after biotite and augite dissolution. In the event that observed Mg was less than that attributed to biotite, than there was assumed to be no dissolution of other ferromagnesian minerals. In the event that there was no biotite in a sample, all Mg release was assumed to come from dissolution of augite, hypersthene and olivine. A maximal stoichiometric rate of dissolution using all residual Mg was determined for each mineral.

4.4.5.2. Comparison of Predicted and Observed Plagioclase Dissolution Stoichiometry

Observed Ca and Na release was plotted over the entire period of record for each sample (Appendix 7, Figures A7.1-A7.21). The ratio of observed Ca: Na release was also plotted and compared to the solid phase Ca: Na in plagioclase as determined by McSwiggen and Associates (Table 11, Appendix 1, Attachment A1.1).

4.4.5.3. Gibbsite Production

In order to determine whether gibbsite dissolution could be responsible for the high percentage of samples whose minimum pH was approximately 4.0, maximal gibbsite production was compared to acidity production. Maximal gibbsite production was determined by assuming that all Al released during plagioclase dissolution was used in gibbsite formation at the rate of two moles Al per mole gibbsite. To determine plagioclase dissolution, all Ca was assumed to come from plagioclase. Each mole of plagioclase was assumed to release 0.5 moles Ca and 1.5 moles Al. The amount of gibbsite was compared to the amount of acidity produced ((cumulative sulfate*2) – (cumulative Ca*2)) by the first time a sample's pH declined below 4.0.

4.4.5.4. Sulfide Mineral Oxidation Rate Determination

Pyrrhotite mass was calculated by estimating the amounts of sulfur associated with copper and nickel sulfides and assuming the remaining sulfur was present as pyrrhotite. The sulfur mass in grams associated with copper sulfides was calculated by multiplying the percent copper of a sample by 1.26 x 75/100. The first factor is a mass ratio of Cu to S, assuming all copper was equally divided between chalcopyrite ($CuFeS_2$) and cubanite ($CuFe_2S_3$), and the second factor determined S mass for the 75 g sample. It should be noted that this calculation differs from the method used to determine chalcopyrite content in the bulk samples, which attributed all Cu to chalcopyrite. Although the assumption of a chalcopyrite:cubanite ratio equal to one does not appear justified by grain analyses showing a 3:1 ratio in unleached samples, the low number of grains analyzed makes this ratio prone to error. Sulfur associated with nickel sulfide was determined by multiplying sample nickel content by 0.97 x 75/100. This formula is similar to the previous one and assumed all nickel was present as pentlandite ($Ni_{4.5}Fe_{4.5}S_8$). Sulfur associated with pyrrhotite was then determined by subtracting the sulfur associated with copper and nickel from the total sulfur content of a sample. Total mass of all sulfide minerals was determined as the sum of copper and nickel sulfides plus pyrrhotite. See Table 50 for specific sulfide mineral values. It should be noted that nickel associated with olivine was not accounted for in these calculations, however this would have a negligible effect on the calculated sulfide oxidation rates since 75% of the nickel, on average, was associated with pentlandite and pyrrhotite (Table 42).

Rates of sulfide mineral oxidation were calculated by dividing observed rates of SO_4 release by the surface area of sulfide minerals present (determined as in section 4.4.5.1.1). The rate of SO_4 release was determined based on observed SO_4 concentrations and the volume of drainage (Appendix 5, Table A5.0). Two calculations were made for sulfide mineral surface area, the first using a calculated pyrrhotite surface area and the second using a calculated surface area for all sulfide minerals present.

Sample sulfur depletion was determined using two methods. First, the amount of sulfur released as sulfate was subtracted from the amount of sulfur determined to be in the solids initially. Second, the amount of sulfur in leached solids from the replicate samples was subtracted from the amount of sulfur in unleached solids to determine sulfur depletion. The second method is likely to underestimate sulfur loss, as the replicate samples were removed from the experiment long before the other samples.

Sulfur present in each unreacted sample was determined using the equation $M_S = (S_0 / 100) * M_R$, where M_S = mass of sulfur present, S_0 = % of sample present as sulfur, and M_R = mass of sample. The mass of released sulfur was subtracted from the total initial sulfur values to determine sulfur depletion.

Five distinct stages during the period of record were selected to determine when S release occurred relative to other changes in sample measurements. These stages were bounded by the point pH drops below six, the point at which a major pH drop or increased SO_4 release begins, the point of minimum pH or maximum SO_4 release, the point pH begins to increase from the minimum, and termination of the experiment (Table 71).

4.4.6. Mineral Sources of Metals

Of the four metals of concern (Co, Cu, Ni, Zn), only Co and Ni had multiple potential mineral sources whose relative importance was not clear. For Co, these were cobaltite and pentlandite. For Ni, these were olivine, pyrrhotite, maucherite and pentlandite. This section describes the methods used to determine the relative importance of the sources.

To determine the maximum percent of cobalt present as cobaltite, (Co, Fe)AsS, in these samples, As:Co ratios were determined. A similar approach was used for maucherite ($Ni_{11}As_8$). Arsenic, Co and Ni values from unleached solid samples (Table 3) were used in this comparison. Due to the small amount of Zn and Co, a quantitative partitioning of mineral sources was not attempted for these metals. Any such analysis would have a high degree of uncertainty due to lack of reasonable surface roughness coefficients and oxidation rates for many potential mineral sources.

Calculation of maximum olivine dissolution rates assumed that all Mg in leachate resulted from olivine dissolution and that Ni was released stoichiometrically. As can be seen in Table 23, most samples have large fractions of Mg present in minerals other than olivine. Thus, the assumption of all Mg coming from olivine is likely to overestimate olivine dissolution.

To calculate the rate of Ni release from sulfide minerals, it was assumed that Ni associated with pyrrhotite and pentlandite (presented in Table 42) was present in the ratio $Ni_{sulfides}:(S_{total}-S_{cp,cb})$. Sulfur associated with copper sulfides (chalcopyrite and cubanite) was calculated by multiplying the percent copper of a sample by $1.26/100 \times 75$. The first factor is a mass ratio of Cu to S, assuming all copper was equally divided between chalcopyrite ($CuFeS_2$) and cubanite ($CuFe_2S_3$), and the second factor determined S mass for the 75 g sample.

5. RESULTS

This analysis of results begins with analysis of unleached solid samples, including particle size, chemical composition, neutralization potential and, finally, mineral abundance and chemistry. The second results section discusses methodological and environmental analyses, including humidity and temperature data, analysis of replicate samples, sample water retention and evaporation rates, and effects of volume of leachate on pH. The third section discusses ways to group samples based on leachate analyses that include characteristic time periods, sample groups with similar behavior, and correlations between sample chemistry and leachate characteristics among different time periods. The fourth section discusses results of silicate dissolution models, gibbsite production calculations, and sulfide mineral oxidation models. It should be noted that for the results section, reactors 39-44 are included with the Dunka blast hole samples as a single collection. These samples were stored for 8 to 20+ years prior to dissolution testing and therefore produced higher drainage pH and lower sulfate and metal release rates. These differences in drainage pH and release rates will be neglected for the following results section.

5.1. Unleached Solid-Phase Analyses

5.1.1. Particle Size

Samples were sieved to obtain the 0.053-0.149 mm fraction desired for the experiment. To determine the success of the particle size selection, Duluth Complex samples were examined to determine the extent to which finer grains were present. Perimeter:area ratios were used as an indicator of particle size for twelve samples. Most unleached samples had perimeter:1000 micron area ratios from 72-111 (Table 17). The 1.63%S sample had a much higher ratio of 255.6. Unleached samples had a mean perimeter:area ratio for all samples of 104. Mean particle diameter, determined assuming spherical particles, was a mean of 42.9 µm.

To assess the extent of very fine particles adsorbed to the surface of other grains, cleaned and uncleaned samples from the 0.22 and 0.40%S were compared. This revealed many small particles on the grain surface of two uncleaned-unleached samples (Figure 57).

5.1.2. Chemistry

5.1.2.1. Whole-Rock Chemical Composition

For Duluth Complex samples, total sulfur content of unleached samples ranged from 0.18-1.71%, with virtually all sulfur present as sulfide. Evolved CO_2 exceeded 0.15% in only the 0.40%S sample (0.64%). Loss on ignition for samples analyzed ranged from 1.1-4.5% (LOI2, adjusted for difference in O between FeO and Fe_2O_3). The major whole rock chemical components were SiO_2 (43.2-50.5%), Al_2O_3 (14.1-21.0%), FeO (8.6-14.4%), MgO (5.7-8.6%) and CaO (6.6-9.8%). Contents of K_2O (0.3-1.7%), Na_2O (0.3-3.0%), TiO_2 (0.5-3.6%), MnO (0.1-0.2%), Fe_2O_3 (0.64-4.85%) and P_2O_5 (0.04-0.3%) were low (Table 2).

In Virginia Formation samples, S and Fe_2O_3 contents differed from Duluth Complex samples. Virginia Formation samples had higher sulfur content (2.06-5.44%S) and lower Fe_2O_3 (below detection-0.51%). Evolved CO_2 was high relative to other samples in the 5.44%S sample, with a measurement of 0.32%. Loss on ignition for samples analyzed ranged from 1.7-3.8% (LOI2, adjusted for difference in O between FeO and Fe_2O_3). The major whole rock chemical components were SiO_2 (42.5-57.8%), Al_2O_3 (11.5-20.5%), FeO (9.9-17.1%), MgO (4.6-10.8%) and CaO (2.2-10.0%). Contents of K_2O (0.4-1.9%), Na_2O (1.3-3.0%), TiO_2 (0.8-1.8%), MnO (0.08-0.2%), Fe_2O_3 (0.64-4.85%) and P_2O_5 (0.1-0.3%) were

low (Table 2).

Trace metal contents were generally low for all samples (Table 3). Metals that were the focus of environmental concern in these samples were fairly uniform, with content decreasing in the order Cu > Ni > Zn > Co. Metals (and their associated maximum values in mg kg^{-1}) present at higher concentrations were Ce (64.1), Rb (69), Bi (90.54), Co (120), Zr (234), V (304), Zn (305), Sr (348), Ba (508), Cr (653), Ni (718), and Cu (2220). The maximum reported concentration for most metals were less than 33 mg kg^{-1} (Ag, As, Cs, Dy, Er, Eu, Ga, Gd, Ge, Hf, Ho, In, La, Lu, Mo, Nb, Nd, Pb, Pr, Sb, Sm, Sn, Ta, Tb, Th, Tl, Tm, U, W, Y, Yb).

When comparing Duluth Complex and Virginia Formation samples, metal contents were similar. However, in light of the large differences in S%, Virginia Formation samples are relatively low in metals compared to Duluth Complex samples.

5.1.2.2. Neutralization Potential

Neutralization potentials for 19 unleached Duluth Complex samples (excluding 0.67(R40) and 0.82%S) were determined using the method of Sobek et al. (1978) and values ranged from 8-21 kg $CaCO_3$/ton of rock (Table 22). NP values determined using the NP_{pH6} test for all 1997 samples (reactors 39-44) ranged from 2.0-5.4 kg $CaCO_3$/MT rock. The neutralization potential as Ca and Mg carbonate for all Duluth Complex samples was <0.7-3.2 g $CaCO_3$ eq (kg rock)$^{-1}$, with the exception of the 0.4%S sample, which had 14.5 g $CaCO_3$ eq (kg rock)$^{-1}$ (Table 22). For comparison, acid potentials calculated for the solids ranged from 6-53 g $CaCO_3$ eq (kg rock)$^{-1}$ (Table 22, Figure 61).

Virginia Formation neutralization potentials determined using the method of Sobek et al. (1978) ranged from 0.5-14 kg $CaCO_3$/ton of rock (Table 22). Neutralization potential as Ca and Mg carbonate was <0.7-7.3 g $CaCO_3$ eq (kg rock)$^{-1}$ (Table 22). For comparison, acid potentials calculated for the solids ranged from 64-170 g $CaCO_3$ eq (kg rock)$^{-1}$ (Table 22).

5.1.3. Mineral Analysis

5.1.3.1. Mineral Abundance and Composition

Scanning electron microscope electron probing and visual point counts of silicate minerals identified plagioclase in all 25 samples, as well as hypersthene and augite in all samples except the 5.44%S. Ilmenite was present in 22 samples, olivine in 20 samples, and 19 samples contained biotite and potassium feldspar. Quartz and potassium feldspar were more abundant in Virginia Formation samples than Duluth Complex (Table 7). Plagioclase, olivine and augite were less common in Virginia Formation samples than Duluth Complex rock.

Abundance of sulfide minerals pyrrhotite, pentlandite, chalcopyrite, pyrite, and cubanite was determined by a point count modal analysis. Pyrrhotite was most common, as it was observed in 18 of 25 total samples, ranging from 0.8-13.9%. Pentlandite, cubanite, chalcopyrite, and pyrite were observed at least once (Table 10). Pyrrhotite, chalcopyrite and pentlandite were more accurately determined using chemical calculations. Using these calculations, pyrrhotite was still the most common sulfide mineral (0.02-13.7%), but all three minerals were found in every sample (Table 10).

Plagioclase (36-65%), augite (3-17%), hypersthene (3-28%), and olivine (0-39%) contributed 65-97% of mineral content in Duluth Complex samples. Fifteen samples contained the plagioclase species labradorite, while six samples contain the more sodium-rich species andesine (0.18, 0.22, 0.57, 1.44,

1.63 and 1.64%S). Unleached Duluth Complex silicate minerals had the following range of stoichiometric coefficients: plagioclase $(Ca_{0.40-0.63}Na_{0.32-0.55})Al_{1.44-1.66}Si_{2.35-2.57}O_8$, augite $((Ca_{0.76-0.95}Na_{0.01-0.03})(Mg_{0.36-0.84}Fe_{0.27-0.63}Ti_{0.01-0.02})(Si_{1.95-2.00}Al_{0.03-0.1})O_6)$, hypersthene $((Mg_{0.91-1.30}Fe_{0.63-1.66})_2Si_{1.97-2.00}O_6)$, and olivine $((Mg_{0.72-1.23}Fe_{0.75-1.25})Si_{0.99-1.01}O_4)$ (Table 11).

Unleached Duluth Complex sulfide minerals had the following range of stoichiometric coefficients: pyrrhotite $Fe_{6.91-7.93}S_8$, pyrite $Fe_{4.06-4.44}S_8$, chalcopyrite $(Cu_{4.16-4.66}Fe_{4.32-4.99})\ S_8$, cubanite $(Cu_{2.71-2.86}Fe_{5.49-5.59})\ S_8$ and pentlandite $(Ni_{4.61-5.16}Fe_{4.39-5.37})\ S_8$ (Table 8).

In unleached Virginia Formation samples, plagioclase (12-35%), augite (0-3%), hypersthene (0-12%), and olivine (0-1%) contribute 12-42% of mineral content. One sample contained the plagioclase species labradorite (3.12%S), two samples contained the more sodium-rich species andesine (2.06 and 3.71%S), and one contained neither (5.44%S). Unleached Virginia Formation silicate minerals had the following range of stoichiometric coefficients: plagioclase $(Ca_{0.32-0.56}Na_{0.38-0.66})Al_{1.34-1.60}Si_{2.41-2.67}O_8$, augite $((Ca_{0.81-0.97}Na_{0.002-0.02})(Mg_{0.34-0.74}Fe_{0.28-0.63}Ti_{0.005-0.01})(Si_{1.97-1.99}Al_{0.04-0.1})O_6)$, hypersthene $((Mg_{0.95-1.11}Fe_{0.78-0.96})_2Si_{1.96-1.98}O_6)$, and olivine $((Mg_{0.89}Fe_{1.06})Si_{1.01}O_4)$ (Table 10).

Unleached Virginia Formation sulfide minerals present in these samples had the following range of stoichiometric coefficients: pyrrhotite $Fe_{7.10-7.53}S_8$, pyrite $Fe_{4.07-4.55}S_8$ and chalcopyrite $(Cu_{4.17}Fe_{4.37})\ S_8$.

5.1.3.2. Mineral Grain Coatings

There were no coatings or dissolution features observed on unleached grains during analysis (Table 14), indicating no situ alteration or weathering of the samples prior to initiation of the experiment.

5.2. Reaction Conditions and Methodological Variables

5.2.1. Temperature and Relative Humidity

Average weekly temperatures ranged from 21-28°C, with the exception of seven weeks below 21°C and eight weeks above 28°C (mean 24.4^{o}C, SD 1.6, 0.09 95% CI). Average weekly relative humidity ranged from 37-86% (mean 57.6%, SD 4.8, 0.3 95% CI) (Appendix 2, Table A2.2). The humidity weekly average only exceeded 75% four times, and only fell below 45% eleven times during the 24 year period of record.

5.2.2. Replication

Two replicate reactors from nineteen samples were compared based on average weekly drainage pH and SO_4, Ca and Mg concentrations for data sets ranging from 78-328 weeks, depending on replicate pair (Appendix 3, Figures A3.45-A3.63). Paired t-tests found no difference for any factor between replicate groups, so replicate samples (even-numbered reactors 2-38) were removed from the experiment to save time and resources. The t-test p-values, which describe the chance that any difference between replicate groups was coincidence, ranged from 0.371-0.822. A p-value ≤ 0.05, typical for these types of tests, was required to reject the null hypothesis. The null hypothesis for these t-tests was that there was no difference in means between groups of replicates. Thus there was a 37-82% chance that differences between replicates were due to random chance, indicating there were no differences between replicate groups. Although there were differences between replicates (Table 19), there were no systematic differences between the two groups when one set of samples was compared to a replicate set of samples.

When minimum drainage pH was compared between replicates, the p-value was 0.371, with a mean pH of 4.30 and the mean difference between replicates 0.04. The t-test for SO_4 had a p-value of 0.822, with a mean concentration of 25.7 mg/L and a mean difference of 0.5 mg/L. Ca ($p = 0.757$) had a mean of 3.06 mg/L and a mean difference of 0.02 mg/L. Mg ($p = 0.757$) had a mean of 1.92 mg/L and a mean difference of 0.02 mg/L.

5.2.3. Analysis of Water Retention and Evaporation of Samples

In a six-week trial, eight 75-g samples retained 11.8-21.1 ml water (mean 16.2 ml, SD 2.4) the day after each weekly 200 ml water addition. This was 15.7-28.1% moisture content (by weight). Approximately 75% of retained water evaporated within three days of rinsing (Table 18), with approximately equal amounts lost each day. Reactors were completely dry before the next weekly water addition and, at the rate of drying from the first two days, would have completely dried less than four days after water was added.

5.2.4. Analysis of Effect of Rinse Volume on Drainage pH

Difference in drainage volume had a significant, though small, effect on drainage pH (Table 20). Mean pH in the 10 ml sample was 0.11 (SD=0.096) lower than the drainage for the entire 200 ml rinse (paired t-test, $p<0.001$). Differences of this magnitude would not effect management recommendations of this research. Total drainage pH measurements of most samples (ten of sixteen) were within the eight-week range prior to the experiment. Of the six that were outside this range, all were lower, but only with a mean -0.0333, making any effect of the sampling method on total drainage pH measurements unlikely.

5.3. Drainage Quality and Geochemical Evolution of Solid Phase

Leachate from the samples had predictable changes that occurred based on sample characteristics and duration of experiment. Changes in Ca and Mg release from samples were used to define time periods A-D. Minimum pH during the experiment was the primary determinant when dividing samples into Groups I-III. Time periods and sample groups were used to organize results and predict water quality effects based on sample characteristics. To increase mechanistic understanding of experimental leaching on samples, particle size, sample chemistry, as well as mineral composition and coatings was analyzed for leached samples.

5.3.1. Sample Classification, Drainage Quality and Mass Release

In general, pH decreased and sulfate release rates increased as solid-phase sulfur content increased. Maximum SO_4, Ca, and Mg release rates for samples containing less than 1.12%S typically occurred during the first 100 weeks, while higher sulfur samples had peak rates after this period. pH gradually decreased to a minimum value in 270-425 weeks for most samples, before gradually increasing. Higher sulfur content produced sharper initial declines in pH, as well as lower pH minima. Coinciding with the pH minima were marked increases in SO_4 release rates. Ca and Mg release rates also rose sharply near the pH minimum, but these increases were less than that observed for SO_4. These trends in drainage quality were typical enough to classify the samples into characteristic time periods and groups.

There were four periods of pH change accompanied by Ca and Mg release changes. Drainage pH declined from period A through period C, before rising in period D. Sample release rates were highest in period A, declined in period B, rose in period C and declined again in period D. pH generally declined in the first three periods and rose in the fourth. Not all samples went through later periods. For example, samples with S content $\leq 0.22\%$ only exhibited periods A and B.

Samples were classified into three groups based on typical minimum drainage pH (Table 25). The drainage pH criterion was selected based on its importance in predicting the amount of treatment for wastewater. Group I (circumneutral) generally had $0.18 \leq S\% \leq 0.22$ and a minimum pH that was generally no lower than 6. Group II (acidic) generally had $0.40 \leq \%S \leq 0.70$ and a minimum pH from 4-6. Group III (strongly acidic) generally had greater than $0.71 \leq \%S \leq 1.64$ and a minimum pH well below 4. Samples comprising each group, as well as representative data for each group, are listed in Table 25. The terms describing groups (circumneutral to strongly acidic) were used to simplify data presentation relative to this experiment, and are not intended to apply to mine waste generally. Similarities in chemical characteristics and drainage quality effects of the sample groups are discussed below. It should be noted that reactors 39-44 have been included in these groupings for the results section. These samples were stored for 8 to 20+ years prior to dissolution testing and therefore produced higher drainage pH and lower sulfate and metal release rates.

5.3.1.1. Drainage Quality Changes over Time

Drainage pH, as well as Ca and Mg release were measured throughout the experiment and exhibited similar patterns among samples. These patterns were used to define four sequential sample periods (see definitions below). Some samples did not exhibit behavior typical of all periods. Calcium and magnesium concentrations were relatively high and variable at the onset of the experiment, with Ca concentrations exceeding Mg concentrations. Release decreased and stabilized for both at lower levels after periods of 35-138 weeks. Mg concentrations during the latter stages of dissolution decreased to a greater extent than Ca concentrations. Ca concentrations generally increased in response to pH declines, and decreased as pH rose. Na and K concentrations were determined starting in 1997 (after week 400 for most reactors), and were typically lower and more variable than calcium or magnesium concentrations (Appendix 3, Figures A3.1-A3.44). For a summary of exact lengths of periods for each sample see Table 24.

Samples were assigned to periods based on similarity of changes in pH and ion release. These periods were labeled Periods A-D, and are described as follows:

1.) Period A

Period A started at week 1 for each sample, and ended as Ca and Mg release rates stabilized. Sample pH gradually decreased throughout this period for all samples. pH was relatively variable among samples, ranging from 4.6-8.7. Group I samples stabilized from approximately 6-7, while sample groups II and III stabilized between 4.5-5.5 by the end of Period A. Initially, each sample had relatively high Ca and Mg release, with Ca concentrations typically exceeding Mg concentrations. Release gradually declined with time. This period lasted from 34-140 weeks, and was present in all samples.

2.) Period B

Period B began when Mg and Ca release rates stabilized at levels typically 20-60 percent of those in Period A. Sample pH typically decreased gradually throughout this period. Magnesium release exceeded Ca release in 11 of 21 samples (0.51, 0.57, 0.58, 0.67(R39), 0.71, 1.16, 1.40, 1.44, 1.63, 1.64, 1.71%S).

3.) Period C

Period C occurred in most group II samples (all but 0.82, 0.92, 1.71%S) and all group III Duluth Complex samples. No group I samples entered period C. It began with a relatively rapid pH decline (below ~4.4) and concurrent increases in SO_4, Ca, and Mg release, with SO_4 release increases typically occurring 10-50 weeks prior to Ca and Mg release increases. During this

period it was common to have Mg spikes exceeding Ca release at the pH minimum. This increase in Ca release was particularly common in group III samples. Calcium, Mg and SO_4 release decreased as Period C ended.

4.) Period D

Period D occurred in most group II (all but 0.4, 0.82, 0.92, 1.71%S) and group III Duluth Complex samples (all but 1.40%S). No group I samples entered period D. The beginning of this period is marked by decreases in ion release rates and increasing pH (above ~4.2-4.4). When it occurred, this period continued to the experiment's termination. This period was characterized by gradually increasing pH (above ~4.2-4.4), and decreasing Ca, Mg, and SO_4 concentrations. Calcium release rates remained relatively stable during Period D, at levels similar to Period B, while Mg release rates gradually decreased to levels less than or equal to rates observed in Period B.

Sodium and K concentrations were typically more variable than Ca and Mg concentrations, and were not determined for the entire period of record for the 19 samples started in 1989 and 1990. In all aspects of drainage quality, the six samples started in 1997 generally had fewer of the fluctuations that characterized the earlier samples.

Iron, Al and Si were sampled similarly to Na and K. Iron and Al release rates were very low until pH reached 4 (Table 35, Figures 52, 53). Below pH 4, Fe release rates were 12.4 times faster than above pH 4. Aluminum rates were 21.5 times faster. Si release was less responsive to pH declines (Table 35, Figures 54). Although average Si release rates were 2.3 times faster below pH 4, this was based on only seven points, making an accurate determination of pH effects on Si release difficult. It should be noted that no metal analyses were performed from weeks 78-360. This should be kept in mind when reviewing the results.

5.3.1.2. Sample Groups

5.3.1.2.1. Group I

Group I samples had typical drainage pH values above 6.0 throughout the period of record (Table 27). This group consisted of three samples with 0.18, 0.22 and 0.72%S. Drainage pH tended to decrease and stabilize from 6.0–6.5. Drainage pH ranged from 7.5–8.0 initially, decreased to roughly 6.5 after 100 weeks, and stabilized between 6.0–6.5 by week 150 (Table 27, Figure 5).

SO_4 release generally decreased with time. Average release rates were initially 8.8×10^{-12} to 1.2×10^{-11} mol/s, with maxima from $1.3\text{-}3.0 \times 10^{-11}$ mol/s (Table 28). Rates decreased after 100-150 weeks to levels an average of 30% of initial rates (Figure 6). Average rates ranged from $2.5\text{-}5.0 \times 10^{-12}$ mol/s until week 350, and continued to decrease until the end of the period of record, when average rates ranged from 1.2×10^{-12} to 1.7×10^{-12} mol/s. Total release for the entire experiment was 1.1-2.3 mmol, with an average of 1.9 mmol (Table 25).

Ca release rates tended to decrease with time as well (Table 29). Average release rates during the initial 150 weeks were $1.1\text{-}1.8 \times 10^{-11}$ mol/s, with maximum release rates from $2.6\text{-}4.2 \times 10^{-11}$ mol/s. Rates decreased and stabilized after week 150, with average rates of $2.4\text{-}3.5 \times 10^{-12}$ mol/s and maximum release rates of 8.7×10^{-12} to 1.2×10^{-11} mol/s for the remainder of the experiment. Rates for each sample remained relatively constant during this final period (Figure 8). Total release for the entire experiment was 1.3-3.9 mmol, with an average of 2.9 mmol (Table 25).

At the beginning of the experiment, average Mg release was 3.5 x 10^{-12} to 1.6 x 10^{-11} mol/s and maxima were 7.7 x 10^{-12} to 5.5 x 10^{-11} mol/s (Table 30). Following week 150, release rates decreased, with values as high as 1.4 x 10^{-11} mol/s until week 350 for the 0.18 and 0.22%S samples and week 200 for the 0.72%S sample (Figure 9). After this period Mg release began to decrease relative to Ca, with average values 20 to 60% lower than Ca release. Total release for the entire experiment was 0.8-3.3 mmol, with an average of 2.0 mmol (Table 25).

Na and K release were determined following week 420 for the 0.18 and 0.22%S samples and for the entire period of record for the 0.72%S sample (Figures 10, 11). Potassium release rates gradually decreased during the experiment, while Na stayed in a range for all samples between 6.1 x 10^{-13} to 3.3 x 10^{-12} mol/s (Table 32, Figure 10). Average potassium release for all samples decreased from 8.2 x 10-^{13} to 4.3 x 10^{-12} initially, to minima rates of 3.6 – 7.2 x 10^{-13} mol/s by the end of the period of record (Table 33, Figure 11).

Sample metal release (Co, Cu, Ni, Zn) was slower than rates of other elements already discussed (Tables 36-39). Average release rates for periods A and B ranged from 4.9 x 10^{-15} to 5.1 x 10^{-14} mol/s for Co, 8.8 x 10^{-15} to 4.3 x 10^{-14} mol/s for Cu, 7.5 x 10^{-15} to 9.2 x 10^{-14} mol/s for Ni, and 2.8-6.6 x 10^{-14} mol/s for Zn. Co generally varied little over time for each sample (Figure 12), although maximum release rates among samples ranged from 5.4 x 10^{-15} to 1.5 x 10^{-13} mol/s. Cu, Ni and Zn had a similar maximum range among samples (1.3 x 10^{-14} to 1.4 x 10^{-13}, 1.0 x 10^{-14} to 3.0 x 10^{-13}, 6.7 x 10^{-14} to 2.3 x 10^{-13} mol/s for Cu, Ni and Zn respectively), but individual samples were more variable over time. There were several spikes of 300-400% increases from preceding release rates (Figures 13-15). These spikes were quite rapid and tended to occur approximately concurrently with increased SO_4 release. Average release rates generally declined from period A to period B. Variation in pH was small over time among group I samples and there was no observed relationship between pH and metal release. Metal release from the 0.72%S sample displayed much less temporal variability than the other two samples in this group.

The empirical neutralization potential above pH 7 (ENP_{pH7}) for Group I samples was 0.3-0.6 mg $CaCO_3$/g rock (Table 21). ENP_{pH6} ranged from >1.5 to >3.1 mg $CaCO_3$/g rock. These ENP values were considerably lower than those determined by the method of Sobek et al. (1978) (16-17 mg $CaCO_3$/g, Table 22). The neutralization potential for the 0.72%S sample (determined by autotitrator method) was 3.4 mg $CaCO_3$/g rock.

5.3.1.2.2. Group II

Group II consisted of eleven samples with sulfur contents from 0.40-0.92% (excluding 0.71 and 0.72%) and 1.71%. Three samples (0.72, 0.92 and 1.17%S) were AMAX samples and had been stored for at least twenty years prior to experimental leaching. The long period of storage likely caused decreased release rates for sulfate, Ca and Mg, as well as slowing the pH decline.

This group includes samples for which drainage pH was below 6, but not below 4 for any sustained period. Drainage pH tended to decrease quickly during the initial weeks and plateau slightly. Gradual decreases were usually observed until weeks 250-300, when short but sharp drops to minimum values occurred at about pH 4.3. Gradual increases were observed shortly thereafter and continued to the end of the experiment. Initial pH values ranged from 6.8-7.9, while minimum pH ranged from 3.8-5.1. Minimum pH was reached during weeks 332-395 for eight of ten reactors, with 0.40 and 0.67%S samples reaching minimum pH at weeks 810 and 151, respectively. The 1.71%S sample was terminated after 144 weeks with a drainage pH value of 5.1. Drainage pH values near the end of the experiment ranged from 4-5 for all reactors (Table 27, Figure 5).

Sulfate release tended to decrease with time initially, only to increase after 200-250 weeks of dissolution, followed by decreases near the end of the experiment (Figure 6). These trends were moderated in samples having less rapid pH declines. SO_4 release rates during the initial period averaged 1.2-3.8 x 10^{-11} mol/s with a range from 1.6 x 10^{-12} to 1.0 x 10^{-10} mol/s (Table 28). Sulfate release began to increase as pH decreased below five, with most samples' maximum release coinciding with minimum pH. Maximum rates in the later years (Period C) ranged from 3.2 x 10^{-11} to 1.1 x 10^{-10} mol/s. These release rates then gradually declined for the remainder of the experiment. Rates ranged from 1.8 x 10^{-13} to 4.5 x 10^{-11} mol/s throughout this interval (Table 28). Total release for the entire experiment was 1.9-10 mmol, with an average of 6.4 mmol (Table 25).

Several reactors displayed periods of decreased SO_4 release either longer or shorter than those previously discussed (Figure 7). Several others did not experience subsequent release rate increases before the end of the experiment. Decreased release was observed for 234 weeks for the 0.67%S (R40) reactor before increases were noted, 80 weeks longer than the other reactors in this group. The 0.40%S reactor also behaved abnormally, undergoing 745 weeks of decreased release before pH dropped below 5, and rates began to increase. No release rate increases were observed for the 0.82 and 0.92%S reactors, both of which had a 750-week period of decreased release prior to the end of the experiment. In contrast to this prolonged release, the 0.58%S reactor experienced a very short period of decreased release only 30 weeks in length before increased release was observed. However, it should be noted that the 0.82 and 0.92%S samples were stored for 8 to 20+ years prior to dissolution testing and therefore produced higher drainage pH and lower sulfate and metal release rates.

For most samples in this group, Ca release rates tended to decrease from the highest average releases initially, increase after a period of relatively stable release, and decrease again near the end of the experiment (Figure 8). At the onset of the experiment (Period A), average Ca release rates ranged from 1.2-3.7 x 10^{-11} mol/s, with a range from 2.2 x 10^{-12} to 1.0 x 10^{-10} mol/s (Table 29). Release rates decreased and stabilized after 150 weeks to rates averaging 30% of those in period A (Table 29). Very little change was noted in release rates until pH levels reached ~4.4, which typically occurred during weeks 220-350. As pH declined below 4.4, release began to increase, with average rates ranging from 3.5 x 10^{-12} to 1.2 x 10^{-11} mol/s. Despite the increase, average rates were only 35% of those in the initial period. Release rates declined as pH rose, and generally stabilized after week 445 at average rates of 2.0-6.5 x 10^{-12} mol/s until the end of the experiment (Table 29). Total release for the entire experiment was 1.2-8.7 mmol, with an average of 3.3 mmol (Table 25).

Magnesium release behaved similarly to Ca release for most of the period of record. Release rates tended to decrease with time, with rates decreasing after an initial, highly variable period to lower, more stable levels, only to increase for a short time before steadily decreasing to the end of the experiment (Figure 9). Initial average rates were 4.3 x 10^{-12} to 1.6 x 10^{-11} and maxima from 8.5 x 10^{-12} to 7.3 x 10^{-11} mol/s (Table 30). Release rates decreased and stabilized after 39-140 weeks of dissolution to levels 20-60% below initial levels in 8 of 11 samples (Table 30). The rise in Mg in response to declining pH typically produced smaller release rates than Ca, although they were, on average, 70% higher than the reduced rates in period B. After this increase, rates stabilized similarly to Ca. However, unlike Ca release, Mg rates began another period of decline. Seven of eight longer-running samples (those started in 1989-90) had concentrations at the Mg detection limit near the end of this study. This equated to a release rate of approximately 6.0 x 10^{-13} mol/s. Total release for the entire experiment was 0.45-2.6 mmol, with an average of 1.4 mmol (Table 25).

For seven samples from group II (0.40, 0.41, 0.51, 0.57, 0.58 and both 0.67%S) Mg release rates increased after pH dropped below 4.5. For most of these samples (all except both 0.67%S), concurrent increases in Ca release were observed. During the concurrent increase, both Ca and Mg rates were, on

average 60 and 80% higher than those in the previous stable release period, respectively (Tables 29, 30). pH stayed relatively constant for the 0.40%S sample much longer than other samples, not starting a noticeable decline until week 715. This was followed by Ca release rate increases at week 775, followed by slightly elevated Mg release rates at week 800 (Figures 8, 9).

The remaining group II samples (0.54, 0.82, 0.92, and 1.71%S) had no major changes in Mg or Ca release following the initial decrease (Tables 29, 30). In these cases, pH gradually decreased with time, reached minimum values of pH 4-5, and remained fairly constant until termination of the experiment. The 0.54%S sample did not display any increases even though pH dropped to 4.2 at week 270 and remained below 4.4 for 130 weeks (Appendix 3, Figure A3.11). As with other Group II reactors, release rates decreased to levels 10-35% of initial rates before termination (Tables 29, 30). Both Ca and Mg release rates tended to decrease with time, with Mg typically decreasing near detection limits.

Group II Na release rates generally remained stable throughout the period of record, with values from 6.3×10^{-13} (below detection limit) to 1.0×10^{-11} mol/s, and a mean for all samples of 2.0×10^{-12} mol/s (Table 32). Potassium release rates were less consistent, as exhibited by the wider range from 3.6×10^{-13} to 6.5×10^{-11} mol/s, with a mean value for all samples of 8.1×10^{-12} mol/s (Table 33). Rates seemed to decrease with time, although this is difficult to conclusively state because of early gaps in the record for many samples.

Trace metal concentrations varied with pH for the Group II reactors. Average release rates for periods A through D ranged from 4.9×10^{-15} to 4.0×10^{-13} mol/s for Co, 2.3×10^{-14} to 3.4×10^{-12} mol/s for Cu, 6.0×10^{-14} to 3.0×10^{-12} mol/s for Ni, and 2.4×10^{-14} to 3.3×10^{-13} mol/s for Zn (Tables 36-39). Peak release rates among samples during periods A through D ranged from 4.9×10^{-15} to 4.1×10^{-13} mol/s for Co, 6.7×10^{-14} to 4.5×10^{-12} mol/s for Cu, 9.7×10^{-14} to 5.9×10^{-12} mol/s for Ni, and 3.7×10^{-14} to 6.3×10^{-13} mol/s for Zn. During period A, when pH remained above 6, average Ni, Co, and Zn release rates ranged from 6.1×10^{-14} to 4.2×10^{-13} mol/s. As drainage pH decreased below six, Co, Ni, and Zn release rates increased by 3, 34, and 15%, respectively (Figures 16, 18, 19). Release of these three metals typically increased quickly, followed by a gradual decrease to previous levels (Figures 12, 14, 15). Copper release behaved differently, remaining below 1.2×10^{-12} mol/s until pH approached five (Figure 17). The Cu increase lagged other metals and formed more of a plateau, with peak values ranging from 2.7×10^{-13} to 4.5×10^{-12} mol/s (Table 37, Figure 13). The 0.54%S reactor differed from the others in that the Cu release never rose above 4.1×10^{-13} mol/s, even though drainage pH decreased to 4.2.

ENP_{pH7} for Group II samples was 0.1-1.2 mg $CaCO_3$/g rock, while ENP_{pH6} was 0.4-8.0 mg $CaCO_3$/g rock (Table 21). All but two samples had drainage pH persistently below 5, with an ENP_{pH5} range of 0.9-8.5 mg $CaCO_3$/g rock. ENP_{pH4} was only calculated for the 0.40 and 0.57%S samples (9.9 and 6.8 mg $CaCO_3$/g rock respectively). For the nine samples with NP determined by the method of Sobek et al. (1978) (12-19 mg $CaCO_3$/g rock except 0.92%S with NP=4 determined by the method of Sobek et al. 1978), ENP values were lower (Table 22). The NP values for the five samples determined by the autotitrator method were in closer agreement at 2.0-5.4 mg $CaCO_3$/g rock.

5.3.1.2.3. Group III

Group III consisted of seven Duluth Complex samples (0.71 and 1.12-1.64%S) and four Virginia Formation samples (2.06-5.44%S). Although Virginia Formation samples are included in this group, discussion focuses on Duluth Complex samples with more complete periods of record.

This group contains samples for which minimum pH values were less than 4. Sample pH initially ranged from 5.5-7.5, and was below 5 by week 75 in six of seven Duluth Complex samples. Drainage pH dropped steadily until reaching approximately 4.3, and subsequently declined rapidly to minimum values of 3.0-4.2 between weeks 274-304 (Table 27). The 0.71% sample exhibited this same behavior with slightly different timing, as pH fell below 5 at week 190, and minimum pH was observed at week 362. Rapid increases were observed shortly after minimum pH was reached until ~pH 5, with pH rising more gradually to the end of the experiment (Figure 5). Virginia Formation samples declined rapidly to pH minima of 2.9-4.2 (Table 41), and did not rise due to the short period of record (78 weeks). For ease of use and to conserve space, only tables containing the calculated dissolution rates of minerals present are included in this report. Tables displaying the release rates from individual reactors are located in Appendix 5.

SO_4 release was highest in group III among Duluth Complex samples. Rates tended to decrease, only to experience marked increases after a short period of fairly constant release, and conclude with gradual decreases toward the end of the period of record (Figure 6). Initial release rates ranged from 3.1 x 10^{-12} to 3.6 x 10^{-10} mol/s (Table 28) over a period lasting 38-127 weeks (Table 24). Release rates subsequently decreased and stabilized at 50-75% of initial release rates for 87-174 weeks. Release rates for this period ranged from 1.4 x 10^{-11} to 1.3 x 10^{-10} mol/s and remained relatively stable until pH reached ~4.6. As pH declined below 4.6, release rates began to increase dramatically, rising for 46-139 weeks before reaching peak values. Maximum release rates from 2.5-3.6 x 10^{-10} mol/s typically occurred within 14 weeks of each sample's minimum pH (the 1.40%S sample took 39 weeks from the time of maximum sulfate release to reach minimum pH of 3.1). Peak rates were 3-9 times higher than the preceding stable period. Release rates began to decline shortly following maximum release, and continued to decrease through the end of the experiment. As pH rose, sulfate release rates decreased to levels at or below those observed prior to the period of increased release, and typically took 35-80 weeks to reach rates that occurred prior to the peaks. The only exception was the 1.40%S reactor, which was terminated 78 weeks after peak release, and did not completely return to pre-elevated values before the end of the experiment. Total release for Duluth Complex samples was 11.3-28.9 mmol, with an average of 21.7 mmol (Table 25).

Due to the Virginia Formation samples' much shorter period of record, total release was smaller at 5.5-14.5 mmol, with an average of 10.5 mmol (Table 25). Virginia Formation sulfate release behaved similarly to Duluth Complex samples in the inverse response to pH, and ranged from 1.5 x 10^{-11} to 3.0 x 10^{-9} mol/s, with a mean of 2.2 x 10^{-10} mol/s (Table 41).

For Duluth Complex samples, Ca release behaved similarly in group III as in groups I and II. Release tended to decrease with time initially, stabilize for a short period, then increase, and finally decrease and stabilize near the end of the experimental period. The initial period of increased Ca release lasted for 39-120 weeks. Rates ranged from 3.6 x 10^{-12} to 1.6 x 10^{-10} mol/s during this initial period (Table 29). Subsequently, a period of relatively stable release had rates from 7.3 x 10^{-13} to 2.3 x 10^{-11} mol/s, or 14-20% of initial rates. This period began between weeks 39-128 for all reactors, and lasted 87-174 weeks. As pH decreased below ~4.3, Ca release rates increased 2-3 times higher than the previous period rates, but still only 25-60% of the initial rates, on average. Release rates ranged from 3.6 x 10^{-13} to 3.2 x 10^{-11} mol/s during this period, and averaged 1.4 x 10^{-11} mol/s. Rates decreased quickly as pH began to rise, typically 80-150 weeks after increases were initially observed. As pH rose above ~4.4, Ca release rates decreased and quickly stabilized at rates similar to pre-elevated values, ranging from 2.2 x 10^{-12} to 1.4 x 10^{-11} mol/s and remained fairly constant for the remainder of the experiment. Total release for the experiment was 3.0-6.1 mmol, with an average of 4.1 mmol (Table 25).

Ca release from the Virginia Formation samples appeared to be similar to the initial erratic period of Duluth Complex samples. Release ranged from 8.4 x 10^{-12} to 8.0 x 10^{-10} mol/s during this period, and averaged 5.2 x 10^{-11} mol/s. Total release for the experiment was 1.6-3.7 mmol, with an average of 2.8 mmol (Table 25).

Group III Duluth Complex samples displayed Mg release patterns similar to those observed in groups I and II. Release rates decreased initially before stabilizing at lower levels. This preceded elevated release during low pH periods, which decreased again near the end of the experiment (Table 30). Initial release rates were 1.2 x 10^{-12} to 8.9 x 10^{-11} mol/s, for a period of 38-127 weeks. Mg release rates then began to decrease, stabilizing at levels 67% of initial rates, on average. These rates ranged from 1.2 x 10^{-12} to 2.5 x 10^{-11} mol/s, and the period of stable release lasted 87-174 weeks. As pH dropped to ~4.4, Mg release rates began to climb, eventually reaching rates as much as 4 times higher than initial rates (Figure 9). The elevated rates ranged from 3.5 x 10^{-12} to 8.2 x 10^{-11} mol/s, and typically took less than 25 weeks after pH decline to reach maximum release rate. Shortly after maximum release values were observed, pH began to rise, and release rates began to decrease. Rates quickly dropped below pre-elevated rates and continued to decline until termination of the experiment, with all reactors displaying release rates below pre-elevated rates (Table 30). This is contrary to Ca release rates, which stabilize following elevated release. This late decline in Mg release relative to Ca is similar to that observed in other samples. Total release for the entire experiment was 2.6-6.9 mmol, with an average of 5.0 mmol (Table 25).

Mg release from the Virginia Formation samples also appeared to be similar to the initial erratic period of Duluth Complex samples. Release ranged from 1.3 x 10^{-11} to 7.5 x 10^{-10} mol/s during this period, and averaged 6.7 x 10^{-11} mol/s. Total release for the experiment was 2.4-4.7 mmol, with an average of 3.5 mmol (Table 25).

Marked concurrent Mg and Ca release rate increases were once again observed in all reactors during low pH periods, with Mg release greatly exceeding Ca release in 6 of the 7 Duluth Complex samples (Table 25). As pH decreased below approximately 4.3, concurrent Ca and Mg release increases were noted. In the Virginia Formation samples, concurrent releases were hard to detect because of the short period and rapid pH decline.

Group III Na release was similar to the group I and II samples, with average values remaining relatively constant below 6.4 x 10^{-12} mol/s (Table 32). Potassium release was more variable, with rates similar to Group II samples (7.6 x 10^{-13} to 2.2 x 10^{-11} mol/s) (Table 33). Potassium release rates seemed to decrease, but the presence of a large number of anomalous values and irregularly collected data made determination of a distinct trend difficult. These results must be viewed with caution, as K and Na data were sparse for most samples when compared to Ca and Mg data. There was no Na or K data collected for Virginia Formation samples.

Group III had higher trace metal release than other group I and II samples (Tables 36-39). Average Duluth Complex release rates for periods A through D ranged from 6.8 x 10^{-15} to 2.1 x 10^{-12} mol/s for Co, 4.8 x 10^{-14} to 7.2 x 10^{-12} mol/s for Cu, 3.7 x 10^{-14} to 2.2 x 10^{-11} mol/s for Ni, and 4.0 x 10^{-14} to 7.2 x 10^{-13} mol/s for Zn. Peak Duluth Complex release rates among samples during periods A through D ranged from 1.8 x 10^{-14} to 2.9 x 10^{-12} mol/s for Co, 9.5 x 10^{-14} to 8.8 x 10^{-12} mol/s for Cu, 1.1 x 10^{-13} to 3.6 x 10^{-11} mol/s for Ni, and 4.7 x 10^{-14} to 9.8 x 10^{-13} mol/s for Zn. Ni release generally had an early peak as pH approached 6.0, and then declined until a similar peak later. This later peak coincided with a dramatic pH decline below 4.0 and a spike in sulfate release. Cu generally did not share the initial peak with Ni, but increased to its highest release rates concurrent with the second Ni peak. Co and Zn had similar patterns as Ni, although at lower release rates.

Virginia Formation and Duluth Complex samples had similar trace metal release behavior over the period of record shared by both. Average Virginia Formation release rates for the entire period of record ranged from 5.5 x 10^{-13} to 2.0 x 10^{-12} mol/s for Co, 5.1 x 10^{-14} to 2.0 x 10^{-12} mol/s for Cu, 2.6 x 10^{-12} to 1.9 x 10^{-11} mol/s for Ni, and 5.8 x 10^{-13} to 2.0 x 10^{-12} mol/s for Zn (Table 40). Peak Virginia Formation release rates during the entire period of record ranged from 1.0-6.7 x 10^{-12} mol/s for Co, 9.2 x 10^{-14} to 4.6 x 10^{-12} mol/s for Cu, 5.2 x 10^{-12} to 6.4 x 10^{-11} mol/s for Ni, and 1.1-7.1 x 10^{-12} mol/s for Zn.

ENP for Duluth Complex samples in Group III followed a similar pattern. The 1.40%S sample, for instance, had drainage below pH 7.0 very early and dropped below 3.5 before the end of the experiment. The ENPs calculated for this reactor at pH 6, 5, 4.5, 4, and 3.5 were 1.5, 3.8, 6.3, 10.4, and 13.1 mg $CaCO_3$/g rock, respectively (Table 21). In contrast to the results of Groups I and II, the NPs determined by the method of Sobek et al. (1978) (12-21 mg $CaCO_3$/g rock), were similar to the ENPs (Table 22). Virginia Formation samples had similar calculated ENPs to Duluth Complex samples (Table 21), despite being run for a much shorter period, but Virginia Formation NPs were generally lower.

5.3.1.3. Regressions Between pH, SO_4, Ca, Mg Release Rates and Solid Phase Composition

Sulfur depletion averaged 54.8% when determined as unleached solid %S minus sulfate release in leachate (Table 71). Sulfur depletion averaged 54.3% when determined as unleached solid %S minus leached solid %S (Table 71). There were large differences between methods for some samples, such as 103.7% for the sulfate method and 12.8% for the solids method with the 0.40%S sample. However, most samples had similar values between the two methods. Fifteen samples that had sulfur depletion determined using both methods. Of those, twelve had both values within 20% of the mean. A potential source of error for the leached solids analysis was that this was done on the replicate samples which were removed from the experiment long before the other samples. This would underestimate the loss of sulfur.

Sulfate release and pH were both correlated to the sample's sulfur content. Sulfate release increased with higher sulfur content (Figures 24-27, Table 34), with the greatest increases in periods B and C (slopes of average release = 1.03-1.35, r^2 = 0.52-0.78). Drainage pH decreased with higher sulfur content (Figures 20-23), with the steepest declines in periods A and B (slopes of average release = -1.69 to -1.37, r^2 = 0.34-0.45). Drainage pH was negatively correlated to average sulfate release (log transformed) for minimum, maximum, and average pH during all periods (Figures 28-31, Table 34). The greatest decreases in average releases occurred in periods A and B. Regression slopes of average release were from -1.65 to –1.17, and r^2 = 0.63-0.78.

Calcium and magnesium release was positively correlated to sulfate release (Figures 32-39, Table 34). The greatest increases were in periods C and D for both Ca (slopes of average releases = 0.59-0.60, r^2 = 0.68-0.94) and Mg (slopes of average releases = 0.94-1.42, r^2 = 0.60-0.88). When Ca and Mg release were added (Figures 40-43), the greatest increases were still in periods C and D (slopes of average releases = 0.71-0.92, r^2 = 0.94-0.97). Despite the rapid increase in Ca and Mg release, sulfate release rates were generally greater than the summation of Ca and Mg release in periods C and D, whereas they were more equal in periods A and B.

Metal content of unleached solids was not correlated to sulfur content for Co, Cu, Ni or Zn (Figures 44-47, r^2=0.03-0.08). However, metal release in leachate increased significantly with lower leachate pH for all four metals (Figures 16-19). Regressions of log metal release (mg/L) versus pH yielded slopes of -0.52 to -0.23, and r^2 of 0.25-0.54. Cobalt had a slope of –0.32, y-intercept of -0.54 and r^2 of 0.25 (Figure 48). Copper had a slope of –0.42, y-intercept of 0.37 and r^2 of 0.54 (Figure 49). Nickel had a

slope of –0.52, y-intercept of 1.0 and r^2 of 0.41 (Figure 50). Zinc had a slope of –0.23, y-intercept of -0.63 and r^2 of 0.40 (Figure 51).

5.3.2. Leaching Effects

5.3.2.1. Particle Size

Leaching decreased the mean perimeter:area ratio of all samples 17% from 104 to 87 (Table 17). Mean particle diameter, determined assuming spherical particles, increased in leached samples. Leached particle diameter (46.6 µm) was 9% higher than unleached (42.9 µm). However, the effect differed among samples, with half showing a decreased ratio and half displaying an increase. Overall, there appeared to be no consistent effect of leaching on grain size and the decreasing ratio of perimeter to area is consistent with decreasing surface roughness.

Samples that had decreased ratios from leaching had higher initial ratios (127.1) than samples with increased ratios (81.3). The decreasing samples had a much larger mean change (-49.3%) than increasing samples (+8.8%). Of these six decreasing samples, three (0.58 (R15 & 16), 1.63 and 1.71%S) were the only photomicrographs which showed a much larger number of small particles in the unleached than leached samples (Figures 55, 56 for 0.58%S as example; Appendix 1, Attachment A1.2 for all samples). There was no obvious difference in drainage quality from these three samples when compared to others without an abundance of finer particles. Because these samples were mounted in epoxy whose surface was then polished (see Appendix 1, Attachment A1.2 for more detail on these methods), these smaller particles are not expected to be of the type examined in the particle cleaning analysis discussed in the next paragraph.

On uncleaned-leached, cleaned-leached and cleaned-unleached samples, adsorbed particulates were rare compared to uncleaned samples. These results were inferred from eight photos comparing washed and unwashed (Figure 57). The absence of detectable particles on the uncleaned-leached particles examined suggests that most surface particle were dissolved or washed off in the first 328 weeks. The samples that were compared likely overestimate the amount of adsorbed fines in the experiment as a whole, since they were both dry-sieved. Eleven of the twenty-five samples were wet-sieved and this would likely remove some of the adsorbed particles.

5.3.2.2. Chemistry

Total sulfur content of leached samples from the Duluth Complex declined to a range of 0.11-1.08%S (Table 4), which was 15-93% of original S, with a mean of 46%. This compares well with S depletion calculations based on measured sulfate in drainage, which had a range of 0-95% original S and a mean of 47% (Table 71). Evolved CO_2 did not exceed 0.11% in any sample and declined for all samples greater than 0.22%S. Loss on ignition (LOI2, adjusted for the difference in O between FeO and Fe_2CO_3) ranged from 1.3-3.4%. The major whole rock chemical constituents for all samples were SiO_2 (44.0-53.3%), Al_2O_3 (13.6-17.2%), CaO (0.6-8.8%), FeO (1.6-15.2%), and MgO (2.7-9.0%). Contents of K_2O (0.6-3.6%), Na_2O (1.2-2.7%), and TiO_2 (0.4-3.9%) were low (Table 4).

Total sulfur content of leached samples from the Virginia Formation declined to a range of 1.77-4.35%S, or 64-86% of original S, with a mean of 76%. Evolved CO_2 increased to greater than 0.62% for three of the four samples. Loss on ignition (LOI2, adjusted for difference in O between FeO and Fe_2CO_3) ranged from 2.8-12.5%. Virginia Formation samples continued to have higher percent Fe_2O_3, when compared to Duluth Complex samples.

Trace metal concentrations in leached samples remained similar to unleached quantities. Once again, the maximum reported value for most metals was less than 30 mg kg^{-1} (Ag, As, Bi, Cs, Dy, Er, Eu, Ga, Gd, Ge, Hf, Ho, In, La, Lu, Mo, Nb, Nd, Pb, Pr, Rb, Sb, Sm, Sn, Ta, Tb, Tl, Tm, Th, U, W, Yb). Metals (and their maximum concentration in mg kg^{-1}) present at higher concentrations were Y (217), Ce (46.8), Co (86), Zr (217), V (306), Zn (261), Sr (321), Ba (405), Cr (455), Ni (483), and Cu (>10000). The trace metal content of the leached samples displayed greater range than that of the unleached samples, especially copper. Cu concentrations for most of the samples ranged from 500 to 2000 ppm, with only one sample exceeding 2500 ppm. This sample (0.71% S) displayed Cu concentrations greater than 10000 ppm, and is therefore suspect due to the extreme nature of the value (Table 5). The only major differences between Duluth Complex and Virginia Formation samples were that Virginia Formation samples had lower Cu and higher Zn.

5.3.2.3. Mineralogy

5.3.2.3.1. Leaching Effects on Mineral Content

Of the 25 samples that underwent unleached chemical analyses, ten (two from 0.58%S) were analyzed using point count to determine the effects of leaching on mineralogy of all minerals. These samples leached for 144-328 weeks prior to termination. Chemical analysis of pyrrhotite, chalcopyrite and pentlandite showed the losses of sulfide mineral when compared to unleached samples, with the exception of the 0.71%S which showed a significant increase in copper content in the leached sample possibly due lack of a representative sample. Average sulfide mineral percent depletion decreased in the order Po>Pn>Cp with values of 64, 40, and 10% depletion, respectively (Table 9). Pyrrhotite grains were generally found in highly altered states, if at all, whereas chalcopyrite remained less altered in all samples (Tables 14, 15). This indicates that pyrrhotite underwent dissolution at a significantly higher rate than chalcopyrite. Microprobe analysis of pentlandite, pyrite and cubanite failed to detect grains in both unleached and leached analyses of any sample, so there was no way to determine oxidation of these minerals using point count.

Samples' silicate minerals displayed similar content to corresponding unleached samples, with plagioclase, augite, hypersthene, and olivine comprising 74-94% weight of samples. The percentage is slightly higher in the leached specimens than in the unleached materials. Other minerals, including amphibole, biotite, muscovite, chlorite, and cordierite were observed in approximately the same quantities as in unleached materials (Tables 7, 8).

5.3.2.3.2. Leaching Effect on Mineral Chemistry

Leached Duluth Complex sulfide minerals had the following range of stoichiometric coefficients: pyrrhotite $Fe_{7.48-7.60}S_8$, pyrite $Fe_{4.59}S_8$, chalcopyrite $(Cu_{4.22-4.38}Fe_{4.43-4.57})S_8$, and cubanite $(Cu_{2.75-2.99}Fe_{5.57-6.13})S_8$ (Table 12).

Leached Duluth Complex silicate minerals had the following range of stoichiometric coefficients: plagioclase $(Ca_{0.43-0.55}Na_{0.37-0.49})Al_{1.47-1.62}Si_{2.42-2.53}O_8$, augite $((Ca_{0.76-0.95}Na_{0.01-0.02})(Mg_{0.45-0.78}Fe_{0.35-0.54}Ti_{0.01-0.02})(Si_{1.95-2.00}Al_{0.06-0.09})O_6)$, hypersthene $((Mg_{0.92-1.11}Fe_{0.72-1.00})_2Si_{1.96-2.00}O_6)$, and olivine $((Mg_{0.70-1.14}Fe_{0.83-1.27})Si_{1.00-1.01}O_4)$ (Table 13).

5.3.2.3.3. Leaching Effects on Mineral Grain Coatings

Evidence for alteration of sulfide minerals during leaching was obtained in microprobe analysis of leached samples, with almost all sulfide minerals displaying oxide coatings and only limited coatings on

some silicate minerals.

When pyrrhotite was observed, Fe-oxy-hydroxide coatings were generally present both on mineral surfaces and along parting planes within the minerals (Figure 58 for example). The latter observation is apparently the "veined" occurrence referred to by Jambor (2003). When measured, coating thicknesses ranged from 2-14 microns. Chalcopyrite coatings were also observed (Figure 59 for example), but these coatings were typically thinner than pyrrhotite at 0-6 microns, although some were observed at 10 microns. Consistent with section 5.3.2.3.1, this indicates that the extent of pyrrhotite leaching was greater than that of chalcopyrite leaching.

The 0.58%S unleached samples contained pyrrhotite and chalcopyrite in a 3:1 ratio, but slightly altered chalcopyrite grains were the dominant sulfides present in the leached solid. In the two leached 0.58%S samples, one sample (R16) had only one pyrrhotite grain, which was nearly completely altered and replaced by oxide minerals. The other sample (R15) had a 2:1 pyrrhotite: chalcopyrite ratio, but the pyrrhotite had oxide coatings 2-14 µm thick. Seven of the remaining ten leached samples (0.41, 0.51, 0.71, 1.16, 1.40, 1.44, and 1.63%S) had similar results, with many slightly- to moderately-altered chalcopyrite grains and few, highly altered pyrrhotite grains. In the 0.22, 0.40 and 1.71%S samples, more pyrrhotite grains were observed, but were once again more highly altered than nearby chalcopyrite grains, with the exception of the 1.71%S sample which had little sulfide alteration. In samples that leached for a short period of time, such as the 1.71%S sample, or a sample with relatively low solid-phase sulfur content (<0.40%S), pyrrhotite grains had a thin Fe-oxy-hydroxide coating, whereas chalcopyrite grains remained completely unreacted. In samples leached for longer periods of time or with higher sulfur content (>0.40% S), pyrrhotite grains showed evidence of advanced dissolution or were completely reacted away, whereas most chalcopyrite grains had a marginal coating, indicating a lesser degree (or slower rate) of dissolution (Table 16).

Coatings on sulfide grains provided information about the nature of leaching, as well as supplying a means of quantifying the extent of leaching within the sample. Coatings were typically a SiO_2-Fe-oxide-hydroxide phase. These coatings varied from just a few microns thick on non-reactive grains to complete replacement of sulfide mineral grains (Figure 58). It was found that sulfide minerals leached at different rates. Iron-bearing sulfides like pyrrhotite reacted much faster than Cu-bearing sulfides, such as chalcopyrite (Table 16).

Silicate minerals showed limited signs of dissolution. The lower sulfur samples (0.22 and 0.40%S) displayed no evidence of mineral dissolution, either from coatings or alteration features. As sulfur content increased, dissolution or alteration features increased in prevalence (Figure 59). The 0.58%S sample included several hypersthene grains with oxide coatings. More alteration was seen in the 1.16 and 1.40%S samples, where dissolution pits and grooves are found in plagioclase, amphibole, hypersthene, and olivine grains. Oxide coatings of 0-3 µm were observed on both olivine and hypersthene grains, and several greenalite-coated olivine grains were also observed. The 1.63%S sample contained several olivine and hypersthene grains with andesine coatings (Appendix 1, Attachment A1.2). Coatings were thicker on olivine grains in this sample, ranging from four to six microns. In general, as solid-phase sulfur content increased dissolution of silicate minerals also increased, in response to increased acid production. This resulted in increased thickness of coatings and extent of alteration features on silicate minerals in high sulfur solids (Table 15).

During microscopic analysis, several hypersthene and olivine grains were found with alteration structures. These alterations are unlike the oxide coatings found on other leached mineral grains, but it is not clear whether the alteration products are pre- or post-leaching. The dominant alteration phases observed are greenalite and serpentine. Pre-leaching weathering of the sample or some form of in situ

alteration (serpentinization) could have produced these phases before any leaching from the experiment occurred. These phases may not have been observed in any unleached samples due to the small overall number of points counted, a possibility discussed earlier in this report (Table 15).

5.4. Mineral Dissolution

Mineral dissolution was examined through the use of several models. Models 1-4 determined silicate mineral dissolution rates based on an initial assumption of specific mineral contributing the observed release of a given solute. From this foundation, rates of dissolution of five to six other minerals were calculated using both logical inference and necessary assumptions. Model 1 was based on Ca, model 2 on Mg, model 3 on Na, and model 4 on K. Results of these models were compared in four different periods (A-D). Models 1-4 were also compared among sample groups and among three pairs of samples with different mineral and chemical composition.

Models 1-4 are numbered from what are considered strongest to weakest. Calcium and Mg-based models are preferred because all samples have analyses of these elements throughout the experimental period (144-1252 weeks). Sodium and potassium analyses were run much later in the experiment than other parameters and for many of the samples measurements were not collected for the first 300-400 weeks of dissolution, making models 3 and 4 less preferred. All sample periods mentioned match those described in earlier sections. Again, it should be noted that reactors 39-44 were stored for 8 to 20+ years prior to dissolution testing and therefore produced higher drainage pH and lower sulfate and metal release rates. These reactors are included in the following results section and the differences in drainage pH and release rates will be neglected.

Potential gibbsite production was also calculated to determine whether this mineral's neutralizing capacity could explain the common sample pH minimum of approximately 4.0. Oxidation of pyrrhotite alone, as well as all sulfide minerals combined was determined and compared among sample groups.

Likely mineral sources of Co and Ni were determined using calculations based on solid and leachate chemistry, as well as mineral abundance. These metals had multiple potential mineral sources, unlike Cu and Zn. Copper was likely from chalcopyrite and cubanite, while Zn was likely from sphalerite (McSwiggen 1999).

5.4.1. Silicate Dissolution Models

5.4.1.1. Plagioclase

Calculated plagioclase rates among the first three silicate dissolution models had similar patterns. Dissolution was usually greatest during period A, declined to its lowest rates in period B, increased in period C, and declined again in period D. Model 1 generally had the highest single and average rates.

Initially, potassium was used to determine plagioclase dissolution for model 4. However, in three samples with no biotite or potassium feldspar detected (0.72, 0.92 and 1.16%S), predicted plagioclase dissolution rates were approximately 20-120 times higher than comparable rates predicted by models 1-3, as well as generally higher than literature dissolution rates at comparable pH (e.g., Oxburgh et al. 1994). Because of this disparity, model 4 plagioclase dissolution was computed in the same manner as model 3.

During period A, average plagioclase dissolution ranged from 6.2×10^{-13} to 5.7×10^{-12} mol m^{-2} s^{-1}, with the highest rate in model 1 and the lowest in model 3 (Tables 52-54). Because of the lack of Na data for

early samples, model 3 had only six samples during period A. The rates for model 3 samples were generally lower than results for the same samples using different models. The highest sample rate was 4.1 x 10^{-11} mol m^{-2} s^{-1} in model 1 and the lowest rate was zero in model 2.

During period B, average plagioclase dissolution ranged from 3.4 x 10^{-13} to 1.5 x 10^{-12} mol m^{-2} s^{-1} in the three models, with the highest rate in model 1 and the lowest in model 2 (Tables 52-54). The lack of Na data for samples initiated in 1989-1990 continued to limit data available for modeling, but there was an increase to nine samples with available Na data for modeling during period B. The highest sample rate was 9.3 x 10^{-12} mol m^{-2} s^{-1} in model 1 the lowest rate was zero in model 2.

During period C, average plagioclase dissolution ranged from 3.3 x 10^{-13} mol m^{-2} s^{-1} for model 2 to 9.2 x 10^{-13} mol m^{-2} s^{-1} for model 3 (Tables 52-54). Model 1 rates, which were the same as the average rates from period B because of model design, were higher than the other two models. Between models 2 and 3, the highest sample rate was 4.1 x 10^{-12} mol m^{-2} s^{-1} in model 2 and the lowest rate was zero in model 2.

During period D, average plagioclase dissolution ranged from 4.1 x 10^{-13} mol m^{-2} s^{-1} for model 2 to 1.3 x 10^{-12} mol m^{-2} s^{-1} for model 1 (Tables 52-54). The highest sample rate was 4.6 x 10^{-12} mol m^{-2} s^{-1} in model 1 and the lowest rate was zero in model 2.

Maximum plagioclase dissolution rates were determined based on sample stoichiometry and cumulative Ca release. Rates increased from group I, with an average of 1.0 x 10^{-9} mol m^{-2} s^{-1}, to group III, with an average of 1.9 x 10^{-9} mol m^{-2} s^{-1} (Table 51).

Observed Ca and Na release was compared to the solid phase Ca: Na ratio in plagioclase (section 4.4.5.2). In general, the Ca:Na ratio was higher in the observed drainage quality than was expected compared to the solid-phase stoichiometric Ca:Na ratio in plagioclase (Appendix 7, Figures A7.1-A7.21). The solid-phase average ratio ranged from 0.7 – 2.0, with an overall average value of 1.2. Whereas the average Ca:Na ratio in the drainage quality ranged from 1.9-23, with an overall average value of 4.7. This may indicate that there was preferential dissolution of calcium. The observed Ca:Na ratio may also be higher due to other sources of Ca release (i.e. augite), which are not accounted for in this analysis. A number of reactors (0.41, 0.51, 0.71, 1.16, 1.40, 1.44%S) displayed stoichiometric release of plagioclase early in the experiment (weeks 100-235). However the observed Ca:Na release ratio increased considerably as drainage pH decreased below 4.4, which could be associated with augite dissolution under acidic conditions. Future reporting should examine this in more detail.

5.4.1.2. Augite

Calculated augite dissolution rates in models 2, 3, and 4 were usually greatest during period A, declined in period B, increased in period C, and declined in period D to levels similar to period B. Model 1 calculated augite dissolution only during acidic conditions (period C). These values were the lowest calculated for all four models. Augite dissolution was calculated in model 4 in the same manner as model 3.

During period A, average augite dissolution ranged from 8.8 x 10^{-12} mol m^{-2} s^{-1} in model 3 to 9.5 x 10^{-12} mol m^{-2} s^{-1} in model 2 (Tables 53-56), with no augite dissolution determined in model 1. Because of the lack of Na and K data for early samples, period A of models 3 and 4 had only six samples. Augite dissolution rates for all samples in model 3 were lower than those in model 2. The highest sample rate was 5.0 x 10^{-11} mol m^{-2} s^{-1} in model 2 and the lowest rate was 1.1 x 10^{-12} in model 3.

During period B, average augite dissolution ranged from 1.9 x 10^{-12} mol m^{-2} s^{-1} in model 3 to 3.5 x 10^{-12}

mol m^{-2} s^{-1} in model 2 (Tables 53-56), with no augite dissolution determined in model 1. Because of the lack of Na and K data for early samples, models 3 and 4 had only nine samples during period B. The highest sample rate was 2.2 x 10^{-11} mol m^{-2} s^{-1} (model 2) and the lowest rate was zero (model 3).

During period C, average augite dissolution ranged from 3.3 x 10^{-12} mol m^{-2} s^{-1} in model 1 to 6.7 x 10^{-12} mol m^{-2} s^{-1} in model 2 (Tables 53-56). The highest rate among samples was 5.9 x 10^{-11} mol m^{-2} s^{-1} in model 2 and the lowest rate was zero in models 1 and 3.

During period D, average augite dissolution ranged from 1.3-2.3 x 10^{-12} mol m^{-2} s^{-1} with the highest rate in model 2 and the lowest in model 4 (Tables 53-56), and no augite dissolution determined in model 1. The highest sample rate was 1.4 x 10^{-11} mol m^{-2} s^{-1} in model 2 and the lowest rate was zero in model 3.

5.4.1.3. Hypersthene, Olivine and Biotite Dissolution

In models 1-3, calculated dissolution rates for ferromagnesian metals increased from periods A-C, and then declined to their lowest levels in period D. Model 4 rates behaved differently between the three minerals. Model 4 hypersthene and olivine rates decreased from period A to B, increased to the highest rates in period C and decreased in period D. Model 4 biotite rates decreased from periods A to C and then increased in period D. Model 1 had the highest average rates for all ferromagnesian metals in all periods.

During period A, average hypersthene dissolution ranged from 6.7 x 10^{-13} mol m^{-2} s^{-1} in model 2 to 5.5 x 10^{-12} mol m^{-2} s^{-1} in model 1 (Tables 52-56). Average olivine dissolution ranged from 4.3 x 10^{-13} mol m^{-2} s^{-1} in model 4 to 1.7 x 10^{-11} mol m^{-2} s^{-1} in model 1. Average biotite dissolution ranged from 1.3 x 10^{-13} mol m^{-2} s^{-1} in model 3 to 1.5 x 10^{-11} mol m^{-2} s^{-1} in model 4. For all samples, the highest rates for hypersthene, olivine, and biotite were 4.0 x 10^{-11}, 2.0 x 10^{-10}, and 6.2 x 10^{-11} mol m^{-2} s^{-1}, respectively, and they all occurred in model 1. All three minerals had a minimum sample rate of zero in models 2 and 3.

During period B, average hypersthene dissolution ranged from 6.9 x 10^{-13} mol m^{-2} s^{-1} to 3.4 x 10^{-12} mol m^{-2} s^{-1} in models 4 and 1, respectively (Tables 52-56). Average olivine dissolution ranged from 3.1 x 10^{-13} mol m^{-2} s^{-1} to 9.1 x 10^{-12} mol m^{-2} s^{-1} in models 4 and 1, respectively. Average biotite dissolution ranged from 2.8 x 10^{-12} to 1.0 x 10^{-11} mol m^{-2} s^{-1} in models 2 and 4, respectively. Hypersthene's highest sample rate was 2.9 x 10^{-11} mol m^{-2} s^{-1} in model 1, olivine's was 5.4 x 10^{-11} mol m^{-2} s^{-1} in model 1, and biotite's was 5.3 x 10^{-11} mol m^{-2} s^{-1} in model 4. All three minerals had a minimum sample rate of zero in models 2 and 3.

During period C, average hypersthene dissolution ranged from 2.7-4.4 x 10^{-12} mol m^{-2} s^{-1} in models 4 and 1, respectively (Tables 52-56). Average olivine dissolution ranged from 7.0 x 10^{-12} mol m^{-2} s^{-1} in model 4 to 1.6 x 10^{-11} mol m^{-2} s^{-1} in model 1. Average biotite dissolution ranged from 3.4-6.0 x 10^{-12} mol m^{-2} s^{-1} in models 2 and 1, respectively. For all samples, the highest rates for hypersthene, olivine, and biotite were 3.1 x 10^{-11}, 6.8 x 10^{-11}, and 3.3 x 10^{-11} mol m^{-2} s^{-1}, respectively, and they all occurred in model 1. All three minerals had a minimum value of zero in models 1, 2 and 3.

During period D, average hypersthene dissolution ranged from 1.3 x 10^{-13} mol m^{-2} s^{-1} in model 2 to 1.1 x 10^{-12} mol m^{-2} s^{-1} in model 1 (Tables 52-56). Average olivine dissolution ranged from 2.4 x 10^{-13} mol m^{-2} s^{-1} in model 4 to 5.0 x 10^{-12} mol m^{-2} s^{-1} in model 1. Average biotite dissolution ranged from 2.6 x 10^{-13} mol m^{-2} s^{-1} in model 2 to 4.4 x 10^{-12} mol m^{-2} s^{-1} in model 4. Hypersthene's highest sample rate was 7.0 x 10^{-12} mol m^{-2} s^{-1} in model 1, olivine's was 3.2 x 10^{-11} mol m^{-2} s^{-1} in model 1, and biotite's was 4.1 x 10^{-11} mol m^{-2} s^{-1} in model 4. All minerals had a minimum sample rate of zero in models 2 and 3.

Maximum olivine dissolution rates were determined based on sample stoichiometry and cumulative Mg release. Group I had an average of 4.1 x 10^{-9} mol m^{-2} s^{-1}, group II had an average of 3.0 x 10^{-9} mol m^{-2} s^{-1}, and group III had an average of 1.0 x 10^{-8} mol m^{-2} s^{-1} (Table 51).

5.4.1.4. Orphan Calcium

The potential for Ca release not attributed to any specific mineral occurred in period C of Model 1 and all periods of model 3. In model 3, average orphan Ca release declined throughout the experiment from 3.7 x 10^{-12} mol m^{-2} s^{-1} in period A to 3.6 x 10^{-13} mol m^{-2} s^{-1} in period D (Table 55). During period C, average orphan Ca in model 1 was 25% of that in model 3 (Table 52).

5.4.1.5. Potassium Feldspar

Potassium feldspar dissolution rates were only determined in model 4 because of its limited importance in modeled elements other than potassium. Average dissolution rates declined every period from 2.5 x 10^{-10} mol m^{-2} s^{-1} in period A to 1.8 x 10^{-11} mol m^{-2} s^{-1} in period D. The maximum dissolution rate of 4.7 x 10^{-10} mol m^{-2} s^{-1} occurred in period A, while the minimum rate of 6.3 x 10^{-13} occurred in period D (Table 56).

5.4.1.6. Comparison of Sample Groups and Specific Samples with Varying Mineralogies

Group III samples usually had the highest release rates regardless of mineral, model or period (Tables 60-63). In periods A and B of models 3 and 4, when there were no data for group III samples, group II generally had the fastest rates.

To compare model consistency among samples with different mineral contents, three pairs of individual samples were chosen. Dissolution rates of plagioclase, augite, hypersthene, olivine, and biotite were compared among models within pairs to determine consistency of model results for similar samples. Dissolution rates among sample pairs for each model were also compared to determine how mineral content differences affected model results (Tables 64-67, Figures 73-77).

The first pair of samples (0.72 and 1.16%S) had the most Ca, Na and K (>76%) contained in plagioclase, while most Mg (51-85%) was in olivine. The second pair of samples (0.67 (R40) and 0.82%S) had Na distribution similar to the first pair (99-100% in plagioclase), but differed in the distribution of Ca (67-68% in plagioclase), Mg (26-29% in augite) and K (67-69% as biotite). The third pair (1.40 and 1.64%S) had Ca (61% in plagioclase), Mg (35-36% in augite) and Na (97-98% in plagioclase) distribution similar to the second pair, but had most K (61-68%) as potassium feldspar.

Plagioclase typically had the best agreement between models 1-4, where in 8 of the 11 cases, the range in calculated plagioclase dissolution was less than a factor of 4. The best agreement in calculated plagioclase dissolution occurred in model 3. Biotite typically had the worst agreement between models 1-4, where in only 1 out of 11 cases, the range in calculated biotite dissolution was less than a factor of 3. It should be noted that the samples in set 1 (0.72 and 1.16%S) did not contain biotite.

Model 3 typically had the best agreement for each sample set where in 10 of the 18 cases, the range in calculated dissolution rates was less than a factor of 5. Agreement in model 3 was the best during period B. Model 2 typically has the worst agreement for each sample set where in only 4 out of 19 cases, the range in calculated dissolution rates were less than a factor of 4.

Rate period A typically had the best agreement for each sample set, where in 8 of the 16 cases, the range in calculated dissolution rates was less than a factor of 5. Agreement during period A was the best for calculated plagioclase dissolution rates. Rate period C typically had the worst agreement for each sample set, where in only 4 out of 18 cases, the range in calculated dissolution rates were less than a factor of 3.

5.4.2. Gibbsite Production

Potential gibbsite production was determined at the point pH dropped below 4.0, or at minimum pH where samples never had pH less than 4.0. Dissolution of maximal amounts of gibbsite was well in excess of acidity production for all samples except 1.44%S (Table 68).

5.4.3. Pyrrhotite and Combined Sulfide Minerals Oxidation Models

Oxidation of pyrrhotite and all sulfide minerals declined from period A to B, rose in period C, then declined in period D. Both pyrrhotite and all sulfide rates were highest in period C with average rates of 4.2×10^{-10} mol m^{-2} s^{-1} and 3.8×10^{-10} mol m^{-2} s^{-1}, respectively (Tables 69, 70). Average oxidation rates were approximately 10-20% higher for pyrrhotite than the combined sulfide minerals in periods B, C, and D.

When comparing results among sample groups, the pyrrhotite model had varied results with the fastest rates in period A for group I and period C for groups II and III. When modeling all sulfide minerals, group III had the fastest oxidation rates in period C, followed by group II in period C. Both the dependence of rate on period and group reflect the correlation of sulfide oxidation and pH.

5.4.4. Mineral Sources of Co and Ni

Cobaltite, (Co, Fe)AsS, is a common mineral source of Co. However, As in unleached solids is quite low for Duluth Complex samples (Table 3). To determine the maximum percent of cobalt present as cobaltite in these samples, As:Co ratios were determined. Thirteen of twenty-one samples had nondetectable As, while the remaining samples had As:Co ratios of 5.4-19.2% (mean 11.2%, 5.0% SD). Virginia Formation samples had ratios of 9.2-31.8% (mean 20.6%, 9.9% SD).

Maucherite ($Ni_{11}As_8$) is not a potentially important source of Ni for reasons similar to cobaltite. Duluth Complex samples had maximum Ni contributions of 1.6-7.9% (mean 3.6%, 2.1% SD) from maucherite. Virginia Formation samples had maximum contributions of 2.0-5.9% (mean 4.1%, 1.6% SD).

The major potential sources of Ni in these samples were pentlandite ($(Ni, Fe)_9S_8$), pyrrhotite ($Fe_{1-x}S$), and olivine ($(Mg,Fe)_2SiO_4$). In order to determine the possible sources, Ni content was analyzed in olivine, pyrrhotite, and pentlandite (only two grains identified in all samples) in the solid-phase samples. Expected Ni release was then determined from samples based on magnesium (olivine originated) and sulfate (sulfide mineral originated) release rates. Because only two grains of pentlandite were located in the Duluth Complex samples, it was necessary to estimate the nickel present in sulfides based on sample composition to estimate nickel release from sulfide minerals. These predicted values were compared to actual Ni release to ascertain likely sources of Ni in leachate. Finally, maximum olivine Ni release rates from the samples were estimated.

Most of the Ni in the solids was calculated to be present in sulfides, ranging from 66 to 100 percent in 18 of 21 samples (Table 42). The two samples in which all Ni was calculated to be present in sulfides contained no olivine. The fraction of Ni present in sulfides for the remaining three samples ranged from

–25% (0.92%S, see methods for possible sources of error), to 28 percent. Solid-phase analysis indicated Ni was a stoichiometrically low proportion of olivine (0.002 mol Ni/ mol ol, Table 42) and pyrrhotite (0.031 mol Ni/ mol po, Table 10). Pentlandite had the highest concentration of Ni at 4.88 mol Ni/ mol pent (Table 10).

The average aqueous release Ni/Mg ratio for all samples was approximately 29 – 87 times higher than the ratio found in olivine (Table 43). The three group I samples (0.18, 0.22, 0.72%S) had average release Ni/Mg ratios more similar to olivine (aqueous average 3-4 times ol ratio). Group I samples were the only samples to never decline below pH 6 for an extended period (Table 27).

When maximum olivine Ni release was calculated based on Mg release in leachate, calculated Ni release from olivine averaged <15% of actual Ni release (Table 44). Linear regression showed a significant positive correlation between sample sulfur content and calculated:observed Ni release in periods C and D (Table 45, Figure 62). In periods C and D, lower sulfur samples were absent. The increase in calculated release from olivine is due to increased Mg release. The increased Mg release likely resulted from increasing Mg-bearing mineral dissolution due to lower pH.

To determine the maximum possible rate of Ni release from olivine, the rate of observed Ni release was divided by the number moles of olivine present. This assumed that all Ni was the result of olivine dissolution. The rate range of group I samples (0.18, 0.22, 0.72 %S) was 4.9×10^{-8} to 2.3×10^{-6} mol Ni · (mol ol)$^{-1}$ · week^{-1} (Table 46). The rate range of the two high-olivine (0.57, 0.92%S) samples with lower pH minima was 7.9×10^{-7} to 7.7×10^{-6} mol Ni · (mol ol)$^{-1}$ · week^{-1}. Linear regression of all rates in Table 46 showed significant positive relationships between (mol Ni) · (mol ol)$^{-1}$ · week^{-1} and %S (Table 47, Figure 63). The slopes of time periods B and C, which occurred after the initial rapid release of period A, were very similar.

Predicted Ni release from sulfide minerals was also determined using observed sulfate release multiplied by the Ni/S ratio in the sulfides. As can be seen in Figure 64, the predicted values are often higher than observed Ni release. For all samples, the median ratio of predicted to observed nickel release was 2.9. This data was broken into the rate periods A through D and cumulative probability distribution graphs were generated to determine the relationship between the ratio of predicted to observed Ni release as a function of pH (Figure 65). Looking at the cumulative probability of predicted vs observed Ni release for each rate period, periods A and B are fairly similar, with median values of 2.6 and 1.8, respectively. As pH decreased in period C, nickel release increased significantly, resulting in predicted values much closer to observed, with a 75th percentile predicted vs. observed nickel release ratio less than 5. As pH increased in period D, nickel release decreased much faster than sulfate release, resulting in higher predicted vs. observed nickel release ratios (median value of 9).

When comparing spikes in Ni release for individual samples in time series data to Mg and sulfate spikes (Appendix 3, Figures A3.1-A3.44), sulfate spikes generally matched the Ni spikes best. Of the twenty samples examined, sulfate spikes were the most similar to Ni spikes in eight samples (0.18, 0.22, 0.40, 0.41, 0.51, 0.54, 0.67 (R39), 0.82%S), while both Mg and sulfate were equally matched with Ni in four samples (0.67 (R40), 1.16, 1.40, 1.44%S). Six samples had Mg spikes more closely matching Ni (0.57, 0.58, 0.71, 1.12, 1.63, 1.71%S). Two samples had neither sulfate nor Mg dynamics that matched Ni (0.92 and 1.64%S). For group II and III samples, when drainage pH declined into acidic conditions, there was generally a corresponding spike in Mg release.

6. DISCUSSION

6.1. Introduction

The general objective of these experiments was to correlate rock composition and drainage water quality to aid in predicting management concerns for mining operations in the Duluth Complex of Northeastern Minnesota. The Duluth Complex is a large copper and nickel resource (Kingston et al. 1970), and contains elevated levels of platinum group elements (Anonymous 1992). Growing interest in mining of these resources highlights the importance of long-term studies, like this one, that will aid in determining drainage quality from mine wastes over time scales of decades to centuries.

Twenty-one samples with sulfur contents from 0.18-1.71% were ground to a well-defined size to maximize liberation of sulfide minerals and characterized with respect to chemistry, mineral content, and mineral chemistry. Weekly water additions over 144-1252 weeks allowed samples to fully dry each week in a sample chamber that was temperature and humidity-controlled. Leachate samples were analyzed to determine pH, sulfate, Ca, Mg, Na, and K (frequency varied among samples and analytes, Table 6). Cobalt, Cu, Ni, Zn, Fe, Al and Si were analyzed less frequently. Photomicrographs of unleached and leached samples were used to assess solid surfaces for evidence of mineral oxidation and dissolution. The results of this study allowed categorization of sample water quality based on duration of leaching and rock chemistry. These categories will aid in predicting treatment needs of mining wastes in the Duluth Complex.

The Duluth Complex samples came from three locations (Table 1). All seventeen Dunka blast hole samples were from the South Kawishiwi intrusion near Babbitt, Minnesota, and the only ARIMETCO and the three AMAX samples came from the Babbitt/Mesaba deposit of the Partridge River intrusion. The four Virginia Formation samples (2.06 ~~%S ~~5.44, all Dunka blast hole) were terminated after 78 weeks of dissolution. During the short period of record, sulfate and pH behaved similarly to the higher %S Duluth Complex samples, though the changes proceeded more rapidly due to the higher S content. The data indicated that waste rock with sulfur content in this range would produce drainage in need of rigorous environmental controls. It was concluded that testing of rock with lower sulfur contents would provide information more beneficial to environmental mine waste management planning than continuation of the higher sulfur samples already known to produce strongly acidic drainage. Because of the differences in chemistry and experimental duration, this discussion is limited to Duluth Complex samples. Section 5.0 briefly addresses the drainage quality from these samples and all drainage quality data is presented in Appendix 3.

It should also be noted that reactors 39-44 have been omitted from the discussion section (with the exception of section 6.7.4). These samples were stored for 8 to 20+ years before dissolution testing, resulting in higher drainage pH and lower release rates compared to other Dunka blast hole samples. In addition, since these were initiated 7 to 8 years after reactors 1-38, these samples represent different analytical periods. Elimination of reactors 39-44 focuses this discussion section specifically on South Kawishiwi intrusion (SKI) samples. It should be noted that two of these reactors contained Dunka blast hole samples. These weathered samples displayed higher drainage pH and lower release compared to "fresh" samples (Appendix 7, Attachment A7.2). For drainage quality results on the Partridge River intrusion (PRI) and South Kawishiwi intrusion (SKI) samples, see results section 5.0.

Although the dissolution tests remain in progress for a subset of samples, the following discussion presents the first 24 years of data. The dissolution tests were conducted to determine the variation of drainage pH with sulfur content, categorize the samples based on this examination, and determine the variation of sulfate and major cation release rates with sulfur content. This data was also used to

determine the variation of metal release with drainage pH and examine possible mineral sources of metals. In addition, these data were used to determine rates of sulfide oxidation and silicate dissolution as a function of time and drainage pH and provide a comparison with reported literature rates.

6.2. Sample Classification

Duluth Complex samples were divided into three groups based on the observation that drainage pH decreased as the solid-phase S content of samples increased.

Group I: $0.18 \leq \%S \leq 0.22$
Group II: $0.40 \leq \%S \leq 0.70$
Group III: $0.70 < \%S \leq 1.64$

Although there were some exceptions to the %S ranges stated above, these were all from samples with prolonged storage periods (0.72%S in group I; 0.82, 0.92, 1.71%S in group II). When sulfate release rates from the three AMAX samples from the present project (0.72, 0.92, 1.71%S), were compared to freshly ground samples of the same rock (0.59, 0.80, 1.87%, respectively) (Lapakko 1993a), rates of the stored samples were only 25-80% of the fresh samples (Appendix 7, Table A7.1). The prolonged storage might have resulted in formation of sulfide mineral coatings, which would cause slowing rates of acid production from sulfide oxidation. Janzen et al. (2000) has reported this phenomenon for pyrrhotite. Janzen stored two samples for 15 months in a desiccator and found the pyrrhotite surface areas had decreased by 85-90 percent.

Sulfate release and attendant acid production increased with sample group %S, while other release rates followed this trend due to decreasing pH. Minimum pH decreased as group solid-phase sulfur content increased. Metal release rates (Co, Cu, Ni, Zn) also increased with group sulfur content. Sodium and potassium had more limited sampling, but were much more similar among groups than other ions.

Group I had sulfur contents ranging from 0.18 to 0.22 % and typically produced drainage pH values of at least 6. Minimum pH ranged from 5.7-5.9 (mean 5.8, Table 80), but pH was generally between pH 6-6.5 by week 150. Average sulfate release rates were from 2.3×10^{-12} to 1.2×10^{-11} mol/s (Table 81). Average Ca release was 3.1×10^{-12} to 1.8×10^{-11} mol/s (Table 82). Average Mg release was 2.3×10^{-12} to 1.6×10^{-11} mol/s (Table 83). The range of average Na release was 1.3-1.4 x 10^{-12} mol/s (Table 85). Average potassium release ranged from 3.3-5.7 x 10^{-12} mol/s (Table 86). Average metal release rates ranged from 6.6×10^{-15} to 5.1×10^{-14} mol/s for Co, 2.1-4.3 x 10^{-14} mol/s for Cu, 1.7-9.2 x 10^{-14} mol/s for Ni, and 4.6-6.6 x 10^{-14} mol/s for Zn (Tables 89-92). The circumneutral pH resulted in relatively low trace metal release, due to low metal solubility.

Group II had sustained minimum pH of 4-5 and $0.40 \leq \%S \leq 0.70$. Minimum pH ranged from 3.8-4.1 (mean 4.3, Table 80), and occurred between weeks 115-770. Average sulfate release rates were from 8.3×10^{-12} to 5.1×10^{-11} mol/s (Table 81). Average Ca release was 3.1×10^{-12} to 3.7×10^{-11} mol/s (Table 82). Average Mg release was 5.9×10^{-13} to 1.6×10^{-11} mol/s (Table 83). Na release rates ranged from 8.0×10^{-13} to 3.7×10^{-12} mol/s (mean 2.3×10^{-12} mol/s) (Table 85). Potassium release rates were from 3.7 x 10^{-13} to 7.6×10^{-12} mol/s (mean 3.9×10^{-12} mol/s) (Table 86). Average metal release rates ranged from 4.9×10^{-15} to 2.5×10^{-13} mol/s for Co, 5.2×10^{-14} to 3.4×10^{-12} mol/s for Cu, 7.3×10^{-14} to 1.6×10^{-12} mol/s for Ni, and 2.4×10^{-14} to 2.7×10^{-13} mol/s for Zn (Tables 89-92).

Group III had sustained minimum pH < 4 and $0.70 < \%S \leq 1.71$. Minimum pH ranged from 3.0-3.5 (mean 3.3, Table 80), and occurred between weeks 274-362. Average sulfate release rates were from 1.1×10^{-11} to 1.9×10^{-10} mol/s (Table 81). Average Ca release was 3.1×10^{-12} to 5.8×10^{-11} mol/s (Table

82). Average Mg release was 2.2 x 10^{-12} to 4.4 x 10^{-11} mol/s (Table 83). Sodium release was similar to that of samples from groups I and II, with average values remaining relatively constant below 6.4 x 10^{-12} mol/s (Table 85). Potassium release rates were similar to Group II samples (1.5 x 10^{-12} to 1.6 x 10^{-11} mol/s) (Table 86). Average metal release rates ranged from 6.8 x 10^{-15} to 2.1 x 10^{-12} mol/s for Co, 4.8 x 10^{-14} to 7.2 x 10^{-12} mol/s for Cu, 3.7 x 10^{-14} to 2.2 x 10^{-11} mol/s for Ni, and 4.0 x 10^{-14} to 7.2 x 10^{-13} mol/s for Zn (Tables 89-92).

6.3. Temporal Variation

There were typical changes in water quality that occurred sequentially within sample groups. These changes were driven by the balance between rates of acid production associated with sulfate release and neutralization from cation release. Water quality changes were characterized using four sequential periods (A-D). These periods were defined by typical changes in pH, SO_4, Ca and Mg (section 5.3.1.1). Drainage pH generally declined throughout the first three periods and rose in the fourth. Periods A and B were shorter for samples with greater %S (i.e., pH declined more rapidly), and some samples did not enter periods C and D. Calcium and Mg release rates were highest in period A, declined in period B, rose in period C and declined again in period D. Sulfate release followed a similar pattern of increases and declines, but the highest release rates were in period C. Because the periods of record for Duluth Complex samples ranged from 144-1252 weeks, use of experimental periods also assists comparison among samples and groups with different periods of record.

Period A started at week 1 for each sample, and ended as Ca and Mg release rates stabilized. SO_4 release tended to fluctuate during this period of cation stabilization. Drainage pH gradually decreased throughout this period, and varied among samples, ranging from 4.5-8.7 (Table 80). Group I samples stabilized at pH 6-7, while sample groups II and III stabilized between pH 4.5-5.5 by the end of Period A. Initially, each sample had relatively high and variable Ca and Mg release. Ca concentrations typically exceeded Mg concentrations, and both gradually declined with time. This period lasted from 34-140 weeks and occurred in all samples.

Period B began when Mg and Ca release rates stabilized at levels 20-30 percent of those in Period A. Average Mg release rates exceeded those of Ca in 11 of 21 samples. Similarly, average SO_4 release rates decreased 22-77 percent of those rates in Period A (Table 81). Drainage pH typically decreased gradually throughout this period (average pH= 6.5 group I, 5.2 group II, 4.6 group III). Period B lasted from 66-1118 weeks and occurred in all samples.

Period C began with a relatively rapid pH decline (below ~4.4) and concurrent increases in Ca, Mg and SO_4 release, with SO_4 release increases typically occurring 15-55 weeks prior to Ca and Mg release increases. During this period it was common to have Mg spikes exceeding Ca release at the pH minimum. This increase in Mg release was particularly common in group III samples. Ca, Mg and SO_4 release decreased as Period C ended. Period C occurred in all group II and III Dunka blast hole samples. No group I samples entered period C.

Period D was characterized by gradually increasing pH (above ~4.2-4.4), and Ca, Mg, and SO_4 release rates lower than period C. Calcium release rates remained relatively stable during Period D, at levels similar to Period B, while Mg release rates gradually decreased to levels less than or equal to rates observed in Period B. Period D occurred in most group II (all but 0.54%S) and group III Duluth Complex samples (all but 1.40%S). No group I samples entered period D.

6.3.1. Initial Flush

The flush of Ca, Mg SO_4 at the beginning of period A might be due to 1) the rapid dissolution of fine particles adhered to the mineral grains, 2) rapid dissolution and depletion of more reactive minerals present in small amounts, 3) rapid ion exchange, 4) exposure of new surfaces to oxidation and dissolution as a result of grinding, or 5) release of solutes generated by dissolution during sample storage. There was physical evidence that these samples might have had higher initial release rates due to adhered particles on mineral grain surfaces. Photomicrographs clearly show that some grains had numerous particles on grain surfaces that were removed by washing (Figure 57). Also, leached grains from the same sample were missing most of these particles, suggesting that these particles dissolved with time. Another indication that smaller particles dissolved was the decrease in the perimeter:area ratio after samples were leached (Table 17).

Smaller particles are likely to be from softer minerals, such as sulfides (Lapakko et al. 2004a), so this factor may contribute more to sulfate release than to release of other ions. On the other hand, fine particles would be more susceptible to loss from the bed by physical transport with the leachate. Crushing may result in less size reduction than expected of some softer minerals, such as biotite, which tend to resist the equally distributed force applied by most crushing methods (i.e. are "tougher", Chodos and Engel 1961). This limited size reduction would likely lower dissolution rates. Finally, there may be error in the perimeter:area measurements due to the how the grains were mounted having increased the likelihood of underestimating the area of larger particles (Appendix 1, Attachment A1.6).

The initial flush of ions is also typical in studies that have wet-sieved and sonicated grains prior to the start of the experiment (e.g., White and Brantley 2003), implying that adhered particles are not the only contributing factor in initial elevated release. Samples in this experiment that were wet-sieved showed a similar initial flush as dry sieved samples, further indicating the relative unimportance of adhered particles. Thus, the flush probably has more to do with other factors.

White and Brantley (2003) attributed an initial Ca flush from granite to dissolution of trace calcite. Further support for this interpretation was provided when they found the Ca flush in unweathered rock, but not weathered rock from the same core. Although point count data from Dunka blast hole samples detected calcite rarely and at low amounts, calcite might be more abundant than these data indicate. Discrepancies in point count data are likely due to error inherent in the point count method. With the number of grains measured (~100), the visual point count method has a 2-σ error level (86.5% confidence) that makes it possible that true values of mineral abundance under 10% could be 60-100% different from estimates. Samples with no grains of a certain mineral detected could have as much as 2% of the sample present as the undetected mineral grains. Evolved CO_2 from unleached samples was almost always more than in leached samples, showing loss of calcite. The calcite depletion may have been more than indicated by the CO_2 data because of either new calcite formation in mineral microsites or CO_2 absorbed into the lattice of minerals. Adsorption of CO_2 has been reported in olivine (Oberheuser et al. 1983, Schott and Berner 1983). Berg and Banwart (2000) have proposed a mechanism for CO_2 adsorption on anorthite where inorganic carbon is rapidly adsorbed, forming a reactive bi-dentate surface Al–carbonate complex that is released to solution in a much slower, irreversible step.

Initial flushes of cations due to rapid ion exchange have been known for a long time (e.g., Garrels and Howard 1959). These initial flushes are very rapid (~ one minute) and are due to reversible exchange of alkali ions on the feldspar surface with hydrogen ions in solution. This and other early research reported parabolic dissolution kinetics, which was thought to be due to armoring of surface layers that limited transport of reactants or products (Phillips et al. 2001). More recent laboratory experiments appear to

favor the silicate dissolution model of Furrer and Stumm (1986), which was developed for simple oxide dissolution (Brantley and Chen 1995). This dissolution rate is a form of ligand-exchange reaction that is proportional to the concentration of surface complexes and valence of the metal ion. This model emphasized the control of silicate dissolution by surface reaction, rather than diffusion through a surface layer of reaction products (Blum and Stillings 1995).

Initial flushes of sulfate and Mg are most likely due to exposure of fresh mineral surfaces and fractures during grinding. Janzen et al. (1997) reported that surface area of pyrrhotite appears to be the most influential variable affecting oxidation rates, and differences in crushing methods were found to affect surface area of mineral grains smaller than 212 μm (the starting size used in this experiment) (Janzen et al. 1997). These differences may be due to differing strains on the mineral grains causing differing amounts of mineral grain fractures (Janzen et al. 1997). Fracture sites are particularly vulnerable to oxidation (Janzen et al. 1997, 2000). Eggleston et al. (1989) found dissolution rates of diopside (end member of the solution series containing augite) decreased with time, showing the importance of fresh surfaces in silicate mineral dissolution.

6.3.2. Biological Mediation

As time passes, sulfide oxidation increases, coincident with declining pH. The rates of sulfate release exceed the combined release of Ca and Mg, resulting in net increases in acidity. The accelerated rate of sulfate release below pH 4 was probably due to increased biologically mediated ferric-iron oxidation of the sulfide minerals. Chemical oxidation may have also increased, although the extent of increase was most likely slight relative to the increase in biological oxidation (Nordstrom 1982). Thus, group III sulfide mineral oxidation creates a positive feedback in which the decreased pH conditions from sulfide oxidation create conditions favorable to increased oxidation by bacteria.

6.3.3. Development of Coating (Shrinking Core)

As sulfide minerals oxidize, the outer layer becomes sulfur-depleted and an Fe^{III}-O layer forms that is resistant to oxidation of the underlying sulfide (Pratt et al. 1994a, b, Figure 66). This coating development, combined with depletion of sulfide minerals and continued dissolution of silicate minerals, results in the increasing pH characteristic of period D. Sulfide mineral abundance measured with point count and chemical analyses revealed a >50% decrease in pyrrhotite, while showing no significant decrease in chalcopyrite. Pyrrhotite oxidizes much faster than chalcopyrite (Jambor et al. 2005), and the relative increase in slower-oxidizing sulfide minerals may partially explain the decreased sulfate release, and consequent pH increase, in period D.

6.3.4. Calcium versus Magnesium

The instances of Mg release exceeding Ca release around the pH minima can be explained by the greater sensitivity of ferromagnesian minerals to decreasing pH. Augite dissolution was the major source of accelerated Mg release at lower pH (<4.5). Plagioclase, which contained the majority of Ca, was less sensitive to pH. This is shown by the consistently steeper regression slopes of Mg versus sulfate when compared to Ca versus sulfate (Table 87). However, average total Ca release was generally 1.4 to 2.5 times higher than total Mg release in the Group I and Group II samples, respectively (Table 79). Average Mg release only exceeded average Ca release for an entire period in group III during periods B and C. This was most likely due to the prolonged period at lower pH which resulted in increased olivine and hypersthene dissolution. Group III samples still had increased Ca release in period C, as did group II. However this increase was not enough to equal the greater increase in Mg release. Group III samples had the lowest pH, and olivine, the source of 28% of Mg in Duluth Complex samples (Table

23), displayed dissolution rates that are more sensitive to decreasing pH than labradorite (Liu et al. 2006, Rosso and Rimstidt 2000, Palandri and Kharaka 2004). Hypersthene, the source of 36% of Mg in Duluth Complex samples (Table 23), did not have rate equation information for comparison. Bronzite, which is similar to hypersthene, is more sensitive to pH than labradorite and has a faster dissolution rate than labradorite at pH 4 (Palandri and Kharaka 2004). The pH dependence of biotite dissolution rates under acidic conditions is roughly the same as that for labradorite and andesine (Tables 73, 77). Labradorite was the main source of Ca for these samples.

6.3.5. Silicate Coatings

Silicate mineral surface layers are modified or coated more slowly than sulfides (Tables 14, 15). This contributes to an increase in Ca and Mg release relative to sulfate release in the latter part of period C and period D. This increase in relative release contributes to the rise in pH starting in the latter part of period C, and the dominant factor influencing the increase is the decline in iron sulfide oxidation rate. White and Brantley (2003) reported slow development of depleted layers in plagioclase, based on lower Na release in weathered granite than unweathered indicating slowdown in plagioclase dissolution. The authors estimated that it would take 22,000 years for a rate decrease of about two orders of magnitude. Substantial Si-rich leached layers have often been found in laboratory dissolution experiments, which are often conducted in acidic solutions without Na, K, Mg, Ca or Al. Nesbitt et al. (1991) reported that in conditions in which cations exceeded the amount of H_3O^+ in solution, as they did in this experiment, labradorite dissolution was congruent and leached layers were small. In samples like the ones in this report, which had greater amounts of Fe from sulfide minerals than the granite samples of White and Brantley (2003), silicate coatings may form faster due to the presence of Fe^{3+} on mineral surfaces (Brantley 2003).

There was evidence of ferromagnesian mineral coating development based on Mg release. The greater decline in release of Mg relative to Ca indicated that olivine or hypersthene was developing coatings. If augite, which had roughly equal amounts of Ca and Mg, were coated, then a drop in Ca would also be expected. However, Ca release remained similar in periods B and D. The importance of olivine as a source of Mg is shown by the fact that the only samples with declining Mg release from period B to period C were the two with no olivine (0.40 and 0.54%S). The increased reaction rate as pH declined in period C could cause formation of coatings on olivine. Rates of fayalite dissolution have been reported to decrease with time, possibly due to formation of iron oxyhydroxide phases (Siever and Woodford 1979, Wogelius and Walther 1992). Welch and Banfield (2002) suggested that adsorption of ferric iron and formation of a thin "laihunite-like surface structure" might be responsible for the observed rate inhibition. Laihunite structures were reported present in field-altered olivine (Banfield et al. 1990, 1992). The presence of iron in olivine or the presence of other iron-bearing minerals might have contributed to slower rates of olivine (Fo_{57}) dissolution from a mixture of minerals (65% olivine (Fo_{57}), 20% plagioclase (An_{50}), 8% magnetite, 4% pyroxene, 3% serpentine) reported by Duro et. al (2005).

6.4. Variation with Sulfur Content

6.4.1. Acid Generation

A major concern associated with mining of the Duluth Complex is acidification of surrounding ground and surface water. The main source of acid from mine wastes is sulfide oxidation (Younger et al. 2002), and virtually all S in these unleached samples was present as sulfide. Solids with sulfur contents ≥ 0.4% produced drainage pH values below 6.0, and those with sulfur contents less than 0.22% produced typical drainage pH above 6.0. These values indicate an empirically demonstrated "acidification cutoff" of 0.22% S. The "critical sulfur content" is actually somewhere between 0.22 and 0.40 percent S but more

refined definition of the value is limited by the absence of tested solids with sulfur contents in this range. Given the uncertainty in quantifying the critical sulfur cutoff more precisely, the empirically demonstrated value (0.22 %S) would be applied as a conservative value for mine waste management decisions.

Furthermore, while the 0.4%S sample generally maintained drainage pH above 6.0 for ~750 weeks, the rate of acid neutralization (reflected by the sum of calcium and magnesium release rates) by this sample was anomalously high relative to its sulfur content (Table 84). The high neutralization was due to the presence of 0.64% calcite (Table 2), which is rare in Duluth Complex rock. Thus, without elevated acid neutralization values, the sample would have acidified earlier than week 750. To illustrate this point, drainage pH from the 0.41%S sample fell below 6.0 after 75 weeks and generated a minimum drainage pH of 4.05 despite rates of sulfate release which were lower than those from the 0.40% S sample. Again, it should be noted that no samples were tested with sulfur content 0.22<%S<0.40. Samples in this range of sulfur content have been observed in other Duluth Complex laboratory experiments (Lapakko 2013b) to approach acidic conditions (pH<6). These data could affect the mine waste management sulfur groupings discussed in this report. Additional testing of samples with sulfur contents in the range of 0.2 to 0.4% could be conducted to refine this critical value.

Sulfide mineral oxidation was the source of acidity in these samples, as confirmed by the significant positive regressions between sulfate release (average and maximum) and %S in periods A-C (Table 34, Figures 24-27), as well as the negative regression relationship between pH and sulfate release in all periods (Table 34, Figures 28-31). This increase in sulfide oxidation drove the significant negative regressions of pH (minimum and average) versus %S in periods A-C (Table 87). The slopes of these regressions were essentially zero during period D (Figure 23). The decline of sulfate release for higher %S samples during period D (Figure 27) is likely due to coating development on sulfide minerals rather than sulfide mineral depletion. This is because the average sulfur depletion, calculated based on (1) sulfate release and (2) %S difference between leached and unleached, ranged from 54-68%. This leaves a large portion of sulfide minerals still potentially available for oxidation (Table 109).

Total sulfate release during the entire experiment differed among groups (Table 79). Total sulfate release for group I was 2.30-2.34 mmol (mean 2.32 mmol). Total sulfate release for Group II was 5.9-10 mmol (mean 8.2 mmol). Total sulfate release for group III was 11.3-28.9 mmol (mean 21.7 mmol). The large increases in sulfate release reflected the differences in pH minima among groups.

The type of sulfide minerals also affected sulfate release rate. Pyrrhotite and pentlandite are more rapid oxidizers than chalcopyrite (Kwong and Ferguson 1990, Ahonen and Tuovinen 1992), the other common sulfide mineral in these samples (Figures 58, 59). Pyrrhotite was by far the most common sulfide mineral in the samples (Table 7). This is supported by 1) the change in mineral content between unleached and leached samples (Table 9), 2) development of coatings, 3) higher sulfate release in solution compared to copper, and 4) fewer pyrrhotite grains were found in the leached samples (Table 15).

6.4.2. Acid Neutralization

6.4.2.1. Major Cation Release

Ultimately, pH is a reflection of the balance between acidity production from sulfide mineral oxidation and neutralization potential from dissolution of Ca and Mg-bearing silicate minerals. There is also a smaller amount of neutralization from Na and K cations. Higher sulfur content samples produced sharper initial declines in pH, as well as lower pH minima (Figure 5). Coinciding with the pH minima

were marked increases in SO_4 release rates. Ca and Mg release rates also rose sharply near the pH minimum, which could be attributed to augite dissolution, but these increases were generally less than that observed for SO_4. Average Ca and Mg release rates doubled going from period B to period C, compared to the 3-fold increase in SO_4 release during this minimum pH period (Tables 81, 84).

Cumulative group release of SO_4 and Ca increased as solid-phase sulfur content increased (Table 79). Average cumulative Mg release was slightly less from group II than group I, but Ca+Mg release increased with increasing group sulfur content. In Group I, total Ca release was 3.4-3.9 mmol (mean 3.7 mmol) and total Mg release was 2.0-3.3 mmol (mean 2.7 mmol). The total of Ca and Mg release exceeded total sulfate release, explaining the circumneutral behavior of group I samples. Groups II and III generally had combined Ca and Mg release that did not exceed sulfate release, explaining the samples' pH decline. The only exception was the 0.40%S sample, which had exceptional neutralization capacity relative to its sulfur content (discussed later in section). The small difference between total Ca+Mg release and total sulfate release could be explained by measurement error or errors associated with interpolation to account for missing sample points. In group II, total Ca release was 2.7-8.7 mmol (mean 4.4 mmol), and total Mg release was 0.9-2.6 mmol (mean 1.8 mmol). In group III, total Ca release was 3.0-6.1 mmol (mean 4.1 mmol) and total Mg release was 2.6-6.9 mmol (mean 5.0 mmol).

Calcite is often present in igneous rock (White et al. 1999), but was not an important long-term source of neutralization in these samples. It was detected by point count in only one sample (0.58%S sample). Using carbon dioxide as a surrogate for calcite, the mineral was below detection limits in eight of twenty-one samples. The rapid dissolution of the calcite that was present likely contributed to the high initial Ca release. The potential importance of calcite is shown by the 0.40%S sample, which had the highest calcite (measured as CO_2, Table 2) and took 750 weeks before a sustained period below pH 6. It is possible that additional calcite formed during the experiment, as implied by the increase in CO_2 for two leached samples. However, this could also be explained by absorption of CO_2 into mineral lattices (Oberheuser et al. 1983) or by errors in sample analysis.

Because of the small amount of calcite, silicate mineral dissolution must have generated the majority of neutralization potential in most samples. However, despite the large percentage of samples occurring as silicate minerals, the ability of silicates to maintain a circumneutral pH is limited by a slow dissolution rate. The small percentage change in silicate mineral point counts after leaching, as well as model dissolution rates that were consistently slower than sulfide mineral oxidation rates, are consistent with the slow rate of silicate mineral dissolution.

Another limit on silicate minerals preventing acidification is that pyrrhotite, the most common mineral, oxidizes much faster at low pH when ferric iron is the oxidant (Janzen 1996), while most silicate minerals are less responsive to pH (Phillips et al. 2001). Rates of plagioclase dissolution in this experiment showed this insensitivity to pH, as indicated by the similarity in Ca release rates from period B through period D (Table 82). As with modeled sulfide mineral oxidation rates, silicate dissolution computed in this experiment was slower than most rates reported elsewhere. Some possible reasons for slower rates include, saturated fluids that became trapped in micro-pits that develop on the grain surface over time, formation of preferential flow paths through the sample, and wet-dry cycles causing intermittent periods of no dissolution (White and Brantley 2003). Additionally, the formation of leached layer or coatings, as discussed earlier, could slow silicate dissolution.

Silicate dissolution can be important in maintaining neutral conditions of low sulfur samples, as shown by an examination of cumulative mass release (Table 79). Each mole of Ca and Mg released by silicate dissolution can neutralize two moles of acidity, and each mole of sulfate released by sulfide oxidation produces two moles of acidity. Table 79 shows group I samples, which stayed above pH 6, had greater

neutralization, indicated by Ca and Mg release, than acidity, indicated by sulfate release. Groups II and III had greater acidity than potential neutralization, explaining their decline in pH.

At the end of the period of record for all samples, there is still a large amount of untapped neutralization potential in silicate minerals. Using Ca release to determine maximal dissolution of plagioclase revealed that the highest potential dissolution of any sample was 16.3 mmol (Table 99), while the average abundance of plagioclase in each sample was 140 mmol, assuming 50% labradorite. Similarly, using Mg release to determine maximal dissolution of olivine revealed that the highest potential dissolution of any sample was 9.6 mmol (Table 99), while the average abundance of olivine in each sample was 49.7 mmol, assuming 11% olivine. It should be noted that olivine is not the only source of Mg release (i.e. augite, hypersthene) therefore the rate of maximum olivine dissolution calculated is probably an overestimation. When compared to the over-50% sulfide mineral depletion (Table 109), it is clear that sulfide mineral depletion occurred much faster than silicate mineral depletion.

In addition to neutralization by dissolution of the silicate minerals initially present, neutralization by some of their solid-phase reaction products must also be considered. Plagioclase dissolves to kaolinite, which dissolves to gibbsite. Gibbsite dissolution neutralizes six moles of acidity, and its dissolution rate increases at pH 4 due to increased solubility of Al. After determining maximal rates of gibbsite formation at pH $\geq$ 4 (Table 68), the potential amount of gibbsite was more than enough to offset all acidity production in the sample. There is a marked increase in Al release rates beginning near pH 4 for the Dunka blast hole samples (Table 88, Figure 53). It has been reported that kaolinite dissolution rates are fast enough to avoid affecting plagioclase dissolution rates at 22ºC and pH 4 (Yang and Steefel 2008). This relationship will likely be unchanged by pH in the acidic range, as the slope of log kaolinite dissolution rate versus pH is –0.09 from pH 1-6 at 25ºC (Nagy 1995). Reported kaolinite dissolution rates of approximately 3×10^{-12} mol m^{-2} s^{-1} (Nagy 1995) are similar to the rates for plagioclase dissolution found in this experiment.

Sodium and K release was not analyzed as frequently as other ions, but it indicates the potential neutralization from silicate mineral dissolution. For the six samples that had Na and K release rates for the entire period, K was higher initially and declined through time, while Na remained essentially constant. This indicates that K ion exchange dissolution may be a more important source of acid neutralization initially. Similar results have been found for Archean Greenstone in Minnesota (Lapakko et al. 2004b).

6.4.2.2. Neutralization Potentials

Offsetting the acidity generation, neutralization can come from dissolution of the calcium and magnesium fractions of carbonate minerals and silicate minerals. Calcite ($CaCO_3$) content of the rock samples tested, as indicated by CO_2 generated from unleached samples, was generally low. In all Duluth Complex samples except 0.4%S, the amount of Ca released in leachate prior to rapid pH decline was in excess of the sample's $CaCO_3$ content, indicating that calcite was likely depleted early in these experiments.

Of the methods used in this report to measure neutralization potential, the empirical neutralization potential (ENP) gives the most accurate measurement of sample neutralization potential. The ENP measures the amount of acid a sample can neutralize before drainage decreases below a certain pH. The ENP_{pH6} for all but one sample did not exceed 2 mg $CaCO_3$/g rock (Table 21). This is in good agreement with the calcium plus magnesium carbonate content implied by the CO_2 content of the rock samples [$NP(CO_2)$]. All but two of these values were less than 2.5 mg $CaCO_3$/g rock (Table 2). The extent of neutralization observed during the dissolution tests reflects a very low capacity for acid neutralization,

which is consistent with the lack of carbonate minerals present and the low values of all NP measurements except those using the Sobek (1978) method (Tables 21, 22). The exception was the 0.4%S sample which had an ENP of 8 mg $CaCO_3$/g rock and an NP(CO_2) of 14 mg $CaCO_3$/g rock.

The pH of drainage from the group I samples (0.18, 0.22%S) rarely dropped below 6.0. The neutralization potentials above pH 6 for these solids were approximately 3 mg $CaCO_3$/g rock. The neutralization above pH 6 by these solids was at the upper end of the range observed for most samples. This was most likely due to the slow rate of acid generation by these low-sulfur solids, and the consequent low rate of silicate mineral dissolution required to neutralize the acid produced. That is, silicate mineral dissolution can neutralize acid to maintain circumneutral pH if and only if the rate of acid production is very slow.

Older methods of determining neutralization potential (NP), such as Sobek et al. (1978), overestimate the amount of NP present as calcium and magnesium carbonate (Lapakko 1994a). At circumneutral pH, carbonates dissolve faster than silicates and are more important in maintaining pH above 6 when the rate of acid production (iron sulfide mineral oxidation) is rapid. Although the rate of silicate mineral dissolution increases at lower pH, it is still very slow relative to rates of calcium and magnesium carbonate dissolution. More realistic estimations of the ability to maintain pH above 6 are given by the autotitrator NP and the NP calculated from CO_2 content (Figure 61).

6.5. Metal Release

Another concern associated with mining is metal contamination through leaching from mine wastes. The main metals of concern in the Duluth Complex are Co, Cu, Ni and Zn. Acidity from sulfide mineral oxidation was the controlling factor in metal concentrations of sample leachate, and all metal release rates increased with declining pH. Although metal release increased with lower pH, there were differences in temporal variation among metals. These differences may be due to differences in mineral sources of each metal and differences in mineral abundance among groups.

6.5.1. Mineral Sources of Metals

Likely mineral sources of Co and Ni were determined using calculations based on solid and leachate chemistry, as well as mineral abundance. These metals had multiple potential mineral sources, unlike Cu and Zn. Copper was likely from chalcopyrite and cubanite, while Zn was likely from sphalerite (McSwiggen 1999). Cobaltite, (Co, Fe)AsS, is a common mineral source of Co. However, As in unleached solids is quite low for Duluth Complex samples (Table 3), making it impossible for the majority of Co to be accounted for by cobaltite. A similar situation exists for Ni in maucherite ($Ni_{11}As_8$). Pentlandite was inferred to be the major mineral source for both of these metals.

Although cobaltite and maucherite could not account for the majority of Co and Ni present in these samples, dissolution of these minerals could have detrimental effects on the environment through arsenic release. No As was detected in these experiments, but waste rock piles or tailings basins with small amounts of cobaltite or maucherite could release significant amounts of As due to their large amounts of rock. Arsenic release at this scale could be potentially problematic.

6.5.1.1. Cobalt

Less than 20% of Co could be present as cobaltite, due to the amount of As present in solid samples. Pentlandite in these samples was common, and has been reported to have 1-1.4% Co by weight in the Duluth Complex (McSwiggen 1999). Other researchers have found pentlandite to occur commonly in

the Duluth Complex, whereas cobaltite has not been found to be widespread in large amounts (Severson and Barnes 1991, Theriault et al. 2000).

6.5.1.2. Copper

Chalcopyrite and cubanite are the dominant sources of Cu in these samples. This is typical of Duluth Complex samples and these minerals are commonly found and widespread in the Duluth Complex (Stevenson et al. 1979).

6.5.1.3. Nickel

Pentlandite is common in the Duluth Complex (Severson and Barnes 1991, Theriault et al. 2000), and it has been reported to be the primary Ni ore mineral in the Duluth Complex (Severson and Barnes 1991). Furthermore, they indicate that most Ni in sulfides is held in pentlandite. Olivine is a relatively minor contributor of Ni (between 5-27%), except in samples with low sulfur content. Analysis of a subset of samples with low sulfur content allowed estimation of a maximal rate of Ni release from olivine. Maucherite was determined to be of minor potential importance due to the low amounts of As in the samples.

6.5.1.4. Zinc

Zinc was likely to have come primarily from sphalerite ((Zn, Fe)S). This mineral has been reported in minor amounts in the Duluth Complex (Stevenson et al. 1979, Theriault et al. 2000) and has been reported to be locally common (Severson and Barnes 1991). There could also be trace amounts of Zn contained in pyrrhotite.

6.5.2. Drainage Quality

6.5.2.1. Changes in Release with pH

Release of Co, Ni and Zn increased concurrently with sulfate release, and both metal and sulfate releases occurred during declines in pH. Copper release also increased as pH declined, but its increase lagged behind that of Co, Ni and Zn and had a more prolonged peak. Increased metal release was mainly due to greater sulfide oxidation and increased solubility of metals at lower pH, typically in the order Cu>Ni>Zn>Co. There was no strong relationship between metal content and S content of the rock samples (Table 3, Section 5.1.2.1), but there was a significant correlation between metal release and leachate pH (Figures 48-51). The trend was reflected by increasing metal release from low-sulfur samples in group I to higher-sulfur samples in groups II and III. That is, metal release from the higher-sulfur samples was a result of their lower pH rather than higher metal content.

6.5.2.2. Changes in Release over Time

Group I samples had similar temporal patterns for all metals (Tables 89-92). Release was highest in period A and declined in period B. Declines in group I average rates decreased in the order Zn<Cu<Ni<Co by 17, 44, 79, 85%, respectively. Drainage pH for these samples stabilized near 6.0 and, therefore, there was no increased metal release under acidic conditions. The decline in metal concentrations over time is consistent with the observed decline in sulfate concentrations. Both trends are assumed to be the result of decreasing amounts of sulfide minerals (and associated metals) available for leaching and coating of sulfide mineral surfaces (Appendix 1, Attachment A1.2).

Within the Group II samples the temporal release patterns for Cu were different from Co, Ni, and Zn (Tables 89-92). Cobalt, Ni, and Zn release rates declined in period B, whereas Cu releases increased steadily from period A through period C. The similarity in Co and Ni release are likely because both are largely released from pentlandite. Although cobalt was not reported in the mineralogical analysis, McSwiggen (1999) found pentlandite to have 1-1.5% Co by weight, making it a major source of Co in the Minnamax Cu-Ni sulfide deposit of the Duluth Complex. Copper is present in chalcopyrite, which oxidizes slower than pentlandite (Shaw et al. 1998, Jambor et al. 2005), perhaps contributing to the lag relative to Co and Ni.

Group III samples had the highest average release for all trace metals during period C, which corresponds to when minimum pH values were observed. These samples also had a similar temporal release pattern for Co, Ni, and Zn which differed from that of Cu (Tables 89-92). Average cobalt and nickel release rates decreased through all periods. Copper release increased through period C, then decreased, while Zn release peaked in period C and decreased.

Iron and Al had increased release rates below pH 4 (Table 88, Figures 52, 53). This was particularly evident with Al, which had an average release rate 18 times faster below pH 4 than at higher pH. The increased solubility of Al may indicate gibbsite dissolution. Although increased Al release from aluminosilicate minerals generally would be expected at lower pH, White and Brantley (2003) attributed increased Al release from plagioclase at lower pH to dissolution of secondary clays.

Cu and Zn release was entirely the result of sulfide oxidation. Nickel, and to a lesser extent cobalt, release could result from both sulfide mineral oxidation and olivine dissolution. The first evidence that sulfide minerals are the major source of nickel is the calculation that olivine only contains an average of 16% of Ni in the Dunka blast hole samples. Eliminating samples with an elevated fraction of Ni present as olivine (i.e. 0.57%S), this average decreases to 12% (Table 93). Second, peaks in Ni release coincided more often with sulfate than Mg release. Third, when Ni/Mg release ratios were analyzed, they were, on average, 60-times higher than the ratio found in olivine (Table 94). It should be noted, however, that samples with lower sulfate release rates, and consequently lower nickel release from sulfide minerals (0.18, 0.22%S) exhibited Ni/Mg release ratios more similar to Ni/Mg ratios in olivine (aqueous average 4 times higher than ol ratio). Fourth, Ni/Mg release ratios were generally similar between samples with olivine and those without (0.40 and 0.54%S), indicating limited importance of olivine as a Ni source in these samples.

Regression of calculated Ni release from olivine versus %S (Table 96, Figure 62) gives a positive correlation with higher sulfur samples at lower pH ranges. The increase in calculated release from olivine is based on increased Mg release. The increased Mg release likely resulted from increasing Mg-bearing mineral dissolution due to lower pH resulting from increased sulfide mineral oxidation rates. Despite the overestimation of olivine dissolution (Section 4.4.6), calculated Ni release from olivine dissolution was far lower than actual Ni release. This discrepancy between the calculation of maximal olivine dissolution and observed Ni release further emphasizes the importance of sulfide minerals as a Ni source. Both Ni and Mg release rates decrease from period A to period C, but Mg release declines faster than Ni. In period D, Mg release increases relative to Ni (Table 95), indicating a likely combination of sulfide mineral depletion and coatings on sulfide grains inhibiting oxidation.

The molar ratios of nickel to sulfur in the solids averaged 0.05 with a standard deviation of 0.03 (omitting the Group I samples). In contrast, the molar ratios of observed nickel to sulfur release averaged 0.09 with a standard deviation of 0.13. This comparison indicates that sulfur was released preferentially to nickel. That is, the ratio of sulfur release to the amount of sulfur in the solid exceeded the corresponding ratio for nickel. One factor contributing to this differential was the presence of some

nickel in olivine (5-27%). A second factor is that pyrrhotite, the dominant sulfide mineral present in the samples, oxidizes more rapidly than pentlandite (Shaw et al. 1998).

Analysis of a subset of samples gave an estimate of maximal Ni release resulting from olivine dissolution. It was determined that low contribution of Ni from sulfide minerals was the main factor that increased agreement between calculated Ni from olivine and observed Ni release. The 0.57 %S sample had a very high percentage of total Ni (>72%) and Mg (>55%) in olivine. The predicted maximum Ni release in the Mg-based model was only 8 percent of the actual Ni release during period B (Table 95). The group I samples (0.18, 0.22%S) had calculated maximum Ni release rates that were 40% of actual release during period B (Table 95), despite variations in percent Ni and Mg from olivine (5-19% Ni, 14-21% Mg from olivine). Limited contribution of Ni from sulfide minerals, indicated by circumneutral pH in these samples, is most likely the controlling factor to the larger relative contributions of Ni from olivine.

Calculated maximum rates of olivine dissolution for group I samples during time period B approximated maximum olivine contribution to Ni release best for these samples. Average observed Ni release rates for group I samples was 1.1×10^{-8} mol Ni · week^{-1}(Table 95) or 3.9×10^{-7} mol Ni ·mol ol^{-1} · week^{-1} (Table 97). During period B, these samples were at steady state with neutral pH, and thus had minimal contributions of Ni from sulfide mineral oxidation. However, these calculations still disregard the likelihood that some Mg came from other sources, such as augite dissolution. Even with these overestimations, calculations still underestimated Ni release by a factor of three, indicating that there are significant sources of Ni from other minerals. Other data, including much higher Ni:Mg in the aqueous phase than in olivine, as well as increasing relative amounts of Ni with higher %S samples, reinforce the importance of sulfide minerals as the major source of Ni release. Thus, it was concluded that most Ni is released from sulfides, specifically pentlandite.

6.6. Sulfide Oxidation Rates

6.6.1. Data Summary

Calculated average pyrrhotite oxidation rates typically ranged from 1×10^{-10} to 3×10^{-10} mol m^{-2} s^{-1} during periods A and B, (Table 107), when drainage pH was typically above 4.5 and almost always above 4.0 (Figures 20, 21). Groups II and III had the highest sulfide oxidation rates in period C, with average rates typically ranging from 3.5×10^{-10} to 6×10^{-10} mol m^{-2} s^{-1}, and were coincident with pH minima typically in the threes (Figure 22). Rates from these two groups declined in period D due to both sulfide mineral depletion and coating of sulfide mineral surfaces. Bulk sulfide oxidation rates were slightly lower, a result of the larger surface area contributing to sulfate release for this calculation (Table 108).

Some systematic variation of rates among the samples was observed. The pyrrhotite oxidation rates reported for the 0.22%S sample were about an order of magnitude higher than the other samples (Table 107), and this is attributed to the low amount of pyrrhotite calculated for this sample. The assumptions that 1) copper was equally divided between chalcopyrite and cubanite (as opposed to entirely present in chalcopyrite) and 2) all nickel was present as pentlandite (ignoring Ni in olivine), would lend to underestimation of pyrrhotite content. Such error would be more pronounced for samples of low sulfur content. Consistent with this concept, the 0.18%S sample was calculated to have a pyrrhotite content less than zero.

The accelerated rates observed for some samples may be due to greater available sulfide surface areas and/or chemical or mineralogical differences for these samples. Some accelerated samples, such as 0.51

and 1.40%S, may have higher sulfide surface area due to subtle differences in sulfide grain size and/or "roughness" of the mineral surfaces. Other accelerated samples, such as 0.40 and 1.12%S, may oxidize faster due to lower copper content than other samples (Table 3). This difference, and/or its implication of a difference in chalcopyrite content, may influence the sulfide oxidation rate for these samples.

It is possible that under certain conditions copper has an inhibitive effect on iron sulfide oxidation. For example, Koch (1975) reported that the presence of copper in a sulfide such as chalcopyrite stabilizes the ferric iron present. It is also possible that copper released from chalcopyrite may have participated in an exchange reaction with iron present in pyrrhotite and, consequently, inhibited its oxidation. Similar reactions have been reported in which metals in solution exchange with sulfide-bound metals of higher solubility (Gaudin et al., 1959). If galvanic interactions of chalcopyrite and pyrrhotite contributed significantly to pyrrhotite oxidation (Ahonen et al., 1986; Natarajan et al., 1982), lower than normal oxidation rates might be expected for these samples.

These rates were generally within or at the lower range of reported rates for pyrrhotite oxidation (Figure 67), and direct comparisons of rates must consider experimental variables such as pH, temperature, and presence of bacteria (Table 72). Janzen et al. (2000) reported rates for oxidation of 12 pyrrhotite samples by oxygen at pH 2.75 and 22°C. The rates determined based on sulfate release varied by about a factor of 70, ranging from 8.15 x 10^{-12} to 5.86 x 10^{-10} mol m^{-2} s^{-1} and averaging 2.0 x 10^{-10}. Kwong (1995) reported a rate above this range, 8.5 x 10^{-10} mol m^{-2} s^{-1}, under the same pH and temperature conditions. Within this range Steger (1982) reported a rate of 6.5 x 10^{-10} mol m^{-2} s^{-1}, for air oxidation of pyrrhotite at 62% relative humidity and 28°C. Rates determined by Janzen et al. (2000) based on iron release were one to two orders of magnitude higher that those based on sulfate release. The associated ratio for rates determined by Kwong (1995) and Steger (1982) was almost always less than 10.

It should be noted that the 8 x 10^{-10} mol m^{-2} s^{-1} rate reported for Lapakko and Antonson (1994) in Table 72 was based on the first 150 weeks of this experiment. The rate was calculated for the 0.71%S sample and used different assumptions for calculating the amount of pyrrhotite present (yielding a value of 0.9 g rather than 0.79 g by methods in the present report) and a pyrrhotite roughness factor of 2.6 rather than 10. Using assumptions from the present report reduces the rate reported by Lapakko and Antonson (1994) to 2 x 10^{-10} mol m^{-2} s^{-1}, within the range for periods A and B in the present report (Table 107).

6.6.2. Factors Contributing to Lower Rates

Reaction rates for individual sulfide minerals were determined under ideal conditions for oxidation in the laboratory. The design of this experiment, in which samples were dry for approximately half of each week, may have reduced rates of oxidation directly, or indirectly, through evaporation creating saturated films at the mineral grain surface that inhibited dissolution. This evaporative process might also favor formation of coatings on the mineral grain that would resist dissolution. Effects of wetting and drying cycles are poorly understood, but Blum and Stillings (1995) state that it might make grain surfaces less reactive. The proposed mechanism for lower reactivity is initiated by dehydration of the surface, which destabilizes surface species. This leads to condensation reactions between surface groups, making the surface less reactive.

The length of the experiment might have reduced rates relative to the typically short experimental periods of most experiments by giving more time for leached layers to form on the mineral grains, thus slowing rates. This mechanism was clearly reflected by the low rates observed in period D (Table 107). Oxygen bonds with Fe^{III} at vacant sites on the pyrrhotite surface, creating an Fe^{III}-O rich layer above a S-depleted layer. As the Fe^{III}-O layer grows, electron transfer will be inhibited and oxidation will slow (Leopold and Lapakko 2001).

Coatings can also inhibit oxidation over time. Newly ground sulfide minerals have many fracture sites that are preferentially oxidized (Janzen et al. 1997). As time since grinding increases, iron oxyhydroxides form on these grains, reducing surface area dramatically and, thus, reducing reaction rates (Janzen et al. 2000). Janzen reported an 85-90% reduction in surface area (section 6.2).

Mineral oxidation, which increases surface area, is often assumed to speed up dissolution. However, microscopic features may trap water that is saturated with reaction products, rendering less of the surface area available to oxidation (Drever and Clow 1995, White and Brantley 2003). Over time, the formation of preferential flow paths through a sample may occur, leaving some mineral grains dry during the weekly water additions. Preferential flow paths are also thought to occur in soils, potentially isolating some mineral grains from oxidation (Blum and Stillings 1995).

High trace metal contents may also slow oxidation (Plumlee 1999), although others have found no effect (Janzen 1996). Finally, the fact that these were mixed mineral samples, as opposed to the relatively pure samples found in most experiments, presents the possibility of interactions with ions released from other minerals affecting oxidation rates.

6.6.3. Rates for Different Sulfide Minerals

Rates of sulfide oxidation for these samples were expected to be relatively rapid due to the generally high percentage of sulfide minerals occurring as pyrrhotite. Pyrrhotite is reported to be more reactive than other sulfide minerals present in these samples (Jambor 1994, Jambor et al. 2005). Pyrrhotite reacted more rapidly in these samples also, as shown by SEM photomicrographs. Images of individual samples showed that pyrrhotite grains were more abundant in unleached samples and far less abundant in leached samples (Appendix 1, Attachment A1.2). Additionally, pyrrhotite was approximately two-thirds of sulfide minerals in unleached samples and less than 10% of leached samples (Tables 7, 10). The few pyrrhotite grains that were found in leached samples were generally highly oxidized. Other sulfide mineral grains, such as chalcopyrite, found in the samples showed a generally small amount of oxidation, indicating a much slower reaction rate (Table 15).

Kwong and Ferguson (1990) supported this difference in oxidation rate, reporting that pyrrhotite in an abandoned Vancouver mine oxidized faster than chalcopyrite, which oxidized faster than pyrite. Pyrrhotite has also been found to oxidize faster than pentlandite (Ahonen and Tuovinen 1992). Jambor et al. (2005) ranked sulfide oxidation rates as pyrrhotite > pentlandite > pyrite > chalcopyrite. Data from this experiment support this ranking, with average sulfide mineral percent depletion decreasing in the order Po>Pn>Cp with values of 64, 40, and 10% depletion, respectively (Table 9).

It should be noted that pyrite was ignored as a potential source of sulfate release even thought it might have been present in small amounts. Pyrite was measured by visual point counts and, with the number of grains measured (~100), the 2-σ error level (86.5% confidence) of this method makes it possible that the pyrite (found in only one leached sample) could have over 100% error (Figure 60). Additionally, pyrite may have been present in other samples but not detected. Low percentage mineral abundances determined by point count are susceptible to large percentage differences between what was counted and what was actually present.

6.7. Silicate Mineral Dissolution Rates

6.7.1. Literature Review

The methods typically used by researchers to determine mineral dissolution rates are likely to find rates on the high end of a sample's potential range. Most experimentally determined dissolution rates used a homogenous mineral sample housed in a flow-through reactor that did not allow for buildup of dissolved elements or drying of grains. Also, samples were typically analyzed for a far shorter period of time than this experiment, reducing the likelihood of coatings forming on mineral grains and slowing dissolution.

6.7.1.1. Plagioclase

Labradorite and andesine are the most common forms of plagioclase in Duluth Complex rock, and the following discussion focuses on these minerals. Labradorite had experimental equations that gave rates ranging from 1.2×10^{-11} to 1.9×10^{-10} mol m^{-2} s^{-1} at pH 4 (Table 73). At pH 6, rates were from 1.2×10^{-12} to 1.9×10^{-11} mol m^{-2} s^{-1}. Andesine had experimental equations that gave rates ranging from 7.9×10^{-12} to 1.0×10^{-11} mol m^{-2} s^{-1} at pH 4. At pH 6, rates were about an order of magnitude slower (Table 73).

Labradorite rate equations presented by Sjöberg (1989), Sverdrup (1990), and Blum and Stillings (1995), yielded rates near 1×10^{-11} mol m^{-2} s^{-1} and 1.0×10^{-12} mol m^{-2} s^{-1} at pH 4 and 6, respectively (Table 73). Rates calculated from data presented by Welch and Ullman (1993) were about an order of magnitude higher at both pH values. In comparison, rates derived from the acid rate equation of Palandri and Kharaka (2004) were roughly three times higher than Blum and Stillings' rate at pH 4 and slightly higher at pH 6. The rate derived from the neutral rate equation of Palandri and Kharaka (2004) was six times higher at pH 6 than the rate using their acid rate equation. Palandri and Kharaka's (2004) acid and neutral equations were calculated to be equal at pH 4.83 for labradorite and pH 4.80 for andesine. Rate equations reported for andesine yielded rates that were roughly 20 to 80 percent of those for labradorite (Table 73).

Plagioclase dissolution rates are dependent on pH and rates can be inhibited by the presence of certain ions in solution. Al^{3+} has been found to be particularly effective in retarding dissolution of plagioclase. Chou and Wollast (1985) found that Al inhibited dissolution of albite at pH 3 with concentrations below 10^{-5} mol/l, while Na and Si had little effect. This inhibition may increase with increased temperature (Chen and Brantley 1997), but Oelkers (2001) disputes this. Nesbitt et al. (1991) believe that Al is more effective than other cations at inhibiting dissolution of labradorite. At typical environmental temperatures, significant Al inhibition is limited to low pH conditions. Chen and Brantley (1997) did not find important Al inhibition at pH 4.5 until temperatures reached 70ºC.

Although coatings can develop on mineral grains, McSwiggen's studies indicate that these develop slowly on plagioclase feldspars (Appendix 1, Attachment A1.2). The literature rates are generally faster than the calculated modeled dissolution rates during experimental leaching of Duluth Complex rock after pH declines below 6. Studies have been conducted to examine the potential of surface precipitates or "leached layers" to develop as plagioclase dissolution progresses and inhibit rates of dissolution. Nugent et al. (1998) observed thin (0.005-1 µm), discontinuous coatings of amorphous and crystalline aluminosilicate forming on the surface of albite during early stages of weathering in acidic soils. They suggested formation of such coatings might inhibit dissolution in the field relative to that in the laboratory.

Schweda et al. (1997) used SIMS to examine the near-surface composition of labradorite leached for 500 to 1200 hours in HCl solutions acidified to pH 1-3. They reported Na, Ca, and Al in the outer 100

nm were completely depleted after 1200 hours of leaching at pH 3. The depth of depletion increased with increasing reaction time and decreasing pH. However, no substantial decrease in dissolution rate resulted from the development of the leached layer.

Hellmann et al. (2003) further examined samples generated by Schweda et al. (1997). The authors concluded the leached layer, or altered zone, at the mineral surface was formed by dissolution and subsequent precipitation. They speculated the precipitate was probably a very permeable silica gel that should not serve to inhibit the dissolution rate.

The aforementioned studies were conducted on isolated plagioclase phases. It is possible that precipitates resulting from dissolution of other minerals present in rocks might inhibit plagioclase dissolution. Hodson (2003) reported that an amorphous iron-rich layer formed on anorthite dissolved in a 0.001 molar $FeCl_3$ solution at pH 2.6 for 250 days, and that the coating did not inhibit the anorthite dissolution rate.

Ganor et al. (2005) conducted flow-through reactor experiments at pH 1 on a bulk granite sample and plagioclase, alkali feldspar, and biotite/chlorite fractions separated from the granite. After about 50 days the experiment on the bulk granite was stopped, the solids were dried at 50°C for approximately 50 days, and replaced into the reactor for additional dissolution. They reported that the rate of plagioclase dissolution decreased by roughly 85 percent as the iron coating increased from 0 to 1.5 x 10^{-5} moles per gram mineral. They noted that such coatings were more likely to develop under conditions in nature as opposed to those typically used in mineral dissolution experiments.

In summary, the rates of plagioclase do not appear to be inhibited by surface precipitates or leached layers formed during plagioclase dissolution. Iron precipitates have been observed to form on the surface of plagioclase minerals at low pH. The iron source in these experiments was either the leachate or other minerals present in the solid phase. Plagioclase dissolution was inhibited in one of the two cases.

6.7.1.2. Olivine

Most studies on olivine dissolution (Table 74) were conducted on forsterite, the Mg-rich end member of the solution series. Only one study examined fayalite, the Fe-rich end member, and one study examined a crushed rock sample of 65% olivine (Fo_{57}). Rates for this mixed sample were based on surface area for the entire sample, including non-olivine minerals, therefore literature results should be interpreted with caution.

Forsterite had experimental equations from seven papers at near 25 ºC, which gave rates ranging from 3.0 x 10^{-10} mol m^{-2} s^{-1} to 3.0 x 10^{-7} mol m^{-2} s^{-1} at pH 4 (Table 74). At pH 6, rates were from 3.0 x 10^{-11} mol m^{-2} s^{-1} to 9.8 x 10^{-8} mol m^{-2} s^{-1}. Palandri and Kharaka (2004) used the acid rate equation to derive rates of 1.9 x 10^{-9} and 2.1 x 10^{-10} mol m^{-2} s^{-1} for pH 4 and 6, respectively, at 25 ^{o}C. The rate derived from the neutral rate equation of Palandri and Kharaka (2004) at pH 6 was one-tenth the rate using their acid rate equation. Palandri and Kharaka's (2004) acid and neutral equations were calculated to be equal at pH 8.06 for forsterite.

The only acid rate equation for fayalite (Sverdrup 1990, also used by Palandri and Kharaka 2004) gives rates of 1.6 x 10^{-9} mol m^{-2} s^{-1} at pH 4 and 1.6x 10^{-11} mol m^{-2} s^{-1} at pH 6. The rate derived from the neutral rate equation of Palandri and Kharaka (2004) was two orders of magnitude slower at pH 6 than the rate using their acid rate equation. Palandri and Kharaka's (2004) acid and neutral equations were calculated to be equal at pH 8.0 for fayalite.

In addition to the effects of pH on olivine dissolution, solute ion concentrations can also affect dissolution. Oelkers (2001) found that forsterite dissolution was independent of dissolved Mg and Si concentrations at pH 2 from 25-65 ºC. Addition of aqueous Al did not affect dissolution rates at pH 3 and 65 ºC (Chen and Brantley 2000). Dissolution of fayalite was found to be incongruous for all major cations at 25 ºC and pH 4.5 (Siever and Woodford 1979).

There have been several papers on olivine dissolution since 1998 (Khisina et al. 1998; Awad et al. 2000; Chen and Brantley 2000; Pokrovsky and Schott 2000a, b; Rosso and Rimstidt 2000; Oelkers 2001; Zakaznova-Herzog et al. 2008). Forsterite dissolution initially proceeds with a relatively rapid preferential release of Mg^{2+} followed by congruent dissolution (e.g. Luce et al. 1972, Siever and Woodford 1979, Zakaznova-Herzog et al. 2008). The initial magnesium release results from reaction with H^+ that yields a magnesium deficient (silica enriched) surface layer (Pokrovsky and Schott 2000b). The magnesium is released as a result of H^+ attack on an oxygen atom bonded with two magnesium atoms and one silicon atom (e.g. Liu et al. 2006). This reaction occurs more readily on the (111) surface than the (223, line above second 2 and 3) surface (Awad et al. 2000). The rate of subsequent dissolution is controlled by a ligand exchange reaction that releases silica to solution (Liu et al. 2006). This reaction is controlled by the breaking of the Mg-OH bond that holds silica, a reaction in which a proton is present in the activated complex (Liu et al. 2006).

Olivine rates of dissolution in nature are complicated by the presence of iron in the olivine and by the presence of other minerals. Rates of fayalite dissolution have been reported to decrease with time, possibly due to the formation of iron oxyhydroxide phases (Siever and Woodford 1979, Wogelius and Walther 1992). Iishi et al. (1997) reported oxidative dissolution of iron-rich olivines (Fa_{100}-$Fa_{70}Fo_{30}$) at 300°C and 100 bar formed laihunite and hematite precipitate layers in alkaline environments and hematite and amorphous silica in acidic environments. Santelli et al. (2001) found that fayalite dissolution in the presence of *acidithiobacillus ferrooxidans* in the pH range of 2 to 4 was 2 to 50 percent of that under abiotic conditions. They concluded that the rate of acid neutralization by silicate hydrolysis reactions could be significantly reduced in the presence of microorganisms. Welch and Banfield (2002) suggested that adsorption of ferric iron and formation of a thin "laihunite-like surface structure" might be responsible for the observed rate inhibition. The laihunite structures were reported present in field-altered olivine (Banfield et al. 1990, 1992). More refined work indicated that, rather than laihunite-like layers, crystal structure itself demonstrated a high degree of oxidation and ordering in a distorted olivine structure (Janney and Banfield 1998). The presence of iron in the olivine or the presence of other iron-bearing minerals might have contributed to the slower rates of olivine (Fo_{57}) dissolution from a mixture of minerals (65% olivine (Fo_{57}), 20% plagioclase (An_{50}), 8% magnetite, 4% pyroxene, 3% serpentine) reported by Duro et. al (2005).

6.7.1.3. Pyroxene Group Silicates

6.7.1.3.1. Hypersthene (Bronzite)

Hypersthene is the orthopyroxene reported to be present in Dunka blast hole samples used for dissolution experiments. However, no references were found that specifically report hypersthene dissolution data. Both hypersthene and bronzite are members of the orthopyroxene solid solution series that extends from enstatite ($MgSiO_3$) to ferrosilite ($FeSiO_3$). Hypersthene is 50-70% Mg, while bronzite is 70-88% Mg (Klein and Hurlburt 1985). Consequently, bronzite is considered an adequate representation of hypersthene.

Bronzite had experimental equations from two papers, which gave rates ranging from 4.6×10^{-12} mol m^{-2} s^{-1} to 3.2×10^{-9} mol m^{-2} s^{-1} at pH 4. At pH 6, rates were from 1.4×10^{-12} mol m^{-2} s^{-1} to 3.2×10^{-10} mol m^{-}

^{2} s^{-1} (Table 75). Palandri and Kharaka's (2004) acid rate equation yields rates of 1.3 x 10^{-11} mol m^{-2} s^{-1} at pH 4 and 6.3 x 10^{-13} mol m^{-2} s^{-1} at pH 6. The neutral rate equation of Palandri and Kharaka (2004) yields a rate three times faster at pH 6 than the rate using their acid rate equation. Palandri and Kharaka's (2004) acid and neutral equations were calculated to be equal at pH 5.23.

At pH 4.5 and 5.5, dissolution of hypersthene was found to be incongruent at all times for all major cations in the solid, although the sample was noted to not be a pure phase (Siever and Woodford 1979). Incongruent dissolution of bronzite was found at pH 1, with Mg being released more slowly than Si (Schott and Berner 1985).

6.7.1.3.2. Augite

Augite ($(Ca,Na)(Mg,Fe,Al)(Si,Al)_2O_6$) is a clinopyroxene that is closely related to the diopside-hedenbergite series but with some substitution of Na for Ca, Al for Mg (or Fe^{2+}), and Al for Si. Sverdrup (1990) reported rates ranging from 3.2 x 10^{-11} mol m^{-2} s^{-1} at pH 4 to 1.3 x 10^{-12} mol m^{-2} s^{-1} at pH 6 (Table 76). Palandri and Kharaka's (2004) acid rate equation gave rates of 2.4 x 10^{-10} mol m^{-2} s^{-1} at pH 4 and 9.5 x 10^{-12} mol m^{-2} s^{-1} at pH 6. The neutral rate equation of Palandri and Kharaka (2004) yields a rate nine times slower at pH 6 than the rate using their acid rate equation. Palandri and Kharaka's (2004) acid and neutral equations were calculated to be equal at pH 7.36.

During dissolution, the mineral surface had increasingly lower Fe:Si as pH declined (White and Yee 1985). Calcium depletion relative to Si was reported for diopside (Schott et al. 1981). Dissolution has also been reported to increase diopside surface area from 860 cm^2 g^{-1} to 1300-1940 cm^2 g^{-1} after 3400 hours (25 ºC, inlet pH 1-3.5) (Chen and Brantley 1998).

6.7.1.4. Biotite

Biotite ($K(Mg,Fe)_3(AlSiO_3O_{10})(OH)_2$) is a phyllosilicate. Kalinowski and Schweda (1996) reported rates ranging from 1.1 x 10^{-12} mol m^{-2} s^{-1} at pH 4 (based on initial mineral surface area) to 1.6 x 10^{-13} mol m^{-2} s^{-1} at pH 6 (based on final mineral surface area) (Table 77). Palandri and Kharaka's (2004) acid rate equation gave rates of 1.1 x 10^{-12} mol m^{-2} s^{-1} at pH 4 and 9.5 x 10^{-14} mol m^{-2} s^{-1} at pH 6. The neutral rate equation of Palandri and Kharaka (2004) yields a rate three times faster at pH 6 than the rate using their acid rate equation. Palandri and Kharaka's (2004) acid and neutral equations were calculated to be equal at pH 5.11.

Biotite does not typically dissolve stoichiometrically in aqueous acidic solutions (Wilson 2004). In the pH range of 5-5.5, Mg is the only detectable cation released (Acker and Bricker 1992). At pH 3, increased Si and Al concentrations indicate significant decomposition of the tetrahedral sheet, while higher Mg and Fe concentrations show continued dissolution of the octahedral sheet (Wilson 2004). At pH 3, initial rates of cation release are highly nonstoichiometric, but Mg/Fe and Mg/Si ratios approached stoichiometric concentrations after an initial period of high release (Taylor et al. 2000). Edge surfaces dissolved more quickly than basal surfaces (Turpault and Trotignon 1994). Biotite is very susceptible to weathering and may progress rapidly to vermiculite and then to kaolinite. Associated with this progression is an increase in mineral surface area (Kalinowski and Schweda 1996).

6.7.2. Silicate Dissolution Modeling Methods

Four modeling approaches were used to determine silicate mineral dissolution rates, all of which assumed stoichiometric release of major cations (Ca, Mg, Na, K) from the most common silicate minerals in Dunka blast hole samples (plagioclase, augite, hypersthene, olivine, biotite) (see section

4.4.5.1 for methods). Model 1 assigned all observed calcium release to plagioclase dissolution and, therefore, yields a plagioclase dissolution rate that is typically higher than those determined by other models. Calcium concentrations were determined frequently throughout the experiment for all samples. Plagioclase was by far the most common mineral in these samples and contained about three fourths of calcium present (Table 23). Augite contained virtually all of the remaining calcium but its dissolution rate was determined only under acidic conditions when calcium and magnesium concentrations increased concurrently. Model 1 calculated dissolution rates of olivine, hypersthene, and biotite in separate calculations that assigned all magnesium release to dissolution of each individual mineral. Thus, Model 1 calculations tend to favor high dissolution rates for all minerals except augite.

Model 2 assigned all observed magnesium release to augite dissolution. Similar to calcium, magnesium concentrations were also determined frequently for all samples. Plagioclase dissolution rates were determined only when the ratio of calcium release to magnesium release exceeded their corresponding ratio in augite. That is, plagioclase dissolution was only detected by this model if calcium release exceeded that which could be attributed to augite dissolution quantified by magnesium release. Dissolution rates of olivine, hypersthene and biotite were calculated only when calcium release limited the rate of augite dissolution. Under this condition the magnesium release was greater than that possible based on observed calcium release and the assumption that augite dissolution was stoichiometric. When the augite rate calculation yielded excess magnesium, separate calculations assigned the excess to each individual mineral. Model 2 calculations tend to favor high augite dissolution rates but limit rates of dissolution for other minerals due to the assignment of large amounts of magnesium and calcium release to augite.

Model 3 assigned all observed sodium release to plagioclase dissolution which, assuming stoichiometric dissolution, inferred calcium release from plagioclase. The observed rate of calcium release almost always exceeded that attributed to plagioclase dissolution (Table 26, Appendix 7, Figures A7.1-A7.21), and the excess was assigned to augite dissolution. The calculated rate of augite dissolution inferred a consequent magnesium release rate. The difference between the rate of magnesium release observed and that inferred by the augite rate calculation was assigned in total to olivine, hypersthene and biotite in separate calculations.

Model 4 calculated plagioclase dissolution rates based on the rate of sodium release and the assumption that all sodium was released due to stoichiometric dissolution of plagioclase. Calcium release from plagioclase was determined based on the plagioclase dissolution rate and the difference between this release and that observed was attributed to augite dissolution. The difference between observed potassium release and that calculated assuming stoichiometric release of K and Na from plagioclase was attributed to biotite dissolution. Magnesium release not attributed to dissolution of augite or biotite was assigned in total to dissolution of olivine and hypersthene dissolution in separate calculations. Thus this model provided well-defined solute-based dissolution rates for plagioclase, augite, and biotite, with rates for olivine and hypersthene both based on the residual magnesium. Sodium and potassium concentrations were determined much later in the experiment than other parameters and for many of the samples measurements were not collected for the first 300-400 weeks of dissolution.

Due to the assumptions used in each model, certain silicate mineral dissolution rates were overestimated and in some cases (i.e. olivine and hypersthene), maximum possible dissolution rates were determined. Sources of error common to all elements and models include: assumptions about the mineral source of elements, surface roughness factors assigned to specific minerals, time of dissolution (reactors were dry for roughly half the week), as well as sampling error associated with determining mineral chemistry, content and mass. The models often overestimated mineral dissolution by assuming the entirety of the element examined comes from one mineral. The models also assume that all minerals are dissolving

stoichiometrically, which is not always the case. In addition, the rate of dissolution for each sample is based on an average mineral content, which was determined by microprobe analysis of 100 to 125 mineral grains. Errors resulting from the number of mineral grains examined in each sample (as explained in section 6.6) could have also lead to errors in determining mineral chemistry and mineral content of each sample, resulting in under or over estimates. These factors should be kept in mind when reviewing the following discussion section.

6.7.3. Comparison among Silicate Dissolution Models

Modeled silicate mineral dissolution rate statistics reported in Tables 100-104 were plotted for each mineral for comparative purposes amongst each model through periods A through D (Figures 68-72). Average model dissolution rates were plotted for each period, with the range representing the minimum and maximum values calculated. Average modeled rates in periods B-D ± a factor of 3 were graphed to show the central tendency. Cases where an absence of dissolution (zero) was calculated are designated with a red arrow. Average modeled rates were compared to the average period rates (top of figure) to show the range in dissolution rates between periods. At the bottom of each figure, the maximum to minimum rate ratios were reported to show the range in rates for each model. Also displayed at the bottom of the figures are the total number of rates (n) used in the statistics, as well as the percent of these values that are equal to zero (Percent of n=0), which represents an absence of dissolution. It should be noted that there were a number cases during periods B-D where more than 50% of the calculated rates used for the statistics were equal to zero, typically resulting in artificially low average dissolution rates. These data have been omitted from Figures 68-72 to improve resolution, but are reported in Table 105 for each silicate mineral.

Model 1 typically resulted in the highest plagioclase dissolution rates (Figure 68). This was due to the model assumption that all Ca was associated with plagioclase. For nonacidic periods (A, B, and D) the model assumed all Mg was available for olivine, hypersthene, and biotite dissolution, resulting in maximum possible dissolution rates of these three minerals. Because of this modeling assumption, olivine and hypersthene dissolution rates were typically the highest during model 1 as well (Figures 70, 71). Exceptions to this occurred during acidic period C where Mg and Ca (beyond that attributed to plagioclase dissolution, assuming a plagioclase dissolution rate equal to that in period B) were assumed to be released stoichiometrically from augite (Figure 69) and any residual Mg was then associated with olivine, hypersthene, and biotite (Figure 70-72). Another exception to this was biotite dissolution rates under non-acidic conditions from model 4 were higher than those from model 1 (discussed below).

Model 2 resulted in the highest augite dissolution rates (Figure 69). This was due to the fact that the model assumed all Mg release was associated with augite. Model 2 typically resulted in intermediate plagioclase dissolution rates, which were calculated based on residual Ca after augite dissolution (Figure 68). Similarly, dissolution rates of hypersthene and olivine, which were calculated based on residual Mg after augite dissolution, were also intermediate compared to models 1 and 4 (Figures 70, 71). Biotite dissolution rates for model 2 were also calculated based on the residual magnesium release rate after augite dissolution (Figure 72). These rates were typically the lowest calculated amongst the four models.

Model 3 typically resulted in intermediate dissolution rates for plagioclase that were higher than model 2, but lower than model 1 (Figure 68). This was due to the fact that the model assigned all Na release to plagioclase. It should be noted that plagioclase dissolution rates between models 1 and 3 where not equal. This indicates that the ratio of calcium to sodium release to solution differed from the stoichiometric ratio for the average plagioclase chemistry. Note that the plagioclase composition of a given sample was variable with respect to Ca/Na ratio. Thus it is conceivable, if not likely, that

anorthitic fractions of the plagioclase dissolved more rapidly. This would yield a Ca/Na release to solution higher than that indicated by the average plagioclase stoichiometry (Table 26, Appendix 7, Figures A7.1-A7.21). Augite dissolution rates, which were based on the difference between Ca release observed and stoichiometric release of Ca and Na from plagioclase, were typically lowest in model 3, although not markedly lower (Figure 69). Hypersthene, olivine, and biotite dissolution rates, which were based on residual Mg after augite dissolution, where typically higher than model 2, but lower than model 1 (Figures 70-72). In a few instances, model 3 results produced the lowest dissolution rates amongst the four models. These included lower plagioclase, olivine, and biotite dissolution rates during period A, and lower biotite dissolution rates during period B. These exceptions may be due to the limited amount of Na drainage quality measurements during the beginning of the experiment.

Model 4 generally resulted in the highest biotite dissolution rates (Figure 72). The model calculated the rate of biotite dissolution based on the difference between K release observed and stoichiometric release of K and Na from plagioclase. Biotite dissolution rates were generally higher in model 4 than model 1 due to K release typically being higher than Mg release during non-acidic periods A, B, and D (Table 59). Mg release during acidic period C was typically higher than K release, resulting in slightly higher calculated biotite dissolution rates in model 1 (Figure 72). Rates of biotite dissolution in model 4 could be higher due to Mg release from other sources (i.e. augite, olivine). Biotite dissolution may also be overestimated in model 4 due to the fact that some of the potassium would be attributed to K-feldspar, but is not partitioned in the model. Potassium feldspar dissolution rates were calculated in model 4 using the same method that was used to determine biotite dissolution rates. That is, all potassium release was attributed to potassium feldspar. Similarly to biotite, K-feldspar dissolution rates where fairly similar throughout the period of record, with a slight increase in dissolution rates during acidic period C (Table 104). Hypersthene and olivine dissolution rates were the lowest in model 4 (Figures 70, 71). This was due to the fact that residual Mg used for the dissolution rate calculations was significantly lower after factoring in Mg release resulting from biotite and augite dissolution. It should be noted that plagioclase and augite dissolution rates were the same as in model 3.

The degree of agreement among models was dependent on the specific mineral rates calculated. For this analysis a factor of 3 from the average rates in periods B, C, and D (period A values omitted from average due to higher initial release rates) was used to develop a threshold of similar dissolution rates. In general, calculated plagioclase and augite dissolution rates were the closest among the four models over time. Calculated plagioclase dissolution rates fell within a factor of 3 from the overall average for periods B, C, and D approximately 80% of the time (Figure 68). Augite dissolution rates also showed good agreement among the four models, where 60% of all values fell within a factor of 3 from the overall average (Figure 69). The range in olivine, hypersthene, and biotite dissolution rates was larger between the four models. In general, the calculated dissolution rates were within a factor of 3 from the overall average approximately 45-55% of the time (Figures 70-72). These ranges in modeled dissolution rates are consistent with literature which reported variations in laboratory results differing by a factor of 2-5 (Blum and Stillings 1995, Lapakko et al. 2012). Future reporting should include additional analysis to further address these ranges in modeled silicate dissolution rates.

The range in calculated silicate mineral dissolution rates between some models during specified periods could be due to a number of factors. These may include, but are not limited to, differences in mineral content between samples, number of drainage quality measurements per sample, and calculated dissolution rates equal to zero which indicate an absence of dissolution. The calculated absence of dissolution may have had the most significant effect on the calculated statistics for each mineral dissolution model. For example, olivine dissolution rates calculated during period C for models 1-4 used at total of 840 values, 79 of those were zero values representing an absence of dissolution (Table 106). Because 90% of the values used in the calculations were greater than zero, the range in olivine

dissolution rates between models is fairly consistent (+/-40%) (Figure 71). In contrast, olivine dissolution rates calculated during period B for models 1-4 used at total of 1,615 values, 376 of those were zero values. Because 25% of the values used in the calculations were equal to zero, the range in olivine dissolution rates between models is significant (1–250%) (Figure 71). Future analysis should examine these calculations in more detail.

In general, silicate mineral dissolution rates were slower than sulfide oxidation rates. Average sulfide oxidation rates were approximately 50 times faster than silicate dissolution rates, ranging from 20 – 400 times faster depending on the silicate mineral. Coatings formed far more slowly on silicate minerals than sulfides (Appendix 1, Attachment A1.2), indicating their reduced reactivity relative to sulfide minerals, particularly pyrrhotite.

6.7.4. Comparison of Modeled Dissolution Rates with Literature Values

In addition to individual experimental rates from the literature, modeled silicate dissolution rates were compared to acid and neutral rate equations from Palandri and Kharaka (2004) (see Figures 78, 79 for plagioclase example). These equations were developed by synthesizing all appropriate literature rates for each mineral, often including rates listed previously in the tables (Tables 73-77). The neutral rate equation indicates independence of pH in the circumneutral pH range. Palandri and Kharaka also list rates for basic pH ranges, although this is of lesser concern to dissolution of Duluth Complex rock associated with mining wastes. In this section, modeled silicate dissolution rates were compared to the summation of acid and neutral rate equations from Palandri and Kharaka (Table 78). Due to the lack of Na and K data during periods A and B for reactors 1-38, the samples stored for 8 years prior to kinetic test work (reactors 39-44) are included in this comparison in order to utilize available Na and K data. It should be noted that SO_4, Ca, and Mg release rates during these earlier periods were reasonably similar to reactors 1-38, with average release rates within 85, 75, and 40% of the overall average, respectively (Tables 28-30).

Additional comparisons with other literature rates are in Tables 73-77. These differences between modeled dissolution rates for each silicate mineral and reported literature rates could be due to a number of model assumptions (i.e. stoichiometric release, surface roughness factor) as well as the mineral purity of the samples analyzed in the literature.

6.7.4.1. Plagioclase

Modeled plagioclase dissolution rates during periods B-D were fairly consistent over time. Calculated rates were within a factor of three from the overall average (9.3×10^{-13} mol m^{-2} s^{-1}) in all four models (Figure 68). Plagioclase dissolution rates displayed little pH dependence, although there appeared to be a subtle dependence in models 3 and 4 when pH dropped below 4.5 (Figure 80). The range in plagioclase dissolution rates was fairly tight in models 1, 3, and 4 (within an order of magnitude) as pH dropped from 7 to 4. However, this range was much larger in model 2 (three orders of magnitude), which is most likely attributed to augite dissolution.

Calcium concentrations reflect the fairly constant modeled plagioclase rates with respect to pH and time. Increases in calcium release rates were often observed as pH decreased near and below 4. These increases were typically paralleled by increases in magnesium concentration. A total of 26 samples displayed drainage pH below 4.4. Out of those, 15 samples displayed increasing Ca and Mg, 5 samples displayed increasing Ca with no associated Mg increase, 2 samples displayed increasing Mg with no associated Ca increase, and 4 samples displayed no increase in Ca or Mg release. This parallel increase in Ca and Mg when pH < 4.4 is most likely attributed to augite dissolution, although plagioclase rates

also might have increased in this pH range. The lack of rate dependence on dissolution time is consistent with conclusions from White and Brantley (2003). They reported very slow decreases in plagioclase dissolution rates, estimating that it would take over 22,000 years for a rate decrease of about two orders of magnitude.

Although modeled plagioclase dissolution rates were slower than reported rates for labradorite, model 1 likely overestimated the plagioclase dissolution rate. Initially, plagioclase dissolution overestimates could have been from rapid dissolution of Ca-rich minerals such as calcite. Later overestimation could have resulted from nonstoichiometric dissolution of intermediate feldspars (%An<75) at pH<5. Nonstoichiometric plagioclase dissolution has been reported in both field and laboratory (Blum and Stillings 1995), resulting in a Si-rich leached layer. This means that Ca could be preferentially released, although some claims of this have been attributed to dissolution of Ca-rich exsolution lamellae (Inskeep et al. 1991). There also could have been error resulting from the formula used to determine density of plagioclase (Table 48). This equation, which depended on percent anorthite, was subject to the error associated with the number of grains examined and could result in either higher or lower rate determinations.

More recent work has reported stoichiometric dissolution of plagioclase (e.g., White and Brantley 2003). Some results from this experiment indicate nonstoichiometric dissolution. Observed Ca:Na was generally higher than predicted by average stoichiometry of the sample (Table 26, Appendix 7, Attachment A7.1, Figures A7.1-A7.21), possibly indicating preferential release of Ca. Observed ratios were typically lower than values predicted from individual grain stoichiometry when pH>4.4, while the relationship was reversed when pH<4.4. Although some of these differences could be based on sampling error brought on by analysis of too few grains, the differences at pH<4.4 are quite large. These large differences between observed and predicted at lower pH could be due to the formation of Ca-poor depleted layers, or the dissolution of augite.

Most experimentally determined rates were conducted under conditions likely to give higher rates than this experiment, as noted earlier in Section 6.7.1. However, modeled plagioclase dissolution rates were approximately an order of magnitude faster than rates in fresh granite with 26% plagioclase, approximately 1.0 x 10-13 mol m^{-2} s^{-1} (White and Brantley 2003). White and Brantley's study is a good comparison to this experiment because it was conducted in a column reactor, ran for an extended period (6.2 years), and was conducted with ground rock rather than a homogeneous sample of a pure mineral.

Palandri and Kharaka (2004) labradorite and andesine dissolution rates were used for comparison with modeled plagioclase dissolution rates. Labradorite rates were 40-160 times higher at pH 4 and 3-20 times higher at pH 6 than modeled plagioclase dissolution rates. Andesine rates were 10-40 times higher at pH 4 and 0.7-7 times higher at pH 6 (Table 78). Model 1 rates were typically the closest to literature rates due to the assumption that all Ca was attributed to plagioclase. In contrast, differences between modeled and literature rates at pH 6 were the greatest in models 3 and 4, because plagioclase dissolution was calculated based on sodium release which was typically lower than calcium release.

6.7.4.2. Olivine

Modeled olivine dissolution rates during periods B-D were less consistent over time. Calculated rates were within a factor of three from the overall average (9.3 x 10^{-12} mol m^{-2} s^{-1}) in 7 out of 9 model runs (Figure 71). It should be noted that in a number of the model runs, a significant portion of the calculated rates were zero (≈3 to 50%), indicating a calculated absence of olivine dissolution. Olivine dissolution rates displayed a slight pH dependence, increasing by about a factor of three as pH dropped below 4.5

(Figure 82). This dependence was more apparent in models 3 and 4. The range in olivine dissolution rates was within approximately two orders of magnitude as pH dropped from 7 to 4 in all four models.

Although literature dissolution rates reported for olivine are much faster than those for plagioclase, the modeled maximum dissolution rates for olivine (generally assuming all Mg release from olivine) are only slightly faster than the modeled plagioclase dissolution rates. Reported rates of olivine dissolution are approximately two orders of magnitude faster than plagioclase, whereas average modeled rates were 1-3 times faster in period A (the first 34-144 weeks of dissolution) and 2-30 times faster in periods B through D (Table 57). Although this is, in part, potentially an artifact of the modeling assumptions, this might be due to inhibition of dissolution by alteration of the olivine surface, a phenomenon reported by several researchers.

The relative rates of plagioclase and olivine dissolution determined by these models must be considered based on the composition of the samples. Rates of calcium release were often greater than those of magnesium release. However, the plagioclase content of the samples was considerably higher than the olivine content. Plagioclase contents were typically near 50 percent. In contrast, olivine contents ranged from 5-20 percent, excluding two samples with no olivine (0.40, 0.54%S) and one sample with 39% olivine (0.92%S). Thus, the plagioclase contents were considerably higher than those of olivine. Consequently, calcium release rates exceeding magnesium release rates did not indicate plagioclase dissolution rates exceeding olivine dissolution rates.

Palandri and Kharaka (2004) forsterite and fayalite dissolution rates were used for comparison with modeled olivine dissolution rates. Forsterite rates were 120-270 times higher at pH 4 and 10-280 times higher at pH 6 than modeled olivine dissolution rates. Fayalite rates were 100-230 times higher at pH 4 and 0.9-20 times higher at pH 6 than modeled olivine dissolution rates (Table 78). Agreement with literature values typically was best in model 1, which assumed that all Mg release was associated with olivine, resulting in the maximum dissolution rates.

6.7.4.3. Pyroxene Group Silicates

6.7.4.3.1. Hypersthene (Bronzite)

Modeled hypersthene dissolution rates during periods B-D were less consistent over time. Calculated rates were within a factor of three from the overall average (2.4×10^{-12} mol m^{-2} s^{-1}) in 5 out of 7 model runs (Figure 70). It should be noted that in a number of the model runs, a significant portion of the calculated rates were zero (≈5 to 50%), indicating a calculated absence of hypersthene dissolution. Hypersthene dissolution rates displayed little pH dependence, increasing slightly (approximately 40%) as pH dropped below 4.5 (Figure 83). The range in hypersthene dissolution rates was within approximately two orders of magnitude as pH dropped from 7 to 4 in models 1 and 3 and approximately three orders of magnitude in models 2 and 4.

Palandri and Kharaka (2004) bronzite dissolution rates were used for comparison with modeled hypersthene dissolution rates. Bronzite rates were 3-5 times higher at pH 4 than modeled hypersthene dissolution rates. Bronzite rates at pH 6 were typically in good agreement with modeled hypersthene dissolution rates, within a factor of 4 (Table 78). Similar to olivine, model 1 hypersthene dissolution rates were the closest to literature values due to the assumption that all Mg was associated with hypersthene, resulting in the maximum dissolution rates.

6.7.4.3.2. Augite

Modeled augite dissolution rates during periods B-D were fairly consistent over time. Calculated rates were within a factor of three from the overall average (3.6×10^{-12} mol m^{-2} s^{-1}) in all four models (Figure 69). Augite dissolution rates displayed a strong pH dependence in model 1, but no significant pH dependence in models 2-4 was observed (Figure 81). The range in augite dissolution rates was typically within 1-2 orders of magnitude as pH dropped from 7 to 4 in all four models.

The concurrent release of Ca and Mg in period C was crucial to model 1, because it indicated dissolution of augite, which was the only common sample mineral with approximately equal proportions of Ca and Mg. This assumption was validated by reported labradorite and augite rates from four studies. Results from these four studies were graphically analyzed to determine that augite rates were faster than labradorite rates at pH<4 (McAdam et al. 2007). This difference in rates, combined with the stoichiometry of augite in these samples (Table 11), gave good justification for the concurrent release assumption.

Palandri and Kharaka (2004) augite dissolution rates were 40-70 times higher than modeled rates at pH 4. Modeled dissolution rates at pH 6 were in very good agreement with literature values, within 80% (Table 78). Model 2 rates were typically the closest to literature rates due to the assumption that all Mg release was associated with augite dissolution.

6.7.4.4. Biotite

Modeled biotite dissolution rates during periods B-D were less consistent over time. Calculated rates were within a factor of three from the overall average (3.8×10^{-12} mol m^{-2} s^{-1}) in 8 out of 10 model runs (Figure 72). It should be noted that in a number of the model runs, a significant portion of the calculated rates were zero (≈6 to 50%), indicating a calculated absence of biotite dissolution. Biotite dissolution rates displayed no significant pH dependence in all four models (Figure 84). The range in biotite dissolution rates was within approximately one order of magnitude as pH dropped from 7 to 4 in models 1 and 4 and approximately 2-3 orders of magnitude in models 2 and 3.

Modeled biotite dissolution rates were 2-5 times higher at pH 4 and 0.3-40 times higher at pH 6 than Palandri and Kharaka (2004) biotite dissolution rates (Table 78). These higher modeled rates were most likely due to the assumption in models 1-3 that all residual Mg could be associated with biotite. However, based on mineralogical analysis, biotite on average contributed approximately 14% of Mg present in the samples (Table 23). Modeled biotite dissolution rates may have also been higher due to an underestimation of surface roughness.

7. CONCLUSIONS

Fifteen 75-gram samples of Duluth Complex rock ($0.053 < d < 0.149$ mm) containing 0.18 to 1.71 percent sulfur were characterized (chemistry, mineral content, mineral chemistry), subjected to laboratory dissolution tests, and examined for dissolution features following these tests. The major elements present as oxides in the samples, in descending order were silicon, aluminum, iron, calcium, magnesium, sodium, and titanium. Heavy metals of greatest concern decreased in the order (with typical percent concentration ranges in parentheses) copper (0.05 – 0.14), nickel (typically 0.03 – 0.05), zinc (0.01 – 0.02), and cobalt (0.005 – 0.01) (Table 3). Plagioclase was the dominant host-rock mineral, comprising 40 to 60 volume percent. Olivine, hypersthene, and augite contents typically ranged from 5 to 20 percent, and ilmenite contents were typically reported as 3 to 8 percent. Lesser amounts of potassium feldspar, biotite, prehnite, and amphibole were also reported in some samples (Table 7). Based on chemical and mineralogical analyses, pyrrhotite was the major sulfide mineral, followed by chalcopyrite, cubanite, pentlandite, and lesser amounts of cobalt and zinc sulfides were present.

The samples were subjected to laboratory dissolution tests in which they were rinsed weekly with 200-mL volumes of distilled-deionized water for a period of 144 -1252 weeks. The volume of drainage was determined and drainage samples were analyzed at varying frequencies for pH, specific conductance, sulfate, silicon, calcium, magnesium, sodium, potassium, iron, aluminum, copper, nickel, cobalt, and zinc.

The objectives of the study were as follows.

1) Determine the variation of laboratory drainage pH with solid-phase sulfur content of Duluth Complex samples.
2) Based on the aforementioned relationship, categorize solids based on sulfur content and associated drainage pH.
3) Determine the variation of major ion release rates with solid-phase sulfur content and dissolution time.
4) Determine the empirical neutralization potentials of solids producing acidic drainage and compare these values to those determined by solid-phase analysis.
5) Determine rates of sulfide and silicate mineral dissolution as a function of dissolution time and drainage pH.

In general, drainage pH tended to decrease and release rates of heavy metals (copper, nickel, cobalt, zinc) and major cations (calcium, magnesium, sodium, potassium) tended to increase with increasing sulfur content (Section 6.4). Samples were divided into three groups based on sulfur content: Group I, 0.18, 0.22 %S; Group II, $0.40 \leq$ %S ≤ 0.70; and Group III, $0.70 <$ %S ≤ 1.64 (Section 6.2). As sulfur content increased, minimum pH decreased, and sulfate, major cation, and trace metal release increased. Drainage pH for the Group I samples was generally above 6.0. Based on drainage pH alone, disposal of waste rock in Group I would probably require no rigorous reclamation. Potential impacts of sulfate, copper, nickel, cobalt and zinc release from rock in this range of sulfur content would require consideration. The minimum drainage pH values for Groups II and III were 3.8 and 3.0, respectively. Minimum pH values and corresponding maximum sulfate release values were typically attained after 300 to 400 weeks (6 to 8 years) of dissolution. The maximum time to minimum pH was roughly 800 weeks, emphasizing the fact that short term drainage quality results do not necessarily reflect long-term trends. The decreasing sulfate release following attainment of maximum values was attributed to development of coatings on the sulfide mineral surface, and this was supported by photographs of

leached sulfide grains. Diminishing sulfide mineral content over time contributed, to a lesser extent, to the decrease.

Neutralization potential (NP) is a measure of the capacity of a sample to neutralize acid, and this capacity was determined empirically based on the drainage quality observed in dissolution tests (Section 6.4.2). This empirical neutralization potential (ENP) was used for comparison with results from three methods used to determine NP based on solid-phase analysis. ENP values typically fell between 0.5 and 2 kg $CaCO_3$/t rock. NP calculated using carbon dioxide analysis to estimate total calcium plus magnesium carbonate content were closest to the ENP values (<0.7 to 2.5 kg $CaCO_3$/t rock), therefore the best estimate of the capacity of these samples to neutralize acid. Values determined on a subset of samples by a direct titration technique (Lapakko 1994a) yielded values roughly 1 to 3 kg $CaCO_3$/t rock higher than the observed values. Values determined by one common NP method (Sobek et al. 1978) almost always fell between 12 and 21 kg $CaCO_3$/t rock, substantially higher than the observed values (Table 22, Figure 61). These results indicate that caution must be exercised when selecting solid-phase methods for determining NP and interpreting the results generated.

These rock samples exhibited four typical periods of behavior characterized by declining pH in periods A and B, minimum pH in period C and increasing pH in period D (Section 6.3). All samples exhibited periods A and B, while only samples from groups II and III exhibited the more acidic pH and subsequent pH rise characteristic of periods C and D, respectively. The declining pH was due to increased acid production with increasing sulfide mineral surface area available for oxidation. The rise in pH after period C was caused by decreasing rates of sulfide mineral oxidation that was due to coating of the sulfide minerals and, to a lesser extent, diminishing amounts of sulfide minerals present. At the end of the experiment, there was typically 50% of sulfide minerals left unoxidized.

Two models were developed to determine rates of sulfide oxidation and four were developed for silicate dissolution. These rates were based on observed solute release, potential mineral sources of solutes, mineral chemistry, and mineral content. For each sample, mineral-specific surface areas were determined based on mineral content, geometric surface area, and surface roughness. Pyrrhotite oxidation rates were based on sulfate release and were highest during period C, coincident with minimum pH values (Section 6.6). Calculated rates of pyrrhotite oxidation during periods A and B were generally in reasonable agreement with rates reported for laboratory tests on pyrrhotite alone (i.e. in the absence of other minerals) (Janzen et al. 2000).

Silicate dissolution rates were generally in fair agreement among the four silicate dissolution models (Section 6.7). There did not appear to be strong dependence of rates on pH, with the exception of augite dissolution rates. Calculated silicate mineral dissolution rates were compared to rates published by Palandri and Kharaka (2004) (Table 78, Figures 80-84). Calcium-based rates of plagioclase dissolution for pH above about 5.5 were lower than published rates but within an order of magnitude. Calcium-based plagioclase rates at lower pH were much lower than published rates, as were plagioclase rates based on sodium dissolution. Similarly, calculated augite dissolution rates for pH above 5.5 tended to be lower than published rates but within an order of magnitude. Under more acidic conditions the calculated augite rates were generally more than an order of magnitude lower than published rates. The upper range of calculated hypersthene dissolution rates tended to be bounded by the published rates. For all pH values calculated olivine dissolution rates were generally far lower than published rates, and calculated biotite dissolution rates, whether based on magnesium or potassium release, tended to be one to two orders of magnitude higher than published rates.

Six Duluth Complex samples ($0.67 \leq \%\ S \leq 1.71$) that had been stored for eight to twenty years were also subjected to testing, typically for 809 weeks. These samples exhibited slower rates of sulfate

release and higher drainage pH values than the aforementioned samples. Based on information presented by Janzen et al. (2000), this was attributed to the development of coatings on sulfide minerals during storage. It should be noted that during operation rock is exposed to elements immediately after blasting. Therefore, the storage time of drill core used to predict drainage quality in the field will affect the laboratory dissolution test results generated. That is, storage time for laboratory predictive test samples should be minimized, especially if the drill core samples have been crushed. Results from the tests on stored Duluth Complex samples were not included in modeling of pyrrhotite oxidation rates but were used in modeling silicate mineral dissolution rates (Section 6.7.4).

Four Virginia formation samples ($1.71 \leq \% S \leq 5.44$) were subjected to testing for 78 weeks. Due to their high sulfur contents, these samples generated acidic drainage relatively quickly. Drainage quality of these tests is briefly addressed in the results section, but is not included in the overall discussion. All drainage quality data are included in Appendix 3.

Practical implications of these results for environmentally sound management of waste rock generated by mining in Minnesota include the potential for managing waste rock based on sulfur content, a relatively simple analytical assessment. Based on drainage pH alone, disposal of waste rock in Group I ($\%S \leq 0.22$ %) would probably require no rigorous reclamation. Potential impacts of sulfate, copper, nickel, cobalt, and zinc release from rock in this range of sulfur content would require consideration. In order to meet water quality standards, waste rock producing drainage pH values similar to Group III samples ($0.70 < \%S \leq 1.64$) would require the most rigorous control measures of the samples tested. Elevated release of sulfate and heavy metals from these groups would likely require mitigation as well. The results have implications of substance to mine waste drainage quality prediction. First, caution that must be exercised when assessing the capacity of rock to neutralize acid based on solid-phase analyses designed to quantify NP. Second, results generated from short-term dissolution tests do not necessarily simulate long-term drainage quality. Emphasizing this point, drainage pH from one sample was circumneutral for 800 weeks and then acidified, reaching a minimum pH of 3.8. Third, tests conducted on samples that had been stored for 8 to 20 years yielded lower rates of sulfate release and higher drainage than samples within about one year of excavation. Therefore, samples used for mine waste drainage quality prediction should be tested as close to the time of their excavation as possible.

Additional work should be conducted to further the benefits generated to date. This includes continued dissolution testing to confidently define long-term trends, analysis of leached solids to increase understanding of chemical reactions controlling solute release (e.g. chemical precipitation and adsorption), and more detailed analysis of the present data and modeled mineral dissolution rates. Furthermore, the methods for calculating mineral dissolution rates and approaches for other aspects of data analysis developed for this project should be applied to experiments examining dissolution of Partridge River Intrusion rocks.

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Table 1. High Sulfur experiment reactors and description of samples.
DC = Duluth Complex and VF = Virginia Formation.

% Sulfur	Reactor	Sample Source	Rock Suite	Period of Record (weeks)
0.18[1]	1	Dunka BH	DC	1252[4]
	2	Dunka BH	DC	328
0.22[1]	3	Dunka BH	DC	1252[4]
	4	Dunka BH	DC	328
0.40[1]	5	Dunka BH	DC	909[5]
	6	Dunka BH	DC	328
0.41[1]	7	Dunka BH	DC	724
	8	Dunka BH	DC	328
0.51[1]	9	Dunka BH	DC	724
	10	Dunka BH	DC	328
0.54[1]	11	Dunka BH	DC	441
	12	Dunka BH	DC	328
0.57[1]	13	Dunka BH	DC	441
	14	Dunka BH	DC	328
0.58[1]	15	Dunka BH	DC	724
	16	Dunka BH	DC	328
0.67[3]	39	ARIMETCO	DC	809[4]
0.67[3]	40	Dunka BH	DC	809[4]
0.71[1]	17	Dunka BH	DC	724
	18	Dunka BH	DC	328
0.72[3]	42	AMAX	DC	809[4]
0.82[3]	43	Dunka BH	DC	809[4]
0.92[3]	41	AMAX	DC	809[4]
1.12[2]	35	Dunka BH	DC	360
	36	Dunka BH	DC	247
1.16[2]	29	Dunka BH	DC	643
	30	Dunka BH	DC	247
1.40[2]	37	Dunka BH	DC	360
	38	Dunka BH	DC	247
1.44[2]	33	Dunka BH	DC	643
	34	Dunka BH	DC	247
1.63[1]	19	Dunka BH	DC	724
	20	Dunka BH	DC	289
1.64[2]	31	Dunka BH	DC	360
	32	Dunka BH	DC	247
1.71[3]	44	AMAX	DC	144
2.06[1]	21	Dunka BH	VF	78
	22	Dunka BH	VF	78
3.12[1]	23	Dunka BH	VF	78
	24	Dunka BH	VF	78
3.72[1]	25	Dunka BH	VF	78
	26	Dunka BH	VF	78
5.44[1]	27	Dunka BH	VF	78
	28	Dunka BH	VF	78

[1] Reactor initiated week of February 14, 1989.
[2] Reactor initiated week of September 4, 1990.
[3] Reactor initiated week of August 12, 1997.
[4] Reactor was still active at time of report.
[5] Reactor was terminated after 909 weeks.

Table 2. Chemical composition (%) of unleached rock samples. Analyses by Actlabs, Inc.

%S	0.18	0.22	0.40	0.41	0.51	0.54	0.57	0.58	0.67	0.67	0.71	0.72	0.82
Reactor	1,2	3,4	5,6	7,8	9,10	11,12	13,14	15,16	39	40	17,18	42	43
CO_2	<0.03	<0.03	0.64	0.06	0.07	0.11	0.10	0.07	<0.03	<0.03	0.05	0.06	<0.03
Al_2O_3	18.92	19.82	17.68	16.20	14.89	16.27	14.75	17.64	18.32	16.26	18.17	20.98	16.76
CaO	8.75	9.18	7.6	8.71	7.76	8.35	7.79	7.77	8.69	8.35	7.60	9.77	8.37
Fe_2O_3	1.56	1.23	1.84	2.51	2.04	2.08	4.03	4.81	3.75	3.28	4.65	0.64	3.52
FeO	10.71	10.63	10.19	11.83	13.3	9.37	11.59	10.06	9.22	10.75	10.27	8.64	9.38
K_2O	0.33	0.50	0.72	0.71	0.89	1.13	0.92	0.99	0.60	0.72	0.78	0.30	0.74
MgO	7.85	6.27	5.81	6.91	6.66	8.61	7.98	5.84	7.53	6.88	6.96	7.24	6.75
MnO	0.139	0.124	0.181	0.177	0.188	0.156	0.185	0.150	0.16	0.21	0.133	0.1	0.15
Na_2O	2.59	3.03	2.45	2.78	2.60	1.80	2.54	2.78	0.33	0.28	2.55	2.78	2.33
P_2O_5	0.11	0.14	0.15	0.19	0.27	0.22	0.33	0.16	0.13	0.13	0.09	0.04	0.12
SiO_2	46.54	46.82	50.49	46.24	46.75	48.49	45.76	43.19	47.7	49.89	44.16	47.58	50.32
TiO_2	1.382	1.102	1.312	2.869	2.650	1.196	2.640	1.891	1.88	2.45	0.964	0.54	1.67
LOI[1]	0.25	0.31	0.66	-0.21	-0.23	1.41	0.26	3.39			2.87		
LOI2[2]	1.45	1.50	1.80	1.11	1.26	2.46	1.56	4.52			4.02		

%S	0.92	1.12	1.16	1.40	1.44	1.63	1.64	1.71	2.06	3.12	3.72	5.44
Reactor	41	35,36	29,30	37,38	33,34	19,20	31,32	44	21,22	23,24	25,26	28
CO_2	<0.03	<0.03	0.08	0.10	<0.03	0.11	0.14	0.06	0.12	<0.03	0.06	0.32
Al_2O_3	19.19	14.14	15.94	15.41	15.22	14.35	14.61	18.66	16.11	16.50	20.47	11.49
CaO	9	7.38	8.25	8.51	7.40	6.60	8.23	7.75	2.22	5.14	9.29	9.98
Fe_2O_3	1.02	2.45	2.15	4.85	3.33	1.73	3.85	4.13	0.40	0.51	nd	0.42
FeO	10.82	12.63	12.84	11.4	12.42	14.38	12.35	9.49	10.23	9.91	10.81	17.06
K_2O	0.41	1.74	0.75	0.67	0.96	1.25	0.86	0.65	1.88	1.09	0.42	0.55
MgO	7.5	5.65	6.73	6.41	7.00	6.60	6.33	8.28	4.56	5.71	6.60	10.79
MnO	0.15	0.184	0.175	0.176	0.163	0.175	0.191	0.14	0.082	0.106	0.113	0.205
Na_2O	0.33	2.44	2.70	2.58	2.58	2.55	2.49	2.3	2.03	2.27	2.99	1.27
P_2O_5	0.06	0.17	0.15	0.20	0.14	0.19	0.19	0.12	0.30	0.13	0.11	0.10
SiO_2	47.07	47.98	46.21	44.25	46.86	46.81	45.35	45.68	57.79	54.68	47.14	42.51
TiO_2	1.39	2.961	2.435	2.946	2.274	3.587	2.728	1.67	1.310	1.438	0.793	1.779
LOI[1]		0.93	0.33	1.08	0.40	1.28	1.48		2.10	1.51	0.44	1.92
LOI2[2]		2.34	1.77	2.36	1.79	2.89	2.86		3.25	2.62	1.65	3.83

[1] Loss on Ignition. LOI<0.01 represent a gain on ignition.
[2] Loss on Ignition adjusted for the difference in oxygen between FeO and Fe_2O_3.
Note: nd = not detected.

Table 3. Trace metal composition (ppm) of unleached rock samples. The four metals of greatest environmental concern (Co, Cu, Ni, Zn) are listed first. Analyses by Actlabs, Inc. Page 1 of 4.

%S	0.18	0.22	0.40	0.41	0.51	0.54	0.57	0.58	0.67	0.67	0.71	0.72	0.82
Reactor	1,2	3,4	5,6	7,8	9,10	11,12	13,14	15,16	39	40	17,18	42	43
Co	84	79	43	70	71	52	81	81	73	56	120	nd	45
Cu	1360	1350	512	788	1430	211	1290	1300	617	490	1160	517	601
Ni	525	484	231	511	473	174	398	396	523	261	388	254	204
Zn	102	103	123	160	103	171	187	183	119	187	126	59.2	148
Ag	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd
As	nd	nd	7	nd	nd	10	nd	nd	nd	3	10	nd	3
Ba	141	166	275	221	259	380	288	290	100	102	256	100	260
Bi	0.5	nd	nd	0.5	0.8	nd	1.8	1.7	49.78	90.54	1.7	12.67	44.06
Cr	167	152	296	266	232	484	185	187	92	166	170	45	190
Cs	0.6	0.5	1.5	1.1	1.4	2.3	1.5	1.5	0.6	0.8	1.9	nd	1.6
Hf	1.7	1.9	2.3	4.2	5.2	2.1	5.3	5.1	2.3	4.3	2.1	0.9	2.7
Mo	2	2	3	nd	3	3	nd	nd	2.13	1.85	3	2.71	2.20
Pb	5	nd	6	7	5	nd	10	10	12.9	19.2	12	15.1	14.8
Rb	7	11	17	22	28	39	30	30	nd	nd	26	nd	27
Sb	0.9	0.5	1.5	nd	0.7	1.0	0.5	0.5	0.2	0.3	0.5	nd	0.2
Sn	1	2	2	2	2	nd	3	3			2		
Sr	299	311	318	268	275	276	260	339	185	256	348	338	283
Ta	0.5	0.5	0.8	1.1	1.2	0.5	1.3	1.2	nd	nd	0.4	nd	nd
V	155	105	237	239	223	304	212	182	16.8	25.1	127	45.6	232
W	nd	nd	2	1	1	nd	2	2	nd	nd	nd	nd	nd
Y	11	13	21	25	30	18	31	16	nd	nd	11	5.17	20.7
Zr	55	64	78	138	166	68	172	92	15.3	28.3	71	21	151
Ce	17.9	23.1	45.1	32.6	43.8	30.1	47.3	47.1	16	28	18.3	8	30

Table 3. Trace metal composition (ppm) of unleached rock samples. The four metals of greatest environmental concern (Co, Cu, Ni, Zn) are listed first. Analyses by Actlabs, Inc. Page 2 of 4.

%S	0.92	1.12	1.16	1.40	1.44	1.63	1.64	1.71	2.06	3.12	3.72	5.44
Reactor	41	35,36	29,30	37,38	33,34	19,20	31,32	44	21,22	23,24	25,26	28
Co	67	70	83	91	87	81	95	82	44	52	87	117
Cu	752	800	1090	1560	1210	1180	2220	660	1300	1540	1280	1290
Ni	324	329	422	503	386	492	718	354	327	436	540	589
Zn	100	305	171	181	191	93	191	86.6	223	123	80	120
Ag	nd	nd	nd	nd	nd	6	8	nd	nd	nd	nd	nd
As	nd	9	nd	nd	nd	6	nd	11	14	13	8	19
Ba	100	405	210	201	251	288	249	135	508	274	142	155
Bi	37.5	0.6	0.8	0.7	0.6	nd	1.2	22.30	0.8	0.6	0.7	0.8
Cr	98	211	209	177	204	227	210	210	277	265	150	653
Cs	nd	2.0	1.5	1.5	1.7	2.2	1.7	1.8	6.5	4.5	0.5	1.4
Hf	1.2	5.7	4.7	6.8	4.1	4.7	5.6	0.5	4.3	3.1	1.8	2.3
Mo	1.21	3	2	2	nd	4	2	2.19	5	4	2	2
Pb	13.4	11	10	16	8	nd	7	22.7	21	13	5	5
Rb	nd	49	23	22	30	37	28	nd	69	45	11	21
Sb	nd	0.8	0.6	nd	nd	0.8	1.6	0.2	1.1	1.2	1.0	1.7
Sn		2	2	4	2	1	3		1	2	2	2
Sr	142	232	270	246	246	209	245	335	239	259	336	148
Ta	nd	1.4	1.2	1.5	1.2	1.4	1.4	nd	1.1	0.7	0.3	0.5
V	10.6	291	201	221	205	293	224	149	225	192	78	301
W	nd	4	nd	nd	nd	nd	4	nd	1	2	6	nd
Y	nd	29	25	28	24	28	31	4.48	32	18	10	20
Zr	11.8	195	152	234	129	157	182	25.5	143	101	64	72
Ce	15	40.7	33.0	35.9	34.1	39.4	39.8	6	64.1	39.2	17.7	24.6

Table 3. Trace metal composition (ppm) of unleached rock samples. Analyses by Actlabs, Inc. Page 3 of 4.

%S	0.18	0.22	0.40	0.41	0.51	0.54	0.57	0.58	0.67	0.67	0.71	0.72	0.82
Reactor	1,2	3,4	5,6	7,8	9,10	11,12	13,14	15,16	39	40	17,18	42	43
La	8.7	11.1	24.8	15.1	20.2	15.7	21.2	21.3	9	14.4	8.8	4.4	6.4
Lu	0.17	0.19	0.39	0.39	0.43	0.29	0.42	0.43	0.25	0.38	0.15	0.08	0.28
Nd	9.5	12.4	20.5	18.4	24.7	15.3	26.6	27.0	8		9.8	nd	14
Tb	0.3	0.4	0.6	0.8	0.9	0.5	1.0	1.0	nd	0.6	0.3	nd	nd
Th	0.8	10	1.5	1.6	2.1	0.9	2.2	2.2	0.8	1.3	1.1	nd	1.1
U	0.3	0.3	0.6	0.5	0.8	0.5	0.7	0.7	0.7	0.7	0.5	nd	nd
Yb	1.1	1.2	2.3	2.4	2.8	1.9	2.7	2.8	1.4	2.5	1.0	0.5	1.8
Ga	20	22	24	22	21	20	22	22			20		
Ge	1	nd	2	1	2	1	1	2			nd		
Nb	5	5	10	15	16	7	17	17			5		
In	nd	nd	nd	nd	nd	nd	nd	nd			nd		
Pr	2.03	2.62	4.79	3.87	5.19	3.41	5.56	5.62			2.09		
Sm	2.2	2.6	3.9	4.3	5.8	3.3	6.0	6.1	2.3	3.2	2.2	0.8	2.9
Gd	2.2	2.6	3.6	4.3	5.6	3.1	5.9	5.9			2.0		
Dy	2.0	2.4	3.5	4.4	5.5	3.2	5.6	5.6			1.8		
Ho	0.4	0.4	0.7	0.8	1.0	0.6	1.0	1.0			0.3		
Er	1.1	1.3	2.1	2.4	2.9	1.8	2.9	3.0			1.0		
Tm	0.16	0.18	0.32	0.36	0.43	0.27	0.44	0.43			0.14		
Tl	nd	nd	nd	0.1	0.1	0.2	0.2	0.2			0.3		
Eu	1.54	1.78	2.49	1.76	2.06	1.67	2.02	2.00	1.3	1.5	1.73	1	1.6

Table 3. Trace metal composition (ppm) of unleached rock samples. Analyses by Actlabs, Inc. Page 4 of 4.

%S	0.92	1.12	1.16	1.40	1.44	1.63	1.64	1.71	2.06	3.12	3.72	5.44
Reactor	41	35,36	29,30	37,38	33,34	19,20	31,32	44	21,22	23,24	25,26	28
La	8	20.5	15.7	16.5	16.7	19.1	18.5	4.5	31.6	19.7	8.5	11.3
Lu	0.15	0.46	0.40	0.47	0.40	0.45	0.47	0.08	0.49	0.29	0.15	0.28
Nd	6	21.8	18.8	20.2	17.9	21.3	22.3	nd	32.6	19.0	9.4	14.3
Tb	nd	0.8	0.8	0.9	0.7	0.8	1.0	nd	1.0	0.6	0.3	0.6
Th	0.5	2.8	1.8	2.0	2.6	2.2	2.5	nd	6.4	4.0	0.9	1.5
U	nd	1.4	0.6	0.7	1.1	1.4	0.6	nd	2.8	2.2	0.3	0.8
Yb	0.9	2.9	2.4	3.0	2.5	2.8	3.0	0.5	3.1	1.8	0.9	1.7
Ga		22	21	21	20	20	22		24	21	22	15
Ge		2	1	1	1	1	2		2	1	1	2
Nb		19	16	19	15	17	18		14	8	4	6
In		nd	nd	nd	nd	nd	nd		nd	nd	nd	nd
Pr		4.73	3.86	4.22	3.89	4.59	4.72		7.22	4.33	2.00	3.03
Sm	1.6	4.8	4.4	4.8	4.1	5.0	5.4	0.7	6.8	3.7	2.0	3.4
Gd		4.8	4.4	4.8	4.0	4.8	5.4		5.9	3.4	1.9	3.5
Dy		4.8	4.4	5.0	4.3	5.0	5.5		5.8	3.1	1.9	3.3
Ho		0.9	0.8	1.0	0.8	0.9	1.0		1.1	0.6	0.3	0.6
Er		2.8	2.5	2.8	2.5	2.8	3.0		3.2	1.8	1.0	1.8
Tm		0.43	0.38	0.42	0.37	0.42	0.46		0.49	0.26	0.14	0.27
Tl		0.3	0.1	0.2	0.2	0.3	0.2		1.9	1.2	nd	0.2
Eu	1.4	2.09	1.85	1.97	1.77	1.98	2.01	1.2	1.65	1.50	1.59	1.34

Note: nd = not detected.

Table 4. Chemical composition (%) of leached rock samples. Analyses by Actlabs, Inc. Page 1 of 2.

Original %S	0.18	0.22	0.40	0.41	0.51	0.54	0.54	0.57	0.57	0.58	0.71	1.12	1.12
Reactor	2	4	6	8	10	11	12	13	14	16	18	35	36
S	0.106	0.152	0.349	0.214	0.233	0.151	0.211	0.192	0.298	0.219	0.317	0.173	0.498
SO_4^{2-} as S	nd	nd	nd	nd	0.12	nd	0.06	nd	nd	nd	0.15	0.12	0.15
CO_2	0.077	0.037	0.059	0.037	0.040	0.106	0.051	0.040	0.033	0.051	nd	0.037	0.026
Al_2O_3	15.17	14.29	17.19	15.69	14.65	16.02	16.06	14.14	14.36	14.88	13.64	13.71	13.96
CaO	8.30	7.26	7.18	8.43	7.60	8.04	8.02	7.52	7.52	7.41	7.33	6.91	7.02
Fe_2O_3	7.57	3.53	1.43	1.00	1.86	0.98	1.37	2.62	2.63	1.25	1.45	2.76	2.50
FeO	6.77	12.27	11.41	13.99	14.22	10.77	10.64	13.74	13.79	12.96	15.22	12.81	12.91
K_2O	0.70	1.04	0.59	0.61	0.78	1.03	1.06	0.76	0.82	0.94	0.70	1.65	1.66
MgO	6.73	6.66	5.98	6.78	6.67	8.59	8.48	7.93	7.83	6.07	9.01	5.65	5.68
MnO	0.184	0.182	0.187	0.182	0.192	0.158	0.157	0.185	0.187	0.185	0.199	0.184	0.186
Na_2O	2.63	2.47	2.33	2.69	2.51	1.76	1.75	2.43	2.44	2.54	2.39	2.38	2.42
P_2O_5	0.25	0.22	0.16	0.18	0.27	0.11	0.12	0.34	0.32	0.20	0.18	0.17	0.16
SiO_2	47.51	44.56	50.58	45.82	47.10	48.49	48.71	44.88	45.46	47.40	46.05	47.89	48.16
TiO_2	2.454	3.900	1.430	2.972	2.751	1.167	1.176	2.851	2.813	3.238	1.993	3.037	3.006
LOI[1]	0.68	1.19	0.56	-0.18	0.18	1.52	1.59	0.67	0.50	0.69	0.28	1.20	1.13
LOI2[2]	1.44	2.56	1.84	1.39	1.77	2.73	2.78	2.21	2.04	2.14	1.98	2.63	2.58

Table 4. Chemical composition (%) of leached rock samples. Analyses by Actlabs, Inc. Page 2 of 2.

Original %S	1.16	1.16	1.40	1.40	1.44	1.63	1.64	1.64	2.06	3.12	3.72	5.44
Reactor	29	30	37	38	34	20	31	32	22	24	26	28
S	1.08	0.592	0.292	0.821	0.663	0.458	0.408	0.662	1.77	1.99	2.83	4.35
SO_4^{2-} as S	nd	0.27	0.09	0.15	0.33	0.24	0.15	0.18	0.05	1.08	1.23	1.86
CO_2	nd	0.011	0.055	0.040	nd	0.110	0.059	0.051	0.070	1.48	1.03	0.62
Al_2O_3	15.83	15.72	14.96	15.25	15.02	14.22	14.09	14.27	14.57	15.51	15.95	13.80
CaO	8.75	8.16	8.14	8.27	7.26	6.53	8.04	8.04	7.73	3.45	1.99	0.64
Fe_2O_3	2.06	1.62	6.24	4.50	3.09	3.63	5.37	5.05	5.64	5.84	8.14	10.51
FeO	13.17	14.03	10.69	12.41	12.80	12.60	11.32	11.51	11.98	5.10	3.43	1.56
K_2O	0.77	0.67	1.65	0.59	0.96	1.14	0.74	0.81	0.70	2.58	2.67	3.63
MgO	6.80	6.65	5.65	6.23	6.95	5.59	6.18	6.17	8.09	5.25	4.34	2.71
MnO	0.174	0.175	0.166	0.176	0.166	0.169	0.176	0.180	0.174	0.079	0.055	0.020
Na_2O	2.68	2.66	2.52	2.54	2.57	2.55	2.38	2.39	2.38	1.52	1.42	1.18
P_2O_5	0.15	0.14	0.19	0.16	0.14	0.17	0.18	0.18	0.18	0.10	0.12	0.09
SiO_2	45.93	45.95	44.15	44.08	46.80	46.82	44.63	44.69	43.97	52.72	52.55	53.32
TiO_2	2.434	2.441	3.074	3.052	2.306	3.572	2.792	2.777	2.169	0.709	0.655	0.408
LOI[1]	1.30	0.57	2.12	1.51	0.83	1.90	2.17	2.12	1.42	6.70	8.61	12.31
LOI2[2]	1.77	2.14	3.32	2.90	2.26	3.31	3.44	3.41	2.76	7.27	8.99	12.48

[1] Loss on Ignition. LOI<0.01 represent a gain on ignition.

[2] Loss on Ignition adjusted for the difference in oxygen between FeO and Fe_2O_3.

Note: nd = not detected.

Table 5. Trace metal composition (ppm) of leached rock samples. The four metals of greatest environmental concern (Co, Cu, Ni, Zn) are listed first. Analyses by Actlabs, Inc. Page 1 of 4.

Original %S	0.18	0.22	0.40	0.41	0.51	0.54	0.54	0.57	0.57	0.58	0.71	1.12	1.12
Reactor	2	4	6	8	10	11	12	13	14	16	18	35	36
Co	59	67	42	62	57	38	42	65	69	44	70	42	47
Cu	560	1140	624	777	1200	159	175	1210	1360	624	>10000	745	877
Ni	220	277	206	304	183	91	116	251	323	125	246	93	174
Zn	145	120	258	106	104	209	289	216	84	109	200	261	233
Ag	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd
As	nd	nd	7	nd	nd	8	8	nd	nd	nd	7	9	10
Ba	278	330	266	199	241	362	367	262	265	282	229	405	400
Bi	1.2	1.6	1.0	0.4	0.9	nd	nd	1.5	0.6	0.8	2.8	0.5	0.8
Cr	162	190	257	207	139	428	455	159	166	159	196	177	172
Cs	1.1	1.3	1.7	0.9	1.3	2.1	3.1	1.4	1.3	1.4	1.1	2.0	2.0
Hf	5.1	5.0	2.6	4.1	5.1	2.1	2.1	6.3	5.2	4.4	4.2	6.1	5.8
Mo	nd	2	3	nd	2	3	3	2	2	2	nd	3	4
Pb	7	6	10	nd	6	nd	6	14	nd	6	11	10	7
Rb	20	34	18	18	26	35	38	25	27	28	21	49	45
Sb	2.7	0.7	nd	nd	0.8	0.9	1.0	0.5	0.7	0.5	nd	0.8	0.9
Sn	2	2	2	2	3	1	1	3	3	2	2	3	2
Sr	290	273	321	259	271	262	267	249	246	271	263	232	228
Ta	1.2	1.5	0.9	1.2	1.2	0.5	0.5	1.4	1.3	1.3	1.0	1.5	1.5
V	209	296	253	236	222	283	296	220	216	259	182	306	293
W	1	1	1	nd	1	nd	nd	1	2	1	1	1.5	1.5
Y	30	29	23	24	31	68	17	217	31	29	135	29	29
Zr	173	158	86	129	162	68	71	217	168	132	135	212	192
Ce	42.3	39.9	44.7	31.7	42.2	28.0	29.1	46.8	46.5	38.3	34.0	40.8	41.0

Table 5. Trace metal composition (ppm) of leached rock samples. The four metals of greatest environmental concern (Co, Cu, Ni, Zn) are listed first. Analyses by Actlabs, Inc. Page 2 of 4.

Original %S	1.16	1.16	1.40	1.40	1.44	1.63	1.64	1.64	2.06	3.12	3.72	5.44
Reactor	29	30	37	38	34	20	31	32	22	24	26	28
Co	86	71	51	73	68	49	45	52	86	25	22	13
Cu	1110	1160	1750	1920	1270	1060	1770	1970	2310	129	102	129
Ni	454	369	210	483	277	275	159	252	403	77	298	44
Zn	181	184	185	42	213	158	66	124	170	187	349	296
Ag	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd
As	nd	nd	nd	nd	9	7	6	6	5	8	20	13
Ba	214	207	194	192	250	300	233	250	228	372	422	465
Bi	1.2	1.6	2.4	0.6	0.8	0.9	1.0	1.8	1.5	0.5	0.6	0.4
Cr	227	184	223	149	172	254	162	163	167	327	754	178
Cs	1.6	1.5	1.3	1.2	1.7	2.6	1.5	1.7	1.7	6.2	7.7	9.0
Hf	4.8	4.8	5.9	4.9	4.5	5.2	5.5	5.8	4.0	2.6	2.9	3.1
Mo	nd	2	3	2	2	5	2	3	2	14	20	22
Pb	11	13	20	nd	12	7	nd	6	12	7	18	12
Rb	25	22	18	19	29	39	27	26	28	97	117	149
Sb	nd	0.5	0.6	0.9	0.7	0.7	0.6	0.7	nd	1.3	1.2	1.8
Sn	3	3	4	3	3	2	3	3	3	1	1	nd
Sr	271	270	242	236	244	229	238	246	237	171	146	136
Ta	1.2	1.2	1.5	1.5	1.2	1.4	1.4	1.4	1.1	0.5	0.6	0.5
V	206	200	229	219	206	316	227	229	194	279	231	248
W	nd	nd	nd	nd	nd	1	2	1	nd	2	3	3
Y	25	25	27	27	25	29	30	31	26	25	22	25
Zr	160	156	199	152	152	166	172	185	133	84	93	95
Ce	32.4	31.9	34.0	34.6	32.6	40.4	39.6	39.8	35.7	43.1	46.3	55.4

Table 5. Trace metal composition (ppm) of leached rock samples. Analyses by Actlabs, Inc. Page 3 of 4.

Original %S	0.18	0.22	0.40	0.41	0.51	0.54	0.54	0.57	0.57	0.58	0.71	1.12	1.12
Reactor	2	4	6	8	10	11	12	13	14	16	18	35	36
La	19.3	18.3	24.5	14.9	19.3	14.6	15.1	21.3	21.4	17.8	15.8	20.6	16.0
Lu	0.42	0.40	0.44	0.38	0.42	0.28	0.30	0.44	0.44	0.45	0.38	0.49	0.48
Nd	24.5	22.6	20.8	18.6	24.5	14.5	14.8	26.4	26.4	22.5	19.4	21.7	17.5
Tb	0.9	0.9	0.6	0.7	0.9	0.5	0.5	1.0	1.0	0.9	0.8	0.8	0.8
Th	1.8	2.6	1.5	1.7	2.3	2.9	1.8	2.5	2.2	1.7	1.7	3.0	3.0
U	0.6	0.7	0.6	0.5	0.8	0.4	0.4	0.7	0.7	0.6	0.5	1.0	1.2
Yb	2.7	2.6	2.6	2.3	2.9	1.8	1.8	2.9	2.9	2.9	2.4	3.0	3.1
Ga	21	21	26	21	21	19	20	21	21	20	21	22	21
Ge	2	2	2	2	2	2	2	2	2	2	3	2	2
Nb	17	18	12	15	16	7	7	18	18	17	13	20	19
In	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd
Pr	5.02	4.66	4.78	3.77	5.00	3.18	3.27	5.59	5.53	4.59	4.06	4.75	4.75
Sm	5.6	5.3	4.0	4.3	5.6	3.0	3.1	6.1	6.0	5.3	4.6	4.9	5.1
Gd	5.6	5.1	3.8	4.3	5.4	3.1	3.1	6.0	5.9	5.2	4.6	4.7	4.8
Dy	5.5	5.1	3.6	4.4	5.4	2.9	3.1	5.8	5.6	5.2	4..6	5.0	4.9
Ho	1.0	1.0	0.7	0.8	1.0	0.6	0.6	1.1	1.0	1.0	0.8	0.9	0.9
Er	2.9	2.7	2.2	2.4	2.9	1.8	1.8	3.0	3.0	2.8	2.4	2.8	2.9
Tm	0.42	0.40	0.35	0.36	0.44	0.25	0.26	0.44	0.44	0.44	0.38	0.44	0.44
Tl	0.2	0.2	0.2	0.1	0.1	0.1	0.2	0.2	0.1	0.1	0.2	0.3	0.3
Eu	2.04	1.99	2.42	1.78	2.01	1.55	1.62	2.00	1.98	2.07	1.77	2.10	2.11

Table 5. Trace metal composition (ppm) of leached rock samples. Analyses by Actlabs, Inc. Page 4 of 4.

Original %S	1.16	1.16	1.40	1.40	1.44	1.63	1.64	1.64	2.06	3.12	3.72	5.44
Reactor	29	30	37	38	34	20	31	32	22	24	26	28
La	15.2	18.3	15.8	16.0	16.0	19.4	18.3	18.4	16.8	23.6	24.0	28.4
Lu	0.41	0.41	0.43	0.44	0.41	0.47	0.47	0.49	0.38	0.46	0.38	0.42
Nd	18.3	18.1	19.6	19.2	17.5	22.6	22.2	22.6	20.2	21.1	21.6	27.2
Tb	0.7	0.7	0.8	0.8	0.7	0.9	0.9	0.9	0.8	0.6	0.6	0.7
Th	1.9	2.4	2.2	1.8	2.3	2.5	2.6	2.7	2.1	5.1	5.6	8.0
U	0.6	0.6	0.6	1.2	0.9	1.4	0.7	0.8	0.6	6.9	7.8	11.1
Yb	2.6	2.6	2.7	2.8	2.5	3.0	3.0	3.1	2.5	2.7	2.3	2.4
Ga	22	21	21	20	20	22	21	21	21	21	22	19
Ge	1	2	2	2	2	2	2	2	1	nd	1	nd
Nb	16	16	21	19	16	19	19	19	15	7	7	6
In	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd	nd
Pr	3.81	3.79	4.03	4.08	3.75	4.72	4.72	4.70	4.21	4.79	5.06	6.30
Sm	4.3	4.3	4.6	4.6	4.0	5.3	5.3	5.4	4.6	4.1	4.2	5.1
Gd	4.4	4.4	4.5	4.7	4.0	5.1	5.3	5.3	4.9	4.0	3.8	4.7
Dy	4.4	4.6	4.8	4.9	4.2	5.3	5.5	5.6	4.6	3.9	3.5	4.0
Ho	0.9	0.8	0.9	0.9	0.8	1.0	1.0	1.1	0.9	0.8	0.7	0.8
Er	2.5	2.5	2.7	2.7	2.4	3.0	3.0	3.0	2.5	2.5	2.0	2.2
Tm	0.40	0.37	0.40	0.42	0.36	0.46	0.45	0.45	0.38	0.40	0.34	0.37
Tl	0.2	0.2	0.2	nd	0.2	0.5	nd	0.1	0.3	1.8	2.9	3.3
Eu	1.91	1.89	1.87	1.88	1.71	2.08	1.98	1.98	1.88	1.48	1.48	1.35

Note: nd = not detected.

Table 6. Leachate sampling frequency for reactors. Column entries indicate sampling frequency and number of weeks sampled at that frequency for each analysis.

Reactor	Alkalinity/Acidity	pH/Specific Conductance	Sulfate	Calcium/Magnesium	Trace Metals	Sodium, Potassium, Iron, Aluminum, Silicon
1-20	Bi-monthly: 0-84 Monthly: 85-577 Bi-monthly: 578-1252	Weekly: 0-523 Monthly: 524-577 Bi-monthly: 578-1252	Weekly: 0-112 Monthly: 113-585 Bi-monthly: 586-1252	Bi-monthly: 0-20 Monthly: 21-625 Bi-monthly: 626-1252	Every three months: 12-60 Every 16-20 weeks: 403-553, 633-1252 Other: approximately 17 times irregularly in other weeks	Similar to Trace Metals starting in week 335
21-28	Monthly: 0-12 Every two months: 13-76	Weekly: 0-78	Bi-monthly: 0-68, 72-78	Bi-monthly: 4-34, 40-62 Other weeks: 0, 70	Bi-monthly: 4-62 Other: weeks 0, 70	None
29-38	Monthly: 0-609 Bi-monthly: 610-866	Weekly: 0-444 Monthly: 445-609 Bi-monthly: 610-866	Monthly: 0-609 Bi-monthly: 610-866	Monthly: 0-609 Bi-monthly: 610-866	Bi-monthly: 9-17 Other: 46 samples irregularly spread through period of record	Similar to Trace Metals starting in week 254
39-44	Monthly: 0-246 Bi-monthly: 247-809	Weekly: 0-81 Monthly: 82-246 Bi-monthly: 247-809	Monthly: 0-246 Bi-monthly: 247-809	Monthly: 0-246 Bi-monthly: 247-809	23 samples taken 8-36 weeks apart	Similar to Trace Metals

Table 7. Reactor mineral composition (%) before leaching. All determined by point count, with the exception of pyrrhotite, chalcopyrite and pentlandite, which were chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Numbers in parentheses following chalcopyrite are Cu ppm and following pentlandite are Ni ppm. Page 1 of 2.

	%S	0.18	0.22	0.4	0.41	0.51	0.54	0.57	0.58	0.67[1]	0.67[2]	0.71	0.72	0.82
Silicate														
Plagioclase		49.5	64.9	47.4	57.4	46.6	62.2	42.6	45.8	56.3	47.7	54.8	61.2	47.8
Hypersthene		17.4	5.4	27.6	8.7	17.8	19.8	5	15	4.2	14.4	12.2	3.3	15.7
Olivine		10.1	4.5	0	13	5.9	0	20.8	2.5	16.8	4.5	14.8	21.5	3.5
Augite		10.1	8.1	4.3	8.7	7.6	8.1	16.8	14.2	14.3	11.7	7.8	2.5	13
Biotite		1.8	2.7	1.7	2.6	2.5	2.7	0	0.8	0	6.3	1.7	0	3.5
Prehnite		1.8	0.9	1.7	0	0	0.9	0	0	0	0	0	4.1	0
Amphibole		0	0.9	0.9	0.9	1.7	0.9	0	0	0	0	0	0	0
K-Spar		0	3.6	3.4	0.9	5.1	0	4	5	1.7	0.9	0.9	0	0
Ilmenite		6.4	6.3	3.4	7.8	9.3	0	5.9	7.5	3.4	4.5	2.6	1.7	2.6
Quartz		0.9	0.9	2.6	0	0.8	1.8	1	0.8	0	1.8	0	0.8	0
Cordierite		0	0	3.4	0	0	0.9	0	0.8	0	7.2	0	0	5.2
Chlorite		0.9	0.8	0	0	0	0	0	0	0	0	0	4.1	2.6
Apatite		0	0	0	0	0	0	3	1.7	0	0	0.9	0	1.7
Stilpnomelane		0	0	0	0	0	0	0	0	0	0	0	0	0
Muscovite		0	0	0	0	0	0	0	0	0	0.9	0	0	0
Magnetite		0	0	0	0	0	0	1.0	0	0.8	0	0	0	0
Calcite		0	0	0	0	0	0	0	0.8	0	0	0	0	0
Garnet		0	0	0.9	0	0	0	0	0	0	0	0	0	0
Sillimanite		0	0	0	0	0	0	0	0	0	0	0	0	0
Scapolite		0	0	0	0	0	0	0	0.8	0	0	0	0	0
Sulfide														
Pyrrhotite		0.02	0.13	0.83	0.72	0.84	1.26	1.04	1.06	1.41	1.5	1.42	1.62	1.86
Chalcopyrite		0.39 (1360)	0.39 (1350)	0.15 (512)	0.23 (788)	0.41 (1430)	0.06 (211)	0.37 (1290)	0.37 (1300)	0.18 (617)	0.14 (490)	0.33 (1160)	0.15 (517)	0.17 (601)
Pentlandite		0.15 (525)	0.14 (484)	0.07 (231)	0.15 (511)	0.14 (473)	0.05 (174)	0.11 (398)	0.11 (396)	0.15 (523)	0.07 (261)	0.11 (388)	0.07 (254)	0.06 (204)
Pyrite		0	0	0.9	0	0	0	0	1.7	0	0	0	0	0
Cubanite		0	0	0	0	0	0	0	0	0	0	0	0	0
Unknown		0	0	0	0	0	0	0	0	0	0	0	0	0

Table 7. Reactor mineral composition (%) before leaching. All determined by point count, with the exception of pyrrhotite, chalcopyrite and pentlandite, which were chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Numbers in parentheses following chalcopyrite are Cu ppm and following pentlandite are Ni ppm. Page 2 of 2.

	%S	0.92	1.12	1.16	1.40	1.44	1.63	1.64	1.71	2.06	3.12	3.72	5.44
Silicate													
Plagioclase		44.6	39.7	58.4	46.2	52.9	36.4	45.8	45.5	27.7	34.8	18	11.5
Hypersthene		3.3	19.8	12.4	8.4	24.4	17.8	10.8	2.5	10.9	11.6	9.6	0
Olivine		38.8	4.1	16.8	5.9	5.9	5.6	3.3	19.8	0.8	0	0	0
Augite		4.1	5.8	9.7	11.8	2.5	5.6	13.3	2.5	2.5	0.9	2.6	0
Biotite		0	1.7	0	0	1.7	1.9	2.5	0.8	0.8	2.7	2.6	0.9
Prehnite		0	0.8	0	2.5	0	0	0.8	0.8	0	0	0.9	0
Amphibole		3.3	0.8	0	0.8	0	0.9	0	0	0	0	0	0
K-Spar		0	11.6	0	2.5	4.2	5.6	4.2	0.8	12.6	8	15	20.4
Ilmenite		2.5	7.4	1.8	8.4	3.4	8.4	8.3	5	4.2	0.9	0	0
Quartz		0	0	0	0	1.7	5.6	0	0	14.3	20.5	24	34.5
Cordierite		0	0.8	0	0	0	0.9	0.8	3.3	20.2	0	4.3	0
Chlorite		0.8	0	0	0	0	0	1.7	8.3	0	0	0.9	0
Apatite		0	0	0	0.8	0	4.7	0.8	0	0	0	0	0
Stilpnomelane		0	0	0	0	0	0	0	0	0	4.5	6.1	0
Muscovite		0	0.8	0	0	0	0	0.8	0	0	2.7	1.7	1.8
Magnetite		0	0	0	1.7	0.8	0	2.5	0	0	0	0	0
Calcite		0	0	0	0	0	0	0	0	0	1.8	0	0
Garnet		0	0	0	0	0	0	0.8	0	0	0	0	0
Sillimanite		0	0	0	0	0	0	0	0	0	0	0	0.9
Scapolite		0	0	0	0	0	0	0	0	0	0	0	0
Sulfide													
Pyrrhotite		2.05	2.54	2.55	3.02	3.23	3.69	3.42	4.04	4.78	7.35	8.9	13.2
Chalcopyrite		0.21 (752)	0.23 (800)	0.31 (1090)	0.45 (1560)	0.35 (1210)	0.34 (1180)	0.63 (2220)	0.19 (660)	0.37 (1300)	0.44 (1540)	0.4 (1280)	0.37 (1290)
Pentlandite		0.09 (324)	0.09 (329)	0.12 (422)	0.14 (503)	0.11 (386)	0.14 (4.92)	0.21 (718)	0.1 (354)	0.09 (327)	0.12 (436)	0.2 (540)	0.17 (589)
Pyrite		0	0.8	0	0	0	0	1.7	0	0	4.5	0.9	1.8
Cubanite		0.8	0	0.9	0	0	1.9	0	0.8	0	0	0	0
Unknown		0	0	0	0	0	0	0	0	0	0	0	15

[1] Reactor 39
[2] Reactor 40

Table 8. Reactor mineral composition (%) after leaching. All determined by point count, with the exception of pyrrhotite, chalcopyrite and pentlandite, which were chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Numbers in parentheses following chalcopyrite are Cu ppm and following pentlandite are Ni ppm. Only minerals detected are listed.

	%S	0.40	0.41	0.51	0.58[1]	0.58[2]	0.71	1.40	1.44	1.63	1.71
Silicate											
Plagioclase		46.7	52.1	44	42.5	54.1	48.6	55.6	45.9	52.3	47.4
Hypersthene		27	8.5	17.4	14.2	6.6	14.4	8.3	16.2	5.6	12.3
Olivine		0	12	10.1	1.8	8.2	20.7	6.5	9.9	1.9	21.9
Augite		5.8	12	11	15	13.1	10.8	17.6	12.6	13.1	0
Biotite		0	0	0	3.5	4.9	0.9	0.9	0.9	0.9	0
Prehnite		0	0	0	0	0	0	0.9	0	0	0
Amphibole		0	2.6	0.9	0	0	0	0	0	0	0
K-Spar		2.2	0.9	1.8	2.7	6.6	0	0.9	6.3	10.3	2.6
Ilmenite		6.6	6.8	11.9	11.5	3.3	1.8	9	5.4	8.4	4.4
Apatite		0	1.7	14.8	1.8	1.6	0	7.4	0	0	0
Cordierite		6.6	0	0	3.5	0	0	0	0	0	2.6
Quartz		0.7	0	0	1.8	0	0	0	0	7.5	0
Magnetite		0.7	0.9	0	0.9	0	0.9	1	0.9	0	0
Chlorite		0	0	0	0	0	0	0	0	0	4.4
Scapolite		0	1.7	0	0	0	0	0	0	0	0
Muscovite		0.7	0	0.9	0	0	0	0	0	0	0
Sulfide											
Pyrrhotite		0.68	0.29	0.27	0.36	0.38	0.05	0.28	1.3	0.84	4.4[4]
Chalcopyrite		0.18 (624)	0.22 (777)	0.34 (1200)	0.39 (624)[3]	0.18 (624)	0.86 (>10000)	0.5 (1920)	0.36 (1270)	0.3 (1060)	0[4]
Pentlandite		0.06 (206)	0.09 (304)	0.05 (183)	0.09 (125)[3]	0.04 (125)	0.07 (246)	0.06 (483)	0.08 (277)	0.08 (275)	0[4]
Pyrite		1.5	0	0	0	0	0	0	0	0	0
Cubanite		0	0.9	0	0.9	0	0.9	0	0	0	0

[1] Reactor 15, Cu and Ni analyses from Reactor 16
[2] Reactor 16
[3] Cu and Ni analyses from Reactor 16
[4] The 1.71%S sample had all values determined by point count.

Table 9. Mineral content calculated based on S, Cu, and Ni content from Actlabs (see calculation methods below). These samples leached for 144-328 weeks prior to termination.

Reactor	5	7	9	15[1]	16[1]	17	37	33	19	31[2]	Average[3]
Before leaching											
S	0.40	0.41	0.51	0.58	0.58	0.71	1.40	1.44	1.63	1.71	0.94
Cu	512	788	1430	1300	1300	1160	1560	1210	1180	660	1110
Ni	231	511	473	396	396	388	503	386	492	354	413
After leaching											
S	0.35	0.21	0.23	0.22	0.22	0.32	0.29	0.66	0.46	N/A	0.33
Cu	624	777	1200	624	624	>10000[4]	1920	1270	1060	N/A	1012
Ni	206	304	183	125	125	246	483	277	275	N/A	247

Reactor	5	7	9	15[1]	16[1]	17	37	33	19	31[2]	Average[3]
Before leaching											
Pyrrhotite	0.84	0.72	0.82	1.05	1.05	1.43	3.06	3.29	3.76	4.13	2.01
Chalcopyrite	0.15	0.23	0.41	0.38	0.38	0.34	0.45	0.35	0.34	0.19	0.32
Pentlandite	0.07	0.15	0.14	0.12	0.12	0.11	0.15	0.11	0.14	0.10	0.12
After leaching											
Pyrrhotite	0.68	0.27	0.24	0.37	0.37	-1.84[4]	0.13	1.30	0.83	N/A	0.53
Chalcopyrite	0.18	0.22	0.35	0.18	0.18	2.89[4]	0.55	0.37	0.31	N/A	0.29
Pentlandite	0.06	0.09	0.05	0.04	0.04	0.07	0.14	0.08	0.08	N/A	0.07
Final/Initial Ratio											
Pyrrhotite	0.82	0.38	0.29	0.35	0.35	-1.29	0.04	0.40	0.22	N/A	0.36
Chalcopyrite	1.22	0.99	0.84	0.48	0.48	8.62[4]	1.23	1.05	0.90	N/A	0.90
Pentlandite	0.89	0.59	0.39	0.32	0.32	0.63	0.96	0.72	0.56	N/A	0.60

[1] Both 0.58%S samples analyzed for solids
[2] No S, Cu, Ni content for leached 1.71%S sample. Sample omitted from average calculations.
[3] Average does not include 0.71%S sample
[4] Anomalous Cu content omitted Po and Cp average calculations

Pyrrhotite: S with no Cu and Ni is $Fe_{0.9}S$. Then {S - [Cu(2x32.06)/63.5 + Ni(8x32.06)/(4.5 x 58.7]} x (0.9 x 55.8 + 32.06)/32.06
Chalcopyrite: Assume all Cu with $CuFeS_2$. Then $CuFeS_2$ = [(2 x 32.06 + 63.5 + 55.8)/63.5] x % Cu = 2.89 x %Cu
Pentlandite: Assume all Ni with $(Ni, Fe)_9S_8$. Then $(Ni,Fe)_9S_8$ = [(4.5 x 58.7 + 4.5 x 55.8 + 8 x 32.06)/(4.5 x 58.7] x % Ni = 2.92 x %Ni

Table 10. Unleached sulfide mineral chemistry determined by microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples where a grain was identified by point count had analyses to determine stoichiometric ratios. Chemical percent abundance was chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine the composition of the minerals in this study, substitute the mean sample values for each element into the stoichiometric formula. Page 1 of 4.

Pyrrhotite – $Fe_{7.18}S_8$

Sulfur %	Reactor	%, chemical	%, point count	N	Fe atoms	Ni atoms	Cu atoms
Duluth Complex							
0.18	1,2	0.02	0.9	1	7.007	0.031	0.033
0.221	3,4	0.13					
0.4	5,6	0.83	1.7	3	7.27	0.035	0.028
0.41	7,8	0.72					
0.51	9,10	0.84	2.5	2	7.041	0.062	0.011
0.54	11,12	1.26	1.8	2	7.017	0.016	0.019
0.57	13,14	1.04					
0.58	15,16	1.06	2.5	3	7.933	0.036	0.039
0.67	39	1.5	0.8	1	6.952	0.007	0.05
0.671	40	1.5					
0.71	17,18	1.42	4.3	5	7.222	0.019	0.008
0.72	42	1.62					
0.82	43	1.86	0.9	1	6.911	0.028	0.054
0.92	41	2.05	0.8	1	6.985	0.018	0.037
1.12	35,36	2.54	5	6	7.093	0.026	0.038
1.16	29,30	2.55					
1.4	37,38	3.02	7.6	9	7.669	0.038	0.013
1.44	33,34	3.23	1.7	2	7.096	0.047	0.007
1.63	19,20	3.69	4.7	6	7.222	0.052	0.026
1.64	31,32	3.42	1.7	2	7.04	0.02	0.075
1.71	44	4.04	9.1	11	7.27	0.037	0.007
	Mean				7.182	0.031	0.03
	Max				7.933	0.062	0.075
	Min				6.911	0.007	0.007
	St. Dev.				0.28	0.015	0.02
Virginia Formation							
2.06	21,22	4.78	5	6	7.144	0.029	0.029
3.12	23,24	7.35	7.1	8	7.095	0.02	0.052
3.72	25,26	8.88	13.9	16	7.526	0.012	0.031
5.44	27,28	13.17	13.3	15	7.489	0.01	0.029
	Mean				7.314	0.018	0.035
	Max				7.526	0.029	0.052
	Min				7.095	0.01	0.029
	St. Dev.				0.225	0.009	0.011

Table 10. Unleached sulfide mineral chemistry determined by microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples where a grain was identified by point count had analyses to determine stoichiometric ratios. Chemical percent abundance was chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine the composition of the minerals in this study, substitute the mean sample values for each element into the stoichiometric formula. Page 2 of 4.

Chalcopyrite - $(Cu_{4.29}Fe_{4.47})S_8$

Sulfur %	Reactor	%, chemical	%, point count	N	Fe atoms	Ni atoms	Cu atoms
Duluth Complex							
0.181	1,2	0.39					
0.221	3,4	0.39					
0.4	5,6	0.15					
0.41	7,8	0.23					
0.51	9,10	0.41					
0.54	11,12	0.06	0.9	1	4.41	0.002	4.204
0.57	13,14	0.37					
0.58	15,16	0.37					
0.671	39	0.18	1.7	2	4.315	0.007	4.163
0.67	40	0.14					
0.71	17,18	0.33					
0.72	42	0.15	0.8	1	4.363	0.012	4.249
0.822	43	0.17	3.5	4	4.79	0.021	1.733
0.92	41	0.21					
1.12	35,36	0.23	0.8	1	4.298	0	4.281
1.16	29,30	0.31					
1.4	37,38	0.45	2.5	3	4.986	0.005	4.659
1.44	33,34	0.35	0.8	1	4.471	0	4.24
1.63	19,20	0.34					
1.64	31,32	0.63					
1.71	44	0.19	0.8	1	4.439	0	4.213
	Mean				4.469	0.004	4.287
	Max				4.986	0.021	4.659
	Min				4.315	0	4.163
	St. Dev.				0.237	0.005	0.168
Virginia Formation							
2.06	21,22	0.37	0.8	1	4.366	0	4.174
3.12	23,24	0.44					
3.72	25,26	0.37					
5.44	27,28	0.37					

Table 10. Unleached sulfide mineral chemistry determined by microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples where a grain was identified by point count had analyses to determine stoichiometric ratios. Chemical percent abundance was chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine the composition of the minerals in this study, substitute the mean sample values for each element into the stoichiometric formula. Page 3 of 4.

Pentlandite - $(Ni_{4.89}Fe_{4.88})S_8$

Sulfur %	Reactor	%, chemical	%, point count	N	Fe atoms	Ni atoms	Cu atoms
Duluth Complex							
0.181	1,2	0.15					
0.221	3,4	0.14					
0.4	5,6	0.07					
0.41	7,8	0.15					
0.51	9,10	0.14					
0.541	11,12	0.05					
0.57	13,14	0.11					
0.58	15,16	0.11					
0.67	39	0.15					
0.671	40	0.07					
0.71	17,18	0.11					
0.72	42	0.07					
0.82	43	0.06					
0.92	41	0.09	0.8	1	4.385	4.609	0.016
1.12	35,36	0.09					
1.16	29,30	0.12					
1.4	37,38	0.14	0.8	1	5.366	5.162	0.043
1.44	33,34	0.11					
1.63	19,20	0.14					
1.64	31,32	0.21					
1.71	44	0.1					
	Mean				4.876	4.886	0.03
	Max				5.366	5.162	0.043
	Min				4.385	4.609	0.016
	St. Dev.				0.694	0.391	0.019
Virginia Formation							
2.06	21,22	0.09					
3.12	23,24	0.12					
3.72	25,26	0.15					
5.44	27,28	0.17					

Table 10. Unleached sulfide mineral chemistry determined by microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine the composition of the minerals in this study, substitute the mean sample values for each element into the stoichiometric formula. Page 4 of 4.

Pyrite - $Fe_{4.28}S_8$

Sulfur %	Reactor	%, point count	N	Fe atoms	Ni atoms	Cu atoms
Duluth Complex						
0.4	5,6	1.5	1	4.296	0.023	0.001
0.58	15,16	1.7	2	4.443	0.027	0.026
1.12	35,36	0.8	1	4.061	0.02	0
1.64	31,32	1.7	2	4.299	0.073	0.231
	Mean			4.275	0.036	0.065
	Max			4.443	0.073	0.231
	Min			4.061	0.001	0
	St. Dev.			0.158	0.025	0.112
Virginia Formation						
3.12	23,24	4.5	5	4.072	0.012	0.007
3.72	25,26	0.9	1	4.356	0.001	0
5.44	27,28	1.8	2	4.546	0.014	0.02
	Mean			4.325	0.009	0.009
	Max			4.546	0.014	0.02
	Min			4.072	0.001	0
	St. Dev.			0.134	0.009	0.014

Cubanite - $Cu_{2.77}Fe_{5.59}S_8$

Sulfur %	Reactor	%, point count	N	Fe atoms	Ni atoms	Cu atoms
Duluth Complex						
0.92	41	0.8	1	5.487	0	2.705
1.16	29,30	0.9	1	5.599	0	2.862
1.63	19,20	1.9	1	5.675	0.019	2.729
1.71	44	0.8	1	5.607	0	2.79
	Mean			5.592	0.005	2.772
	Max			5.675	0.019	2.862
	Min			5.487	0	2.705
	St. Dev.			0.078	0.01	0.07

Note: N = number of samples analyzed for each Sulfur %

Note: Pyrrhotite, chalcopyrite and pentlandite percentages were determined using calculations from bulk chemistry. All other mineral percentages were determined using point counts.

[1] No chemical analysis performed

[2] These samples were listed as chalcopyrite in the McSwiggen analysis. However, due to the large deviation in expected Cu stoichiometry, they were not included in the calculation.

Table 11. Unleached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 1 of 8.

Plagioclase - $(Ca_{0.51}Na_{0.45})Al_{1.54}Si_{2.46}O_8$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.18	1,2	49.5	54	0.471	0.002	0.032	0.468	2.505	1.501	0.012	0.001
0.22	3,4	64.9	72	0.446	0.002	0.033	0.521	2.532	1.46	0.017	0.002
0.4	5,6	47.4	55	0.537	0.001	0.026	0.422	2.441	1.563	0.009	0
0.41	7,8	57.4	65	0.525	0.002	0.019	0.43	2.45	1.554	0.008	0.001
0.51	9,10	46.6	55	0.51	0.003	0.02	0.459	2.472	1.524	0.012	0.001
0.54	11,12	62.2	59	0.628	0.012	0.017	0.318	2.347	1.655	0.014	0
0.57	13,14	42.6	43	0.473	0.004	0.022	0.462	2.497	1.51	0.02	0.001
0.58	15,16	45.8	55	0.492	0.002	0.022	0.453	2.483	1.522	0.016	0.002
0.67	39	56.3	67	0.53	0.003	0.021	0.472	2.417	1.578	0.016	0.001
0.67	40	47.7	53	0.514	0.001	0.015	0.48	2.495	1.49	0.01	0.001
0.71	17,18	54.8	63	0.507	0.005	0.023	0.452	2.47	1.527	0.017	0.001
0.72	42	61.2	74	0.602	0.002	0.015	0.393	2.38	1.611	0.009	0.003
0.82	43	47.8	55	0.508	0.013	0.019	0.425	2.456	1.551	0.017	0.001
0.92	41	44.6	54	0.568	0.002	0.024	0.387	2.423	1.578	0.01	0.001
1.12	35,36	39.7	48	0.49	0.002	0.018	0.462	2.501	1.5	0.014	0.001
1.16	29,30	58.4	66	0.53	0.003	0.02	0.436	2.449	1.546	0.017	0.002
1.4	37,38	46.2	55	0.501	0.003	0.018	0.445	2.477	1.526	0.018	0.002
1.44	33,34	52.9	61	0.48	0.002	0.024	0.459	2.473	1.538	0.02	0.002
1.63	19,20	36.4	19	0.396	0.003	0.02	0.55	2.568	1.444	0.011	0.001
1.64	31,32	45.8	55	0.478	0.002	0.018	0.478	2.513	1.485	0.016	0.001
1.71	44	45.5	53	0.555	0.011	0.027	0.374	2.394	1.617	0.017	0.002
	Mean			0.514	0.004	0.022	0.445	2.464	1.537	0.014	0.001
	Max			0.628	0.013	0.033	0.55	2.568	1.655	0.02	0.003
	Min			0.396	0.001	0.015	0.318	2.347	1.444	0.008	0
	St. Dev.			0.052	0.004	0.005	0.05	0.052	0.052	0.004	0.001
Virginia Formation											
2.06	21,22	27.7	31	0.325	0.003	0.02	0.625	2.665	1.339	0.009	0.001
3.12	23,24	34.8	39	0.561	0.001	0.029	0.381	2.411	1.6	0.008	0.001
3.72	25,26	18.3	21	0.407	0.001	0.02	0.583	2.56	1.441	0.007	0
5.44	27,28	11.5	13	0.322	0.001	0.031	0.663	2.645	1.351	0.01	0.001
	Mean			0.404	0.002	0.025	0.563	2.57	1.433	0.009	0.001
	Max			0.561	0.003	0.029	0.663	2.665	1.6	0.01	0.001
	Min			0.322	0.001	0.02	0.381	2.411	1.339	0.007	0
	St. Dev.			0.112	0.001	0.006	0.126	0.116	0.12	0.001	0.001

Table 11. Unleached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 2 of 8.

Hypersthene - $(Mg_{1.06}Fe_{0.89})_2Si_{1.99}O_6$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.18	1,2	17.4	19	0.048	1.018	0.001	0.001	1.991	0.027	0.874	0.007
0.22	3,4	5.4	6	0.06	1.03	0.001	0.002	1.979	0.027	0.876	0.007
0.4	5,6	27.6	32	0.022	0.98	0	0.002	1.967	0.057	0.943	0.007
0.41	7,8	8.7	10	0.066	1.074	0.001	0.001	1.988	0.032	0.806	0.005
0.51	9,10	17.8	21	0.08	0.978	0	0.002	1.985	0.029	0.896	0.005
0.54	11,12	19.8	22	0.031	1.123	0.001	0.001	1.997	0.027	0.783	0.006
0.57	13,14	5	5	0.04	1.106	0	0	1.985	0.029	0.812	0.005
0.58	15,16	15	16	0.045	0.936	0.001	0	1.98	0.026	0.985	0.007
0.67	39	4.2	5	0.029	1.233	0	0	1.987	0.026	0.703	0.003
0.67	40	14.4	16	0.042	0.947	0.001	0.001	1.979	0.049	0.946	0.006
0.71	17,18	12.2	14	0.049	1.118	0	0.002	1.985	0.031	0.785	0.007
0.72	42	3.3	4	0.035	1.304	0	0.001	1.977	0.029	0.63	0.006
0.82	43	15.7	18	0.031	1.161	0.001	0.001	1.987	0.043	0.836	0.006
0.92	41	3.3	4	0.045	1.187	0	0	1.998	0.021	0.709	0.004
1.12	35,36	19.8	24	0.043	0.907	0	0.001	1.998	0.023	0.987	0.005
1.16	29,30	12.4	14	0.048	1.044	0.001	0.001	1.985	0.027	0.866	0.006
1.4	37,38	8.4	10	0.043	1.029	0.001	0.001	1.98	0.027	0.897	0.004
1.44	33,34	24.4	29	0.04	1.064	0.001	0.056	1.98	0.025	1.661	0.006
1.63	19,20	17.8	19	0.037	0.94	0.001	0.001	1.982	0.03	0.983	0.005
1.64	31,32	10.8	13	0.046	0.979	0	0.001	1.994	0.024	0.916	0.006
1.71	44	2.5	3	0.051	1.177	0	0	1.965	0.034	0.744	0.015
	Mean			0.044	1.064	0.001	0.004	1.984	0.031	0.888	0.006
	Max			0.08	1.304	0.001	0.056	1.998	0.057	1.661	0.015
	Min			0.022	0.907	0	0	1.965	0.021	0.63	0.003
	St. Dev.			0.013	0.107	0.001	0.012	0.009	0.009	0.203	0.002
Virginia Formation											
2.06	21,22	10.9	13	0.014	0.949	0.001	0.001	1.973	0.062	0.961	0.009
3.12	23,24	11.6	13	0.024	1.112	0.002	0.001	1.982	0.057	0.777	0.006
3.72	25,26	9.6	11	0.019	1.033	0.001	0.001	1.964	0.064	0.896	0.005
	Mean			0.019	1.031	0.001	0.001	1.973	0.061	0.878	0.007
	Max			0.024	1.112	0.002	0.001	1.982	0.064	0.961	0.009
	Min			0.014	0.949	0.001	0.001	1.964	0.057	0.777	0.005
	St. Dev.			0.005	0.082	0.001	0	0.009	0.004	0.093	0.002

Table 11. Unleached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 3 of 8.

Olivine - $(Mg_{0.93}Fe_{1.05})Si_{1.00}O_4$											
Sulfur %	**Reactor**	**%Olivine**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.18	1,2	10.1	10	0.003	0.763	0	0.001	1.01	0.002	1.194	0.001
0.22	3,4	4.5	5	0.002	0.857	0	0.001	0.987	0	1.147	0.001
0.41	7,8	13	15	0.002	0.897	0	0.001	1.003	0	1.074	0.001
0.51	9,10	5.9	7	0.001	0.812	0	0.001	1.001	0	1.168	0.001
0.57	13,14	20.8	21	0.001	1.003	0	0.001	1.008	0.001	0.96	0.001
0.58	15,16	2.5	3	0.002	0.851	0	0	0.997	0.001	1.135	0
0.67	39	16.8	20	0.001	1.09	0	0	1.004	0	0.881	0.002
0.67	40	4.5	5	0.002	0.785	0	0.001	1.009	0	1.178	0
0.71	17,18	14.8	17	0.002	0.958	0	0.001	1.005	0.001	1.01	0.001
0.72	42	21.5	26	0.002	1.233	0.001	0	0.996	0	0.751	0.002
0.82	43	3.5	4	0.001	0.8	0.001	0.001	1.009	0.001	1.156	0.001
0.92	41	38.8	47	0.003	1.035	0.001	0	1.009	0.008	0.914	0.001
1.12	35,36	4.1	5	0.002	0.953	0	0	1.007	0.001	1.013	0
1.16	29,30	16.8	19	0.002	0.948	0	0.001	1.004	0.001	1.023	0.001
1.4	37,38	5.9	7	0.002	0.954	0	0.001	0.992	0	1.038	0.001
1.44	33,34	5.9	7	0.002	0.953	0	0.002	0.999	0.003	1.023	0.001
1.63	19,20	5.6	6	0.002	0.722	0	0.002	1	0.001	1.248	0.002
1.64	31,32	3.3	4	0.002	0.828	0	0	1.008	0	1.128	0.001
1.71	44	19.8	24	0.002	1.139	0	0.001	0.997	0.001	0.849	0.001
	Mean			0.002	0.925	0	0.001	1.002	0.001	1.047	0.001
	Max			0.003	1.233	0.001	0.002	1.01	0.008	1.248	0.002
	Min			0.001	0.722	0	0	0.987	0	0.751	0
	St. Dev.			0.001	0.135	0	0.001	0.006	0.002	0.132	0.001
Virginia Formation											
2.06	21,22	0.8	1	0.002	0.894	0	0	1.014	0	1.058	0.001

Table 11. Unleached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 4 of 8.

Augite - $(Ca_{0.80}Na_{0.02})(Mg_{0.71}Fe_{0.40}Ti_{0.01})(Si_{1.97}Al_{0.07})O_6$											
Sulfur %	**Reactor**	**%Augite**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.18	1,2	10.1	11	0.789	0.73	0	0.014	1.979	0.058	0.398	0.009
0.22	3,4	8.1	9	0.816	0.717	0	0.023	1.962	0.068	0.393	0.012
0.4	5,6	4.3	6	0.945	0.362	0	0.008	1.971	0.043	0.626	0.009
0.41	7,8	8.7	10	0.787	0.715	0.011	0.024	1.952	0.09	0.393	0.018
0.51	9,10	7.6	9	0.814	0.709	0.001	0.014	1.968	0.056	0.415	0.01
0.54	11,12	8.1	9	0.861	0.789	0.003	0.01	2	0.031	0.273	0.007
0.57	13,14	16.8	17	0.792	0.706	0.001	0.016	1.971	0.075	0.398	0.013
0.58	15,16	14.2	17	0.756	0.694	0.001	0.019	1.967	0.072	0.461	0.012
0.67	39	14.3	19	0.755	0.697	0	0.022	1.982	0.06	0.444	0.012
0.67	40	11.7	13	0.823	0.665	0.002	0.017	1.968	0.063	0.428	0.014
0.71	17,18	7.8	9	0.795	0.78	0	0.02	1.963	0.07	0.349	0.011
0.72	42	2.5	3	0.76	0.839	0.002	0.015	1.949	0.083	0.313	0.021
0.82	43	13	15	0.789	0.712	0.004	0.029	1.953	0.094	0.387	0.018
0.92	41	4.1	5	0.822	0.766	0.001	0.018	1.958	0.083	0.317	0.016
1.12	35,36	5.8	7	0.814	0.613	0.001	0.011	1.989	0.043	0.49	0.009
1.16	29,30	9.7	11	0.822	0.745	0	0.02	1.957	0.075	0.358	0.013
1.4	37,38	11.8	14	0.793	0.737	0.001	0.022	1.951	0.079	0.397	0.015
1.44	33,34	2.5	3	0.761	0.773	0.001	0.024	1.949	0.095	0.362	0.019
1.63	19,20	5.6	6	0.822	0.653	0.001	0.012	1.985	0.046	0.45	0.008
1.64	31,32	13.3	16	0.811	0.686	0.001	0.019	1.962	0.071	0.422	0.013
1.71	44	2.5	3	0.767	0.741	0.001	0.019	1.97	0.066	0.397	0.015
	Mean			0.804	0.706	0.002	0.018	1.967	0.068	0.403	0.013
	Max			0.945	0.839	0.011	0.029	2	0.095	0.626	0.021
	Min			0.755	0.362	0	0.008	1.949	0.031	0.273	0.007
	St. Dev.			0.042	0.093	0.002	0.005	0.014	0.017	0.073	0.004
Virginia Formation											
2.06	21,22	2.5	3	0.808	0.589	0	0.009	1.981	0.051	0.526	0.009
3.12	23,24	0.6	1	0.966	0.343	0.002	0.002	1.987	0.044	0.616	0.005
3.72	25,26	2.6	3	0.821	0.742	0.026	0.016	1.974	0.104	0.284	0.007
	Mean			0.865	0.558	0.009	0.009	1.981	0.066	0.475	0.007
	Max			0.966	0.742	0.026	0.016	1.987	0.104	0.616	0.009
	Min			0.808	0.343	0	0.002	1.974	0.044	0.284	0.005
	St. Dev.			0.088	0.201	0.014	0.007	0.007	0.033	0.172	0.002

Table 11. Unleached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 5 of 8.

Potassium-Feldspar - $K_{0.77}Al_{1.05}Si_{2.96}O_8$											
Sulfur %	**Reactor**	**%K-Spar**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.22	3,4	3.6	4	0.014	0.001	0.778	0.195	2.973	1.023	0.013	0.001
0.40	5,6	3.4	3	0.082	0.000	0.721	0.139	2.815	1.229	0.009	0.002
0.41	7,8	0.9	1	0.010	0.000	0.882	0.097	2.972	1.031	0.008	0.000
0.51	9,10	5.1	6	0.013	0.001	0.781	0.209	2.967	1.027	0.012	0.002
0.57	13,14	4.0	4	0.006	0.000	0.877	0.121	2.971	1.026	0.011	0.001
0.58	15,16	5.0	6	0.023	0.000	0.791	0.161	2.948	1.060	0.010	0.003
0.67	39	1.7	2	0.026	0.001	0.754	0.202	2.957	1.047	0.002	0.003
0.67	40	0.9	1	0.009	0.000	0.867	0.145	2.963	1.037	0.004	0.000
0.71	17,18	0.9	1	0.003	0.000	0.772	0.218	2.987	1.008	0.013	0.000
1.12	35,36	11.6	14	0.013	0.001	0.738	0.240	2.972	1.029	0.007	0.001
1.40	37,38	2.5	3	0.021	0.000	0.757	0.206	2.960	1.041	0.014	0.001
1.44	33,34	4.2	5	0.017	0.000	0.800	0.163	2.964	1.043	0.007	0.001
1.63	19,20	5.6	6	0.012	0.003	0.808	0.138	2.970	1.036	0.014	0.001
1.64	31,32	4.2	5	0.014	0.001	0.826	0.141	2.959	1.041	0.017	0.002
1.71	44	0.8	1	0.027	0.001	0.443	0.486	2.956	1.049	0.020	0.001
	Mean			0.019	0.001	0.773	0.191	2.956	1.048	0.011	0.001
	Max			0.082	0.003	0.882	0.486	2.987	1.229	0.020	0.003
	Min			0.003	0.000	0.443	0.097	2.815	1.008	0.002	0.000
	St. Dev.			0.019	0.001	0.104	0.091	0.040	0.051	0.005	0.001
Virginia Formation											
2.06	21,22	12.6	15	0.010	0.001	0.742	0.239	2.969	1.033	0.005	0.001
3.12	23,24	8.0	9	0.046	0.009	0.745	0.189	2.897	1.106	0.021	0.001
3.72	25,26	14.8	17	0.002	0.001	0.834	0.143	2.971	1.034	0.006	0.005
5.44	27,28	20.4	23	0.004	0.001	0.838	0.133	2.980	1.024	0.009	0.002
	Mean			0.016	0.003	0.790	0.176	2.954	1.049	0.010	0.002
	Max			0.046	0.009	0.838	0.239	2.980	1.106	0.021	0.005
	Min			0.002	0.001	0.742	0.133	2.897	1.024	0.005	0.001
	St. Dev.			0.021	0.004	0.053	0.049	0.038	0.038	0.007	0.002

Table 11. Unleached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 6 of 8.

Biotite - $K_{1.78}(Mg_{3.07}Fe_{2.03})(Al_{2.53}Si_{5.54}O_{20})(OH)_2$											
Sulfur %	**Reactor**	**%Biotite**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.18	1,2	1.8	2	0.006	2.652	1.851	0.047	5.532	2.544	2.288	0.611
0.22	3,4	2.7	3	0.005	3.294	1.829	0.049	5.527	2.509	1.907	0.516
0.4	5,6	1.7	2	0.009	2.834	1.672	0.058	5.52	2.747	2.239	0.442
0.41	7,8	2.6	3	0.006	3.188	1.824	0.05	5.532	2.589	1.816	0.551
0.51	9,10	2.5	3	0.006	3.215	1.807	0.056	5.576	2.5	1.879	0.526
0.54	11,12	2.7	3	0.004	3.02	1.804	0.035	5.548	2.574	1.886	0.605
0.58	15,16	0.8	1	0	2.163	1.837	0.024	5.499	2.421	2.804	0.731
0.67	40	6.3	7	0.023	2.929	1.782	0.081	5.497	2.554	2.159	0.563
0.71	17,18	1.7	2	0.016	3.405	1.691	0.082	5.497	2.497	1.712	0.616
0.82	43	3.5	4	0.01	2.74	1.753	0.057	5.55	2.553	2.295	0.553
1.12	35,36	1.7	2	0.007	3.279	1.849	0.048	5.648	2.44	1.786	0.511
1.44	33,34	1.7	2	0.001	3.688	1.839	0.056	5.491	2.584	1.661	0.421
1.63	19,20	1.9	2	0.002	3.145	1.645	0.059	5.498	2.609	2.292	0.395
1.64	31,32	2.5	3	0.015	3.265	1.767	0.029	5.549	2.502	1.95	0.503
1.71	44	0.8	1	0.014	3.212	1.739	0.085	5.608	2.353	1.723	0.696
	Mean			0.008	3.069	1.779	0.054	5.538	2.532	2.026	0.549
	Max			0.023	3.688	1.851	0.085	5.648	2.747	2.804	0.731
	Min			0	2.163	1.645	0.024	5.491	2.353	1.661	0.395
	St. Dev.			0.006	0.366	0.067	0.018	0.045	0.092	0.312	0.094
Virginia Formation											
2.06	21,22	0.8	1	0.016	2.585	1.609	0.028	5.585	2.947	1.902	0.545
3.12	23,24	2.7	3	0.005	2.849	1.779	0.031	5.515	2.789	2.098	0.459
3.72	25,26	2.6	3	0.004	4.515	1.816	0.019	5.744	2.514	0.79	0.254
5.44	27,28	0.9	1	0.009	2.623	1.76	0.023	5.478	2.799	2.346	0.485
	Mean			0.009	3.143	1.741	0.025	5.581	2.762	1.784	0.436
	Max			0.016	4.515	1.816	0.031	5.744	2.947	2.346	0.545
	Min			0.004	2.585	1.609	0.019	5.478	2.514	0.79	0.254
	St. Dev.			0.005	0.922	0.091	0.005	0.118	0.181	0.687	0.126

Table 11. Unleached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 7 of 8.

Prehnite - $Ca_{1.92}Al_{1.96}(Si_{3.03}O_9)(OH)_2$											
Sulfur %	**Reactor**	**%Prehnite**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.18	1,2	1.8	2	1.936	0.003	0.001	0	3.015	1.997	0.029	0.002
0.22	3,4	0.9	1	1.957	0.002	0.001	0.009	3.026	1.966	0.029	0.001
0.4	5,6	1.7	2	1.959	0.003	0.001	0.009	2.979	2.029	0.022	0.002
0.54	11,12	0.9	1	1.891	0	0.001	0.007	3.074	1.968	0.006	0
0.72	42	4.1	5	1.915	0.005	0.004	0.022	2.99	2.035	0.022	0.002
1.12	35,36	0.8	1	1.959	0.003	0	0.006	3.039	1.958	0.02	0
1.4	37,38	2.5	3	1.893	0.034	0.001	0	3.024	1.949	0.095	0.003
1.64	31,32	0.8	1	1.94	0.007	0.001	0.003	3.086	1.707	0.309	0.004
1.71	44	0.8	1	1.864	0	0	0.01	3.047	2.022	0.005	0
	Average			1.924	0.006	0.001	0.007	3.031	1.959	0.06	0.002
	Max			1.959	0.034	0.004	0.022	3.086	2.035	0.309	0.004
	Min			1.864	0	0	0	2.979	1.707	0.005	0
	St. Dev.			0.035	0.011	0.001	0.007	0.035	0.1	0.097	0.001
Virginia Formation											
3.72	25,26	0.9	1	1.855	0.029	0.003	0.003	3.058	1.968	0.041	0.001

Table 11. Unleached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 8 of 8.

Amphibole - $(Ca_{1.78}Na_{0.25})(Mg_{2.96}Fe_{2.15})(Al_{0.99}Si_{7.12})_8O_{22}(OH)$											
Sulfur %	**Reactor**	**% Amphibole**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.22	3,4	0.9	1	1.691	2.436	0.106	0.504	6.631	2.056	2.17	0.017
0.4	5,6	0.9	1	1.687	3.48	0.038	0.109	7.626	0.522	1.574	0.062
0.41	7,8	0.9	1	1.837	3.047	0.074	0.397	6.802	1.242	1.904	0.244
0.51	9,10	1.7	2	1.889	2.707	0.01	0.032	7.25	0.553	2.778	0.114
0.54	11,12	1	1	1.482	4.264	0.001	0.015	7.269	0.075	2.552	0.008
0.92	41	3.3	4	1.267	3.098	0.006	0.168	7.578	0.708	2.259	0.017
1.12	35,36	0.8	1	2.237	3.077	0	0.326	7.16	1.04	1.388	0.113
1.4	37,38	0.8	1	2.25	2.606	0.165	0.357	6.788	1.365	1.806	0.217
1.63	19,20	0.9	1	1.654	1.962	0.142	0.316	6.974	1.381	2.883	0.114
	Mean			1.777	2.964	0.06	0.247	7.12	0.994	2.146	0.101
	Max			2.25	4.264	0.165	0.504	7.626	2.056	2.883	0.244
	Min			1.267	1.962	0	0.015	6.631	0.075	1.388	0.008
	St. Dev.			0.322	0.659	0.064	0.172	0.35	0.594	0.524	0.086

Note: N = number of samples analyzed for each Sulfur %
All element units are moles.

Table 12. Leached sulfide mineral abundance determined by microprobe analysis and point count. Only minerals likely to be affected by leaching are displayed. Only samples where a grain was identified by point count had analyses to determine stoichiometric ratios. Chemical percent abundance was chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine the composition of the minerals in this study, substitute the mean sample values for each element into the stoichiometric formula. Page 1 of 4.

Pyrrhotite – $Fe_{7.18}S_8$							
Sulfur %	**Reactor**	**%, chemical**	**%, point count**	**N**	**Fe atoms**	**Ni atoms**	**Cu atoms**
Duluth Complex							
0.18	2	0.09					
0.22	4	0.05					
0.4	6	0.68	0.7	1	7.6	0.036	0.031
0.41	8	0.29					
0.51	10	0.27					
0.54	11	0.32					
0.54	12	0.46					
0.57	13	0.14					
0.57	14	0.36					
0.58	16	0.38					
0.71	18	0.05					
1.12	35	0.24					
1.12	36	1					
1.16	29	2.34					
1.16	30	1.13					
1.4	37	0.28					
1.4	38	1.5					
1.44	34	1.3					
1.63	20	0.84					
1.64	31	1.22					
1.64	32	1.15					
1.71	44	4.4	4.4	5	7.48	0.035	0.022
	Mean				7.54	0.036	0.027
	Max				7.48	0.035	0.022
	Min				7.6	0.036	0.031
	St. Dev.				0.085	0.001	0.006
Virginia Formation							
2.06	22	3.8					
3.12	24	4.93					
3.72	26	6.98					
5.44	28	10.84					

Table 12. Leached sulfide mineral abundance determined by microprobe analysis and point count. Only minerals likely to be affected by leaching are displayed. Only samples where a grain was identified by point count had analyses to determine stoichiometric ratios. Chemical percent abundance was chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine the composition of the minerals in this study, substitute the mean sample values for each element into the stoichiometric formula. Page 2 of 4.

Chalcopyrite - $(Cu_{4.29}Fe_{4.47})S_8$							
Sulfur %	**Reactor**	**%, chemical**	**%, point count**	**N**	**Fe atoms**	**Ni atoms**	**Cu atoms**
Duluth Complex							
0.18	2	0.16					
0.22	4	0.33					
0.4	6	0.18					
0.41	8	0.22					
0.51	10	0.34					
0.54	11	0.05					
0.54	12	0.05					
0.57	13	0.35					
0.57	14	0.39					
0.58	16	0.39	0.0 (R15), 0.9 (R16)	1	4.43	0	4.378
0.71	18	0.86	0.9	1	4.441	0	4.221
1.12	35	0.21					
1.12	36	0.25					
1.16	29	0.32					
1.16	30	0.33					
1.4	37	0.5					
1.4	38	0.55					
1.44	34	0.36	0.9	1	4.566	0	4.247
1.63	20	0.3					
1.64	31	0.87					
1.64	32	0.56					
	Mean				4.479	0	4.282
	Max				4.43	0	4.221
	Min				4.566	0	4.378
	St. Dev.				0.076	0	0.084
Virginia Formation							
2.06	22	0.66					
3.12	24	0.04					
3.72	26	0.03					
5.44	28	0.04					

Table 12. Leached sulfide mineral abundance determined by microprobe analysis. There were no instances of this mineral found during point count of leached samples. Only minerals likely to be affected by leaching are displayed. Only samples where a grain was identified by point count had analyses to determine stoichiometric ratios. Chemical percent abundance was chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Stoichiometric formula at top of table is the same as the unleached sample, since there was no leached stoichiometric analysis. Page 3 of 4.

Pentlandite - $(Ni_{4.89}Fe_{4.88})S_8$						
Sulfur %	**Reactor**	**%, chemical**	**N**	**Fe atoms**	**Ni atoms**	**Cu atoms**
Duluth Complex						
0.18	2	0.06				
0.22	4	0.08				
0.4	6	0.06				
0.41	8	0.09				
0.51	10	0.05				
0.54	11	0.03				
0.54	12	0.03				
0.57	13	0.07				
0.57	14	0.09				
0.58	16	0.04				
0.71	18	0.07				
1.12	35	0.03				
1.12	36	0.05				
1.16	29	0.13				
1.16	30	0.11				
1.4	37	0.06				
1.4	38	0.14				
1.44	34	0.08				
1.63	20	0.08				
1.64	31	0.22				
1.64	32	0.07				
Virginia Formation						
2.06	22	0.12				
3.12	24	0.02				
3.72	26	0.09				
5.44	28	0.01				

Table 12. Leached sulfide mineral data determined by microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples where a grain was identified by point count had analyses to determine stoichiometric ratios. Chemical percent abundance was chemically determined by assuming that all Cu was present as chalcopyrite and all Ni as pentlandite. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine the composition of the minerals in this study, substitute the mean sample values for each element into the stoichiometric formula. Page 4 of 4.

Pyrite - $Fe_{4.59}S_8$						
Sulfur %	Reactor	%Pyrite	N	Fe atoms	Ni atoms	Cu atoms
Duluth Complex						
0.4	6	1.5	2	4.592	0.078	2.113

Cubanite - $Cu_{2.88}Fe_{5.79}S_8$						
Sulfur %	Reactor	%Cubanite	N	Fe atoms	Ni atoms	Cu atoms
Duluth Complex						
0.41	8	0.9	1	5.657	0.003	2.908
0.58	16	0.9	1	6.134	0.002	2.989
0.71	18	0.9	1	5.565	0	2.752
	Mean			5.785	0.001	2.883
	Max			6.134	0.003	2.989
	Min			5.565	0.002	2.752
	St. Dev.			0.305	0.002	0.12

Note: N = number of samples analyzed for each Sulfur %

Note: Pyrrhotite and chalcopyrite percentages were determined using calculations from bulk chemistry except for pyrrhotite in the 1.71%S sample, which was determined by point count. All other mineral percentages were determined using point counts. There were no analyses of leached samples for five samples that were still being run at the time of analysis (0.67 (reactors 39, 40), 0.72, 0.82 and 0.92%S).

Table 13. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 1 of 8.

Plagioclase - $(Ca_{0.48}Na_{0.45})Al_{1.53}Si_{2.48}O_8$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.4	6	46.7	64	0.546	0.001	0.021	0.416	2.439	1.561	0.011	0.001
0.41	7	52.1	61	0.522	0.002	0.023	0.437	2.463	1.534	0.017	0.001
0.51	9	44	48	0.503	0.002	0.021	0.466	2.482	1.512	0.017	0.001
0.58	15	42.5	48	0.465	0.001	0.031	0.470	2.513	1.493	0.014	0.002
0.58	16	54.1	59	0.432	0.004	0.032	0.467	2.525	1.494	0.019	0.001
0.71	17	48.6	54	0.471	0.001	0.025	0.469	2.493	1.515	0.017	0.001
1.4	37	55.6	60	0.460	0.001	0.021	0.481	2.516	1.493	0.014	0.001
1.44	33	45.9	51	0.499	0.001	0.013	0.451	2.459	1.552	0.018	0.001
1.63	19	52.3	56	0.439	0.002	0.030	0.493	2.534	1.474	0.015	0.002
1.71	44	47.4	54	0.504	0.005	0.041	0.366	2.419	1.617	0.016	0.002
	Mean			0.484	0.002	0.026	0.452	2.484	1.525	0.016	0.001
	Max			0.546	0.005	0.041	0.493	2.534	1.617	0.019	0.002
	Min			0.432	0.001	0.013	0.366	2.419	1.474	0.011	0.001
	St. Dev.			0.037	0.001	0.008	0.037	0.039	0.043	0.002	0.000

Table 13. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 2 of 8.

Hypersthene - $(Mg_{1.01}Fe_{0.89})_2Si_{1.98}O_6$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.4	6	27	37	0.018	0.984	0.000	0.001	1.957	0.065	0.952	0.008
0.41	7	8.5	10	0.045	1.051	0.001	0.002	1.991	0.032	0.841	0.006
0.51	9	17.4	19	0.055	1.000	0.000	0.002	1.974	0.035	0.901	0.012
0.58	15	14.2	16	0.047	0.922	0.000	0.001	1.974	0.038	0.991	0.007
0.58	16	6.6	20	0.039	0.926	0.001	0.001	1.992	0.037	0.960	0.007
0.71	17	14.4	16	0.086	1.112	0.001	0.001	1.979	0.034	0.761	0.006
1.4	37	8.3	9	0.046	1.021	0.001	0.001	1.985	0.025	0.896	0.005
1.44	33	16.2	18	0.036	1.024	0.001	0.001	1.974	0.028	0.916	0.009
1.63	19	5.6	6	0.048	0.929	0.001	0.001	1.983	0.024	0.984	0.007
1.71	44	12.3	14	0.077	1.105	0.000	0.002	1.958	0.098	0.717	0.009
	Mean			0.050	1.007	0.001	0.001	1.977	0.042	0.892	0.008
	Max			0.086	1.112	0.001	0.002	1.992	0.098	0.991	0.012
	Min			0.018	0.922	0.000	0.001	1.957	0.024	0.717	0.005
	St. Dev.			0.020	0.070	0.000	0.001	0.012	0.023	0.093	0.002

Table 13. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 3 of 8.

Olivine - $(Mg_{0.89}Fe_{1.08})Si_{1.00}O_4$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.4	6	0									
0.41	7	12	14	0.002	0.904	0.000	0.001	1.006	0.001	1.062	0.001
0.51	9	10.1	11	0.002	0.822	0.000	0.001	1.007	0.003	1.140	0.001
0.58	15	1.8	2	0.002	0.806	0.000	0.001	1.001	0.000	1.176	0.001
0.58	16	8.2	5	0.002	0.698	0.000	0.000	1.007	0.000	1.268	0.001
0.71	17	20.7	23	0.002	1.029	0.000	0.001	1.005	0.001	0.941	0.001
1.4	37	6.5	7	0.002	0.923	0.000	0.001	0.998	0.001	1.056	0.001
1.44	33	9.9	11	0.001	0.969	0.000	0.001	0.997	0.001	1.015	0.001
1.63	19	1.9	2	0.001	0.731	0.001	0.001	0.995	0.000	1.260	0.000
1.71	44	21.9	25	0.002	1.137	0.000	0.000	1.007	0.000	0.828	0.002
	Mean			0.002	0.891	0.000	0.001	1.003	0.001	1.083	0.001
	Max			0.002	1.137	0.001	0.001	1.007	0.003	1.268	0.002
	Min			0.001	0.698	0.000	0.000	0.995	0.000	0.828	0.000
	St. Dev.			0.000	0.142	0.000	0.000	0.005	0.001	0.145	0.000

Table 13. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 4 of 8.

Augite - $(Ca_{0.80}Na_{0.02})(Mg_{0.69}Fe_{0.42}Ti_{0.01})(Si_{1.96}Al_{0.08})O_6$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.4	6	5.8	8	0.945	0.451	0.001	0.003	1.963	0.063	0.538	0.007
0.41	7	12	14	0.845	0.726	0.001	0.021	1.955	0.076	0.355	0.014
0.51	9	11	12	0.772	0.706	0.009	0.013	1.953	0.089	0.439	0.012
0.58	15	15	17	0.759	0.708	0.000	0.015	1.956	0.074	0.459	0.014
0.58	16	13.1	7	0.759	0.676	0.001	0.017	1.995	0.063	0.434	0.011
0.71	17	10.8	12	0.767	0.784	0.000	0.021	1.963	0.076	0.355	0.014
1.4	37	17.6	19	0.780	0.723	0.003	0.024	1.949	0.085	0.411	0.017
1.44	33	12.6	14	0.756	0.766	0.000	0.021	1.962	0.083	0.381	0.014
1.63	19	13.1	14	0.786	0.695	0.002	0.017	1.963	0.070	0.438	0.012
1.71	44	0									
	Mean			0.797	0.693	0.002	0.017	1.962	0.075	0.423	0.013
	Max			0.945	0.784	0.009	0.024	1.995	0.089	0.538	0.017
	Min			0.756	0.451	0.000	0.003	1.949	0.063	0.355	0.007
	St. Dev.			0.062	0.097	0.003	0.006	0.013	0.009	0.057	0.003

Table 13. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 5 of 8.

Potassium-Feldspar - $K_{0.77}Al_{1.08}Si_{2.93}O_8$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.4	6	2.2	3	0.006	0.001	0.877	0.104	2.978	1.019	0.016	0.001
0.41	7	0.9	1	0.006	0.010	0.878	0.116	2.973	1.011	0.025	0.000
0.51	9	1.8	2	0.008	0.001	0.850	0.129	2.979	1.018	0.012	0.002
0.58	15	2.7	3	0.019	0.056	0.687	0.168	2.888	1.122	0.036	0.001
0.58	16	6.6	2	0.077	0.009	0.683	0.125	2.747	1.320	0.023	0.005
0.71	17	0									
1.4	37	0.9	1	0.004	0.000	0.848	0.121	2.960	1.053	0.003	0.003
1.44	33	6.3	7	0.013	0.001	0.799	0.176	2.967	1.034	0.011	0.001
1.63	19	10.3	11	0.014	0.012	0.728	0.210	2.946	1.056	0.024	0.002
1.71	44	2.6	3	0.037	0.027	0.531	0.375	2.925	1.067	0.031	0.000
	Mean			0.021	0.013	0.765	0.169	2.929	1.078	0.020	0.002
	Max			0.077	0.056	0.878	0.375	2.979	1.320	0.036	0.005
	Min			0.004	0.000	0.531	0.104	2.747	1.011	0.003	0.000
	St. Dev.			0.024	0.018	0.117	0.084	0.074	0.097	0.011	0.002

Table 13. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 6 of 8.

Biotite - $K_{1.78}(Mg_{2.72}Fe_{2.25})(Al_{2.53}Si_{5.54}O_{20})(OH)_2$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.4	6	0									
0.41	7	0									
0.51	9	0									
0.58	15	3.5	4	0.010	2.477	1.816	0.059	5.544	2.487	2.485	0.633
0.58	16	4.9	1	0.006	3.303	1.875	0.065	5.598	2.554	1.718	0.484
0.71	17	0.9	1	0.009	2.854	1.791	0.034	5.489	2.503	2.064	0.708
1.4	37	0.9	1	0.014	3.142	1.690	0.068	5.557	2.586	1.852	0.559
1.44	33	0.9	1	0.000	2.815	1.811	0.006	5.524	2.558	2.160	0.615
1.63	19	0.9	1	0.011	1.744	1.695	0.077	5.524	2.513	3.236	0.640
1.71	44	0									
	Mean			0.008	2.723	1.780	0.051	5.539	2.534	2.253	0.607
	Max			0.014	3.303	1.875	0.077	5.598	2.586	3.236	0.708
	Min			0.000	1.744	1.690	0.006	5.489	2.487	1.718	0.484
	St. Dev.			0.005	0.558	0.073	0.027	0.037	0.038	0.550	0.077

Table 13. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 7 of 8.

Prehnite - $Ca_{1.90}Al_{1.99}(Si_{3.03}O_9)(OH)_2$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.4	6										
0.41	7										
0.51	9										
0.58	15										
0.58	16										
0.71	17										
1.4	37	0.9	1	1.901	0.005	0.002	0.019	3.027	1.99	0.042	0.001
1.44	33										
1.63	19										
1.71	44										
	Mean			1.901	0.005	0.002	0.019	3.027	1.99	0.042	0.001
	Max										
	Min										
	St. Dev.										

Table 13. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on the mean ratios of major elements for Duluth Complex samples. To determine sample stoichiometry, substitute mean sample values for each element into the stoichiometric formula. Page 8 of 8.

Amphibole - $(Ca_{1.61}Na_{0.31})(Mg_{1.66}Fe_{2.16})(Al_{1.64}Si_{7.08})_8O_{22}(OH)$											
Sulfur %	**Reactor**	**%, point count**	**N**	**Ca**	**Mg**	**K**	**Na**	**Si**	**Al**	**Fe**	**Ti**
Duluth Complex											
0.4	6	0									
0.41	7	2.6	3	1.453	1.536	0.383	0.582	7.417	1.781	1.443	0.278
0.51	9	0.9	1	1.762	1.778	0.228	0.488	6.747	1.506	2.880	0.209
0.58	15	0									
0.58	16	0									
0.71	17	0									
1.4	37	0									
1.44	33	0									
1.63	19	0									
1.71	44	0									
	Mean			1.608	1.657	0.305	0.535	7.082	1.643	2.161	0.244
	Max			1.762	1.778	0.383	0.582	7.417	1.781	2.880	0.278
	Min			1.453	1.536	0.228	0.488	6.747	1.506	1.443	0.209
	St. Dev.			0.218	0.171	0.110	0.067	0.474	0.194	1.016	0.049

Note: N = number of samples analyzed for each Sulfur %
All element units are moles.

Table 14. Particle surface area and coating analysis for unleached samples (Analysis by McSwiggen and Associates).

Sulfur %	Reactor	Perimeter / 1 mm area	Sulfides present	Ratio	Coating (microns)	Silicate Comments	Other Information
0.22	4	106	chalcopyrite pyrrhotite	4 cp:1 po		no alteration	
0.4	5,6	76.7	chalcopyrite pyrrhotite	1 cp:3 po		no alteration	
0.41	7,8	110.9	chalcopyrite pyrrhotite	1 cp:1 po		no alteration	Sulfides unaltered
0.51	9,10	99.9	chalcopyrite pyrrhotite	2 cp:3 po		no alteration	
0.58	15,16	103.9	chalcopyrite pyrrhotite	1 cp:3 po		no alteration	
0.58	16	85.4	chalcopyrite pyrrhotite pentlandite	1 cp:2 po		no alteration	
0.71	17,18	81.8	chalcopyrite pyrrhotite	1 cp:1 po		no alteration	
1.16	29,30	72	chalcopyrite pyrrhotite	1 cp: 3 po			
1.4	37,38	72	chalcopyrite pyrrhotite pentlandite	1 cp: 3 po		no alteration	
1.44	33,34	80.9	chalcopyrite pyrrhotite	1 cp:8 po		no alteration	
1.63	19,20	255.6	chalcopyrite pyrrhotite	1 cp:3 po		no alteration	
1.71	44	105.3	chalcopyrite pyrrhotite	1 cp:3 po		no alteration	

Table 15. Particle surface area and coating analysis for leached samples (Analysis by McSwiggen and Associates).

Sulfur %	Reactor	Reaction time (wks)	Perimeter/ 1 mm area	Sulfides present	Ratio	Coating (microns)	Silicate Comments	Other Information
0.22	4	328	94.1	chalcopyrite pyrrhotite pyrite	4 cp:1 po	0-2 7	no coatings	Po more reacted than cp; oxide coating compositions reported
0.4	6	328	92.9	pyrrhotite			no reaction evidence	Sulfides clean to half reacted to oxide; oxide coating compositions reported
0.41	7	724	83	chalcopyrite		thin - 2	hypersthene shows some alteration	One chalcopyrite grain had small serpentine inclusion
0.51	9	724	111.3	chalcopyrite		2-6	no alteration or coatings	Cp is very altered, and oxide coating compositions are reported
0.58	15	724	80.1	chalcopyrite pyrrhotite	Only 1 pyrrhotite grain found	0-3 2-14	few coatings; hyp alteration	po is much more reacted than cp; sulfide and silicate oxide coating compositions
0.58	16	328	87.1	chalcopyrite pyrrhotite	Only 1 pyrrhotite grain found			pyrrhotite coating composition; po much more reacted than cp
0.71	17	724	85.7	chalcopyrite		2-4.6		Oxide coating compositions reported
1.16	29	643	75.5	chalcopyrite		2.3-5	greenalite coatings on oli	Oxide coating compositions, cp is only sulfide found (variably altered), pyrrhotite is gone
1.4	37	360	78.3	chalcopyrite		0-1	coatings on hyp (0-3 microns)	Oxide coating compositions, hyp coatings are post-crush, po gone, cp variably altered
1.44	33	643	79.9	chalcopyrite		4-10	1-3 micron oli coatings, hyp, plag, amph dissolution pits	large degree of sulfide alteration; oxide coatings compositions; first indication of silicate dissolution (grooves and pits)
1.63	19	724	85.9	chalcopyrite		3-10	Andesine coating on hyp (4 microns), 4-6 micron coating on oli	Highly altered sulfides, dominated by chalcopyrite, very little pyrrhotite left
1.71	44	144	87	pyrrhotite			no reaction evidence	little or no sulfide alteration

Table 16. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on average ratio of major elements to a constant amount of oxygen. Page 1 of 4.

Plagioclase - $(Ca_{0.48}Na_{0.45})Al_{1.53}Si_{2.48}O_8$

Sulfur %	Reactor	% Plagioclase	N	Ca	Mg	K	Na	Si	Al	Fe	Ti
0.40	6	46.7	64	0.546	0.001	0.021	0.416	2.439	1.561	0.011	0.001
0.41	7	52.1	61	0.522	0.002	0.023	0.437	2.463	1.534	0.017	0.001
0.51	9	44.0	48	0.503	0.002	0.021	0.466	2.482	1.512	0.017	0.001
0.58	15	42.5	48	0.465	0.001	0.031	0.470	2.513	1.493	0.014	0.002
0.58	16	52.2	59	0.432	0.004	0.032	0.467	2.525	1.494	0.019	0.001
0.71	17	48.7	54	0.471	0.001	0.025	0.469	2.493	1.515	0.017	0.001
1.40	37	55.6	60	0.460	0.001	0.021	0.481	2.516	1.493	0.014	0.001
1.44	33	46.0	51	0.499	0.001	0.013	0.451	2.459	1.552	0.018	0.001
1.63	19	52.3	56	0.439	0.002	0.030	0.493	2.534	1.474	0.015	0.002
1.71	44	47.4	54	0.504	0.005	0.041	0.366	2.419	1.617	0.016	0.002
	Mean			0.484	0.002	0.026	0.452	2.484	1.525	0.016	0.001
	Max			0.546	0.005	0.041	0.493	2.534	1.617	0.019	0.002
	Min			0.432	0.001	0.013	0.366	2.419	1.474	0.011	0.001
	St Dev			0.037	0.001	0.008	0.037	0.039	0.043	0.002	0.000

Hypersthene - $(Mg_{1.01}Fe_{0.89})Si_{1.98}O_6$

Sulfur %	Reactor	%Hypersthene	N	Ca	Mg	K	Na	Si	Al	Fe	Ti
0.40	6	27.0	37	0.018	0.984	0.000	0.001	1.957	0.065	0.952	0.008
0.41	7	8.6	10	0.045	1.051	0.001	0.002	1.991	0.032	0.841	0.006
0.51	9	17.4	19	0.055	1.000	0.000	0.002	1.974	0.035	0.901	0.012
0.58	15	14.2	16	0.047	0.922	0.000	0.001	1.974	0.038	0.991	0.007
0.58	16	17.7	20	0.039	0.926	0.001	0.001	1.992	0.037	0.960	0.007
0.71	17	14.4	15	0.086	1.112	0.001	0.001	1.979	0.034	0.761	0.006
1.40	37	8.3	9	0.046	1.021	0.001	0.001	1.985	0.025	0.896	0.005
1.44	33	16.2	18	0.036	1.024	0.001	0.001	1.974	0.028	0.916	0.009
1.63	19	5.6	6	0.048	0.929	0.001	0.001	1.983	0.024	0.984	0.007
1.71	44	12.3	14	0.077	1.105	0.000	0.002	1.958	0.0.98	0.717	0.009
	Mean			0.050	1.007	0.001	0.001	1.977	0.035	0.892	0.008
	Max			0.086	1.112	0.001	0.002	1.992	0.0.98	0.991	0.012
	Min			0.018	0.922	0	0.001	1.957	0.024	0.717	0.005
	St Dev			0.020	0.070	0.001	0.000	0.012	0.012	0.093	0.002

Table 16. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on average ratio of major elements to a constant amount of oxygen. Page 2 of 4.

Augite - $(Ca_{0.80}Na_{0.02})(Mg_{0.69}Fe_{0.42}Ti_{0.01})(Si_{1.96}Al_{0.08})O_6$

Sulfur %	Reactor	%Augite	N	Ca	Mg	K	Na	Si	Al	Fe	Ti
0.40	6	5.8	8	0.945	0.451	0.001	0.003	1.963	0.063	0.538	0.007
0.41	7	12.0	14	0.845	0.726	0.001	0.021	1.955	0.076	0.355	0.014
0.51	9	11.0	12	0.772	0.706	0.009	0.013	1.953	0.089	0.439	0.012
0.58	15	15.0	17	0.759	0.708	0.000	0.015	1.956	0.074	0.459	0.014
0.58	16	6.2	7	0.759	0.676	0.001	0.017	1.995	0.063	0.434	0.011
0.71	17	10.8	11	0.767	0.784	0.000	0.021	1.963	0.076	0.355	0.014
1.40	37	17.6	19	0.780	0.723	0.003	0.024	1.949	0.085	0.411	0.017
1.44	33	12.6	14	0.756	0.766	0.000	0.021	1.962	0.083	0.381	0.014
1.63	19	13.1	14	0.786	0.695	0.002	0.017	1.963	0.070	0.438	0.012
	Mean			0.797	0.693	0.002	0.017	1.962	0.075	0.423	0.013
	Max			0.945	0.784	0.009	0.024	1.995	0.089	0.538	0.017
	Min			0.756	0.451	0	0.003	1.949	0.063	0.355	0.007
	St. Dev			0.062	0.097	0.003	0.006	0.013	0.009	0.057	0.003

Olivine - $(Mg_{0.89}Fe_{1.08})Si_{1.00}O_4$

Sulfur %	Reactor	%Olivine	N	Ca	Mg	K	Na	Si	Al	Fe	Ti
0.41	7	12.0	14	0.002	0.904	0.000	0.001	1.006	0.001	1.062	0.001
0.51	9	10.1	11	0.002	0.822	0.000	0.001	1.007	0.003	1.140	0.001
0.58	15	1.8	2	0.002	0.806	0.000	0.001	1.001	0.000	1.176	0.001
0.58	16	4.4	5	0.002	0.698	0.000	0.000	1.007	0.000	1.268	0.001
0.71	17	20.7	23	0.002	1.029	0.000	0.001	1.005	0.001	0.941	0.001
1.40	37	6.5	7	0.002	0.923	0.000	0.001	0.998	0.001	1.056	0.001
1.44	33	9.9	11	0.001	0.969	0.000	0.001	0.997	0.001	1.015	0.001
1.63	19	1.9	3	0.001	0.731	0.001	0.001	0.995	0.000	1.260	0.000
1.71	44	21.9	25	0.002	1.137	0.000	0.000	1.007	0.000	0.828	0.002
	Mean			0.002	0.891	0.000	0.001	1.003	0.001	1.083	0.001
	Max			0.002	1.137	0.001	0.001	1.007	0.003	1.268	0.002
	Min			0.001	0.698	0	0	0.995	0	0.828	0
	St Dev			0.000	0.142	0.000	0.000	0.005	0.001	0.145	0.001

Table 16. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on average ratio of major elements to a constant amount of oxygen. Page 3 of 4.

Potassium Feldspar - $K_{0.77}Al_{1.08}Si_{2.93}O_8$

Sulfur %	Reactor	%K-Spar	N	Ca	Mg	K	Na	Si	Al	Fe	Ti
0.40	6	2.2	3	0.006	0.001	0.877	0.104	2.978	1.019	0.016	0.001
0.41	7	0.9	1	0.006	0.010	0.878	0.116	2.973	1.011	0.025	0.000
0.51	9	1.8	2	0.008	0.001	0.850	0.129	2.979	1.018	0.012	0.002
0.58	15	2.7	3	0.019	0.056	0.687	0.168	2.888	1.122	0.036	0.001
0.58	16	1.8	2	0.077	0.009	0.683	0.125	2.747	1.320	0.023	0.005
1.40	37	0.9	1	0.004	0.000	0.848	0.121	2.960	1.053	0.003	0.003
1.44	33	6.3	7	0.013	0.001	0.799	0.176	2.967	1.034	0.011	0.001
1.63	19	10.3	11	0.014	0.012	0.728	0.210	2.946	1.056	0.024	0.002
1.71	44	2.6	3	0.037	0.027	0.531	0.375	2.925	1.067	0.031	0.000
	Mean			0.020	0.013	0.765	0.169	2.929	1.078	0.020	0.002
	Max			0.077	0.056	0.878	0.375	2.979	1.32	0.036	0.005
	Min			0.004	0	0.531	0.104	2.747	1.011	0.003	0
	St Dev			0.023	0.018	0.117	0.084	0.074	0.097	0.010	0.002

Biotite - $K_{1.78}(Mg_{2.72}Fe_{2.25})(Al_{2.53}Si_{5.53}O_{20})(OH)_2$

Sulfur %	Reactor	%Biotite	N	Ca	Mg	K	Na	Si	Al	Fe	Ti
0.58	15	3.5	4	0.010	2.477	1.816	0.059	5.544	2.487	2.485	0.633
0.58	16	0.9	1	0.006	3.303	1.875	0.065	5.598	2.554	1.718	0.484
0.71	17	0.9	1	0.009	2.854	1.791	0.034	5.489	2.503	2.064	0.708
1.40	37	0.9	1	0.014	3.142	1.690	0.068	5.557	2.586	1.852	0.559
1.44	33	0.9	1	0.000	2.815	1.811	0.006	5.524	2.558	2.160	0.615
1.63	19	0.9	1	0.011	1.744	1.695	0.077	5.524	2.513	3.236	0.640
	Mean			0.008	2.723	1.780	0.051	5.539	2.534	2.252	0.607
	Max			0.014	3.303	1.875	0.077	5.598	2.586	3.236	0.708
	Min			0.000	1.744	1.690	0.006	5.489	2.487	1.718	0.484
	St Dev			0.005	0.558	0.073	0.027	0.037	0.038	0.550	0.077

Table 16. Leached silicate mineral chemistry, as determined through microprobe analysis. Only minerals likely to be affected by leaching are displayed. Only samples containing each mineral are displayed. Stoichiometric formulas at top of table are based on average ratio of major elements to a constant amount of oxygen. Page 4 of 4.

Amphibole - $(Ca_{1.61}Na_{0.54})(Mg_{1.66}Fe_{2.16})(Al_{1.64}Si_{7.08})_8O_{22}(OH)$

Sulfur %	Reactor	%Amphibole	N	Ca	Mg	K	Na	Si	Al	Fe	Ti
0.41	7	2.6	3	1.453	1.536	0.383	0.582	7.417	1.781	1.443	0.278
0.51	9	0.9	1	1.762	1.778	0.228	0.488	6.747	1.506	2.880	0.209
	Mean			1.608	1.657	0.306	0.535	7.082	1.644	2.162	0.244
	Max			1.762	1.778	0.383	0.582	7.417	1.781	2.88	0.278
	Min			1.453	1.536	0.228	0.488	6.747	1.506	1.443	0.209
	St Dev			0.218	0.171	0.110	0.066	0.474	0.194	1.016	0.049

Prehnite - $Ca_{1.90}Al_{1.99}(Si_{3.03}O_9)(OH)_2$

Sulfur %	Reactor	%Prehnite	N	Ca	Mg	K	Na	Si	Al	Fe	Ti
1.40	37	0.9	1	1.901	0.005	0.002	0.019	3.027	1.990	0.042	0.001

Note: N = number of samples analyzed for each Sulfur %

Table 17. Perimeter/1000 micron area of mineral grains from different samples. Analysis done by McSwiggen and Associates.

	Sample 1	Sample 2	Sample 3	Sample 4	Mean P	Particle Diameter[1] (μm)
0.22 R4 unleached	99.1	112.1	106.8		106.0	37.7
0.22 R4 leach	96.7	91.5			94.1	42.5
unleached/leach					1.13	0.87
0.40 R5&6 unleached	73.4	79.9			76.7	52.2
0.40 R6 leach	96.3	93.3	89.2		92.9	43.1
unleached/leach					0.83	1.21
0.41 R7&8 unleached	115.3	115.8	101.7		110.9	36.1
0.41 R7 leached	88.1	79.8	81.1		83.0	48.2
unleached/leach					1.34	0.75
0.51 R9&10 unleached	103.0	102.6	94.2		99.9	40.0
0.51 R9 leached	120.1	113.3	100.6		111.3	35.9
unleached/leach					0.90	1.11
0.58 R15&16 unleached	122.0	78.8	111.0		103.9	38.5
0.58 R15 leached	79.2	80.9			80.1	49.9
unleached/leach					1.30	0.77
0.58 R16 unleached	84.1	78.6	93.4		85.4	46.8
0.58 R16 leached	90.1	78.1	93.0		87.1	45.9
unleached/leach					0.98	1.02
0.71 R17&18 unleached	80.8	82.0	82.5		81.8	48.9
0.71 R17 leached	81.5	86.0	89.6		85.7	46.7
unleached/leach					0.95	1.05
1.16 R29&30 unleached	72.1	73.2	70.8		72.0	55.6
1.16 R29 leached	76.3	77.0	77.7	70.8	75.5	53.0
unleached/leach					0.95	1.05
1.40 R37&38 unleached	73.9	74.1	68.1		72.0	55.6
1.40 R37 leached	80.9	77.0	77.0		78.3	51.1
unleached/leach					0.92	1.09
1.44 R33&34 unleached	80.7	78.3	83.6		80.9	49.4
1.44 R33 leached	74.7	85.1			79.9	50.1
unleached/leach					1.01	0.99
1.63 R19&20 unleached	225.4	339.6	201.7		255.6	15.6
1.63 R19 leached	79.5	86.3	92.0		85.9	46.6
unleached/leach					2.98	0.34
1.71 R44 unleached	142.1	90.9	83.0		105.3	38.0
1.71 R44 leached	96.6	79.1	85.2		87.0	46.0
unleached/leach					1.21	0.83
Average unleached					104.2	42.9
Average leached					86.7	46.6
Average unleached/leached					1.21	0.92

[1] Particle diameter, d, was determined by assuming particles are spherical and smooth.
$P/A = \pi dn/[(\pi d^2/4)n] = 4\pi dn/(\pi d^2 n) = 4/d$, and $d = 4A/P$. $A=1000\ \mu m^2$, therefore $d= 4000/P$.

Table 18. Water retention/drying time of Duluth Complex rock used in the experiment. Cells were rinsed on Tuesdays after weighing. Numbers are weights of rock and reactor cell (plus water) in grams.

Week start date	Reactor	Mon	Tue	Wed	Tue-Wed	Moisture content	Thu	Wed - Thur	Fri	Thu-Fri	Percent evap
10/28/2003	Cell 1		147.7	165.9	18.2	24.3%	159	6.9	153.4	5.6	68.7%
	Cell 3		144.9	160.7	15.8	21.1%	154.4	6.3	149.2	5.2	72.8%
	Cell 5		154.1	174.6	20.5	27.3%	168.3	6.3	163	5.3	56.6%
	Cell 39		157.5	178.6	21.1	28.1%	171.3	7.3	166	5.3	59.7%
	Cell 40		154	174.3	20.3	27.1%	167.5	6.8	162.2	5.3	59.6%
	Cell 41		155.5	172.6	17.1	22.8%	165.9	6.7	160.8	5.1	69.0%
	Cell 42		157.4	178.2	20.8	27.7%	171.9	6.3	167	4.9	53.8%
	Cell 43		157.7	177.9	20.2	26.9%	170	7.9	163.8	6.2	69.8%
11/3/2003	Cell 1	147.7	147.7	163.7	16	21.3%	156.9	6.8	150.2	6.7	84.4%
	Cell 3	144.9	144.8	160.6	15.8	21.1%	154.7	5.9	149.3	5.4	71.5%
	Cell 5	154.1	154	171.4	17.4	23.2%	165.9	5.5	159.7	6.2	67.2%
	Cell 39	157.6	157.5	171.3	13.8	18.4%	165.9	5.4	160.5	5.4	78.3%
	Cell 40	154	153.9	169.7	15.8	21.1%	164.8	4.9	159.7	5.1	63.3%
	Cell 41	155.7	155.5	167.9	12.4	16.5%	161.6	6.3	156.6	5	91.1%
	Cell 42	157.4	157.4	171.4	14	18.7%	164.9	6.5	159.1	5.8	87.9%
	Cell 43	157.7	157.7	174.6	16.9	22.5%	168.7	5.9	163.2	5.5	67.5%
11/10/2003	Cell 1	147.7	147.7	165.1	17.4	23.2%	157.8	7.3	150.7	7.1	82.8%
	Cell 3	144.9	144.9	159.3	14.4	19.2%	153.3	6	147.5	5.8	81.9%
	Cell 5	154.1	154.1	174	19.9	26.5%	167.7	6.3	161.5	6.2	62.8%
	Cell 39	157.6	157.5	173.1	15.6	20.8%	166.8	6.3	161.2	5.6	76.3%
	Cell 40	154	154	169.2	15.2	20.3%	162.9	6.3	156.6	6.3	82.9%
	Cell 41	155.5	155.5	170.6	15.1	20.1%	164.8	5.8	158.8	6	78.1%
	Cell 42	157.4	157.4	175	17.6	23.5%	169.4	5.6	163.1	6.3	67.6%
	Cell 43	157.7	157.7	174.8	17.1	22.8%	168.3	6.5	162.9	5.4	69.6%
11/18/2003	Cell 1		147.2	164.8	17.6	23.5%	158.3	6.5	152	6.3	72.7%
	Cell 3		144.9	157.7	12.8	17.1%	151.7	6	146.2	5.5	89.8%
	Cell 5		154.1	169.5	15.4	20.5%	163	6.5	156.8	6.2	82.5%
	Cell 39		157.5	170.5	13	17.3%	164.5	6	159.1	5.4	87.7%
	Cell 40		153.9	169.5	15.6	20.8%	162.6	6.9	156.6	6	82.7%
	Cell 41		155.5	168.6	13.1	17.5%	162.2	6.4	157.2	5	87.0%
	Cell 42		157.4	172.9	15.5	20.7%	166.7	6.2	160.9	5.8	77.4%
	Cell 43		157.7	173.4	15.7	20.9%	166.5	6.9	160.9	5.6	79.6%
11/24/2003	Cell 1	147.7	Cells (dry) not weighed before rinsing	163.7	16	21.3%					
	Cell 3	144.9		157.9	13	17.3%					
	Cell 5	154		172.1	18.1	24.1%					
	Cell 39	157.5		175.5	18	24.0%					
	Cell 40	153.9		173.7	19.8	26.4%					
	Cell 41	155.5		169.5	14	18.7%					
	Cell 42	157.4		173.8	16.4	21.9%					
	Cell 43	157.7		174.2	16.5	22.0%					
12/1/2003	Cell 1	147.7	147.7	162.9	15.2	20.3%	154.6	8.3	149.4	5.2	88.8%
	Cell 3	144.8	144.8	156.6	11.8	15.7%	149.2	7.4	145.3	3.9	95.8%
	Cell 5	154	154	169.4	15.4	20.5%	162	7.4	156.8	5.2	81.8%
	Cell 39	157.5	157.5	170.6	13.1	17.5%	163.1	7.5	158.9	4.2	89.3%
	Cell 40	153.9	153.9	169.4	15.5	20.7%	160.1	9.3	154.8	5.3	94.2%
	Cell 41	155.5	155.5	168.9	13.4	17.9%	161.1	7.8	156.8	4.3	90.3%
	Cell 42	157.4	157.4	173.5	16.1	21.5%	166.5	7	161.5	5	74.5%
	Cell 43	157.7	157.7	173.8	16.1	21.5%	166.5	7.3	161.3	5.2	77.6%
Average		153.60	153.57	169.73	16.16	21.5%	163.03	6.63	157.51	5.52	76.9%
Min		144.80	144.80	156.60	11.80	15.7%	149.20	4.90	145.30	3.90	53.8%
Max		157.70	157.70	178.60	21.10	28.1%	171.90	9.30	167.00	7.10	95.8%
St Dev		4.571	4.569	5.552	2.359	3.1%	5.582	0.839	5.559	0.647	10.8%

Table 19. Differences between replicates for analyzed data. Minimum pH and average drainage concentrations (mg/L) for SO_4, Ca and Mg.

%S	Reactor pair	# weeks run	min pH mean	pH % diff	SO_4 mean	SO_4 % diff	Ca mean	Ca % diff	Mg mean	Mg % diff
0.18	1,2	328	6.07	0.33	3.03	6.50	0.95	0.38	0.45	5.34
0.22	3,4	328	6.16	0.08	2.88	2.21	1.01	0.14	0.63	3.63
0.4	5,6	328	5.96	0.42	7.79	7.28	2.61	3.76	0.26	0.07
0.41	7,8	328	4.79	2.61	3.58	0.89	0.84	0.91	0.30	1.86
0.51	9,10	328	4.08	0.74	7.03	12.38	1.32	5.01	0.49	8.97
0.54	11,12	328	4.14	0.48	9.70	6.14	3.03	0.38	0.76	7.38
0.57	13,14	328	4.24	2.01	8.10	6.28	1.28	0.86	0.84	2.24
0.58	15,16	328	4.06	0.74	9.89	4.14	1.30	0.39	0.90	3.36
0.71	17,18	328	4.12	1.21	7.76	11.56	0.92	7.09	0.96	15.71
1.12	35,36	247	3.64	3.99	25.86	14.59	4.64	3.19	0.83	11.75
1.16	29,30	247	4.00	0.63	13.52	16.06	1.46	5.43	0.98	12.85
1.4	37,38	247	4.02	0.25	15.02	4.17	2.15	0.05	1.20	0.59
1.44	33,34	247	3.82	0.39	21.01	0.81	1.69	0.13	1.08	1.19
1.63	19,20	289	3.62	7.73	26.39	11.80	2.33	7.74	2.34	2.23
1.64	31,32	247	3.83	1.96	30.04	13.98	3.67	10.89	2.10	5.30
2.06	21,22	78	4.25	1.18	35.89	3.10	4.45	2.78	4.37	1.76
3.12	23,24	78	3.60	2.78	114.48	14.62	9.79	3.61	5.25	8.55
3.72	25,26	78	3.73	0.67	57.64	10.49	7.10	1.01	5.48	1.34
5.44	27,28	78	3.13	7.20	93.90	6.44	7.39	2.66	7.38	1.75

Note: All % diff columns are % difference of each reactor from the mean of the replicates

Table 20. Effect of drainage volume on pH of effluent.

%S	Reactor	Mean pH (and range) for 8 weeks before test	pH, 10 ml drainage	pH, total drainage	pH difference (10 ml-total)
0.41	8	5.20 (5.12-5.39)	5.23	5.28	-0.05
0.51	10	4.65 (4.58-4.73)	4.61	4.58	0.03
0.54	11	4.7 (4.68-4.79)	4.58	4.69	-0.11
0.54	12	4.85 (4.87-4.98)	4.88	4.95	-0.07
0.57	13	5.05 (5.05-5.21)	4.92	5.04	-0.12
0.58	16	4.22 (4.14-4.22)	4.31	4.17	0.14
0.71	17	4.90 (4.86-4.96)	4.82	4.87	-0.05
0.71	18	4.45 (4.48-4.73)	4.27	4.42	-0.15
1.12	35	4.20 (4.16-4.21)	3.96	4.16	-0.13
1.16	29	4.35 (4.33-4.40)	4.15	4.30	-0.11
1.16	30	4.50 4.47-4.54)	4.24	4.43	-0.15
1.4	37	4.75 (4.7-4.79)	4.47	4.72	-0.19
1.63	19	4.08 (4.03-4.11)	3.90	4.03	-0.17
1.63	20	4.15 (4.12-4.18)	4.05	4.16	-0.18
1.64	31	4.30 (4.26-4.33)	4.06	4.23	-0.20
1.64	32	4.33 (4.31-4.37)	4.10	4.28	-0.25
Average		4.54	4.41	4.52	-0.11

Table 21. Empirical neutralization potentials[1] mg $CaCO_3$/g rock of drainage from samples.

Sulfur %	Reactor	pH 7	Week	pH 6	Week	pH 5	Week	pH 4.5	Week	pH 4	Week	pH 3.5	Week
Group I													
0.18	1	0.56	55	>3.07	1252[2]	>3.07	1252[2]	>3.07	1252[2]	>3.07	1252[2]	>3.07	1252[2]
0.22	3	0.46	55	>3.12	1252[2]	>3.12	1252[2]	>3.12	1252[2]	>3.12	1252[2]	>3.12	1252[2]
0.72[3]	42	0.31	51	>1.48	809[2]	>1.48	809[2]	>1.48	809[2]	>1.48	809[2]	>1.48	809[2]
Group II													
0.40	5	0.71	18	7.98	669	8.53	746	9.03	780	9.85	804	>12.93	909[2]
0.41	7	0.26	55	0.77	80	2.16	278	3.10	353	>7.83	724[2]	>7.83	724[2]
0.51	9	0.61	53	1.02	67	2.77	229	3.22	279	>11.42	724[2]	>11.42	724[2]
0.54	11	0.89	31	1.84	68	4.98	221	5.85	265	>10.60	441[2]	>10.60	441[2]
0.57	13	0.93	31	2.01	80	4.43	259	4.93	282	6.81	331	>9.81	441[2]
0.58	15	0.42	4	1.45	54	2.93	146	3.58	181	>13.37	724[2]	>13.37	724[2]
0.67[3]	39	0	0	0.41	12	0.94	31	2.02	100	>6.37	809[2]	>6.37	809[2]
0.67[4]	40	0.29	11	1.00	39	4.28	307	5.07	343	>10.44	809[2]	>10.44	809[2]
0.82[4]	43	0.23	12	0.56	23	2.48	277	>4.40	809[2]	>4.40	809[2]	>4.40	809[2]
0.92[3]	41	0.14	12	0.49	39	>2.50	809[2]	>2.50	809[2]	>2.50	809[2]	>2.50	809[2]
1.71[3]	44	1.17	33	2.00	58	4.25	144[2]	>4.25	144[2]	>4.25	144[2]	>4.25	144[2]
Group III													
0.71	17	0.27	16	0.79	54	3.17	229	3.81	280	6.41	341	>15.04	724[2]
1.12	35	0	0	0	0	4.60	57	7.45	131	11.52	200	16.26	249
1.16	29	0	0	1.54	32	2.48	59	3.92	131	10.15	255	13.48	276
1.40	37	0.62	4	1.49	34	3.78	95	6.29	194	10.39	259	13.11	276
1.44	33	0.55	2	1.01	8	2.20	51	3.88	104	13.40	252	14.27	259
1.63	19	0.57	4	1.66	22	4.32	102	6.61	164	18.16	275	19.29	283
1.64	31	0	0	0	0	3.02	10	9.14	131	15.24	235	17.68	259
Virginia Formation													
2.06	21	0	0	0.51	4	1.65	12	5.38	65	7.30	78[2]	>7.30	78[2]
3.12	23	0	0	2.11	3	3.56	15	4.30	17	17.34	75	>19.34	78[2]
3.72	25	0	0	0	0	0	0	2.65	13	10.63	75	>11.71	78[2]
5.44	27	0	0	0	0	0	0	0	0	6.13	13	>17.68	78[2]

[1] Empirical neutralization potentials (ENP) were calculated using cumulative sulfate release at the point after which the pH was consistently below the given threshold value. ENP = [(cum. mg SO_4)(100 mg $CaCO_3$/96.1 mg SO_4)]/(75 g rock).
[2] Full period of record. Samples did not consistently fall below threshold value.
[3] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[4] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 22. Neutralization Potential (NP) of rock samples.

S%	Reactor	CO_2(%)	kg $CaCO_3$/t[1] rock				
			Acid Production	NP (Sobek 1978)	NP(pH6)[2]	NP[3] [(Ca/Mg)CO_3]	ENP(pH 6)
Group I, Duluth Complex							
0.18	1	<0.03	6	17		<0.7	>3.07
0.22	3	<0.03	7	16		<0.7	>3.12
0.72[4]	42	0.06	22	17	3.4	1.4	>1.48
Group II, Duluth Complex							
0.40	5	0.64	12	14		14.5	7.98
0.41	7	0.06	13	13		1.4	0.77
0.51	9	0.07	16	16		1.6	1.02
0.54	11	0.11	17	12		2.5	1.84
0.57	13	0.01	18	19		2.3	2.01
0.58	15	0.07	18	14		1.6	1.45
0.67[4]	39	<0.03	21	14	3.2	<0.7	0.41
0.67[5]	40	<0.03	21		2	<0.7	1.00
0.82[5]	43	<0.03	26		2.2	<0.7	0.56
0.92[4]	41	<0.03	29	4	2.1	<0.7	0.49
1.71[4]	44	0.06	53	21	5.4	1.4	2.00
Group III, Duluth Complex							
0.71	17	0.05	22	21		1.1	0.79
1.12	35	<0.03	35	15		<0.7	1.54
1.16	29	0.08	36	16		1.8	1.49
1.4	37	0.1	44	18		2.3	1.01
1.44	33	<0.03	45	16		<0.7	1.66
1.63	19	0.11	51	12		2.5	0
1.64	31	0.14	51	12		3.2	1.54
Virginia Formation							
2.06	21	0.12	64	14		2.7	0.51
3.12	23	<0.03	97	8		<0.7	2.11
3.72	25	0.06	116	5		1.4	0
5.44	27	0.32	170	0.5		7.3	0

[1] Metric ton = 1000 kg
[2] Autotitrator test slightly modified from (Lapakko 1994a). NP after 4 hours of less than or equal to 0.1 ml addition.
[3] Calculated from CO_2 content. NP [(Ca/Mg)CO_3] = 22.7 x percent CO_2
[4] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[5] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 23. Mineralogical breakdown of cations present in samples (%). Blank entries indicate no mineral present in sample.

%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
		Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
Group I, Duluth Complex																
0.18	1	67.06	22.92	10.02		19.63	47.15	20.51	12.71		100		32.22	67.78		
0.22	3	74.53	17.02	4.53	3.92	22.07	21.14	14.66	33.80	8.33	97.97	2.03	21.47	49.50	28.07	0.96
0.72[1]	42	79.07	4.08	16.85		6.37	13.08	80.55			100		100			
Group II, Duluth Complex																
0.4	5	74.07	11.82	9.69	4.42	4.26	73.99		13.18	8.57	97.69	2.31	18.79	43.33	37.37	0.52
0.41	7	78.00	17.72		4.28	16.26	24.42	30.48	21.67	7.17	99.65	0.35	16.29	70.85	11.86	1.00
0.51	9	71.66	18.65		9.68	13.39	43.28	11.91	19.98	11.44	95.25	4.75	9.86	47.81	42.15	0.18
0.54	11	79.36	14.17	3.46	3.01	15.57	54.17		19.87	10.39	100		17.83	82.15		0.02
0.57	13	60.23	39.77			31.01	14.46	54.54			97.60	2.40	21.08		78.92	
0.58	15	67.73	32.27			35.51	50.59	7.67	6.24		96.26	3.74	15.66	22.85	61.49	
0.67[1]	39	73.43	26.57			29.79	15.48	54.73			98.72	1.28	47.98		52.02	
0.67[2]	40	71.80	28.20			17.93	31.42	8.14	42.52		99.43	0.57	5.62	88.24	6.13	
0.82[2]	43	70.30	29.70			23.21	45.71	7.02	24.05		100		12.89	87.11		
0.92[1]	41	77.04	10.25		12.71	5.47	6.82	69.91		17.80	100		98.18			1.82
1.71[1]	44	88.11	6.69	5.20		6.19	9.84	75.38	8.59		97.77	2.23	41.31	46.78	11.92	
Group III, Duluth Complex																
0.71	17	81.75	18.25			15.33	34.36	35.72	14.58		99.21	0.79	26.10	59.52	14.39	
1.12	35	70.66	17.15	5.69	6.50	10.63	53.68	11.68	16.66	7.36	86.82	13.18	5.75	25.31	68.93	
1.16	29	79.52	20.48			20.02	35.86	44.12			100		100			
1.4	37	59.29	23.97	12.12	4.61	34.71	34.50	22.47		8.32	97.56	2.44	29.12		66.26	4.62
1.44	33	93.03	6.97			4.86	65.25	14.13	15.76		97.26	2.74	16.37	40.31	43.32	
1.63	19	70.29	22.45		7.26	11.37	52.01	12.57	18.57	5.49	96.28	3.72	8.56	36.74	53.19	1.50
1.64	31	63.96	31.51	4.53		29.82	34.56	8.93	26.68		97.37	2.63	9.46	50.71	39.83	
Virginia Formation																
2.06	21	81.67	18.33			10.09	70.85	4.90	14.16		85.18	14.82	4.95	11.50	83.55	
3.12	23	97.12	2.88			0.99	62.02		36.99		89.76	10.24	8.57	40.80	50.63	
3.72	25	77.72	22.28			8.18	42.05		49.77		83.45	16.55	2.10	27.09	70.81	
5.44	27	100							100		73.75	26.25	1.87	8.32	89.81	
Min (Duluth Complex)		59.29	4.08	0.00	0.00	4.26	6.82	0.00	0.00	0.00	86.82	0.00	5.62	0.00	0.00	0.00
Max (Duluth Complex)		93.03	39.77	16.85	12.71	35.51	73.99	80.55	42.52	17.80	100.00	13.18	100.00	88.24	78.92	4.62
Avg (Duluth Complex)		73.85	20.03	3.43	2.69	17.78	36.27	27.86	14.04	4.04	97.85	2.15	31.17	39.00	29.33	0.51
Avg (Virginia Formation)		89.13	10.87	0.00	0.00	4.81	43.73	1.22	50.23	0.00	83.04	16.96	4.37	21.93	73.70	0.00
Average for all samples		76.30	18.56	2.88	2.26	15.71	37.47	23.60	19.83	3.39	95.48	4.52	26.88	36.27	36.43	0.42

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 24. Sample time periods beginning and duration for each sample. See section 5.3.1.1 for definition of periods.

		Weeks comprising each period				Length of Period			
		A	B	C	D	A	B	C	D
%S	**Reactor**								
Group I									
0.18	1	1-137	138-1252			137	1115		
0.22	3	1-134	135-1252			134	1118		
0.72[1]	42	1-34	35-809			34	775		
Group II									
0.40	5	1-140	141-769	770-867	868-909	140	629	98	42
0.41	7	1-129	130-351	352-439	440-724	129	222	88	285
0.51	9	1-140	141-277	278-383	384-724	140	137	106	341
0.54	11	1-137	138-278	279-441		137	141	163	
0.57	13	1-103	104-294	295-351	352-441	103	191	57	90
0.58	15	1-101	102-217	218-335	336-724	101	116	118	389
0.67[1]	39	1-39	40-111	112-195	196-809	39	72	84	614
0.67[2]	40	1-58	59-345	346-431	432-809	58	287	86	378
0.82[2]	43	1-54	55-809			54	755		
0.92[1]	41	1-54	55-809			54	755		
1.71[1]	44	1-78	79-144			78	66		
Group III									
0.71	17	1-100	101-274	275-426	427-724	100	174	152	298
1.12	35	1-127	128-214	215-348	349-360	127	87	134	12
1.16	29	1-38	39-189	190-382	383-643	38	151	193	261
1.40	37	1-74	75-209	210-360		74	135	151	
1.44	33	1-76	77-212	213-444	445-643	76	136	232	199
1.63	19	1-66	67-155	156-410	411-724	66	89	255	314
1.64	31	1-64	65-228	229-340	341-360	64	164	112	20

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 25. Minimum drainage pH, sulfate and major cation cumulative mass release (mmol) over the entire period of record.

%S	Reactor	Period of Record (weeks)	Minimum pH	SO_4	Ca	Mg	Na	K
Group I, 3 samples								
0.18	**1**	1252	5.67	2.30	3.37	2.02	1.60	5.69
0.22	**3**	1252	5.91	2.34	3.94	3.28	1.22	3.45
0.72[1]	**42**	809	5.60	1.11	1.26	0.80	0.65	1.70
Average			5.73	1.92	2.86	2.03	1.16	3.61
Group II, 11 samples								
0.4	**5**	909	3.78	9.70	8.74	0.93	0.48	2.99
0.41	**7**	724	4.05	5.87	2.71	1.48	0.93	2.00
0.51	**9**	724	3.95	8.56	3.80	1.71	1.13	1.38
0.54	**11**	441	4.08	7.95	5.12	1.56	0.48	0.84
0.57	**13**	441	3.86	7.36	2.88	2.60	0.72	1.35
0.58	**15**	724	3.98	10.0	3.13	2.45	1.06	2.69
0.67[1]	**39**	809	4.22	4.78	1.27	1.12	0.69	2.45
0.67[2]	**40**	809	4.04	7.83	3.80	1.52	0.96	1.93
0.82[2]	**43**	809	4.49	3.30	1.68	0.55	0.42	1.08
0.92[1]	**41**	809	4.89	1.88	1.16	0.45	0.67	1.37
1.71[1]	**44**	144	5.06	3.19	1.60	1.45	0.18	1.54
Average			4.22	6.40	3.26	1.44	0.70	1.78
Group III, Duluth Complex, 7 samples								
0.71	**17**	724	3.53	11.3	3.02	4.06	1.16	2.73
1.12	**35**	360	3.23	20.6	6.06	2.55	0.86	1.61
1.16	**29**	643	3.24	19.4	3.47	4.75	1.58	2.76
1.40	**37**	360	3.06	22.7	3.59	5.85	1.30	4.62
1.44	**33**	643	3.04	28.3	3.62	6.00	1.95	5.16
1.63	**19**	724	3.32	28.9	4.75	6.94	1.35	3.02
1.64	**31**	360	3.38	20.6	4.43	4.69	0.93	0.96
Average			3.26	21.7	4.13	4.98	1.30	2.98
Virginia Formation, 4 samples								
2.06	21	78	4.20	5.47	1.63	2.42	N/A	N/A
3.12	23	78	3.70	14.5	3.69	3.68	N/A	N/A
3.72	25	78	3.70	8.78	2.85	3.23	N/A	N/A
5.44	27	78	3.35	13.3	3.10	4.74	N/A	N/A
Average			3.74	10.5	2.82	3.52	N/A	N/A

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 26. Ca/Na drainage quality vs plagioclase stoichiometry

Reactor	%S	Ca:Na (obs)[1]	Ca:Na (stoic)[2]	Obs/stoic
1	0.18	2.06	1.01	2.05
3	0.22	3.05	0.86	3.55
5	0.4	23.11	1.27	18.16
7	0.41	2.89	1.22	2.37
9	0.51	3.47	1.11	3.12
11	0.54	10.76	1.97	5.45
13	0.57	4.07	1.02	3.97
15	0.58	3.05	1.09	2.81
39	0.67	1.87	1.12	1.66
40	0.67	4.03	1.07	3.75
17	0.71	2.53	1.12	2.26
42	0.72	1.98	1.53	1.29
43	0.82	3.60	1.20	3.00
41	0.92	1.86	1.47	1.26
35	1.12	7.36	1.06	6.94
29	1.16	2.15	1.22	1.77
37	1.4	2.74	1.13	2.43
33	1.44	1.90	1.05	1.81
19	1.63	3.28	0.72	4.55
44	1.71	8.82	1.48	5.95
Average		4.73	1.19	3.91
Min		1.86	0.72	1.26
Max		23.11	1.97	18.16

[1] Cumulative mass release ratio for Ca to Na for entire period of record
[2] Stoichiometric ratio of Ca to Na in plagioclase for each sample

Table 27. Sample pH by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	6.90	7.90	6.20	134	6.42	7.18	5.67	565								
0.22	**3**	6.91	7.75	6.40	132	6.51	7.05	5.91	560								
0.72[1]	**42**	7.30	8.09	6.95	34	6.37	7.34	5.60	243								
Average		7.04	7.91	6.52	100	6.43	7.19	5.73	456								
Group II																	
0.40	**5**	6.60	7.40	6.16	139	6.13	6.69	4.58	453	4.21	4.53	3.78	45	4.67	4.98	4.50	21
0.41	**7**	6.51	8.35	5.34	128	5.13	6.13	4.47	222	4.22	4.56	4.05	87	4.75	5.82	4.26	127
0.51	**9**	6.30	8.70	5.08	139	5.07	5.45	4.61	137	4.18	4.51	3.95	105	4.64	6.04	4.07	182
0.54	**11**	6.13	8.05	5.13	136	4.98	5.55	4.24	141	4.39	4.77	4.08	162				
0.57	**13**	6.44	7.75	5.50	101	5.21	5.72	4.32	190	4.29	4.50	3.86	57	4.40	4.81	4.21	89
0.58	**15**	5.89	7.55	5.20	100	4.82	5.49	4.20	116	4.22	4.38	3.98	118	4.61	5.73	4.11	229
0.67[1]	**39**	5.59	6.81	4.82	39	4.67	4.81	4.32	56	4.36	4.60	4.22	27	5.34	6.17	4.37	148
0.67[2]	**40**	6.27	7.19	5.52	58	5.62	6.20	4.46	98	4.22	4.40	4.04	22	5.41	6.35	4.46	100
0.82[2]	**43**	6.09	7.52	5.31	54	5.28	6.12	4.49	217								
0.92[1]	**41**	6.43	8.13	5.62	53	5.55	6.25	4.89	219								
1.71[1]	**44**	6.62	7.60	5.66	76	5.42	6.12	5.06	32								
Average		6.26	7.73	5.39	93	5.26	5.87	4.51	171	4.26	4.53	4.00	78	4.83	5.70	4.28	128
Group III																	
0.71	**17**	6.15	7.65	5.11	100	5.08	5.45	4.41	173	4.22	4.95	3.53	151	4.90	5.43	4.52	140
1.12	**35**	4.87	5.85	4.48	125	4.13	4.52	3.81	87	3.61	4.17	3.23	133	4.09	4.18	3.98	12
1.16	**29**	6.38	6.95	5.69	37	4.62	5.70	4.14	150	3.93	4.78	3.24	192	4.95	5.64	4.50	106
1.40	**37**	5.92	7.30	4.78	73	4.81	5.18	4.01	134	3.66	4.49	3.06	150				
1.44	**33**	5.23	7.20	4.59	74	4.42	4.83	4.19	135	3.89	4.90	3.04	229	4.75	5.09	4.52	46
1.63	**19**	5.73	7.50	4.95	66	4.90	5.35	4.37	88	3.89	4.72	3.32	254	4.77	5.71	4.35	156
1.64	**31**	4.80	5.45	4.51	63	4.39	4.80	4.02	163	3.72	4.19	3.38	111	4.03	4.14	3.85	20
Average		5.58	6.84	4.87	77	4.62	5.12	4.14	133	3.84	4.60	3.26	174	4.58	5.03	4.29	80

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 28. Sample weekly sulfate release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	1.2E-11	3.0E-11	2.9E-12	59	2.3E-12	1.0E-11	1.9E-13	185								
0.22	**3**	1.0E-11	2.9E-11	3.0E-12	57	2.6E-12	9.3E-12	1.9E-13	187								
0.72[1]	**42**	8.8E-12	1.3E-11	3.9E-12	12	2.2E-12	7.9E-12	1.9E-13	120								
Average		1.0E-11	2.4E-11	3.3E-12	43	2.4E-12	9.2E-12	1.9E-13	164								
Group II																	
0.40	**5**	3.3E-11	7.4E-11	1.3E-11	61	9.9E-12	3.1E-11	3.0E-12	132	5.0E-11	1.1E-10	1.8E-11	21	1.2E-11	1.9E-11	7.9E-12	10
0.41	**7**	1.2E-11	3.8E-11	1.6E-12	58	9.5E-12	2.2E-11	3.0E-12	52	3.7E-11	5.0E-11	2.5E-11	21	1.1E-11	4.3E-11	2.3E-12	54
0.51	**9**	1.8E-11	4.7E-11	3.1E-12	61	8.3E-12	1.5E-11	3.0E-12	34	5.1E-11	7.7E-11	3.8E-11	25	1.6E-11	4.5E-11	3.4E-12	67
0.54	**11**	3.3E-11	7.0E-11	1.2E-11	60	2.3E-11	5.6E-11	6.0E-12	36	3.1E-11	5.9E-11	1.5E-11	38				
0.57	**13**	2.8E-11	4.2E-11	1.4E-11	50	1.8E-11	5.3E-11	3.0E-12	48	4.6E-11	7.4E-11	3.3E-11	13	3.1E-11	3.8E-11	2.3E-11	22
0.58	**15**	2.5E-11	1.0E-10	3.0E-12	49	2.6E-11	6.0E-11	1.3E-11	31	4.5E-11	7.1E-11	3.2E-11	27	1.5E-11	3.8E-11	3.4E-12	78
0.67[1]	**39**	3.2E-11	4.1E-11	1.4E-11	13	2.0E-11	2.6E-11	1.4E-11	16	2.6E-11	3.2E-11	2.0E-11	20	5.5E-12	2.2E-11	1.9E-13	79
0.67[2]	**40**	2.9E-11	6.0E-11	1.3E-11	18	1.5E-11	4.2E-11	9.3E-12	57	4.7E-11	6.3E-11	2.5E-11	10	6.2E-12	2.3E-11	1.8E-13	45
0.82[2]	**43**	2.2E-11	4.6E-11	1.0E-11	17	6.0E-12	1.8E-11	1.9E-13	113								
0.92[1]	**41**	1.3E-11	2.3E-11	7.4E-12	17	3.7E-12	1.2E-11	1.9E-13	112								
1.71[1]	**44**	3.8E-11	5.8E-11	2.8E-11	23	3.0E-11	4.0E-11	1.9E-11	17								
Average		2.6E-11	5.4E-11	1.1E-11	39	1.5E-11	3.4E-11	6.8E-12	59	4.2E-11	6.7E-11	2.6E-11	22	1.4E-11	3.3E-11	5.7E-12	51
Group III																	
0.71	**17**	2.0E-11	3.8E-11	3.1E-12	48	1.4E-11	2.9E-11	3.0E-12	45	7.4E-11	2.5E-10	1.4E-11	35	1.2E-11	3.3E-11	4.2E-12	58
1.12	**35**	7.3E-11	2.4E-10	2.6E-11	39	7.2E-11	9.0E-11	3.1E-11	20	1.4E-10	2.5E-10	3.9E-11	33	4.9E-11	5.2E-11	4.4E-11	3
1.16	**29**	4.2E-11	1.1E-10	2.0E-11	19	3.1E-11	5.8E-11	1.4E-11	37	1.2E-10	3.0E-10	1.8E-11	46	1.1E-11	2.8E-11	4.7E-12	47
1.40	**37**	4.5E-11	8.2E-11	2.7E-11	27	3.6E-11	8.9E-11	1.7E-11	32	1.9E-10	3.6E-10	6.2E-11	37				
1.44	**33**	4.9E-11	1.4E-10	2.3E-11	27	6.1E-11	1.3E-10	3.1E-11	32	1.4E-10	3.4E-10	1.8E-11	57	1.5E-11	2.6E-11	8.9E-12	32
1.63	**19**	5.4E-11	1.4E-10	1.8E-11	33	3.8E-11	5.8E-11	2.2E-11	29	1.4E-10	3.3E-10	3.5E-11	57	1.7E-11	4.7E-11	3.9E-12	61
1.64	**31**	1.2E-10	3.6E-10	6.0E-11	24	6.1E-11	9.0E-11	3.6E-11	38	1.3E-10	2.6E-10	5.2E-11	28	5.9E-11	6.7E-11	5.2E-11	5
Average		5.8E-11	1.6E-10	2.5E-11	31	4.5E-11	7.7E-11	2.2E-11	33	1.3E-10	3.0E-10	3.4E-11	42	2.7E-11	4.2E-11	2.0E-11	34

Note: The end of Period 1 for the 0.72%S sample was defined early relative to other samples. This means that Period 2 sulfate releases may be higher than expected.

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.

[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 29. Sample weekly calcium release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	1.6E-11	2.6E-11	1.0E-11	8	3.1E-12	1.2E-11	3.5E-13	189								
0.22	**3**	1.8E-11	3.1E-11	8.7E-12	7	3.5E-12	1.0E-11	3.6E-13	191								
0.72[1]	**42**	1.1E-11	4.2E-11	7.3E-12	12	2.4E-12	8.7E-12	3.5E-13	115								
Average		1.5E-11	3.3E-11	8.7E-12	9	3.0E-12	1.0E-11	3.6E-13	165								
Group II																	
0.40	**5**	3.7E-11	6.1E-11	2.7E-11	8	1.2E-11	2.8E-11	5.5E-12	133	1.2E-11	2.4E-11	4.8E-12	21	4.6E-12	6.8E-12	2.8E-12	9
0.41	**7**	1.7E-11	3.3E-11	3.6E-12	10	3.1E-12	1.2E-11	7.8E-13	52	5.7E-12	1.0E-11	3.5E-12	22	3.4E-12	7.2E-12	1.9E-12	54
0.51	**9**	1.9E-11	5.6E-11	2.2E-12	11	3.1E-12	5.2E-12	1.4E-12	33	8.8E-12	1.5E-11	5.8E-12	29	4.7E-12	1.6E-11	2.3E-12	66
0.54	**11**	3.4E-11	1.0E-10	1.1E-11	32	1.3E-11	2.2E-11	2.2E-12	37	9.0E-12	1.5E-11	5.2E-12	40				
0.57	**13**	2.5E-11	5.6E-11	8.5E-12	7	5.2E-12	1.2E-11	7.3E-13	45	8.6E-12	1.4E-11	5.7E-12	14	6.5E-12	8.6E-12	4.2E-12	21
0.58	**15**	2.0E-11	5.0E-11	5.2E-12	7	5.4E-12	1.0E-11	7.3E-13	24	6.8E-12	1.3E-11	3.6E-12	30	4.6E-12	1.1E-11	2.5E-12	80
0.67[1]	**39**	1.4E-11	3.1E-11	3.0E-12	13	2.8E-12	3.7E-12	1.5E-12	16	3.5E-12	4.3E-12	2.9E-12	21	2.0E-12	4.3E-12	3.4E-13	80
0.67[2]	**40**	1.8E-11	3.2E-11	3.6E-12	18	1.0E-11	1.4E-11	7.2E-12	56	9.3E-12	1.2E-11	7.9E-12	10	3.5E-12	7.9E-12	7.9E-13	44
0.82[2]	**43**	1.6E-11	3.1E-11	5.1E-12	17	3.0E-12	9.8E-12	3.3E-13	114								
0.92[1]	**41**	1.2E-11	2.2E-11	4.4E-12	17	1.9E-12	5.9E-12	9.0E-14	114								
1.71[1]	**44**	2.7E-11	5.0E-11	1.2E-11	23	6.2E-12	1.0E-11	3.6E-12	17								
Average		2.2E-11	4.8E-11	7.8E-12	15	5.9E-12	1.2E-11	2.2E-12	58	8.0E-12	1.3E-11	4.9E-12	23	4.2E-12	8.7E-12	2.1E-12	51
Group III																	
0.71	**17**	1.9E-11	2.7E-11	8.7E-12	6	3.1E-12	1.2E-11	7.3E-13	39	8.1E-12	2.8E-11	3.6E-13	38	4.3E-12	7.4E-12	2.2E-12	58
1.12	**35**	4.3E-11	8.2E-11	3.6E-12	27	1.3E-11	1.8E-11	8.0E-12	23	1.9E-11	3.2E-11	7.1E-12	33	8.7E-12	9.5E-12	7.8E-12	3
1.16	**29**	2.2E-11	3.4E-11	1.2E-11	10	5.7E-12	2.2E-11	2.3E-12	37	1.3E-11	2.8E-11	5.6E-12	48	4.6E-12	8.0E-12	2.9E-12	47
1.40	**37**	2.8E-11	5.6E-11	1.2E-11	18	8.3E-12	1.9E-11	4.4E-12	35	1.6E-11	2.7E-11	6.6E-12	38				
1.44	**33**	2.4E-11	6.1E-11	5.1E-12	16	5.0E-12	1.2E-11	2.2E-12	34	1.3E-11	2.7E-11	4.2E-12	57	3.8E-12	6.3E-12	2.5E-12	32
1.63	**19**	2.2E-11	5.0E-11	8.7E-12	22	7.6E-12	2.3E-11	3.6E-12	8	1.3E-11	2.7E-11	5.3E-12	64	4.9E-12	1.1E-11	2.5E-12	62
1.64	**31**	5.8E-11	1.6E-10	1.2E-11	12	9.9E-12	1.6E-11	6.4E-12	40	1.6E-11	2.4E-11	1.1E-11	28	1.1E-11	1.4E-11	9.9E-12	5
Average		3.1E-11	6.7E-11	8.8E-12	16	7.4E-12	1.7E-11	4.0E-12	31	1.4E-11	2.8E-11	5.7E-12	44	6.2E-12	9.4E-12	4.6E-12	35

Note: The end of Period 1 for the 0.72%S sample was defined early relative to other samples. This means that Period 2 Ca releases may be higher than expected.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 30. Sample weekly magnesium release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	8.2E-12	1.2E-11	1.2E-12	9	2.3E-12	8.9E-12	5.5E-13	187								
0.22	**3**	1.6E-11	5.5E-11	3.5E-12	8	3.5E-12	1.4E-11	5.6E-13	191								
0.72[1]	**42**	3.5E-12	7.7E-12	6.0E-13	12	1.6E-12	6.1E-12	5.8E-13	118								
Average		9.1E-12	2.5E-11	1.8E-12	10	2.5E-12	9.8E-12	5.6E-13	165								
Group II																	
0.40	**5**	5.0E-12	9.8E-12	1.2E-12	7	1.3E-12	9.5E-12	5.7E-13	129	1.1E-12	2.7E-12	5.8E-13	21	5.9E-13	6.0E-13	5.8E-13	9
0.41	**7**	5.4E-12	9.5E-12	1.2E-12	8	2.9E-12	9.8E-12	5.9E-13	51	7.2E-12	1.8E-11	5.9E-13	22	1.3E-12	5.9E-12	5.5E-13	54
0.51	**9**	6.1E-12	1.2E-11	1.2E-12	10	3.5E-12	1.0E-11	1.2E-12	33	7.1E-12	1.7E-11	6.1E-13	29	1.5E-12	7.1E-12	5.6E-13	66
0.54	**11**	1.5E-11	7.3E-11	1.2E-12	30	4.5E-12	1.1E-11	1.2E-12	36	1.7E-12	7.5E-12	5.9E-13	41				
0.57	**13**	1.4E-11	2.1E-11	7.0E-12	8	7.8E-12	1.9E-11	1.2E-12	45	1.4E-11	3.5E-11	4.7E-12	14	4.6E-12	9.4E-12	5.9E-13	22
0.58	**15**	1.6E-11	3.1E-11	3.7E-12	8	8.5E-12	1.4E-11	1.2E-12	24	8.5E-12	1.3E-11	3.5E-12	29	1.3E-12	5.9E-12	5.6E-13	79
0.67[1]	**39**	1.1E-11	1.7E-11	4.8E-12	13	4.9E-12	9.6E-12	1.2E-12	16	6.2E-12	8.1E-12	3.3E-12	21	9.3E-13	3.6E-12	1.5E-13	79
0.67[2]	**40**	1.0E-11	1.7E-11	3.6E-12	18	3.0E-12	5.6E-12	6.0E-13	57	7.0E-12	9.3E-12	2.8E-12	10	9.4E-13	3.9E-12	4.1E-13	44
0.82[2]	**43**	6.4E-12	9.6E-12	2.4E-12	17	7.1E-13	2.5E-12	1.5E-13	114								
0.92[1]	**41**	4.3E-12	8.5E-12	6.1E-13	18	6.5E-13	2.4E-12	1.5E-13	112								
1.71[1]	**44**	1.4E-11	2.8E-11	2.4E-12	23	1.7E-11	2.1E-11	1.2E-11	17								
Average		9.8E-12	2.1E-11	2.7E-12	15	5.0E-12	1.0E-11	1.8E-12	58	6.6E-12	1.4E-11	2.1E-12	23	1.6E-12	5.2E-12	4.8E-13	50
Group III																	
0.71	**17**	1.6E-11	2.8E-11	9.6E-12	7	6.9E-12	2.2E-11	1.2E-12	39	2.1E-11	6.5E-11	5.9E-12	38	2.2E-12	9.4E-12	5.9E-13	58
1.12	**35**	7.5E-12	1.9E-11	1.2E-12	25	9.6E-12	1.4E-11	4.7E-12	24	1.4E-11	2.5E-11	7.0E-12	33	8.3E-12	9.5E-12	7.0E-12	3
1.16	**29**	1.2E-11	2.3E-11	1.2E-12	9	1.0E-11	2.5E-11	6.1E-12	36	2.7E-11	5.5E-11	5.8E-12	48	2.4E-12	1.1E-11	5.8E-13	47
1.40	**37**	1.0E-11	2.2E-11	1.2E-12	16	1.5E-11	1.9E-11	6.1E-12	33	4.4E-11	8.2E-11	7.2E-12	38				
1.44	**33**	1.3E-11	2.1E-11	7.3E-12	12	1.1E-11	1.7E-11	6.0E-12	35	2.9E-11	6.5E-11	3.5E-12	57	3.2E-12	7.0E-12	5.8E-13	32
1.63	**19**	3.3E-11	8.1E-11	1.4E-11	26	1.9E-11	2.2E-11	1.4E-11	7	2.4E-11	4.1E-11	8.3E-12	65	4.3E-12	1.4E-11	5.3E-13	62
1.64	**31**	3.9E-11	8.9E-11	7.1E-12	12	1.5E-11	2.4E-11	7.2E-12	41	2.4E-11	3.9E-11	1.3E-11	28	1.1E-11	1.4E-11	8.1E-12	5
Average		1.8E-11	4.0E-11	6.0E-12	15	1.2E-11	2.0E-11	6.5E-12	31	2.6E-11	5.3E-11	7.2E-12	44	5.2E-12	1.1E-11	2.9E-12	35

Note: The end of Period 1 for the 0.72%S sample was defined early relative to other samples. This means that Period 2 Mg releases may be higher than expected.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 31. Sample weekly calcium+magnesium release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	2.4E-11	3.8E-11	1.4E-11	8	5.3E-12	1.7E-11	6.4E-13	187								
0.22	**3**	3.4E-11	8.0E-11	1.5E-11	7	7.0E-12	2.2E-11	9.7E-13	190								
0.72[1]	**42**	1.5E-11	4.7E-11	7.9E-12	12	3.9E-12	1.3E-11	4.1E-13	115								
Average		2.4E-11	5.5E-11	1.2E-11	9	5.4E-12	1.7E-11	6.7E-13	164								
Group II																	
0.40	**5**	4.3E-11	6.3E-11	3.1E-11	6	1.3E-11	3.0E-11	6.1E-12	128	1.3E-11	2.6E-11	5.4E-12	21	5.2E-12	7.4E-12	3.4E-12	9
0.41	**7**	2.5E-11	4.2E-11	7.2E-12	7	6.2E-12	1.9E-11	2.7E-12	48	1.3E-11	2.4E-11	6.3E-12	22	4.7E-12	1.1E-11	2.5E-12	54
0.51	**9**	2.6E-11	6.6E-11	3.4E-12	9	6.6E-12	1.2E-11	3.5E-12	32	1.6E-11	3.2E-11	8.7E-12	29	6.2E-12	2.3E-11	2.9E-12	66
0.54	**11**	5.0E-11	1.4E-10	2.0E-11	28	1.7E-11	2.9E-11	5.7E-12	36	1.1E-11	2.1E-11	5.8E-12	40				
0.57	**13**	3.8E-11	7.8E-11	1.8E-11	7	1.3E-11	2.7E-11	1.9E-12	45	2.3E-11	4.9E-11	1.0E-11	14	1.1E-11	1.7E-11	5.5E-12	21
0.58	**15**	3.5E-11	8.1E-11	8.8E-12	7	1.4E-11	2.4E-11	1.9E-12	24	1.5E-11	2.6E-11	1.0E-11	29	5.8E-12	1.4E-11	3.1E-12	79
0.67[1]	**39**	2.5E-11	4.8E-11	9.6E-12	12	7.7E-12	1.2E-11	4.2E-12	16	9.8E-12	1.2E-11	6.8E-12	21	2.9E-12	7.2E-12	1.5E-13	79
0.67[2]	**40**	2.8E-11	4.9E-11	7.3E-12	18	1.3E-11	1.9E-11	8.7E-12	56	1.6E-11	2.0E-11	1.1E-11	10	4.4E-12	1.0E-11	4.9E-13	44
0.82[2]	**43**	2.2E-11	4.0E-11	1.2E-11	17	3.7E-12	1.1E-11	1.3E-13	114								
0.92[1]	**41**	1.6E-11	2.7E-11	9.6E-12	17	2.5E-12	6.9E-12	6.0E-14	112								
1.71[1]	**44**	4.1E-11	7.2E-11	2.7E-11	23	2.4E-11	3.1E-11	1.5E-11	17								
Average		3.2E-11	6.4E-11	1.4E-11	14	1.1E-11	2.0E-11	4.6E-12	57	1.5E-11	2.6E-11	8.0E-12	23	5.8E-12	1.3E-11	2.6E-12	50
Group III																	
0.71	**17**	3.2E-11	4.3E-11	1.8E-11	6	1.0E-11	3.4E-11	1.9E-12	39	2.9E-11	8.9E-11	7.5E-12	37	6.6E-12	1.5E-11	2.7E-12	58
1.12	**35**	5.0E-11	1.0E-10	1.3E-11	25	2.2E-11	3.0E-11	1.5E-11	23	3.4E-11	5.5E-11	1.4E-11	33	1.7E-11	1.8E-11	1.5E-11	3
1.16	**29**	3.5E-11	4.8E-11	1.3E-11	9	1.6E-11	4.6E-11	9.7E-12	36	4.0E-11	7.7E-11	1.1E-11	48	7.0E-12	1.9E-11	3.5E-12	47
1.40	**37**	3.9E-11	6.4E-11	2.3E-11	16	2.3E-11	3.1E-11	1.6E-11	33	6.0E-11	1.1E-10	1.5E-11	38				
1.44	**33**	4.2E-11	7.6E-11	1.8E-11	12	1.7E-11	2.2E-11	1.0E-11	34	4.2E-11	8.9E-11	7.7E-12	57	7.0E-12	1.3E-11	3.1E-12	32
1.63	**19**	5.3E-11	9.3E-11	2.6E-11	22	2.6E-11	4.2E-11	1.8E-11	7	3.7E-11	5.9E-11	2.0E-11	64	9.2E-12	2.6E-11	3.1E-12	62
1.64	**31**	9.6E-11	2.4E-10	2.3E-11	12	2.5E-11	3.8E-11	1.8E-11	40	4.0E-11	6.3E-11	2.6E-11	28	2.1E-11	2.8E-11	1.8E-11	5
Average		5.0E-11	9.6E-11	1.9E-11	15	2.0E-11	3.5E-11	1.3E-11	30	4.0E-11	7.7E-11	1.4E-11	44	1.1E-11	2.0E-11	7.5E-12	35

Note: The end of Period 1 for the 0.72%S sample was defined early relative to other samples. This means that Period 2 Mg and Ca releases may be higher than expected.

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.

[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 32. Sample weekly sodium release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**					1.4E-12	3.2E-12	6.1E-13	32								
0.22	**3**					1.3E-12	2.3E-12	6.2E-13	34								
0.72[1]	**42**	3.1E-12	3.3E-12	2.8E-12	2	1.2E-12	1.9E-12	6.2E-13	33								
Average		3.1E-12	3.3E-12	2.8E-12	2	1.3E-12	2.5E-12	6.2E-13	33								
Group II																	
0.40	**5**					8.0E-13	2.1E-12	6.2E-13	20	2.2E-12	3.4E-12	1.5E-12	7	8.7E-13	1.4E-12	6.2E-13	3
0.41	**7**									2.5E-12	2.8E-12	2.2E-12	2	2.0E-12	3.6E-12	1.4E-12	14
0.51	**9**													2.5E-12	5.1E-12	1.5E-12	17
0.54	**11**									2.1E-12	2.4E-12	1.8E-12	2				
0.57	**13**													3.7E-12	4.8E-12	2.6E-12	2
0.58	**15**													2.4E-12	1.0E-11	1.3E-12	18
0.67[1]	**39**	2.3E-12	2.7E-12	2.0E-12	2	2.6E-12	4.8E-12	1.7E-12	4	2.4E-12	2.8E-12	2.0E-12	7	1.2E-12	2.3E-12	6.2E-13	25
0.67[2]	**40**	1.9E-12	2.4E-12	1.7E-12	3	1.9E-12	3.6E-12	6.3E-13	18	2.7E-12	3.1E-12	2.4E-12	4	2.1E-12	3.7E-12	6.2E-13	10
0.82[2]	**43**	1.7E-12	2.4E-12	6.3E-13	3	7.9E-13	1.8E-12	6.2E-13	32								
0.92[1]	**41**	2.1E-12	2.2E-12	2.0E-12	3	1.2E-12	4.4E-12	6.3E-13	33								
1.71[1]	**44**	2.1E-12	2.8E-12	1.6E-12	4	1.8E-12	2.2E-12	1.4E-12	2								
Average		2.0E-12	2.5E-12	1.6E-12	3	1.5E-12	3.1E-12	9.2E-13	18	2.4E-12	2.9E-12	2.0E-12	4	2.1E-12	4.5E-12	1.2E-12	13
Group III																	
0.71	**17**									2.6E-12	3.0E-12	2.3E-12	2	2.3E-12	3.3E-12	1.6E-12	16
1.12	**35**									6.4E-12	9.2E-12	3.6E-12	2	3.0E-12	3.0E-12	3.0E-12	1
1.16	**29**									4.0E-12	5.5E-12	2.9E-12	3	2.4E-12	3.7E-12	1.5E-12	15
1.40	**37**									5.2E-12	5.9E-12	4.1E-12	3				
1.44	**33**									3.6E-12	7.1E-12	2.3E-12	7	2.2E-12	3.6E-12	1.5E-12	11
1.63	**19**									3.5E-12	3.5E-12	3.5E-12	1	2.4E-12	4.0E-12	1.6E-12	18
1.64	**31**									4.2E-12	4.2E-12	4.2E-12	1	4.1E-12	4.5E-12	3.8E-12	2
Average										4.2E-12	5.5E-12	3.3E-12	3	2.7E-12	3.7E-12	2.2E-12	11

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 33. Sample weekly potassium release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**					5.7E-12	1.6E-11	7.2E-13	36								
0.22	**3**					3.3E-12	1.2E-11	3.6E-13	36								
0.72[1]	**42**	1.4E-11	1.8E-11	1.1E-11	2	2.9E-12	1.4E-11	3.6E-13	34								
Average		1.4E-11	1.8E-11	1.1E-11	2	4.0E-12	1.4E-11	4.8E-13	35								
Group II																	
0.40	**5**					4.6E-12	1.3E-11	1.3E-12	18	1.1E-12	2.2E-12	3.6E-13	7	3.7E-13	3.7E-13	3.7E-13	2
0.41	**7**									6.3E-12	8.0E-12	4.5E-12	2	3.7E-12	1.0E-11	1.2E-12	13
0.51	**9**													3.2E-12	9.2E-12	3.7E-13	17
0.54	**11**									3.6E-12	4.0E-12	3.2E-12	2				
0.57	**13**													7.6E-12	1.0E-11	4.9E-12	2
0.58	**15**													4.9E-12	1.7E-11	3.7E-13	17
0.67[1]	**39**	3.5E-11	6.5E-11	4.8E-12	2	1.4E-11	2.7E-11	2.6E-12	4	7.2E-12	1.3E-11	4.6E-12	7	1.4E-12	5.6E-12	3.6E-13	27
0.67[2]	**40**	1.3E-11	2.7E-11	5.8E-12	3	5.3E-12	1.8E-11	2.2E-12	19	3.5E-12	4.8E-12	1.5E-12	4	1.1E-12	2.4E-12	3.6E-13	12
0.82[2]	**43**	1.1E-11	1.8E-11	3.7E-12	3	1.1E-12	1.1E-11	3.6E-13	31								
0.92[1]	**41**	1.3E-11	2.5E-11	5.8E-12	3	1.5E-12	2.0E-11	3.7E-13	33								
1.71[1]	**44**	1.4E-11	1.9E-11	1.1E-11	4	1.8E-11	2.1E-11	1.5E-11	2								
Average		1.7E-11	3.1E-11	6.2E-12	3	7.4E-12	1.8E-11	3.7E-12	18	4.3E-12	6.4E-12	2.8E-12	4	3.2E-12	7.9E-12	1.1E-12	13
Group III																	
0.71	**17**									5.0E-12	6.1E-12	3.9E-12	2	6.0E-12	1.9E-11	9.0E-13	17
1.12	**35**									6.0E-12	7.4E-12	4.7E-12	2	6.2E-12	6.2E-12	6.2E-12	1
1.16	**29**									7.1E-12	9.8E-12	4.9E-12	3	3.4E-12	1.1E-11	1.0E-12	15
1.40	**37**									1.6E-11	2.1E-11	7.7E-12	3				
1.44	**33**									9.5E-12	2.2E-11	2.4E-12	7	1.5E-12	2.4E-12	7.6E-13	10
1.63	**19**									6.5E-12	6.5E-12	6.5E-12	1	6.2E-12	2.1E-11	1.5E-12	17
1.64	**31**									4.1E-12	4.1E-12	4.1E-12	1	6.0E-12	6.7E-12	5.3E-12	2
Average										7.7E-12	1.1E-11	4.9E-12	3	4.9E-12	1.1E-11	2.6E-12	10

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 34. Slopes, y-intercepts, and r^2 values from regression analysis of relationships among %S, sulfate, pH, Ca and Mg. See Figures 20-43 for data used.

independent	dependent	Period A			Period B			Period C			Period D		
		slope	y-int	r^2	slope	y-int	r^2	slope	y-int	r^2	slope	y-int	r^2
log %S	pH avg	-1.37	5.96	0.340	-1.69	4.98	0.452	-1.00	3.97	0.693	-0.46	4.62	0.070
log %S	pH max	-1.36	7.27	0.238	-1.84	5.55	0.387	-0.02	4.56	0.000	-1.63	5.23	0.254
log %S	pH min	-1.34	5.20	0.314	-1.21	4.39	0.339	-1.58	3.50	0.704	0.02	4.25	0.000
log %S	log SO_4 avg	0.82	1.28	0.525	1.35	1.09	0.521	1.03	1.68	0.779	0.76	1.00	0.140
log %S	log SO_4 max	0.71	1.67	0.317	0.85	1.43	0.413	1.37	2.03	0.671	0.33	1.35	0.163
log %S	log SO_4 min	1.19	0.93	0.563	1.93	0.64	0.399	0.37	1.25	0.172	1.49	0.62	0.147
log SO_4 avg	pH avg	-1.65	8.07	0.625	-1.17	6.27	0.757	-0.98	5.61	0.904	-1.06	5.69	0.961
log SO_4 avg	pH max	-1.97	9.76	0.638	-1.37	7.04	0.751	-0.13	4.76	0.019	-1.74	6.93	0.906
log SO_4 avg	pH min	-1.43	7.05	0.464	-0.94	5.41	0.718	-1.52	6.04	0.879	-0.45	4.71	0.512
log SO_4 avg	log Ca avg	0.43	0.55	0.533	0.38	0.10	0.473	0.60	-0.20	0.681	0.60	-0.13	0.937
log SO_4 avg	log Ca max	0.48	0.89	0.485	0.17	0.72	0.149	0.73	-0.12	0.608	0.28	0.43	0.541
log SO_4 avg	log Ca min	0.22	0.37	0.063	0.82	-0.82	0.610	0.38	-0.17	0.083	1.02	-0.78	0.917
log SO_4 avg	log Mg avg	0.56	0.08	0.379	0.75	-0.27	0.675	1.42	-1.49	0.604	0.94	-0.73	0.877
log SO_4 avg	log Mg max	0.56	0.45	0.284	0.36	0.52	0.391	1.24	-0.82	0.535	0.33	0.40	0.453
log SO_4 avg	log Mg min	0.54	-0.47	0.169	0.91	-0.87	0.636	1.28	-1.77	0.478	1.31	-1.57	0.811
log SO_4 avg	log Ca+Mg avg	0.50	0.68	0.696	0.55	0.28	0.764	0.92	-0.34	0.936	0.71	-0.03	0.971
log SO_4 avg	log Ca+Mg max	0.46	1.05	0.447	0.28	0.88	0.423	0.97	-0.14	0.718	0.33	0.67	0.538
log SO_4 avg	log Ca+Mg min	0.42	0.42	0.301	0.93	-0.48	0.716	0.61	-0.18	0.610	1.09	-0.70	0.954

Table 35. Iron, aluminum and silicon release rates for all Duluth Complex samples combined. Average, minimum and maximum rates are presented for entire period of record, as well as for pH>4 and pH≤4 (n = number of measurements).

Criteria	Stats	Fe (mol/s)	Al (mol/s)	Si (mol/s)
Entire period of record	Avg	5.00E-13	3.10E-12	1.21E-11
	Min	4.81E-15	1.03E-14	1.22E-13
	Max	3.67E-11	8.25E-11	1.73E-10
	n	441	445	425
pH > 4	Avg	2.73E-13	1.23E-12	1.18E-11
	Min	4.81E-15	1.03E-14	1.22E-13
	Max	3.67E-11	2.31E-11	1.73E-10
	n	410	414	418
pH ≤ 4	Avg	3.51E-12	2.81E-11	2.93E-11
	Min	1.20E-13	2.13E-14	4.51E-12
	Max	2.39E-11	8.25E-11	4.03E-11
	n	31	31	7

Table 36. Sample weekly cobalt release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	5.1E-14	1.5E-13	2.4E-14	9	6.6E-15	2.4E-14	4.8E-15	38								
0.22	**3**	4.0E-14	9.8E-14	2.4E-14	8	7.2E-15	2.0E-14	4.8E-15	39								
0.72[1]	**42**	5.0E-15	5.0E-15	5.0E-15	2	4.9E-15	5.4E-15	4.7E-15	33								
Average		3.2E-14	8.3E-14	1.8E-14	6	6.2E-15	1.7E-14	4.8E-15	37								
Group II																	
0.40	**5**	9.0E-14	2.0E-13	2.5E-14	8	1.1E-14	2.1E-14	4.9E-15	22	3.6E-14	7.3E-14	1.2E-14	6	4.9E-15	4.9E-15	4.8E-15	3
0.41	**7**	9.9E-14	2.0E-13	2.4E-14	11	7.4E-14	1.0E-13	5.1E-14	4	9.0E-14	1.2E-13	6.0E-14	4	1.1E-14	3.2E-14	4.6E-15	13
0.51	**9**	9.2E-14	2.1E-13	2.5E-14	11	3.7E-14	4.9E-14	2.6E-14	2	1.4E-13	2.4E-13	7.6E-14	4	1.1E-14	4.4E-14	4.8E-15	17
0.54	**11**	7.5E-14	2.0E-13	2.4E-14	33	5.0E-14	5.0E-14	5.0E-14	2	1.4E-14	2.3E-14	5.0E-15	6				
0.57	**13**	9.3E-14	2.4E-13	2.4E-14	8	1.4E-13	2.5E-13	4.9E-14	6	1.5E-13	2.1E-13	9.2E-14	2	3.7E-14	7.9E-14	1.5E-14	4
0.58	**15**	2.5E-13	3.4E-13	1.5E-13	8	1.1E-13	1.5E-13	9.8E-14	4	4.3E-14	4.3E-14	4.3E-14	1	8.7E-15	4.0E-14	4.7E-15	18
0.67[1]	**39**	4.0E-13	4.1E-13	3.9E-13	2	9.3E-14	1.5E-13	6.0E-14	4	4.8E-14	6.5E-14	3.8E-14	3	7.8E-15	3.1E-14	4.8E-15	27
0.67[2]	**40**	4.8E-14	8.9E-14	4.9E-15	3	5.8E-14	1.1E-13	3.7E-14	15	5.6E-14	8.7E-14	3.7E-14	4	6.3E-15	1.8E-14	4.7E-15	12
0.82[2]	**43**	1.0E-13	2.1E-13	5.0E-15	3	1.5E-14	8.7E-14	4.8E-15	31								
0.92[1]	**41**	2.5E-14	3.7E-14	5.0E-15	3	1.6E-14	5.0E-14	4.9E-15	31								
1.71[1]	**44**	5.0E-14	1.7E-13	4.9E-15	4	2.3E-13	2.3E-13	2.2E-13	2								
Average		1.2E-13	2.1E-13	6.2E-14	9	7.5E-14	1.1E-13	5.5E-14	11	7.2E-14	1.1E-13	4.5E-14	4	1.2E-14	3.6E-14	6.3E-15	13
Group III																	
0.71	**17**	1.8E-13	4.0E-13	2.5E-14	7	2.2E-13	3.0E-13	9.9E-14	5	1.2E-13	2.5E-13	1.2E-14	6	6.8E-15	1.8E-14	4.7E-15	15
1.12	**35**	1.0E-12	1.9E-12	9.9E-14	11	1.7E-13	2.4E-13	9.8E-14	2	1.1E-13	2.0E-13	2.8E-14	5	2.1E-14	2.1E-14	2.1E-14	1
1.16	**29**	3.6E-14	5.2E-14	2.6E-14	10	4.9E-13	5.7E-13	3.5E-13	4	1.8E-13	3.4E-13	6.2E-14	7	1.2E-14	3.5E-14	4.8E-15	13
1.40	**37**	1.6E-13	5.6E-13	2.5E-14	15	6.0E-13	6.0E-13	6.0E-13	1	4.4E-13	7.8E-13	2.0E-13	7				
1.44	**33**	2.7E-13	4.5E-13	2.6E-14	12	2.7E-13	3.0E-13	2.5E-13	2	1.9E-13	4.6E-13	3.3E-14	10	9.7E-15	2.1E-14	4.8E-15	10
1.63	**19**	6.6E-13	9.4E-13	2.0E-13	26	3.7E-13	5.5E-13	2.6E-13	3	1.2E-13	2.1E-13	3.3E-14	7	8.1E-15	2.1E-14	4.4E-15	16
1.64	**31**	2.1E-12	2.9E-12	4.9E-13	10	1.5E-13	1.5E-13	1.5E-13	1	1.3E-13	1.8E-13	5.8E-14	4	2.9E-14	3.0E-14	2.9E-14	2
Average		6.3E-13	1.0E-12	1.3E-13	13	3.3E-13	3.9E-13	2.6E-13	3	1.8E-13	3.4E-13	6.1E-14	7	1.5E-14	2.4E-14	1.1E-14	10

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 37. Sample weekly copper release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	4.3E-14	1.4E-13	2.2E-14	9	2.1E-14	5.6E-14	4.5E-15	37								
0.22	**3**	4.3E-14	1.4E-13	2.2E-14	8	2.6E-14	6.0E-14	1.4E-14	37								
0.72[1]	**42**	8.8E-15	1.3E-14	4.6E-15	2	2.2E-14	7.7E-14	4.5E-15	32								
Average		3.1E-14	9.5E-14	1.6E-14	6	2.3E-14	6.4E-14	7.5E-15	35								
Group II																	
0.40	**5**	5.4E-14	1.8E-13	2.3E-14	8	2.2E-13	9.0E-13	3.5E-14	22	8.4E-13	1.7E-12	2.9E-13	6	1.8E-13	2.1E-13	1.7E-13	3
0.41	**7**	1.5E-13	9.2E-13	2.3E-14	11	1.3E-12	1.8E-12	9.0E-13	4	1.5E-12	1.9E-12	9.7E-13	4	5.8E-13	1.2E-12	2.1E-13	13
0.51	**9**	7.9E-13	3.1E-12	2.2E-14	11	1.7E-12	2.1E-12	1.3E-12	2	3.4E-12	4.5E-12	2.8E-12	4	1.3E-12	3.3E-12	5.2E-14	17
0.54	**11**	5.2E-14	2.7E-13	2.2E-14	34	2.8E-13	2.8E-13	2.8E-13	2	2.2E-13	4.1E-13	7.6E-14	6				
0.57	**13**	1.3E-13	3.1E-13	2.2E-14	8	1.9E-12	4.1E-12	7.1E-13	6	2.4E-12	2.4E-12	2.4E-12	1	2.5E-12	3.0E-12	1.8E-12	4
0.58	**15**	2.0E-13	4.5E-13	2.3E-14	8	8.5E-13	1.1E-12	5.5E-13	4	1.1E-12	1.1E-12	1.1E-12	1	6.8E-13	1.3E-12	1.4E-13	18
0.67[1]	**39**	6.8E-13	1.3E-12	6.1E-14	2	1.4E-12	1.7E-12	1.1E-12	4	1.7E-12	2.2E-12	1.4E-12	3	6.4E-13	1.7E-12	1.9E-14	27
0.67[2]	**40**	2.7E-14	6.7E-14	4.5E-15	3	4.0E-13	1.3E-12	7.1E-14	15	1.3E-12	1.8E-12	8.3E-13	4	4.5E-13	1.0E-12	5.1E-14	12
0.82[2]	**43**	2.3E-13	3.6E-13	9.2E-15	3	4.9E-13	8.8E-13	1.9E-13	32								
0.92[1]	**41**	6.5E-14	1.6E-13	9.2E-15	3	7.7E-13	1.4E-12	3.3E-13	29								
1.71[1]	**44**	2.3E-14	7.3E-14	4.6E-15	4	2.0E-13	2.7E-13	1.4E-13	2								
Average		2.2E-13	6.6E-13	2.0E-14	9	8.6E-13	1.4E-12	5.1E-13	11	1.5E-12	2.0E-12	1.2E-12	4	9.1E-13	1.7E-12	3.5E-13	13
Group III																	
0.71	**17**	2.3E-13	6.9E-13	2.3E-14	7	2.0E-12	2.5E-12	1.3E-12	5	3.2E-12	5.4E-12	6.3E-13	6	1.9E-12	3.8E-12	8.3E-13	15
1.12	**35**	2.0E-13	7.8E-13	2.3E-14	11	1.3E-12	1.3E-12	1.2E-12	2	2.8E-12	4.7E-12	1.1E-12	5	9.1E-13	9.1E-13	9.1E-13	1
1.16	**29**	4.8E-14	9.5E-14	2.4E-14	10	1.2E-12	2.1E-12	3.2E-13	4	2.1E-12	4.5E-12	4.2E-13	7	1.2E-12	2.3E-12	9.0E-13	12
1.40	**37**	1.5E-13	9.8E-13	2.4E-14	15	1.7E-12	1.7E-12	1.7E-12	1	3.8E-12	6.7E-12	1.4E-12	7				
1.44	**33**	1.5E-13	6.0E-13	2.3E-14	12	7.4E-13	1.0E-12	4.6E-13	2	2.9E-12	5.4E-12	7.2E-13	10	3.7E-12	5.0E-12	1.2E-12	10
1.63	**19**	2.8E-13	7.5E-13	2.3E-14	26	9.9E-13	1.3E-12	4.2E-13	3	4.2E-12	7.6E-12	9.7E-13	8	1.8E-12	4.5E-12	3.6E-13	16
1.64	**31**	7.1E-13	1.3E-12	4.6E-14	10	3.2E-12	3.2E-12	3.2E-12	1	7.2E-12	8.8E-12	6.2E-12	4	3.4E-12	3.6E-12	3.3E-12	2
Average		2.5E-13	7.4E-13	2.6E-14	13	1.6E-12	1.9E-12	1.2E-12	3	3.7E-12	6.2E-12	1.6E-12	7	2.2E-12	3.4E-12	1.2E-12	9

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 38. Sample weekly nickel release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	9.2E-14	3.0E-13	2.4E-14	9	1.9E-14	5.5E-14	4.8E-15	37								
0.22	**3**	7.7E-14	2.0E-13	2.4E-14	8	1.7E-14	5.4E-14	4.8E-15	37								
0.72[1]	**42**	7.5E-15	1.0E-14	5.0E-15	2	1.0E-14	2.1E-14	4.7E-15	31								
Average		5.9E-14	1.7E-13	1.8E-14	6	1.5E-14	4.3E-14	4.8E-15	35								
Group II																	
0.40	**5**	4.5E-13	7.1E-13	1.0E-13	8	9.7E-14	2.9E-13	4.3E-14	22	6.1E-13	1.3E-12	1.9E-13	6	7.3E-14	9.7E-14	5.4E-14	3
0.41	**7**	1.2E-12	2.9E-12	2.4E-14	11	1.6E-12	2.7E-12	9.3E-13	4	1.6E-12	2.2E-12	1.1E-12	4	1.8E-13	5.9E-13	2.0E-14	13
0.51	**9**	9.1E-13	2.6E-12	2.6E-14	10	3.8E-13	4.1E-13	3.5E-13	2	1.6E-12	2.0E-12	1.1E-12	4	1.1E-13	6.0E-13	4.9E-15	17
0.54	**11**	3.7E-13	1.9E-12	2.5E-14	33	3.5E-13	3.5E-13	3.5E-13	2	1.2E-13	2.0E-13	4.6E-14	6				
0.57	**13**	8.3E-13	2.0E-12	2.4E-14	8	1.2E-12	1.9E-12	7.7E-13	6	1.3E-12	1.3E-12	1.3E-12	1	5.1E-13	1.1E-12	2.1E-13	4
0.58	**15**	1.5E-12	2.3E-12	1.0E-12	8	8.5E-13	1.1E-12	7.0E-13	4	4.7E-13	4.7E-13	4.7E-13	1	7.7E-14	4.6E-13	4.9E-15	18
0.67[1]	**39**	3.0E-12	5.9E-12	9.9E-15	2	1.6E-12	2.4E-12	1.2E-12	4	9.1E-13	1.2E-12	7.1E-13	3	9.4E-14	5.3E-13	4.8E-15	27
0.67[2]	**40**	4.3E-13	7.1E-13	6.8E-14	3	4.6E-13	7.9E-13	2.9E-13	15	6.7E-13	1.0E-12	4.4E-13	4	6.0E-14	2.2E-13	4.7E-15	12
0.82[2]	**43**	9.9E-13	2.0E-12	1.0E-13	3	1.7E-13	8.7E-13	2.7E-14	31								
0.92[1]	**41**	2.3E-13	3.7E-13	5.0E-15	3	2.2E-13	5.1E-13	5.1E-15	31								
1.71[1]	**44**	2.5E-13	8.8E-13	4.9E-15	4	1.1E-12	1.1E-12	1.1E-12	2								
Average		9.1E-13	2.0E-12	1.3E-13	8	7.3E-13	1.1E-12	5.3E-13	11	9.2E-13	1.2E-12	6.7E-13	4	1.6E-13	5.1E-13	4.4E-14	13
Group III																	
0.71	**17**	1.2E-12	3.2E-12	2.5E-14	7	2.3E-12	3.2E-12	8.0E-13	5	1.3E-12	2.2E-12	1.3E-13	6	4.9E-14	1.8E-13	4.7E-15	15
1.12	**35**	7.3E-12	1.3E-11	1.2E-12	11	1.4E-12	1.5E-12	1.4E-12	2	1.4E-12	2.6E-12	2.9E-13	5	2.1E-13	2.1E-13	2.1E-13	1
1.16	**29**	1.3E-13	2.6E-13	5.1E-14	10	5.6E-12	6.8E-12	3.6E-12	4	2.6E-12	4.9E-12	8.3E-13	7	1.4E-13	4.5E-13	3.5E-14	13
1.40	**37**	1.5E-12	6.4E-12	1.0E-13	15	6.6E-12	6.6E-12	6.6E-12	1	5.9E-12	1.1E-11	2.6E-12	6				
1.44	**33**	2.4E-12	3.7E-12	2.7E-13	12	2.5E-12	2.6E-12	2.5E-12	2	1.9E-12	4.6E-12	1.9E-13	10	7.2E-14	1.9E-13	3.4E-14	10
1.63	**19**	5.1E-12	7.1E-12	1.4E-12	26	3.0E-12	3.6E-12	2.7E-12	3	1.4E-12	2.8E-12	2.1E-13	8	3.7E-14	1.1E-13	4.7E-15	16
1.64	**31**	2.2E-11	3.6E-11	5.8E-12	10	2.2E-12	2.2E-12	2.2E-12	1	2.0E-12	2.9E-12	8.1E-13	4	3.9E-13	3.9E-13	3.8E-13	2
Average		5.7E-12	9.9E-12	1.3E-12	13	3.4E-12	3.8E-12	2.8E-12	3	2.3E-12	4.4E-12	7.2E-13	7	1.5E-13	2.6E-13	1.1E-13	10

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 39. Sample weekly zinc release (mol/second) by groups and time periods for all Duluth Complex samples (%S≤1.71).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	6.6E-14	1.3E-13	2.2E-14	9	5.2E-14	2.2E-13	9.0E-15	38								
0.22	**3**	5.2E-14	8.8E-14	2.2E-14	8	4.6E-14	2.3E-13	1.3E-14	38								
0.72[1]	**42**	6.6E-14	6.7E-14	6.6E-14	2	2.8E-14	8.5E-14	4.4E-15	33								
Average		6.1E-14	9.5E-14	3.6E-14	6	4.2E-14	1.8E-13	8.9E-15	36								
Group II																	
0.40	**5**	1.0E-13	1.8E-13	4.5E-14	8	8.7E-14	2.6E-13	2.9E-14	22	1.0E-13	1.8E-13	4.8E-14	6	2.4E-14	3.7E-14	1.7E-14	3
0.41	**7**	1.4E-13	3.1E-13	2.2E-14	11	8.3E-14	1.8E-13	4.6E-14	4	1.8E-13	3.5E-13	7.0E-14	4	8.8E-14	1.8E-13	2.9E-14	13
0.51	**9**	1.8E-13	6.3E-13	2.3E-14	11	9.0E-14	9.2E-14	8.9E-14	2	1.6E-13	2.2E-13	1.2E-13	4	1.2E-13	3.5E-13	3.1E-14	17
0.54	**11**	1.4E-13	5.7E-13	2.2E-14	34	1.8E-13	1.8E-13	1.8E-13	2	1.4E-13	2.3E-13	6.2E-14	6				
0.57	**13**	1.7E-13	4.8E-13	2.2E-14	8	2.0E-13	2.2E-13	1.3E-13	6	2.3E-13	2.4E-13	2.2E-13	2	1.9E-13	2.5E-13	1.2E-13	4
0.58	**15**	2.7E-13	4.8E-13	2.2E-14	8	2.6E-13	4.0E-13	1.3E-13	4	8.8E-14	8.8E-14	8.8E-14	1	9.5E-14	2.0E-13	2.2E-14	18
0.67[1]	**39**	3.3E-13	4.1E-13	2.5E-13	2	2.0E-13	3.4E-13	9.0E-14	3	9.0E-14	1.1E-13	7.9E-14	3	3.9E-14	1.1E-13	1.1E-14	27
0.67[2]	**40**	1.4E-13	1.9E-13	8.5E-14	3	1.5E-13	3.1E-13	8.6E-14	15	1.1E-13	1.6E-13	8.4E-14	4	7.1E-14	1.8E-13	2.4E-14	12
0.82[2]	**43**	2.0E-13	3.4E-13	1.0E-13	3	5.4E-14	2.0E-13	4.5E-15	31								
0.92[1]	**41**	1.3E-13	1.5E-13	8.4E-14	3	3.7E-14	1.8E-13	4.4E-15	32								
1.71[1]	**44**	1.2E-13	2.5E-13	6.5E-14	4	1.2E-13	1.4E-13	9.0E-14	2								
Average		1.8E-13	3.6E-13	6.8E-14	9	1.3E-13	2.3E-13	8.0E-14	11	1.4E-13	2.0E-13	9.7E-14	4	8.9E-14	1.9E-13	3.6E-14	13
Group III																	
0.71	**17**	1.8E-13	4.0E-13	2.2E-14	7	1.8E-13	3.2E-13	9.0E-14	5	1.7E-13	2.5E-13	1.0E-13	6	8.9E-14	2.4E-13	4.6E-14	15
1.12	**35**	4.1E-13	8.0E-13	1.3E-13	11	3.1E-13	4.4E-13	1.8E-13	2	4.3E-13	5.9E-13	2.9E-13	5	2.4E-13	2.4E-13	2.4E-13	1
1.16	**29**	4.0E-14	4.7E-14	2.3E-14	10	3.7E-13	6.2E-13	1.8E-13	4	2.7E-13	5.0E-13	1.4E-13	7	1.4E-13	9.1E-13	3.5E-14	13
1.40	**37**	1.2E-13	5.0E-13	2.3E-14	15	2.7E-13	2.7E-13	2.7E-13	1	3.1E-13	5.0E-13	1.2E-13	7				
1.44	**33**	2.3E-13	3.6E-13	9.5E-14	12	1.8E-13	2.2E-13	1.4E-13	2	2.4E-13	5.0E-13	1.1E-13	10	7.5E-14	1.3E-13	3.7E-14	9
1.63	**19**	3.2E-13	6.3E-13	4.4E-14	26	4.3E-13	5.0E-13	3.3E-13	3	4.9E-13	8.5E-13	1.8E-13	8	1.7E-13	4.2E-13	4.0E-14	16
1.64	**31**	7.2E-13	9.8E-13	2.7E-13	10	2.2E-13	2.2E-13	2.2E-13	1	3.6E-13	4.0E-13	2.9E-13	4	2.2E-13	2.8E-13	1.6E-13	2
Average		2.9E-13	5.3E-13	8.7E-14	13	2.8E-13	3.7E-13	2.0E-13	3	3.2E-13	5.2E-13	1.8E-13	7	1.6E-13	3.7E-13	9.2E-14	9

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 40. Observed metal release from Virginia Formation samples. All samples had 28-32 measurements of metal release taken throughout the 78-week experimental period.

Reactor	%S	Co (mol/s)			Cu (mol/s)			Ni (mol/s)			Zn (mol/s)		
		Ave	Max	Min	Ave	Max	Min	Ave	Max	Min	Ave	Max	Min
21	2.06	1.79E-12	4.57E-12	8.86E-13	1.77E-12	4.56E-12	2.28E-14	1.65E-11	4.51E-11	7.12E-12	5.81E-13	1.11E-12	1.81E-13
22	2.06	2.02E-12	6.42E-12	9.06E-13	1.98E-12	4.45E-12	4.67E-14	1.87E-11	6.43E-11	8.19E-12	6.10E-13	1.16E-12	2.25E-13
23	3.12	5.48E-13	1.45E-12	9.97E-14	9.36E-14	2.71E-13	2.26E-14	2.60E-12	6.90E-12	2.55E-13	1.44E-12	4.13E-12	1.83E-13
24	3.12	5.86E-13	1.03E-12	2.42E-14	1.03E-13	3.08E-13	2.20E-14	2.87E-12	5.16E-12	2.42E-13	1.64E-12	2.70E-12	8.71E-14
25	3.72	7.16E-13	1.41E-12	4.80E-13	5.09E-14	9.37E-14	2.23E-14	2.90E-12	7.95E-12	1.47E-12	9.84E-13	2.41E-12	6.06E-13
26	3.72	7.40E-13	1.49E-12	3.78E-13	5.26E-14	9.23E-14	2.19E-14	2.99E-12	8.33E-12	1.61E-12	8.69E-13	2.29E-12	4.69E-13
27	5.44	8.84E-13	4.07E-12	3.45E-13	7.71E-14	4.00E-13	2.25E-14	4.59E-12	2.62E-11	1.52E-12	2.03E-12	7.12E-12	1.27E-12
28	5.44	1.09E-12	6.69E-12	4.20E-13	8.94E-14	4.39E-13	2.24E-14	4.81E-12	2.63E-11	1.20E-12	1.92E-12	7.00E-12	1.22E-12
Average		1.05E-12	3.39E-12	4.42E-13	5.28E-13	1.33E-12	2.54E-14	7.00E-12	2.38E-11	2.70E-12	1.26E-12	3.49E-12	5.30E-13

Table 41. Observed drainage pH, sulfate, calcium, and magnesium release from Virginia Formation samples.

Reactor	%S	pH			SO_4 (mol/s)			Ca (mol/s)			Mg (mol/s)		
		Ave	Max	Min	Ave	Max	Min	Ave	Max	Min	Ave	Max	Min
21	2.06	4.85	7.70	4.20	1.13E-10	3.10E-10	1.51E-11	3.18E-11	1.95E-10	1.02E-11	5.19E-11	1.29E-10	1.91E-11
22	2.06	4.85	7.80	4.30	1.06E-10	2.10E-10	4.85E-11	3.35E-11	2.15E-10	1.16E-11	5.37E-11	1.37E-10	2.16E-11
23	3.12	4.51	6.75	3.70	2.94E-10	1.42E-09	4.77E-11	6.87E-11	3.88E-10	2.65E-11	6.87E-11	5.28E-10	1.45E-11
24	3.12	4.38	6.50	3.50	3.86E-10	9.03E-10	7.11E-11	7.23E-11	4.03E-10	3.17E-11	5.63E-11	3.09E-10	1.41E-11
25	3.72	4.30	5.55	3.70	1.92E-10	1.30E-09	6.48E-11	5.21E-11	5.38E-10	1.84E-11	6.51E-11	4.28E-10	1.28E-11
26	3.72	4.31	5.55	3.75	1.53E-10	5.54E-10	7.25E-11	5.02E-11	5.46E-10	1.11E-11	6.55E-11	4.34E-10	1.37E-11
27	5.44	3.88	4.85	3.35	2.96E-10	3.03E-09	5.87E-11	5.12E-11	7.37E-10	8.35E-12	8.55E-11	7.50E-10	2.51E-11
28	5.44	3.84	5.00	2.90	2.53E-10	9.03E-10	8.00E-11	5.30E-11	8.03E-10	1.09E-11	8.64E-11	7.24E-10	2.68E-11
Average		4.37	6.21	3.68	2.24E-10	1.08E-09	5.73E-11	5.16E-11	4.78E-10	1.61E-11	6.66E-11	4.30E-10	1.85E-11

Table 42. Ni, Mg content and Ni-bearing mineral abundance in Duluth Complex solid samples.

%S	Ni ppm	Ni (g/sample)	Ni (mol/sample)	Olivine (mol/sample)	Ni ol (mols Ni/ mol ol)	Ni in ol (mol/sample)	fraction Ni as olivine	fraction Ni as pent/po	Mg (mols/mol ol)	ol Ni/Mg (molar)
Group I										
0.18	525	0.039	6.71E-04	0.043	0.003	1.28E-04	0.191	0.809	0.763	0.0039
0.22	484	0.036	6.18E-04	0.019	0.002	3.82E-05	0.062	0.938	0.857	0.0023
0.72[2]	254	0.019	3.25E-04	0.098	0.003	2.95E-04	0.910	0.090	1.233	0.0024
Group II										
0.40[1]	231	0.017	2.95E-04					1.000		
0.41	511	0.038	6.53E-04	0.056	0.003	1.68E-04	0.258	0.742	0.897	0.0033
0.51	473	0.035	6.04E-04	0.025	0.002	5.00E-05	0.083	0.917	0.812	0.0025
0.54[1]	174	0.013	2.22E-04					1.000		
0.57	398	0.030	5.09E-04	0.092	0.004	3.66E-04	0.721	0.279	1.003	0.0040
0.58	396	0.030	5.06E-04	0.011	0.001	1.07E-05	0.021	0.979	0.851	0.0012
0.67[2]	523	0.039	6.68E-04	0.075	0.003	2.25E-04	0.337	0.663	1.09	0.0028
0.67[3]	261	0.020	3.34E-04	0.019	0.001	1.52E-05	0.046	0.954	0.947	0.0008
0.82[3]	204	0.015	2.61E-04	0.015	0.001	1.49E-05	0.057	0.943	0.8	0.0013
0.92[2]	324	0.024	4.14E-04	0.173	0.003	5.18E-04	1.251	-0.251	1.035	0.0029
1.71[2]	354	0.027	4.52E-04	0.089	0.001	8.89E-05	0.196	0.804	1.139	0.0009
Group III										
0.71	388	0.029	4.96E-04	0.065	0.002	1.29E-04	0.260	0.740	0.958	0.0021
1.12	329	0.025	4.20E-04	0.018	0.002	3.58E-05	0.085	0.915	0.953	0.0021
1.16	422	0.032	5.39E-04	0.073	0.002	1.46E-04	0.271	0.729	0.958	0.0021
1.4	503	0.038	6.43E-04	0.026	0.003	7.67E-05	0.119	0.881	0.954	0.0031
1.44	386	0.029	4.93E-04	0.026	0.003	7.70E-05	0.156	0.844	0.953	0.0031
1.63	492	0.037	6.29E-04	0.023	0.002	4.68E-05	0.075	0.925	0.722	0.0028
1.64	718	0.054	9.17E-04	0.014	0.003	4.23E-05	0.046	0.954	0.828	0.0036
Average	398	0.030	5.08E-04	0.050	0.002	1.30E-04	0.245	0.755	0.934	0.0025

[1] Sample had no olivine detected by point count.

[2] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.

[3] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 43. Average Ni/Mg ratio in Duluth Complex leachate during different sample periods. See appendix for details on period definition. Numbers in parentheses are the number of data points in each period. Blank cells indicate sample did not enter that time period.

%S	Reactor	Ni/Mg olivine	Average Ni/Mg ratio in leachate (molar)			
			Period A	Period B	Period C	Period D
Group I						
0.18	1	0.0038	0.015 (9)	0.011 (37)		
0.22	3	0.002	0.008 (8)	0.006 (37)		
0.72[2]	42	0.0024	0.002 (2)	0.009 (31)		
Group II						
0.40[1]	5	No ol	0.152 (7)	0.122 (21)	0.631 (6)	0.123 (3)
0.41	7	0.0033	0.493 (8)	0.461 (4)	0.166 (4)	0.134 (13)
0.51	9	0.0025	0.244 (8)	0.155 (2)	0.231 (4)	0.061 (17)
0.54[1]	11	No ol	0.042 (29)	0.064 (2)	0.086 (6)	
0.57	13	0.004	0.086 (8)	0.247 (6)	0.093 (1)	0.177 (4)
0.58	15	0.0012	0.124 (8)	0.204 (4)	0.079 (1)	0.073 (18)
0.67[3]	39	0.0028	0.208 (2)	0.483 (4)	0.147 (3)	0.102 (27)
0.67[3]	40	0.0008	0.039 (3)	0.186 (15)	0.086 (4)	0.040 (12)
0.82[2]	43	0.0013	0.124 (3)	0.270 (31)		
0.92[2]	41	0.0029	0.028 (3)	0.357 (30)		
1.71[2]	44	0.0009	0.014 (4)	0.063 (2)		
Group III						
0.71	17	0.0021	0.076 (7)	0.556 (5)	0.084 (6)	0.29 (15)
1.12	35	0.0021	0.989 (10)	0.215 (2)	0.075 (5)	0.030 (1)
1.16	29	0.0021	0.032 (9)	0.311 (3)	0.089 (7)	0.076 (13)
1.4	37	0.0031	0.184 (13)	no Ni:Mg measurements	0.135 (6)	
1.44	33	0.0031	0.214 (9)	0.295 (2)	0.055 (10)	0.041 (10)
1.63	19	0.0028	0.190 (26)	0.164 (2)	0.053 (8)	0.012 (16)
1.64	31	0.0036	0.603 (10)	0.159 (1)	0.075 (4)	0.037 (2)
Total Average		0.0025	0.184	0.217	0.139	0.072

[1] Sample had no olivine detected by point count.
[2] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[3] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 44. Calculated and observed Ni release. Calculated based on Mg in leachate on days Ni was measured, multiplied by Ni:Mg in olivine. This method assumes all Mg comes from ol dissolution therefore it overestimates both Ni release and olivine dissolution. Blank cells indicate sample did not enter that time period.

%S	Calculated Ni release from ol (mol/week)				Observed Ni release rate in leachate (mol/week)				Calculated:Observed Ni from olivine			
	Period A	Period B	Period C	Period D	Period A	Period B	Period C	Period D	Period A	Period B	Period C	Period D
Group I												
0.18	1.96E-08	4.68E-09			5.58E-08	1.14E-08			35.1%	41.0%		
0.22	2.21E-08	4.26E-09			4.65E-08	1.03E-08			47.6%	41.5%		
0.72[2]	6.23E-09	2.29E-09			4.54E-09	6.08E-09			137.1%	37.6%		
Group II												
0.40[1]					2.73E-07	5.86E-08	3.69E-07	4.40E-08				
0.41	1.10E-08	8.59E-09	2.04E-08	2.74E-09	6.99E-07	9.81E-07	9.94E-07	1.07E-07	1.6%	0.9%	2.1%	2.6%
0.51	9.41E-09	3.62E-09	1.23E-08	2.67E-09	5.53E-07	2.28E-07	9.67E-07	6.94E-08	1.7%	1.6%	1.3%	3.9%
0.54[1]					2.25E-07	2.13E-07	7.03E-08					
0.57	3.32E-08	1.99E-08	3.40E-08	9.25E-09	5.00E-07	6.96E-07	7.93E-07	3.11E-07	6.6%	2.9%	4.3%	3.0%
0.58	1.14E-08	6.37E-09	4.18E-09	1.05E-09	8.90E-07	5.17E-07	2.82E-07	4.67E-08	1.3%	1.2%	1.5%	2.2%
0.67[2]	2.09E-08	9.01E-09	1.03E-08	1.45E-09	1.80E-06	9.86E-07	5.53E-07	5.66E-08	1.2%	0.9%	1.9%	2.6%
0.67[3]	5.28E-09	1.48E-09	3.94E-09		2.58E-07	2.79E-07	4.03E-07	6.05E-14	2.0%	0.5%	1.0%	
0.82[3]	5.44E-09	5.05E-10			5.96E-07	1.05E-07			0.9%	0.5%		
0.92[2]	1.20E-08	1.10E-09			1.41E-07	1.33E-07			8.5%	0.8%		
1.71[2]	6.98E-09	9.32E-09			1.50E-07	6.67E-07			4.7%	1.4%		
Group III												
0.71	1.98E-08	1.14E-08	2.56E-08	3.26E-09	7.03E-07	1.41E-06	8.01E-07	2.94E-08	2.8%	0.8%	3.2%	11.1%
1.12	1.26E-08	1.06E-08	2.11E-08	8.88E-09	4.39E-06	8.75E-07	8.28E-07	1.27E-07	0.3%	1.2%	2.5%	7.0%
1.16	1.47E-08	2.58E-08	3.67E-08	2.92E-09	7.83E-08	3.39E-06	1.60E-06	8.59E-08	18.8%	0.8%	2.3%	3.4%
1.4	1.61E-08		9.25E-08		9.26E-07	3.98E-06	3.55E-06		1.7%		2.6%	
1.44	2.40E-08	1.96E-08	6.07E-08	3.88E-09	1.46E-06	1.53E-06	1.13E-06	4.37E-08	1.6%	1.3%	5.4%	8.9%
1.63	5.48E-08	3.27E-08	4.18E-08	7.04E-09	3.07E-06	1.83E-06	8.49E-07	2.26E-08	1.8%	1.8%	4.9%	31.2%
1.64	9.07E-08	3.10E-08	5.59E-08	2.31E-08	1.34E-05	1.36E-06	1.19E-06	2.34E-07	0.7%	2.3%	4.7%	9.9%
Average	2.09E-08	1.12E-08	3.23E-08	6.02E-09	1.44E-06	9.17E-07	9.58E-07	9.06E-08	14.5%	7.7%	2.9%	7.8%

[1] Sample had no olivine by point count.
[2] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[3] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 45. Regression results for calculated fraction of Ni from olivine versus percent sulfur in sample (data in Table 5). Periods A and B did not have a significant regression.

	slope	y-intercept	r^2
Period A			
Period B			
Period C	0.022	0.0074	0.463
Period D	0.120	-0.037	0.441

Table 46. Average sample weekly nickel release per mole olivine by time period. Blank cells indicate sample did not enter that time period.

%S	Olivine (mol/sample)	mol Ni/mol olivine /week in leachate			
		Period A	Period B	Period C	Period D
Group I					
0.18	0.044	1.27E-06	2.59E-07		
0.22	0.02	2.32E-06	5.13E-07		
0.72[1]	0.093	4.88E-08	6.54E-08		
Group II					
0.40	No ol				
0.41	0.056	1.25E-05	1.75E-05	1.77E-05	1.91E-06
0.51	0.026	2.13E-05	8.79E-06	3.72E-05	2.67E-06
0.54	No ol				
0.57	0.09	5.55E-06	7.74E-06	8.81E-06	3.46E-06
0.58	0.011	8.09E-05	4.70E-05	2.56E-05	4.25E-06
0.67[1]	0.073	2.47E-05	1.35E-05	7.58E-06	7.75E-07
0.67[2]	0.02	1.29E-05	1.40E-05	2.01E-05	
0.82[2]	0.015	3.97E-05	6.99E-06		
0.92[1]	0.168	8.37E-07	7.92E-07		
1.71[1]	0.086	1.74E-06	7.75E-06		
Group III					
0.71	0.064	1.10E-05	2.21E-05	1.25E-05	4.60E-07
1.12	0.018	2.44E-04	4.86E-05	4.60E-05	7.06E-06
1.16	0.073	1.07E-06	4.64E-05	2.19E-05	1.18E-06
1.4	0.026	3.56E-05	1.53E-04	1.37E-04	
1.44	0.026	5.62E-05	5.87E-05	4.34E-05	1.68E-06
1.63	0.024	1.28E-04	7.62E-05	3.54E-05	9.40E-07
1.64	0.014	9.54E-04	9.70E-05	8.49E-05	1.67E-05
Total Average		8.59E-05	3.30E-05	3.83E-05	3.73E-06

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 47. Regression results for average sample weekly nickel release per mole olivine (data in Table 7) by time period versus %S. Period D did not have a significant regression.

	slope	y-intercept	r^2
Period A	2.03E-4	-9.68E-5	0.200
Period B	5.39E-5	-1.54E-5	0.408
Period C	5.02E-5	-1.00E-5	0.393
Period D	4.20E-6	-2.50E-7	0.171

Table 48: Values used in the calculation of predicted mineral dissolution rates of mineral grains.

Mineral	Surface Roughness		Density	
	Coefficient	Source	ρ	Source
Labradorite	9.1	Brantley and Mellott, 2000	2.62+(%An * 0.0014)	Klein and Hurlbut, 1985
Andesine	11	Blum, A.E, 1994	2.62+(%An * 0.0014)	Klein and Hurlbut, 1985
Hypersthene	17	Sverdrup, 1990	3.25 + %$FeSiO_3$ * 0.006	Klein and Hurlbut, 1985
Olivine	12	Brantley and Mellott, 2000	3.27 + Fe_2SiO_4 * 0.012	Klein and Hurlbut, 1985
Augite	16	Average[3]	3.2 - 3.4 (3.3)[1]	Klein and Hurlbut, 1985
Biotite	17	Sverdrup, 1990	2.8 - 3.2 (3)[1]	Klein and Hurlbut, 1985
Amphibole	16	Brantley and Mellott, 2000	3.0 - 3.4[2] (3.2)[1]	Klein and Hurlbut, 1985
Orthoclase	5.8	Blum, A.E, 1994	2.57	Klein and Hurlbut, 1985
Prehnite	30	Rose, 1991	2.8 - 2.95 (2.88)[1]	Klein and Hurlbut, 1985

Note: Geometric Mean Grain Diameter (53 - 149um) = 89 microns

[1]Value in parentheses selected for calculation

[2]Range for Hornblende

[3]Average of values reported by Brantley and Mellott, 2000 (SR=14) and Brantley and Chen, 1995 (SR=18)

Table 49. Determination of mineral surface areas used for determining silicate mineral dissolution rates in models. Page 1 of 2.

%S	Reactor	Plagioclase				K-Spar				Hypersthene			
		SR = 9.1		d = 0.0089 cm[1]		SR = 5.8		d = 0.0089 cm[1]		SR = 17.0		d = 0.0089 cm[1]	
		Density	Modal%	Mass	Area*SR (m^2)	Density	Modal%	Mass	Area*SR (m^2)	Density	Modal%	Mass	Area*SR (m^2)
0.18	1	2.69	49.50	37.13	8.467	2.57				3.53	17.40	13.05	4.237
0.22	3	2.68	64.90	48.68	11.142	2.57	3.60	2.70	0.411	3.53	5.40	4.05	1.315
0.40	5	2.70	47.40	35.55	8.078	2.57	3.40	2.55	0.388	3.54	27.60	20.70	6.702
0.41	7	2.70	57.40	43.05	9.782	2.57	0.90	0.68	0.103	3.51	8.70	6.53	2.131
0.51	9	2.69	46.60	34.95	7.971	2.57	5.10	3.83	0.582	3.54	17.80	13.35	4.322
0.54	11	2.67	62.20	46.65	10.719	2.57				3.50	19.80	14.85	4.863
0.57	13	2.69	42.60	31.95	7.287	2.57	4.00	3.00	0.456	3.54	5.00	3.75	1.214
0.58	15	2.69	45.80	34.35	7.834	2.57	5.00	3.75	0.571	3.56	15.00	11.25	3.622
0.67	39	2.69	56.30	42.23	9.630	2.57	1.70	1.28	0.194	3.53	4.20	3.15	1.023
0.67	40	2.69	47.70	35.78	8.159	2.57	0.90	0.68	0.103	3.55	14.40	10.80	3.487
0.71	17	2.69	54.80	41.10	9.373	2.57	0.90	0.68	0.103	3.50	12.20	9.15	2.996
0.72	42	2.70	61.20	45.90	10.429	2.57				3.45	3.30	2.48	0.822
0.82	43	2.69	47.80	35.85	8.176	2.57				3.51	15.70	11.78	3.845
0.92	41	2.70	44.60	33.45	7.600	2.57				3.47	3.30	2.48	0.817
1.12	35	2.69	39.70	29.78	6.791	2.57	11.60	8.70	1.324	3.56	19.80	14.85	4.781
1.16	39	2.70	58.40	43.80	9.952	2.57				3.52	12.40	9.30	3.028
1.40	37	2.69	46.20	34.65	7.902	2.57	2.50	1.88	0.285	3.53	8.40	6.30	2.045
1.44	33	2.69	52.90	39.68	9.048	2.57	4.20	3.15	0.479	3.54	24.40	18.30	5.925
1.63	19	2.68	36.40	27.30	6.249	2.57	5.60	4.20	0.639	3.56	17.80	13.35	4.298
1.64	31	2.69	45.80	34.35	7.834	2.57	4.20	3.15	0.479	3.54	10.80	8.10	2.622
1.71	44	2.70	45.50	34.13	7.754	2.57	0.80	0.60	0.091	3.48	2.50	1.88	0.617
2.06	31	2.67	27.70	20.78	4.773	2.57	12.60	9.45	1.438	3.55	10.90	8.18	2.639
3.12	23	2.70	34.80	26.10	5.930	2.57	8.00	6.00	0.913	3.50	11.60	8.70	2.849
3.72	25	2.68	18.30	13.73	3.142	2.57	14.80	11.10	1.689	3.52	9.60	7.20	2.344
5.44	27	2.66	11.50	8.63	1.989	2.57	20.40	15.30	2.328	3.53			

[1] Geometric mean diameter

Table 49. Determination of mineral surface areas used for determining silicate mineral dissolution rates in models. Page 2 of 2.

%S	Reactor	Augite				Olivine				Biotite			
		SR = 16.0		d = 0.0089 cm[1]		SR = 12.0		d = 0.0089 cm[1]		SR = 17.0		d = 0.0089 cm[1]	
		Density	Modal%	Mass	Area*SR (m^2)	Density	Modal%	Mass	Area*SR (m^2)	Density	Modal%	Mass	Area*SR (m^2)
0.18	1	3.30	10.10	7.58	2.476	4.00	10.10	7.58	1.532	3.00	1.80	1.35	0.516
0.22	3	3.30	8.10	6.08	1.986	3.96	4.50	3.38	0.689	3.00	2.70	2.03	0.774
0.40	5	3.30	4.30	3.23	1.054					3.00	1.70	1.28	0.487
0.41	7	3.30	8.70	6.53	2.133	3.92	13.00	9.75	2.012				
0.51	9	3.30	7.60	5.70	1.863	3.98	5.90	4.43	0.899				
0.54	11	3.30	8.10	6.08	1.986					3.00	2.70	2.03	0.774
0.57	13	3.30	16.80	12.60	4.118	3.86	20.80	15.60	3.269				
0.58	15	3.30	14.20	10.65	3.481	4.00	2.50	1.88	0.379	3.00	0.80	0.60	0.229
0.67	39	3.30	14.30	10.73	3.506	3.89	16.80	12.60	2.620				
0.67	40	3.30	11.70	8.78	2.868	3.74	4.50	3.38	0.730	3.00	6.30	4.73	1.805
0.71	17	3.30	7.80	5.85	1.912	3.72	14.80	11.10	2.414	3.00	1.70	1.28	0.487
0.72	42	3.30	2.50	1.88	0.613	3.98	21.50	16.13	3.278				
0.82	43	3.30	13.00	9.75	3.187	3.83	3.50	2.63	0.554	3.00	3.50	2.63	1.003
0.92	41	3.30	4.10	3.08	1.005	3.89	38.80	29.10	6.052	3.00			
1.12	35	3.30	5.80	4.35	1.422	3.89	4.10	3.08	0.639	3.00	1.70	1.28	0.487
1.16	39	3.30	9.70	7.28	2.378	3.90	16.80	12.60	2.614				
1.40	37	3.30	11.80	8.85	2.893	3.56	5.90	4.43	1.006				
1.44	33	3.30	2.50	1.88	0.613	4.03	5.90	4.43	0.888	3.00	1.70	1.28	0.487
1.63	19	3.30	5.60	4.20	1.373	3.96	5.60	4.20	0.858	3.00	1.90	1.43	0.544
1.64	31	3.30	13.30	9.98	3.260	3.78	3.30	2.48	0.530	3.00	2.50	1.88	0.716
1.71	44	3.30	2.50	1.88	0.613	3.78	19.80	14.85	3.178	3.00	0.80	0.60	0.229
2.06	31	3.30	2.50	1.88	0.613	3.92	0.80	0.60	0.124	3.00	0.80	0.60	0.229
3.12	23	3.30	0.90	0.68	0.221					3.00	2.70	2.03	0.774
3.72	25	3.30	2.60	1.95	0.637					3.00	2.60	1.95	0.745
5.44	27									3.00	0.90	0.68	0.258

[1] Geometric mean diameter

Table 50. Determination of mineral surface areas used for determining sulfide mineral dissolution rates in models. Page 1 of 2.

Reactor	Sulfide Mineral			Pentlandite			Chalcopyrite		
				d = 0.0089 cm^2, SR = 10[1] density = 4.8, ratio = 0.97[2]			d = 0.0089 cm^2, SR = 10[1] density = 4.19, ratio = 1.01[3]		
	%S	%Cu	%Ni	%	Mass (g)	SA (m^2)	%	Mass (g)	SA (m^2)
1	0.18	0.136	0.0525	0.0509	0.0382	0.0054	0.1374	0.1030	0.0166
3	0.22	0.135	0.0484	0.0469	0.0352	0.0050	0.1364	0.1023	0.0165
5	0.4	0.0512	0.0231	0.0224	0.0168	0.0024	0.0517	0.0388	0.0062
7	0.41	0.0788	0.0511	0.0496	0.0372	0.0052	0.0796	0.0597	0.0096
9	0.51	0.143	0.0473	0.0459	0.0344	0.0048	0.1444	0.1083	0.0175
11	0.54	0.0211	0.0174	0.0169	0.0127	0.0018	0.0213	0.0160	0.0026
13	0.57	0.129	0.0398	0.0386	0.0290	0.0041	0.1303	0.0977	0.0157
15	0.58	0.13	0.0396	0.0384	0.0288	0.0041	0.1313	0.0985	0.0159
39	0.67	0.0617	0.0523	0.0507	0.0380	0.0054	0.0623	0.0467	0.0075
40	0.67	0.049	0.0523	0.0507	0.0380	0.0054	0.0495	0.0371	0.0060
17	0.71	0.116	0.0388	0.0376	0.0282	0.0040	0.1172	0.0879	0.0142
42	0.72	0.0517	0.0254	0.0246	0.0185	0.0026	0.0522	0.0392	0.0063
43	0.82	0.0601	0.0204	0.0198	0.0148	0.0021	0.0607	0.0455	0.0073
41	0.92	0.0752	0.0324	0.0314	0.0236	0.0033	0.0760	0.0570	0.0092
35	1.12	0.08	0.0324	0.0314	0.0236	0.0033	0.0808	0.0606	0.0098
29	1.16	0.109	0.0329	0.0319	0.0239	0.0034	0.1101	0.0826	0.0133
37	1.4	0.156	0.0503	0.0488	0.0366	0.0051	0.1576	0.1182	0.0190
33	1.44	0.121	0.0386	0.0374	0.0281	0.0040	0.1222	0.0917	0.0148
19	1.63	0.118	0.0492	0.0477	0.0358	0.0050	0.1192	0.0894	0.0144
20	1.63	0.118	0.0492	0.0477	0.0358	0.0050	0.1192	0.0894	0.0144
31	1.64	0.222	0.0718	0.0696	0.0522	0.0073	0.2242	0.1682	0.0271
44	1.71	0.066	0.0354	0.0343	0.0258	0.0036	0.0667	0.0500	0.0081

[1] Surface roughness factor of 10 in the lower range of those reported in Janzen 2000
[2] Pentlandite $(Fe, Ni)_9S_8$; Ni *4.5/S*8 = 264.11 / 256.56 = 0.97
[3] Chalcopyrite $(CuFe)S_2$; Cu/S*2 = 63.55 / 64.14 = 1.01

Table 50. Determination of mineral surface areas used for determining sulfide mineral dissolution rates in models. Page 2 of 2.

Reactor	Sulfide Mineral			Cubanite			Pyrrhotite			Sulfide SA
				d = 0.0089 cm2, SR = 10[1] density = 4.7, ratio = 1.51[2]			d = 0.0089 cm2, SR = 10[1] density = 4.61, ratio = 2.57[3]			
	%S	%Cu	%Ni	%	Mass (g)	SA (m^2)	%	Mass (g)	SA (m^2)	m^2
1	0.18	0.136	0.0525	0.2054	0.1540	0.0221	-0.0218	-0.0164	0.0000	0.0441
3	0.22	0.135	0.0484	0.2039	0.1529	0.0220	0.0941	0.0705	0.0103	0.0537
5	0.4	0.0512	0.0231	0.0773	0.0580	0.0083	0.8370	0.6278	0.0919	0.1089
7	0.41	0.0788	0.0511	0.1190	0.0892	0.0128	0.7199	0.5399	0.0791	0.1067
9	0.51	0.143	0.0473	0.2159	0.1619	0.0233	0.8216	0.6162	0.0903	0.1358
11	0.54	0.0211	0.0174	0.0319	0.0239	0.0034	1.2889	0.9666	0.1416	0.1494
13	0.57	0.129	0.0398	0.1948	0.1461	0.0210	1.0311	0.7733	0.1133	0.1541
15	0.58	0.13	0.0396	0.1963	0.1472	0.0211	1.0547	0.7910	0.1159	0.1569
39	0.67	0.0617	0.0523	0.0932	0.0699	0.0100	1.4289	1.0717	0.1570	0.1799
40	0.67	0.049	0.0523	0.0740	0.0555	0.0080	1.4616	1.0962	0.1605	0.1798
17	0.71	0.116	0.0388	0.1752	0.1314	0.0189	1.4269	1.0701	0.1567	0.1937
42	0.72	0.0517	0.0254	0.0781	0.0586	0.0084	1.6523	1.2392	0.1815	0.1988
43	0.82	0.0601	0.0204	0.0908	0.0681	0.0098	1.9005	1.4254	0.2088	0.2280
41	0.92	0.0752	0.0324	0.1136	0.0852	0.0122	2.0879	1.5659	0.2293	0.2541
35	1.12	0.08	0.0324	0.1208	0.0906	0.0130	2.5895	1.9421	0.2844	0.3105
29	1.16	0.109	0.0329	0.1646	0.1234	0.0177	2.6165	1.9624	0.2874	0.3218
37	1.4	0.156	0.0503	0.2356	0.1767	0.0254	3.0678	2.3009	0.3370	0.3866
33	1.44	0.121	0.0386	0.1827	0.1370	0.0197	3.2906	2.4680	0.3615	0.3999
19	1.63	0.118	0.0492	0.1782	0.1336	0.0192	3.7594	2.8195	0.4130	0.4516
20	1.63	0.118	0.0492	0.1782	0.1336	0.0192	3.7594	2.8195	0.4130	0.4516
31	1.64	0.222	0.0718	0.3352	0.2514	0.0361	3.4597	2.5948	0.3800	0.4506
44	1.71	0.066	0.0354	0.0997	0.0747	0.0107	4.1341	3.1006	0.4541	0.4765

[1] Surface roughness factor of 10 in the lower range of those reported in Janzen 2000
[2] Cubanite ($CuFe_2S_3$); Cu/S*3 = 63.55 / 96.21 = 1.51
[3] Pyrrhotite (Fe(0.9)S); ((0.9*Fe+S)/S) = (0.9*55.85+32.07)/32.07 = 2.57

Table 51. Maximum amount of plagioclase and olivine dissolution based on cumulative Ca and Mg release (Table 25) and sample stoichiometry (Table 11).

%S	Reactor	Period of Record (weeks)	Minimum pH	Maximum plagioclase dissolution		Maximum olivine dissolution	
				(mmol)	(mol m^{-2} s^{-1})	(mmol)	(mol m^{-2} s^{-1})
Group I, 3 samples							
0.18	**1**	1252	5.67	7.165	1.40E-09	2.649	2.85E-09
0.22	**3**	1252	5.91	8.833	1.31E-09	3.830	9.17E-09
0.72[1]	**42**	809	5.60	2.089	3.31E-10	0.646	3.26E-10
Average			5.73	6.029	1.01E-09	2.375	4.12E-09
Group II, 11 samples							
0.4	**5**	909	3.78	16.266	3.32E-09		
0.41	**7**	724	4.05	5.167	8.72E-10	1.652	1.36E-09
0.51	**9**	724	3.95	7.447	1.54E-09	2.110	3.87E-09
0.54	**11**	441	4.08	8.153	1.26E-09		
0.57	**13**	441	3.86	6.081	1.38E-09	2.593	1.31E-09
0.58	**15**	724	3.98	6.364	1.34E-09	2.873	1.25E-08
0.67[1]	**39**	809	4.22	2.403	4.12E-10	1.029	6.48E-10
0.67[2]	**40**	809	4.04	7.386	1.49E-09	1.934	4.37E-09
0.82[2]	**43**	809	4.49	3.312	6.69E-10	0.691	2.06E-09
0.92[1]	**41**	809	4.89	2.037	4.42E-10	0.439	1.20E-10
1.71[1]	**44**	144	5.06	2.881	6.13E-10	1.275	6.62E-10
Average			4.22	6.136	1.21E-09	1.622	2.99E-09
Group III, Duluth Complex, 7 samples							
0.71	**17**	724	3.53	5.954	1.05E-09	4.237	2.90E-09
1.12	**35**	360	3.23	12.361	3.01E-09	2.676	6.91E-09
1.16	**29**	643	3.24	6.556	1.09E-09	5.014	3.17E-09
1.4	**37**	360	4.03	7.168	1.50E-09	6.132	1.01E-08
1.44	**33**	643	3.04	7.532	1.37E-09	6.293	1.17E-08
1.63	**19**	724	3.32	11.983	3.17E-09	9.617	1.85E-08
1.64	**31**	360	3.38	9.263	1.95E-09	5.668	1.77E-08
Average			3.40	8.688	1.88E-09	5.663	1.01E-08
Virginia Formation, 4 samples							
2.06	21	78	4.20	5.014	1.73E-09	2.706	3.61E-08
3.12	23	78	3.70	6.574	1.83E-09		
3.72	25	78	3.70	7.004	3.68E-09		
5.44	27	78	3.35	9.639	8.00E-09		
Average			3.74	7.058	3.81E-09	2.706	3.61E-08

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 52. Model 1, Periods A-D Silicate Mineral Dissolution Rates. Page 1 of 4.

Model 1, Period A																		
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	8	4.0E-12	1.4E-12	6.6E-12	2.5E-12	1.9E-12	7.4E-13	2.7E-12	2.7E-13	7.0E-12	2.7E-12	1.0E-11	9.9E-13	6.0E-12	2.3E-12	8.6E-12	8.5E-13
3	0.22	7	3.7E-12	1.7E-12	6.2E-12	1.7E-12	1.2E-11	1.2E-11	4.0E-11	2.6E-12	2.6E-11	2.8E-11	9.2E-11	5.9E-12	6.1E-12	6.5E-12	2.1E-11	1.4E-12
5	0.40	8	8.4E-12	2.6E-12	1.4E-11	6.2E-12	7.6E-13	5.1E-13	1.5E-12	1.8E-13					3.6E-12	2.4E-12	7.1E-12	8.4E-13
7	0.41	10	3.3E-12	2.0E-12	6.4E-12	7.0E-13	2.4E-12	1.2E-12	4.1E-12	5.2E-13	3.0E-12	1.5E-12	5.3E-12	6.6E-13	2.3E-12	1.2E-12	4.0E-12	5.0E-13
9	0.51	11	4.7E-12	4.1E-12	1.4E-11	5.4E-13	1.4E-12	9.6E-13	2.9E-12	2.8E-13	8.4E-12	5.6E-12	1.7E-11	1.6E-12	2.7E-12	1.8E-12	5.4E-12	5.1E-13
11	0.54	32	5.0E-12	2.5E-12	1.5E-11	1.7E-12	2.8E-12	2.6E-12	1.3E-11	2.2E-13					6.6E-12	6.0E-12	3.1E-11	5.0E-13
13	0.57	7	7.2E-12	4.5E-12	1.6E-11	2.4E-12	1.0E-11	3.4E-12	1.6E-11	5.2E-12	4.2E-12	1.4E-12	6.4E-12	2.1E-12				
15	0.58	7	5.3E-12	3.7E-12	1.3E-11	1.3E-12	4.7E-12	2.8E-12	9.1E-12	1.1E-12	5.0E-11	2.9E-11	9.5E-11	1.1E-11	3.2E-11	1.9E-11	6.2E-11	7.4E-12
39[1]	0.67	13	2.6E-12	1.8E-12	6.1E-12	5.8E-13	9.0E-12	2.8E-12	1.3E-11	3.8E-12	4.0E-12	1.2E-12	5.9E-12	1.7E-12				
40[2]	0.67	18	4.2E-12	1.4E-12	7.7E-12	8.7E-13	3.1E-12	1.1E-12	5.1E-12	1.1E-12	1.8E-11	6.3E-12	2.9E-11	6.3E-12	1.9E-12	6.9E-13	3.2E-12	6.8E-13
17	0.71	6	3.9E-12	1.6E-12	5.6E-12	1.8E-12	4.7E-12	2.0E-12	8.4E-12	2.9E-12	6.8E-12	2.9E-12	1.2E-11	4.1E-12	9.4E-12	4.0E-12	1.7E-11	5.8E-12
42[1]	0.72	12	1.8E-12	1.6E-12	6.7E-12	1.2E-12	3.3E-12	2.0E-12	7.2E-12	5.6E-13	8.6E-13	5.3E-13	1.9E-12	1.5E-13				
43[2]	0.82	17	3.8E-12	1.6E-12	7.5E-12	1.2E-12	1.4E-12	5.4E-13	2.1E-12	5.3E-13	1.4E-11	5.4E-12	2.2E-11	5.3E-12	2.3E-12	8.7E-13	3.5E-12	8.6E-13
41[1]	0.92	17	2.8E-12	1.0E-12	5.1E-12	1.0E-12	4.4E-12	2.3E-12	8.7E-12	6.3E-13	6.8E-13	3.5E-13	1.4E-12	9.7E-14				
35	1.12	27	1.3E-11	6.4E-12	2.5E-11	1.1E-12	1.7E-12	9.4E-13	4.4E-12	2.7E-13	1.2E-11	6.7E-12	3.2E-11	1.9E-12	4.7E-12	2.6E-12	1.2E-11	7.4E-13
29	1.16	10	4.2E-12	1.3E-12	6.5E-12	2.3E-12	3.7E-12	2.0E-12	7.1E-12	3.9E-13	4.7E-12	2.6E-12	9.1E-12	5.0E-13				
37	1.40	18	7.1E-12	2.9E-12	1.4E-11	2.9E-12	4.8E-12	2.7E-12	1.0E-11	5.8E-13	1.0E-11	5.9E-12	2.3E-11	1.3E-12				
33	1.44	16	5.4E-12	4.7E-12	1.4E-11	1.2E-12	2.1E-12	6.3E-13	3.3E-12	1.2E-12	1.6E-11	4.7E-12	2.4E-11	8.6E-12	7.3E-12	2.2E-12	1.1E-11	4.0E-12
19	1.63	22	8.7E-12	5.2E-12	2.0E-11	3.5E-12	8.1E-12	4.4E-12	2.0E-11	3.5E-12	5.3E-11	2.9E-11	1.3E-10	2.3E-11	1.9E-11	1.0E-11	4.7E-11	8.3E-12
31	1.64	12	1.5E-11	1.1E-11	4.1E-11	3.1E-12	1.5E-11	8.2E-12	3.4E-11	2.8E-12	8.8E-11	4.8E-11	2.0E-10	1.6E-11	1.6E-11	9.0E-12	3.8E-11	3.1E-12
44[1]	1.71	23	6.3E-12	2.0E-12	1.2E-11	2.9E-12	2.0E-11	9.0E-12	3.8E-11	3.3E-12	3.9E-12	1.8E-12	7.6E-12	6.6E-13	1.9E-11	8.9E-12	3.8E-11	3.3E-12
Summary			5.7E-12	3.1E-12	4.1E-11	5.4E-13	5.5E-12	3.0E-12	4.0E-11	1.8E-13	1.7E-11	9.6E-12	2.0E-10	9.7E-14	9.3E-12	5.2E-12	6.2E-11	5.0E-13

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 52. Model 1, Periods A-D Silicate Mineral Dissolution Rates. Page 2 of 4.

Model 1, Period B																		
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	189	7.7E-13	3.8E-13	3.0E-12	8.8E-14	5.3E-13	4.8E-13	2.1E-12	1.3E-13	2.0E-12	1.8E-12	7.6E-12	4.7E-13	1.7E-12	1.5E-12	6.5E-12	4.0E-13
3	0.22	191	7.0E-13	3.1E-13	2.1E-12	7.3E-14	2.6E-12	1.8E-12	1.1E-11	4.1E-13	6.0E-12	4.2E-12	2.4E-11	9.5E-13	1.4E-12	9.8E-13	5.7E-12	2.2E-13
5	0.40	133	2.7E-12	1.0E-12	6.4E-12	1.3E-12	1.9E-13	2.1E-13	1.5E-12	8.6E-14					9.1E-13	9.9E-13	6.9E-12	4.1E-13
7	0.41	52	6.1E-13	3.8E-13	2.4E-12	1.5E-13	1.3E-12	8.1E-13	4.3E-12	2.6E-13	1.6E-12	1.0E-12	5.4E-12	3.3E-13	1.2E-12	7.8E-13	4.1E-12	2.5E-13
9	0.51	33	7.5E-13	2.2E-13	1.3E-12	3.5E-13	8.2E-13	4.4E-13	2.4E-12	2.8E-13	4.7E-12	2.6E-12	1.4E-11	1.6E-12	1.5E-12	8.1E-13	4.4E-12	5.2E-13
11	0.54	37	1.9E-12	5.7E-13	3.2E-12	3.2E-13	8.1E-13	4.3E-13	2.0E-12	2.1E-13					1.9E-12	1.0E-12	4.7E-12	5.0E-13
13	0.57	45	1.5E-12	7.2E-13	3.5E-12	2.1E-13	5.8E-12	2.0E-12	1.4E-11	8.6E-13	2.4E-12	8.3E-13	5.7E-12	3.5E-13				
15	0.58	24	1.4E-12	4.8E-13	2.6E-12	1.9E-13	2.5E-12	8.0E-13	4.2E-12	3.5E-13	2.6E-11	8.4E-12	4.4E-11	3.7E-12	1.7E-11	5.5E-12	2.9E-11	2.4E-12
39[1]	0.67	16	5.4E-13	1.3E-13	7.3E-13	2.9E-13	3.9E-12	1.7E-12	7.6E-12	9.6E-13	1.7E-12	7.4E-13	3.4E-12	4.3E-13				
40[2]	0.67	56	2.4E-12	3.6E-13	3.4E-12	1.7E-12	9.1E-13	3.6E-13	1.7E-12	1.8E-13	5.2E-12	2.1E-12	9.7E-12	1.1E-12	5.7E-13	2.2E-13	1.0E-12	1.1E-13
17	0.71	39	6.4E-13	4.2E-13	2.5E-12	1.5E-13	2.1E-12	1.0E-12	6.6E-12	3.6E-13	3.0E-12	1.5E-12	9.5E-12	5.2E-13	4.1E-12	2.1E-12	1.3E-11	7.2E-13
42[1]	0.72	115	3.8E-13	2.1E-13	1.4E-12	5.6E-14	1.5E-12	1.0E-12	5.6E-12	5.4E-13	3.9E-13	2.7E-13	1.5E-12	1.4E-13				
43[2]	0.82	114	7.2E-13	5.4E-13	2.4E-12	8.0E-14	1.6E-13	8.1E-14	5.6E-13	3.3E-14	1.6E-12	8.2E-13	5.6E-12	3.3E-13	2.6E-13	1.3E-13	9.0E-13	5.4E-14
41[1]	0.92	114	4.3E-13	2.9E-13	1.4E-12	2.1E-14	6.7E-13	2.4E-13	2.5E-12	1.5E-13	1.0E-13	3.7E-14	3.9E-13	2.4E-14				
35	1.12	23	3.8E-12	7.9E-13	5.3E-12	2.4E-12	2.2E-12	5.8E-13	3.3E-12	1.1E-12	1.6E-11	4.1E-12	2.4E-11	7.7E-12	6.0E-12	1.6E-12	9.0E-12	2.9E-12
29	1.16	37	1.1E-12	7.7E-13	4.1E-12	4.3E-13	3.1E-12	1.2E-12	7.8E-12	1.9E-12	4.0E-12	1.6E-12	9.9E-12	2.4E-12				
37	1.40	35	2.1E-12	7.9E-13	4.8E-12	1.1E-12	7.1E-12	1.4E-12	9.2E-12	2.9E-12	1.5E-11	3.0E-12	2.0E-11	6.3E-12				
33	1.44	34	1.2E-12	3.9E-13	2.7E-12	5.1E-13	1.8E-12	4.7E-13	2.7E-12	9.4E-13	1.3E-11	3.5E-12	2.0E-11	7.0E-12	6.3E-12	1.7E-12	9.5E-12	3.3E-12
19	1.63	8	3.1E-12	2.6E-12	9.3E-12	1.5E-12	4.6E-12	5.8E-13	5.4E-12	3.6E-12	3.0E-11	3.8E-12	3.5E-11	2.3E-11	1.1E-11	1.4E-12	1.3E-11	8.4E-12
31	1.64	40	2.6E-12	6.5E-13	4.3E-12	1.7E-12	5.9E-12	1.5E-12	9.3E-12	2.8E-12	3.4E-11	8.5E-12	5.4E-11	1.6E-11	6.4E-12	1.6E-12	1.0E-11	3.1E-12
44[1]	1.71	17	1.4E-12	3.7E-13	2.4E-12	8.4E-13	2.4E-11	3.5E-12	2.9E-11	1.6E-11	4.8E-12	7.1E-13	5.8E-12	3.3E-12	2.4E-11	3.5E-12	2.9E-11	1.6E-11
Summary			1.5E-12	5.9E-13	9.3E-12	2.1E-14	3.4E-12	9.8E-13	2.9E-11	3.3E-14	9.1E-12	2.6E-12	5.4E-11	2.4E-14	5.6E-12	1.6E-12	2.9E-11	5.4E-14

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 52. Model 1, Periods A-D Silicate Mineral Dissolution Rates. Page 3 of 4.

Model 1, Period C																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)				Orphan Ca (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	21	1.5E-12	2.4E-12	7.0E-12	0.0E+00	8.1E-14	6.3E-14	2.1E-13	0.0E+00					3.9E-13	3.0E-13	1.0E-12	0.0E+00	7.2E-13	1.8E-12	6.3E-12	0.0E+00
7	0.41	22	1.4E-12	7.6E-13	3.1E-12	2.3E-13	2.2E-12	1.8E-12	6.8E-12	0.0E+00	2.8E-12	2.2E-12	8.6E-12	0.0E+00	2.1E-12	1.7E-12	6.5E-12	0.0E+00	3.3E-13	8.4E-13	3.4E-12	0.0E+00
9	0.51	29	3.1E-12	1.4E-12	7.4E-12	4.6E-13	7.1E-13	5.8E-13	2.0E-12	0.0E+00	4.1E-12	3.4E-12	1.2E-11	0.0E+00	1.3E-12	1.1E-12	3.6E-12	0.0E+00	1.1E-12	2.2E-12	8.1E-12	0.0E+00
11	0.54	40	1.0E-13	3.0E-13	1.5E-12	0.0E+00	2.6E-13	2.6E-13	1.1E-12	0.0E+00					6.1E-13	6.1E-13	2.6E-12	0.0E+00	8.2E-14	2.8E-13	1.3E-12	0.0E+00
13	0.57	14	1.0E-12	6.9E-13	2.6E-12	1.6E-13	8.3E-12	4.8E-12	2.1E-11	3.1E-12	3.4E-12	2.0E-12	8.5E-12	1.3E-12					0.0E+00	0.0E+00	0.0E+00	0.0E+00
15	0.58	30	6.1E-13	7.6E-13	2.9E-12	0.0E+00	2.1E-12	9.2E-13	3.9E-12	6.5E-13	2.2E-11	9.7E-12	4.1E-11	6.8E-12	1.4E-11	6.3E-12	2.7E-11	4.4E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00
39[1]	0.67	21	2.8E-13	1.6E-13	5.7E-13	5.1E-14	4.4E-12	1.1E-12	6.1E-12	2.0E-12	1.9E-12	4.9E-13	2.7E-12	8.8E-13					0.0E+00	0.0E+00	0.0E+00	0.0E+00
40[2]	0.67	10	6.6E-14	2.0E-13	6.4E-13	0.0E+00	2.1E-12	6.5E-13	2.8E-12	8.5E-13	1.2E-11	3.7E-12	1.6E-11	4.9E-12	1.3E-12	4.1E-13	1.7E-12	5.3E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00
17	0.71	38	3.3E-12	3.1E-12	1.6E-11	0.0E+00	4.8E-12	4.1E-12	1.6E-11	1.4E-13	7.0E-12	6.0E-12	2.3E-11	2.0E-13	9.7E-12	8.3E-12	3.3E-11	2.8E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00
35	1.12	33	6.4E-12	6.1E-12	1.7E-11	0.0E+00	2.0E-12	7.6E-13	3.8E-12	7.8E-13	1.4E-11	5.4E-12	2.7E-11	5.5E-12	5.5E-12	2.1E-12	1.0E-11	2.1E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00
29	1.16	48	3.9E-12	3.1E-12	1.1E-11	0.0E+00	6.2E-12	3.1E-12	1.3E-11	0.0E+00	8.0E-12	4.0E-12	1.7E-11	0.0E+00					1.5E-14	1.0E-13	7.0E-13	0.0E+00
37	1.40	38	3.5E-12	2.6E-12	8.0E-12	0.0E+00	1.7E-11	7.5E-12	3.1E-11	3.4E-12	3.8E-11	1.6E-11	6.8E-11	7.5E-12					0.0E+00	0.0E+00	0.0E+00	0.0E+00
33	1.44	57	1.8E-11	1.6E-11	4.8E-11	0.0E+00	3.2E-12	1.8E-12	7.1E-12	5.5E-13	2.4E-11	1.3E-11	5.3E-11	4.1E-12	1.1E-11	6.2E-12	2.5E-11	1.9E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00
19	1.63	64	4.7E-12	4.4E-12	1.7E-11	0.0E+00	4.9E-12	1.5E-12	8.2E-12	3.1E-13	3.2E-11	1.0E-11	5.3E-11	2.0E-12	1.1E-11	3.6E-12	1.9E-11	7.3E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00
31	1.64	28	2.2E-12	1.5E-12	5.3E-12	2.6E-13	7.4E-12	1.9E-12	1.1E-11	4.0E-12	4.3E-11	1.1E-11	6.7E-11	2.4E-11	8.2E-12	2.1E-12	1.3E-11	4.4E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00
Summary			3.3E-12	2.9E-12	4.8E-11	0.0E+00	4.4E-12	2.1E-12	3.1E-11	0.0E+00	1.6E-11	6.7E-12	6.8E-11	0.0E+00	6.0E-12	3.0E-12	3.3E-11	0.0E+00	1.5E-13	3.5E-13	8.1E-12	0.0E+00

Note: For this period, model assumptions dictate plagioclase dissolution rates to be the same as Period B averages for each sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 52. Model 1, Periods A-D Silicate Mineral Dissolution Rates. Page 4 of 4.

Model 1, Period D																		
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	9	1.1E-12	2.6E-13	1.6E-12	6.4E-13	8.9E-14	9.3E-16	9.1E-14	8.8E-14					4.2E-13	4.4E-15	4.3E-13	4.2E-13
7	0.41	54	6.7E-13	2.1E-13	1.4E-12	3.6E-13	5.6E-13	5.6E-13	2.6E-12	2.4E-13	7.1E-13	7.1E-13	3.3E-12	3.1E-13	5.4E-13	5.4E-13	2.5E-12	2.3E-13
9	0.51	66	1.2E-12	5.6E-13	3.9E-12	5.6E-13	3.5E-13	3.9E-13	1.7E-12	1.3E-13	2.0E-12	2.2E-12	9.7E-12	7.7E-13	6.4E-13	7.1E-13	3.1E-12	2.4E-13
13	0.57	21	1.9E-12	4.1E-13	2.5E-12	1.2E-12	3.4E-12	1.7E-12	7.0E-12	4.4E-13	1.4E-12	7.0E-13	2.9E-12	1.8E-13				
15	0.58	80	1.2E-12	4.3E-13	2.8E-12	6.5E-13	3.9E-13	3.8E-13	1.7E-12	1.6E-13	4.1E-12	4.0E-12	1.8E-11	1.7E-12	2.7E-12	2.6E-12	1.2E-11	1.1E-12
39[1]	0.67	80	3.9E-13	2.2E-13	8.5E-13	6.7E-14	7.4E-13	6.1E-13	2.9E-12	1.2E-13	3.3E-13	2.7E-13	1.3E-12	5.2E-14				
40[2]	0.67	44	8.4E-13	4.6E-13	1.9E-12	1.9E-13	2.8E-13	2.2E-13	1.2E-12	1.2E-13	1.6E-12	1.3E-12	6.8E-12	7.1E-13	1.8E-13	1.4E-13	7.4E-13	7.7E-14
17	0.71	58	9.1E-13	2.6E-13	1.6E-12	4.5E-13	6.7E-13	6.4E-13	2.8E-12	1.8E-13	9.7E-13	9.2E-13	4.1E-12	2.5E-13	1.3E-12	1.3E-12	5.7E-12	3.5E-13
35	1.12	3	2.6E-12	2.6E-13	2.8E-12	2.3E-12	1.9E-12	2.9E-13	2.2E-12	1.6E-12	1.4E-11	2.1E-12	1.6E-11	1.1E-11	5.2E-12	8.0E-13	6.0E-12	4.4E-12
29	1.16	47	8.8E-13	2.2E-13	1.5E-12	5.4E-13	7.5E-13	7.6E-13	3.4E-12	1.8E-13	9.5E-13	9.7E-13	4.3E-12	2.4E-13				
33	1.44	32	8.7E-13	1.5E-13	1.5E-12	5.8E-13	5.1E-13	2.3E-13	1.1E-12	9.2E-14	3.8E-12	1.7E-12	8.2E-12	6.9E-13	1.8E-12	7.9E-13	3.9E-12	3.2E-13
19	1.63	62	2.0E-12	8.7E-13	4.6E-12	9.9E-13	1.1E-12	8.6E-13	3.5E-12	1.3E-13	7.0E-12	5.6E-12	2.3E-11	8.5E-13	2.5E-12	2.0E-12	8.3E-12	3.1E-13
31	1.64	5	2.9E-12	4.2E-13	3.6E-12	2.6E-12	4.1E-12	8.6E-13	5.5E-12	3.2E-12	2.4E-11	5.0E-12	3.2E-11	1.8E-11	4.5E-12	9.5E-13	6.0E-12	3.5E-12
Summary			1.3E-12	3.6E-13	4.6E-12	6.7E-14	1.1E-12	5.8E-13	7.0E-12	8.8E-14	5.0E-12	2.1E-12	3.2E-11	5.2E-14	2.0E-12	9.9E-13	1.2E-11	7.7E-14

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 53. Model 2, Periods A-D Silicate Mineral Dissolution Rates. Page 1 of 4.

Model 2, Period A																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	8	4.4E-12	1.8E-12	6.5E-12	6.4E-13	1.9E-12	1.2E-12	3.4E-12	0.0E+00	3.5E-14	9.8E-14	2.8E-13	0.0E+00	1.3E-13	3.6E-13	1.0E-12	0.0E+00	1.1E-13	3.1E-13	8.8E-13	0.0E+00
3	0.22	7	7.7E-12	4.9E-12	1.6E-11	2.5E-12	1.2E-12	1.2E-12	3.0E-12	0.0E+00	3.6E-12	9.0E-12	2.4E-11	0.0E+00	8.2E-12	2.1E-11	5.5E-11	0.0E+00	1.9E-12	4.8E-12	1.3E-11	0.0E+00
5	0.40	8	1.9E-11	1.3E-11	4.2E-11	3.0E-12	5.5E-12	4.1E-12	1.3E-11	2.1E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	10	4.7E-12	2.4E-12	8.5E-12	7.8E-13	2.6E-12	1.9E-12	4.9E-12	0.0E+00	1.8E-14	4.8E-14	1.3E-13	0.0E+00	2.3E-14	6.1E-14	1.6E-13	0.0E+00	1.8E-14	4.6E-14	1.2E-13	0.0E+00
9	0.51	11	5.1E-12	5.0E-12	1.8E-11	8.9E-13	3.4E-12	3.7E-12	1.1E-11	1.2E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
11	0.54	32	8.9E-12	5.6E-12	2.1E-11	7.5E-13	3.1E-12	2.1E-12	9.8E-12	0.0E+00	4.5E-13	1.7E-12	8.4E-12	0.0E+00					1.1E-12	3.9E-12	2.0E-11	0.0E+00
13	0.57	7	4.2E-12	1.7E-12	7.3E-12	2.4E-12	3.2E-12	3.3E-12	9.4E-12	0.0E+00	6.1E-13	1.1E-12	2.9E-12	0.0E+00	2.5E-13	4.6E-13	1.2E-12	0.0E+00				
15	0.58	7	5.6E-12	3.7E-12	1.3E-11	1.5E-12	1.4E-12	1.5E-12	4.2E-12	0.0E+00	2.5E-13	6.7E-13	1.8E-12	0.0E+00	2.7E-12	7.1E-12	1.9E-11	0.0E+00	1.7E-12	4.6E-12	1.2E-11	0.0E+00
39[1]	0.67	13	3.6E-12	1.8E-12	6.9E-12	1.1E-12	8.5E-13	1.1E-12	3.0E-12	0.0E+00	1.8E-12	2.6E-12	7.7E-12	0.0E+00	7.9E-13	1.1E-12	3.4E-12	0.0E+00				
40[2]	0.67	18	5.2E-12	2.0E-12	8.8E-12	1.5E-12	1.3E-12	8.0E-13	2.7E-12	0.0E+00	3.4E-14	1.0E-13	4.1E-13	0.0E+00	1.9E-13	6.0E-13	2.3E-12	0.0E+00	2.1E-14	6.5E-14	2.5E-13	0.0E+00
17	0.71	6	8.6E-12	2.2E-12	1.1E-11	5.7E-12	1.1E-12	1.2E-12	3.1E-12	0.0E+00	2.2E-13	4.2E-13	1.0E-12	0.0E+00	3.3E-13	6.1E-13	1.5E-12	0.0E+00	4.5E-13	8.4E-13	2.1E-12	0.0E+00
42[1]	0.72	12	6.8E-12	4.2E-12	1.5E-11	1.2E-12	1.3E-12	1.5E-12	6.0E-12	5.0E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
43[2]	0.82	17	2.7E-12	1.1E-12	4.2E-12	1.0E-12	2.1E-12	1.4E-12	5.3E-12	0.0E+00	3.4E-14	1.4E-13	5.8E-13	0.0E+00	3.4E-13	1.4E-12	5.9E-12	0.0E+00	5.6E-14	2.3E-13	9.4E-13	0.0E+00
41[1]	0.92	17	5.2E-12	2.6E-12	1.1E-11	7.9E-13	1.8E-12	1.1E-12	3.8E-12	0.0E+00	2.6E-13	1.1E-12	4.5E-12	0.0E+00	4.1E-14	1.7E-13	7.0E-13	0.0E+00				
35	1.12	27	1.0E-11	8.6E-12	3.9E-11	1.4E-12	1.0E-11	5.8E-12	2.3E-11	0.0E+00	1.4E-13	5.1E-13	2.0E-12	0.0E+00	1.0E-12	3.6E-12	1.5E-11	0.0E+00	3.9E-13	1.4E-12	5.6E-12	0.0E+00
29	1.16	10	6.7E-12	3.4E-12	1.3E-11	7.0E-13	2.0E-12	1.4E-12	4.4E-12	3.7E-14	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
37	1.40	18	5.1E-12	2.8E-12	1.1E-11	5.8E-13	4.7E-12	3.5E-12	1.2E-11	0.0E+00	2.3E-13	6.2E-13	1.9E-12	0.0E+00	5.0E-13	1.4E-12	4.2E-12	0.0E+00				
33	1.44	16	2.2E-11	9.7E-12	4.3E-11	1.1E-11	4.1E-12	4.1E-12	1.2E-11	0.0E+00	3.6E-13	6.4E-13	1.5E-12	0.0E+00	2.6E-12	4.8E-12	1.1E-11	0.0E+00	1.2E-12	2.2E-12	5.2E-12	0.0E+00
19	1.63	22	1.8E-11	9.9E-12	4.1E-11	7.7E-12	3.9E-13	9.9E-13	4.2E-12	0.0E+00	3.6E-12	4.6E-12	1.8E-11	0.0E+00	2.4E-11	3.0E-11	1.2E-10	0.0E+00	8.5E-12	1.1E-11	4.2E-11	0.0E+00
31	1.64	12	1.6E-11	1.0E-11	4.0E-11	3.2E-12	4.2E-12	4.8E-12	1.3E-11	0.0E+00	1.2E-12	2.1E-12	6.0E-12	0.0E+00	6.9E-12	1.2E-11	3.5E-11	0.0E+00	1.3E-12	2.3E-12	6.6E-12	0.0E+00
44[1]	1.71	23	2.9E-11	1.3E-11	5.0E-11	5.3E-12	3.1E-12	2.1E-12	6.1E-12	0.0E+00	1.2E-12	3.0E-12	1.0E-11	0.0E+00	2.5E-13	6.1E-13	2.0E-12	0.0E+00	1.2E-12	3.0E-12	9.9E-12	0.0E+00
Summary			9.5E-12	5.2E-12	5.0E-11	5.8E-13	2.8E-12	2.3E-12	2.3E-11	0.0E+00	6.7E-13	1.4E-12	2.4E-11	0.0E+00	2.5E-12	4.5E-12	1.2E-10	0.0E+00	1.2E-12	2.3E-12	4.2E-11	0.0E+00

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 53. Model 2, Periods A-D Silicate Mineral Dissolution Rates. Page 2 of 4.

Model 2, Period B																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	189	1.0E-12	7.6E-13	3.9E-12	1.8E-13	2.7E-13	3.2E-13	2.7E-12	0.0E+00	1.0E-13	2.6E-13	1.3E-12	0.0E+00	3.9E-13	9.7E-13	4.9E-12	0.0E+00	3.3E-13	8.3E-13	4.2E-12	0.0E+00
3	0.22	191	1.8E-12	9.1E-13	6.1E-12	2.2E-13	1.2E-13	2.4E-13	1.9E-12	0.0E+00	7.2E-13	1.2E-12	8.3E-12	0.0E+00	1.7E-12	2.8E-12	1.9E-11	0.0E+00	3.9E-13	6.5E-13	4.4E-12	0.0E+00
5	0.40	133	3.4E-12	3.3E-12	1.5E-11	1.5E-12	2.0E-12	1.1E-12	6.1E-12	0.0E+00	1.3E-14	8.2E-14	8.6E-13	0.0E+00					6.0E-14	3.9E-13	4.1E-12	0.0E+00
7	0.41	52	1.4E-12	8.5E-13	5.3E-12	3.9E-13	1.7E-13	3.4E-13	1.9E-12	0.0E+00	3.6E-13	5.7E-13	2.3E-12	0.0E+00	4.6E-13	7.2E-13	2.9E-12	0.0E+00	3.5E-13	5.4E-13	2.2E-12	0.0E+00
9	0.51	33	1.8E-12	5.4E-13	2.8E-12	9.1E-13	9.2E-14	1.5E-13	5.9E-13	0.0E+00	2.7E-13	4.8E-13	2.1E-12	0.0E+00	1.5E-12	2.8E-12	1.2E-11	0.0E+00	4.9E-13	8.8E-13	3.8E-12	0.0E+00
11	0.54	37	2.9E-12	1.7E-12	8.4E-12	7.5E-13	1.2E-12	5.9E-13	2.6E-12	0.0E+00	1.2E-14	5.3E-14	2.9E-13	0.0E+00					2.8E-14	1.2E-13	6.7E-13	0.0E+00
13	0.57	45	1.6E-12	7.1E-13	3.7E-12	2.2E-13	3.4E-14	1.2E-13	6.3E-13	0.0E+00	2.4E-12	1.7E-12	8.3E-12	0.0E+00	9.8E-13	7.0E-13	3.4E-12	0.0E+00				
15	0.58	24	2.0E-12	7.8E-13	3.9E-12	2.8E-13	4.0E-14	2.0E-13	9.7E-13	0.0E+00	1.1E-12	5.4E-13	2.0E-12	0.0E+00	1.1E-11	5.6E-12	2.1E-11	0.0E+00	7.5E-12	3.7E-12	1.4E-11	0.0E+00
39[1]	0.67	16	1.0E-12	2.9E-13	1.4E-12	5.0E-13	2.0E-14	8.0E-14	3.2E-13	0.0E+00	2.0E-12	1.8E-12	6.6E-12	0.0E+00	8.7E-13	7.7E-13	2.9E-12	0.0E+00				
40[2]	0.67	56	1.6E-12	6.2E-13	2.9E-12	3.2E-13	1.5E-12	3.3E-13	2.2E-12	1.0E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
17	0.71	39	2.0E-12	1.2E-12	7.8E-12	4.8E-13	1.7E-14	1.0E-13	6.4E-13	0.0E+00	1.2E-12	8.1E-13	3.4E-12	0.0E+00	1.7E-12	1.2E-12	5.0E-12	0.0E+00	2.4E-12	1.6E-12	6.9E-12	0.0E+00
42[1]	0.72	115	2.9E-12	1.8E-12	9.3E-12	7.6E-13	1.7E-13	1.6E-13	8.8E-13	0.0E+00	6.6E-14	3.1E-13	1.9E-12	0.0E+00	1.8E-14	8.1E-14	5.0E-13	0.0E+00				
43[2]	0.82	114	3.0E-13	1.7E-13	1.1E-12	6.5E-14	5.4E-13	4.9E-13	2.0E-12	0.0E+00	6.2E-15	1.8E-14	6.1E-14	0.0E+00	6.3E-14	1.8E-13	6.2E-13	0.0E+00	1.0E-14	3.0E-14	9.9E-14	0.0E+00
41[1]	0.92	114	8.1E-13	4.7E-13	3.5E-12	1.1E-13	2.8E-13	2.5E-13	1.2E-12	0.0E+00	6.2E-14	1.1E-13	2.7E-13	0.0E+00	9.7E-15	1.7E-14	4.2E-14	0.0E+00				
35	1.12	23	9.4E-12	2.0E-12	1.3E-11	5.4E-12	4.9E-13	6.5E-13	1.9E-12	0.0E+00	2.8E-13	3.5E-13	1.3E-12	0.0E+00	2.0E-12	2.5E-12	9.2E-12	0.0E+00	7.7E-13	9.6E-13	3.5E-12	0.0E+00
29	1.16	37	2.9E-12	2.1E-12	1.1E-11	1.2E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	1.5E-12	6.5E-13	3.2E-12	2.2E-14	1.9E-12	8.3E-13	4.0E-12	2.8E-14				
37	1.40	35	3.4E-12	1.1E-12	5.5E-12	1.9E-12	9.6E-14	5.5E-13	3.2E-12	0.0E+00	3.6E-12	1.3E-12	6.8E-12	0.0E+00	8.0E-12	2.9E-12	1.5E-11	0.0E+00				
33	1.44	34	1.0E-11	2.6E-12	1.8E-11	4.7E-12	4.2E-14	2.2E-13	1.3E-12	0.0E+00	1.1E-12	4.9E-13	2.1E-12	0.0E+00	7.8E-12	3.7E-12	1.6E-11	0.0E+00	3.7E-12	1.7E-12	7.4E-12	0.0E+00
19	1.63	8	6.7E-12	5.8E-12	2.0E-11	3.2E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	3.1E-12	1.3E-12	4.4E-12	3.2E-13	2.0E-11	8.7E-12	2.9E-11	2.1E-12	7.4E-12	3.2E-12	1.0E-11	7.6E-13
31	1.64	40	3.6E-12	8.1E-13	5.5E-12	2.4E-12	6.3E-14	2.3E-13	1.0E-12	0.0E+00	2.8E-12	1.4E-12	5.0E-12	0.0E+00	1.6E-11	8.1E-12	2.9E-11	0.0E+00	3.0E-12	1.5E-12	5.5E-12	0.0E+00
44[1]	1.71	17	1.3E-11	3.4E-12	2.2E-11	7.7E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	1.6E-11	2.8E-12	2.1E-11	8.4E-12	3.1E-12	5.5E-13	4.1E-12	1.7E-12	1.5E-11	2.7E-12	2.0E-11	8.2E-12
Summary			3.5E-12	1.5E-12	2.2E-11	6.5E-14	3.4E-13	2.9E-13	6.1E-12	0.0E+00	1.7E-12	7.7E-13	2.1E-11	0.0E+00	4.1E-12	2.3E-12	2.9E-11	0.0E+00	2.8E-12	1.3E-12	2.0E-11	0.0E+00

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 53. Model 2, Periods A-D Silicate Mineral Dissolution Rates. Page 3 of 4.

Model 2, Period C																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	21	2.9E-12	1.8E-12	7.0E-12	1.5E-12	2.1E-12	8.9E-13	4.1E-12	7.6E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	22	2.8E-12	1.0E-12	4.3E-12	3.9E-13	1.9E-13	3.8E-13	1.3E-12	0.0E+00	1.2E-12	1.5E-12	5.6E-12	0.0E+00	1.6E-12	1.9E-12	7.0E-12	0.0E+00	1.2E-12	1.4E-12	5.4E-12	0.0E+00
9	0.51	29	4.5E-12	1.9E-12	9.4E-12	4.6E-13	5.0E-13	8.3E-13	2.7E-12	0.0E+00	2.9E-13	3.7E-13	1.4E-12	0.0E+00	1.7E-12	2.2E-12	7.8E-12	0.0E+00	5.3E-13	6.9E-13	2.5E-12	0.0E+00
11	0.54	40	1.0E-12	9.6E-13	3.9E-12	3.8E-13	1.1E-12	4.2E-13	2.1E-12	8.8E-14	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
13	0.57	14	2.6E-12	7.2E-13	4.1E-12	1.6E-12	3.1E-14	6.2E-14	1.6E-13	0.0E+00	5.0E-12	4.7E-12	1.7E-11	0.0E+00	2.0E-12	1.9E-12	7.1E-12	0.0E+00				
15	0.58	30	2.4E-12	7.2E-13	4.9E-12	1.4E-12	1.4E-13	2.6E-13	7.9E-13	0.0E+00	8.3E-13	8.0E-13	2.9E-12	0.0E+00	8.7E-12	8.4E-12	3.0E-11	0.0E+00	5.6E-12	5.4E-12	2.0E-11	0.0E+00
39[1]	0.67	21	1.3E-12	1.6E-13	1.6E-12	1.1E-12	5.3E-16	2.4E-15	1.1E-14	0.0E+00	2.3E-12	1.1E-12	4.1E-12	0.0E+00	1.0E-12	4.9E-13	1.8E-12	0.0E+00				
40[2]	0.67	10	3.4E-12	9.1E-13	4.3E-12	1.5E-12	3.1E-13	4.3E-13	1.1E-12	0.0E+00	1.7E-13	1.8E-13	5.4E-13	0.0E+00	9.9E-13	1.0E-12	3.1E-12	0.0E+00	1.1E-13	1.1E-13	3.4E-13	0.0E+00
17	0.71	38	5.2E-12	3.2E-12	1.8E-11	2.4E-13	2.8E-14	1.1E-13	5.4E-13	0.0E+00	4.0E-12	4.1E-12	1.5E-11	0.0E+00	5.8E-12	5.9E-12	2.2E-11	0.0E+00	8.1E-12	8.2E-12	3.1E-11	0.0E+00
35	1.12	33	1.5E-11	6.0E-12	2.6E-11	6.1E-12	5.5E-13	7.1E-13	2.4E-12	0.0E+00	3.0E-13	5.1E-13	1.6E-12	0.0E+00	2.1E-12	3.6E-12	1.1E-11	0.0E+00	8.1E-13	1.4E-12	4.3E-12	0.0E+00
29	1.16	48	6.7E-12	3.1E-12	1.4E-11	2.9E-12	2.5E-14	1.7E-13	1.2E-12	0.0E+00	4.7E-12	3.1E-12	1.2E-11	0.0E+00	5.9E-12	3.9E-12	1.5E-11	0.0E+00				
37	1.40	38	7.0E-12	2.7E-12	1.2E-11	2.9E-12	1.7E-15	1.1E-14	6.5E-14	0.0E+00	1.4E-11	7.4E-12	2.7E-11	0.0E+00	3.0E-11	1.6E-11	6.0E-11	0.0E+00				
33	1.44	57	2.8E-11	1.6E-11	5.9E-11	7.4E-12	7.8E-15	3.7E-14	2.1E-13	0.0E+00	2.4E-12	1.7E-12	6.3E-12	0.0E+00	1.8E-11	1.3E-11	4.7E-11	0.0E+00	8.6E-12	6.1E-12	2.2E-11	0.0E+00
19	1.63	64	1.1E-11	4.4E-12	2.4E-11	4.6E-12	4.5E-14	3.1E-13	2.4E-12	0.0E+00	3.4E-12	1.5E-12	6.7E-12	0.0E+00	2.2E-11	9.8E-12	4.3E-11	0.0E+00	8.1E-12	3.6E-12	1.6E-11	0.0E+00
31	1.64	28	5.9E-12	1.5E-12	9.0E-12	4.0E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	4.2E-12	1.9E-12	8.2E-12	8.1E-13	2.4E-11	1.1E-11	4.8E-11	4.7E-12	4.6E-12	2.1E-12	9.0E-12	8.9E-13
Summary			6.7E-12	3.0E-12	5.9E-11	2.4E-13	3.3E-13	3.1E-13	4.1E-12	0.0E+00	2.8E-12	1.9E-12	2.7E-11	0.0E+00	9.6E-12	6.1E-12	6.0E-11	0.0E+00	3.4E-12	2.6E-12	3.1E-11	0.0E+00

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 53. Model 2, Periods A-D Silicate Mineral Dissolution Rates. Page 4 of 4.

Model 2, Period D																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	9	1.5E-12	1.6E-14	1.6E-12	1.5E-12	7.0E-13	2.6E-13	1.2E-12	2.9E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	54	8.2E-13	7.7E-13	3.1E-12	3.6E-13	4.0E-13	2.0E-13	8.9E-13	0.0E+00	1.8E-14	9.3E-14	5.9E-13	0.0E+00	2.2E-14	1.2E-13	7.5E-13	0.0E+00	1.7E-14	9.0E-14	5.7E-13	0.0E+00
9	0.51	66	1.1E-12	1.1E-12	5.4E-12	4.2E-13	7.7E-13	4.3E-13	2.7E-12	0.0E+00	1.3E-14	8.6E-14	6.6E-13	0.0E+00	7.8E-14	5.0E-13	3.8E-12	0.0E+00	2.5E-14	1.6E-13	1.2E-12	0.0E+00
13	0.57	21	1.5E-12	6.7E-13	2.6E-12	2.0E-13	5.1E-13	5.8E-13	2.1E-12	0.0E+00	2.5E-13	7.2E-13	3.2E-12	0.0E+00	1.0E-13	3.0E-13	1.3E-12	0.0E+00				
15	0.58	80	5.8E-13	6.3E-13	3.8E-12	2.3E-13	7.9E-13	3.9E-13	2.4E-12	0.0E+00	8.7E-15	6.9E-14	6.1E-13	0.0E+00	9.1E-14	7.2E-13	6.4E-12	0.0E+00	5.9E-14	4.7E-13	4.2E-12	0.0E+00
39[1]	0.67	80	3.7E-13	3.3E-13	1.4E-12	6.1E-14	2.0E-13	1.9E-13	6.4E-13	0.0E+00	4.7E-14	1.1E-13	5.8E-13	0.0E+00	2.1E-14	4.8E-14	2.6E-13	0.0E+00				
40[2]	0.67	44	4.9E-13	3.8E-13	2.0E-12	2.1E-13	5.6E-13	3.4E-13	1.4E-12	1.4E-14	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
17	0.71	58	1.4E-12	1.3E-12	4.3E-12	3.9E-13	4.5E-13	2.1E-13	8.1E-13	0.0E+00	3.1E-14	1.6E-13	1.1E-12	0.0E+00	4.6E-14	2.3E-13	1.6E-12	0.0E+00	6.4E-14	3.1E-13	2.3E-12	0.0E+00
35	1.12	3	7.5E-12	7.3E-13	8.2E-12	6.7E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	4.2E-13	2.4E-13	6.9E-13	2.6E-13	3.0E-12	1.7E-12	4.9E-12	1.9E-12	1.1E-12	6.5E-13	1.9E-12	7.1E-13
29	1.16	47	1.1E-12	9.2E-13	4.1E-12	3.3E-13	4.5E-13	2.5E-13	9.6E-13	0.0E+00	1.1E-13	3.2E-13	1.8E-12	0.0E+00	1.4E-13	4.1E-13	2.2E-12	0.0E+00				
33	1.44	32	6.4E-12	2.6E-12	1.4E-11	1.2E-12	1.8E-13	2.0E-13	7.1E-13	0.0E+00	3.1E-14	5.7E-14	2.4E-13	0.0E+00	2.3E-13	4.2E-13	1.8E-12	0.0E+00	1.1E-13	2.0E-13	8.6E-13	0.0E+00
19	1.63	62	3.7E-12	2.3E-12	1.0E-11	5.9E-13	2.6E-13	3.2E-13	1.2E-12	0.0E+00	2.4E-13	4.1E-13	1.4E-12	0.0E+00	1.6E-12	2.7E-12	9.0E-12	0.0E+00	5.7E-13	9.7E-13	3.2E-12	0.0E+00
31	1.64	5	4.1E-12	6.1E-13	5.1E-12	3.6E-12	1.3E-14	3.0E-14	6.7E-14	0.0E+00	5.6E-13	4.0E-13	1.0E-12	0.0E+00	3.3E-12	2.3E-12	6.0E-12	0.0E+00	6.2E-13	4.4E-13	1.1E-12	0.0E+00
Summary			2.3E-12	9.5E-13	1.4E-11	6.1E-14	4.1E-13	2.6E-13	2.7E-12	0.0E+00	1.3E-13	2.1E-13	3.2E-12	0.0E+00	7.1E-13	7.9E-13	9.0E-12	0.0E+00	2.6E-13	3.3E-13	4.2E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 54. Model 3, Periods A-D Silicate Mineral Dissolution Rates. Page 1 of 4.

Model 3, Period A																						
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
39[1]	0.67	2	5.1E-13	9.9E-14	5.8E-13	4.4E-13	2.7E-12	2.3E-12	4.4E-12	1.1E-12	4.7E-12	6.5E-12	9.3E-12	9.8E-14	2.1E-12	2.9E-12	4.1E-12	4.3E-14				
40[2]	0.67	3	4.9E-13	9.8E-14	6.0E-13	4.2E-13	5.4E-12	9.8E-13	6.2E-12	4.3E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
42[1]	0.72	2	7.5E-13	8.7E-14	8.1E-13	6.9E-13	7.2E-12	1.0E-12	7.9E-12	6.5E-12	4.8E-13	3.0E-13	6.9E-13	2.7E-13	1.3E-13	7.9E-14	1.8E-13	7.1E-14				
43[2]	0.82	3	4.9E-13	2.7E-13	6.9E-13	1.8E-13	2.7E-12	9.9E-13	3.7E-12	1.7E-12	2.4E-13	4.2E-13	7.3E-13	0.0E+00	2.5E-12	4.3E-12	7.4E-12	0.0E+00	4.0E-13	6.9E-13	1.2E-12	0.0E+00
41[1]	0.92	3	7.2E-13	2.6E-14	7.4E-13	6.9E-13	5.7E-12	4.8E-12	1.1E-11	1.5E-12	2.5E-12	4.4E-12	7.6E-12	0.0E+00	3.9E-13	6.8E-13	1.2E-12	0.0E+00				
44[1]	1.71	4	7.3E-13	1.7E-13	9.6E-13	5.6E-13	2.9E-11	8.8E-12	4.2E-11	2.4E-11	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
Summary			6.2E-13	1.2E-13	9.6E-13	1.8E-13	8.8E-12	3.2E-12	4.2E-11	1.1E-12	1.3E-12	1.9E-12	9.3E-12	0.0E+00	8.4E-13	1.3E-12	7.4E-12	0.0E+00	1.3E-13	2.3E-13	1.2E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 54. Model 3, Periods A-D Silicate Mineral Dissolution Rates. Page 2 of 4.

Model 3, Period B																						
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	32	3.6E-13	1.6E-13	8.1E-13	1.5E-13	5.1E-13	3.8E-13	1.8E-12	0.0E+00	1.7E-13	3.4E-13	1.7E-12	0.0E+00	6.2E-13	1.3E-12	6.1E-12	0.0E+00	5.3E-13	1.1E-12	5.2E-12	0.0E+00
3	0.22	34	2.2E-13	8.9E-14	4.0E-13	1.1E-13	1.1E-12	4.7E-13	2.4E-12	2.1E-13	8.8E-13	1.2E-12	5.2E-12	0.0E+00	2.0E-12	2.7E-12	1.2E-11	0.0E+00	4.7E-13	6.2E-13	2.7E-12	0.0E+00
5	0.40	20	2.3E-13	1.2E-13	6.3E-13	1.8E-13	3.4E-12	3.2E-12	1.1E-11	1.5E-12	1.9E-14	6.4E-14	2.7E-13	0.0E+00					9.2E-14	3.0E-13	1.3E-12	0.0E+00
39[1]	0.67	4	5.7E-13	3.2E-13	1.0E-12	3.7E-13	1.6E-13	2.0E-13	4.1E-13	0.0E+00	4.4E-12	3.3E-12	8.1E-12	1.8E-13	2.0E-12	1.4E-12	3.6E-12	7.9E-14				
40[2]	0.67	18	4.8E-13	1.7E-13	9.2E-13	1.6E-13	1.5E-12	6.4E-13	2.8E-12	3.2E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
42[1]	0.72	33	2.9E-13	1.3E-13	4.7E-13	1.5E-13	1.2E-12	1.3E-12	4.1E-12	0.0E+00	1.2E-12	1.1E-12	4.9E-12	0.0E+00	3.2E-13	2.9E-13	1.3E-12	0.0E+00				
43[2]	0.82	32	2.3E-13	9.8E-14	5.2E-13	1.8E-13	2.0E-13	1.3E-13	5.2E-13	0.0E+00	3.7E-14	5.8E-14	1.5E-13	0.0E+00	3.7E-13	5.8E-13	1.5E-12	0.0E+00	6.0E-14	9.4E-14	2.4E-13	0.0E+00
41[1]	0.92	33	4.0E-13	2.7E-13	1.5E-12	2.1E-13	2.8E-13	3.1E-13	8.0E-13	0.0E+00	6.3E-13	6.4E-13	3.1E-12	0.0E+00	9.8E-14	9.8E-14	4.9E-13	0.0E+00				
44[1]	1.71	2	6.1E-13	2.0E-13	7.5E-13	4.7E-13	8.9E-12	4.7E-12	1.2E-11	5.6E-12	1.9E-11	1.6E-12	2.0E-11	1.7E-11	3.7E-12	3.3E-13	4.0E-12	3.5E-12	1.8E-11	1.6E-12	1.9E-11	1.7E-11
Summary			3.8E-13	1.7E-13	1.5E-12	1.1E-13	1.9E-12	1.3E-12	1.2E-11	0.0E+00	2.9E-12	9.2E-13	2.0E-11	0.0E+00	1.1E-12	8.4E-13	1.2E-11	0.0E+00	3.2E-12	6.2E-13	1.9E-11	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 54. Model 3, Periods A-D Silicate Mineral Dissolution Rates. Page 3 of 4.

Model 3, Period C																						
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	7	6.4E-13	2.2E-13	9.9E-13	4.3E-13	2.8E-12	2.0E-12	7.0E-12	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	2	5.9E-13	1.1E-13	6.7E-13	5.1E-13	1.5E-12	8.7E-15	1.5E-12	1.4E-12	2.7E-12	1.5E-12	3.7E-12	1.7E-12	3.4E-12	1.8E-12	4.7E-12	2.1E-12	2.6E-12	1.4E-12	3.6E-12	1.6E-12
11	0.54	2	6.2E-13	1.3E-13	7.2E-13	5.3E-13	8.8E-13	6.9E-13	1.4E-12	3.9E-13	2.5E-13	3.5E-13	5.0E-13	0.0E+00					5.8E-13	8.2E-13	1.2E-12	0.0E+00
39[1]	0.67	7	5.3E-13	5.5E-14	6.2E-13	4.4E-13	3.7E-13	1.8E-13	7.0E-13	1.6E-13	4.0E-12	7.6E-13	4.8E-12	2.6E-12	1.8E-12	3.3E-13	2.1E-12	1.2E-12				
40[2]	0.67	4	7.0E-13	7.8E-14	7.9E-13	6.1E-13	2.7E-12	3.7E-13	3.1E-12	2.2E-12	7.6E-13	5.1E-13	1.1E-12	0.0E+00	4.4E-12	2.9E-12	6.1E-12	0.0E+00	4.7E-13	3.2E-13	6.6E-13	0.0E+00
17	0.71	2	6.2E-13	1.2E-13	7.0E-13	5.3E-13	1.9E-12	3.9E-13	2.2E-12	1.7E-12	2.0E-12	9.0E-13	2.7E-12	1.4E-12	2.9E-12	1.3E-12	3.9E-12	2.0E-12	4.1E-12	1.8E-12	5.4E-12	2.8E-12
35	1.12	2	2.0E-12	1.3E-12	2.9E-12	1.2E-12	4.5E-12	6.4E-12	9.0E-12	0.0E+00	2.0E-12	3.8E-13	2.3E-12	1.7E-12	1.4E-11	2.7E-12	1.6E-11	1.2E-11	5.4E-12	1.0E-12	6.1E-12	4.7E-12
29	1.16	3	9.1E-13	3.0E-13	1.3E-12	6.8E-13	3.2E-12	2.0E-12	5.5E-12	1.9E-12	6.9E-12	5.8E-12	1.4E-11	2.7E-12	8.8E-12	7.5E-12	1.7E-11	3.4E-12				
37	1.40	3	1.5E-12	2.6E-13	1.7E-12	1.2E-12	5.3E-12	1.5E-12	6.2E-12	3.6E-12	1.8E-11	5.7E-12	2.5E-11	1.4E-11	4.0E-11	1.2E-11	5.4E-11	3.2E-11				
33	1.44	7	8.6E-13	4.0E-13	1.7E-12	5.6E-13	1.4E-11	6.5E-12	2.4E-11	6.7E-12	3.5E-12	1.8E-12	6.2E-12	9.6E-13	2.6E-11	1.3E-11	4.6E-11	7.2E-12	1.2E-11	6.4E-12	2.2E-11	3.4E-12
19	1.63	1	1.0E-12	N/A	N/A	N/A	7.8E-12	N/A	N/A	N/A	2.6E-12	N/A	N/A	N/A	1.7E-11	N/A	N/A	N/A	6.1E-12	N/A	N/A	N/A
31	1.64	1	1.1E-12	N/A	N/A	N/A	3.7E-12	N/A	N/A	N/A	4.9E-12	N/A	N/A	N/A	2.8E-11	N/A	N/A	N/A	5.3E-12	N/A	N/A	N/A
Summary			9.2E-13	2.9E-13	2.9E-12	4.3E-13	4.0E-12	2.0E-12	2.4E-11	0.0E+00	4.0E-12	1.8E-12	2.5E-11	0.0E+00	1.5E-11	5.3E-12	5.4E-11	0.0E+00	4.1E-12	1.7E-12	2.2E-11	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 54. Model 3, Periods A-D Silicate Mineral Dissolution Rates. Page 4 of 4.

Model 3, Period D																						
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	3	2.6E-13	1.2E-13	4.0E-13	1.8E-13	1.5E-12	1.7E-14	1.6E-12	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	14	4.8E-13	1.6E-13	8.5E-13	3.2E-13	3.6E-13	2.2E-13	8.3E-13	0.0E+00	3.3E-13	5.9E-13	1.6E-12	0.0E+00	4.2E-13	7.4E-13	2.0E-12	0.0E+00	3.2E-13	5.6E-13	1.5E-12	0.0E+00
9	0.51	17	6.9E-13	2.6E-13	1.4E-12	4.1E-13	6.0E-13	5.2E-13	2.1E-12	0.0E+00	2.0E-13	3.7E-13	1.2E-12	0.0E+00	1.2E-12	2.1E-12	7.2E-12	0.0E+00	3.7E-13	6.7E-13	2.3E-12	0.0E+00
13	0.57	2	1.1E-12	4.6E-13	1.4E-12	7.7E-13	6.0E-13	1.5E-13	7.1E-13	4.9E-13	2.2E-12	8.7E-13	2.8E-12	1.6E-12	9.0E-13	3.5E-13	1.2E-12	6.5E-13				
15	0.58	18	6.7E-13	5.6E-13	2.9E-12	3.6E-13	3.2E-13	2.0E-13	9.3E-13	0.0E+00	2.2E-13	5.6E-13	2.0E-12	0.0E+00	2.4E-12	5.9E-12	2.1E-11	0.0E+00	1.5E-12	3.8E-12	1.4E-11	0.0E+00
39[1]	0.67	25	2.6E-13	1.1E-13	5.0E-13	1.4E-13	2.0E-13	1.7E-13	7.4E-13	0.0E+00	3.5E-13	4.5E-13	1.6E-12	0.0E+00	1.5E-13	2.0E-13	7.0E-13	0.0E+00				
40[2]	0.67	10	5.4E-13	2.1E-13	9.5E-13	1.6E-13	4.7E-13	4.7E-13	1.5E-12	0.0E+00	6.3E-14	1.0E-13	3.1E-13	0.0E+00	3.6E-13	5.9E-13	1.8E-12	0.0E+00	3.9E-14	6.4E-14	1.9E-13	0.0E+00
17	0.71	16	5.5E-13	1.3E-13	7.9E-13	3.9E-13	9.9E-13	7.6E-13	2.7E-12	0.0E+00	3.2E-13	6.0E-13	2.1E-12	0.0E+00	4.6E-13	8.6E-13	3.1E-12	0.0E+00	6.4E-13	1.2E-12	4.3E-12	0.0E+00
35	1.12	1	9.4E-13	N/A	N/A	N/A	4.0E-12	N/A	N/A	N/A	8.1E-13	N/A	N/A	N/A	5.7E-12	N/A	N/A	N/A	2.2E-12	N/A	N/A	N/A
29	1.16	15	5.5E-13	1.3E-13	8.5E-13	3.5E-13	5.9E-13	2.9E-13	1.1E-12	3.3E-13	3.4E-13	5.8E-13	1.7E-12	0.0E+00	4.4E-13	7.4E-13	2.2E-12	0.0E+00				
33	1.44	11	5.2E-13	1.5E-13	8.7E-13	3.6E-13	2.6E-12	1.4E-12	4.3E-12	0.0E+00	1.8E-13	2.1E-13	6.8E-13	0.0E+00	1.3E-12	1.6E-12	5.1E-12	0.0E+00	6.3E-13	7.5E-13	2.4E-12	0.0E+00
19	1.63	18	6.9E-13	2.0E-13	1.2E-12	4.7E-13	2.5E-12	1.8E-12	6.4E-12	5.9E-13	4.3E-13	6.7E-13	1.8E-12	0.0E+00	2.8E-12	4.4E-12	1.2E-11	0.0E+00	1.0E-12	1.6E-12	4.3E-12	0.0E+00
31	1.64	2	1.1E-12	1.2E-13	1.2E-12	1.0E-12	2.3E-12	4.1E-14	2.4E-12	2.3E-12	2.1E-12	5.8E-15	2.1E-12	2.1E-12	1.2E-11	3.4E-14	1.2E-11	1.2E-11	2.3E-12	6.4E-15	2.3E-12	2.3E-12
Summary			6.4E-13	2.2E-13	2.9E-12	1.4E-13	1.3E-12	5.0E-13	6.4E-12	0.0E+00	5.8E-13	4.2E-13	2.8E-12	0.0E+00	2.4E-12	1.6E-12	2.1E-11	0.0E+00	9.0E-13	9.7E-13	1.4E-11	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 55. Model 3, Periods A-D Orphan Calcium Rates.

Model 3, Orphan Calcium (mol $m^{-2}s^{-1}$)																	
R#	%S	Period A				Period B				Period C				Period D			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18					1.7E-13	3.1E-13	1.2E-12	0.0E+00								
3	0.22					3.1E-14	1.7E-13	9.7E-13	0.0E+00								
5	0.4					4.6E-12	2.4E-12	9.0E-12	0.0E+00	6.5E-12	3.2E-12	1.2E-11	2.7E-12	1.6E-12	1.3E-13	1.7E-12	1.5E-12
7	0.41									0.0E+00	0.0E+00	0.0E+00	0.0E+00	2.3E-13	3.6E-13	1.2E-12	0.0E+00
9	0.51													3.2E-13	4.2E-13	1.4E-12	0.0E+00
11	0.54									9.5E-13	1.3E-12	1.9E-12	0.0E+00				
13	0.57													0.0E+00	0.0E+00	0.0E+00	0.0E+00
15	0.58													8.0E-13	5.9E-13	1.6E-12	0.0E+00
39[1]	0.67	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	2.8E-13	5.3E-13	1.9E-12	0.0E+00
40[2]	0.67	2.2E-12	2.6E-12	5.1E-12	1.5E-13	5.2E-12	1.3E-12	6.8E-12	2.1E-12	2.8E-13	5.6E-13	1.1E-12	0.0E+00	4.9E-13	6.8E-13	1.9E-12	0.0E+00
17	0.71									0.0E+00	0.0E+00	0.0E+00	0.0E+00	2.7E-13	3.2E-13	1.0E-12	0.0E+00
42[1]	0.72	0.0E+00	0.0E+00	0.0E+00	0.0E+00	4.9E-14	2.3E-13	1.3E-12	0.0E+00								
43[2]	0.82	7.5E-12	1.0E-11	1.9E-11	0.0E+00	1.5E-12	1.8E-12	6.8E-12	0.0E+00								
41[1]	0.92	3.4E-12	5.5E-12	9.8E-12	0.0E+00	7.7E-14	2.1E-13	8.4E-13	0.0E+00								
35	1.12									0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00		0.0E+00	0.0E+00
29	1.16									0.0E+00	0.0E+00	0.0E+00	0.0E+00	4.3E-13	5.0E-13	1.5E-12	0.0E+00
37	1.4									0.0E+00	0.0E+00	0.0E+00	0.0E+00				
33	1.44									0.0E+00	0.0E+00	0.0E+00	0.0E+00	1.3E-13	4.1E-13	1.4E-12	0.0E+00
19	1.63									0.0E+00		0.0E+00	0.0E+00	1.2E-13	3.0E-13	1.1E-12	0.0E+00
31	1.64									0.0E+00		0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
44[1]	1.71	9.2E-12	5.7E-12	1.2E-11	5.9E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00								
Summary		3.7E-12	4.0E-12	1.9E-11	0.0E+00	1.3E-12	7.1E-13	9.0E-12	0.0E+00	6.4E-13	5.1E-13	1.2E-11	0.0E+00	3.6E-13	3.5E-13	1.9E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 56. Model 4, Periods A-D Silicate Mineral Dissolution Rates. Page 1 of 4.

Model 4, Period A																						
R#	%S	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
39[1]	0.67	2					2.4E-10	2.9E-10	4.4E-10	3.2E-11	2.7E-12	2.3E-12	4.4E-12	1.1E-12	4.7E-12	6.5E-12	9.3E-12	9.8E-14	2.1E-12	2.9E-12	4.1E-12	4.3E-14
40[2]	0.67	3	4.1E-12	3.6E-12	8.2E-12	1.8E-12	1.5E-10	1.3E-10	3.0E-10	6.4E-11	5.4E-12	9.8E-13	6.2E-12	4.3E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
42[1]	0.72	2									7.2E-12	1.0E-12	7.9E-12	6.5E-12	4.8E-13	3.0E-13	6.9E-13	2.7E-13	1.3E-13	7.9E-14	1.8E-13	7.1E-14
43[2]	0.82	3	6.4E-12	4.1E-12	1.0E-11	2.1E-12					2.7E-12	9.9E-13	3.7E-12	1.7E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
41[1]	0.92	3									5.7E-12	4.8E-12	1.1E-11	1.5E-12	2.5E-12	4.4E-12	7.6E-12	0.0E+00	3.9E-13	6.8E-13	1.2E-12	0.0E+00
44[1]	1.71	4	3.6E-11	1.0E-11	4.7E-11	2.7E-11	3.5E-10	1.0E-10	4.7E-10	2.7E-10	2.9E-11	8.8E-12	4.2E-11	2.4E-11	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
Summary			1.5E-11	5.9E-12	4.7E-11	1.8E-12	2.5E-10	1.7E-10	4.7E-10	3.2E-11	8.8E-12	3.2E-12	4.2E-11	1.1E-12	1.3E-12	1.9E-12	9.3E-12	0.0E+00	4.3E-13	6.1E-13	4.1E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table AZ).
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 56. Model 4, Periods A-D Silicate Mineral Dissolution Rates. Page 2 of 4.

Model 4, Period B																						
R#	%S	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	32	6.0E-12	5.3E-12	1.7E-11	7.0E-13					5.1E-13	3.8E-13	1.8E-12	0.0E+00	3.1E-15	1.8E-14	1.0E-13	0.0E+00	1.2E-14	6.5E-14	3.7E-13	0.0E+00
3	0.22	33	2.3E-12	2.4E-12	8.6E-12	2.0E-13	1.0E-11	1.1E-11	3.8E-11	8.9E-13	1.1E-12	4.7E-13	2.4E-12	2.1E-13	2.2E-14	1.3E-13	7.3E-13	0.0E+00	5.0E-14	2.9E-13	1.7E-12	0.0E+00
5	0.40	18	5.6E-12	4.2E-12	1.5E-11	1.5E-12	1.6E-11	1.2E-11	4.5E-11	4.3E-12	3.6E-12	3.3E-12	1.1E-11	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
39[1]	0.67	4					9.3E-11	7.1E-11	1.8E-10	1.7E-11	1.6E-13	2.0E-13	4.1E-13	0.0E+00	4.4E-12	3.3E-12	8.1E-12	1.8E-13	2.0E-12	1.4E-12	3.6E-12	7.9E-14
40[2]	0.67	18	1.6E-12	1.2E-12	5.6E-12	6.5E-13	5.8E-11	4.3E-11	2.0E-10	2.3E-11	1.5E-12	6.4E-13	2.8E-12	3.2E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
42[1]	0.72	32									1.1E-12	1.2E-12	3.8E-12	0.0E+00	1.2E-12	1.1E-12	4.9E-12	0.0E+00	3.2E-13	2.9E-13	1.3E-12	0.0E+00
43[2]	0.82	28	6.2E-13	1.2E-12	6.3E-12	1.7E-13					1.8E-13	1.2E-13	2.7E-13	0.0E+00	2.8E-15	7.1E-15	2.8E-14	0.0E+00	2.8E-14	7.2E-14	2.8E-13	0.0E+00
41[1]	0.92	31									2.7E-13	3.1E-13	8.0E-13	0.0E+00	5.7E-13	4.4E-13	1.5E-12	0.0E+00	8.8E-14	6.9E-14	2.4E-13	0.0E+00
44[1]	1.71	2	4.5E-11	1.1E-11	5.3E-11	3.8E-11	4.5E-10	1.1E-10	5.2E-10	3.7E-10	8.9E-12	4.7E-12	1.2E-11	5.6E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
Summary			1.0E-11	4.2E-12	5.3E-11	1.7E-13	1.3E-10	4.9E-11	5.2E-10	8.9E-13	1.9E-12	1.3E-12	1.2E-11	0.0E+00	6.9E-13	5.5E-13	8.1E-12	0.0E+00	3.1E-13	2.8E-13	3.6E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table AZ).
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 56. Model 4, Periods A-D Silicate Mineral Dissolution Rates. Page 3 of 4.

Model 4, Period C																						
R#	%S	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	7	1.2E-12	9.9E-13	2.5E-12	3.2E-13	3.5E-12	2.9E-12	7.2E-12	9.4E-13	2.8E-12	2.0E-12	7.0E-12	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
7	0.41	2	4.5E-12	1.8E-12	5.8E-12	3.2E-12	6.8E-11	2.8E-11	8.7E-11	4.8E-11	1.5E-12	8.7E-15	1.5E-12	1.4E-12	2.0E-13	2.8E-13	3.9E-13	0.0E+00	2.5E-13	3.5E-13	5.0E-13	0.0E+00
11	0.54	2	2.5E-12	4.1E-13	2.8E-12	2.2E-12					8.8E-13	6.9E-13	1.4E-12	3.9E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
39[1]	0.67	7					4.8E-11	1.9E-11	8.8E-11	3.1E-11	3.7E-13	1.8E-13	7.0E-13	1.6E-13	4.0E-12	7.6E-13	4.8E-12	2.6E-12	1.8E-12	3.3E-13	2.1E-12	1.2E-12
40[2]	0.67	4	1.1E-12	4.6E-13	1.5E-12	4.6E-13	3.9E-11	1.7E-11	5.3E-11	1.6E-11	2.7E-12	3.7E-13	3.1E-12	2.2E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
17	0.71	2	5.9E-12	1.8E-12	7.2E-12	4.6E-12	6.1E-11	1.9E-11	7.5E-11	4.8E-11	1.9E-12	3.9E-13	2.2E-12	1.7E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
35	1.12	2	6.4E-12	2.3E-12	8.0E-12	4.8E-12	5.9E-12	2.1E-12	7.4E-12	4.4E-12	4.5E-12	6.4E-12	9.0E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
29	1.16	3									3.2E-12	2.0E-12	5.5E-12	1.9E-12	6.9E-12	5.8E-12	1.4E-11	2.7E-12	8.8E-12	7.5E-12	1.7E-11	3.4E-12
37	1.40	3					7.2E-11	3.3E-11	9.7E-11	3.5E-11	5.3E-12	1.5E-12	6.2E-12	3.6E-12	1.8E-11	5.7E-12	2.5E-11	1.4E-11	4.0E-11	1.2E-11	5.4E-11	3.2E-11
33	1.44	7	1.0E-11	6.9E-12	2.4E-11	2.5E-12	2.4E-11	1.6E-11	5.5E-11	6.0E-12	1.4E-11	6.5E-12	2.4E-11	6.7E-12	9.0E-13	8.8E-13	2.0E-12	0.0E+00	6.7E-12	6.5E-12	1.5E-11	0.0E+00
19	1.63	1	7.1E-12	N/A	N/A	N/A	1.2E-11	N/A	N/A	N/A	7.8E-12	N/A	N/A	N/A	0.0E+00	N/A	N/A	N/A	0.0E+00	N/A	N/A	N/A
31	1.64	1	3.1E-12	N/A	N/A	N/A	1.0E-11	N/A	N/A	N/A	3.7E-12	N/A	N/A	N/A	2.0E-12	N/A	N/A	N/A	1.2E-11	N/A	N/A	N/A
Summary			4.7E-12	2.1E-12	2.4E-11	3.2E-13	3.4E-11	1.7E-11	9.7E-11	9.4E-13	4.0E-12	2.0E-12	2.4E-11	0.0E+00	2.7E-12	1.3E-12	2.5E-11	0.0E+00	7.0E-12	3.4E-12	5.4E-11	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table AZ).
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 56. Model 4, Periods A-D Silicate Mineral Dissolution Rates. Page 4 of 4.

Model 4, Period D																						
R#	%S	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	2	4.0E-13	3.4E-15	4.1E-13	4.0E-13	1.2E-12	1.0E-14	1.2E-12	1.2E-12	1.6E-12	1.3E-14	1.6E-12	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
7	0.41	12	2.8E-12	2.0E-12	7.5E-12	8.1E-13	4.1E-11	3.0E-11	1.1E-10	1.2E-11	3.9E-13	2.1E-13	8.3E-13	0.0E+00	2.0E-14	7.0E-14	2.4E-13	0.0E+00	2.6E-14	8.8E-14	3.1E-13	0.0E+00
9	0.51	16	2.3E-12	1.8E-12	6.9E-12	2.2E-13	6.7E-12	5.2E-12	2.0E-11	6.3E-13	6.1E-13	5.4E-13	2.1E-12	0.0E+00	2.8E-14	1.1E-13	4.5E-13	0.0E+00	1.6E-13	6.5E-13	2.6E-12	0.0E+00
13	0.57	2					1.8E-11	9.4E-12	2.5E-11	1.2E-11	6.0E-13	1.5E-13	7.1E-13	4.9E-13	2.2E-12	8.7E-13	2.8E-12	1.6E-12	9.0E-13	3.5E-13	1.2E-12	6.5E-13
15	0.58	17	1.1E-11	1.3E-11	4.1E-11	6.9E-13	1.1E-11	1.2E-11	3.8E-11	6.4E-13	3.2E-13	2.1E-13	9.3E-13	0.0E+00	3.5E-14	1.4E-13	6.0E-13	0.0E+00	3.7E-13	1.5E-12	6.3E-12	0.0E+00
39[1]	0.67	25					9.3E-12	1.0E-11	3.8E-11	2.1E-12	2.0E-13	1.7E-13	7.4E-13	0.0E+00	3.5E-13	4.5E-13	1.6E-12	0.0E+00	1.5E-13	2.0E-13	7.0E-13	0.0E+00
40[2]	0.67	10	2.6E-13	1.1E-13	4.3E-13	8.8E-14	9.5E-12	3.8E-12	1.5E-11	3.2E-12	4.7E-13	4.7E-13	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
17	0.71	16	7.4E-12	7.8E-12	2.3E-11	9.9E-13	7.7E-11	8.1E-11	2.3E-10	1.0E-11	9.9E-13	7.6E-13	2.7E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
35	1.12	1	6.8E-12	N/A	N/A	N/A	6.3E-12	N/A	N/A	N/A	4.0E-12	N/A	N/A	N/A	0.0E+00	N/A	N/A	N/A	0.0E+00	N/A	N/A	N/A
29	1.16	15									5.9E-13	2.9E-13	1.1E-12	3.3E-13	3.4E-13	5.8E-13	1.7E-12	0.0E+00	4.4E-13	7.4E-13	2.2E-12	0.0E+00
33	1.44	9	1.4E-12	6.0E-13	2.5E-12	7.2E-13	3.3E-12	1.4E-12	5.9E-12	1.7E-12	2.7E-12	1.5E-12	4.3E-12	0.0E+00	5.6E-14	1.2E-13	3.2E-13	0.0E+00	4.2E-13	8.6E-13	2.4E-12	0.0E+00
19	1.63	17	6.9E-12	6.3E-12	2.4E-11	1.6E-12	1.2E-11	1.1E-11	4.1E-11	2.7E-12	2.3E-12	1.6E-12	6.4E-12	5.9E-13	6.0E-14	2.4E-13	1.0E-12	0.0E+00	3.9E-13	1.6E-12	6.6E-12	0.0E+00
31	1.64	2	4.6E-12	7.5E-13	5.1E-12	4.1E-12	1.5E-11	2.4E-12	1.6E-11	1.3E-11	2.3E-12	4.1E-14	2.4E-12	2.3E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
Summary			4.4E-12	3.6E-12	4.1E-11	8.8E-14	1.8E-11	1.5E-11	2.3E-10	6.3E-13	1.3E-12	5.0E-13	6.4E-12	0.0E+00	2.4E-13	2.1E-13	2.8E-12	0.0E+00	2.4E-13	5.5E-13	6.6E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table AZ).
[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 57. Comparison of silicate mineral dissolution model statistics (values reported in Tables 52-56)

Period	Model	Plagioclase (mol $m^{-2}s^{-1}$)				Augite (mol $m^{-2}s^{-1}$)				Hypersthene (mol $m^{-2}s^{-1}$)				Olivine (mol $m^{-2}s^{-1}$)				Biotite (mol $m^{-2}s^{-1}$)			
		Ave	S.D.	Max	Min[1]	Ave	S.D.	Max	Min[1]	Ave	S.D.	Max	Min[1]	Ave	S.D.	Max	Min[1]	Ave	S.D.	Max	Min[1]
A	1	5.7E-12	3.1E-12	4.1E-11	5.4E-13					5.5E-12	3.0E-12	4.0E-11	1.8E-13	1.7E-11	9.6E-12	2.0E-10	9.7E-14	9.3E-12	5.2E-12	6.2E-11	5.0E-13
	2	2.8E-12	2.3E-12	2.3E-11	0.0E+00	9.5E-12	5.2E-12	5.0E-11	5.8E-13	6.7E-13	1.4E-12	2.4E-11	0.0E+00	2.5E-12	4.5E-12	1.2E-10	0.0E+00	1.2E-12	2.3E-12	4.2E-11	0.0E+00
	3	6.2E-13	1.2E-13	9.6E-13	1.8E-13	8.8E-12	3.2E-12	4.2E-11	1.1E-12	1.3E-12	1.9E-12	9.3E-12	0.0E+00	8.4E-13	1.3E-12	7.4E-12	0.0E+00	1.3E-13	2.3E-13	1.2E-12	0.0E+00
	4									1.3E-12	1.9E-12	9.3E-12	0.0E+00	4.3E-13	6.1E-13	4.1E-12	0.0E+00	1.5E-11	5.9E-12	4.7E-11	1.8E-12
	max /min	9.3	24.9	43.3	N/A	1.1	1.7	1.2	1.8	8.3	2.2	4.3	N/A	40.2	15.9	49.0	N/A	116.0	25.8	52.1	N/A
B	1	1.5E-12	5.9E-13	9.3E-12	2.1E-14					3.4E-12	9.8E-13	2.9E-11	3.3E-14	9.1E-12	2.6E-12	5.4E-11	2.4E-14	5.6E-12	1.6E-12	2.9E-11	5.4E-14
	2	3.4E-13	2.9E-13	6.1E-12	0.0E+00	3.5E-12	1.5E-12	2.2E-11	6.5E-14	1.7E-12	7.7E-13	2.1E-11	0.0E+00	4.1E-12	2.3E-12	2.9E-11	0.0E+00	2.8E-12	1.3E-12	2.0E-11	0.0E+00
	3	3.8E-13	1.7E-13	1.5E-12	1.1E-13	1.9E-12	1.3E-12	1.2E-11	0.0E+00	2.9E-12	9.2E-13	2.0E-11	0.0E+00	1.1E-12	8.4E-13	1.2E-11	0.0E+00	3.2E-12	6.2E-13	1.9E-11	0.0E+00
	4									6.9E-13	5.5E-13	8.1E-12	0.0E+00	3.1E-13	2.8E-13	3.6E-12	0.0E+00	1.0E-11	4.2E-12	5.3E-11	1.7E-13
	max /min	4.3	3.4	6.3	N/A	1.8	1.2	1.8	N/A	5.0	1.8	3.6	N/A	29.7	9.3	15.1	N/A	3.7	6.9	2.7	N/A
C	1	1.5E-12	5.9E-13	9.3E-12	2.1E-14	3.3E-12	2.9E-12	4.8E-11	0.0E+00	4.4E-12	2.1E-12	3.1E-11	0.0E+00	1.6E-11	6.7E-12	6.8E-11	0.0E+00	6.0E-12	3.0E-12	3.3E-11	0.0E+00
	2	3.3E-13	3.1E-13	4.1E-12	0.0E+00	6.7E-12	3.0E-12	5.9E-11	2.4E-13	2.8E-12	1.9E-12	2.7E-11	0.0E+00	9.6E-12	6.1E-12	6.0E-11	0.0E+00	3.4E-12	2.6E-12	3.1E-11	0.0E+00
	3	9.2E-13	2.9E-13	2.9E-12	4.3E-13	4.0E-12	2.0E-12	2.4E-11	0.0E+00	4.0E-12	1.8E-12	2.5E-11	0.0E+00	1.5E-11	5.3E-12	5.4E-11	0.0E+00	4.1E-12	1.7E-12	2.2E-11	0.0E+00
	4									2.7E-12	1.3E-12	2.5E-11	0.0E+00	7.0E-12	3.4E-12	5.4E-11	0.0E+00	4.7E-12	2.1E-12	2.4E-11	3.2E-13
	max /min	4.4	2.0	3.2	N/A	2.0	1.5	2.5	N/A	1.6	1.5	1.2	N/A	2.4	2.0	1.2	N/A	1.8	1.8	1.5	N/A
D	1	1.3E-12	3.6E-13	4.6E-12	6.7E-14					1.1E-12	5.8E-13	7.0E-12	8.8E-14	5.0E-12	2.1E-12	3.2E-11	5.2E-14	2.0E-12	9.9E-13	1.2E-11	7.7E-14
	2	4.1E-13	2.6E-13	2.7E-12	0.0E+00	2.3E-12	9.5E-13	1.4E-11	6.1E-14	1.3E-13	2.1E-13	3.2E-12	0.0E+00	7.1E-13	7.9E-13	9.0E-12	0.0E+00	2.6E-13	3.3E-13	4.2E-12	0.0E+00
	3	6.4E-13	2.2E-13	2.9E-12	1.4E-13	1.3E-12	5.0E-13	6.4E-12	0.0E+00	5.8E-13	4.2E-13	2.8E-12	0.0E+00	2.4E-12	1.6E-12	2.1E-11	0.0E+00	9.0E-13	9.7E-13	1.4E-11	0.0E+00
	4									2.4E-13	2.1E-13	2.8E-12	0.0E+00	2.4E-13	5.5E-13	6.6E-12	0.0E+00	4.4E-12	3.6E-12	4.1E-11	8.8E-14
	max /min	3.3	1.7	1.7	N/A	1.8	1.9	2.1	N/A	8.6	2.8	2.5	N/A	21.2	3.9	4.9	N/A	17.1	10.9	9.8	N/A

[1] No range in minimum values reported due to zero values calculated

Note: Plagioclase dissolution calculations in Model 1, Period C are the same as Period B.

Note: Plagioclase and augite dissolution calculations in Model 4 are the same as Model 3.

Table 58. Total number of calculated mineral dissolution rates for each model during the specified rate period and the total number of cases where the models calculated an absence of dissolution (zero value). Associated percentage is the comparison of zero values cases to total number of cases.

Sorted by model, than period

Model	Period	Plagioclase			Augite			Hypersthene			Olivine			Biotite		
		Total n	zero	%	Total n	zero	%	Total n	zero	%	Total n	zero	%	Total n	zero	%
1	A	301	0	0%				296	0	0%	259	0	0%	220	0	0%
	B	1352	0	0%				1345	0	0%	1180	0	0%	985	0	0%
	C				493	93	19%	491	22	4%	430	13	3%	370	21	6%
	D	561	0	0%				560	0	0%	551	0	0%	412	0	0%
2	A	280	56	20%	301	0	0%	280	224	80%	246	193	78%	207	162	78%
	B	1332	528	40%	1352	0	0%	1332	804	60%	1168	649	56%	975	607	62%
	C	491	368	75%	493	0	0%	491	123	25%	430	62	14%	370	117	32%
	D	559	91	16%	561	0	0%	559	468	84%	550	459	83%	412	351	85%
3	A	17	0	0%	17	0	0%	17	11	65%	17	11	65%	10	9	90%
	B	208	0	0%	208	34	16%	206	77	37%	187	60	32%	137	70	51%
	C	41	0	0%	41	1	2%	41	9	22%	32	1	3%	28	9	32%
	D	152	0	0%	152	11	7%	152	72	47%	149	69	46%	110	55	50%
4	A	17	0	0%	17	0	0%	17	12	71%	17	12	71%	10	0	0%
	B	208	0	0%	208	34	16%	196	129	66%	179	112	63%	131	0	0%
	C	41	0	0%	41	1	2%	41	22	54%	32	13	41%	28	0	0%
	D	152	0	0%	152	11	7%	144	112	78%	142	110	77%	102	0	0%

Sorted by period, than model

Period	Model	Plagioclase			Augite			Hypersthene			Olivine			Biotite		
		Total n	zero	%	Total n	zero	%	Total n	zero	%	Total n	zero	%	Total n	zero	%
A	1	301	0	0%				296	0	0%	259	0	0%	220	0	0%
	2	280	56	20%	301	0	0%	280	224	80%	246	193	78%	207	162	78%
	3	17	0	0%	17	0	0%	17	11	65%	17	11	65%	10	9	90%
	4	17	0	0%	17	0	0%	17	12	71%	17	12	71%	10	0	0%
B	1	1352	0	0%				1345	0	0%	1180	0	0%	985	0	0%
	2	1332	528	40%	1352	0	0%	1332	804	60%	1168	649	56%	975	607	62%
	3	208	0	0%	208	34	16%	206	77	37%	187	60	32%	137	70	51%
	4	208	0	0%	208	34	16%	196	129	66%	179	112	63%	131	0	0%
C	1				493	93	19%	491	22	4%	430	13	3%	370	21	6%
	2	491	368	75%	493	0	0%	491	123	25%	430	62	14%	370	117	32%
	3	41	0	0%	41	1	2%	41	9	22%	32	1	3%	28	9	32%
	4	41	0	0%	41	1	2%	41	22	54%	32	13	41%	28	0	0%
D	1	561	0	0%				560	0	0%	551	0	0%	412	0	0%
	2	559	91	16%	561	0	0%	559	468	84%	550	459	83%	412	351	85%
	3	152	0	0%	152	11	7%	152	72	47%	149	69	46%	110	55	50%
	4	152	0	0%	152	11	7%	144	112	78%	142	110	77%	102	0	0%

Table 59. Comparison of Model 1 and 4 biotite dissolution rates and ratio of K/Mg release rates. Instances where the calculated biotite dissolution rate was higher in model 1 than model 4 typically correspond to lower K/Mg release ratio (highlighted values).

R#	%S	Period A			Period B			Period C			Period D		
		Model 1	Model 4	M4/M1	Model 1	Model 4	M4/M1	Model 1	Model 4	M4/M1	Model 1	Model 4	M4/M1
1	0.18	6.0E-12			1.7E-12	6.0E-12	3.58						
3	0.22	6.1E-12			1.4E-12	2.3E-12	1.68						
5	0.40	3.6E-12			9.1E-13	5.6E-12	6.22	3.9E-13	1.2E-12	3.10	4.2E-13	4.0E-13	0.95
7	0.41	2.3E-12			1.2E-12			2.1E-12	4.5E-12	2.10	5.4E-13	2.8E-12	5.10
9	0.51	2.7E-12			1.5E-12			1.3E-12			6.4E-13	2.3E-12	3.69
15	0.58	3.2E-11			1.7E-11			1.4E-11			2.7E-12	1.1E-11	4.20
17	0.71	9.4E-12			4.1E-12			9.7E-12	5.9E-12	0.61	1.3E-12	7.4E-12	5.49
33	1.44	7.3E-12			6.3E-12			1.1E-11	1.0E-11	0.91	1.8E-12	1.4E-12	0.79
19	1.63	1.9E-11			1.1E-11			1.1E-11	7.1E-12	0.62	2.5E-12	6.9E-12	2.73

R#	%S	Period A			Period B			Period C			Period D		
		Mg	K	K/Mg	Mg	K	K/Mg	Mg	K	K/Mg	Mg	K	K/Mg
1	0.18	8.2E-12			2.3E-12	5.7E-12	2.52						
3	0.22	1.6E-11			3.5E-12	3.3E-12	0.92						
5	0.4	5.0E-12			1.3E-12	4.6E-12	3.71	1.1E-12	1.1E-12	0.98	5.9E-13	3.7E-13	0.63
7	0.41	5.4E-12			2.9E-12			7.2E-12	6.3E-12	0.87	1.3E-12	3.7E-12	2.86
9	0.51	6.1E-12			3.5E-12			7.1E-12			1.5E-12	3.2E-12	2.17
15	0.58	1.6E-11			8.5E-12			8.5E-12			1.3E-12	4.9E-12	3.65
17	0.71	1.6E-11			6.9E-12			2.1E-11	5.0E-12	0.24	2.2E-12	6.0E-12	2.70
33	1.44	1.3E-11			1.1E-11			2.9E-11	9.5E-12	0.33	3.2E-12	1.5E-12	0.46
19	1.63	3.3E-11			1.9E-11			2.4E-11	6.5E-12	0.27	4.3E-12	6.2E-12	1.45

Table 60. Model 1, Sample Group Average Silicate Mineral Dissolution Rates.

Model 1, Average rates by sample group																						
Period	Group	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
A	I	9	3.2E-12	1.6E-12	6.5E-12	1.8E-12					5.6E-12	5.0E-12	1.7E-11	1.1E-12	1.1E-11	1.0E-11	3.5E-11	2.3E-12	6.1E-12	4.4E-12	1.5E-11	1.1E-12
	II	15	4.9E-12	2.5E-12	1.1E-11	1.8E-12					5.4E-12	2.5E-12	1.0E-11	1.5E-12	1.2E-11	5.8E-12	2.1E-11	3.3E-12	8.9E-12	5.1E-12	1.9E-11	1.8E-12
	III	16	8.2E-12	4.8E-12	1.8E-11	2.3E-12					5.7E-12	3.0E-12	1.3E-11	1.6E-12	2.7E-11	1.4E-11	6.2E-11	7.9E-12	1.1E-11	5.6E-12	2.5E-11	4.4E-12
B	I	165	6.2E-13	3.0E-13	2.2E-12	7.3E-14					1.5E-12	1.1E-12	6.1E-12	3.6E-13	2.8E-12	2.1E-12	1.1E-11	5.2E-13	1.5E-12	1.2E-12	6.1E-12	3.1E-13
	II	58	1.3E-12	4.6E-13	2.7E-12	4.9E-13					3.7E-12	9.7E-13	6.3E-12	1.8E-12	5.4E-12	1.9E-12	1.0E-11	1.2E-12	5.9E-12	1.6E-12	9.9E-12	2.5E-12
	III	31	2.1E-12	9.2E-13	4.7E-12	1.1E-12					3.8E-12	9.6E-13	6.3E-12	1.9E-12	1.7E-11	3.7E-12	2.5E-11	9.1E-12	6.7E-12	1.7E-12	1.1E-11	3.7E-12
C	I		6.2E-13	3.0E-13	2.2E-12	7.3E-14																
	II	23	1.3E-12	4.6E-13	2.7E-12	4.9E-13	1.0E-12	8.3E-13	3.2E-12	1.1E-13	2.5E-12	1.3E-12	5.5E-12	8.2E-13	7.7E-12	3.6E-12	1.5E-11	2.3E-12	3.3E-12	1.7E-12	7.1E-12	8.3E-13
	III	44	2.1E-12	9.2E-13	4.7E-12	1.1E-12	6.0E-12	5.3E-12	1.8E-11	3.8E-14	6.6E-12	3.0E-12	1.3E-11	1.3E-12	2.4E-11	9.5E-12	4.4E-11	6.1E-12	9.3E-12	4.5E-12	2.0E-11	1.9E-12
D	I																					
	II	51	1.0E-12	3.6E-13	2.1E-12	5.3E-13					8.4E-13	5.5E-13	2.4E-12	1.9E-13	1.7E-12	1.5E-12	7.0E-12	6.2E-13	9.0E-13	8.0E-13	3.7E-12	4.2E-13
	III	35	1.7E-12	3.6E-13	2.6E-12	1.3E-12					1.5E-12	6.1E-13	3.1E-12	8.9E-13	8.4E-12	2.7E-12	1.5E-11	5.3E-12	3.1E-12	1.2E-12	6.0E-12	1.8E-12

Table 61. Model 2, Sample Group Average Silicate Mineral Dissolution Rates.

Model 2, Average rates by sample group																						
Period	Group	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
A	I	9	6.3E-12	3.6E-12	1.2E-11	1.4E-12	1.5E-12	1.3E-12	4.1E-12	1.7E-13	1.2E-12	3.0E-12	8.1E-12	0.0E+00	2.8E-12	7.0E-12	1.9E-11	0.0E+00	1.0E-12	2.5E-12	6.8E-12	0.0E+00
	II	15	8.5E-12	4.7E-12	1.7E-11	1.7E-12	2.6E-12	2.1E-12	6.7E-12	2.1E-13	4.3E-13	9.5E-13	3.3E-12	0.0E+00	5.1E-13	1.3E-12	3.8E-12	0.0E+00	5.1E-13	1.5E-12	5.4E-12	0.0E+00
	III	16	1.2E-11	6.7E-12	2.8E-11	4.3E-12	3.8E-12	3.1E-12	1.0E-11	5.3E-15	8.2E-13	1.3E-12	4.3E-12	0.0E+00	5.0E-12	7.5E-12	2.6E-11	0.0E+00	2.4E-12	3.5E-12	1.2E-11	0.0E+00
B	I	165	1.9E-12	1.2E-12	6.4E-12	3.9E-13	1.9E-13	2.4E-13	1.8E-12	0.0E+00	3.0E-13	6.0E-13	3.8E-12	0.0E+00	6.9E-13	1.3E-12	8.1E-12	0.0E+00	3.6E-13	7.4E-13	4.3E-12	0.0E+00
	II	58	2.7E-12	1.2E-12	6.3E-12	1.2E-12	5.3E-13	3.3E-13	1.7E-12	9.2E-14	2.0E-12	7.3E-13	3.9E-12	7.6E-13	2.1E-12	1.3E-12	5.2E-12	1.9E-13	3.0E-12	1.0E-12	5.6E-12	1.0E-12
	III	31	5.5E-12	2.2E-12	1.2E-11	2.8E-12	1.0E-13	2.5E-13	1.1E-12	0.0E+00	1.9E-12	9.1E-13	3.7E-12	4.9E-14	8.3E-12	4.0E-12	1.5E-11	3.0E-13	3.5E-12	1.8E-12	6.7E-12	1.5E-13
C	I																					
	II	23	2.6E-12	1.0E-12	5.0E-12	1.0E-12	5.4E-13	4.1E-13	1.5E-12	1.1E-13	1.2E-12	1.1E-12	4.0E-12	0.0E+00	2.7E-12	2.6E-12	9.5E-12	0.0E+00	1.2E-12	1.3E-12	4.6E-12	0.0E+00
	III	44	1.1E-11	5.3E-12	2.3E-11	4.0E-12	9.4E-14	1.9E-13	9.8E-13	0.0E+00	4.7E-12	2.9E-12	1.1E-11	1.2E-13	1.6E-11	9.1E-12	3.5E-11	6.8E-13	6.0E-12	4.3E-12	1.6E-11	1.8E-13
D	I																					
	II	51	9.0E-13	5.6E-13	2.8E-12	4.3E-13	5.6E-13	3.4E-13	1.6E-12	4.3E-14	4.8E-14	1.5E-13	8.1E-13	0.0E+00	5.2E-14	2.8E-13	2.1E-12	0.0E+00	2.0E-14	1.4E-13	1.2E-12	0.0E+00
	III	35	4.0E-12	1.4E-12	7.6E-12	2.1E-12	2.3E-13	1.7E-13	6.2E-13	0.0E+00	2.3E-13	2.6E-13	1.0E-12	4.4E-14	1.4E-12	1.3E-12	4.3E-12	3.1E-13	5.0E-13	5.1E-13	1.9E-12	1.4E-13

Table 62. Model 3, Sample Group Average Silicate Mineral Dissolution Rates.

Model 3, Average rates by sample group																						
Period	Group	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
A	I	2	7.5E-13	8.7E-14	8.1E-13	6.9E-13	7.2E-12	1.0E-12	7.9E-12	6.5E-12	4.8E-13	3.0E-13	6.9E-13	2.7E-13	1.3E-13	7.9E-14	1.8E-13	7.1E-14				
	II	3	5.9E-13	1.3E-13	7.2E-13	4.6E-13	9.1E-12	3.6E-12	1.3E-11	6.4E-12	1.5E-12	2.3E-12	3.5E-12	2.0E-14	9.9E-13	1.6E-12	2.5E-12	8.6E-15	1.3E-13	2.3E-13	4.0E-13	0.0E+00
	III																					
B	I	33	2.9E-13	1.3E-13	5.6E-13	1.4E-13	9.4E-13	7.2E-13	2.8E-12	6.9E-14	7.5E-13	8.7E-13	3.9E-12	0.0E+00	9.8E-13	1.4E-12	6.4E-12	0.0E+00	5.0E-13	8.5E-13	4.0E-12	0.0E+00
	II	18	4.2E-13	2.0E-13	8.9E-13	2.6E-13	2.4E-12	1.5E-12	4.6E-12	1.2E-12	3.9E-12	9.4E-13	5.2E-12	2.9E-12	1.2E-12	4.9E-13	1.9E-12	7.1E-13	4.6E-12	5.0E-13	5.2E-12	4.3E-12
	III																					
C	I																					
	II	4	6.1E-13	1.2E-13	7.6E-13	5.0E-13	1.6E-12	6.5E-13	2.7E-12	1.1E-12	1.5E-12	6.1E-13	2.0E-12	8.6E-13	3.2E-12	1.7E-12	4.3E-12	1.1E-12	9.1E-13	6.3E-13	1.4E-12	4.0E-13
	III	3	1.1E-12	4.7E-13	1.5E-12	8.9E-13	5.8E-12	3.4E-12	8.3E-12	3.6E-12	5.7E-12	2.9E-12	8.1E-12	4.1E-12	2.0E-11	7.5E-12	2.6E-11	1.5E-11	6.6E-12	3.1E-12	9.0E-12	4.5E-12
D	I																					
	II	13	5.7E-13	2.7E-13	1.2E-12	3.3E-13	5.9E-13	2.5E-13	1.2E-12	2.9E-13	4.8E-13	4.2E-13	1.4E-12	2.3E-13	8.9E-13	1.7E-12	5.7E-12	1.1E-13	4.5E-13	1.0E-12	3.6E-12	0.0E+00
	III	11	7.3E-13	1.5E-13	9.7E-13	5.9E-13	2.2E-12	8.4E-13	3.5E-12	1.2E-12	6.9E-13	4.1E-13	1.5E-12	4.8E-13	3.8E-12	1.5E-12	6.7E-12	3.0E-12	1.3E-12	8.9E-13	3.1E-12	8.9E-13

Table 63. Model 4, Sample Group Average Silicate Mineral Dissolution Rates.

Model 4, Average rates by sample group																						
Period	Group	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
A	I	2									7.2E-12	1.0E-12	7.9E-12	6.5E-12	4.8E-13	3.0E-13	6.9E-13	2.7E-13	1.3E-13	7.9E-14	1.8E-13	7.1E-14
	II	3	1.5E-11	5.9E-12	2.2E-11	1.0E-11	2.5E-10	1.7E-10	4.0E-10	1.2E-10	9.1E-12	3.6E-12	1.3E-11	6.4E-12	1.4E-12	2.2E-12	3.4E-12	2.0E-14	4.9E-13	7.1E-13	1.1E-12	8.6E-15
	III																					
B	I	32	4.1E-12	3.9E-12	1.3E-11	4.5E-13	1.0E-11	1.1E-11	3.8E-11	8.9E-13	9.1E-13	6.9E-13	2.7E-12	6.9E-14	4.1E-13	4.1E-13	1.9E-12	0.0E+00	1.3E-13	2.1E-13	1.1E-12	0.0E+00
	II	17	1.3E-11	4.4E-12	2.0E-11	9.9E-12	1.5E-10	5.9E-11	2.4E-10	1.0E-10	2.4E-12	1.6E-12	4.6E-12	1.2E-12	8.3E-13	6.2E-13	1.6E-12	3.0E-14	4.1E-13	3.2E-13	8.2E-13	1.6E-14
	III																					
C	I																					
	II	4	2.3E-12	9.3E-13	3.1E-12	1.5E-12	4.0E-11	1.7E-11	5.9E-11	2.4E-11	1.6E-12	6.5E-13	2.7E-12	1.1E-12	8.5E-13	2.1E-13	1.0E-12	5.2E-13	6.8E-13	2.3E-13	8.7E-13	3.8E-13
	III	3	6.6E-12	3.7E-12	9.8E-12	4.4E-12	3.1E-11	1.8E-11	4.3E-11	1.9E-11	5.8E-12	3.4E-12	8.3E-12	3.6E-12	4.0E-12	2.5E-12	6.1E-12	2.7E-12	9.6E-12	5.3E-12	1.4E-11	6.7E-12
D	I																					
	II	12	3.4E-12	3.4E-12	1.1E-11	4.4E-13	1.4E-11	1.0E-11	3.6E-11	4.5E-12	5.9E-13	2.5E-13	1.2E-12	2.9E-13	3.8E-13	2.3E-13	8.1E-13	2.3E-13	2.7E-13	4.7E-13	1.8E-12	1.1E-13
	III	10	5.4E-12	3.9E-12	1.2E-11	2.8E-12	2.3E-11	2.4E-11	6.1E-11	6.8E-12	2.1E-12	8.4E-13	3.5E-12	1.2E-12	7.7E-14	1.9E-13	5.0E-13	0.0E+00	2.1E-13	6.4E-13	1.9E-12	0.0E+00

Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table BE).

Table 64. Model 1, Comparison of Silicate Mineral Dissolution Rates from Samples with Differing Mineralogy.

Model 1, Comparison of rates from samples with differing mineralogy																					
Period	%S, Reactor	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
A	0.72, 1.16	3.0E-12	1.4E-12	6.6E-12	1.7E-12					3.5E-12	2.0E-12	7.2E-12	4.8E-13	2.8E-12	1.6E-12	5.5E-12	3.2E-13				
	0.67 (R40), 0.82	4.0E-12	1.5E-12	7.6E-12	1.0E-12					2.2E-12	8.2E-13	3.6E-12	8.1E-13	1.6E-11	5.9E-12	2.5E-11	5.8E-12	2.1E-12	7.8E-13	3.3E-12	7.7E-13
	1.4, 1.64	1.1E-11	7.1E-12	2.8E-11	3.0E-12					9.9E-12	5.4E-12	2.2E-11	1.7E-12	4.9E-11	2.7E-11	1.1E-10	8.8E-12	1.6E-11	9.0E-12	3.8E-11	3.1E-12
B	0.72, 1.16	7.2E-13	4.9E-13	2.7E-12	2.4E-13					2.3E-12	1.1E-12	6.7E-12	1.2E-12	2.2E-12	9.2E-13	5.7E-12	1.3E-12				
	0.67 (R40), 0.82	1.6E-12	4.5E-13	2.9E-12	9.0E-13					5.3E-13	2.2E-13	1.1E-12	1.1E-13	3.4E-12	1.4E-12	7.6E-12	6.9E-13	4.1E-13	1.8E-13	9.8E-13	8.4E-14
	1.4, 1.64	2.4E-12	7.2E-13	4.6E-12	1.4E-12					6.5E-12	1.4E-12	9.2E-12	2.8E-12	2.5E-11	5.8E-12	3.7E-11	1.1E-11	6.4E-12	1.6E-12	1.0E-11	3.1E-12
C	0.72, 1.16	7.2E-13	4.9E-13	2.7E-12	2.4E-13	3.9E-12	3.1E-12	1.1E-11	0.0E+00	6.2E-12	3.1E-12	1.3E-11	0.0E+00	8.0E-12	4.0E-12	1.7E-11	0.0E+00				
	0.67 (R40), 0.82	1.6E-12	4.5E-13	2.9E-12	9.0E-13	6.6E-14	2.0E-13	6.4E-13	0.0E+00	2.1E-12	6.5E-13	2.8E-12	8.5E-13	1.2E-11	3.7E-12	1.6E-11	4.9E-12	1.3E-12	4.1E-13	1.7E-12	5.3E-13
	1.4, 1.64	2.4E-12	7.2E-13	4.6E-12	1.4E-12	2.8E-12	2.0E-12	6.7E-12	1.3E-13	1.2E-11	4.7E-12	2.1E-11	3.7E-12	4.1E-11	1.4E-11	6.7E-11	1.6E-11	8.2E-12	2.1E-12	1.3E-11	4.4E-12
D	0.72, 1.16	8.8E-13	2.2E-13	1.5E-12	5.4E-13					7.5E-13	7.6E-13	3.4E-12	1.8E-13	9.5E-13	9.7E-13	4.3E-12	2.4E-13				
	0.67 (R40), 0.82	8.4E-13	4.6E-13	1.9E-12	1.9E-13					2.8E-13	2.2E-13	1.2E-12	1.2E-13	1.6E-12	1.3E-12	6.8E-12	7.1E-13	1.8E-13	1.4E-13	7.4E-13	7.7E-14
	1.4, 1.64	2.9E-12	4.2E-13	3.6E-12	2.6E-12					4.1E-12	8.6E-13	5.5E-12	3.2E-12	2.4E-11	5.0E-12	3.2E-11	1.8E-11	4.5E-12	9.5E-13	6.0E-12	3.5E-12

Mineralogical breakdown of cations present (%) for specific samples and minerals (see Table U for full mineralogical breakdown)

%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
		Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

Table 65. Model 2, Comparison of Silicate Mineral Dissolution Rates from Samples with Differing Mineralogy.

Model 2, Comparison of rates from samples with differing mineralogy																					
Period	%S, Reactor	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
A	0.72, 1.16	1.6E-12	1.4E-12	5.2E-12	2.7E-13	6.7E-12	3.8E-12	1.4E-11	9.3E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
	0.67 (R40), 0.82	1.7E-12	1.1E-12	4.0E-12	0.0E+00	4.0E-12	1.5E-12	6.5E-12	1.3E-12	3.4E-14	1.2E-13	4.9E-13	0.0E+00	2.7E-13	1.0E-12	4.1E-12	0.0E+00	3.8E-14	1.5E-13	6.0E-13	0.0E+00
	1.4, 1.64	4.4E-12	4.1E-12	1.3E-11	0.0E+00	1.0E-11	6.6E-12	2.5E-11	1.9E-12	7.0E-13	1.3E-12	3.9E-12	0.0E+00	3.7E-12	6.8E-12	2.0E-11	0.0E+00	1.3E-12	2.3E-12	6.6E-12	0.0E+00
B	0.72, 1.16	8.3E-14	8.0E-14	4.4E-13	0.0E+00	2.9E-12	1.9E-12	1.0E-11	9.6E-13	7.9E-13	4.8E-13	2.5E-12	1.1E-14	9.8E-13	4.5E-13	2.3E-12	1.4E-14				
	0.67 (R40), 0.82	1.0E-12	4.1E-13	2.1E-12	5.1E-13	9.3E-13	4.0E-13	2.0E-12	1.9E-13	3.1E-15	9.1E-15	3.1E-14	0.0E+00	3.1E-14	9.2E-14	3.1E-13	0.0E+00	5.1E-15	1.5E-14	5.0E-14	0.0E+00
	1.4, 1.64	8.0E-14	3.9E-13	2.1E-12	0.0E+00	3.5E-12	9.6E-13	5.5E-12	2.2E-12	3.2E-12	1.4E-12	5.9E-12	0.0E+00	1.2E-11	5.5E-12	2.2E-11	0.0E+00	3.0E-12	1.5E-12	5.5E-12	0.0E+00
C	0.72, 1.16	2.5E-14	1.7E-13	1.2E-12	0.0E+00	6.7E-12	3.1E-12	1.4E-11	2.9E-12	4.7E-12	3.1E-12	1.2E-11	0.0E+00	5.9E-12	3.9E-12	1.5E-11	0.0E+00				
	0.67 (R40), 0.82	3.1E-13	4.3E-13	1.1E-12	0.0E+00	3.4E-12	9.1E-13	4.3E-12	1.5E-12	1.7E-13	1.8E-13	5.4E-13	0.0E+00	9.9E-13	1.0E-12	3.1E-12	0.0E+00	1.1E-13	1.1E-13	3.4E-13	0.0E+00
	1.4, 1.64	8.6E-16	5.3E-15	3.3E-14	0.0E+00	6.5E-12	2.1E-12	1.0E-11	3.4E-12	9.0E-12	4.7E-12	1.8E-11	4.0E-13	2.7E-11	1.4E-11	5.4E-11	2.4E-12	4.6E-12	2.1E-12	9.0E-12	8.9E-13
D	0.72, 1.16	4.5E-13	2.5E-13	9.6E-13	0.0E+00	1.1E-12	9.2E-13	4.1E-12	3.3E-13	1.1E-13	3.2E-13	1.8E-12	0.0E+00	1.4E-13	4.1E-13	2.2E-12	0.0E+00				
	0.67 (R40), 0.82	5.6E-13	3.4E-13	1.4E-12	1.4E-14	4.9E-13	3.8E-13	2.0E-12	2.1E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
	1.4, 1.64	1.3E-14	3.0E-14	6.7E-14	0.0E+00	4.1E-12	6.1E-13	5.1E-12	3.6E-12	5.6E-13	4.0E-13	1.0E-12	0.0E+00	3.3E-12	2.3E-12	6.0E-12	0.0E+00	6.2E-13	4.4E-13	1.1E-12	0.0E+00

Mineralogical breakdown of cations present (%) for specific samples and minerals (see Table U for full mineralogical breakdown)

%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
		Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

Table 66. Model 3, Comparison of Silicate Mineral Dissolution Rates from Samples with Differing Mineralogy.

Model 3, Comparison of rates from samples with differing mineralogy																					
Period	%S, Reactor	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
A	0.72, 1.16	7.5E-13	8.7E-14	8.1E-13	6.9E-13	7.2E-12	1.0E-12	7.9E-12	6.5E-12	4.8E-13	3.0E-13	6.9E-13	2.7E-13	1.3E-13	7.9E-14	1.8E-13	7.1E-14				
	0.67 (R40), 0.82	4.9E-13	1.8E-13	6.5E-13	3.0E-13	4.0E-12	9.9E-13	5.0E-12	3.0E-12	1.2E-13	2.1E-13	3.7E-13	0.0E+00	1.2E-12	2.1E-12	3.7E-12	0.0E+00	2.0E-13	3.4E-13	6.0E-13	0.0E+00
	1.4, 1.64																				
B	0.72, 1.16	2.9E-13	1.3E-13	4.7E-13	1.5E-13	1.2E-12	1.3E-12	4.1E-12	0.0E+00	1.2E-12	1.1E-12	4.9E-12	0.0E+00	3.2E-13	2.9E-13	1.3E-12	0.0E+00				
	0.67 (R40), 0.82	3.6E-13	1.4E-13	7.2E-13	1.7E-13	8.5E-13	3.9E-13	1.6E-12	1.6E-13	1.8E-14	2.9E-14	7.3E-14	0.0E+00	1.8E-13	2.9E-13	7.4E-13	0.0E+00	3.0E-14	4.7E-14	1.2E-13	0.0E+00
	1.4, 1.64																				
C	0.72, 1.16	9.1E-13	3.0E-13	1.3E-12	6.8E-13	3.2E-12	2.0E-12	5.5E-12	1.9E-12	6.9E-12	5.8E-12	1.4E-11	2.7E-12	8.8E-12	7.5E-12	1.7E-11	3.4E-12				
	0.67 (R40), 0.82	7.0E-13	7.8E-14	7.9E-13	6.1E-13	2.7E-12	3.7E-13	3.1E-12	2.2E-12	7.6E-13	5.1E-13	1.1E-12	0.0E+00	4.4E-12	2.9E-12	6.1E-12	0.0E+00	4.7E-13	3.2E-13	6.6E-13	0.0E+00
	1.4, 1.64	1.3E-12	2.6E-13	1.4E-12	1.1E-12	4.5E-12	1.5E-12	5.0E-12	3.7E-12	1.2E-11	5.7E-12	1.5E-11	9.6E-12	3.4E-11	1.2E-11	4.1E-11	3.0E-11	5.3E-12		5.3E-12	5.3E-12
D	0.72, 1.16	5.5E-13	1.3E-13	8.5E-13	3.5E-13	5.9E-13	2.9E-13	1.1E-12	3.3E-13	3.4E-13	5.8E-13	1.7E-12	0.0E+00	4.4E-13	7.4E-13	2.2E-12	0.0E+00				
	0.67 (R40), 0.82	5.4E-13	2.1E-13	9.5E-13	1.6E-13	4.7E-13	4.7E-13	1.5E-12	0.0E+00	6.3E-14	1.0E-13	3.1E-13	0.0E+00	3.6E-13	5.9E-13	1.8E-12	0.0E+00	3.9E-14	6.4E-14	1.9E-13	0.0E+00
	1.4, 1.64	1.1E-12	1.2E-13	1.2E-12	1.0E-12	2.3E-12	4.1E-14	2.4E-12	2.3E-12	2.1E-12	5.8E-15	2.1E-12	2.1E-12	1.2E-11	3.4E-14	1.2E-11	1.2E-11	2.3E-12	6.4E-15	2.3E-12	2.3E-12

Mineralogical breakdown of cations present (%) for specific samples and minerals (see Table U for full mineralogical breakdown)

%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
		Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

Table 67. Model 4, Comparison of Silicate Mineral Dissolution Rates from Samples with Differing Mineralogy.

Model 4, Comparison of rates from samples with differing mineralogy																					
Period	%S, Reactor	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
A	0.72, 1.16									7.2E-12	1.0E-12	7.9E-12	6.5E-12	4.8E-13	3.0E-13	6.9E-13	2.7E-13	1.3E-13	7.9E-14	1.8E-13	7.1E-14
	0.67 (R40), 0.82	5.3E-12	3.8E-12	9.2E-12	1.9E-12	1.5E-10	1.3E-10	3.0E-10	6.4E-11	4.0E-12	9.9E-13	5.0E-12	3.0E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
	1.4, 1.64																				
B	0.72, 1.16									1.1E-12	1.2E-12	3.8E-12	0.0E+00	1.2E-12	1.1E-12	4.9E-12	0.0E+00	3.2E-13	2.9E-13	1.3E-12	0.0E+00
	0.67 (R40), 0.82	1.1E-12	1.2E-12	5.9E-12	4.1E-13	5.8E-11	4.3E-11	2.0E-10	2.3E-11	8.4E-13	3.8E-13	1.5E-12	1.6E-13	1.4E-15	3.6E-15	1.4E-14	0.0E+00	1.4E-14	3.6E-14	1.4E-13	0.0E+00
	1.4, 1.64																				
C	0.72, 1.16									3.2E-12	2.0E-12	5.5E-12	1.9E-12	6.9E-12	5.8E-12	1.4E-11	2.7E-12	8.8E-12	7.5E-12	1.7E-11	3.4E-12
	0.67 (R40), 0.82	1.1E-12	4.6E-13	1.5E-12	4.6E-13	3.9E-11	1.7E-11	5.3E-11	1.6E-11	2.7E-12	3.7E-13	3.1E-12	2.2E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
	1.4, 1.64	3.1E-12		3.1E-12	3.1E-12	4.1E-11	3.3E-11	5.4E-11	2.2E-11	4.5E-12	1.5E-12	5.0E-12	3.7E-12	1.0E-11	5.7E-12	1.3E-11	8.2E-12	2.6E-11	1.2E-11	3.3E-11	2.2E-11
D	0.72, 1.16									5.9E-13	2.9E-13	1.1E-12	3.3E-13	3.4E-13	5.8E-13	1.7E-12	0.0E+00	4.4E-13	7.4E-13	2.2E-12	0.0E+00
	0.67 (R40), 0.82	2.6E-13	1.1E-13	4.3E-13	8.8E-14	9.5E-12	3.8E-12	1.5E-11	3.2E-12	4.7E-13	4.7E-13	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
	1.4, 1.64	4.6E-12	7.5E-13	5.1E-12	4.1E-12	1.5E-11	2.4E-12	1.6E-11	1.3E-11	2.3E-12	4.1E-14	2.4E-12	2.3E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00

Mineralogical breakdown of cations present (%) for specific samples and minerals (see Table U for full mineralogical breakdown)

%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
		Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table BI).

Table 68. Potential gibbsite production and neutralization capacity (NC) for experiment until pH minima or initial decline of pH below 4.0.

%S	Reactor	week of P5 pH	5th percentile pH	first week below pH 4	Ca release (mmol)	SO4 release (mmol)	Acidity (mmol H+)	Gibbsite at pH 4 or pH min (mmol)	Gibbsite NC	Gibbsite NC-acidity
Group I, Duluth Complex										
0.18	**1**	754	6.12		2.72	2.06	-1.31	4.08	24.5	25.8
0.22	**3**	523	6.25		2.73	1.82	-1.82	4.09	24.6	26.4
0.72[1]	**42**	615	5.95		1.13	1.02	-0.22	1.70	10.2	10.4
	Average		6.11		2.19	1.63	-1.12	3.29	19.7	20.9
Group II, Duluth Complex										
0.40	**5**	770	4.37	804	8.14	7.39	-1.50	12.2	73.3	74.8
0.41	**7**	374	4.17		1.93	2.73	1.60	2.90	17.4	15.8
0.51	**9**	287	4.1	357	2.75	4.88	4.25	4.13	24.8	20.5
0.54	**11**	276	4.24		4.21	4.66	0.92	6.31	37.9	36.9
0.57	**13**	369	4.27	331	2.40	5.11	5.42	3.60	21.6	16.2
0.58	**15**	293	4.15	330	2.06	6.68	9.24	3.09	18.6	9.33
0.67[1]	**39**	135	4.36		0.44	2.05	3.23	0.66	3.95	0.72
0.67[2]	**40**	411	4.26		2.84	5.99	6.29	4.26	25.6	19.3
0.82[2]	**43**	325	4.78		1.25	2.03	1.58	1.87	11.2	9.63
0.92[1]	**41**	291	5.21		0.80	1.25	0.91	1.20	7.21	6.30
1.71[1]	**44**	115	5.25		1.50	2.69	2.37	2.25	13.5	11.1
	Average		4.47		2.57	4.13	3.12	3.86	23.2	20.1
Group III, Duluth Complex										
0.71	**17**	374	3.88	331	1.73	4.34	5.22	2.60	15.6	10.4
1.12	**35**	264	3.33	194	4.27	8.40	8.27	6.40	38.4	30.1
1.16	**29**	319	3.39	234	1.49	6.38	9.78	2.24	13.5	3.68
1.40	**37**	295	3.16	257	2.37	7.64	10.5	3.56	21.3	10.8
1.44	**33**	274	3.18	234	1.42	8.64	14.4	2.13	12.8	-1.68
1.63	**19**	289	3.43	257	2.40	12.4	19.9	3.59	21.6	1.63
1.64	**31**	268	3.48	235	3.28	11.4	16.3	4.92	29.5	13.3
	Average		3.41		2.42	8.46	12.1	3.63	21.8	9.7
Virginia Formation										
2.06	21	66	4.3		1.47	4.10	5.25	2.21	13.3	8.01
3.12	23	45	3.85	43	2.70	5.82	6.24	4.05	24.3	18.0
3.72	25	63	3.9	60	2.40	5.74	6.69	3.60	21.6	14.9
5.44	27	36	3.65	3	2.03	3.40	2.74	3.05	18.3	15.5
Average			3.93		2.15	4.76	5.23	3.22	19.3	14.1

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 69. Calculated pyrrhotite oxidation rates for all Duluth Complex samples.

R#	%S	Period A				Period B				Period C				Period D			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
Group I																	
1	0.18[1]	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A								
3	0.22	1.25E-08	6.55E-09	3.47E-08	3.55E-09	3.10E-09	2.55E-09	1.12E-08	2.25E-10								
42	0.72[2]	4.94E-11	1.49E-11	7.32E-11	2.21E-11	1.25E-11	8.01E-12	4.47E-11	1.05E-12								
Average		6.28E-09	3.28E-09	3.47E-08	2.21E-11	1.56E-09	1.28E-09	1.12E-08	1.05E-12								
Group II																	
5	0.4	3.78E-10	1.47E-10	8.34E-10	1.50E-10	1.12E-10	4.93E-11	3.51E-10	3.39E-11	5.62E-10	3.34E-10	1.23E-09	1.99E-10	1.30E-10	3.63E-11	2.09E-10	8.92E-11
7	0.41	1.62E-10	1.35E-10	5.12E-10	2.12E-11	1.29E-10	7.49E-11	3.03E-10	4.06E-11	5.08E-10	9.97E-11	6.83E-10	3.41E-10	1.52E-10	1.36E-10	5.90E-10	3.17E-11
9	0.51	2.28E-10	1.42E-10	5.81E-10	3.89E-11	1.04E-10	4.63E-11	1.86E-10	3.76E-11	6.37E-10	1.15E-10	9.56E-10	4.79E-10	2.04E-10	1.51E-10	5.65E-10	4.19E-11
11	0.54	2.37E-10	9.74E-11	4.97E-10	8.59E-11	1.61E-10	8.10E-11	4.02E-10	4.28E-11	2.23E-10	7.50E-11	4.25E-10	1.10E-10				
13	0.57	2.65E-10	7.20E-11	4.00E-10	1.35E-10	1.74E-10	9.00E-11	5.10E-10	2.87E-11	4.46E-10	1.10E-10	7.15E-10	3.17E-10	2.96E-10	4.97E-11	3.66E-10	2.17E-10
15	0.58	2.32E-10	1.89E-10	9.57E-10	2.86E-11	2.48E-10	9.54E-11	5.61E-10	1.27E-10	4.18E-10	8.96E-11	6.65E-10	3.01E-10	1.43E-10	9.46E-11	3.60E-10	3.15E-11
39	0.67[2]	2.10E-10	5.61E-11	2.71E-10	9.06E-11	1.33E-10	2.46E-11	1.73E-10	9.26E-11	1.73E-10	1.92E-11	2.10E-10	1.33E-10	3.58E-11	3.84E-11	1.43E-10	1.23E-12
40	0.67[3]	1.83E-10	6.77E-11	3.81E-10	8.52E-11	9.78E-11	3.35E-11	2.64E-10	5.94E-11	2.98E-10	8.50E-11	3.98E-10	1.57E-10	3.97E-11	3.79E-11	1.49E-10	1.17E-12
43	0.82[3]	1.08E-10	4.95E-11	2.24E-10	5.04E-11	2.91E-11	1.59E-11	8.61E-11	9.15E-13								
41	0.92[2]	5.59E-11	2.00E-11	1.02E-10	3.28E-11	1.64E-11	1.04E-11	5.21E-11	8.39E-13								
44	1.71[2]	8.52E-11	2.04E-11	1.28E-10	6.15E-11	6.58E-11	1.13E-11	8.81E-11	4.28E-11								
Average		1.95E-10	9.06E-11	9.57E-10	2.12E-11	1.15E-10	4.84E-11	5.61E-10	8.39E-13	4.08E-10	1.16E-10	1.23E-09	1.10E-10	1.43E-10	7.76E-11	5.90E-10	1.17E-12
Group III																	
17	0.71	1.36E-10	5.69E-11	2.54E-10	2.06E-11	9.32E-11	3.64E-11	1.94E-10	2.04E-11	5.01E-10	4.33E-10	1.67E-09	9.74E-11	8.17E-11	4.52E-11	2.19E-10	2.84E-11
35	1.12	2.61E-10	1.49E-11	8.73E-10	9.22E-11	2.57E-10	8.01E-12	3.22E-10	1.12E-10	4.86E-10	2.64E-10	9.05E-10	1.39E-10	1.75E-10	1.46E-11	1.86E-10	1.59E-10
29	1.16	1.52E-10	4.95E-11	4.03E-10	7.33E-11	1.12E-10	1.59E-11	2.07E-10	5.12E-11	4.23E-10	2.80E-10	1.07E-09	6.57E-11	4.00E-11	2.10E-11	1.00E-10	1.70E-11
37	1.4	1.40E-10	2.00E-11	2.53E-10	8.21E-11	1.09E-10	1.04E-11	2.74E-10	5.23E-11	5.88E-10	3.02E-10	1.11E-09	1.91E-10				
33	1.44	1.38E-10	1.36E-10	3.92E-10	6.57E-11	1.74E-10	5.57E-11	3.58E-10	8.69E-11	3.91E-10	2.64E-10	9.52E-10	5.15E-11	4.38E-11	1.46E-11	7.51E-11	2.52E-11
19	1.63	1.33E-10	8.07E-11	3.36E-10	4.51E-11	9.33E-11	3.84E-11	1.43E-10	5.32E-11	3.45E-10	2.80E-10	8.14E-10	8.77E-11	4.23E-11	2.10E-11	1.15E-10	9.54E-12
31	1.64	3.34E-10	5.65E-11	9.80E-10	1.65E-10	1.69E-10	4.02E-11	2.46E-10	9.86E-11	3.60E-10	3.02E-10	7.21E-10	1.44E-10	1.61E-10	1.61E-11	1.85E-10	1.43E-10
Average		1.85E-10	5.92E-11	9.80E-10	2.06E-11	1.44E-10	2.93E-11	3.58E-10	2.04E-11	4.42E-10	3.04E-10	1.67E-09	5.15E-11	9.07E-11	2.21E-11	2.19E-10	9.54E-12
Summary[4]		1.91E-10	7.84E-11	9.80E-10	2.06E-11	1.27E-10	4.10E-11	5.61E-10	8.39E-13	4.24E-10	2.04E-10	1.67E-09	5.15E-11	1.19E-10	5.20E-11	5.90E-10	1.17E-12

[1] Reactor 1 calculated to have pyrrhotite content less than zero
[2] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[3] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.
[4] Overall calculations does not include group I due to the low amount of pyrrhotite calculated for reactor 3

Table 70. Calculated sulfide oxidation rates (all sulfide minerals combined) for all Duluth Complex samples.

R#	%S	Period A				Period B				Period C				Period D			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
Group I																	
1	0.18	4.81E-10	2.82E-10	1.21E-09	1.19E-10	9.48E-11	9.23E-11	4.20E-10	7.70E-12								
3	0.22	4.17E-10	2.18E-10	1.16E-09	1.18E-10	1.03E-10	8.51E-11	3.74E-10	7.51E-12								
42	0.72[1]	4.68E-11	1.32E-10	6.93E-11	2.09E-11	1.18E-11	4.44E-11	4.23E-11	9.99E-13								
Average		3.15E-10	2.11E-10	1.21E-09	2.09E-11	7.00E-11	7.39E-11	4.20E-10	9.99E-13								
Group II																	
5	0.4	3.41E-10	1.32E-10	7.52E-10	1.35E-10	1.01E-10	4.44E-11	3.17E-10	3.05E-11	5.07E-10	3.01E-10	1.11E-09	1.79E-10	1.17E-10	3.27E-11	1.89E-10	8.04E-11
7	0.41	1.32E-10	1.10E-10	4.18E-10	1.73E-11	1.06E-10	6.12E-11	2.47E-10	3.32E-11	4.15E-10	8.15E-11	5.58E-10	2.79E-10	1.24E-10	1.11E-10	4.82E-10	2.59E-11
9	0.51	1.74E-10	1.08E-10	4.42E-10	2.96E-11	7.89E-11	3.53E-11	1.41E-10	2.86E-11	4.85E-10	8.78E-11	7.27E-10	3.64E-10	1.55E-10	1.15E-10	4.30E-10	3.19E-11
11	0.54	2.29E-10	9.42E-11	4.81E-10	8.31E-11	1.56E-10	7.83E-11	3.89E-10	4.13E-11	2.16E-10	7.26E-11	4.11E-10	1.06E-10				
13	0.57	2.18E-10	5.93E-11	3.29E-10	1.11E-10	1.43E-10	7.41E-11	4.19E-10	2.36E-11	3.67E-10	9.06E-11	5.88E-10	2.61E-10	2.43E-10	4.09E-11	3.01E-10	1.79E-10
15	0.58	1.92E-10	1.56E-10	7.90E-10	2.36E-11	2.05E-10	7.87E-11	4.63E-10	1.05E-10	3.45E-10	7.39E-11	5.49E-10	2.49E-10	1.18E-10	7.81E-11	2.97E-10	2.60E-11
39	0.67[1]	1.92E-10	5.14E-11	2.48E-10	8.29E-11	1.21E-10	2.26E-11	1.59E-10	8.48E-11	1.58E-10	1.76E-11	1.92E-10	1.22E-10	3.28E-11	3.52E-11	1.31E-10	1.13E-12
40	0.67[2]	1.70E-10	6.27E-11	3.53E-10	7.90E-11	9.06E-11	3.10E-11	2.45E-10	5.51E-11	2.77E-10	7.88E-11	3.69E-10	1.46E-10	3.68E-11	3.51E-11	1.38E-10	1.08E-12
43	0.82[2]	1.02E-10	4.70E-11	2.13E-10	4.79E-11	2.77E-11	1.51E-11	8.18E-11	8.70E-13								
41	0.92[1]	5.26E-11	1.88E-11	9.58E-11	3.09E-11	1.54E-11	9.76E-12	4.90E-11	7.90E-13								
44	1.71[1]	8.28E-11	1.99E-11	1.25E-10	5.97E-11	6.40E-11	1.10E-11	8.57E-11	4.16E-11								
Average		1.71E-10	7.82E-11	7.90E-10	1.73E-11	1.01E-10	4.19E-11	4.63E-10	7.90E-13	3.46E-10	1.01E-10	1.11E-09	1.06E-10	1.18E-10	6.39E-11	4.82E-10	1.08E-12
Group III																	
17	0.71	1.19E-10	5.00E-11	2.23E-10	1.81E-11	8.19E-11	3.20E-11	1.71E-10	1.79E-11	4.41E-10	3.81E-10	1.47E-09	8.56E-11	7.18E-11	3.97E-11	1.93E-10	2.49E-11
35	1.12	2.48E-10	1.29E-10	8.29E-10	8.76E-11	2.45E-10	5.30E-11	3.06E-10	1.07E-10	4.62E-10	2.51E-10	8.60E-10	1.32E-10	1.67E-10	1.39E-11	1.77E-10	1.51E-10
29	1.16	1.42E-10	7.56E-11	3.77E-10	6.86E-11	1.05E-10	3.59E-11	1.93E-10	4.79E-11	3.96E-10	2.63E-10	1.00E-09	6.16E-11	3.75E-11	1.96E-11	9.40E-11	1.59E-11
37	1.4	1.29E-10	5.22E-11	2.33E-10	7.58E-11	1.01E-10	3.71E-11	2.53E-10	4.83E-11	5.42E-10	2.79E-10	1.02E-09	1.76E-10				
33	1.44	1.30E-10	8.35E-11	3.70E-10	6.20E-11	1.64E-10	6.23E-11	3.38E-10	8.20E-11	3.69E-10	2.66E-10	8.98E-10	4.86E-11	4.13E-11	9.70E-12	7.08E-11	2.38E-11
19	1.63	1.26E-10	6.34E-11	3.19E-10	4.28E-11	8.86E-11	2.51E-11	1.35E-10	5.05E-11	3.27E-10	1.47E-10	7.73E-10	8.32E-11	4.01E-11	2.80E-11	1.10E-10	9.06E-12
31	1.64	3.02E-10	1.83E-10	8.86E-10	1.49E-10	1.52E-10	3.90E-11	2.22E-10	8.91E-11	3.26E-10	1.46E-10	6.51E-10	1.30E-10	1.45E-10	1.46E-11	1.67E-10	1.29E-10
Average		1.71E-10	9.09E-11	8.86E-10	1.81E-11	1.34E-10	4.06E-11	3.38E-10	1.79E-11	4.09E-10	2.48E-10	1.47E-09	4.86E-11	8.38E-11	2.09E-11	1.93E-10	9.06E-12
Summary		1.92E-10	1.01E-10	1.21E-09	1.73E-11	1.07E-10	4.61E-11	4.63E-10	7.90E-13	3.76E-10	1.69E-10	1.47E-09	4.86E-11	1.02E-10	4.41E-11	4.82E-10	1.08E-12

[1] AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
[2] Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

Table 71. Sulfur depletion statistics. The first method was based on initial %S minus S lost as determined by sulfate concentration in leachate. The cumulative S depletion for this sulfate-based method (%S depletion (sulfate)) is displayed next to the right-most column (%S Depletion (solids)), which determines S depletion as unleached sample %S minus leached sample %S. Leached solids ran for less time than full experimental period, so are likely to underestimate S loss.

Reactor	%S	Start of experiment until pH<6		Major pH drop and/or major sulfate release starts		Minimum pH and/or max sulfate release		pH increase begins		End of experiment		%S Depletion (sulfate)	% S Depletion (solids)	
		Week	%S remaining	Week	%S remaining	Week	%S remaining	Week	%S remaining	Week	%S remaining		Week	% S depletion
1	0.18	#								1252	45.3	54.7	328	41.1
3	0.22	#								1252	54.6	45.4	328	30.9
5	0.40	400	48.5	720	33.3	810	17.4	825	8.5	909	-3.7	103.7	328	12.8
7	0.41	80	94.0	265	83.9	395	65.9	480	48.5	724	38.8	61.2	328	47.8
9	0.51	70	93.0	250	81.8	355	59.8	404	48.6	724	28.2	71.8	328	54.3
11	0.54	57	90.9	215	71.0	290	59.3	385	42.7	441	37.0	63.0	441	72
13	0.57	75	89.4	257	75.2	340	59.4	390	51.1	441	44.8	55.2	441	66.3
15	0.58	55	91.9	140	84.3	280	60.3	335	49.9	724	26.1	73.9	328	62.2
39	0.67[1]	15	97.6	80	92.2	120	88.6	180	82.2	809	69.5	30.5		(a)
40	0.67[2]	140	88.8	250	83.3	380	67.8	420	60.6	809	50.0	50.0		(a)
17	0.71	50	96.7	275	83.4	350	67.3	375	50.4	724	32.1	67.9	328	55.4
42	0.72[1]	#								809	93.4	6.6		(a)
43	0.82[2]	25	97.6							809	82.8	17.2		(a)
41	0.92[1]	102	96.8							809	91.3	8.7		(a)
35	1.12	*		205	66.2	270	42.6	290	32.3	360	21.3	78.7	360	84.6
29	1.16	35	95.5	140	88.7	280	60.2	315	43.5	643	28.3	71.7	643	6.9
37	1.40	35	96.5	192	85.8	282	66.2	345	34.9	360	30.8	69.2	360	79.1
33	1.44	10	97.6	220	76.9	280	57.2	330	34.2	643	16.0	84.0	247	54
19	1.63	20	96.9	120	90.7	290	59.5	355	38.8	724	24.3	75.7	289	71.9
31	1.64	*		220	72.3	275	60.0	290	55.2	360	46.2	53.8	360	75.1
44	1.71[1]	70	95.4							144	92.0	8.0		(a)
Average			91.7		77.9		59.4		45.4		45.2	54.8		54.3
Std Dev			11.9		14.4		14.9		16.0		26.9	26.9		23.2

* Started at pH<6.0
# pH never <6.0 for sustained period
(a) Leached solids not analyzed to date
1 AMAX and ARIMETCO samples originated from core drilled between 1974 and 1977 and were stored for at least 20 years prior to initiation.
2 Dunka blast hole samples originated from a blast on August 30, 1989 at the Dunka Mine and were stored for 8 years prior to initiation.

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Table 72. Reported rates of pyrrhotite oxidation

Samples	Oxidant	Conditions	Rate (mol m^{-2} s^{-1})		Reference
			Iron-based	Sulfate-based	
Monoclinic	Air	68% r.h.[1], 52 ºC	1.5E-8	6.0E-10	Steger and Desjardins 1978[2]
Monoclinic	Air	62% r.h.[1], 28 ºC 62% r.h.[1], 35 ºC 62% r.h.[1], 43 ºC 62% r.h.[1], 50 ºC 37% r.h.[1], 50 ºC 50% r.h.[1], 50 ºC 55% r.h.[1], 50 ºC 75% r.h.[1], 50 ºC	3.9E-9 5.0E-9 6.3E-9 8.9E-9 3.2E-9 4.4E-9 5.2E-9 1.1E-8	6.5E-10 7.1E-10 7.8E-10 8.4E-10 8.0E-10 9.1E-10 9.1E-10 9.0E-10	Steger 1982[2]
Museum-grade	Air flow reactor	pH= 2-6 10 ºC 22 ºC 33 ºC	 3.1E-9 8.5E-9 3.3E-8		Nicholson and Sharer 1994[3]
Museum-grade samples from N. America	Air flow reactor	Abiotic 22 ºC, pH= 2 35 ºC, pH= 2 35 ºC, pH= 3 35 ºC, pH= 4	 2.28E-9 2.87E-9 3.86E-9 2.51E-9	 8.51E-10	Kwong 1995
Museum-grade samples from N. America	Air flow reactor	Biotic 22 ºC, pH= 2 35 ºC, pH= 2 35 ºC, pH= 4	 6.74E-9 8.76E-9 6.91E-9	 6.97E-9	Kwong 1995
12 samples, various locations from N. America	Air flow reactor	22 ºC, pH= 2	4.0E-9	2.0E-10	Janzen et al. 2000
Dunka blast holes, Duluth Complex	Air	pH=3.5-7.4 22.5-29 ºC r.h.=38-88%		8E-10	Lapakko and Antonson 1994
12 samples, various locations from N. America	Fe^{3+}	25 ºC Fe^{3+}=0.2 mM, pH=2.75 Fe^{3+}=0.2 mM, pH=2. 5 Fe^{3+}=1.0 mM, pH=2. 5	 3.5E-8 3.1E-8 6.8E-8		Janzen et al. 2000
Museum-grade samples from N. America	Fe^{3+}	pH= 2, 22 ºC Fe^{3+} ~10^{-3} M	1.44E-8	2.05E-9	Kwong 1995
12 samples, various locations from N. America	Non-oxidative dissolution	25 ºC, pH= 2.75	5E-10		Janzen et al. 2000
Monoclinic	log k= -8.04, n= -0.597				Palandri and Kharaka 2004[4]
Hexagonal	log k= -6.79, n= -0.090				Palandri and Kharaka 2004[4]
				MNDNR average modeled pyrrhotite oxidation rates	
				1.91E-10	Period A
				1.27E-10	Period B
				4.24E-10	Period C
				1.19E-10	Period D

[1] relative humidity
[2] Mean estimated using and average particle specific surface area from Janzen et al. 2000
[3] Average value of oxidation rates
[4] Log(rate) = log k – n·pH

Table 73. Report rates of labradorite and andesine dissolution and dependences on pH.
Log(rate) = log k – n·pH

Element measured	Log k	n	pH[a]	rate at pH 4[b]	rate at pH 6[b]	Reference
Labradorite						
Si	-7.73	0.5	3.1-5.3	1.9E-10	1.9E-11	Welch and Ullman 1993; calculated using labradorite B sample
Si	-8.86	0.5	1-6	1.4E-11	1.4E-12	Sjöberg 1989
Ca, Mg, Na, K	-8.93	0.5		1.2E-11	1.2E-12	Sverdrup 1990 (rate from Blum and Stillings 1995), 25 C
	-9.33	0.4	2-5.1	1.2E-11	1.9E-12	From Blum and Stillings 1995 (listed as Oxburgh et al. 1994 which has no labradorite rates)
	-7.87	0.63	0.84-6	4.1E-11	2.2E-12	Palandri and Kharaka 2004
	-10.91[c]	0	NR		1.2E-11	Palandri and Kharaka 2004
Andesine						
Si	-9.10	0.5	3.1-5.1	7.9E-12	7.9E-13	Oxburgh et al. 1994
	-9.0	0.5	2-5	1.0E-11	1.0E-12	Blum and Stillings 1995, from figure, p. 307 (rates from Oxburgh et al. 1994 and Casey et al. 1991)
	-8.88	0.54	2-4.11	9.1E-12	7.6E-13	Palandri and Kharaka 2004
	-11.47[c]	0	5.1-7.7		3.4E-12	Palandri and Kharaka 2004

[a] pH range over which experiments were conducted
[b] Rate calculated at specified pH, mol m^{-2} s^{-1}
[c] Neutral mechanism rate
NR: Not reported

Table 74. Reported rates of forsteritic and fayalitic olivine dissolution. Log(rate) = log k – n·pH

Mineral	Element measured	Log k	n	pH [a]	rate at pH 4 [b]	Rate at pH 6 [b]	Reference
Olivine (Fo)		-7.0	0.56	1.8-4.6 (approx)	5.8E-10	4.4E-11	Blum and Lasaga 1988, calculated from data in Chen and Brantley 2000
Olivine (Fo_{91})	Mg	-7.00	0.53	2-5.7	7.6E-10	6.6E-11	Wogelius and Walther 1991, calculated from atmospheric CO_2 data (batch and flow-through)
Olivine (Fo_{91})	Si	-7.10	0.53	2-5.7	6.0E-10	5.2E-11	Wogelius and Walther 1991, calculated from atmospheric CO_2 data (batch and flow-through)
Olivine (Fo_{57})	Si	-7.52	0.50	acid	3.0E-10	3.0E-11	Duro et al. 2005, sample was 65% olivine
Olivine (Fo_{91})	Mg	-5.57	0.24	1.66-3.92	3.0E-07	9.8E-08	Luce et al. 1972 (batch)
Olivine (Fo_{91})	Si	-6.54	0.20	1.66-3.92	4.6E-08	1.8E-08	Luce et al. 1972 (batch)
Olivine (Fo_{100})	Mg	-7.72	0.16	1.98-5.25	4.4E-09	2.1E-09	Golubev et al. 2005, CO2 free
Olivine (Fo_{100})	Si	-7.59	0.19	1.98-5.25	4.5E-09	1.9E-09	Golubev et al. 2005, CO2 free
Olivine (Fo_{92})	Mg, Si	-6.86	0.53	1.8-3.8	1.0E-09	9.1E-11	Rosso and Rimstidt 2000, weighted averaged data at 25C
Olivine (Fo_{91})	Mg	-6.53	0.60	1.03-4.2	1.2E-09	7.4E-11	Pokrovsky and Schott 2000b, 25C, CO2 free
Olivine (Fo_{91})	Si	-6.60	0.51	1.03-4.95	2.3E-09	2.2E-10	Pokrovsky and Schott 2000b, 25C, CO2 free
Olivine (Fo_6)	Si, Fe	10.0	0.69	2-3	2.3E-09	4.9E-10	Wogelius and Walther 1992, rate = $1.1 \times 10^{-10} a_{H+}^{0.69} + 3.2 \times 10^{-14} + 1.2 \times 10^{-16} a_{H+}^{-0.31}$
Olivine (Fo)		-6.85	0.47	1.03-8.71	1.9E-09	2.1E-10	Palandri and Kharaka 2004
Olivine (Fo)		-10.64		9.3-12.06		2.3E-11	Palandri and Kharaka 2004
Olivine (Fa)		-4.80	1	acid [c]	1.6E-09	1.6E-11	Palandri and Kharaka 2004, Sverdrup 1990 is only source
Olivine (Fa)		-12.80		neutral [c]		1.6E-13	Palandri and Kharaka 2004

[a] pH range over which experiments were conducted
[b] Rate calculated at specified pH, mol m^{-2} s^{-1}
[c] range not specified by Palandri and Kharaka

Table 75. Reported rates of bronzite dissolution. Log(rate) = log k – n·pH

Mineral	Element measured	Log k	n	pH [a]	rate at pH 4 [b]	rate at pH 6 [b]	Reference
Bronzite	Si	-10.3	0.26	1-6	4.6E-12	1.4E-12	Schott and Berner 1983
Bronzite	Si	-6.5	0.5	1-4.5	3.2E-9	3.2E-10	Grandstaff 1977
Bronzite	Mg	-5.8	0.5	1-4.5	1.6E-8	1.6E-9	Grandstaff 1977
Bronzite		-8.30	0.65	acid [c]	1.3E-11	6.3E-13	Palandri and Kharaka 2004, from Sverdrup 1990 using two papers above
Bronzite		-11.70		neutral [c]		2.0E-12	Palandri and Kharaka 2004, from Sverdrup 1990 using two papers above

[a] pH range over which experiments were conducted
[b] Rate calculated at specified pH, mol m^{-2} s^{-1}
[c] range not specified by Palandri and Kharaka

Table 76. Reported rates of augite dissolution. Log(rate) = log k – n·pH

Mineral	Element measured	Log k	n	pH [a]	rate at pH 4 [b]	rate at pH 6 [b]	Reference
Augite	Ca, Mg	-7.7	0.7	2.5-6	3.2E-11	1.3E-12	Sverdrup 1990
Augite		-6.82	0.7	acid [a]	2.4E-10	9.5E-12	Palandri and Kharaka 2004, includes Sverdrup 1990
Augite		-11.97		neutral [c]		1.1E-12	Palandri and Kharaka 2004, includes Sverdrup 1990

[a] pH range over which experiments were conducted
[b] Rate calculated at specified pH, mol m^{-2} s^{-1}
[c] range not specified by Palandri and Kharaka, but original papers had pH range of 1-7.5

Table 77. Reported rates of biotite dissolution. Log(rate) = log k – n·pH

Mineral	Element measured	Log k	n	pH [c]	rate at pH 4 [d]	rate at pH 6 [d]	Reference
Biotite	Weighted avg of several elements	-9.5[a] -10.7[b]	0.61[a] 0.35[b]	1-4	1.1E-12 7.9E-13	6.9E-14 1.6E-13	Kalinowski and Schweda 1996
Biotite		-9.84	0.53	acid [e]	1.1E-12	9.5E-14	Palandri and Kharaka 2004
Biotite		-12.55		neutral [e]		2.8E-13	Palandri and Kharaka 2004

[a] Normalized by initial surface area
[b] Normalized by final surface area
[c] pH range over which experiments were conducted
[d] Rate calculated at specified pH, mol m^{-2} s^{-1}
[e] range not specified by Palandri and Kharaka

Table 78. Comparison of modeled silicate dissolution rates and Palandri and Kharaka rates (mol m^{-2} s^{-1}).

MNDNR modeled dissolution rates							Palandri & Kharaka 2004 rates				P&K / MNDNR	
Mineral	Model	Periods	Avg pH	rate at pH 4	rate at pH 6	rates at pH 4 / pH6	Mineral	rate at pH 4	rate at pH 6	rates at pH 4 / pH6	pH 4	pH 6
Plagioclase	1	A, D	6.15, 4.72	1.3E-12	5.7E-12	0.23	Labradorite	5.3E-11	1.5E-11	3.6	41	2.6
	2	A, C	6.15, 4.07	3.3E-13	2.8E-12	0.12					161	5.2
	3, 4	A, C	6.32, 4.11	9.2E-13	6.2E-13	1.5					58	23
	1	A, D	6.15, 4.72	1.3E-12	5.7E-12	0.23	Andesine	1.3E-11	4.1E-12	3.0	9.6	0.73
	2	A, C	6.15, 4.07	3.3E-13	2.8E-12	0.12					38	1.5
	3, 4	A, C	6.32, 4.11	9.2E-13	6.2E-13	1.5					14	6.7
Augite	1	C	4.07	3.3E-12			Augite	2.4E-10	1.1E-11	23	73	
	2	A, C	6.15, 4.07	6.7E-12	9.5E-12	0.71					36	1.1
	3, 4	A, C	6.32, 4.11	4E-12	8.8E-12	0.45					60	1.2
Olivine	1	A, C	6.15, 4.07	1.6E-11	1.7E-11	0.94	Forsterite	1.9E-09	2.4E-10	8.0	118	14
	2	A, C	6.15, 4.07	9.6E-12	2.5E-12	3.8					196	95
	3	A, C	6.32, 4.11	1.5E-11	8.4E-13	17.86					126	282
	4	A, C	6.32, 4.11	7E-12	1.4E-12	5.0					269	169
	1	A, C	6.15, 4.07	1.6E-11	1.7E-11	0.94	Fayalite	1.6E-09	1.6E-11	99	99	0.94
	2	A, C	6.15, 4.07	9.6E-12	2.5E-12	3.8					165	6.4
	3	A, C	6.32, 4.11	1.5E-11	8.4E-13	17.86					106	19
	4	A, C	6.32, 4.11	7E-12	1.4E-12	5.0					226	11
Hypersthene	1	A, C	6.15, 4.07	4.4E-12	5.5E-12	0.80	Bronzite	1.5E-11	2.6E-12	5.6	3.3	0.48
	2	A, C	6.15, 4.07	2.8E-12	6.7E-13	4.2					5.2	3.9
	3	A, C	6.32, 4.11	4E-12	1.3E-12	3.1					3.6	2.0
	4	A, C	6.32, 4.11	2.7E-12	3.5E-12	0.77					5.4	0.75
Biotite	1	A, C	6.15, 4.07	6E-12	9.3E-12	0.65	Biotite	1.4E-12	3.8E-13	3.7	0.23	0.041
	2	A, C	6.15, 4.07	3.4E-12	1.2E-12	2.8					0.41	0.31
	3	A, C	6.32, 4.11	4.1E-12	1.3E-13	31.54					0.34	2.9
	4	A, C	6.32, 4.11	4.7E-12	1.5E-11	0.31					0.29	0.025
K-spar	4	A, C	6.32, 4.11	3.44E-11	2.45E-10	0.14	K-spar	1.3E-12	4.8E-13	2.6	0.037	0.0019

Table 79. Minimum drainage pH, sulfate and major cation cumulative mass release (mmol) over the entire period of record (excludes R39-44).

%S	Reactor	Period of Record (weeks)	Minimum pH	SO_4	Ca	Mg	Na	K
Group I, 2 samples								
0.18	**1**	1252	5.67	2.304	3.375	2.021	1.598	5.689
0.22	**3**	1252	5.91	2.337	3.940	3.282	1.223	3.450
Average			5.79	2.321	3.657	2.652	1.410	4.569
Group II, 6 samples								
0.4	**5**	909	3.78	9.701	8.735	0.930	0.477	2.987
0.41	**7**	724	4.05	5.872	2.713	1.482	0.930	2.003
0.51	**9**	724	3.95	8.562	3.798	1.713	1.126	1.383
0.54	**11**	441	4.08	7.953	5.120	1.564	0.480	0.845
0.57	**13**	441	3.86	7.359	2.877	2.601	0.717	1.353
0.58	**15**	724	3.98	10.030	3.131	2.445	1.055	2.694
Average			3.95	8.246	4.396	1.789	0.798	1.877
Group III, Duluth Complex, 7 samples								
0.71	**17**	724	3.53	11.282	3.019	4.059	1.163	2.735
1.12	**35**	360	3.23	20.618	6.057	2.550	0.861	1.609
1.16	**29**	643	3.24	19.443	3.475	4.754	1.577	2.760
1.4	**37**	360	3.06	22.652	3.591	5.850	1.305	4.624
1.44	**33**	643	3.04	28.302	3.615	5.998	1.946	5.162
1.63	**19**	724	3.32	28.860	4.745	6.943	1.349	3.022
1.64	**31**	360	3.38	20.624	4.427	4.694	0.932	0.957
Average			3.26	21.683	4.133	4.978	1.305	2.981
Virginia Formation, 4 samples								
2.06	21	78	4.20	5.471	1.629	2.419	N/A	N/A
3.12	23	78	3.70	14.506	3.688	3.679	N/A	N/A
3.72	25	78	3.70	8.779	2.851	3.234	N/A	N/A
5.44	27	78	3.35	13.262	3.104	4.742	N/A	N/A
Average			3.74	10.504	2.818	3.519	N/A	N/A

Table 80. Sample pH by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	6.90	7.90	6.20	134	6.42	7.18	5.67	565								
0.22	**3**	6.91	7.75	6.40	132	6.51	7.05	5.91	560								
Average		6.91	7.83	6.30	133	6.47	7.12	5.79	563								
Group II																	
0.4	**5**	6.60	7.40	6.16	139	6.13	6.69	4.58	453	4.21	4.53	3.78	45	4.67	4.98	4.50	21
0.41	**7**	6.51	8.35	5.34	128	5.13	6.13	4.47	222	4.22	4.56	4.05	87	4.75	5.82	4.26	127
0.51	**9**	6.30	8.70	5.08	139	5.07	5.45	4.61	137	4.18	4.51	3.95	105	4.64	6.04	4.07	182
0.54	**11**	6.13	8.05	5.13	136	4.98	5.55	4.24	141	4.39	4.77	4.08	162				
0.57	**13**	6.44	7.75	5.50	101	5.21	5.72	4.32	190	4.29	4.50	3.86	57	4.40	4.81	4.21	89
0.58	**15**	5.89	7.55	5.20	100	4.82	5.49	4.20	116	4.22	4.38	3.98	118	4.61	5.73	4.11	229
Average		6.31	7.97	5.40	124	5.22	5.84	4.40	210	4.25	4.54	3.95	96	4.61	5.48	4.23	130
Group III																	
0.71	**17**	6.15	7.65	5.11	100	5.08	5.45	4.41	173	4.22	4.95	3.53	151	4.90	5.43	4.52	140
1.12	**35**	4.87	5.85	4.48	125	4.13	4.52	3.81	87	3.61	4.17	3.23	133	4.09	4.18	3.98	12
1.16	**29**	6.38	6.95	5.69	37	4.62	5.70	4.14	150	3.93	4.78	3.24	192	4.95	5.64	4.50	106
1.4	**37**	5.92	7.30	4.78	73	4.81	5.18	4.01	134	3.66	4.49	3.06	150				
1.44	**33**	5.23	7.20	4.59	74	4.42	4.83	4.19	135	3.89	4.90	3.04	229	4.75	5.09	4.52	46
1.63	**19**	5.73	7.50	4.95	66	4.90	5.35	4.37	88	3.89	4.72	3.32	254	4.77	5.71	4.35	156
1.64	**31**	4.80	5.45	4.51	63	4.39	4.80	4.02	163	3.72	4.19	3.38	111	4.03	4.14	3.85	20
Average		5.58	6.84	4.87	77	4.62	5.12	4.14	133	3.84	4.60	3.26	174	4.58	5.03	4.29	80

Table 81. Sample weekly sulfate release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	1.2E-11	3.0E-11	2.9E-12	59	2.3E-12	1.0E-11	1.9E-13	185								
0.22	**3**	1.0E-11	2.9E-11	3.0E-12	57	2.6E-12	9.3E-12	1.9E-13	187								
Average		1.1E-11	2.9E-11	2.9E-12	58	2.5E-12	9.9E-12	1.9E-13	186								
Group II																	
0.4	**5**	3.3E-11	7.4E-11	1.3E-11	61	9.9E-12	3.1E-11	3.0E-12	132	5.0E-11	1.1E-10	1.8E-11	21	1.2E-11	1.9E-11	7.9E-12	10
0.41	**7**	1.2E-11	3.8E-11	1.6E-12	58	9.5E-12	2.2E-11	3.0E-12	52	3.7E-11	5.0E-11	2.5E-11	21	1.1E-11	4.3E-11	2.3E-12	54
0.51	**9**	1.8E-11	4.7E-11	3.1E-12	61	8.3E-12	1.5E-11	3.0E-12	34	5.1E-11	7.7E-11	3.8E-11	25	1.6E-11	4.5E-11	3.4E-12	67
0.54	**11**	3.3E-11	7.0E-11	1.2E-11	60	2.3E-11	5.6E-11	6.0E-12	36	3.1E-11	5.9E-11	1.5E-11	38				
0.57	**13**	2.8E-11	4.2E-11	1.4E-11	50	1.8E-11	5.3E-11	3.0E-12	48	4.6E-11	7.4E-11	3.3E-11	13	3.1E-11	3.8E-11	2.3E-11	22
0.58	**15**	2.5E-11	1.0E-10	3.0E-12	49	2.6E-11	6.0E-11	1.3E-11	31	4.5E-11	7.1E-11	3.2E-11	27	1.5E-11	3.8E-11	3.4E-12	78
Average		2.5E-11	6.2E-11	7.8E-12	57	1.6E-11	4.0E-11	5.2E-12	56	4.3E-11	7.3E-11	2.7E-11	24	1.7E-11	3.7E-11	7.9E-12	46
Group III																	
0.71	**17**	2.0E-11	3.8E-11	3.1E-12	48	1.4E-11	2.9E-11	3.0E-12	45	7.4E-11	2.5E-10	1.4E-11	35	1.2E-11	3.3E-11	4.2E-12	58
1.12	**35**	7.3E-11	2.4E-10	2.6E-11	39	7.2E-11	9.0E-11	3.1E-11	20	1.4E-10	2.5E-10	3.9E-11	33	4.9E-11	5.2E-11	4.4E-11	3
1.16	**29**	4.2E-11	1.1E-10	2.0E-11	19	3.1E-11	5.8E-11	1.4E-11	37	1.2E-10	3.0E-10	1.8E-11	46	1.1E-11	2.8E-11	4.7E-12	47
1.4	**37**	4.5E-11	8.2E-11	2.7E-11	27	3.6E-11	8.9E-11	1.7E-11	32	1.9E-10	3.6E-10	6.2E-11	37				
1.44	**33**	4.9E-11	1.4E-10	2.3E-11	27	6.1E-11	1.3E-10	3.1E-11	32	1.4E-10	3.4E-10	1.8E-11	57	1.5E-11	2.6E-11	8.9E-12	32
1.63	**19**	5.4E-11	1.4E-10	1.8E-11	33	3.8E-11	5.8E-11	2.2E-11	29	1.4E-10	3.3E-10	3.5E-11	57	1.7E-11	4.7E-11	3.9E-12	61
1.64	**31**	1.2E-10	3.6E-10	6.0E-11	24	6.1E-11	9.0E-11	3.6E-11	38	1.3E-10	2.6E-10	5.2E-11	28	5.9E-11	6.7E-11	5.2E-11	5
Average		5.8E-11	1.6E-10	2.5E-11	31	4.5E-11	7.7E-11	2.2E-11	33	1.3E-10	3.0E-10	3.4E-11	42	2.7E-11	4.2E-11	2.0E-11	34

Table 82. Sample weekly calcium release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	1.6E-11	2.6E-11	1.0E-11	8	3.1E-12	1.2E-11	3.5E-13	189								
0.22	**3**	1.8E-11	3.1E-11	8.7E-12	7	3.5E-12	1.0E-11	3.6E-13	191								
Average		1.7E-11	2.9E-11	9.4E-12	8	3.3E-12	1.1E-11	3.6E-13	190								
Group II																	
0.4	**5**	3.7E-11	6.1E-11	2.7E-11	8	1.2E-11	2.8E-11	5.5E-12	133	1.2E-11	2.4E-11	4.8E-12	21	4.6E-12	6.8E-12	2.8E-12	9
0.41	**7**	1.7E-11	3.3E-11	3.6E-12	10	3.1E-12	1.2E-11	7.8E-13	52	5.7E-12	1.0E-11	3.5E-12	22	3.4E-12	7.2E-12	1.9E-12	54
0.51	**9**	1.9E-11	5.6E-11	2.2E-12	11	3.1E-12	5.2E-12	1.4E-12	33	8.8E-12	1.5E-11	5.8E-12	29	4.7E-12	1.6E-11	2.3E-12	66
0.54	**11**	3.4E-11	1.0E-10	1.1E-11	32	1.3E-11	2.2E-11	2.2E-12	37	9.0E-12	1.5E-11	5.2E-12	40				
0.57	**13**	2.5E-11	5.6E-11	8.5E-12	7	5.2E-12	1.2E-11	7.3E-13	45	8.6E-12	1.4E-11	5.7E-12	14	6.5E-12	8.6E-12	4.2E-12	21
0.58	**15**	2.0E-11	5.0E-11	5.2E-12	7	5.4E-12	1.0E-11	7.3E-13	24	6.8E-12	1.3E-11	3.6E-12	30	4.6E-12	1.1E-11	2.5E-12	80
Average		2.5E-11	6.0E-11	9.6E-12	13	6.9E-12	1.5E-11	1.9E-12	54	8.5E-12	1.5E-11	4.8E-12	26	4.8E-12	9.8E-12	2.7E-12	46
Group III																	
0.71	**17**	1.9E-11	2.7E-11	8.7E-12	6	3.1E-12	1.2E-11	7.3E-13	39	8.1E-12	2.8E-11	3.6E-13	38	4.3E-12	7.4E-12	2.2E-12	58
1.12	**35**	4.3E-11	8.2E-11	3.6E-12	27	1.3E-11	1.8E-11	8.0E-12	23	1.9E-11	3.2E-11	7.1E-12	33	8.7E-12	9.5E-12	7.8E-12	3
1.16	**29**	2.2E-11	3.4E-11	1.2E-11	10	5.7E-12	2.2E-11	2.3E-12	37	1.3E-11	2.8E-11	5.6E-12	48	4.6E-12	8.0E-12	2.9E-12	47
1.4	**37**	2.8E-11	5.6E-11	1.2E-11	18	8.3E-12	1.9E-11	4.4E-12	35	1.6E-11	2.7E-11	6.6E-12	38				
1.44	**33**	2.4E-11	6.1E-11	5.1E-12	16	5.0E-12	1.2E-11	2.2E-12	34	1.3E-11	2.7E-11	4.2E-12	57	3.8E-12	6.3E-12	2.5E-12	32
1.63	**19**	2.2E-11	5.0E-11	8.7E-12	22	7.6E-12	2.3E-11	3.6E-12	8	1.3E-11	2.7E-11	5.3E-12	64	4.9E-12	1.1E-11	2.5E-12	62
1.64	**31**	5.8E-11	1.6E-10	1.2E-11	12	9.9E-12	1.6E-11	6.4E-12	40	1.6E-11	2.4E-11	1.1E-11	28	1.1E-11	1.4E-11	9.9E-12	5
Average		3.1E-11	6.7E-11	8.8E-12	16	7.4E-12	1.7E-11	4.0E-12	31	1.4E-11	2.8E-11	5.7E-12	44	6.2E-12	9.4E-12	4.6E-12	35

Table 83. Sample weekly magnesium release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	8.2E-12	1.2E-11	1.2E-12	9	2.3E-12	8.9E-12	5.5E-13	187								
0.22	**3**	1.6E-11	5.5E-11	3.5E-12	8	3.5E-12	1.4E-11	5.6E-13	191								
Average		1.2E-11	3.3E-11	2.3E-12	9	2.9E-12	1.2E-11	5.6E-13	189								
Group II																	
0.4	**5**	5.0E-12	9.8E-12	1.2E-12	7	1.3E-12	9.5E-12	5.7E-13	129	1.1E-12	2.7E-12	5.8E-13	21	5.9E-13	6.0E-13	5.8E-13	9
0.41	**7**	5.4E-12	9.5E-12	1.2E-12	8	2.9E-12	9.8E-12	5.9E-13	51	7.2E-12	1.8E-11	5.9E-13	22	1.3E-12	5.9E-12	5.5E-13	54
0.51	**9**	6.1E-12	1.2E-11	1.2E-12	10	3.5E-12	1.0E-11	1.2E-12	33	7.1E-12	1.7E-11	6.1E-13	29	1.5E-12	7.1E-12	5.6E-13	66
0.54	**11**	1.5E-11	7.3E-11	1.2E-12	30	4.5E-12	1.1E-11	1.2E-12	36	1.7E-12	7.5E-12	5.9E-13	41				
0.57	**13**	1.4E-11	2.1E-11	7.0E-12	8	7.8E-12	1.9E-11	1.2E-12	45	1.4E-11	3.5E-11	4.7E-12	14	4.6E-12	9.4E-12	5.9E-13	22
0.58	**15**	1.6E-11	3.1E-11	3.7E-12	8	8.5E-12	1.4E-11	1.2E-12	24	8.5E-12	1.3E-11	3.5E-12	29	1.3E-12	5.9E-12	5.6E-13	79
Average		1.0E-11	2.6E-11	2.6E-12	12	4.7E-12	1.2E-11	9.8E-13	53	6.6E-12	1.6E-11	1.8E-12	26	1.9E-12	5.8E-12	5.7E-13	46
Group III																	
0.71	**17**	1.6E-11	2.8E-11	9.6E-12	7	6.9E-12	2.2E-11	1.2E-12	39	2.1E-11	6.5E-11	5.9E-12	38	2.2E-12	9.4E-12	5.9E-13	58
1.12	**35**	7.5E-12	1.9E-11	1.2E-12	25	9.6E-12	1.4E-11	4.7E-12	24	1.4E-11	2.5E-11	7.0E-12	33	8.3E-12	9.5E-12	7.0E-12	3
1.16	**29**	1.2E-11	2.3E-11	1.2E-12	9	1.0E-11	2.5E-11	6.1E-12	36	2.7E-11	5.5E-11	5.8E-12	48	2.4E-12	1.1E-11	5.8E-13	47
1.4	**37**	1.0E-11	2.2E-11	1.2E-12	16	1.5E-11	1.9E-11	6.1E-12	33	4.4E-11	8.2E-11	7.2E-12	38				
1.44	**33**	1.3E-11	2.1E-11	7.3E-12	12	1.1E-11	1.7E-11	6.0E-12	35	2.9E-11	6.5E-11	3.5E-12	57	3.2E-12	7.0E-12	5.8E-13	32
1.63	**19**	3.3E-11	8.1E-11	1.4E-11	26	1.9E-11	2.2E-11	1.4E-11	7	2.4E-11	4.1E-11	8.3E-12	65	4.3E-12	1.4E-11	5.3E-13	62
1.64	**31**	3.9E-11	8.9E-11	7.1E-12	12	1.5E-11	2.4E-11	7.2E-12	41	2.4E-11	3.9E-11	1.3E-11	28	1.1E-11	1.4E-11	8.1E-12	5
Average		1.8E-11	4.0E-11	6.0E-12	15	1.2E-11	2.0E-11	6.5E-12	31	2.6E-11	5.3E-11	7.2E-12	44	5.2E-12	1.1E-11	2.9E-12	35

Table 84. Sample weekly calcium+magnesium release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	2.4E-11	3.8E-11	1.4E-11	8	5.3E-12	1.7E-11	6.4E-13	187								
0.22	**3**	3.4E-11	8.0E-11	1.5E-11	7	7.0E-12	2.2E-11	9.7E-13	190								
Average		2.9E-11	5.9E-11	1.4E-11	8	6.2E-12	1.9E-11	8.0E-13	189								
Group II																	
0.4	**5**	4.3E-11	6.3E-11	3.1E-11	6	1.3E-11	3.0E-11	6.1E-12	128	1.3E-11	2.6E-11	5.4E-12	21	5.2E-12	7.4E-12	3.4E-12	9
0.41	**7**	2.5E-11	4.2E-11	7.2E-12	7	6.2E-12	1.9E-11	2.7E-12	48	1.3E-11	2.4E-11	6.3E-12	22	4.7E-12	1.1E-11	2.5E-12	54
0.51	**9**	2.6E-11	6.6E-11	3.4E-12	9	6.6E-12	1.2E-11	3.5E-12	32	1.6E-11	3.2E-11	8.7E-12	29	6.2E-12	2.3E-11	2.9E-12	66
0.54	**11**	5.0E-11	1.4E-10	2.0E-11	28	1.7E-11	2.9E-11	5.7E-12	36	1.1E-11	2.1E-11	5.8E-12	40				
0.57	**13**	3.8E-11	7.8E-11	1.8E-11	7	1.3E-11	2.7E-11	1.9E-12	45	2.3E-11	4.9E-11	1.0E-11	14	1.1E-11	1.7E-11	5.5E-12	21
0.58	**15**	3.5E-11	8.1E-11	8.8E-12	7	1.4E-11	2.4E-11	1.9E-12	24	1.5E-11	2.6E-11	1.0E-11	29	5.8E-12	1.4E-11	3.1E-12	79
Average		3.6E-11	7.7E-11	1.5E-11	11	1.2E-11	2.4E-11	3.6E-12	52	1.5E-11	3.0E-11	7.8E-12	26	6.6E-12	1.4E-11	3.5E-12	46
Group III																	
0.71	**17**	3.2E-11	4.3E-11	1.8E-11	6	1.0E-11	3.4E-11	1.9E-12	39	2.9E-11	8.9E-11	7.5E-12	37	6.6E-12	1.5E-11	2.7E-12	58
1.12	**35**	5.0E-11	1.0E-10	1.3E-11	25	2.2E-11	3.0E-11	1.5E-11	23	3.4E-11	5.5E-11	1.4E-11	33	1.7E-11	1.8E-11	1.5E-11	3
1.16	**29**	3.5E-11	4.8E-11	1.3E-11	9	1.6E-11	4.6E-11	9.7E-12	36	4.0E-11	7.7E-11	1.1E-11	48	7.0E-12	1.9E-11	3.5E-12	47
1.4	**37**	3.9E-11	6.4E-11	2.3E-11	16	2.3E-11	3.1E-11	1.6E-11	33	6.0E-11	1.1E-10	1.5E-11	38				
1.44	**33**	4.2E-11	7.6E-11	1.8E-11	12	1.7E-11	2.2E-11	1.0E-11	34	4.2E-11	8.9E-11	7.7E-12	57	7.0E-12	1.3E-11	3.1E-12	32
1.63	**19**	5.3E-11	9.3E-11	2.6E-11	22	2.6E-11	4.2E-11	1.8E-11	7	3.7E-11	5.9E-11	2.0E-11	64	9.2E-12	2.6E-11	3.1E-12	62
1.64	**31**	9.6E-11	2.4E-10	2.3E-11	12	2.5E-11	3.8E-11	1.8E-11	40	4.0E-11	6.3E-11	2.6E-11	28	2.1E-11	2.8E-11	1.8E-11	5
Average		5.0E-11	9.6E-11	1.9E-11	15	2.0E-11	3.5E-11	1.3E-11	30	4.0E-11	7.7E-11	1.4E-11	44	1.1E-11	2.0E-11	7.5E-12	35

Table 85. Sample weekly sodium release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**					1.4E-12	3.2E-12	6.1E-13	32								
0.22	**3**					1.3E-12	2.3E-12	6.2E-13	34								
Average						1.4E-12	2.8E-12	6.1E-13	33								
Group II																	
0.4	**5**					8.0E-13	2.1E-12	6.2E-13	20	2.2E-12	3.4E-12	1.5E-12	7	8.7E-13	1.4E-12	6.2E-13	3
0.41	**7**									2.5E-12	2.8E-12	2.2E-12	2	2.0E-12	3.6E-12	1.4E-12	14
0.51	**9**													2.5E-12	5.1E-12	1.5E-12	17
0.54	**11**									2.1E-12	2.4E-12	1.8E-12	2				
0.57	**13**													3.7E-12	4.8E-12	2.6E-12	2
0.58	**15**													2.4E-12	1.0E-11	1.3E-12	18
Average						8.0E-13	2.1E-12	6.2E-13	20	2.3E-12	2.9E-12	1.8E-12	4	2.3E-12	5.0E-12	1.5E-12	11
Group III																	
0.71	**17**									2.6E-12	3.0E-12	2.3E-12	2	2.3E-12	3.3E-12	1.6E-12	16
1.12	**35**									6.4E-12	9.2E-12	3.6E-12	2	3.0E-12	3.0E-12	3.0E-12	1
1.16	**29**									4.0E-12	5.5E-12	2.9E-12	3	2.4E-12	3.7E-12	1.5E-12	15
1.4	**37**									5.2E-12	5.9E-12	4.1E-12	3				
1.44	**33**									3.6E-12	7.1E-12	2.3E-12	7	2.2E-12	3.6E-12	1.5E-12	11
1.63	**19**									3.5E-12	3.5E-12	3.5E-12	1	2.4E-12	4.0E-12	1.6E-12	18
1.64	**31**									4.2E-12	4.2E-12	4.2E-12	1	4.1E-12	4.5E-12	3.8E-12	2
Average										4.2E-12	5.5E-12	3.3E-12	3	2.7E-12	3.7E-12	2.2E-12	11

Table 86. Sample weekly potassium release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**					5.7E-12	1.6E-11	7.2E-13	36								
0.22	**3**					3.3E-12	1.2E-11	3.6E-13	36								
Average						4.5E-12	1.4E-11	5.4E-13	36								
Group II																	
0.4	**5**					4.6E-12	1.3E-11	1.3E-12	18	1.1E-12	2.2E-12	3.6E-13	7	3.7E-13	3.7E-13	3.7E-13	2
0.41	**7**									6.3E-12	8.0E-12	4.5E-12	2	3.7E-12	1.0E-11	1.2E-12	13
0.51	**9**													3.2E-12	9.2E-12	3.7E-13	17
0.54	**11**									3.6E-12	4.0E-12	3.2E-12	2				
0.57	**13**													7.6E-12	1.0E-11	4.9E-12	2
0.58	**15**													4.9E-12	1.7E-11	3.7E-13	17
Average						4.6E-12	1.3E-11	1.3E-12	18	3.7E-12	4.8E-12	2.7E-12	4	3.9E-12	9.5E-12	1.4E-12	10
Group III																	
0.71	**17**									5.0E-12	6.1E-12	3.9E-12	2	6.0E-12	1.9E-11	9.0E-13	17
1.12	**35**									6.0E-12	7.4E-12	4.7E-12	2	6.2E-12	6.2E-12	6.2E-12	1
1.16	**29**									7.1E-12	9.8E-12	4.9E-12	3	3.4E-12	1.1E-11	1.0E-12	15
1.4	**37**									1.6E-11	2.1E-11	7.7E-12	3				
1.44	**33**									9.5E-12	2.2E-11	2.4E-12	7	1.5E-12	2.4E-12	7.6E-13	10
1.63	**19**									6.5E-12	6.5E-12	6.5E-12	1	6.2E-12	2.1E-11	1.5E-12	17
1.64	**31**									4.1E-12	4.1E-12	4.1E-12	1	6.0E-12	6.7E-12	5.3E-12	2
Average										7.7E-12	1.1E-11	4.9E-12	3	4.9E-12	1.1E-11	2.6E-12	10

Table 87. Slopes, y-intercepts, and r^2 values from regression analysis of relationships among %S, sulfate, pH, Ca and Mg (excludes R39-44).

independent	dependent	Period A			Period B			Period C			Period D		
		slope	y-int	r^2	slope	y-int	r^2	slope	y-int	r^2	slope	y-int	r^2
log %S	pH avg	-1.75	5.76	0.656	-2.03	4.77	0.751	-0.96	3.95	0.722	-0.30	4.57	0.046
log %S	pH max	-1.66	7.14	0.359	-2.27	5.29	0.772	-0.03	4.57	0.001	-1.47	5.17	0.237
log %S	pH min	-1.65	5.00	0.672	-1.57	4.20	0.681	-1.48	3.45	0.793	0.05	4.24	0.003
log %S	log SO_4 avg	0.92	1.36	0.727	1.53	1.25	0.874	0.98	1.70	0.868	0.36	1.07	0.051
log %S	log SO_4 max	0.88	1.81	0.656	1.01	1.57	0.762	1.28	2.08	0.800	0.21	1.37	0.083
log %S	log SO_4 min	1.22	0.95	0.593	2.21	0.90	0.872	0.36	1.26	0.162	0.58	0.79	0.051
log SO_4 avg	pH avg	-1.65	8.03	0.672	-1.32	6.42	0.850	-1.02	5.67	0.897	-1.12	5.76	0.942
log SO_4 avg	pH max	-2.08	9.93	0.650	-1.45	7.11	0.842	-0.18	4.86	0.032	-2.05	7.29	0.919
log SO_4 avg	pH min	-1.24	6.76	0.437	-1.08	5.53	0.854	-1.47	5.95	0.861	-0.71	5.01	0.699
log SO_4 avg	log Ca avg	0.38	0.65	0.575	0.30	0.20	0.391	0.51	-0.03	0.652	0.52	-0.04	0.899
log SO_4 avg	log Ca max	0.51	0.89	0.582	0.14	0.80	0.147	0.51	0.27	0.580	0.17	0.56	0.230
log SO_4 avg	log Ca min	0.21	0.41	0.067	0.68	-0.67	0.578	0.43	-0.27	0.091	0.79	-0.50	0.926
log SO_4 avg	log Mg avg	0.40	0.30	0.241	0.62	-0.09	0.639	1.64	-1.86	0.652	1.08	-0.90	0.866
log SO_4 avg	log Mg max	0.43	0.67	0.191	0.19	0.78	0.360	1.20	-0.74	0.474	0.15	0.61	0.117
log SO_4 avg	log Mg min	0.33	-0.18	0.067	0.80	-0.72	0.670	1.63	-2.37	0.612	1.48	-1.77	0.768
log SO_4 avg	log Ca+Mg avg	0.42	0.81	0.771	0.44	0.43	0.812	0.92	-0.34	0.921	0.70	-0.01	0.947
log SO_4 avg	log Ca+Mg max	0.44	1.13	0.461	0.19	1.02	0.352	0.83	0.13	0.669	0.18	0.85	0.208
log SO_4 avg	log Ca+Mg min	0.39	0.48	0.278	0.77	-0.30	0.709	0.68	-0.30	0.636	1.02	-0.62	0.918

Table 88. Iron, aluminum and silicon release rates for all Dunka blast hole samples combined. Average, minimum and maximum rates are presented for entire period of record, as well as for pH>4 and pH≤4 (n = number of measurements) (excludes R39-44).

Criteria	Stats	Fe (mol/s)	Al (mol/s)	Si (mol/s)
Entire period of record	Avg	6.75E-13	4.58E-12	1.60E-11
	Min	4.81E-15	1.05E-14	1.26E-13
	Max	3.67E-11	8.25E-11	9.58E-11
	n	270	271	232
pH > 4	Avg	3.07E-13	1.54E-12	1.56E-11
	Min	4.81E-15	1.05E-14	1.26E-13
	Max	3.67E-11	2.31E-11	9.58E-11
	n	239	240	225
pH ≤ 4	Avg	3.51E-12	2.81E-11	2.93E-11
	Min	1.20E-13	2.13E-14	4.51E-12
	Max	2.39E-11	8.25E-11	4.03E-11
	n	31	31	7

Table 89. Sample weekly cobalt release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	5.1E-14	1.5E-13	2.4E-14	9	6.6E-15	2.4E-14	4.8E-15	38								
0.22	**3**	4.0E-14	9.8E-14	2.4E-14	8	7.2E-15	2.0E-14	4.8E-15	39								
Average		4.5E-14	1.2E-13	2.4E-14	9	6.9E-15	2.2E-14	4.8E-15	39								
Group II																	
0.4	**5**	9.0E-14	2.0E-13	2.5E-14	8	1.1E-14	2.1E-14	4.9E-15	22	3.6E-14	7.3E-14	1.2E-14	6	4.9E-15	4.9E-15	4.8E-15	3
0.41	**7**	9.9E-14	2.0E-13	2.4E-14	11	7.4E-14	1.0E-13	5.1E-14	4	9.0E-14	1.2E-13	6.0E-14	4	1.1E-14	3.2E-14	4.6E-15	13
0.51	**9**	9.2E-14	2.1E-13	2.5E-14	11	3.7E-14	4.9E-14	2.6E-14	2	1.4E-13	2.4E-13	7.6E-14	4	1.1E-14	4.4E-14	4.8E-15	17
0.54	**11**	7.5E-14	2.0E-13	2.4E-14	33	5.0E-14	5.0E-14	5.0E-14	2	1.4E-14	2.3E-14	5.0E-15	6				
0.57	**13**	9.3E-14	2.4E-13	2.4E-14	8	1.4E-13	2.5E-13	4.9E-14	6	1.5E-13	2.1E-13	9.2E-14	2	3.7E-14	7.9E-14	1.5E-14	4
0.58	**15**	2.5E-13	3.4E-13	1.5E-13	8	1.1E-13	1.5E-13	9.8E-14	4	4.3E-14	4.3E-14	4.3E-14	1	8.7E-15	4.0E-14	4.7E-15	18
Average		1.2E-13	2.3E-13	4.6E-14	13	7.0E-14	1.0E-13	4.7E-14	7	7.9E-14	1.2E-13	4.8E-14	4	1.5E-14	4.0E-14	6.9E-15	11
Group III																	
0.71	**17**	1.8E-13	4.0E-13	2.5E-14	7	2.2E-13	3.0E-13	9.9E-14	5	1.2E-13	2.5E-13	1.2E-14	6	6.8E-15	1.8E-14	4.7E-15	15
1.12	**35**	1.0E-12	1.9E-12	9.9E-14	11	1.7E-13	2.4E-13	9.8E-14	2	1.1E-13	2.0E-13	2.8E-14	5	2.1E-14	2.1E-14	2.1E-14	1
1.16	**29**	3.6E-14	5.2E-14	2.6E-14	10	4.9E-13	5.7E-13	3.5E-13	4	1.8E-13	3.4E-13	6.2E-14	7	1.2E-14	3.5E-14	4.8E-15	13
1.4	**37**	1.6E-13	5.6E-13	2.5E-14	15	6.0E-13	6.0E-13	6.0E-13	1	4.4E-13	7.8E-13	2.0E-13	7				
1.44	**33**	2.7E-13	4.5E-13	2.6E-14	12	2.7E-13	3.0E-13	2.5E-13	2	1.9E-13	4.6E-13	3.3E-14	10	9.7E-15	2.1E-14	4.8E-15	10
1.63	**19**	6.6E-13	9.4E-13	2.0E-13	26	3.7E-13	5.5E-13	2.6E-13	3	1.2E-13	2.1E-13	3.3E-14	7	8.1E-15	2.1E-14	4.4E-15	16
1.64	**31**	2.1E-12	2.9E-12	4.9E-13	10	1.5E-13	1.5E-13	1.5E-13	1	1.3E-13	1.8E-13	5.8E-14	4	2.9E-14	3.0E-14	2.9E-14	2
Average		6.3E-13	1.0E-12	1.3E-13	13	3.3E-13	3.9E-13	2.6E-13	3	1.8E-13	3.4E-13	6.1E-14	7	1.5E-14	2.4E-14	1.1E-14	10

Table 90. Sample weekly copper release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	4.3E-14	1.4E-13	2.2E-14	9	2.1E-14	5.6E-14	4.5E-15	37								
0.22	**3**	4.3E-14	1.4E-13	2.2E-14	8	2.6E-14	6.0E-14	1.4E-14	37								
Average		4.3E-14	1.4E-13	2.2E-14	9	2.4E-14	5.8E-14	9.0E-15	37								
Group II																	
0.4	**5**	5.4E-14	1.8E-13	2.3E-14	8	2.2E-13	9.0E-13	3.5E-14	22	8.4E-13	1.7E-12	2.9E-13	6	1.8E-13	2.1E-13	1.7E-13	3
0.41	**7**	1.5E-13	9.2E-13	2.3E-14	11	1.3E-12	1.8E-12	9.0E-13	4	1.5E-12	1.9E-12	9.7E-13	4	5.8E-13	1.2E-12	2.1E-13	13
0.51	**9**	7.9E-13	3.1E-12	2.2E-14	11	1.7E-12	2.1E-12	1.3E-12	2	3.4E-12	4.5E-12	2.8E-12	4	1.3E-12	3.3E-12	5.2E-14	17
0.54	**11**	5.2E-14	2.7E-13	2.2E-14	34	2.8E-13	2.8E-13	2.8E-13	2	2.2E-13	4.1E-13	7.6E-14	6				
0.57	**13**	1.3E-13	3.1E-13	2.2E-14	8	1.9E-12	4.1E-12	7.1E-13	6	2.4E-12	2.4E-12	2.4E-12	1	2.5E-12	3.0E-12	1.8E-12	4
0.58	**15**	2.0E-13	4.5E-13	2.3E-14	8	8.5E-13	1.1E-12	5.5E-13	4	1.1E-12	1.1E-12	1.1E-12	1	6.8E-13	1.3E-12	1.4E-13	18
Average		2.3E-13	8.8E-13	2.3E-14	13	1.0E-12	1.7E-12	6.2E-13	7	1.6E-12	2.0E-12	1.3E-12	4	1.1E-12	1.8E-12	4.8E-13	11
Group III																	
0.71	**17**	2.3E-13	6.9E-13	2.3E-14	7	2.0E-12	2.5E-12	1.3E-12	5	3.2E-12	5.4E-12	6.3E-13	6	1.9E-12	3.8E-12	8.3E-13	15
1.12	**35**	2.0E-13	7.8E-13	2.3E-14	11	1.3E-12	1.3E-12	1.2E-12	2	2.8E-12	4.7E-12	1.1E-12	5	9.1E-13	9.1E-13	9.1E-13	1
1.16	**29**	4.8E-14	9.5E-14	2.4E-14	10	1.2E-12	2.1E-12	3.2E-13	4	2.1E-12	4.5E-12	4.2E-13	7	1.2E-12	2.3E-12	9.0E-13	12
1.4	**37**	1.5E-13	9.8E-13	2.4E-14	15	1.7E-12	1.7E-12	1.7E-12	1	3.8E-12	6.7E-12	1.4E-12	7				
1.44	**33**	1.5E-13	6.0E-13	2.3E-14	12	7.4E-13	1.0E-12	4.6E-13	2	2.9E-12	5.4E-12	7.2E-13	10	3.7E-12	5.0E-12	1.2E-12	10
1.63	**19**	2.8E-13	7.5E-13	2.3E-14	26	9.9E-13	1.3E-12	4.2E-13	3	4.2E-12	7.6E-12	9.7E-13	8	1.8E-12	4.5E-12	3.6E-13	16
1.64	**31**	7.1E-13	1.3E-12	4.6E-14	10	3.2E-12	3.2E-12	3.2E-12	1	7.2E-12	8.8E-12	6.2E-12	4	3.4E-12	3.6E-12	3.3E-12	2
Average		2.5E-13	7.4E-13	2.6E-14	13	1.6E-12	1.9E-12	1.2E-12	3	3.7E-12	6.2E-12	1.6E-12	7	2.2E-12	3.4E-12	1.2E-12	9

Table 91. Sample weekly nickel release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	9.2E-14	3.0E-13	2.4E-14	9	1.9E-14	5.5E-14	4.8E-15	37								
0.22	**3**	7.7E-14	2.0E-13	2.4E-14	8	1.7E-14	5.4E-14	4.8E-15	37								
Average		8.5E-14	2.5E-13	2.4E-14	9	1.8E-14	5.5E-14	4.8E-15	37								
Group II																	
0.4	**5**	4.5E-13	7.1E-13	1.0E-13	8	9.7E-14	2.9E-13	4.3E-14	22	6.1E-13	1.3E-12	1.9E-13	6	7.3E-14	9.7E-14	5.4E-14	3
0.41	**7**	1.2E-12	2.9E-12	2.4E-14	11	1.6E-12	2.7E-12	9.3E-13	4	1.6E-12	2.2E-12	1.1E-12	4	1.8E-13	5.9E-13	2.0E-14	13
0.51	**9**	9.1E-13	2.6E-12	2.6E-14	10	3.8E-13	4.1E-13	3.5E-13	2	1.6E-12	2.0E-12	1.1E-12	4	1.1E-13	6.0E-13	4.9E-15	17
0.54	**11**	3.7E-13	1.9E-12	2.5E-14	33	3.5E-13	3.5E-13	3.5E-13	2	1.2E-13	2.0E-13	4.6E-14	6				
0.57	**13**	8.3E-13	2.0E-12	2.4E-14	8	1.2E-12	1.9E-12	7.7E-13	6	1.3E-12	1.3E-12	1.3E-12	1	5.1E-13	1.1E-12	2.1E-13	4
0.58	**15**	1.5E-12	2.3E-12	1.0E-12	8	8.5E-13	1.1E-12	7.0E-13	4	4.7E-13	4.7E-13	4.7E-13	1	7.7E-14	4.6E-13	4.9E-15	18
Average		8.6E-13	2.1E-12	2.0E-13	13	7.4E-13	1.1E-12	5.2E-13	7	9.6E-13	1.2E-12	7.0E-13	4	1.9E-13	5.7E-13	5.9E-14	11
Group III																	
0.71	**17**	1.2E-12	3.2E-12	2.5E-14	7	2.3E-12	3.2E-12	8.0E-13	5	1.3E-12	2.2E-12	1.3E-13	6	4.9E-14	1.8E-13	4.7E-15	15
1.12	**35**	7.3E-12	1.3E-11	1.2E-12	11	1.4E-12	1.5E-12	1.4E-12	2	1.4E-12	2.6E-12	2.9E-13	5	2.1E-13	2.1E-13	2.1E-13	1
1.16	**29**	1.3E-13	2.6E-13	5.1E-14	10	5.6E-12	6.8E-12	3.6E-12	4	2.6E-12	4.9E-12	8.3E-13	7	1.4E-13	4.5E-13	3.5E-14	13
1.4	**37**	1.5E-12	6.4E-12	1.0E-13	15	6.6E-12	6.6E-12	6.6E-12	1	5.9E-12	1.1E-11	2.6E-12	6				
1.44	**33**	2.4E-12	3.7E-12	2.7E-13	12	2.5E-12	2.6E-12	2.5E-12	2	1.9E-12	4.6E-12	1.9E-13	10	7.2E-14	1.9E-13	3.4E-14	10
1.63	**19**	5.1E-12	7.1E-12	1.4E-12	26	3.0E-12	3.6E-12	2.7E-12	3	1.4E-12	2.8E-12	2.1E-13	8	3.7E-14	1.1E-13	4.7E-15	16
1.64	**31**	2.2E-11	3.6E-11	5.8E-12	10	2.2E-12	2.2E-12	2.2E-12	1	2.0E-12	2.9E-12	8.1E-13	4	3.9E-13	3.9E-13	3.8E-13	2
Average		5.7E-12	9.9E-12	1.3E-12	13	3.4E-12	3.8E-12	2.8E-12	3	2.3E-12	4.4E-12	7.2E-13	7	1.5E-13	2.6E-13	1.1E-13	10

Table 92. Sample weekly zinc release (mol/second) by groups and time periods for all Dunka blast hole samples (excludes R39-44).

%S	Reactor	Period A				Period B				Period C				Period D			
		Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n	Ave	Max	Min	n
Group I																	
0.18	**1**	6.6E-14	1.3E-13	2.2E-14	9	5.2E-14	2.2E-13	9.0E-15	38								
0.22	**3**	5.2E-14	8.8E-14	2.2E-14	8	4.6E-14	2.3E-13	1.3E-14	38								
Average		5.9E-14	1.1E-13	2.2E-14	9	4.9E-14	2.2E-13	1.1E-14	38								
Group II																	
0.4	**5**	1.0E-13	1.8E-13	4.5E-14	8	8.7E-14	2.6E-13	2.9E-14	22	1.0E-13	1.8E-13	4.8E-14	6	2.4E-14	3.7E-14	1.7E-14	3
0.41	**7**	1.4E-13	3.1E-13	2.2E-14	11	8.3E-14	1.8E-13	4.6E-14	4	1.8E-13	3.5E-13	7.0E-14	4	8.8E-14	1.8E-13	2.9E-14	13
0.51	**9**	1.8E-13	6.3E-13	2.3E-14	11	9.0E-14	9.2E-14	8.9E-14	2	1.6E-13	2.2E-13	1.2E-13	4	1.2E-13	3.5E-13	3.1E-14	17
0.54	**11**	1.4E-13	5.7E-13	2.2E-14	34	1.8E-13	1.8E-13	1.8E-13	2	1.4E-13	2.3E-13	6.2E-14	6				
0.57	**13**	1.7E-13	4.8E-13	2.2E-14	8	2.0E-13	2.2E-13	1.3E-13	6	2.3E-13	2.4E-13	2.2E-13	2	1.9E-13	2.5E-13	1.2E-13	4
0.58	**15**	2.7E-13	4.8E-13	2.2E-14	8	2.6E-13	4.0E-13	1.3E-13	4	8.8E-14	8.8E-14	8.8E-14	1	9.5E-14	2.0E-13	2.2E-14	18
Average		1.7E-13	4.4E-13	2.6E-14	13	1.5E-13	2.2E-13	1.0E-13	7	1.5E-13	2.2E-13	1.0E-13	4	1.0E-13	2.0E-13	4.3E-14	11
Group III																	
0.71	**17**	1.8E-13	4.0E-13	2.2E-14	7	1.8E-13	3.2E-13	9.0E-14	5	1.7E-13	2.5E-13	1.0E-13	6	8.9E-14	2.4E-13	4.6E-14	15
1.12	**35**	4.1E-13	8.0E-13	1.3E-13	11	3.1E-13	4.4E-13	1.8E-13	2	4.3E-13	5.9E-13	2.9E-13	5	2.4E-13	2.4E-13	2.4E-13	1
1.16	**29**	4.0E-14	4.7E-14	2.3E-14	10	3.7E-13	6.2E-13	1.8E-13	4	2.7E-13	5.0E-13	1.4E-13	7	1.4E-13	9.1E-13	3.5E-14	13
1.4	**37**	1.2E-13	5.0E-13	2.3E-14	15	2.7E-13	2.7E-13	2.7E-13	1	3.1E-13	5.0E-13	1.2E-13	7				
1.44	**33**	2.3E-13	3.6E-13	9.5E-14	12	1.8E-13	2.2E-13	1.4E-13	2	2.4E-13	5.0E-13	1.1E-13	10	7.5E-14	1.3E-13	3.7E-14	9
1.63	**19**	3.2E-13	6.3E-13	4.4E-14	26	4.3E-13	5.0E-13	3.3E-13	3	4.9E-13	8.5E-13	1.8E-13	8	1.7E-13	4.2E-13	4.0E-14	16
1.64	**31**	7.2E-13	9.8E-13	2.7E-13	10	2.2E-13	2.2E-13	2.2E-13	1	3.6E-13	4.0E-13	2.9E-13	4	2.2E-13	2.8E-13	1.6E-13	2
Average		2.9E-13	5.3E-13	8.7E-14	13	2.8E-13	3.7E-13	2.0E-13	3	3.2E-13	5.2E-13	1.8E-13	7	1.6E-13	3.7E-13	9.2E-14	9

Table 93. Ni, Mg content and Ni-bearing mineral abundance in Dunka blast hole samples (excludes R39-44).

%S	Ni ppm	Ni (g/sample)	Ni (mol/sample)	Olivine (mol/sample)	Ni ol (mols Ni/ mol ol)	Ni in ol (mol/sample)	fraction Ni as olivine	fraction Ni as pent/po	Mg (mols/mol ol)	ol Ni/Mg (molar)
Group I										
0.18	525	0.039	6.71E-04	0.043	0.003	1.28E-04	0.191	0.809	0.763	0.0039
0.22	484	0.036	6.18E-04	0.019	0.002	3.82E-05	0.062	0.938	0.857	0.0023
Group II										
0.40[1]	231	0.017	2.95E-04					1.000		
0.41	511	0.038	6.53E-04	0.056	0.003	1.68E-04	0.258	0.742	0.897	0.0033
0.51	473	0.035	6.04E-04	0.025	0.002	5.00E-05	0.083	0.917	0.812	0.0025
0.54[1]	174	0.013	2.22E-04					1.000		
0.57	398	0.030	5.09E-04	0.092	0.004	3.66E-04	0.721	0.279	1.003	0.0040
0.58	396	0.030	5.06E-04	0.011	0.001	1.07E-05	0.021	0.979	0.851	0.0012
Group III										
0.71	388	0.029	4.96E-04	0.065	0.002	1.29E-04	0.260	0.740	0.958	0.0021
1.12	329	0.025	4.20E-04	0.018	0.002	3.58E-05	0.085	0.915	0.953	0.0021
1.16	422	0.032	5.39E-04	0.073	0.002	1.46E-04	0.271	0.729	0.958	0.0021
1.4	503	0.038	6.43E-04	0.026	0.003	7.67E-05	0.119	0.881	0.954	0.0031
1.44	386	0.029	4.93E-04	0.026	0.003	7.70E-05	0.156	0.844	0.953	0.0031
1.63	492	0.037	6.29E-04	0.023	0.002	4.68E-05	0.075	0.925	0.722	0.0028
1.64	718	0.054	9.17E-04	0.014	0.003	4.23E-05	0.046	0.954	0.828	0.0036
Average	429	0.032	3.42E-04	0.038	0.002	1.01E-04	0.156	0.844	0.885	0.0028

[1] Sample had no olivine detected by point count.

Table 94. Average Ni/Mg ratio in Dunka blast hole sample leachate during different sample periods. See appendix for details on period definition. Numbers in parentheses are the number of data points in each period. Blank cells indicate sample did not enter that time period (excludes R39-44).

%S	Reactor	Ni/Mg olivine	Average Ni/Mg ratio in leachate (molar)							
			Period A		Period B		Period C		Period D	
Group I										
0.18	1	0.0038	0.015	(9)	0.011	(37)				
0.22	3	0.002	0.008	(8)	0.006	(37)				
Group II										
0.40[1]	5	No ol	0.152	(7)	0.122	(21)	0.631	(6)	0.123	(3)
0.41	7	0.0033	0.493	(8)	0.461	(4)	0.166	(4)	0.134	(13)
0.51	9	0.0025	0.244	(8)	0.155	(2)	0.231	(4)	0.061	(17)
0.54[1]	11	No ol	0.042	(29)	0.064	(2)	0.086	(6)		
0.57	13	0.004	0.086	(8)	0.247	(6)	0.093	(1)	0.177	(4)
0.58	15	0.0012	0.124	(8)	0.204	(4)	0.079	(1)	0.073	(18)
Group III										
0.71	17	0.0021	0.076	(7)	0.556	(5)	0.084	(6)	0.29	(15)
1.12	35	0.0021	0.989	(10)	0.215	(2)	0.075	(5)	0.030	(1)
1.16	29	0.0021	0.032	(9)	0.311	(3)	0.089	(7)	0.076	(13)
1.4	37	0.0031	0.184	(13)	no Ni:Mg measurements		0.135	(6)		
1.44	33	0.0031	0.214	(9)	0.295	(2)	0.055	(10)	0.041	(10)
1.63	19	0.0028	0.190	(26)	0.164	(2)	0.053	(8)	0.012	(16)
1.64	31	0.0036	0.603	(10)	0.159	(1)	0.075	(4)	0.037	(2)
Total Average		0.0028	0.230		0.212		0.142		0.072	

[1] Sample had no olivine detected by point count.

Table 95. Calculated and observed Ni release for Dunka blast hole samples. Calculated based on Mg in leachate on days Ni was measured, multiplied by Ni:Mg in olivine. This method assumes all Mg comes from ol dissolution therefore it overestimates both Ni release and olivine dissolution. Blank cells indicate sample did not enter that time period (excludes R39-44).

%S	Calculated Ni release from ol (mol/week)				Observed Ni release rate in leachate (mol/week)				Calculated:Observed Ni from olivine			
	Period A	Period B	Period C	Period D	Period A	Period B	Period C	Period D	Period A	Period B	Period C	Period D
Group I												
0.18	1.96E-08	4.68E-09			5.58E-08	1.14E-08			35.1%	41.0%		
0.22	2.21E-08	4.26E-09			4.65E-08	1.03E-08			47.6%	41.5%		
Group II												
0.40[1]					2.73E-07	5.86E-08	3.69E-07	4.40E-08				
0.41	1.10E-08	8.59E-09	2.04E-08	2.74E-09	6.99E-07	9.81E-07	9.94E-07	1.07E-07	1.6%	0.9%	2.1%	2.6%
0.51	9.41E-09	3.62E-09	1.23E-08	2.67E-09	5.53E-07	2.28E-07	9.67E-07	6.94E-08	1.7%	1.6%	1.3%	3.9%
0.54[1]					2.25E-07	2.13E-07	7.03E-08					
0.57	3.32E-08	1.99E-08	3.40E-08	9.25E-09	5.00E-07	6.96E-07	7.93E-07	3.11E-07	6.6%	2.9%	4.3%	3.0%
0.58	1.14E-08	6.37E-09	4.18E-09	1.05E-09	8.90E-07	5.17E-07	2.82E-07	4.67E-08	1.3%	1.2%	1.5%	2.2%
Group III												
0.71	1.98E-08	1.14E-08	2.56E-08	3.26E-09	7.03E-07	1.41E-06	8.01E-07	2.94E-08	2.8%	0.8%	3.2%	11.1%
1.12	1.26E-08	1.06E-08	2.11E-08	8.88E-09	4.39E-06	8.75E-07	8.28E-07	1.27E-07	0.3%	1.2%	2.5%	7.0%
1.16	1.47E-08	2.58E-08	3.67E-08	2.92E-09	7.83E-08	3.39E-06	1.60E-06	8.59E-08	18.8%	0.8%	2.3%	3.4%
1.4	1.61E-08		9.25E-08		9.26E-07	3.98E-06	3.55E-06		1.7%		2.6%	
1.44	2.40E-08	1.96E-08	6.07E-08	3.88E-09	1.46E-06	1.53E-06	1.13E-06	4.37E-08	1.6%	1.3%	5.4%	8.9%
1.63	5.48E-08	3.27E-08	4.18E-08	7.04E-09	3.07E-06	1.83E-06	8.49E-07	2.26E-08	1.8%	1.8%	4.9%	31.2%
1.64	9.07E-08	3.10E-08	5.59E-08	2.31E-08	1.34E-05	1.36E-06	1.19E-06	2.34E-07	0.7%	2.3%	4.7%	9.9%
Average	2.61E-08	1.49E-08	3.68E-08	6.48E-09	1.81E-06	1.14E-06	1.03E-06	1.02E-07	9.4%	8.1%	3.2%	8.3%

[1] Sample had no olivine by point count.

Table 96. Regression results for calculated fraction of Ni from olivine versus percent sulfur in sample (data in Table 5). Periods A and B did not have a significant regression (excludes R39-44).

	slope	y-intercept	r^2
Period A			
Period B			
Period C	0.020	0.012	0.423
Period D	0.120	-0.032	0.422

Table 97. Average sample weekly nickel release per mole olivine by time period for Dunka blast hole samples. Blank cells indicate sample did not enter that time period (excludes R39-44).

%S	Olivine (mol/sample)	mol Ni/mol olivine /week in leachate			
		Period A	Period B	Period C	Period D
Group I					
0.18	0.044	1.27E-06	2.59E-07		
0.22	0.02	2.32E-06	5.13E-07		
Group II					
0.40	No ol				
0.41	0.056	1.25E-05	1.75E-05	1.77E-05	1.91E-06
0.51	0.026	2.13E-05	8.79E-06	3.72E-05	2.67E-06
0.54	No ol				
0.57	0.09	5.55E-06	7.74E-06	8.81E-06	3.46E-06
0.58	0.011	8.09E-05	4.70E-05	2.56E-05	4.25E-06
Group III					
0.71	0.064	1.10E-05	2.21E-05	1.25E-05	4.60E-07
1.12	0.018	2.44E-04	4.86E-05	4.60E-05	7.06E-06
1.16	0.073	1.07E-06	4.64E-05	2.19E-05	1.18E-06
1.4	0.026	3.56E-05	1.53E-04	1.37E-04	
1.44	0.026	5.62E-05	5.87E-05	4.34E-05	1.68E-06
1.63	0.024	1.28E-04	7.62E-05	3.54E-05	9.40E-07
1.64	0.014	9.54E-04	9.70E-05	8.49E-05	1.67E-05
Total Average		1.19E-04	4.49E-05	4.27E-05	4.03E-06

Table 98. Regression results for average sample weekly nickel release per mole olivine (data in Table 7) by time period versus %S. Period D did not have a significant regression (excludes R39-44).

	slope	y-intercept	r^2
Period A	2.59E-4	-1.11E-4	0.278
Period B	6.85E-5	-1.60E-5	0.669
Period C	4.72E-5	-5.15E-6	0.352
Period D	3.93E-6	1.87E-7	0.151

Table 99. Maximum amount of plagioclase and olivine dissolution based on cumulative Ca and Mg release (Table 79) and sample stoichiometry (Table 11) (excludes R39-44).

%S	Reactor	Period of Record (weeks)	Minimum pH	Maximum plagioclase dissolution		Maximum olivine dissolution	
				(mmol)	(mol m^{-2} s^{-1})	(mmol)	(mol m^{-2} s^{-1})
Group I, 2 samples							
0.18	**1**	1252	5.67	7.165	1.40E-09	2.649	2.85E-09
0.22	**3**	1252	5.91	8.833	1.31E-09	3.830	9.17E-09
Average			5.79	7.999	1.35E-09	3.240	6.01E-09
Group II, 6 samples							
0.4	**5**	909	3.78	16.266	3.32E-09		
0.41	**7**	724	4.05	5.167	8.72E-10	1.652	1.36E-09
0.51	**9**	724	3.95	7.447	1.54E-09	2.110	3.87E-09
0.54	**11**	441	4.08	8.153	1.26E-09		
0.57	**13**	441	3.86	6.081	1.38E-09	2.593	1.31E-09
0.58	**15**	724	3.98	6.364	1.34E-09	2.873	1.25E-08
Average			3.95	8.247	1.62E-09	2.307	4.76E-09
Group III, Duluth Complex, 7 samples							
0.71	**17**	724	3.53	5.954	1.05E-09	4.237	2.90E-09
1.12	**35**	360	3.23	12.361	3.01E-09	2.676	6.91E-09
1.16	**29**	643	3.24	6.556	1.09E-09	5.014	3.17E-09
1.4	**37**	360	3.06	7.168	1.50E-09	6.132	1.01E-08
1.44	**33**	643	3.04	7.532	1.37E-09	6.293	1.17E-08
1.63	**19**	724	3.32	11.983	3.17E-09	9.617	1.85E-08
1.64	**31**	360	3.38	9.263	1.95E-09	5.668	1.77E-08
Average			3.26	8.688	1.88E-09	5.663	1.01E-08
Virginia Formation, 4 samples							
2.06	21	78	4.20	5.014	1.73E-09	2.706	3.61E-08
3.12	23	78	3.70	6.574	1.83E-09		
3.72	25	78	3.70	7.004	3.68E-09		
5.44	27	78	3.35	9.639	8.00E-09		
Average			3.74	7.058	3.81E-09	2.706	3.61E-08

Table 100. Model 1, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 1 of 4.

Model 1, Period A																		
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	8	4.0E-12	1.4E-12	6.6E-12	2.5E-12	1.9E-12	7.4E-13	2.7E-12	2.7E-13	7.0E-12	2.7E-12	1.0E-11	9.9E-13	6.0E-12	2.3E-12	8.6E-12	8.5E-13
3	0.22	7	3.7E-12	1.7E-12	6.2E-12	1.7E-12	1.2E-11	1.2E-11	4.0E-11	2.6E-12	2.6E-11	2.8E-11	9.2E-11	5.9E-12	6.1E-12	6.5E-12	2.1E-11	1.4E-12
5	0.40	8	8.4E-12	2.6E-12	1.4E-11	6.2E-12	7.6E-13	5.1E-13	1.5E-12	1.8E-13					3.6E-12	2.4E-12	7.1E-12	8.4E-13
7	0.41	10	3.3E-12	2.0E-12	6.4E-12	7.0E-13	2.4E-12	1.2E-12	4.1E-12	5.2E-13	3.0E-12	1.5E-12	5.3E-12	6.6E-13	2.3E-12	1.2E-12	4.0E-12	5.0E-13
9	0.51	11	4.7E-12	4.1E-12	1.4E-11	5.4E-13	1.4E-12	9.6E-13	2.9E-12	2.8E-13	8.4E-12	5.6E-12	1.7E-11	1.6E-12	2.7E-12	1.8E-12	5.4E-12	5.1E-13
11	0.54	32	5.0E-12	2.5E-12	1.5E-11	1.7E-12	2.8E-12	2.6E-12	1.3E-11	2.2E-13					6.6E-12	6.0E-12	3.1E-11	5.0E-13
13	0.57	7	7.2E-12	4.5E-12	1.6E-11	2.4E-12	1.0E-11	3.4E-12	1.6E-11	5.2E-12	4.2E-12	1.4E-12	6.4E-12	2.1E-12				
15	0.58	7	5.3E-12	3.7E-12	1.3E-11	1.3E-12	4.7E-12	2.8E-12	9.1E-12	1.1E-12	5.0E-11	2.9E-11	9.5E-11	1.1E-11	3.2E-11	1.9E-11	6.2E-11	7.4E-12
17	0.71	6	3.9E-12	1.6E-12	5.6E-12	1.8E-12	4.7E-12	2.0E-12	8.4E-12	2.9E-12	6.8E-12	2.9E-12	1.2E-11	4.1E-12	9.4E-12	4.0E-12	1.7E-11	5.8E-12
35	1.12	27	1.3E-11	6.4E-12	2.5E-11	1.1E-12	1.7E-12	9.4E-13	4.4E-12	2.7E-13	1.2E-11	6.7E-12	3.2E-11	1.9E-12	4.7E-12	2.6E-12	1.2E-11	7.4E-13
29	1.16	10	4.2E-12	1.3E-12	6.5E-12	2.3E-12	3.7E-12	2.0E-12	7.1E-12	3.9E-13	4.7E-12	2.6E-12	9.1E-12	5.0E-13				
37	1.40	18	7.1E-12	2.9E-12	1.4E-11	2.9E-12	4.8E-12	2.7E-12	1.0E-11	5.8E-13	1.0E-11	5.9E-12	2.3E-11	1.3E-12				
33	1.44	16	5.4E-12	4.7E-12	1.4E-11	1.2E-12	2.1E-12	6.3E-13	3.3E-12	1.2E-12	1.6E-11	4.7E-12	2.4E-11	8.6E-12	7.3E-12	2.2E-12	1.1E-11	4.0E-12
19	1.63	22	8.7E-12	5.2E-12	2.0E-11	3.5E-12	8.1E-12	4.4E-12	2.0E-11	3.5E-12	5.3E-11	2.9E-11	1.3E-10	2.3E-11	1.9E-11	1.0E-11	4.7E-11	8.3E-12
31	1.64	12	1.5E-11	1.1E-11	4.1E-11	3.1E-12	1.5E-11	8.2E-12	3.4E-11	2.8E-12	8.8E-11	4.8E-11	2.0E-10	1.6E-11	1.6E-11	9.0E-12	3.8E-11	3.1E-12
Summary			6.6E-12	3.7E-12	4.1E-11	5.4E-13	5.1E-12	3.0E-12	4.0E-11	1.8E-13	2.2E-11	1.3E-11	2.0E-10	5.0E-13	9.7E-12	5.6E-12	6.2E-11	5.0E-13

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.

Table 100. Model 1, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 2 of 4.

Model 1, Period B																		
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	189	7.7E-13	3.8E-13	3.0E-12	8.8E-14	5.3E-13	4.8E-13	2.1E-12	1.3E-13	2.0E-12	1.8E-12	7.6E-12	4.7E-13	1.7E-12	1.5E-12	6.5E-12	4.0E-13
3	0.22	191	7.0E-13	3.1E-13	2.1E-12	7.3E-14	2.6E-12	1.8E-12	1.1E-11	4.1E-13	6.0E-12	4.2E-12	2.4E-11	9.5E-13	1.4E-12	9.8E-13	5.7E-12	2.2E-13
5	0.40	133	2.7E-12	1.0E-12	6.4E-12	1.3E-12	1.9E-13	2.1E-13	1.5E-12	8.6E-14					9.1E-13	9.9E-13	6.9E-12	4.1E-13
7	0.41	52	6.1E-13	3.8E-13	2.4E-12	1.5E-13	1.3E-12	8.1E-13	4.3E-12	2.6E-13	1.6E-12	1.0E-12	5.4E-12	3.3E-13	1.2E-12	7.8E-13	4.1E-12	2.5E-13
9	0.51	33	7.5E-13	2.2E-13	1.3E-12	3.5E-13	8.2E-13	4.4E-13	2.4E-12	2.8E-13	4.7E-12	2.6E-12	1.4E-11	1.6E-12	1.5E-12	8.1E-13	4.4E-12	5.2E-13
11	0.54	37	1.9E-12	5.7E-13	3.2E-12	3.2E-13	8.1E-13	4.3E-13	2.0E-12	2.1E-13					1.9E-12	1.0E-12	4.7E-12	5.0E-13
13	0.57	45	1.5E-12	7.2E-13	3.5E-12	2.1E-13	5.8E-12	2.0E-12	1.4E-11	8.6E-13	2.4E-12	8.3E-13	5.7E-12	3.5E-13				
15	0.58	24	1.4E-12	4.8E-13	2.6E-12	1.9E-13	2.5E-12	8.0E-13	4.2E-12	3.5E-13	2.6E-11	8.4E-12	4.4E-11	3.7E-12	1.7E-11	5.5E-12	2.9E-11	2.4E-12
17	0.71	39	6.4E-13	4.2E-13	2.5E-12	1.5E-13	2.1E-12	1.0E-12	6.6E-12	3.6E-13	3.0E-12	1.5E-12	9.5E-12	5.2E-13	4.1E-12	2.1E-12	1.3E-11	7.2E-13
35	1.12	23	3.8E-12	7.9E-13	5.3E-12	2.4E-12	2.2E-12	5.8E-13	3.3E-12	1.1E-12	1.6E-11	4.1E-12	2.4E-11	7.7E-12	6.0E-12	1.6E-12	9.0E-12	2.9E-12
29	1.16	37	1.1E-12	7.7E-13	4.1E-12	4.3E-13	3.1E-12	1.2E-12	7.8E-12	1.9E-12	4.0E-12	1.6E-12	9.9E-12	2.4E-12				
37	1.40	35	2.1E-12	7.9E-13	4.8E-12	1.1E-12	7.1E-12	1.4E-12	9.2E-12	2.9E-12	1.5E-11	3.0E-12	2.0E-11	6.3E-12				
33	1.44	34	1.2E-12	3.9E-13	2.7E-12	5.1E-13	1.8E-12	4.7E-13	2.7E-12	9.4E-13	1.3E-11	3.5E-12	2.0E-11	7.0E-12	6.3E-12	1.7E-12	9.5E-12	3.3E-12
19	1.63	8	3.1E-12	2.6E-12	9.3E-12	1.5E-12	4.6E-12	5.8E-13	5.4E-12	3.6E-12	3.0E-11	3.8E-12	3.5E-11	2.3E-11	1.1E-11	1.4E-12	1.3E-11	8.4E-12
31	1.64	40	2.6E-12	6.5E-13	4.3E-12	1.7E-12	5.9E-12	1.5E-12	9.3E-12	2.8E-12	3.4E-11	8.5E-12	5.4E-11	1.6E-11	6.4E-12	1.6E-12	1.0E-11	3.1E-12
Summary			1.6E-12	7.0E-13	9.3E-12	7.3E-14	2.8E-12	9.2E-13	1.4E-11	8.6E-14	1.2E-11	3.5E-12	5.4E-11	3.3E-13	5.0E-12	1.7E-12	2.9E-11	2.2E-13

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.

Table 100. Model 1, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 3 of 4.

Model 1, Period C																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)				Orphan Ca (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	21	1.5E-12	2.4E-12	7.0E-12	0.0E+00	8.1E-14	6.3E-14	2.1E-13	0.0E+00					3.9E-13	3.0E-13	1.0E-12	0.0E+00	7.2E-13	1.8E-12	6.3E-12	0.0E+00
7	0.41	22	1.4E-12	7.6E-13	3.1E-12	2.3E-13	2.2E-12	1.8E-12	6.8E-12	0.0E+00	2.8E-12	2.2E-12	8.6E-12	0.0E+00	2.1E-12	1.7E-12	6.5E-12	0.0E+00	3.3E-13	8.4E-13	3.4E-12	0.0E+00
9	0.51	29	3.1E-12	1.4E-12	7.4E-12	4.6E-13	7.1E-13	5.8E-13	2.0E-12	0.0E+00	4.1E-12	3.4E-12	1.2E-11	0.0E+00	1.3E-12	1.1E-12	3.6E-12	0.0E+00	1.1E-12	2.2E-12	8.1E-12	0.0E+00
11	0.54	40	1.0E-13	3.0E-13	1.5E-12	0.0E+00	2.6E-13	2.6E-13	1.1E-12	0.0E+00					6.1E-13	6.1E-13	2.6E-12	0.0E+00	8.2E-14	2.8E-13	1.3E-12	0.0E+00
13	0.57	14	1.0E-12	6.9E-13	2.6E-12	1.6E-13	8.3E-12	4.8E-12	2.1E-11	3.1E-12	3.4E-12	2.0E-12	8.5E-12	1.3E-12					0.0E+00	0.0E+00	0.0E+00	0.0E+00
15	0.58	30	6.1E-13	7.6E-13	2.9E-12	0.0E+00	2.1E-12	9.2E-13	3.9E-12	6.5E-13	2.2E-11	9.7E-12	4.1E-11	6.8E-12	1.4E-11	6.3E-12	2.7E-11	4.4E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00
17	0.71	38	3.3E-12	3.1E-12	1.6E-11	0.0E+00	4.8E-12	4.1E-12	1.6E-11	1.4E-13	7.0E-12	6.0E-12	2.3E-11	2.0E-13	9.7E-12	8.3E-12	3.3E-11	2.8E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00
35	1.12	33	6.4E-12	6.1E-12	1.7E-11	0.0E+00	2.0E-12	7.6E-13	3.8E-12	7.8E-13	1.4E-11	5.4E-12	2.7E-11	5.5E-12	5.5E-12	2.1E-12	1.0E-11	2.1E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00
29	1.16	48	3.9E-12	3.1E-12	1.1E-11	0.0E+00	6.2E-12	3.1E-12	1.3E-11	0.0E+00	8.0E-12	4.0E-12	1.7E-11	0.0E+00					1.5E-14	1.0E-13	7.0E-13	0.0E+00
37	1.40	38	3.5E-12	2.6E-12	8.0E-12	0.0E+00	1.7E-11	7.5E-12	3.1E-11	3.4E-12	3.8E-11	1.6E-11	6.8E-11	7.5E-12					0.0E+00	0.0E+00	0.0E+00	0.0E+00
33	1.44	57	1.8E-11	1.6E-11	4.8E-11	0.0E+00	3.2E-12	1.8E-12	7.1E-12	5.5E-13	2.4E-11	1.3E-11	5.3E-11	4.1E-12	1.1E-11	6.2E-12	2.5E-11	1.9E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00
19	1.63	64	4.7E-12	4.4E-12	1.7E-11	0.0E+00	4.9E-12	1.5E-12	8.2E-12	3.1E-13	3.2E-11	1.0E-11	5.3E-11	2.0E-12	1.1E-11	3.6E-12	1.9E-11	7.3E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00
31	1.64	28	2.2E-12	1.5E-12	5.3E-12	2.6E-13	7.4E-12	1.9E-12	1.1E-11	4.0E-12	4.3E-11	1.1E-11	6.7E-11	2.4E-11	8.2E-12	2.1E-12	1.3E-11	4.4E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00
Summary			3.8E-12	3.3E-12	4.8E-11	0.0E+00	4.6E-12	2.2E-12	3.1E-11	0.0E+00	1.8E-11	7.6E-12	6.8E-11	0.0E+00	6.5E-12	3.2E-12	3.3E-11	0.0E+00	1.7E-13	4.0E-13	8.1E-12	0.0E+00

Note: For this period, model assumptions dictate plagioclase dissolution rates to be the same as Period B averages for each sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.

Table 100. Model 1, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 4 of 4.

Model 1, Period D																		
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	9	1.1E-12	2.6E-13	1.6E-12	6.4E-13	8.9E-14	9.3E-16	9.1E-14	8.8E-14					4.2E-13	4.4E-15	4.3E-13	4.2E-13
7	0.41	54	6.7E-13	2.1E-13	1.4E-12	3.6E-13	5.6E-13	5.6E-13	2.6E-12	2.4E-13	7.1E-13	7.1E-13	3.3E-12	3.1E-13	5.4E-13	5.4E-13	2.5E-12	2.3E-13
9	0.51	66	1.2E-12	5.6E-13	3.9E-12	5.6E-13	3.5E-13	3.9E-13	1.7E-12	1.3E-13	2.0E-12	2.2E-12	9.7E-12	7.7E-13	6.4E-13	7.1E-13	3.1E-12	2.4E-13
13	0.57	21	1.9E-12	4.1E-13	2.5E-12	1.2E-12	3.4E-12	1.7E-12	7.0E-12	4.4E-13	1.4E-12	7.0E-13	2.9E-12	1.8E-13				
15	0.58	80	1.2E-12	4.3E-13	2.8E-12	6.5E-13	3.9E-13	3.8E-13	1.7E-12	1.6E-13	4.1E-12	4.0E-12	1.8E-11	1.7E-12	2.7E-12	2.6E-12	1.2E-11	1.1E-12
17	0.71	58	9.1E-13	2.6E-13	1.6E-12	4.5E-13	6.7E-13	6.4E-13	2.8E-12	1.8E-13	9.7E-13	9.2E-13	4.1E-12	2.5E-13	1.3E-12	1.3E-12	5.7E-12	3.5E-13
35	1.12	3	2.6E-12	2.6E-13	2.8E-12	2.3E-12	1.9E-12	2.9E-13	2.2E-12	1.6E-12	1.4E-11	2.1E-12	1.6E-11	1.1E-11	5.2E-12	8.0E-13	6.0E-12	4.4E-12
29	1.16	47	8.8E-13	2.2E-13	1.5E-12	5.4E-13	7.5E-13	7.6E-13	3.4E-12	1.8E-13	9.5E-13	9.7E-13	4.3E-12	2.4E-13				
33	1.44	32	8.7E-13	1.5E-13	1.5E-12	5.8E-13	5.1E-13	2.3E-13	1.1E-12	9.2E-14	3.8E-12	1.7E-12	8.2E-12	6.9E-13	1.8E-12	7.9E-13	3.9E-12	3.2E-13
19	1.63	62	2.0E-12	8.7E-13	4.6E-12	9.9E-13	1.1E-12	8.6E-13	3.5E-12	1.3E-13	7.0E-12	5.6E-12	2.3E-11	8.5E-13	2.5E-12	2.0E-12	8.3E-12	3.1E-13
31	1.64	5	2.9E-12	4.2E-13	3.6E-12	2.6E-12	4.1E-12	8.6E-13	5.5E-12	3.2E-12	2.4E-11	5.0E-12	3.2E-11	1.8E-11	4.5E-12	9.5E-13	6.0E-12	3.5E-12
Summary			1.5E-12	3.7E-13	4.6E-12	3.6E-13	1.3E-12	6.1E-13	7.0E-12	8.8E-14	5.9E-12	2.4E-12	3.2E-11	1.8E-13	2.2E-12	1.1E-12	1.2E-11	2.3E-13

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.

Table 101. Model 2, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 1 of 4.

Model 2, Period A																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	8	4.4E-12	1.8E-12	6.5E-12	6.4E-13	1.9E-12	1.2E-12	3.4E-12	0.0E+00	3.5E-14	9.8E-14	2.8E-13	0.0E+00	1.3E-13	3.6E-13	1.0E-12	0.0E+00	1.1E-13	3.1E-13	8.8E-13	0.0E+00
3	0.22	7	7.7E-12	4.9E-12	1.6E-11	2.5E-12	1.2E-12	1.2E-12	3.0E-12	0.0E+00	3.6E-12	9.0E-12	2.4E-11	0.0E+00	8.2E-12	2.1E-11	5.5E-11	0.0E+00	1.9E-12	4.8E-12	1.3E-11	0.0E+00
5	0.40	8	1.9E-11	1.3E-11	4.2E-11	3.0E-12	5.5E-12	4.1E-12	1.3E-11	2.1E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	10	4.7E-12	2.4E-12	8.5E-12	7.8E-13	2.6E-12	1.9E-12	4.9E-12	0.0E+00	1.8E-14	4.8E-14	1.3E-13	0.0E+00	2.3E-14	6.1E-14	1.6E-13	0.0E+00	1.8E-14	4.6E-14	1.2E-13	0.0E+00
9	0.51	11	5.1E-12	5.0E-12	1.8E-11	8.9E-13	3.4E-12	3.7E-12	1.1E-11	1.2E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
11	0.54	32	8.9E-12	5.6E-12	2.1E-11	7.5E-13	3.1E-12	2.1E-12	9.8E-12	0.0E+00	4.5E-13	1.7E-12	8.4E-12	0.0E+00					1.1E-12	3.9E-12	2.0E-11	0.0E+00
13	0.57	7	4.2E-12	1.7E-12	7.3E-12	2.4E-12	3.2E-12	3.3E-12	9.4E-12	0.0E+00	6.1E-13	1.1E-12	2.9E-12	0.0E+00	2.5E-13	4.6E-13	1.2E-12	0.0E+00				
15	0.58	7	5.6E-12	3.7E-12	1.3E-11	1.5E-12	1.4E-12	1.5E-12	4.2E-12	0.0E+00	2.5E-13	6.7E-13	1.8E-12	0.0E+00	2.7E-12	7.1E-12	1.9E-11	0.0E+00	1.7E-12	4.6E-12	1.2E-11	0.0E+00
17	0.71	6	8.6E-12	2.2E-12	1.1E-11	5.7E-12	1.1E-12	1.2E-12	3.1E-12	0.0E+00	2.2E-13	4.2E-13	1.0E-12	0.0E+00	3.3E-13	6.1E-13	1.5E-12	0.0E+00	4.5E-13	8.4E-13	2.1E-12	0.0E+00
35	1.12	27	1.0E-11	8.6E-12	3.9E-11	1.4E-12	1.0E-11	5.8E-12	2.3E-11	0.0E+00	1.4E-13	5.1E-13	2.0E-12	0.0E+00	1.0E-12	3.6E-12	1.5E-11	0.0E+00	3.9E-13	1.4E-12	5.6E-12	0.0E+00
29	1.16	10	6.7E-12	3.4E-12	1.3E-11	7.0E-13	2.0E-12	1.4E-12	4.4E-12	3.7E-14	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
37	1.40	18	5.1E-12	2.8E-12	1.1E-11	5.8E-13	4.7E-12	3.5E-12	1.2E-11	0.0E+00	2.3E-13	6.2E-13	1.9E-12	0.0E+00	5.0E-13	1.4E-12	4.2E-12	0.0E+00				
33	1.44	16	2.2E-11	9.7E-12	4.3E-11	1.1E-11	4.1E-12	4.1E-12	1.2E-11	0.0E+00	3.6E-13	6.4E-13	1.5E-12	0.0E+00	2.6E-12	4.8E-12	1.1E-11	0.0E+00	1.2E-12	2.2E-12	5.2E-12	0.0E+00
19	1.63	22	1.8E-11	9.9E-12	4.1E-11	7.7E-12	3.9E-13	9.9E-13	4.2E-12	0.0E+00	3.6E-12	4.6E-12	1.8E-11	0.0E+00	2.4E-11	3.0E-11	1.2E-10	0.0E+00	8.5E-12	1.1E-11	4.2E-11	0.0E+00
31	1.64	12	1.6E-11	1.0E-11	4.0E-11	3.2E-12	4.2E-12	4.8E-12	1.3E-11	0.0E+00	1.2E-12	2.1E-12	6.0E-12	0.0E+00	6.9E-12	1.2E-11	3.5E-11	0.0E+00	1.3E-12	2.3E-12	6.6E-12	0.0E+00
Summary			9.7E-12	5.7E-12	4.3E-11	5.8E-13	3.3E-12	2.7E-12	2.3E-11	0.0E+00	7.1E-13	1.4E-12	2.4E-11	0.0E+00	3.6E-12	6.2E-12	1.2E-10	0.0E+00	1.4E-12	2.6E-12	4.2E-11	0.0E+00

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.

Table 101. Model 2, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 2 of 4.

Model 2, Period B																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	189	1.0E-12	7.6E-13	3.9E-12	1.8E-13	2.7E-13	3.2E-13	2.7E-12	0.0E+00	1.0E-13	2.6E-13	1.3E-12	0.0E+00	3.9E-13	9.7E-13	4.9E-12	0.0E+00	3.3E-13	8.3E-13	4.2E-12	0.0E+00
3	0.22	191	1.8E-12	9.1E-13	6.1E-12	2.2E-13	1.2E-13	2.4E-13	1.9E-12	0.0E+00	7.2E-13	1.2E-12	8.3E-12	0.0E+00	1.7E-12	2.8E-12	1.9E-11	0.0E+00	3.9E-13	6.5E-13	4.4E-12	0.0E+00
5	0.40	133	3.4E-12	3.3E-12	1.5E-11	1.5E-12	2.0E-12	1.1E-12	6.1E-12	0.0E+00	1.3E-14	8.2E-14	8.6E-13	0.0E+00					6.0E-14	3.9E-13	4.1E-12	0.0E+00
7	0.41	52	1.4E-12	8.5E-13	5.3E-12	3.9E-13	1.7E-13	3.4E-13	1.9E-12	0.0E+00	3.6E-13	5.7E-13	2.3E-12	0.0E+00	4.6E-13	7.2E-13	2.9E-12	0.0E+00	3.5E-13	5.4E-13	2.2E-12	0.0E+00
9	0.51	33	1.8E-12	5.4E-13	2.8E-12	9.1E-13	9.2E-14	1.5E-13	5.9E-13	0.0E+00	2.7E-13	4.8E-13	2.1E-12	0.0E+00	1.5E-12	2.8E-12	1.2E-11	0.0E+00	4.9E-13	8.8E-13	3.8E-12	0.0E+00
11	0.54	37	2.9E-12	1.7E-12	8.4E-12	7.5E-13	1.2E-12	5.9E-13	2.6E-12	0.0E+00	1.2E-14	5.3E-14	2.9E-13	0.0E+00					2.8E-14	1.2E-13	6.7E-13	0.0E+00
13	0.57	45	1.6E-12	7.1E-13	3.7E-12	2.2E-13	3.4E-14	1.2E-13	6.3E-13	0.0E+00	2.4E-12	1.7E-12	8.3E-12	0.0E+00	9.8E-13	7.0E-13	3.4E-12	0.0E+00				
15	0.58	24	2.0E-12	7.8E-13	3.9E-12	2.8E-13	4.0E-14	2.0E-13	9.7E-13	0.0E+00	1.1E-12	5.4E-13	2.0E-12	0.0E+00	1.1E-11	5.6E-12	2.1E-11	0.0E+00	7.5E-12	3.7E-12	1.4E-11	0.0E+00
17	0.71	39	2.0E-12	1.2E-12	7.8E-12	4.8E-13	1.7E-14	1.0E-13	6.4E-13	0.0E+00	1.2E-12	8.1E-13	3.4E-12	0.0E+00	1.7E-12	1.2E-12	5.0E-12	0.0E+00	2.4E-12	1.6E-12	6.9E-12	0.0E+00
35	1.12	23	9.4E-12	2.0E-12	1.3E-11	5.4E-12	4.9E-13	6.5E-13	1.9E-12	0.0E+00	2.8E-13	3.5E-13	1.3E-12	0.0E+00	2.0E-12	2.5E-12	9.2E-12	0.0E+00	7.7E-13	9.6E-13	3.5E-12	0.0E+00
29	1.16	37	2.9E-12	2.1E-12	1.1E-11	1.2E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	1.5E-12	6.5E-13	3.2E-12	2.2E-14	1.9E-12	8.3E-13	4.0E-12	2.8E-14				
37	1.40	35	3.4E-12	1.1E-12	5.5E-12	1.9E-12	9.6E-14	5.5E-13	3.2E-12	0.0E+00	3.6E-12	1.3E-12	6.8E-12	0.0E+00	8.0E-12	2.9E-12	1.5E-11	0.0E+00				
33	1.44	34	1.0E-11	2.6E-12	1.8E-11	4.7E-12	4.2E-14	2.2E-13	1.3E-12	0.0E+00	1.1E-12	4.9E-13	2.1E-12	0.0E+00	7.8E-12	3.7E-12	1.6E-11	0.0E+00	3.7E-12	1.7E-12	7.4E-12	0.0E+00
19	1.63	8	6.7E-12	5.8E-12	2.0E-11	3.2E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	3.1E-12	1.3E-12	4.4E-12	3.2E-13	2.0E-11	8.7E-12	2.9E-11	2.1E-12	7.4E-12	3.2E-12	1.0E-11	7.6E-13
31	1.64	40	3.6E-12	8.1E-13	5.5E-12	2.4E-12	6.3E-14	2.3E-13	1.0E-12	0.0E+00	2.8E-12	1.4E-12	5.0E-12	0.0E+00	1.6E-11	8.1E-12	2.9E-11	0.0E+00	3.0E-12	1.5E-12	5.5E-12	0.0E+00
Summary			3.6E-12	1.7E-12	2.0E-11	1.8E-13	3.0E-13	3.2E-13	6.1E-12	0.0E+00	1.2E-12	7.5E-13	8.3E-12	0.0E+00	5.7E-12	3.2E-12	2.9E-11	0.0E+00	2.2E-12	1.3E-12	1.4E-11	0.0E+00

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.

Table 101. Model 2, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 3 of 4.

Model 2, Period C																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	21	2.9E-12	1.8E-12	7.0E-12	1.5E-12	2.1E-12	8.9E-13	4.1E-12	7.6E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	22	2.8E-12	1.0E-12	4.3E-12	3.9E-13	1.9E-13	3.8E-13	1.3E-12	0.0E+00	1.2E-12	1.5E-12	5.6E-12	0.0E+00	1.6E-12	1.9E-12	7.0E-12	0.0E+00	1.2E-12	1.4E-12	5.4E-12	0.0E+00
9	0.51	29	4.5E-12	1.9E-12	9.4E-12	4.6E-13	5.0E-13	8.3E-13	2.7E-12	0.0E+00	2.9E-13	3.7E-13	1.4E-12	0.0E+00	1.7E-12	2.2E-12	7.8E-12	0.0E+00	5.3E-13	6.9E-13	2.5E-12	0.0E+00
11	0.54	40	1.0E-12	9.6E-13	3.9E-12	3.8E-13	1.1E-12	4.2E-13	2.1E-12	8.8E-14	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
13	0.57	14	2.6E-12	7.2E-13	4.1E-12	1.6E-12	3.1E-14	6.2E-14	1.6E-13	0.0E+00	5.0E-12	4.7E-12	1.7E-11	0.0E+00	2.0E-12	1.9E-12	7.1E-12	0.0E+00				
15	0.58	30	2.4E-12	7.2E-13	4.9E-12	1.4E-12	1.4E-13	2.6E-13	7.9E-13	0.0E+00	8.3E-13	8.0E-13	2.9E-12	0.0E+00	8.7E-12	8.4E-12	3.0E-11	0.0E+00	5.6E-12	5.4E-12	2.0E-11	0.0E+00
17	0.71	38	5.2E-12	3.2E-12	1.8E-11	2.4E-13	2.8E-14	1.1E-13	5.4E-13	0.0E+00	4.0E-12	4.1E-12	1.5E-11	0.0E+00	5.8E-12	5.9E-12	2.2E-11	0.0E+00	8.1E-12	8.2E-12	3.1E-11	0.0E+00
35	1.12	33	1.5E-11	6.0E-12	2.6E-11	6.1E-12	5.5E-13	7.1E-13	2.4E-12	0.0E+00	3.0E-13	5.1E-13	1.6E-12	0.0E+00	2.1E-12	3.6E-12	1.1E-11	0.0E+00	8.1E-13	1.4E-12	4.3E-12	0.0E+00
29	1.16	48	6.7E-12	3.1E-12	1.4E-11	2.9E-12	2.5E-14	1.7E-13	1.2E-12	0.0E+00	4.7E-12	3.1E-12	1.2E-11	0.0E+00	5.9E-12	3.9E-12	1.5E-11	0.0E+00				
37	1.40	38	7.0E-12	2.7E-12	1.2E-11	2.9E-12	1.7E-15	1.1E-14	6.5E-14	0.0E+00	1.4E-11	7.4E-12	2.7E-11	0.0E+00	3.0E-11	1.6E-11	6.0E-11	0.0E+00				
33	1.44	57	2.8E-11	1.6E-11	5.9E-11	7.4E-12	7.8E-15	3.7E-14	2.1E-13	0.0E+00	2.4E-12	1.7E-12	6.3E-12	0.0E+00	1.8E-11	1.3E-11	4.7E-11	0.0E+00	8.6E-12	6.1E-12	2.2E-11	0.0E+00
19	1.63	64	1.1E-11	4.4E-12	2.4E-11	4.6E-12	4.5E-14	3.1E-13	2.4E-12	0.0E+00	3.4E-12	1.5E-12	6.7E-12	0.0E+00	2.2E-11	9.8E-12	4.3E-11	0.0E+00	8.1E-12	3.6E-12	1.6E-11	0.0E+00
31	1.64	28	5.9E-12	1.5E-12	9.0E-12	4.0E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	4.2E-12	1.9E-12	8.2E-12	8.1E-13	2.4E-11	1.1E-11	4.8E-11	4.7E-12	4.6E-12	2.1E-12	9.0E-12	8.9E-13
Summary			7.4E-12	3.4E-12	5.9E-11	2.4E-13	3.6E-13	3.2E-13	4.1E-12	0.0E+00	3.1E-12	2.1E-12	2.7E-11	0.0E+00	1.1E-11	7.1E-12	6.0E-11	0.0E+00	3.7E-12	2.9E-12	3.1E-11	0.0E+00

Note: N=number of experimental observations used to model dissolution rates.
Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.

Table 101. Model 2, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 4 of 4.

Model 2, Period D																						
R#	%S	N	Aug (mol $m^{-2}s^{-1}$)				Plag (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	9	1.5E-12	1.6E-14	1.6E-12	1.5E-12	7.0E-13	2.6E-13	1.2E-12	2.9E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	54	8.2E-13	7.7E-13	3.1E-12	3.6E-13	4.0E-13	2.0E-13	8.9E-13	0.0E+00	1.8E-14	9.3E-14	5.9E-13	0.0E+00	2.2E-14	1.2E-13	7.5E-13	0.0E+00	1.7E-14	9.0E-14	5.7E-13	0.0E+00
9	0.51	66	1.1E-12	1.1E-12	5.4E-12	4.2E-13	7.7E-13	4.3E-13	2.7E-12	0.0E+00	1.3E-14	8.6E-14	6.6E-13	0.0E+00	7.8E-14	5.0E-13	3.8E-12	0.0E+00	2.5E-14	1.6E-13	1.2E-12	0.0E+00
13	0.57	21	1.5E-12	6.7E-13	2.6E-12	2.0E-13	5.1E-13	5.8E-13	2.1E-12	0.0E+00	2.5E-13	7.2E-13	3.2E-12	0.0E+00	1.0E-13	3.0E-13	1.3E-12	0.0E+00				
15	0.58	80	5.8E-13	6.3E-13	3.8E-12	2.3E-13	7.9E-13	3.9E-13	2.4E-12	0.0E+00	8.7E-15	6.9E-14	6.1E-13	0.0E+00	9.1E-14	7.2E-13	6.4E-12	0.0E+00	5.9E-14	4.7E-13	4.2E-12	0.0E+00
17	0.71	58	1.4E-12	1.3E-12	4.3E-12	3.9E-13	4.5E-13	2.1E-13	8.1E-13	0.0E+00	3.1E-14	1.6E-13	1.1E-12	0.0E+00	4.6E-14	2.3E-13	1.6E-12	0.0E+00	6.4E-14	3.1E-13	2.3E-12	0.0E+00
35	1.12	3	7.5E-12	7.3E-13	8.2E-12	6.7E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	4.2E-13	2.4E-13	6.9E-13	2.6E-13	3.0E-12	1.7E-12	4.9E-12	1.9E-12	1.1E-12	6.5E-13	1.9E-12	7.1E-13
29	1.16	47	1.1E-12	9.2E-13	4.1E-12	3.3E-13	4.5E-13	2.5E-13	9.6E-13	0.0E+00	1.1E-13	3.2E-13	1.8E-12	0.0E+00	1.4E-13	4.1E-13	2.2E-12	0.0E+00				
33	1.44	32	6.4E-12	2.6E-12	1.4E-11	1.2E-12	1.8E-13	2.0E-13	7.1E-13	0.0E+00	3.1E-14	5.7E-14	2.4E-13	0.0E+00	2.3E-13	4.2E-13	1.8E-12	0.0E+00	1.1E-13	2.0E-13	8.6E-13	0.0E+00
19	1.63	62	3.7E-12	2.3E-12	1.0E-11	5.9E-13	2.6E-13	3.2E-13	1.2E-12	0.0E+00	2.4E-13	4.1E-13	1.4E-12	0.0E+00	1.6E-12	2.7E-12	9.0E-12	0.0E+00	5.7E-13	9.7E-13	3.2E-12	0.0E+00
31	1.64	5	4.1E-12	6.1E-13	5.1E-12	3.6E-12	1.3E-14	3.0E-14	6.7E-14	0.0E+00	5.6E-13	4.0E-13	1.0E-12	0.0E+00	3.3E-12	2.3E-12	6.0E-12	0.0E+00	6.2E-13	4.4E-13	1.1E-12	0.0E+00
Summary			2.7E-12	1.1E-12	1.4E-11	2.0E-13	4.1E-13	2.6E-13	2.7E-12	0.0E+00	1.5E-13	2.3E-13	3.2E-12	0.0E+00	8.5E-13	9.4E-13	9.0E-12	0.0E+00	2.9E-13	3.7E-13	4.2E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.

Table 102. Model 3, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 1 of 4.

Model 3, Period A																						
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
N/A			N/A				N/A				N/A				N/A				N/A			
Summary			N/A				N/A				N/A				N/A				N/A			

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.

Table 102. Model 3, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 2 of 4.

Model 3, Period B																						
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	32	3.6E-13	1.6E-13	8.1E-13	1.5E-13	5.1E-13	3.8E-13	1.8E-12	0.0E+00	1.7E-13	3.4E-13	1.7E-12	0.0E+00	6.2E-13	1.3E-12	6.1E-12	0.0E+00	5.3E-13	1.1E-12	5.2E-12	0.0E+00
3	0.22	34	2.2E-13	8.9E-14	4.0E-13	1.1E-13	1.1E-12	4.7E-13	2.4E-12	2.1E-13	8.8E-13	1.2E-12	5.2E-12	0.0E+00	2.0E-12	2.7E-12	1.2E-11	0.0E+00	4.7E-13	6.2E-13	2.7E-12	0.0E+00
5	0.40	20	2.3E-13	1.2E-13	6.3E-13	1.8E-13	3.4E-12	3.2E-12	1.1E-11	1.5E-12	1.9E-14	6.4E-14	2.7E-13	0.0E+00					9.2E-14	3.0E-13	1.3E-12	0.0E+00
Summary			2.7E-13	1.2E-13	8.1E-13	1.1E-13	1.7E-12	1.3E-12	1.1E-11	0.0E+00	3.6E-13	5.2E-13	5.2E-12	0.0E+00	1.3E-12	2.0E-12	1.2E-11	0.0E+00	3.6E-13	6.7E-13	5.2E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.

Table 102. Model 3, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 3 of 4.

Model 3, Period C																						
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	7	6.4E-13	2.2E-13	9.9E-13	4.3E-13	2.8E-12	2.0E-12	7.0E-12	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	2	5.9E-13	1.1E-13	6.7E-13	5.1E-13	1.5E-12	8.7E-15	1.5E-12	1.4E-12	2.7E-12	1.5E-12	3.7E-12	1.7E-12	3.4E-12	1.8E-12	4.7E-12	2.1E-12	2.6E-12	1.4E-12	3.6E-12	1.6E-12
11	0.54	2	6.2E-13	1.3E-13	7.2E-13	5.3E-13	8.8E-13	6.9E-13	1.4E-12	3.9E-13	2.5E-13	3.5E-13	5.0E-13	0.0E+00					5.8E-13	8.2E-13	1.2E-12	0.0E+00
17	0.71	2	6.2E-13	1.2E-13	7.0E-13	5.3E-13	1.9E-12	3.9E-13	2.2E-12	1.7E-12	2.0E-12	9.0E-13	2.7E-12	1.4E-12	2.9E-12	1.3E-12	3.9E-12	2.0E-12	4.1E-12	1.8E-12	5.4E-12	2.8E-12
35	1.12	2	2.0E-12	1.3E-12	2.9E-12	1.2E-12	4.5E-12	6.4E-12	9.0E-12	0.0E+00	2.0E-12	3.8E-13	2.3E-12	1.7E-12	1.4E-11	2.7E-12	1.6E-11	1.2E-11	5.4E-12	1.0E-12	6.1E-12	4.7E-12
29	1.16	3	9.1E-13	3.0E-13	1.3E-12	6.8E-13	3.2E-12	2.0E-12	5.5E-12	1.9E-12	6.9E-12	5.8E-12	1.4E-11	2.7E-12	8.8E-12	7.5E-12	1.7E-11	3.4E-12				
37	1.40	3	1.5E-12	2.6E-13	1.7E-12	1.2E-12	5.3E-12	1.5E-12	6.2E-12	3.6E-12	1.8E-11	5.7E-12	2.5E-11	1.4E-11	4.0E-11	1.2E-11	5.4E-11	3.2E-11				
33	1.44	7	8.6E-13	4.0E-13	1.7E-12	5.6E-13	1.4E-11	6.5E-12	2.4E-11	6.7E-12	3.5E-12	1.8E-12	6.2E-12	9.6E-13	2.6E-11	1.3E-11	4.6E-11	7.2E-12	1.2E-11	6.4E-12	2.2E-11	3.4E-12
19	1.63	1	1.0E-12	N/A	N/A	N/A	7.8E-12	N/A	N/A	N/A	2.6E-12	N/A	N/A	N/A	1.7E-11	N/A	N/A	N/A	6.1E-12	N/A	N/A	N/A
31	1.64	1	1.1E-12	N/A	N/A	N/A	3.7E-12	N/A	N/A	N/A	4.9E-12	N/A	N/A	N/A	2.8E-11	N/A	N/A	N/A	5.3E-12	N/A	N/A	N/A
Summary			9.9E-13	3.5E-13	2.9E-12	4.3E-13	4.5E-12	2.4E-12	2.4E-11	0.0E+00	4.3E-12	2.1E-12	2.5E-11	0.0E+00	1.8E-11	6.5E-12	5.4E-11	2.0E-12	4.5E-12	1.9E-12	2.2E-11	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.

Table 102. Model 3, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 4 of 4.

Model 3, Period D																						
R#	%S	N	Plag (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)				Bio (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	3	2.6E-13	1.2E-13	4.0E-13	1.8E-13	1.5E-12	1.7E-14	1.6E-12	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00					0.0E+00	0.0E+00	0.0E+00	0.0E+00
7	0.41	14	4.8E-13	1.6E-13	8.5E-13	3.2E-13	3.6E-13	2.2E-13	8.3E-13	0.0E+00	3.3E-13	5.9E-13	1.6E-12	0.0E+00	4.2E-13	7.4E-13	2.0E-12	0.0E+00	3.2E-13	5.6E-13	1.5E-12	0.0E+00
9	0.51	17	6.9E-13	2.6E-13	1.4E-12	4.1E-13	6.0E-13	5.2E-13	2.1E-12	0.0E+00	2.0E-13	3.7E-13	1.2E-12	0.0E+00	1.2E-12	2.1E-12	7.2E-12	0.0E+00	3.7E-13	6.7E-13	2.3E-12	0.0E+00
13	0.57	2	1.1E-12	4.6E-13	1.4E-12	7.7E-13	6.0E-13	1.5E-13	7.1E-13	4.9E-13	2.2E-12	8.7E-13	2.8E-12	1.6E-12	9.0E-13	3.5E-13	1.2E-12	6.5E-13				
15	0.58	18	6.7E-13	5.6E-13	2.9E-12	3.6E-13	3.2E-13	2.0E-13	9.3E-13	0.0E+00	2.2E-13	5.6E-13	2.0E-12	0.0E+00	2.4E-12	5.9E-12	2.1E-11	0.0E+00	1.5E-12	3.8E-12	1.4E-11	0.0E+00
17	0.71	16	5.5E-13	1.3E-13	7.9E-13	3.9E-13	9.9E-13	7.6E-13	2.7E-12	0.0E+00	3.2E-13	6.0E-13	2.1E-12	0.0E+00	4.6E-13	8.6E-13	3.1E-12	0.0E+00	6.4E-13	1.2E-12	4.3E-12	0.0E+00
35	1.12	1	9.4E-13	N/A	N/A	N/A	4.0E-12	N/A	N/A	N/A	8.1E-13	N/A	N/A	N/A	5.7E-12	N/A	N/A	N/A	2.2E-12	N/A	N/A	N/A
29	1.16	15	5.5E-13	1.3E-13	8.5E-13	3.5E-13	5.9E-13	2.9E-13	1.1E-12	3.3E-13	3.4E-13	5.8E-13	1.7E-12	0.0E+00	4.4E-13	7.4E-13	2.2E-12	0.0E+00				
33	1.44	11	5.2E-13	1.5E-13	8.7E-13	3.6E-13	2.6E-12	1.4E-12	4.3E-12	0.0E+00	1.8E-13	2.1E-13	6.8E-13	0.0E+00	1.3E-12	1.6E-12	5.1E-12	0.0E+00	6.3E-13	7.5E-13	2.4E-12	0.0E+00
19	1.63	18	6.9E-13	2.0E-13	1.2E-12	4.7E-13	2.5E-12	1.8E-12	6.4E-12	5.9E-13	4.3E-13	6.7E-13	1.8E-12	0.0E+00	2.8E-12	4.4E-12	1.2E-11	0.0E+00	1.0E-12	1.6E-12	4.3E-12	0.0E+00
31	1.64	2	1.1E-12	1.2E-13	1.2E-12	1.0E-12	2.3E-12	4.1E-14	2.4E-12	2.3E-12	2.1E-12	5.8E-15	2.1E-12	2.1E-12	1.2E-11	3.4E-14	1.2E-11	1.2E-11	2.3E-12	6.4E-15	2.3E-12	2.3E-12
Summary			6.9E-13	2.3E-13	2.9E-12	1.8E-13	1.5E-12	5.3E-13	6.4E-12	0.0E+00	6.5E-13	4.4E-13	2.8E-12	0.0E+00	2.8E-12	1.9E-12	2.1E-11	0.0E+00	1.0E-12	1.1E-12	1.4E-11	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.

Table 103. Model 3, Periods A-D Orphan Calcium Rates for all Dunka blast hole samples (excludes R39-44).

Model 3, Orphan Calcium (mol $m^{-2}s^{-1}$)																	
R#	%S	Period A				Period B				Period C				Period D			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18					1.7E-13	3.1E-13	1.2E-12	0.0E+00								
3	0.22					3.1E-14	1.7E-13	9.7E-13	0.0E+00								
5	0.4					4.6E-12	2.4E-12	9.0E-12	0.0E+00	6.5E-12	3.2E-12	1.2E-11	2.7E-12	1.6E-12	1.3E-13	1.7E-12	1.5E-12
7	0.41									0.0E+00	0.0E+00	0.0E+00	0.0E+00	2.3E-13	3.6E-13	1.2E-12	0.0E+00
9	0.51													3.2E-13	4.2E-13	1.4E-12	0.0E+00
11	0.54									9.5E-13	1.3E-12	1.9E-12	0.0E+00				
13	0.57													0.0E+00	0.0E+00	0.0E+00	0.0E+00
15	0.58													8.0E-13	5.9E-13	1.6E-12	0.0E+00
17	0.71									0.0E+00	0.0E+00	0.0E+00	0.0E+00	2.7E-13	3.2E-13	1.0E-12	0.0E+00
35	1.12									0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00		0.0E+00	0.0E+00
29	1.16									0.0E+00	0.0E+00	0.0E+00	0.0E+00	4.3E-13	5.0E-13	1.5E-12	0.0E+00
37	1.4									0.0E+00	0.0E+00	0.0E+00	0.0E+00				
33	1.44									0.0E+00	0.0E+00	0.0E+00	0.0E+00	1.3E-13	4.1E-13	1.4E-12	0.0E+00
19	1.63									0.0E+00		0.0E+00	0.0E+00	1.2E-13	3.0E-13	1.1E-12	0.0E+00
31	1.64									0.0E+00		0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
Summary		N/A	N/A	N/A	N/A	1.6E-12	9.6E-13	9.0E-12	0.0E+00	7.4E-13	5.7E-13	1.2E-11	0.0E+00	3.5E-13	3.0E-13	1.7E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.

Table 104. Model 4, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 1 of 4.

Model 4, Period A																					
R#	%S	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
N/A			N/A				N/A				N/A				N/A				N/A			
Summary			N/A				N/A				N/A				N/A				N/A			

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table AZ).

Table 104. Model 4, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 2 of 4.

Model 4, Period B																						
R#	%S	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
1	0.18	32	6.0E-12	5.3E-12	1.7E-11	7.0E-13					5.1E-13	3.8E-13	1.8E-12	0.0E+00	3.1E-15	1.8E-14	1.0E-13	0.0E+00	1.2E-14	6.5E-14	3.7E-13	0.0E+00
3	0.22	33	2.3E-12	2.4E-12	8.6E-12	2.0E-13	1.0E-11	1.1E-11	3.8E-11	8.9E-13	1.1E-12	4.7E-13	2.4E-12	2.1E-13	2.2E-14	1.3E-13	7.3E-13	0.0E+00	5.0E-14	2.9E-13	1.7E-12	0.0E+00
5	0.40	18	5.6E-12	4.2E-12	1.5E-11	1.5E-12	1.6E-11	1.2E-11	4.5E-11	4.3E-12	3.6E-12	3.3E-12	1.1E-11	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
Summary			4.6E-12	4.0E-12	1.7E-11	2.0E-13	1.3E-11	1.1E-11	4.5E-11	8.9E-13	1.7E-12	1.3E-12	1.1E-11	0.0E+00	8.4E-15	4.8E-14	7.3E-13	0.0E+00	3.1E-14	1.8E-13	1.7E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table AZ).

Table 104. Model 4, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 3 of 4.

Model 4, Period C																						
R#	%S	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	7	1.2E-12	9.9E-13	2.5E-12	3.2E-13	3.5E-12	2.9E-12	7.2E-12	9.4E-13	2.8E-12	2.0E-12	7.0E-12	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
7	0.41	2	4.5E-12	1.8E-12	5.8E-12	3.2E-12	6.8E-11	2.8E-11	8.7E-11	4.8E-11	1.5E-12	8.7E-15	1.5E-12	1.4E-12	2.0E-13	2.8E-13	3.9E-13	0.0E+00	2.5E-13	3.5E-13	5.0E-13	0.0E+00
11	0.54	2	2.5E-12	4.1E-13	2.8E-12	2.2E-12					8.8E-13	6.9E-13	1.4E-12	3.9E-13	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
17	0.71	2	5.9E-12	1.8E-12	7.2E-12	4.6E-12	6.1E-11	1.9E-11	7.5E-11	4.8E-11	1.9E-12	3.9E-13	2.2E-12	1.7E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
35	1.12	2	6.4E-12	2.3E-12	8.0E-12	4.8E-12	5.9E-12	2.1E-12	7.4E-12	4.4E-12	4.5E-12	6.4E-12	9.0E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
29	1.16	3									3.2E-12	2.0E-12	5.5E-12	1.9E-12	6.9E-12	5.8E-12	1.4E-11	2.7E-12	8.8E-12	7.5E-12	1.7E-11	3.4E-12
37	1.40	3					7.2E-11	3.3E-11	9.7E-11	3.5E-11	5.3E-12	1.5E-12	6.2E-12	3.6E-12	1.8E-11	5.7E-12	2.5E-11	1.4E-11	4.0E-11	1.2E-11	5.4E-11	3.2E-11
33	1.44	7	1.0E-11	6.9E-12	2.4E-11	2.5E-12	2.4E-11	1.6E-11	5.5E-11	6.0E-12	1.4E-11	6.5E-12	2.4E-11	6.7E-12	9.0E-13	8.8E-13	2.0E-12	0.0E+00	6.7E-12	6.5E-12	1.5E-11	0.0E+00
19	1.63	1	7.1E-12	N/A	N/A	N/A	1.2E-11	N/A	N/A	N/A	7.8E-12	N/A	N/A	N/A	0.0E+00	N/A	N/A	N/A	0.0E+00	N/A	N/A	N/A
31	1.64	1	3.1E-12	N/A	N/A	N/A	1.0E-11	N/A	N/A	N/A	3.7E-12	N/A	N/A	N/A	2.0E-12	N/A	N/A	N/A	1.2E-11	N/A	N/A	N/A
Summary			5.1E-12	2.4E-12	2.4E-11	3.2E-13	3.2E-11	1.7E-11	9.7E-11	9.4E-13	4.5E-12	2.4E-12	2.4E-11	0.0E+00	2.8E-12	1.6E-12	2.5E-11	0.0E+00	8.5E-12	4.5E-12	5.4E-11	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table AZ).

Table 104. Model 4, Periods A-D Silicate Mineral Dissolution Rates for all Dunka blast hole samples (excludes R39-44). Page 4 of 4.

Model 4, Period D																						
R#	%S	N	Bio (mol $m^{-2}s^{-1}$)				K-spar (mol $m^{-2}s^{-1}$)				Aug (mol $m^{-2}s^{-1}$)				Hyp (mol $m^{-2}s^{-1}$)				Ol (mol $m^{-2}s^{-1}$)			
			Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
5	0.40	2	4.0E-13	3.4E-15	4.1E-13	4.0E-13	1.2E-12	1.0E-14	1.2E-12	1.2E-12	1.6E-12	1.3E-14	1.6E-12	1.5E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00				
7	0.41	12	2.8E-12	2.0E-12	7.5E-12	8.1E-13	4.1E-11	3.0E-11	1.1E-10	1.2E-11	3.9E-13	2.1E-13	8.3E-13	0.0E+00	2.0E-14	7.0E-14	2.4E-13	0.0E+00	2.6E-14	8.8E-14	3.1E-13	0.0E+00
9	0.51	16	2.3E-12	1.8E-12	6.9E-12	2.2E-13	6.7E-12	5.2E-12	2.0E-11	6.3E-13	6.1E-13	5.4E-13	2.1E-12	0.0E+00	2.8E-14	1.1E-13	4.5E-13	0.0E+00	1.6E-13	6.5E-13	2.6E-12	0.0E+00
13	0.57	2					1.8E-11	9.4E-12	2.5E-11	1.2E-11	6.0E-13	1.5E-13	7.1E-13	4.9E-13	2.2E-12	8.7E-13	2.8E-12	1.6E-12	9.0E-13	3.5E-13	1.2E-12	6.5E-13
15	0.58	17	1.1E-11	1.3E-11	4.1E-11	6.9E-13	1.1E-11	1.2E-11	3.8E-11	6.4E-13	3.2E-13	2.1E-13	9.3E-13	0.0E+00	3.5E-14	1.4E-13	6.0E-13	0.0E+00	3.7E-13	1.5E-12	6.3E-12	0.0E+00
17	0.71	16	7.4E-12	7.8E-12	2.3E-11	9.9E-13	7.7E-11	8.1E-11	2.3E-10	1.0E-11	9.9E-13	7.6E-13	2.7E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
35	1.12	1	6.8E-12	N/A	N/A	N/A	6.3E-12	N/A	N/A	N/A	4.0E-12	N/A	N/A	N/A	0.0E+00	N/A	N/A	N/A	0.0E+00	N/A	N/A	N/A
29	1.16	15									5.9E-13	2.9E-13	1.1E-12	3.3E-13	3.4E-13	5.8E-13	1.7E-12	0.0E+00	4.4E-13	7.4E-13	2.2E-12	0.0E+00
33	1.44	9	1.4E-12	6.0E-13	2.5E-12	7.2E-13	3.3E-12	1.4E-12	5.9E-12	1.7E-12	2.7E-12	1.5E-12	4.3E-12	0.0E+00	5.6E-14	1.2E-13	3.2E-13	0.0E+00	4.2E-13	8.6E-13	2.4E-12	0.0E+00
19	1.63	17	6.9E-12	6.3E-12	2.4E-11	1.6E-12	1.2E-11	1.1E-11	4.1E-11	2.7E-12	2.3E-12	1.6E-12	6.4E-12	5.9E-13	6.0E-14	2.4E-13	1.0E-12	0.0E+00	3.9E-13	1.6E-12	6.6E-12	0.0E+00
31	1.64	2	4.6E-12	7.5E-13	5.1E-12	4.1E-12	1.5E-11	2.4E-12	1.6E-11	1.3E-11	2.3E-12	4.1E-14	2.4E-12	2.3E-12	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00	0.0E+00
Summary			4.9E-12	4.0E-12	4.1E-11	2.2E-13	1.9E-11	1.7E-11	2.3E-10	6.3E-13	1.5E-12	5.3E-13	6.4E-12	0.0E+00	2.5E-13	2.1E-13	2.8E-12	0.0E+00	2.7E-13	6.4E-13	6.6E-12	0.0E+00

Note: Zeroes indicate a calculated absence of dissolution. Blank cells indicate there was none of that mineral present in the sample.
Note: N=number of experimental observations used to model dissolution rates.
Note: Plagioclase dissolution calculations in Model 4 are the same as Model 3 (see Table AZ).

Table 105. Comparison of silicate mineral dissolution model statistics (values reported in Tables 100-104) (excludes R39-44).

Period	Model	Plagioclase (mol $m^{-2}s^{-1}$)				Augite (mol $m^{-2}s^{-1}$)				Hypersthene (mol $m^{-2}s^{-1}$)				Olivine (mol $m^{-2}s^{-1}$)				Biotite (mol $m^{-2}s^{-1}$)			
		Ave	S.D.	Max	Min[1]	Ave	S.D.	Max	Min[1]	Ave	S.D.	Max	Min[1]	Ave	S.D.	Max	Min[1]	Ave	S.D.	Max	Min[1]
A	1	6.6E-12	3.7E-12	4.1E-11	5.4E-13					5.1E-12	3.0E-12	4.0E-11	1.8E-13	2.2E-11	1.3E-11	2.0E-10	5.0E-13	9.7E-12	5.6E-12	6.2E-11	5.0E-13
	2	3.3E-12	2.7E-12	2.3E-11	0.0E+00	9.7E-12	5.7E-12	4.3E-11	5.8E-13	7.1E-13	1.4E-12	2.4E-11	0.0E+00	3.6E-12	6.2E-12	1.2E-10	0.0E+00	1.4E-12	2.6E-12	4.2E-11	0.0E+00
	3	N/A	N/A	N/A	N/A	N/A`	N/A`	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	4									N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	max /min	2.0	1.4	1.8	N/A	1.0	1.0	1.0	1.0	7.1	2.1	1.7	N/A	6.3	2.1	1.7	N/A	7.0	2.1	1.5	N/A
B	1	1.6E-12	7.0E-13	9.3E-12	7.3E-14					2.8E-12	9.2E-13	1.4E-11	8.6E-14	1.2E-11	3.5E-12	5.4E-11	3.3E-13	5.0E-12	1.7E-12	2.9E-11	2.2E-13
	2	3.0E-13	3.2E-13	6.1E-12	0.0E+00	3.6E-12	1.7E-12	2.0E-11	1.8E-13	1.2E-12	7.5E-13	8.3E-12	0.0E+00	5.7E-12	3.2E-12	2.9E-11	0.0E+00	2.2E-12	1.3E-12	1.4E-11	0.0E+00
	3	2.7E-13	1.2E-13	8.1E-13	1.1E-13	1.7E-12	1.3E-12	1.1E-11	0.0E+00	3.6E-13	5.2E-13	5.2E-12	0.0E+00	1.3E-12	2.0E-12	1.2E-11	0.0E+00	3.6E-13	6.7E-13	5.2E-12	0.0E+00
	4									8.4E-15	4.8E-14	7.3E-13	0.0E+00	3.1E-14	1.8E-13	1.7E-12	0.0E+00	4.6E-12	4.0E-12	1.7E-11	2.0E-13
	max /min	6.1	5.6	11.4	N/A	2.2	1.2	1.8	N/A	328.5	19.1	19.2	N/A	395.1	19.5	32.7	N/A	13.7	6.0	5.5	N/A
C	1					3.8E-12	3.3E-12	4.8E-11	0.0E+00	4.6E-12	2.2E-12	3.1E-11	0.0E+00	1.8E-11	7.6E-12	6.8E-11	0.0E+00	6.5E-12	3.2E-12	3.3E-11	0.0E+00
	2	3.6E-13	3.2E-13	4.1E-12	0.0E+00	7.4E-12	3.4E-12	5.9E-11	2.4E-13	3.1E-12	2.1E-12	2.7E-11	0.0E+00	1.1E-11	7.1E-12	6.0E-11	0.0E+00	3.7E-12	2.9E-12	3.1E-11	0.0E+00
	3	9.9E-13	3.5E-13	2.9E-12	4.3E-13	4.5E-12	2.4E-12	2.4E-11	0.0E+00	4.3E-12	2.1E-12	2.5E-11	0.0E+00	1.8E-11	6.5E-12	5.4E-11	2.0E-12	4.5E-12	1.9E-12	2.2E-11	0.0E+00
	4									2.8E-12	1.6E-12	2.5E-11	0.0E+00	8.5E-12	4.5E-12	5.4E-11	0.0E+00	5.1E-12	2.4E-12	2.4E-11	3.2E-13
	max /min	2.7	1.1	1.4	N/A	1.9	1.4	2.5	N/A	1.6	1.4	1.2	N/A	2.1	1.7	1.2	N/A	1.7	1.7	1.5	N/A
D	1	1.5E-12	3.7E-13	4.6E-12	3.6E-13					1.3E-12	6.1E-13	7.0E-12	8.8E-14	5.9E-12	2.4E-12	3.2E-11	1.8E-13	2.2E-12	1.1E-12	1.2E-11	2.3E-13
	2	4.1E-13	2.6E-13	2.7E-12	0.0E+00	2.7E-12	1.1E-12	1.4E-11	2.0E-13	1.5E-13	2.3E-13	3.2E-12	0.0E+00	8.5E-13	9.4E-13	9.0E-12	0.0E+00	2.9E-13	3.7E-13	4.2E-12	0.0E+00
	3	6.9E-13	2.3E-13	2.9E-12	1.8E-13	1.5E-12	5.3E-13	6.4E-12	0.0E+00	6.5E-13	4.4E-13	2.8E-12	0.0E+00	2.8E-12	1.9E-12	2.1E-11	0.0E+00	1.0E-12	1.1E-12	1.4E-11	0.0E+00
	4									2.5E-13	2.1E-13	2.8E-12	0.0E+00	2.7E-13	6.4E-13	6.6E-12	0.0E+00	4.9E-12	4.0E-12	4.1E-11	2.2E-13
	max /min	3.5	1.6	1.7	N/A	1.8	2.0	2.1	N/A	8.3	2.9	2.5	N/A	21.6	3.7	4.9	N/A	17.0	11.0	9.8	N/A

[1] No range in minimum values reported due to zero values calculated
Note: Plagioclase dissolution calculations in Model 1, Period C are the same as Period B.
Note: Plagioclase and augite dissolution calculations in Model 4 are the same as Model 3.

Table 106. Total number of calculated mineral dissolution rates for each model during the specified rate period and the total number of cases where the models calculated an absence of dissolution (zero value). Associated percentage is the comparison of zero values cases to total number of cases (excludes R39-44).

Sorted by model, than period

Model	Period	Plagioclase			Augite			Hypersthene			Olivine			Biotite		
		Total n	zero	%	Total n	zero	%	Total n	zero	%	Total n	zero	%	Total n	zero	%
1	A	201	0	0%				195	0	0%	158	0	0%	162	0	0%
	B	920	0	0%				911	0	0%	746	0	0%	797	0	0%
	C				462	85	18%	460	22	5%	399	13	3%	360	21	6%
	D	437	0	0%				437	0	0%	428	0	0%	368	0	0%
2	A	181	42	23%	201	0	0%	181	139	77%	147	108	73%	149	111	74%
	B	902	447	50%	920	0	0%	902	455	50%	738	300	41%	788	449	57%
	C	460	342	74%	462	0	0%	460	118	26%	399	57	14%	360	113	31%
	D	436	75	17%	437	0	0%	436	361	83%	427	352	82%	368	307	83%
3	A	0	0	N/A	0	0	N/A	0	0	N/A	0	0	N/A	0	0	N/A
	B	86	0	0%	86	1	1%	85	30	35%	66	13	20%	85	30	35%
	C	30	0	0%	30	1	3%	30	8	27%	21	0	0%	24	8	33%
	D	117	0	0%	117	6	5%	117	58	50%	114	55	48%	100	49	49%
4	A	0	0	N/A	0	0	N/A	0	0	N/A	0	0	N/A	0	0	N/A
	B	86	0	0%	86	1	1%	82	80	98%	65	63	97%	83	0	0%
	C	30	0	0%	30	1	3%	30	18	60%	21	9	43%	24	0	0%
	D	117	0	0%	117	6	5%	109	94	86%	107	92	86%	92	0	0%

Sorted by period, than model

Model	Period	Plagioclase			Augite			Hypersthene			Olivine			Biotite		
		Total n	zero	%	Total n	zero	%	Total n	zero	%	Total n	zero	%	Total n	zero	%
1	A	201	0	0%				195	0	0%	158	0	0%	162	0	0%
2		181	42	23%	201	0	0%	181	139	77%	147	108	73%	149	111	74%
3		0	0	N/A	0	0	N/A	0	0	N/A	0	0	N/A	0	0	N/A
4		0	0	N/A	0	0	N/A	0	0	N/A	0	0	N/A	0	0	N/A
1	B	920	0	0%				911	0	0%	746	0	0%	797	0	0%
2		902	447	50%	920	0	0%	902	455	50%	738	300	41%	788	449	57%
3		86	0	0%	86	1	1%	85	30	35%	66	13	20%	85	30	35%
4		86	0	0%	86	1	1%	82	80	98%	65	63	97%	83	0	0%
1	C				462	85	18%	460	22	5%	399	13	3%	360	21	6%
2		460	342	74%	462	0	0%	460	118	26%	399	57	14%	360	113	31%
3		30	0	0%	30	1	3%	30	8	27%	21	0	0%	24	8	33%
4		30	0	0%	30	1	3%	30	18	60%	21	9	43%	24	0	0%
1	D	437	0	0%				437	0	0%	428	0	0%	368	0	0%
2		436	75	17%	437	0	0%	436	361	83%	427	352	82%	368	307	83%
3		117	0	0%	117	6	5%	117	58	50%	114	55	48%	100	49	49%
4		117	0	0%	117	6	5%	109	94	86%	107	92	86%	92	0	0%

Table 107. Calculated pyrrhotite oxidation rates for all Dunka blast hole samples (excludes R39-44).

R#	%S	Period A				Period B				Period C				Period D			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
Group I																	
1	0.18	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A								
3	0.22	1.25E-08	6.55E-09	3.47E-08	3.55E-09	3.10E-09	2.55E-09	1.12E-08	2.25E-10								
Average		1.25E-08	6.55E-09	3.47E-08	3.55E-09	3.10E-09	2.55E-09	1.12E-08	2.25E-10								
Group II																	
5	0.4	3.78E-10	1.47E-10	8.34E-10	1.50E-10	1.12E-10	4.93E-11	3.51E-10	3.39E-11	5.62E-10	3.34E-10	1.23E-09	1.99E-10	1.30E-10	3.63E-11	2.09E-10	8.92E-11
7	0.41	1.62E-10	1.35E-10	5.12E-10	2.12E-11	1.29E-10	7.49E-11	3.03E-10	4.06E-11	5.08E-10	9.97E-11	6.83E-10	3.41E-10	1.52E-10	1.36E-10	5.90E-10	3.17E-11
9	0.51	2.28E-10	1.42E-10	5.81E-10	3.89E-11	1.04E-10	4.63E-11	1.86E-10	3.76E-11	6.37E-10	1.15E-10	9.56E-10	4.79E-10	2.04E-10	1.51E-10	5.65E-10	4.19E-11
11	0.54	2.37E-10	9.74E-11	4.97E-10	8.59E-11	1.61E-10	8.10E-11	4.02E-10	4.28E-11	2.23E-10	7.50E-11	4.25E-10	1.10E-10				
13	0.57	2.65E-10	7.20E-11	4.00E-10	1.35E-10	1.74E-10	9.00E-11	5.10E-10	2.87E-11	4.46E-10	1.10E-10	7.15E-10	3.17E-10	2.96E-10	4.97E-11	3.66E-10	2.17E-10
15	0.58	2.32E-10	1.89E-10	9.57E-10	2.86E-11	2.48E-10	9.54E-11	5.61E-10	1.27E-10	4.18E-10	8.96E-11	6.65E-10	3.01E-10	1.43E-10	9.46E-11	3.60E-10	3.15E-11
Average		2.51E-10	1.30E-10	9.57E-10	2.12E-11	1.55E-10	7.28E-11	5.61E-10	2.87E-11	4.66E-10	1.37E-10	1.23E-09	1.10E-10	1.85E-10	9.34E-11	5.90E-10	3.15E-11
Group III																	
17	0.71	1.36E-10	5.69E-11	2.54E-10	2.06E-11	9.32E-11	3.64E-11	1.94E-10	2.04E-11	5.01E-10	4.33E-10	1.67E-09	9.74E-11	8.17E-11	4.52E-11	2.19E-10	2.84E-11
35	1.12	2.61E-10		8.73E-10	9.22E-11	2.57E-10		3.22E-10	1.12E-10	4.86E-10	2.64E-10	9.05E-10	1.39E-10	1.75E-10	1.46E-11	1.86E-10	1.59E-10
29	1.16	1.52E-10		4.03E-10	7.33E-11	1.12E-10		2.07E-10	5.12E-11	4.23E-10	2.80E-10	1.07E-09	6.57E-11	4.00E-11	2.10E-11	1.00E-10	1.70E-11
37	1.4	1.40E-10		2.53E-10	8.21E-11	1.09E-10		2.74E-10	5.23E-11	5.88E-10	3.02E-10	1.11E-09	1.91E-10				
33	1.44	1.38E-10	1.36E-10	3.92E-10	6.57E-11	1.74E-10	5.57E-11	3.58E-10	8.69E-11	3.91E-10	2.64E-10	9.52E-10	5.15E-11	4.38E-11	1.46E-11	7.51E-11	2.52E-11
19	1.63	1.33E-10	8.07E-11	3.36E-10	4.51E-11	9.33E-11	3.84E-11	1.43E-10	5.32E-11	3.45E-10	2.80E-10	8.14E-10	8.77E-11	4.23E-11	2.10E-11	1.15E-10	9.54E-12
31	1.64	3.34E-10	5.65E-11	9.80E-10	1.65E-10	1.69E-10	4.02E-11	2.46E-10	9.86E-11	3.60E-10	3.02E-10	7.21E-10	1.44E-10	1.61E-10	1.61E-11	1.85E-10	1.43E-10
Average		1.85E-10	8.25E-11	9.80E-10	2.06E-11	1.44E-10	4.27E-11	3.58E-10	2.04E-11	4.42E-10	3.04E-10	1.67E-09	5.15E-11	9.07E-11	2.21E-11	2.19E-10	9.54E-12
Summary[2]		2.15E-10	1.11E-10	9.80E-10	2.06E-11	1.49E-10	6.08E-11	5.61E-10	2.04E-11	4.53E-10	2.27E-10	1.67E-09	5.15E-11	1.34E-10	5.45E-11	5.90E-10	9.54E-12

[1] Reactor 1 calculated to have pyrrhotite content less than zero
[2] Overall calculations does not include group I due to the low amount of pyrrhotite calculated for reactor 3

Table 108. Calculated sulfide oxidation rates (all sulfide minerals combined) for all Dunka blast hole samples (excludes R39-44).

R#	%S	Period A				Period B				Period C				Period D			
		Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min	Ave	S.D.	Max	Min
Group I																	
1	0.18	4.81E-10	2.82E-10	1.21E-09	1.19E-10	9.48E-11	9.23E-11	4.20E-10	7.70E-12								
3	0.22	4.17E-10	2.18E-10	1.16E-09	1.18E-10	1.03E-10	8.51E-11	3.74E-10	7.51E-12								
Average		4.49E-10	2.50E-10	1.21E-09	1.18E-10	9.91E-11	8.87E-11	4.20E-10	7.51E-12								
Group II																	
5	0.4	3.41E-10	1.32E-10	7.52E-10	1.35E-10	1.01E-10	4.44E-11	3.17E-10	3.05E-11	5.07E-10	3.01E-10	1.11E-09	1.79E-10	1.17E-10	3.27E-11	1.89E-10	8.04E-11
7	0.41	1.32E-10	1.10E-10	4.18E-10	1.73E-11	1.06E-10	6.12E-11	2.47E-10	3.32E-11	4.15E-10	8.15E-11	5.58E-10	2.79E-10	1.24E-10	1.11E-10	4.82E-10	2.59E-11
9	0.51	1.74E-10	1.08E-10	4.42E-10	2.96E-11	7.89E-11	3.53E-11	1.41E-10	2.86E-11	4.85E-10	8.78E-11	7.27E-10	3.64E-10	1.55E-10	1.15E-10	4.30E-10	3.19E-11
11	0.54	2.29E-10	9.42E-11	4.81E-10	8.31E-11	1.56E-10	7.83E-11	3.89E-10	4.13E-11	2.16E-10	7.26E-11	4.11E-10	1.06E-10				
13	0.57	2.18E-10	5.93E-11	3.29E-10	1.11E-10	1.43E-10	7.41E-11	4.19E-10	2.36E-11	3.67E-10	9.06E-11	5.88E-10	2.61E-10	2.43E-10	4.09E-11	3.01E-10	1.79E-10
15	0.58	1.92E-10	1.56E-10	7.90E-10	2.36E-11	2.05E-10	7.87E-11	4.63E-10	1.05E-10	3.45E-10	7.39E-11	5.49E-10	2.49E-10	1.18E-10	7.81E-11	2.97E-10	2.60E-11
Average		2.14E-10	1.10E-10	7.90E-10	1.73E-11	1.32E-10	6.20E-11	4.63E-10	2.36E-11	3.89E-10	1.18E-10	1.11E-09	1.06E-10	1.52E-10	7.55E-11	4.82E-10	2.59E-11
Group III																	
17	0.71	1.19E-10	5.00E-11	2.23E-10	1.81E-11	8.19E-11	3.20E-11	1.71E-10	1.79E-11	4.41E-10	3.81E-10	1.47E-09	8.56E-11	7.18E-11	3.97E-11	1.93E-10	2.49E-11
35	1.12	2.48E-10	1.29E-10	8.29E-10	8.76E-11	2.45E-10	5.30E-11	3.06E-10	1.07E-10	4.62E-10	2.51E-10	8.60E-10	1.32E-10	1.67E-10	1.39E-11	1.77E-10	1.51E-10
29	1.16	1.42E-10	7.56E-11	3.77E-10	6.86E-11	1.05E-10	3.59E-11	1.93E-10	4.79E-11	3.96E-10	2.63E-10	1.00E-09	6.16E-11	3.75E-11	1.96E-11	9.40E-11	1.59E-11
37	1.4	1.29E-10	5.22E-11	2.33E-10	7.58E-11	1.01E-10	3.71E-11	2.53E-10	4.83E-11	5.42E-10	2.79E-10	1.02E-09	1.76E-10				
33	1.44	1.30E-10	8.35E-11	3.70E-10	6.20E-11	1.64E-10	6.23E-11	3.38E-10	8.20E-11	3.69E-10	2.66E-10	8.98E-10	4.86E-11	4.13E-11	9.70E-12	7.08E-11	2.38E-11
19	1.63	1.26E-10	6.34E-11	3.19E-10	4.28E-11	8.86E-11	2.51E-11	1.35E-10	5.05E-11	3.27E-10	1.47E-10	7.73E-10	8.32E-11	4.01E-11	2.80E-11	1.10E-10	9.06E-12
31	1.64	3.02E-10	1.83E-10	8.86E-10	1.49E-10	1.52E-10	3.90E-11	2.22E-10	8.91E-11	3.26E-10	1.46E-10	6.51E-10	1.30E-10	1.45E-10	1.46E-11	1.67E-10	1.29E-10
Average		1.71E-10	9.09E-11	8.86E-10	1.81E-11	1.34E-10	4.06E-11	3.38E-10	1.79E-11	4.09E-10	2.48E-10	1.47E-09	4.86E-11	8.38E-11	2.09E-11	1.93E-10	9.06E-12
Summary		2.25E-10	1.20E-10	1.21E-09	1.73E-11	1.28E-10	5.56E-11	4.63E-10	7.51E-12	4.00E-10	1.88E-10	1.47E-09	4.86E-11	1.15E-10	4.57E-11	4.82E-10	9.06E-12

Table 109. Sulfur depletion statistics. The first method was based on initial %S minus S lost as determined by sulfate concentration in leachate. The cumulative S depletion for this sulfate-based method (%S depletion (sulfate)) is displayed next to the right-most column (%S Depletion (solids)), which determines S depletion as unleached sample %S minus leached sample %S. Leached solids ran for less time than full experimental period, so are likely to underestimate S loss (excludes R39-44).

Reactor	%S	Start of experiment until pH<6		Major pH drop and/or major sulfate release starts		Minimum pH and/or max sulfate release		pH increase begins		End of experiment		%S Depletion (sulfate)	% S Depletion (solids)	
		Week	%S remaining	Week	%S remaining	Week	%S remaining	Week	%S remaining	Week	%S remaining		Week	% S depletion
1	0.18	#								1252	45.3	54.7	328	41.1
3	0.22	#								1252	54.6	45.4	328	30.9
5	0.40	400	48.5	720	33.3	810	17.4	825	8.5	909	-3.7	103.7	328	12.8
7	0.41	80	94.0	265	83.9	395	65.9	480	48.5	724	38.8	61.2	328	47.8
9	0.51	70	93.0	250	81.8	355	59.8	404	48.6	724	28.2	71.8	328	54.3
11	0.54	57	90.9	215	71.0	290	59.3	385	42.7	441	37.0	63.0	441	72
13	0.57	75	89.4	257	75.2	340	59.4	390	51.1	441	44.8	55.2	441	66.3
15	0.58	55	91.9	140	84.3	280	60.3	335	49.9	724	26.1	73.9	328	62.2
17	0.71	50	96.7	275	83.4	350	67.3	375	50.4	724	32.1	67.9	328	55.4
35	1.12	*		205	66.2	270	42.6	290	32.3	360	21.3	78.7	360	84.6
29	1.16	35	95.5	140	88.7	280	60.2	315	43.5	643	28.3	71.7	643	6.9
37	1.40	35	96.5	192	85.8	282	66.2	345	34.9	360	30.8	69.2	360	79.1
33	1.44	10	97.6	220	76.9	280	57.2	330	34.2	643	16.0	84.0	247	54
19	1.63	20	96.9	120	90.7	290	59.5	355	38.8	724	24.3	75.7	289	71.9
31	1.64	*		220	72.3	275	60.0	290	55.2	360	46.2	53.8	360	75.1
Average			90.1		76.4		56.5		41.4		31.3	68.7		54.3
Std Dev			14.1		14.8		13.2		12.3		14.4	14.4		23.2

* Started at pH<6.0
# pH never <6.0 for sustained period

Figure 1. The Duluth Complex and associated intrusions in the area of the Dunka Mine and orientation of western footwall rock suites (Miller et al, 2001).

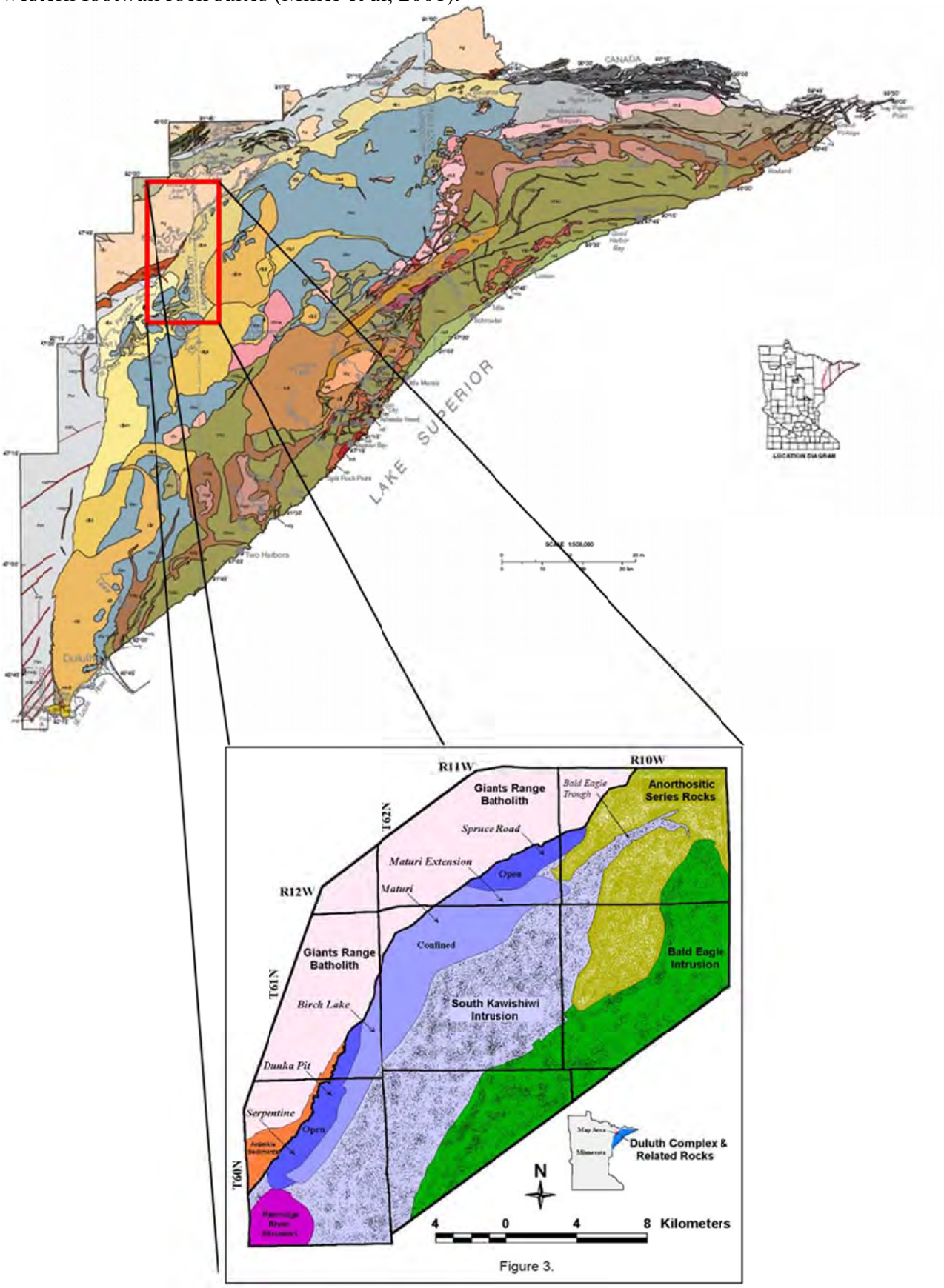
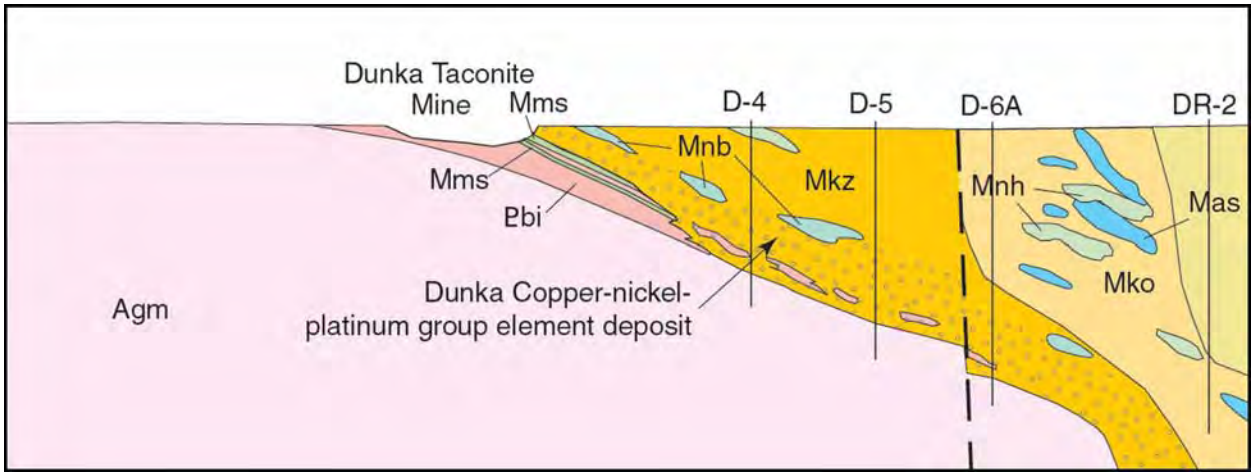


Figure 2. Cross Section from the northwest to the southeast across the Dunka Pit (Miller et al, 2005).



Relevant Geologic Map and Cross Section Symbols (After Miller et al, 2005):

Mesoproterozoic-- Duluth Complex, Layered Series--South Kawishiwi intrusion:

Mnb: Basaltic hornfels—Fine-grained, granoblastic to poikiloblastic basaltic hornfels; consists of variable amounts of plagioclase, augite, olivine, hypersthene, and inverted pigeonite.

Mkz: Contact zone—Varitextured olivine gabbro, augite troctolite, gabbronorite, norite, and footwall inclusions; commonly sulfide-bearing.

Mms: Mafic footwall sill—Mafic sills in the Paleoproterozoic footwall near the base of the Virginia Formation. Contains variable amounts of plagioclase, orthopyroxene, clinopyroxene, olivine, amphibole, and lesser amounts of biotite.

Paleoproterozoic--Animikie Group:

Pvf: Virginia Formation—A well bedded sequence of graywacke and slate, locally carbonaceous and sulfidic, that are strongly recrystallized and locally partially melted (metatexite and diatexite) by the Duluth Complex. Also occurs as hornfels inclusions in the contact zone of the South Kawishiwi intrusion.

Pbi: Biwabik Iron Formation—Well bedded, iron-bearing strata of alternating cherty intervals and slaty intervals.

Neoarchean--Giants Range batholith:

Agm: Quartz monzonite, granite, and monzodiorite—Pink to dark greenish-gray, hornblendebearing, coarse-grained and variably porphyritic, variably magnetic.

Figure 3. Map of the Duluth Complex basal contact and associated sulfide mineral deposits.

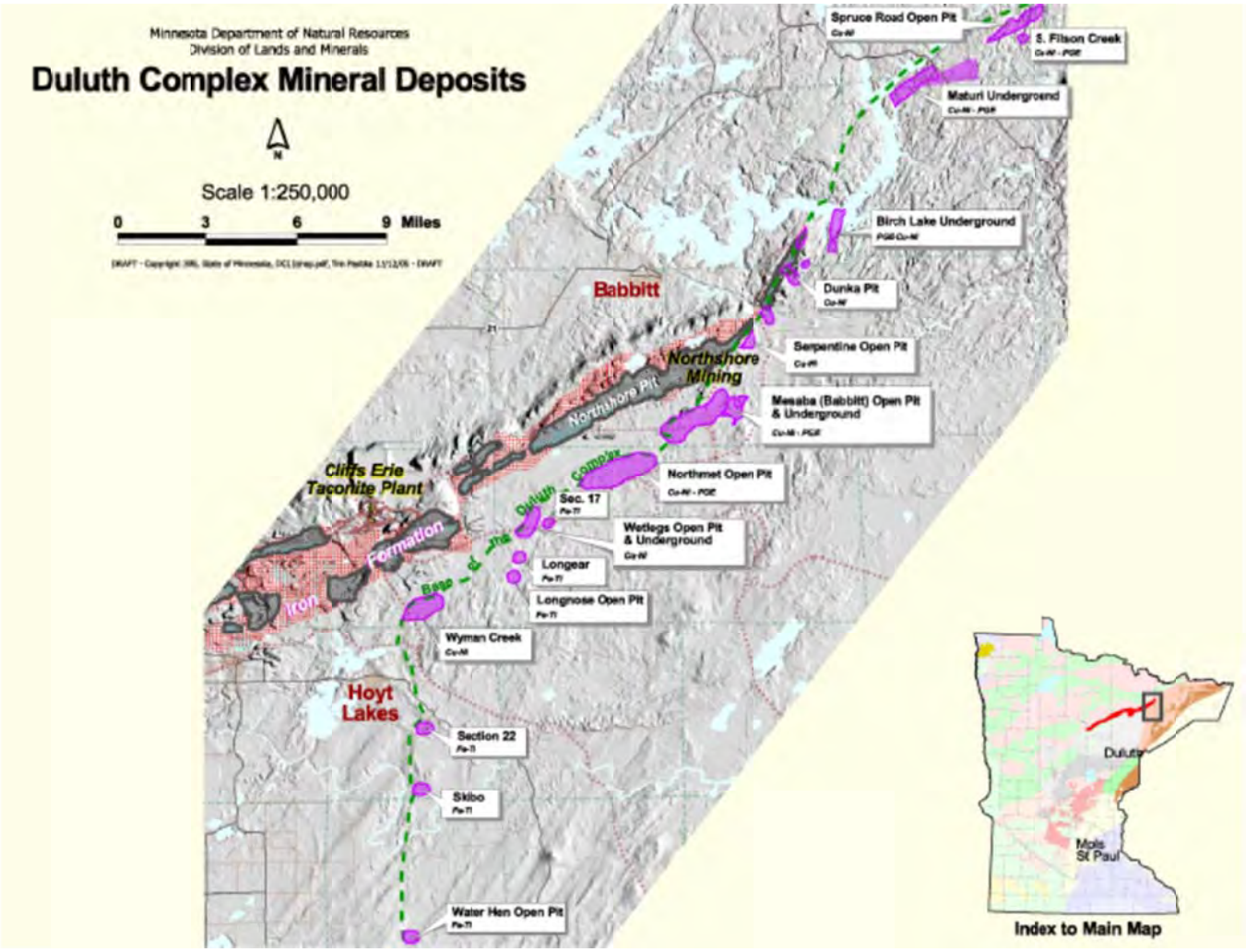


Figure 4. Experiment reactor diagram and picture

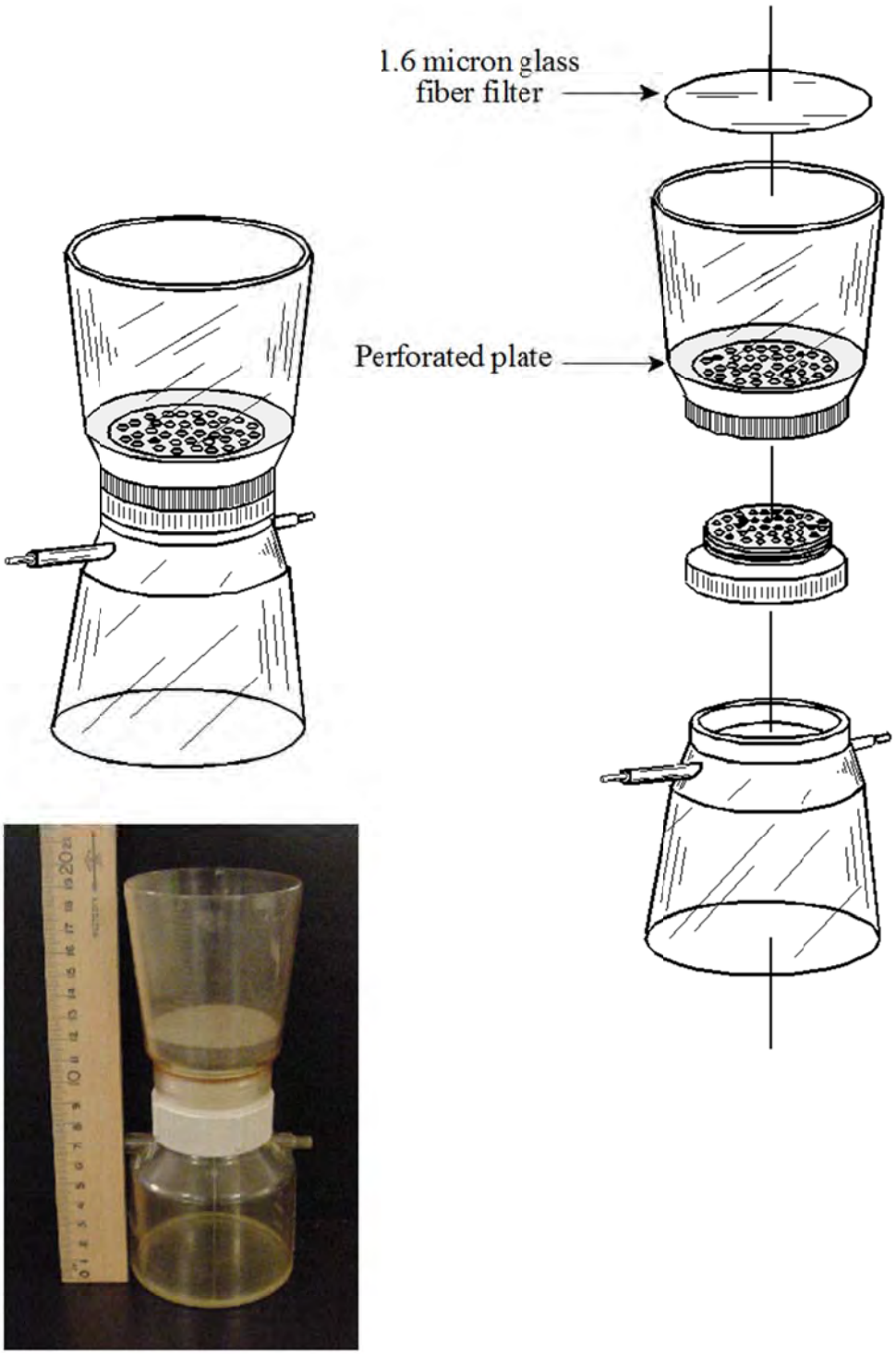
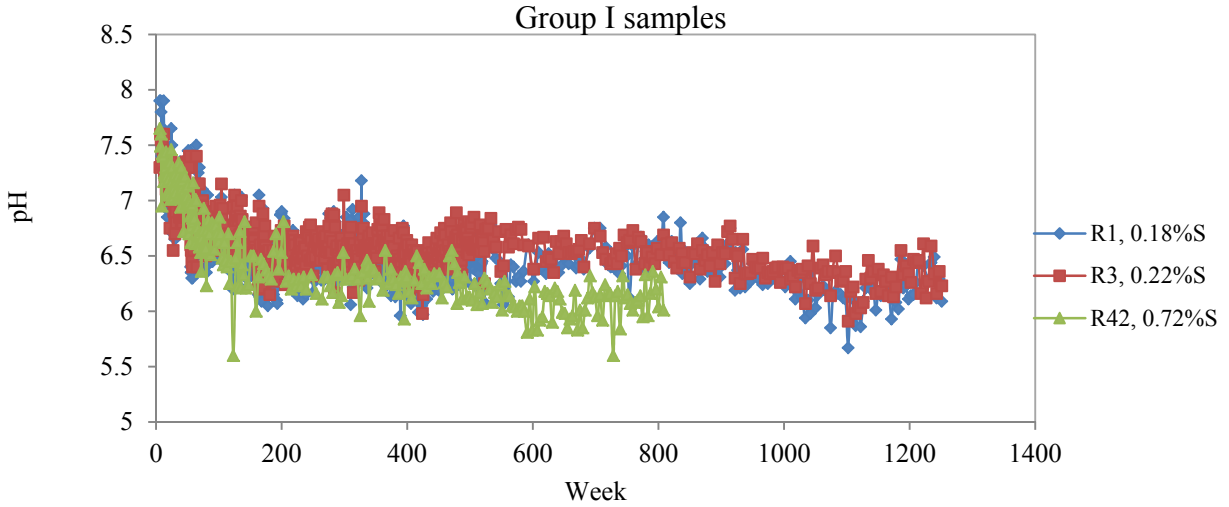
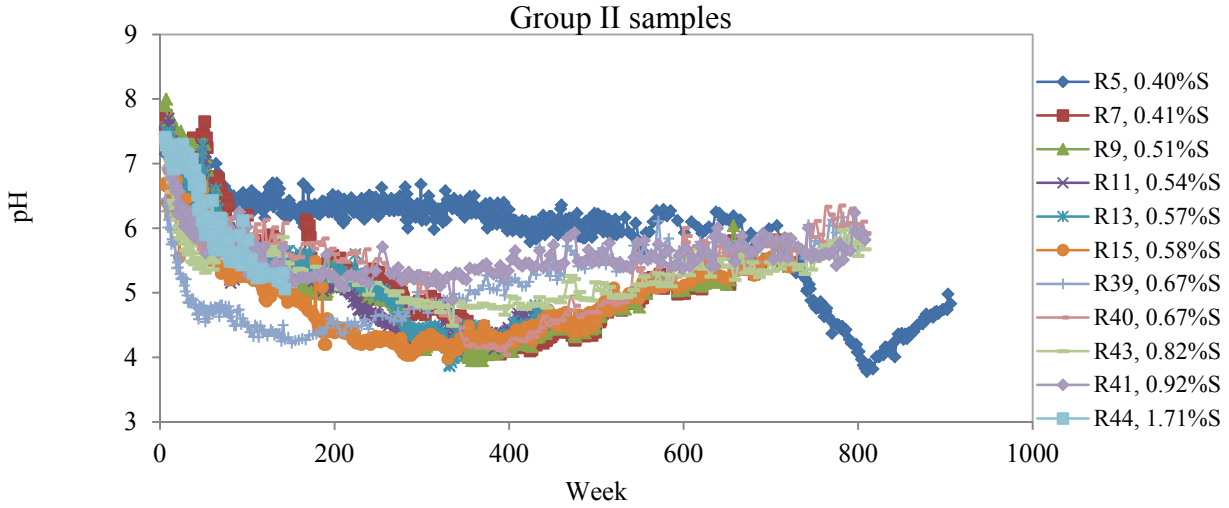


Figure 5. pH vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.





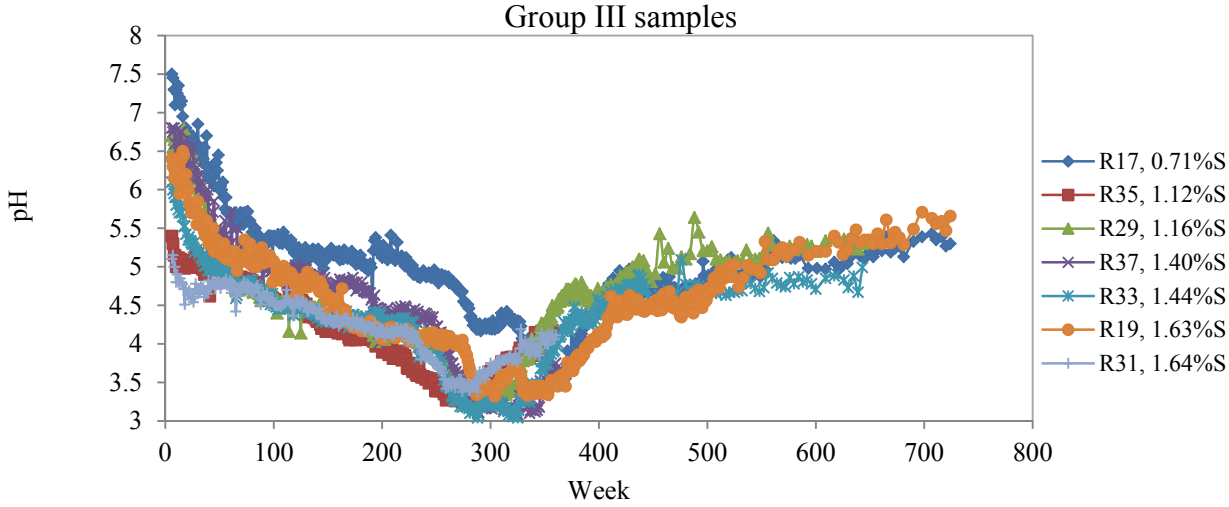


Figure 6. Sulfate release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

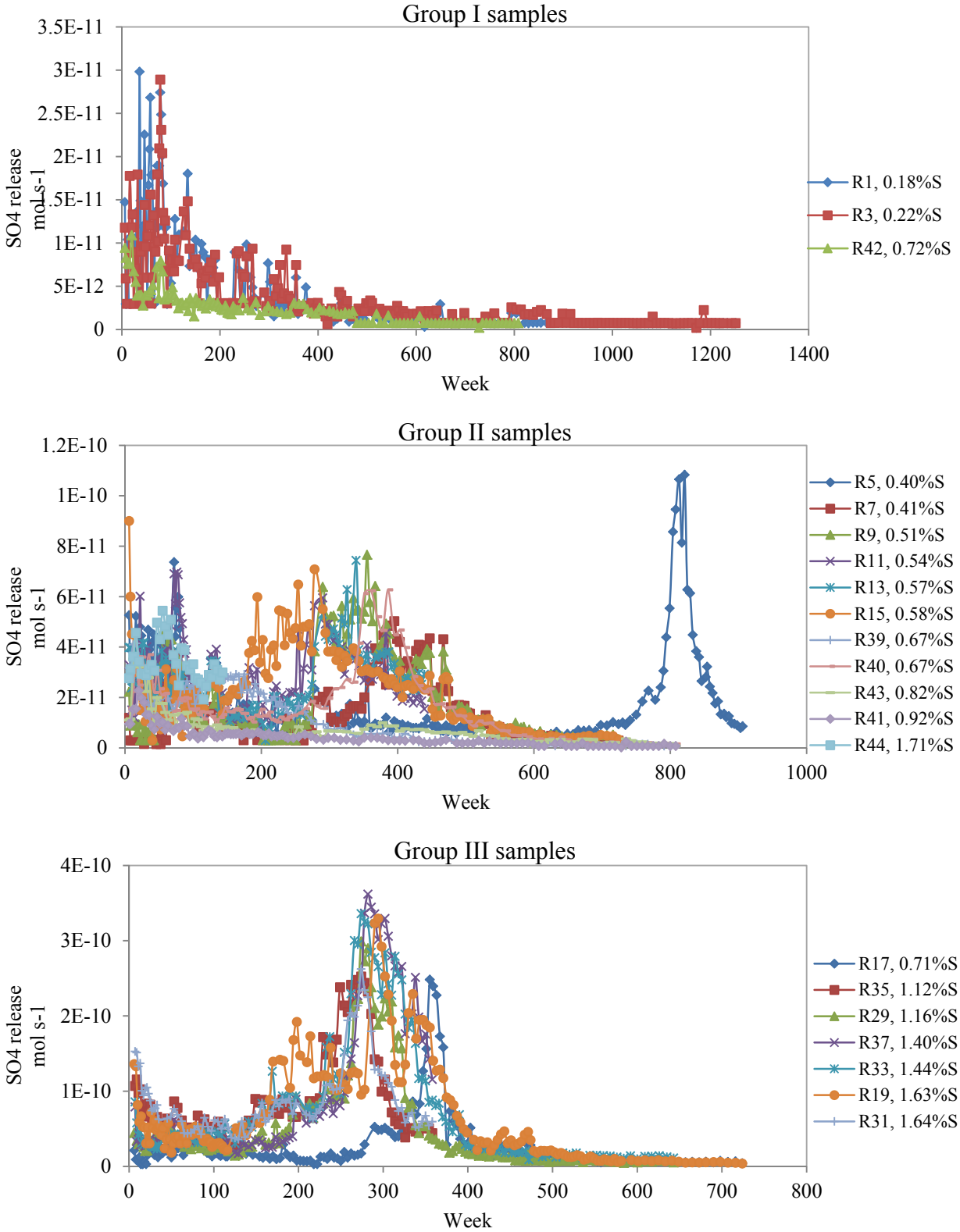


Figure 7. Sulfate release vs. time for Group II samples (0.40, 0.58, 0.67 (R40), 0.82, 0.92 %S). Weeks 0-5 were omitted to improve resolution.

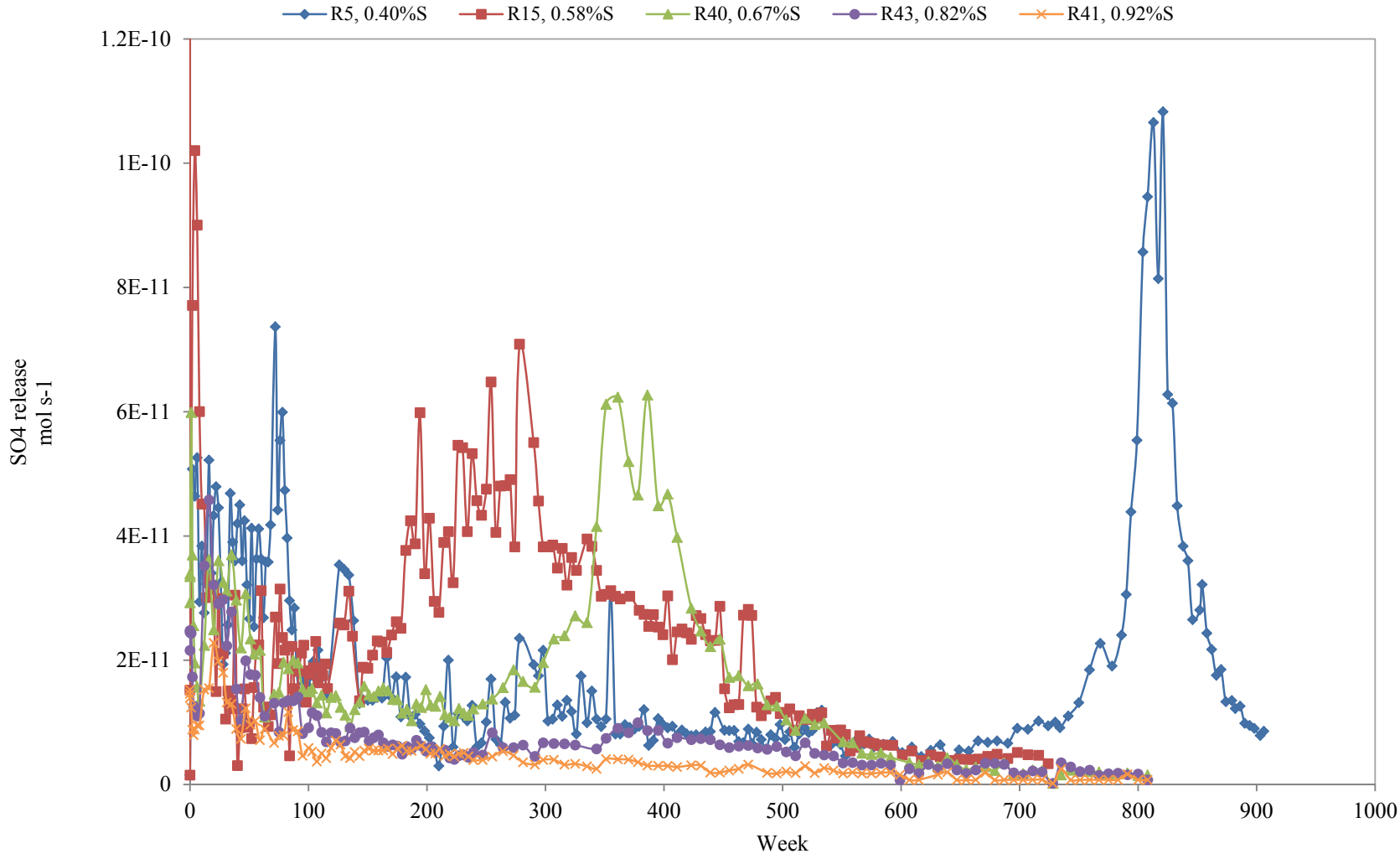


Figure 8. Calcium release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

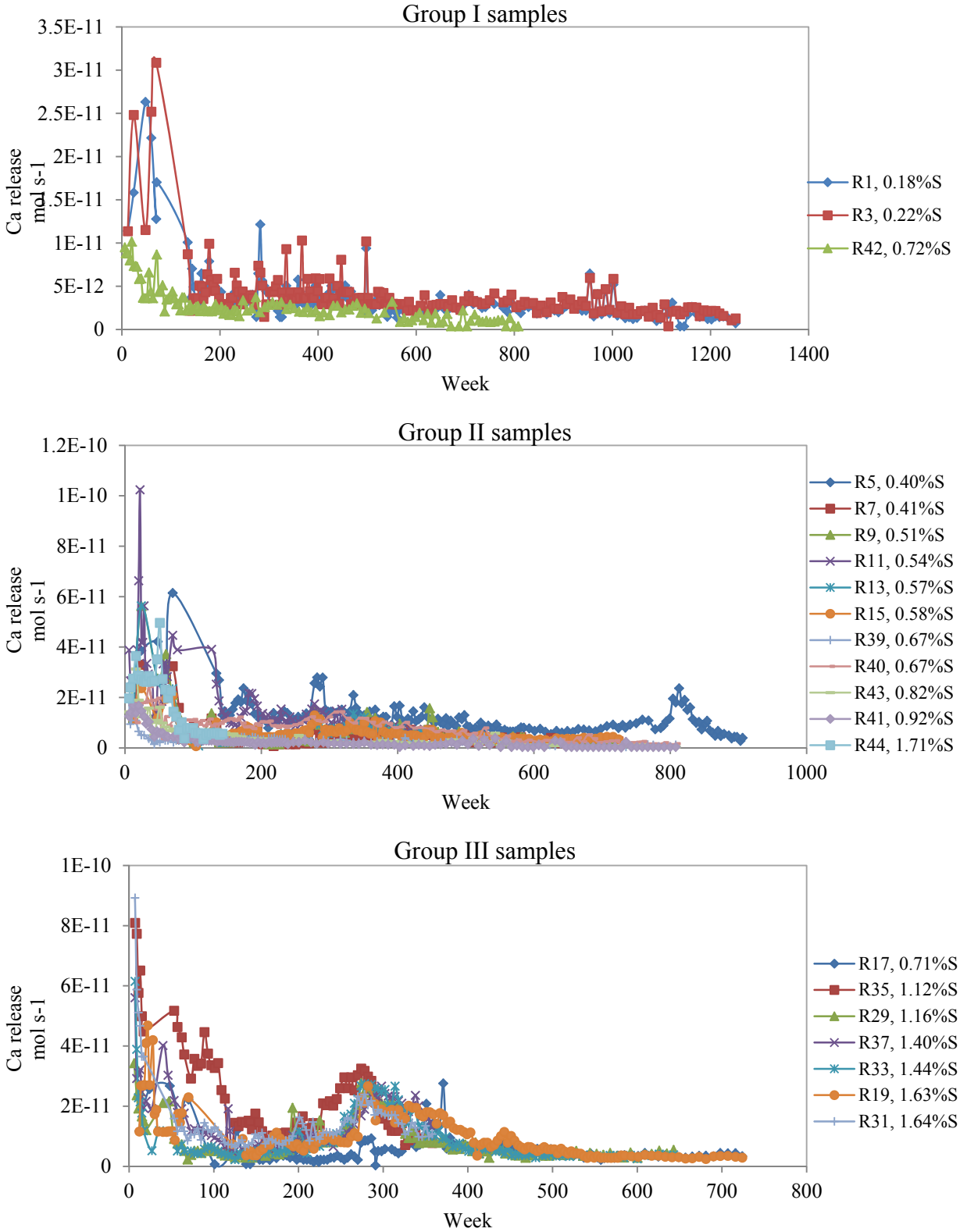


Figure 9. Magnesium release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

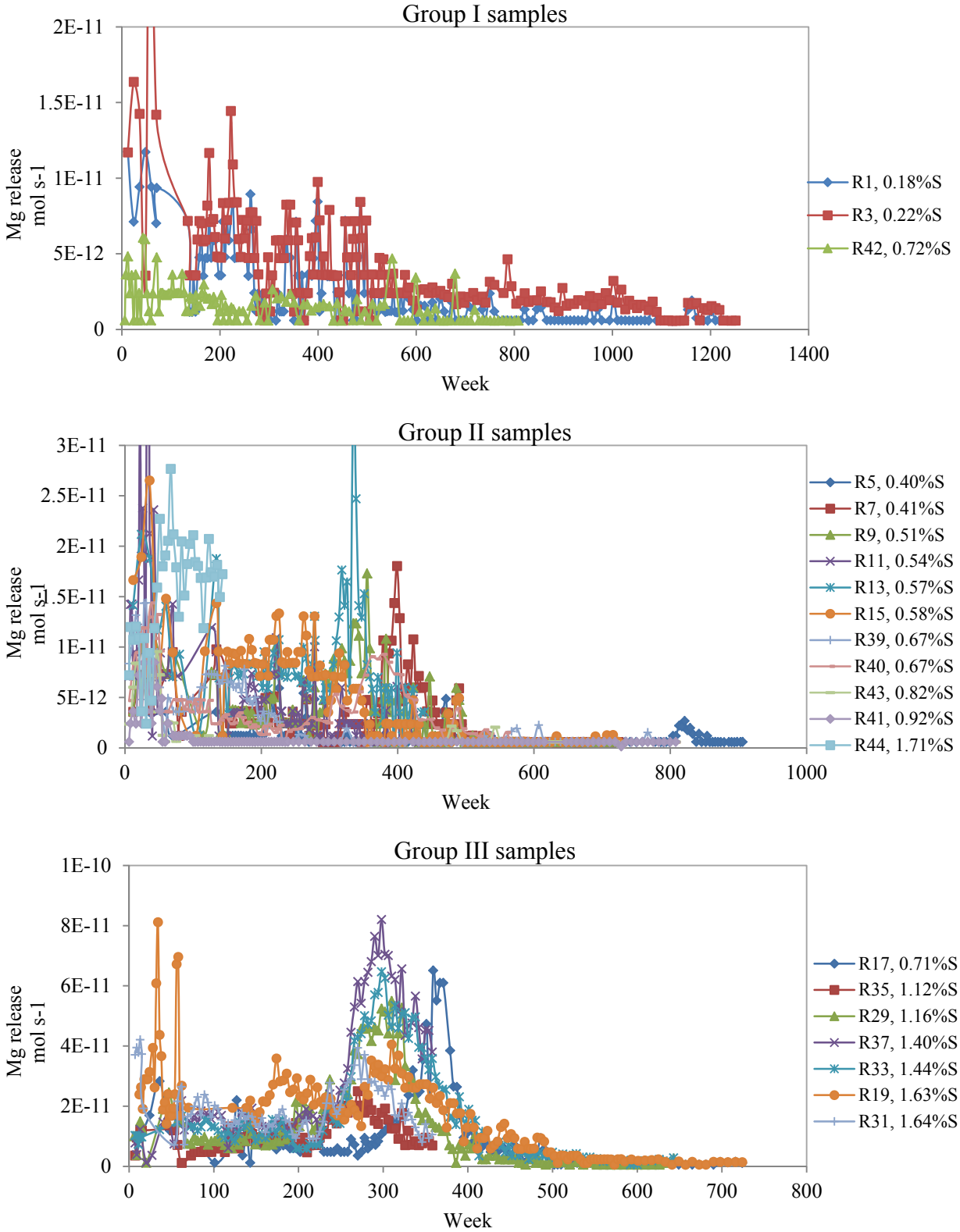


Figure 10. Sodium release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

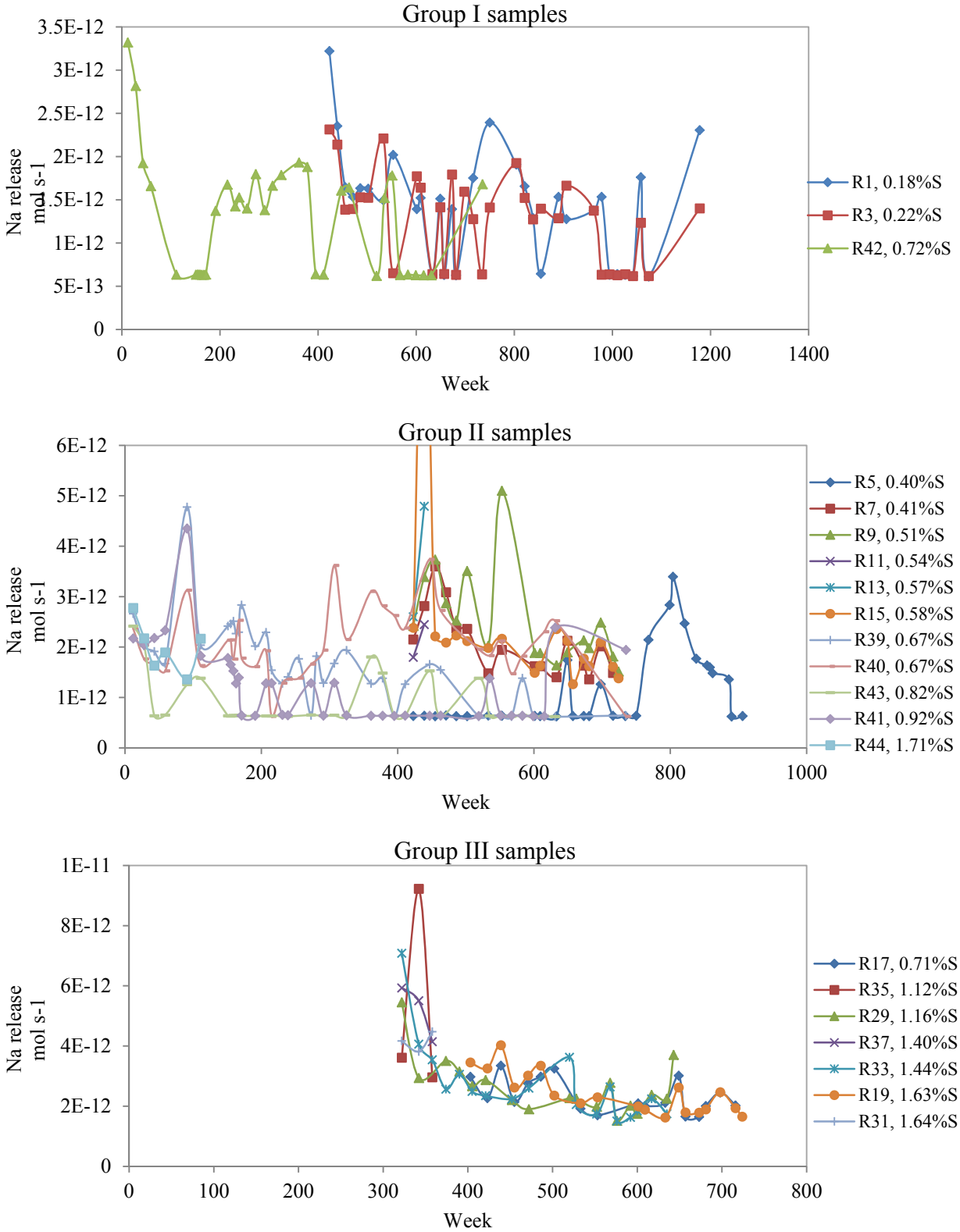


Figure 11. Potassium release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

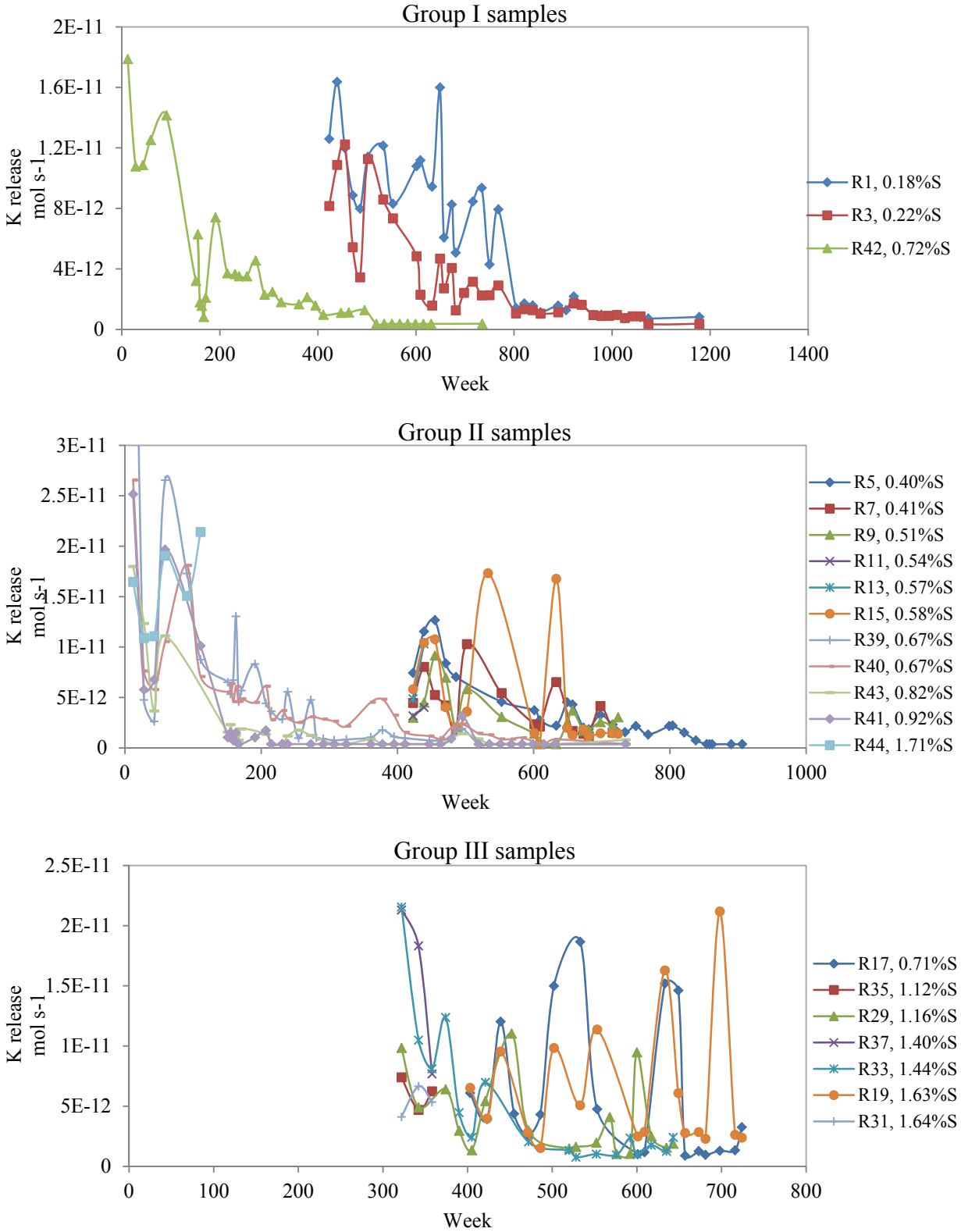


Figure 12. Cobalt release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

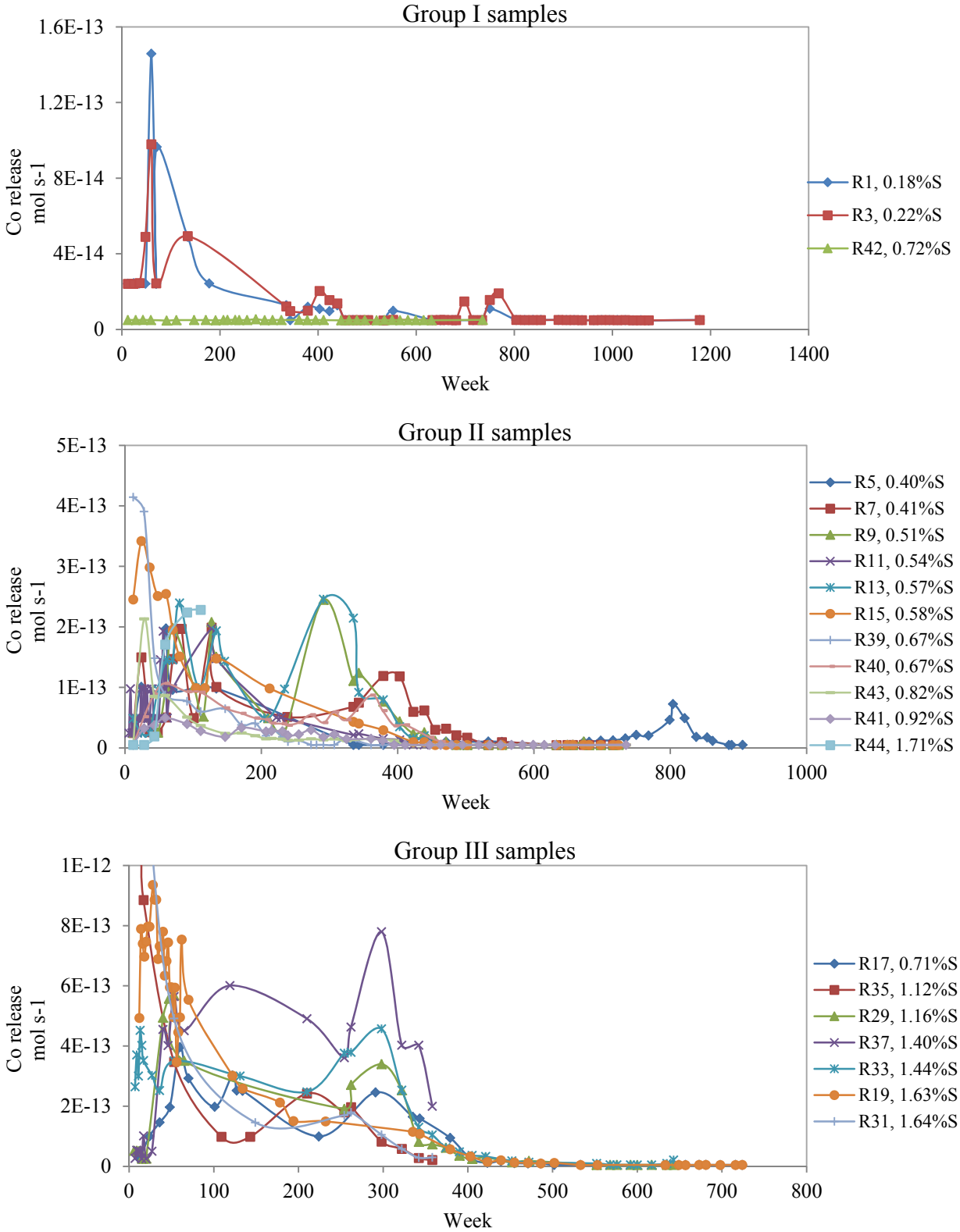


Figure 13. Copper release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

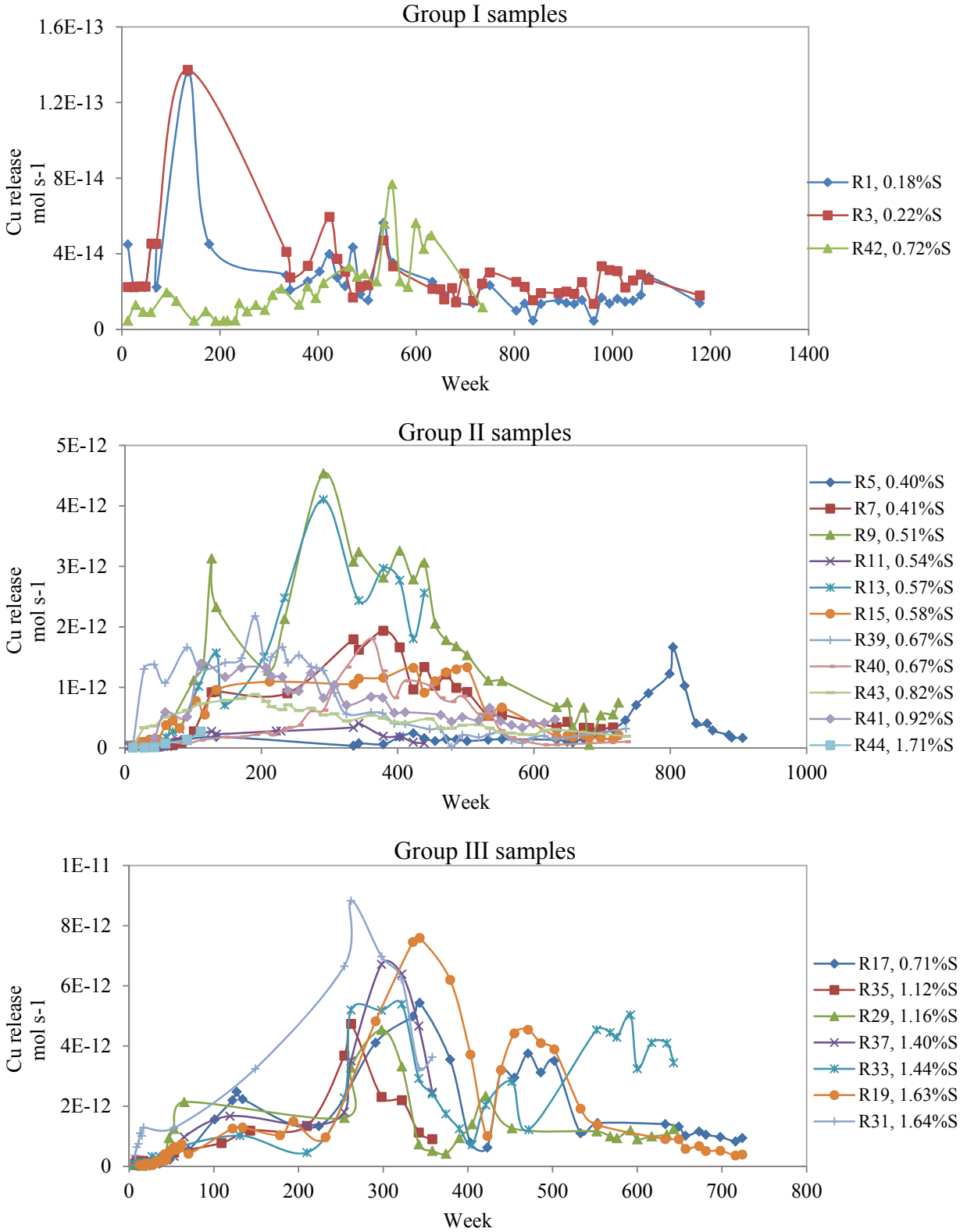


Figure 14. Nickel release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

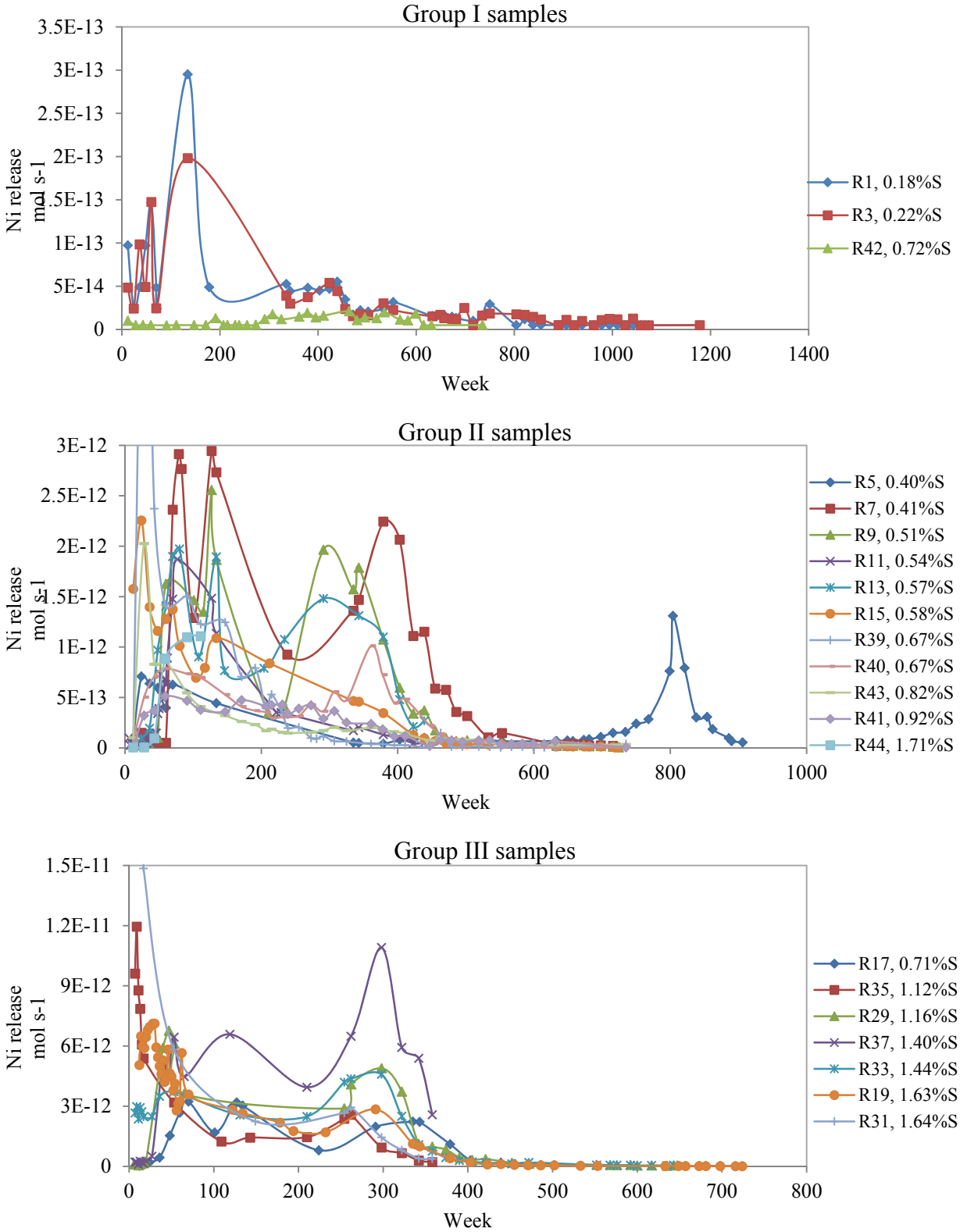


Figure 15. Zinc release vs. time for each sample group with similar behavior. Weeks 0-5 were omitted to improve resolution.

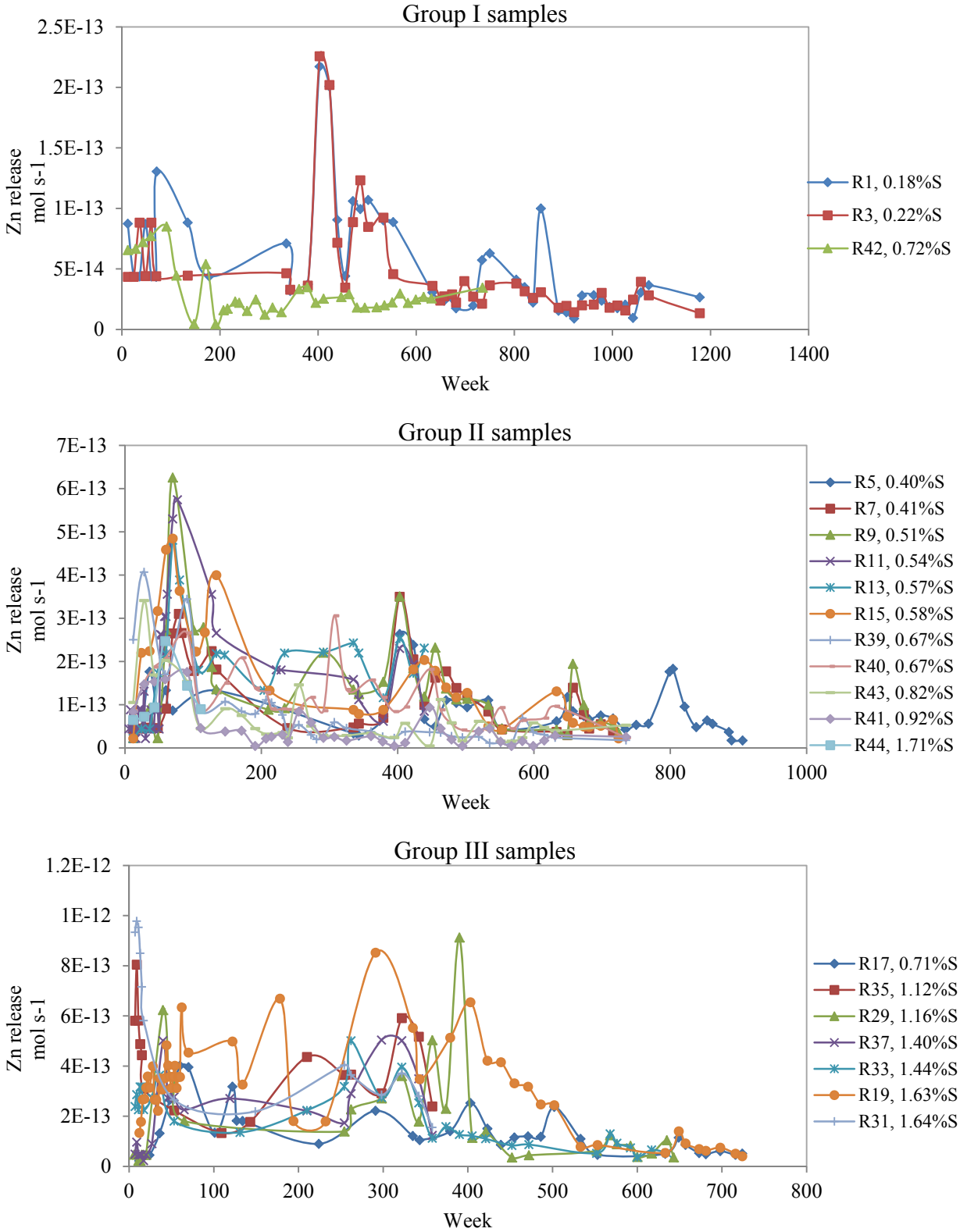


Figure 16. Cobalt release vs. pH for group II reactors

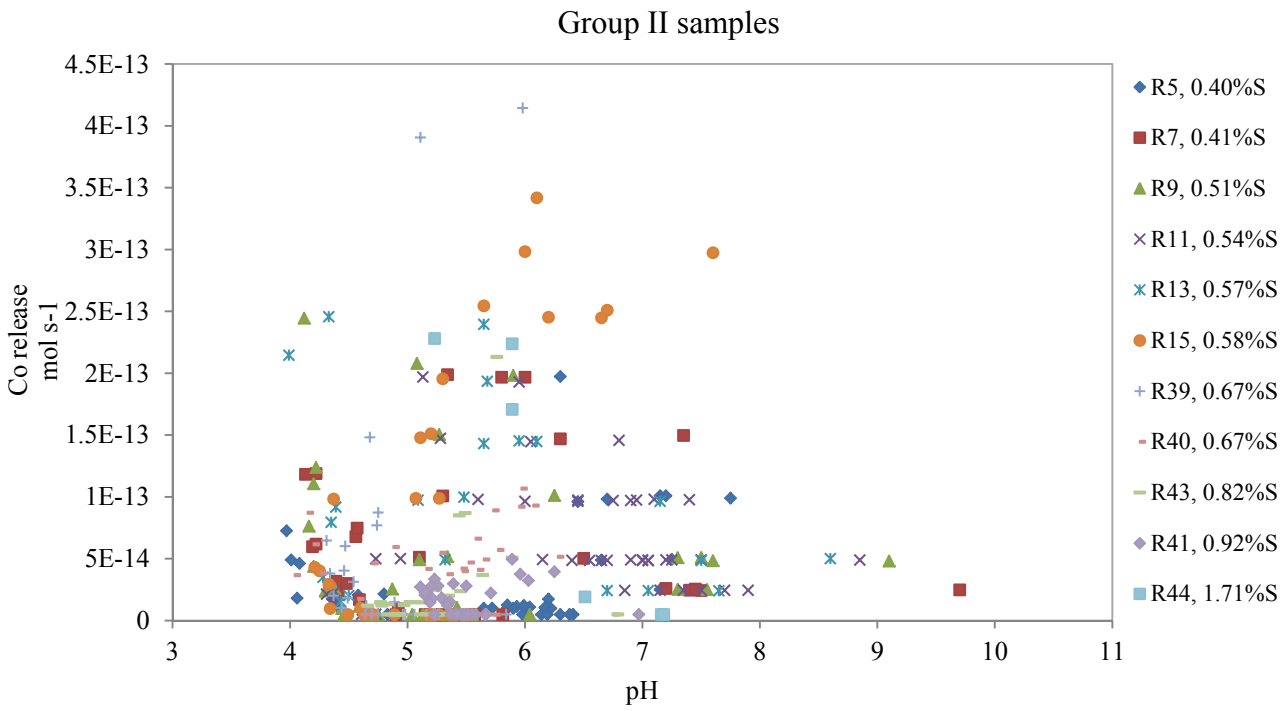


Figure 17. Copper release vs. pH for group II reactors

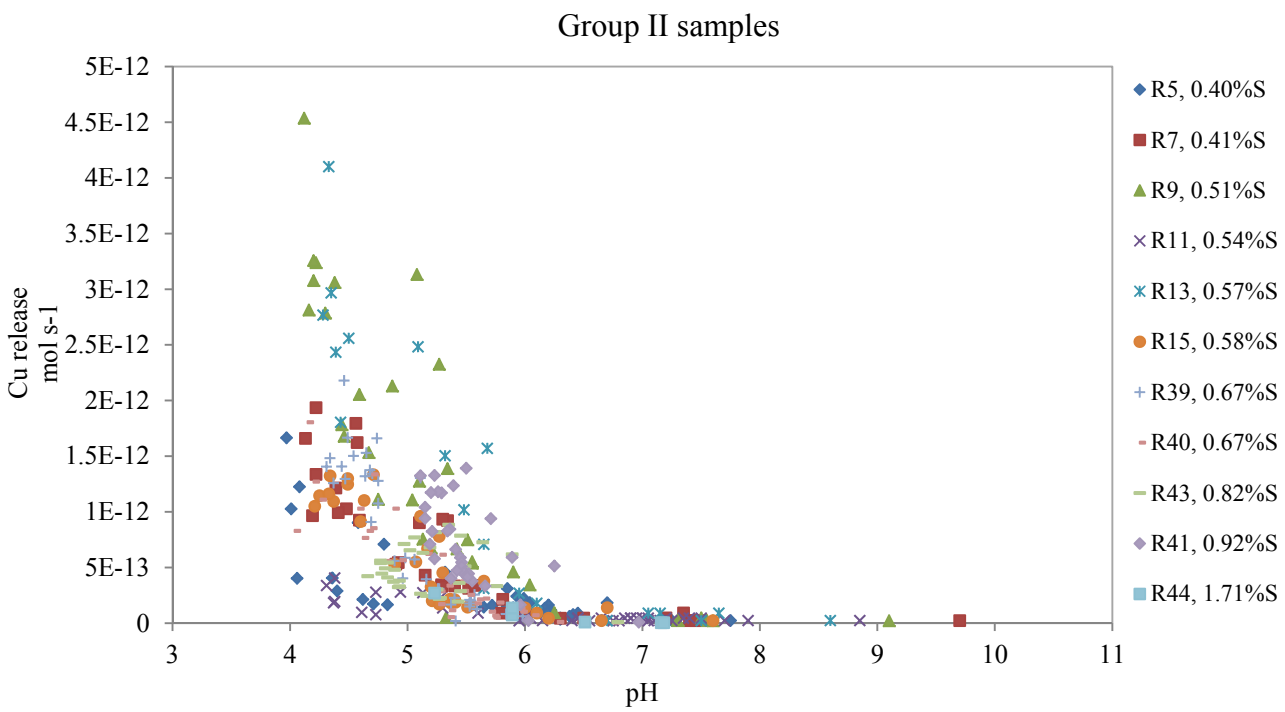


Figure 18. Nickel release vs. pH for group II reactors

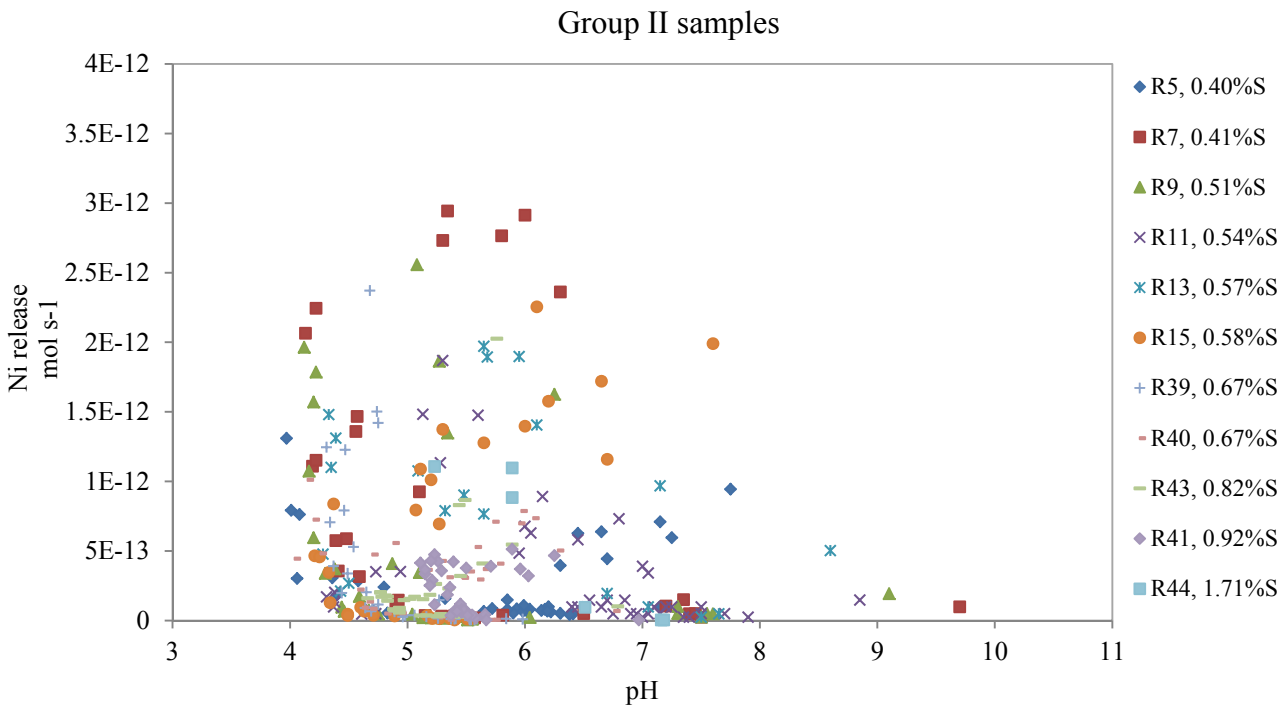


Figure 19. Zinc release vs. pH for group II reactors

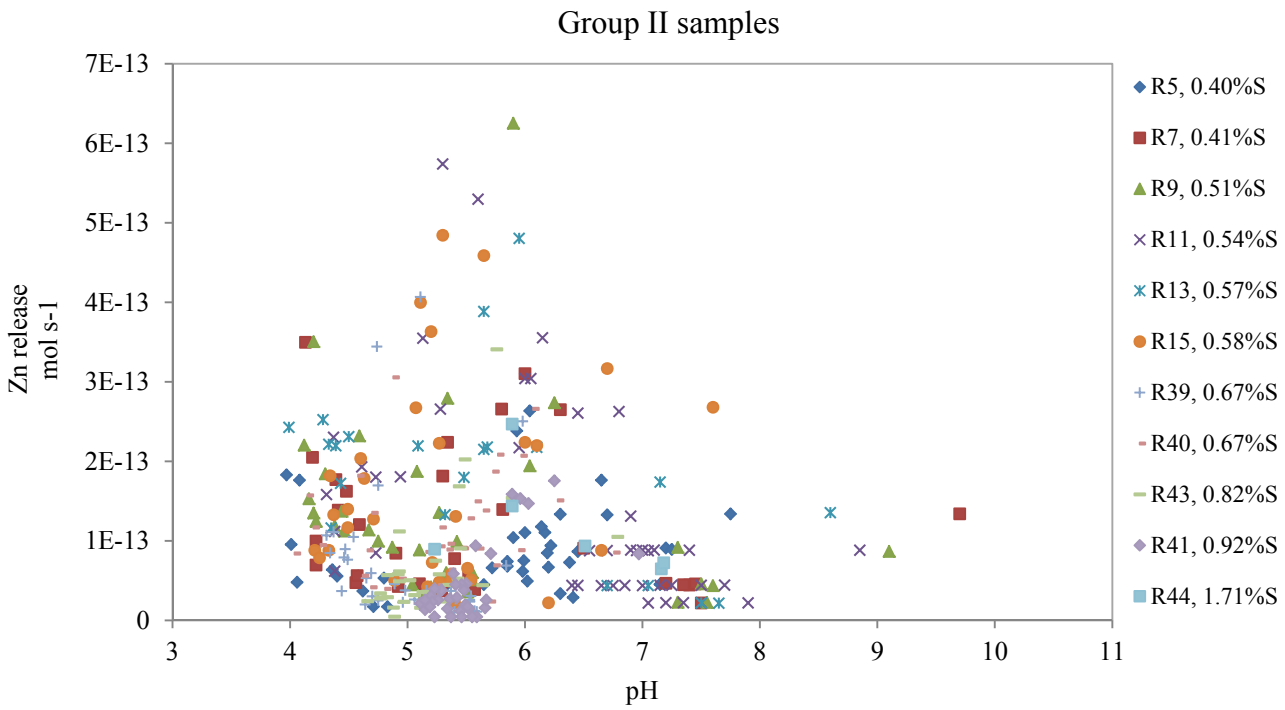


Figure 20. pH vs. sulfur content, period A

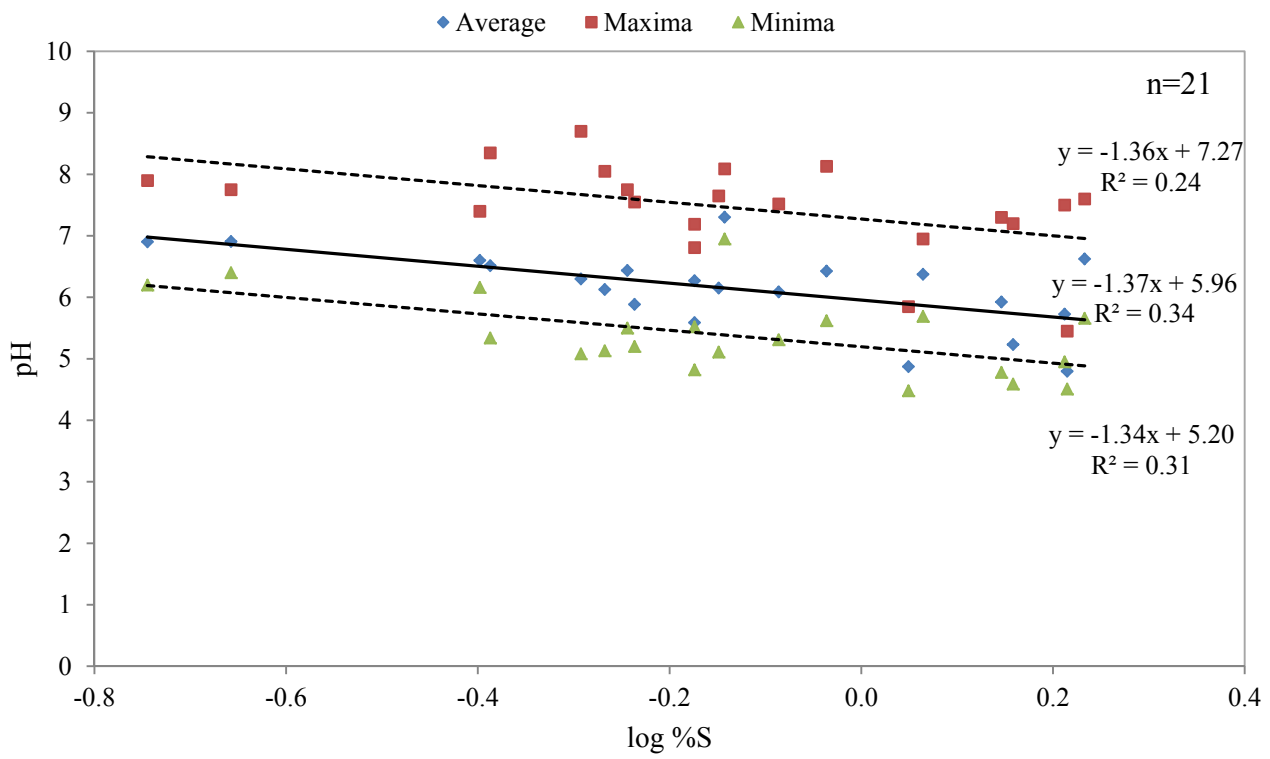


Figure 21. pH vs. sulfur content, period B

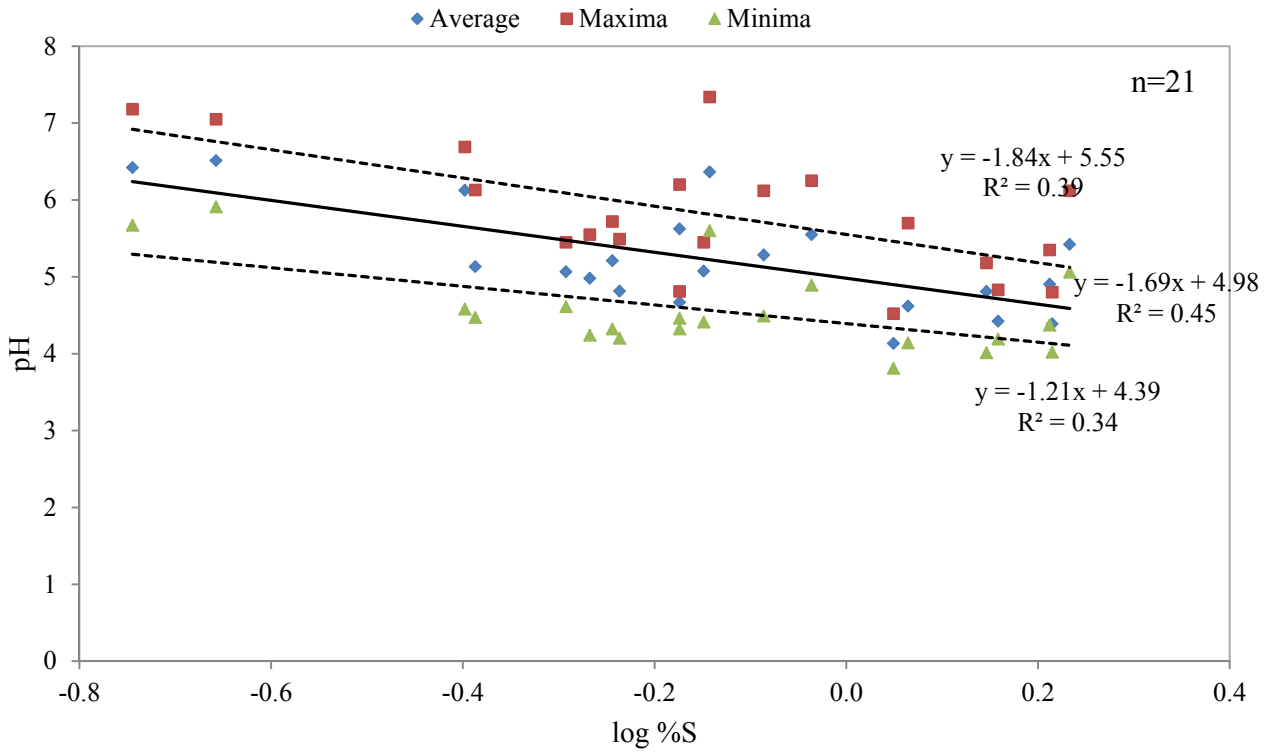


Figure 22. pH vs. sulfur content, period C

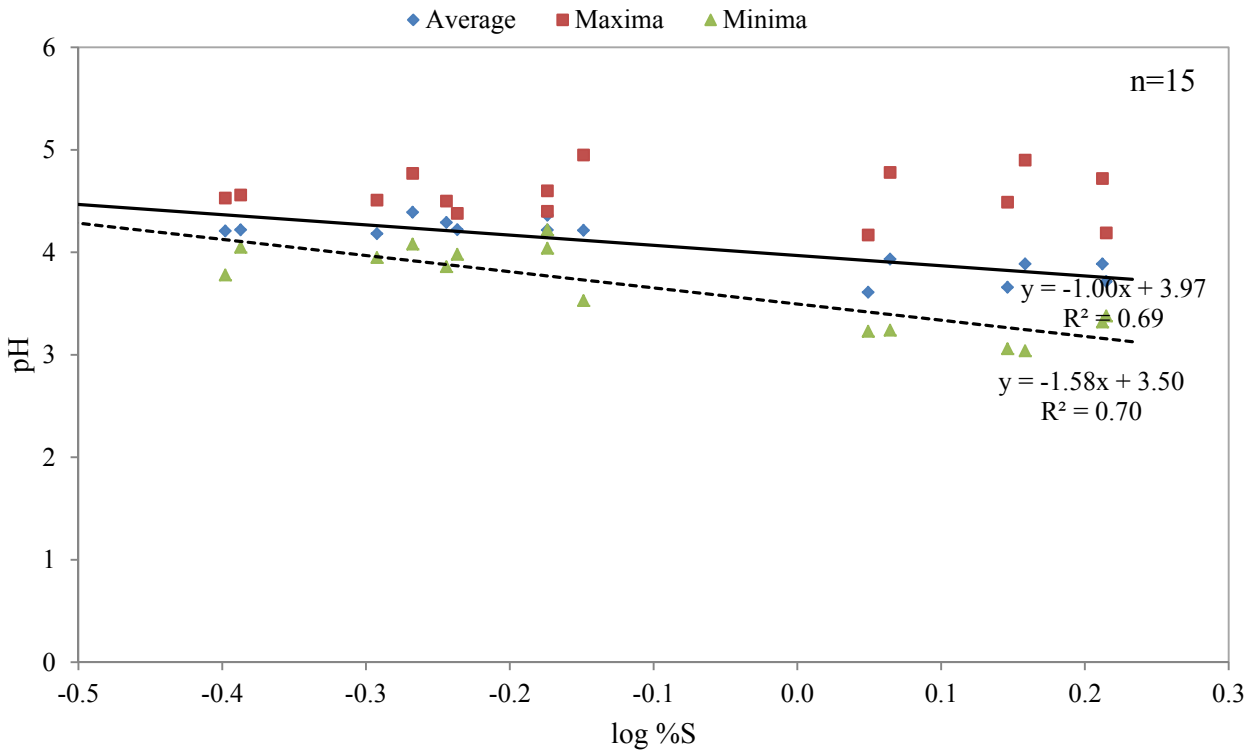


Figure 23. pH vs. sulfur content, period D

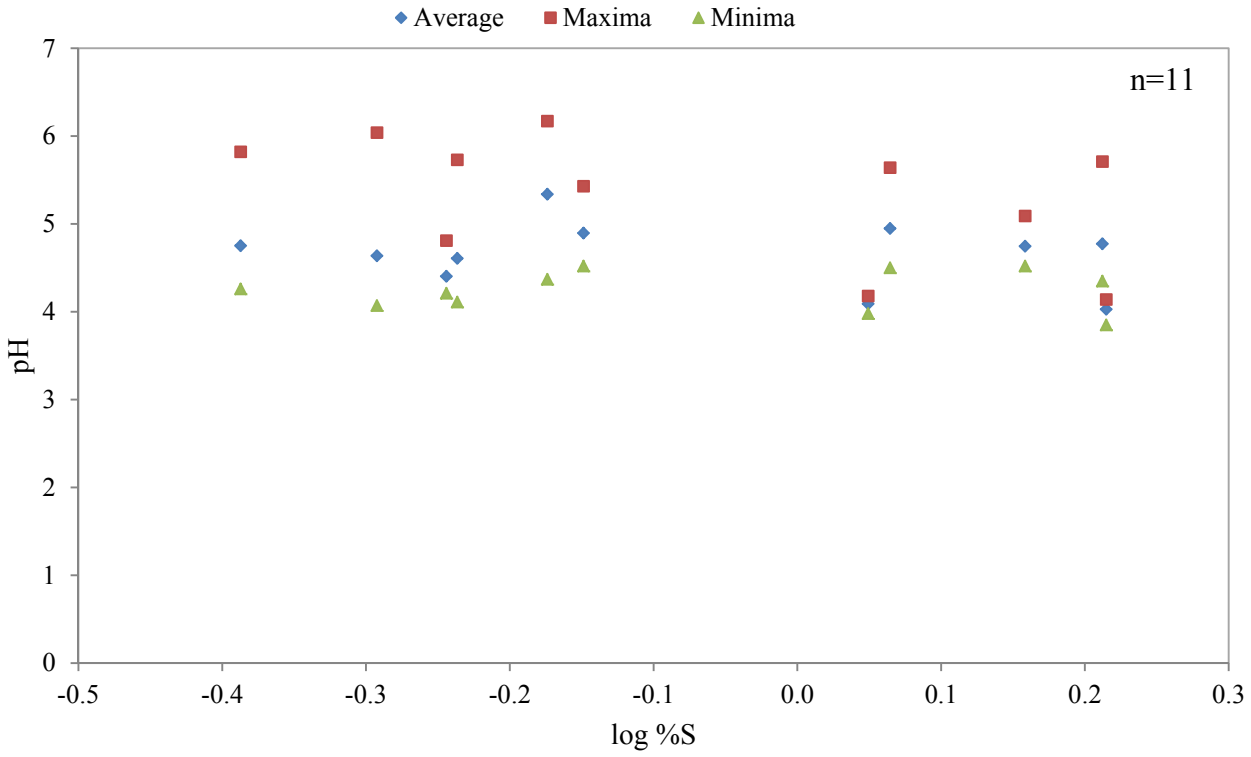


Figure 24. Sulfate release (umol/wk) vs. sulfur content, period A

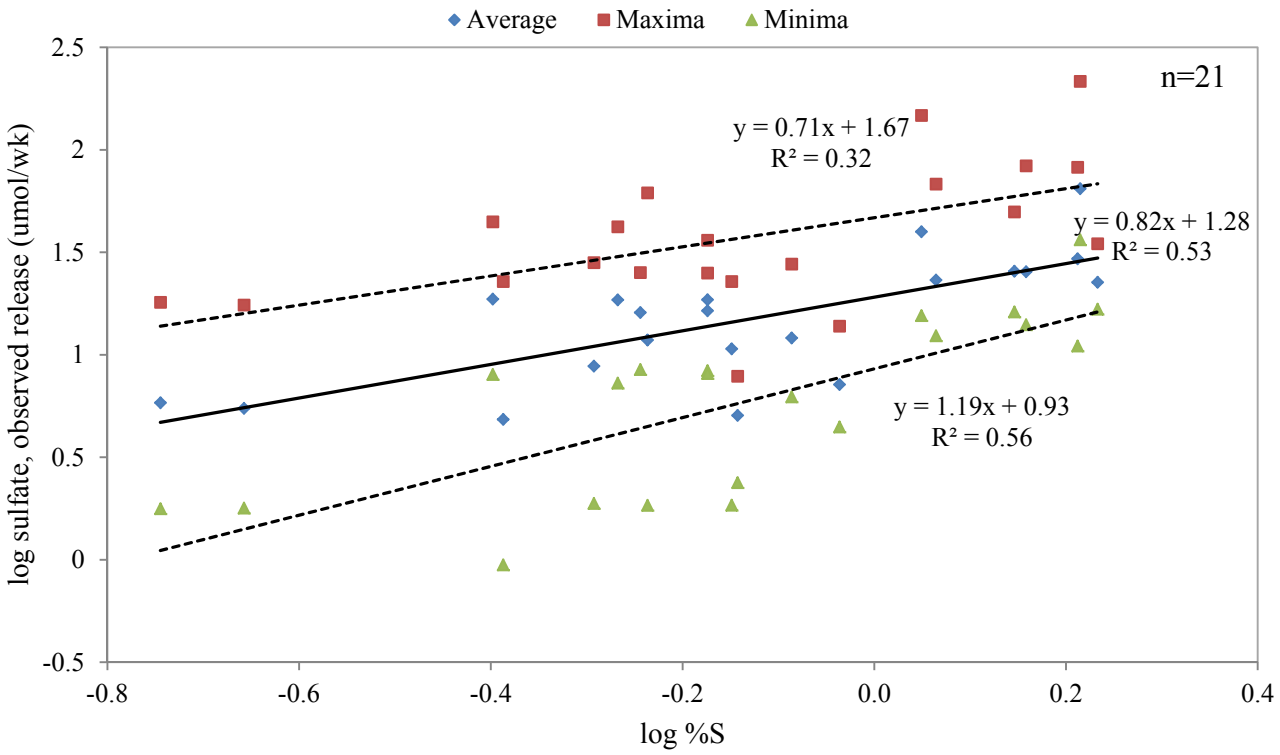


Figure 25. Sulfate release (umol/wk) vs. sulfur content, period B

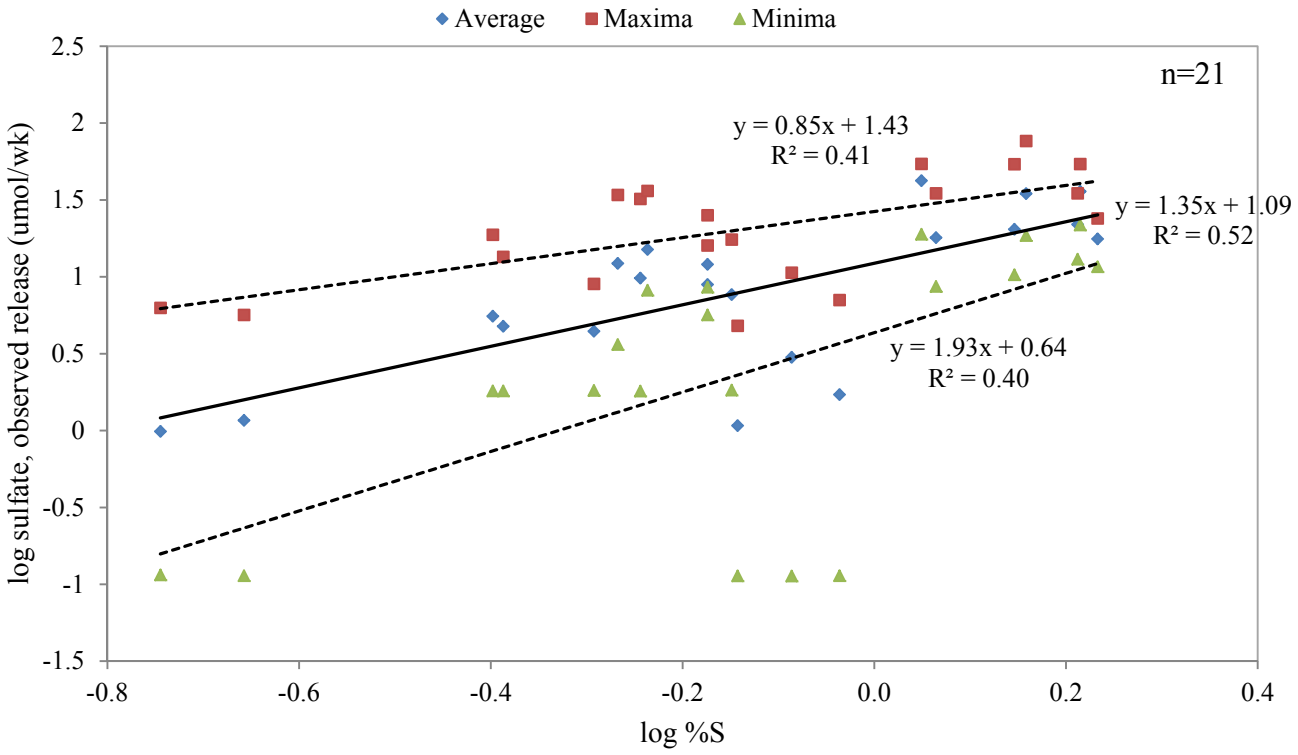


Figure 26. Sulfate release (umol/wk) vs. sulfur content, period C

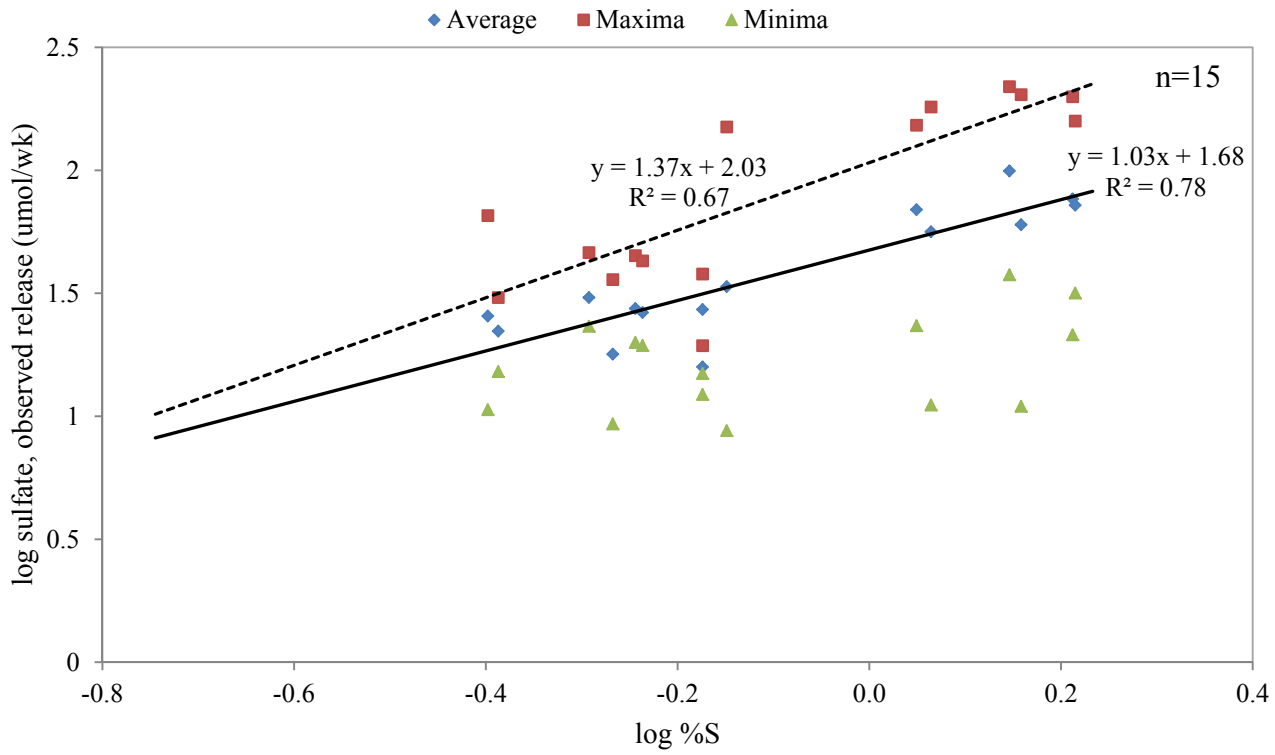


Figure 27. Sulfate release (umol/wk) vs. sulfur content, period D

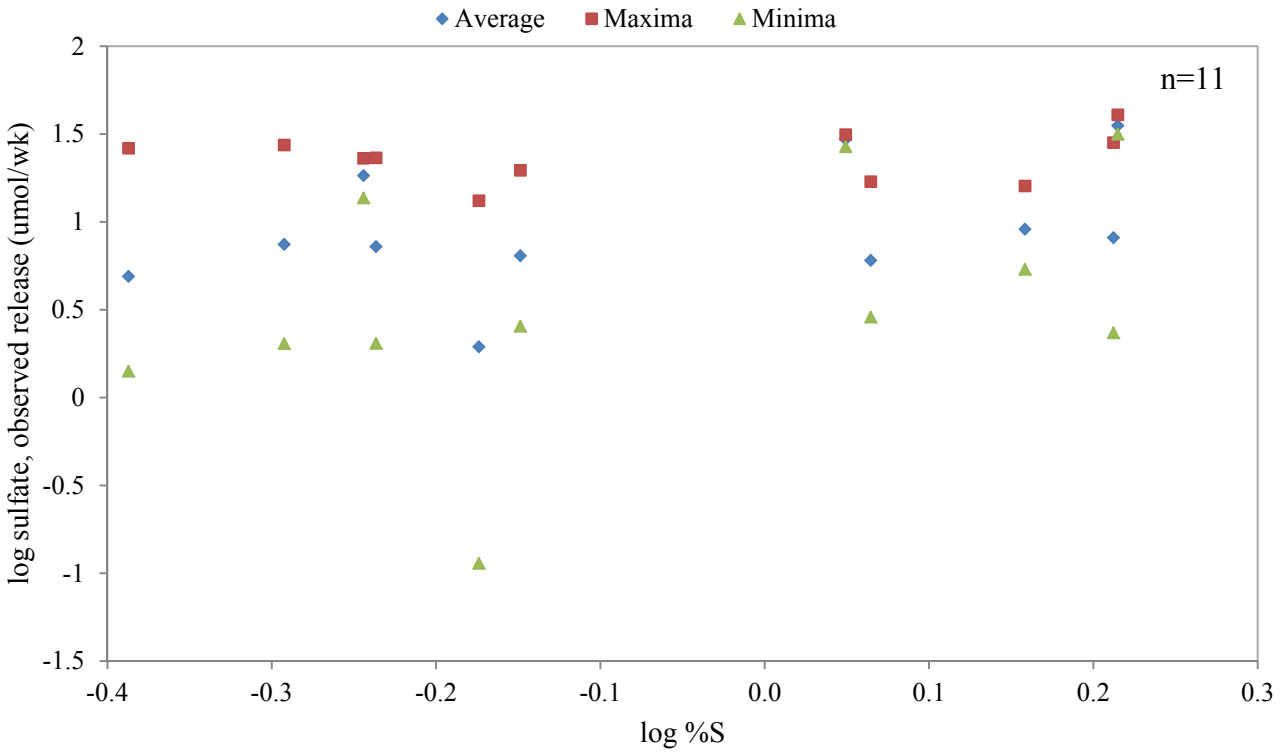


Figure 28. pH vs. sulfate release (umol/wk), period A

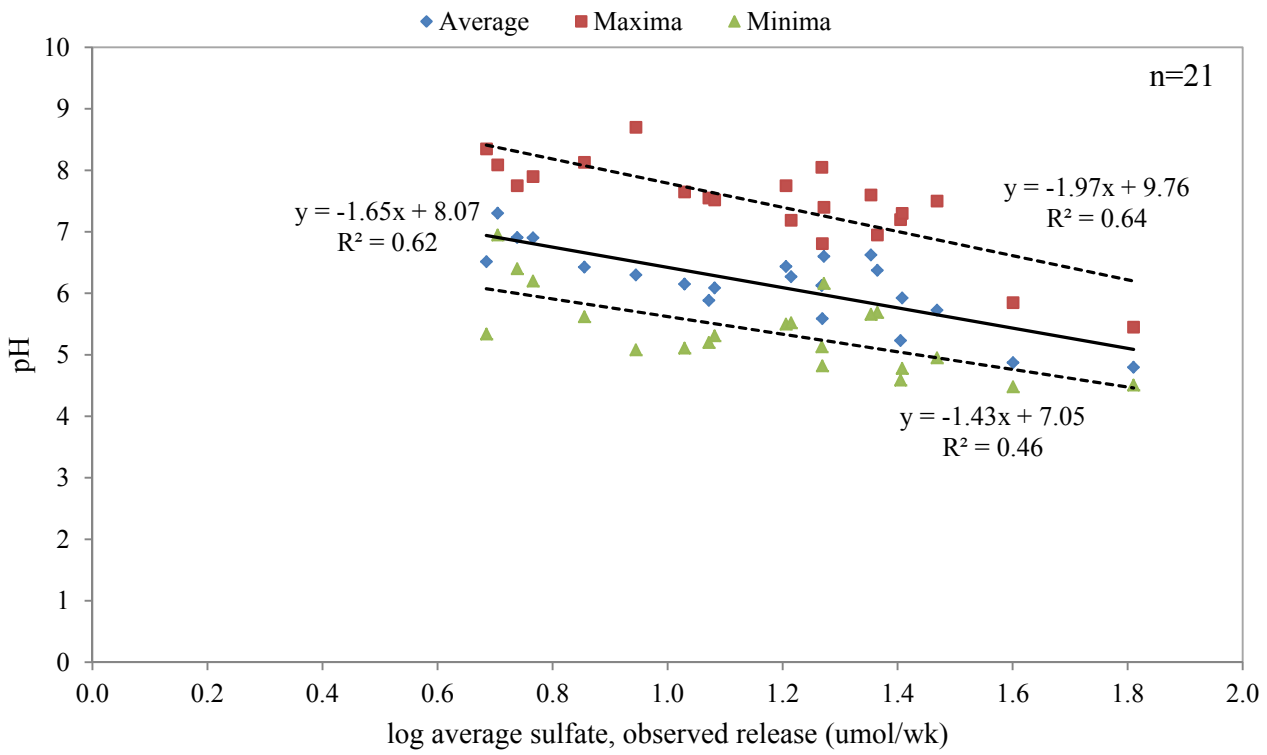


Figure 29. pH vs. sulfate release (umol/wk), period B

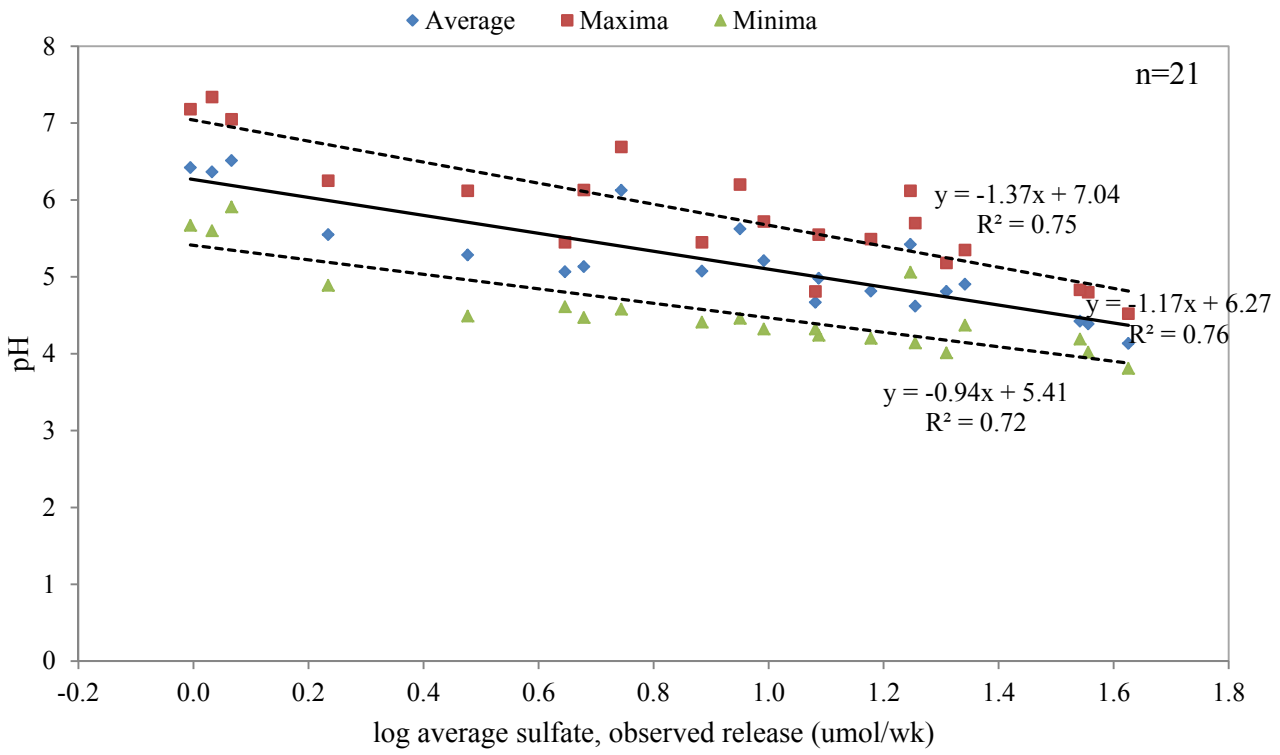


Figure 30. pH vs. sulfate release (umol/wk), period C

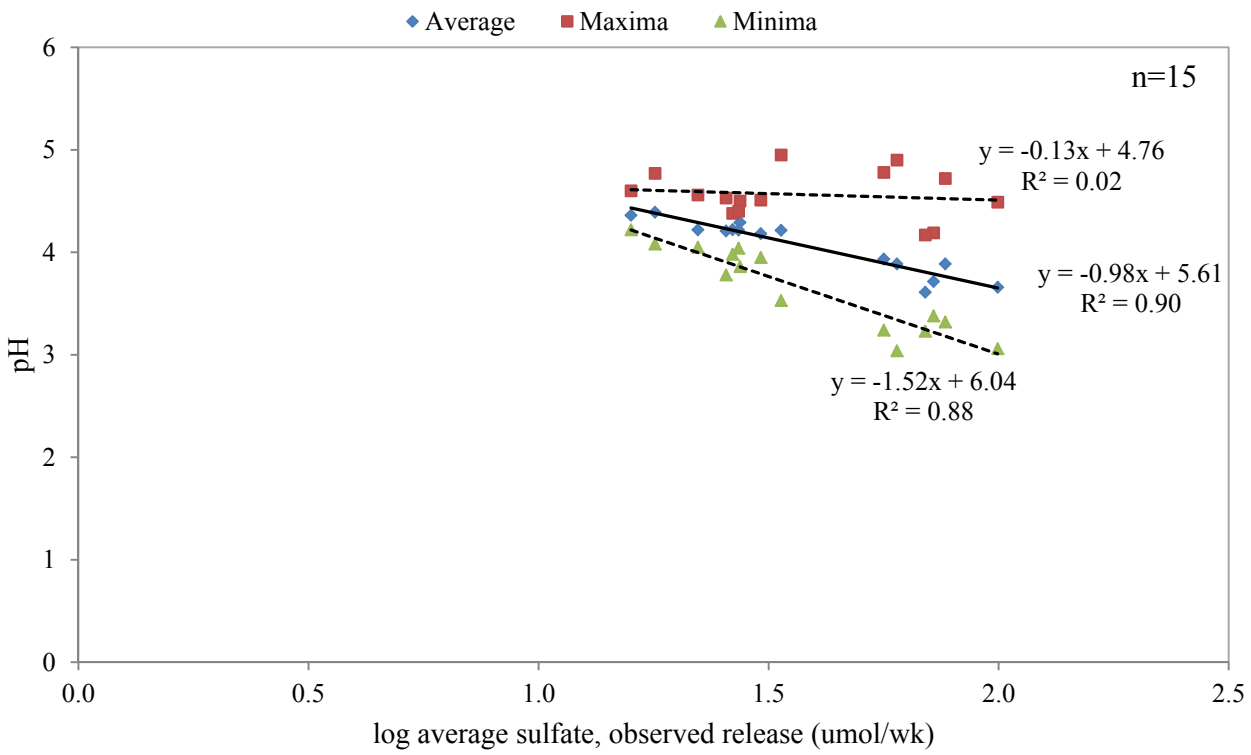


Figure 31. pH vs. sulfate release (umol/wk), period D

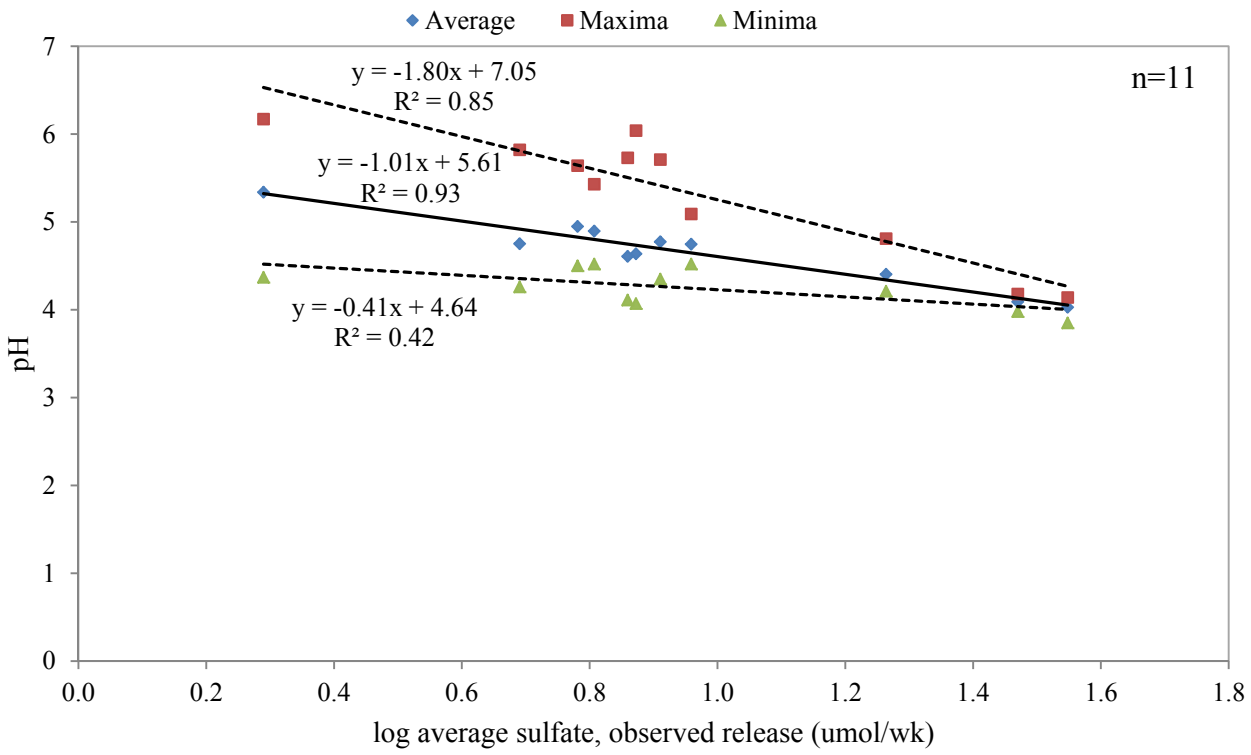


Figure 32. Calcium release vs. sulfate release (umol/wk), period A

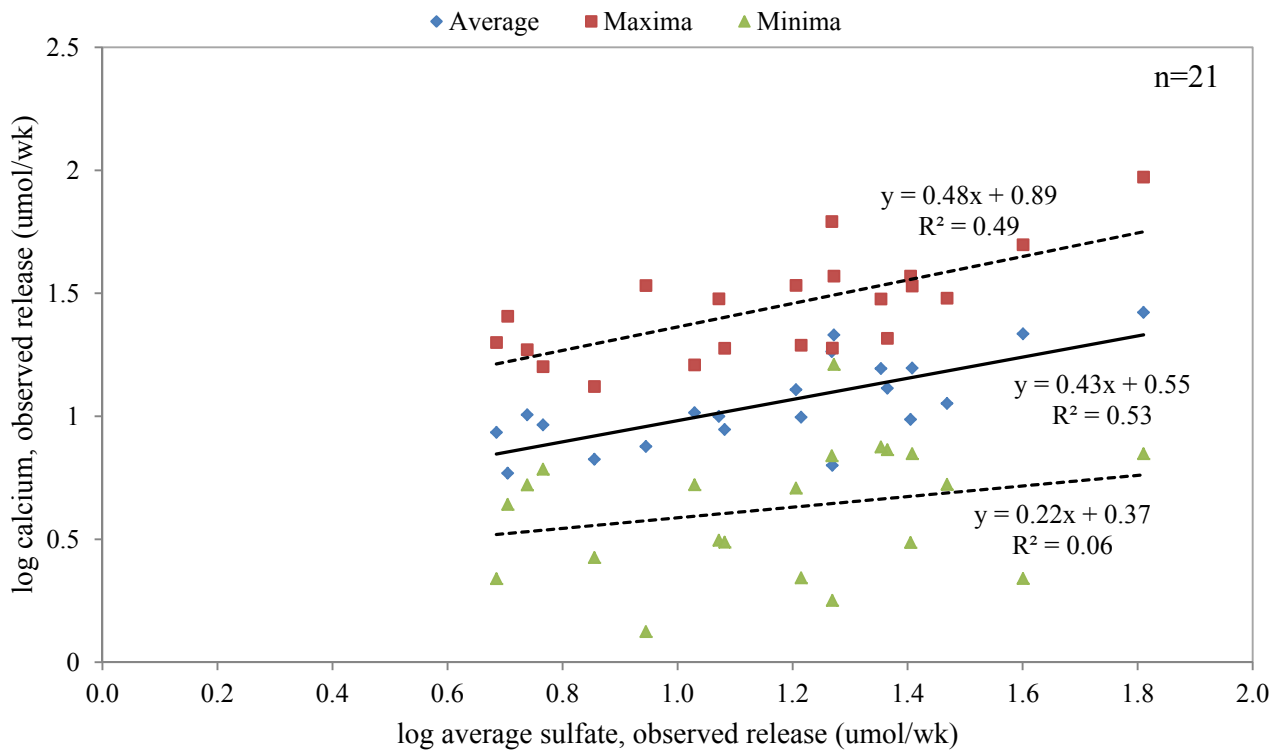


Figure 33. Calcium release vs. sulfate release (umol/wk), period B

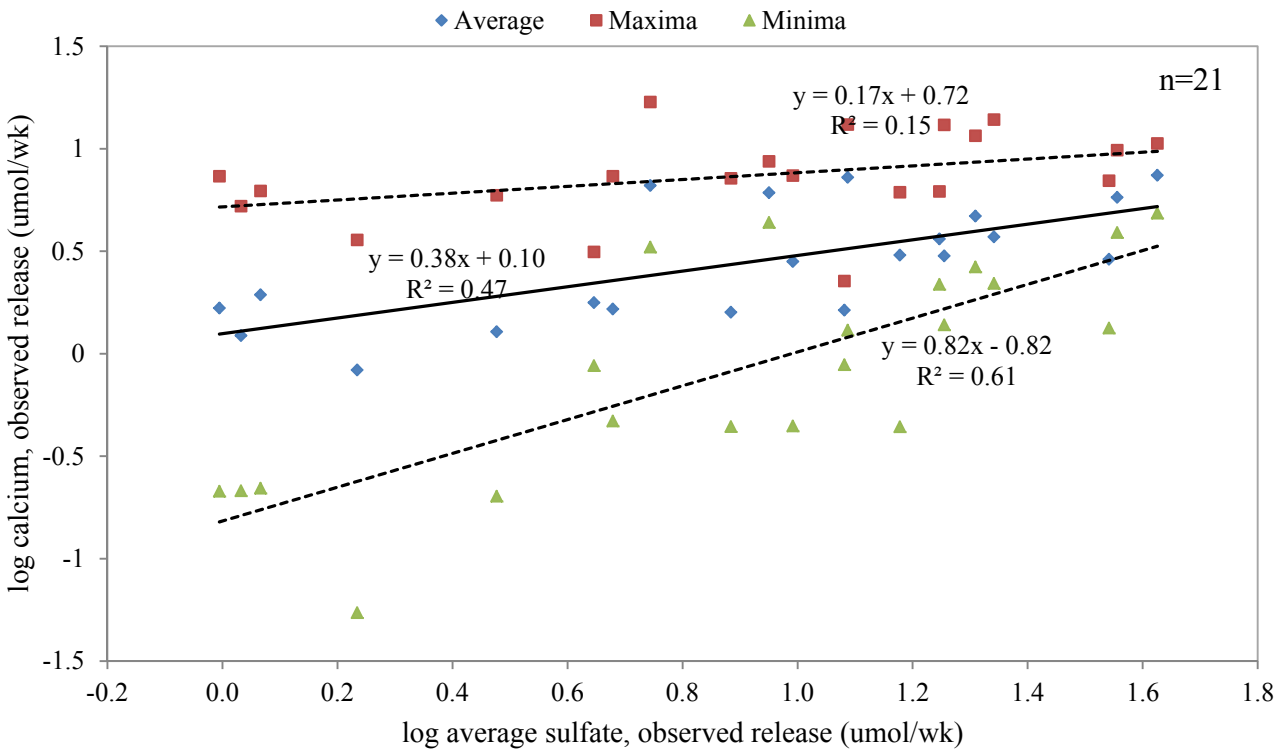


Figure 34. Calcium release vs. sulfate release (umol/wk), period C

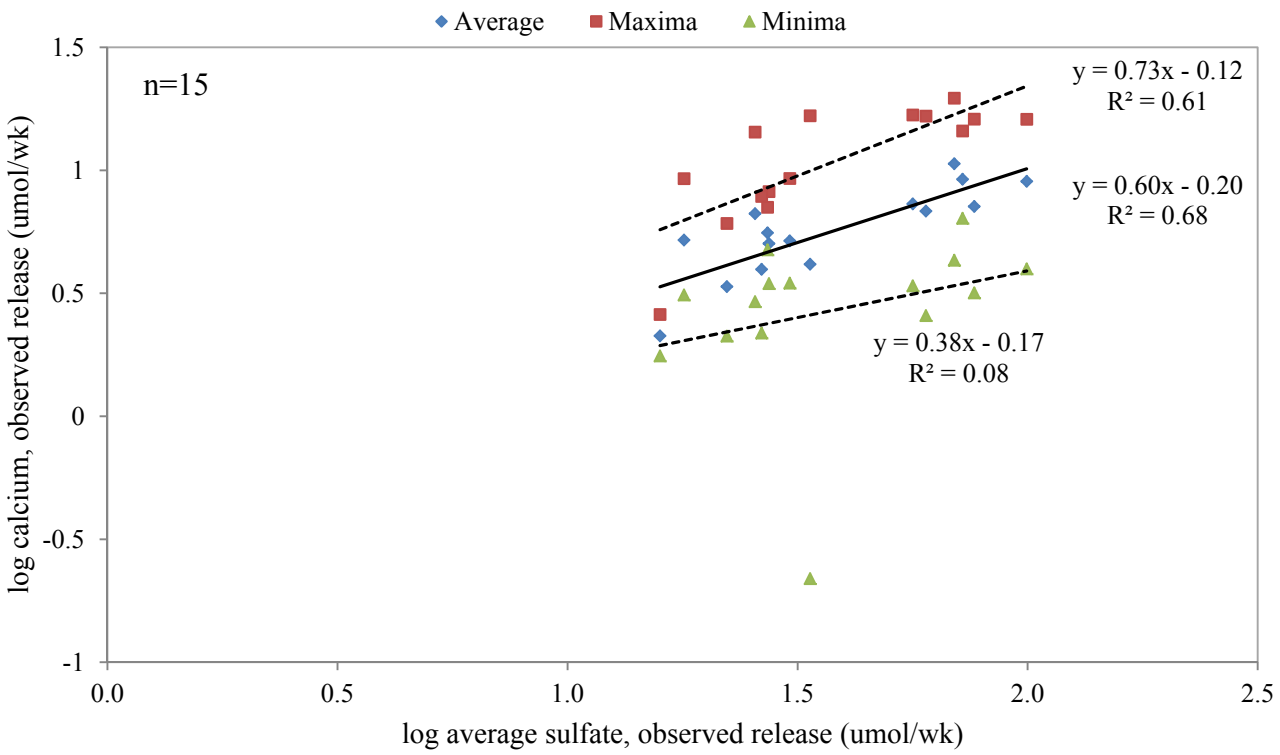


Figure 35. Calcium release vs. sulfate release (umol/wk), period D

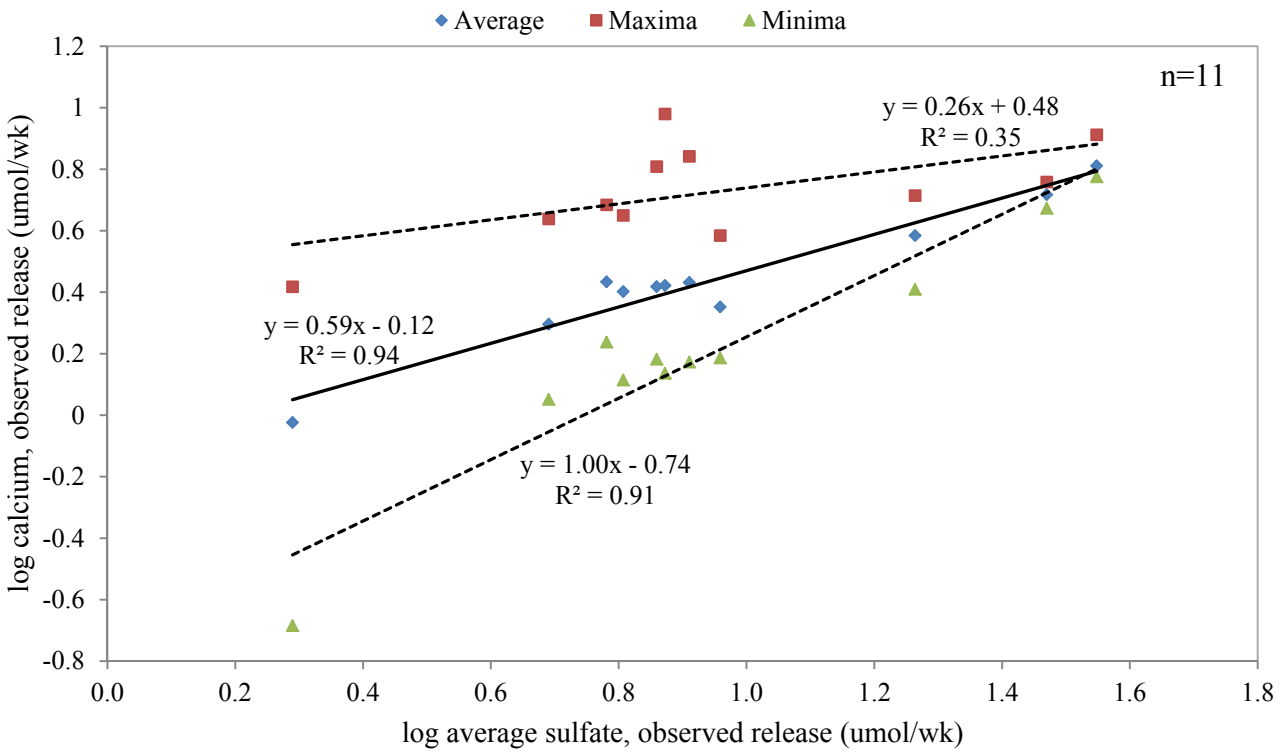


Figure 36. Magnesium release vs. sulfate release (umol/wk), period A

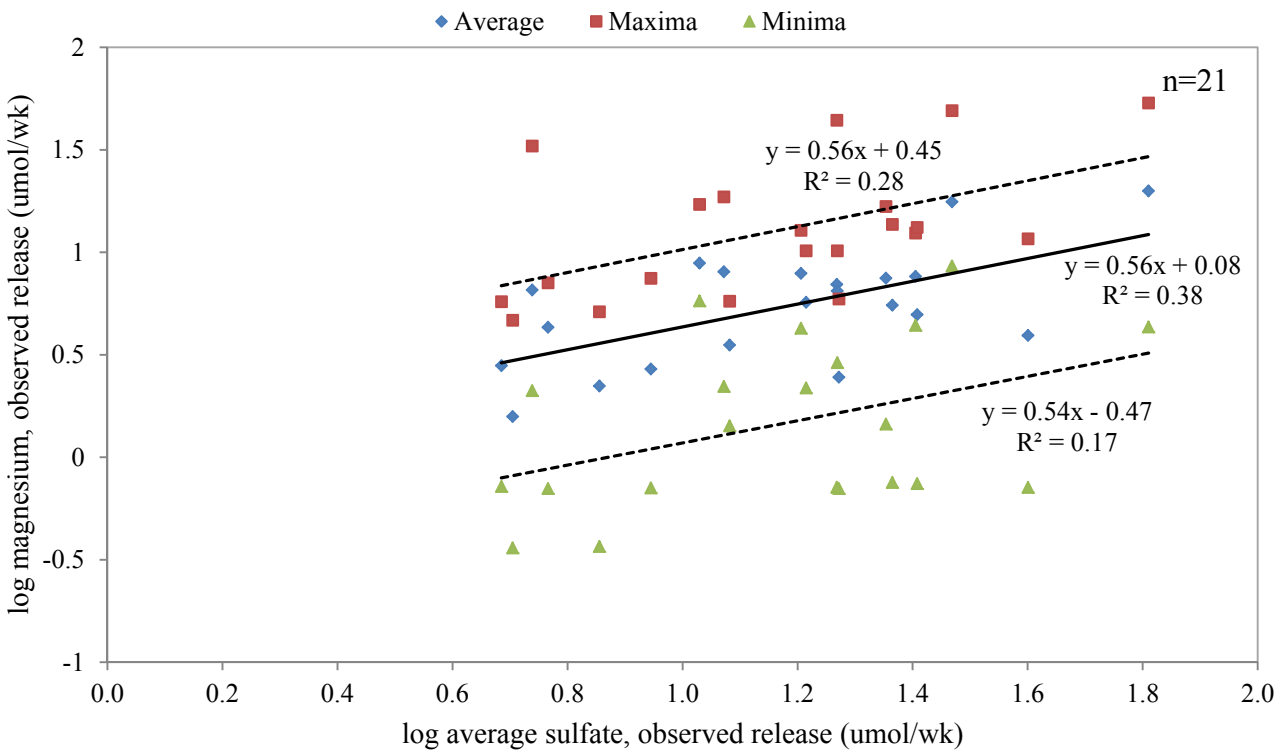


Figure 37. Magnesium release vs. sulfate release (umol/wk), period B

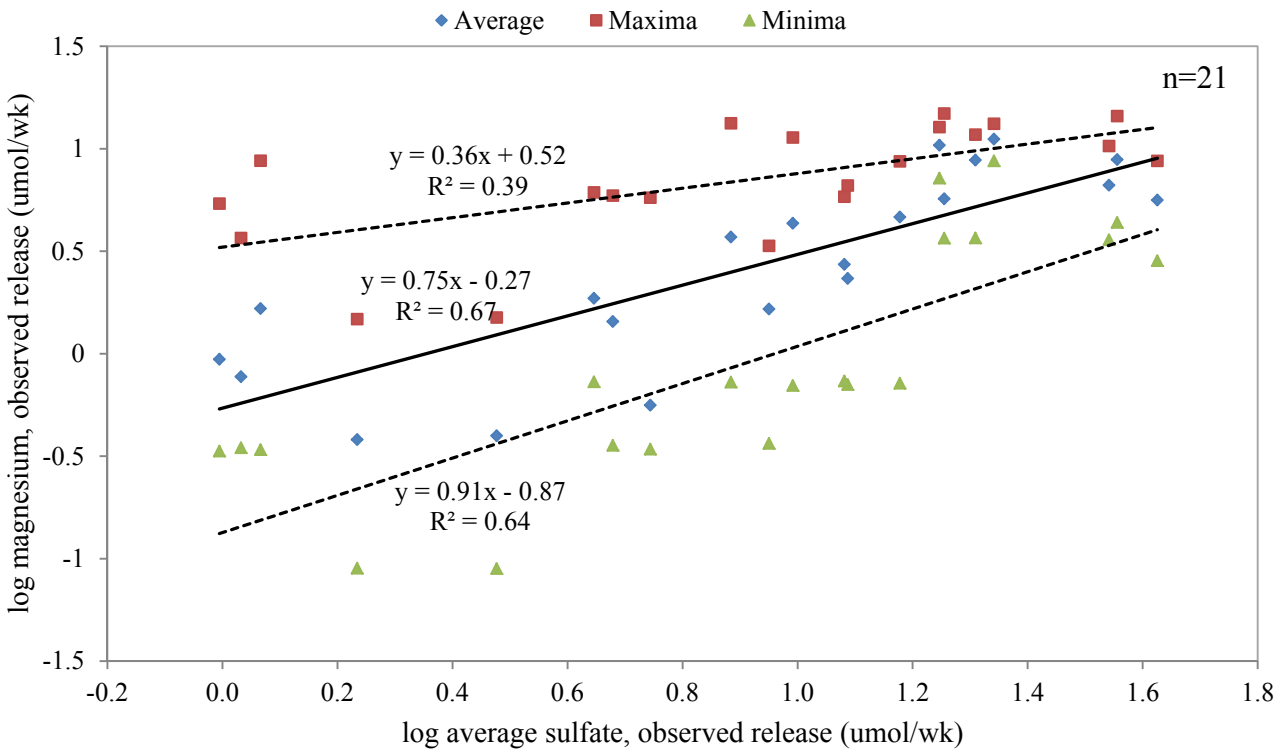


Figure 38. Magnesium release vs. sulfate release (umol/wk), period C

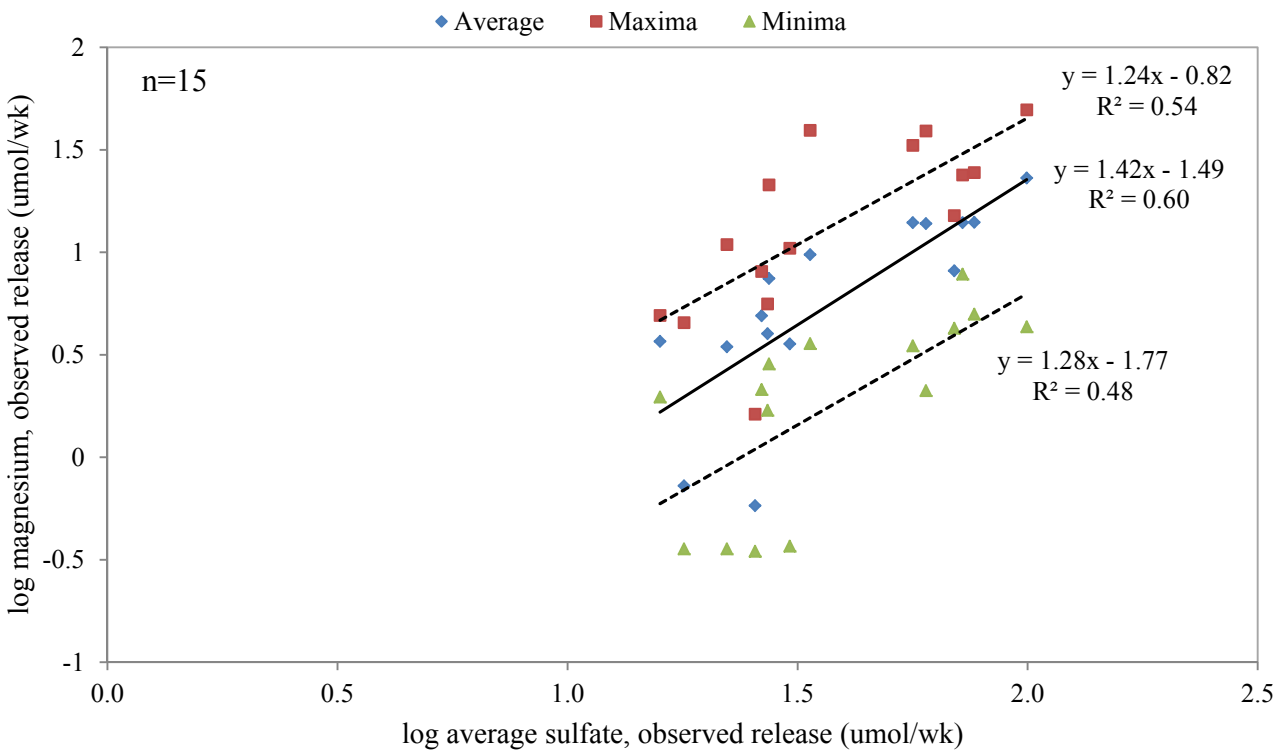


Figure 39. Magnesium release vs. sulfate release (umol/wk), period D

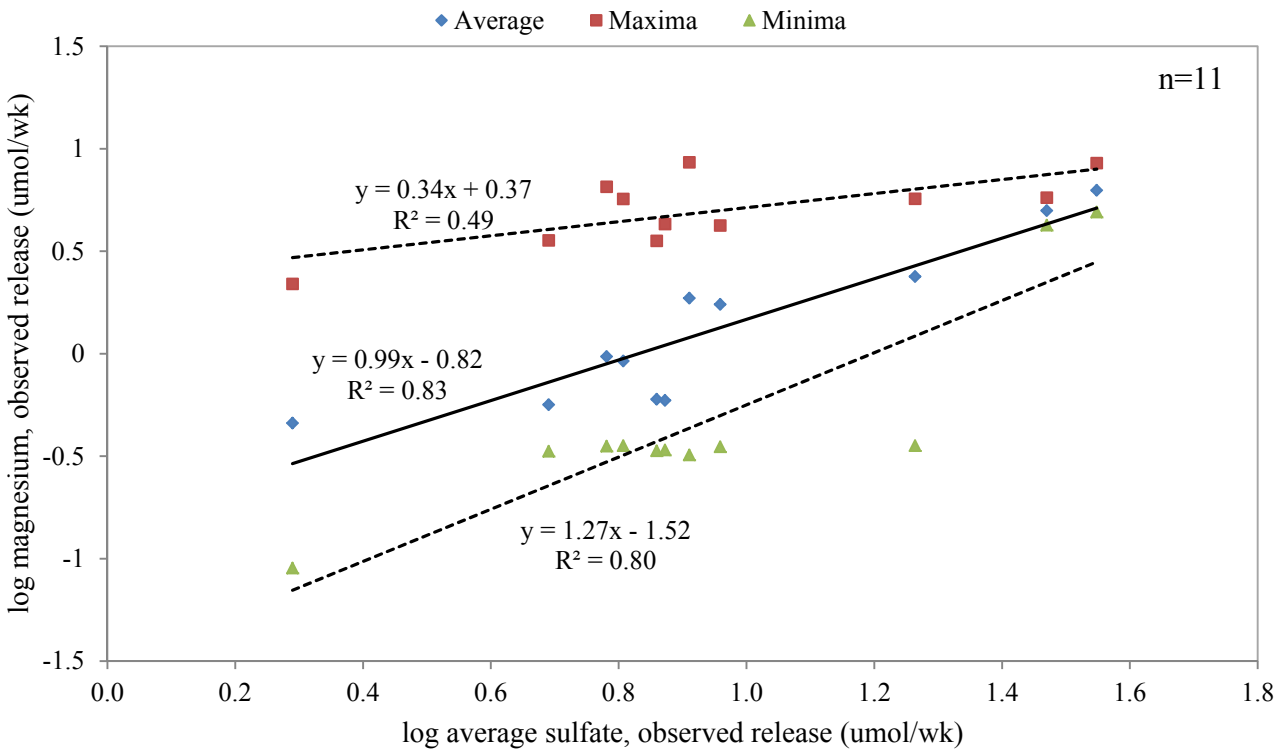


Figure 40. Ca + Mg release vs. sulfate release (umol/wk), period A

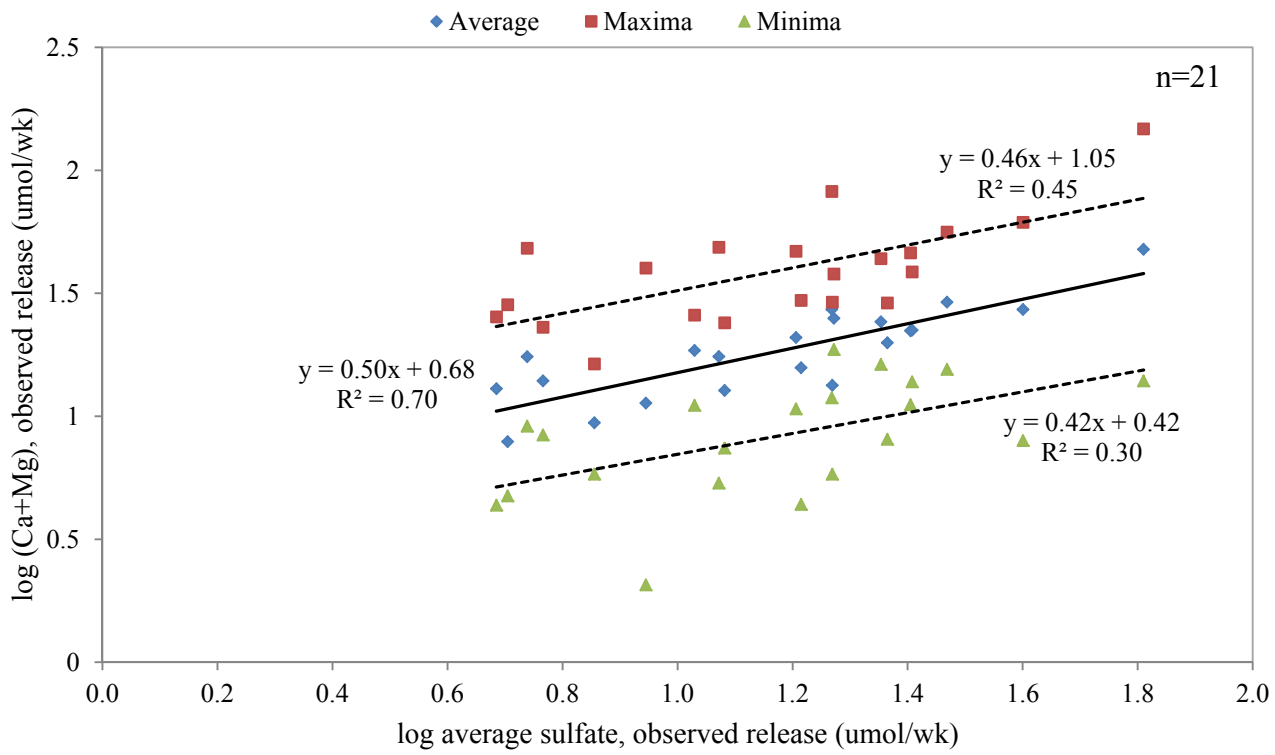


Figure 41. Ca + Mg release vs. sulfate release (umol/wk), period B

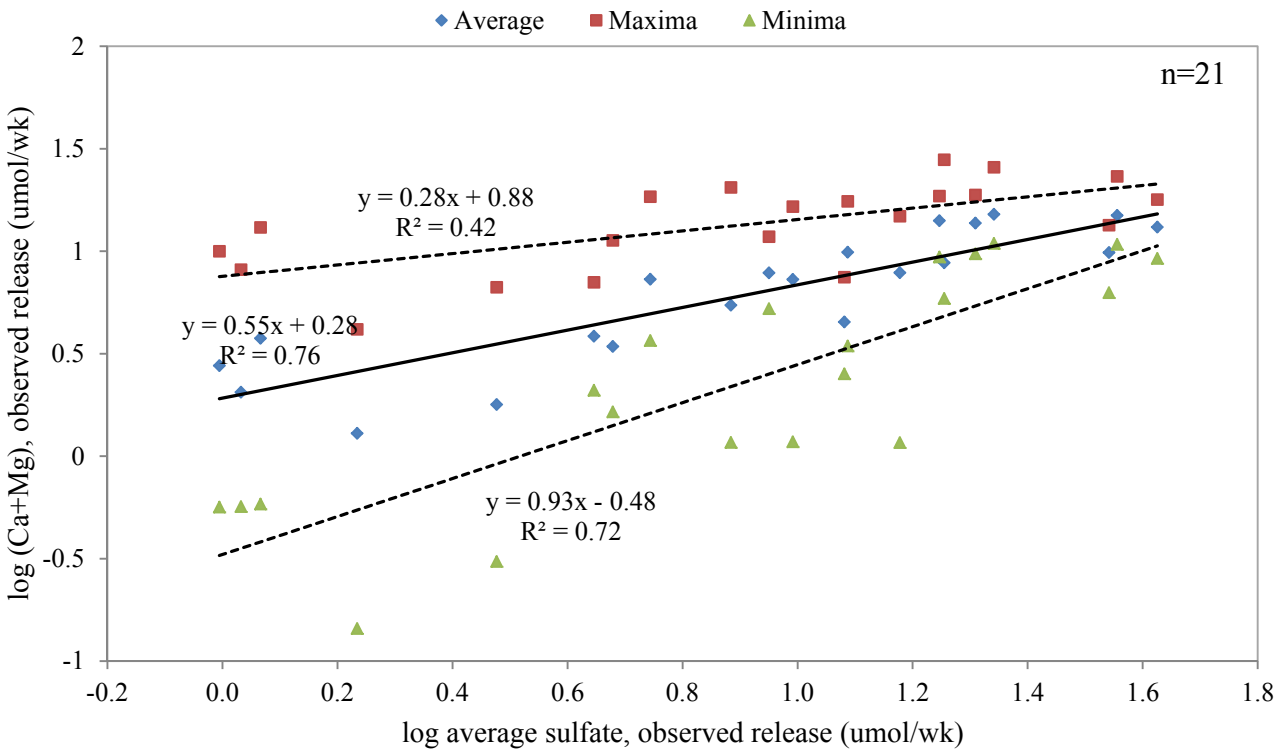


Figure 42. Ca + Mg release vs. sulfate release (umol/wk), period C

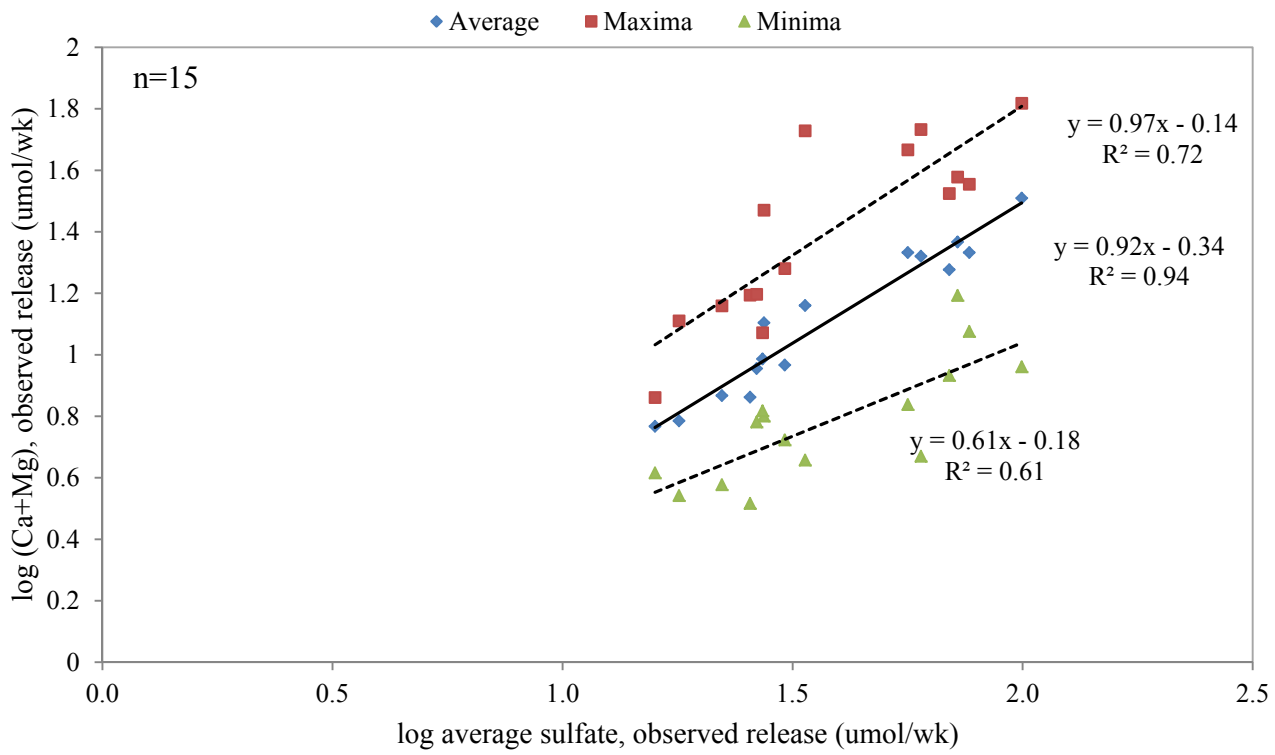


Figure 43. Ca + Mg release vs. sulfate release (umol/wk), period D

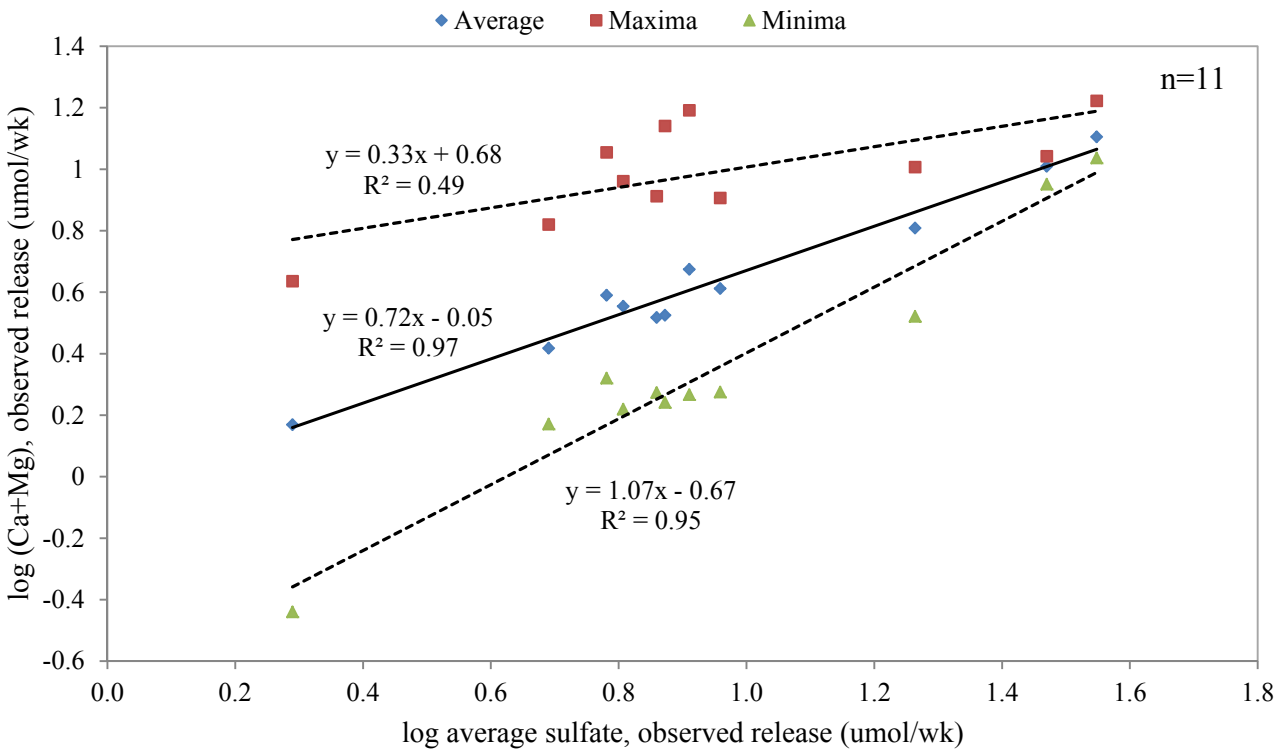


Figure 44. Cobalt composition vs. % sulfur for unleached solids

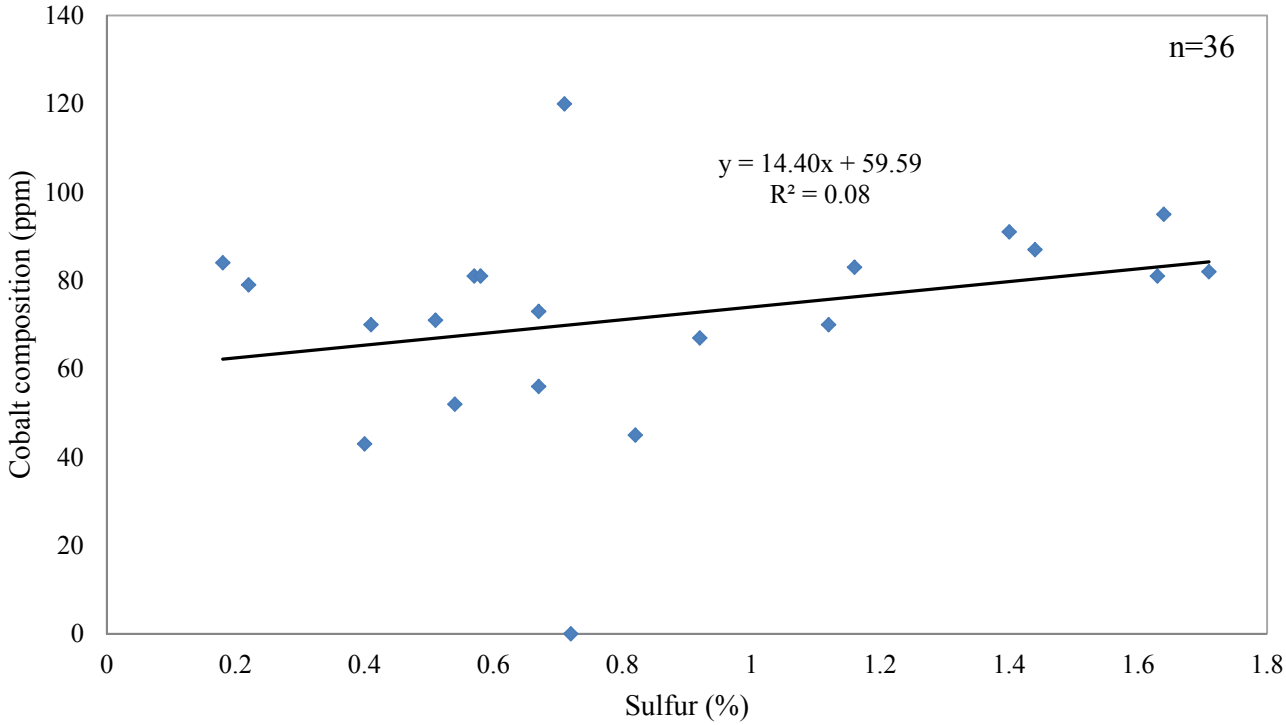


Figure 45. Copper composition vs. % sulfur for unleached solids

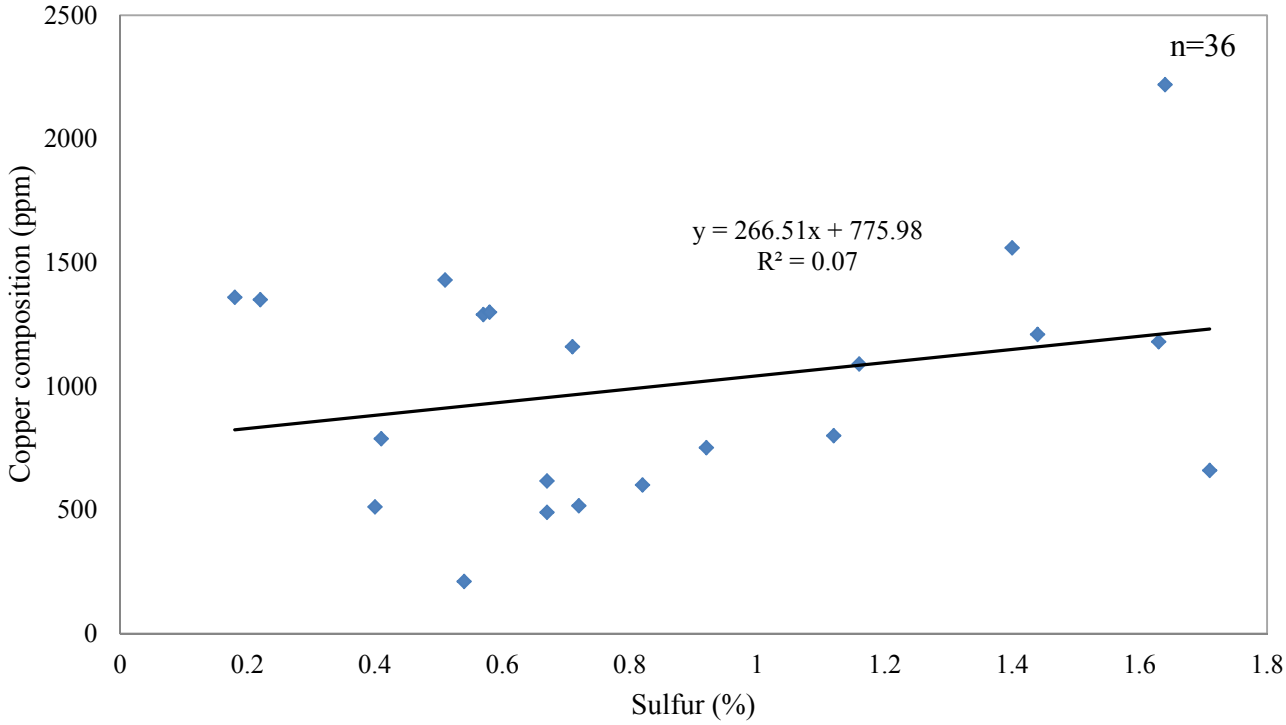


Figure 46. Nickel composition vs. % sulfur for unleached solids

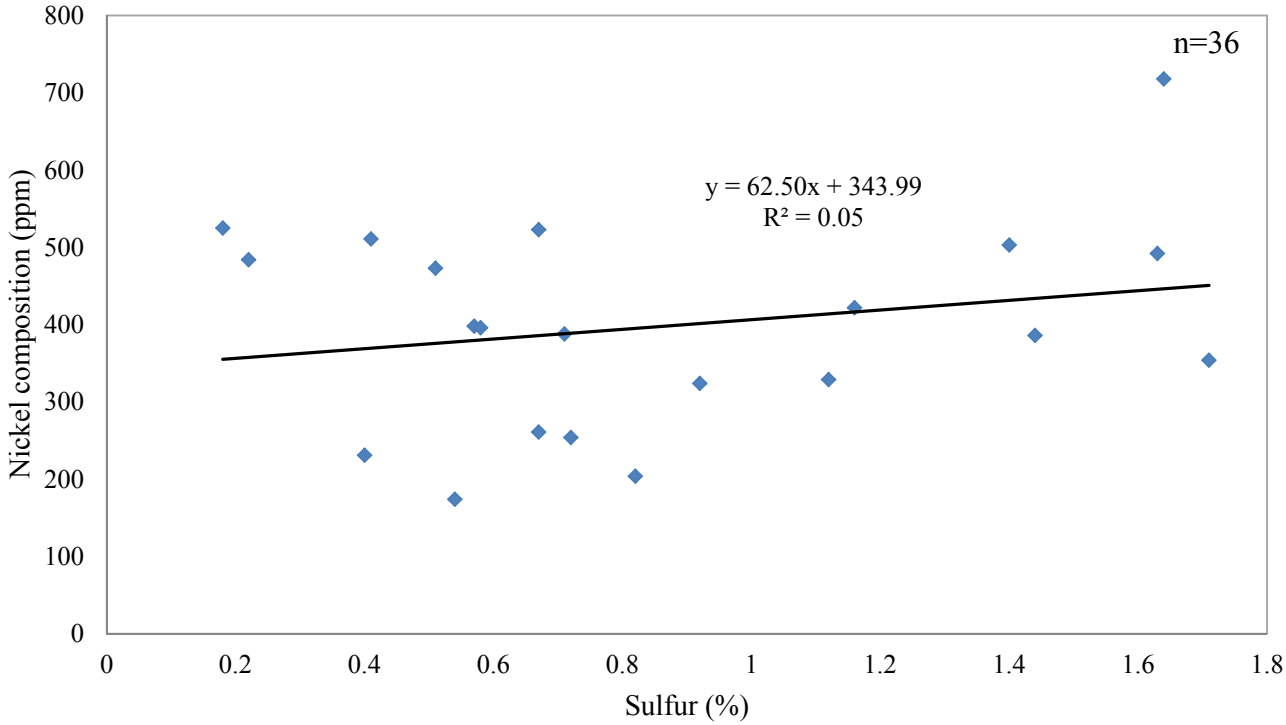


Figure 47. Zinc composition vs. % sulfur for unleached solids

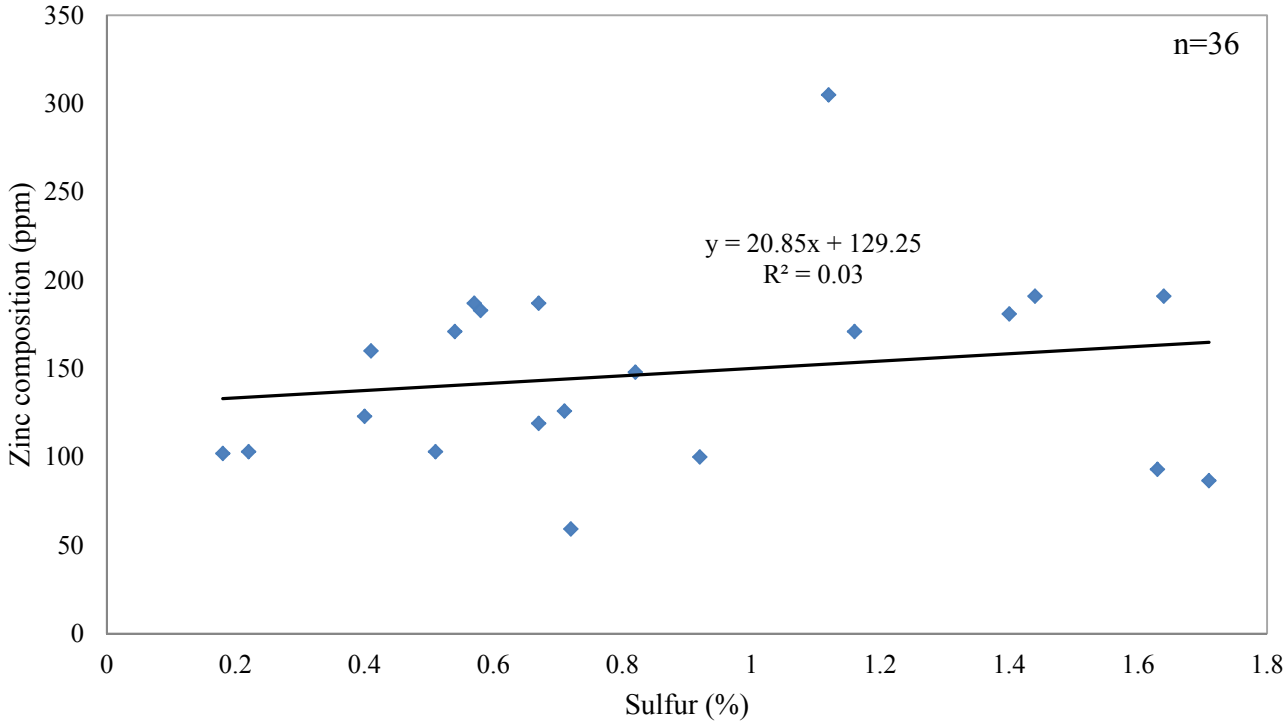


Figure 48. Cobalt concentration vs. minimum pH for all Duluth Complex samples. Metal concentrations plotted are those closest to the week of minimum pH.

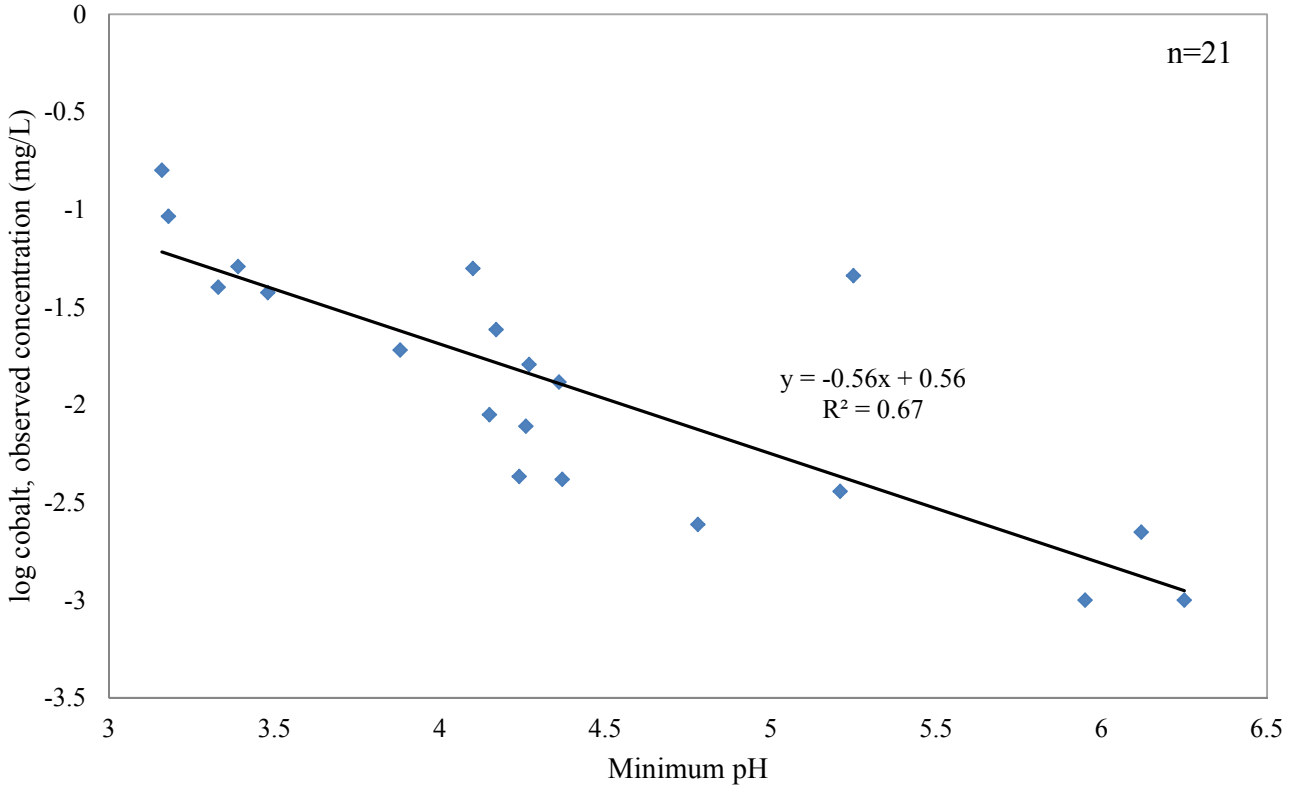


Figure 49. Copper concentration vs. minimum pH for all Duluth Complex samples. Metal concentrations plotted are those closest to the week of minimum pH.

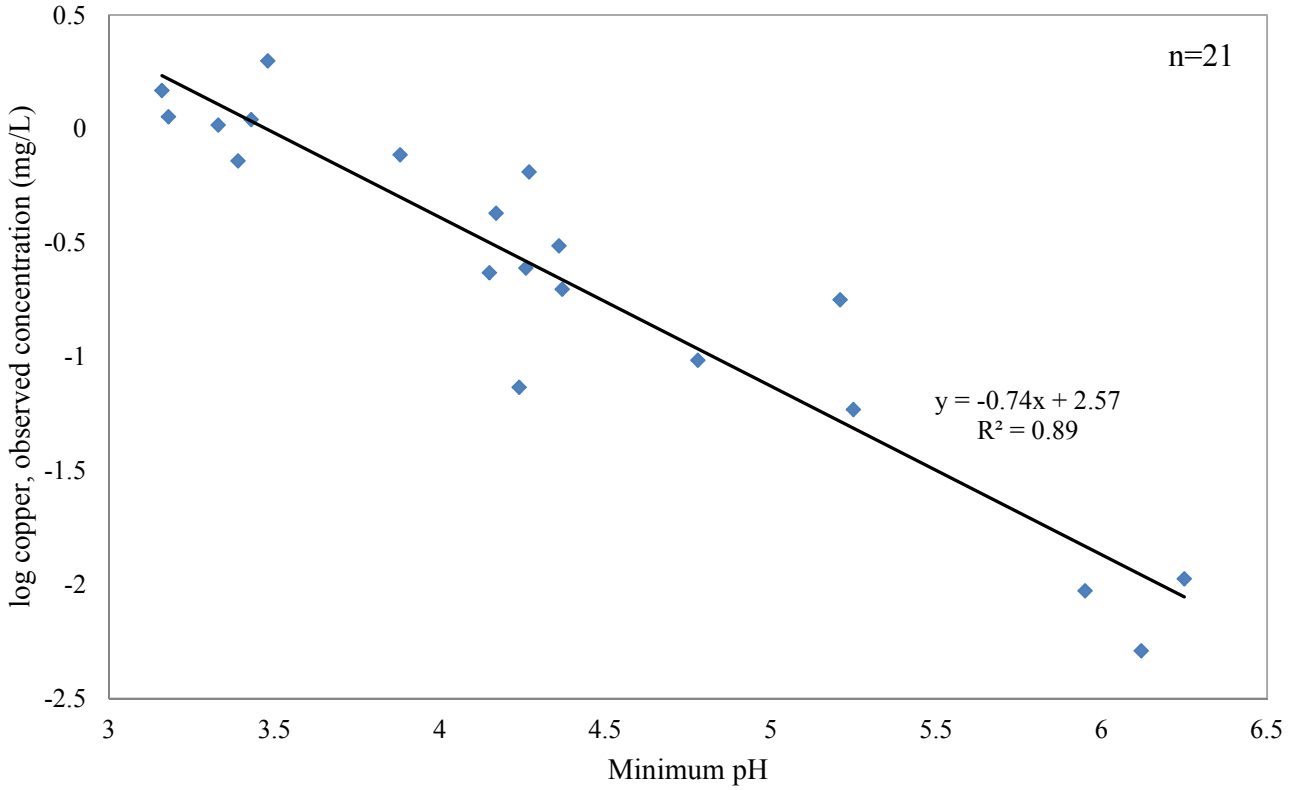


Figure 50. Nickel concentration vs. minimum pH for all Duluth Complex samples. Metal concentrations plotted are those closest to the week of minimum pH.

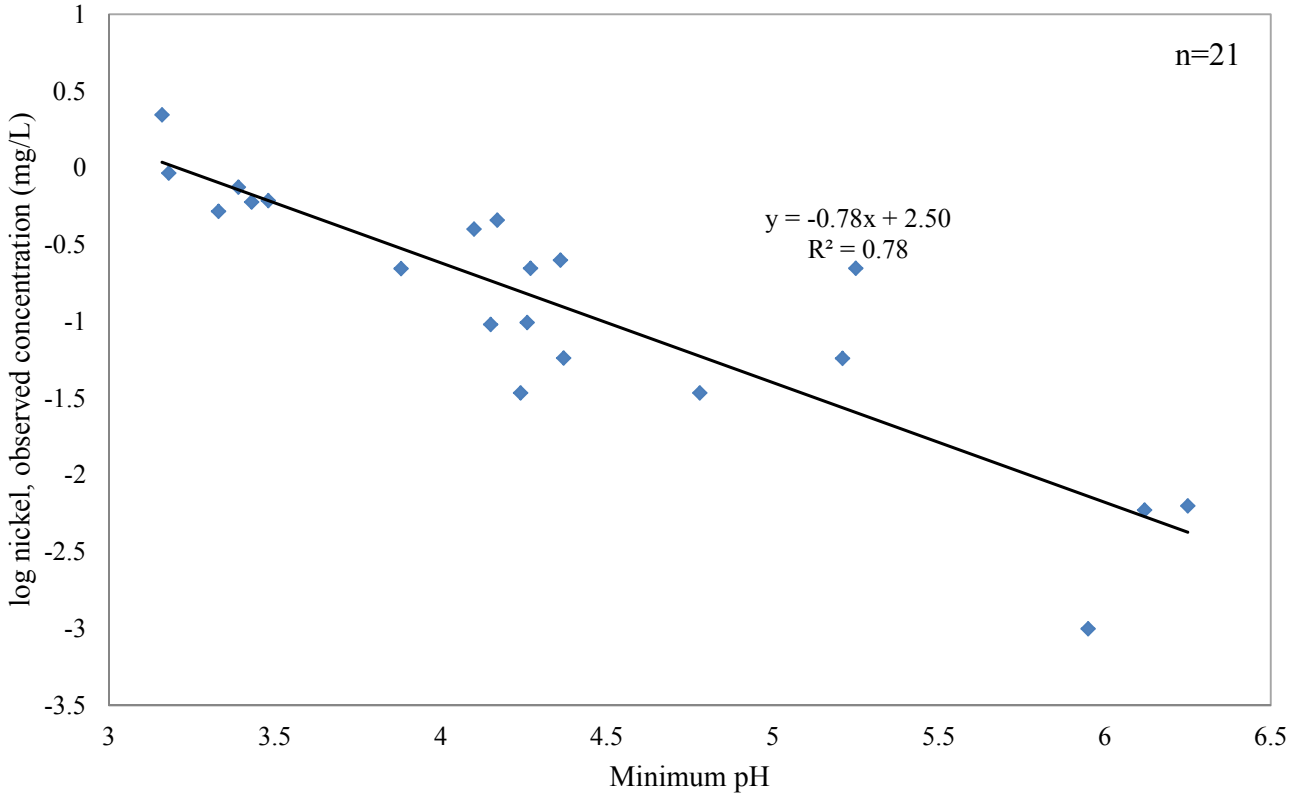


Figure 51. Zinc concentration vs. minimum pH for all Duluth Complex samples. Metal concentrations plotted are those closest to the week of minimum pH.

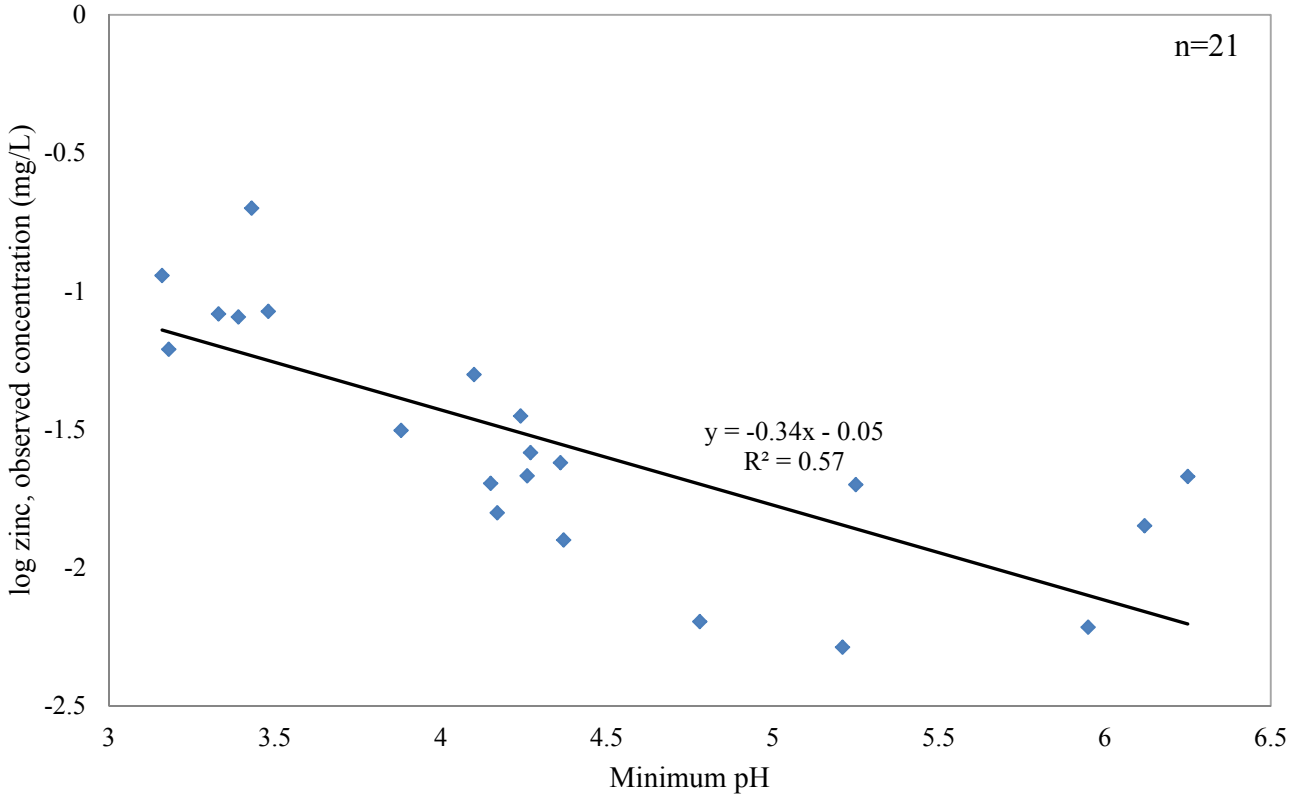


Figure 52. Iron release rate vs. drainage pH, all Duluth Complex samples, entire period of record.

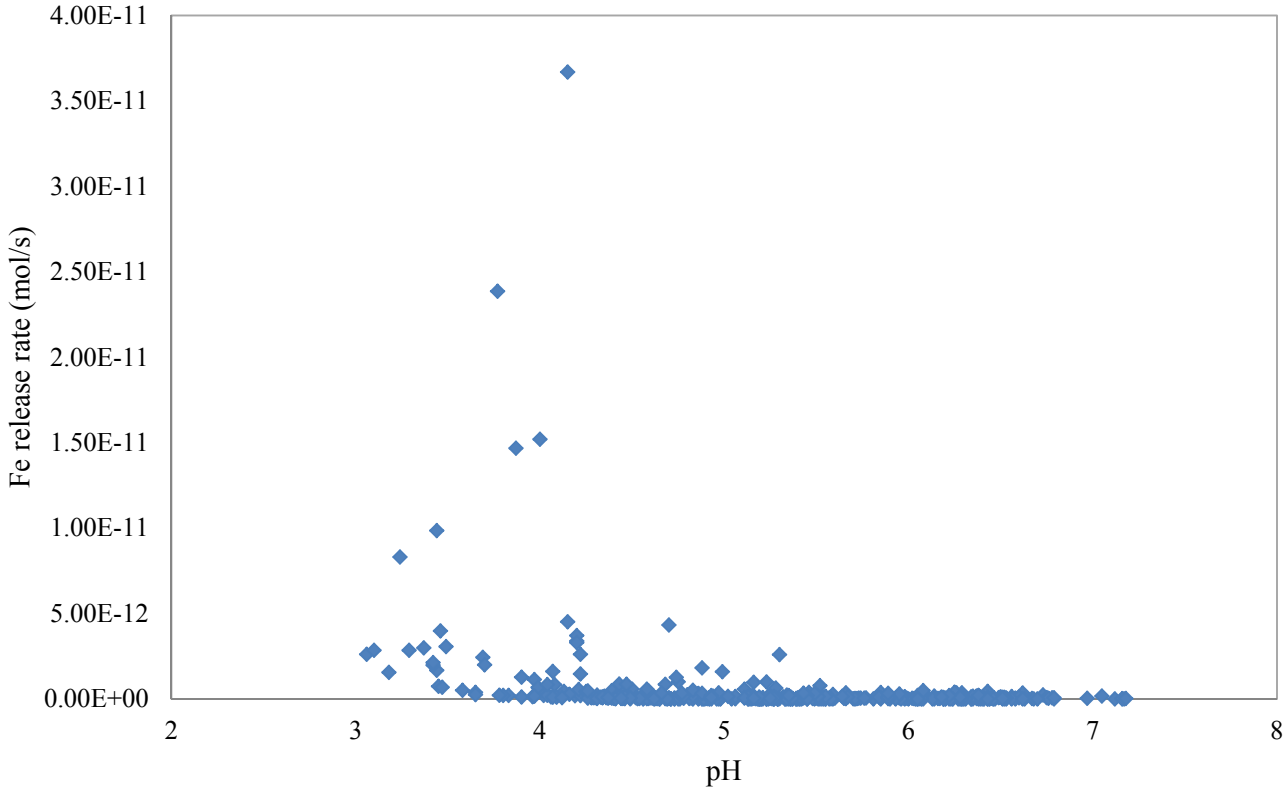


Figure 53. Aluminum release rate vs. drainage pH, all Duluth Complex samples, entire period of record.

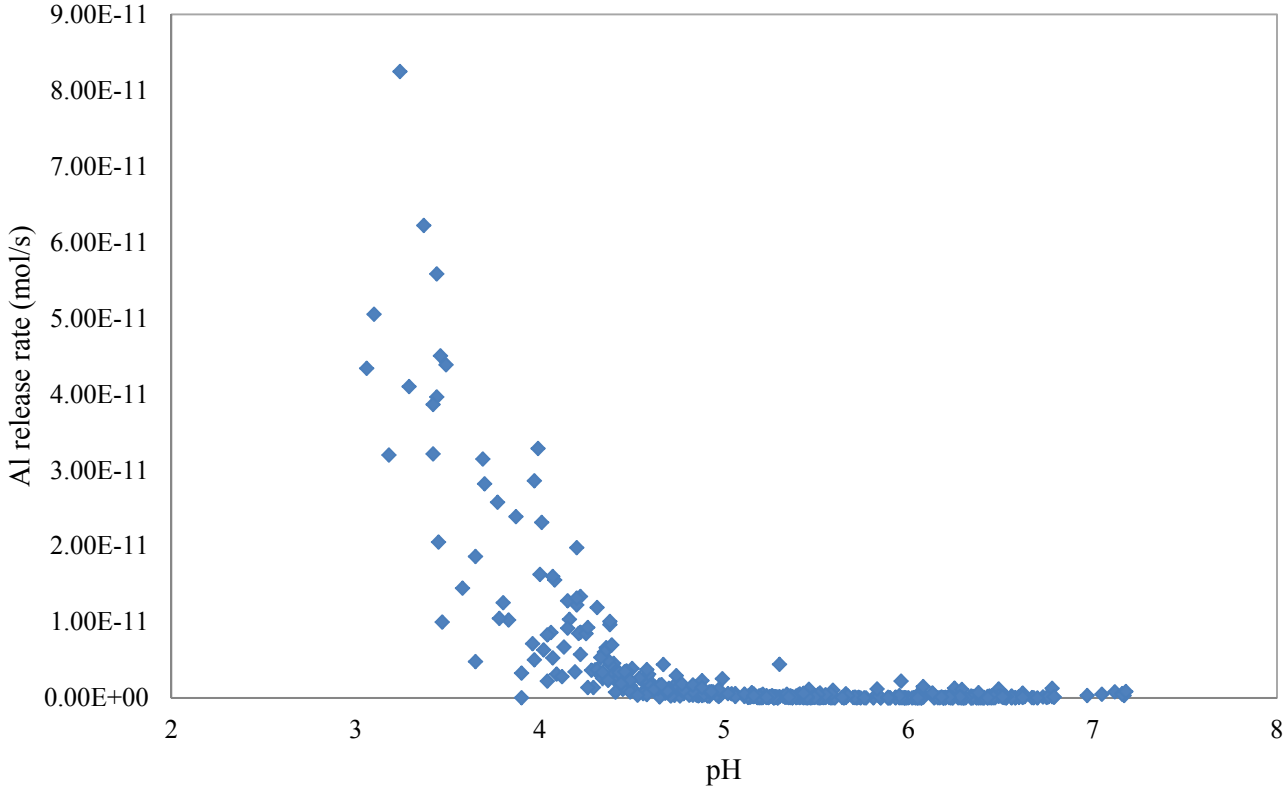


Figure 54. Silicon release rate vs. drainage pH, all Duluth Complex samples, entire period of record.

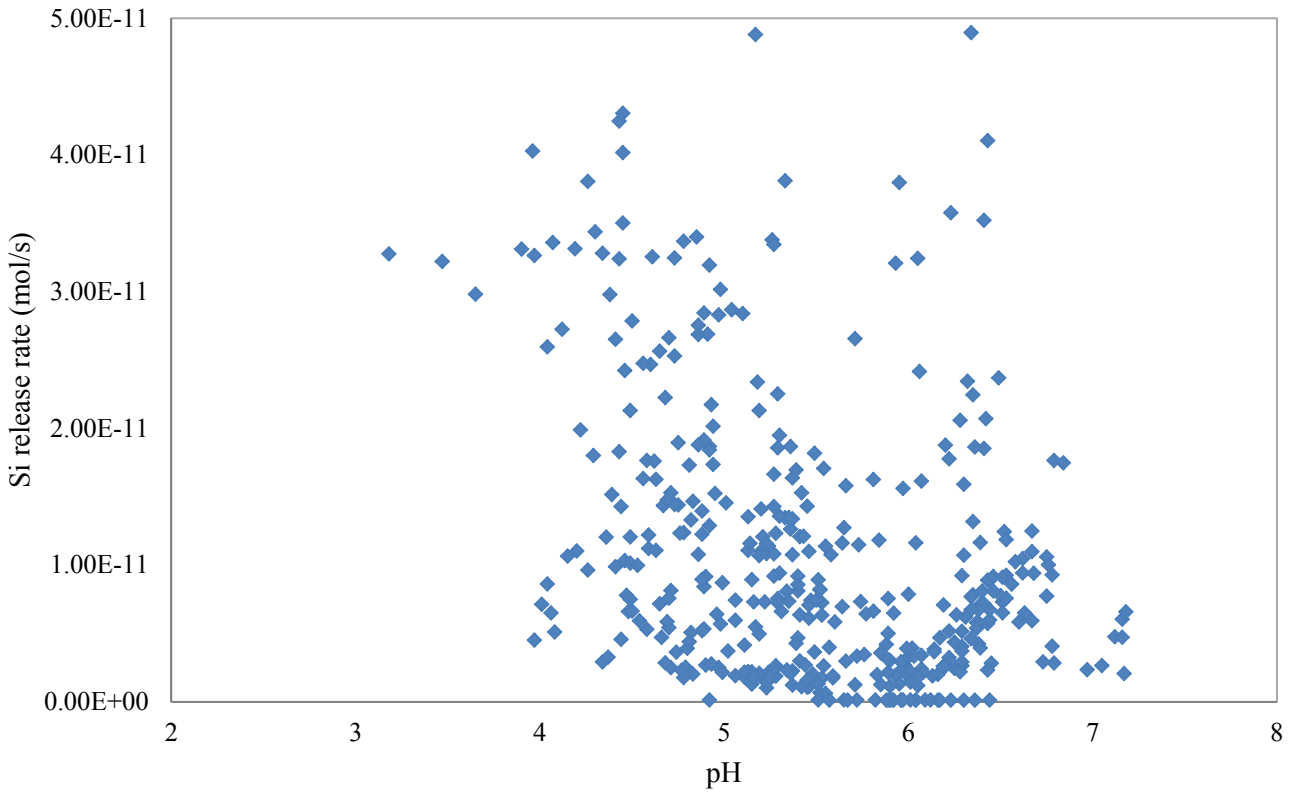


Figure 55. 0.58%S unleached sample (reactors 15 and 16).

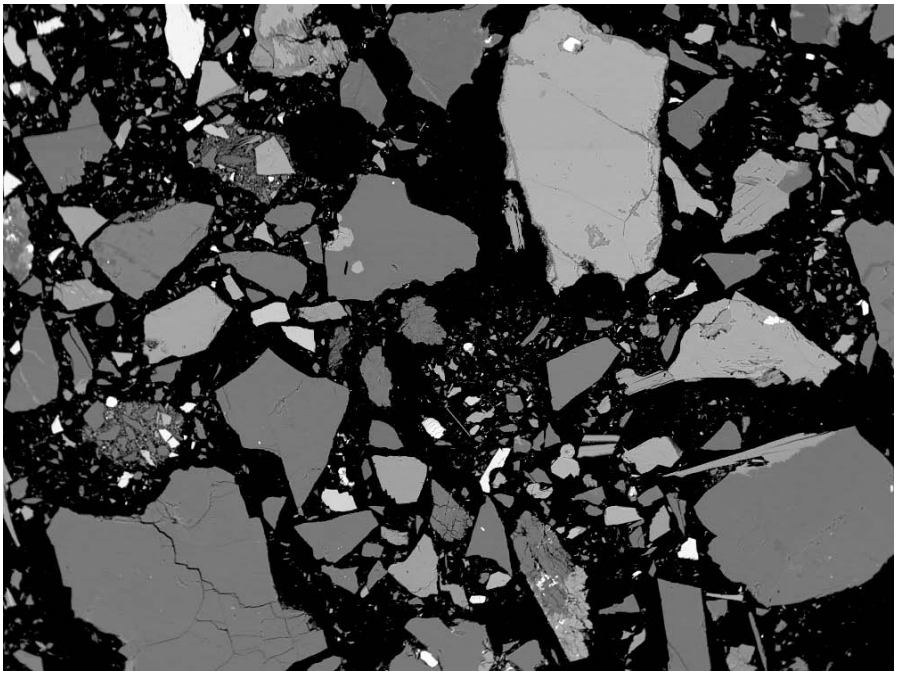


Figure 56. 0.58%S leached sample (reactor 16).

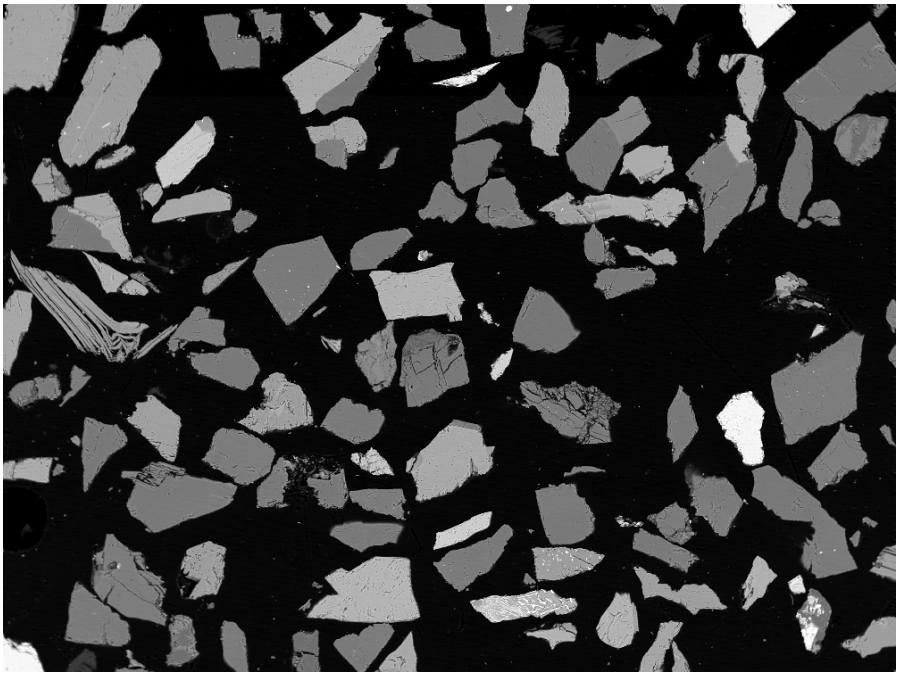


Figure 57. Small particles adhering to mineral grain surfaces as shown by comparison of washed and unwashed grains form unleached (upper) and leached (lower) samples.

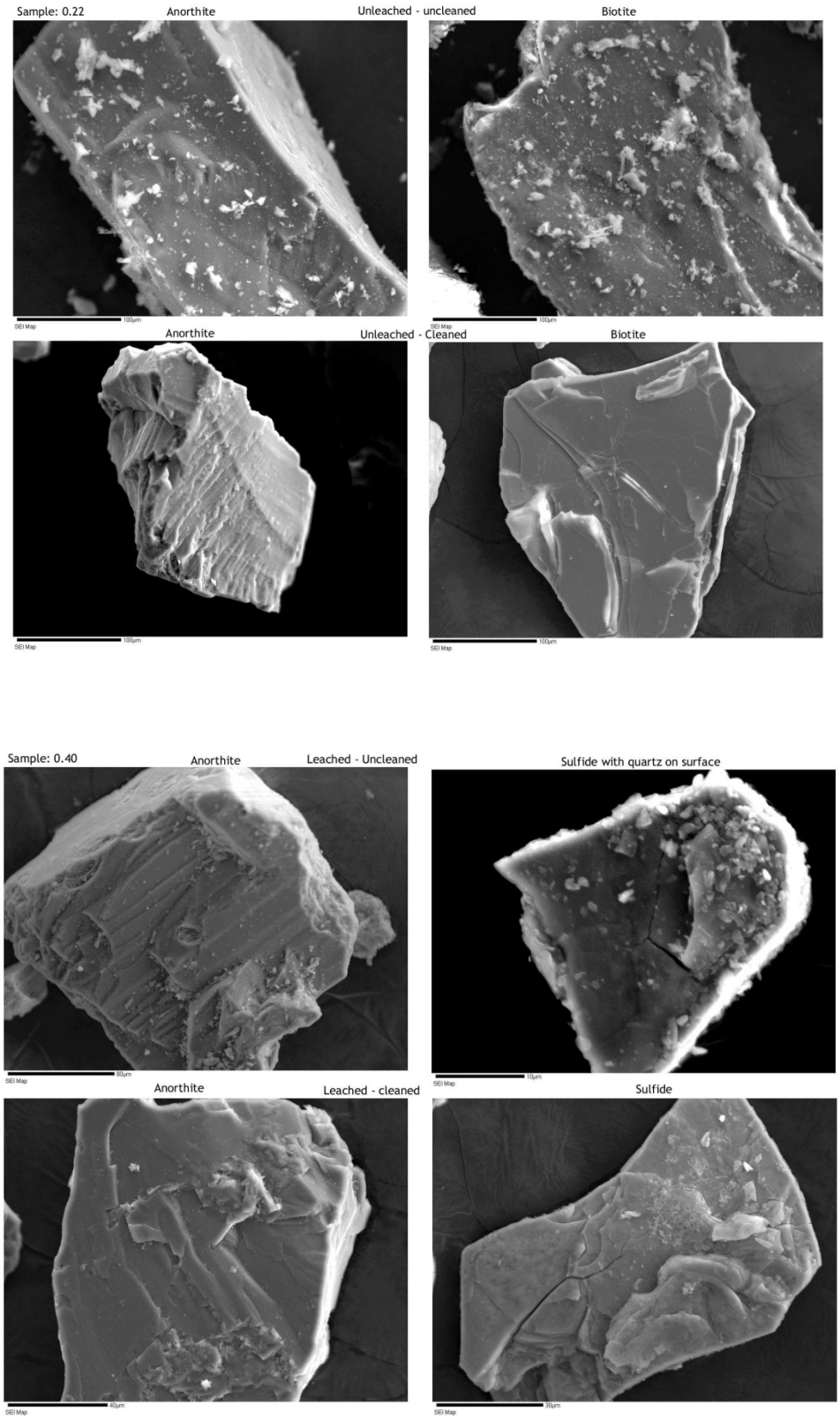


Figure 58. Pyrrhotite grain highly oxidized (dark area, 0.40%S leached sample).

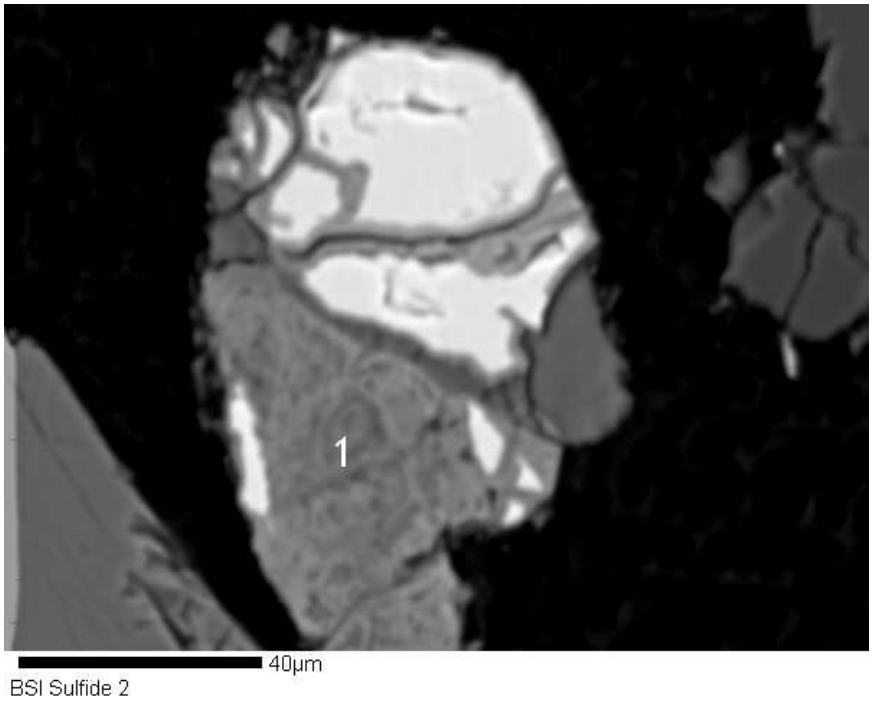


Figure 59. Chalcopyrite grain with thin coating and little oxidation (0.41%S leached sample).

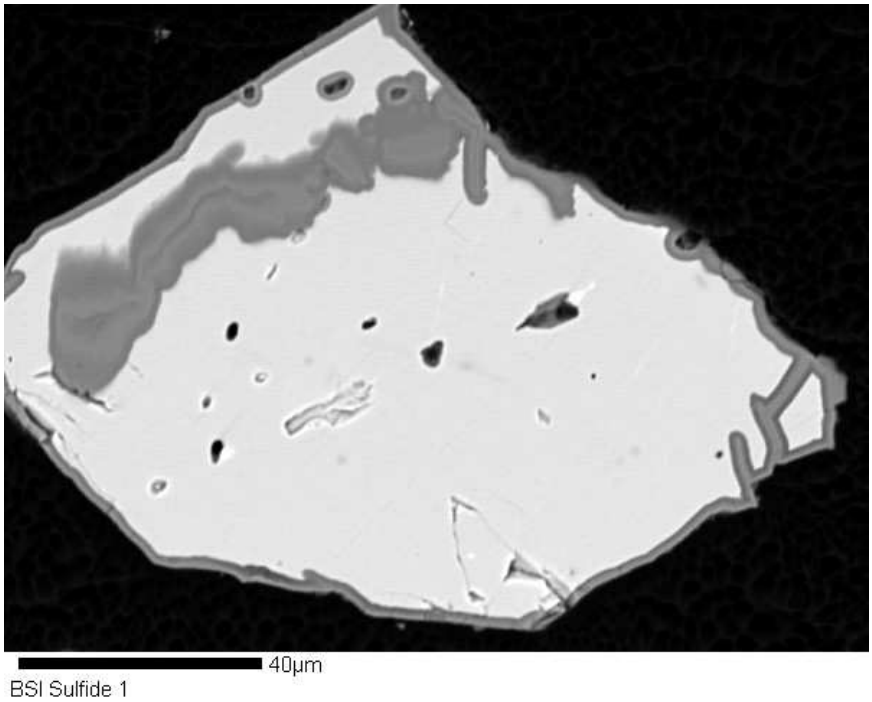


Figure 60. Potential percentage error in point counts. With the number of grains measured in this report (~100), the visual point count method has a 2-σ error level (86.5% confidence) that makes it possible that true values of mineral abundance under 10% could be 60-100% different from estimates. Samples with no grains of a certain mineral detected could have as much as 2% of the sample present as the undetected mineral grains

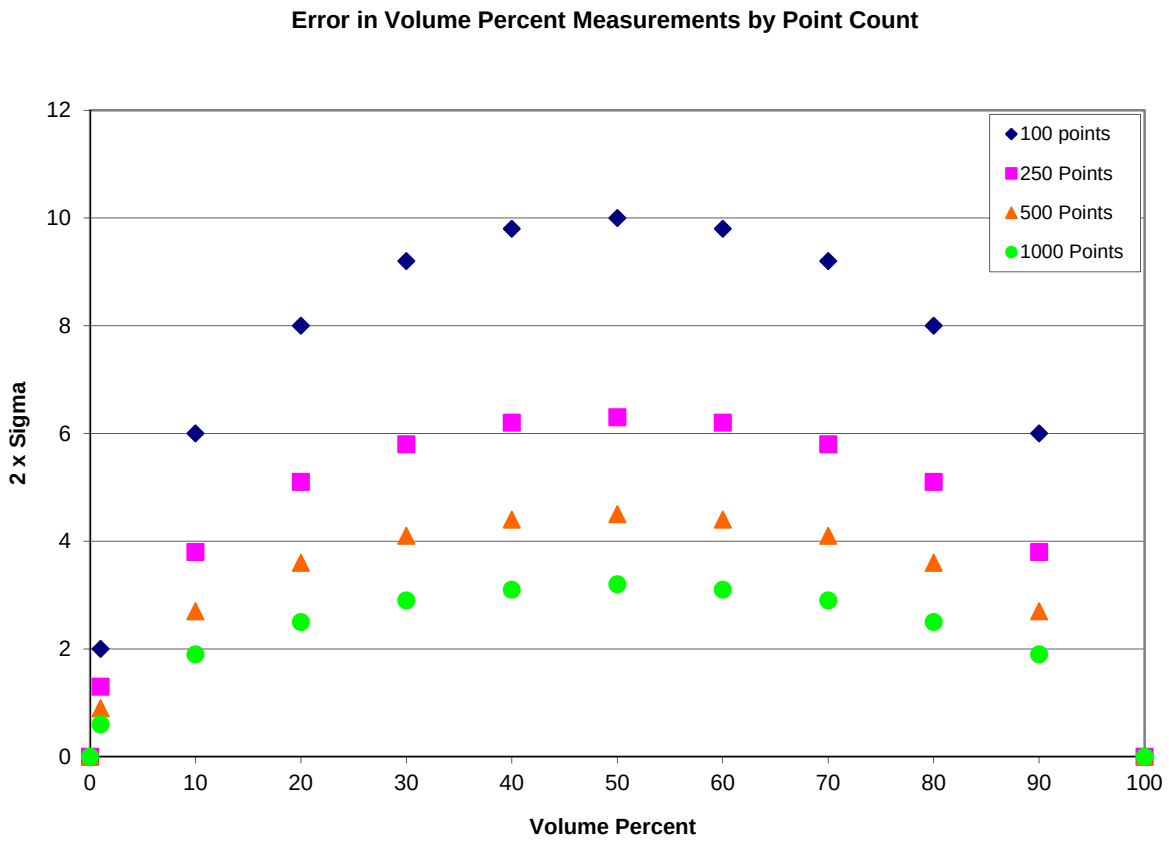


Figure 61. Comparison of neutralization potential calculations for Dunka blast hole samples.

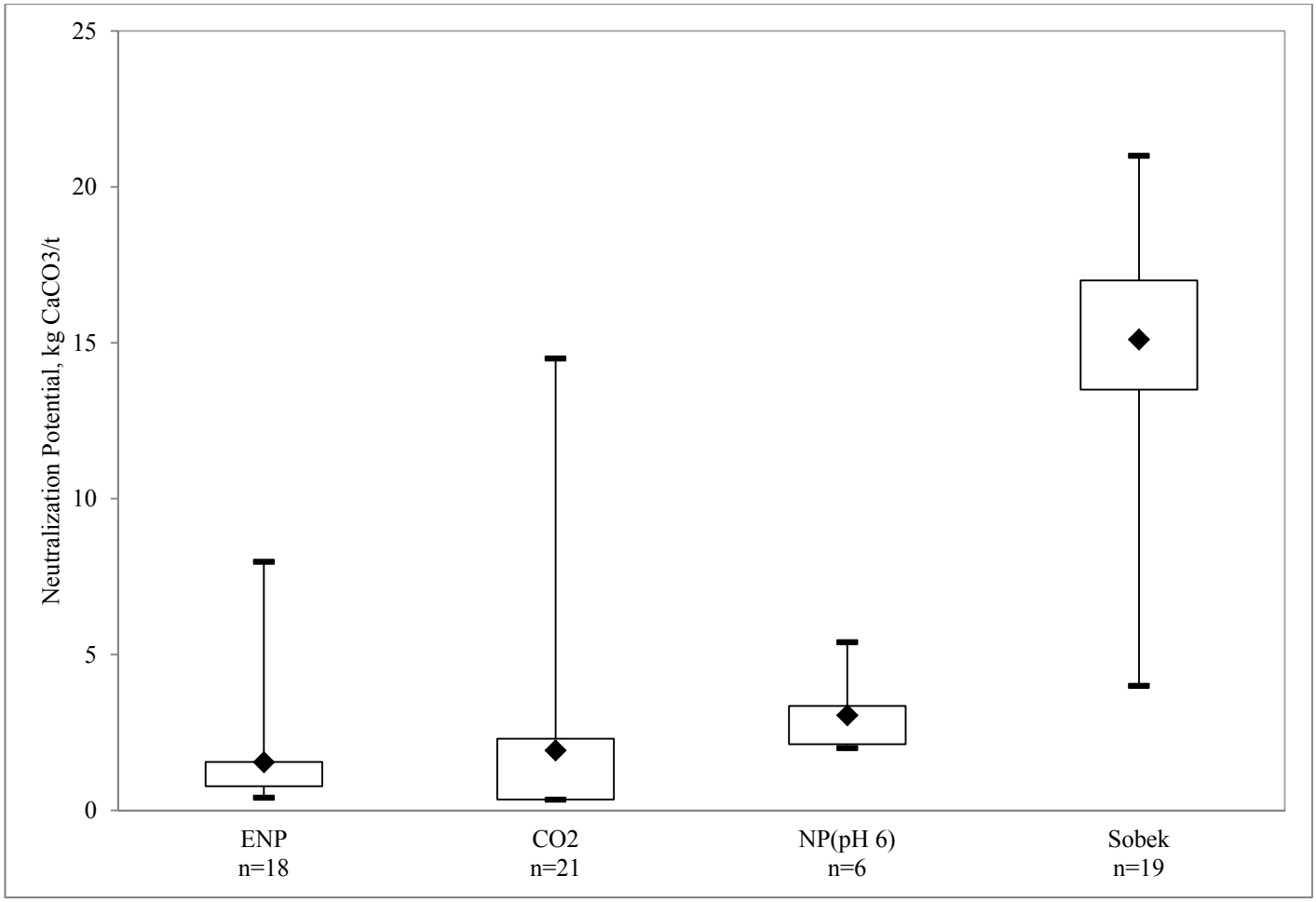


Figure 62. Ratio of calculated: observed nickel release from olivine vs sulfur content. Good correlation with data in periods C and D.

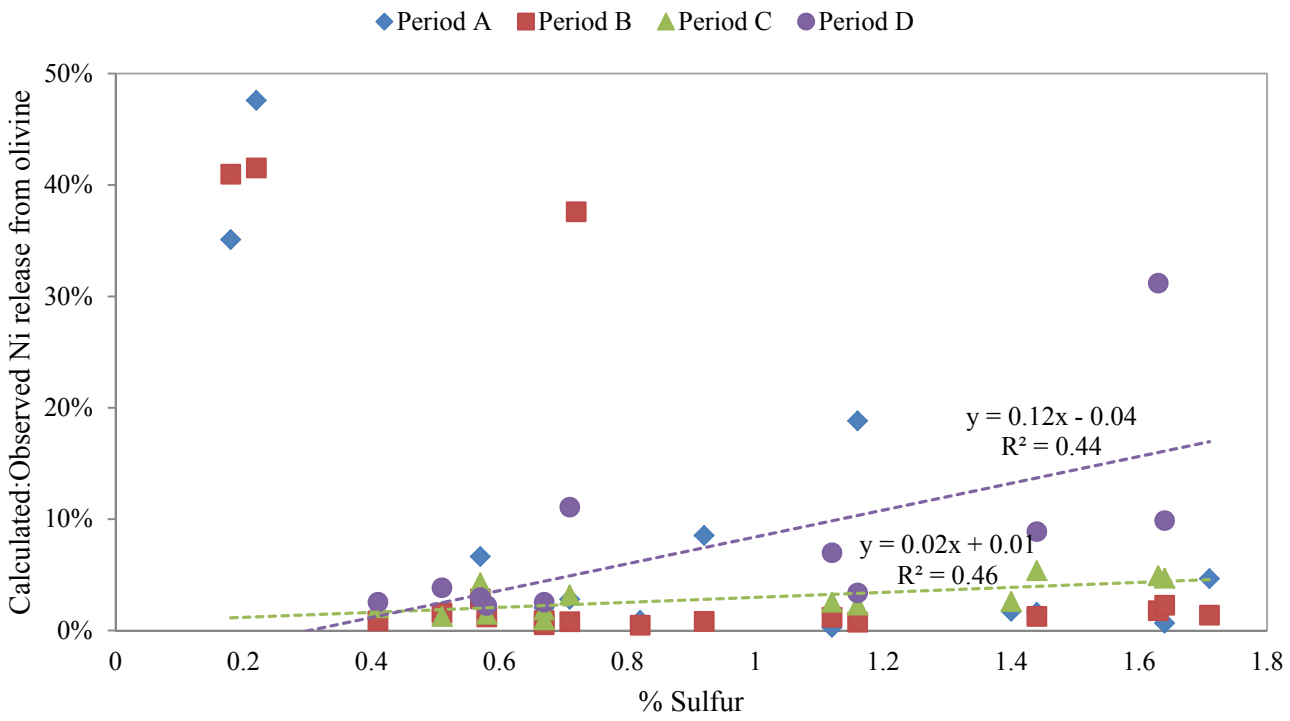


Figure 63. Moles of Nickel / moles of olivine / week vs sulfur content. Good correlation with data in periods B and C.

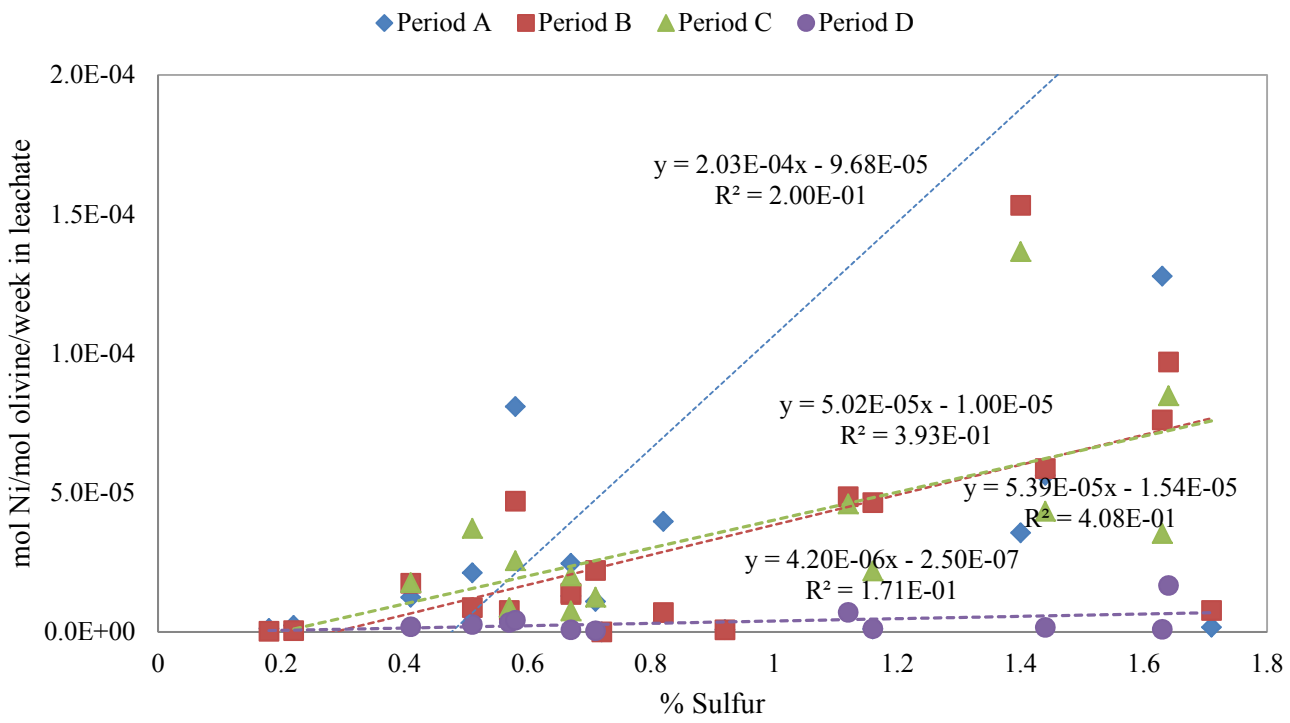


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Figure 64. Ratio of predicted Ni release (based on leachate sulfate) to observed Ni release at days when Ni was sampled. Data for lower dNi/dt are inset.

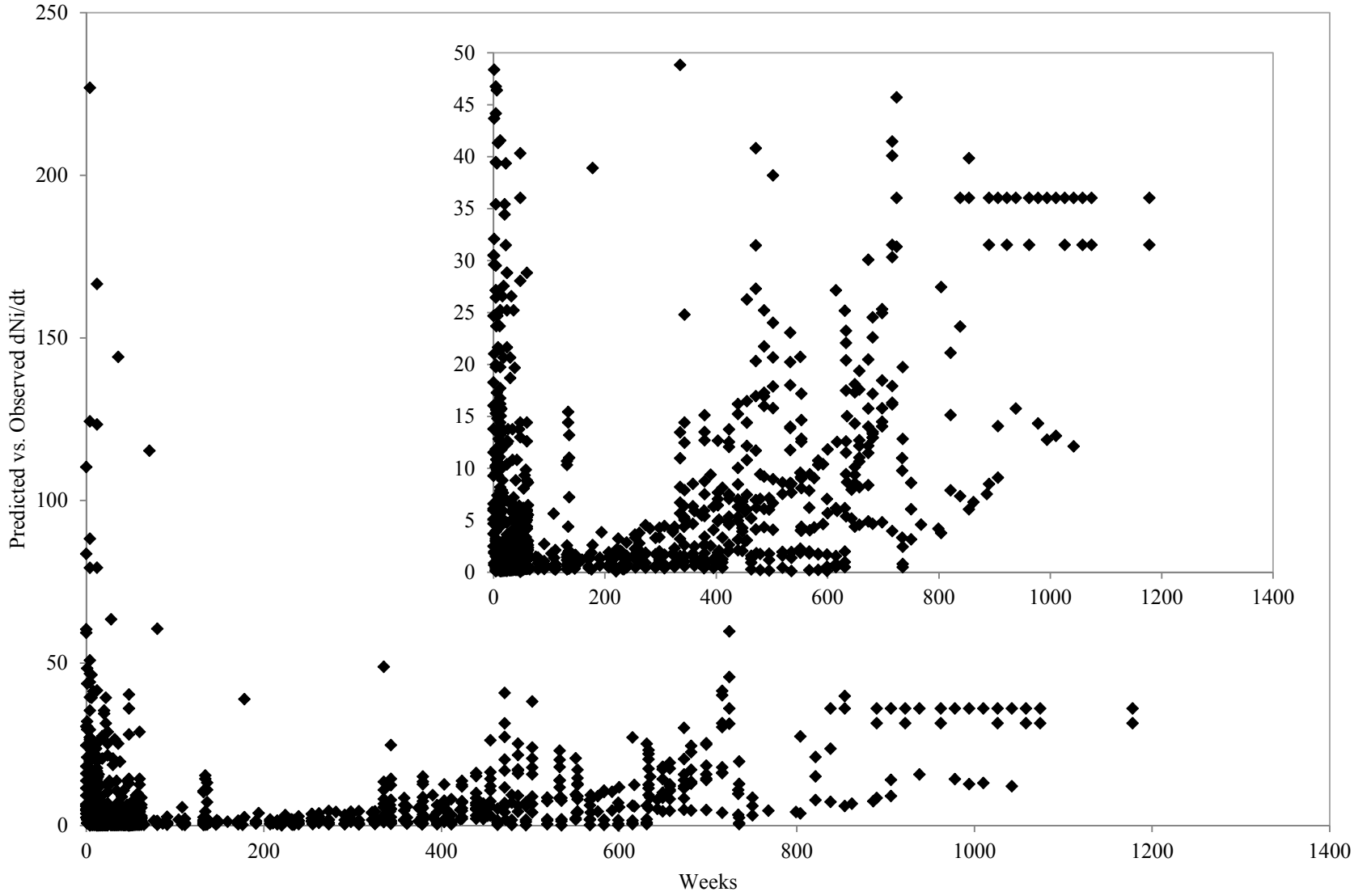


Figure 65. Cumulative probability distribution for ratio of predicted Ni release (based on leachate sulfate) to observed Ni release at days when Ni was sampled. Data compares ratios for each rate period and overall.

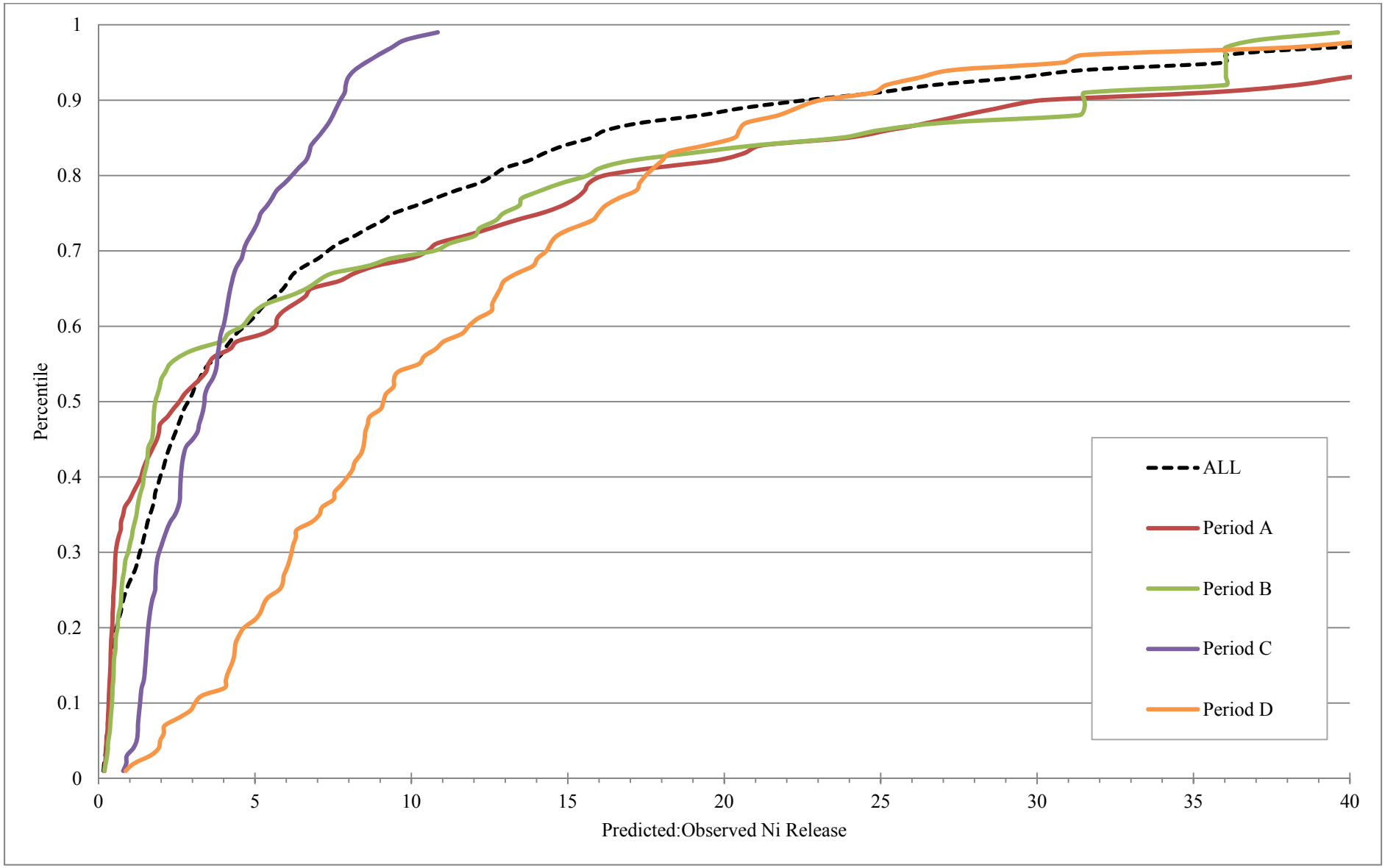


Figure 66. Model illustrating the sequence of oxidation products at the surface of pyrrhotite. Lower panels are at later times. Taken from Belzile et al. 2004.

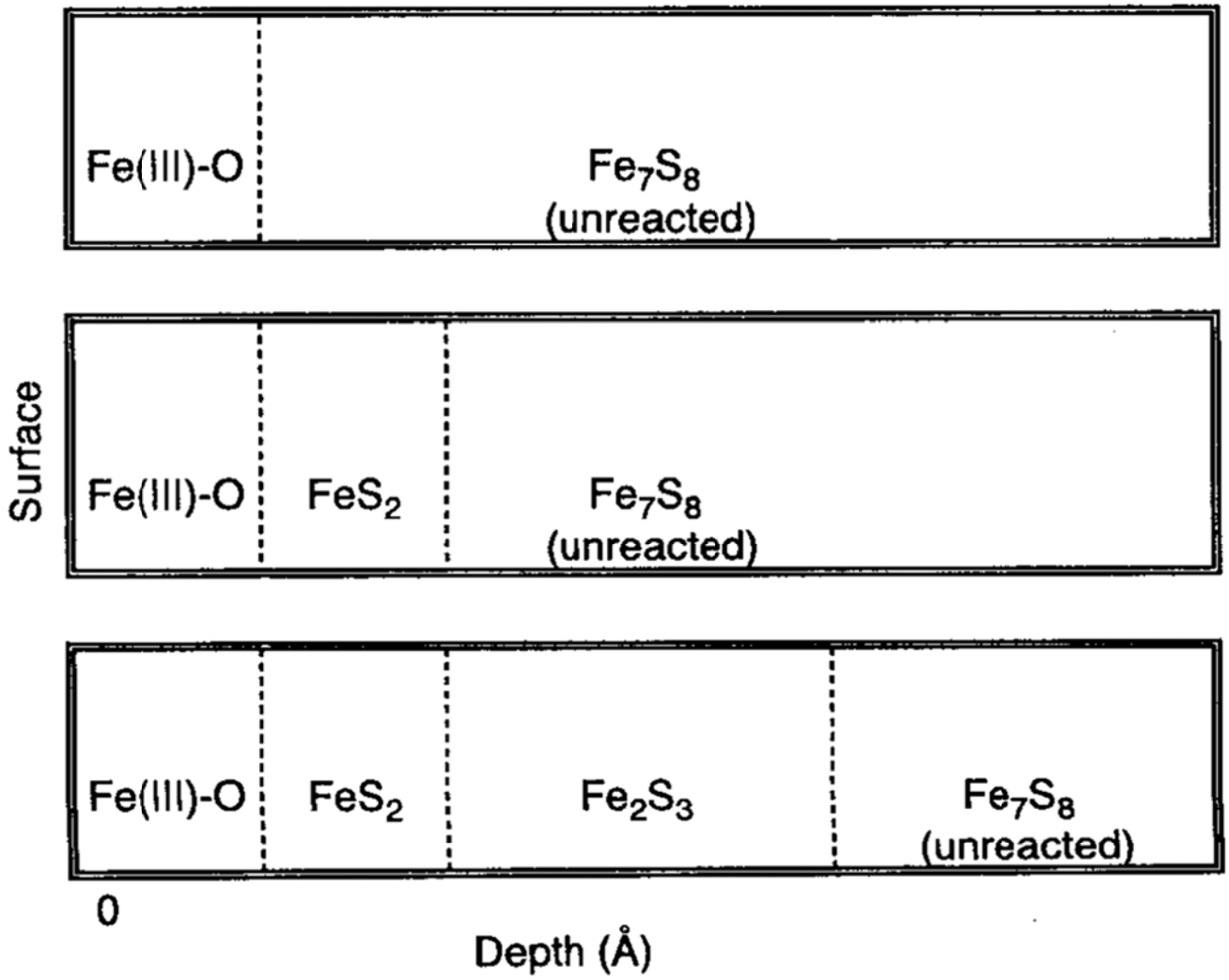
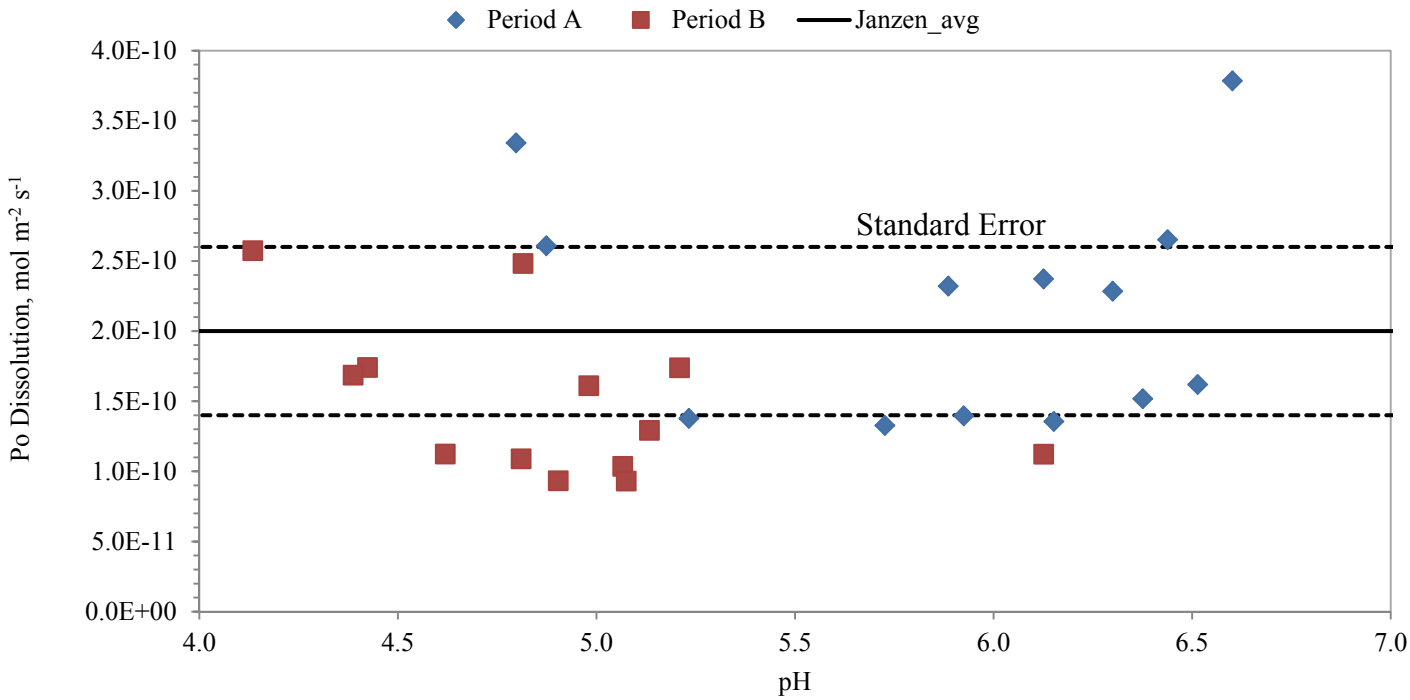


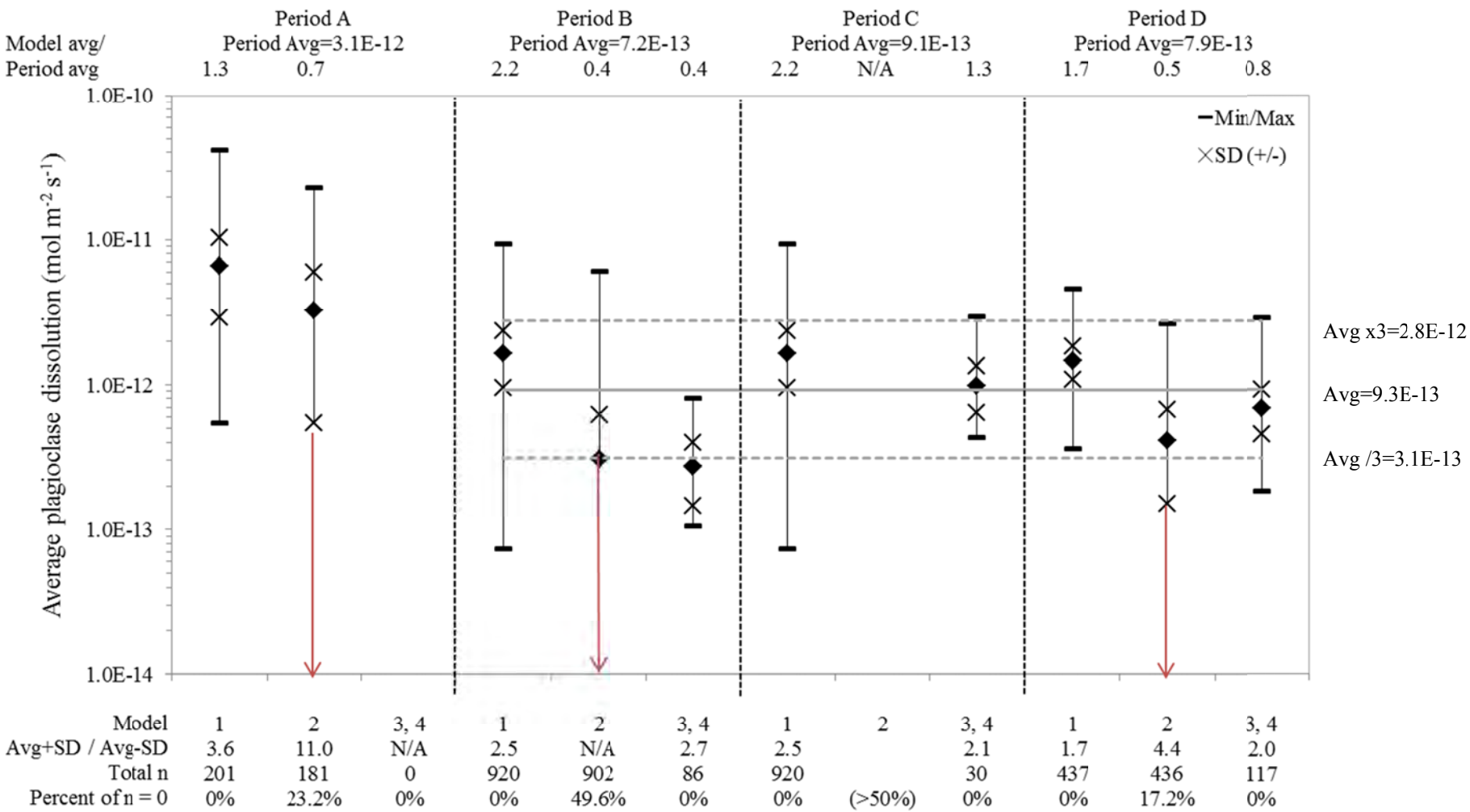
Figure 67. Comparison of modeled pyrrhotite oxidation rates with literature values (excludes R39-44). Calculated pyrrhotite oxidation rates were typically within 1 standard error of the average presented by Janzen et al. (2000). The solid horizontal line represents the mean of 12 reported values and the dashed lines represent one S.E. from the mean. Overall the reported range was from 8.2 x 10^{-12} to 5.9 x 10^{-10} mol m^{-2} s^{-1}.



R#	%S	Period A			Period B		
		weeks	Avg pH	Ave po	weeks	Avg pH	Ave po
1[1]	0.18	1-137	6.90	N/A	138-1252	6.42	N/A
3[2]	0.22	1-134	6.91	1.3E-08	135-1252	6.51	3.1E-09
5	0.4	1-140	6.60	3.8E-10	141-769	6.13	1.1E-10
7	0.41	1-129	6.51	1.6E-10	130-351	5.13	1.3E-10
9	0.51	1-140	6.30	2.3E-10	141-277	5.07	1.0E-10
11	0.54	1-137	6.13	2.4E-10	138-278	4.98	1.6E-10
13	0.57	1-103	6.44	2.7E-10	104-294	5.21	1.7E-10
15	0.58	1-101	5.89	2.3E-10	102-217	4.82	2.5E-10
17	0.71	1-100	6.15	1.4E-10	101-274	5.08	9.3E-11
35	1.12	1-127	4.87	2.6E-10	128-214	4.13	2.6E-10
29	1.16	1-38	6.38	1.5E-10	39-189	4.62	1.1E-10
37	1.40	1-74	5.93	1.4E-10	75-209	4.81	1.1E-10
33	1.44	1-76	5.23	1.4E-10	77-212	4.42	1.7E-10
19	1.63	1-66	5.73	1.3E-10	67-155	4.90	9.3E-11
31	1.64	1-64	4.80	3.3E-10	65-228	4.39	1.7E-10

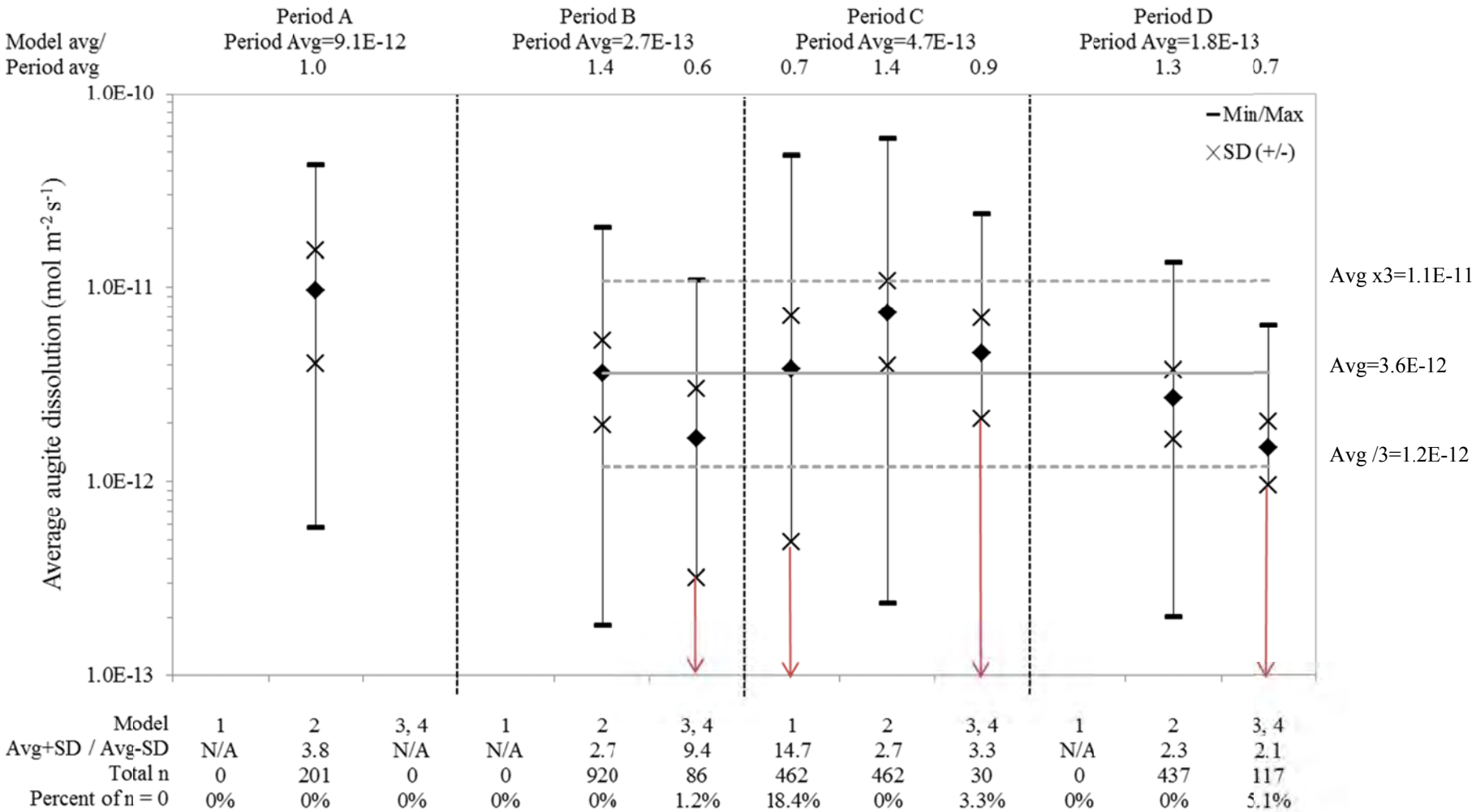
[1] Reactor 1 calculated to have pyrrhotite content less than zero
[2] Data from reactor 3 (0.22%S) omitted from figure due to significantly lower calculated po content

Figure 68. Comparison of plagioclase dissolution rates among models and rate periods (excludes R39-44). See Table 105 for values.



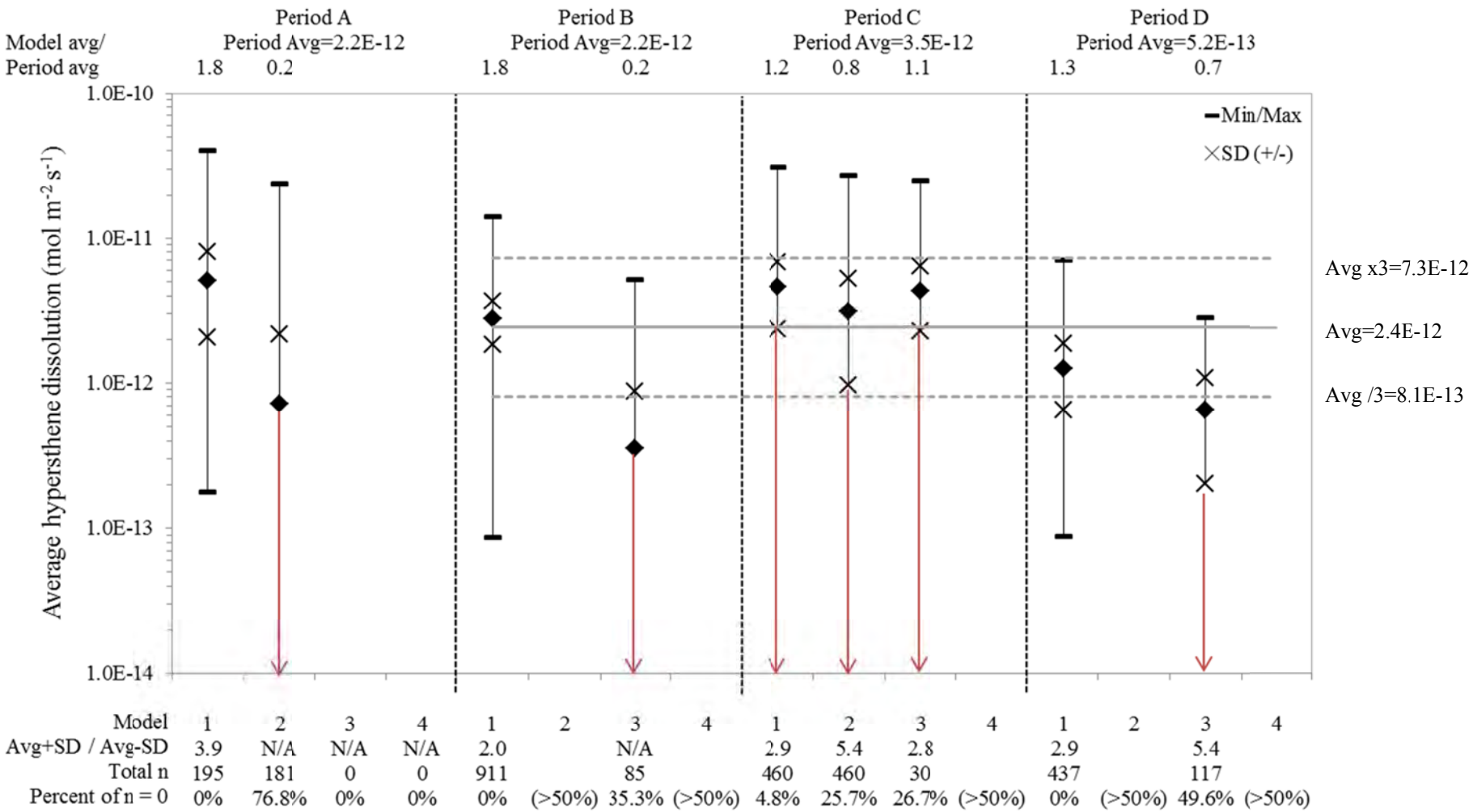
Note: Solid line = average plagioclase dissolution for periods B, C, and D (period A omitted due to initial increase in major cation release). Dashed lines = factor of 3 from average value. Average values from 7 out of 8 model runs during periods B, C, and D were within a factor of 3.
Note: Plagioclase dissolution in Model 1, Period C is calculated in the same manner as Period B. Plagioclase dissolution in Model 4 is the same as in Model 3, and therefore not included in average calculation.
Note: Zero values were calculated in model 2 (17-50% of total values), indicating a calculated absence of plagioclase dissolution (identified by a red arrow).

Figure 69. Comparison of augite dissolution rates among models and rate periods (excludes R39-44). See Table 105 for values.



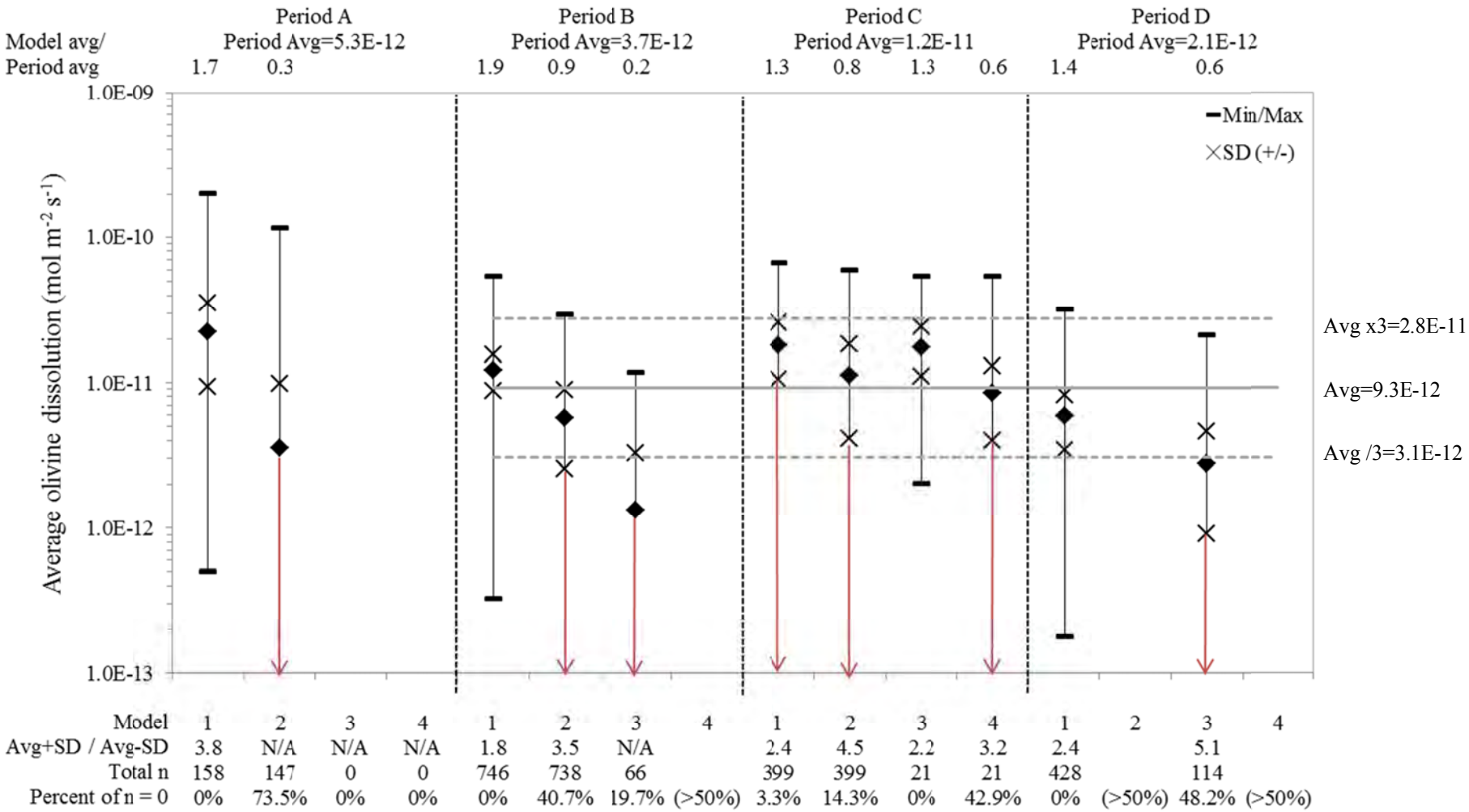
Note: Solid line = average augite dissolution for periods B, C, and D (period A omitted due to initial increase in major cation release). Dashed lines = factor of 3 from average value. Average values from all model runs during periods B, C, and D were within a factor of 3.
Note: Augite dissolution in Model 4 is the same as in Model 3, and therefore not included in average calculation.
Note: Range in average dissolution rates with no lower tail occurred when the average – standard deviation < 0.
Note: Zero values were calculated in model 1 (18% of total values) and models 3, 4 (1-5% of total values), indicating a calculated absence of augite dissolution (identified by a red arrow).

Figure 70. Comparison of hypersthene dissolution rates among models and rate periods (excludes R39-44). See Table 105 for values.



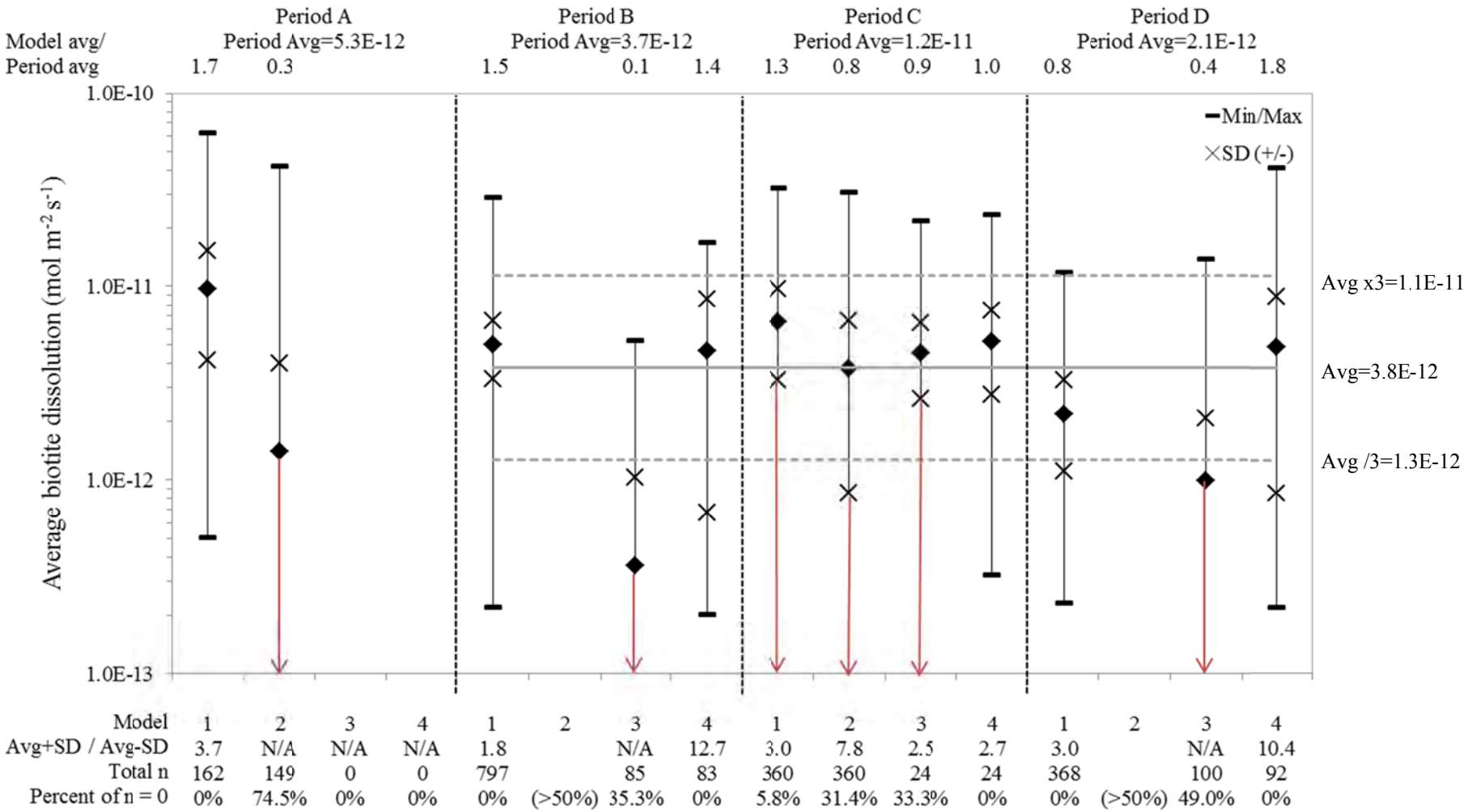
Note: Solid line = average hypersthene dissolution for periods B, C, and D (period A omitted due to initial increase in major cation release). Dashed lines = factor of 3 from average value. Average values from 5 out of 7 model runs during periods B, C, and D were within a factor of 3.
Note: Range in average dissolution rates with no lower tail occurred when the average – standard deviation < 0.
Note: Zero values were calculated in model 2 (26-77% of total values) and model 3 (27-50% of total values) indicating a calculated absence of hypersthene dissolution (identified by a red arrow). Zero values were calculated in model 1, only during period C (5% of the total values).

Figure 71. Comparison of olivine dissolution rates among models and rate periods (excludes R39-44). See Table 105 for values.



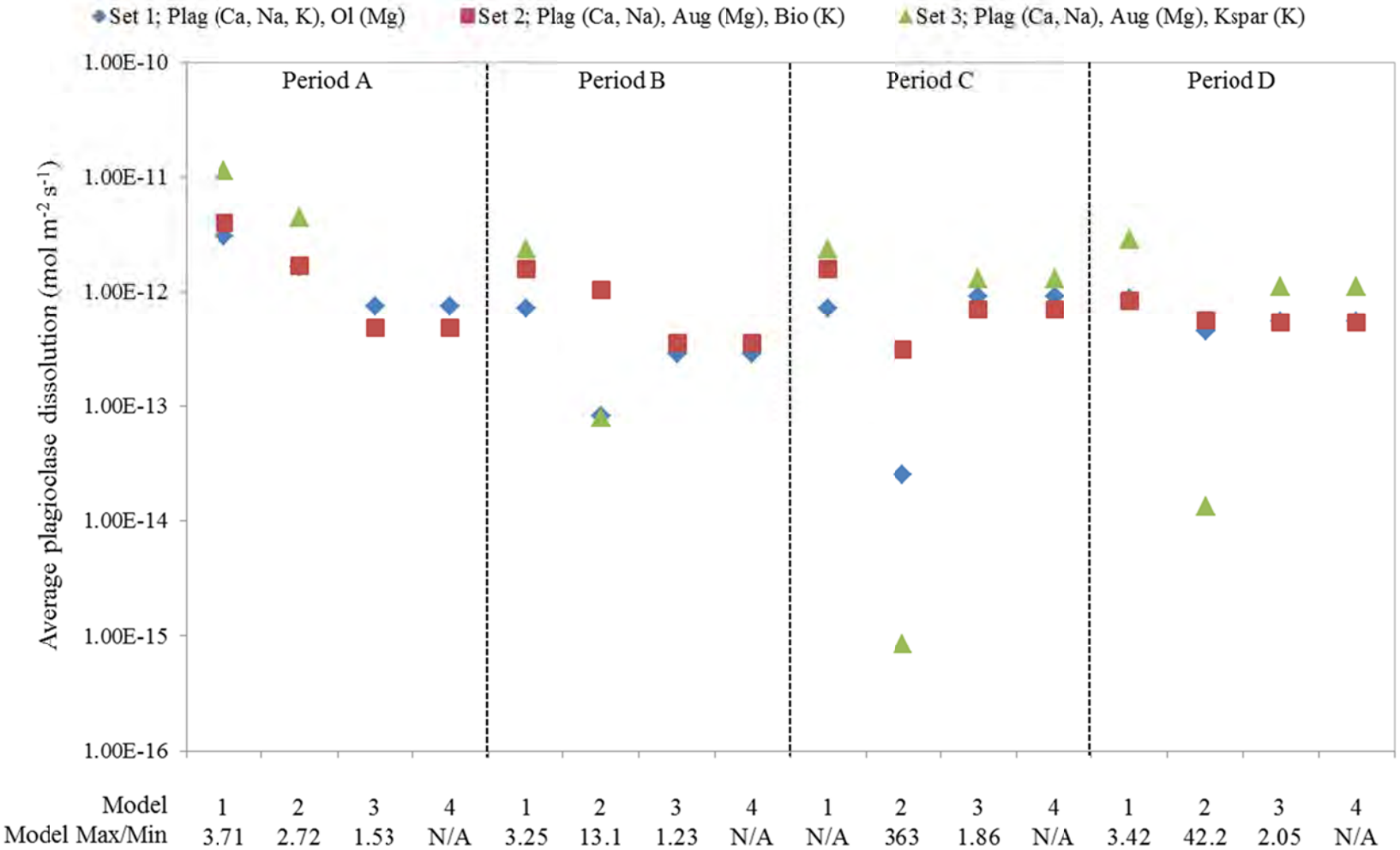
Note: Solid line = average olivine dissolution for periods B, C, and D (period A omitted due to initial increase in major cation release). Dashed lines = factor of 3 from average value. Average values from 7 out of 9 model runs during periods B, C, and D were within a factor of 3.
Note: Range in average dissolution rates with no lower tail occurred when the average – standard deviation < 0.
Note: Zero values were calculated in model 2 (14-74% of total values), model 3 (20-48% of total values), and model 4 (43% of total values) indicating a calculated absence of olivine dissolution (identified by a red arrow). Zero values were calculated in model 1, only during period C (3% of the total values).

Figure 72. Comparison of biotite dissolution rates among models and rate periods (excludes R39-44). See Table 105 for values.



Note: Solid line = average biotite dissolution for periods B, C, and D (period A omitted due to initial increase in major cation release). Dashed lines = factor of 3 from average value. Average values from 8 out of 10 model runs during periods B, C, and D were within a factor of 3.
Note: Range in average dissolution rates with no lower tail occurred when the average – standard deviation < 0.
Note: Zero values were calculated in model 2 (31-75% of total values) and model 3 (33-49% of total values) indicating a calculated absence of biotite dissolution (identified by a red arrow). Zero values were calculated in model 1, only during period C (6% of the total values).

Figure 73. Comparison of plagioclase dissolution rates from samples with differing mineralogy.

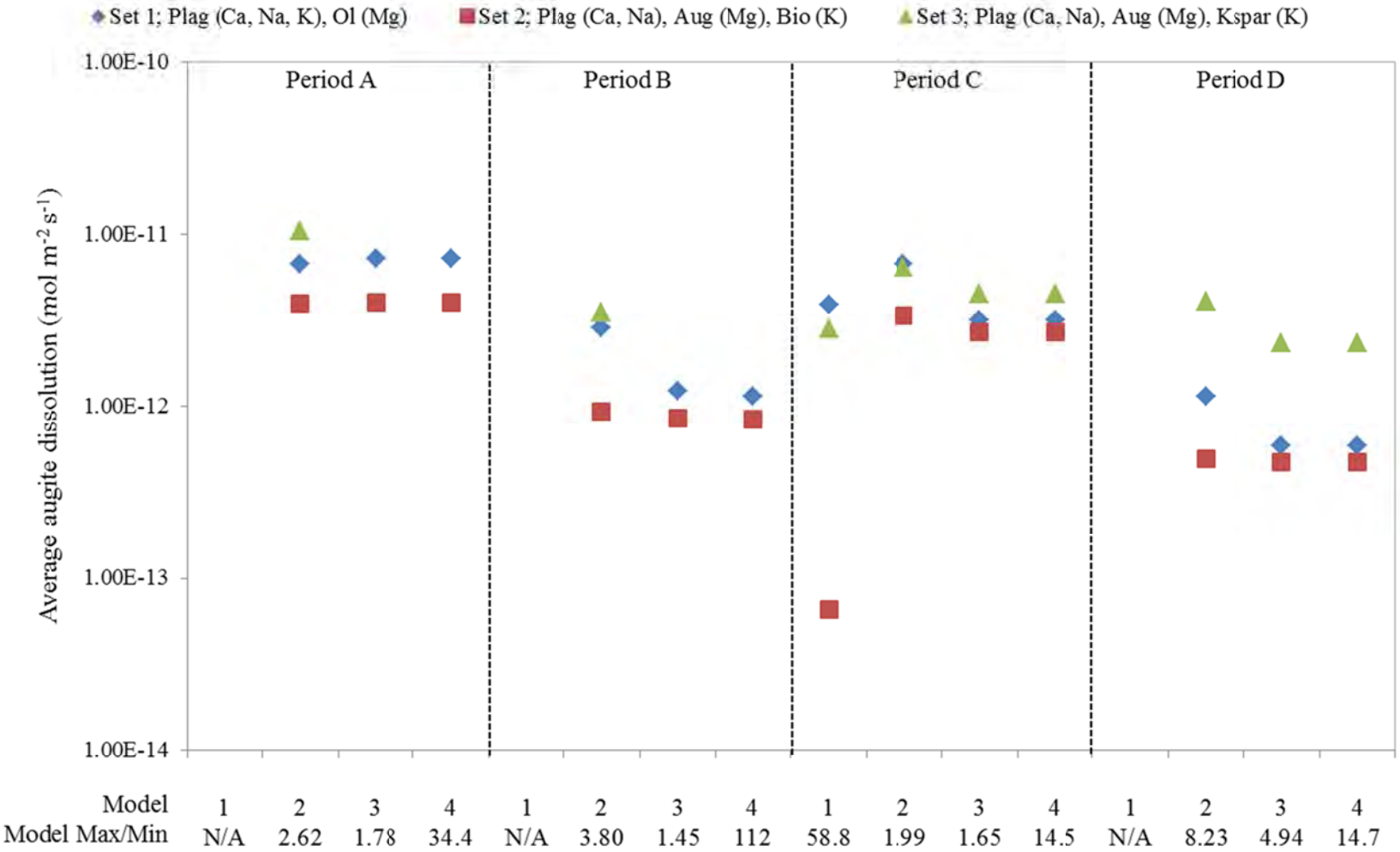


Note: Plagioclase dissolution in Model 1, Period C is the same as Period B. Plagioclase dissolution in Model 4 is the same as in Model 3.

Mineralogical breakdown of cations present (%) for specific samples (see Table U for full mineralogical breakdown)

Set	%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
			Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
1	0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
	1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
2	0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
	0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
3	1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
	1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

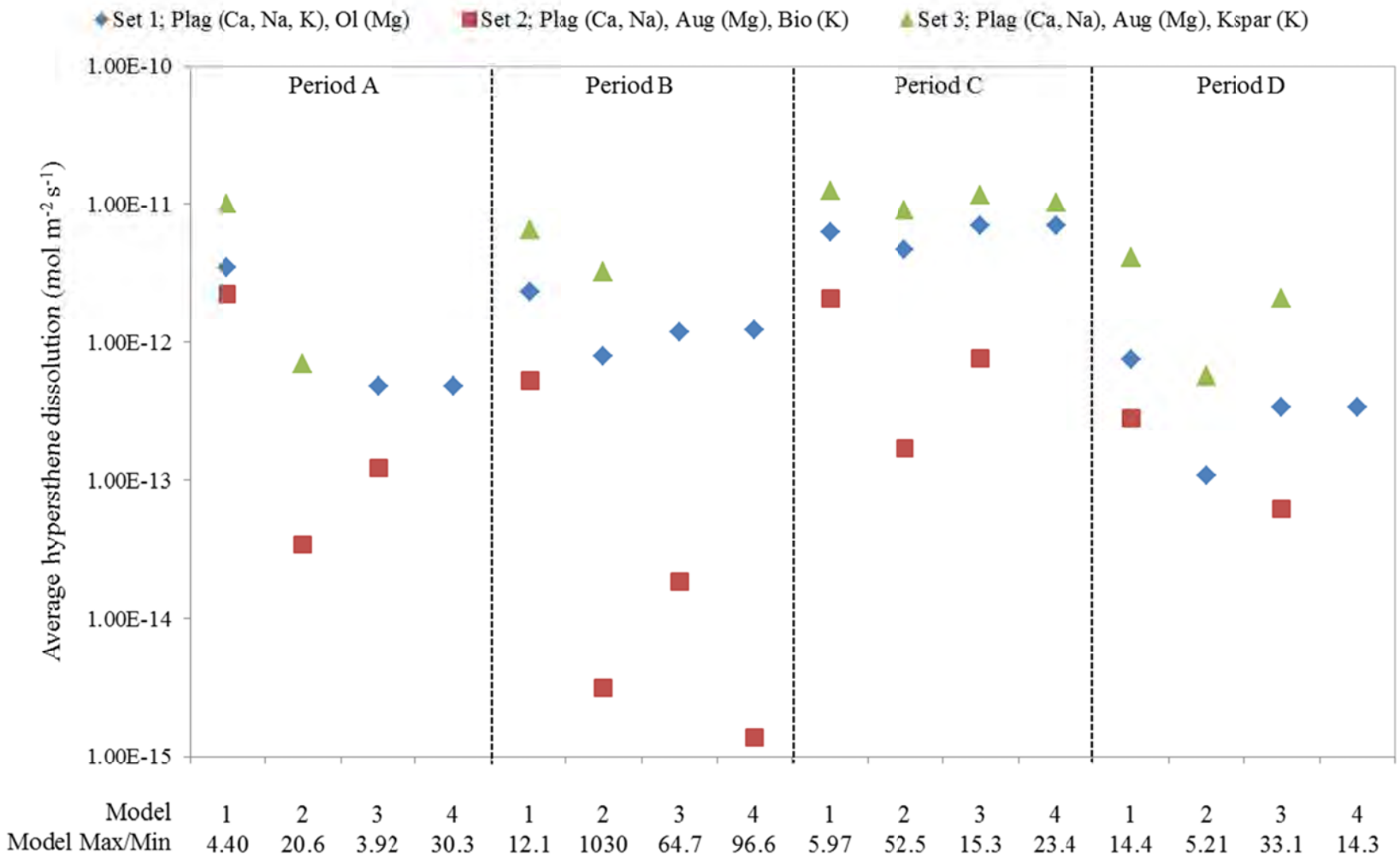
Figure 74. Comparison of augite dissolution rates from samples with differing mineralogy.



Mineralogical breakdown of cations present (%) for specific samples (see Table U for full mineralogical breakdown)

Set	%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
			Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
1	0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
	1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
2	0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
	0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
3	1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
	1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

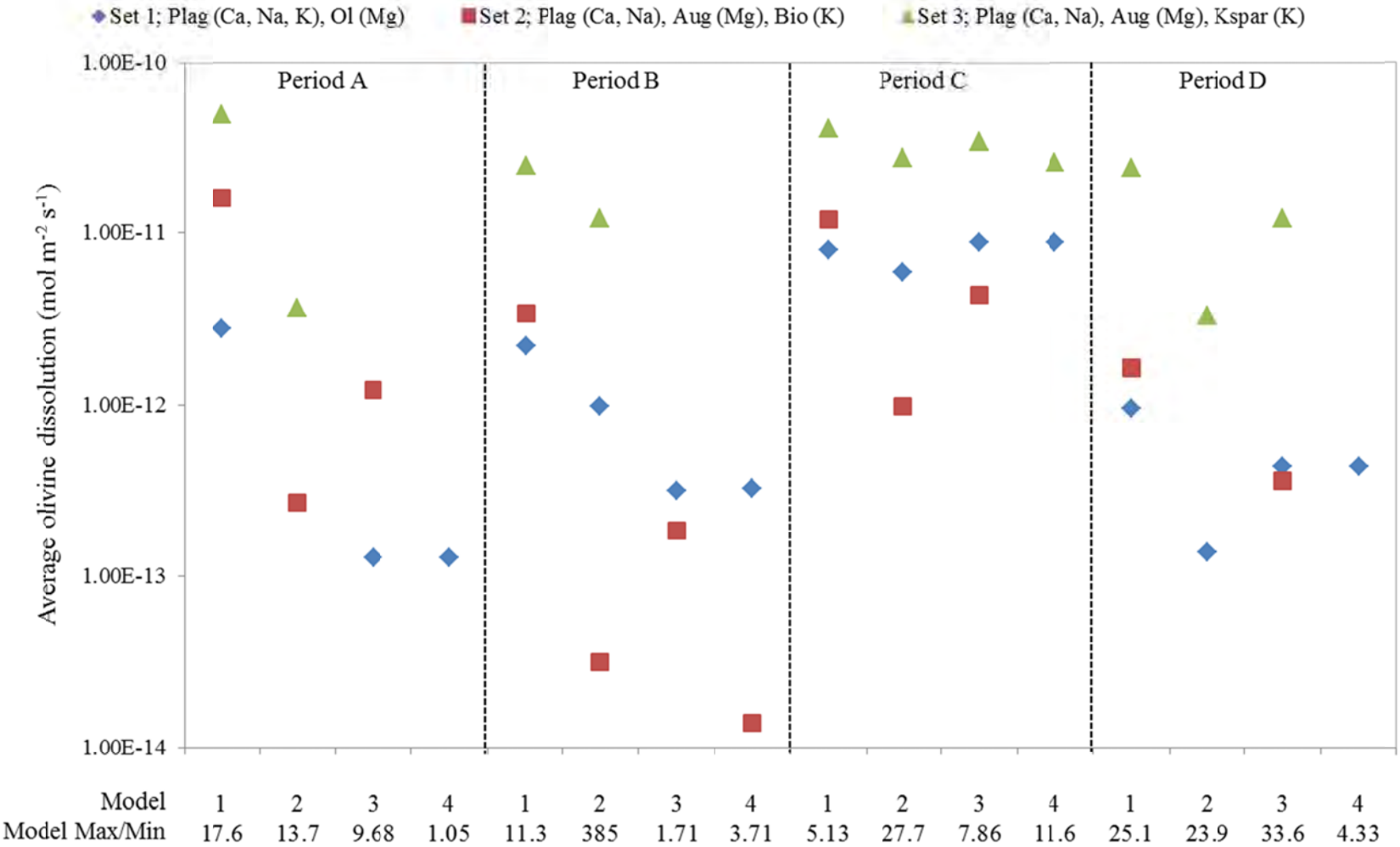
Figure 75. Comparison of hypersthene dissolution rates from samples with differing mineralogy.



Mineralogical breakdown of cations present (%) for specific samples (see Table U for full mineralogical breakdown)

Set	%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
			Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
1	0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
	1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
2	0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
	0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
3	1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
	1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

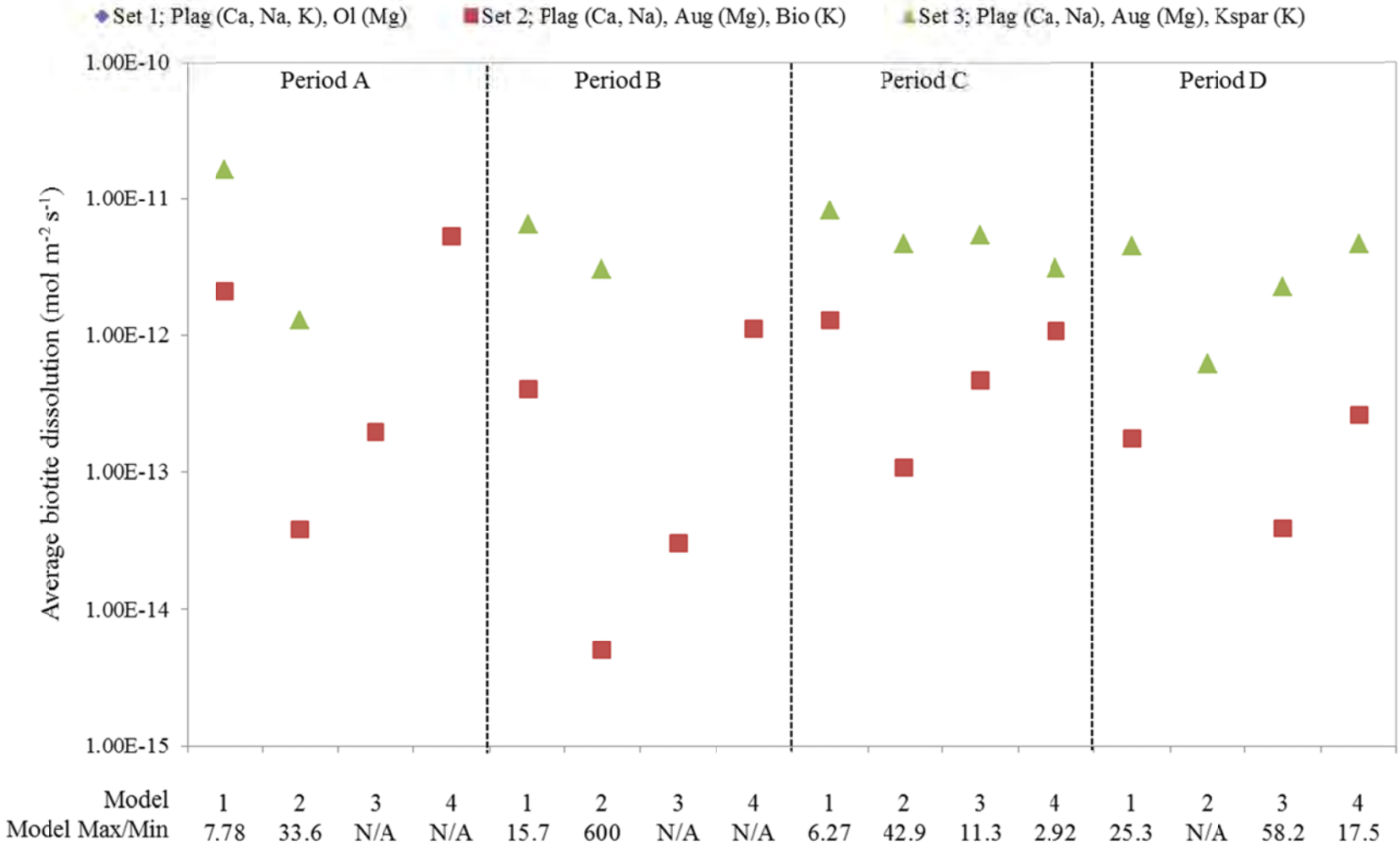
Figure 76. Comparison of olivine dissolution rates from samples with differing mineralogy.



Mineralogical breakdown of cations present (%) for specific samples (see Table U for full mineralogical breakdown)

Set	%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
			Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
1	0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
	1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
2	0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
	0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
3	1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
	1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

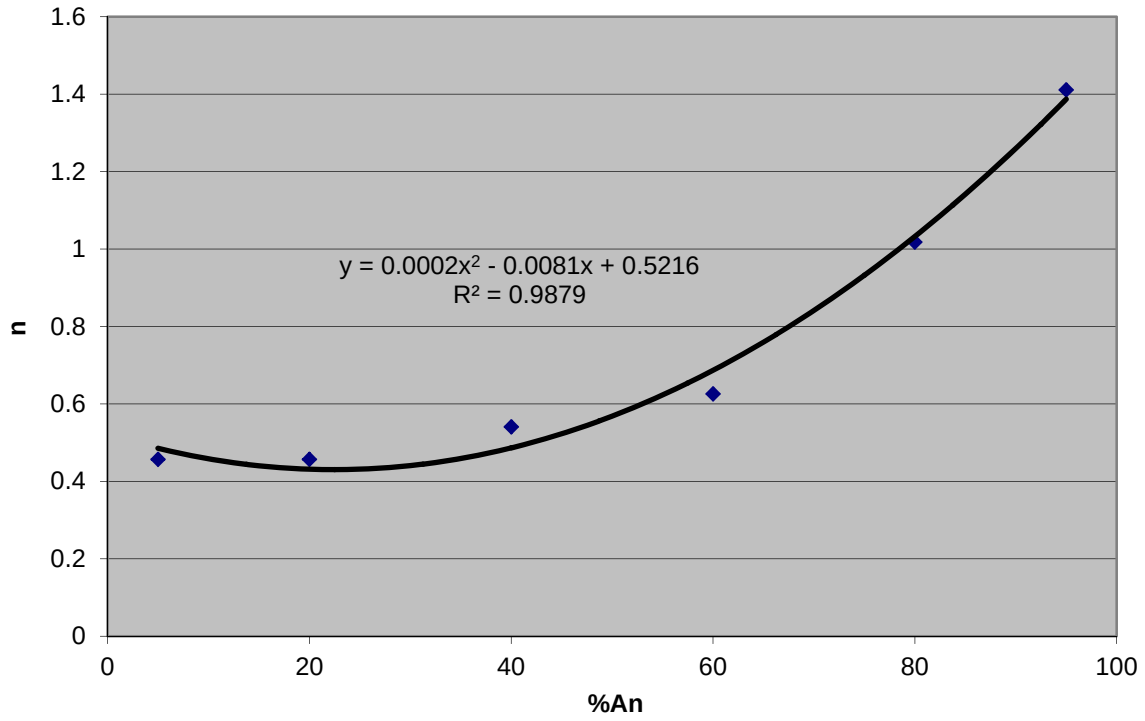
Figure 77. Comparison of biotite dissolution rates from samples with differing mineralogy.



Mineralogical breakdown of cations present (%) for specific samples (see Table U for full mineralogical breakdown)

Set	%S	Reactor	Calcium				Magnesium					Sodium		Potassium			
			Plag	Augite	Prehnite	Amphibole	Augite	Hyper	Olivine	Biotite	Amphibole	Plag	K-Spar	Plag	Biotite	K-Spar	Amphibole
1	0.72	42	94.09	5.91			4.90	10.17	84.93			100.00		100.00			
	1.16	29	76.36	23.64			17.72	31.30	50.98			100.00		100.00			
2	0.67	40	68.15	31.85			25.90	44.55	15.03	14.52		99.44	0.56	15.02	68.73	16.25	
	0.82	43	66.52	33.48			29.18	52.09	11.23	7.50		100.00		33.24	66.76		
3	1.40	37	60.97	29.45	8.15	1.44	34.57	33.96	29.37		2.10	97.59	2.41	30.83		67.72	1.45
	1.64	31	61.29	35.95	2.76		36.46	41.43	13.93	8.18		97.39	2.61	15.08	23.74	61.18	

Figure 78. Below are the equations to determine coefficients for the dissolution rates of plagioclase based on percent anorthite (%An) in the acidic range (pH<4.4). The equation was generated from data in Palandri and Kharaka (2004) and is in the form: log(rate) = log K – n*pH.



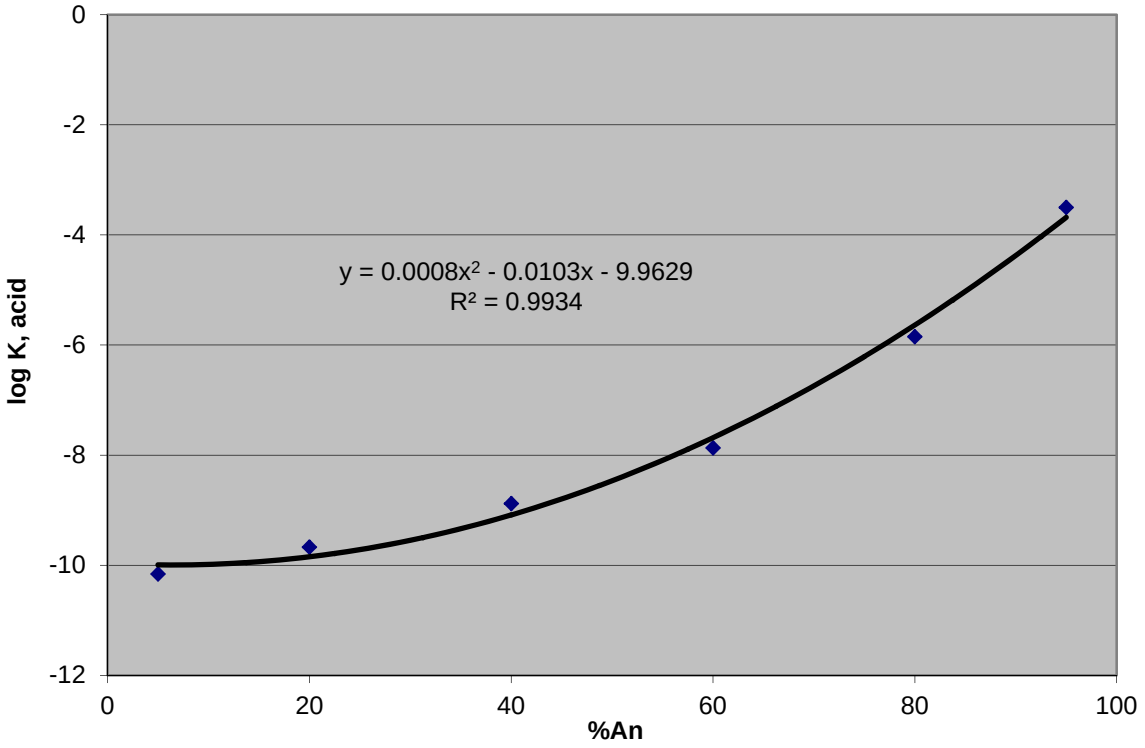


Figure 79. Below are the equations to determine coefficients for the dissolution rates of plagioclase based on percent anorthite (%An) in the neutral range (pH≥4.4). The equation was generated from data in Palandri and Kharaka (2004) and is in the form: log(rate) = log K.

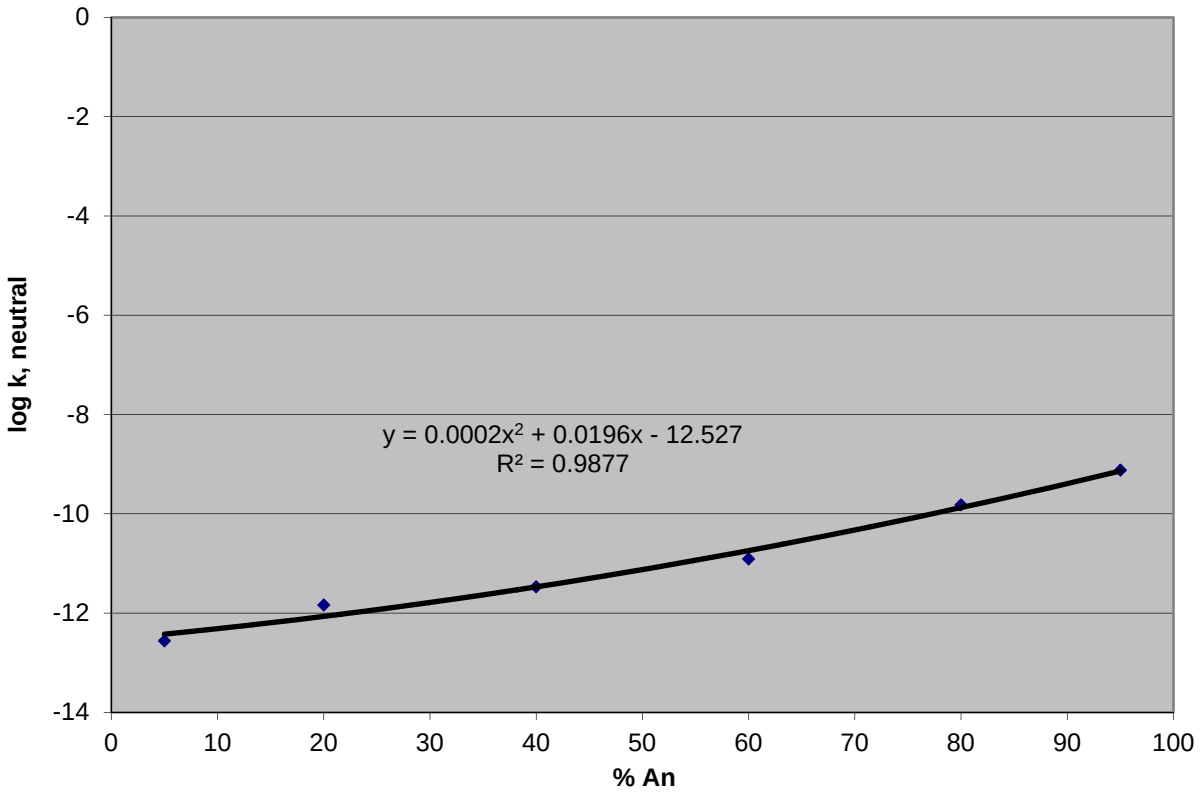
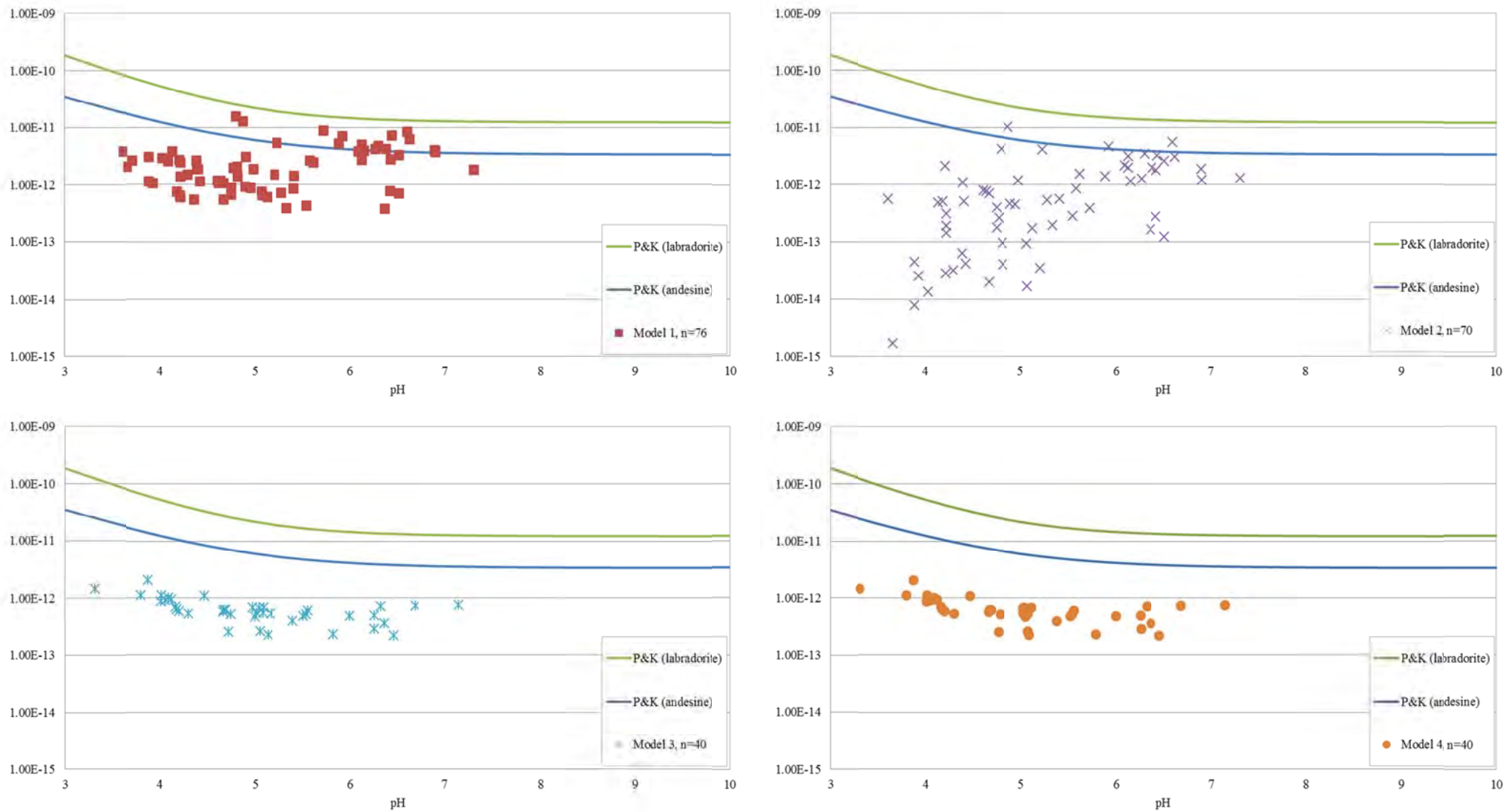


Figure 80. Modeled plagioclase dissolution rates (mol m^{-2} s^{-1}) vs drainage pH. Average plagioclase dissolution rates for each reactor are plotted for each rate period (Table 52-56) as a function of average pH for each of the four silicate dissolution models (Table 27). Palandri and Kharaka acidic+neutral rates for labradorite and andesine are plotted for comparative purposes (Table 73).



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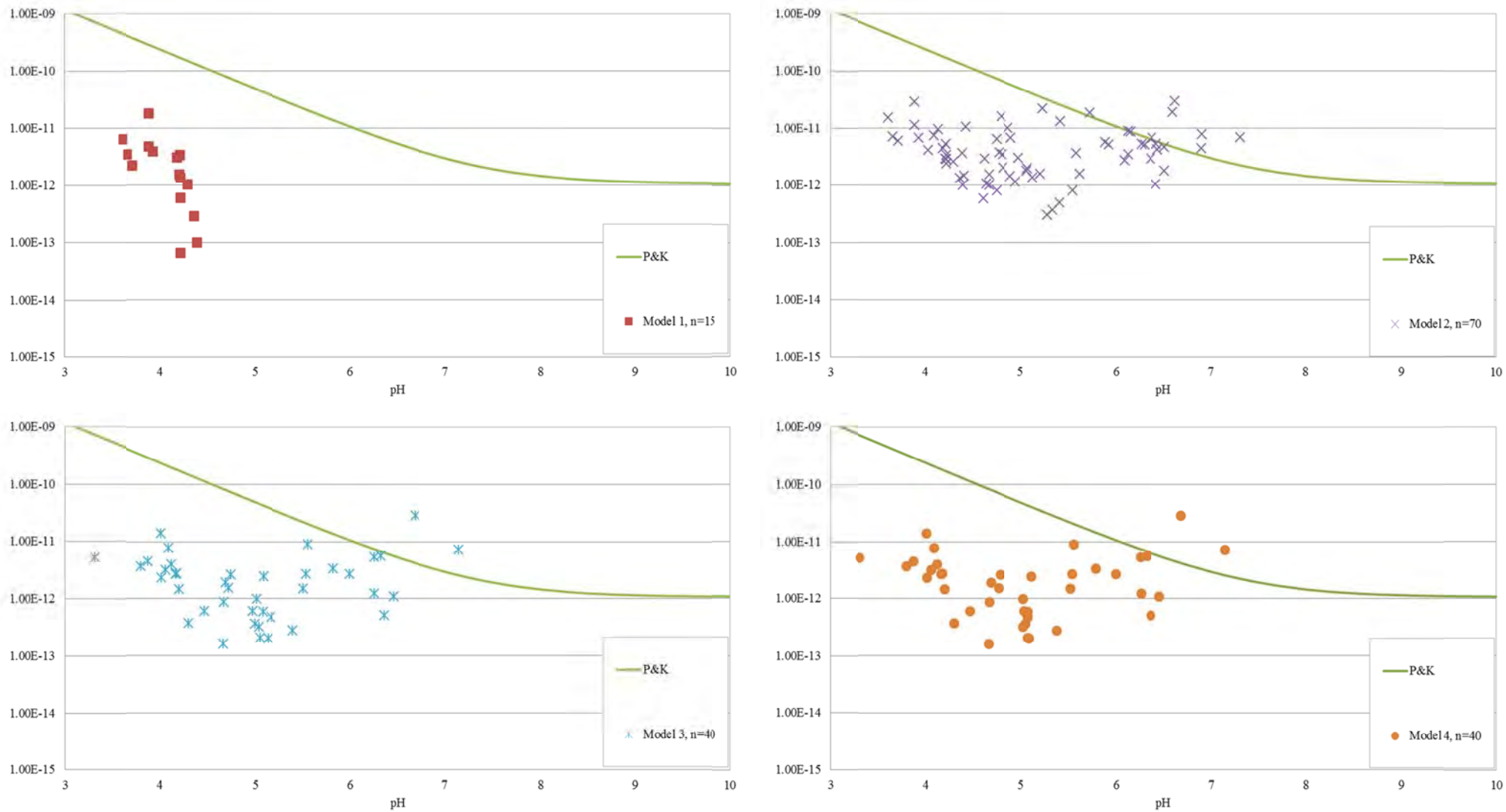

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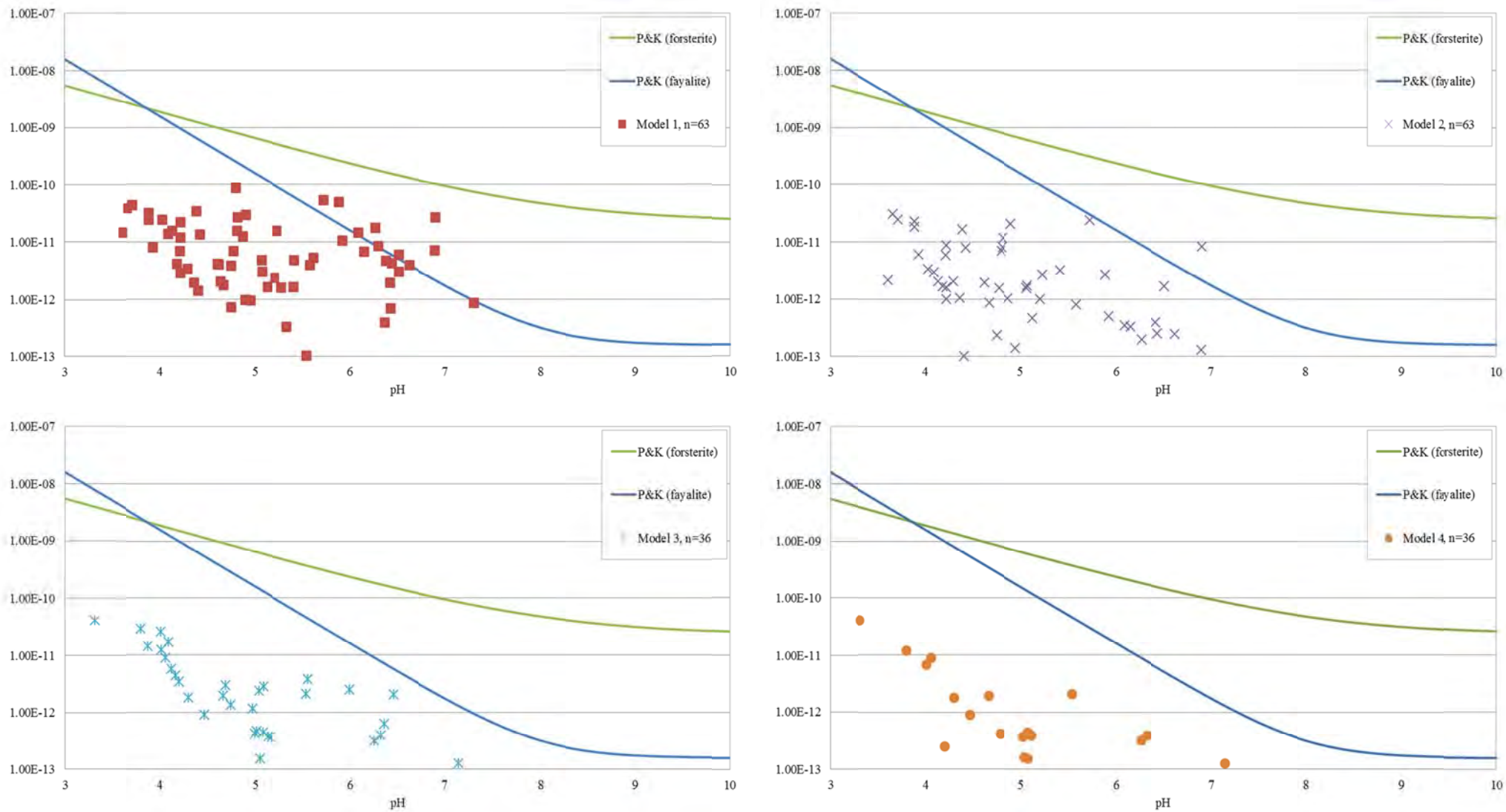

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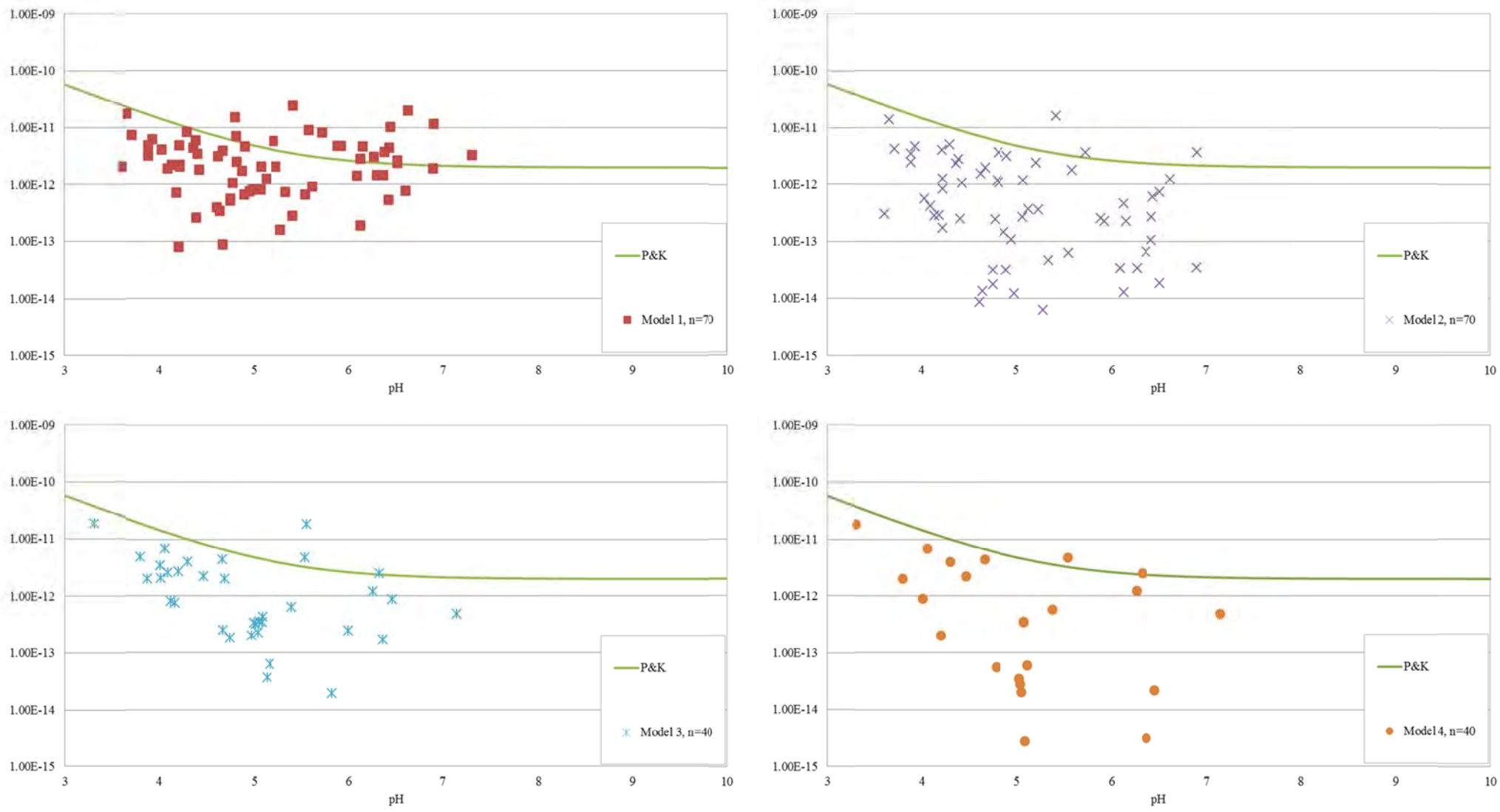

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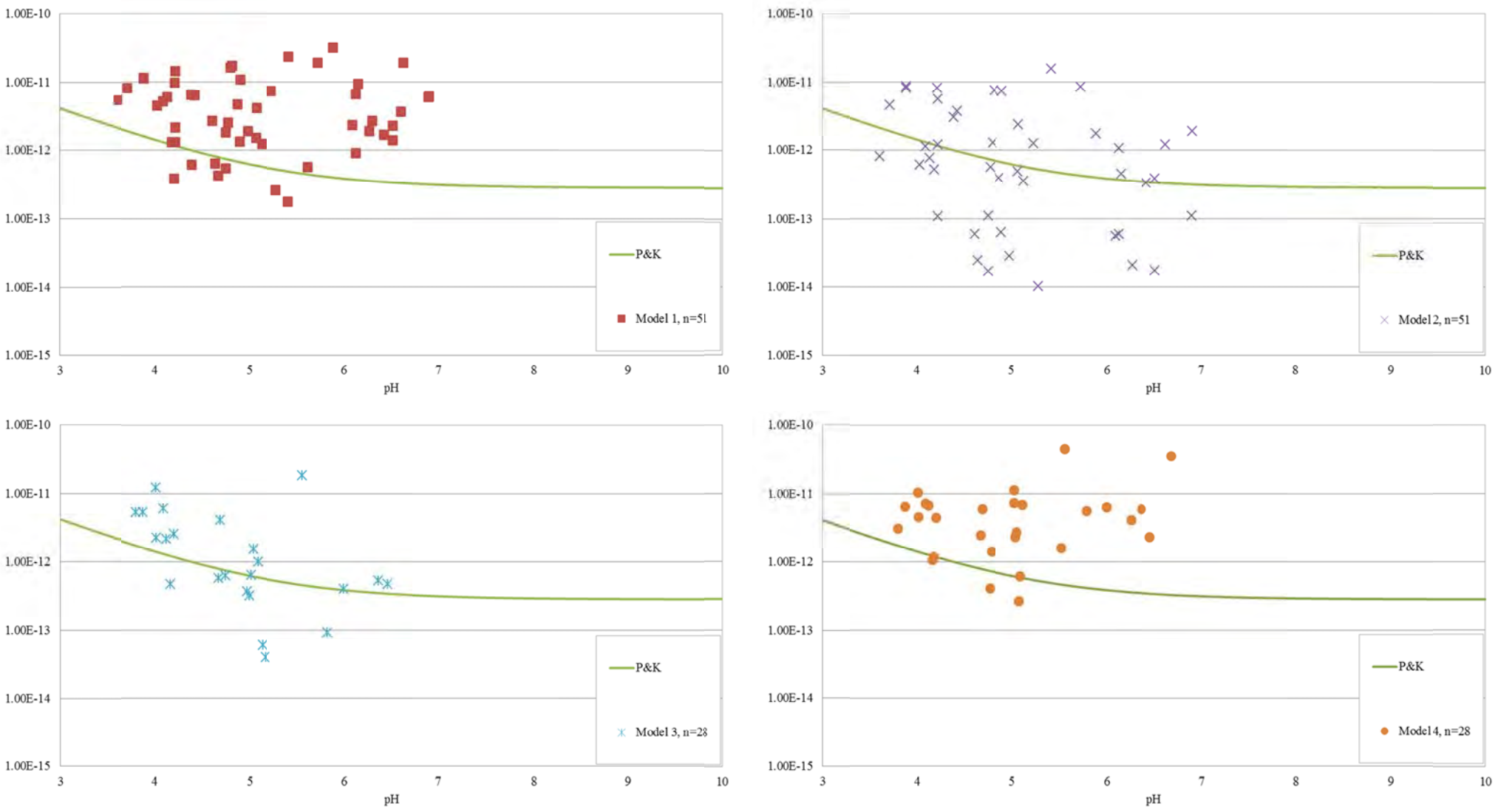

Attachment 15

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Dissolution of Duluth Complex Rock from the Babbitt and Dunka Road Prospects: Eight-year laboratory experiment

June 2013

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Dissolution of Duluth Complex Rock from the Babbitt and Dunka Road Prospects: Eight-year laboratory experiment

June 2013

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0. EXECUTIVE SUMMARY

The *research reported will be applied* in the environmental review and permitting of future nonferrous mines in Minnesota, particularly those in the Duluth Complex. These regulatory activities require prediction of solute release from waste rock during mine operation and in the decades and centuries following mine closure. As a foundation for such predictions, the reactions occurring in waste rock stockpiles and variables affecting these reactions must be understood. This knowledge and a description of the proposed method of waste rock disposal are necessary to 1) design appropriate tests for predicting the release of potential pollutants from waste rock, 2) extrapolate results of such tests to field conditions, and 3) if necessary, implement mitigation measures that will ensure that water resources are not adversely impacted. In the present project, laboratory studies were conducted to examine solute release and variables affecting this release from waste rock from the Partridge River intrusion of the Duluth Complex. The studies provide scientific and technical information necessary to address questions inherent to environmental review and permitting of nonferrous metal mines. The project also provides empirical data describing eight years of rock dissolution for comparison with and extrapolation of data submitted for environmental review and permitting by companies proposing mining in the Duluth Complex.

To *summarize* the laboratory work, dissolution tests were initiated in August 2003 on 10 well-characterized (particle size, chemical composition, mineral content, mineral chemistry) Duluth Complex rock samples from the Babbitt and Dunka Road prospects of the Partridge River intrusion. Dissolution tests were conducted on 10 coarse samples ($d < 6.35$ mm = 0.25 in.; $0.13 \leq \%S \leq 1.36$) and seven fine samples ($0.053 < d < 0.149$ mm; $0.07 \leq \%S \leq 0.94$). The experiments remain in progress and the following report presents data generated through week 400. This report summarizes results and provides a conceptual structure to frame these results.

In general, drainage pH tended to decrease and release rates of heavy metals (copper, nickel, cobalt, zinc) and major cations (calcium, magnesium, sodium, potassium) tended to increase with increasing sulfur content. These results were consistent with the previously reported 1) dependence of sulfide mineral oxidation (as reflected by sulfate release) and the attendant acid production on the amount of sulfide minerals present and 2) importance of sulfide mineral oxidation as a driving force for release of heavy metals and major cations. More specifically, numerous relationships between drainage quality and both solid-phase sulfur content and dissolution time were identified and quantified to quantitatively describe solute release and its dependence on solid-phase composition and dissolution time.

For the 10 coarse samples, drainage pH decreased with increasing sulfur content, with a minimum value of 4.1. For all but the two highest sulfur samples, pH decreased throughout the eight-year experiment. Rates of sulfate release, reflecting sulfide mineral oxidation, increased with increasing sulfur content. The strong correlation between average annual rates for the 10 samples and corresponding sulfur contents was quantified for each year using regression analysis ($0.91 < r^2 < 0.94$). This analysis was used to derive composite annual sulfate release rates (per gram sulfur), which were found to decrease slightly over the eight-year experiment.

Release rates for copper, nickel, cobalt, and zinc also correlated highly with sulfur content (typically $0.80 < r^2 < 0.95$). Magnesium was the dominant cation released, followed by calcium, and sodium. Their releases tended to parallel that of sulfate, reflecting silicate minerals dissolving to neutralize acid produced by iron sulfide mineral oxidation.

Similar analysis was conducted on data generated in the seven fine-grained samples. As with the larger particles, drainage pH decreased and release rates for sulfate, heavy metals, and major cations increased with increasing sulfur content. Although the relationship between minimum drainage pH and sulfur content was similar to that for the larger particles, pH from the finer samples declined more rapidly, reaching minimums between weeks 60 and 120. Sulfate release from the fine particles were initially three times those from the coarse particles, reflecting the greater sulfide mineral surface area per unit mass of the fine particles, and this rapid release contributed to the aforementioned early pH depression. Sulfate release rates from the fine particles decreased fairly quickly over time and by year seven equaled those from the coarse particles, for which rates decreased more slowly. The rapid decline of rates from the fine particles was attributed to sulfur depletion and probable coating of sulfide minerals.

Dissolution testing of Dunka Road samples was limited to one sample in the 0.053-0.149 mm size fraction and three samples in the $d < 6.35$ mm fraction. The most notable difference between Dunka Road and Babbitt prospect rock was elevated heavy metal release from the latter. Whereas some of the difference could be ascribed to higher metal content of the Babbitt prospect rock examined, differences between prospects in total sulfide mineral oxidation rates and drainage pH also played a role. Because the number of samples for comparison was limited, these results should be viewed as preliminary.

The Partridge River rock samples yielded higher sulfate rates than Partridge River tailings of similar sulfur content, due to greater exposure of sulfide minerals in the rock samples. Heavy metal release from the rock was also higher than that from the tailings. This was due to the aforementioned elevated sulfide mineral exposure, higher heavy metal contents, and lower drainage pH associated with the rock samples. Drainage pH from the Partridge River rock tended to be about 0.7 units higher than that from previously tested South Kawishiwi rock of similar sulfur content. This was attributed to the higher heavy metal content (and consequent lower iron sulfide content) of the Partridge River rock examined in the present study.

In *conclusion,* for both ten coarse and seven fine samples, testing generated data describing drainage pH and release of sulfate, heavy metals, and major cations over a period of eight years. Drainage pH and solute release were correlated with solid-phase sulfur content and time of dissolution. Furthermore, the two data sets allowed comparison of solute release from the two size fractions, consequently providing insight on the influence of particle size. In total, the project 1) quantifies the dependence of drainage pH and release rates of sulfate, heavy metals, and major cations on solid-phase sulfur content, dissolution time, and particle size; 2) interprets results with respect to mineral dissolution reaction; and 3) provides context for the results generated by comparisons with dissolution data on Partridge River tailings and South Kawishiwi intrusion rocks. Thus, the project will help inform design of predictive test programs for

environmental review and permitting and interpretation of results generated by such program. It also provides a robust empirical data base to provide a benchmark for data submitted.

Additional work should be conducted to further the benefits generated to date. This includes continued dissolution testing to confidently define long-term trends, analysis of leached solids to increase understanding of chemical reactions controlling solute release (e.g. chemical precipitation and adsorption), and more detailed analysis of the present data, including calculation of mineral dissolution rates. Calculation of mineral dissolution rates will facilitate application of the data generated to a wider range of mineralogical compositions of Duluth Complex rock.

1. INTRODUCTION

1.1. Background

Several areas of the Duluth Complex are presently being considered for mineral resource development. Prior to development, potential mine wastes must be characterized (particle size, chemistry, mineralogy, petrology) and well-characterized samples must be subjected to dissolution studies. These studies assess the potential of mine wastes (e.g. waste rock, mine walls, tailings, and processing residues) to adversely impact water resources. The Minnesota Department of Natural Resources is conducting dissolution studies on well-characterized samples of South Kawishiwi intrusion (SKI) Duluth Complex rock. The period of record for some of these samples exceeds 24 years. Although there is some homogeneity in the bulk mineral content of rock throughout the Duluth Complex, the degree to which compositional variations within the complex might affect the water quality resulting from rock dissolution has not been determined.

The present project examines laboratory dissolution of rock from a bulk sample in the Babbitt prospect and from drill core samples from the Dunka Road prospect, both in the Partridge River intrusion (PRI) of the Duluth Complex. These samples might simulate waste rock or mine walls resulting from mineral resource development. Two different methods of dissolution testing were used, one of which allows comparison of results from the Partridge River intrusion samples with those from long-term dissolution studies conducted on rock from the Dunka Mine in the South Kawishiwi intrusion of the Duluth Complex. Thus, the present project addresses a comparison of test methods and the dissolution of rock from different locations in the Duluth Complex. The present report summarizes data generated over the first 400 weeks of dissolution testing.

1.2. Duluth Complex Geology Summary

Miller et al. (2002) reported on the mineral occurrences and geology of the Duluth Complex, and the following summarizes information on the Partridge River and South Kawishiwi intrusions from that report. Dissolution of rocks from these two intrusions is the focus of the present report.

The Partridge River intrusion hosts at least four copper-nickel prospects (Babbitt, Dunka Road, Wetlegs, Wyman Creek) and at least seven potential iron-titanium prospects (Miller et. al., 2002) (Section 17, Longear, Longnose, Wyman Creek, Section 22, Skibo, Water Hen) (Figure 1). The Section 22, Skibo, and Water Hen prospects all lie south, and within 12 miles, of the Wyman Creek prospect and are not depicted in Figure 1.

The Partridge River intrusion consists mainly of troctolitic rocks that form an arc-shaped exposure (Figure 1). It is estimated to be 2.5 km thick, and is underlain by the Virginia Formation slates and greywackes and the Biwabik Iron Formation. The basal 900 meters of the intrusion has been subdivided into seven stratigraphic units (Severson and Hauck 1990, 1997; Geerts 1991; Severson 1991, 1994) that can be correlated over a strike length of 24 km. These units dip 10 to 20 degrees to the southeast and have been described in detail due to the presence

of copper-nickel (Cu-Ni) mineralization. Rocks overlying the Partridge River intrusion include other Duluth Complex rocks and metamorphosed basalts (probably North Shore Volcanic Group). These rocks are also present in the interior of the intrusion as large inclusions.

The contact zone between the Partridge River and South Kawishiwi intrusions is poorly exposed and sparsely drilled. The seven basal units become unrecognizable in the contact zone. This contact is complicated by the presence of a fault (Grano fault of Severson 1994) that was repeatedly activated during and after the emplacement of the two intrusions.

The South Kawishiwi intrusion hosts at least five copper-nickel prospects (Spruce Road, South Filson Creek, Maturi, Dunka Pit, Serpentine) and a potential platinum group element copper-nickel prospect (Birch Lake).

The South Kawishiwi intrusion is composed of troctolitic rocks exposed in an 8-by-32 kilometer arc-shaped band (Figure 1). The Giant's Range batholith is the dominant footwall rock, although Virginia Formation and Biwabik Iron Formation underlie the intrusion in some areas. Five major map units in the South Kawishiwi intrusion have been identified, although some geologists have further subdivided these units (e.g. Severson 1994). The units are not continuous throughout the intrusion. Major sulfide mineralization occurs at the base of the intrusion. Virginia Formation and Biwabik Iron Formation inclusions occur throughout the intrusion. Inclusions of basaltic hornfels (probably from the North Shore Volcanic Group) and quartz sandstone hornfels (probably from the Puckwunge and Nopeming sandstones) occur near some of the copper-nickel prospects. The lower ultramafic unit contains the majority of the high platinum group element values.

2. OBJECTIVES

The objectives of the present research on samples from the Partridge River intrusion of the Duluth Complex are as follows.

1. Determine the variation of laboratory drainage quality and rates of chemical release as a function of sulfur content and time for $0.053 < d < 0.149$ mm samples.

2. Determine the variation of laboratory drainage quality and rates of chemical release as a function of sulfur content and time for $d < 6.35$ mm samples.

3. Compare the dependences of drainage quality and chemical release rates on sulfur content and time determined for $0.053 < d < 0.149$ mm rock with those determined for $d < 6.35$ mm rock.

4. Compare the dependences of drainage quality and chemical release rates on sulfur content and time determined for Babbitt prospect rock samples with those generated by Dunka Road rock.

5. Compare results for $0.053 < d < 0.149$ mm rock samples with those for Duluth Complex tailings sample (unsaturated reactor) produced by processing of Duluth Complex rock from the Babbitt prospect on which testing was initiated in 2002.

6. Compare results for $0.053 < d < 0.149$ mm rock samples with those for basal Duluth Complex rock samples from the South Kawishiwi intrusion on which testing was initiated in 1989 and 1990.

3. METHODS

3.1. Materials

3.1.1. Babbitt Prospect

Duluth Complex rock samples from the Babbitt prospect were obtained from a bulk sample taken in 2001. The bulk sample site was in the SE of the NW, Section 29, Township 60N, Range 12W, near the approximate location of AMAX drill hole number 321. The excavated area for the bulk sample was approximately 300 feet by 200 feet and had roughly 25 feet of glacial overburden above the bedrock. Although the overburden contained large rocks and boulders, it appeared to be fairly silt-rich. There was a very distinct boundary between the oxidized upper layer of the soil and the lower reducing layer of soil, the upper layer being red and the lower layer being grey. There was at least five feet of the reducing layer above the bedrock, which was not weathered and seemed very competent (Eger 2001).

After the bulk sample was blasted, rock was trucked to a loading pocket, crushed to approximately minus six inches, and a sample was taken and transported by rail for metallurgical testing. From the remaining pile, several buckets of rock were collected for possible dissolution testing. Highly mineralized rock, as determined by visual examination, was excluded from this collection. Fourteen samples, approximately two to four kg each, were selected from the buckets. The objective of this sampling was to obtain, by visual examination, samples with a range of sulfur contents.

The 14 samples were sent to Lerch Brothers Inc. (Hibbing, MN) where they were stage crushed to −0.64-cm to limit generation of fines. The process entailed screening the drill core sample to −0.64 cm and three crushing steps (large jaw crusher set at 1.92 cm, small jaw crusher set at 0.95 cm, roll crusher set at 0.64 cm). After each of the first two crushing steps, the −0.64 cm fraction was collected and the oversize was passed to the next crushing phase. Five 250-gram samples and one 100-gram sample were split from each bulk sample. The five 250-gram samples were stored in sealed plastic bags until filling the cells. The 100-gram split was pulped for subsequent analysis. Sulfur and carbon dioxide contents of the pulps were determined, and seven samples were selected for dissolution testing in humidity cells based on these analyses. In addition, a 400-gram sample was split from six of the samples, crushed to –100 / +270 mesh ($0.053 < d < 0.149$ mm) and wet sieved for use in the small reactors.

3.1.2. Dunka Road Prospect

The Dunka Road prospect is approximately three kilometers west southwest of the Babbitt prospect (Figure 1). On 16 August 2001, Minnesota Department of Natural Resources (MN DNR) personnel examined information (rock unit, mineral grain size, sulfur and copper content, and visually estimated plagioclase, olivine, and pyroxene contents) on drill holes from the Dunka Road Cu-Ni prospect for the purpose of selecting samples for laboratory waste characterization experiments. The information was from a total of 32 drill holes that were drilled in 1999 and 2000. The drill core had a diameter of either 1 3/4" or 2" (depending on the year of drilling) and

the required footage needed for a single laboratory cell was five feet, yielding a mass of approximately 1200 to 2000 grams.

Based on the drill core logs and mass requirements 26 intervals were selected, and the core (half of the original) was brought to the MN DNR Hibbing office and sawed in half. One quarter of the core was packaged, labeled, and inventoried in the core building for future work. The remaining core was returned to the mining company. Ten of the inventoried samples were selected for possible use in the laboratory experiment. All samples were stage crushed to –6.35 mm and one sample was crushed and sieved to obtain the 0.053-0.149 mm fraction (see section 3.1.1 for methods). A split of each sample was analyzed for sulfur content at Lerch Brothers Inc. Samples were selected for laboratory testing based on sulfur content.

The majority of the samples had relatively low sulfur contents and these samples were excluded from the experiment. A 0.23% sulfur sample was selected for testing in a humidity cell. The remaining higher sulfur samples lacked sufficient mass to run in large cells. Due to the sample mass available a 0.25% sulfur sample was mixed with a 0.36% sulfur sample and a 1.30% sulfur sample was mixed with a 0.03% sulfur sample (Table 1). The two mixed samples were analyzed at Lerch Brothers for sulfur and carbon dioxide. Sulfur contents for the 0.25% and 0.36% S composite and the 1.30% and 0.03%S composite were 0.31% and 0.61% S, respectively.

All three samples (0.23, 0.31, 0.61% S) were run in humidity cells using a 1000-gram mass of rock finer than 6.35 mm rock. The 0.31% S sample was also run in a small reactor using a 75-gram mass of 0.053-0.149 mm rock.

3.2. Apparatus

3.2.1. Small Reactors (75 g charge)

The samples were placed onto a 1.6-micron Whatman GF/A glass fiber filter (5.5 cm diameter) resting on a perforated plastic plate in the upper section of a two-stage filter unit or reactor (Figure 2). On 19 August 2003 the reactors were filled. A plastic weighing boat was placed on a scale and 75 grams was weighed out and added to the top section of the small reactors. The calculated bed depth was about 15 mm (density = 3 g cm^{-3}, 40% voids, d = 5.66 cm), but the depth was measured as roughly 18 mm after 400 weeks (Appendix 2, Attachment A2.2).

3.2.2. Humidity Cells (1 kg charge)

Cylindrical cells (I.D. = 10.2 cm, h = 19.0 cm) constructed of acrylic plastic were used in the experiment. A 0.64-cm high acrylic sleeve was glued into the bottom of the cell to support a 0.16-cm thick PVC plate, which was perforated with 0.32-cm holes spaced 0.48 cm on center. The plate was covered with a polypropylene filter (National Filter Media Corporation, Polymax B, Style 226-075-2) that supported the solids. The bottom was sealed with an acrylic plate with a 0.85-cm outlet port in the center and a removable cover, with a 0.85-cm vent hole, was place on top of the cell (Figure 3). Prior to filling, the cells were acid washed with 10% HNO_3 and

then rinsed three times with distilled water. The cells were weighed and the weight was recorded on the cell.

On 20 August 2003 the cells were each filled with 1000 g of rock. A plastic pan was placed on a scale and four of the five 250-gram splits were added. The weight was generally about 50 grams over the desired 1000-gram sample size. A plastic spoon was used to randomly remove the excess and add it to the remaining 250-gram split. The 1000-gram sample was then poured into the cell. The samples were very dry and some of the very fine dust-like particles were lost due to suction from the exhaust hood. The plastic pan and spoon were rinsed with distilled water after each reactor was filled. The cells with dry solids were weighed and the tare weight recorded. Measured bed depth was in the range of 6.8 to 7.2 cm, and the depth was calculated as 6.9 cm, assuming a density of 3 g cm^{-3} and 40% voids used for calculation (Appendix 2, Attachment A2.2).

3.3. Experimental Procedure

A total of eight small reactors and eleven humidity cells were used. The small reactors were used to allow direct correlation of data with those generated previously using these reactors to examine the dissolution of Duluth Complex rock. The eleven humidity cells were intended to provide a comparison of the small reactors and the ASTM D5744 method that uses these cells (ASTM 2007). Both methods provided for weekly rinsing of solutes from the rock samples. Between rinses the rock samples were maintained at fairly constant temperature and humidity.

The solids were rinsed three times (week 0) to remove oxidation products that accumulated on the solids from the time they were processed until the beginning of the experiment. Distilled water (200 mL) was slowly dripped into the small reactors from a buret and was allowed to drain freely from the cell. The reactors have since been rinsed weekly with one 200-mL volume of distilled water. Between rinses these samples are exposed to air and virtually all the pore water evaporates during the one-week oxidation period.

For the humidity cells, a rubber cap was placed on the outlet port and 500 mL of distilled water was slowly dripped into each cell from a separatory funnel. The water remained in the cell for ten minutes after the last one was filled, and then the cells were uncapped and allowed to drain. The cells were subsequently rinsed weekly with one 500 mL volume of distilled water. These cells were covered during the oxidation period, and evaporation from them was minimal.

Effluent (or drainage) samples were weighed and analyzed for pH, specific conductance, and alkalinity or acidity at the MN DNR laboratory. From the remaining drainage, samples were collected and filtered for subsequent determinations of sulfate and metal concentrations. After filtration, samples to be analyzed for metals were preserved with 0.2 mL Baker Instra-Analyzed nitric acid per 50 mL sample.

3.4. Reaction Conditions

Between rinses the samples were retained in the cells to oxidize. The cells were stored in a room (8.5 × 10.5 × 9.5 ft) in which temperature and humidity were controlled by commercially available equipment. Temperature was maintained by a thermostatically controlled heater (Lakewood 750) and humidity was maintained by a Holmes HM 3500 humidifier and a Sears Kenmore dehumidifier, both of which were controlled by Honeywell humidistats.

Water retention in the small reactors and large cells was determined by weighing. During the first seven weeks of the experiment the units were weighed four to five times a week. Subsequently, weights were determined immediately before and one day after rinsing.

3.5. Analyses

3.5.1. Solid Phase Analyses

Samples were analyzed at Lerch Brothers for particle size, total sulfur and carbon dioxide. Particle size distribution for the –6.35-mm samples was determined using method ASTM E 276-93. Total sulfur contents of all samples were determined using a LECO combustion furnace (method ASTM E395-95A) and carbon dioxide content was determined using a gas evolution method (ASTM E350-89C). The remaining solid-phase constituents were determined by Chemex (ALS Minerals) in Reno, NV (Appendix 1, Tables A1.3-A1.8). Whole rock constituents were determined using a lithium borate fusion method (ME-ICP06) and analysis by inductively coupled plasma-atomic emission spectroscopy (ICP-AES). Concentrations of Ag, As, Cd, Co, Cu, Mo, Ni, Pb, Sc, and Zn were determined using a 4-acid digestion for base metals (ME-4ACD81), with analysis by ICP-MS. Other trace elements were determined using a lithium metaborate fusion for resistive elements (ME-MS81). Chemex also measured total sulfur and carbon (by combustion furnace) as well as copper and nickel (aqua regia digestion and ICP) for five size fractions of the -6.35-mm samples (-270 mesh, +270/-100 mesh, +100/-35 mesh, +35/-10 mesh, +10 mesh/-6/35 mm).

Peter McSwiggen (McSwiggen & Associates, Minneapolis, MN) determined mineral content and chemistries (Appendix 1, Attachment A1.2). Grains from seven waste rock samples were mounted into a one-inch epoxy mount, which was then polished and carbon coated. Over one hundred grains were randomly selected for analysis. Each grain was analyzed using a JEOL 8600 Electron Probe Microanalyzer (15 kV accelerating voltage, 20 namp beam current, 5 second dwell time, 5 μm beam diameter).

McSwiggen determined the grain size distribution of the particles using image analysis techniques. Using the program NIH ImageJ, a threshold was applied to the backscattered electron image to separate the particles from the mounting medium using the intensity of the backscattered electron signal. ImageJ was then used to process the resulting image to determine the actual particles, and then measure each particle's longest length, perimeter, and area. In Microsoft Excel, the particles were sorted based on their area, and a cumulative area was calculated. The percent cumulative area was than calculated, as well as the circular diameter of

each particle. This data was plotted onto percent cumulative area vs. circular diameter graphs in order to determine P5, median, P95 grain size for each sample. This process was done for both the sulfide grains and all grains (Appendix 1, Attachment A1.3). Using the individually measured sulfide particle sizes, the probability of a sulfide grain to be exposed at the surface of a silicate particle was calculated using the radial differences of the sulfide and silicate grains (Appendix 1, Attachment A1.4).

3.5.2. Drainage Quality Analyses

Water samples were analyzed for specific conductance, pH, and alkalinity or acidity at the MN DNR laboratory. Specific conductance was analyzed using a Myron L conductivity meter, and an Orion SA720 meter, equipped with a Ross combination pH electrode (8165), was used for pH analyses. Alkalinity (for pH $\geq$ 6.3) and acidity were determined using standard titration techniques for endpoints of 4.5 and 8.3, respectively (APHA et al. 1992). The remaining sample was filtered for metals and sulfate analyses. Metals samples were acidified with 0.2 mL ultra-pure nitric acid per 60 mL sample.

Metals and sulfate samples were analyzed at Minnesota Department of Agriculture (MDA) in St. Paul, MN. Ca, Mg, Na, and K were determined with a Varian 400 SPECTRAA; inductively coupled plasma mass spectrometry (ICP-MS, Hewlett Packard HP4500 Series, model #G1820A) was used for the remaining metals analyses. Sulfate concentrations were determined using a Lachat QuickChem 8000 or, for $[SO_4] < 5$ mg/L, a Dionex ion chromatograph. More detailed analyses were conducted on samples collected on weeks 88, 126, 158, and 174 using a Perkin Elmer Elan 6000 ICP-Mass Spectrometer. These analyses were conducted by ACME Analytical Laboratories Ltd., Vancouver, BC. The results are not discussed but are presented in Appendix 1, Attachment A1.1.

3.6. Release Rate Calculation Methods

Weekly chemical mass release for each solute was calculated as the product of drainage volume and solute concentration. For weeks that solute concentrations were not determined, the concentration was estimated as the average of previous and subsequent measured values. Cumulative release was calculated as the sum of weekly release values.

Rates of solute release were calculated for every week drainage quality samples were analyzed for a specified solute. For each solute, the weekly rate of release was calculated as the product of the volume and solute concentration divided by the rock mass originally present. The weekly release rates were averaged over 52 week periods to determine temporal variability over the 400 week period of record. Standard deviations were calculated to provide a measure of rate variability during the periods.

In order to gain a better understanding of the rates of release as a function of sulfur content, the annual average release rates (Tables 9, 10) were plotted as a function of sulfur content for both fine and coarse particles (see Figure 20 for example). A set of graphs for each parameter (SO_4, Ca, Mg, Na, K, Cu, Ni, Co, Zn) were developed for each year in the 400 week period of record

and a linear regression was fit for each data set (Appendix 4, Figures A4.20-A4.29). The slopes of these regressions were normalized per gram of sulfur (mass of rock x g sulfur/g rock) to calculate an annual composite release rate per gram of sulfur for both particle sizes. These composite rates were plotted annually for comparative purposes between the particle sizes (Figures 21-25).

This examination of annual composite release rates per gram of sulfur was done for all parameters (SO_4, Ca, Mg, Na, K, Cu, Ni, Co, Zn). The regression fit is very good for those parameters that exhibit a strong correlation with sulfur (i.e. sulfate), but not as good for those that have a weak correlation with sulfur, such as potassium (Table 12). It should be noted that for all parameters these linear regressions have associated y-intercept values (Appendix 4, Table A4.21) that are not accounted for in Figures 21-25. It should also be noted that magnesium release from reactor 7 was omitted due to lower than expected drainage quality which resulted in a weak correlation with respect to sulfur. This does not necessarily mean the drainage quality data from reactor 7 is inaccurate, but rather the magnesium release from this sample does not represent the bulk release from the finer particles as a function of sulfur content.

4. RESULTS

4.1. Introduction

In August 2003, tests were initiated on the dissolution of Duluth Complex rock samples from the Babbitt and Dunka Road prospects of the Partridge River intrusion. Seven –100/+270 mesh ($0.053 < d < 0.149$ mm) samples were tested, six from the Babbitt prospect and one from the Dunka Road prospect. Sulfur and carbon dioxide content ranges were 0.07 to 0.94 percent and 0.07 to 0.2 percent, respectively. Ten –0.25-inch ($d < 6.35$ mm) samples were tested, seven from the Babbitt prospect and three from the Dunka road prospect. Sulfur and carbon dioxide content ranges were 0.13 to 1.36 percent and 0.05 to 0.21 percent, respectively. One sample in each size fraction was run in duplicate and drainage quality for both pairs replicated well (Figures 4, 5). These experiments remain in progress and the following reporting represent an assessment of the data through week 400.

4.2. Solid-Phase Analyses

Particle size distribution of the –6.35-mm humidity cell samples was determined. Roughly 29 to 57 percent of the sample masses were finer than 10 mesh (2 mm). The corresponding ranges for 35 and 200 mesh (0.5 and 0.074 mm, respectively) were 12 to 23 percent and 3.4 to 5.6 percent (Table 2). Total sulfur content for the bulk samples ranged from 0.07 to 1.36 percent and correspondingly the carbon dioxide contents ranged from 0.05 to 0.2 percent (Tables 3 and 4). There were no major differences in whole rock chemistry, heavy metal composition, and mineral contents between the reactors and humidity cells. The average major whole rock components were SiO_2 (44.8%), Al_2O_3 (17.2%), and Fe_2O_3 (14.0%) (Table 4). Average copper, nickel, cobalt and zinc contents were determined as 0.21, 0.06, 0.008, 0.01%, respectively (Table 5). Mineral contents and chemistries were 36-65% plagioclase $(Ca_{0.54\text{-}0.67}Na_{0.35\text{-}0.46})Al_{1.52\text{-}1.61}Si_{2.34\text{-}2.46}O_8$, 12-38% olivine $(Mg_{1.05\text{-}1.27}Fe_{0.73\text{-}0.98})Si_{0.97\text{-}1.00}O_4$, and 2-19% augite $(Ca_{0.73\text{-}0.90}Na_{0.017\text{-}0.025})(Mg_{0.73\text{-}0.82}Fe_{0.27\text{-}0.41}Ti_{0.013\text{-}0.035})(Si_{1.91\text{-}1.96}Al_{0.07\text{-}0.10})O_6$. Ilmenite, biotite, potassium feldspar, and orthopyroxene were also reported present, on average, at 1-6% (Appendix 1, Attachment A1.2).

The Babbitt prospect samples were marginally finer than Dunka Road prospect samples of similar sulfur content (Table 2). Differences in major cation contents were generally small, For example CaO contents of the Babbitt prospect rock that were about 0.8 to 0.9 and times those of the Dunka Road samples (Table 6). The largest difference was observed for potassium, for which Babbitt prospect contents were 1.6 - 2.2 times those of the Dunka Road samples. These differences were reflected in the relative biotite contents of the samples (Table 7). Copper, nickel, and zinc contents of the Babbitt prospect rock were consistently higher than those from the Dunka Road samples and cobalt contents were similar (Table 6).

4.3. Reaction Conditions

4.3.1. Temperature and Relative Humidity

Over the 400-week period of record temperature and relative humidity (in the room in which reactors and humidity cells were stored) were measured three to four times a week with a Taylor wet-bulb, dry-bulb hygrometer. Average weekly values for temperature ranged from 20 to 28 ^{o}C, averaging 23.6 ^{o}C (s.d. = 1.3 ^{o}C, n = 392). The corresponding range for relative humidity was 42% to 75%, averaging 57.9% (s.d. = 4.2%, n = 392) (Appendix 2, Table A2.2). Data from 3 May to 28 June 2004 were lost.

4.3.2 Water Retention

Water retention did not replicate exceptionally well for the duplicate pairs of small reactors or humidity cells. In both cases, the typical variation among different solids appeared to be less than that between the duplicates (Appendix 2, Figures A2.1 and A2.2). The reason for this is unclear. As indicated by results from the following section, drainage quality from the paired reactors was in good agreement, suggesting that the magnitude of the differences in water retention did not substantially affect rock dissolution and the consequent drainage quality.

For the small reactors, the average water retention after the rinse ranged from 17 to 27 g. Water retention after the rinse continued to decrease to approximately 5 mL by week 300 and plateaued. The average water retention before the rinse was less than -2 mL (Table 8). The low water retention before the rinse indicated that virtually all water evaporated during the weekly cycle. Although the rate of evaporation was not measured for these samples, daily measurements were made during a six-week trial on the 75 gram, -100/+270 mesh Duluth Complex waste rock samples from South Kawishiwi blast hole samples from the Dunka Mine (MNDNR 2013). These measurements determined that approximately 75% of retained water evaporated within three days of rinsing, with approximately equal amounts lost each day. Reactors were completely dry before the next weekly water addition and, at the rate of drying from the first two days, would have completely dried less than four days after water was added.

For the large cells, average water retention after the rinse ranged from 100 to 166 g, and the variation was apparently related to particle size distribution (Table 2). Additional data analysis was conducted to examine the relationship between water retention after the rinse and the fraction of a sample finer than 2 mm. The average water retained after the rinse was plotted against the sample mass occurring in the -10 mesh (2 mm) fraction. Linear regression analysis was conducted and yielded respective slope and intercept values of 0.22 and 34 (n = 11, r^2 = 0.713) (Figure 6). The slope suggests that 0.22 g water were retained per gram of -10 mesh rock. Water retention in the -10 mesh fraction was determined using a rock density of 3 g cm^{-3}, estimating a bed with 40% voids, and assuming all pores in the -10 fraction are filled yields a water retention value of 0.22 g per gram -10 mesh rock (Appendix 2, Attachment A2.3). This interpretation is reasonably consistent with interpretation of the linear regression analysis. Furthermore, the y-axis intercept derived by the regression suggests that 34 grams of water were associated with the +10 mesh fraction and the humidity cell itself. The 34 grams represents from

20 to 34 percent of the entire water retention, with most values near 30 percent. The range before the rinse was 94 to 151 g, with most values between roughly 100 and 110 g (Table 8). Water loss during the week ranged from roughly 4 to 15 g, indicating that little evaporation occurred during the weekly cycle.

4.3.3 Leachate Volume

The drainage volumes for the duplicate reactors were in good agreement for the period of record. There were some excursions in Reactor 5, where drainage volume would drop 30-40 mL during a weekly rinse period (Appendix 2, Figure A2.7). However, the average drainage volumes were in very good agreement for reactors 4 and 5 at 167.8±4.3 mL and 168.0±4.8 mL respectively. The average drainage volume for the 8 reactors for the 400 week period of record was 164.6±2.7 mL (Appendix 2, Table A2.5).

The drainage volumes for the duplicate humidity cells were also in good agreement. The average drainage volumes for reactors 12 and 13 were 480.3±6.5 and 486.4±6.3 mL respectively. Drainage volumes from reactor 13 were typically 5-7 mL higher than reactor 12 for the entire period of record (Appendix 2, Figure A2.8). The average drainage volume for the 11 humidity cells for the 400 week period of record was 481.2±3.1 mL (Appendix 2, Table A2.5).

4.4. Dissolution of Babbitt and Dunka Road Prospects rock

4.4.1 Water Quality Replication

For the following discussion, differences in dissolution behaviors of the Babbitt and Dunka Road samples identified in Section 5.3 will be neglected, and the samples will be considered as a single collection of Partridge River intrusion samples. Duplicate 75-g reactors and humidity cells were run with the 0.31%S and 0.55% S, respectively, Babbitt prospect samples and drainage quality agreement between duplicates was generally very good (Figures 4 and 5).

For the reactors (finer particles), replication of pH and concentrations of sulfate and major cations (Ca, Mg, Na, K) was excellent (Figure 4). This agreement is corroborated by average rates of mass release calculated over the period of record. Replicate rates for all five solutes were within 5% of the mean value (Table 9). Rates of nickel, cobalt and zinc release were similarly in good agreement between duplicates. However, copper rates were roughly 20 percent of the mean value, due to higher rates in reactor 4 from year 2 to 6 (Table 10).

For the humidity cells (coarser particles), pH replication was excellent. Small deviations were observed for concentrations of sulfate, calcium, and magnesium, but agreement between replicates was nonetheless very good (Figure 5). This agreement was quantified by average solute release rates determined over the period of record. Replicate rates for sulfate, calcium, magnesium, sodium, and potassium differed by no more than 4% of the respective mean (Table 9). Although the agreement in nickel and cobalt release rates was similar, the difference from the mean for copper and zinc rates was 10 and 8 percent, respectively, of the mean value (Table 10).

4.4.2 Drainage pH

Considering the Babbitt prospect and Dunka Road samples as a whole, drainage pH from both size fractions tended to decrease with time and sulfur content. For the reactors, drainage pH typically decreased over periods of 30 to 50 weeks and then become relatively stable (Figure 7). Drainage pH values from samples with sulfur contents less than 0.31% were generally greater than 6.0, although the 0.20%-S sample did produce several drainage pH values in the range of 5.8 to 6.0. Samples with sulfur contents of 0.63% and 0.94% produced minimum drainage pH values of near 5.2 and 4.3, respectively (Figure 7).

For the humidity cells, drainage pH tended to decrease steadily over time (Figure 7). Drainage pH from samples with sulfur contents of 0.13% to 0.72% typically plateaued after 150-300 weeks and reached minimum pH values ranging from 5.3 to 6.4. Samples with sulfur contents of 1.03% and 1.36% went acidic after 20 weeks and produced minimum pH values of 4.2 and 4.3, respectively, by week 150 (Figure 7).

4.4.3 Sulfate Release (Acid Production)

Sulfate release rates in drainage from the finer particles increased with increasing solid-phase sulfur content. Sulfate release rates tended to peak between weeks 30 and 50 and decreased over the remaining period of record (Figure 8). This behavior coincides with specific conductance over time (Figure 9). Sulfate release rates for most reactors after year 8 were approximately 3-11% of those in year 1. However, the higher sulfur samples (0.63 and 0.94%S) displayed sulfate rates after year 8 that were about 22% of those in year 1 (Table 9). The fraction of solid-phase sulfur depleted during the 8-year period ranged from 13 to 57% (Table 11).

As with the finer particles, sulfate release rates in the humidity cells tended to increase with solid-phase sulfur content. In contrast with the sulfate peaks observed for the finer particles, sulfate release in drainage from the d < 6.35 mm rock tended to be relatively stable over time (Figure 8). Sulfate release rates for most humidity cells after year 8 were approximately 29-83% of those in year 1. One exception was the 1.03%S sample, for which sulfate release at year 8 was the same as year 1 (Table 9). Unlike the other humidity cells, this sample displayed an increasing trend in sulfate release from weeks 50 to 200. The fraction of solid-phase sulfur depleted during the 8-year period ranged from 8 to 37% (Table 11).

4.4.4 Major Cation Release (Acid Neutralization)

For the reactors, molar concentrations of major cations initially decreased in the order Ca > Mg > K > Na and the ordering subsequently evolved to Mg > Ca > Na > K. Release rates tended to parallel those of sulfate, where release rates peaked during weeks 30-50 and decreased for the remainder of the period of record (Figures 10-13). Calcium and potassium release rates in year 8 were on average, 9 and 5 % of those in year 1, respectively (Table 9). In contrast, magnesium and sodium release rates were slightly less variable over the period of record, with release rates at year 8 averaging 29 and 39 % respectively, of year 1 release rates (Table 9). Total molar release of magnesium was the highest, measuring 1.2±0.4 times that of calcium and roughly

3.0±1.2 and 3.3±1.3 times that of sodium and potassium, respectively. Total release of sodium was 1.2±0.3 times that of potassium (Table 11). Because calcium and magnesium are divalent their neutralization potential was roughly six times higher than sodium and potassium.

For the humidity cells, molar concentrations of major cations initially decreased in the order Ca > Mg > Na > K and the ordering subsequently evolved to Mg > Ca > Na > K. Similar to the reactors, major cation release rates tended to parallel sulfate. Calcium, magnesium, sodium, and potassium release decreased steadily from weeks 50 to 100 and began to plateau (Figures 10-13). Release rates in year 8 were on average, 21, 65, 62, and 38 % of those in year 1, respectively (Table 9). Total molar release of magnesium was the highest, measuring 1.1±0.4 times that of calcium and roughly 4.3±1.9 and 5.2±1.7 times that of sodium and potassium, respectively. Total release of sodium was 1.5±1.0 times that of potassium (Table 11). Because calcium and magnesium are divalent their neutralization potential was roughly ten times higher than sodium and potassium.

4.4.5 Heavy Metal Release

For the reactors, nickel contributed the majority of heavy metal release, followed by copper, then cobalt and zinc (Table 11). Copper release tended to gradually increase and peaked between weeks 100-150 (Figure 14). Release of nickel, cobalt, and zinc tended to peak between 30 to 50 weeks and decreased for the remaining period of record (Figures 15-17). Heavy metal release tended to increase with increasing sulfur content (Table 10). Heavy metal release also tended to increase with relatively small decreases in pH (0.3 to 0.5 units). As pH decreased from 7.0 to 6.5, copper, nickel, cobalt, and zinc release rates increased, on average, by 10, 30, 3, 6 times respectively (Table 10). Total release of copper, on average, was less than 9 % of that present initially and, correspondingly, nickel was less than 29 %. The fractions of cobalt and zinc depletion were intermediate to these values (Table 11).

For the humidity cells, nickel contributed the majority of heavy metal release, followed by copper and cobalt, than zinc (Table 11). Release rates tended to gradually increase and level off by week 200 to 250 (Figures 14-17). However, the higher sulfur samples (1.03 and 1.36 %S), peaked around week 30 for nickel, cobalt, and zinc and remained at higher rates than the lower sulfur samples. Copper release in these higher sulfur humidity cells increased up until week 300 and began to plateau (Figure 14). Similar to the reactors, heavy metal release tended to increase with increasing sulfur content and small decreases in pH. As pH decreased from 7.0 to 6.5, copper, nickel, cobalt, and zinc release rates increased, on average, by 12, 16, 9, 3 times respectively (Table 10). Total release of copper, on average, was less than 2 % of that present initially and, correspondingly, nickel was less than 16 %. The fractions of cobalt and zinc depletion were intermediate to these values (Table 11).

5. DISCUSSION

5.1 Introduction

Duluth Complex waste rock was characterized (particle size distribution, chemistry, mineralogy) and subjected to laboratory dissolution testing. Although the dissolution tests remain in progress, the following discussion presents the first 400 weeks of data. The dissolution tests were conducted to examine 1) drainage quality from fine particles (0.053-0.149 mm, 75 grams), 2) drainage quality from coarser particles (–6.35-mm, 1000 grams) and 3) differences in drainage quality between these size fractions. The data also provided a comparison between 4) Babbitt and Dunka Road prospect samples, 5) waste rock and tailings, and 6) Partridge River and South Kawishiwi Intrusion samples.

5.2 Conceptual Overview

This section presents a brief description of the mineral dissolution reactions that occur during chemical weathering of Duluth Complex rock. This summary is intended to provide a conceptual structure to frame the discussion of results. It should be noted that detailed analysis of mineral dissolution reactions, such as calculation of mineral dissolution rates, is beyond the scope of this report.

Iron sulfide and base metal sulfide minerals present in Duluth Complex rock oxidize when in contact with oxygen and water. Pyrrhotite is the predominant iron sulfide mineral present and its oxidation releases acid and sulfate. Some or all of the acid released can be neutralized by dissolution of silicate minerals present, including plagioclase, olivine, and augite. The balance of acid production and acid neutralization reactions determines the solution pH. Oxidation of base metal sulfides such as chalcopyrite and pentlandite releases copper and nickel, respectively. The metals released can remain in solution or be removed in secondary phases due to chemical precipitation, adsorption, or coprecipitation. The extent to which metals released from the sulfide phase remain in solution increases as solution pH decreases, that is, as the solution becomes more acidic base metal concentrations tend to increase.

Sulfide mineral oxidation and silicate mineral dissolution are surface reactions and, therefore, the rates of solute release depend on the mineral surface area exposed. All other factors being equal, the mineral surface area exposed will increase as the mineral abundance increases. In addition, two factors contribute to mineral surface area exposed increasing as rock particle size decreases. First, for a given mineral mass the total mineral surface area increases as the mineral particle size decreases. That is, a kilogram of fine mineral particles will have more surface area than a kilogram of coarse particles. Second, the mineral surface area exposed increases as the degree of mineral "liberation" from the mineral matrix of the rock increases, and the extent of liberation increases as the rock particle size decreases. That is, a mineral occurring between or within other minerals in a large rock fragment will become exposed if the fragment is crushed to a finer size.

Sulfide mineral oxidation and silicate mineral dissolution rates are also affected by solution pH. Assuming oxidation of sulfide minerals in general parallels that of pyrite, abiotic rates of oxidation by oxygen decreases slightly as pH decreases. However, the overall abiotic rate increases as pH decreases into a range where ferric iron becomes the dominant oxidant (Williamson and Rimstidt 1994; Nordstrom 1982). As pH further decreases, bacterial oxidation of ferrous iron becomes the rate limiting step in the oxidation by ferric iron (Singer and Stumm

1970). In laboratory tests on Duluth Complex rock, the pyrrhotite oxidation rate in the pH range of 3.5 to 4.05 was roughly six to seven times that in the range of 5.35 to 6.1, and the higher rate was attributed to bacterially mediated oxidation (Lapakko and Antonson 1994). Silicate mineral dissolution rates also tend to increase as pH decreases but to a lesser degree (White and Brantley 1995).

5.3 Comparison of Fine-Grained and Coarse-Grained Particles

Certain general trends related to solid-phase sulfur content and time were apparent for both the fine ($0.053 < d < 0.149$ mm) and coarse ($d < 6.35$ mm) particles. For both particle sizes, as sulfur content increased, the rates of sulfate release increased and drainage pH decreased (Figures 18, 19). The increasing sulfur content reflected increasing sulfide mineral surface area, which yielded increased sulfide mineral oxidation and consequent acid production. The more rapid acid production tended to result in decreasing drainage pH and more rapid release of both major cations and heavy metals.

With respect to general temporal variations, sulfate release rates for both size fractions tended to decrease over time and plateau. Rates of major cation release tended to parallel sulfate release rates, although peak sodium release occurred later than peaks for other solutes. The major cation release was a response to the variation of rates of acid production reflected by sulfate concentrations. Heavy metal concentrations were largely controlled by drainage pH and, consequently, tended to increase as sulfur content increased. Heavy metal release from the fines tended to peak within the first 50 weeks and then decline and plateau, whereas release from the coarse particles was more gradual in response to slowly declining pH.

Although similar trends were observed for the two size fraction samples, the variations in solute release with respect to sulfur content and time differed quantitatively. Per unit mineral mass, the finer particles had greater sulfide mineral surface area exposed and available for oxidation. As a result, sulfate release rates from the finer fraction were approximately three times those from the coarse fraction during the first year of dissolution (Figure 21). Sulfate release rates from the fine fraction tended to peak within 30 to 50 weeks of reaction and declined fairly rapidly over time, influenced by depletion of the sulfide minerals and probably coating of the sulfide minerals remaining (Figure 8). Corresponding drainage pH minimums were reached between weeks 60 and 120, lagging about a year behind the observed maximums for sulfate release (Figure 7). In contrast, sulfate release from the coarse particles declined fairly slowly, and drainage pH from all but the two highest sulfur samples declined over the entire period of record (Figures 7, 8). In response to the associated decreasing rates of acid production the rates of calcium and magnesium release declined correspondingly, hence the parallelism between the release of major cations and sulfate (Figures 10, 11).

For both particle sizes, the dissolution of calcium and magnesium-bearing minerals contributed the most acid neutralization (Appendix 4, Figures A4.1-A4.19). The elevated sulfide oxidation in the finer fraction resulted in more rapid silicate mineral dissolution and consequent acid neutralization. Release of calcium and magnesium from the fine fraction in the first year was 1.6 and 1.9 times that from the coarse fraction, respectively (Figure 22). Total mass release of

calcium and magnesium from the fine fraction for the period of record averaged 1.4 and 1.6 times that from the coarse fraction, respectively (Table 11). It should be noted, however, that calcium release rates from the coarse fraction exceeded that from the fine fraction after year 4 (Figure 22).

Sulfate and heavy metal release result from oxidation of sulfide mineral phases, such as pyrrhotite and chalcopyrite. Roughly 75 to 90 percent of the calcium occurs in plagioclase and the majority of the remainder is with augite (App 1, Attach A1.2 - Table A1.2.8). The small amount of carbon dioxide present is likely associated with calcium, and the rapid dissolution of this phase contributes to the initially elevated calcium release (Figure 10). Plagioclase also hosts virtually all of the sodium present (App 1, Attach A1.2 - Table A1.2.8). If plagioclase were dissolving stoichiometrically, calcium would be released at a ratio of 1.5:1 to sodium (App 1, Table A1.2.1). However the ratio of Ca: Na total molar release was approximately 1.5 to 3 times the stoichiometric ratio of plagioclase dissolution after 400 weeks. This indicates calcium was being released by other minerals such as augite, the second largest host of calcium. Potential sources of magnesium release were olivine, augite, and biotite dissolution (App 1, Attach A1.2 - Table A1.2.8). Additional data analysis should be conducted in the future to better assess major cation contributions from individual mineral phases and rates of mineral dissolution. Such efforts are beyond the scope of the present project.

This rapid decrease of drainage pH in the finer fraction resulted in heavy metal release rates approximately three to four times those from the coarser fraction during the first two years of dissolution (Figure 26). In contrast, sulfide oxidation from the coarser fraction remained fairly constant throughout the period of record, and drainage pH declined slowly over time. Consequently by year 7 the nickel and cobalt release rates from the coarser particles were roughly five times higher than the fines (Figures 24, 25). Although it appeared that drainage pH was the largest influence on heavy metal release, it also was likely influenced by the greater degree of metal depletion from the fine grained samples with sulfur contents of at least 0.2 % (Table 11). Similar dependences on particle size and sulfur content were observed for copper and zinc, although release rates were higher in the reactors over the entire period of record (Figures 24, 25).

5.4 Comparison of Babbitt and Dunka Road Samples

Comparison of the Babbitt prospect and Dunka Road prospect rocks is tenuous because of the limited number of Dunka Road samples tested. Only one Dunka Road sample was tested in the 0.053-0.149 mm size fraction and three samples in the d < 6.35 mm fraction. For the finer fraction, results for the 0.31 %S Dunka Road sample were compared to those from the duplicated 0.31 %S Babbitt prospect samples. For the coarser fraction, Dunka Road samples with sulfur contents of 0.23, 0.31, and 0.61 % were compared with Babbitt prospect samples with sulfur contents of 0.21, 0.33, and 0.55 % (duplicated). Differences in dissolution of rocks from the two prospects would be expected to result from differences in mineral content, mineral chemistries, and modes of mineral occurrence.

5.4.1 Solids

The total sulfide mineral content of each pair was largely normalized by selecting samples of similar sulfur content for comparison of the two prospects. However the heavy metal content of the Babbitt Prospect samples tended to be higher than that of the Dunka Road samples, especially for the fine samples. This was particularly true for copper, for which Babbitt prospect sample contents were 1.3 to 5.5 times those of Dunka Road samples (Table 13). This indicates that chalcopyrite and cubanite constitute a larger fraction of the sulfides in the Babbitt Prospect samples. Nickel, cobalt, and zinc contents of the Babbitt Prospect samples were typically 0.9 to 1.4 times the associated Dunka Road contents, with only three ratios being below 1.0.

Evolved carbon dioxide contents of the paired fine samples (~0.20 %) and the low sulfur coarse samples (~0.08 %) were similar. However, for the two highest sulfur content coarse samples, the CO_2 contents of Dunka Road samples were 2 to 2.5 times their associated Babbitt prospect pair (Table 4). This suggests that the Dunka Road humidity cell samples had higher calcite contents than their Babbitt prospect pairs. The only consistent trends regarding silicate mineral content were higher biotite and potassium feldspar contents of the Babbitt Prospect samples, reflecting potassium contents that were about twice those in the Dunka Road samples (Table 7).

5.4.2 Solute Release

For *fine particles*, solute release is largely dependent on mineral content and mineral chemistry. The particle size is fairly uniform (0.053 to 0.149 mm) and virtually all mineral surfaces are exposed rather than being obstructed due to occurrence among or within other minerals in larger rock particles. Although drainage pH from the fine Babbitt prospect sample was slightly higher than that from the fine Dunka Road samples during the first 30 weeks, values from the two prospects were quite close subsequently (Figure 27). Concentrations of sulfate, major cations (Ca, Mg, Na, K), and base metals (Cu, Ni, Co, Zn) in drainage from the Babbitt prospect sample were typically higher than associated values in drainage from the Dunka Road sample (Figure 27). For sulfate and major cations the elevated release was most marked during the first year, whereas this trend extended beyond two years for base metals (Figure 27).

These trends were driven largely by more rapid sulfide mineral oxidation for Babbitt Prospect samples. Accelerated iron sulfide mineral oxidation enhanced acid generation, which in turn accelerated silicate mineral dissolution and the consequent major cation release. The cause of the accelerated oxidation of sulfide minerals in the Babbitt Prospect rock cannot be determined conclusively. As mentioned above, base metal sulfides (especially chalcopyrite and cubanite) comprised a larger fraction of the total sulfide content in the Babbitt Prospect samples. A similar phenomenon was reported for Dunka Road waste rock with a higher copper sulfide mineral fraction (SRK 2007). Either the sulfide chemistry or features related to this chemistry (e.g. grain size, surface roughness, mineral lattice dislocation density) apparently accelerated oxidation of all sulfide minerals present. Accelerated iron sulfide oxidation and the consequent acid production enhanced silicate mineral dissolution and the associated major cation release. Accelerated base metal sulfide mineral oxidation, due in part to a higher base metal sulfide content, enhanced release of copper, nickel, cobalt, and zinc from the Babbitt prospect samples.

It should be noted that the higher base metal sulfide content of the Babbitt prospect samples does not entirely explain the higher base metal sulfide release. The copper, nickel, cobalt, and zinc contents of the Babbitt prospect sample were 1.3 to 5.5 times the respective values in the Dunka Road sample. The corresponding metal mass releases from the Babbitt prospect sample during the initial 400 weeks of reaction were 2.6 to 42 times those from the Dunka Road samples (Table 13). Thus, the mass release ratios were roughly 2 to 9 times those expected based on the differences in base metal content alone (Table 13).

Some of the additional acceleration can be attributed to the more rapid rate of sulfate release from the Babbitt prospect samples. Specifically, the more rapid release of nickel and zinc from the Babbitt prospect samples is roughly bounded by consideration of differences in metal content and sulfate release rate (Table 13). The degree of copper release acceleration was well beyond that inferred by the compounded influences of solid-phase concentration and sulfate release rates (Table 13). The acceleration cannot be attributed to differences in pH because the pH of drainages from the two samples was similar (Figure 27). This suggests that the copper sulfide minerals present in the Babbitt prospect samples oxidized more rapidly than those in the Dunka Road samples.

For *coarse particles*, relative magnitudes of pH and sulfate and major cation release varied among the three samples subjected to humidity cell testing. For the two lower sulfur Dunka Road samples, pH was higher and rates of sulfate, calcium, magnesium, and sodium release were generally lower than or equal to corresponding values for Babbitt prospect samples (Figures 28, 29). For the highest sulfur sample pair these trends were generally reversed, magnesium being an exception (Figure 30). It should be noted that the difference in sulfur contents for this pair was the largest of the four pairs examined. Thus, there were no consistent major differences in drainage pH, acid production rates, or acid neutralization rates among the three pairs of samples examined. This is consistent with the general mineralogical similarity between the Babbitt and Dunka Road prospect samples. The only common trend was that potassium release from the Dunka Road samples was roughly half that from the Babbitt prospect samples of similar sulfur content (Tables 11). As mentioned above, this was likely due to the lower potassium-bearing minerals in the Dunka Road rock, as was the case for the fine-grained samples.

The lack of a consistent relationship between copper content and sulfate release from the coarse samples indicates that the sulfide mineral composition did not greatly influence the rate of total sulfide mineral oxidation. However, heavy metal release from the Babbitt prospect samples tended to exceed that from the Dunka Road samples. For the two lower sulfur samples, the relative difference in heavy metal release through 400 weeks was beyond that which would be expected based on differences in heavy metal content and sulfate release rate (Table 13). Lower drainage pH from the Babbitt prospect samples contributed to part of the elevated release. That is, more acidic conditions associated with the Babbitt prospect samples, in addition to higher metal contents and sulfate release rates, likely enhanced metal release. In contrast, drainage pH from the Dunka Road sample was lower than that from the Babbitt prospect sample for the high sulfur pair (Figure 30). Despite this “advantage” for the Dunka Road rock, the relative metal release from the Babbitt Prospect samples was roughly equal to the differential indicated by the differences in metal content and sulfate release rate (Table 13). As was the case for the fine

particles, it appears that the oxidation rates of heavy metal sulfides present in the Babbitt prospect exceeded that of heavy metal sulfides in the Dunka Road prospect.

In summary, there were no consistent differences between the Dunka Road and Babbitt prospects in drainage pH or rates of sulfate, calcium, and sodium release. Magnesium release from the Babbitt prospect samples was consistently 1.1 to 1.2 times that from the Dunka Road samples (Table 14), but there was no obvious solid-phase attribute to which this difference could be correlated. Although the largest observed ratio (2.8) was associated with the largest difference in olivine content, no trend between magnesium release and olivine content was apparent (Table 7). Potassium release from the Dunka Road fine and coarse samples was roughly half that from their paired Babbitt prospect sample. This reasonably reflected differences in biotite contents between the two prospects.

Heavy metal release from the Dunka Road rock tended to be lower than that from the Babbitt prospect rock. Whereas some of the difference could be ascribed to higher metal content of the Babbitt prospect rock, differences in total sulfide mineral oxidation, and drainage pH also played a role. Data from the fine sample pair suggested that heavy metal sulfides from the Babbitt prospect oxidized more rapidly (when normalized per unit mass). This was particularly the case for copper sulfides. Whereas the aforementioned trends were observed for the sample pairs tested, it should be emphasized that these comparisons consider only a few samples and may not represent general trends for the two prospects.

5.5 Comparison of Duluth Complex Waste Rock and Tailings

A sample of 0.2% S tailings from the Babbitt Prospect of the Partridge River Intrusion was subjected to dissolution in duplicate under the same conditions to which the waste rock samples were exposed (unsaturated tailings reactors 1 and 2). This section compares results from this tailings sample to those from the reactor containing the 0.20 %S waste rock sample. The waste rock particles ranged from 0.053 to 0.149 mm in diameter and were coarser than the tailings, for which roughly 51 % were in the aforementioned range, 29 % were finer than 0.053 mm, and 16 % were finer than 0.025 mm (Lapakko 2013, Table 2). The plagioclase contents of the waste rock and tailings were similar (50 and 57 percent, respectively), as were the calcium contents of the plagioclase present (Appendix 1, Table A1.11). Augite and biotite contents of the waste rock were reported to be slightly lower than those of the tailings (11 vs. 7.8% and 5.7 vs. 2.3%, respectively). In contrast, the olivine content of the waste rock was considerably higher than that in the tailings (26 vs. 9%). Respective concentrations of copper, nickel, and cobalt in the waste rock were 2.0, 1.2, and 1.2 times those in the tailings (Appendix 1, Table A1.10).

The pH of drainage from the waste rock sample was typically 0.8 to 1 unit lower than that of the tailings, with respective minimum values of 5.7 and 6.5 (Figure 31). Both higher sulfide mineral oxidation rates and lower silicate mineral dissolution rates contributed to the lower pH values from the waste rock.

The rates of sulfate release from the tailings averaged about 60 μmol (kg rock·wk)$^{-1}$ over the first eight years of the experiment (Lapakko et al. 2013), 80 percent of the 75 μmol (kg rock·wk)$^{-1}$

reported for the 0.20 %S waste rock over the same period (Table 9). Sulfate release rates for the tailings and rock were very close in year 1 (260 vs 240 μmol (kg rock·wk)$^{-1}$) and year 8 (16 vs 15 μmol (kg rock·wk)$^{-1}$). Rates for the rock tended to be somewhat higher in the intermediate years (Figure 31).

Sulfate release rates are influenced by the sulfide mineral surface area exposed, the availability of oxygen, and the sulfide mineral chemistry. Because the rates of sulfate release from the waste rock and tailings were not substantially different, these factors did not appear to exert a major influence. Nonetheless, these factors are briefly discussed in the interest of providing perspective on the oxidation of sulfide minerals present in the two solids.

Considering the finer particle size of the tailings, and associated higher specific surface area, it would be expected that they would exhibit higher sulfate release rates. The lower observed sulfate release rates observed for the tailings are apparently the result of removal of "liberated" sulfide minerals during mineral processing. Consequently, relative to unprocessed waste rock, a greater fraction of sulfide minerals present in the tailings are included within other rock particles and, therefore, less available for reaction. This is supported by mineralogical analyses that reported respective liberation probabilities of 100 and 87 percent for the rock and tailings (Appendix 1, Attachment A1.4).

The finer grained tailings also might have limited oxygen diffusion to sulfide mineral surfaces and thereby inhibited sulfide mineral oxidation. Both the finer grain size itself and increased water retention with finer grain size would tend to reduce diffusion. However limitation of oxygen diffusion was probably not substantial due to the thin tailings beds used in the experiment and the fact that water retained after the rinse evaporated within about three days (Lapakko et al. 2013). Although evaporation rates were not determined in the present study, similar evaporation rates were measured for 75-gram samples of -100/+270 mesh rock in previous work (see Section 4.3.2).

It should be noted that there were also differences in the specific sulfide minerals present in the two samples. Respective copper contents of 0.1 and 0.051 percent for the waste rock and tailings were used to estimate the fraction of sulfur present with chalcopyrite. Assuming all copper was present as chalcopyrite indicated an estimated 50 percent of the sulfur in the waste rock and 25 percent of the sulfur in the tailings was associated with chalcopyrite. This infers that the pyrrhotite content of the tailings was about 1.5 times that of the waste rock. Pyrrhotite is reported to oxidize more rapidly than chalcopyrite (e.g. Steger and Desjardins 1978), so this would tend to accelerate sulfate release from the tailings relative to the waste rock. However, as discussed in the previous section, samples of higher copper sulfide mineral content tended to have higher sulfate release rates. Thus, it is unclear if the difference in sulfide mineral partitioning between the rock and tailings samples was of substantial influence.

Average rates of calcium and magnesium release from the tailings were on the order of twice those from the rock (Figure 31). For calcium this was apparently largely due to the finer particle size of the tailings, resulting in a greater surface area of calcium-bearing silicate minerals. Plagioclase was the major calcium-bearing mineral in both the waste rock and tailings. There

was little difference between the waste rock and tailings in plagioclase contents (50 vs. 57%) or the atom fractions of calcium in the plagioclase (0.54 vs. 0.57; Appendix 1, Table A1.11). Thus, mineralogical differences between the samples were small and any associated influences on calcium release would likely be minimal.

There were differences in magnesium-bearing contents of the waste rock and tailings. Relative to the tailings, the waste rock had less augite and biotite (7.8 vs. 11%, 2.3 vs. 5.7%, respectively) and more olivine (26% vs. 9.8%). Particle size, and its associated influence on silicate mineral surface area, would be expected to have a similar effect on magnesium release. That is, if there were no differences in magnesium-bearing mineral content, magnesium release from the tailings would be expected to be twice that from the waste rock due to particle size influences alone. If this is so, mineralogical differences between the tailings and rock had little influence on magnesium release. This further suggests, given that the olivine content of the waste rock was more than 2.5 times that of the tailings, that olivine was not a major contributor of magnesium to solution. That is, if olivine were a major magnesium contributor, the waste rock would yield much higher magnesium release than the tailings.

Respective annual rates of copper and nickel from the waste rock, on average, were 40 and 150 times greater than those from the tailings. Similarly annual rates of zinc and cobalt release from the waste rock averaged 4 and 8 times those from than the tailings, respectively. These rates tended to be highest during years 2 and 3 (Figure 32). The differences are attributed to pH values from the waste rock that was near 1 unit lower than those from the tailings. Concentrations of copper, nickel, and cobalt in the waste rock were 2.0, 1.2, and 1.2 times those in the tailings, respectively (Appendix 1, Table A1.10). These differences and a lower degree of base metal sulfide liberation in the tailings probably contributed to a lesser degree.

In summary, the rate of sulfate release from the tailings averaged 80 percent of that from the waste rock sample of similar sulfur content. The difference is believed to be largely the result of a higher degree of sulfide mineral liberation in the rock. In contrast, rates of calcium and magnesium release from the tailings were roughly twice those from the waste rock. The combination of lower sulfide mineral oxidation (and consequent acid production) and higher calcium and magnesium silicate mineral dissolution (and consequent acid neutralization) resulted in drainage pH values from the tailings that were roughly a unit higher than those from the waste rock. Due to the lower waste rock drainage pH (and to a lesser extent, elevated base metal content of the waste rock), base metal release rates from the waste rock averaged 4 to 150 times those from the tailings.

5.6 Comparison of Partridge River and South Kawishiwi Intrusion Samples

The dissolution behavior of Partridge River Intrusion rock (PRI), samples from the Babbitt and Dunka Road prospects, was compared to that of the South Kawishiwi Intrusion rock (SKI), samples from the Dunka Mine. Data from laboratory experiments on samples with diameters of 0.053 to 0.149 mm were used for the comparison because that was the only dissolution test to which the South Kawishiwi samples were subjected. Data from seven PRI samples ($0.07 \leq$ percent sulfur ≤ 0.94) and eleven SKI samples ($0.18 \leq$ percent sulfur ≤ 1.16) were used.

Whereas comparisons of the intrusions were made based on normalized sulfur content, the variation of base metal contents with sulfur for the intrusions differed. Copper and nickel contents in the PRI rock tended to increase with sulfur content but not in the SKI samples. Consequently, copper and nickel contents of the PRI higher sulfur samples were 2 to 5 times higher than respective contents of the SKI samples with similar sulfur content (Figures 33, 34). Conversely, the zinc content of SKI samples increased with sulfur content, but the PRI samples did not respond in a similar manner (Figure 35). Cobalt content did not vary systematically with sulfur content for samples in either intrusion (Figure 36).

PRI samples on average had higher olivine content than the SKI samples, 31 vs 8%. SKI samples on average had higher plagioclase and hypersthene (compared to Opx), 52 vs 45 and 15 vs 3.5%, respectively than the PRI. Average contents between the two intrusion samples were similar for augite, biotite, potassium feldspar, and ilmenite (Appendix 1, Table A1.14).

Drainage pH from the PRI samples tended to be higher than that from the SKI samples. For a specified sulfur content, the minimum drainage pH from the PRI samples was roughly 0.7 units higher than that from the SKI samples (Figure 37). Unlike the PRI samples which were mainly circumneutral (exceptions being the 0.63 and 0.94%S samples), SKI samples with sulfur content as low as 0.41% generated acidic drainage within 400 weeks of reaction. However, the drainage pH from PRI samples reached minimums by week 50-100, the SKI samples declined more slowly, taking as long as 300-350 weeks to reach minimum pH values (Figure 38).

During the first three years, the annual rate of sulfate release from the PRI samples averaged 1.2 to 2 those from the SKI samples of similar sulfur content (Figure 39, 40, Table 15). Sulfate release from the PRI samples peaked in the first 30-50 weeks, a trend associated with the relatively rapid initial decrease in pH. After 250-300 weeks, as drainage pH from the SKI samples continued to decline, sulfate responded with release rates as much as 10 times those from the PRI samples (Figure 40, Table 15). This rapid increase in sulfate release occurred when pH fell below 4, and the increase likely reflected biological mediation of sulfide mineral oxidation. During the 8 year period of record, average sulfur depletion in the PRI and SKI samples was 41 and 54 percent respectively.

The cause of differences in drainage pH from SKI and PRI samples was investigated further. Solid-phase and drainage quality data for the two highest sulfur PRI samples (0.63 and 0.94% S) was compared with associated data for the three SKI samples with comparable sulfur contents (0.57, 0.58, and 0.71%; 0.71, 1.12, and 1.16%, respectively). The minimum drainage pH for the

0.63% S PRI sample was 5.2 as compared to a range of 3.6 to 4.0 for the corresponding SKI samples (Appendix 6, Figure A6.4). Similarly the minimum drainage pH for the 0.94% S PRI sample was 4.9, as compared to a range of 3.2 to 3.6 for the corresponding SKI samples (Appendix 6, Figure A6.5). This indicates that the SKI samples had either a higher capacity to produce acid or a lower capacity to neutralize acid.

Because the SKI samples had lower contents of copper and nickel, a larger fraction of their sulfur was associated with iron. Thus, for samples of the same sulfur content the SKI samples had more iron sulfide and therefore more potential to produce acid. This hypothesis was pursued by comparing solid-phase and reactor release rate data for PRI samples (0.63 and 0.94% S) with associated data for the three SKI samples with comparable sulfur contents (0.57, 0.58, and 0.71%; 0.71, 1.12, and 1.16%, respectively). This comparison indicated that, per unit sulfur present, acid producing fractions of sulfides for the 0.63% S and 0.94% S PRI samples were 0.70 and 0.65, respectively. For comparison the average acid producing fractions of sulfides for the 0.57% S, 0.58% S, and 0.71% S SKI samples was 0.85 and for the 0.71% S, 1.12% S, and 1.16 % S SKI samples was 0.91 (Appendix 6, Table A6.4). Thus, the acid producing fraction of sulfide present in the two PRI sample was roughly 80 and 70 percent, respectively, of the average for the SKI samples with similar sulfur contents. Thus, the amount of acid-producing iron sulfide per unit sulfur present in the SKI samples was greater than that present in the PRI samples. This likely contributed to the lower drainage pH values observed for the SKI samples. It should be noted that this solid-phase difference is rather subtle relative to the large difference in minimum pH values, suggesting a high sensitivity of drainage pH to small changes in sulfur content.

For the five SKI and two PRI samples discussed above, there was no indication that the potential of SKI samples to neutralize acid was less than that of the PRI samples. The CO_2 content range for SKI samples was <0.03 – 0.10% as opposed to 0.2% and 0.07%, respectively, for the PRI samples with sulfur contents of 0.63% and 0.94%. Although the 0.2% CO_2 concentration for the 0.63% S PRI sample is easily the highest reported, the 0.07% concentration for the second sample is in the middle of the range for the SKI samples. Consequently, the calcite contents inferred by the CO_2 contents do not provide compelling insight into the lower drainage pH from the SKI samples.

Dissolution of minerals releasing calcium and magnesium provided the majority of acid neutralization. Besides trace amounts of calcite, calcium was largely present in plagioclase (~70-80% of Ca) and augite. There were some differences in the neutralizing silicate mineral contents and chemistries for the two intrusions. The plagioclase contents (volume percent) of the five SKI samples averaged 48% as compared to 40% for the two PRI samples. The respective average mole fractions of calcium present were 0.50 and 0.55 (Appendix 6, Table A6.5). Thus, the plagioclase content suggested slightly greater neutralization capacity for the SKI samples while the opposite was implied by the calcium contents of the plagioclase. Augite contents and chemistries of the SKI and PRI samples were similar. Thus there is no evidence suggesting that calcium-bearing mineral dissolution would limit acid neutralization by the SKI rocks. This is consistent with the observation that average rates of calcium release as a function of sulfur

content for the SKI samples tended to exceed those for the PRI samples (Appendix 6, Figure A6.13).

The largest silicate mineral difference between rocks from the two intrusions was that of olivine content, which averaged 12 percent for the five SKI samples and 34 percent for the two PRI samples (Appendix 6, Table A6.5). This suggests that the deficiency of olivine might have contributed to the lower drainage pH of the SKI samples. However, magnesium release rates as a function of sulfur release rates were not substantially different between the intrusions (Appendix 6, Figure A6.14), indicating that the difference in olivine contents was not a major factor in the observed acidification of drainage from the SKI samples.

Comparison of heavy metal release from the PRI and SKI intrusions is limited to data prior to week 74 during which drainage pH was circumneutral for all cases (Table 16). Heavy metal concentrations from the SKI samples were determined only sporadically after this time. Prior to week 74, nickel release from the PRI intrusion samples was roughly three times that from the SKI samples (Figure 41). Copper, cobalt, and zinc release from the PRI intrusion samples was not obviously different from that from the SKI samples (Figures 42-44). These trends in heavy metal release coincide with the initial compositions of the intrusion samples (Appendix 1, Table A1.13).

6. SUMMARY

In August 2003 laboratory dissolution tests were initiated on 10 Duluth Complex rock samples from the Babbitt and Dunka Road prospects of the Partridge River intrusion. These experiments remain in progress and the present report analyzes and summarizes data generated through week 400.

Dissolution tests were conducted on 10 coarse samples ($d < 6.35$ mm = 0.25 in.; $0.13 \leq \%S_T \leq 1.36$; $0.05 \leq \%CO_2 \leq 0.21$) and seven fine samples ($0.053 < d < 0.149$ mm; $0.07 \leq \%S_T \leq 0.94$; $0.07 \leq \%CO_2 \leq 0.21$). All samples were characterized with respect to chemistry, mineral content, and mineral chemistry. Of the 10 coarse samples, seven were from the Babbitt prospect and three from the Dunka Road prospect. For these samples particle size distribution and abbreviated chemistry (S, CO_2, Cu, Ni) as a function of particle size were also determined (Appendix 1, Table A1.2). The seven 0.053-0.149 mm samples consisted of six samples from the Babbitt prospect and one from the Dunka Road prospect.

The tests generated data that described mineral dissolution reactions that release acid and heavy metals (sulfide mineral oxidation reactions) and reactions that neutralize acid (silicate mineral dissolution reactions). In general, drainage pH tended to decrease and release rates of heavy metals (copper, nickel, cobalt, zinc) and major cations (calcium, magnesium, sodium, potassium) tended to increase with increasing sulfur content. These results were consistent with the previously reported 1) dependence of sulfide mineral oxidation (as reflected by sulfate release) and the attendant acid production on the amount of sulfide minerals present and 2) importance of sulfide mineral oxidation as a driving force for release of heavy metals and major cations.

The pH of drainage from the 10 coarse samples decreased with increasing sulfur content, with a minimum value of 4.1. For all but the two highest sulfur samples, pH decreased throughout the eight-year experiment. Numerous relationships between drainage quality and both solid-phase sulfur content and dissolution time were identified and quantified to accurately describe solute release and its dependence on solid-phase composition and dissolution time. Regression analysis was used to quantify a strong relationship between rates of sulfate release (reflecting sulfide mineral oxidation) and sulfur content, reflecting exposed sulfide mineral surface area ($0.91 < r^2 < 0.94$). This analysis was used to derive composite annual sulfate release rates (per gram sulfur), which were found to decrease slightly over the eight-year experiment. Release rates for copper, nickel, cobalt, and zinc also correlated highly with sulfur content (typically $0.80 < r^2 < 0.95$). Copper release rates for year 8 were roughly five times those during year 1. In contrast rates of nickel, cobalt, and zinc release at the end of the experiment were roughly half those at the beginning. Magnesium was the dominant cation released, followed by calcium, and sodium. Their releases tended to parallel that of sulfate, reflecting silicate minerals dissolving to neutralize acid produced by iron sulfide mineral oxidation.

Similar analysis was conducted on data generated in the seven finer-sized samples. As with the larger particles, drainage pH decreased and release rates for sulfate, major cations, and heavy metals increased with increasing sulfur content. Although the relationship between minimum drainage pH and sulfur content was similar to that for the larger particles, pH from the finer

samples declined more rapidly, reaching minimums between weeks 60 and 120. Sulfate release from the fine particles were initially three times those from the coarse particles, reflecting the greater sulfide mineral surface area per unit mass of the fine particles, and this rapid release contributed to the aforementioned early pH depression. Sulfate release rates from the fine particles decreased fairly quickly over time and by year seven equaled those from the coarse particles, for which rates decreased more slowly. The rapid decline of rates from the fine particles was attributed to sulfur depletion and probable coating of sulfide minerals.

Dissolution testing of Dunka Road samples was limited to one sample in the 0.053-0.149 mm size fraction and three samples in the d < 6.35 mm fraction. The most notable difference between Dunka Road and Babbitt prospect rock was elevated heavy metal release from the latter. Whereas some of the difference could be ascribed to higher metal content of the Babbitt prospect rock, differences between prospects in total sulfide mineral oxidation rates and drainage pH also played a role. Because the number of samples for comparison was limited, these results should be viewed as preliminary.

The Partridge River rock samples yielded higher sulfate rates than Partridge River tailings of similar sulfur content, due to greater sulfide mineral exposure in the rock samples. Heavy metal release from the rock was also higher than that from the tailings. This was due to the aforementioned elevated sulfide mineral exposure, higher heavy metal contents, and lower drainage pH associated with the rock samples. Drainage pH from the Partridge River rock tended to be about 0.7 units higher than that from previously tested South Kawishiwi rock of similar sulfur content. This was attributed to the higher heavy metal content (and consequent lower iron sulfide content) of the Partridge River rock examined in the present study.

Additional work should be conducted to further benefit the application of results generated to date. This includes continued dissolution testing to confidently define long-term trends, analysis of leached solids to increase understanding of chemical reactions controlling solute release (e.g. chemical precipitation and adsorption), and more detailed analysis of the present data, including calculation of mineral dissolution rates. The more detailed analysis should include calculation of mineral dissolution rates to allow application of the data generated to a wider range of mineralogical compositions.

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Table 1. Dunka Road drill hole data obtained from PolyMet (16 August 2001) for samples selected for laboratory dissolution tests.

DNR Sample #	Drill Hole	Footage		Lerch Total %S	Pmet Total %S[1]	Cu ppm	Grain Size[2]	% Plag	% Olivine	% Pyroxene	Rock Unit
12046 [3]	99318	60	65	0.03	0.15	290	m-vcg	50	40	10	3
	99318	55	60	1.30	0.93	1835	mg	55	45	0	3
12049 [3]	00340	165	170	0.25	0.31	391	m-fg	80	20	trace	3
	00340	55	60	0.36	0.33	539	f-cg	50	50	0	3
12051	00366	110	115	0.23	0.45	1365	m-fg	77	20	3	3

[1] Total %S reported for drill core by PolyMet were included for comparison with analyses of –6.25-mm samples by Lerch Brothers.

[2] f = fine, m = medium, c = coarse, vc = very coarse, g = grain

[3] Drill hole footages were mixed to produce samples 12046 and 12049 with sulfur contents of 0.61 and 0.31 percent, respectively.

Table 2. Particle size distribution data for the d < 6.35 mm samples (percent passing). Analyses by Lerch Brothers Inc.

Size Fraction		Sample / Cell #									
(mesh)	(microns)	12037/9	12040/10	12041/11	12042/12	12033/14	12112/15	12110/16	12051/17	12046/18	12049/19
%S		0.13	0.21	0.33	0.55	0.72	1.03	1.36	0.23 (D)	0.31 (D)	0.61 (D)
10	2000	42.26	39.90	35.75	40.78	36.83	56.99	53.00	28.67	31.07	32.97
20	841	27.01	24.75	22.06	26.22	23.02	35.23	30.25	18.36	19.68	20.64
35	500	17.72	16.27	14.79	17.77	15.09	23.28	19.18	12.74	13.21	13.88
48	297	14.37	13.22	11.81	14.42	12.42	18.69	15.86	10.41	11.01	11.50
65	210	11.32	10.37	9.44	11.47	9.82	14.52	12.33	8.24	8.78	9.49
100	149	8.83	7.84	7.13	8.61	7.57	10.90	9.26	6.17	6.74	7.29
150	105	6.53	5.79	5.50	6.74	5.59	7.82	6.99	4.76	5.02	5.46
200	74	4.80	4.29	3.93	5.00	4.05	5.65	5.20	3.43	3.80	3.81
270	53	3.36	2.89	2.77	3.63	2.74	3.96	3.76	2.41	2.57	2.49

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Note: All samples had 100% passing ¼ inch.

Table 3. Total sulfur analyses. Analyses by Lerch Brothers Inc.

Sample #	Reactor	%S
75 g, 0.053 < d < 0.149mm rock		
12037	1	0.07
12040	2	0.11
12041	3	0.20
12042	4	0.31
12042	5 (dup. of 4)	0.31
12033	6	0.63
12110	7	0.94
12046	8 (D)	0.31
1000 g, d < 6.35 mm rock		
12037	9	0.13
12040	10	0.21
12041	11	0.33
12042	12	0.55
12042	13 (dup. of 12)	0.55
12033	14	0.72
12112	15	1.03
12110	16	1.36
12051	17 (D)	0.23
12046	18 (D)	0.31
12049	19 (D)	0.61

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 4. Whole rock chemistry (%) for Duluth Complex waste rock reactors (0.053<d<0.149 mm) and humidity cells (d<6.35 mm). Analysis performed by Lerch Brothers Inc. and ALS USA Inc.

Reactor	1	2	3	4	6	7	8 (D)
%S	0.07	0.11	0.2	0.31	0.63	0.94	0.31
CO_2	0.08	0.13	0.12	0.2	0.2	0.07	0.08
SiO_2	45.1	44.5	44.9	43.9	43.3	43.7	46
Al_2O_3	17.35	16.3	17.35	14.75	13.7	16.1	19.9
Fe_2O_3	14.15	14.4	14.35	14.25	15.8	16.25	11.2
CaO	8.29	8.35	8.53	7.28	6.58	8.18	9.5
MgO	10.35	8.66	8.12	9.59	9.67	8.09	7.53
Na_2O	2.3	2.46	2.51	2.25	2.25	2.38	2.56
K_2O	0.39	0.55	0.52	0.52	0.57	0.54	0.31
Cr_2O_3	0.02	0.03	0.03	0.02	0.02	0.02	0.04
TiO_2	0.95	2.4	1.83	1.91	1.78	2.01	0.93
MnO	0.16	0.16	0.16	0.16	0.17	0.17	0.13
P_2O_5	0.08	0.14	0.14	0.18	0.22	0.18	0.08
SrO	0.03	0.03	0.03	0.03	0.03	0.03	0.03
BaO	0.01	0.02	0.02	0.02	0.02	0.02	0.01

HC	9	10	11	12	14	15	16	17 (D)	18 (D)	19 (D)
%S	0.13	0.21	0.33	0.55	0.72	1.03	1.36	0.23	0.31	0.61
CO_2	0.05	0.08	0.08	0.08	0.08	0.06	0.1	0.08	0.07	0.21
SiO_2	45	45.4	44.8	44.2	43.7	44.3	43.5	45.6	47.4	46.2
Al_2O_3	18.15	17.3	18	16.65	14.95	17.8	16.05	19.3	20.7	18.8
Fe_2O_3	12.6	13.4	13.2	13.7	16.5	14.8	16.45	12.4	11.15	13.4
CaO	8.6	8.66	8.68	8.09	7.25	8.75	8.36	9.25	10.25	9.65
MgO	9.29	8.14	7.42	9.24	10.05	6.97	7.84	9.16	7.31	7.96
Na_2O	2.4	2.58	2.57	2.37	2.26	2.52	2.33	2.34	2.6	2.49
K_2O	0.42	0.62	0.58	0.63	0.65	0.53	0.56	0.28	0.34	0.39
Cr_2O_3	0.02	0.02	0.02	0.01	0.02	0.04	0.04	0.03	0.04	0.05
TiO_2	0.96	2.25	1.69	1.84	1.85	1.91	2.11	0.61	0.85	1.62
MnO	0.14	0.15	0.15	0.15	0.17	0.15	0.16	0.14	0.12	0.14
P_2O_5	0.13	0.18	0.17	0.2	0.22	0.16	0.2	0.09	0.11	0.11
SrO	0.04	0.03	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03
BaO	0.01	0.02	0.02	0.02	0.02	0.02	0.02	0.01	0.01	0.01

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 5. Heavy metal composition (%) for Duluth Complex waste rock reactors (0.053<d<0.149 mm) and humidity cells (d<6.35 mm). Analysis performed by ALS USA Inc.

Reactor	1	2	3	4	6	7	8 (D)
%S	0.07	0.11	0.2	0.31	0.63	0.94	0.31
Ag	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	0.00012	<0.00005
As	0.0009	<0.0005	<0.0005	0.0007	0.0011	0.0008	0.0006
Cd	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005
Co	0.0079	0.0069	0.0071	0.0081	0.0095	0.0096	0.0063
Cu	0.0381	0.0686	0.1	0.1835	0.269	0.529	0.0335
Mo	<0.0001	0.0001	0.0001	<0.0001	<0.0001	<0.0001	0.0001
Ni	0.0416	0.0386	0.0419	0.0626	0.0868	0.1055	0.0293
Pb	<0.0002	<0.0002	<0.0002	0.0003	0.0004	0.0004	0.0004
Sc	0.001	0.001	0.0014	0.0014	0.0013	0.0016	0.0009
Zn	0.0101	0.0101	0.0108	0.0109	0.0122	0.0121	0.0077

HC	9	10	11	12	14	15	16	17 (D)	18 (D)	19 (D)
%S	0.13	0.21	0.33	0.55	0.72	1.03	1.36	0.23	0.31	0.61
Ag	<0.00005	<0.00005	<0.00005	<0.00005	0.00006	0.00011	0.00017	<0.00005	<0.00005	<0.00005
As	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	0.0018	<0.0005	0.0005	<0.0005
Cd	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005
Co	0.0071	0.0067	0.0069	0.0078	0.0101	0.0098	0.0101	0.007	0.0069	0.0083
Cu	0.0779	0.1175	0.157	0.281	0.393	0.464	0.648	0.0899	0.0478	0.0763
Mo	<0.0001	0.0001	<0.0001	<0.0001	0.0001	0.0001	0.0001	<0.0001	<0.0001	<0.0001
Ni	0.0436	0.0426	0.0456	0.0612	0.0986	0.1095	0.125	0.0477	0.04	0.0513
Pb	<0.0002	0.0003	0.0002	0.0002	0.0006	0.0004	0.0005	<0.0002	0.0002	0.0002
Sc	0.0009	0.0017	0.0012	0.0013	0.0014	0.0013	0.0016	0.0006	0.0008	0.0014
Zn	0.0095	0.0104	0.0106	0.0104	0.013	0.0108	0.0118	0.008	0.0078	0.0091

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 6. Chemical composition (%) comparison for Babbitt and Dunka Road prospect samples

%S	CaO	MgO	Na_2O	K_2O	Cu	Ni	Co	Zn
Fine rock (75 g, 0.053<d<0.149 mm): Reactors 4, 5 vs 8								
0.31[1] (B)	7.28	9.59	2.25	0.52	0.184	0.0626	0.0081	0.0109
0.31 (D)	9.50	7.53	2.57	0.31	0.0335	0.0293	0.0063	0.0077
B/D	0.77	1.27	0.88	1.68	5.49	2.14	1.29	1.42
Coarse rock (1000g, d<6.35 mm): H. Cell 10 vs 17								
0.21 (B)	8.66	8.14	2.58	0.62	0.118	0.0426	0.0067	0.0104
0.23 (D)	9.25	9.16	2.34	0.28	0.0899	0.0477	0.007	0.008
B/D	0.94	0.89	1.10	2.21	1.31	0.89	0.96	1.3
Coarse rock (1000g, d<6.35 mm): H. Cell 11 vs 18								
0.33 (B)	8.68	7.42	2.57	0.58	0.157	0.0456	0.0069	0.0106
0.31 (D)	10.25	7.31	2.6	0.34	0.0478	0.04	0.0069	0.0078
B/D	0.85	1.02	0.99	1.71	3.28	1.14	1.00	1.36
Coarse rock (1000g, d<6.35 mm): H. Cells 12, 13 vs 19								
0.55[1] (B)	8.09	9.24	2.37	0.63	0.281	0.0612	0.0078	0.0104
0.61 (D)	9.65	7.96	2.49	0.39	0.0763	0.0513	0.0083	0.0091
B/D	0.84	1.16	0.95	1.62	3.68	1.19	0.94	1.14

B, D indicate Babbitt prospect and Dunka Road prospect, respectively
[1] Average values for duplicates

Table 7. Mineral content and chemistry comparison for Babbitt and Dunka Road prospect samples.

75 g, 0.053<d<0.149 mm rock, 0.31 %S.

Mineral	Babbitt Prospect, 0.31 %S		Dunka Road Prospect, 0.31 %S	
	Vol %	Formula	Vol %	Formula
Plagioclase	40	$Ca_{0.56}Na_{0.39}$	57	$Ca_{0.63}Na_{0.38}$
Olivine	30	$Mg_{1.25}Fe_{0.75}$	28	$Mg_{1.21}Fe_{0.80}$
Augite	14	$Ca_{0.78}Mg_{0.74}$	3.9	$Ca_{0.86}Mg_{0.82}$
Hypersthene	3.1	$Mg_{1.27}$	2.4	$Mg_{1.26}$
Biotite	2.3	$Mg_{3.95}K_{1.7}$	0.0	
K-spar	2.3	$K_{0.87}Na_{0.085}$	1.6	$K_{0.80}Na_{0.11}$

1000 g, d<6.35 mm rock, ~0.2 %S.

Mineral	Babbitt Prospect, 0.21 %S		Dunka Road Prospect, 0.23 %S	
	Vol %	Formula	Vol %	Formula
Plagioclase	55	$Ca_{0.55}Na_{0.41}$	53	$Ca_{0.61}Na_{0.36}$
Olivine	15	$Mg_{1.19}Fe_{0.80}$	42	$Mg_{1.23}Fe_{0.76}$
Augite	7.2	$Ca_{0.81}Mg_{0.75}$	1.7	$Ca_{0.80}Mg_{0.79}$
Hypersthene	3.6	$Mg_{1.08}$	0.8	$Mg_{1.37}$
Biotite	4.5	$Mg_{3.68}K_{1.75}$	0.8	$Mg_{3.75}K_{1.72}$
K-spar	1.8	$K_{0.81}Na_{0.16}$	0.0	

1000g, d<6.35 mm rock, ~0.3 %S.

Mineral	Babbitt Prospect, 0.33 %S		Dunka Road Prospect, 0.31 %S	
	Vol %	Formula	Vol %	Formula
Plagioclase	46	$Ca_{0.55}Na_{0.41}$	60	$Ca_{0.60}Na_{0.37}$
Olivine	28	$Mg_{1.11}Fe_{0.87}$	23	$Mg_{1.22}Fe_{0.77}$
Augite	8.0	$Ca_{0.80}Mg_{0.78}$	3.4	$Ca_{0.86}Mg_{0.78}$
Hypersthene	1.8	$Mg_{1.17}$	2.5	$Mg_{1.23}$
Biotite	2.7	$Mg_{3.54}K_{1.71}$	0.0	
K-spar	1.8	$K_{0.78}Na_{0.17}$	1.7	$K_{0.92}Na_{0.081}$

1000g, d<6.35 mm rock, ~0.6 %S.

Mineral	Babbitt Prospect, 0.55 %S		Dunka Road Prospect, 0.61 %S	
	Vol %	Formula	Vol %	Formula
Plagioclase	49	$Ca_{0.58}Na_{0.39}$	48	$Ca_{0.57}Na_{0.38}$
Olivine	20	$Mg_{1.27}Fe_{0.73}$	27	$Mg_{1.16}Fe_{0.84}$
Augite	3.6	$Ca_{0.81}Mg_{0.78}$	5.0	$Ca_{0.78}Mg_{0.76}$
Hypersthene	7.1	$Mg_{1.18}$	2.5	$Mg_{1.09}$
Biotite	1.8	$Mg_{3.96}K_{1.68}$	0.0	
K-spar	2.7	$K_{0.74}Na_{0.22}$	0.8	$K_{0.82}Na_{0.16}$

Table 8. Water retention (g) for the Duluth Complex waste rock samples for weeks 1-400.

Reactor	% S	Water retention before rinse[1]					Water retention after rinse				
		count	min	max	mean	SD	count	min	max	mean	SD
75-g Reactors											
1	0.07	389	-4.9	4.6	-2.0	1.6	399	8.8	38.0	25.0	5.6
2	0.11	396	-7.9	4.2	-2.6	2.6	397	7.6	33.1	22.2	5.4
3	0.20	396	-8.3	1.1	-2.6	2.5	395	5.7	31.0	17.8	5.7
4	0.31	398	-10.9	5.5	-3.0	3.4	398	2.3	36.6	16.7	7.5
5	0.31	399	-11.1	0.3	-6.0	4.0	399	-0.4	29.6	13.8	7.0
6	0.63	398	-3.0	5.7	-0.8	1.3	397	16.9	36.1	27.0	3.3
7	0.94	396	-7.9	9.1	-1.8	2.9	394	7.4	37.8	24.3	6.4
8 (D)	0.31	395	-3.1	2.6	-1.3	1.4	396	14.7	34.8	23.9	4.1
1000-g Cells											
9	0.13	395	90	133	108.1	8.0	397	92	150	117.1	10.0
10	0.21	399	83	126	106.0	6.7	398	88	136	114.8	7.1
11	0.33	398	82	124	101.2	6.9	397	96	147	111.8	6.6
12	0.55	399	96	126	107.9	5.1	400	101	138	119.4	5.9
13	0.55	399	74	110	93.8	5.6	400	80	115	99.7	5.6
14	0.72	397	98	119	108.8	3.4	398	103	132	118.1	4.4
15	1.03	397	133	164	151.3	4.9	399	136	176	166.4	6.5
16	1.36	399	135	161	148.4	4.5	400	140	178	163.4	6.4
17 (D)	0.23	398	79	123	99.1	6.4	399	86	161	108.3	7.6
18 (D)	0.31	398	71	111	88.9	4.2	399	84	119	99.3	4.4
19 (D)	0.61	399	105	135	119.1	5.1	400	107	145	125.7	5.8

[1] To calculate water retention the original mass of the reactor and dry solids (at the beginning of the experiment) was subtracted from the mass before the rinse. Negative values indicate that some solids were lost during the course of the experiment.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 1 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 1 0.07 %S	1	6.48	65.6	26.2	64.9	18.8	41.6	8.59	17	4.35	47.1	18.6
	2	6.42	68.9	21.8	38.6	10.3	37.8	14.1	14.4	3.32	11.4	6.1
	3	6.42	32.9	6.61	23.1	3.1	26.1	2.21	10.5	2.98	2.73*	0.0168
	4	6.41	22.1	3.96	15.9	3.55	20.8	5.78	8.36	4.23	2.73*	0.0239
	5	6.58	14	1.92	12.3	1.42	22.4	1.41	4.66*	0.0633	5.43	6.03
	6	6.65	7.58*	5.26	17	5.31	22.4	4.32	6.69	5.32	4.25	2.93
	7	6.42	6.86*	3.15	18.6	5.14	23.4	4.7	5.68	2.25	2.79*	0.0303
	8[2]	6.48	7.63*	3.82	12.8	3.46	21.1	2.5	4.79*	0.0504	2.82*	0.0268
	Total[3]	6.42	28.2	9.37	25.4	5.68	26.9	4.19	9.02	1.94	9.9	6.38
Reactor 2 0.11 %S	1	6.38	145	80.2	106	21.7	73.7	29.8	19.3	4.78	52.8	22.5
	2	6.23	91.7	31.2	40.5	12.6	50.4	14.6	15.9	2.72	16.8	8.04
	3	6.45	44.6	8.33	25.3	3.49	35.3	4.11	12.4	1.46	3.45	1.34
	4	6.34	29.1	3.53	15.4	3.75	24.1	3.99	11.5	4.22	5.46	3.09
	5	6.53	21.1	2.85	11.9	1.62	26.1	1.93	4.82*	0.0324	4.98	3.09
	6	6.49	16.7	3.3	13.8	3.32	26.6	2.64	6.17	2.33	5.81	4.39
	7	6.36	11.3	5.28	19.8	8.6	29.3	8.45	6.64	3.06	6.67	4.71
	8[2]	6.43	10.1	4.94	12.1	5.07	23.7	4.5	4.86*	0.0228	2.86*	0.0134
	Total[3]	6.35	46.2	27.1	30.5	6.73	36.2	9.45	10.2	1.76	12.4	7.11
Reactor 3 0.20 %S	1	6.06	245	129	145	53.1	88.9	39.6	32.1	8.56	96.7	47.9
	2	5.75	133	31.6	45.8	11.3	34.2	12.6	27.8	7.93	20.3	11.8
	3	5.68	72.6	11.5	26.8	3.01	20	3.05	15.8	3.57	3.81	1.49
	4	5.71	50	3.76	19.6	3.98	14.7	2.54	10.3	4.34	2.8*	0.129
	5	5.83	34.6	6.05	10.7	0.646	12.2	1.34	4.92*	0.0301	2.89*	0.0189
	6	5.86	25.5	3.04	10.4	3.96	9.7	3.73	5.83	2.59	3.42*	1.5
	7	5.79	21	3.92	13.8	4.18	9.1	3.53	7.81	5.88	2.87*	0.0302
	8[2]	5.98	16.5	2.47	6.46	0.394	4.68*	0.0801	10.3	10.7	2.91	2.91
	Total[3]	5.73	74.8	43.6	34.8	17.7	24.2	13.2	14.6	3.51	17	17
Reactor 4 0.31 %S	1	6.34	256	189	179	63	114	69.6	27.4	9.01	78.8	39.2
	2	5.84	183	37.8	48.6	13.4	91.1	17.5	23.1	4.21	18.6	10.7
	3	6.09	102	13.1	26.5	4.48	67.4	8.4	14.8	2.86	4.46	1.99
	4	6.07	68.9	11.8	17.4	2.31	51.2	7.86	8.66	4.36	2.88*	0.052
	5	6.21	45.8	7.16	10.3	2.09	45.9	5.67	4.91*	0.0297	2.9*	0.0366
	6	6.28	30.8	5.74	10.5	3.11	36.2	4.25	5.95*	2.64	3.5*	1.54
	7	6.23	25.4	5.62	11.9	6.28	36.4	8	8.2	7.89	2.95*	0.0426
	8[2]	6.29	19.8	3.04	6.35	1.19	27.3	4.52	8.54	7.03	2.95*	0.00858
	Total[3]	6.07	91.4	63.6	38.8	21	58.7	22.2	12.7	3.02	14.6	13.6

[1] 5th percentile value
[2] Year 8 represented by weeks 365-400
[3] Average and standard deviation for entire period of record
* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.
Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 2 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 5 0.31 %S (dup. of 4)	1	6.30	238	178	172	54.2	107	62.6	25.1	7.26	70.2	30.4
	2	5.93	163	30.4	43.9	14	81.3	17.4	22.3	1.93	15	7.69
	3	6.16	96.9	12.4	24.6	2.94	61.1	8.75	17.5	2.14	11.4	3.8
	4	6.09	62.7	13.2	17.1	4.08	44.3	10.2	11.5	4.48	7.59	3.9
	5	6.34	38.7	6.8	9.68	0.837	38.5	4.76	6.93	4.38	3.73	1.8
	6	6.28	24.1	2.93	9.34	2.51	33.7	2.7	4.92*	0.0316	2.89*	0.0186
	7	6.34	21.2	7.17	11.9	5.18	41.7	6.8	12.3	9.29	2.9*	0.0199
	8[2]	6.38	19.1	2.79	9.35	9.24	31.8	6.54	4.94*	0.0288	2.9*	0.0169
	Total[3]	6.10	83	59.6	37.3	17.7	55	19.7	13.2	3.33	14.6	10.2
Reactor 6 0.63 %S	1	5.96	322	278	138	73.2	151	110	24.2	9.91	65.5	24.3
	2	5.37	320	54.7	69.3	14.9	133	29.7	38.7	2.3	34.8	6.72
	3	5.17	232	35.3	47.8	6.14	89.5	15.8	30.3	4.64	17.7	6.64
	4	5.19	159	21.4	29.7	4.23	65	13.1	22	2.87	21.8	3.4
	5	5.31	130	19.7	19.4	2.11	64.6	9.92	9.32	5.95	9.89	10.2
	6	5.40	90.2	10.3	15.2	6.08	45.6	18.7	10.5	4.12	4.94	3.46
	7	5.69	74	9.86	16.6	3.85	45.9	4.61	17.7	3.57	4.35	2.59
	8[2]	5.67	71.5	10.7	14.6	4.28	42.6	5.12	23.5	9.17	5.43	5.28
	Total[3]	5.19	175	91.3	43.9	24.1	79.8	34.9	22	2.84	20.6	7.1
Reactor 7 0.94 %S	1	4.53	530	263	191	65.3	93.7	47.7	49.9	26.6	81.4	32.3
	2	4.42	452	44.9	83.8	16.3	45.2	11.6	56.4	6.33	35.2	19.2
	3	4.50	314	34	54.3	7.9	28	4.14	38.3	6.06	12.2	8.14
	4	4.55	239	26.8	39.7	3.98	21.3	4.21	30.1	4.78	9.42	2.22
	5	4.55	193	26.6	26.1	5.03	18.2	2.69	17.6	5.12	2.8*	0.0214
	6	4.60	154	15.9	28.7	6.65	18.3	2.2	17.1	2.2	6.17	6.15
	7	4.49	140	19.5	26.7	6.91	15	6.5	25.9	6.67	5.13	4.49
	8[2]	4.69	125	5.67	22.9	3.02	15.8	1.21	24.6	9.76	2.81*	0.0119
	Total[3]	4.47	268	85	59.1	21	31.9	15.6	32.5	7.62	19.4	11.2
Reactor 8(D) 0.31 %S	1	6.34	156	78.7	127	32.3	57.3	30.7	14.3	3.71	32.5	14.8
	2	5.89	135	36.3	48.5	18.3	74.8	16.7	12.9	3.2	7.63	4.16
	3	6.19	69	24.2	22.6	5.19	51.9	12.9	7.76	2.94	3.14*	1.1
	4	6.15	42.7	6.9	15.5	2.86	35.2	6.45	4.73*	0.0521	2.78*	0.0307
	5	6.28	24.8	4.56	8.57	0.659	30.3	2.57	4.78*	0.0223	5.29	4.14
	6	6.32	19.8	4.92	11.1	3.17	29.6	4.78	5.44*	1.79	2.8*	0.023
	7	6.23	15.1	5.55	14.2	6.21	29	6.35	4.76*	0.068	3.5*	1.82
	8[2]	6.18	13.7	6.27	8.55	0.492	25.4	3.85	4.8*	0.0358	2.82*	0.0211
	Total[3]	6.05	59.5	26.1	32	11.1	41.7	9.45	7.44	1.62	7.55	4.99

[1] 5th percentile value
[2] Year 8 represented by weeks 365-400
[3] Average and standard deviation for entire period of record
* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.
Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 3 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 9 0.13 %S	1	6.92	32.9	6.46	34.5	3.83	29.4	11.3	7.67	5.21	7.97	2.15
	2	6.76	32	3.91	20.8	3.83	17.4	1.7	3.02	0.486	5.76	0.39
	3	6.54	27.1	2.82	14.6	2.17	17.2	2.31	2.74	0.424	5.61	0.522
	4	6.47	22.1	2.12	10.3	1.38	14.7	1.35	1.68	0.701	4.3	0.33
	5	6.49	18.5	1.6	7.09	0.587	14.7	0.773	1.06*	0.0124	3.97	0.911
	6	6.44	14.6	1.07	6.11	0.798	13.2	0.558	1.27*	0.556	3.61	0.524
	7	6.37	14.4	2.11	6.67	1.09	14.1	1.85	1.66	0.792	4.21	0.509
	8[2]	6.30	13.2	0.935	4.69	0.296	12.9	0.888	1.06*	0.00281	3.69	0.398
	Total[3]	6.42	21.8	1.82	13.1	1.4	16.7	3.57	2.52	1.72	4.89	0.607
Humidity cell 10 0.21 %S	1	6.92	67.6	12.9	54	11.3	38.4	9.96	8.43	4.44	8.94	1.41
	2	6.61	50.3	5.73	27.1	5.68	23.5	2.94	3.76	0.778	7.38	0.818
	3	6.39	43.2	5.95	19.4	1.95	22.9	2.93	3.99	0.616	6.83	0.537
	4	6.27	33.6	4.15	12.8	1.87	18.3	2.36	3.78	0.534	5.16	0.514
	5	6.27	27.8	2.11	9.17	0.868	16.9	1.06	2.04	0.916	3.99	0.64
	6	6.19	21.6	1.16	7.94	0.845	14.9	0.907	2.64	0.571	3.92	0.496
	7	6.12	19.8	3.13	7.71	0.939	14.7	1.47	2.42	0.742	4.22	0.366
	8[2]	6.16	20.3	2.66	6.25	0.715	15	1.51	1.39	0.65	4.35	0.377
	Total[3]	6.18	35.5	3.7	18.1	3.73	20.6	2.96	3.56	1.33	5.6	0.342
Humidity cell 11 0.33 %S	1	7.00	50.9	8.86	42	3.75	31.8	10.9	8.66	5.61	11.9	2.15
	2	6.44	52.7	2.66	28.3	4.05	21.7	1.4	3.51	0.92	9.81	0.788
	3	5.99	54.7	7.03	23.1	3.05	23.7	3.59	5.06	1.51	10.1	0.932
	4	5.69	48.4	3.66	16.4	2.19	19.9	2.34	5.85	0.43	7.74	0.828
	5	5.57	46.3	4.98	13.7	1.32	19.4	1.62	5.15	0.497	5.99	0.705
	6	5.45	37	3.09	12.1	1.82	16.1	1.96	4.8	0.805	5.56	1.24
	7	5.28	34.9	5.53	11.9	2	15.7	2.12	5.98	1.57	5.29	0.981
	8[2]	4.88	35	4.22	10.1	0.775	14.4	1.25	8.5	2.47	4.57	0.51
	Total[3]	5.31	45	2.1	19.7	1.15	20.3	3.2	5.94	1.71	7.63	0.504
Humidity cell 12 0.55 %S	1	7.06	71.8	9.31	59.8	5.62	40.4	12.8	9.09	5.6	10.6	1.91
	2	6.54	68.8	8.25	42.9	7.92	28.4	2.53	3.77	0.625	8.18	0.518
	3	6.27	68.5	7.99	29.7	3.72	32.5	5.76	4.76	1.12	7.84	0.695
	4	6.02	63.8	3.56	20.9	2.39	32.2	1.67	5.04	0.367	6.21	0.356
	5	5.87	59.1	7.33	13.9	1.98	32.5	3.44	3.32	0.926	5.01	0.899
	6	5.76	50.3	3.84	11.5	1.1	30.2	2.42	3.62	0.323	5.13	0.83
	7	5.67	47.1	5.15	10.3	1.43	30.9	3.53	4.33	1.34	5.03	0.497
	8[2]	5.47	43.4	3.73	8.22	0.954	28.3	1.75	3.92	1.98	5.13	0.69
	Total[3]	5.70	59.1	2.33	24.7	2.48	31.9	3.71	4.73	1.73	6.64	0.484

[1] 5th percentile value
[2] Year 8 represented by weeks 365-400
[3] Average and standard deviation for entire period of record
* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.
Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 4 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 13 0.55 %S	1	6.92	72	12.2	59.9	7.85	39.9	14.3	10	6.69	10.2	2.32
	2	6.64	61.6	5.9	38.4	5.94	25.7	1.93	3.58	0.627	7.73	0.82
	3	6.34	62.6	10.9	27.5	4.33	30.6	6.52	4.35	1.23	7.26	0.911
	4	6.04	57.3	4.83	18.7	2.72	29.1	2.93	4.47	0.419	5.67	0.464
	5	5.90	57.1	5.69	13.7	2.09	31.8	3.24	2.83	1.23	4.78	0.885
	6	5.85	44.7	3.7	10.4	1	27.7	3.46	3.18	0.298	4.48	0.574
	7	5.62	40.9	6.22	9.31	1.57	26.6	3.71	3.05	1.29	4.63	0.39
	8[2]	5.66	40.2	3.6	7.4	0.582	26.4	1.52	3.49	1.76	4.38	0.254
	Total[3]	5.75	54.5	3.2	23.2	2.57	29.7	4.15	4.37	2.08	6.14	0.648
Humidity cell 14 0.72 %S	1	6.69	103	12.7	53.2	7.64	67.5	13.7	8.41	6.3	12.9	4.06
	2	6.11	113	6.33	43.5	4.71	59	3.43	4.58	0.72	10.5	0.57
	3	5.90	103	11.2	29.4	4.14	57.5	8.43	6.79	1.29	9.39	0.91
	4	5.67	89.9	7.36	21.2	2.98	50.1	5.23	6.47	0.519	6.91	0.675
	5	5.61	87.7	7.4	17	1.9	48.8	5.26	5.91	1.05	5.53	0.824
	6	5.55	70.7	3.82	14.4	1.43	41.3	3.14	6.1	0.883	4.93	0.65
	7	5.37	66.9	9.24	12.9	2.02	40	5.53	6.25	1.45	5.13	0.518
	8[2]	5.23	66.9	5.08	12	0.494	39.6	2.47	8.11	2.37	4.7	0.286
	Total[3]	5.43	87.7	3.01	25.4	2.28	50.5	3.67	6.58	1.9	7.49	1.22
Humidity cell 15 1.03 %S	1	4.95	175	24.1	89.2	38.9	43.4	7.37	14.3	3.33	16	3.81
	2	4.50	136	16.2	40.7	3.67	39.6	5.51	20.5	2.18	9.13	1.16
	3	4.40	165	20.6	42	5.29	58.4	10	28.2	4.1	7.18	0.764
	4	4.16	180	15.1	39	2.96	58	7.38	29.2	3.99	5.71	0.864
	5	4.17	185	15.4	33.5	3.64	50.1	9.41	24	4.19	4.01	0.903
	6	4.18	157	8.63	29.6	2.39	48.9	7.72	19.7	3.68	5.01	2.11
	7	4.09	161	31.1	30.4	5.33	49.9	7.48	22.2	4.38	4.87	0.704
	8[2]	4.07	148	12.2	27.4	1.56	45	3.14	19.8	2.82	4.74	0.266
	Total[3]	4.17	164	7.12	41.5	12.6	49.2	2.15	22.2	0.76	7.08	1.14
Humidity cell 16 1.36 %S	1	4.91	242	24.1	121	50.8	67.1	13.6	17.2	4.52	21.8	4.8
	2	4.61	189	10.4	51.2	6.04	50	4.7	23.2	1.28	11.4	1.43
	3	4.46	188	15.6	46.9	4.51	52.1	6.69	27.1	3	8.28	0.869
	4	4.34	174	6.3	38.4	1.5	44.6	4.25	24	1.87	5.71	0.675
	5	4.28	177	10.1	35.1	2.92	37.8	4.99	22	1.99	4.08	0.692
	6	4.31	154	6.71	29.2	5.05	36.4	5.94	20.3	3.8	4.46	1.66
	7	4.19	173	41.8	34.5	7.95	40.2	8.24	25.3	5.24	5.21	0.885
	8[2]	4.13	144	12.5	28.9	2.05	34.6	2.96	20.5	3.16	5.07	1.4
	Total[3]	4.28	180	11.9	48.2	16.6	45.3	3.32	22.4	1.37	8.25	1.36

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 5 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 17(D) 0.23 %S	1	7.19	32	12.6	43.3	10.4	17.6	7.01	8.45	6.98	4.91	1.18
	2	6.97	13.2	1.64	21.1	4.09	8.7	0.801	1.63	0.62	3.51	0.351
	3	6.84	10.7	1.26	14.6	2.33	8.87	1.76	1.42	0.555	3.39	0.479
	4	6.64	8.5	0.472	9.39	1.21	7.11	1.18	1.04*	0.0147	2.48	0.203
	5	6.60	8.75	0.612	7.57	0.424	8.9	0.452	1.06*	0.0206	2.08	0.404
	6	6.47	7.27	0.703	6.86	0.668	8.28	0.34	1.06*	0.00937	2.53	0.824
	7	6.35	7.76	1.47	7.23	1.12	9.31	1.42	1.05*	0.0102	2.5	0.383
	8[2]	6.30	7.07	0.35	5.06	0.211	8.59	0.557	1.06*	0.0109	2.24	0.48
	Total[3]	6.40	11.9	4.14	14.4	3.4	9.66	2.21	2.1	2.42	2.96	0.314
Humidity cell 18(D) 0.31 %S	1	7.07	62.4	9.32	65.9	9.31	16.7	5.7	12.4	11.4	5.52	1.05
	2	6.44	52.2	4.83	37.7	7.04	15.7	2.23	3.24	0.447	4.75	0.438
	3	6.28	49	6.33	26.2	3.87	22.2	4.02	2.96	0.521	4.54	0.726
	4	6.10	40.1	5.09	16.6	2.59	20.7	2.58	1.94	0.898	3.31	0.359
	5	6.09	36	3.74	12.7	1.37	21.2	1.65	1.05*	0.02	2.57	0.586
	6	6.00	28.3	1.64	10.4	0.953	18.5	1.72	1.38	0.581	2.64	0.344
	7	5.83	30.5	10.7	11.8	3.25	20.9	6.34	1.78	0.958	2.86	0.648
	8[2]	5.53	27.6	3.98	8.58	0.865	18.5	1.87	1.36	0.625	2.56	0.521
	Total[3]	5.85	40.8	3	23.7	3.05	19.3	1.87	3.27	3.83	3.6	0.231
Humidity cell 19(D) 0.61 %S	1	7.00	92.6	12	85	18.4	26	5.98	20.4	22.1	5.52	1.42
	2	6.43	109	7.56	94.9	11.7	23.7	3.49	4.92	0.823	5.74	0.717
	3	6.14	105	11.7	67.7	11.2	32.1	5.15	6.83	1.41	5.72	1.47
	4	5.83	87.7	6.6	45.6	5.33	29.5	3.7	7.37	0.824	3.42	0.46
	5	5.64	78.1	11.1	35.9	4.2	28.2	2.18	6.07	0.861	2.02	0.569
	6	5.36	65.9	6.21	29.2	2.68	24.8	1.82	6.66	1.02	2.23	0.268
	7	5.06	71.9	8.44	31.8	4.61	30.3	3.65	9.97	1.93	1.85	1.02
	8[2]	4.96	68.9	6.32	26.9	1.77	29.2	1.86	8.66	1.46	1.75	0.188
	Total[3]	5.09	84.8	2.48	52.1	5.74	28	1.53	8.86	7.41	3.53	0.494

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 1 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 1 0.07 %S	1	6.48	65.6	26.2	3.92E-02	1.63E-02	7.66E-02	7.21E-02	3.60E-02*	3.61E-04	4.14E-02	2.53E-02
	2	6.42	68.9	21.8	3.35E-02*	3.59E-04	6.09E-01	6.77E-02	3.61E-02*	3.87E-04	1.02E-01	6.00E-02
	3	6.42	32.9	6.61	5.04E-02	3.40E-02	4.78E-01	1.02E-01	3.59E-02*	6.24E-04	1.34E-01	4.35E-02
	4	6.41	22.1	3.96	7.42E-02	5.42E-02	3.58E-01	1.21E-01	3.63E-02*	3.67E-04	1.27E-01	7.32E-02
	5	6.58	14	1.92	3.37E-02*	4.75E-04	2.19E-01	5.37E-02	3.63E-02*	5.12E-04	6.09E-02	4.02E-02
	6	6.65	7.58*	5.26	4.93E-02	2.71E-02	1.25E-01	2.82E-02	3.63E-02*	1.07E-04	8.82E-02	5.01E-02
	7	6.42	6.86*	3.15	3.43E-02*	4.04E-04	6.39E-02	3.71E-02	3.70E-02*	4.36E-04	4.73E-02*	3.14E-02
	8[2]	6.48	7.63*	3.82	6.18E-02	3.12E-02	1.60E-01	1.46E-02	3.74E-02*	3.93E-04	6.66E-02	5.67E-02
	Total[3]	6.42	28.2	9.37	4.70E-02	1.97E-02	2.61E-01	3.65E-02	3.64E-02	1.48E-04	8.34E-02	1.57E-02
Reactor 2 0.11 %S	1	6.38	145	80.2	4.66E-02	2.10E-02	9.91E-01	1.84E+00	7.18E-02	7.54E-02	9.34E-02	1.05E-01
	2	6.23	91.7	31.2	1.07E-01	1.70E-02	2.66E+00	7.82E-01	1.72E-01	3.69E-02	2.44E-01	1.59E-01
	3	6.45	44.6	8.33	1.16E-01	2.10E-02	1.21E+00	3.30E-01	9.39E-02	2.56E-02	1.72E-01	6.80E-02
	4	6.34	29.1	3.53	1.08E-01	1.07E-01	4.73E-01	3.25E-01	4.78E-02	2.02E-02	1.89E-01	5.97E-02
	5	6.53	21.1	2.85	7.18E-02	5.26E-02	3.84E-01	1.24E-01	3.76E-02*	4.16E-04	8.31E-02	2.25E-02
	6	6.49	16.7	3.3	1.55E-01	1.04E-01	3.84E-01	6.41E-02	3.74E-02*	6.79E-05	1.78E-01	1.53E-02
	7	6.36	11.3	5.28	2.74E-01	1.46E-01	4.31E-01	1.81E-01	3.78E-02*	2.56E-04	3.01E-01	2.32E-01
	8[2]	6.43	10.1	4.94	2.41E-01	8.24E-02	2.91E-01	4.47E-02	3.79E-02*	1.78E-04	9.48E-02	8.24E-02
	Total[3]	6.35	46.2	27.1	1.40E-01	4.83E-02	8.54E-01	6.05E-01	6.70E-02	2.66E-02	1.69E-01	7.25E-02
Reactor 3 0.20 %S	1	6.06	245	129	4.54E-01	7.60E-01	7.38E+00	1.22E+01	5.80E-01	9.27E-01	4.48E-01	6.76E-01
	2	5.75	133	31.6	1.21E+01	2.76E+00	1.32E+01	3.09E+00	8.98E-01	2.58E-01	8.51E-01	3.97E-01
	3	5.68	72.6	11.5	1.30E+01	1.73E+00	6.75E+00	1.90E+00	4.15E-01	1.25E-01	4.56E-01	8.63E-02
	4	5.71	50	3.76	8.82E+00	6.98E-01	3.10E+00	5.54E-01	1.86E-01	3.23E-02	1.96E-01	6.49E-02
	5	5.83	34.6	6.05	7.41E+00	1.39E+00	1.54E+00	5.08E-01	9.37E-02	1.81E-02	4.17E-01	8.89E-02
	6	5.86	25.5	3.04	5.63E+00	5.23E-01	8.06E-01	8.88E-02	3.78E-02*	1.83E-04	3.04E-01	3.06E-02
	7	5.79	21	3.92	5.25E+00	7.50E-01	6.79E-01	6.16E-02	3.82E-02*	4.01E-04	1.38E-01	1.51E-01
	8[2]	5.98	16.5	2.47	3.46E+00	1.76E+00	4.13E-01	9.59E-02	3.86E-02*	6.61E-04	1.48E-01	2.27E-01
	Total[3]	5.73	74.8	43.6	7.02E+00	7.64E-01	4.24E+00	4.15E+00	2.86E-01	3.19E-01	3.70E-01	2.20E-01
Reactor 4 0.31 %S	1	6.34	256	189	4.32E-01	8.32E-01	1.43E+01	2.47E+01	7.89E-01	1.24E+00	4.60E-01	7.40E-01
	2	5.84	183	37.8	5.28E+00	6.92E-01	1.86E+01	4.75E+00	1.11E+00	2.72E-01	8.56E-01	2.65E-01
	3	6.09	102	13.1	6.00E+00	9.08E-01	8.96E+00	2.33E+00	5.51E-01	1.41E-01	4.97E-01	1.49E-01
	4	6.07	68.9	11.8	4.45E+00	1.17E+00	3.88E+00	8.63E-01	2.59E-01	4.90E-02	2.79E-01	1.45E-01
	5	6.21	45.8	7.16	1.91E+00	2.28E+00	1.55E+00	1.22E+00	1.09E-01	9.86E-02	1.87E-01	2.65E-02
	6	6.28	30.8	5.74	1.05E+00	1.02E+00	9.24E-01	1.32E-01	1.03E-01	9.22E-03	2.70E-01	4.17E-02
	7	6.23	25.4	5.62	1.51E+00	1.60E+00	6.77E-01	1.08E-01	5.70E-02	2.43E-02	3.89E-01	1.12E-01
	8[2]	6.29	19.8	3.04	7.74E-01	4.61E-01	4.55E-01	8.68E-02	3.92E-02*	1.14E-04	2.40E-01	8.92E-02
	Total[3]	6.07	91.4	63.6	2.67E+00	5.77E-01	6.18E+00	8.39E+00	3.78E-01	4.18E-01	3.97E-01	2.32E-01

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 2 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 5 0.31 %S (dup. of 4)	1	6.30	238	178	3.85E-01	8.03E-01	1.25E+01	2.39E+01	6.50E-01	1.25E+00	3.68E-01	6.31E-01
	2	5.93	163	30.4	3.17E+00	1.40E+00	1.61E+01	3.89E+00	9.65E-01	2.11E-01	7.62E-01	2.24E-01
	3	6.16	96.9	12.4	3.02E+00	9.76E-01	8.70E+00	2.18E+00	5.63E-01	1.37E-01	6.77E-01	1.34E-01
	4	6.09	62.7	13.2	1.82E+00	1.22E+00	3.80E+00	1.08E+00	2.50E-01	6.51E-02	4.26E-01	5.70E-02
	5	6.34	38.7	6.8	6.80E-01	2.77E-01	1.58E+00	6.53E-01	1.17E-01	2.14E-02	3.22E-01	4.33E-02
	6	6.28	24.1	2.93	1.02E+00	3.80E-01	7.84E-01	2.38E-01	7.91E-02	3.70E-02	2.15E-01	8.16E-02
	7	6.34	21.2	7.17	9.56E-01	6.14E-01	6.28E-01	9.54E-02	3.81E-02*	9.63E-04	1.27E-01	1.42E-01
	8[2]	6.38	19.1	2.79	1.73E+00	1.83E+00	4.60E-01	4.85E-02	1.49E-01	2.21E-01	2.13E-01	1.52E-01
	Total[3]	6.10	83	59.6	1.60E+00	5.29E-01	5.57E+00	8.16E+00	3.51E-01	4.16E-01	3.89E-01	1.90E-01
Reactor 6 0.63 %S	1	5.96	322	278	6.50E-01	8.94E-01	3.72E+01	5.15E+01	2.08E+00	2.92E+00	1.10E+00	1.38E+00
	2	5.37	320	54.7	9.91E+00	5.58E+00	5.43E+01	1.99E+01	3.31E+00	1.22E+00	2.21E+00	7.81E-01
	3	5.17	232	35.3	2.38E+01	2.97E+00	2.60E+01	5.61E+00	1.47E+00	3.88E-01	1.09E+00	3.37E-01
	4	5.19	159	21.4	2.31E+01	1.37E+00	1.18E+01	2.46E+00	6.53E-01	1.17E-01	5.72E-01	7.19E-02
	5	5.31	130	19.7	2.25E+01	3.70E+00	7.92E+00	3.31E+00	4.62E-01	1.93E-01	4.50E-01	6.11E-02
	6	5.40	90.2	10.3	1.40E+01	5.08E+00	3.42E+00	4.40E-01	2.12E-01	3.47E-02	4.86E-01	2.12E-01
	7	5.69	74	9.86	1.49E+01	4.54E+00	2.41E+00	4.92E-01	6.27E-02	5.76E-02	6.96E-01	2.67E-01
	8[2]	5.67	71.5	10.7	1.17E+01	1.53E+00	1.79E+00	3.08E-01	1.23E-01	1.86E-02	3.27E-01	5.39E-02
	Total[3]	5.19	175	91.3	1.51E+01	1.80E+00	1.81E+01	1.78E+01	1.05E+00	1.01E+00	8.68E-01	4.64E-01
Reactor 7 0.94 %S	1	4.53	530	263	4.47E+01	4.65E+01	9.48E+01	1.02E+02	4.71E+00	5.09E+00	2.30E+00	2.54E+00
	2	4.42	452	44.9	6.35E+01	9.26E+00	6.99E+01	1.71E+01	2.97E+00	8.53E-01	1.23E+00	4.71E-01
	3	4.50	314	34	5.50E+01	3.55E+00	3.41E+01	7.00E+00	1.33E+00	2.91E-01	5.57E-01	1.46E-01
	4	4.55	239	26.8	5.14E+01	3.33E+00	1.74E+01	4.49E+00	7.02E-01	1.49E-01	4.01E-01	6.49E-02
	5	4.55	193	26.6	5.37E+01	6.88E+00	9.06E+00	3.64E+00	3.94E-01	1.53E-01	3.22E-01	5.28E-02
	6	4.60	154	15.9	4.55E+01	5.36E+00	4.46E+00	1.65E+00	1.81E-01	4.90E-02	2.61E-01	1.81E-03
	7	4.49	140	19.5	4.63E+01	2.80E+00	3.83E+00	7.23E-01	3.76E-02*	4.96E-04	7.48E-01	1.32E-01
	8[2]	4.69	125	5.67	3.38E+01	1.54E+00	1.99E+00	1.24E+00	1.23E-01	1.97E-02	4.00E-01	5.18E-02
	Total[3]	4.47	268	85	4.92E+01	1.50E+01	2.94E+01	3.45E+01	1.31E+00	1.75E+00	7.77E-01	8.64E-01
Reactor 8(D) 0.31 %S	1	6.34	156	78.7	7.44E-02	8.29E-02	9.08E-01	1.97E+00	8.37E-02	1.16E-01	1.34E-01	1.76E-01
	2	5.89	135	36.3	4.42E-02	2.45E-02	5.33E+00	1.28E+00	3.18E-01	9.47E-02	3.27E-01	2.20E-01
	3	6.19	69	24.2	7.91E-02	2.00E-02	2.98E+00	1.44E+00	1.91E-01	9.35E-02	2.11E-01	7.71E-02
	4	6.15	42.7	6.9	7.67E-02	5.04E-02	9.63E-01	5.72E-01	9.89E-02	2.12E-02	1.17E-01	5.59E-02
	5	6.28	24.8	4.56	3.48E-02*	2.97E-05	4.37E-01	1.06E-01	3.76E-02*	3.20E-05	1.66E-01	4.93E-03
	6	6.32	19.8	4.92	4.69E-02	2.20E-02	4.14E-01	6.46E-02	3.69E-02*	1.33E-04	1.71E-01	2.97E-02
	7	6.23	15.1	5.55	7.12E-02	6.28E-02	4.12E-01	2.40E-01	3.73E-02*	3.79E-04	1.03E-01	9.59E-02
	8[2]	6.18	13.7	6.27	7.93E-02	2.99E-02	3.26E-01	1.40E-01	6.28E-02	3.12E-02	5.83E-02	4.28E-02
	Total[3]	6.05	59.5	26.1	6.33E-02	2.68E-02	1.47E+00	7.36E-01	1.08E-01	4.87E-02	1.61E-01	7.43E-02

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 3 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 9 0.13 %S	1	6.92	32.9	6.46	7.52E-03*	9.01E-05	2.58E-02	1.01E-02	8.11E-03*	9.71E-05	7.31E-03*	8.75E-05
	2	6.76	32	3.91	7.58E-03*	9.25E-05	7.85E-02	1.37E-02	8.17E-03*	9.98E-05	1.00E-02*	6.48E-03
	3	6.54	27.1	2.82	8.95E-03*	4.00E-03	1.75E-01	3.60E-02	8.23E-03*	6.46E-05	9.31E-03	3.82E-03
	4	6.47	22.1	2.12	1.84E-02	1.44E-02	2.16E-01	2.15E-02	8.20E-03*	4.87E-05	7.39E-03*	4.39E-05
	5	6.49	18.5	1.6	7.65E-03*	1.67E-04	1.47E-01	5.42E-02	8.25E-03*	1.80E-04	7.44E-03*	1.62E-04
	6	6.44	14.6	1.07	1.73E-02	8.90E-03	2.09E-01	4.69E-02	8.23E-03*	9.80E-06	1.36E-02	5.66E-03
	7	6.37	14.4	2.11	1.26E-02*	9.96E-03	1.85E-01	2.57E-02	8.21E-03*	6.10E-05	7.40E-03*	5.50E-05
	8[2]	6.30	13.2	0.935	9.75E-03*	4.23E-03	1.52E-01	9.61E-03	8.24E-03*	2.19E-05	7.42E-03*	1.97E-05
	Total[3]	6.44	21.8	1.82	1.12E-02	5.35E-03	1.49E-01	1.70E-02	8.21E-03	5.37E-05	8.74E-03	2.81E-03
Humidity cell 10 0.21 %S	1	6.92	67.6	12.9	7.61E-03	8.41E-05	6.77E-02	5.39E-02	8.20E-03*	9.07E-05	7.39E-03*	8.17E-05
	2	6.61	50.3	5.73	1.32E-02	6.20E-03	3.22E-01	4.86E-02	8.29E-03*	7.15E-05	2.11E-02	2.16E-02
	3	6.39	43.2	5.95	5.39E-02	2.39E-02	7.44E-01	2.46E-01	3.37E-02	1.21E-02	1.93E-02	9.65E-03
	4	6.27	33.6	4.15	9.05E-02	6.08E-03	9.76E-01	9.96E-02	4.75E-02	6.10E-03	1.82E-02	7.66E-03
	5	6.27	27.8	2.11	1.10E-01	1.55E-02	8.67E-01	1.59E-01	4.43E-02	5.15E-03	1.57E-02	1.17E-02
	6	6.19	21.6	1.16	1.42E-01	2.76E-02	7.00E-01	2.63E-02	4.03E-02	7.15E-04	2.45E-02	5.94E-03
	7	6.12	19.8	3.13	1.67E-01	3.34E-02	6.58E-01	6.61E-02	3.52E-02	4.14E-03	1.15E-02*	9.41E-03
	8[2]	6.16	20.3	2.66	1.39E-01	9.40E-03	5.57E-01	6.40E-02	3.30E-02	3.20E-03	9.69E-03*	4.46E-03
	Total[3]	6.18	35.5	3.7	9.03E-02	1.19E-02	6.12E-01	7.31E-02	3.13E-02	4.03E-03	1.59E-02	6.31E-03
Humidity cell 11 0.33 %S	1	7.00	50.9	8.86	1.10E-02	5.44E-03	4.61E-02	1.93E-02	8.16E-03*	5.03E-05	7.35E-03*	4.54E-05
	2	6.44	52.7	2.66	2.91E-02	1.21E-02	4.77E-01	2.35E-01	1.47E-02	1.07E-02	7.44E-03*	1.13E-04
	3	5.99	54.7	7.03	2.04E-01	1.38E-01	2.30E+00	1.01E+00	1.10E-01	5.92E-02	5.47E-02	3.35E-02
	4	5.69	48.4	3.66	6.04E-01	1.53E-01	3.38E+00	3.95E-01	2.05E-01	3.58E-02	1.06E-01	1.96E-02
	5	5.57	46.3	4.98	1.14E+00	2.47E-01	3.43E+00	5.60E-01	2.43E-01	2.15E-02	1.35E-01	1.45E-03
	6	5.45	37	3.09	1.69E+00	3.00E-01	3.04E+00	1.90E-01	2.09E-01	8.58E-03	1.37E-01	9.64E-03
	7	5.28	34.9	5.53	1.93E+00	3.29E-01	2.71E+00	4.43E-01	1.78E-01	1.85E-02	1.41E-01	3.46E-02
	8[2]	4.88	35	4.22	2.44E+00	1.14E-01	2.46E+00	1.06E-01	1.54E-01	2.01E-02	1.01E-01	1.01E-02
	Total[3]	5.31	45	2.1	1.01E+00	1.22E-01	2.23E+00	3.16E-01	1.40E-01	1.84E-02	8.61E-02	1.42E-02
Humidity cell 12 0.55 %S	1	7.06	71.8	9.31	1.12E-02	5.74E-03	1.23E-01	4.22E-02	8.09E-03*	6.76E-05	7.29E-03*	6.09E-05
	2	6.54	68.8	8.25	1.04E-01	6.15E-02	8.44E-01	3.29E-01	4.10E-02	1.48E-02	2.17E-02	2.21E-02
	3	6.27	68.5	7.99	4.01E-01	1.29E-01	2.98E+00	1.03E+00	1.46E-01	5.56E-02	8.32E-02	3.52E-02
	4	6.02	63.8	3.56	9.62E-01	2.14E-01	5.10E+00	7.61E-01	2.85E-01	5.15E-02	1.77E-01	2.42E-02
	5	5.87	59.1	7.33	1.55E+00	2.59E-01	5.27E+00	6.79E-01	3.24E-01	2.25E-02	2.18E-01	6.85E-03
	6	5.76	50.3	3.84	1.76E+00	9.26E-02	4.51E+00	3.66E-01	2.55E-01	1.29E-02	1.73E-01	9.90E-03
	7	5.67	47.1	5.15	1.82E+00	2.22E-01	3.98E+00	3.97E-01	2.15E-01	1.75E-02	1.64E-01	2.75E-02
	8[2]	5.47	43.4	3.73	1.97E+00	1.03E-01	3.26E+00	1.73E-01	1.78E-01	1.22E-02	1.23E-01	1.93E-02
	Total[3]	5.70	59.1	2.33	1.07E+00	8.80E-02	3.26E+00	3.27E-01	1.81E-01	1.97E-02	1.21E-01	1.17E-02

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 4 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 13 0.55 %S	1	6.92	72	12.2	1.15E-02	6.16E-03	9.47E-02	2.58E-02	8.28E-03*	2.77E-05	8.78E-03*	3.22E-03
	2	6.64	61.6	5.9	3.88E-02	1.78E-02	6.11E-01	3.24E-01	2.38E-02	1.44E-02	2.28E-02	1.04E-02
	3	6.34	62.6	10.9	2.07E-01	1.04E-01	2.84E+00	1.04E+00	1.31E-01	5.54E-02	6.37E-02	3.02E-02
	4	6.04	57.3	4.83	6.22E-01	1.39E-01	4.72E+00	7.88E-01	2.63E-01	5.43E-02	1.63E-01	3.22E-02
	5	5.90	57.1	5.69	1.37E+00	1.41E-01	5.03E+00	1.16E+00	3.07E-01	5.08E-02	2.08E-01	7.60E-03
	6	5.85	44.7	3.7	1.63E+00	1.49E-01	4.02E+00	2.29E-01	2.38E-01	4.04E-03	1.60E-01	1.61E-02
	7	5.62	40.9	6.22	1.71E+00	3.31E-01	3.42E+00	4.31E-01	1.91E-01	1.55E-02	1.41E-01	4.79E-02
	8[2]	5.66	40.2	3.6	1.83E+00	2.49E-02	2.99E+00	1.21E-01	1.67E-01	9.58E-03	1.02E-01	3.52E-03
	Total[3]	5.75	54.5	3.2	9.27E-01	1.06E-01	2.96E+00	4.29E-01	1.66E-01	2.38E-02	1.09E-01	1.62E-02
Humidity cell 14 0.72 %S	1	6.69	103	12.7	2.20E-02	1.92E-02	3.31E-01	3.32E-01	1.26E-02*	1.09E-02	7.37E-03*	4.59E-05
	2	6.11	113	6.33	3.54E-01	1.13E-01	4.69E+00	1.33E+00	1.90E-01	6.03E-02	6.89E-02	2.36E-02
	3	5.90	103	11.2	7.69E-01	1.26E-01	7.38E+00	6.86E-01	3.34E-01	3.46E-02	1.32E-01	2.80E-02
	4	5.67	89.9	7.36	1.34E+00	2.69E-01	7.56E+00	7.45E-01	3.78E-01	5.27E-02	1.72E-01	3.22E-02
	5	5.61	87.7	7.4	2.01E+00	2.34E-01	8.44E+00	2.23E-01	4.45E-01	2.16E-03	2.13E-01	2.54E-02
	6	5.55	70.7	3.82	2.50E+00	1.23E-01	7.02E+00	6.35E-01	3.76E-01	6.81E-03	1.79E-01	2.31E-02
	7	5.37	66.9	9.24	2.93E+00	5.16E-01	6.48E+00	9.06E-01	3.47E-01	4.11E-02	1.81E-01	5.38E-02
	8[2]	5.23	66.9	5.08	3.57E+00	2.61E-01	6.21E+00	3.33E-01	3.36E-01	1.97E-02	1.58E-01	3.95E-03
	Total[3]	5.43	87.7	3.01	1.69E+00	1.51E-01	6.01E+00	3.63E-01	3.02E-01	2.19E-02	1.39E-01	1.67E-02
Humidity cell 15 1.03 %S	1	4.95	175	24.1	2.29E+00	1.75E+00	2.62E+01	1.79E+01	1.26E+00	8.98E-01	3.89E-01	2.94E-01
	2	4.50	136	16.2	3.97E+00	4.84E-01	2.57E+01	3.74E+00	1.08E+00	1.79E-01	4.09E-01	5.40E-02
	3	4.40	165	20.6	5.47E+00	5.46E-01	1.88E+01	2.42E+00	8.06E-01	8.76E-02	3.23E-01	5.24E-02
	4	4.16	180	15.1	1.03E+01	3.44E+00	1.64E+01	1.52E+00	7.58E-01	6.86E-02	2.94E-01	2.64E-02
	5	4.17	185	15.4	1.33E+01	1.83E-02	2.44E+01	7.72E+00	7.95E-01	4.08E-02	2.82E-01	1.67E-02
	6	4.18	157	8.63	1.67E+01	5.22E-01	1.58E+01	1.33E+00	6.92E-01	7.60E-02	2.06E-01	9.47E-03
	7	4.09	161	31.1	2.42E+01	8.61E+00	1.60E+01	2.52E+00	8.45E-01	1.33E-01	3.57E-01	1.43E-01
	8[2]	4.07	148	12.2	2.03E+01	5.73E-01	1.29E+01	9.33E-01	6.03E-01	5.14E-02	1.98E-01	2.79E-02
	Total[3]	4.17	164	7.12	1.21E+01	2.89E+00	1.95E+01	5.72E+00	8.55E-01	2.89E-01	3.07E-01	9.70E-02
Humidity cell 16 1.36 %S	1	4.91	242	24.1	6.23E+00	5.61E+00	3.07E+01	2.38E+01	1.93E+00	1.84E+00	7.96E-01	7.94E-01
	2	4.61	189	10.4	1.29E+01	1.27E+00	3.92E+01	4.95E+00	1.68E+00	2.55E-01	7.89E-01	1.32E-01
	3	4.46	188	15.6	1.50E+01	6.11E-01	2.93E+01	4.16E+00	1.26E+00	1.80E-01	6.57E-01	2.90E-01
	4	4.34	174	6.3	1.78E+01	2.24E+00	2.28E+01	9.95E-01	9.57E-01	4.80E-02	4.18E-01	2.12E-02
	5	4.28	177	10.1	2.19E+01	2.85E-01	2.17E+01	2.89E+00	8.20E-01	1.61E-01	3.77E-01	4.47E-02
	6	4.31	154	6.71	2.79E+01	1.11E+00	1.73E+01	2.02E+00	6.52E-01	8.07E-02	2.89E-01	2.06E-02
	7	4.19	173	41.8	4.16E+01	1.70E+01	1.66E+01	4.31E+00	7.23E-01	1.80E-01	4.08E-01	1.66E-01
	8[2]	4.13	144	12.5	2.65E+01	3.29E+00	1.25E+01	1.31E+00	5.43E-01	6.08E-02	2.35E-01	2.57E-02
	Total[3]	4.28	180	11.7	2.12E+01	5.57E+00	2.38E+01	7.51E+00	1.07E+00	6.05E-01	4.96E-01	2.63E-01

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 5 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 17(D) 0.23 %S	1	7.19	32	12.6	7.51E-03*	1.00E-04	8.13E-03*	1.08E-04	8.10E-03*	1.08E-04	7.30E-03*	9.73E-05
	2	6.97	13.2	1.64	7.58E-03*	6.20E-05	8.21E-03*	6.71E-05	8.17E-03*	6.68E-05	7.37E-03*	6.02E-05
	3	6.84	10.7	1.26	9.49E-03	3.68E-03	1.39E-02	1.28E-02	8.21E-03*	6.88E-05	1.37E-02	9.51E-03
	4	6.64	8.5	0.472	1.23E-02	5.69E-03	2.53E-02	1.25E-02	8.15E-03*	1.44E-04	7.35E-03*	1.29E-04
	5	6.60	8.75	0.612	2.97E-02	3.10E-02	8.33E-03	1.45E-04	8.30E-03*	1.44E-04	7.48E-03*	1.30E-04
	6	6.47	7.27	0.703	1.67E-02	7.89E-03	3.09E-02	1.37E-02	8.20E-03*	9.45E-05	1.09E-02*	6.10E-03
	7	6.35	7.76	1.47	1.12E-02*	8.69E-03	1.39E-02	8.90E-03	8.20E-03*	8.70E-05	7.39E-03*	8.65E-05
	8[2]	6.30	7.07	0.35	7.68E-03*	4.81E-05	1.77E-02	8.26E-03	8.28E-03*	5.18E-05	7.46E-03*	4.67E-05
	Total[3]	6.40	11.9	4.14	1.28E-02	1.03E-02	1.58E-02	6.05E-03	8.20E-03	3.45E-05	8.62E-03	3.69E-03
Humidity cell 18(D) 0.31 %S	1	7.07	62.4	9.32	9.64E-03*	5.05E-03	1.61E-02	9.31E-03	8.20E-03*	1.08E-04	1.06E-02	7.86E-03
	2	6.44	52.2	4.83	7.57E-03*	3.68E-05	2.64E-01	1.53E-01	1.03E-02*	5.16E-03	1.49E-02	1.22E-02
	3	6.28	49	6.33	1.51E-02	5.78E-03	1.02E+00	2.36E-01	4.23E-02	9.03E-03	1.08E-02	5.66E-03
	4	6.10	40.1	5.09	1.51E-02	1.30E-02	1.26E+00	8.75E-02	5.81E-02	3.13E-03	1.19E-02	7.86E-03
	5	6.09	36	3.74	2.41E-02	1.04E-02	1.13E+00	2.41E-01	5.85E-02	6.74E-03	7.43E-03*	3.24E-05
	6	6.00	28.3	1.64	4.10E-02	2.95E-02	9.22E-01	7.65E-02	5.63E-02	3.89E-03	1.37E-02	6.08E-03
	7	5.83	30.5	10.7	1.20E-01	5.75E-02	1.13E+00	4.47E-01	7.64E-02	3.25E-02	1.25E-02*	1.24E-02
	8[2]	5.53	27.6	3.98	7.32E-02	1.14E-03	7.89E-01	8.60E-02	5.43E-02	4.91E-03	7.39E-03*	2.34E-05
	Total[3]	5.85	40.8	3	3.82E-02	1.94E-02	8.17E-01	1.39E-01	4.55E-02	1.02E-02	1.12E-02	4.71E-03
Humidity cell 19(D) 0.61 %S	1	7.00	92.6	12	7.56E-03*	1.14E-04	6.00E-02	2.22E-02	8.15E-03*	1.23E-04	7.35E-03*	1.11E-04
	2	6.43	109	7.56	1.22E-02	7.69E-03	9.45E-01	7.43E-01	4.40E-02	3.66E-02	1.21E-02	8.32E-03
	3	6.14	105	11.7	5.82E-02	2.17E-02	3.75E+00	6.85E-01	2.22E-01	5.27E-02	3.24E-02	2.02E-02
	4	5.83	87.7	6.6	1.64E-01	4.83E-02	4.47E+00	4.73E-01	3.44E-01	5.66E-02	7.58E-02	1.24E-02
	5	5.64	78.1	11.1	3.15E-01	6.00E-02	4.70E+00	9.69E-02	3.85E-01	1.93E-02	1.03E-01	2.75E-02
	6	5.36	65.9	6.21	7.24E-01	3.67E-01	4.60E+00	5.58E-01	4.01E-01	6.62E-02	1.60E-01	4.86E-02
	7	5.06	71.9	8.44	1.05E+00	1.43E-01	4.87E+00	7.04E-01	4.35E-01	5.27E-02	1.94E-01	4.95E-02
	8[2]	4.96	68.9	6.32	1.03E+00	7.66E-02	4.25E+00	4.42E-01	3.94E-01	3.43E-02	1.29E-01	9.64E-03
	Total[3]	5.09	84.8	2.48	4.20E-01	1.21E-01	3.46E+00	2.73E-01	2.79E-01	2.19E-02	8.92E-02	1.86E-02

[1] 5th percentile value
[2] Year 8 represented by weeks 365-400
[3] Average and standard deviation for entire period of record
* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.
Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 11. Total mass release, sulfur and heavy metal depletion for Duluth Complex waste rock reactors and humidity cells for weeks 0-400.

ID, %S	Total mass release after 400 weeks, mmol/kg rock									Percent remaining after 400 weeks				
	SO_4	Ca	Mg	Na	K	Cu	Ni	Co	Zn	S	Cu	Ni	Co	Zn
75 g reactors														
1, 0.07	12	11	11.8	4.64	4.31	0.0185	0.106	0.0147	0.035	45.2%	99.7%	98.5%	98.9%	97.7%
2, 0.11	19.7	13.3	15.6	5.24	5.36	0.0555	0.404	0.0294	0.0756	42.7%	99.5%	93.9%	97.5%	95.1%
3, 0.20	31.5	15.2	10.7	7.01	6.95	2.82	2.07	0.144	0.173	49.4%	82.1%	71.0%	88.0%	89.5%
4, 0.31	39.1	16.9	25.2	6.42	6.11	1.09	3.24	0.191	0.182	59.5%	96.2%	69.6%	86.1%	89.1%
5, 0.31	35.5	16.3	23.6	6.85	6.51	0.671	2.93	0.179	0.178	63.3%	97.7%	72.5%	87.0%	89.3%
6, 0.63	73.7	18.9	33.8	9.72	8.84	6.20	7.4	0.426	0.347	62.5%	85.4%	49.9%	73.6%	81.4%
7, 0.94	111	25.4	14	14	7.84	19.9	12.5	0.556	0.323	62.2%	76.0%	30.6%	65.9%	82.6%
8(D), 0.31	24.7	14.1	17.5	3.96	3.22	0.0261	0.641	0.0466	0.0712	87.0%	99.5%	87.2%	95.6%	94.0%
1000 g humidity cells														
9, 0.13	9.1	5.43	7	1.28	1.99	0.00426	0.0581	0.00329	0.00341	77.6%	100.0%	99.2%	99.7%	99.8%
10, 0.21	14.7	7.44	8.63	1.74	2.27	0.0345	0.245	0.0124	0.00661	77.5%	99.8%	96.6%	98.9%	99.6%
11, 0.33	18.6	8.09	8.5	2.64	3.14	0.382	0.895	0.0564	0.0344	82.0%	98.5%	88.5%	95.2%	97.9%
12, 0.55	24.5	10.2	13.3	2.22	2.71	0.412	1.3	0.0726	0.049	87.9%	99.1%	87.5%	94.5%	96.9%
13, 0.55	22.5	9.56	12.3	2.02	2.52	0.361	1.19	0.0666	0.0441	88.9%	99.2%	88.6%	95.0%	97.2%
14, 0.72	36.1	10.5	20.8	2.81	3.04	0.65	2.4	0.12	0.0557	83.9%	98.9%	85.7%	93.0%	97.2%
15, 1.03	66.9	17.2	20.4	9.34	2.89	4.65	7.89	0.344	0.123	79.2%	93.6%	57.7%	79.3%	92.6%
16, 1.36	74	20	19.2	9.4	3.36	8.31	9.73	0.436	0.201	82.6%	91.9%	54.3%	74.6%	88.9%
17(D), 0.23	5.01	5.98	3.98	1.44	1.21	0.00516	0.00626	0.0033	0.00341	93.0%	100.0%	99.9%	99.7%	99.7%
18(D), 0.31	16.6	9.65	7.81	2.13	1.45	0.0143	0.324	0.0179	0.00447	91.3%	99.8%	95.2%	98.5%	99.6%
19(D), 0.61	34.4	21.1	11.2	4.42	1.45	0.151	1.35	0.108	0.0337	64.4%	98.7%	84.5%	92.3%	97.6%

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 12. Annual linear regression slopes ((µmol/kg*wk)/%S) and R^2 values for reactors and humidity cells

	Year	SO4		Ca		Mg		Na		K		Cu		Ni		Co		Zn	
		slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2
Reactors	1	454	0.883	94.4	0.433	170	0.710	29.2	0.576	22.9	0.103	44.1	0.663	106	0.905	5.26	0.915	2.51	0.920
	2	442	0.985	53.7	0.970	178	0.883	46.9	0.856	28.9	0.688	63.0	0.729	85.3	0.938	3.91	0.822	1.67	0.514
	3	338	0.973	39.8	0.870	121	0.809	33.5	0.858	14.7	0.569	60.7	0.858	41.2	0.944	1.70	0.825	0.670	0.372
	4	257	0.969	28.7	0.899	86.8	0.785	25.9	0.796	13.7	0.372	58.2	0.880	20.6	0.945	0.850	0.899	0.391	0.484
	5	215	0.950	18.2	0.762	81.2	0.758	14.5	0.880	0.710	0.008	60.8	0.868	11.6	0.894	0.518	0.790	0.287	0.318
	6	169	0.946	15.5	0.509	46.5	0.575	12.8	0.808	2.07	0.214	49.4	0.831	5.33	0.930	0.213	0.769	0.222	0.298
	7	152	0.939	9.32	0.296	45.3	0.461	23.7	0.890	1.27	0.069	50.7	0.841	4.34	0.942	0.00986	0.076	0.815	0.750
	8	139	0.940	14.0	0.557	44.6	0.515	26.2	0.812	1.13	0.131	37.4	0.857	2.36	0.902	0.111	0.463	0.387	0.808
Humidity Cells	1	163	0.933	59.6	0.790	33.1	0.538	7.03	0.391	11.4	0.721	4.34	0.715	26.5	0.749	1.52	0.760	0.580	0.744
	2	132	0.926	24.6	0.205	31.7	0.618	17.8	0.803	4.60	0.486	8.74	0.704	31.6	0.813	1.34	0.812	0.586	0.783
	3	143	0.941	27.5	0.431	38.2	0.753	22.8	0.808	2.22	0.172	10.5	0.739	23.4	0.899	1.01	0.916	0.482	0.844
	4	145	0.917	26.3	0.628	35.8	0.731	21.9	0.777	1.41	0.108	13.7	0.801	18.7	0.952	0.809	0.976	0.337	0.922
	5	151	0.919	24.9	0.707	29.0	0.648	19.7	0.811	0.530	0.022	17.1	0.816	20.9	0.875	0.749	0.946	0.313	0.857
	6	132	0.924	21.0	0.714	28.7	0.711	17.1	0.858	1.10	0.128	21.7	0.816	15.5	0.945	0.616	0.909	0.229	0.833
	7	144	0.928	23.8	0.696	30.5	0.782	20.7	0.859	1.25	0.151	32.1	0.802	15.1	0.930	0.709	0.875	0.355	0.934
	8	123	0.914	21.5	0.716	26.6	0.713	17.4	0.846	1.41	0.196	22.1	0.829	11.8	0.926	0.528	0.870	0.203	0.888

* Note that Mg release from reactor 7 was omitted due to low values that resulted in poor correlation with sulfur. Mg slopes and R^2 values when including reactor 7 ranged from 0.27-62.2 and 0.0004-0.273, respectively.

Considerably lower cobalt slope and R^2 values for reactors in year 7 due to drainage quality reaching detection limit values

Table 13. Factor analysis of heavy metal release from Babbitt and Dunka Road prospect samples.

%S (B, D)	min pH_B	min pH_D	M_{SO4B}/M_{SO4D}	Cu	Ni	Co	Zn
Fine (0.31, 0.31%)	6.07	6.05	1.58				
M_B/M_D				41.76	5.05	4.10	2.56
C_B/C_D				5.49	2.14	1.29	1.42
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				8.69	3.38	2.04	2.24
Coarse (0.21, 0.23%)	6.18	6.4	2.93				
M_B/M_D				6.69	39.14	3.76	1.94
C_B/C_D				1.31	0.89	0.96	1.30
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				3.85	2.62	2.81	3.81
Coarse (0.33, 0.31%)	5.31	5.85	1.12				
M_B/M_D				26.71	2.76	3.15	7.70
C_B/C_D				3.28	1.14	1.00	1.36
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				3.68	1.28	1.12	1.52
Coarse (0.55, 0.61%)	5.7	5.09	0.68				
M_B/M_D				2.56	0.92	0.64	1.38
C_B/C_D				3.68	1.19	0.94	1.14
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				2.52	0.81	0.64	0.78

B, D indicate Babbitt prospect and Dunka Road prospect, respectively
M_B/M_D Ratio of Babbitt prospect to Dunka Road prospect mass release after 400 weeks
C_B/C_D Ratio of Babbitt prospect to Dunka Road prospect solid-phase chemical content
min pH 5th percentile pH value for 400-week period of record

Note: Highlighted values are those which still have a high ratio of metal release when compared to the ratio of sulfate release multiplied by the ratio of chemical content

Table 14. Factor analysis of major cation release from Babbitt and Dunka Road prospect samples.

%S (B, D)	min pH_B	min pH_D	M_{SO4B}/M_{SO4D}	Ca	Mg	Na	K
Fine (0.31, 0.31%)	6.07	6.05	1.58				
M_B/M_D				1.20	1.44	1.62	1.90
C_B/C_D				0.77	1.27	0.88	1.68
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				1.21	2.02	1.39	2.66
Coarse (0.21, 0.23%)	6.18	6.4	2.93				
M_B/M_D				1.24	2.17	1.21	1.88
C_B/C_D				0.94	0.89	1.10	2.21
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				2.75	2.61	3.24	6.50
Coarse (0.33, 0.31%)	5.31	5.85	1.12				
M_B/M_D				0.84	1.09	1.24	2.17
C_B/C_D				0.85	1.02	0.99	1.71
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				0.95	1.14	1.11	1.91
Coarse (0.55, 0.61%)	5.7	5.09	0.68				
M_B/M_D				0.47	1.14	0.48	1.80
C_B/C_D				0.84	1.16	0.95	1.62
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				0.57	0.79	0.65	1.10

B, D indicate Babbitt prospect and Dunka Road prospect, respectively
M_B/M_D Ratio of Babbitt prospect to Dunka Road prospect mass release after 400 weeks
C_B/C_D Ratio of Babbitt prospect to Dunka Road prospect solid-phase chemical content
min pH 5th percentile pH value for 400-week period of record

Table 15. Annual average sulfate release rates (µmol/kg*wk) for Partridge River intrusion (PRI) and South Kawishiwi intrusion (SKI) samples

Partridge River Intrusion reactors (75 grams, 0.053<d<0.149 mm)								
ID #	1	2	3	4	5	8 (D)	6	7
%S	0.07	0.11	0.2	0.31	0.31	0.31	0.63	0.94
Yr 1	65.6	145	245	256	238	156	322	530
Yr 2	68.9	91.7	133	183	163	135	320	452
Yr 3	32.9	44.6	72.6	102	96.9	69	232	314
Yr 4	22.1	29.1	50	68.9	62.7	42.7	159	239
Yr 5	14	21.1	34.6	45.8	38.7	24.8	130	193
Yr 6	7.58	16.7	25.5	30.8	24.1	19.8	90.2	154
Yr 7	6.86	11.3	21	25.4	21.2	15.1	74	140
Yr 8	7.63	10.1	16.5	19.8	19.1	13.7	71.5	125

South Kawishiwi Intrusion reactors (75 grams, 0.053<d<0.149 mm)											
ID #	1,2	3,4	5,6	7,8	9,10	11,12	13,14	15,16	17,18	35,36	29,30
%S	0.18	0.22	0.4	0.41	0.51	0.54	0.57	0.58	0.71	1.12	1.16
Yr 1	56.5	64.8	277	39.2	107	219	230	199	161	624	279
Yr 2	117	103	250	159	215	274	194	149	188	468	213
Yr 3	81.7	75.5	147	90.6	119	174	151	166	127	384	165
Yr 4	48.3	48.9	93.5	52	77.7	140	95.9	260	101	553	316
Yr 5	42.6	39.4	86.2	49.2	100	159	117	353	153	873	607
Yr 6	50.1	45.9	124	88.7	372	393	301	349	373	1430	1810
Yr 7	25.3	38.9	95.9	89.2	391	365	340	255	654	412	659
Yr 8	20.8	19.7	73.2	NA[1]	312	216	262	211	469	NA[1]	182

[1] Reactors terminated after year 7

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 16. Rates of metal release from the Dunka Mine samples, weeks 6-74.

Reactor	%S	Release rate, μmol kg^{-1} wk^{-1}			
		Cu	Ni	Co	Zn
1	0.18	0.254	0.551	0.431	0.565
2	0.18	0.366	0.713	0.315	0.426
3	0.22	0.255	0.513	0.314	0.460
4	0.22	0.294	0.438	0.317	0.465
5	0.40	0.295	4.118	0.717	0.790
6	0.40	0.371	3.699	0.481	1.191
7	0.41	0.377	3.792	0.569	0.732
8	0.41	0.371	4.897	0.520	0.793
9	0.51	0.905	2.980	0.610	1.466
10	0.51	1.041	9.501	1.042	1.373
11	0.54	0.292	2.216	0.591	0.959
12	0.54	0.336	1.173	0.443	0.617
13	0.57	0.904	6.188	0.663	1.301
14	0.57	0.821	5.574	0.603	1.124
15	0.58	1.677	12.225	2.130	2.318
16	0.58	1.553	12.409	2.193	2.623
17	0.71	2.176	10.943	1.551	1.685
18	0.71	2.845	10.002	1.634	1.903
35	1.12	1.511	60.638	9.694	3.939
36	1.12	2.231	53.380	8.419	3.577
29	1.16	3.692	17.226	1.642	1.295
30	1.16	4.328	16.942	1.675	1.258

Figure 1. Map of Duluth Complex mineral prospects.

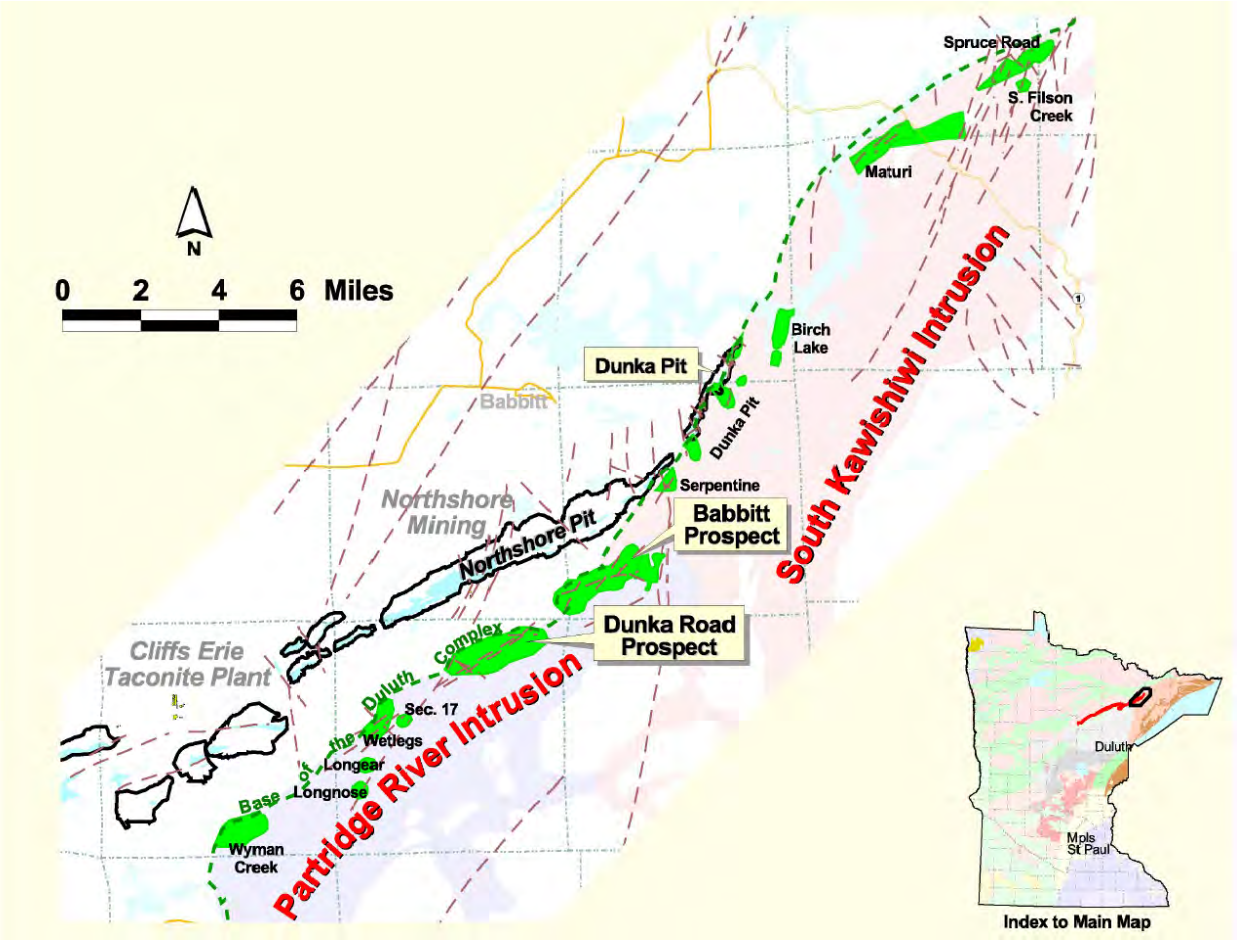


Figure 2. Schematic and photo of small reactor.

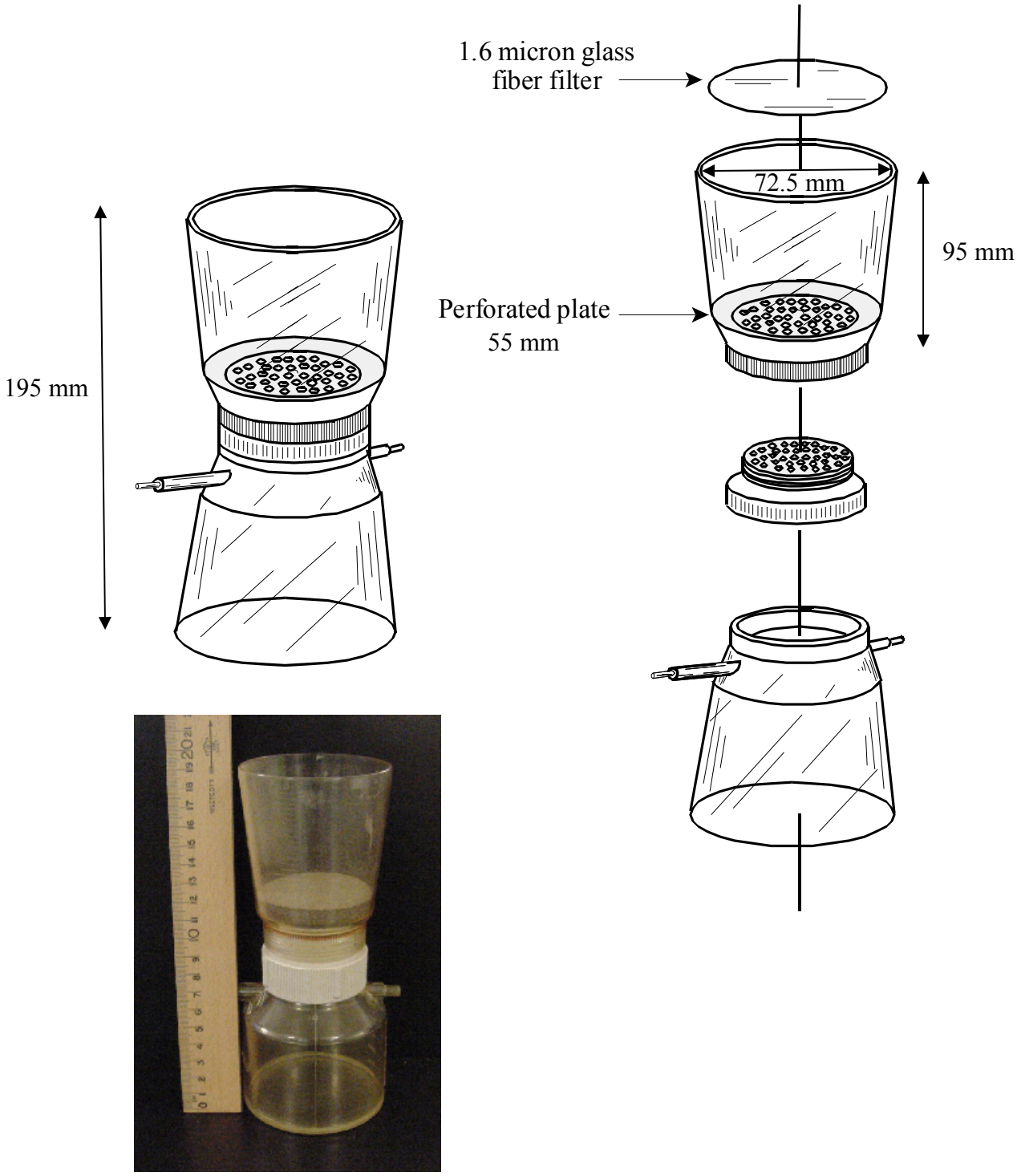


Figure 3. Schematic and photo of humidity cell apparatus. All humidity cell materials are acrylic except the perforated plate (polyvinyl chloride) and the outlet pipe (high density polyethylene).

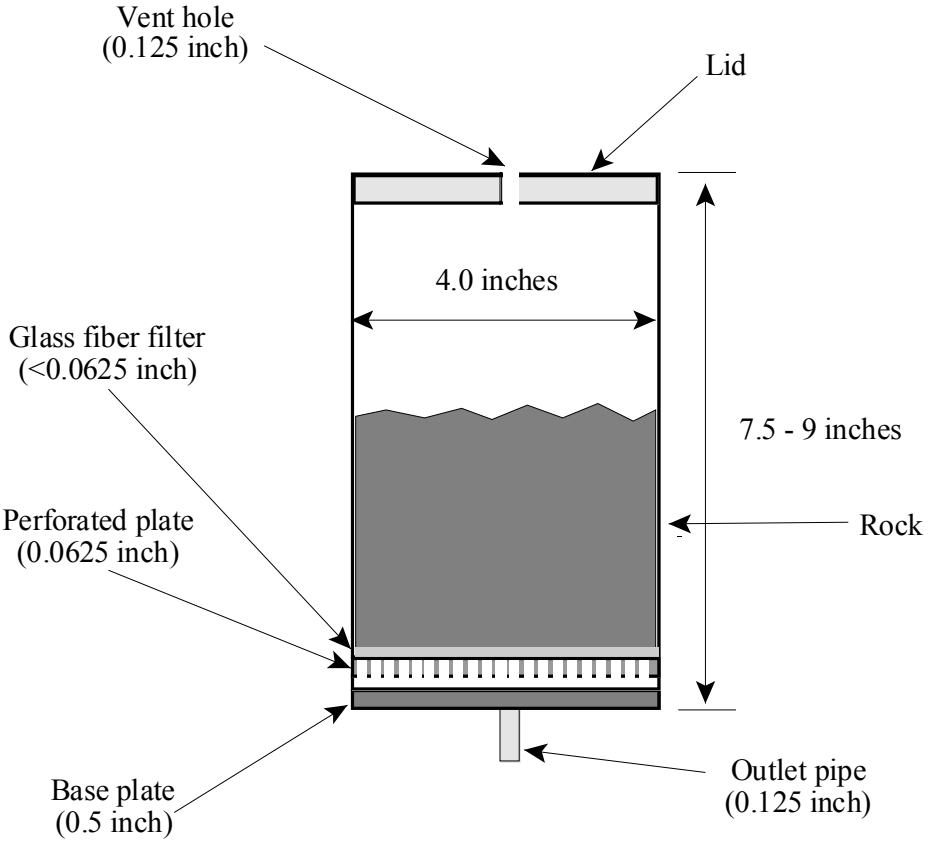


Figure 4. Drainage quality replicated well for duplicate 75-g sample of 0.31 %S Babbitt prospect rock. Weeks 0-5 were excluded to improve resolution.

Red = Reactor 4, Blue = Reactor 5

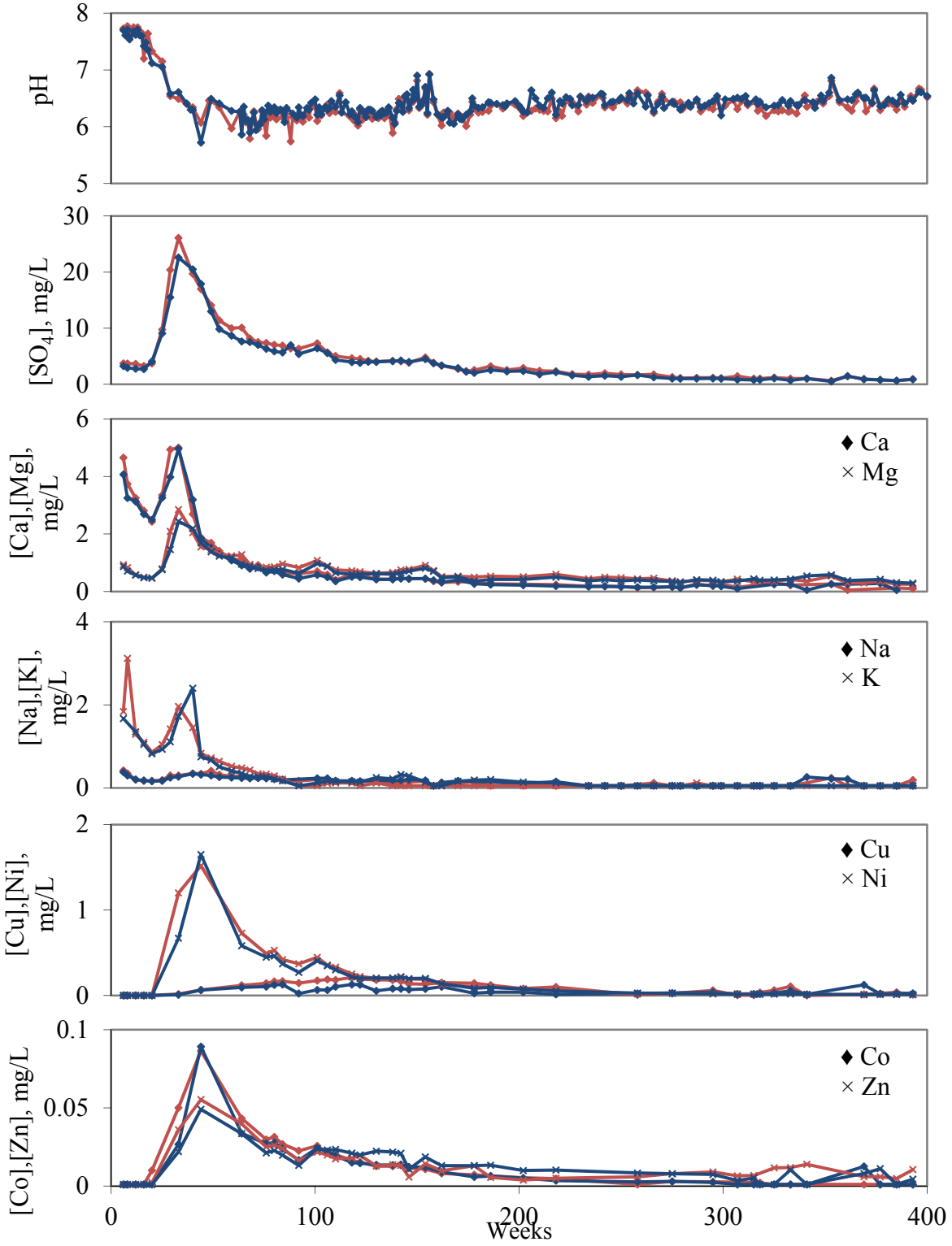


Figure 5. Drainage quality replicated well for duplicate 1000-g sample of 0.55 %S Babbitt prospect rock. Weeks 0-5 were excluded to improve resolution.

Red = Humidity cell 12, Blue = Humidity cell 13

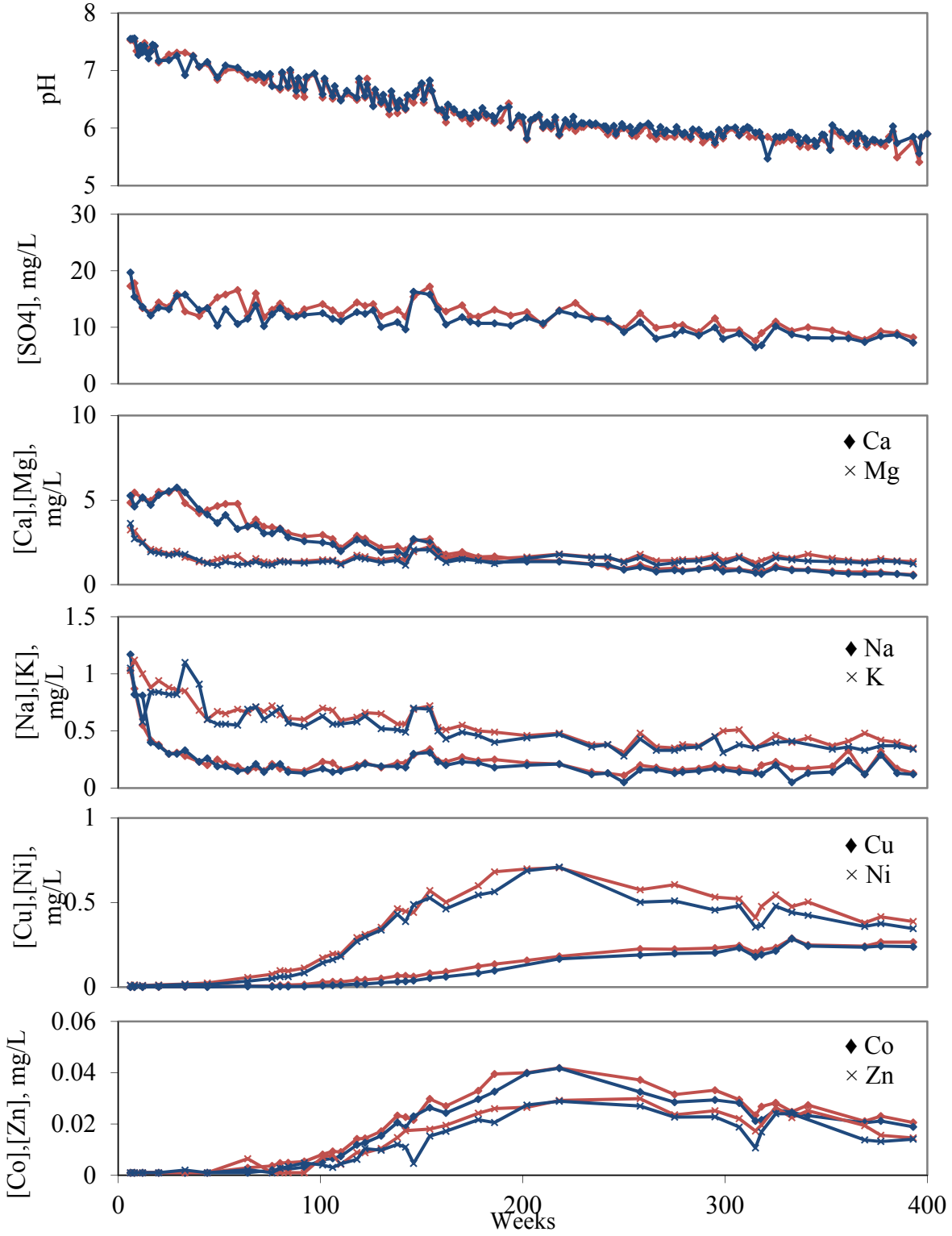
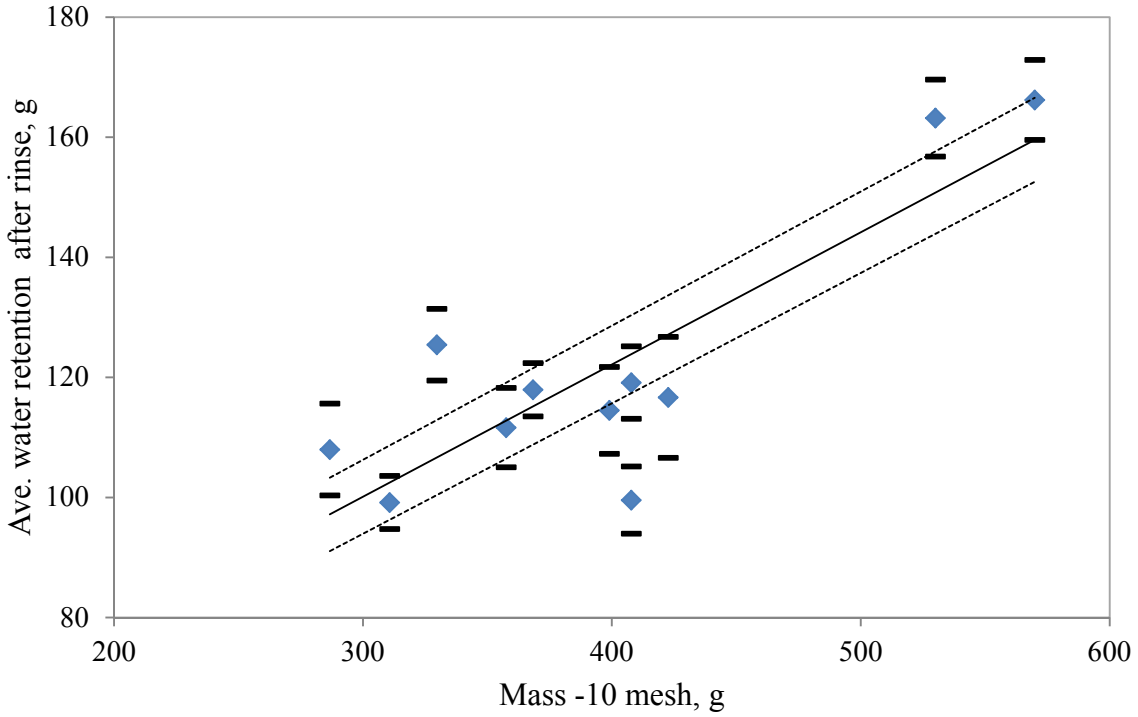
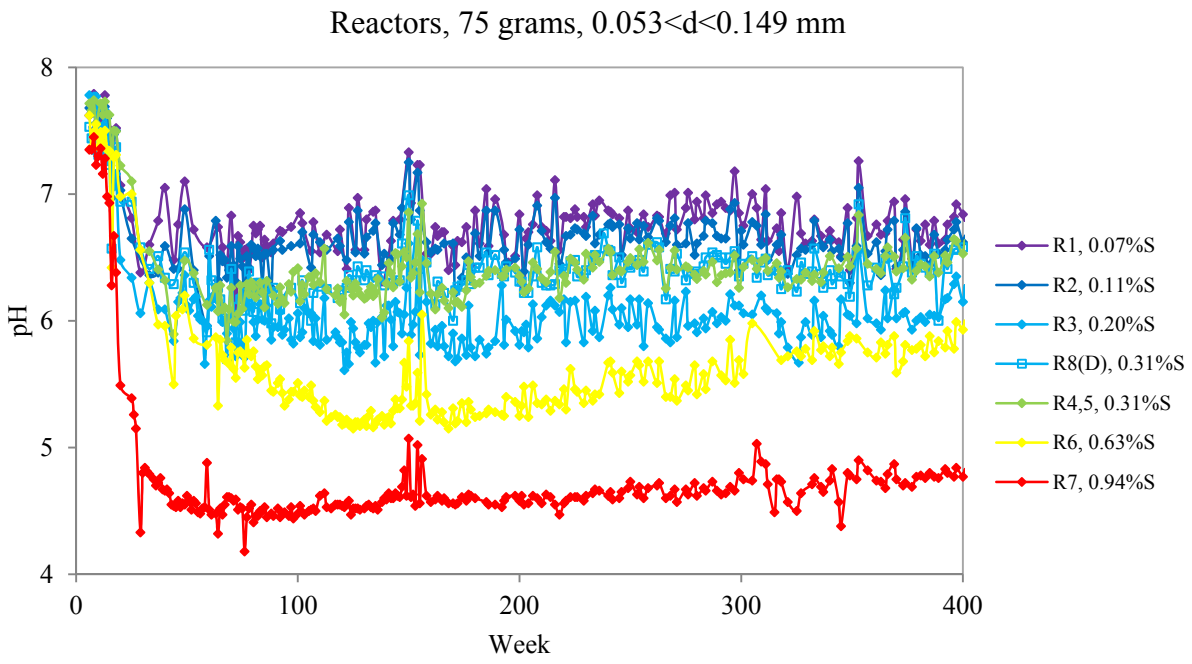


Figure 6. Water retention after rinse vs. mass – 10 mesh rock



	m	b	R^2
Average +SD	0.223	39.2	0.720
Average	0.220	34.0	0.713
Average -SD	0.217	28.8	0.699

Figure 7. pH vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



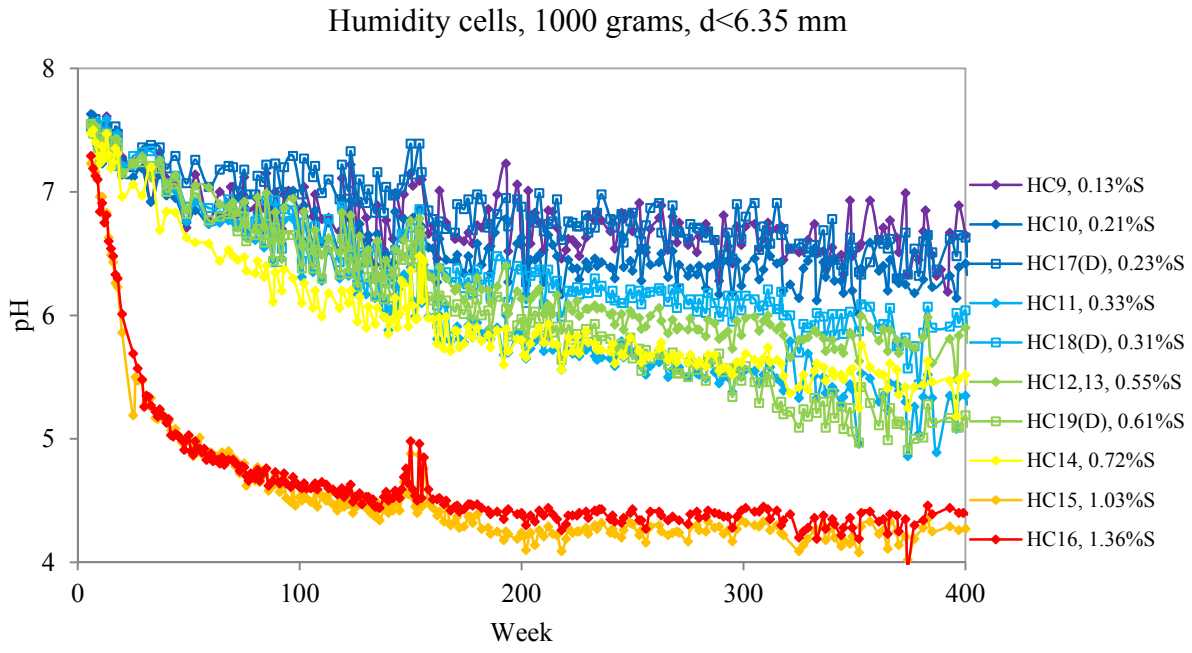
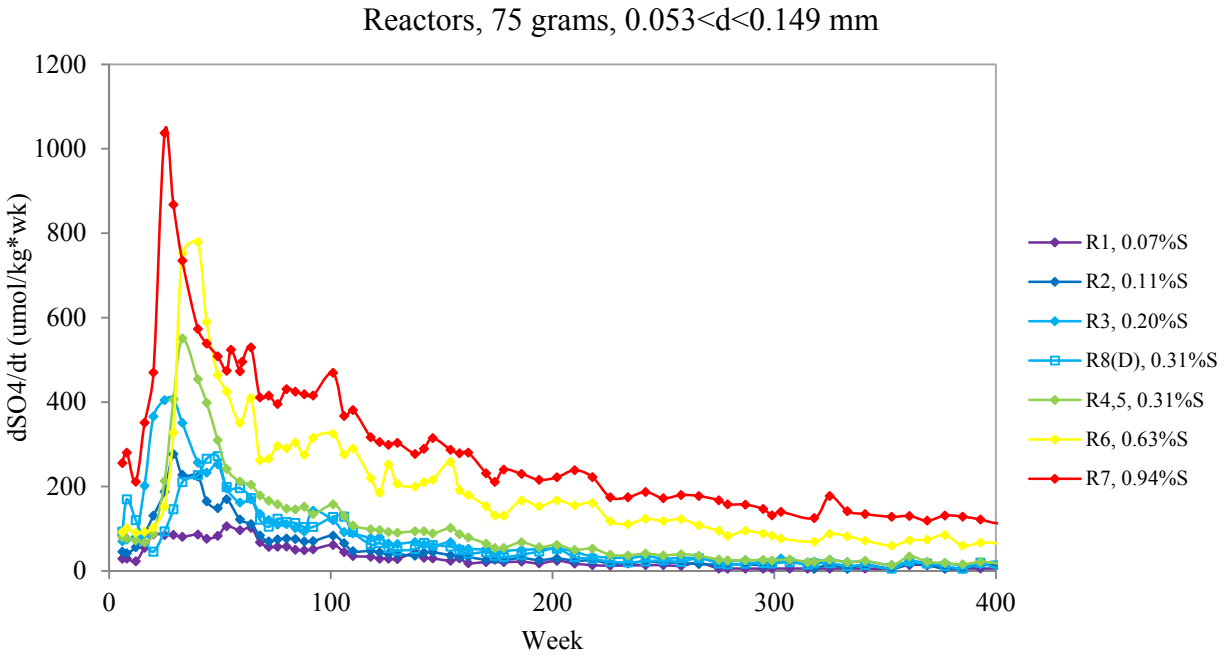


Figure 8. Sulfate release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



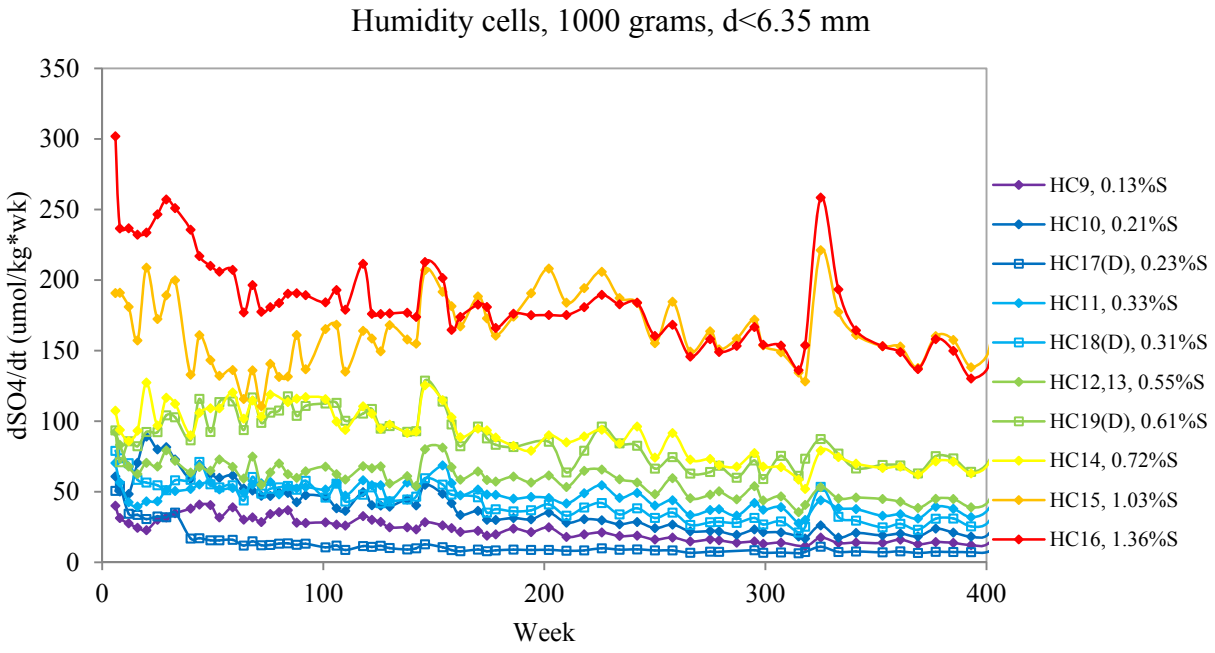
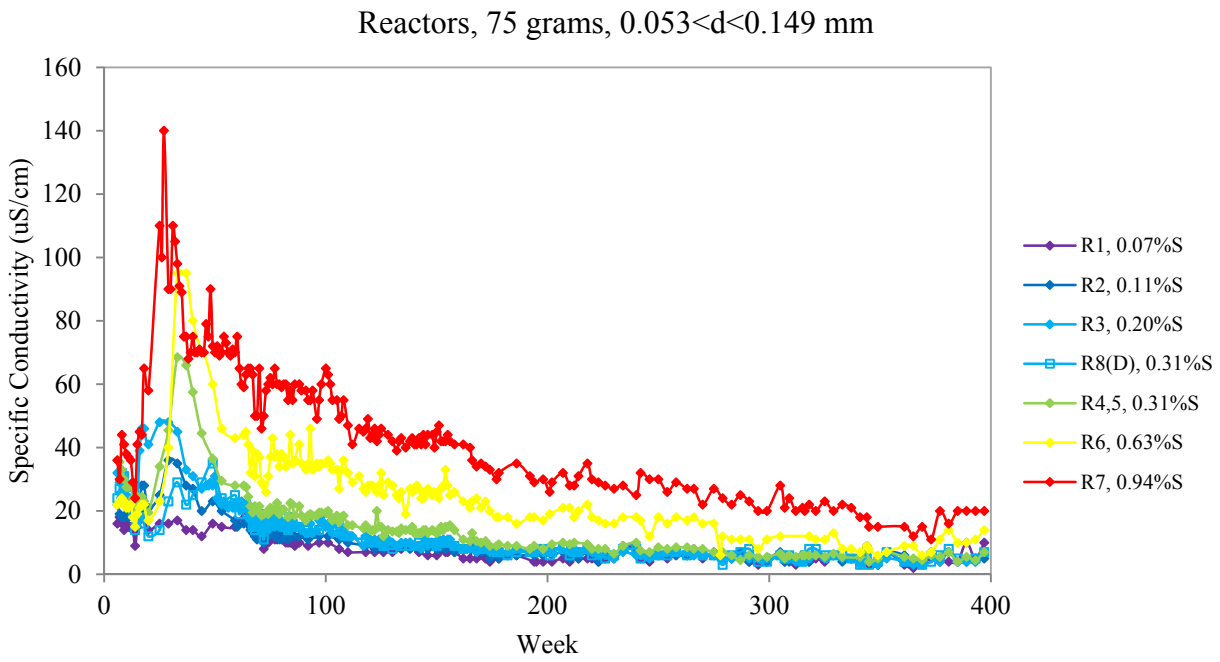


Figure 9. Specific conductivity vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



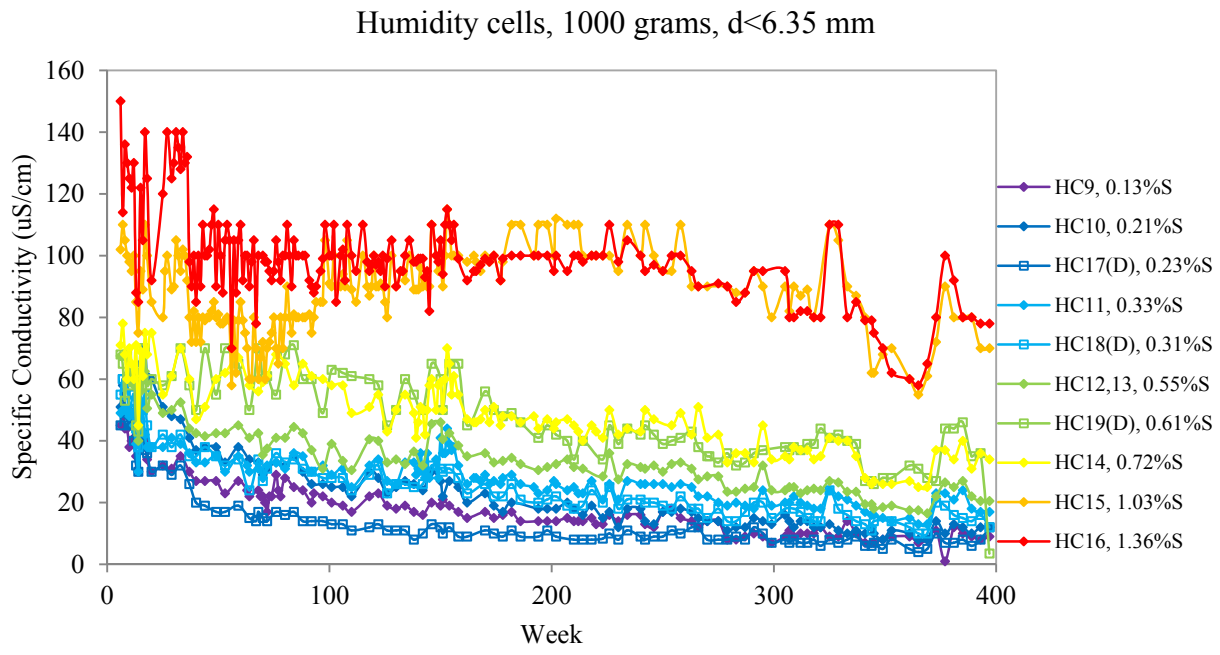
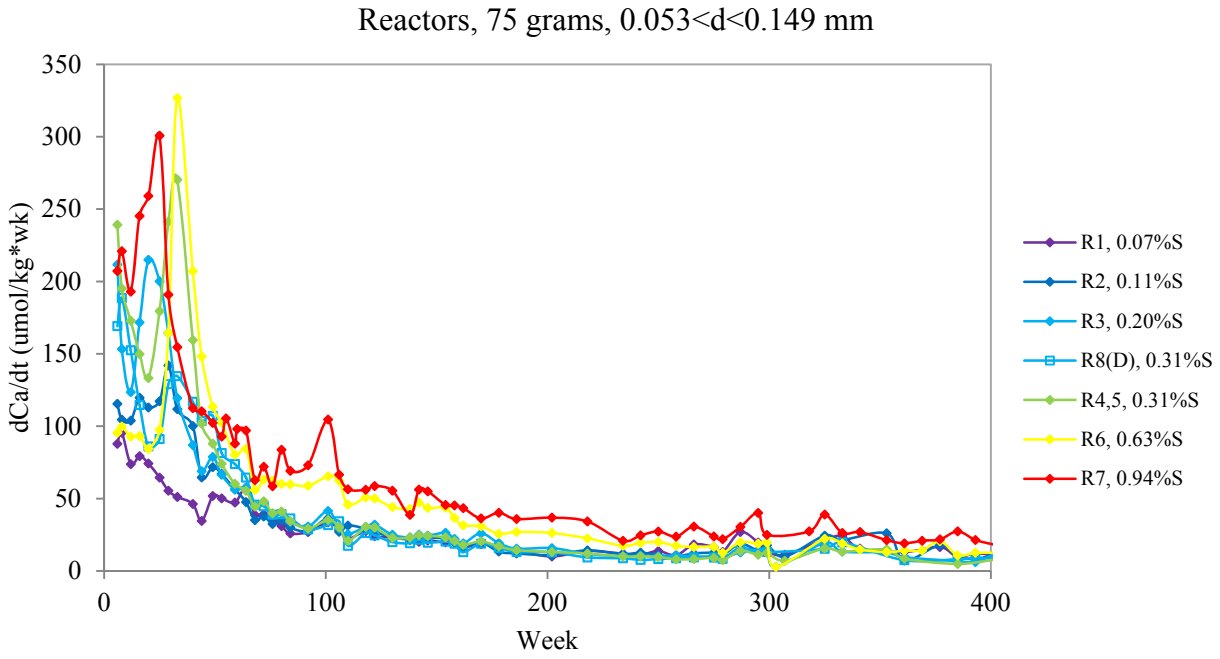


Figure 10. Calcium release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



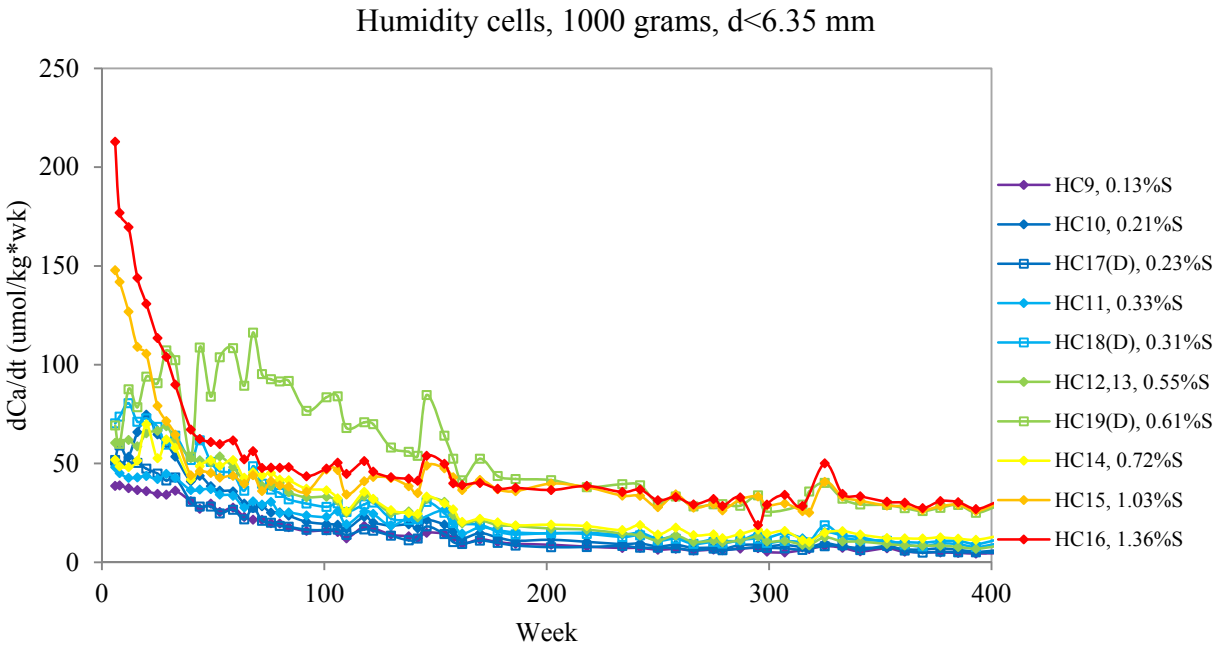
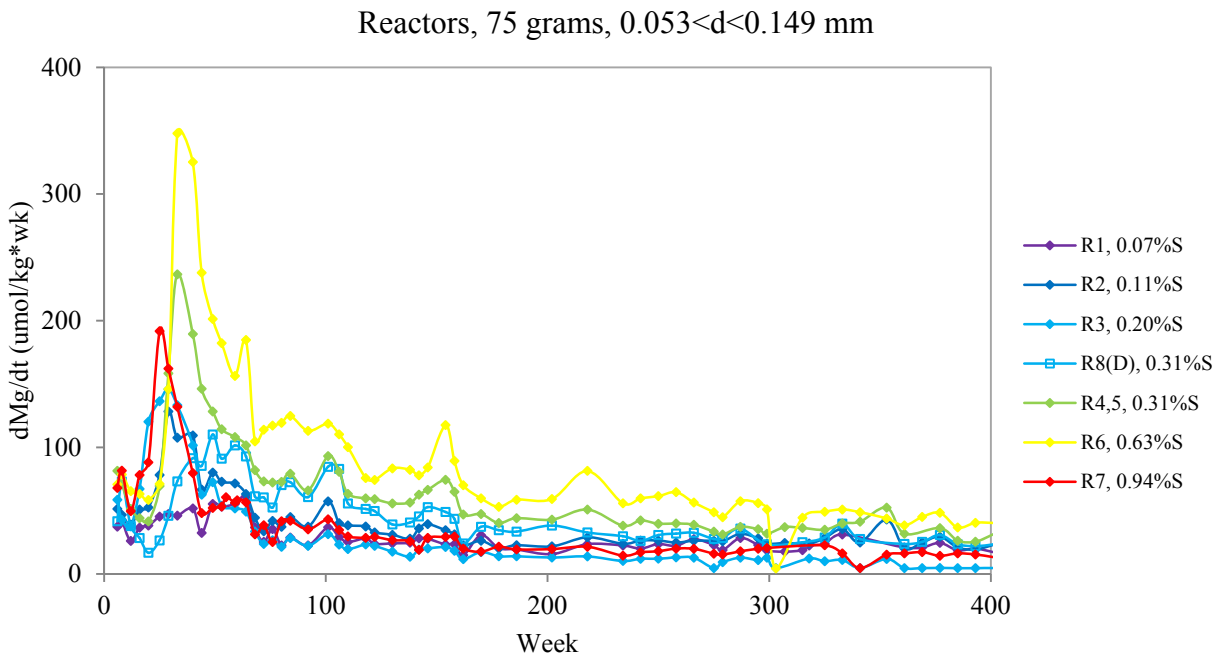


Figure 11. Magnesium release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



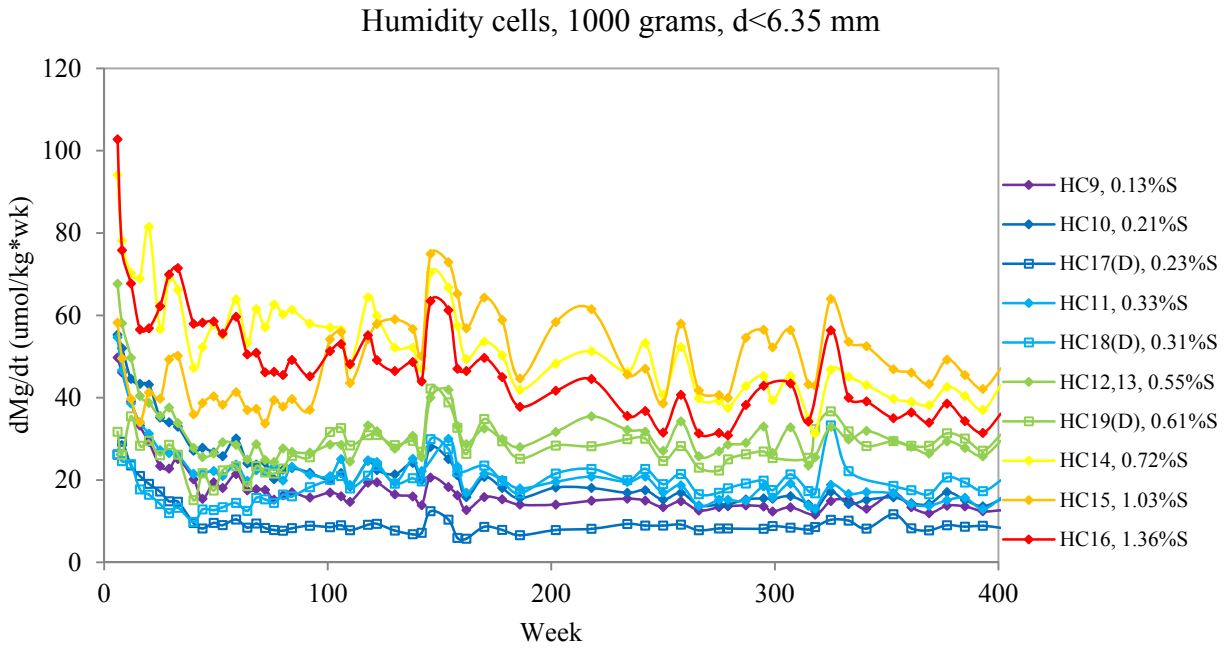
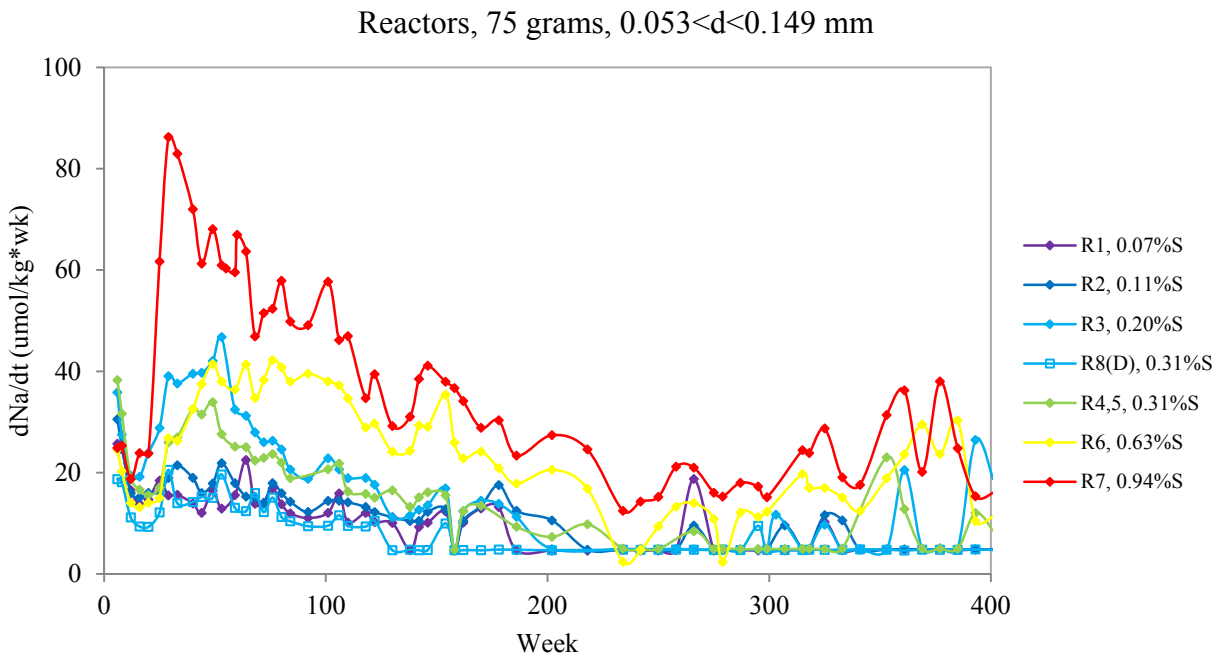


Figure 12. Sodium release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



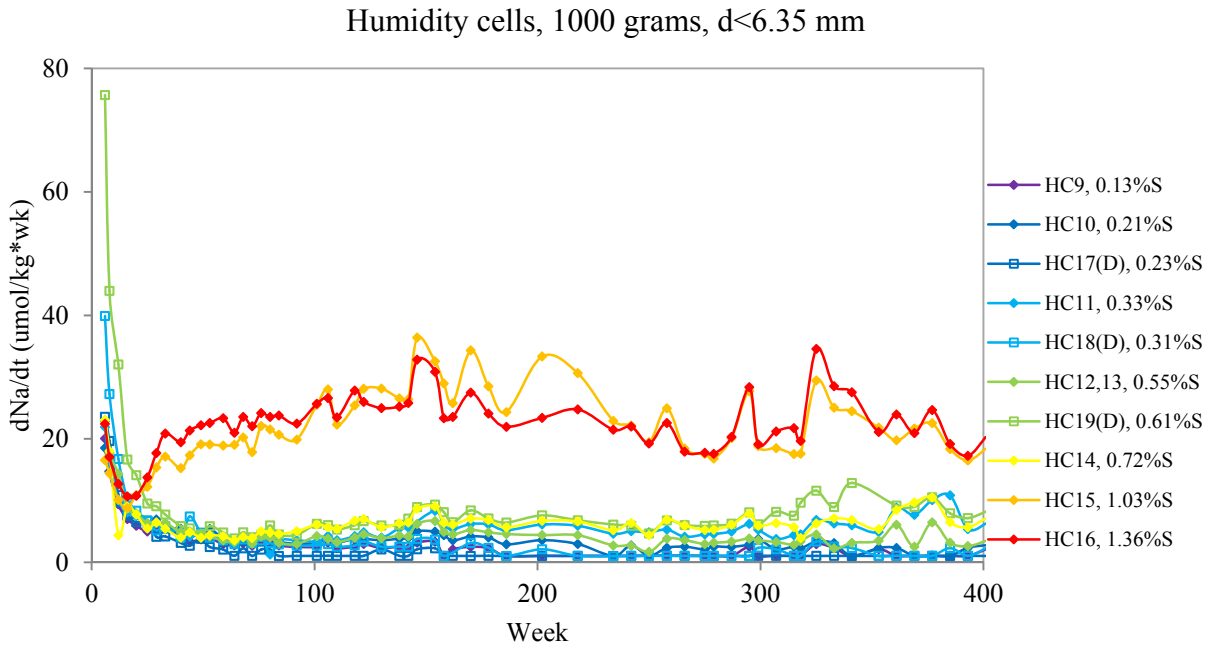
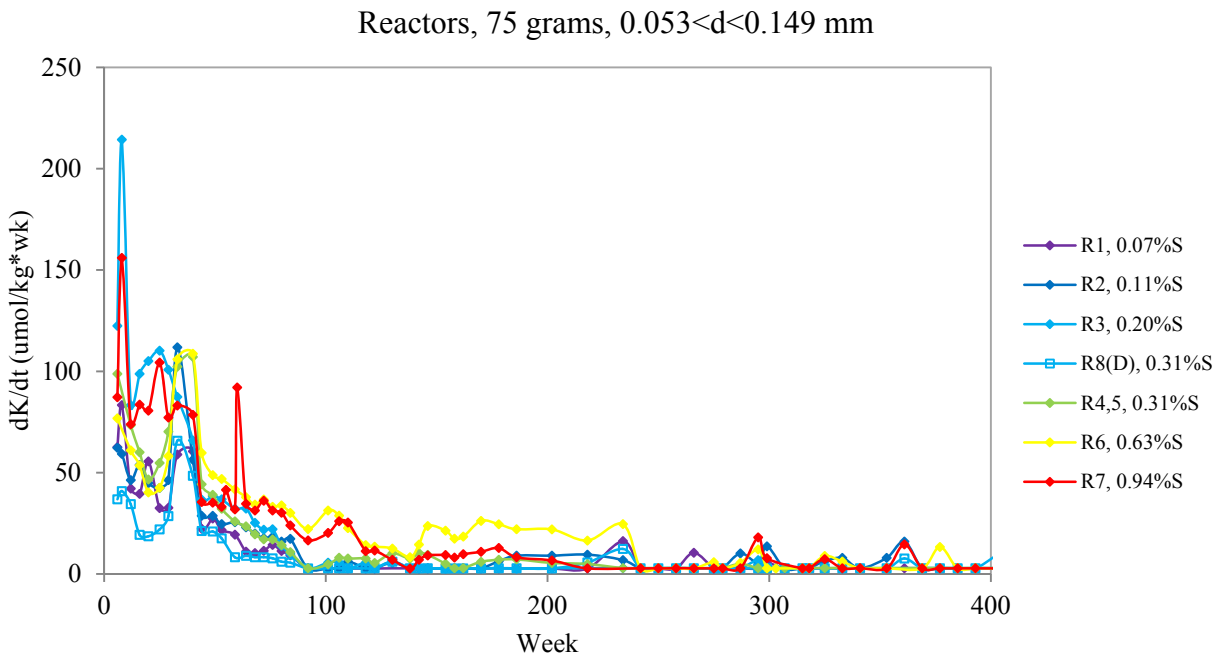


Figure 13. Potassium release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



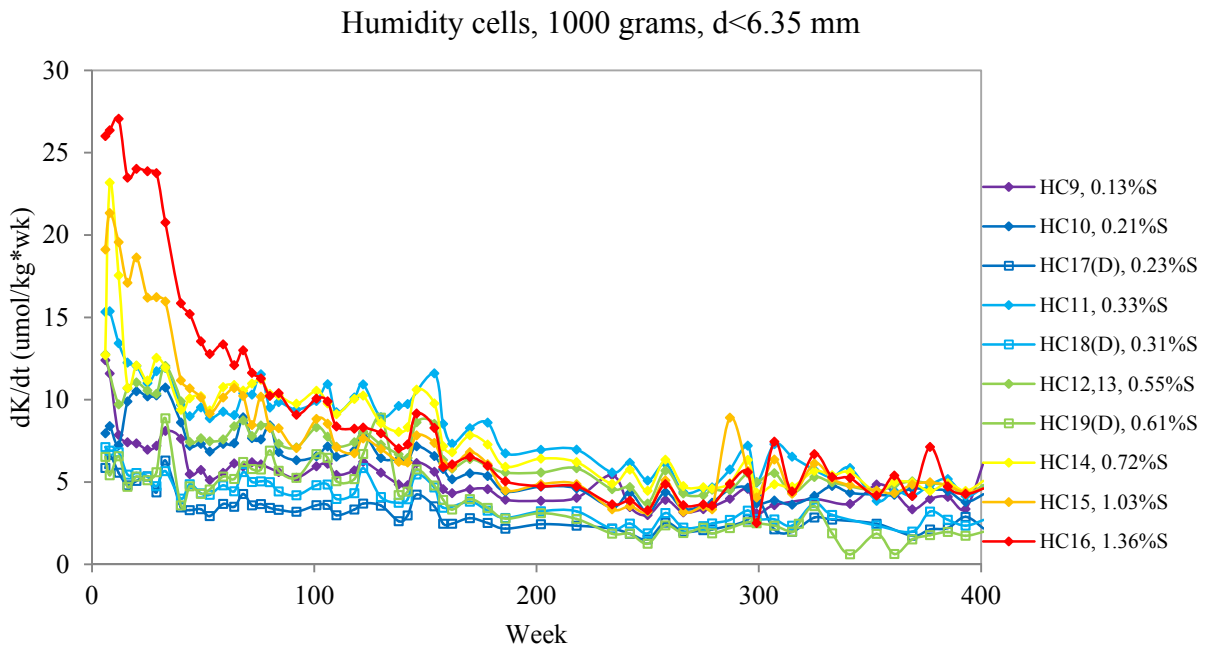
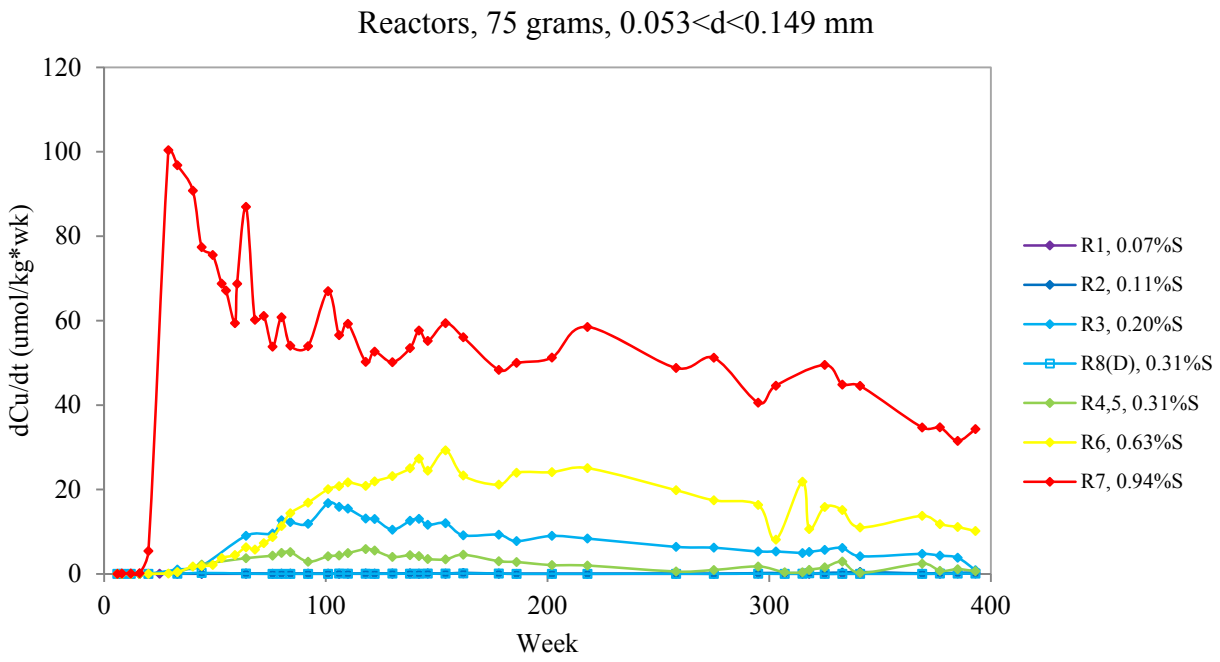


Figure 14. Copper release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



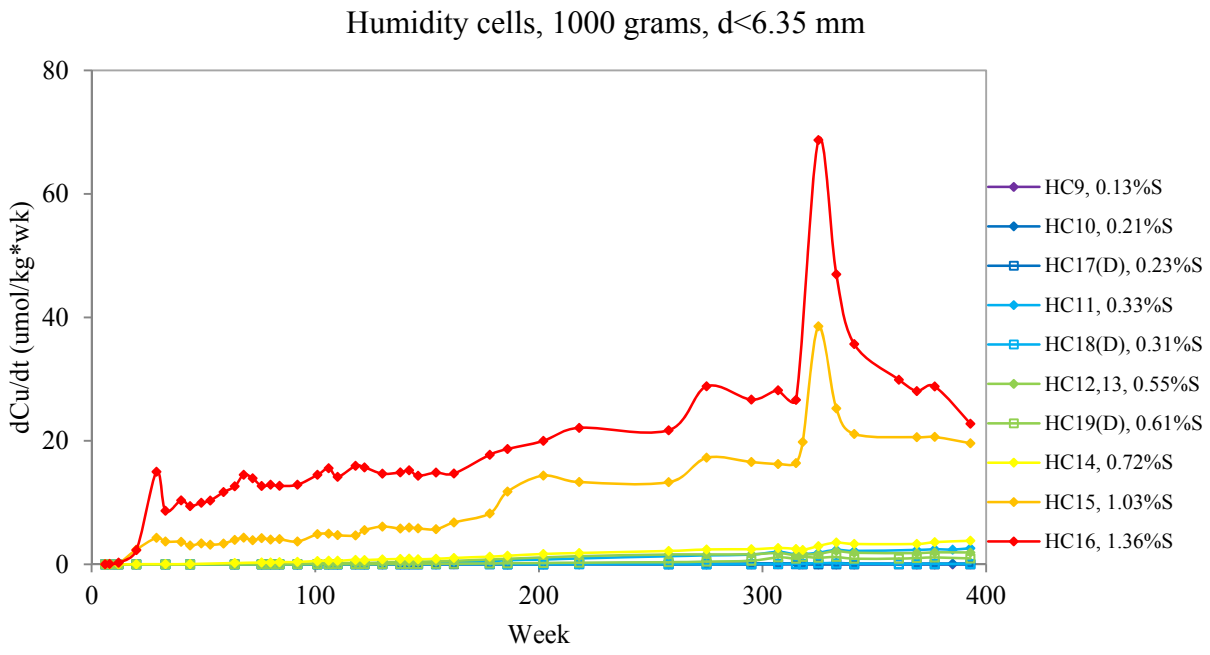
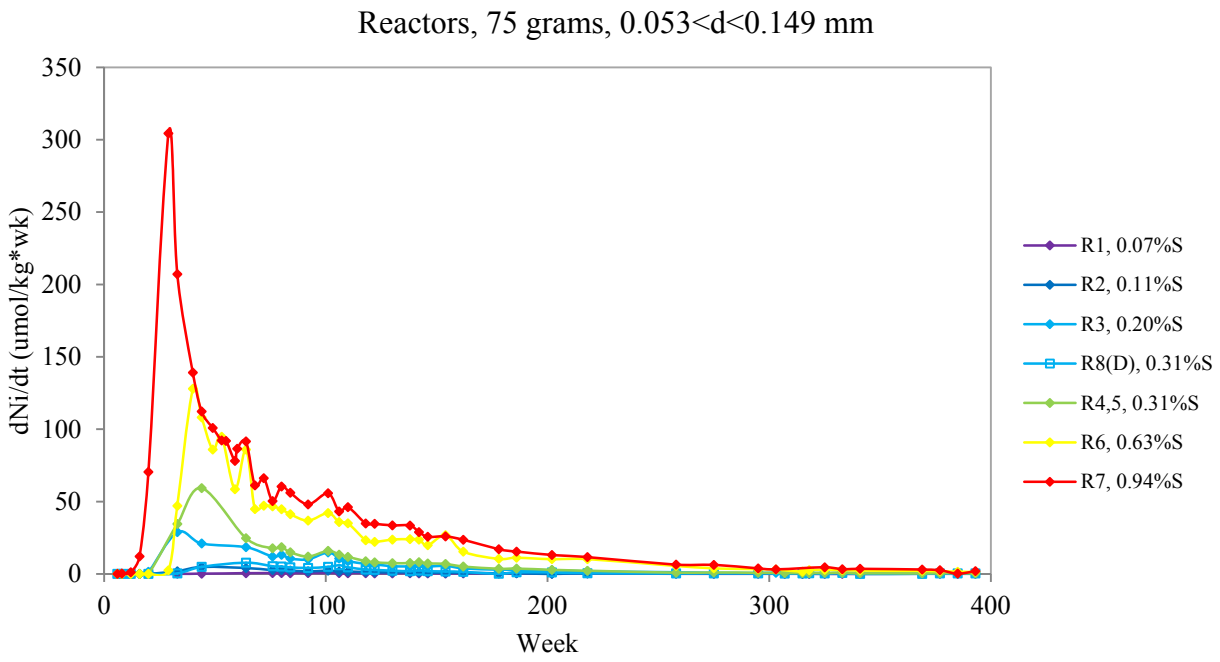


Figure 15. Nickel release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



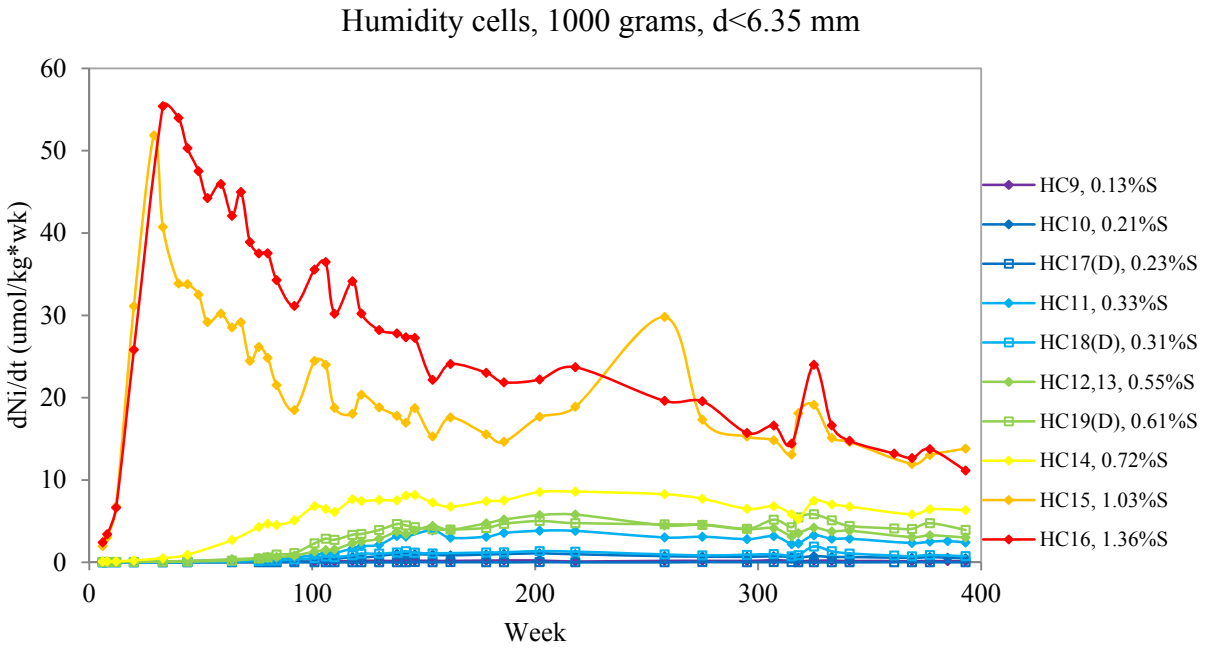
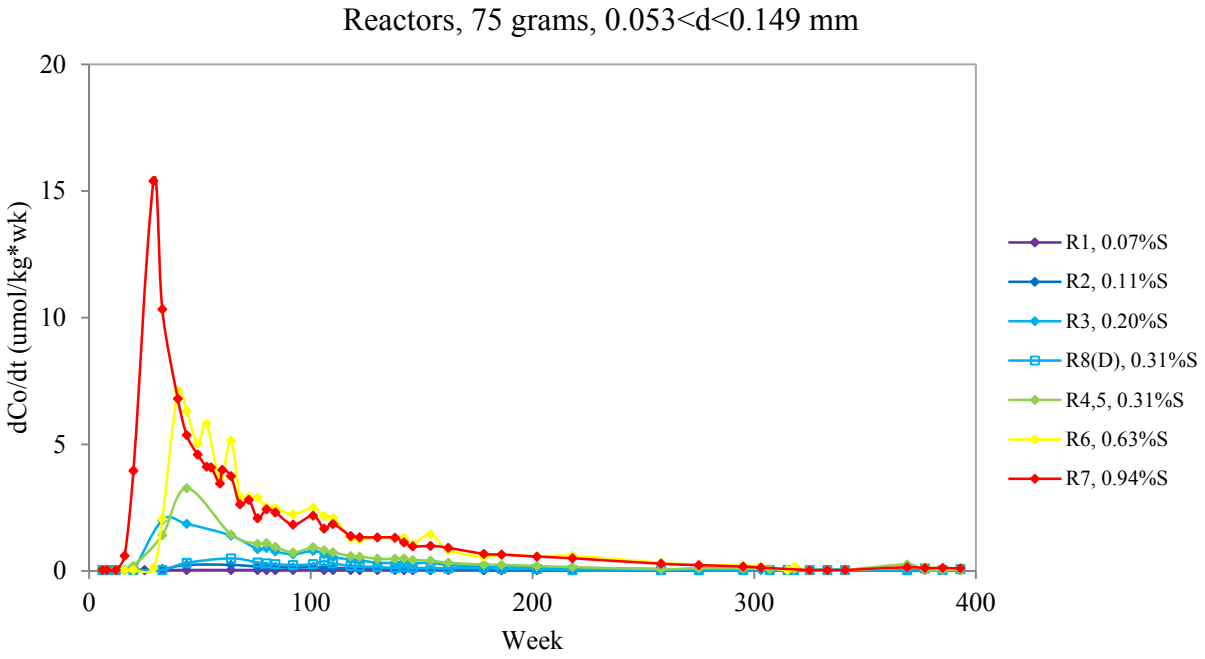


Figure 16. Cobalt release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



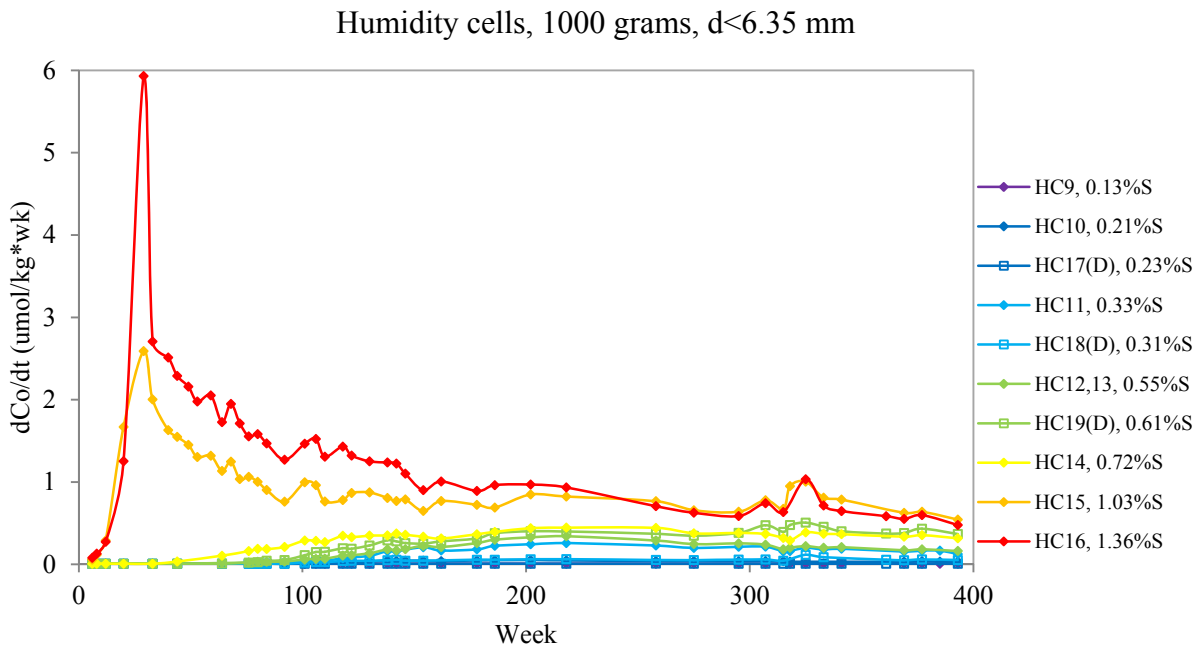
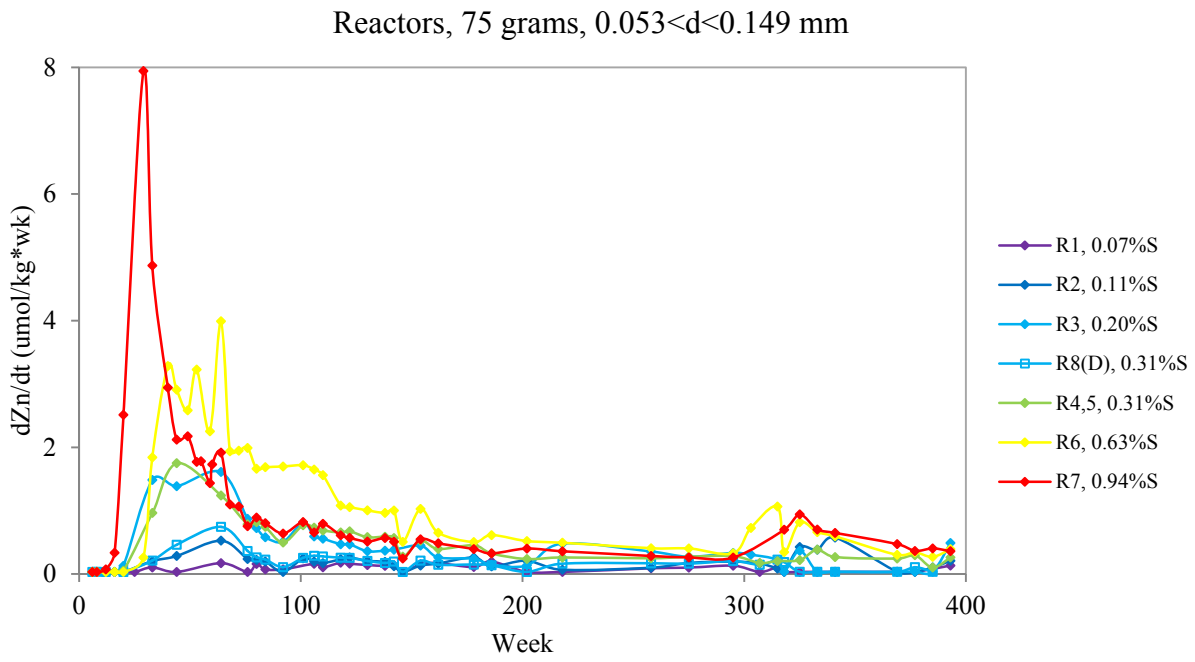


Figure 17. Zinc release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



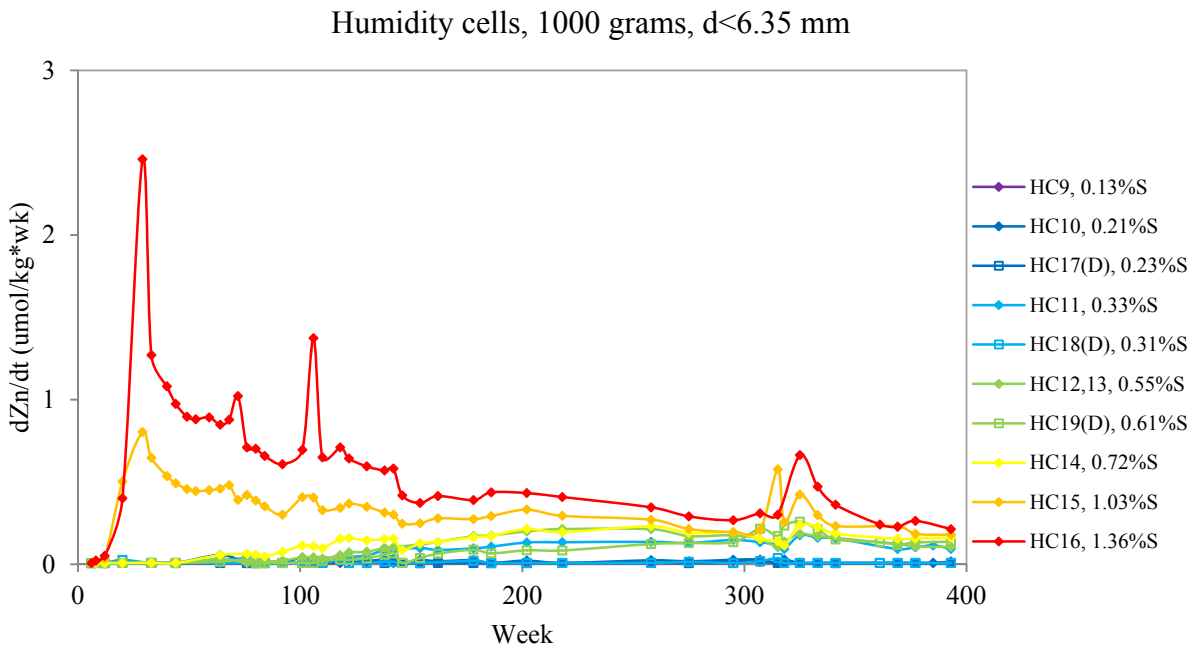


Figure 18. Minimum pH vs. %S for reactors (-100/+270 mesh) and humidity cells (-1/4 inch) for weeks 6-400.

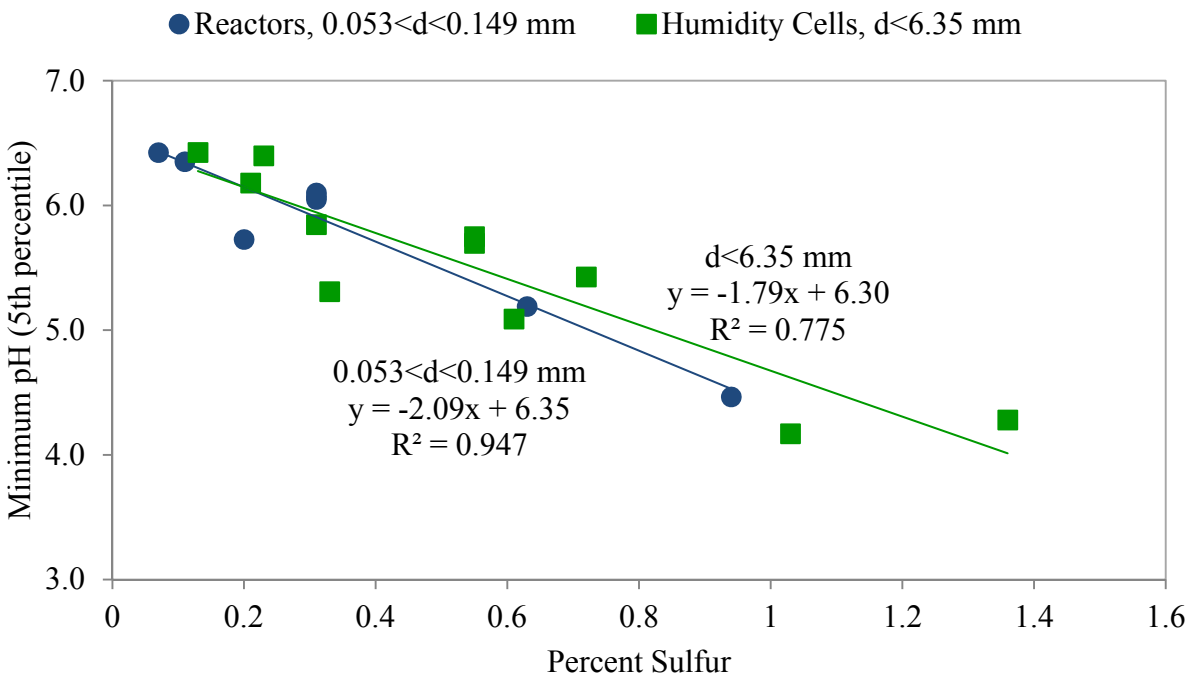


Figure 19. Average sulfate release vs. %S for reactors (-100/+270 mesh) and humidity cells (-1/4 inch) for weeks 6-400.

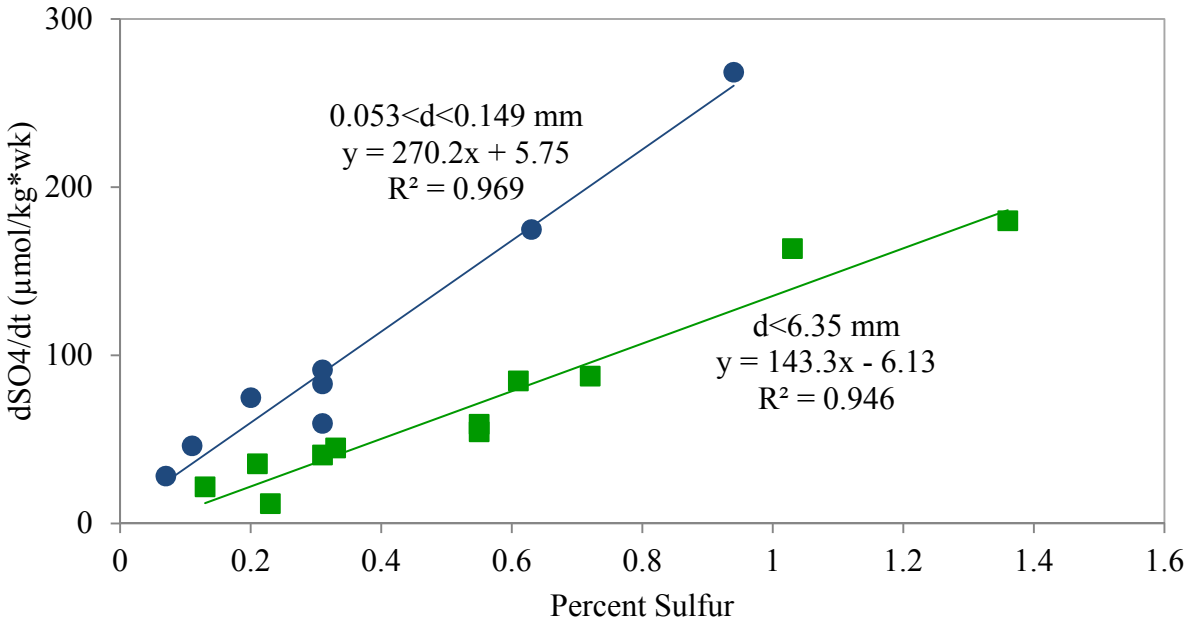


Figure 20. Annual average sulfate release (µmol/kg*wk) vs. %S for Partridge River intrusion samples. Annual rates are shown for Years 2 and 4.
Blue = reactors (0.053<d<0.149), Green = humidity cells (d<6.35 mm)

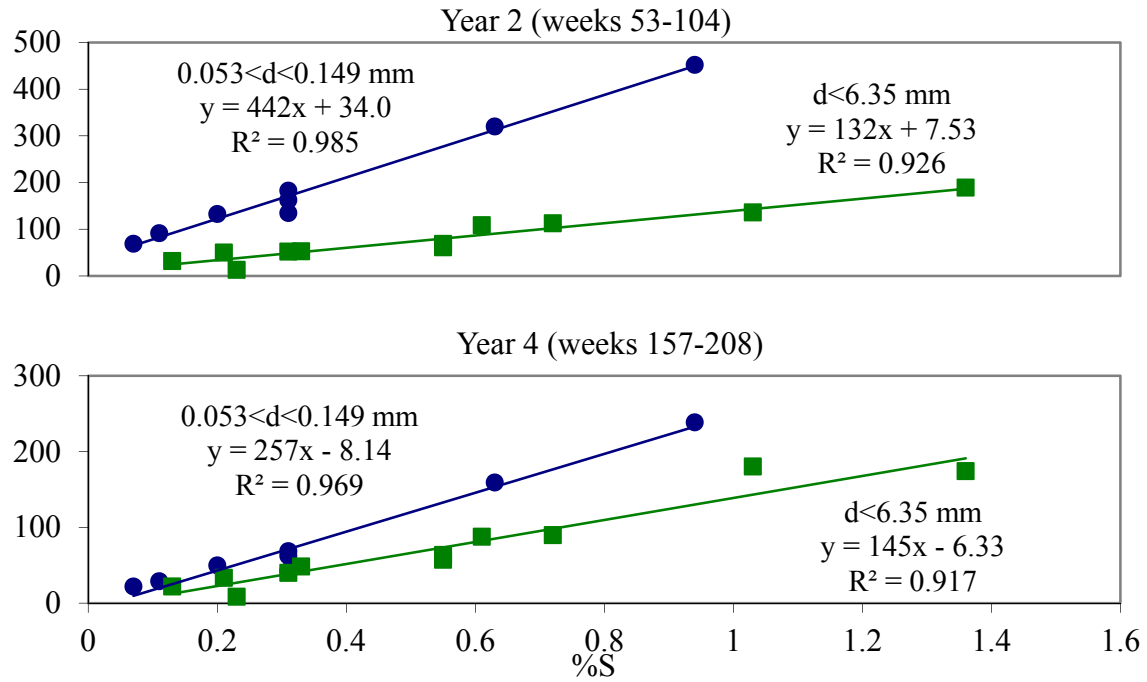


Figure 21. Annual average sulfate release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.948 and 0.925 respectively. Solid line = reactors, dashed line = humidity cells.

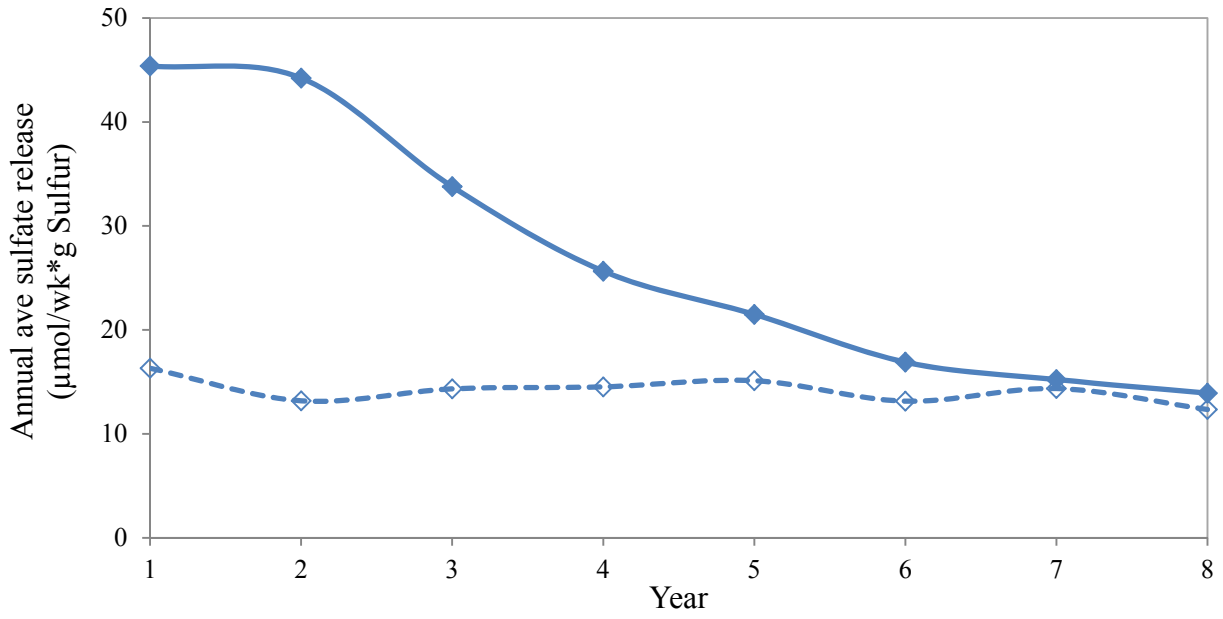


Figure 22. Annual average calcium and magnesium release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.662 and 0.611 for Ca and 0.687 and 0.687 for Mg, respectively. Solid line = reactors, dashed line = humidity cells.

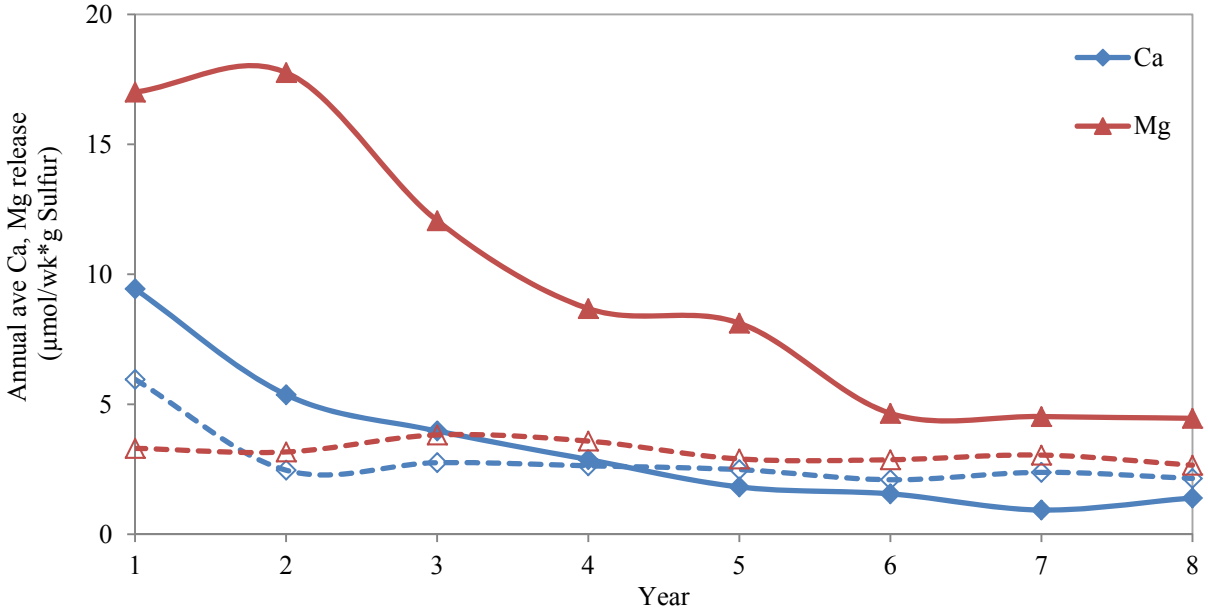


Figure 23. Annual average sodium and potassium release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.810 and 0.769 for Na and 0.269 and 0.248 for K, respectively. Solid line = reactors, dashed line = humidity cells.

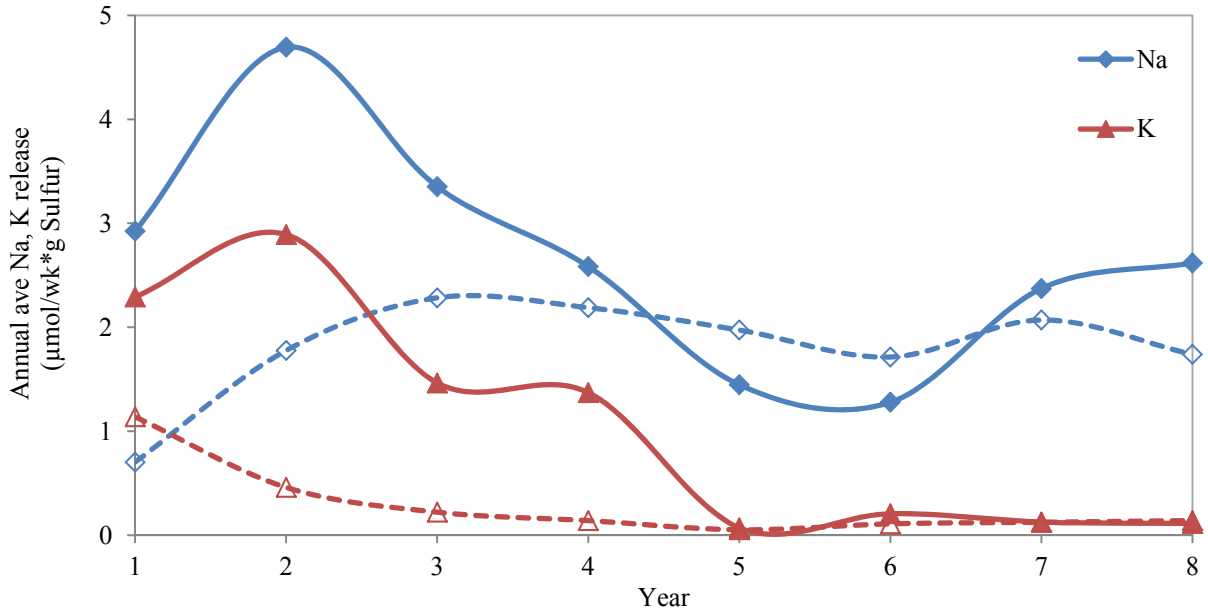


Figure 24. Annual average copper and nickel release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.816 and 0.778 for Cu and 0.925 and 0.886 for Ni, respectively. Solid line = reactors, dashed line = humidity cells.

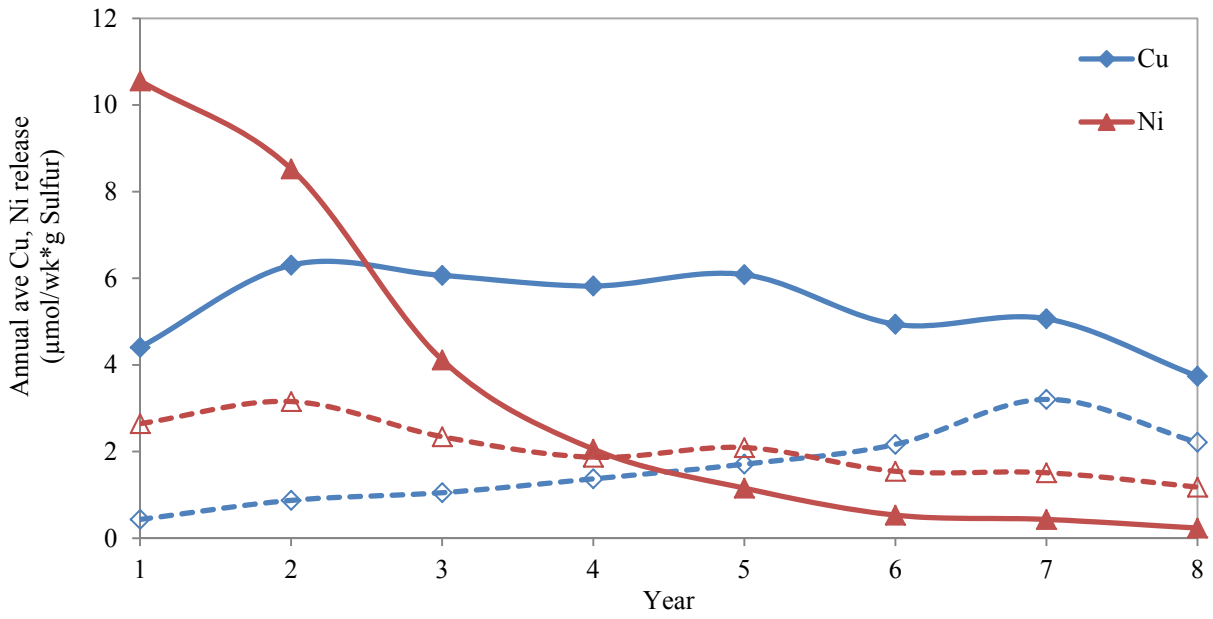


Figure 25. Annual average cobalt and zinc release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.695 and 0.883 for Co and 0.558 and 0.851 for Zn, respectively. Solid line = reactors, dashed line = humidity cells.

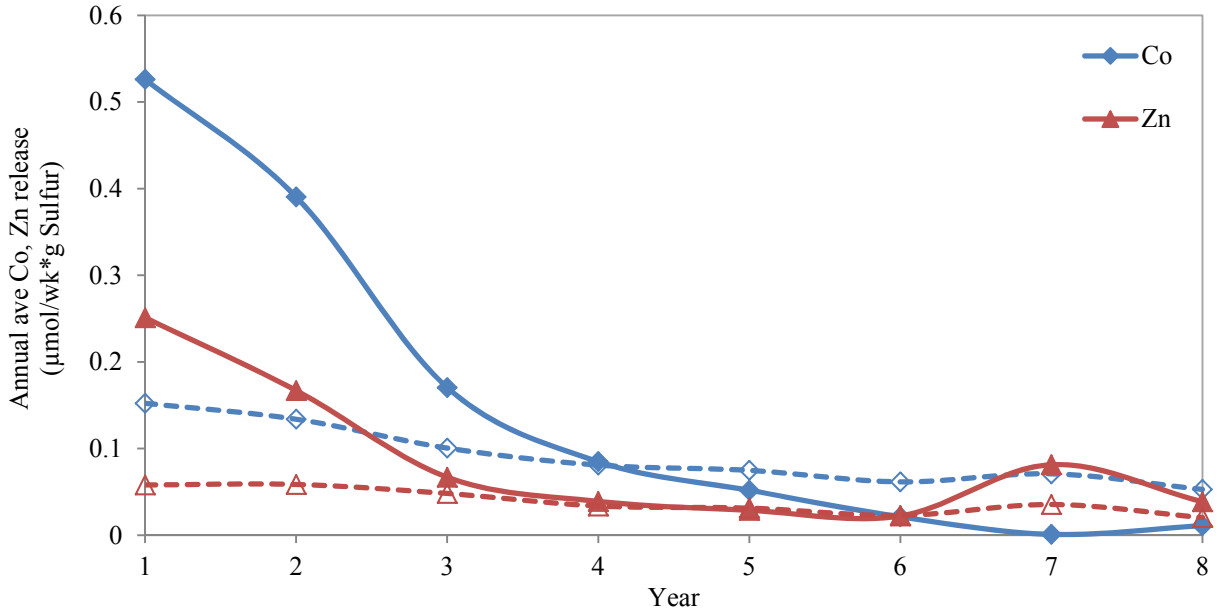


Figure 26. Average nickel and cobalt release vs. %S for reactors (-100/+270 mesh) and humidity cells (-1/4 inch) for weeks 6-104.

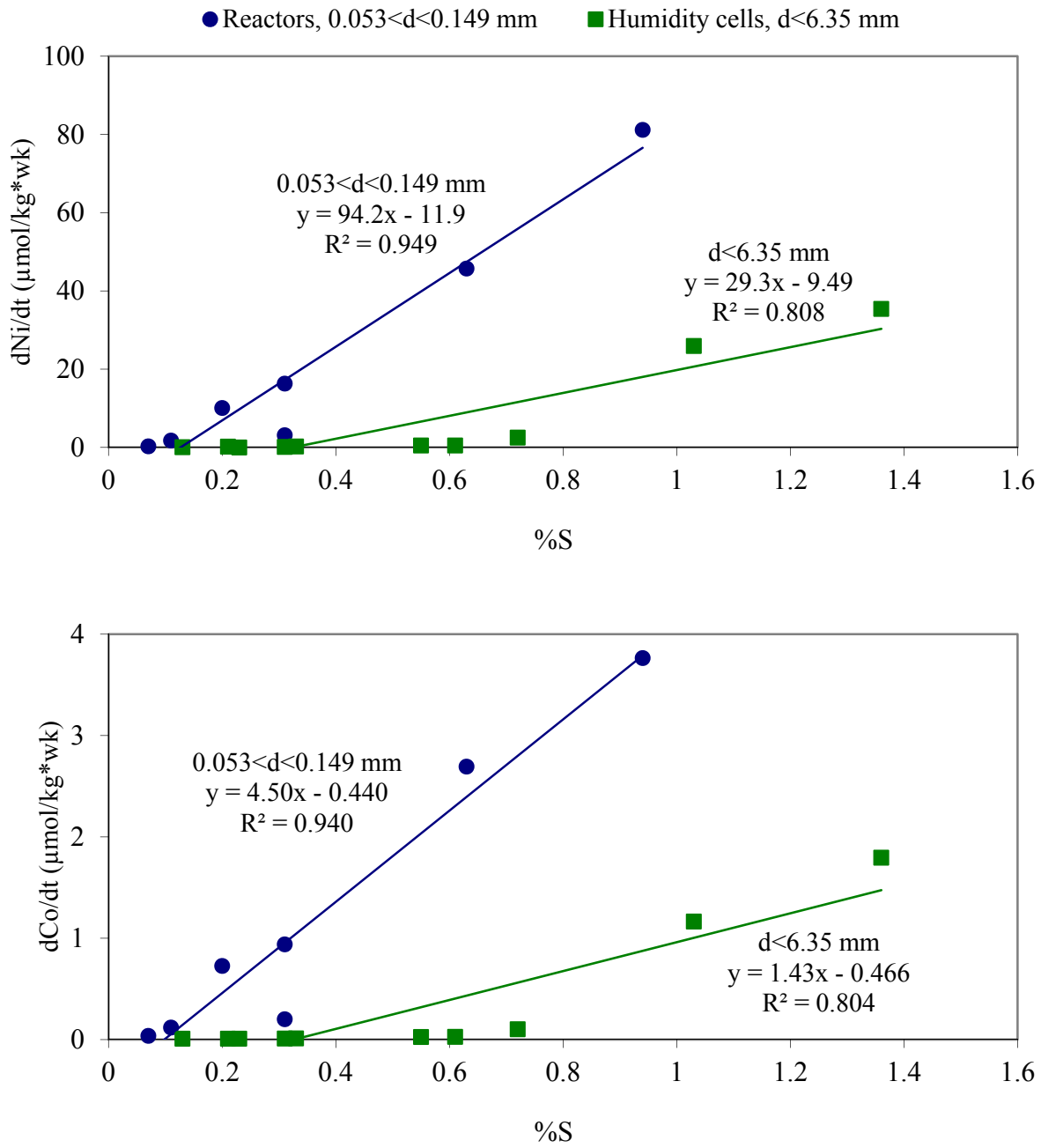


Figure 27. Drainage quality from 75-g samples of 0.31 %S Babbitt Prospect and Dunka Road rocks. Weeks 0-5 were excluded to improve resolution.

Red = Babbitt prospect, Blue = Dunka Road prospect

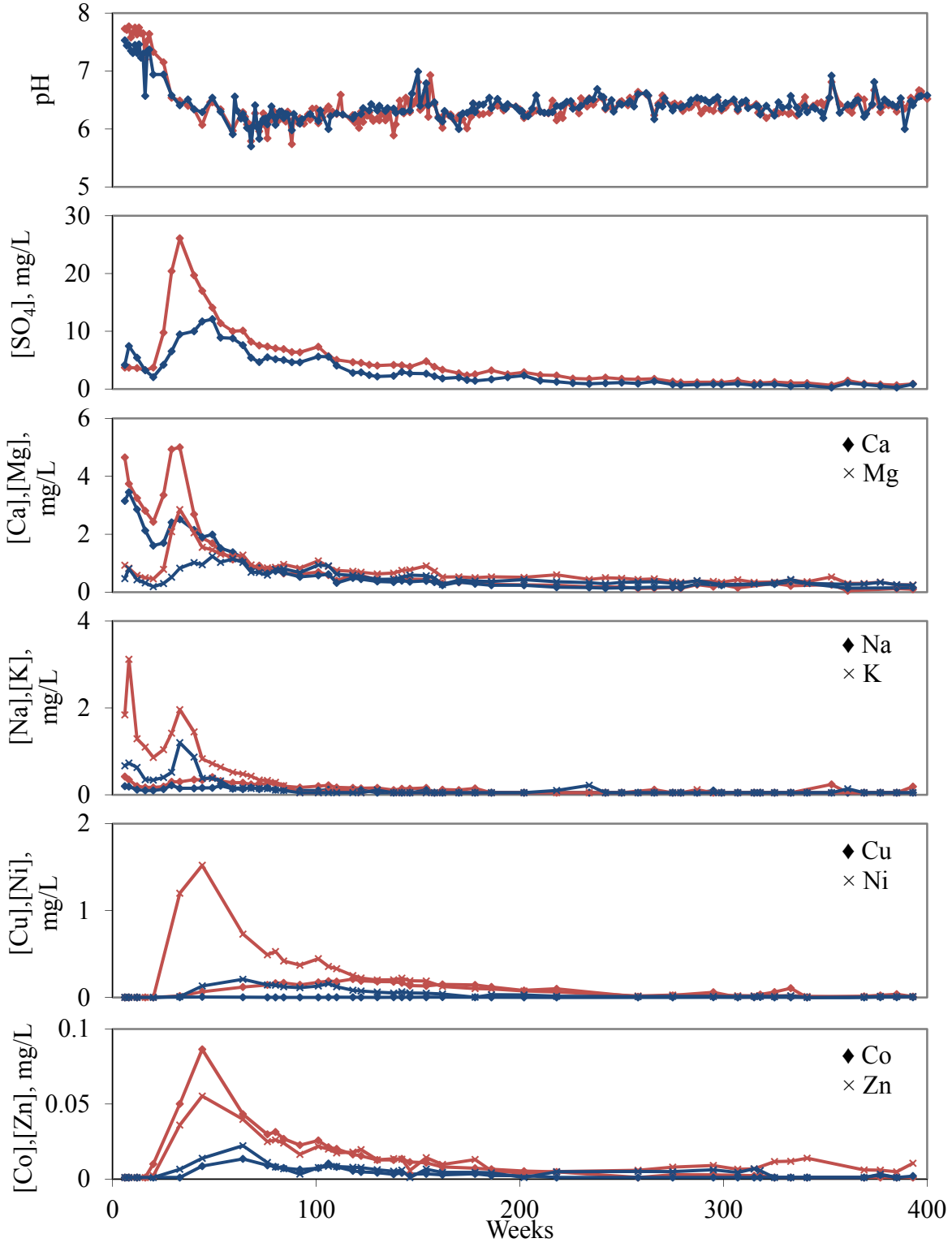


Figure 28. Drainage quality from 1000-g samples of 0.21 %S Babbitt Prospect rock and 0.23 %S Dunka Road rock. Weeks 0-5 were excluded to improve resolution.

Red = Babbitt prospect, Blue = Dunka Road prospect

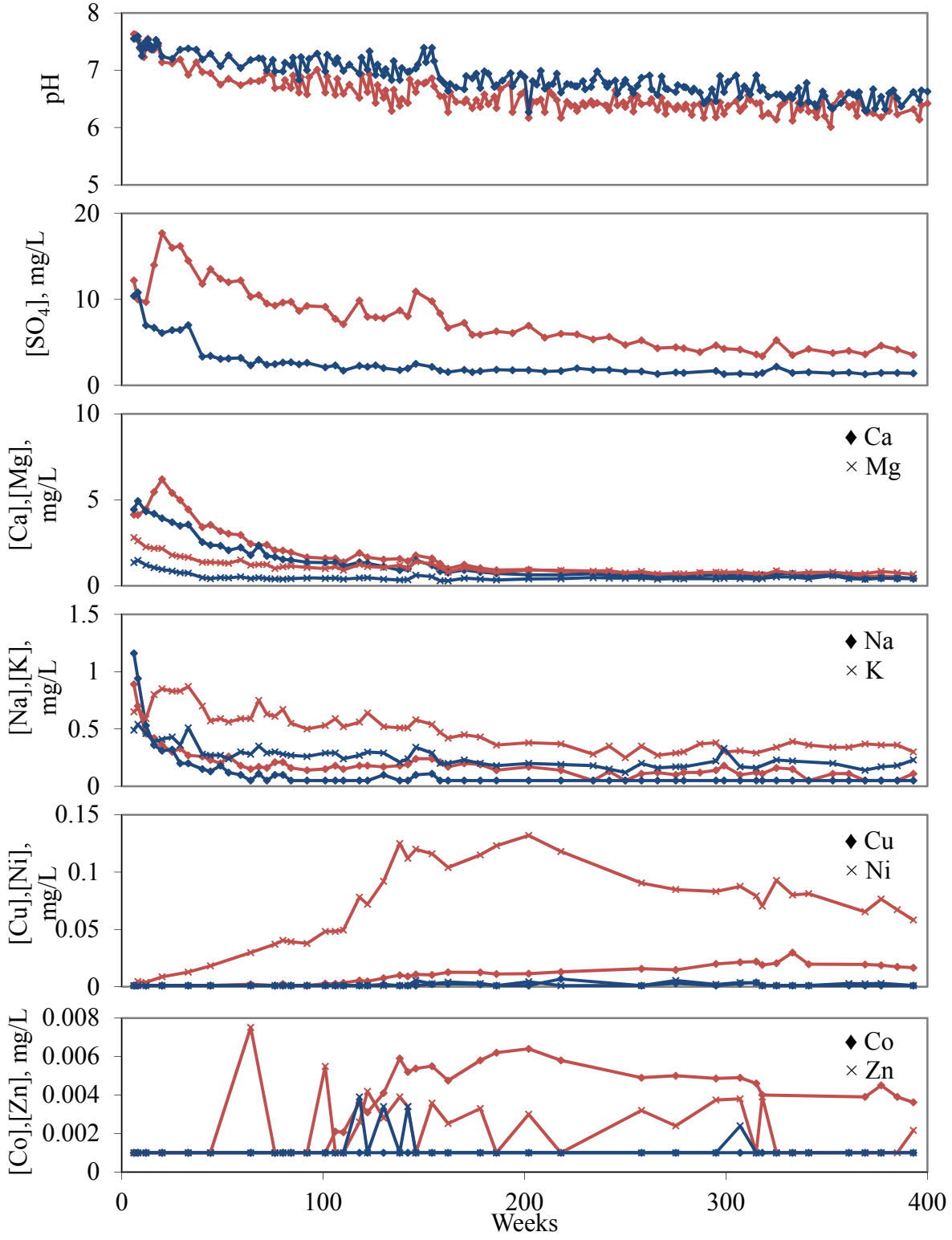


Figure 29. Drainage quality from 1000-g samples of Babbitt prospect (0.33 %S) and Dunka Road rock (0.31 %S). Weeks 0-5 were excluded to improve resolution.

Red = Babbitt prospect, Blue = Dunka Road prospect

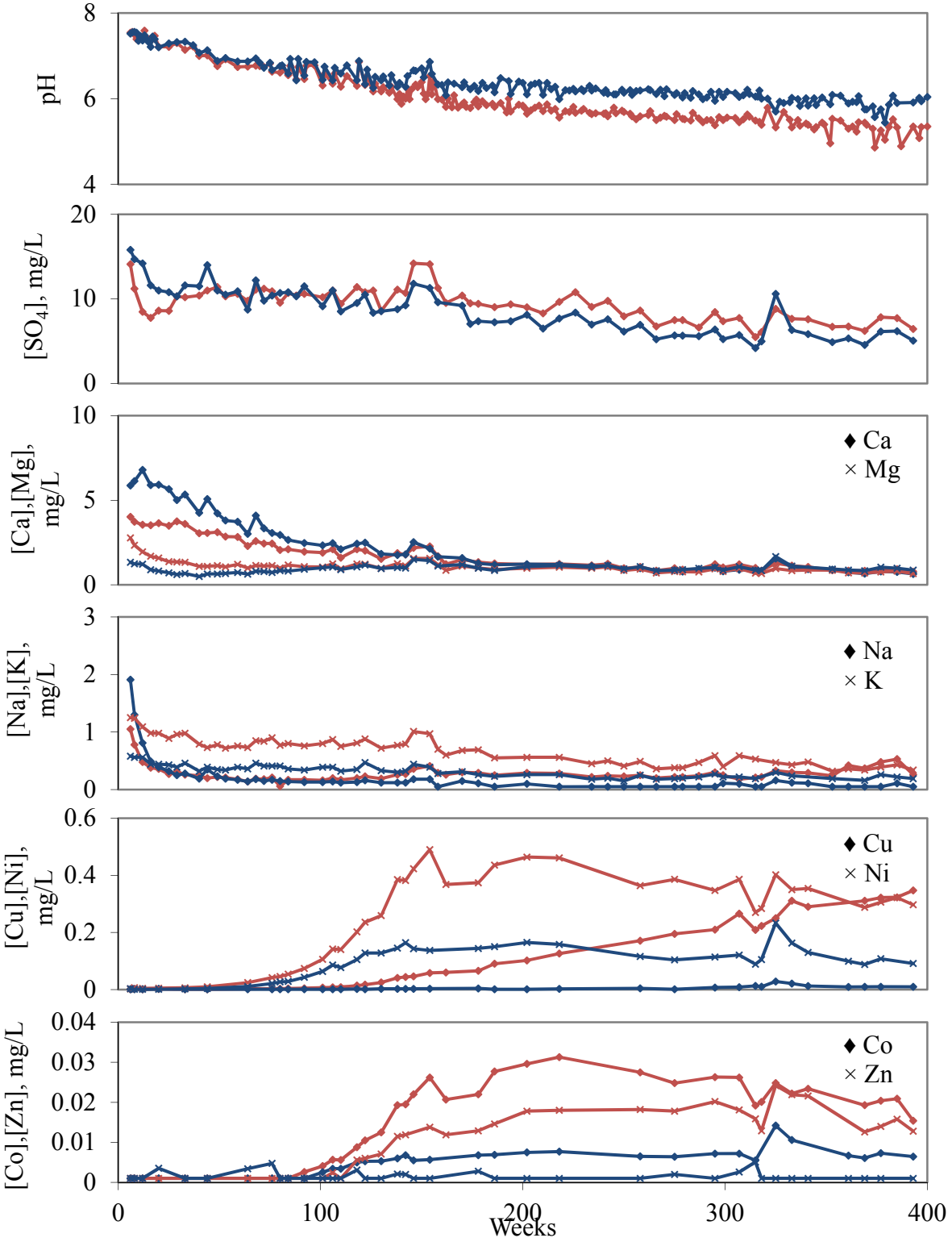


Figure 30. Drainage quality from the 1000-g sample of 0.55 %S Babbitt Prospect and 0.61 %S Dunka Road rock. Weeks 0-5 were excluded to improve resolution.

Red = Babbitt prospect, Blue = Dunka Road prospect

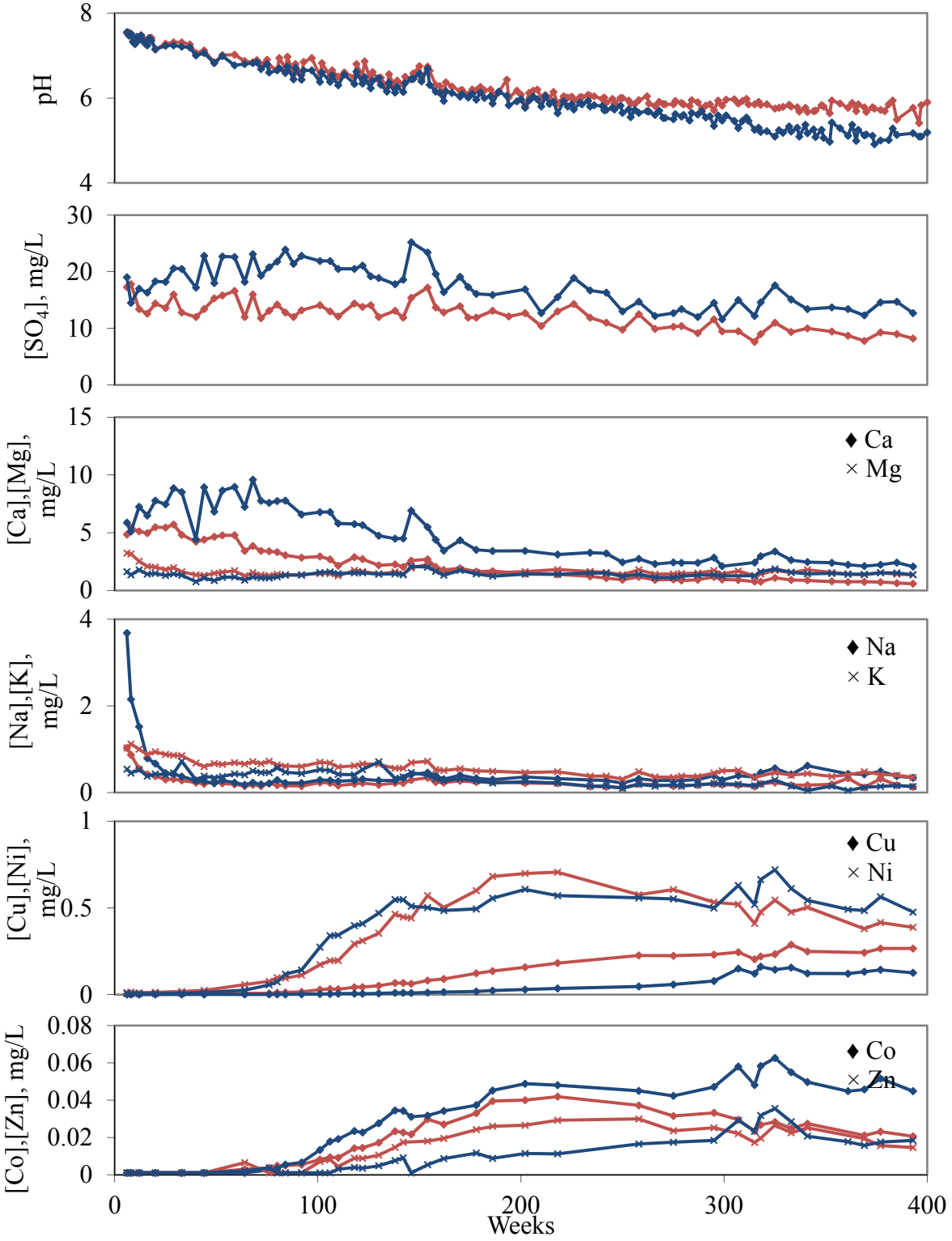


Figure 31. Drainage pH and sulfate, calcium, and magnesium release (µmol/kg*wk) vs. time from the 75-g sample of 0.20 %S Babbitt Prospect waste rock and unsaturated sample of 0.20 %S Duluth Complex tailings rock. Weeks 0-5 were excluded to improve resolution.

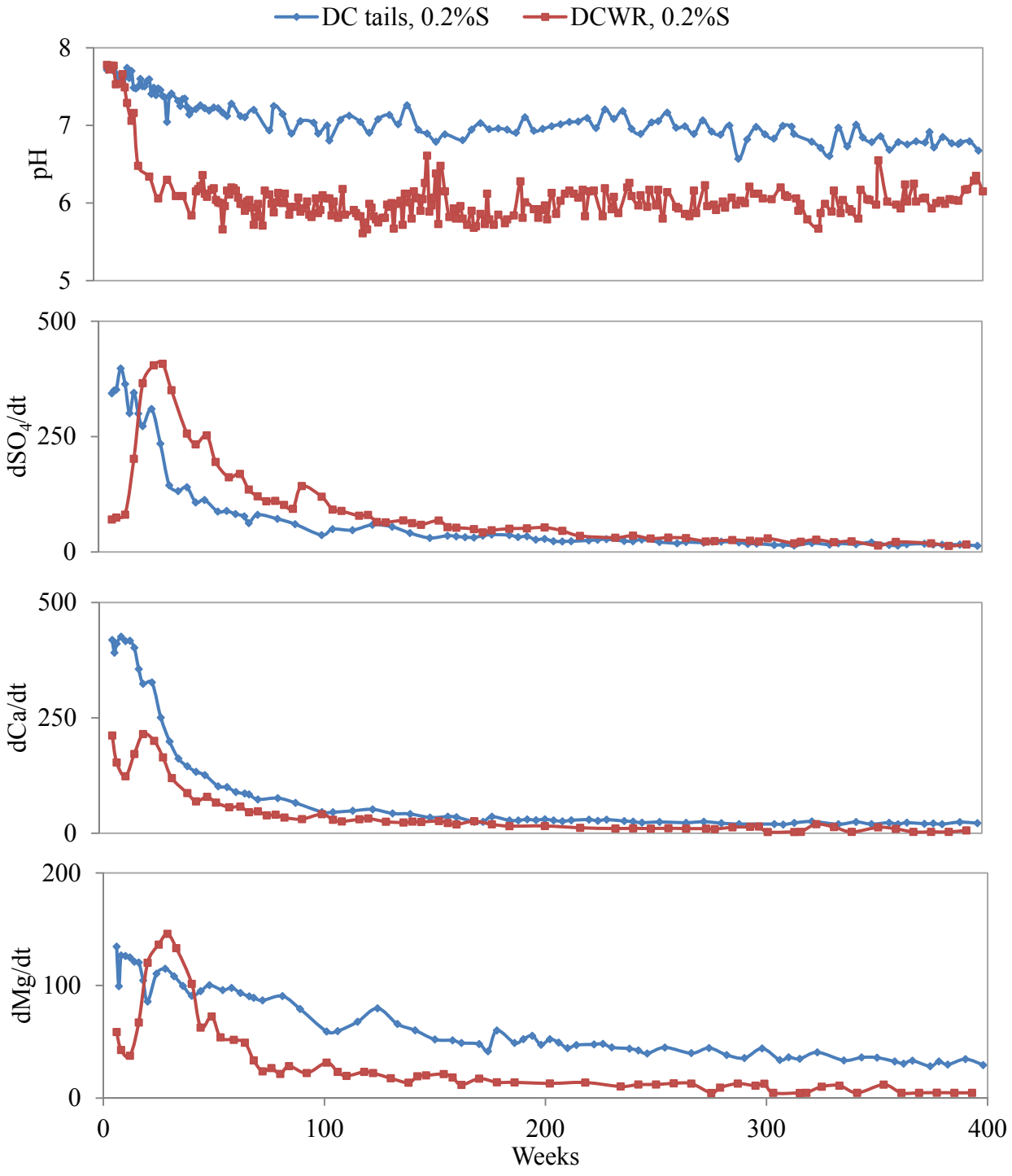


Figure 32. Heavy metal release (µmol/kg*wk) vs. time from the 75-g sample of 0.20 %S Babbitt Prospect waste rock and unsaturated sample of 0.20 %S Duluth Complex tailings rock. Weeks 0-5 were excluded to improve resolution. Note logarithmic scale on y-axis.

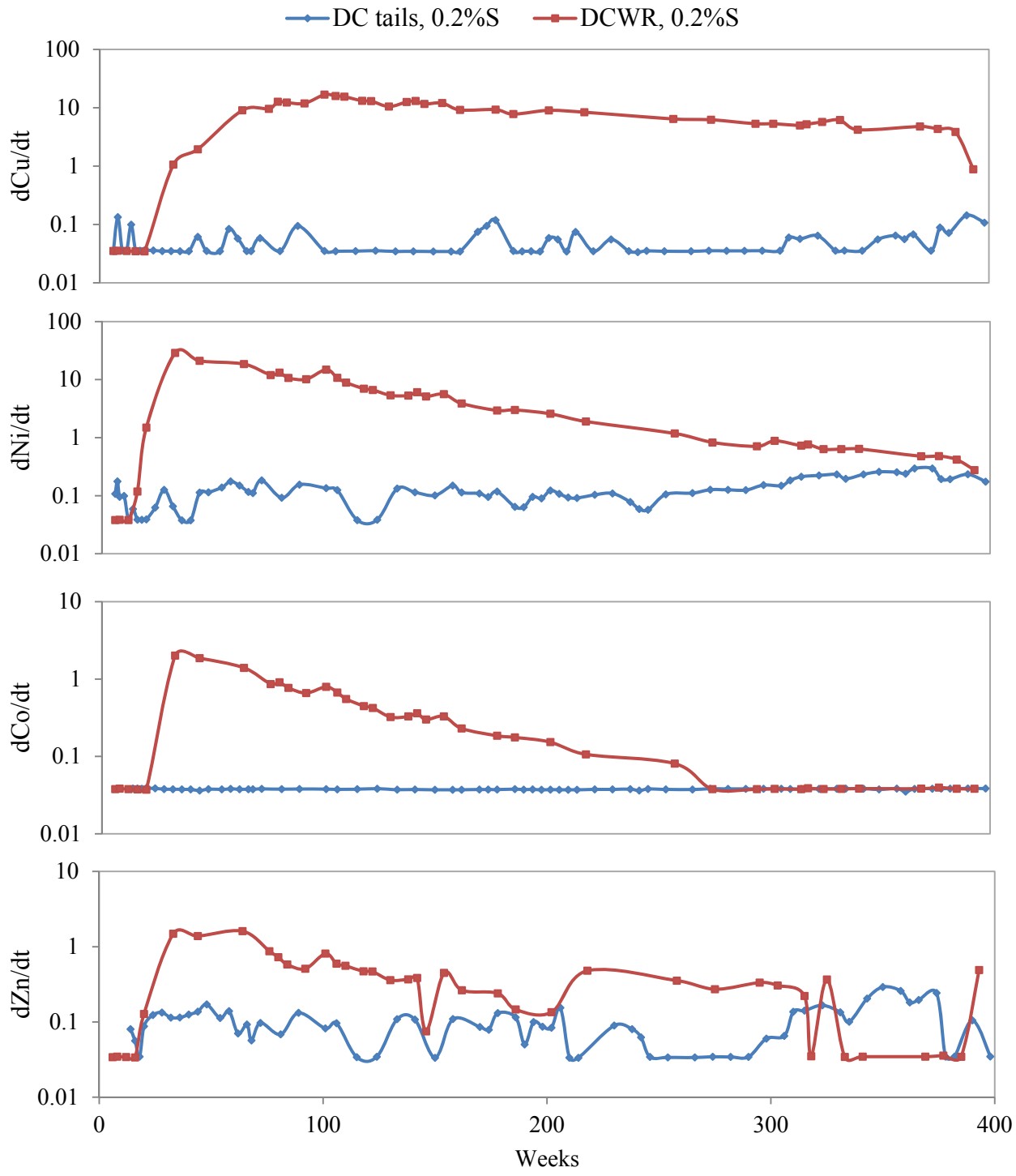


Figure 33. Copper content vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI.

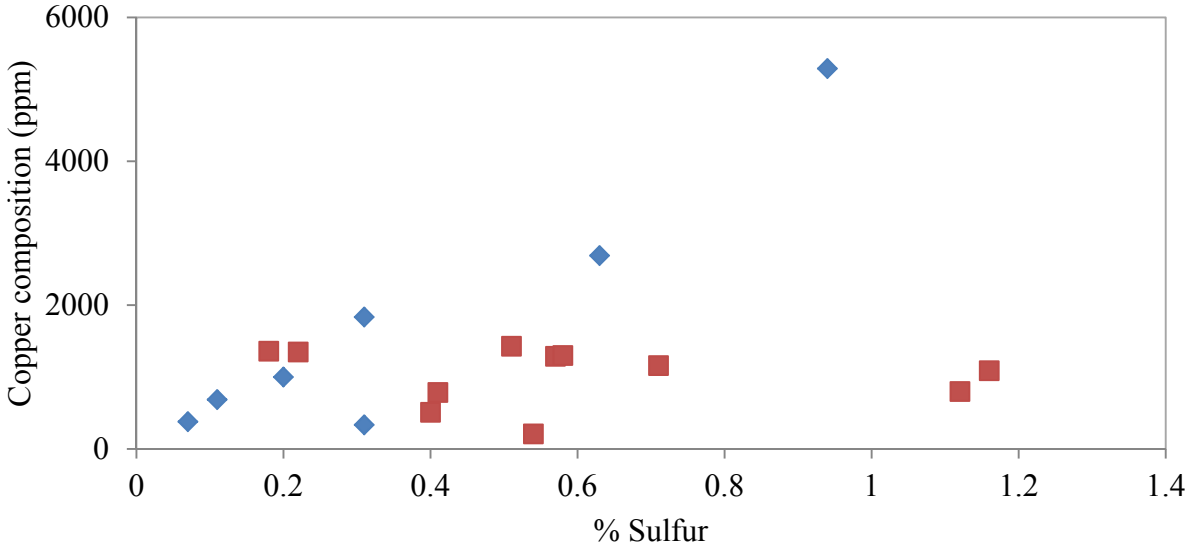


Figure 34. Nickel content vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI.

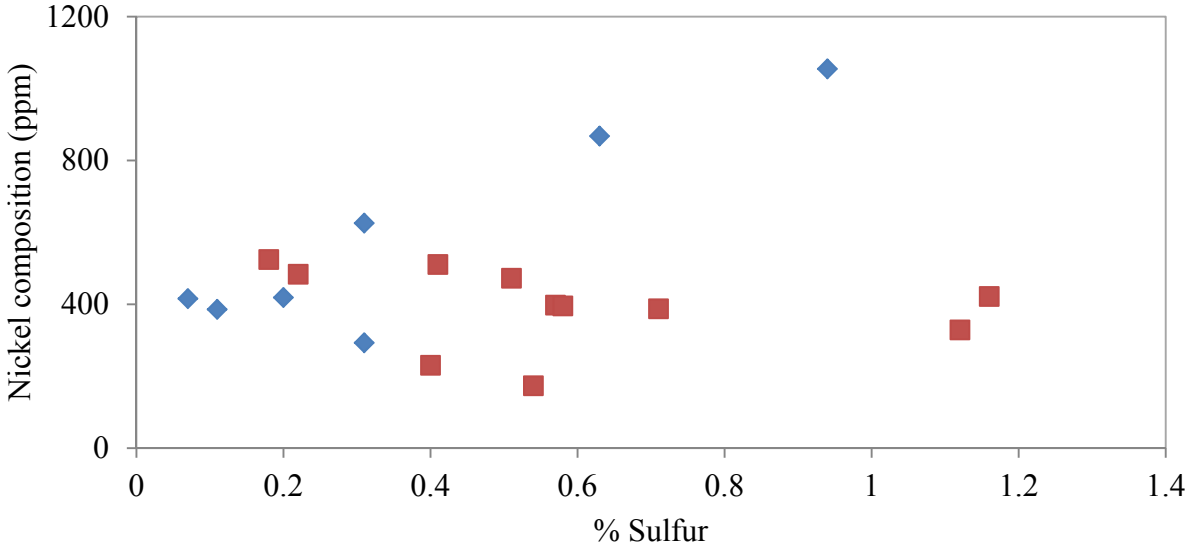


Figure 35. Zinc content vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI.

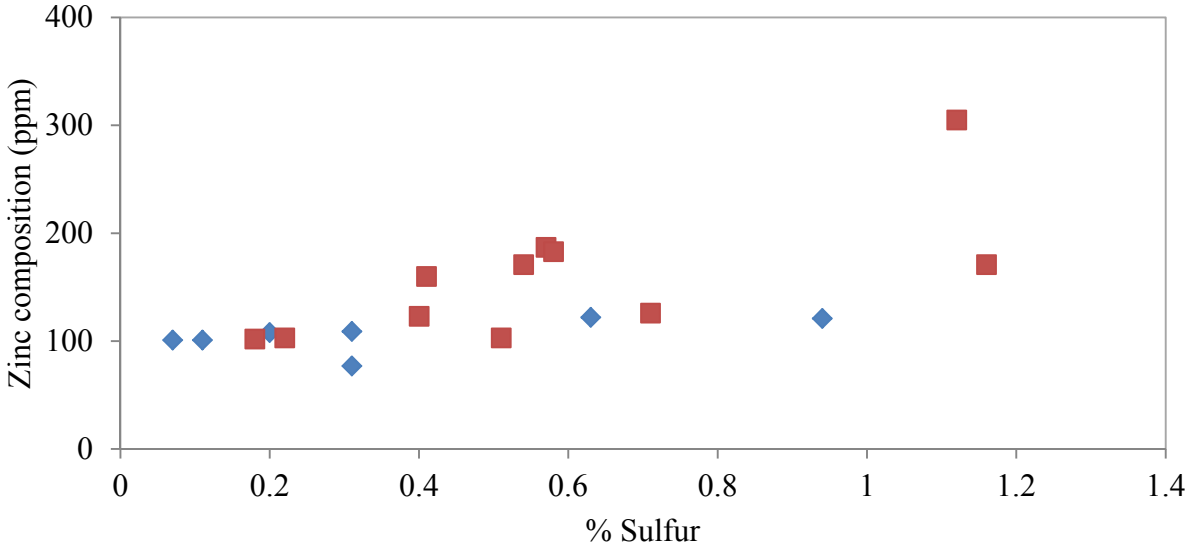


Figure 36. Cobalt content vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI.

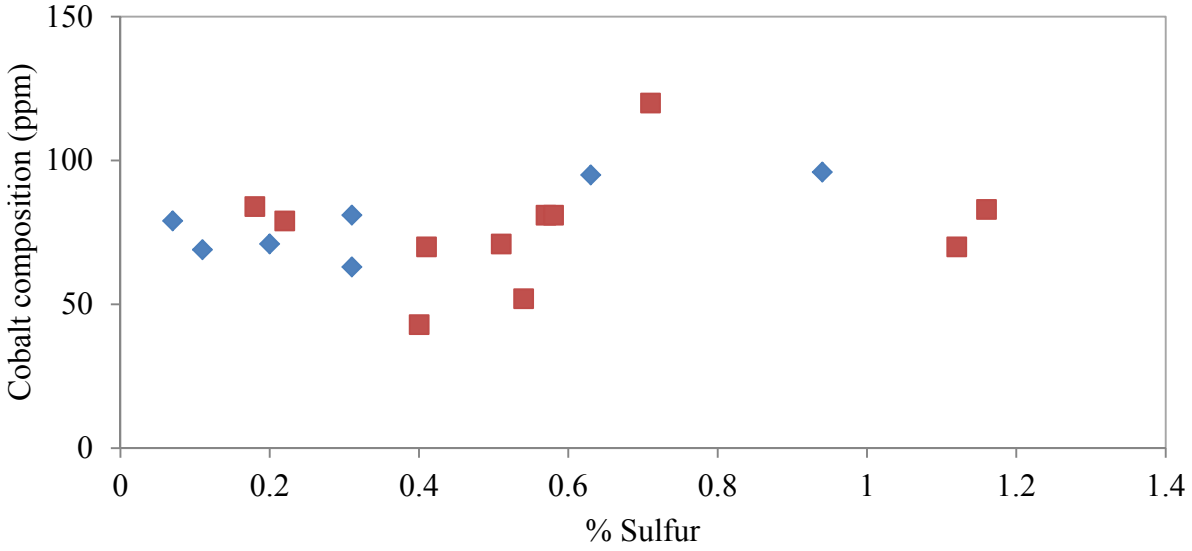


Figure 37. Minimum pH vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples through 400 weeks. Blue = PRI, Red = SKI

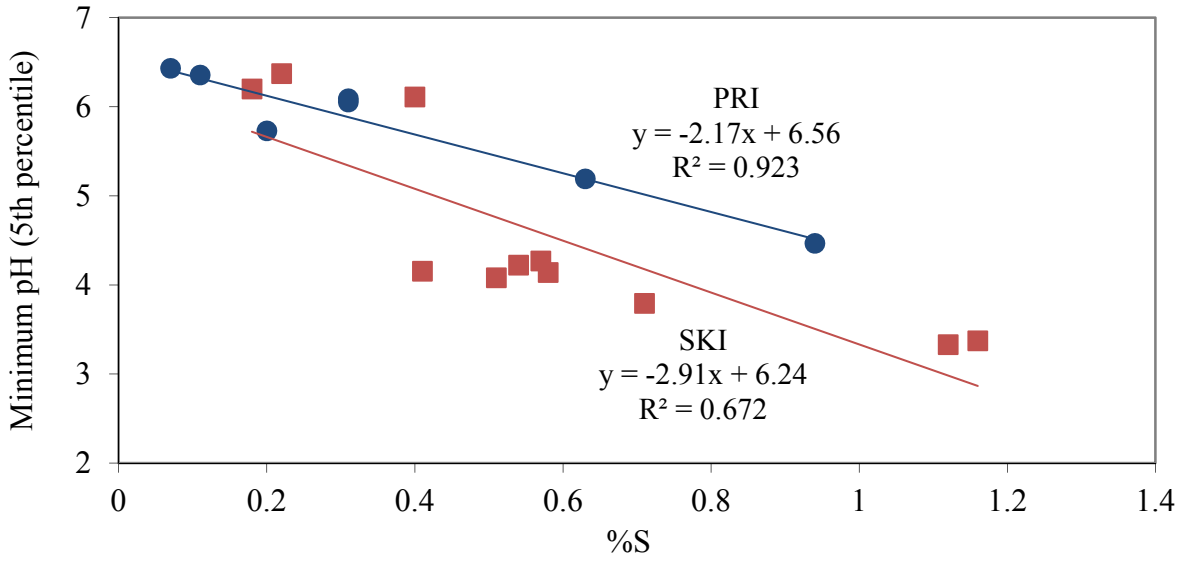


Figure 38. Week of minimum pH vs. %S for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI. Higher sulfur SKI samples typically took 250-350 weeks to reach minimum pH values.

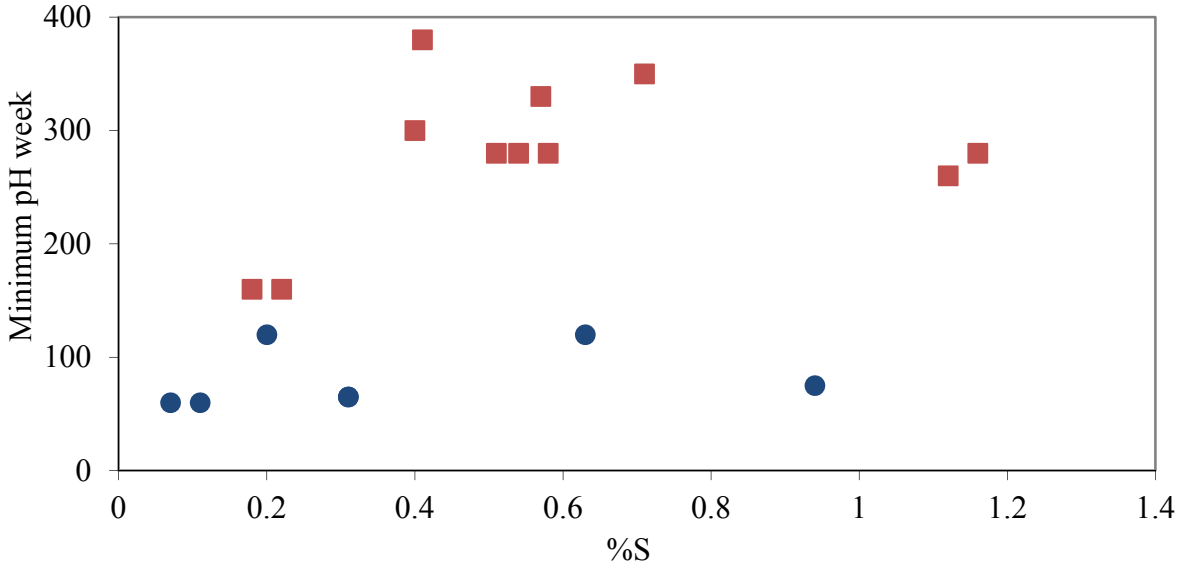


Figure 39. Annual average sulfate release (µmol/kg*wk) vs. %S for Partridge River (PRI) intrusion and South Kawishiwi intrusion (SKI) samples. Annual rates are shown for Years 2 and 6. Blue = PRI, Red = SKI

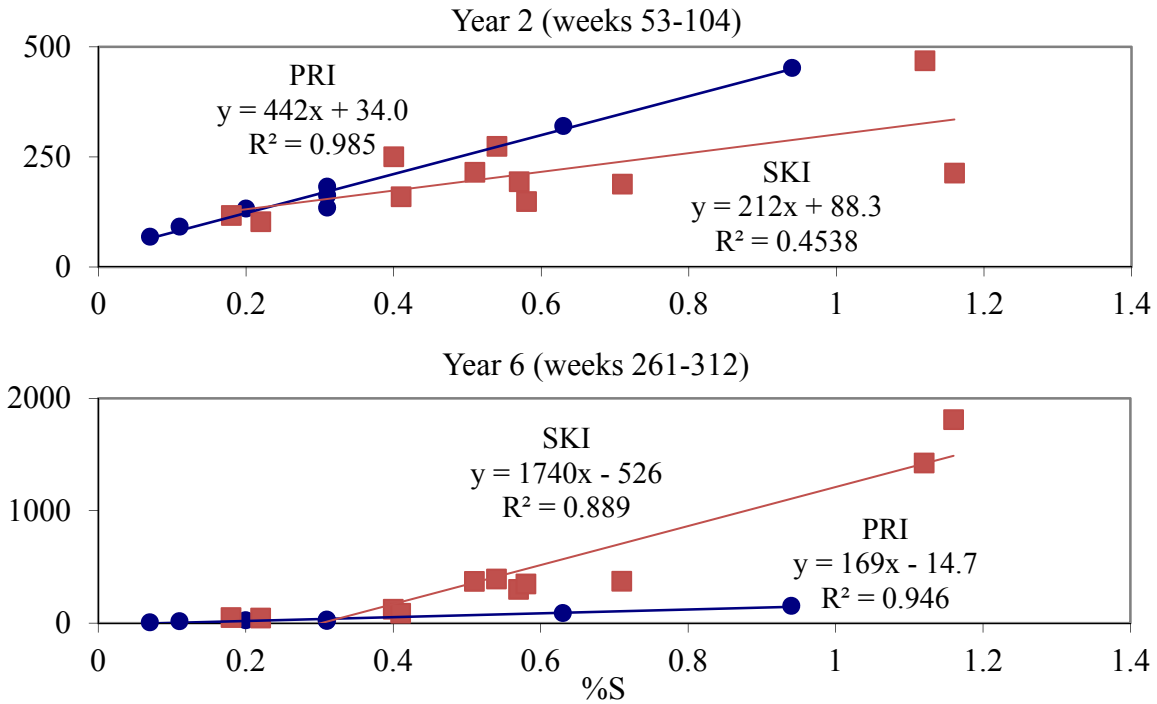


Figure 40. Annual average sulfate release per gram of sulfur vs. time between Babbitt and Dunka Road prospect rock (PRI) and Dunka Mine rock (SKI).

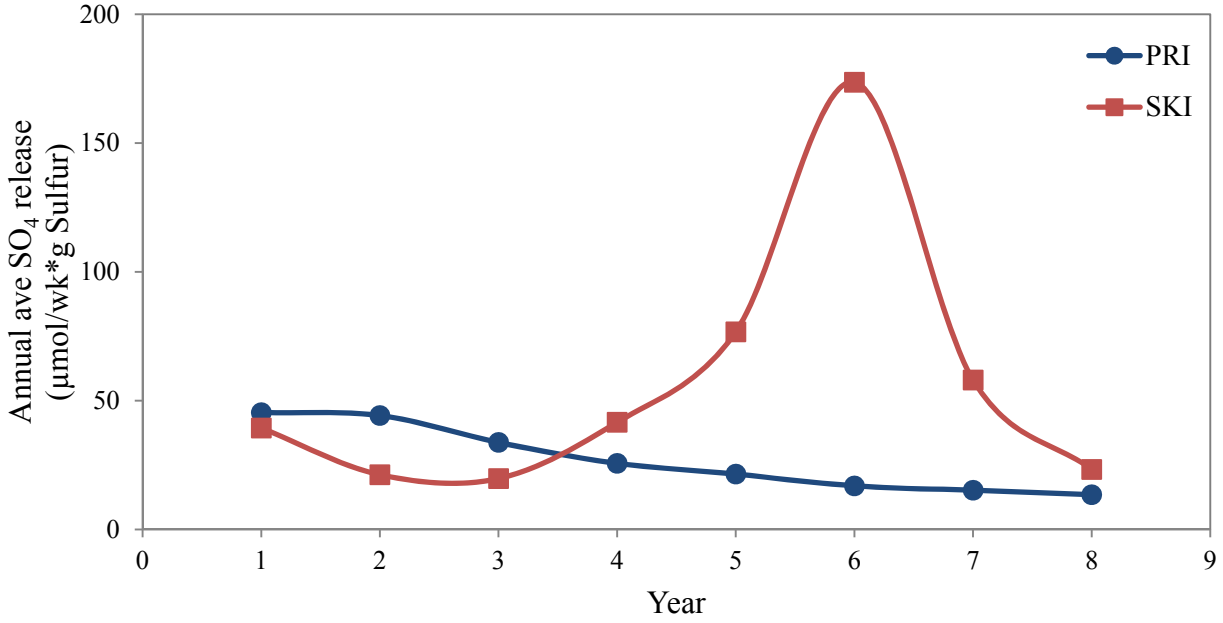


Figure 41. dNi/dt vs %S for 0.053 < d < 0.149 mm Partridge River (PRI) and South Kawishiwi (SKI) intrusion samples, weeks 6-74.

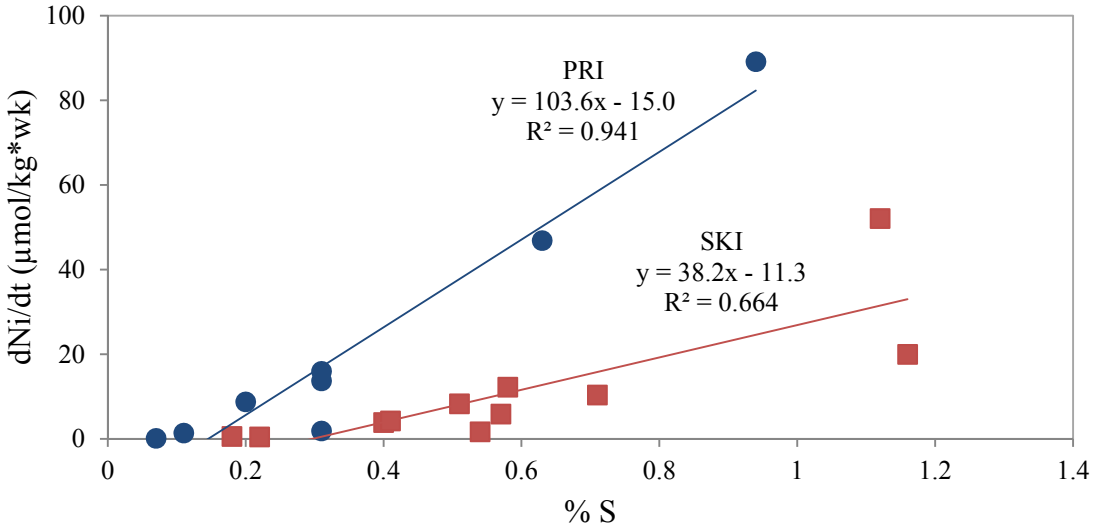


Figure 42. dCu/dt vs %S for 0.053 < d < 0.149 mm Partridge River (PRI) and South Kawishiwi (SKI) intrusion samples, weeks 6-74. Note logarithmic scale on y-axis.

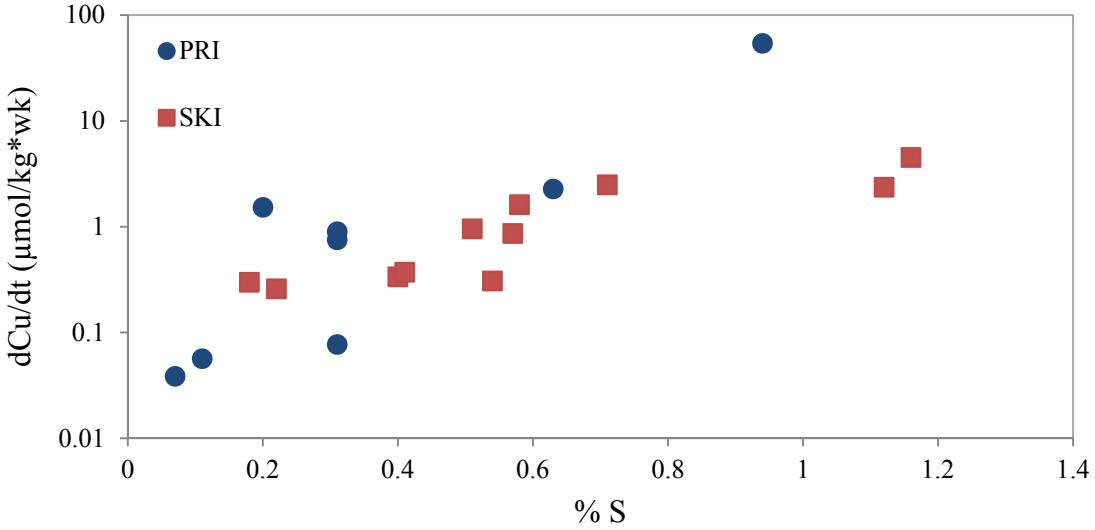


Figure 43. dCo/dt vs %S for 0.053 < d < 0.149 mm Partridge River (PRI) and South Kawishiwi (SKI) intrusion samples, weeks 6-74.

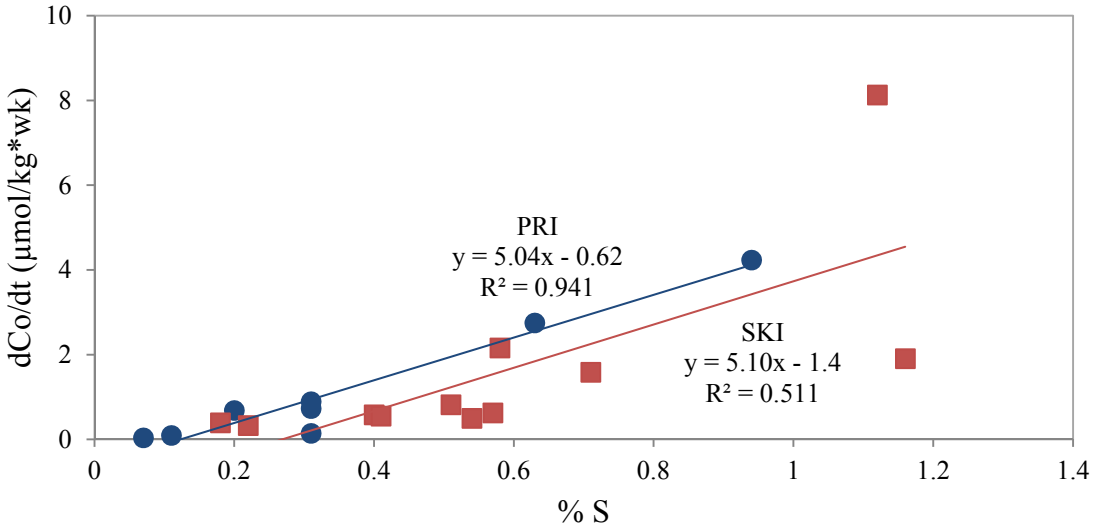
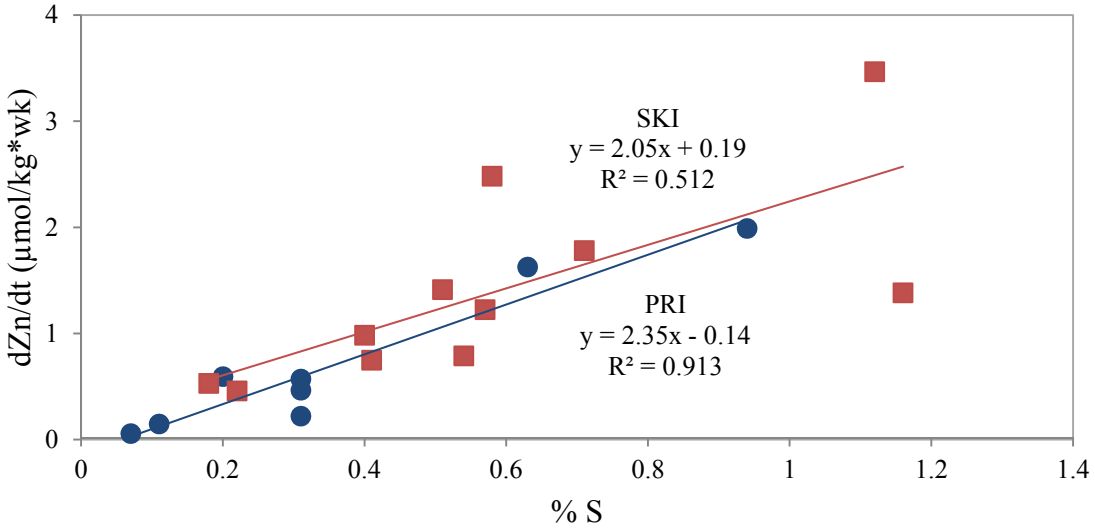


Figure 44. dZn/dt vs %S for 0.053 < d < 0.149 mm Partridge River (PRI) and South Kawishiwi (SKI) intrusion samples, weeks 6-74.



Attachment 16

Wenz Z, Lapakko K, Antonson D (2013) Rock composition, leachate quality and solute release as a function of particle size for three waste rock types: an 18-year laboratory experiment. MN Dept. Nat. Resour., Div. Lands and Minerals. St. Paul, MN.

Dissolution of Duluth Complex Rock from the Babbitt and Dunka Road Prospects: Eight-year laboratory experiment

June 2013

Minnesota Department of Natural Resources
Division of Lands and Minerals
500 Lafayette Road, Box 45
Saint Paul, MN 55155-4045

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June 2013

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0. EXECUTIVE SUMMARY

The *research reported will be applied* in the environmental review and permitting of future nonferrous mines in Minnesota, particularly those in the Duluth Complex. These regulatory activities require prediction of solute release from waste rock during mine operation and in the decades and centuries following mine closure. As a foundation for such predictions, the reactions occurring in waste rock stockpiles and variables affecting these reactions must be understood. This knowledge and a description of the proposed method of waste rock disposal are necessary to 1) design appropriate tests for predicting the release of potential pollutants from waste rock, 2) extrapolate results of such tests to field conditions, and 3) if necessary, implement mitigation measures that will ensure that water resources are not adversely impacted. In the present project, laboratory studies were conducted to examine solute release and variables affecting this release from waste rock from the Partridge River intrusion of the Duluth Complex. The studies provide scientific and technical information necessary to address questions inherent to environmental review and permitting of nonferrous metal mines. The project also provides empirical data describing eight years of rock dissolution for comparison with and extrapolation of data submitted for environmental review and permitting by companies proposing mining in the Duluth Complex.

To *summarize* the laboratory work, dissolution tests were initiated in August 2003 on 10 well-characterized (particle size, chemical composition, mineral content, mineral chemistry) Duluth Complex rock samples from the Babbitt and Dunka Road prospects of the Partridge River intrusion. Dissolution tests were conducted on 10 coarse samples ($d < 6.35$ mm = 0.25 in.; $0.13 \leq \%S \leq 1.36$) and seven fine samples ($0.053 < d < 0.149$ mm; $0.07 \leq \%S \leq 0.94$). The experiments remain in progress and the following report presents data generated through week 400. This report summarizes results and provides a conceptual structure to frame these results.

In general, drainage pH tended to decrease and release rates of heavy metals (copper, nickel, cobalt, zinc) and major cations (calcium, magnesium, sodium, potassium) tended to increase with increasing sulfur content. These results were consistent with the previously reported 1) dependence of sulfide mineral oxidation (as reflected by sulfate release) and the attendant acid production on the amount of sulfide minerals present and 2) importance of sulfide mineral oxidation as a driving force for release of heavy metals and major cations. More specifically, numerous relationships between drainage quality and both solid-phase sulfur content and dissolution time were identified and quantified to quantitatively describe solute release and its dependence on solid-phase composition and dissolution time.

For the 10 coarse samples, drainage pH decreased with increasing sulfur content, with a minimum value of 4.1. For all but the two highest sulfur samples, pH decreased throughout the eight-year experiment. Rates of sulfate release, reflecting sulfide mineral oxidation, increased with increasing sulfur content. The strong correlation between average annual rates for the 10 samples and corresponding sulfur contents was quantified for each year using regression analysis ($0.91 < r^2 < 0.94$). This analysis was used to derive composite annual sulfate release rates (per gram sulfur), which were found to decrease slightly over the eight-year experiment.

Release rates for copper, nickel, cobalt, and zinc also correlated highly with sulfur content (typically $0.80 < r^2 < 0.95$). Magnesium was the dominant cation released, followed by calcium, and sodium. Their releases tended to parallel that of sulfate, reflecting silicate minerals dissolving to neutralize acid produced by iron sulfide mineral oxidation.

Similar analysis was conducted on data generated in the seven fine-grained samples. As with the larger particles, drainage pH decreased and release rates for sulfate, heavy metals, and major cations increased with increasing sulfur content. Although the relationship between minimum drainage pH and sulfur content was similar to that for the larger particles, pH from the finer samples declined more rapidly, reaching minimums between weeks 60 and 120. Sulfate release from the fine particles were initially three times those from the coarse particles, reflecting the greater sulfide mineral surface area per unit mass of the fine particles, and this rapid release contributed to the aforementioned early pH depression. Sulfate release rates from the fine particles decreased fairly quickly over time and by year seven equaled those from the coarse particles, for which rates decreased more slowly. The rapid decline of rates from the fine particles was attributed to sulfur depletion and probable coating of sulfide minerals.

Dissolution testing of Dunka Road samples was limited to one sample in the 0.053-0.149 mm size fraction and three samples in the $d < 6.35$ mm fraction. The most notable difference between Dunka Road and Babbitt prospect rock was elevated heavy metal release from the latter. Whereas some of the difference could be ascribed to higher metal content of the Babbitt prospect rock examined, differences between prospects in total sulfide mineral oxidation rates and drainage pH also played a role. Because the number of samples for comparison was limited, these results should be viewed as preliminary.

The Partridge River rock samples yielded higher sulfate rates than Partridge River tailings of similar sulfur content, due to greater exposure of sulfide minerals in the rock samples. Heavy metal release from the rock was also higher than that from the tailings. This was due to the aforementioned elevated sulfide mineral exposure, higher heavy metal contents, and lower drainage pH associated with the rock samples. Drainage pH from the Partridge River rock tended to be about 0.7 units higher than that from previously tested South Kawishiwi rock of similar sulfur content. This was attributed to the higher heavy metal content (and consequent lower iron sulfide content) of the Partridge River rock examined in the present study.

In *conclusion,* for both ten coarse and seven fine samples, testing generated data describing drainage pH and release of sulfate, heavy metals, and major cations over a period of eight years. Drainage pH and solute release were correlated with solid-phase sulfur content and time of dissolution. Furthermore, the two data sets allowed comparison of solute release from the two size fractions, consequently providing insight on the influence of particle size. In total, the project 1) quantifies the dependence of drainage pH and release rates of sulfate, heavy metals, and major cations on solid-phase sulfur content, dissolution time, and particle size; 2) interprets results with respect to mineral dissolution reaction; and 3) provides context for the results generated by comparisons with dissolution data on Partridge River tailings and South Kawishiwi intrusion rocks. Thus, the project will help inform design of predictive test programs for

environmental review and permitting and interpretation of results generated by such program. It also provides a robust empirical data base to provide a benchmark for data submitted.

Additional work should be conducted to further the benefits generated to date. This includes continued dissolution testing to confidently define long-term trends, analysis of leached solids to increase understanding of chemical reactions controlling solute release (e.g. chemical precipitation and adsorption), and more detailed analysis of the present data, including calculation of mineral dissolution rates. Calculation of mineral dissolution rates will facilitate application of the data generated to a wider range of mineralogical compositions of Duluth Complex rock.

1. INTRODUCTION

1.1. Background

Several areas of the Duluth Complex are presently being considered for mineral resource development. Prior to development, potential mine wastes must be characterized (particle size, chemistry, mineralogy, petrology) and well-characterized samples must be subjected to dissolution studies. These studies assess the potential of mine wastes (e.g. waste rock, mine walls, tailings, and processing residues) to adversely impact water resources. The Minnesota Department of Natural Resources is conducting dissolution studies on well-characterized samples of South Kawishiwi intrusion (SKI) Duluth Complex rock. The period of record for some of these samples exceeds 24 years. Although there is some homogeneity in the bulk mineral content of rock throughout the Duluth Complex, the degree to which compositional variations within the complex might affect the water quality resulting from rock dissolution has not been determined.

The present project examines laboratory dissolution of rock from a bulk sample in the Babbitt prospect and from drill core samples from the Dunka Road prospect, both in the Partridge River intrusion (PRI) of the Duluth Complex. These samples might simulate waste rock or mine walls resulting from mineral resource development. Two different methods of dissolution testing were used, one of which allows comparison of results from the Partridge River intrusion samples with those from long-term dissolution studies conducted on rock from the Dunka Mine in the South Kawishiwi intrusion of the Duluth Complex. Thus, the present project addresses a comparison of test methods and the dissolution of rock from different locations in the Duluth Complex. The present report summarizes data generated over the first 400 weeks of dissolution testing.

1.2. Duluth Complex Geology Summary

Miller et al. (2002) reported on the mineral occurrences and geology of the Duluth Complex, and the following summarizes information on the Partridge River and South Kawishiwi intrusions from that report. Dissolution of rocks from these two intrusions is the focus of the present report.

The Partridge River intrusion hosts at least four copper-nickel prospects (Babbitt, Dunka Road, Wetlegs, Wyman Creek) and at least seven potential iron-titanium prospects (Miller et. al., 2002) (Section 17, Longear, Longnose, Wyman Creek, Section 22, Skibo, Water Hen) (Figure 1). The Section 22, Skibo, and Water Hen prospects all lie south, and within 12 miles, of the Wyman Creek prospect and are not depicted in Figure 1.

The Partridge River intrusion consists mainly of troctolitic rocks that form an arc-shaped exposure (Figure 1). It is estimated to be 2.5 km thick, and is underlain by the Virginia Formation slates and greywackes and the Biwabik Iron Formation. The basal 900 meters of the intrusion has been subdivided into seven stratigraphic units (Severson and Hauck 1990, 1997; Geerts 1991; Severson 1991, 1994) that can be correlated over a strike length of 24 km. These units dip 10 to 20 degrees to the southeast and have been described in detail due to the presence

of copper-nickel (Cu-Ni) mineralization. Rocks overlying the Partridge River intrusion include other Duluth Complex rocks and metamorphosed basalts (probably North Shore Volcanic Group). These rocks are also present in the interior of the intrusion as large inclusions.

The contact zone between the Partridge River and South Kawishiwi intrusions is poorly exposed and sparsely drilled. The seven basal units become unrecognizable in the contact zone. This contact is complicated by the presence of a fault (Grano fault of Severson 1994) that was repeatedly activated during and after the emplacement of the two intrusions.

The South Kawishiwi intrusion hosts at least five copper-nickel prospects (Spruce Road, South Filson Creek, Maturi, Dunka Pit, Serpentine) and a potential platinum group element copper-nickel prospect (Birch Lake).

The South Kawishiwi intrusion is composed of troctolitic rocks exposed in an 8-by-32 kilometer arc-shaped band (Figure 1). The Giant's Range batholith is the dominant footwall rock, although Virginia Formation and Biwabik Iron Formation underlie the intrusion in some areas. Five major map units in the South Kawishiwi intrusion have been identified, although some geologists have further subdivided these units (e.g. Severson 1994). The units are not continuous throughout the intrusion. Major sulfide mineralization occurs at the base of the intrusion. Virginia Formation and Biwabik Iron Formation inclusions occur throughout the intrusion. Inclusions of basaltic hornfels (probably from the North Shore Volcanic Group) and quartz sandstone hornfels (probably from the Puckwunge and Nopeming sandstones) occur near some of the copper-nickel prospects. The lower ultramafic unit contains the majority of the high platinum group element values.

2. OBJECTIVES

The objectives of the present research on samples from the Partridge River intrusion of the Duluth Complex are as follows.

1. Determine the variation of laboratory drainage quality and rates of chemical release as a function of sulfur content and time for $0.053 < d < 0.149$ mm samples.

2. Determine the variation of laboratory drainage quality and rates of chemical release as a function of sulfur content and time for $d < 6.35$ mm samples.

3. Compare the dependences of drainage quality and chemical release rates on sulfur content and time determined for $0.053 < d < 0.149$ mm rock with those determined for $d < 6.35$ mm rock.

4. Compare the dependences of drainage quality and chemical release rates on sulfur content and time determined for Babbitt prospect rock samples with those generated by Dunka Road rock.

5. Compare results for $0.053 < d < 0.149$ mm rock samples with those for Duluth Complex tailings sample (unsaturated reactor) produced by processing of Duluth Complex rock from the Babbitt prospect on which testing was initiated in 2002.

6. Compare results for $0.053 < d < 0.149$ mm rock samples with those for basal Duluth Complex rock samples from the South Kawishiwi intrusion on which testing was initiated in 1989 and 1990.

3. METHODS

3.1. Materials

3.1.1. Babbitt Prospect

Duluth Complex rock samples from the Babbitt prospect were obtained from a bulk sample taken in 2001. The bulk sample site was in the SE of the NW, Section 29, Township 60N, Range 12W, near the approximate location of AMAX drill hole number 321. The excavated area for the bulk sample was approximately 300 feet by 200 feet and had roughly 25 feet of glacial overburden above the bedrock. Although the overburden contained large rocks and boulders, it appeared to be fairly silt-rich. There was a very distinct boundary between the oxidized upper layer of the soil and the lower reducing layer of soil, the upper layer being red and the lower layer being grey. There was at least five feet of the reducing layer above the bedrock, which was not weathered and seemed very competent (Eger 2001).

After the bulk sample was blasted, rock was trucked to a loading pocket, crushed to approximately minus six inches, and a sample was taken and transported by rail for metallurgical testing. From the remaining pile, several buckets of rock were collected for possible dissolution testing. Highly mineralized rock, as determined by visual examination, was excluded from this collection. Fourteen samples, approximately two to four kg each, were selected from the buckets. The objective of this sampling was to obtain, by visual examination, samples with a range of sulfur contents.

The 14 samples were sent to Lerch Brothers Inc. (Hibbing, MN) where they were stage crushed to −0.64-cm to limit generation of fines. The process entailed screening the drill core sample to −0.64 cm and three crushing steps (large jaw crusher set at 1.92 cm, small jaw crusher set at 0.95 cm, roll crusher set at 0.64 cm). After each of the first two crushing steps, the −0.64 cm fraction was collected and the oversize was passed to the next crushing phase. Five 250-gram samples and one 100-gram sample were split from each bulk sample. The five 250-gram samples were stored in sealed plastic bags until filling the cells. The 100-gram split was pulped for subsequent analysis. Sulfur and carbon dioxide contents of the pulps were determined, and seven samples were selected for dissolution testing in humidity cells based on these analyses. In addition, a 400-gram sample was split from six of the samples, crushed to –100 / +270 mesh ($0.053 < d < 0.149$ mm) and wet sieved for use in the small reactors.

3.1.2. Dunka Road Prospect

The Dunka Road prospect is approximately three kilometers west southwest of the Babbitt prospect (Figure 1). On 16 August 2001, Minnesota Department of Natural Resources (MN DNR) personnel examined information (rock unit, mineral grain size, sulfur and copper content, and visually estimated plagioclase, olivine, and pyroxene contents) on drill holes from the Dunka Road Cu-Ni prospect for the purpose of selecting samples for laboratory waste characterization experiments. The information was from a total of 32 drill holes that were drilled in 1999 and 2000. The drill core had a diameter of either 1 3/4" or 2" (depending on the year of drilling) and

the required footage needed for a single laboratory cell was five feet, yielding a mass of approximately 1200 to 2000 grams.

Based on the drill core logs and mass requirements 26 intervals were selected, and the core (half of the original) was brought to the MN DNR Hibbing office and sawed in half. One quarter of the core was packaged, labeled, and inventoried in the core building for future work. The remaining core was returned to the mining company. Ten of the inventoried samples were selected for possible use in the laboratory experiment. All samples were stage crushed to –6.35 mm and one sample was crushed and sieved to obtain the 0.053-0.149 mm fraction (see section 3.1.1 for methods). A split of each sample was analyzed for sulfur content at Lerch Brothers Inc. Samples were selected for laboratory testing based on sulfur content.

The majority of the samples had relatively low sulfur contents and these samples were excluded from the experiment. A 0.23% sulfur sample was selected for testing in a humidity cell. The remaining higher sulfur samples lacked sufficient mass to run in large cells. Due to the sample mass available a 0.25% sulfur sample was mixed with a 0.36% sulfur sample and a 1.30% sulfur sample was mixed with a 0.03% sulfur sample (Table 1). The two mixed samples were analyzed at Lerch Brothers for sulfur and carbon dioxide. Sulfur contents for the 0.25% and 0.36% S composite and the 1.30% and 0.03%S composite were 0.31% and 0.61% S, respectively.

All three samples (0.23, 0.31, 0.61% S) were run in humidity cells using a 1000-gram mass of rock finer than 6.35 mm rock. The 0.31% S sample was also run in a small reactor using a 75-gram mass of 0.053-0.149 mm rock.

3.2. Apparatus

3.2.1. Small Reactors (75 g charge)

The samples were placed onto a 1.6-micron Whatman GF/A glass fiber filter (5.5 cm diameter) resting on a perforated plastic plate in the upper section of a two-stage filter unit or reactor (Figure 2). On 19 August 2003 the reactors were filled. A plastic weighing boat was placed on a scale and 75 grams was weighed out and added to the top section of the small reactors. The calculated bed depth was about 15 mm (density = 3 g cm^{-3}, 40% voids, d = 5.66 cm), but the depth was measured as roughly 18 mm after 400 weeks (Appendix 2, Attachment A2.2).

3.2.2. Humidity Cells (1 kg charge)

Cylindrical cells (I.D. = 10.2 cm, h = 19.0 cm) constructed of acrylic plastic were used in the experiment. A 0.64-cm high acrylic sleeve was glued into the bottom of the cell to support a 0.16-cm thick PVC plate, which was perforated with 0.32-cm holes spaced 0.48 cm on center. The plate was covered with a polypropylene filter (National Filter Media Corporation, Polymax B, Style 226-075-2) that supported the solids. The bottom was sealed with an acrylic plate with a 0.85-cm outlet port in the center and a removable cover, with a 0.85-cm vent hole, was place on top of the cell (Figure 3). Prior to filling, the cells were acid washed with 10% HNO_3 and

then rinsed three times with distilled water. The cells were weighed and the weight was recorded on the cell.

On 20 August 2003 the cells were each filled with 1000 g of rock. A plastic pan was placed on a scale and four of the five 250-gram splits were added. The weight was generally about 50 grams over the desired 1000-gram sample size. A plastic spoon was used to randomly remove the excess and add it to the remaining 250-gram split. The 1000-gram sample was then poured into the cell. The samples were very dry and some of the very fine dust-like particles were lost due to suction from the exhaust hood. The plastic pan and spoon were rinsed with distilled water after each reactor was filled. The cells with dry solids were weighed and the tare weight recorded. Measured bed depth was in the range of 6.8 to 7.2 cm, and the depth was calculated as 6.9 cm, assuming a density of 3 g cm^{-3} and 40% voids used for calculation (Appendix 2, Attachment A2.2).

3.3. Experimental Procedure

A total of eight small reactors and eleven humidity cells were used. The small reactors were used to allow direct correlation of data with those generated previously using these reactors to examine the dissolution of Duluth Complex rock. The eleven humidity cells were intended to provide a comparison of the small reactors and the ASTM D5744 method that uses these cells (ASTM 2007). Both methods provided for weekly rinsing of solutes from the rock samples. Between rinses the rock samples were maintained at fairly constant temperature and humidity.

The solids were rinsed three times (week 0) to remove oxidation products that accumulated on the solids from the time they were processed until the beginning of the experiment. Distilled water (200 mL) was slowly dripped into the small reactors from a buret and was allowed to drain freely from the cell. The reactors have since been rinsed weekly with one 200-mL volume of distilled water. Between rinses these samples are exposed to air and virtually all the pore water evaporates during the one-week oxidation period.

For the humidity cells, a rubber cap was placed on the outlet port and 500 mL of distilled water was slowly dripped into each cell from a separatory funnel. The water remained in the cell for ten minutes after the last one was filled, and then the cells were uncapped and allowed to drain. The cells were subsequently rinsed weekly with one 500 mL volume of distilled water. These cells were covered during the oxidation period, and evaporation from them was minimal.

Effluent (or drainage) samples were weighed and analyzed for pH, specific conductance, and alkalinity or acidity at the MN DNR laboratory. From the remaining drainage, samples were collected and filtered for subsequent determinations of sulfate and metal concentrations. After filtration, samples to be analyzed for metals were preserved with 0.2 mL Baker Instra-Analyzed nitric acid per 50 mL sample.

3.4. Reaction Conditions

Between rinses the samples were retained in the cells to oxidize. The cells were stored in a room (8.5 × 10.5 × 9.5 ft) in which temperature and humidity were controlled by commercially available equipment. Temperature was maintained by a thermostatically controlled heater (Lakewood 750) and humidity was maintained by a Holmes HM 3500 humidifier and a Sears Kenmore dehumidifier, both of which were controlled by Honeywell humidistats.

Water retention in the small reactors and large cells was determined by weighing. During the first seven weeks of the experiment the units were weighed four to five times a week. Subsequently, weights were determined immediately before and one day after rinsing.

3.5. Analyses

3.5.1. Solid Phase Analyses

Samples were analyzed at Lerch Brothers for particle size, total sulfur and carbon dioxide. Particle size distribution for the –6.35-mm samples was determined using method ASTM E 276-93. Total sulfur contents of all samples were determined using a LECO combustion furnace (method ASTM E395-95A) and carbon dioxide content was determined using a gas evolution method (ASTM E350-89C). The remaining solid-phase constituents were determined by Chemex (ALS Minerals) in Reno, NV (Appendix 1, Tables A1.3-A1.8). Whole rock constituents were determined using a lithium borate fusion method (ME-ICP06) and analysis by inductively coupled plasma-atomic emission spectroscopy (ICP-AES). Concentrations of Ag, As, Cd, Co, Cu, Mo, Ni, Pb, Sc, and Zn were determined using a 4-acid digestion for base metals (ME-4ACD81), with analysis by ICP-MS. Other trace elements were determined using a lithium metaborate fusion for resistive elements (ME-MS81). Chemex also measured total sulfur and carbon (by combustion furnace) as well as copper and nickel (aqua regia digestion and ICP) for five size fractions of the -6.35-mm samples (-270 mesh, +270/-100 mesh, +100/-35 mesh, +35/-10 mesh, +10 mesh/-6/35 mm).

Peter McSwiggen (McSwiggen & Associates, Minneapolis, MN) determined mineral content and chemistries (Appendix 1, Attachment A1.2). Grains from seven waste rock samples were mounted into a one-inch epoxy mount, which was then polished and carbon coated. Over one hundred grains were randomly selected for analysis. Each grain was analyzed using a JEOL 8600 Electron Probe Microanalyzer (15 kV accelerating voltage, 20 namp beam current, 5 second dwell time, 5 μm beam diameter).

McSwiggen determined the grain size distribution of the particles using image analysis techniques. Using the program NIH ImageJ, a threshold was applied to the backscattered electron image to separate the particles from the mounting medium using the intensity of the backscattered electron signal. ImageJ was then used to process the resulting image to determine the actual particles, and then measure each particle's longest length, perimeter, and area. In Microsoft Excel, the particles were sorted based on their area, and a cumulative area was calculated. The percent cumulative area was than calculated, as well as the circular diameter of

each particle. This data was plotted onto percent cumulative area vs. circular diameter graphs in order to determine P5, median, P95 grain size for each sample. This process was done for both the sulfide grains and all grains (Appendix 1, Attachment A1.3). Using the individually measured sulfide particle sizes, the probability of a sulfide grain to be exposed at the surface of a silicate particle was calculated using the radial differences of the sulfide and silicate grains (Appendix 1, Attachment A1.4).

3.5.2. Drainage Quality Analyses

Water samples were analyzed for specific conductance, pH, and alkalinity or acidity at the MN DNR laboratory. Specific conductance was analyzed using a Myron L conductivity meter, and an Orion SA720 meter, equipped with a Ross combination pH electrode (8165), was used for pH analyses. Alkalinity (for pH $\geq$ 6.3) and acidity were determined using standard titration techniques for endpoints of 4.5 and 8.3, respectively (APHA et al. 1992). The remaining sample was filtered for metals and sulfate analyses. Metals samples were acidified with 0.2 mL ultra-pure nitric acid per 60 mL sample.

Metals and sulfate samples were analyzed at Minnesota Department of Agriculture (MDA) in St. Paul, MN. Ca, Mg, Na, and K were determined with a Varian 400 SPECTRAA; inductively coupled plasma mass spectrometry (ICP-MS, Hewlett Packard HP4500 Series, model #G1820A) was used for the remaining metals analyses. Sulfate concentrations were determined using a Lachat QuickChem 8000 or, for $[SO_4] < 5$ mg/L, a Dionex ion chromatograph. More detailed analyses were conducted on samples collected on weeks 88, 126, 158, and 174 using a Perkin Elmer Elan 6000 ICP-Mass Spectrometer. These analyses were conducted by ACME Analytical Laboratories Ltd., Vancouver, BC. The results are not discussed but are presented in Appendix 1, Attachment A1.1.

3.6. Release Rate Calculation Methods

Weekly chemical mass release for each solute was calculated as the product of drainage volume and solute concentration. For weeks that solute concentrations were not determined, the concentration was estimated as the average of previous and subsequent measured values. Cumulative release was calculated as the sum of weekly release values.

Rates of solute release were calculated for every week drainage quality samples were analyzed for a specified solute. For each solute, the weekly rate of release was calculated as the product of the volume and solute concentration divided by the rock mass originally present. The weekly release rates were averaged over 52 week periods to determine temporal variability over the 400 week period of record. Standard deviations were calculated to provide a measure of rate variability during the periods.

In order to gain a better understanding of the rates of release as a function of sulfur content, the annual average release rates (Tables 9, 10) were plotted as a function of sulfur content for both fine and coarse particles (see Figure 20 for example). A set of graphs for each parameter (SO_4, Ca, Mg, Na, K, Cu, Ni, Co, Zn) were developed for each year in the 400 week period of record

and a linear regression was fit for each data set (Appendix 4, Figures A4.20-A4.29). The slopes of these regressions were normalized per gram of sulfur (mass of rock x g sulfur/g rock) to calculate an annual composite release rate per gram of sulfur for both particle sizes. These composite rates were plotted annually for comparative purposes between the particle sizes (Figures 21-25).

This examination of annual composite release rates per gram of sulfur was done for all parameters (SO_4, Ca, Mg, Na, K, Cu, Ni, Co, Zn). The regression fit is very good for those parameters that exhibit a strong correlation with sulfur (i.e. sulfate), but not as good for those that have a weak correlation with sulfur, such as potassium (Table 12). It should be noted that for all parameters these linear regressions have associated y-intercept values (Appendix 4, Table A4.21) that are not accounted for in Figures 21-25. It should also be noted that magnesium release from reactor 7 was omitted due to lower than expected drainage quality which resulted in a weak correlation with respect to sulfur. This does not necessarily mean the drainage quality data from reactor 7 is inaccurate, but rather the magnesium release from this sample does not represent the bulk release from the finer particles as a function of sulfur content.

4. RESULTS

4.1. Introduction

In August 2003, tests were initiated on the dissolution of Duluth Complex rock samples from the Babbitt and Dunka Road prospects of the Partridge River intrusion. Seven –100/+270 mesh ($0.053 < d < 0.149$ mm) samples were tested, six from the Babbitt prospect and one from the Dunka Road prospect. Sulfur and carbon dioxide content ranges were 0.07 to 0.94 percent and 0.07 to 0.2 percent, respectively. Ten –0.25-inch ($d < 6.35$ mm) samples were tested, seven from the Babbitt prospect and three from the Dunka road prospect. Sulfur and carbon dioxide content ranges were 0.13 to 1.36 percent and 0.05 to 0.21 percent, respectively. One sample in each size fraction was run in duplicate and drainage quality for both pairs replicated well (Figures 4, 5). These experiments remain in progress and the following reporting represent an assessment of the data through week 400.

4.2. Solid-Phase Analyses

Particle size distribution of the –6.35-mm humidity cell samples was determined. Roughly 29 to 57 percent of the sample masses were finer than 10 mesh (2 mm). The corresponding ranges for 35 and 200 mesh (0.5 and 0.074 mm, respectively) were 12 to 23 percent and 3.4 to 5.6 percent (Table 2). Total sulfur content for the bulk samples ranged from 0.07 to 1.36 percent and correspondingly the carbon dioxide contents ranged from 0.05 to 0.2 percent (Tables 3 and 4). There were no major differences in whole rock chemistry, heavy metal composition, and mineral contents between the reactors and humidity cells. The average major whole rock components were SiO_2 (44.8%), Al_2O_3 (17.2%), and Fe_2O_3 (14.0%) (Table 4). Average copper, nickel, cobalt and zinc contents were determined as 0.21, 0.06, 0.008, 0.01%, respectively (Table 5). Mineral contents and chemistries were 36-65% plagioclase $(Ca_{0.54\text{-}0.67}Na_{0.35\text{-}0.46})Al_{1.52\text{-}1.61}Si_{2.34\text{-}2.46}O_8$, 12-38% olivine $(Mg_{1.05\text{-}1.27}Fe_{0.73\text{-}0.98})Si_{0.97\text{-}1.00}O_4$, and 2-19% augite $(Ca_{0.73\text{-}0.90}Na_{0.017\text{-}0.025})(Mg_{0.73\text{-}0.82}Fe_{0.27\text{-}0.41}Ti_{0.013\text{-}0.035})(Si_{1.91\text{-}1.96}Al_{0.07\text{-}0.10})O_6$. Ilmenite, biotite, potassium feldspar, and orthopyroxene were also reported present, on average, at 1-6% (Appendix 1, Attachment A1.2).

The Babbitt prospect samples were marginally finer than Dunka Road prospect samples of similar sulfur content (Table 2). Differences in major cation contents were generally small, For example CaO contents of the Babbitt prospect rock that were about 0.8 to 0.9 and times those of the Dunka Road samples (Table 6). The largest difference was observed for potassium, for which Babbitt prospect contents were 1.6 - 2.2 times those of the Dunka Road samples. These differences were reflected in the relative biotite contents of the samples (Table 7). Copper, nickel, and zinc contents of the Babbitt prospect rock were consistently higher than those from the Dunka Road samples and cobalt contents were similar (Table 6).

4.3. Reaction Conditions

4.3.1. Temperature and Relative Humidity

Over the 400-week period of record temperature and relative humidity (in the room in which reactors and humidity cells were stored) were measured three to four times a week with a Taylor wet-bulb, dry-bulb hygrometer. Average weekly values for temperature ranged from 20 to 28 ^{o}C, averaging 23.6 ^{o}C (s.d. = 1.3 ^{o}C, n = 392). The corresponding range for relative humidity was 42% to 75%, averaging 57.9% (s.d. = 4.2%, n = 392) (Appendix 2, Table A2.2). Data from 3 May to 28 June 2004 were lost.

4.3.2 Water Retention

Water retention did not replicate exceptionally well for the duplicate pairs of small reactors or humidity cells. In both cases, the typical variation among different solids appeared to be less than that between the duplicates (Appendix 2, Figures A2.1 and A2.2). The reason for this is unclear. As indicated by results from the following section, drainage quality from the paired reactors was in good agreement, suggesting that the magnitude of the differences in water retention did not substantially affect rock dissolution and the consequent drainage quality.

For the small reactors, the average water retention after the rinse ranged from 17 to 27 g. Water retention after the rinse continued to decrease to approximately 5 mL by week 300 and plateaued. The average water retention before the rinse was less than -2 mL (Table 8). The low water retention before the rinse indicated that virtually all water evaporated during the weekly cycle. Although the rate of evaporation was not measured for these samples, daily measurements were made during a six-week trial on the 75 gram, -100/+270 mesh Duluth Complex waste rock samples from South Kawishiwi blast hole samples from the Dunka Mine (MNDNR 2013). These measurements determined that approximately 75% of retained water evaporated within three days of rinsing, with approximately equal amounts lost each day. Reactors were completely dry before the next weekly water addition and, at the rate of drying from the first two days, would have completely dried less than four days after water was added.

For the large cells, average water retention after the rinse ranged from 100 to 166 g, and the variation was apparently related to particle size distribution (Table 2). Additional data analysis was conducted to examine the relationship between water retention after the rinse and the fraction of a sample finer than 2 mm. The average water retained after the rinse was plotted against the sample mass occurring in the -10 mesh (2 mm) fraction. Linear regression analysis was conducted and yielded respective slope and intercept values of 0.22 and 34 (n = 11, r^2 = 0.713) (Figure 6). The slope suggests that 0.22 g water were retained per gram of -10 mesh rock. Water retention in the -10 mesh fraction was determined using a rock density of 3 g cm^{-3}, estimating a bed with 40% voids, and assuming all pores in the -10 fraction are filled yields a water retention value of 0.22 g per gram -10 mesh rock (Appendix 2, Attachment A2.3). This interpretation is reasonably consistent with interpretation of the linear regression analysis. Furthermore, the y-axis intercept derived by the regression suggests that 34 grams of water were associated with the +10 mesh fraction and the humidity cell itself. The 34 grams represents from

20 to 34 percent of the entire water retention, with most values near 30 percent. The range before the rinse was 94 to 151 g, with most values between roughly 100 and 110 g (Table 8). Water loss during the week ranged from roughly 4 to 15 g, indicating that little evaporation occurred during the weekly cycle.

4.3.3 Leachate Volume

The drainage volumes for the duplicate reactors were in good agreement for the period of record. There were some excursions in Reactor 5, where drainage volume would drop 30-40 mL during a weekly rinse period (Appendix 2, Figure A2.7). However, the average drainage volumes were in very good agreement for reactors 4 and 5 at 167.8±4.3 mL and 168.0±4.8 mL respectively. The average drainage volume for the 8 reactors for the 400 week period of record was 164.6±2.7 mL (Appendix 2, Table A2.5).

The drainage volumes for the duplicate humidity cells were also in good agreement. The average drainage volumes for reactors 12 and 13 were 480.3±6.5 and 486.4±6.3 mL respectively. Drainage volumes from reactor 13 were typically 5-7 mL higher than reactor 12 for the entire period of record (Appendix 2, Figure A2.8). The average drainage volume for the 11 humidity cells for the 400 week period of record was 481.2±3.1 mL (Appendix 2, Table A2.5).

4.4. Dissolution of Babbitt and Dunka Road Prospects rock

4.4.1 Water Quality Replication

For the following discussion, differences in dissolution behaviors of the Babbitt and Dunka Road samples identified in Section 5.3 will be neglected, and the samples will be considered as a single collection of Partridge River intrusion samples. Duplicate 75-g reactors and humidity cells were run with the 0.31%S and 0.55% S, respectively, Babbitt prospect samples and drainage quality agreement between duplicates was generally very good (Figures 4 and 5).

For the reactors (finer particles), replication of pH and concentrations of sulfate and major cations (Ca, Mg, Na, K) was excellent (Figure 4). This agreement is corroborated by average rates of mass release calculated over the period of record. Replicate rates for all five solutes were within 5% of the mean value (Table 9). Rates of nickel, cobalt and zinc release were similarly in good agreement between duplicates. However, copper rates were roughly 20 percent of the mean value, due to higher rates in reactor 4 from year 2 to 6 (Table 10).

For the humidity cells (coarser particles), pH replication was excellent. Small deviations were observed for concentrations of sulfate, calcium, and magnesium, but agreement between replicates was nonetheless very good (Figure 5). This agreement was quantified by average solute release rates determined over the period of record. Replicate rates for sulfate, calcium, magnesium, sodium, and potassium differed by no more than 4% of the respective mean (Table 9). Although the agreement in nickel and cobalt release rates was similar, the difference from the mean for copper and zinc rates was 10 and 8 percent, respectively, of the mean value (Table 10).

4.4.2 Drainage pH

Considering the Babbitt prospect and Dunka Road samples as a whole, drainage pH from both size fractions tended to decrease with time and sulfur content. For the reactors, drainage pH typically decreased over periods of 30 to 50 weeks and then become relatively stable (Figure 7). Drainage pH values from samples with sulfur contents less than 0.31% were generally greater than 6.0, although the 0.20%-S sample did produce several drainage pH values in the range of 5.8 to 6.0. Samples with sulfur contents of 0.63% and 0.94% produced minimum drainage pH values of near 5.2 and 4.3, respectively (Figure 7).

For the humidity cells, drainage pH tended to decrease steadily over time (Figure 7). Drainage pH from samples with sulfur contents of 0.13% to 0.72% typically plateaued after 150-300 weeks and reached minimum pH values ranging from 5.3 to 6.4. Samples with sulfur contents of 1.03% and 1.36% went acidic after 20 weeks and produced minimum pH values of 4.2 and 4.3, respectively, by week 150 (Figure 7).

4.4.3 Sulfate Release (Acid Production)

Sulfate release rates in drainage from the finer particles increased with increasing solid-phase sulfur content. Sulfate release rates tended to peak between weeks 30 and 50 and decreased over the remaining period of record (Figure 8). This behavior coincides with specific conductance over time (Figure 9). Sulfate release rates for most reactors after year 8 were approximately 3-11% of those in year 1. However, the higher sulfur samples (0.63 and 0.94%S) displayed sulfate rates after year 8 that were about 22% of those in year 1 (Table 9). The fraction of solid-phase sulfur depleted during the 8-year period ranged from 13 to 57% (Table 11).

As with the finer particles, sulfate release rates in the humidity cells tended to increase with solid-phase sulfur content. In contrast with the sulfate peaks observed for the finer particles, sulfate release in drainage from the d < 6.35 mm rock tended to be relatively stable over time (Figure 8). Sulfate release rates for most humidity cells after year 8 were approximately 29-83% of those in year 1. One exception was the 1.03%S sample, for which sulfate release at year 8 was the same as year 1 (Table 9). Unlike the other humidity cells, this sample displayed an increasing trend in sulfate release from weeks 50 to 200. The fraction of solid-phase sulfur depleted during the 8-year period ranged from 8 to 37% (Table 11).

4.4.4 Major Cation Release (Acid Neutralization)

For the reactors, molar concentrations of major cations initially decreased in the order Ca > Mg > K > Na and the ordering subsequently evolved to Mg > Ca > Na > K. Release rates tended to parallel those of sulfate, where release rates peaked during weeks 30-50 and decreased for the remainder of the period of record (Figures 10-13). Calcium and potassium release rates in year 8 were on average, 9 and 5 % of those in year 1, respectively (Table 9). In contrast, magnesium and sodium release rates were slightly less variable over the period of record, with release rates at year 8 averaging 29 and 39 % respectively, of year 1 release rates (Table 9). Total molar release of magnesium was the highest, measuring 1.2±0.4 times that of calcium and roughly

3.0±1.2 and 3.3±1.3 times that of sodium and potassium, respectively. Total release of sodium was 1.2±0.3 times that of potassium (Table 11). Because calcium and magnesium are divalent their neutralization potential was roughly six times higher than sodium and potassium.

For the humidity cells, molar concentrations of major cations initially decreased in the order Ca > Mg > Na > K and the ordering subsequently evolved to Mg > Ca > Na > K. Similar to the reactors, major cation release rates tended to parallel sulfate. Calcium, magnesium, sodium, and potassium release decreased steadily from weeks 50 to 100 and began to plateau (Figures 10-13). Release rates in year 8 were on average, 21, 65, 62, and 38 % of those in year 1, respectively (Table 9). Total molar release of magnesium was the highest, measuring 1.1±0.4 times that of calcium and roughly 4.3±1.9 and 5.2±1.7 times that of sodium and potassium, respectively. Total release of sodium was 1.5±1.0 times that of potassium (Table 11). Because calcium and magnesium are divalent their neutralization potential was roughly ten times higher than sodium and potassium.

4.4.5 Heavy Metal Release

For the reactors, nickel contributed the majority of heavy metal release, followed by copper, then cobalt and zinc (Table 11). Copper release tended to gradually increase and peaked between weeks 100-150 (Figure 14). Release of nickel, cobalt, and zinc tended to peak between 30 to 50 weeks and decreased for the remaining period of record (Figures 15-17). Heavy metal release tended to increase with increasing sulfur content (Table 10). Heavy metal release also tended to increase with relatively small decreases in pH (0.3 to 0.5 units). As pH decreased from 7.0 to 6.5, copper, nickel, cobalt, and zinc release rates increased, on average, by 10, 30, 3, 6 times respectively (Table 10). Total release of copper, on average, was less than 9 % of that present initially and, correspondingly, nickel was less than 29 %. The fractions of cobalt and zinc depletion were intermediate to these values (Table 11).

For the humidity cells, nickel contributed the majority of heavy metal release, followed by copper and cobalt, than zinc (Table 11). Release rates tended to gradually increase and level off by week 200 to 250 (Figures 14-17). However, the higher sulfur samples (1.03 and 1.36 %S), peaked around week 30 for nickel, cobalt, and zinc and remained at higher rates than the lower sulfur samples. Copper release in these higher sulfur humidity cells increased up until week 300 and began to plateau (Figure 14). Similar to the reactors, heavy metal release tended to increase with increasing sulfur content and small decreases in pH. As pH decreased from 7.0 to 6.5, copper, nickel, cobalt, and zinc release rates increased, on average, by 12, 16, 9, 3 times respectively (Table 10). Total release of copper, on average, was less than 2 % of that present initially and, correspondingly, nickel was less than 16 %. The fractions of cobalt and zinc depletion were intermediate to these values (Table 11).

5. DISCUSSION

5.1 Introduction

Duluth Complex waste rock was characterized (particle size distribution, chemistry, mineralogy) and subjected to laboratory dissolution testing. Although the dissolution tests remain in progress, the following discussion presents the first 400 weeks of data. The dissolution tests were conducted to examine 1) drainage quality from fine particles (0.053-0.149 mm, 75 grams), 2) drainage quality from coarser particles (–6.35-mm, 1000 grams) and 3) differences in drainage quality between these size fractions. The data also provided a comparison between 4) Babbitt and Dunka Road prospect samples, 5) waste rock and tailings, and 6) Partridge River and South Kawishiwi Intrusion samples.

5.2 Conceptual Overview

This section presents a brief description of the mineral dissolution reactions that occur during chemical weathering of Duluth Complex rock. This summary is intended to provide a conceptual structure to frame the discussion of results. It should be noted that detailed analysis of mineral dissolution reactions, such as calculation of mineral dissolution rates, is beyond the scope of this report.

Iron sulfide and base metal sulfide minerals present in Duluth Complex rock oxidize when in contact with oxygen and water. Pyrrhotite is the predominant iron sulfide mineral present and its oxidation releases acid and sulfate. Some or all of the acid released can be neutralized by dissolution of silicate minerals present, including plagioclase, olivine, and augite. The balance of acid production and acid neutralization reactions determines the solution pH. Oxidation of base metal sulfides such as chalcopyrite and pentlandite releases copper and nickel, respectively. The metals released can remain in solution or be removed in secondary phases due to chemical precipitation, adsorption, or coprecipitation. The extent to which metals released from the sulfide phase remain in solution increases as solution pH decreases, that is, as the solution becomes more acidic base metal concentrations tend to increase.

Sulfide mineral oxidation and silicate mineral dissolution are surface reactions and, therefore, the rates of solute release depend on the mineral surface area exposed. All other factors being equal, the mineral surface area exposed will increase as the mineral abundance increases. In addition, two factors contribute to mineral surface area exposed increasing as rock particle size decreases. First, for a given mineral mass the total mineral surface area increases as the mineral particle size decreases. That is, a kilogram of fine mineral particles will have more surface area than a kilogram of coarse particles. Second, the mineral surface area exposed increases as the degree of mineral "liberation" from the mineral matrix of the rock increases, and the extent of liberation increases as the rock particle size decreases. That is, a mineral occurring between or within other minerals in a large rock fragment will become exposed if the fragment is crushed to a finer size.

Sulfide mineral oxidation and silicate mineral dissolution rates are also affected by solution pH. Assuming oxidation of sulfide minerals in general parallels that of pyrite, abiotic rates of oxidation by oxygen decreases slightly as pH decreases. However, the overall abiotic rate increases as pH decreases into a range where ferric iron becomes the dominant oxidant (Williamson and Rimstidt 1994; Nordstrom 1982). As pH further decreases, bacterial oxidation of ferrous iron becomes the rate limiting step in the oxidation by ferric iron (Singer and Stumm

1970). In laboratory tests on Duluth Complex rock, the pyrrhotite oxidation rate in the pH range of 3.5 to 4.05 was roughly six to seven times that in the range of 5.35 to 6.1, and the higher rate was attributed to bacterially mediated oxidation (Lapakko and Antonson 1994). Silicate mineral dissolution rates also tend to increase as pH decreases but to a lesser degree (White and Brantley 1995).

5.3 Comparison of Fine-Grained and Coarse-Grained Particles

Certain general trends related to solid-phase sulfur content and time were apparent for both the fine ($0.053 < d < 0.149$ mm) and coarse ($d < 6.35$ mm) particles. For both particle sizes, as sulfur content increased, the rates of sulfate release increased and drainage pH decreased (Figures 18, 19). The increasing sulfur content reflected increasing sulfide mineral surface area, which yielded increased sulfide mineral oxidation and consequent acid production. The more rapid acid production tended to result in decreasing drainage pH and more rapid release of both major cations and heavy metals.

With respect to general temporal variations, sulfate release rates for both size fractions tended to decrease over time and plateau. Rates of major cation release tended to parallel sulfate release rates, although peak sodium release occurred later than peaks for other solutes. The major cation release was a response to the variation of rates of acid production reflected by sulfate concentrations. Heavy metal concentrations were largely controlled by drainage pH and, consequently, tended to increase as sulfur content increased. Heavy metal release from the fines tended to peak within the first 50 weeks and then decline and plateau, whereas release from the coarse particles was more gradual in response to slowly declining pH.

Although similar trends were observed for the two size fraction samples, the variations in solute release with respect to sulfur content and time differed quantitatively. Per unit mineral mass, the finer particles had greater sulfide mineral surface area exposed and available for oxidation. As a result, sulfate release rates from the finer fraction were approximately three times those from the coarse fraction during the first year of dissolution (Figure 21). Sulfate release rates from the fine fraction tended to peak within 30 to 50 weeks of reaction and declined fairly rapidly over time, influenced by depletion of the sulfide minerals and probably coating of the sulfide minerals remaining (Figure 8). Corresponding drainage pH minimums were reached between weeks 60 and 120, lagging about a year behind the observed maximums for sulfate release (Figure 7). In contrast, sulfate release from the coarse particles declined fairly slowly, and drainage pH from all but the two highest sulfur samples declined over the entire period of record (Figures 7, 8). In response to the associated decreasing rates of acid production the rates of calcium and magnesium release declined correspondingly, hence the parallelism between the release of major cations and sulfate (Figures 10, 11).

For both particle sizes, the dissolution of calcium and magnesium-bearing minerals contributed the most acid neutralization (Appendix 4, Figures A4.1-A4.19). The elevated sulfide oxidation in the finer fraction resulted in more rapid silicate mineral dissolution and consequent acid neutralization. Release of calcium and magnesium from the fine fraction in the first year was 1.6 and 1.9 times that from the coarse fraction, respectively (Figure 22). Total mass release of

calcium and magnesium from the fine fraction for the period of record averaged 1.4 and 1.6 times that from the coarse fraction, respectively (Table 11). It should be noted, however, that calcium release rates from the coarse fraction exceeded that from the fine fraction after year 4 (Figure 22).

Sulfate and heavy metal release result from oxidation of sulfide mineral phases, such as pyrrhotite and chalcopyrite. Roughly 75 to 90 percent of the calcium occurs in plagioclase and the majority of the remainder is with augite (App 1, Attach A1.2 - Table A1.2.8). The small amount of carbon dioxide present is likely associated with calcium, and the rapid dissolution of this phase contributes to the initially elevated calcium release (Figure 10). Plagioclase also hosts virtually all of the sodium present (App 1, Attach A1.2 - Table A1.2.8). If plagioclase were dissolving stoichiometrically, calcium would be released at a ratio of 1.5:1 to sodium (App 1, Table A1.2.1). However the ratio of Ca: Na total molar release was approximately 1.5 to 3 times the stoichiometric ratio of plagioclase dissolution after 400 weeks. This indicates calcium was being released by other minerals such as augite, the second largest host of calcium. Potential sources of magnesium release were olivine, augite, and biotite dissolution (App 1, Attach A1.2 - Table A1.2.8). Additional data analysis should be conducted in the future to better assess major cation contributions from individual mineral phases and rates of mineral dissolution. Such efforts are beyond the scope of the present project.

This rapid decrease of drainage pH in the finer fraction resulted in heavy metal release rates approximately three to four times those from the coarser fraction during the first two years of dissolution (Figure 26). In contrast, sulfide oxidation from the coarser fraction remained fairly constant throughout the period of record, and drainage pH declined slowly over time. Consequently by year 7 the nickel and cobalt release rates from the coarser particles were roughly five times higher than the fines (Figures 24, 25). Although it appeared that drainage pH was the largest influence on heavy metal release, it also was likely influenced by the greater degree of metal depletion from the fine grained samples with sulfur contents of at least 0.2 % (Table 11). Similar dependences on particle size and sulfur content were observed for copper and zinc, although release rates were higher in the reactors over the entire period of record (Figures 24, 25).

5.4 Comparison of Babbitt and Dunka Road Samples

Comparison of the Babbitt prospect and Dunka Road prospect rocks is tenuous because of the limited number of Dunka Road samples tested. Only one Dunka Road sample was tested in the 0.053-0.149 mm size fraction and three samples in the d < 6.35 mm fraction. For the finer fraction, results for the 0.31 %S Dunka Road sample were compared to those from the duplicated 0.31 %S Babbitt prospect samples. For the coarser fraction, Dunka Road samples with sulfur contents of 0.23, 0.31, and 0.61 % were compared with Babbitt prospect samples with sulfur contents of 0.21, 0.33, and 0.55 % (duplicated). Differences in dissolution of rocks from the two prospects would be expected to result from differences in mineral content, mineral chemistries, and modes of mineral occurrence.

5.4.1 Solids

The total sulfide mineral content of each pair was largely normalized by selecting samples of similar sulfur content for comparison of the two prospects. However the heavy metal content of the Babbitt Prospect samples tended to be higher than that of the Dunka Road samples, especially for the fine samples. This was particularly true for copper, for which Babbitt prospect sample contents were 1.3 to 5.5 times those of Dunka Road samples (Table 13). This indicates that chalcopyrite and cubanite constitute a larger fraction of the sulfides in the Babbitt Prospect samples. Nickel, cobalt, and zinc contents of the Babbitt Prospect samples were typically 0.9 to 1.4 times the associated Dunka Road contents, with only three ratios being below 1.0.

Evolved carbon dioxide contents of the paired fine samples (~0.20 %) and the low sulfur coarse samples (~0.08 %) were similar. However, for the two highest sulfur content coarse samples, the CO_2 contents of Dunka Road samples were 2 to 2.5 times their associated Babbitt prospect pair (Table 4). This suggests that the Dunka Road humidity cell samples had higher calcite contents than their Babbitt prospect pairs. The only consistent trends regarding silicate mineral content were higher biotite and potassium feldspar contents of the Babbitt Prospect samples, reflecting potassium contents that were about twice those in the Dunka Road samples (Table 7).

5.4.2 Solute Release

For *fine particles*, solute release is largely dependent on mineral content and mineral chemistry. The particle size is fairly uniform (0.053 to 0.149 mm) and virtually all mineral surfaces are exposed rather than being obstructed due to occurrence among or within other minerals in larger rock particles. Although drainage pH from the fine Babbitt prospect sample was slightly higher than that from the fine Dunka Road samples during the first 30 weeks, values from the two prospects were quite close subsequently (Figure 27). Concentrations of sulfate, major cations (Ca, Mg, Na, K), and base metals (Cu, Ni, Co, Zn) in drainage from the Babbitt prospect sample were typically higher than associated values in drainage from the Dunka Road sample (Figure 27). For sulfate and major cations the elevated release was most marked during the first year, whereas this trend extended beyond two years for base metals (Figure 27).

These trends were driven largely by more rapid sulfide mineral oxidation for Babbitt Prospect samples. Accelerated iron sulfide mineral oxidation enhanced acid generation, which in turn accelerated silicate mineral dissolution and the consequent major cation release. The cause of the accelerated oxidation of sulfide minerals in the Babbitt Prospect rock cannot be determined conclusively. As mentioned above, base metal sulfides (especially chalcopyrite and cubanite) comprised a larger fraction of the total sulfide content in the Babbitt Prospect samples. A similar phenomenon was reported for Dunka Road waste rock with a higher copper sulfide mineral fraction (SRK 2007). Either the sulfide chemistry or features related to this chemistry (e.g. grain size, surface roughness, mineral lattice dislocation density) apparently accelerated oxidation of all sulfide minerals present. Accelerated iron sulfide oxidation and the consequent acid production enhanced silicate mineral dissolution and the associated major cation release. Accelerated base metal sulfide mineral oxidation, due in part to a higher base metal sulfide content, enhanced release of copper, nickel, cobalt, and zinc from the Babbitt prospect samples.

It should be noted that the higher base metal sulfide content of the Babbitt prospect samples does not entirely explain the higher base metal sulfide release. The copper, nickel, cobalt, and zinc contents of the Babbitt prospect sample were 1.3 to 5.5 times the respective values in the Dunka Road sample. The corresponding metal mass releases from the Babbitt prospect sample during the initial 400 weeks of reaction were 2.6 to 42 times those from the Dunka Road samples (Table 13). Thus, the mass release ratios were roughly 2 to 9 times those expected based on the differences in base metal content alone (Table 13).

Some of the additional acceleration can be attributed to the more rapid rate of sulfate release from the Babbitt prospect samples. Specifically, the more rapid release of nickel and zinc from the Babbitt prospect samples is roughly bounded by consideration of differences in metal content and sulfate release rate (Table 13). The degree of copper release acceleration was well beyond that inferred by the compounded influences of solid-phase concentration and sulfate release rates (Table 13). The acceleration cannot be attributed to differences in pH because the pH of drainages from the two samples was similar (Figure 27). This suggests that the copper sulfide minerals present in the Babbitt prospect samples oxidized more rapidly than those in the Dunka Road samples.

For *coarse particles*, relative magnitudes of pH and sulfate and major cation release varied among the three samples subjected to humidity cell testing. For the two lower sulfur Dunka Road samples, pH was higher and rates of sulfate, calcium, magnesium, and sodium release were generally lower than or equal to corresponding values for Babbitt prospect samples (Figures 28, 29). For the highest sulfur sample pair these trends were generally reversed, magnesium being an exception (Figure 30). It should be noted that the difference in sulfur contents for this pair was the largest of the four pairs examined. Thus, there were no consistent major differences in drainage pH, acid production rates, or acid neutralization rates among the three pairs of samples examined. This is consistent with the general mineralogical similarity between the Babbitt and Dunka Road prospect samples. The only common trend was that potassium release from the Dunka Road samples was roughly half that from the Babbitt prospect samples of similar sulfur content (Tables 11). As mentioned above, this was likely due to the lower potassium-bearing minerals in the Dunka Road rock, as was the case for the fine-grained samples.

The lack of a consistent relationship between copper content and sulfate release from the coarse samples indicates that the sulfide mineral composition did not greatly influence the rate of total sulfide mineral oxidation. However, heavy metal release from the Babbitt prospect samples tended to exceed that from the Dunka Road samples. For the two lower sulfur samples, the relative difference in heavy metal release through 400 weeks was beyond that which would be expected based on differences in heavy metal content and sulfate release rate (Table 13). Lower drainage pH from the Babbitt prospect samples contributed to part of the elevated release. That is, more acidic conditions associated with the Babbitt prospect samples, in addition to higher metal contents and sulfate release rates, likely enhanced metal release. In contrast, drainage pH from the Dunka Road sample was lower than that from the Babbitt prospect sample for the high sulfur pair (Figure 30). Despite this “advantage” for the Dunka Road rock, the relative metal release from the Babbitt Prospect samples was roughly equal to the differential indicated by the differences in metal content and sulfate release rate (Table 13). As was the case for the fine

particles, it appears that the oxidation rates of heavy metal sulfides present in the Babbitt prospect exceeded that of heavy metal sulfides in the Dunka Road prospect.

In summary, there were no consistent differences between the Dunka Road and Babbitt prospects in drainage pH or rates of sulfate, calcium, and sodium release. Magnesium release from the Babbitt prospect samples was consistently 1.1 to 1.2 times that from the Dunka Road samples (Table 14), but there was no obvious solid-phase attribute to which this difference could be correlated. Although the largest observed ratio (2.8) was associated with the largest difference in olivine content, no trend between magnesium release and olivine content was apparent (Table 7). Potassium release from the Dunka Road fine and coarse samples was roughly half that from their paired Babbitt prospect sample. This reasonably reflected differences in biotite contents between the two prospects.

Heavy metal release from the Dunka Road rock tended to be lower than that from the Babbitt prospect rock. Whereas some of the difference could be ascribed to higher metal content of the Babbitt prospect rock, differences in total sulfide mineral oxidation, and drainage pH also played a role. Data from the fine sample pair suggested that heavy metal sulfides from the Babbitt prospect oxidized more rapidly (when normalized per unit mass). This was particularly the case for copper sulfides. Whereas the aforementioned trends were observed for the sample pairs tested, it should be emphasized that these comparisons consider only a few samples and may not represent general trends for the two prospects.

5.5 Comparison of Duluth Complex Waste Rock and Tailings

A sample of 0.2% S tailings from the Babbitt Prospect of the Partridge River Intrusion was subjected to dissolution in duplicate under the same conditions to which the waste rock samples were exposed (unsaturated tailings reactors 1 and 2). This section compares results from this tailings sample to those from the reactor containing the 0.20 %S waste rock sample. The waste rock particles ranged from 0.053 to 0.149 mm in diameter and were coarser than the tailings, for which roughly 51 % were in the aforementioned range, 29 % were finer than 0.053 mm, and 16 % were finer than 0.025 mm (Lapakko 2013, Table 2). The plagioclase contents of the waste rock and tailings were similar (50 and 57 percent, respectively), as were the calcium contents of the plagioclase present (Appendix 1, Table A1.11). Augite and biotite contents of the waste rock were reported to be slightly lower than those of the tailings (11 vs. 7.8% and 5.7 vs. 2.3%, respectively). In contrast, the olivine content of the waste rock was considerably higher than that in the tailings (26 vs. 9%). Respective concentrations of copper, nickel, and cobalt in the waste rock were 2.0, 1.2, and 1.2 times those in the tailings (Appendix 1, Table A1.10).

The pH of drainage from the waste rock sample was typically 0.8 to 1 unit lower than that of the tailings, with respective minimum values of 5.7 and 6.5 (Figure 31). Both higher sulfide mineral oxidation rates and lower silicate mineral dissolution rates contributed to the lower pH values from the waste rock.

The rates of sulfate release from the tailings averaged about 60 μmol (kg rock·wk)$^{-1}$ over the first eight years of the experiment (Lapakko et al. 2013), 80 percent of the 75 μmol (kg rock·wk)$^{-1}$

reported for the 0.20 %S waste rock over the same period (Table 9). Sulfate release rates for the tailings and rock were very close in year 1 (260 vs 240 μmol (kg rock·wk)$^{-1}$) and year 8 (16 vs 15 μmol (kg rock·wk)$^{-1}$). Rates for the rock tended to be somewhat higher in the intermediate years (Figure 31).

Sulfate release rates are influenced by the sulfide mineral surface area exposed, the availability of oxygen, and the sulfide mineral chemistry. Because the rates of sulfate release from the waste rock and tailings were not substantially different, these factors did not appear to exert a major influence. Nonetheless, these factors are briefly discussed in the interest of providing perspective on the oxidation of sulfide minerals present in the two solids.

Considering the finer particle size of the tailings, and associated higher specific surface area, it would be expected that they would exhibit higher sulfate release rates. The lower observed sulfate release rates observed for the tailings are apparently the result of removal of "liberated" sulfide minerals during mineral processing. Consequently, relative to unprocessed waste rock, a greater fraction of sulfide minerals present in the tailings are included within other rock particles and, therefore, less available for reaction. This is supported by mineralogical analyses that reported respective liberation probabilities of 100 and 87 percent for the rock and tailings (Appendix 1, Attachment A1.4).

The finer grained tailings also might have limited oxygen diffusion to sulfide mineral surfaces and thereby inhibited sulfide mineral oxidation. Both the finer grain size itself and increased water retention with finer grain size would tend to reduce diffusion. However limitation of oxygen diffusion was probably not substantial due to the thin tailings beds used in the experiment and the fact that water retained after the rinse evaporated within about three days (Lapakko et al. 2013). Although evaporation rates were not determined in the present study, similar evaporation rates were measured for 75-gram samples of -100/+270 mesh rock in previous work (see Section 4.3.2).

It should be noted that there were also differences in the specific sulfide minerals present in the two samples. Respective copper contents of 0.1 and 0.051 percent for the waste rock and tailings were used to estimate the fraction of sulfur present with chalcopyrite. Assuming all copper was present as chalcopyrite indicated an estimated 50 percent of the sulfur in the waste rock and 25 percent of the sulfur in the tailings was associated with chalcopyrite. This infers that the pyrrhotite content of the tailings was about 1.5 times that of the waste rock. Pyrrhotite is reported to oxidize more rapidly than chalcopyrite (e.g. Steger and Desjardins 1978), so this would tend to accelerate sulfate release from the tailings relative to the waste rock. However, as discussed in the previous section, samples of higher copper sulfide mineral content tended to have higher sulfate release rates. Thus, it is unclear if the difference in sulfide mineral partitioning between the rock and tailings samples was of substantial influence.

Average rates of calcium and magnesium release from the tailings were on the order of twice those from the rock (Figure 31). For calcium this was apparently largely due to the finer particle size of the tailings, resulting in a greater surface area of calcium-bearing silicate minerals. Plagioclase was the major calcium-bearing mineral in both the waste rock and tailings. There

was little difference between the waste rock and tailings in plagioclase contents (50 vs. 57%) or the atom fractions of calcium in the plagioclase (0.54 vs. 0.57; Appendix 1, Table A1.11). Thus, mineralogical differences between the samples were small and any associated influences on calcium release would likely be minimal.

There were differences in magnesium-bearing contents of the waste rock and tailings. Relative to the tailings, the waste rock had less augite and biotite (7.8 vs. 11%, 2.3 vs. 5.7%, respectively) and more olivine (26% vs. 9.8%). Particle size, and its associated influence on silicate mineral surface area, would be expected to have a similar effect on magnesium release. That is, if there were no differences in magnesium-bearing mineral content, magnesium release from the tailings would be expected to be twice that from the waste rock due to particle size influences alone. If this is so, mineralogical differences between the tailings and rock had little influence on magnesium release. This further suggests, given that the olivine content of the waste rock was more than 2.5 times that of the tailings, that olivine was not a major contributor of magnesium to solution. That is, if olivine were a major magnesium contributor, the waste rock would yield much higher magnesium release than the tailings.

Respective annual rates of copper and nickel from the waste rock, on average, were 40 and 150 times greater than those from the tailings. Similarly annual rates of zinc and cobalt release from the waste rock averaged 4 and 8 times those from than the tailings, respectively. These rates tended to be highest during years 2 and 3 (Figure 32). The differences are attributed to pH values from the waste rock that was near 1 unit lower than those from the tailings. Concentrations of copper, nickel, and cobalt in the waste rock were 2.0, 1.2, and 1.2 times those in the tailings, respectively (Appendix 1, Table A1.10). These differences and a lower degree of base metal sulfide liberation in the tailings probably contributed to a lesser degree.

In summary, the rate of sulfate release from the tailings averaged 80 percent of that from the waste rock sample of similar sulfur content. The difference is believed to be largely the result of a higher degree of sulfide mineral liberation in the rock. In contrast, rates of calcium and magnesium release from the tailings were roughly twice those from the waste rock. The combination of lower sulfide mineral oxidation (and consequent acid production) and higher calcium and magnesium silicate mineral dissolution (and consequent acid neutralization) resulted in drainage pH values from the tailings that were roughly a unit higher than those from the waste rock. Due to the lower waste rock drainage pH (and to a lesser extent, elevated base metal content of the waste rock), base metal release rates from the waste rock averaged 4 to 150 times those from the tailings.

5.6 Comparison of Partridge River and South Kawishiwi Intrusion Samples

The dissolution behavior of Partridge River Intrusion rock (PRI), samples from the Babbitt and Dunka Road prospects, was compared to that of the South Kawishiwi Intrusion rock (SKI), samples from the Dunka Mine. Data from laboratory experiments on samples with diameters of 0.053 to 0.149 mm were used for the comparison because that was the only dissolution test to which the South Kawishiwi samples were subjected. Data from seven PRI samples ($0.07 \leq$ percent sulfur ≤ 0.94) and eleven SKI samples ($0.18 \leq$ percent sulfur ≤ 1.16) were used.

Whereas comparisons of the intrusions were made based on normalized sulfur content, the variation of base metal contents with sulfur for the intrusions differed. Copper and nickel contents in the PRI rock tended to increase with sulfur content but not in the SKI samples. Consequently, copper and nickel contents of the PRI higher sulfur samples were 2 to 5 times higher than respective contents of the SKI samples with similar sulfur content (Figures 33, 34). Conversely, the zinc content of SKI samples increased with sulfur content, but the PRI samples did not respond in a similar manner (Figure 35). Cobalt content did not vary systematically with sulfur content for samples in either intrusion (Figure 36).

PRI samples on average had higher olivine content than the SKI samples, 31 vs 8%. SKI samples on average had higher plagioclase and hypersthene (compared to Opx), 52 vs 45 and 15 vs 3.5%, respectively than the PRI. Average contents between the two intrusion samples were similar for augite, biotite, potassium feldspar, and ilmenite (Appendix 1, Table A1.14).

Drainage pH from the PRI samples tended to be higher than that from the SKI samples. For a specified sulfur content, the minimum drainage pH from the PRI samples was roughly 0.7 units higher than that from the SKI samples (Figure 37). Unlike the PRI samples which were mainly circumneutral (exceptions being the 0.63 and 0.94%S samples), SKI samples with sulfur content as low as 0.41% generated acidic drainage within 400 weeks of reaction. However, the drainage pH from PRI samples reached minimums by week 50-100, the SKI samples declined more slowly, taking as long as 300-350 weeks to reach minimum pH values (Figure 38).

During the first three years, the annual rate of sulfate release from the PRI samples averaged 1.2 to 2 those from the SKI samples of similar sulfur content (Figure 39, 40, Table 15). Sulfate release from the PRI samples peaked in the first 30-50 weeks, a trend associated with the relatively rapid initial decrease in pH. After 250-300 weeks, as drainage pH from the SKI samples continued to decline, sulfate responded with release rates as much as 10 times those from the PRI samples (Figure 40, Table 15). This rapid increase in sulfate release occurred when pH fell below 4, and the increase likely reflected biological mediation of sulfide mineral oxidation. During the 8 year period of record, average sulfur depletion in the PRI and SKI samples was 41 and 54 percent respectively.

The cause of differences in drainage pH from SKI and PRI samples was investigated further. Solid-phase and drainage quality data for the two highest sulfur PRI samples (0.63 and 0.94% S) was compared with associated data for the three SKI samples with comparable sulfur contents (0.57, 0.58, and 0.71%; 0.71, 1.12, and 1.16%, respectively). The minimum drainage pH for the

0.63% S PRI sample was 5.2 as compared to a range of 3.6 to 4.0 for the corresponding SKI samples (Appendix 6, Figure A6.4). Similarly the minimum drainage pH for the 0.94% S PRI sample was 4.9, as compared to a range of 3.2 to 3.6 for the corresponding SKI samples (Appendix 6, Figure A6.5). This indicates that the SKI samples had either a higher capacity to produce acid or a lower capacity to neutralize acid.

Because the SKI samples had lower contents of copper and nickel, a larger fraction of their sulfur was associated with iron. Thus, for samples of the same sulfur content the SKI samples had more iron sulfide and therefore more potential to produce acid. This hypothesis was pursued by comparing solid-phase and reactor release rate data for PRI samples (0.63 and 0.94% S) with associated data for the three SKI samples with comparable sulfur contents (0.57, 0.58, and 0.71%; 0.71, 1.12, and 1.16%, respectively). This comparison indicated that, per unit sulfur present, acid producing fractions of sulfides for the 0.63% S and 0.94% S PRI samples were 0.70 and 0.65, respectively. For comparison the average acid producing fractions of sulfides for the 0.57% S, 0.58% S, and 0.71% S SKI samples was 0.85 and for the 0.71% S, 1.12% S, and 1.16 % S SKI samples was 0.91 (Appendix 6, Table A6.4). Thus, the acid producing fraction of sulfide present in the two PRI sample was roughly 80 and 70 percent, respectively, of the average for the SKI samples with similar sulfur contents. Thus, the amount of acid-producing iron sulfide per unit sulfur present in the SKI samples was greater than that present in the PRI samples. This likely contributed to the lower drainage pH values observed for the SKI samples. It should be noted that this solid-phase difference is rather subtle relative to the large difference in minimum pH values, suggesting a high sensitivity of drainage pH to small changes in sulfur content.

For the five SKI and two PRI samples discussed above, there was no indication that the potential of SKI samples to neutralize acid was less than that of the PRI samples. The CO_2 content range for SKI samples was <0.03 – 0.10% as opposed to 0.2% and 0.07%, respectively, for the PRI samples with sulfur contents of 0.63% and 0.94%. Although the 0.2% CO_2 concentration for the 0.63% S PRI sample is easily the highest reported, the 0.07% concentration for the second sample is in the middle of the range for the SKI samples. Consequently, the calcite contents inferred by the CO_2 contents do not provide compelling insight into the lower drainage pH from the SKI samples.

Dissolution of minerals releasing calcium and magnesium provided the majority of acid neutralization. Besides trace amounts of calcite, calcium was largely present in plagioclase (~70-80% of Ca) and augite. There were some differences in the neutralizing silicate mineral contents and chemistries for the two intrusions. The plagioclase contents (volume percent) of the five SKI samples averaged 48% as compared to 40% for the two PRI samples. The respective average mole fractions of calcium present were 0.50 and 0.55 (Appendix 6, Table A6.5). Thus, the plagioclase content suggested slightly greater neutralization capacity for the SKI samples while the opposite was implied by the calcium contents of the plagioclase. Augite contents and chemistries of the SKI and PRI samples were similar. Thus there is no evidence suggesting that calcium-bearing mineral dissolution would limit acid neutralization by the SKI rocks. This is consistent with the observation that average rates of calcium release as a function of sulfur

content for the SKI samples tended to exceed those for the PRI samples (Appendix 6, Figure A6.13).

The largest silicate mineral difference between rocks from the two intrusions was that of olivine content, which averaged 12 percent for the five SKI samples and 34 percent for the two PRI samples (Appendix 6, Table A6.5). This suggests that the deficiency of olivine might have contributed to the lower drainage pH of the SKI samples. However, magnesium release rates as a function of sulfur release rates were not substantially different between the intrusions (Appendix 6, Figure A6.14), indicating that the difference in olivine contents was not a major factor in the observed acidification of drainage from the SKI samples.

Comparison of heavy metal release from the PRI and SKI intrusions is limited to data prior to week 74 during which drainage pH was circumneutral for all cases (Table 16). Heavy metal concentrations from the SKI samples were determined only sporadically after this time. Prior to week 74, nickel release from the PRI intrusion samples was roughly three times that from the SKI samples (Figure 41). Copper, cobalt, and zinc release from the PRI intrusion samples was not obviously different from that from the SKI samples (Figures 42-44). These trends in heavy metal release coincide with the initial compositions of the intrusion samples (Appendix 1, Table A1.13).

6. SUMMARY

In August 2003 laboratory dissolution tests were initiated on 10 Duluth Complex rock samples from the Babbitt and Dunka Road prospects of the Partridge River intrusion. These experiments remain in progress and the present report analyzes and summarizes data generated through week 400.

Dissolution tests were conducted on 10 coarse samples ($d < 6.35$ mm = 0.25 in.; $0.13 \leq \%S_T \leq 1.36$; $0.05 \leq \%CO_2 \leq 0.21$) and seven fine samples ($0.053 < d < 0.149$ mm; $0.07 \leq \%S_T \leq 0.94$; $0.07 \leq \%CO_2 \leq 0.21$). All samples were characterized with respect to chemistry, mineral content, and mineral chemistry. Of the 10 coarse samples, seven were from the Babbitt prospect and three from the Dunka Road prospect. For these samples particle size distribution and abbreviated chemistry (S, CO_2, Cu, Ni) as a function of particle size were also determined (Appendix 1, Table A1.2). The seven 0.053-0.149 mm samples consisted of six samples from the Babbitt prospect and one from the Dunka Road prospect.

The tests generated data that described mineral dissolution reactions that release acid and heavy metals (sulfide mineral oxidation reactions) and reactions that neutralize acid (silicate mineral dissolution reactions). In general, drainage pH tended to decrease and release rates of heavy metals (copper, nickel, cobalt, zinc) and major cations (calcium, magnesium, sodium, potassium) tended to increase with increasing sulfur content. These results were consistent with the previously reported 1) dependence of sulfide mineral oxidation (as reflected by sulfate release) and the attendant acid production on the amount of sulfide minerals present and 2) importance of sulfide mineral oxidation as a driving force for release of heavy metals and major cations.

The pH of drainage from the 10 coarse samples decreased with increasing sulfur content, with a minimum value of 4.1. For all but the two highest sulfur samples, pH decreased throughout the eight-year experiment. Numerous relationships between drainage quality and both solid-phase sulfur content and dissolution time were identified and quantified to accurately describe solute release and its dependence on solid-phase composition and dissolution time. Regression analysis was used to quantify a strong relationship between rates of sulfate release (reflecting sulfide mineral oxidation) and sulfur content, reflecting exposed sulfide mineral surface area ($0.91 < r^2 < 0.94$). This analysis was used to derive composite annual sulfate release rates (per gram sulfur), which were found to decrease slightly over the eight-year experiment. Release rates for copper, nickel, cobalt, and zinc also correlated highly with sulfur content (typically $0.80 < r^2 < 0.95$). Copper release rates for year 8 were roughly five times those during year 1. In contrast rates of nickel, cobalt, and zinc release at the end of the experiment were roughly half those at the beginning. Magnesium was the dominant cation released, followed by calcium, and sodium. Their releases tended to parallel that of sulfate, reflecting silicate minerals dissolving to neutralize acid produced by iron sulfide mineral oxidation.

Similar analysis was conducted on data generated in the seven finer-sized samples. As with the larger particles, drainage pH decreased and release rates for sulfate, major cations, and heavy metals increased with increasing sulfur content. Although the relationship between minimum drainage pH and sulfur content was similar to that for the larger particles, pH from the finer

samples declined more rapidly, reaching minimums between weeks 60 and 120. Sulfate release from the fine particles were initially three times those from the coarse particles, reflecting the greater sulfide mineral surface area per unit mass of the fine particles, and this rapid release contributed to the aforementioned early pH depression. Sulfate release rates from the fine particles decreased fairly quickly over time and by year seven equaled those from the coarse particles, for which rates decreased more slowly. The rapid decline of rates from the fine particles was attributed to sulfur depletion and probable coating of sulfide minerals.

Dissolution testing of Dunka Road samples was limited to one sample in the 0.053-0.149 mm size fraction and three samples in the d < 6.35 mm fraction. The most notable difference between Dunka Road and Babbitt prospect rock was elevated heavy metal release from the latter. Whereas some of the difference could be ascribed to higher metal content of the Babbitt prospect rock, differences between prospects in total sulfide mineral oxidation rates and drainage pH also played a role. Because the number of samples for comparison was limited, these results should be viewed as preliminary.

The Partridge River rock samples yielded higher sulfate rates than Partridge River tailings of similar sulfur content, due to greater sulfide mineral exposure in the rock samples. Heavy metal release from the rock was also higher than that from the tailings. This was due to the aforementioned elevated sulfide mineral exposure, higher heavy metal contents, and lower drainage pH associated with the rock samples. Drainage pH from the Partridge River rock tended to be about 0.7 units higher than that from previously tested South Kawishiwi rock of similar sulfur content. This was attributed to the higher heavy metal content (and consequent lower iron sulfide content) of the Partridge River rock examined in the present study.

Additional work should be conducted to further benefit the application of results generated to date. This includes continued dissolution testing to confidently define long-term trends, analysis of leached solids to increase understanding of chemical reactions controlling solute release (e.g. chemical precipitation and adsorption), and more detailed analysis of the present data, including calculation of mineral dissolution rates. The more detailed analysis should include calculation of mineral dissolution rates to allow application of the data generated to a wider range of mineralogical compositions.

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Table 1. Dunka Road drill hole data obtained from PolyMet (16 August 2001) for samples selected for laboratory dissolution tests.

DNR Sample #	Drill Hole	Footage		Lerch Total %S	Pmet Total %S[1]	Cu ppm	Grain Size[2]	% Plag	% Olivine	% Pyroxene	Rock Unit
12046 [3]	99318	60	65	0.03	0.15	290	m-vcg	50	40	10	3
	99318	55	60	1.30	0.93	1835	mg	55	45	0	3
12049 [3]	00340	165	170	0.25	0.31	391	m-fg	80	20	trace	3
	00340	55	60	0.36	0.33	539	f-cg	50	50	0	3
12051	00366	110	115	0.23	0.45	1365	m-fg	77	20	3	3

[1] Total %S reported for drill core by PolyMet were included for comparison with analyses of –6.25-mm samples by Lerch Brothers.

[2] f = fine, m = medium, c = coarse, vc = very coarse, g = grain

[3] Drill hole footages were mixed to produce samples 12046 and 12049 with sulfur contents of 0.61 and 0.31 percent, respectively.

Table 2. Particle size distribution data for the d < 6.35 mm samples (percent passing). Analyses by Lerch Brothers Inc.

Size Fraction		Sample / Cell #									
(mesh)	(microns)	12037/9	12040/10	12041/11	12042/12	12033/14	12112/15	12110/16	12051/17	12046/18	12049/19
%S		0.13	0.21	0.33	0.55	0.72	1.03	1.36	0.23 (D)	0.31 (D)	0.61 (D)
10	2000	42.26	39.90	35.75	40.78	36.83	56.99	53.00	28.67	31.07	32.97
20	841	27.01	24.75	22.06	26.22	23.02	35.23	30.25	18.36	19.68	20.64
35	500	17.72	16.27	14.79	17.77	15.09	23.28	19.18	12.74	13.21	13.88
48	297	14.37	13.22	11.81	14.42	12.42	18.69	15.86	10.41	11.01	11.50
65	210	11.32	10.37	9.44	11.47	9.82	14.52	12.33	8.24	8.78	9.49
100	149	8.83	7.84	7.13	8.61	7.57	10.90	9.26	6.17	6.74	7.29
150	105	6.53	5.79	5.50	6.74	5.59	7.82	6.99	4.76	5.02	5.46
200	74	4.80	4.29	3.93	5.00	4.05	5.65	5.20	3.43	3.80	3.81
270	53	3.36	2.89	2.77	3.63	2.74	3.96	3.76	2.41	2.57	2.49

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Note: All samples had 100% passing ¼ inch.

Table 3. Total sulfur analyses. Analyses by Lerch Brothers Inc.

Sample #	Reactor	%S
75 g, 0.053 < d < 0.149mm rock		
12037	1	0.07
12040	2	0.11
12041	3	0.20
12042	4	0.31
12042	5 (dup. of 4)	0.31
12033	6	0.63
12110	7	0.94
12046	8 (D)	0.31
1000 g, d < 6.35 mm rock		
12037	9	0.13
12040	10	0.21
12041	11	0.33
12042	12	0.55
12042	13 (dup. of 12)	0.55
12033	14	0.72
12112	15	1.03
12110	16	1.36
12051	17 (D)	0.23
12046	18 (D)	0.31
12049	19 (D)	0.61

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 4. Whole rock chemistry (%) for Duluth Complex waste rock reactors (0.053<d<0.149 mm) and humidity cells (d<6.35 mm). Analysis performed by Lerch Brothers Inc. and ALS USA Inc.

Reactor	1	2	3	4	6	7	8 (D)
%S	0.07	0.11	0.2	0.31	0.63	0.94	0.31
CO_2	0.08	0.13	0.12	0.2	0.2	0.07	0.08
SiO_2	45.1	44.5	44.9	43.9	43.3	43.7	46
Al_2O_3	17.35	16.3	17.35	14.75	13.7	16.1	19.9
Fe_2O_3	14.15	14.4	14.35	14.25	15.8	16.25	11.2
CaO	8.29	8.35	8.53	7.28	6.58	8.18	9.5
MgO	10.35	8.66	8.12	9.59	9.67	8.09	7.53
Na_2O	2.3	2.46	2.51	2.25	2.25	2.38	2.56
K_2O	0.39	0.55	0.52	0.52	0.57	0.54	0.31
Cr_2O_3	0.02	0.03	0.03	0.02	0.02	0.02	0.04
TiO_2	0.95	2.4	1.83	1.91	1.78	2.01	0.93
MnO	0.16	0.16	0.16	0.16	0.17	0.17	0.13
P_2O_5	0.08	0.14	0.14	0.18	0.22	0.18	0.08
SrO	0.03	0.03	0.03	0.03	0.03	0.03	0.03
BaO	0.01	0.02	0.02	0.02	0.02	0.02	0.01

HC	9	10	11	12	14	15	16	17 (D)	18 (D)	19 (D)
%S	0.13	0.21	0.33	0.55	0.72	1.03	1.36	0.23	0.31	0.61
CO_2	0.05	0.08	0.08	0.08	0.08	0.06	0.1	0.08	0.07	0.21
SiO_2	45	45.4	44.8	44.2	43.7	44.3	43.5	45.6	47.4	46.2
Al_2O_3	18.15	17.3	18	16.65	14.95	17.8	16.05	19.3	20.7	18.8
Fe_2O_3	12.6	13.4	13.2	13.7	16.5	14.8	16.45	12.4	11.15	13.4
CaO	8.6	8.66	8.68	8.09	7.25	8.75	8.36	9.25	10.25	9.65
MgO	9.29	8.14	7.42	9.24	10.05	6.97	7.84	9.16	7.31	7.96
Na_2O	2.4	2.58	2.57	2.37	2.26	2.52	2.33	2.34	2.6	2.49
K_2O	0.42	0.62	0.58	0.63	0.65	0.53	0.56	0.28	0.34	0.39
Cr_2O_3	0.02	0.02	0.02	0.01	0.02	0.04	0.04	0.03	0.04	0.05
TiO_2	0.96	2.25	1.69	1.84	1.85	1.91	2.11	0.61	0.85	1.62
MnO	0.14	0.15	0.15	0.15	0.17	0.15	0.16	0.14	0.12	0.14
P_2O_5	0.13	0.18	0.17	0.2	0.22	0.16	0.2	0.09	0.11	0.11
SrO	0.04	0.03	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03
BaO	0.01	0.02	0.02	0.02	0.02	0.02	0.02	0.01	0.01	0.01

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 5. Heavy metal composition (%) for Duluth Complex waste rock reactors (0.053<d<0.149 mm) and humidity cells (d<6.35 mm). Analysis performed by ALS USA Inc.

Reactor	1	2	3	4	6	7	8 (D)
%S	0.07	0.11	0.2	0.31	0.63	0.94	0.31
Ag	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	0.00012	<0.00005
As	0.0009	<0.0005	<0.0005	0.0007	0.0011	0.0008	0.0006
Cd	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005
Co	0.0079	0.0069	0.0071	0.0081	0.0095	0.0096	0.0063
Cu	0.0381	0.0686	0.1	0.1835	0.269	0.529	0.0335
Mo	<0.0001	0.0001	0.0001	<0.0001	<0.0001	<0.0001	0.0001
Ni	0.0416	0.0386	0.0419	0.0626	0.0868	0.1055	0.0293
Pb	<0.0002	<0.0002	<0.0002	0.0003	0.0004	0.0004	0.0004
Sc	0.001	0.001	0.0014	0.0014	0.0013	0.0016	0.0009
Zn	0.0101	0.0101	0.0108	0.0109	0.0122	0.0121	0.0077

HC	9	10	11	12	14	15	16	17 (D)	18 (D)	19 (D)
%S	0.13	0.21	0.33	0.55	0.72	1.03	1.36	0.23	0.31	0.61
Ag	<0.00005	<0.00005	<0.00005	<0.00005	0.00006	0.00011	0.00017	<0.00005	<0.00005	<0.00005
As	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	0.0018	<0.0005	0.0005	<0.0005
Cd	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005
Co	0.0071	0.0067	0.0069	0.0078	0.0101	0.0098	0.0101	0.007	0.0069	0.0083
Cu	0.0779	0.1175	0.157	0.281	0.393	0.464	0.648	0.0899	0.0478	0.0763
Mo	<0.0001	0.0001	<0.0001	<0.0001	0.0001	0.0001	0.0001	<0.0001	<0.0001	<0.0001
Ni	0.0436	0.0426	0.0456	0.0612	0.0986	0.1095	0.125	0.0477	0.04	0.0513
Pb	<0.0002	0.0003	0.0002	0.0002	0.0006	0.0004	0.0005	<0.0002	0.0002	0.0002
Sc	0.0009	0.0017	0.0012	0.0013	0.0014	0.0013	0.0016	0.0006	0.0008	0.0014
Zn	0.0095	0.0104	0.0106	0.0104	0.013	0.0108	0.0118	0.008	0.0078	0.0091

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 6. Chemical composition (%) comparison for Babbitt and Dunka Road prospect samples

%S	CaO	MgO	Na_2O	K_2O	Cu	Ni	Co	Zn
Fine rock (75 g, 0.053<d<0.149 mm): Reactors 4, 5 vs 8								
0.31[1] (B)	7.28	9.59	2.25	0.52	0.184	0.0626	0.0081	0.0109
0.31 (D)	9.50	7.53	2.57	0.31	0.0335	0.0293	0.0063	0.0077
B/D	0.77	1.27	0.88	1.68	5.49	2.14	1.29	1.42
Coarse rock (1000g, d<6.35 mm): H. Cell 10 vs 17								
0.21 (B)	8.66	8.14	2.58	0.62	0.118	0.0426	0.0067	0.0104
0.23 (D)	9.25	9.16	2.34	0.28	0.0899	0.0477	0.007	0.008
B/D	0.94	0.89	1.10	2.21	1.31	0.89	0.96	1.3
Coarse rock (1000g, d<6.35 mm): H. Cell 11 vs 18								
0.33 (B)	8.68	7.42	2.57	0.58	0.157	0.0456	0.0069	0.0106
0.31 (D)	10.25	7.31	2.6	0.34	0.0478	0.04	0.0069	0.0078
B/D	0.85	1.02	0.99	1.71	3.28	1.14	1.00	1.36
Coarse rock (1000g, d<6.35 mm): H. Cells 12, 13 vs 19								
0.55[1] (B)	8.09	9.24	2.37	0.63	0.281	0.0612	0.0078	0.0104
0.61 (D)	9.65	7.96	2.49	0.39	0.0763	0.0513	0.0083	0.0091
B/D	0.84	1.16	0.95	1.62	3.68	1.19	0.94	1.14

B, D indicate Babbitt prospect and Dunka Road prospect, respectively
[1] Average values for duplicates

Table 7. Mineral content and chemistry comparison for Babbitt and Dunka Road prospect samples.

75 g, 0.053<d<0.149 mm rock, 0.31 %S.

Mineral	Babbitt Prospect, 0.31 %S		Dunka Road Prospect, 0.31 %S	
	Vol %	Formula	Vol %	Formula
Plagioclase	40	$Ca_{0.56}Na_{0.39}$	57	$Ca_{0.63}Na_{0.38}$
Olivine	30	$Mg_{1.25}Fe_{0.75}$	28	$Mg_{1.21}Fe_{0.80}$
Augite	14	$Ca_{0.78}Mg_{0.74}$	3.9	$Ca_{0.86}Mg_{0.82}$
Hypersthene	3.1	$Mg_{1.27}$	2.4	$Mg_{1.26}$
Biotite	2.3	$Mg_{3.95}K_{1.7}$	0.0	
K-spar	2.3	$K_{0.87}Na_{0.085}$	1.6	$K_{0.80}Na_{0.11}$

1000 g, d<6.35 mm rock, ~0.2 %S.

Mineral	Babbitt Prospect, 0.21 %S		Dunka Road Prospect, 0.23 %S	
	Vol %	Formula	Vol %	Formula
Plagioclase	55	$Ca_{0.55}Na_{0.41}$	53	$Ca_{0.61}Na_{0.36}$
Olivine	15	$Mg_{1.19}Fe_{0.80}$	42	$Mg_{1.23}Fe_{0.76}$
Augite	7.2	$Ca_{0.81}Mg_{0.75}$	1.7	$Ca_{0.80}Mg_{0.79}$
Hypersthene	3.6	$Mg_{1.08}$	0.8	$Mg_{1.37}$
Biotite	4.5	$Mg_{3.68}K_{1.75}$	0.8	$Mg_{3.75}K_{1.72}$
K-spar	1.8	$K_{0.81}Na_{0.16}$	0.0	

1000g, d<6.35 mm rock, ~0.3 %S.

Mineral	Babbitt Prospect, 0.33 %S		Dunka Road Prospect, 0.31 %S	
	Vol %	Formula	Vol %	Formula
Plagioclase	46	$Ca_{0.55}Na_{0.41}$	60	$Ca_{0.60}Na_{0.37}$
Olivine	28	$Mg_{1.11}Fe_{0.87}$	23	$Mg_{1.22}Fe_{0.77}$
Augite	8.0	$Ca_{0.80}Mg_{0.78}$	3.4	$Ca_{0.86}Mg_{0.78}$
Hypersthene	1.8	$Mg_{1.17}$	2.5	$Mg_{1.23}$
Biotite	2.7	$Mg_{3.54}K_{1.71}$	0.0	
K-spar	1.8	$K_{0.78}Na_{0.17}$	1.7	$K_{0.92}Na_{0.081}$

1000g, d<6.35 mm rock, ~0.6 %S.

Mineral	Babbitt Prospect, 0.55 %S		Dunka Road Prospect, 0.61 %S	
	Vol %	Formula	Vol %	Formula
Plagioclase	49	$Ca_{0.58}Na_{0.39}$	48	$Ca_{0.57}Na_{0.38}$
Olivine	20	$Mg_{1.27}Fe_{0.73}$	27	$Mg_{1.16}Fe_{0.84}$
Augite	3.6	$Ca_{0.81}Mg_{0.78}$	5.0	$Ca_{0.78}Mg_{0.76}$
Hypersthene	7.1	$Mg_{1.18}$	2.5	$Mg_{1.09}$
Biotite	1.8	$Mg_{3.96}K_{1.68}$	0.0	
K-spar	2.7	$K_{0.74}Na_{0.22}$	0.8	$K_{0.82}Na_{0.16}$

Table 8. Water retention (g) for the Duluth Complex waste rock samples for weeks 1-400.

Reactor	% S	Water retention before rinse[1]					Water retention after rinse				
		count	min	max	mean	SD	count	min	max	mean	SD
75-g Reactors											
1	0.07	389	-4.9	4.6	-2.0	1.6	399	8.8	38.0	25.0	5.6
2	0.11	396	-7.9	4.2	-2.6	2.6	397	7.6	33.1	22.2	5.4
3	0.20	396	-8.3	1.1	-2.6	2.5	395	5.7	31.0	17.8	5.7
4	0.31	398	-10.9	5.5	-3.0	3.4	398	2.3	36.6	16.7	7.5
5	0.31	399	-11.1	0.3	-6.0	4.0	399	-0.4	29.6	13.8	7.0
6	0.63	398	-3.0	5.7	-0.8	1.3	397	16.9	36.1	27.0	3.3
7	0.94	396	-7.9	9.1	-1.8	2.9	394	7.4	37.8	24.3	6.4
8 (D)	0.31	395	-3.1	2.6	-1.3	1.4	396	14.7	34.8	23.9	4.1
1000-g Cells											
9	0.13	395	90	133	108.1	8.0	397	92	150	117.1	10.0
10	0.21	399	83	126	106.0	6.7	398	88	136	114.8	7.1
11	0.33	398	82	124	101.2	6.9	397	96	147	111.8	6.6
12	0.55	399	96	126	107.9	5.1	400	101	138	119.4	5.9
13	0.55	399	74	110	93.8	5.6	400	80	115	99.7	5.6
14	0.72	397	98	119	108.8	3.4	398	103	132	118.1	4.4
15	1.03	397	133	164	151.3	4.9	399	136	176	166.4	6.5
16	1.36	399	135	161	148.4	4.5	400	140	178	163.4	6.4
17 (D)	0.23	398	79	123	99.1	6.4	399	86	161	108.3	7.6
18 (D)	0.31	398	71	111	88.9	4.2	399	84	119	99.3	4.4
19 (D)	0.61	399	105	135	119.1	5.1	400	107	145	125.7	5.8

[1] To calculate water retention the original mass of the reactor and dry solids (at the beginning of the experiment) was subtracted from the mass before the rinse. Negative values indicate that some solids were lost during the course of the experiment.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 1 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 1 0.07 %S	1	6.48	65.6	26.2	64.9	18.8	41.6	8.59	17	4.35	47.1	18.6
	2	6.42	68.9	21.8	38.6	10.3	37.8	14.1	14.4	3.32	11.4	6.1
	3	6.42	32.9	6.61	23.1	3.1	26.1	2.21	10.5	2.98	2.73*	0.0168
	4	6.41	22.1	3.96	15.9	3.55	20.8	5.78	8.36	4.23	2.73*	0.0239
	5	6.58	14	1.92	12.3	1.42	22.4	1.41	4.66*	0.0633	5.43	6.03
	6	6.65	7.58*	5.26	17	5.31	22.4	4.32	6.69	5.32	4.25	2.93
	7	6.42	6.86*	3.15	18.6	5.14	23.4	4.7	5.68	2.25	2.79*	0.0303
	8[2]	6.48	7.63*	3.82	12.8	3.46	21.1	2.5	4.79*	0.0504	2.82*	0.0268
	Total[3]	6.42	28.2	9.37	25.4	5.68	26.9	4.19	9.02	1.94	9.9	6.38
Reactor 2 0.11 %S	1	6.38	145	80.2	106	21.7	73.7	29.8	19.3	4.78	52.8	22.5
	2	6.23	91.7	31.2	40.5	12.6	50.4	14.6	15.9	2.72	16.8	8.04
	3	6.45	44.6	8.33	25.3	3.49	35.3	4.11	12.4	1.46	3.45	1.34
	4	6.34	29.1	3.53	15.4	3.75	24.1	3.99	11.5	4.22	5.46	3.09
	5	6.53	21.1	2.85	11.9	1.62	26.1	1.93	4.82*	0.0324	4.98	3.09
	6	6.49	16.7	3.3	13.8	3.32	26.6	2.64	6.17	2.33	5.81	4.39
	7	6.36	11.3	5.28	19.8	8.6	29.3	8.45	6.64	3.06	6.67	4.71
	8[2]	6.43	10.1	4.94	12.1	5.07	23.7	4.5	4.86*	0.0228	2.86*	0.0134
	Total[3]	6.35	46.2	27.1	30.5	6.73	36.2	9.45	10.2	1.76	12.4	7.11
Reactor 3 0.20 %S	1	6.06	245	129	145	53.1	88.9	39.6	32.1	8.56	96.7	47.9
	2	5.75	133	31.6	45.8	11.3	34.2	12.6	27.8	7.93	20.3	11.8
	3	5.68	72.6	11.5	26.8	3.01	20	3.05	15.8	3.57	3.81	1.49
	4	5.71	50	3.76	19.6	3.98	14.7	2.54	10.3	4.34	2.8*	0.129
	5	5.83	34.6	6.05	10.7	0.646	12.2	1.34	4.92*	0.0301	2.89*	0.0189
	6	5.86	25.5	3.04	10.4	3.96	9.7	3.73	5.83	2.59	3.42*	1.5
	7	5.79	21	3.92	13.8	4.18	9.1	3.53	7.81	5.88	2.87*	0.0302
	8[2]	5.98	16.5	2.47	6.46	0.394	4.68*	0.0801	10.3	10.7	2.91	2.91
	Total[3]	5.73	74.8	43.6	34.8	17.7	24.2	13.2	14.6	3.51	17	17
Reactor 4 0.31 %S	1	6.34	256	189	179	63	114	69.6	27.4	9.01	78.8	39.2
	2	5.84	183	37.8	48.6	13.4	91.1	17.5	23.1	4.21	18.6	10.7
	3	6.09	102	13.1	26.5	4.48	67.4	8.4	14.8	2.86	4.46	1.99
	4	6.07	68.9	11.8	17.4	2.31	51.2	7.86	8.66	4.36	2.88*	0.052
	5	6.21	45.8	7.16	10.3	2.09	45.9	5.67	4.91*	0.0297	2.9*	0.0366
	6	6.28	30.8	5.74	10.5	3.11	36.2	4.25	5.95*	2.64	3.5*	1.54
	7	6.23	25.4	5.62	11.9	6.28	36.4	8	8.2	7.89	2.95*	0.0426
	8[2]	6.29	19.8	3.04	6.35	1.19	27.3	4.52	8.54	7.03	2.95*	0.00858
	Total[3]	6.07	91.4	63.6	38.8	21	58.7	22.2	12.7	3.02	14.6	13.6

[1] 5th percentile value
[2] Year 8 represented by weeks 365-400
[3] Average and standard deviation for entire period of record
* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.
Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 2 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 5 0.31 %S (dup. of 4)	1	6.30	238	178	172	54.2	107	62.6	25.1	7.26	70.2	30.4
	2	5.93	163	30.4	43.9	14	81.3	17.4	22.3	1.93	15	7.69
	3	6.16	96.9	12.4	24.6	2.94	61.1	8.75	17.5	2.14	11.4	3.8
	4	6.09	62.7	13.2	17.1	4.08	44.3	10.2	11.5	4.48	7.59	3.9
	5	6.34	38.7	6.8	9.68	0.837	38.5	4.76	6.93	4.38	3.73	1.8
	6	6.28	24.1	2.93	9.34	2.51	33.7	2.7	4.92*	0.0316	2.89*	0.0186
	7	6.34	21.2	7.17	11.9	5.18	41.7	6.8	12.3	9.29	2.9*	0.0199
	8[2]	6.38	19.1	2.79	9.35	9.24	31.8	6.54	4.94*	0.0288	2.9*	0.0169
	Total[3]	6.10	83	59.6	37.3	17.7	55	19.7	13.2	3.33	14.6	10.2
Reactor 6 0.63 %S	1	5.96	322	278	138	73.2	151	110	24.2	9.91	65.5	24.3
	2	5.37	320	54.7	69.3	14.9	133	29.7	38.7	2.3	34.8	6.72
	3	5.17	232	35.3	47.8	6.14	89.5	15.8	30.3	4.64	17.7	6.64
	4	5.19	159	21.4	29.7	4.23	65	13.1	22	2.87	21.8	3.4
	5	5.31	130	19.7	19.4	2.11	64.6	9.92	9.32	5.95	9.89	10.2
	6	5.40	90.2	10.3	15.2	6.08	45.6	18.7	10.5	4.12	4.94	3.46
	7	5.69	74	9.86	16.6	3.85	45.9	4.61	17.7	3.57	4.35	2.59
	8[2]	5.67	71.5	10.7	14.6	4.28	42.6	5.12	23.5	9.17	5.43	5.28
	Total[3]	5.19	175	91.3	43.9	24.1	79.8	34.9	22	2.84	20.6	7.1
Reactor 7 0.94 %S	1	4.53	530	263	191	65.3	93.7	47.7	49.9	26.6	81.4	32.3
	2	4.42	452	44.9	83.8	16.3	45.2	11.6	56.4	6.33	35.2	19.2
	3	4.50	314	34	54.3	7.9	28	4.14	38.3	6.06	12.2	8.14
	4	4.55	239	26.8	39.7	3.98	21.3	4.21	30.1	4.78	9.42	2.22
	5	4.55	193	26.6	26.1	5.03	18.2	2.69	17.6	5.12	2.8*	0.0214
	6	4.60	154	15.9	28.7	6.65	18.3	2.2	17.1	2.2	6.17	6.15
	7	4.49	140	19.5	26.7	6.91	15	6.5	25.9	6.67	5.13	4.49
	8[2]	4.69	125	5.67	22.9	3.02	15.8	1.21	24.6	9.76	2.81*	0.0119
	Total[3]	4.47	268	85	59.1	21	31.9	15.6	32.5	7.62	19.4	11.2
Reactor 8(D) 0.31 %S	1	6.34	156	78.7	127	32.3	57.3	30.7	14.3	3.71	32.5	14.8
	2	5.89	135	36.3	48.5	18.3	74.8	16.7	12.9	3.2	7.63	4.16
	3	6.19	69	24.2	22.6	5.19	51.9	12.9	7.76	2.94	3.14*	1.1
	4	6.15	42.7	6.9	15.5	2.86	35.2	6.45	4.73*	0.0521	2.78*	0.0307
	5	6.28	24.8	4.56	8.57	0.659	30.3	2.57	4.78*	0.0223	5.29	4.14
	6	6.32	19.8	4.92	11.1	3.17	29.6	4.78	5.44*	1.79	2.8*	0.023
	7	6.23	15.1	5.55	14.2	6.21	29	6.35	4.76*	0.068	3.5*	1.82
	8[2]	6.18	13.7	6.27	8.55	0.492	25.4	3.85	4.8*	0.0358	2.82*	0.0211
	Total[3]	6.05	59.5	26.1	32	11.1	41.7	9.45	7.44	1.62	7.55	4.99

[1] 5th percentile value
[2] Year 8 represented by weeks 365-400
[3] Average and standard deviation for entire period of record
* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.
Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 3 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 9 0.13 %S	1	6.92	32.9	6.46	34.5	3.83	29.4	11.3	7.67	5.21	7.97	2.15
	2	6.76	32	3.91	20.8	3.83	17.4	1.7	3.02	0.486	5.76	0.39
	3	6.54	27.1	2.82	14.6	2.17	17.2	2.31	2.74	0.424	5.61	0.522
	4	6.47	22.1	2.12	10.3	1.38	14.7	1.35	1.68	0.701	4.3	0.33
	5	6.49	18.5	1.6	7.09	0.587	14.7	0.773	1.06*	0.0124	3.97	0.911
	6	6.44	14.6	1.07	6.11	0.798	13.2	0.558	1.27*	0.556	3.61	0.524
	7	6.37	14.4	2.11	6.67	1.09	14.1	1.85	1.66	0.792	4.21	0.509
	8[2]	6.30	13.2	0.935	4.69	0.296	12.9	0.888	1.06*	0.00281	3.69	0.398
	Total[3]	6.42	21.8	1.82	13.1	1.4	16.7	3.57	2.52	1.72	4.89	0.607
Humidity cell 10 0.21 %S	1	6.92	67.6	12.9	54	11.3	38.4	9.96	8.43	4.44	8.94	1.41
	2	6.61	50.3	5.73	27.1	5.68	23.5	2.94	3.76	0.778	7.38	0.818
	3	6.39	43.2	5.95	19.4	1.95	22.9	2.93	3.99	0.616	6.83	0.537
	4	6.27	33.6	4.15	12.8	1.87	18.3	2.36	3.78	0.534	5.16	0.514
	5	6.27	27.8	2.11	9.17	0.868	16.9	1.06	2.04	0.916	3.99	0.64
	6	6.19	21.6	1.16	7.94	0.845	14.9	0.907	2.64	0.571	3.92	0.496
	7	6.12	19.8	3.13	7.71	0.939	14.7	1.47	2.42	0.742	4.22	0.366
	8[2]	6.16	20.3	2.66	6.25	0.715	15	1.51	1.39	0.65	4.35	0.377
	Total[3]	6.18	35.5	3.7	18.1	3.73	20.6	2.96	3.56	1.33	5.6	0.342
Humidity cell 11 0.33 %S	1	7.00	50.9	8.86	42	3.75	31.8	10.9	8.66	5.61	11.9	2.15
	2	6.44	52.7	2.66	28.3	4.05	21.7	1.4	3.51	0.92	9.81	0.788
	3	5.99	54.7	7.03	23.1	3.05	23.7	3.59	5.06	1.51	10.1	0.932
	4	5.69	48.4	3.66	16.4	2.19	19.9	2.34	5.85	0.43	7.74	0.828
	5	5.57	46.3	4.98	13.7	1.32	19.4	1.62	5.15	0.497	5.99	0.705
	6	5.45	37	3.09	12.1	1.82	16.1	1.96	4.8	0.805	5.56	1.24
	7	5.28	34.9	5.53	11.9	2	15.7	2.12	5.98	1.57	5.29	0.981
	8[2]	4.88	35	4.22	10.1	0.775	14.4	1.25	8.5	2.47	4.57	0.51
	Total[3]	5.31	45	2.1	19.7	1.15	20.3	3.2	5.94	1.71	7.63	0.504
Humidity cell 12 0.55 %S	1	7.06	71.8	9.31	59.8	5.62	40.4	12.8	9.09	5.6	10.6	1.91
	2	6.54	68.8	8.25	42.9	7.92	28.4	2.53	3.77	0.625	8.18	0.518
	3	6.27	68.5	7.99	29.7	3.72	32.5	5.76	4.76	1.12	7.84	0.695
	4	6.02	63.8	3.56	20.9	2.39	32.2	1.67	5.04	0.367	6.21	0.356
	5	5.87	59.1	7.33	13.9	1.98	32.5	3.44	3.32	0.926	5.01	0.899
	6	5.76	50.3	3.84	11.5	1.1	30.2	2.42	3.62	0.323	5.13	0.83
	7	5.67	47.1	5.15	10.3	1.43	30.9	3.53	4.33	1.34	5.03	0.497
	8[2]	5.47	43.4	3.73	8.22	0.954	28.3	1.75	3.92	1.98	5.13	0.69
	Total[3]	5.70	59.1	2.33	24.7	2.48	31.9	3.71	4.73	1.73	6.64	0.484

[1] 5th percentile value
[2] Year 8 represented by weeks 365-400
[3] Average and standard deviation for entire period of record
* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.
Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 4 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 13 0.55 %S	1	6.92	72	12.2	59.9	7.85	39.9	14.3	10	6.69	10.2	2.32
	2	6.64	61.6	5.9	38.4	5.94	25.7	1.93	3.58	0.627	7.73	0.82
	3	6.34	62.6	10.9	27.5	4.33	30.6	6.52	4.35	1.23	7.26	0.911
	4	6.04	57.3	4.83	18.7	2.72	29.1	2.93	4.47	0.419	5.67	0.464
	5	5.90	57.1	5.69	13.7	2.09	31.8	3.24	2.83	1.23	4.78	0.885
	6	5.85	44.7	3.7	10.4	1	27.7	3.46	3.18	0.298	4.48	0.574
	7	5.62	40.9	6.22	9.31	1.57	26.6	3.71	3.05	1.29	4.63	0.39
	8[2]	5.66	40.2	3.6	7.4	0.582	26.4	1.52	3.49	1.76	4.38	0.254
	Total[3]	5.75	54.5	3.2	23.2	2.57	29.7	4.15	4.37	2.08	6.14	0.648
Humidity cell 14 0.72 %S	1	6.69	103	12.7	53.2	7.64	67.5	13.7	8.41	6.3	12.9	4.06
	2	6.11	113	6.33	43.5	4.71	59	3.43	4.58	0.72	10.5	0.57
	3	5.90	103	11.2	29.4	4.14	57.5	8.43	6.79	1.29	9.39	0.91
	4	5.67	89.9	7.36	21.2	2.98	50.1	5.23	6.47	0.519	6.91	0.675
	5	5.61	87.7	7.4	17	1.9	48.8	5.26	5.91	1.05	5.53	0.824
	6	5.55	70.7	3.82	14.4	1.43	41.3	3.14	6.1	0.883	4.93	0.65
	7	5.37	66.9	9.24	12.9	2.02	40	5.53	6.25	1.45	5.13	0.518
	8[2]	5.23	66.9	5.08	12	0.494	39.6	2.47	8.11	2.37	4.7	0.286
	Total[3]	5.43	87.7	3.01	25.4	2.28	50.5	3.67	6.58	1.9	7.49	1.22
Humidity cell 15 1.03 %S	1	4.95	175	24.1	89.2	38.9	43.4	7.37	14.3	3.33	16	3.81
	2	4.50	136	16.2	40.7	3.67	39.6	5.51	20.5	2.18	9.13	1.16
	3	4.40	165	20.6	42	5.29	58.4	10	28.2	4.1	7.18	0.764
	4	4.16	180	15.1	39	2.96	58	7.38	29.2	3.99	5.71	0.864
	5	4.17	185	15.4	33.5	3.64	50.1	9.41	24	4.19	4.01	0.903
	6	4.18	157	8.63	29.6	2.39	48.9	7.72	19.7	3.68	5.01	2.11
	7	4.09	161	31.1	30.4	5.33	49.9	7.48	22.2	4.38	4.87	0.704
	8[2]	4.07	148	12.2	27.4	1.56	45	3.14	19.8	2.82	4.74	0.266
	Total[3]	4.17	164	7.12	41.5	12.6	49.2	2.15	22.2	0.76	7.08	1.14
Humidity cell 16 1.36 %S	1	4.91	242	24.1	121	50.8	67.1	13.6	17.2	4.52	21.8	4.8
	2	4.61	189	10.4	51.2	6.04	50	4.7	23.2	1.28	11.4	1.43
	3	4.46	188	15.6	46.9	4.51	52.1	6.69	27.1	3	8.28	0.869
	4	4.34	174	6.3	38.4	1.5	44.6	4.25	24	1.87	5.71	0.675
	5	4.28	177	10.1	35.1	2.92	37.8	4.99	22	1.99	4.08	0.692
	6	4.31	154	6.71	29.2	5.05	36.4	5.94	20.3	3.8	4.46	1.66
	7	4.19	173	41.8	34.5	7.95	40.2	8.24	25.3	5.24	5.21	0.885
	8[2]	4.13	144	12.5	28.9	2.05	34.6	2.96	20.5	3.16	5.07	1.4
	Total[3]	4.28	180	11.9	48.2	16.6	45.3	3.32	22.4	1.37	8.25	1.36

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 9. Page 5 of 5. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 17(D) 0.23 %S	1	7.19	32	12.6	43.3	10.4	17.6	7.01	8.45	6.98	4.91	1.18
	2	6.97	13.2	1.64	21.1	4.09	8.7	0.801	1.63	0.62	3.51	0.351
	3	6.84	10.7	1.26	14.6	2.33	8.87	1.76	1.42	0.555	3.39	0.479
	4	6.64	8.5	0.472	9.39	1.21	7.11	1.18	1.04*	0.0147	2.48	0.203
	5	6.60	8.75	0.612	7.57	0.424	8.9	0.452	1.06*	0.0206	2.08	0.404
	6	6.47	7.27	0.703	6.86	0.668	8.28	0.34	1.06*	0.00937	2.53	0.824
	7	6.35	7.76	1.47	7.23	1.12	9.31	1.42	1.05*	0.0102	2.5	0.383
	8[2]	6.30	7.07	0.35	5.06	0.211	8.59	0.557	1.06*	0.0109	2.24	0.48
	Total[3]	6.40	11.9	4.14	14.4	3.4	9.66	2.21	2.1	2.42	2.96	0.314
Humidity cell 18(D) 0.31 %S	1	7.07	62.4	9.32	65.9	9.31	16.7	5.7	12.4	11.4	5.52	1.05
	2	6.44	52.2	4.83	37.7	7.04	15.7	2.23	3.24	0.447	4.75	0.438
	3	6.28	49	6.33	26.2	3.87	22.2	4.02	2.96	0.521	4.54	0.726
	4	6.10	40.1	5.09	16.6	2.59	20.7	2.58	1.94	0.898	3.31	0.359
	5	6.09	36	3.74	12.7	1.37	21.2	1.65	1.05*	0.02	2.57	0.586
	6	6.00	28.3	1.64	10.4	0.953	18.5	1.72	1.38	0.581	2.64	0.344
	7	5.83	30.5	10.7	11.8	3.25	20.9	6.34	1.78	0.958	2.86	0.648
	8[2]	5.53	27.6	3.98	8.58	0.865	18.5	1.87	1.36	0.625	2.56	0.521
	Total[3]	5.85	40.8	3	23.7	3.05	19.3	1.87	3.27	3.83	3.6	0.231
Humidity cell 19(D) 0.61 %S	1	7.00	92.6	12	85	18.4	26	5.98	20.4	22.1	5.52	1.42
	2	6.43	109	7.56	94.9	11.7	23.7	3.49	4.92	0.823	5.74	0.717
	3	6.14	105	11.7	67.7	11.2	32.1	5.15	6.83	1.41	5.72	1.47
	4	5.83	87.7	6.6	45.6	5.33	29.5	3.7	7.37	0.824	3.42	0.46
	5	5.64	78.1	11.1	35.9	4.2	28.2	2.18	6.07	0.861	2.02	0.569
	6	5.36	65.9	6.21	29.2	2.68	24.8	1.82	6.66	1.02	2.23	0.268
	7	5.06	71.9	8.44	31.8	4.61	30.3	3.65	9.97	1.93	1.85	1.02
	8[2]	4.96	68.9	6.32	26.9	1.77	29.2	1.86	8.66	1.46	1.75	0.188
	Total[3]	5.09	84.8	2.48	52.1	5.74	28	1.53	8.86	7.41	3.53	0.494

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 1 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 1 0.07 %S	1	6.48	65.6	26.2	3.92E-02	1.63E-02	7.66E-02	7.21E-02	3.60E-02*	3.61E-04	4.14E-02	2.53E-02
	2	6.42	68.9	21.8	3.35E-02*	3.59E-04	6.09E-01	6.77E-02	3.61E-02*	3.87E-04	1.02E-01	6.00E-02
	3	6.42	32.9	6.61	5.04E-02	3.40E-02	4.78E-01	1.02E-01	3.59E-02*	6.24E-04	1.34E-01	4.35E-02
	4	6.41	22.1	3.96	7.42E-02	5.42E-02	3.58E-01	1.21E-01	3.63E-02*	3.67E-04	1.27E-01	7.32E-02
	5	6.58	14	1.92	3.37E-02*	4.75E-04	2.19E-01	5.37E-02	3.63E-02*	5.12E-04	6.09E-02	4.02E-02
	6	6.65	7.58*	5.26	4.93E-02	2.71E-02	1.25E-01	2.82E-02	3.63E-02*	1.07E-04	8.82E-02	5.01E-02
	7	6.42	6.86*	3.15	3.43E-02*	4.04E-04	6.39E-02	3.71E-02	3.70E-02*	4.36E-04	4.73E-02*	3.14E-02
	8[2]	6.48	7.63*	3.82	6.18E-02	3.12E-02	1.60E-01	1.46E-02	3.74E-02*	3.93E-04	6.66E-02	5.67E-02
	Total[3]	6.42	28.2	9.37	4.70E-02	1.97E-02	2.61E-01	3.65E-02	3.64E-02	1.48E-04	8.34E-02	1.57E-02
Reactor 2 0.11 %S	1	6.38	145	80.2	4.66E-02	2.10E-02	9.91E-01	1.84E+00	7.18E-02	7.54E-02	9.34E-02	1.05E-01
	2	6.23	91.7	31.2	1.07E-01	1.70E-02	2.66E+00	7.82E-01	1.72E-01	3.69E-02	2.44E-01	1.59E-01
	3	6.45	44.6	8.33	1.16E-01	2.10E-02	1.21E+00	3.30E-01	9.39E-02	2.56E-02	1.72E-01	6.80E-02
	4	6.34	29.1	3.53	1.08E-01	1.07E-01	4.73E-01	3.25E-01	4.78E-02	2.02E-02	1.89E-01	5.97E-02
	5	6.53	21.1	2.85	7.18E-02	5.26E-02	3.84E-01	1.24E-01	3.76E-02*	4.16E-04	8.31E-02	2.25E-02
	6	6.49	16.7	3.3	1.55E-01	1.04E-01	3.84E-01	6.41E-02	3.74E-02*	6.79E-05	1.78E-01	1.53E-02
	7	6.36	11.3	5.28	2.74E-01	1.46E-01	4.31E-01	1.81E-01	3.78E-02*	2.56E-04	3.01E-01	2.32E-01
	8[2]	6.43	10.1	4.94	2.41E-01	8.24E-02	2.91E-01	4.47E-02	3.79E-02*	1.78E-04	9.48E-02	8.24E-02
	Total[3]	6.35	46.2	27.1	1.40E-01	4.83E-02	8.54E-01	6.05E-01	6.70E-02	2.66E-02	1.69E-01	7.25E-02
Reactor 3 0.20 %S	1	6.06	245	129	4.54E-01	7.60E-01	7.38E+00	1.22E+01	5.80E-01	9.27E-01	4.48E-01	6.76E-01
	2	5.75	133	31.6	1.21E+01	2.76E+00	1.32E+01	3.09E+00	8.98E-01	2.58E-01	8.51E-01	3.97E-01
	3	5.68	72.6	11.5	1.30E+01	1.73E+00	6.75E+00	1.90E+00	4.15E-01	1.25E-01	4.56E-01	8.63E-02
	4	5.71	50	3.76	8.82E+00	6.98E-01	3.10E+00	5.54E-01	1.86E-01	3.23E-02	1.96E-01	6.49E-02
	5	5.83	34.6	6.05	7.41E+00	1.39E+00	1.54E+00	5.08E-01	9.37E-02	1.81E-02	4.17E-01	8.89E-02
	6	5.86	25.5	3.04	5.63E+00	5.23E-01	8.06E-01	8.88E-02	3.78E-02*	1.83E-04	3.04E-01	3.06E-02
	7	5.79	21	3.92	5.25E+00	7.50E-01	6.79E-01	6.16E-02	3.82E-02*	4.01E-04	1.38E-01	1.51E-01
	8[2]	5.98	16.5	2.47	3.46E+00	1.76E+00	4.13E-01	9.59E-02	3.86E-02*	6.61E-04	1.48E-01	2.27E-01
	Total[3]	5.73	74.8	43.6	7.02E+00	7.64E-01	4.24E+00	4.15E+00	2.86E-01	3.19E-01	3.70E-01	2.20E-01
Reactor 4 0.31 %S	1	6.34	256	189	4.32E-01	8.32E-01	1.43E+01	2.47E+01	7.89E-01	1.24E+00	4.60E-01	7.40E-01
	2	5.84	183	37.8	5.28E+00	6.92E-01	1.86E+01	4.75E+00	1.11E+00	2.72E-01	8.56E-01	2.65E-01
	3	6.09	102	13.1	6.00E+00	9.08E-01	8.96E+00	2.33E+00	5.51E-01	1.41E-01	4.97E-01	1.49E-01
	4	6.07	68.9	11.8	4.45E+00	1.17E+00	3.88E+00	8.63E-01	2.59E-01	4.90E-02	2.79E-01	1.45E-01
	5	6.21	45.8	7.16	1.91E+00	2.28E+00	1.55E+00	1.22E+00	1.09E-01	9.86E-02	1.87E-01	2.65E-02
	6	6.28	30.8	5.74	1.05E+00	1.02E+00	9.24E-01	1.32E-01	1.03E-01	9.22E-03	2.70E-01	4.17E-02
	7	6.23	25.4	5.62	1.51E+00	1.60E+00	6.77E-01	1.08E-01	5.70E-02	2.43E-02	3.89E-01	1.12E-01
	8[2]	6.29	19.8	3.04	7.74E-01	4.61E-01	4.55E-01	8.68E-02	3.92E-02*	1.14E-04	2.40E-01	8.92E-02
	Total[3]	6.07	91.4	63.6	2.67E+00	5.77E-01	6.18E+00	8.39E+00	3.78E-01	4.18E-01	3.97E-01	2.32E-01

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 2 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 5 0.31 %S (dup. of 4)	1	6.30	238	178	3.85E-01	8.03E-01	1.25E+01	2.39E+01	6.50E-01	1.25E+00	3.68E-01	6.31E-01
	2	5.93	163	30.4	3.17E+00	1.40E+00	1.61E+01	3.89E+00	9.65E-01	2.11E-01	7.62E-01	2.24E-01
	3	6.16	96.9	12.4	3.02E+00	9.76E-01	8.70E+00	2.18E+00	5.63E-01	1.37E-01	6.77E-01	1.34E-01
	4	6.09	62.7	13.2	1.82E+00	1.22E+00	3.80E+00	1.08E+00	2.50E-01	6.51E-02	4.26E-01	5.70E-02
	5	6.34	38.7	6.8	6.80E-01	2.77E-01	1.58E+00	6.53E-01	1.17E-01	2.14E-02	3.22E-01	4.33E-02
	6	6.28	24.1	2.93	1.02E+00	3.80E-01	7.84E-01	2.38E-01	7.91E-02	3.70E-02	2.15E-01	8.16E-02
	7	6.34	21.2	7.17	9.56E-01	6.14E-01	6.28E-01	9.54E-02	3.81E-02*	9.63E-04	1.27E-01	1.42E-01
	8[2]	6.38	19.1	2.79	1.73E+00	1.83E+00	4.60E-01	4.85E-02	1.49E-01	2.21E-01	2.13E-01	1.52E-01
	Total[3]	6.10	83	59.6	1.60E+00	5.29E-01	5.57E+00	8.16E+00	3.51E-01	4.16E-01	3.89E-01	1.90E-01
Reactor 6 0.63 %S	1	5.96	322	278	6.50E-01	8.94E-01	3.72E+01	5.15E+01	2.08E+00	2.92E+00	1.10E+00	1.38E+00
	2	5.37	320	54.7	9.91E+00	5.58E+00	5.43E+01	1.99E+01	3.31E+00	1.22E+00	2.21E+00	7.81E-01
	3	5.17	232	35.3	2.38E+01	2.97E+00	2.60E+01	5.61E+00	1.47E+00	3.88E-01	1.09E+00	3.37E-01
	4	5.19	159	21.4	2.31E+01	1.37E+00	1.18E+01	2.46E+00	6.53E-01	1.17E-01	5.72E-01	7.19E-02
	5	5.31	130	19.7	2.25E+01	3.70E+00	7.92E+00	3.31E+00	4.62E-01	1.93E-01	4.50E-01	6.11E-02
	6	5.40	90.2	10.3	1.40E+01	5.08E+00	3.42E+00	4.40E-01	2.12E-01	3.47E-02	4.86E-01	2.12E-01
	7	5.69	74	9.86	1.49E+01	4.54E+00	2.41E+00	4.92E-01	6.27E-02	5.76E-02	6.96E-01	2.67E-01
	8[2]	5.67	71.5	10.7	1.17E+01	1.53E+00	1.79E+00	3.08E-01	1.23E-01	1.86E-02	3.27E-01	5.39E-02
	Total[3]	5.19	175	91.3	1.51E+01	1.80E+00	1.81E+01	1.78E+01	1.05E+00	1.01E+00	8.68E-01	4.64E-01
Reactor 7 0.94 %S	1	4.53	530	263	4.47E+01	4.65E+01	9.48E+01	1.02E+02	4.71E+00	5.09E+00	2.30E+00	2.54E+00
	2	4.42	452	44.9	6.35E+01	9.26E+00	6.99E+01	1.71E+01	2.97E+00	8.53E-01	1.23E+00	4.71E-01
	3	4.50	314	34	5.50E+01	3.55E+00	3.41E+01	7.00E+00	1.33E+00	2.91E-01	5.57E-01	1.46E-01
	4	4.55	239	26.8	5.14E+01	3.33E+00	1.74E+01	4.49E+00	7.02E-01	1.49E-01	4.01E-01	6.49E-02
	5	4.55	193	26.6	5.37E+01	6.88E+00	9.06E+00	3.64E+00	3.94E-01	1.53E-01	3.22E-01	5.28E-02
	6	4.60	154	15.9	4.55E+01	5.36E+00	4.46E+00	1.65E+00	1.81E-01	4.90E-02	2.61E-01	1.81E-03
	7	4.49	140	19.5	4.63E+01	2.80E+00	3.83E+00	7.23E-01	3.76E-02*	4.96E-04	7.48E-01	1.32E-01
	8[2]	4.69	125	5.67	3.38E+01	1.54E+00	1.99E+00	1.24E+00	1.23E-01	1.97E-02	4.00E-01	5.18E-02
	Total[3]	4.47	268	85	4.92E+01	1.50E+01	2.94E+01	3.45E+01	1.31E+00	1.75E+00	7.77E-01	8.64E-01
Reactor 8(D) 0.31 %S	1	6.34	156	78.7	7.44E-02	8.29E-02	9.08E-01	1.97E+00	8.37E-02	1.16E-01	1.34E-01	1.76E-01
	2	5.89	135	36.3	4.42E-02	2.45E-02	5.33E+00	1.28E+00	3.18E-01	9.47E-02	3.27E-01	2.20E-01
	3	6.19	69	24.2	7.91E-02	2.00E-02	2.98E+00	1.44E+00	1.91E-01	9.35E-02	2.11E-01	7.71E-02
	4	6.15	42.7	6.9	7.67E-02	5.04E-02	9.63E-01	5.72E-01	9.89E-02	2.12E-02	1.17E-01	5.59E-02
	5	6.28	24.8	4.56	3.48E-02*	2.97E-05	4.37E-01	1.06E-01	3.76E-02*	3.20E-05	1.66E-01	4.93E-03
	6	6.32	19.8	4.92	4.69E-02	2.20E-02	4.14E-01	6.46E-02	3.69E-02*	1.33E-04	1.71E-01	2.97E-02
	7	6.23	15.1	5.55	7.12E-02	6.28E-02	4.12E-01	2.40E-01	3.73E-02*	3.79E-04	1.03E-01	9.59E-02
	8[2]	6.18	13.7	6.27	7.93E-02	2.99E-02	3.26E-01	1.40E-01	6.28E-02	3.12E-02	5.83E-02	4.28E-02
	Total[3]	6.05	59.5	26.1	6.33E-02	2.68E-02	1.47E+00	7.36E-01	1.08E-01	4.87E-02	1.61E-01	7.43E-02

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 3 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 9 0.13 %S	1	6.92	32.9	6.46	7.52E-03*	9.01E-05	2.58E-02	1.01E-02	8.11E-03*	9.71E-05	7.31E-03*	8.75E-05
	2	6.76	32	3.91	7.58E-03*	9.25E-05	7.85E-02	1.37E-02	8.17E-03*	9.98E-05	1.00E-02*	6.48E-03
	3	6.54	27.1	2.82	8.95E-03*	4.00E-03	1.75E-01	3.60E-02	8.23E-03*	6.46E-05	9.31E-03	3.82E-03
	4	6.47	22.1	2.12	1.84E-02	1.44E-02	2.16E-01	2.15E-02	8.20E-03*	4.87E-05	7.39E-03*	4.39E-05
	5	6.49	18.5	1.6	7.65E-03*	1.67E-04	1.47E-01	5.42E-02	8.25E-03*	1.80E-04	7.44E-03*	1.62E-04
	6	6.44	14.6	1.07	1.73E-02	8.90E-03	2.09E-01	4.69E-02	8.23E-03*	9.80E-06	1.36E-02	5.66E-03
	7	6.37	14.4	2.11	1.26E-02*	9.96E-03	1.85E-01	2.57E-02	8.21E-03*	6.10E-05	7.40E-03*	5.50E-05
	8[2]	6.30	13.2	0.935	9.75E-03*	4.23E-03	1.52E-01	9.61E-03	8.24E-03*	2.19E-05	7.42E-03*	1.97E-05
	Total[3]	6.44	21.8	1.82	1.12E-02	5.35E-03	1.49E-01	1.70E-02	8.21E-03	5.37E-05	8.74E-03	2.81E-03
Humidity cell 10 0.21 %S	1	6.92	67.6	12.9	7.61E-03	8.41E-05	6.77E-02	5.39E-02	8.20E-03*	9.07E-05	7.39E-03*	8.17E-05
	2	6.61	50.3	5.73	1.32E-02	6.20E-03	3.22E-01	4.86E-02	8.29E-03*	7.15E-05	2.11E-02	2.16E-02
	3	6.39	43.2	5.95	5.39E-02	2.39E-02	7.44E-01	2.46E-01	3.37E-02	1.21E-02	1.93E-02	9.65E-03
	4	6.27	33.6	4.15	9.05E-02	6.08E-03	9.76E-01	9.96E-02	4.75E-02	6.10E-03	1.82E-02	7.66E-03
	5	6.27	27.8	2.11	1.10E-01	1.55E-02	8.67E-01	1.59E-01	4.43E-02	5.15E-03	1.57E-02	1.17E-02
	6	6.19	21.6	1.16	1.42E-01	2.76E-02	7.00E-01	2.63E-02	4.03E-02	7.15E-04	2.45E-02	5.94E-03
	7	6.12	19.8	3.13	1.67E-01	3.34E-02	6.58E-01	6.61E-02	3.52E-02	4.14E-03	1.15E-02*	9.41E-03
	8[2]	6.16	20.3	2.66	1.39E-01	9.40E-03	5.57E-01	6.40E-02	3.30E-02	3.20E-03	9.69E-03*	4.46E-03
	Total[3]	6.18	35.5	3.7	9.03E-02	1.19E-02	6.12E-01	7.31E-02	3.13E-02	4.03E-03	1.59E-02	6.31E-03
Humidity cell 11 0.33 %S	1	7.00	50.9	8.86	1.10E-02	5.44E-03	4.61E-02	1.93E-02	8.16E-03*	5.03E-05	7.35E-03*	4.54E-05
	2	6.44	52.7	2.66	2.91E-02	1.21E-02	4.77E-01	2.35E-01	1.47E-02	1.07E-02	7.44E-03*	1.13E-04
	3	5.99	54.7	7.03	2.04E-01	1.38E-01	2.30E+00	1.01E+00	1.10E-01	5.92E-02	5.47E-02	3.35E-02
	4	5.69	48.4	3.66	6.04E-01	1.53E-01	3.38E+00	3.95E-01	2.05E-01	3.58E-02	1.06E-01	1.96E-02
	5	5.57	46.3	4.98	1.14E+00	2.47E-01	3.43E+00	5.60E-01	2.43E-01	2.15E-02	1.35E-01	1.45E-03
	6	5.45	37	3.09	1.69E+00	3.00E-01	3.04E+00	1.90E-01	2.09E-01	8.58E-03	1.37E-01	9.64E-03
	7	5.28	34.9	5.53	1.93E+00	3.29E-01	2.71E+00	4.43E-01	1.78E-01	1.85E-02	1.41E-01	3.46E-02
	8[2]	4.88	35	4.22	2.44E+00	1.14E-01	2.46E+00	1.06E-01	1.54E-01	2.01E-02	1.01E-01	1.01E-02
	Total[3]	5.31	45	2.1	1.01E+00	1.22E-01	2.23E+00	3.16E-01	1.40E-01	1.84E-02	8.61E-02	1.42E-02
Humidity cell 12 0.55 %S	1	7.06	71.8	9.31	1.12E-02	5.74E-03	1.23E-01	4.22E-02	8.09E-03*	6.76E-05	7.29E-03*	6.09E-05
	2	6.54	68.8	8.25	1.04E-01	6.15E-02	8.44E-01	3.29E-01	4.10E-02	1.48E-02	2.17E-02	2.21E-02
	3	6.27	68.5	7.99	4.01E-01	1.29E-01	2.98E+00	1.03E+00	1.46E-01	5.56E-02	8.32E-02	3.52E-02
	4	6.02	63.8	3.56	9.62E-01	2.14E-01	5.10E+00	7.61E-01	2.85E-01	5.15E-02	1.77E-01	2.42E-02
	5	5.87	59.1	7.33	1.55E+00	2.59E-01	5.27E+00	6.79E-01	3.24E-01	2.25E-02	2.18E-01	6.85E-03
	6	5.76	50.3	3.84	1.76E+00	9.26E-02	4.51E+00	3.66E-01	2.55E-01	1.29E-02	1.73E-01	9.90E-03
	7	5.67	47.1	5.15	1.82E+00	2.22E-01	3.98E+00	3.97E-01	2.15E-01	1.75E-02	1.64E-01	2.75E-02
	8[2]	5.47	43.4	3.73	1.97E+00	1.03E-01	3.26E+00	1.73E-01	1.78E-01	1.22E-02	1.23E-01	1.93E-02
	Total[3]	5.70	59.1	2.33	1.07E+00	8.80E-02	3.26E+00	3.27E-01	1.81E-01	1.97E-02	1.21E-01	1.17E-02

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 4 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 13 0.55 %S	1	6.92	72	12.2	1.15E-02	6.16E-03	9.47E-02	2.58E-02	8.28E-03*	2.77E-05	8.78E-03*	3.22E-03
	2	6.64	61.6	5.9	3.88E-02	1.78E-02	6.11E-01	3.24E-01	2.38E-02	1.44E-02	2.28E-02	1.04E-02
	3	6.34	62.6	10.9	2.07E-01	1.04E-01	2.84E+00	1.04E+00	1.31E-01	5.54E-02	6.37E-02	3.02E-02
	4	6.04	57.3	4.83	6.22E-01	1.39E-01	4.72E+00	7.88E-01	2.63E-01	5.43E-02	1.63E-01	3.22E-02
	5	5.90	57.1	5.69	1.37E+00	1.41E-01	5.03E+00	1.16E+00	3.07E-01	5.08E-02	2.08E-01	7.60E-03
	6	5.85	44.7	3.7	1.63E+00	1.49E-01	4.02E+00	2.29E-01	2.38E-01	4.04E-03	1.60E-01	1.61E-02
	7	5.62	40.9	6.22	1.71E+00	3.31E-01	3.42E+00	4.31E-01	1.91E-01	1.55E-02	1.41E-01	4.79E-02
	8[2]	5.66	40.2	3.6	1.83E+00	2.49E-02	2.99E+00	1.21E-01	1.67E-01	9.58E-03	1.02E-01	3.52E-03
	Total[3]	5.75	54.5	3.2	9.27E-01	1.06E-01	2.96E+00	4.29E-01	1.66E-01	2.38E-02	1.09E-01	1.62E-02
Humidity cell 14 0.72 %S	1	6.69	103	12.7	2.20E-02	1.92E-02	3.31E-01	3.32E-01	1.26E-02*	1.09E-02	7.37E-03*	4.59E-05
	2	6.11	113	6.33	3.54E-01	1.13E-01	4.69E+00	1.33E+00	1.90E-01	6.03E-02	6.89E-02	2.36E-02
	3	5.90	103	11.2	7.69E-01	1.26E-01	7.38E+00	6.86E-01	3.34E-01	3.46E-02	1.32E-01	2.80E-02
	4	5.67	89.9	7.36	1.34E+00	2.69E-01	7.56E+00	7.45E-01	3.78E-01	5.27E-02	1.72E-01	3.22E-02
	5	5.61	87.7	7.4	2.01E+00	2.34E-01	8.44E+00	2.23E-01	4.45E-01	2.16E-03	2.13E-01	2.54E-02
	6	5.55	70.7	3.82	2.50E+00	1.23E-01	7.02E+00	6.35E-01	3.76E-01	6.81E-03	1.79E-01	2.31E-02
	7	5.37	66.9	9.24	2.93E+00	5.16E-01	6.48E+00	9.06E-01	3.47E-01	4.11E-02	1.81E-01	5.38E-02
	8[2]	5.23	66.9	5.08	3.57E+00	2.61E-01	6.21E+00	3.33E-01	3.36E-01	1.97E-02	1.58E-01	3.95E-03
	Total[3]	5.43	87.7	3.01	1.69E+00	1.51E-01	6.01E+00	3.63E-01	3.02E-01	2.19E-02	1.39E-01	1.67E-02
Humidity cell 15 1.03 %S	1	4.95	175	24.1	2.29E+00	1.75E+00	2.62E+01	1.79E+01	1.26E+00	8.98E-01	3.89E-01	2.94E-01
	2	4.50	136	16.2	3.97E+00	4.84E-01	2.57E+01	3.74E+00	1.08E+00	1.79E-01	4.09E-01	5.40E-02
	3	4.40	165	20.6	5.47E+00	5.46E-01	1.88E+01	2.42E+00	8.06E-01	8.76E-02	3.23E-01	5.24E-02
	4	4.16	180	15.1	1.03E+01	3.44E+00	1.64E+01	1.52E+00	7.58E-01	6.86E-02	2.94E-01	2.64E-02
	5	4.17	185	15.4	1.33E+01	1.83E-02	2.44E+01	7.72E+00	7.95E-01	4.08E-02	2.82E-01	1.67E-02
	6	4.18	157	8.63	1.67E+01	5.22E-01	1.58E+01	1.33E+00	6.92E-01	7.60E-02	2.06E-01	9.47E-03
	7	4.09	161	31.1	2.42E+01	8.61E+00	1.60E+01	2.52E+00	8.45E-01	1.33E-01	3.57E-01	1.43E-01
	8[2]	4.07	148	12.2	2.03E+01	5.73E-01	1.29E+01	9.33E-01	6.03E-01	5.14E-02	1.98E-01	2.79E-02
	Total[3]	4.17	164	7.12	1.21E+01	2.89E+00	1.95E+01	5.72E+00	8.55E-01	2.89E-01	3.07E-01	9.70E-02
Humidity cell 16 1.36 %S	1	4.91	242	24.1	6.23E+00	5.61E+00	3.07E+01	2.38E+01	1.93E+00	1.84E+00	7.96E-01	7.94E-01
	2	4.61	189	10.4	1.29E+01	1.27E+00	3.92E+01	4.95E+00	1.68E+00	2.55E-01	7.89E-01	1.32E-01
	3	4.46	188	15.6	1.50E+01	6.11E-01	2.93E+01	4.16E+00	1.26E+00	1.80E-01	6.57E-01	2.90E-01
	4	4.34	174	6.3	1.78E+01	2.24E+00	2.28E+01	9.95E-01	9.57E-01	4.80E-02	4.18E-01	2.12E-02
	5	4.28	177	10.1	2.19E+01	2.85E-01	2.17E+01	2.89E+00	8.20E-01	1.61E-01	3.77E-01	4.47E-02
	6	4.31	154	6.71	2.79E+01	1.11E+00	1.73E+01	2.02E+00	6.52E-01	8.07E-02	2.89E-01	2.06E-02
	7	4.19	173	41.8	4.16E+01	1.70E+01	1.66E+01	4.31E+00	7.23E-01	1.80E-01	4.08E-01	1.66E-01
	8[2]	4.13	144	12.5	2.65E+01	3.29E+00	1.25E+01	1.31E+00	5.43E-01	6.08E-02	2.35E-01	2.57E-02
	Total[3]	4.28	180	11.7	2.12E+01	5.57E+00	2.38E+01	7.51E+00	1.07E+00	6.05E-01	4.96E-01	2.63E-01

[1] 5th percentile value

[2] Year 8 represented by weeks 365-400

[3] Average and standard deviation for entire period of record

* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 10. Page 5 of 5. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in µmol*(kg rock*week)$^{-1}$. Period of record is 6 – 400 weeks for reactors and humidity cells. During the first year, 8-11 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 3-5 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 17(D) 0.23 %S	1	7.19	32	12.6	7.51E-03*	1.00E-04	8.13E-03*	1.08E-04	8.10E-03*	1.08E-04	7.30E-03*	9.73E-05
	2	6.97	13.2	1.64	7.58E-03*	6.20E-05	8.21E-03*	6.71E-05	8.17E-03*	6.68E-05	7.37E-03*	6.02E-05
	3	6.84	10.7	1.26	9.49E-03	3.68E-03	1.39E-02	1.28E-02	8.21E-03*	6.88E-05	1.37E-02	9.51E-03
	4	6.64	8.5	0.472	1.23E-02	5.69E-03	2.53E-02	1.25E-02	8.15E-03*	1.44E-04	7.35E-03*	1.29E-04
	5	6.60	8.75	0.612	2.97E-02	3.10E-02	8.33E-03	1.45E-04	8.30E-03*	1.44E-04	7.48E-03*	1.30E-04
	6	6.47	7.27	0.703	1.67E-02	7.89E-03	3.09E-02	1.37E-02	8.20E-03*	9.45E-05	1.09E-02*	6.10E-03
	7	6.35	7.76	1.47	1.12E-02*	8.69E-03	1.39E-02	8.90E-03	8.20E-03*	8.70E-05	7.39E-03*	8.65E-05
	8[2]	6.30	7.07	0.35	7.68E-03*	4.81E-05	1.77E-02	8.26E-03	8.28E-03*	5.18E-05	7.46E-03*	4.67E-05
	Total[3]	6.40	11.9	4.14	1.28E-02	1.03E-02	1.58E-02	6.05E-03	8.20E-03	3.45E-05	8.62E-03	3.69E-03
Humidity cell 18(D) 0.31 %S	1	7.07	62.4	9.32	9.64E-03*	5.05E-03	1.61E-02	9.31E-03	8.20E-03*	1.08E-04	1.06E-02	7.86E-03
	2	6.44	52.2	4.83	7.57E-03*	3.68E-05	2.64E-01	1.53E-01	1.03E-02*	5.16E-03	1.49E-02	1.22E-02
	3	6.28	49	6.33	1.51E-02	5.78E-03	1.02E+00	2.36E-01	4.23E-02	9.03E-03	1.08E-02	5.66E-03
	4	6.10	40.1	5.09	1.51E-02	1.30E-02	1.26E+00	8.75E-02	5.81E-02	3.13E-03	1.19E-02	7.86E-03
	5	6.09	36	3.74	2.41E-02	1.04E-02	1.13E+00	2.41E-01	5.85E-02	6.74E-03	7.43E-03*	3.24E-05
	6	6.00	28.3	1.64	4.10E-02	2.95E-02	9.22E-01	7.65E-02	5.63E-02	3.89E-03	1.37E-02	6.08E-03
	7	5.83	30.5	10.7	1.20E-01	5.75E-02	1.13E+00	4.47E-01	7.64E-02	3.25E-02	1.25E-02*	1.24E-02
	8[2]	5.53	27.6	3.98	7.32E-02	1.14E-03	7.89E-01	8.60E-02	5.43E-02	4.91E-03	7.39E-03*	2.34E-05
	Total[3]	5.85	40.8	3	3.82E-02	1.94E-02	8.17E-01	1.39E-01	4.55E-02	1.02E-02	1.12E-02	4.71E-03
Humidity cell 19(D) 0.61 %S	1	7.00	92.6	12	7.56E-03*	1.14E-04	6.00E-02	2.22E-02	8.15E-03*	1.23E-04	7.35E-03*	1.11E-04
	2	6.43	109	7.56	1.22E-02	7.69E-03	9.45E-01	7.43E-01	4.40E-02	3.66E-02	1.21E-02	8.32E-03
	3	6.14	105	11.7	5.82E-02	2.17E-02	3.75E+00	6.85E-01	2.22E-01	5.27E-02	3.24E-02	2.02E-02
	4	5.83	87.7	6.6	1.64E-01	4.83E-02	4.47E+00	4.73E-01	3.44E-01	5.66E-02	7.58E-02	1.24E-02
	5	5.64	78.1	11.1	3.15E-01	6.00E-02	4.70E+00	9.69E-02	3.85E-01	1.93E-02	1.03E-01	2.75E-02
	6	5.36	65.9	6.21	7.24E-01	3.67E-01	4.60E+00	5.58E-01	4.01E-01	6.62E-02	1.60E-01	4.86E-02
	7	5.06	71.9	8.44	1.05E+00	1.43E-01	4.87E+00	7.04E-01	4.35E-01	5.27E-02	1.94E-01	4.95E-02
	8[2]	4.96	68.9	6.32	1.03E+00	7.66E-02	4.25E+00	4.42E-01	3.94E-01	3.43E-02	1.29E-01	9.64E-03
	Total[3]	5.09	84.8	2.48	4.20E-01	1.21E-01	3.46E+00	2.73E-01	2.79E-01	2.19E-02	8.92E-02	1.86E-02

[1] 5th percentile value
[2] Year 8 represented by weeks 365-400
[3] Average and standard deviation for entire period of record
* Majority of concentrations during this time interval were below detection limit. One half of the detection limit was used to calculate the annual average release rate.
Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 11. Total mass release, sulfur and heavy metal depletion for Duluth Complex waste rock reactors and humidity cells for weeks 0-400.

ID, %S	Total mass release after 400 weeks, mmol/kg rock									Percent remaining after 400 weeks				
	SO_4	Ca	Mg	Na	K	Cu	Ni	Co	Zn	S	Cu	Ni	Co	Zn
75 g reactors														
1, 0.07	12	11	11.8	4.64	4.31	0.0185	0.106	0.0147	0.035	45.2%	99.7%	98.5%	98.9%	97.7%
2, 0.11	19.7	13.3	15.6	5.24	5.36	0.0555	0.404	0.0294	0.0756	42.7%	99.5%	93.9%	97.5%	95.1%
3, 0.20	31.5	15.2	10.7	7.01	6.95	2.82	2.07	0.144	0.173	49.4%	82.1%	71.0%	88.0%	89.5%
4, 0.31	39.1	16.9	25.2	6.42	6.11	1.09	3.24	0.191	0.182	59.5%	96.2%	69.6%	86.1%	89.1%
5, 0.31	35.5	16.3	23.6	6.85	6.51	0.671	2.93	0.179	0.178	63.3%	97.7%	72.5%	87.0%	89.3%
6, 0.63	73.7	18.9	33.8	9.72	8.84	6.20	7.4	0.426	0.347	62.5%	85.4%	49.9%	73.6%	81.4%
7, 0.94	111	25.4	14	14	7.84	19.9	12.5	0.556	0.323	62.2%	76.0%	30.6%	65.9%	82.6%
8(D), 0.31	24.7	14.1	17.5	3.96	3.22	0.0261	0.641	0.0466	0.0712	87.0%	99.5%	87.2%	95.6%	94.0%
1000 g humidity cells														
9, 0.13	9.1	5.43	7	1.28	1.99	0.00426	0.0581	0.00329	0.00341	77.6%	100.0%	99.2%	99.7%	99.8%
10, 0.21	14.7	7.44	8.63	1.74	2.27	0.0345	0.245	0.0124	0.00661	77.5%	99.8%	96.6%	98.9%	99.6%
11, 0.33	18.6	8.09	8.5	2.64	3.14	0.382	0.895	0.0564	0.0344	82.0%	98.5%	88.5%	95.2%	97.9%
12, 0.55	24.5	10.2	13.3	2.22	2.71	0.412	1.3	0.0726	0.049	87.9%	99.1%	87.5%	94.5%	96.9%
13, 0.55	22.5	9.56	12.3	2.02	2.52	0.361	1.19	0.0666	0.0441	88.9%	99.2%	88.6%	95.0%	97.2%
14, 0.72	36.1	10.5	20.8	2.81	3.04	0.65	2.4	0.12	0.0557	83.9%	98.9%	85.7%	93.0%	97.2%
15, 1.03	66.9	17.2	20.4	9.34	2.89	4.65	7.89	0.344	0.123	79.2%	93.6%	57.7%	79.3%	92.6%
16, 1.36	74	20	19.2	9.4	3.36	8.31	9.73	0.436	0.201	82.6%	91.9%	54.3%	74.6%	88.9%
17(D), 0.23	5.01	5.98	3.98	1.44	1.21	0.00516	0.00626	0.0033	0.00341	93.0%	100.0%	99.9%	99.7%	99.7%
18(D), 0.31	16.6	9.65	7.81	2.13	1.45	0.0143	0.324	0.0179	0.00447	91.3%	99.8%	95.2%	98.5%	99.6%
19(D), 0.61	34.4	21.1	11.2	4.42	1.45	0.151	1.35	0.108	0.0337	64.4%	98.7%	84.5%	92.3%	97.6%

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 12. Annual linear regression slopes ((µmol/kg*wk)/%S) and R^2 values for reactors and humidity cells

	Year	SO4		Ca		Mg		Na		K		Cu		Ni		Co		Zn	
		slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2	slope	R^2
Reactors	1	454	0.883	94.4	0.433	170	0.710	29.2	0.576	22.9	0.103	44.1	0.663	106	0.905	5.26	0.915	2.51	0.920
	2	442	0.985	53.7	0.970	178	0.883	46.9	0.856	28.9	0.688	63.0	0.729	85.3	0.938	3.91	0.822	1.67	0.514
	3	338	0.973	39.8	0.870	121	0.809	33.5	0.858	14.7	0.569	60.7	0.858	41.2	0.944	1.70	0.825	0.670	0.372
	4	257	0.969	28.7	0.899	86.8	0.785	25.9	0.796	13.7	0.372	58.2	0.880	20.6	0.945	0.850	0.899	0.391	0.484
	5	215	0.950	18.2	0.762	81.2	0.758	14.5	0.880	0.710	0.008	60.8	0.868	11.6	0.894	0.518	0.790	0.287	0.318
	6	169	0.946	15.5	0.509	46.5	0.575	12.8	0.808	2.07	0.214	49.4	0.831	5.33	0.930	0.213	0.769	0.222	0.298
	7	152	0.939	9.32	0.296	45.3	0.461	23.7	0.890	1.27	0.069	50.7	0.841	4.34	0.942	0.00986	0.076	0.815	0.750
	8	139	0.940	14.0	0.557	44.6	0.515	26.2	0.812	1.13	0.131	37.4	0.857	2.36	0.902	0.111	0.463	0.387	0.808
Humidity Cells	1	163	0.933	59.6	0.790	33.1	0.538	7.03	0.391	11.4	0.721	4.34	0.715	26.5	0.749	1.52	0.760	0.580	0.744
	2	132	0.926	24.6	0.205	31.7	0.618	17.8	0.803	4.60	0.486	8.74	0.704	31.6	0.813	1.34	0.812	0.586	0.783
	3	143	0.941	27.5	0.431	38.2	0.753	22.8	0.808	2.22	0.172	10.5	0.739	23.4	0.899	1.01	0.916	0.482	0.844
	4	145	0.917	26.3	0.628	35.8	0.731	21.9	0.777	1.41	0.108	13.7	0.801	18.7	0.952	0.809	0.976	0.337	0.922
	5	151	0.919	24.9	0.707	29.0	0.648	19.7	0.811	0.530	0.022	17.1	0.816	20.9	0.875	0.749	0.946	0.313	0.857
	6	132	0.924	21.0	0.714	28.7	0.711	17.1	0.858	1.10	0.128	21.7	0.816	15.5	0.945	0.616	0.909	0.229	0.833
	7	144	0.928	23.8	0.696	30.5	0.782	20.7	0.859	1.25	0.151	32.1	0.802	15.1	0.930	0.709	0.875	0.355	0.934
	8	123	0.914	21.5	0.716	26.6	0.713	17.4	0.846	1.41	0.196	22.1	0.829	11.8	0.926	0.528	0.870	0.203	0.888

* Note that Mg release from reactor 7 was omitted due to low values that resulted in poor correlation with sulfur. Mg slopes and R^2 values when including reactor 7 ranged from 0.27-62.2 and 0.0004-0.273, respectively.

Considerably lower cobalt slope and R^2 values for reactors in year 7 due to drainage quality reaching detection limit values

Table 13. Factor analysis of heavy metal release from Babbitt and Dunka Road prospect samples.

%S (B, D)	min pH_B	min pH_D	M_{SO4B}/M_{SO4D}	Cu	Ni	Co	Zn
Fine (0.31, 0.31%)	6.07	6.05	1.58				
M_B/M_D				41.76	5.05	4.10	2.56
C_B/C_D				5.49	2.14	1.29	1.42
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				8.69	3.38	2.04	2.24
Coarse (0.21, 0.23%)	6.18	6.4	2.93				
M_B/M_D				6.69	39.14	3.76	1.94
C_B/C_D				1.31	0.89	0.96	1.30
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				3.85	2.62	2.81	3.81
Coarse (0.33, 0.31%)	5.31	5.85	1.12				
M_B/M_D				26.71	2.76	3.15	7.70
C_B/C_D				3.28	1.14	1.00	1.36
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				3.68	1.28	1.12	1.52
Coarse (0.55, 0.61%)	5.7	5.09	0.68				
M_B/M_D				2.56	0.92	0.64	1.38
C_B/C_D				3.68	1.19	0.94	1.14
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				2.52	0.81	0.64	0.78

B, D indicate Babbitt prospect and Dunka Road prospect, respectively
M_B/M_D Ratio of Babbitt prospect to Dunka Road prospect mass release after 400 weeks
C_B/C_D Ratio of Babbitt prospect to Dunka Road prospect solid-phase chemical content
min pH 5th percentile pH value for 400-week period of record

Note: Highlighted values are those which still have a high ratio of metal release when compared to the ratio of sulfate release multiplied by the ratio of chemical content

Table 14. Factor analysis of major cation release from Babbitt and Dunka Road prospect samples.

%S (B, D)	min pH_B	min pH_D	M_{SO4B}/M_{SO4D}	Ca	Mg	Na	K
Fine (0.31, 0.31%)	6.07	6.05	1.58				
M_B/M_D				1.20	1.44	1.62	1.90
C_B/C_D				0.77	1.27	0.88	1.68
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				1.21	2.02	1.39	2.66
Coarse (0.21, 0.23%)	6.18	6.4	2.93				
M_B/M_D				1.24	2.17	1.21	1.88
C_B/C_D				0.94	0.89	1.10	2.21
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				2.75	2.61	3.24	6.50
Coarse (0.33, 0.31%)	5.31	5.85	1.12				
M_B/M_D				0.84	1.09	1.24	2.17
C_B/C_D				0.85	1.02	0.99	1.71
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				0.95	1.14	1.11	1.91
Coarse (0.55, 0.61%)	5.7	5.09	0.68				
M_B/M_D				0.47	1.14	0.48	1.80
C_B/C_D				0.84	1.16	0.95	1.62
$(M_{SO4B}/M_{SO4D})(C_B/C_D)$				0.57	0.79	0.65	1.10

B, D indicate Babbitt prospect and Dunka Road prospect, respectively
M_B/M_D Ratio of Babbitt prospect to Dunka Road prospect mass release after 400 weeks
C_B/C_D Ratio of Babbitt prospect to Dunka Road prospect solid-phase chemical content
min pH 5th percentile pH value for 400-week period of record

Table 15. Annual average sulfate release rates (µmol/kg*wk) for Partridge River intrusion (PRI) and South Kawishiwi intrusion (SKI) samples

Partridge River Intrusion reactors (75 grams, 0.053<d<0.149 mm)								
ID #	1	2	3	4	5	8 (D)	6	7
%S	0.07	0.11	0.2	0.31	0.31	0.31	0.63	0.94
Yr 1	65.6	145	245	256	238	156	322	530
Yr 2	68.9	91.7	133	183	163	135	320	452
Yr 3	32.9	44.6	72.6	102	96.9	69	232	314
Yr 4	22.1	29.1	50	68.9	62.7	42.7	159	239
Yr 5	14	21.1	34.6	45.8	38.7	24.8	130	193
Yr 6	7.58	16.7	25.5	30.8	24.1	19.8	90.2	154
Yr 7	6.86	11.3	21	25.4	21.2	15.1	74	140
Yr 8	7.63	10.1	16.5	19.8	19.1	13.7	71.5	125

South Kawishiwi Intrusion reactors (75 grams, 0.053<d<0.149 mm)											
ID #	1,2	3,4	5,6	7,8	9,10	11,12	13,14	15,16	17,18	35,36	29,30
%S	0.18	0.22	0.4	0.41	0.51	0.54	0.57	0.58	0.71	1.12	1.16
Yr 1	56.5	64.8	277	39.2	107	219	230	199	161	624	279
Yr 2	117	103	250	159	215	274	194	149	188	468	213
Yr 3	81.7	75.5	147	90.6	119	174	151	166	127	384	165
Yr 4	48.3	48.9	93.5	52	77.7	140	95.9	260	101	553	316
Yr 5	42.6	39.4	86.2	49.2	100	159	117	353	153	873	607
Yr 6	50.1	45.9	124	88.7	372	393	301	349	373	1430	1810
Yr 7	25.3	38.9	95.9	89.2	391	365	340	255	654	412	659
Yr 8	20.8	19.7	73.2	NA[1]	312	216	262	211	469	NA[1]	182

[1] Reactors terminated after year 7

Note: Samples are from Babbitt prospect unless noted with (D), which indicates Dunka Road prospect.

Table 16. Rates of metal release from the Dunka Mine samples, weeks 6-74.

Reactor	%S	Release rate, μmol kg^{-1} wk^{-1}			
		Cu	Ni	Co	Zn
1	0.18	0.254	0.551	0.431	0.565
2	0.18	0.366	0.713	0.315	0.426
3	0.22	0.255	0.513	0.314	0.460
4	0.22	0.294	0.438	0.317	0.465
5	0.40	0.295	4.118	0.717	0.790
6	0.40	0.371	3.699	0.481	1.191
7	0.41	0.377	3.792	0.569	0.732
8	0.41	0.371	4.897	0.520	0.793
9	0.51	0.905	2.980	0.610	1.466
10	0.51	1.041	9.501	1.042	1.373
11	0.54	0.292	2.216	0.591	0.959
12	0.54	0.336	1.173	0.443	0.617
13	0.57	0.904	6.188	0.663	1.301
14	0.57	0.821	5.574	0.603	1.124
15	0.58	1.677	12.225	2.130	2.318
16	0.58	1.553	12.409	2.193	2.623
17	0.71	2.176	10.943	1.551	1.685
18	0.71	2.845	10.002	1.634	1.903
35	1.12	1.511	60.638	9.694	3.939
36	1.12	2.231	53.380	8.419	3.577
29	1.16	3.692	17.226	1.642	1.295
30	1.16	4.328	16.942	1.675	1.258

Figure 1. Map of Duluth Complex mineral prospects.

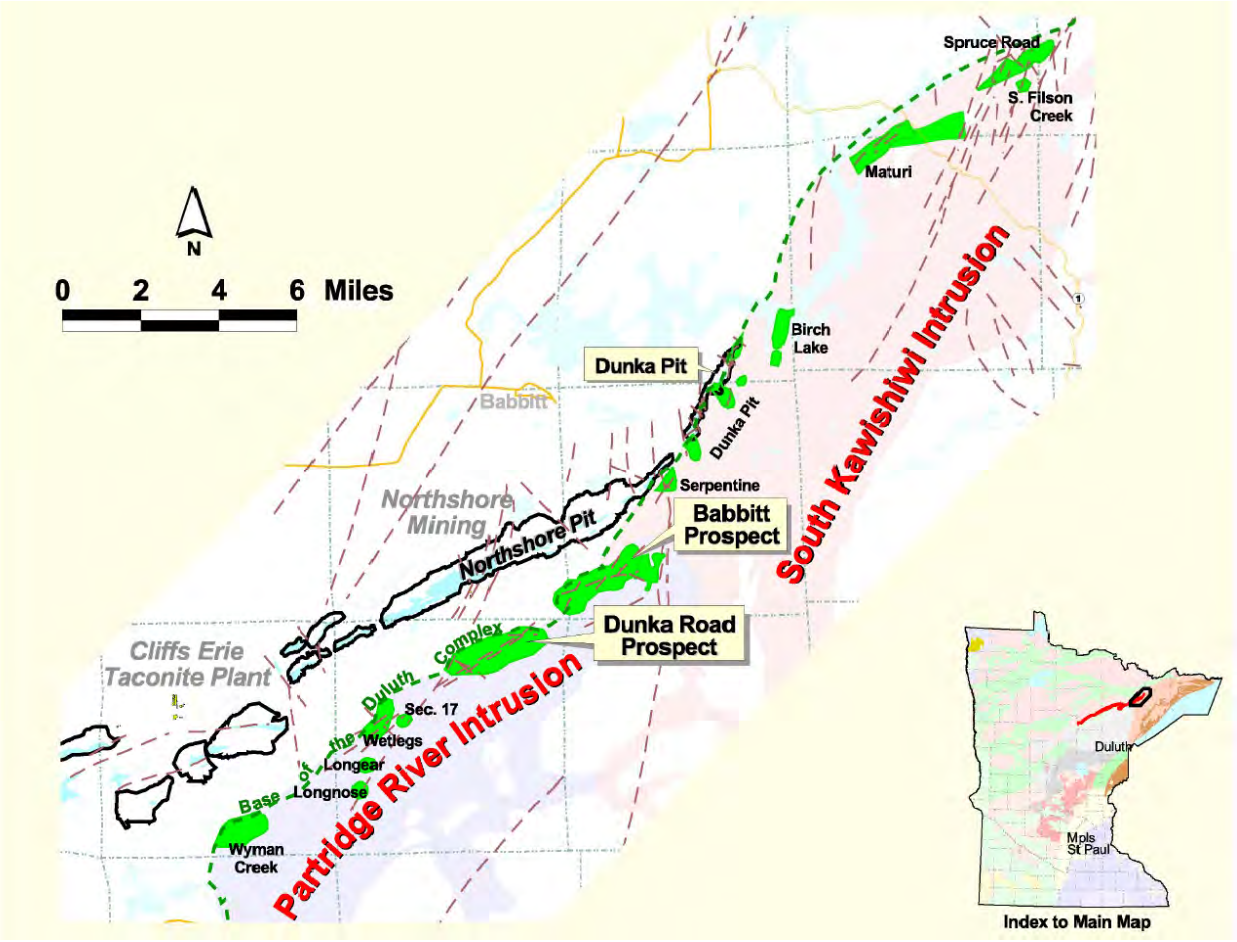


Figure 2. Schematic and photo of small reactor.

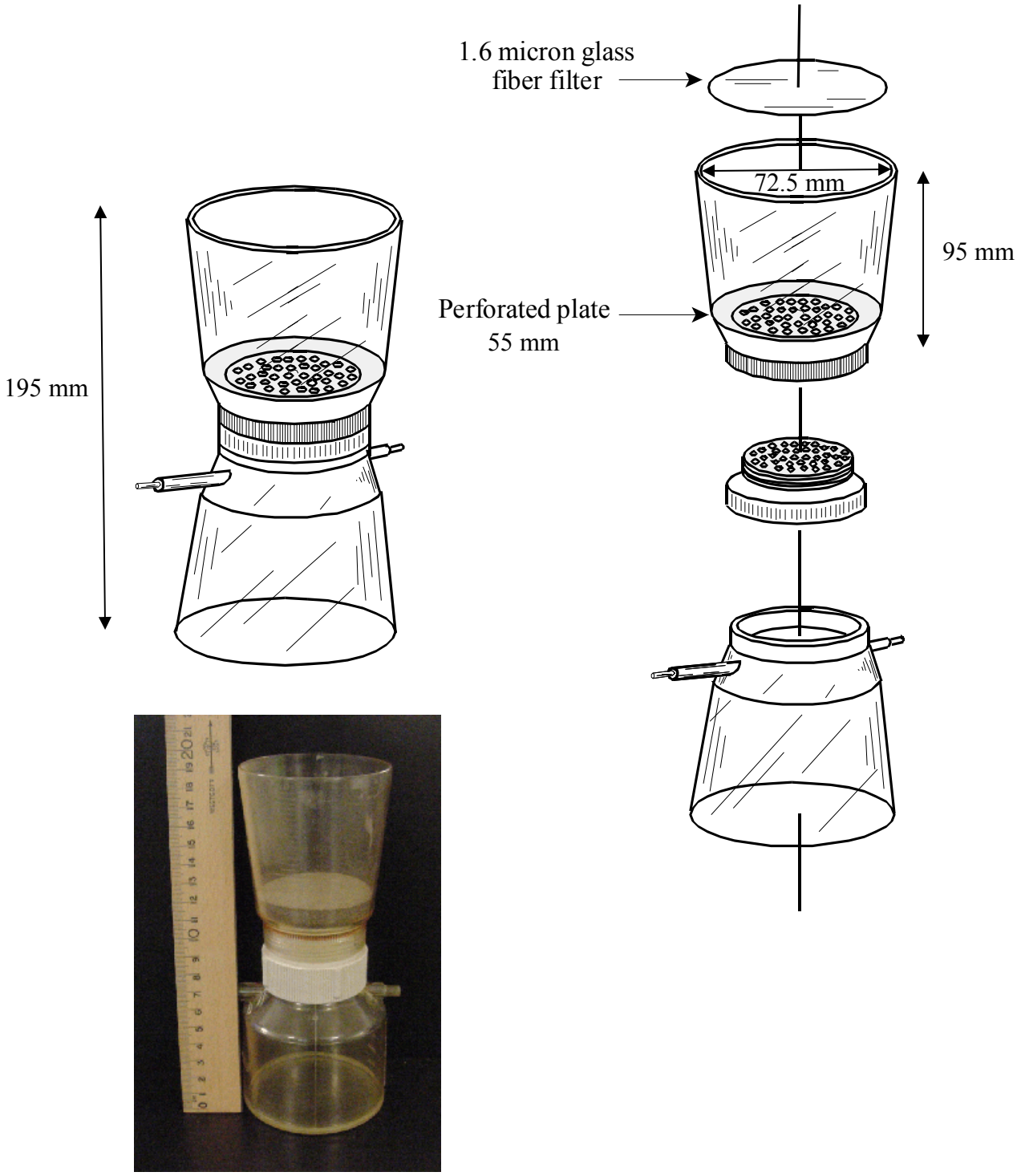


Figure 3. Schematic and photo of humidity cell apparatus. All humidity cell materials are acrylic except the perforated plate (polyvinyl chloride) and the outlet pipe (high density polyethylene).

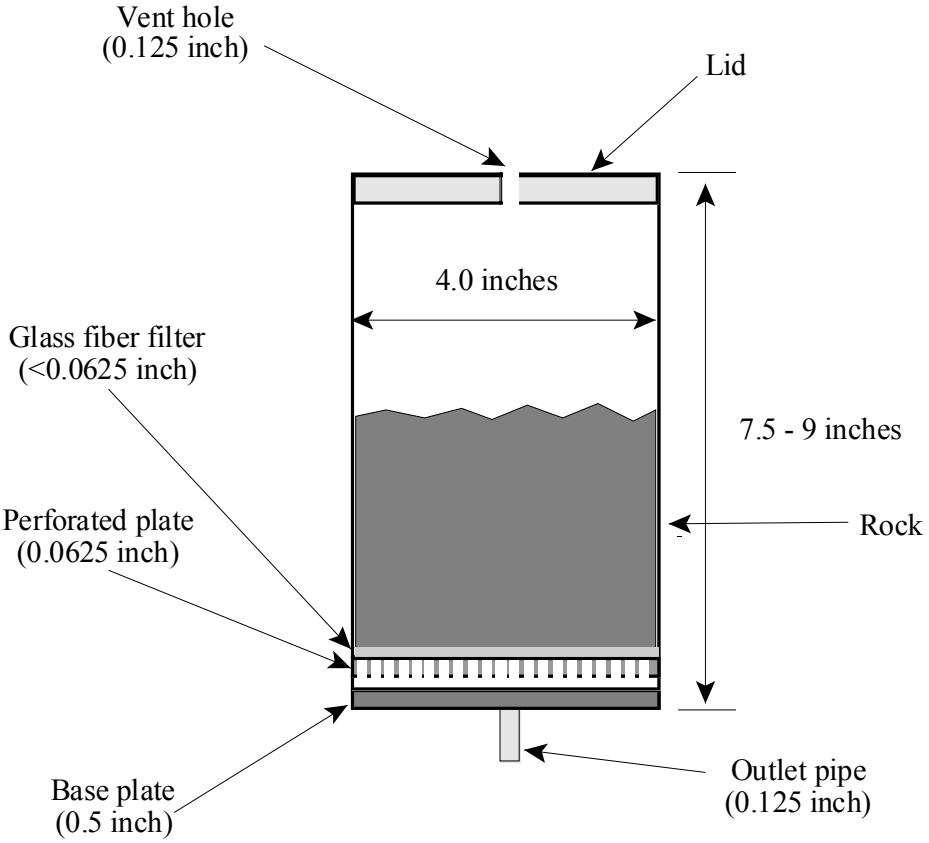


Figure 4. Drainage quality replicated well for duplicate 75-g sample of 0.31 %S Babbitt prospect rock. Weeks 0-5 were excluded to improve resolution.

Red = Reactor 4, Blue = Reactor 5

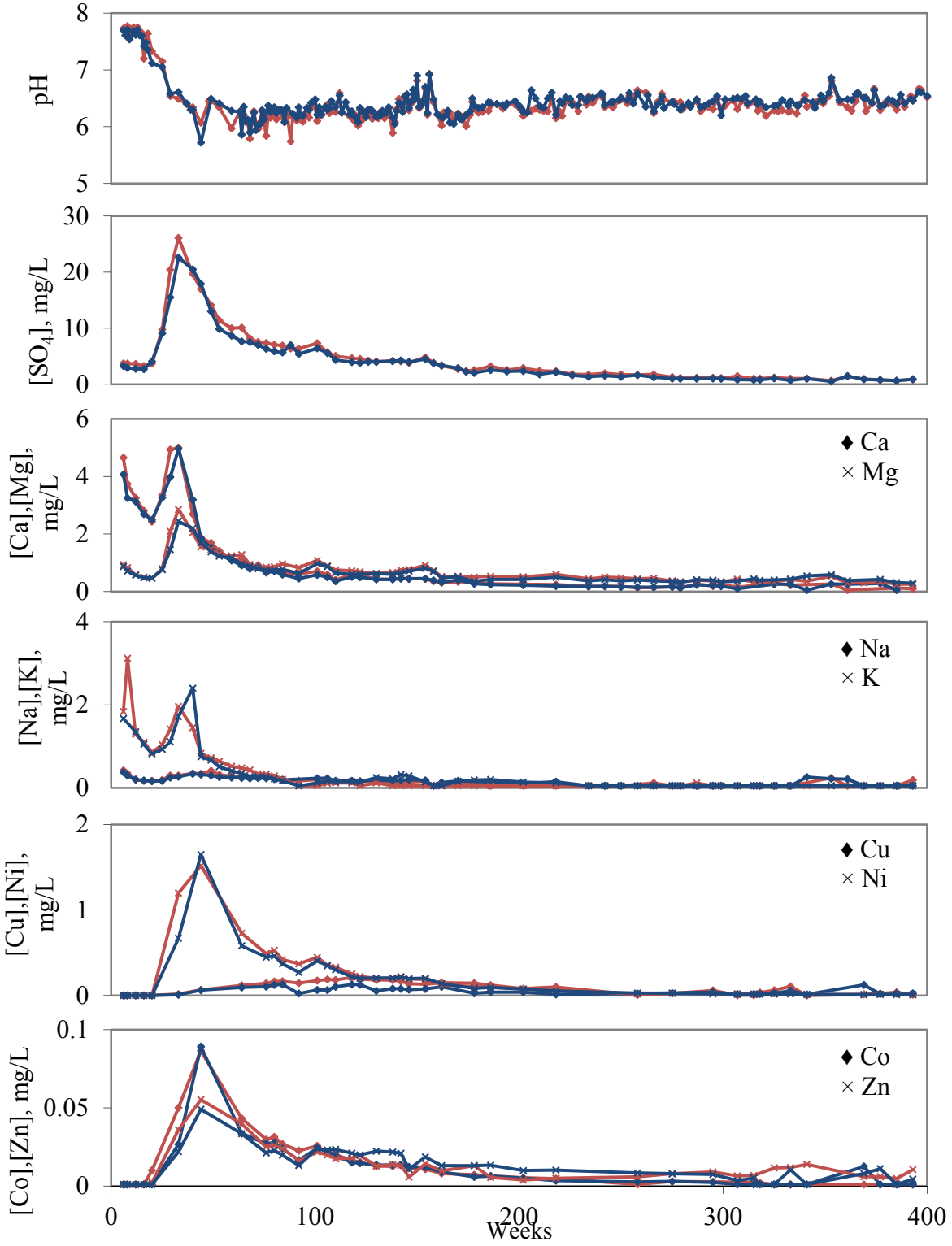


Figure 5. Drainage quality replicated well for duplicate 1000-g sample of 0.55 %S Babbitt prospect rock. Weeks 0-5 were excluded to improve resolution.

Red = Humidity cell 12, Blue = Humidity cell 13

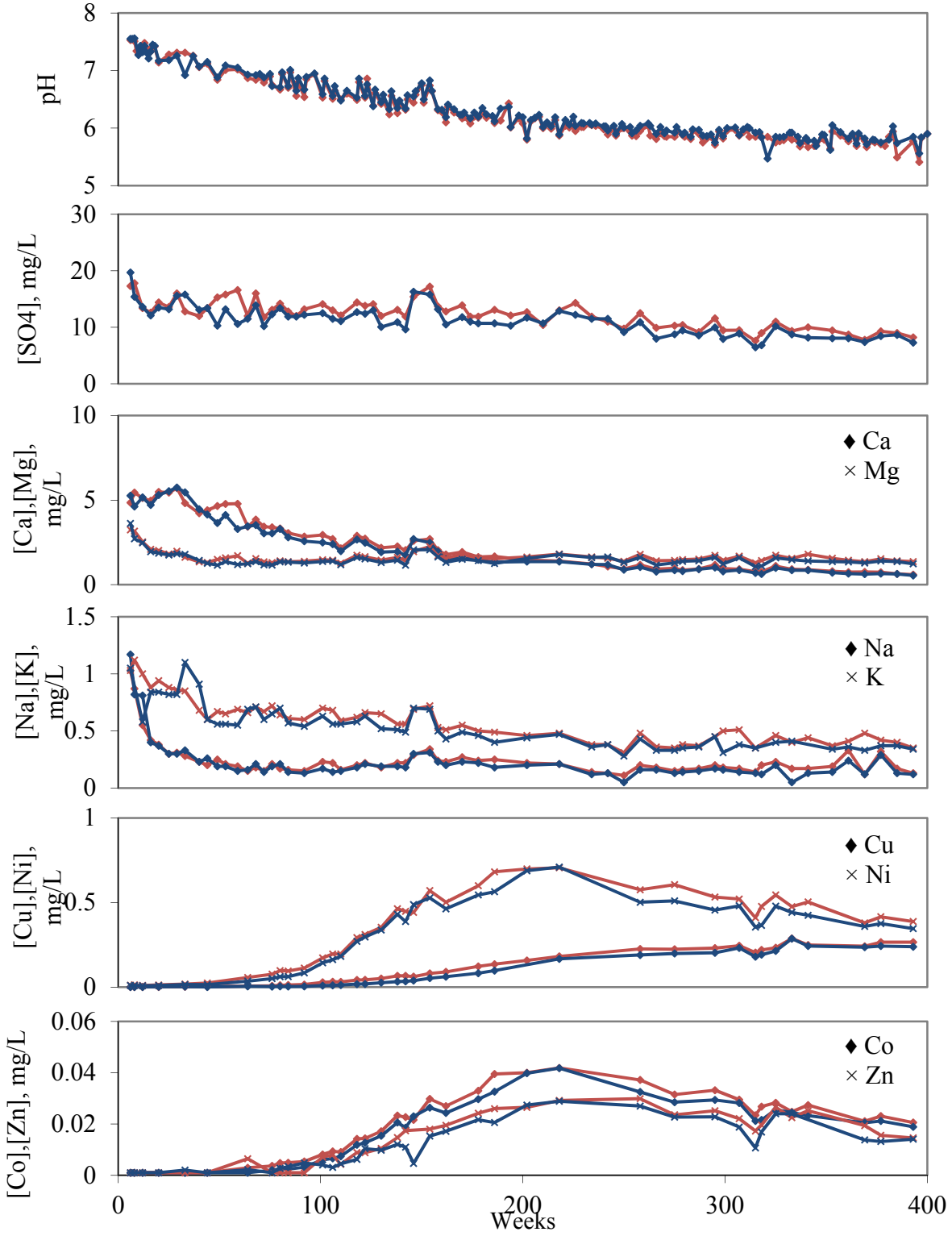
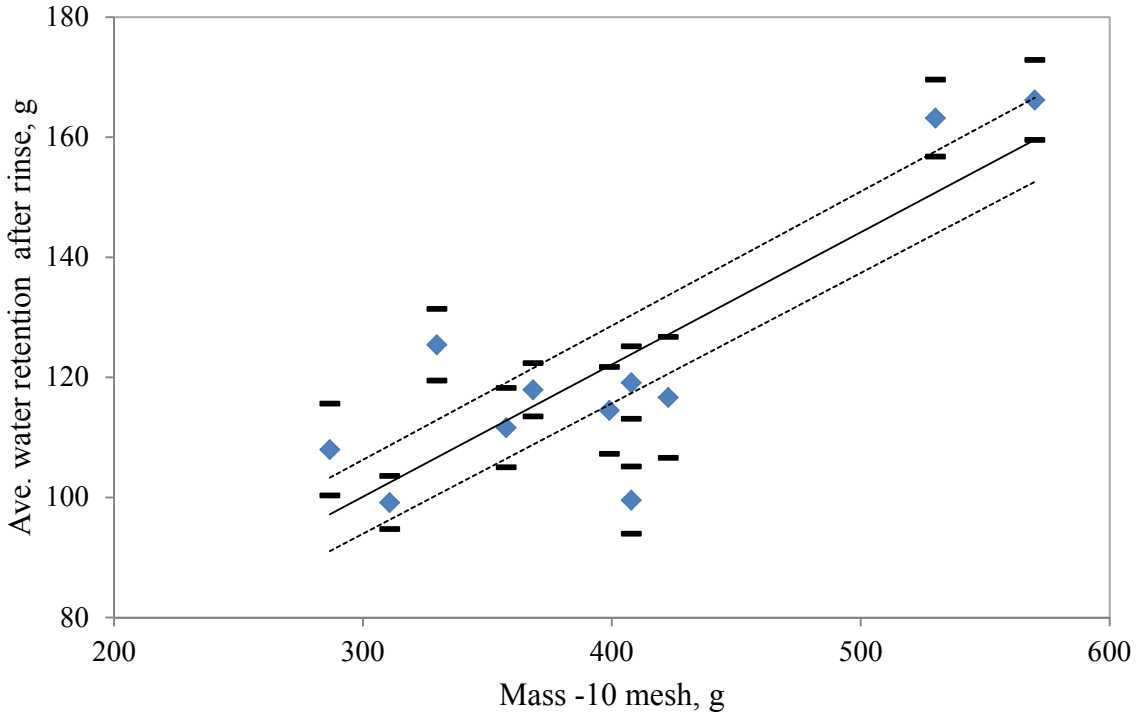
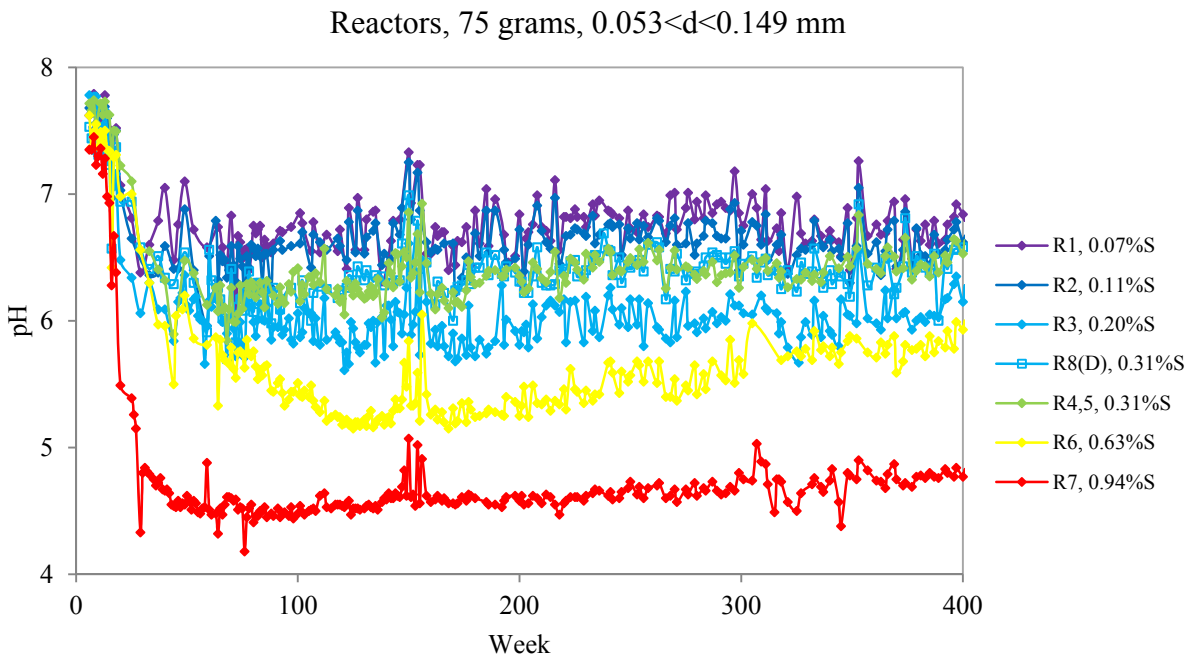


Figure 6. Water retention after rinse vs. mass – 10 mesh rock



	m	b	R^2
Average +SD	0.223	39.2	0.720
Average	0.220	34.0	0.713
Average -SD	0.217	28.8	0.699

Figure 7. pH vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



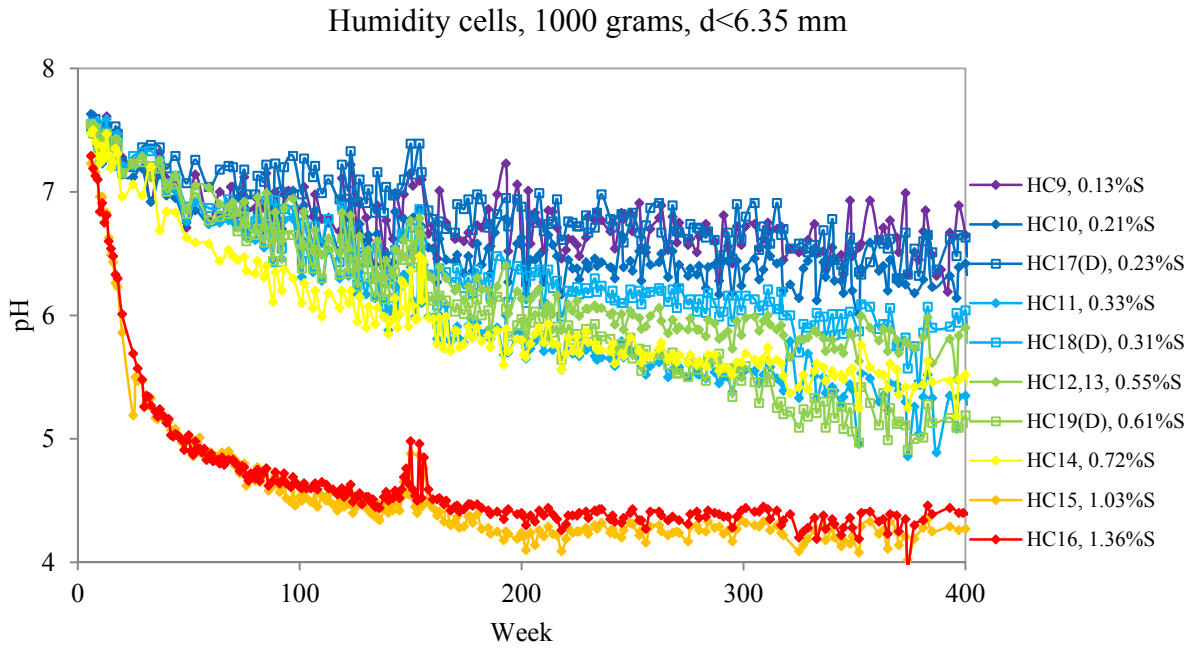
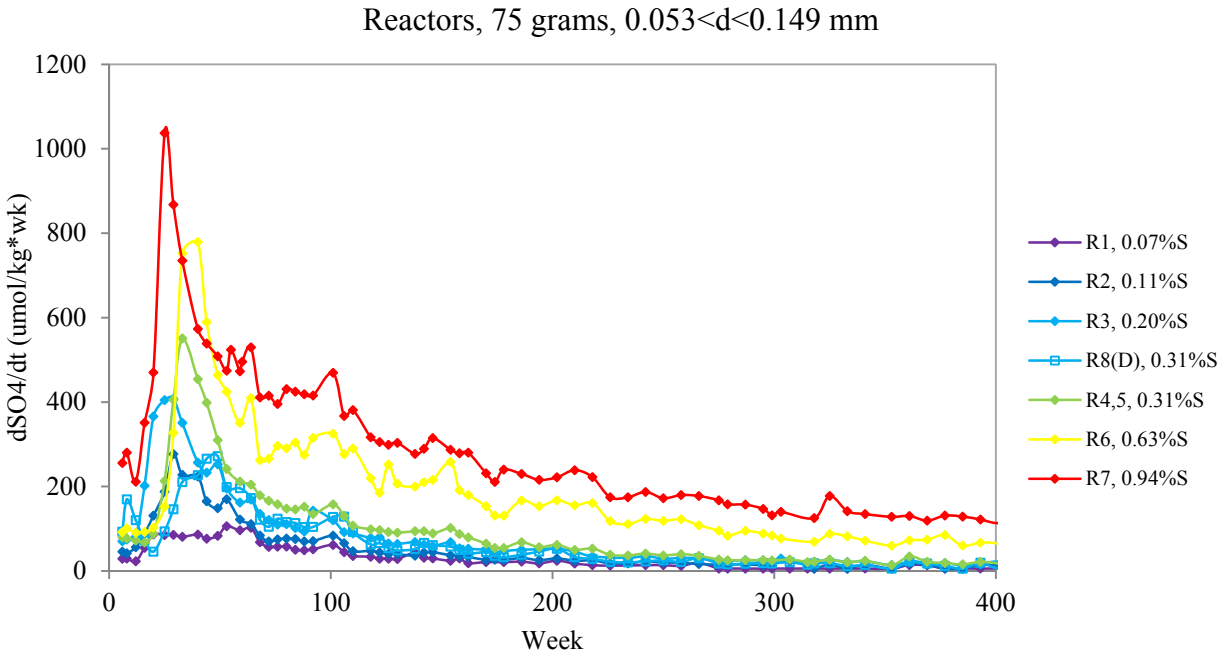


Figure 8. Sulfate release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



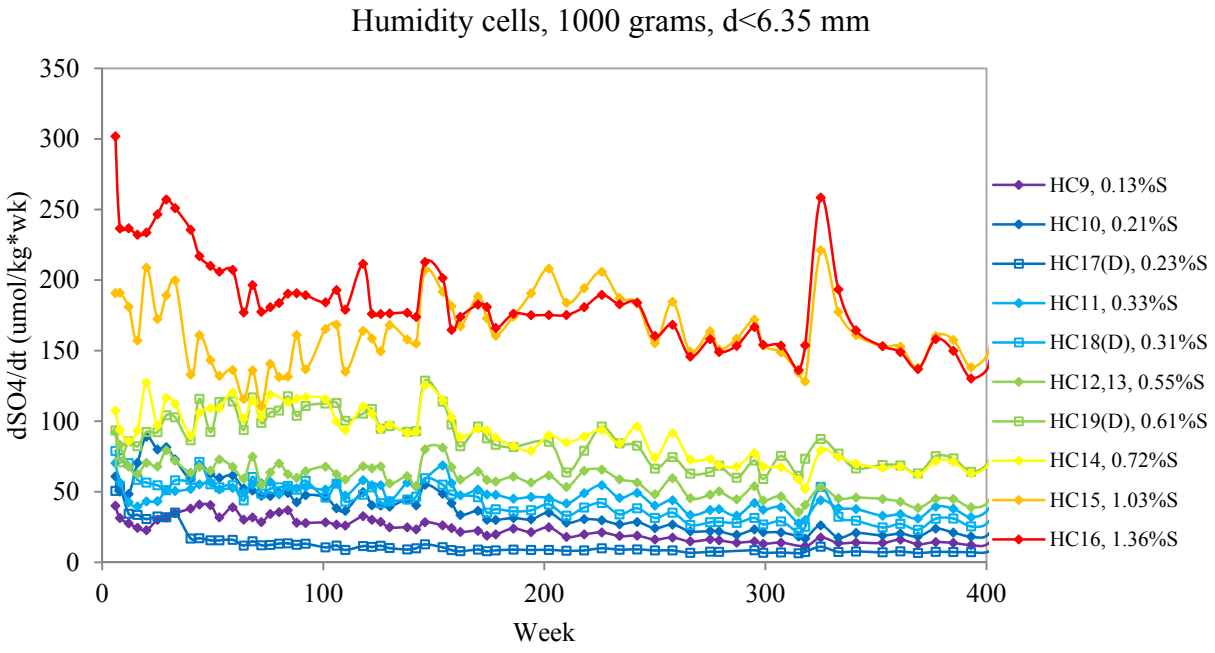
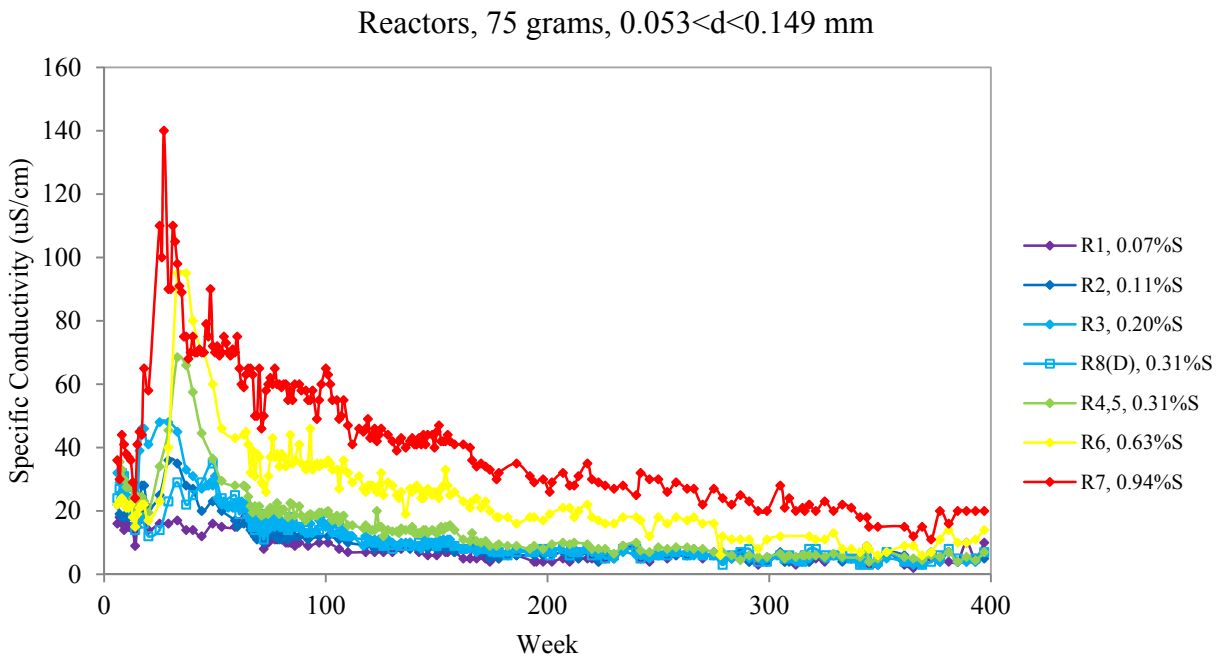


Figure 9. Specific conductivity vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



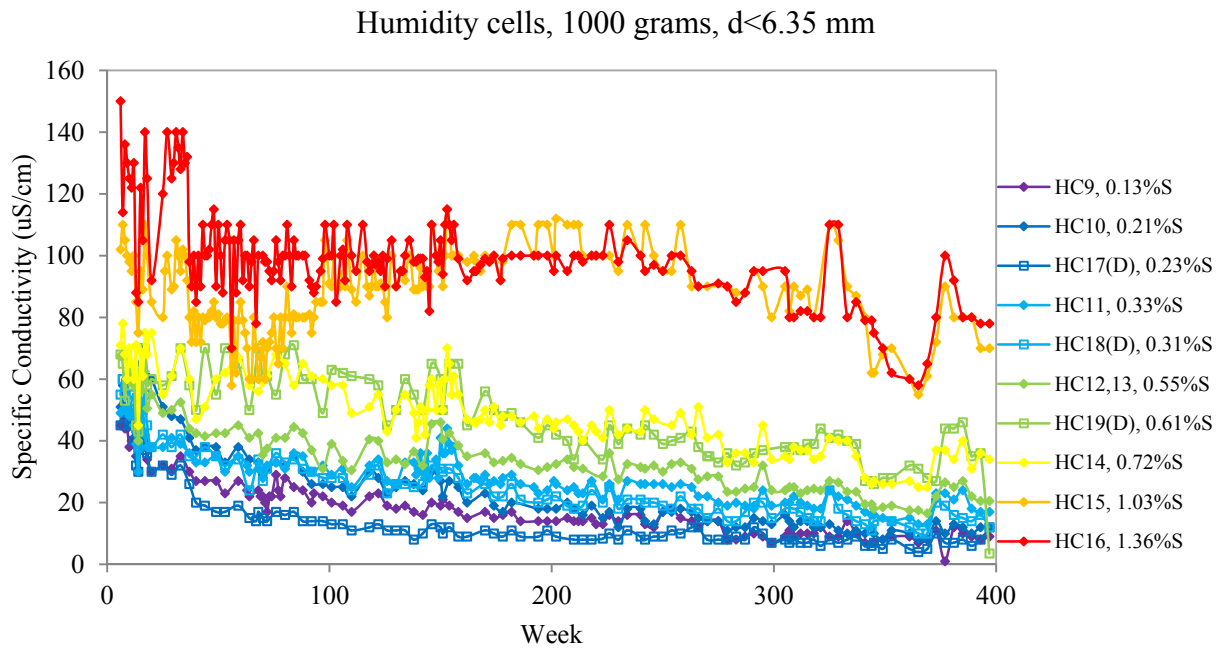
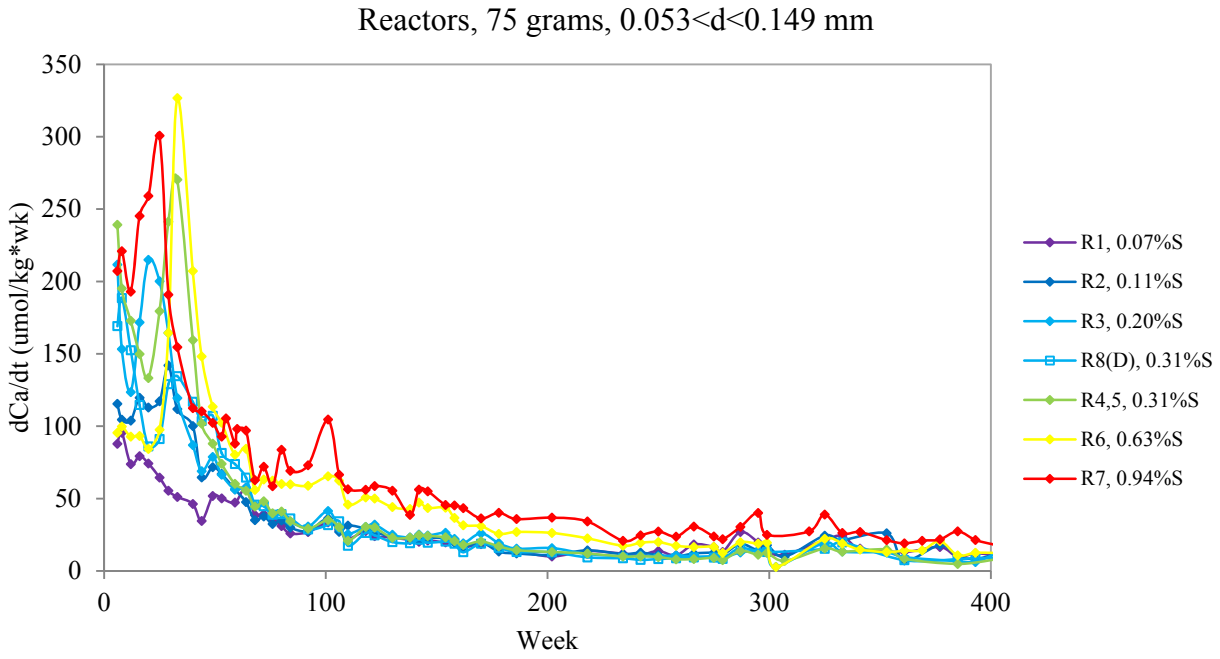


Figure 10. Calcium release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



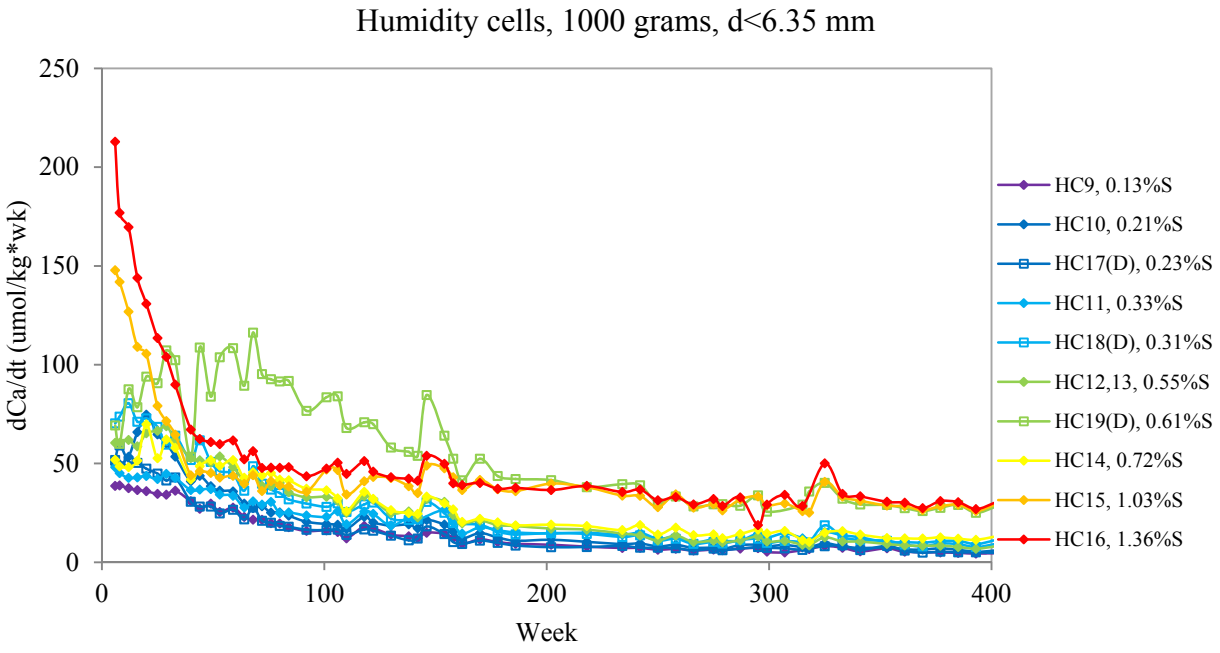
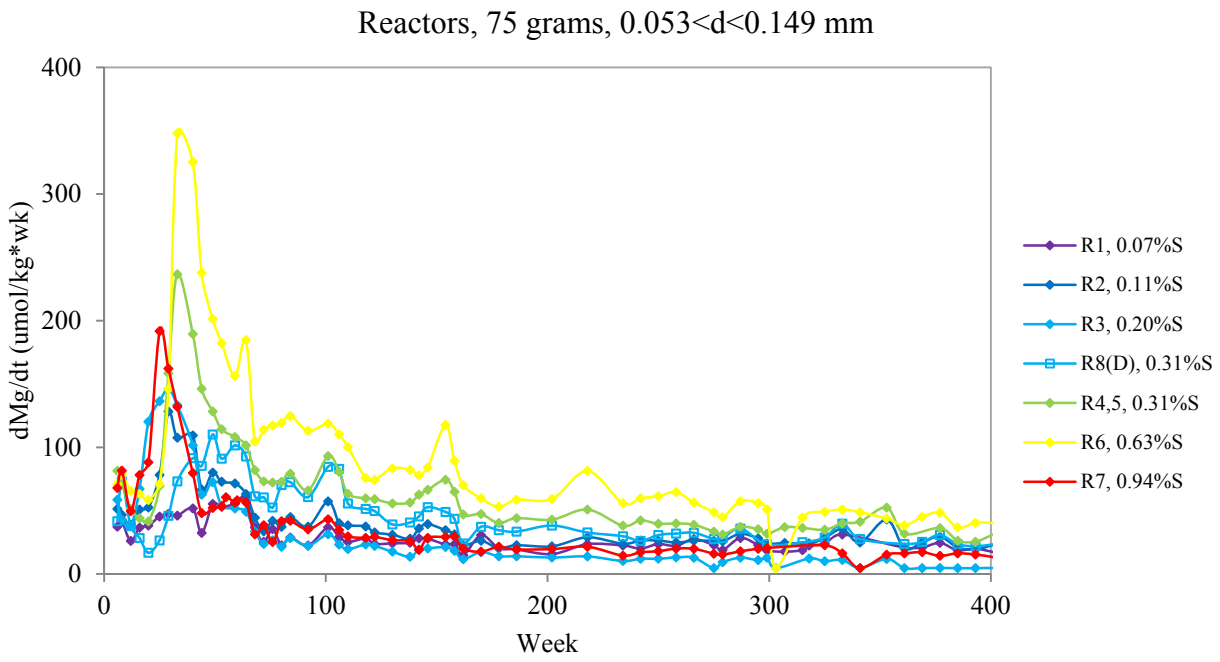


Figure 11. Magnesium release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



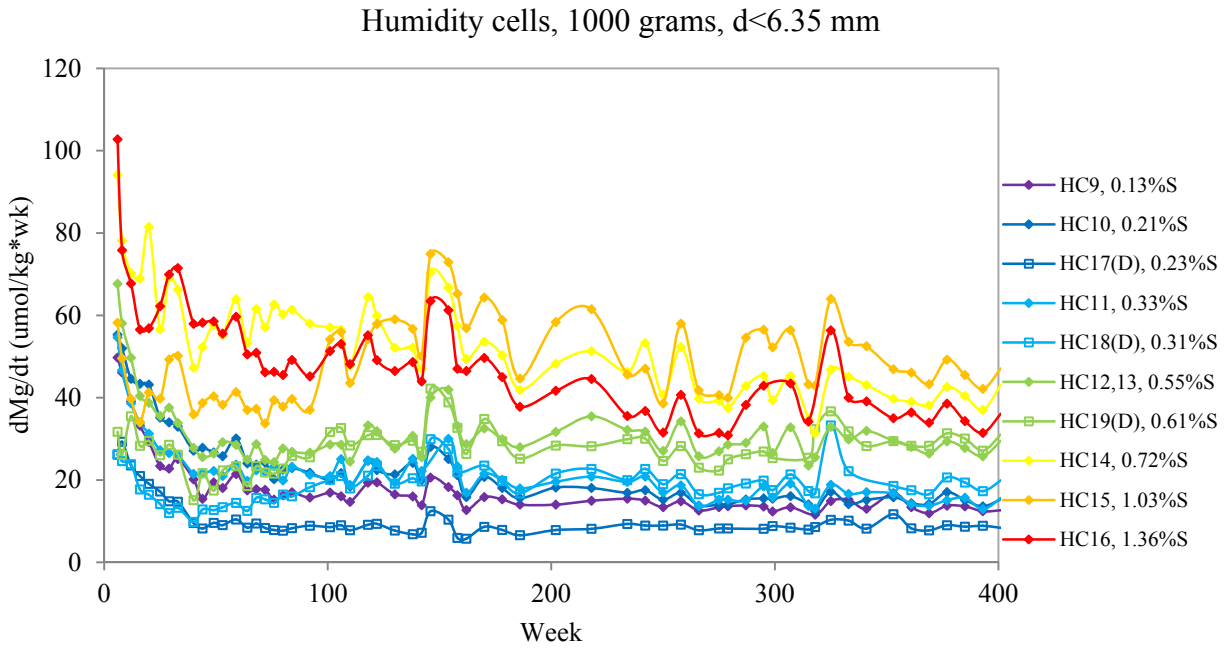
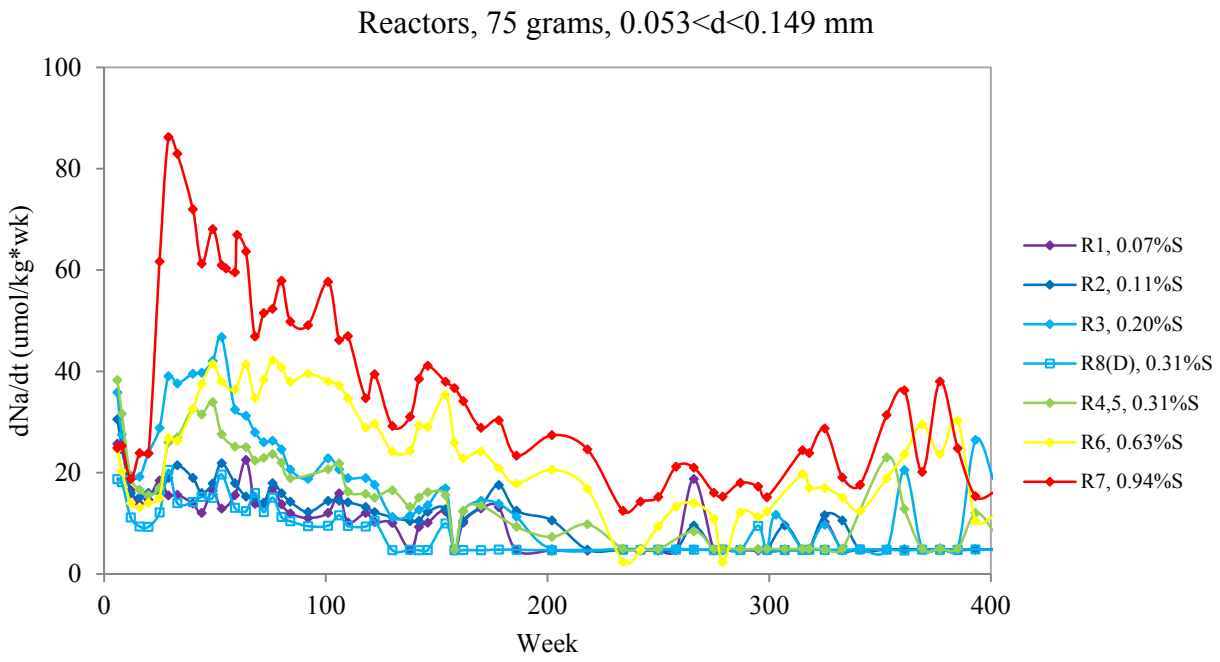


Figure 12. Sodium release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



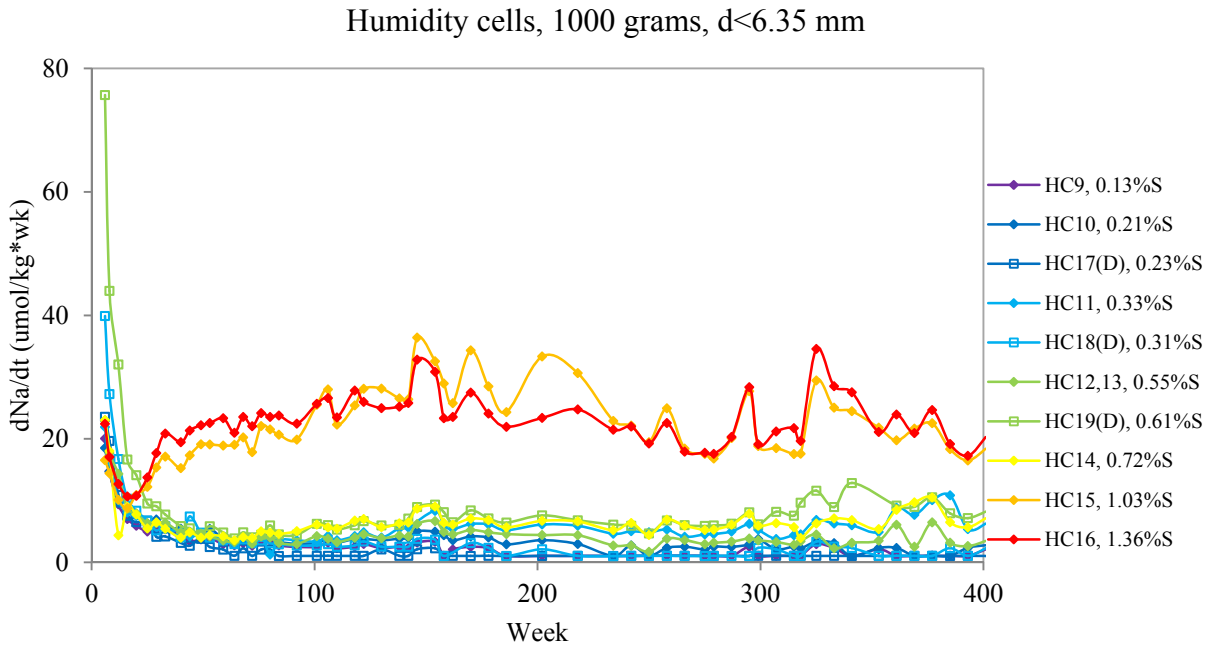
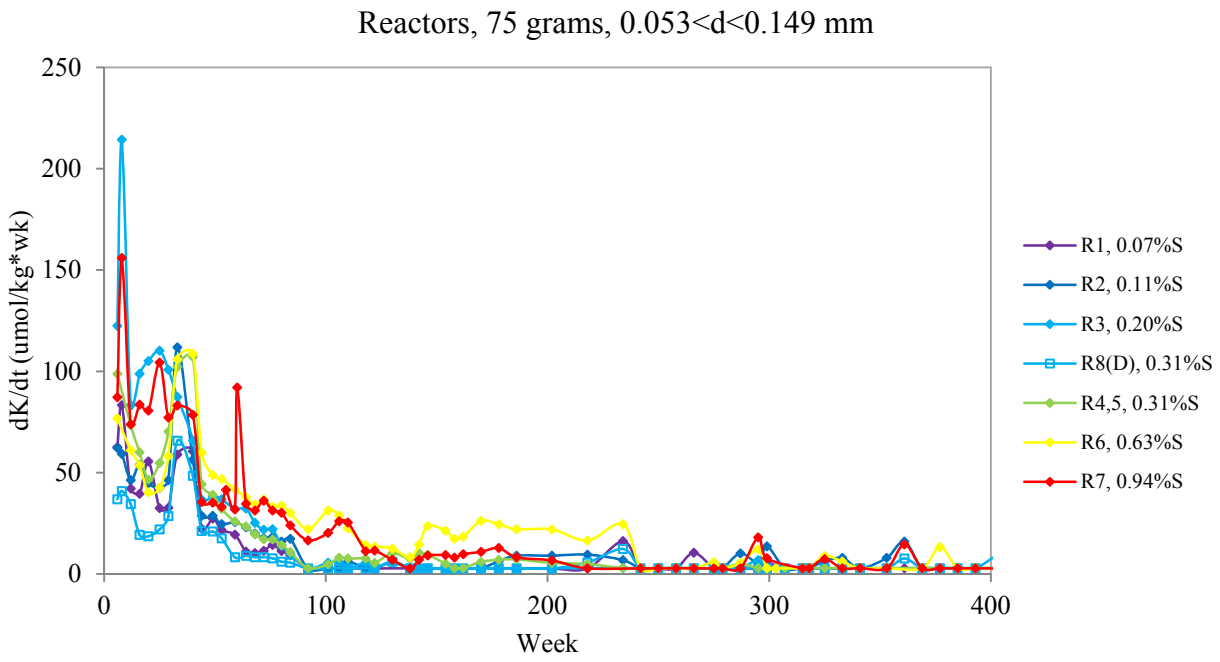


Figure 13. Potassium release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



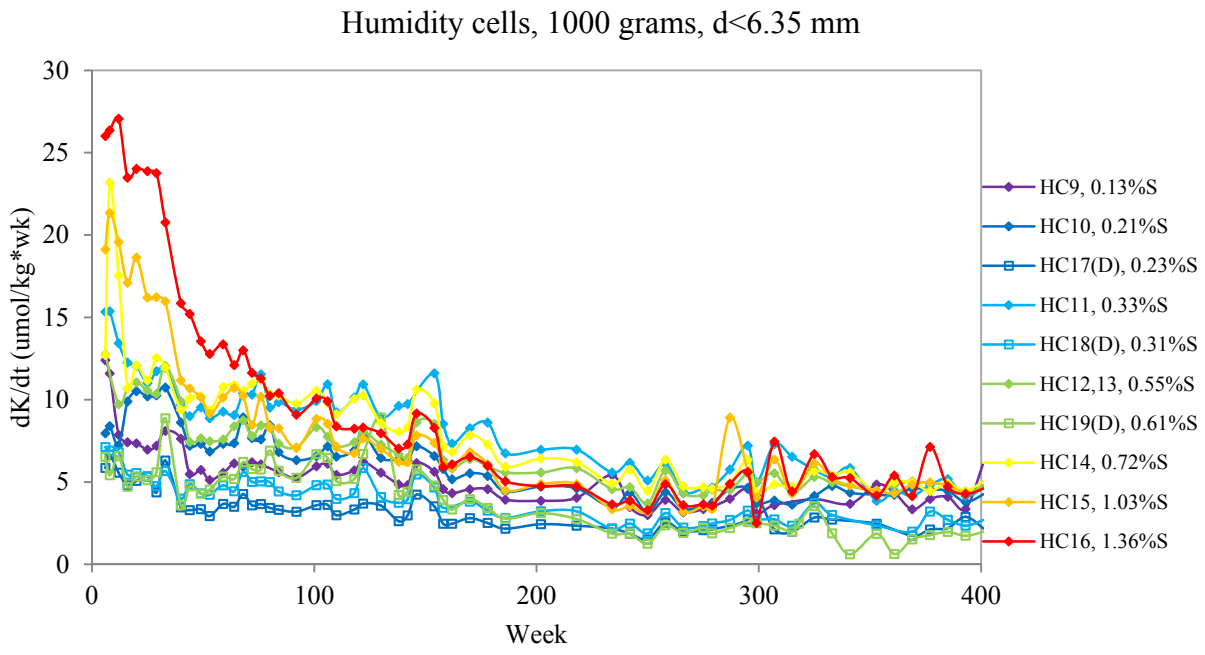
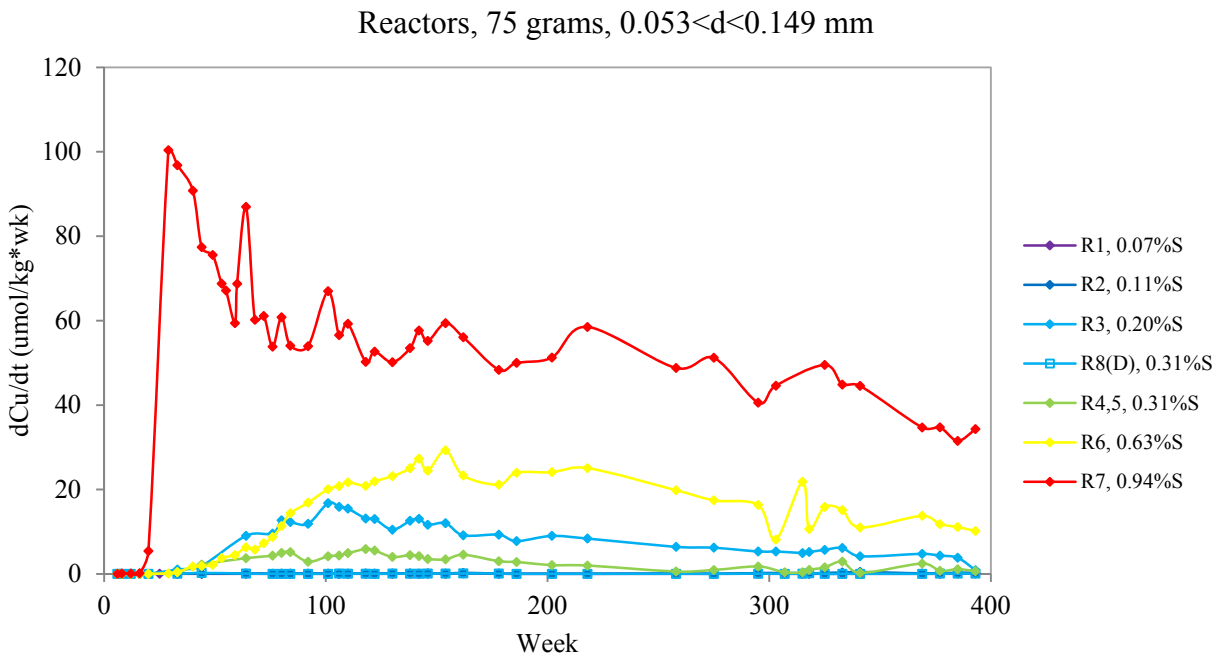


Figure 14. Copper release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



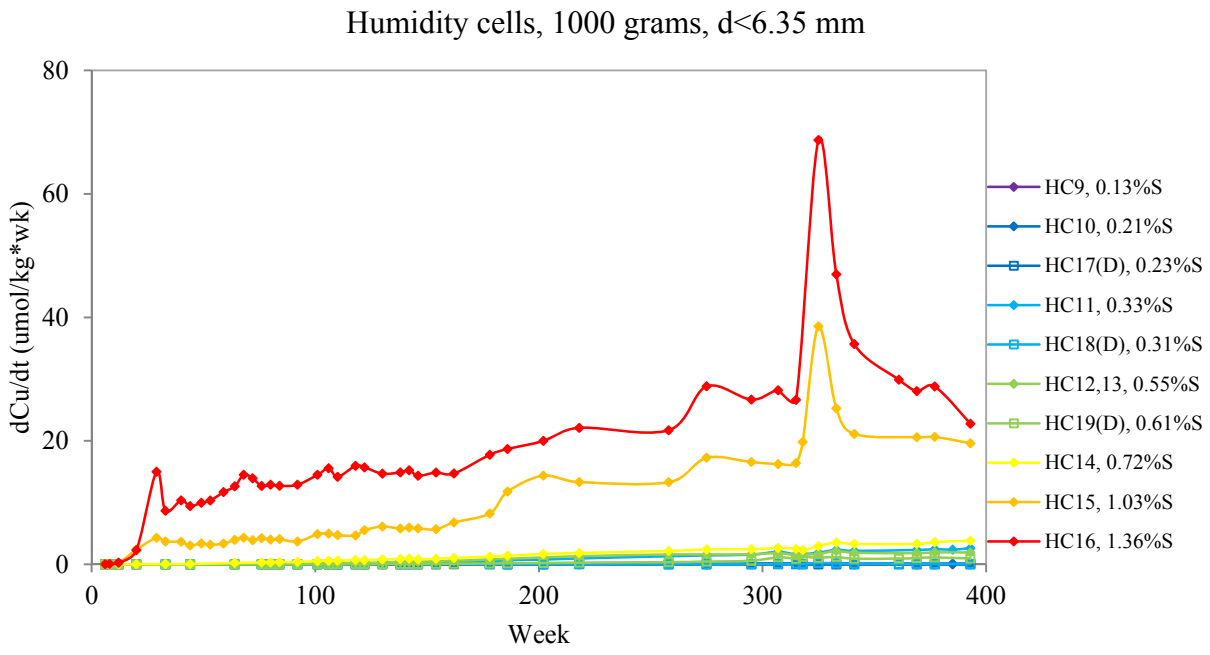
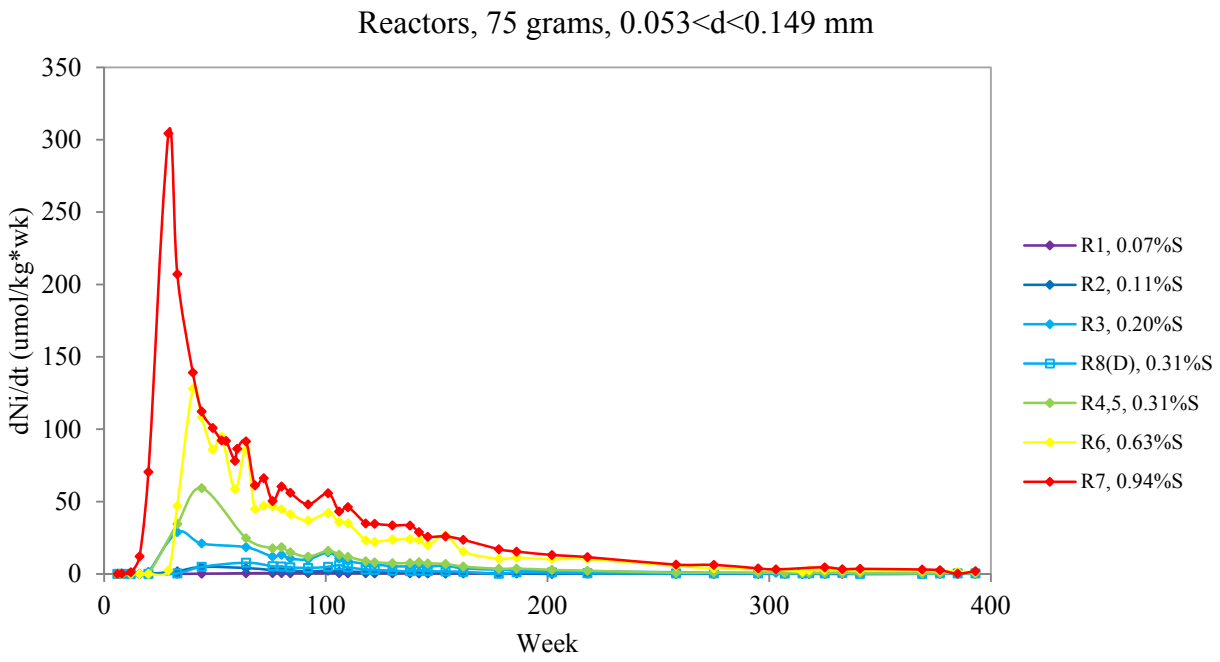


Figure 15. Nickel release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



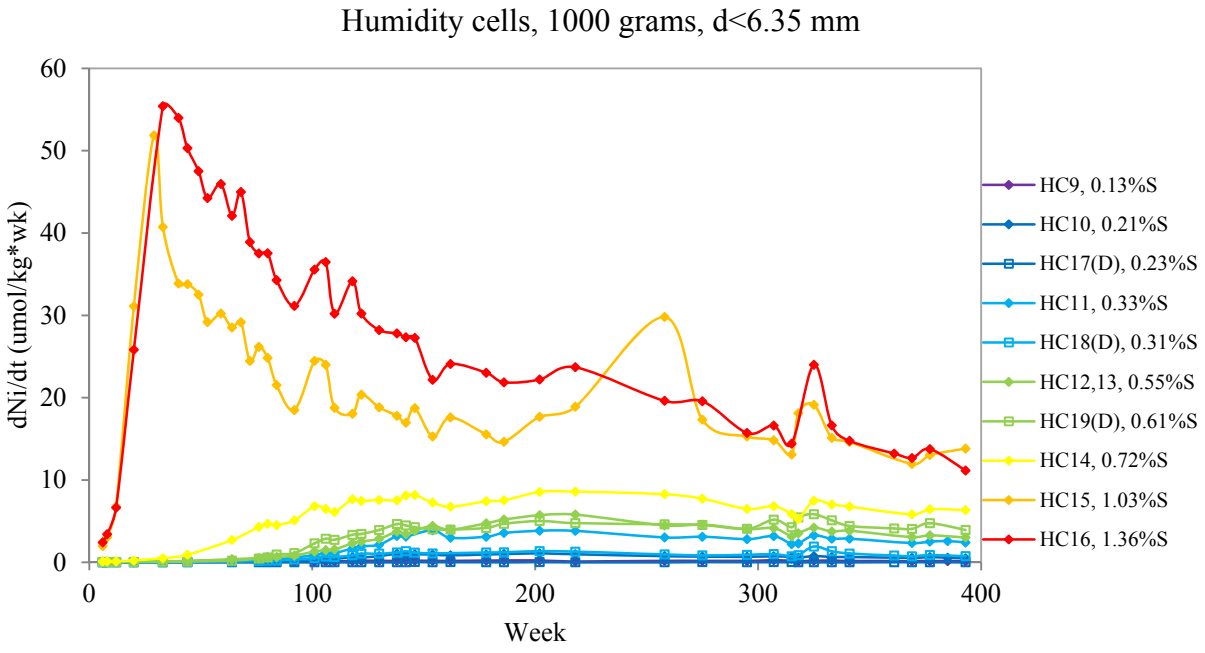
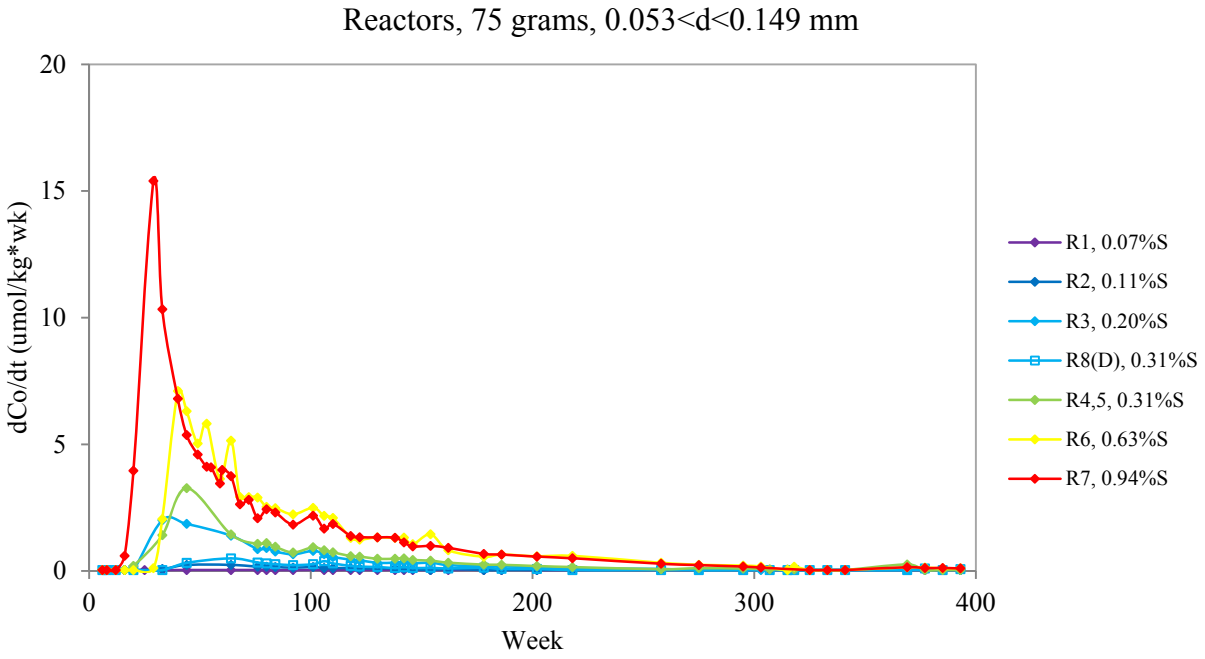


Figure 16. Cobalt release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



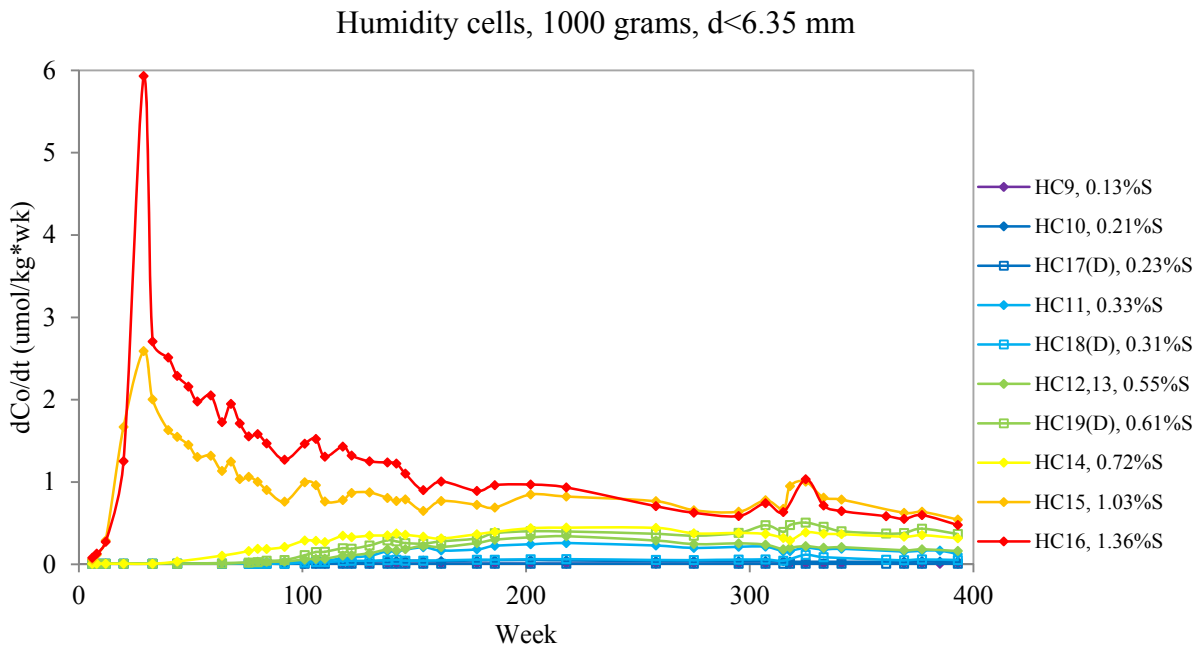
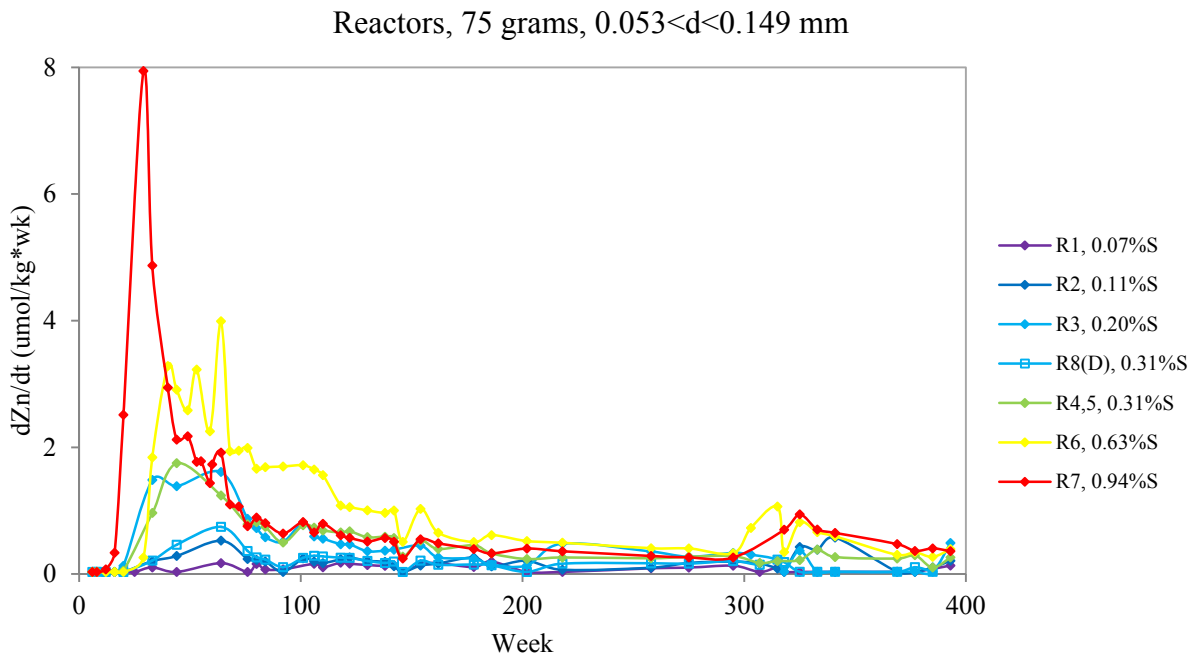


Figure 17. Zinc release vs. time for reactors and humidity cells. Weeks 0-5 excluded to improve resolution and duplicates are averaged.



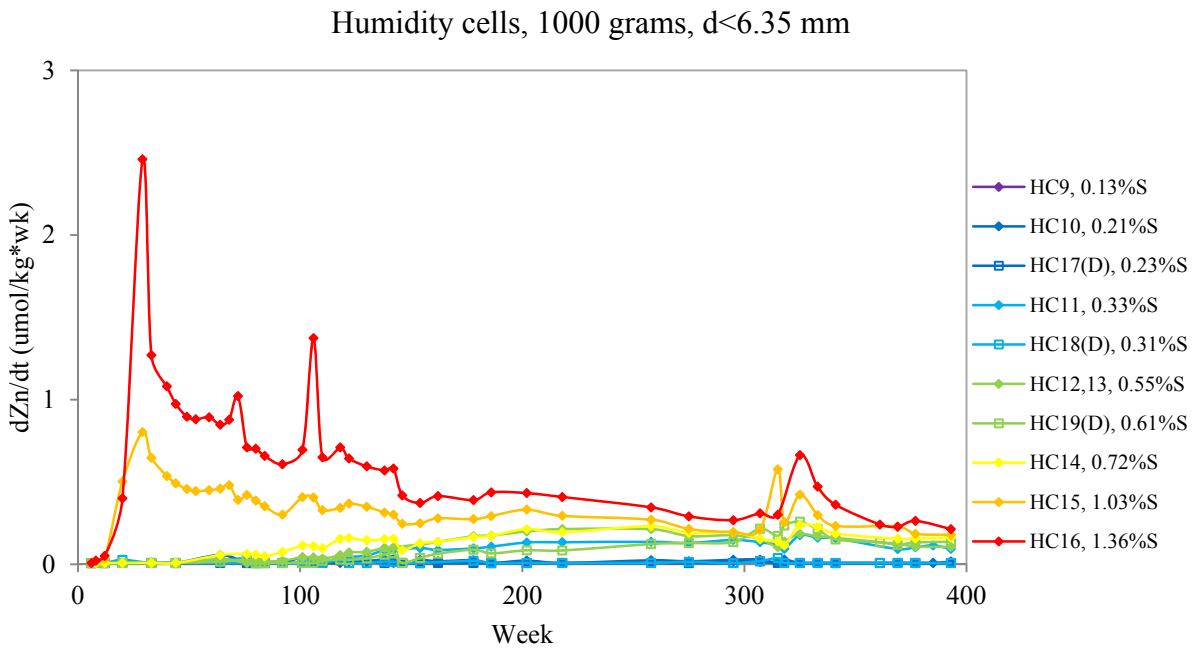


Figure 18. Minimum pH vs. %S for reactors (-100/+270 mesh) and humidity cells (-1/4 inch) for weeks 6-400.

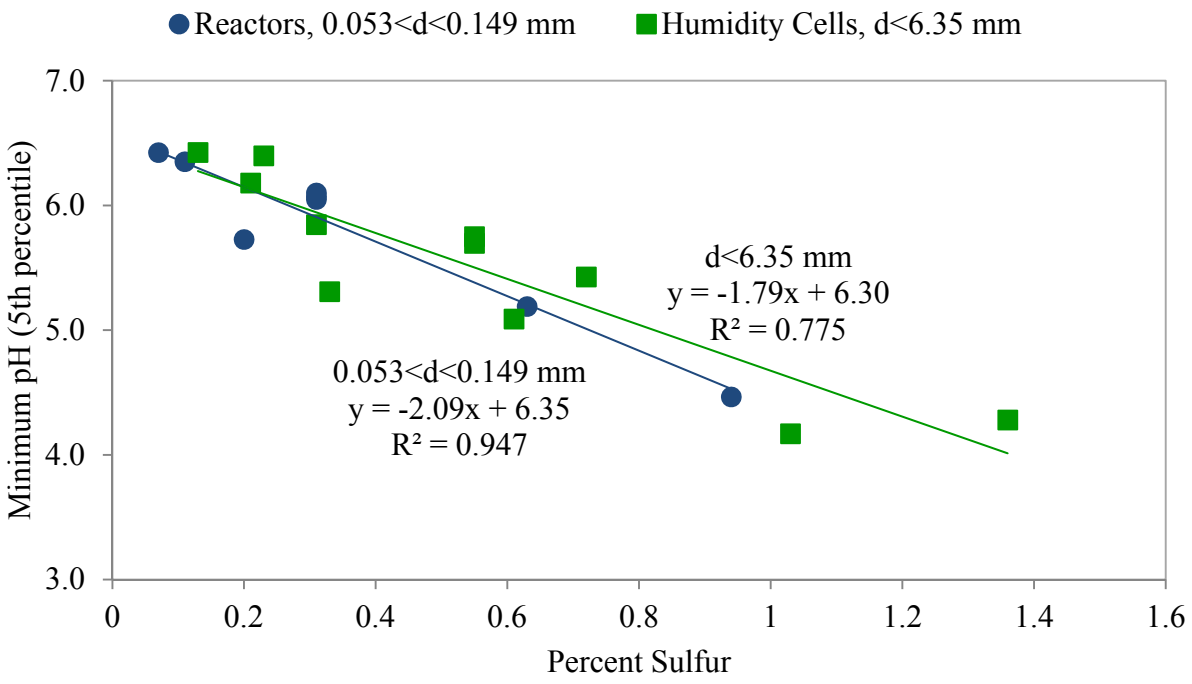


Figure 19. Average sulfate release vs. %S for reactors (-100/+270 mesh) and humidity cells (-1/4 inch) for weeks 6-400.

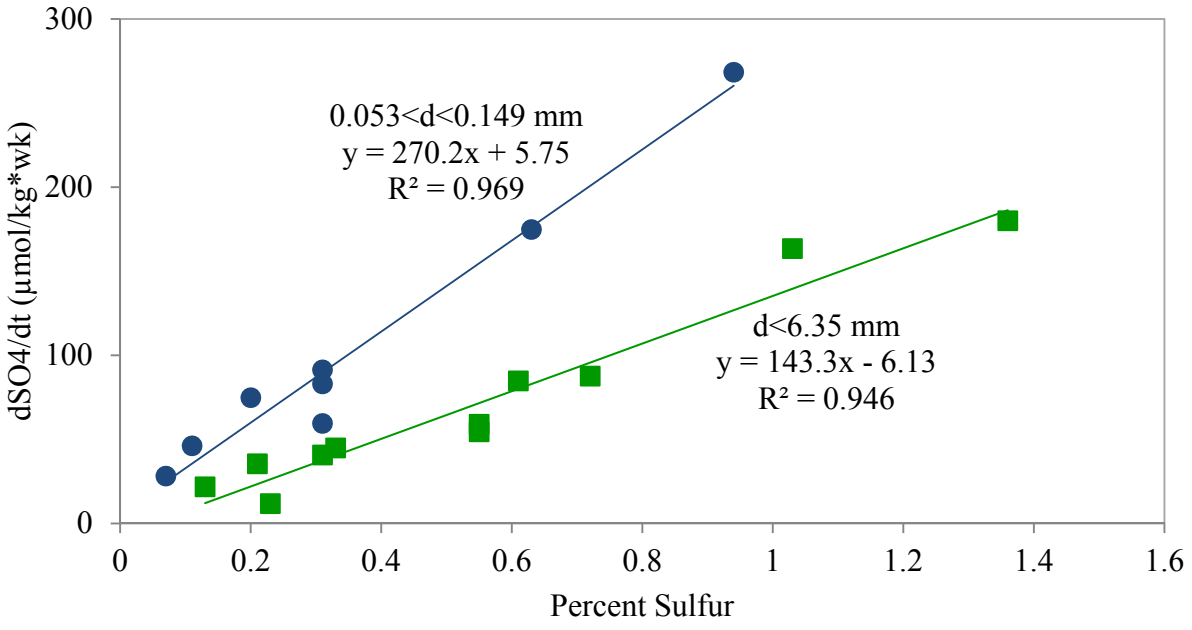


Figure 20. Annual average sulfate release (µmol/kg*wk) vs. %S for Partridge River intrusion samples. Annual rates are shown for Years 2 and 4.
Blue = reactors (0.053<d<0.149), Green = humidity cells (d<6.35 mm)

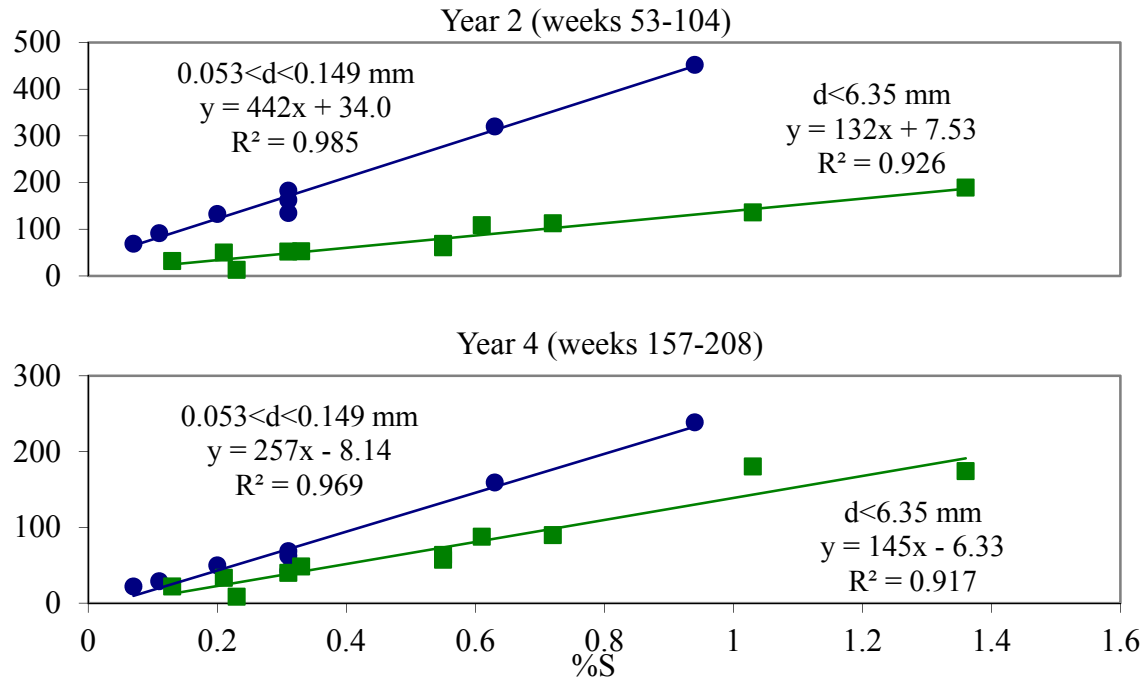


Figure 21. Annual average sulfate release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.948 and 0.925 respectively. Solid line = reactors, dashed line = humidity cells.

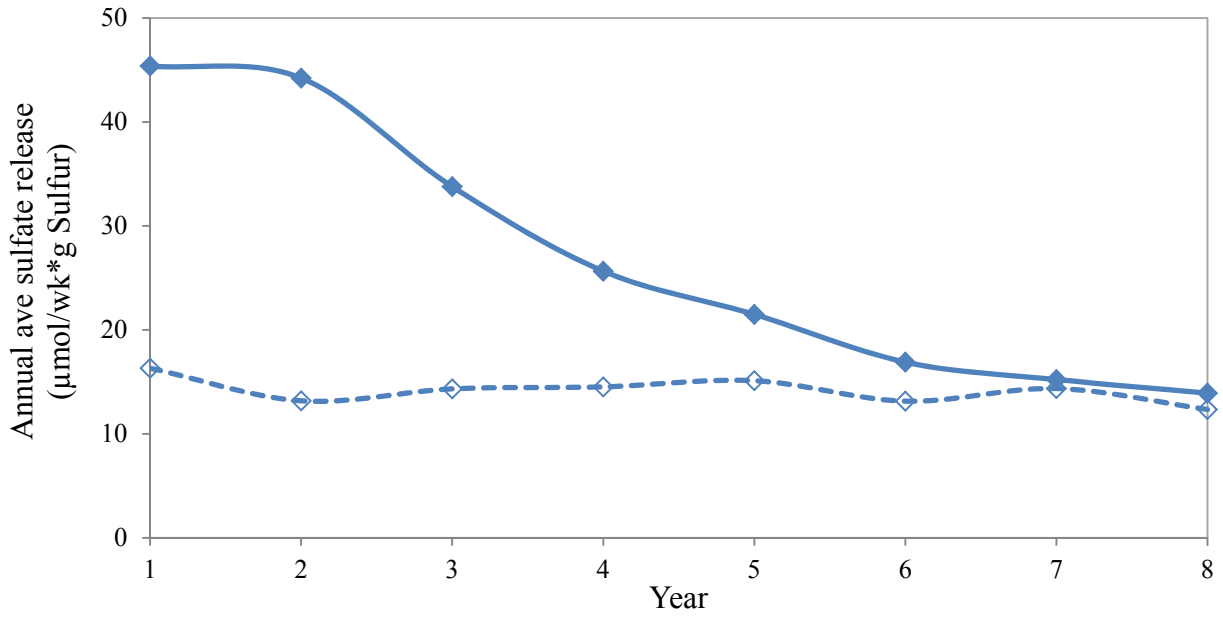


Figure 22. Annual average calcium and magnesium release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.662 and 0.611 for Ca and 0.687 and 0.687 for Mg, respectively. Solid line = reactors, dashed line = humidity cells.

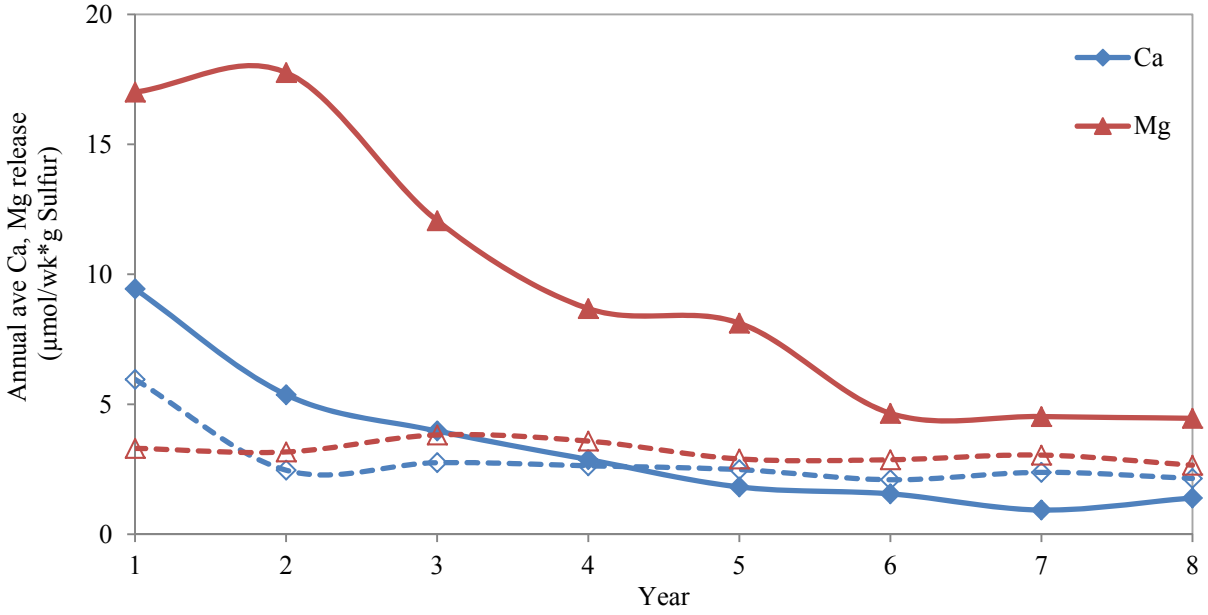


Figure 23. Annual average sodium and potassium release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.810 and 0.769 for Na and 0.269 and 0.248 for K, respectively. Solid line = reactors, dashed line = humidity cells.

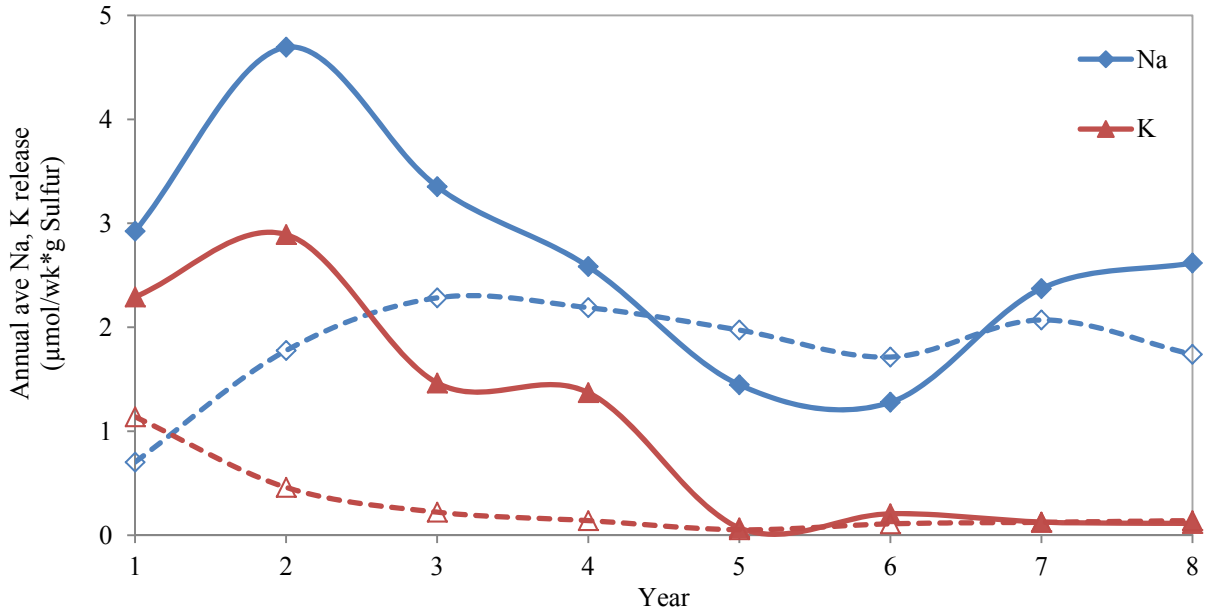


Figure 24. Annual average copper and nickel release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.816 and 0.778 for Cu and 0.925 and 0.886 for Ni, respectively. Solid line = reactors, dashed line = humidity cells.

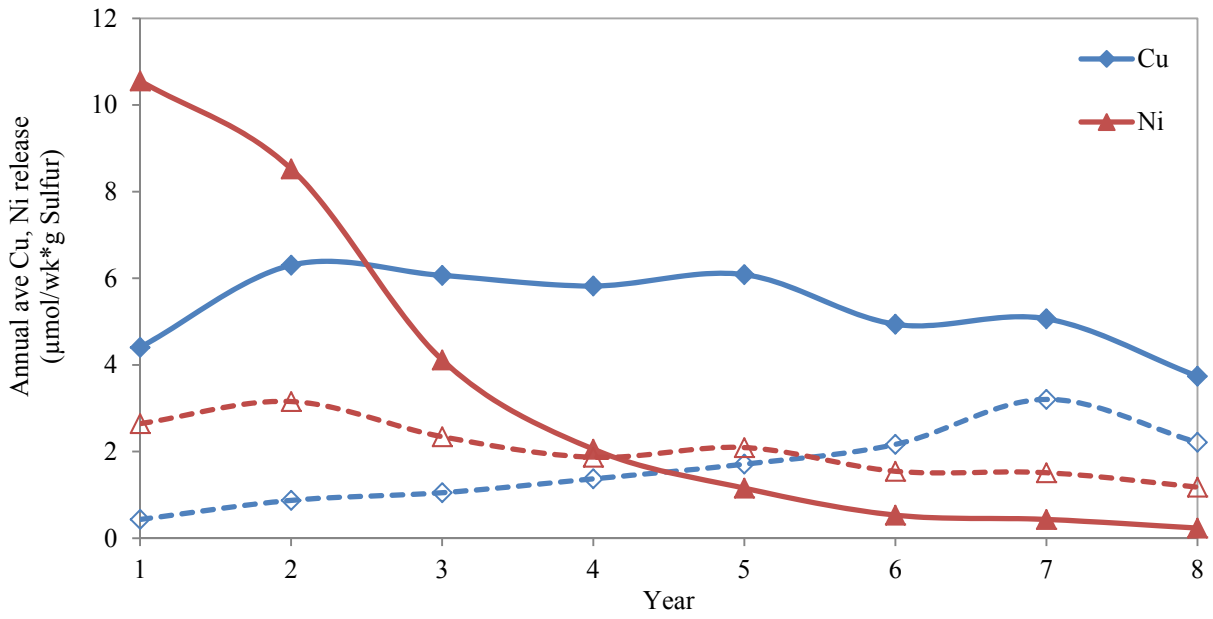


Figure 25. Annual average cobalt and zinc release per gram of sulfur vs. time. Average R^2 values for the reactors and humidity cells were 0.695 and 0.883 for Co and 0.558 and 0.851 for Zn, respectively. Solid line = reactors, dashed line = humidity cells.

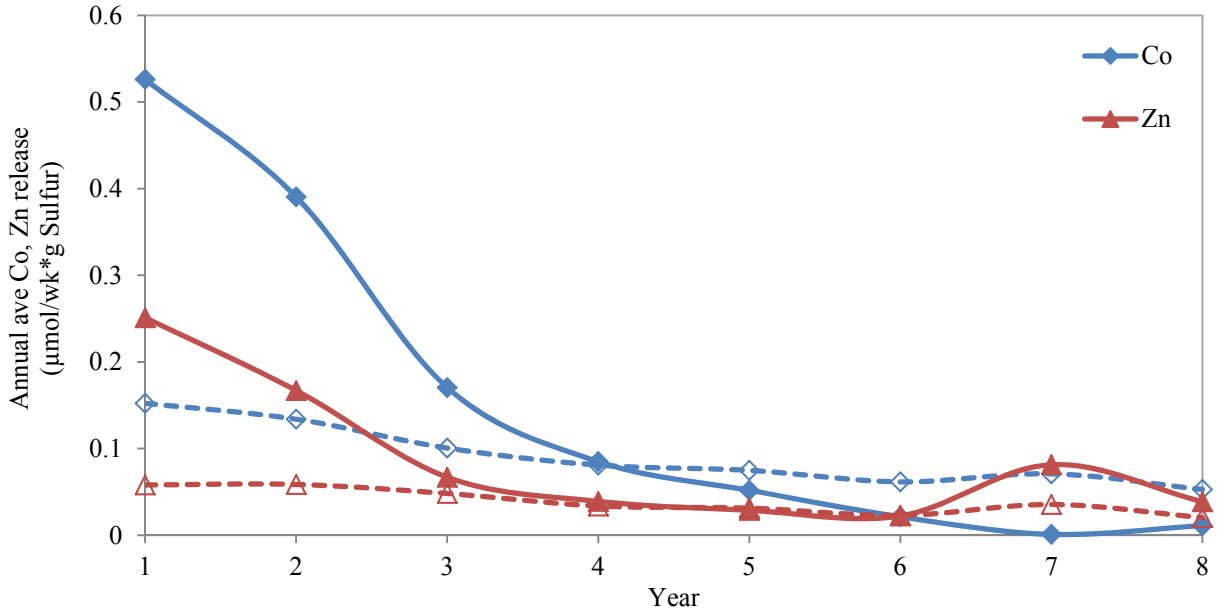


Figure 26. Average nickel and cobalt release vs. %S for reactors (-100/+270 mesh) and humidity cells (-1/4 inch) for weeks 6-104.

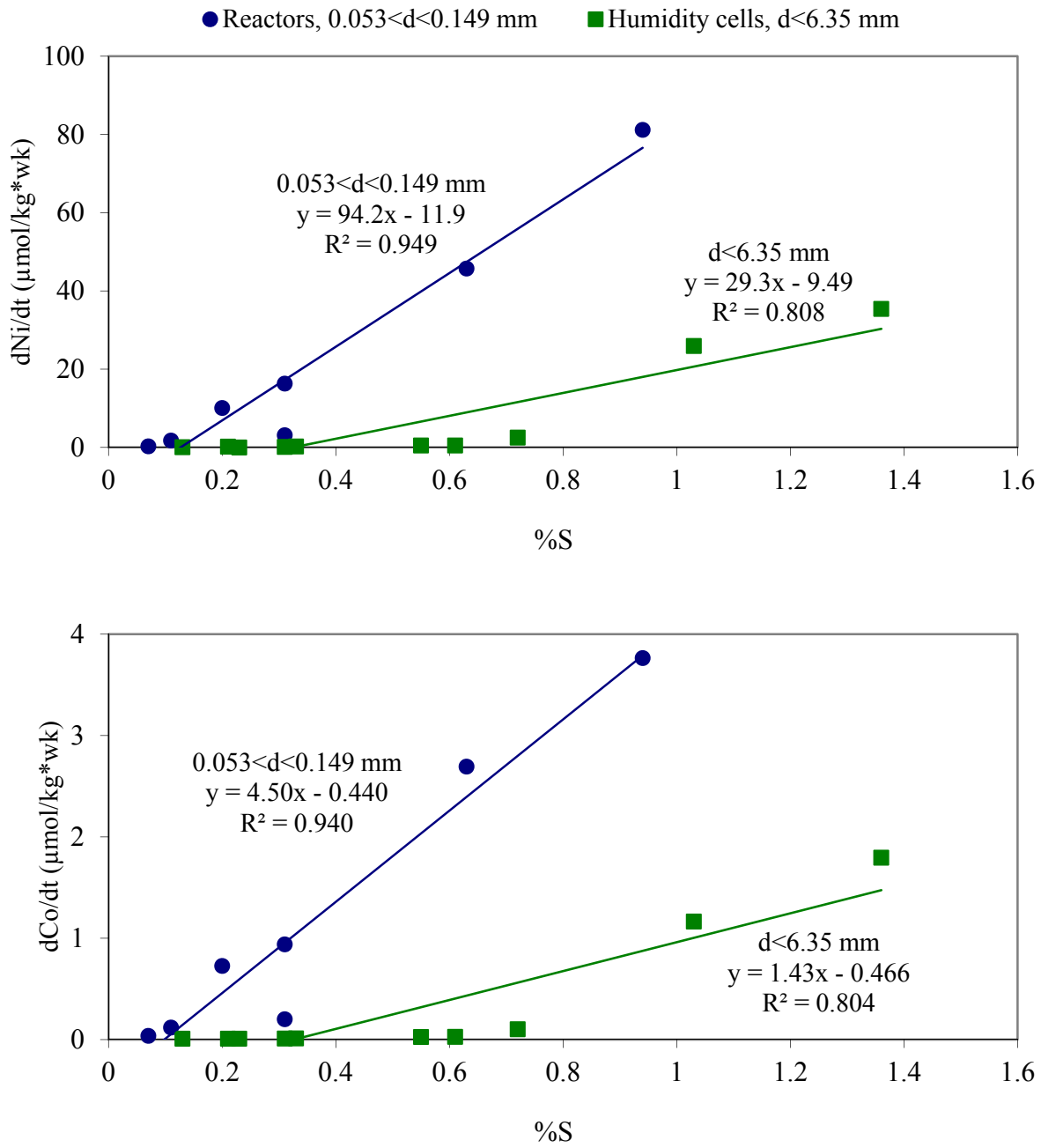


Figure 27. Drainage quality from 75-g samples of 0.31 %S Babbitt Prospect and Dunka Road rocks. Weeks 0-5 were excluded to improve resolution.

Red = Babbitt prospect, Blue = Dunka Road prospect

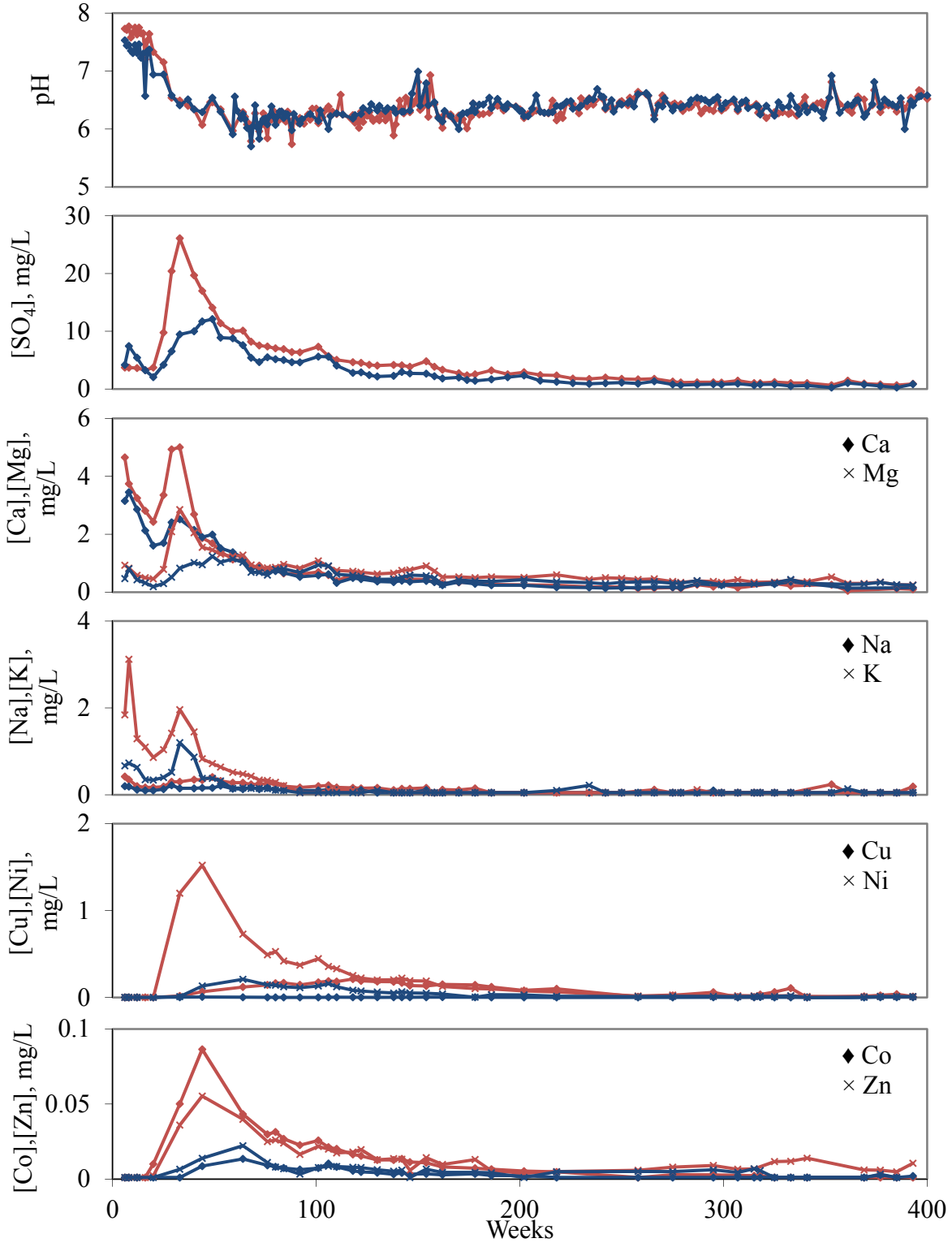


Figure 28. Drainage quality from 1000-g samples of 0.21 %S Babbitt Prospect rock and 0.23 %S Dunka Road rock. Weeks 0-5 were excluded to improve resolution.

Red = Babbitt prospect, Blue = Dunka Road prospect

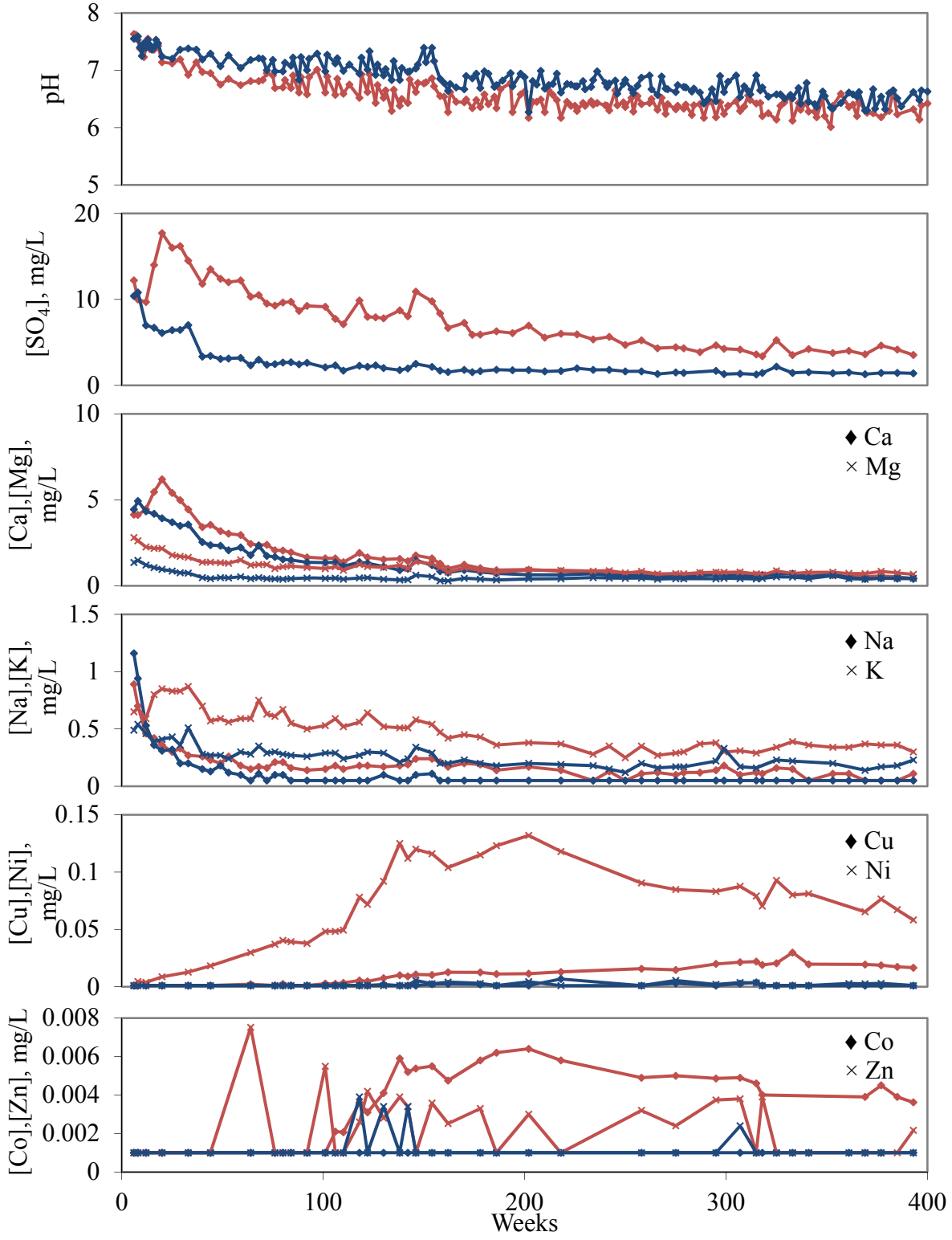


Figure 29. Drainage quality from 1000-g samples of Babbitt prospect (0.33 %S) and Dunka Road rock (0.31 %S). Weeks 0-5 were excluded to improve resolution.

Red = Babbitt prospect, Blue = Dunka Road prospect

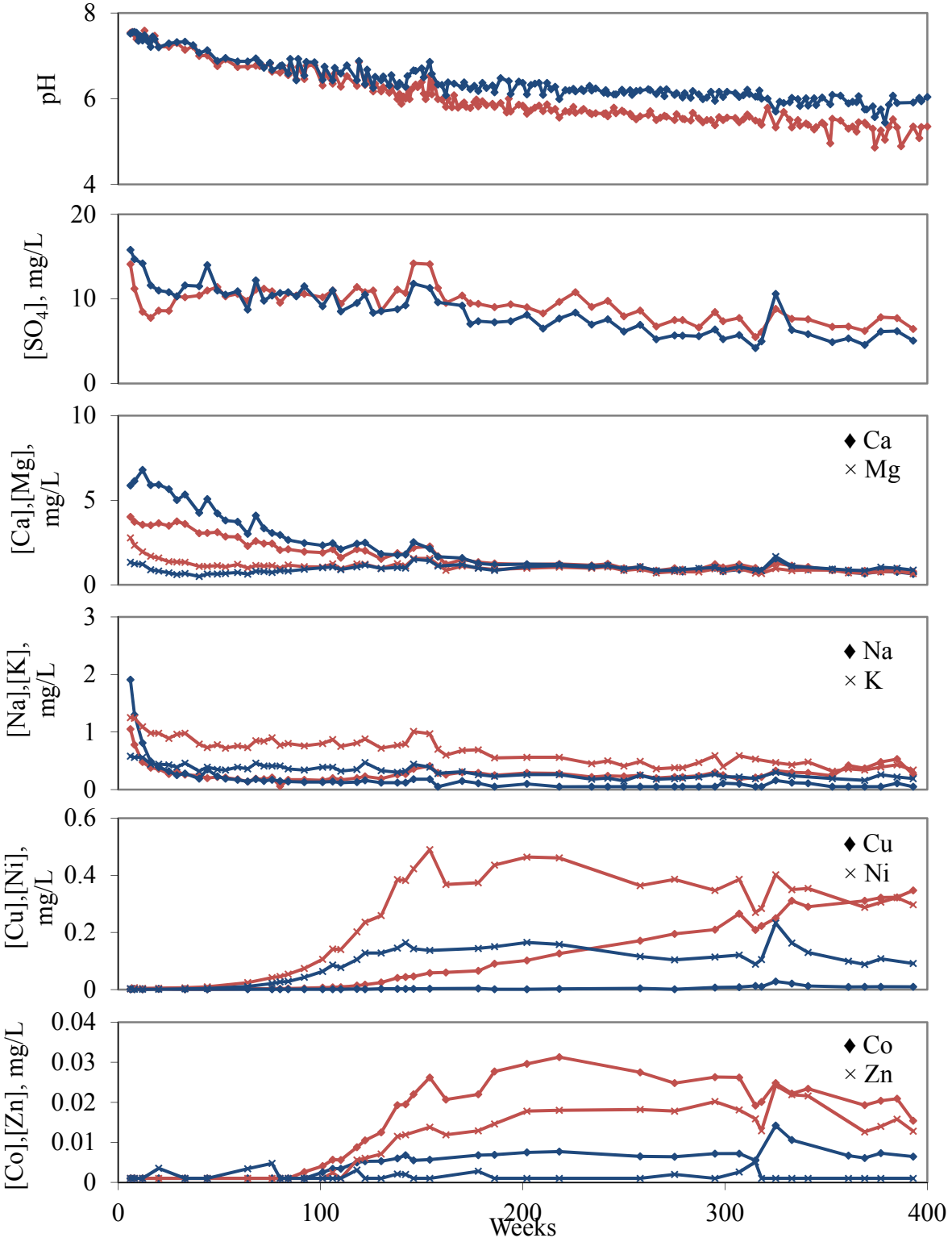


Figure 30. Drainage quality from the 1000-g sample of 0.55 %S Babbitt Prospect and 0.61 %S Dunka Road rock. Weeks 0-5 were excluded to improve resolution.

Red = Babbitt prospect, Blue = Dunka Road prospect

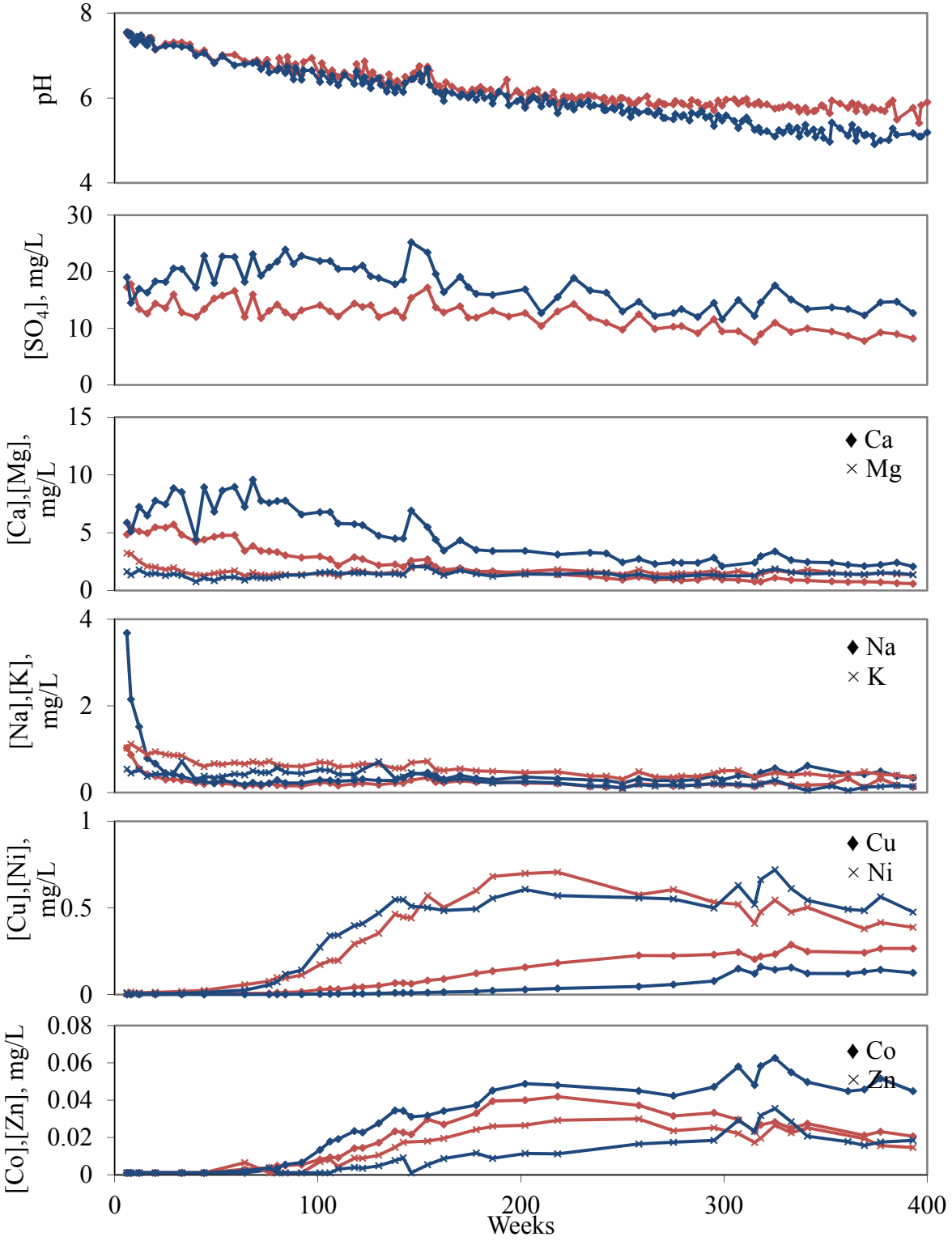


Figure 31. Drainage pH and sulfate, calcium, and magnesium release (µmol/kg*wk) vs. time from the 75-g sample of 0.20 %S Babbitt Prospect waste rock and unsaturated sample of 0.20 %S Duluth Complex tailings rock. Weeks 0-5 were excluded to improve resolution.

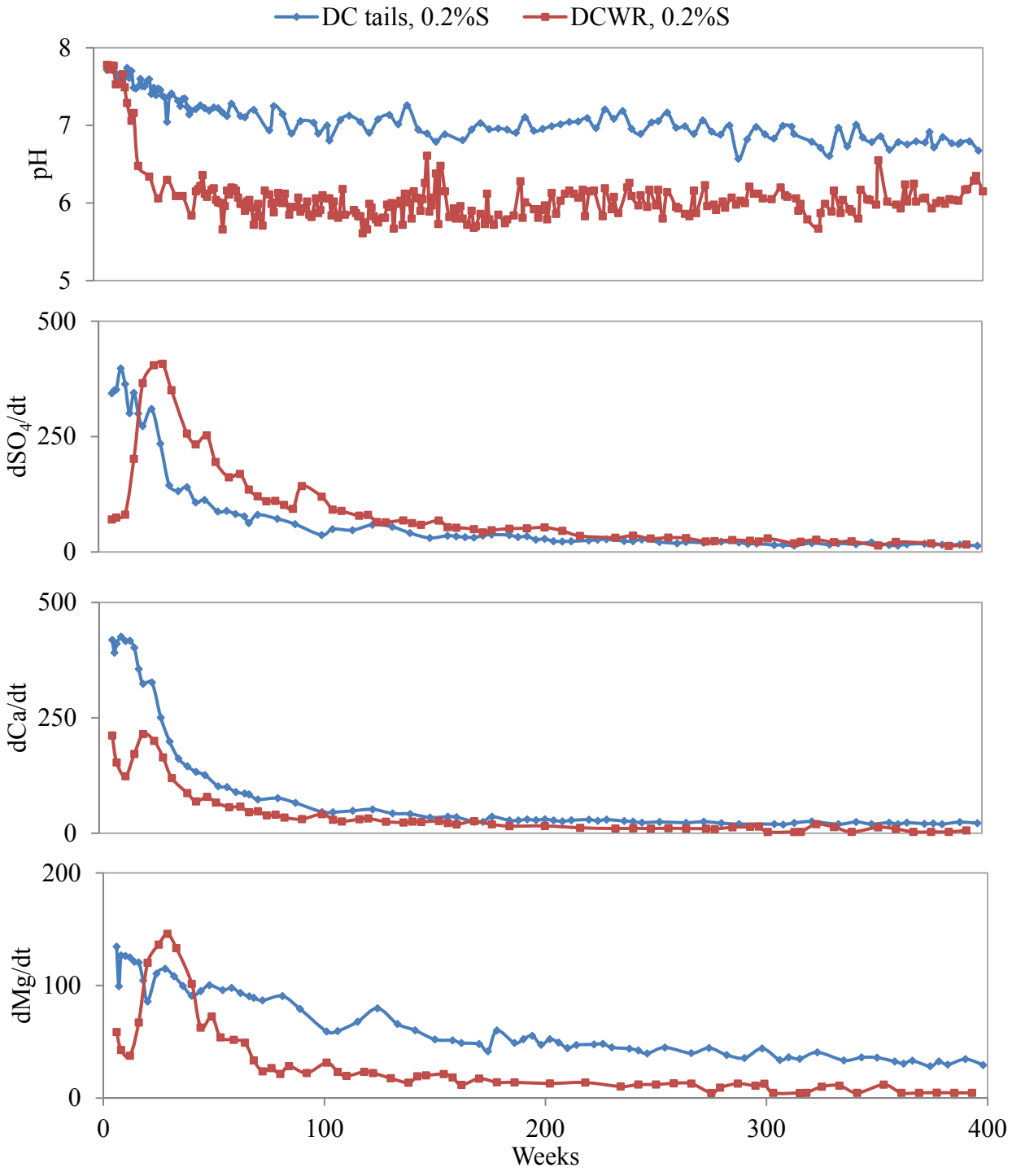


Figure 32. Heavy metal release (µmol/kg*wk) vs. time from the 75-g sample of 0.20 %S Babbitt Prospect waste rock and unsaturated sample of 0.20 %S Duluth Complex tailings rock. Weeks 0-5 were excluded to improve resolution. Note logarithmic scale on y-axis.

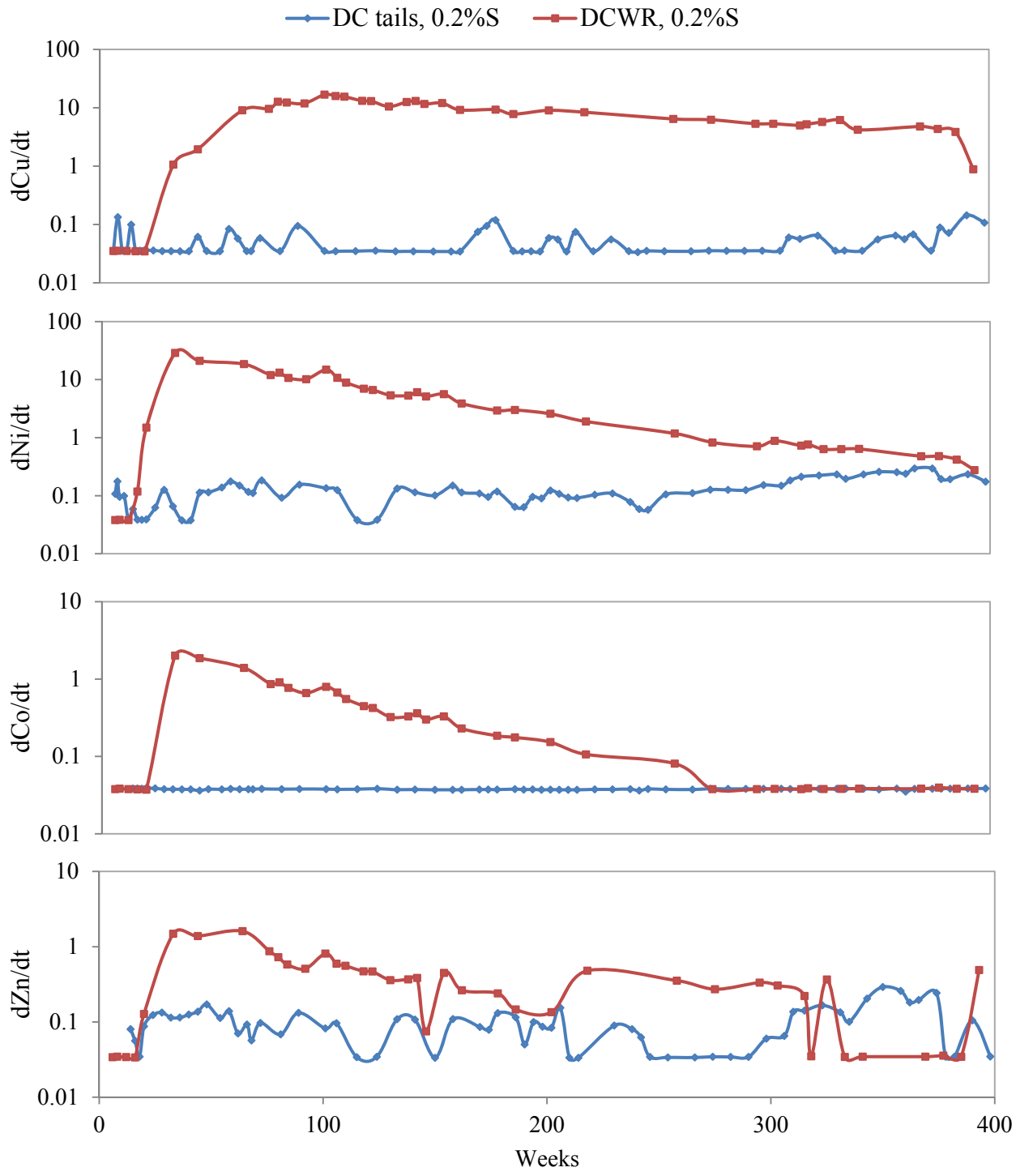


Figure 33. Copper content vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI.

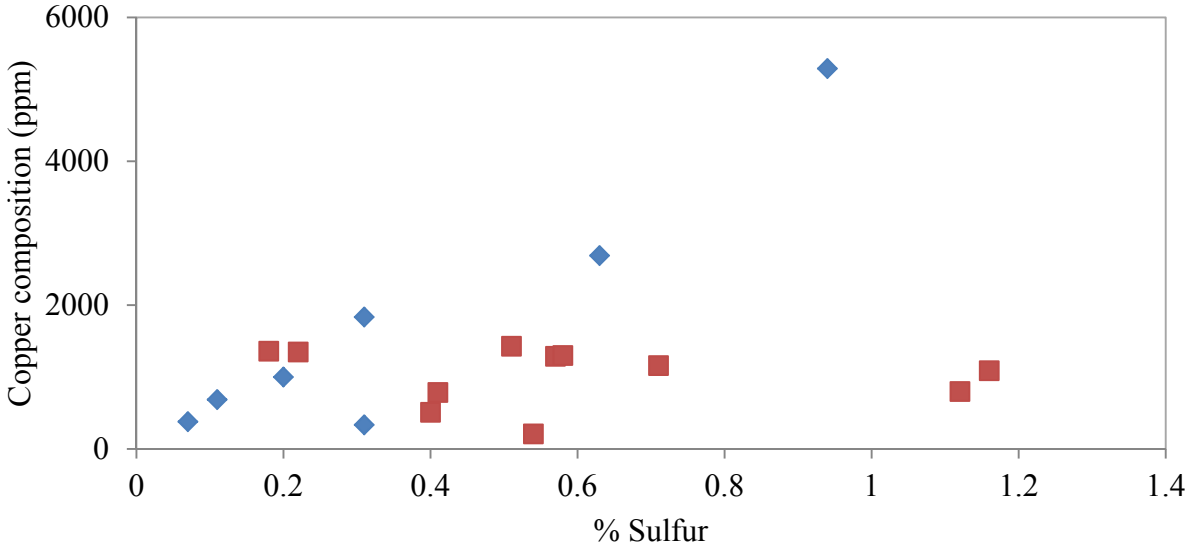


Figure 34. Nickel content vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI.

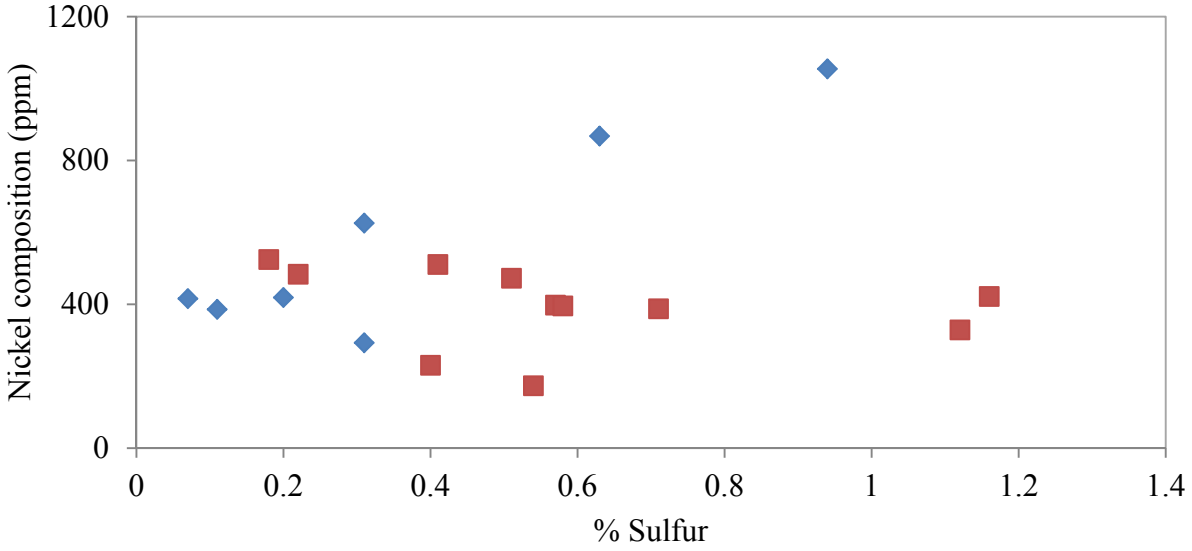


Figure 35. Zinc content vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI.

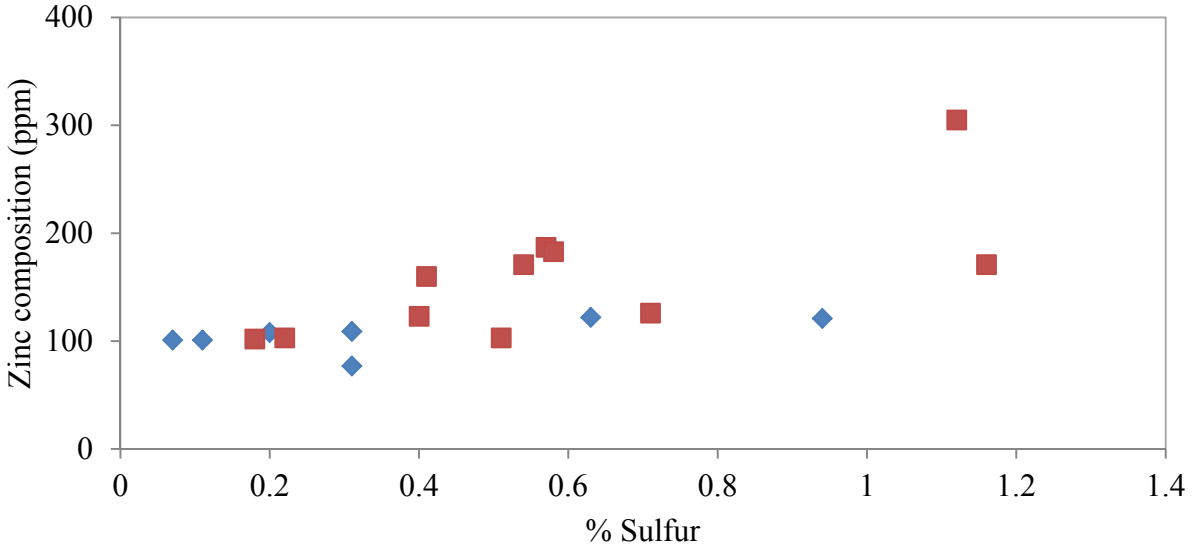


Figure 36. Cobalt content vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI.

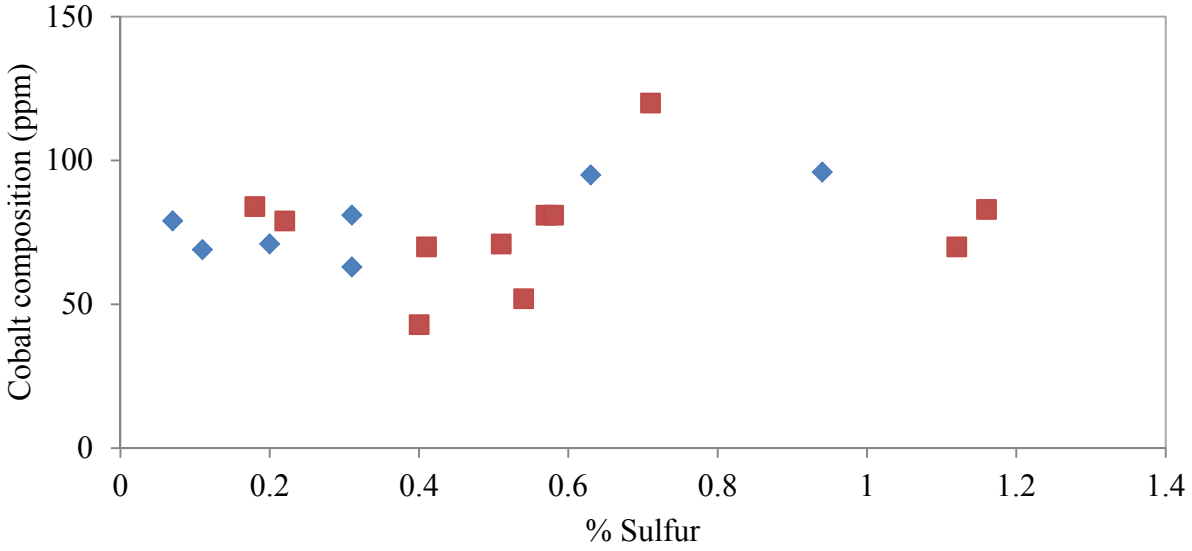


Figure 37. Minimum pH vs. % sulfur for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples through 400 weeks. Blue = PRI, Red = SKI

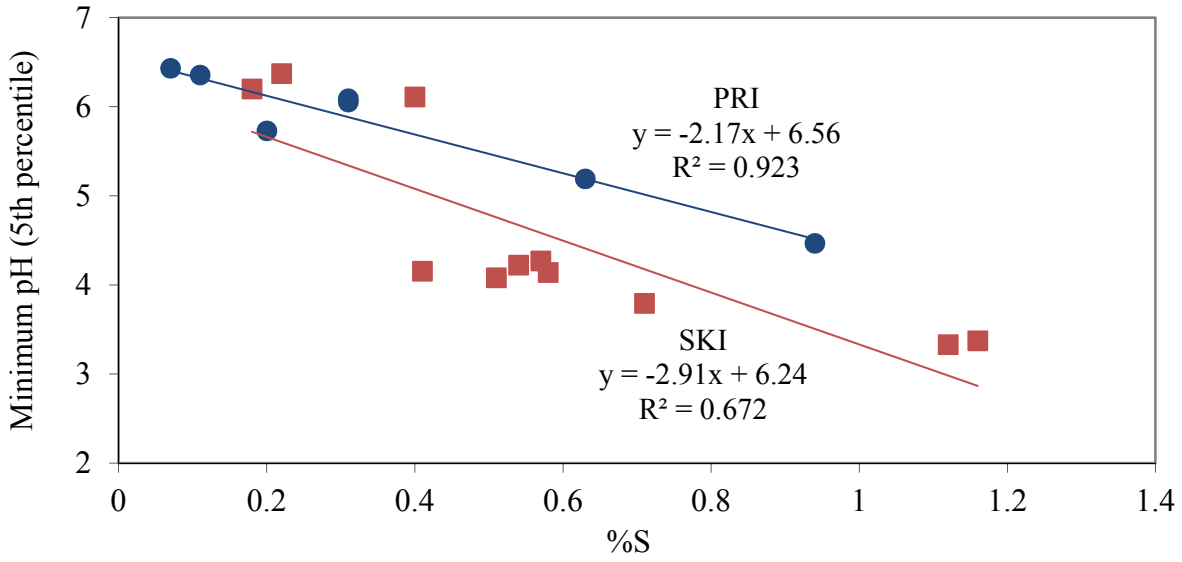


Figure 38. Week of minimum pH vs. %S for Partridge River (PRI) and South Kawishawi (SKI) intrusion samples. Blue = PRI, Red = SKI. Higher sulfur SKI samples typically took 250-350 weeks to reach minimum pH values.

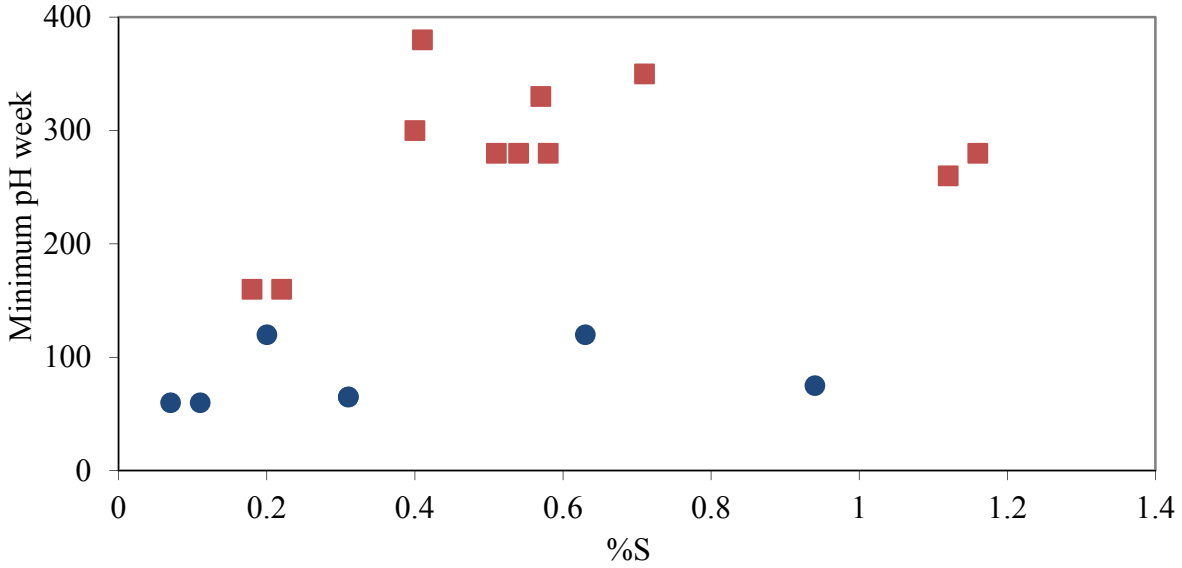


Figure 39. Annual average sulfate release (µmol/kg*wk) vs. %S for Partridge River (PRI) intrusion and South Kawishiwi intrusion (SKI) samples. Annual rates are shown for Years 2 and 6. Blue = PRI, Red = SKI

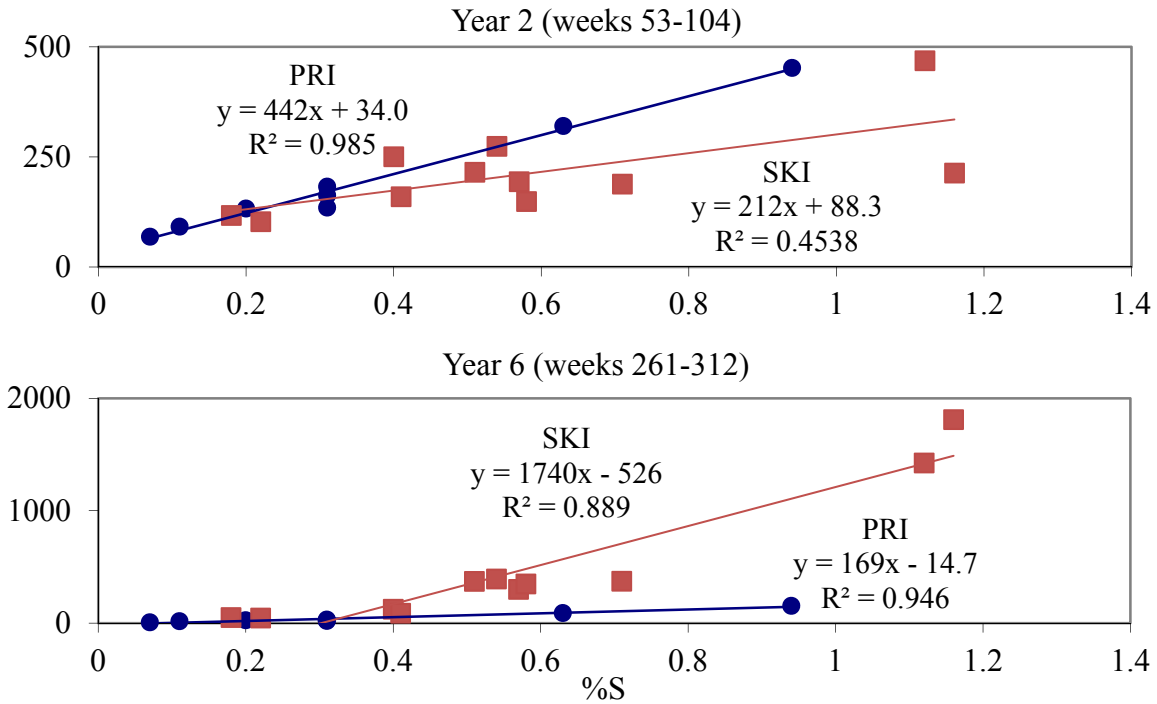


Figure 40. Annual average sulfate release per gram of sulfur vs. time between Babbitt and Dunka Road prospect rock (PRI) and Dunka Mine rock (SKI).

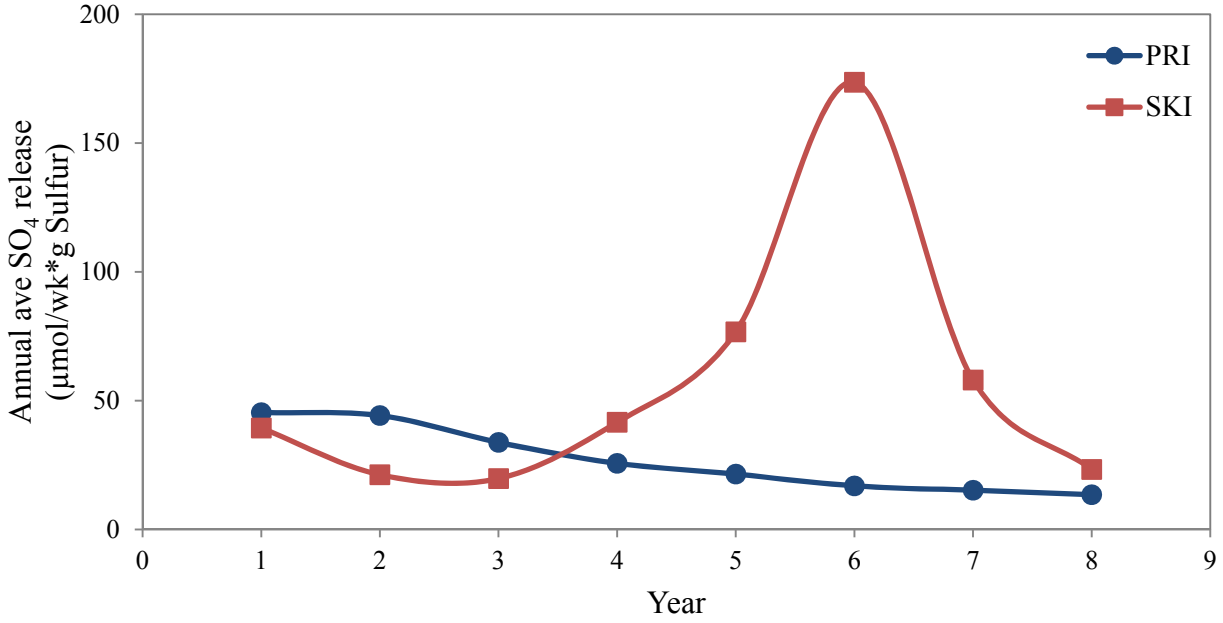


Figure 41. dNi/dt vs %S for 0.053 < d < 0.149 mm Partridge River (PRI) and South Kawishiwi (SKI) intrusion samples, weeks 6-74.

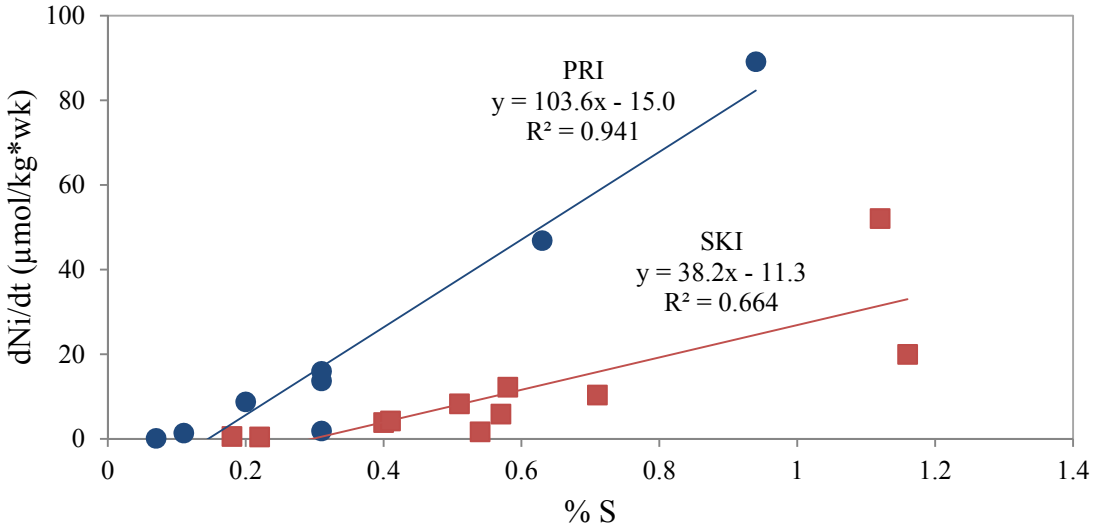


Figure 42. dCu/dt vs %S for 0.053 < d < 0.149 mm Partridge River (PRI) and South Kawishiwi (SKI) intrusion samples, weeks 6-74. Note logarithmic scale on y-axis.

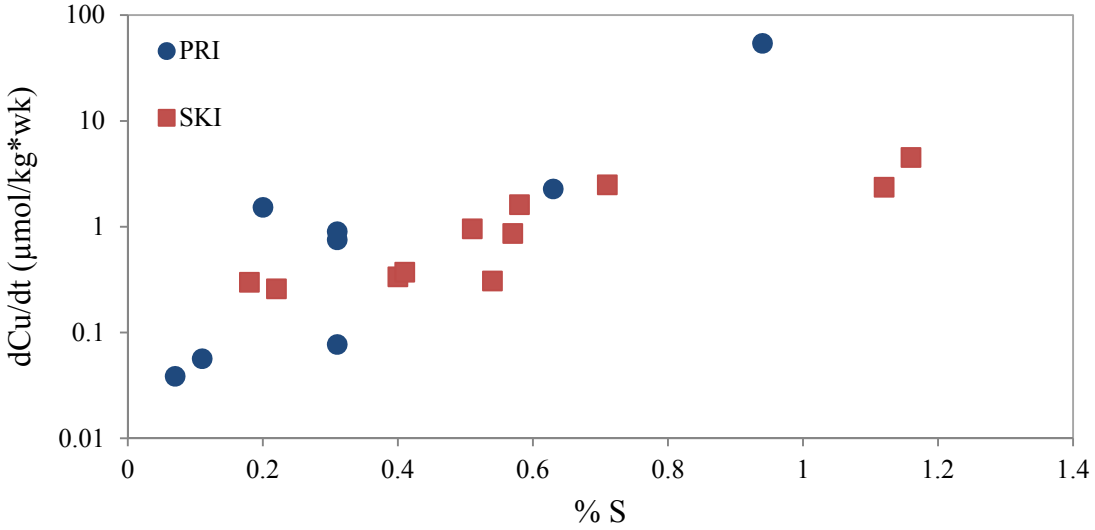
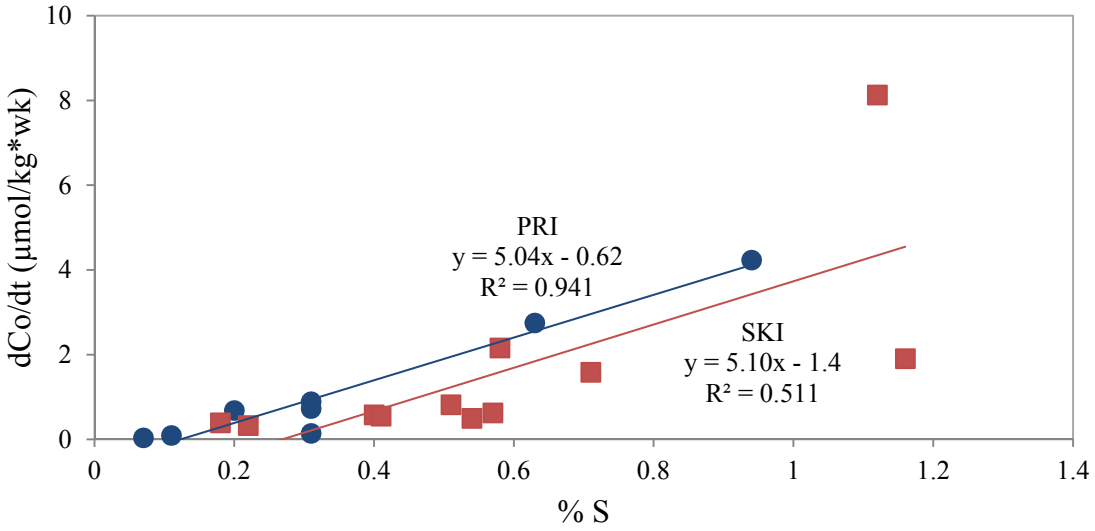


Figure 43. dCo/dt vs %S for 0.053 < d < 0.149 mm Partridge River (PRI) and South Kawishiwi (SKI) intrusion samples, weeks 6-74.



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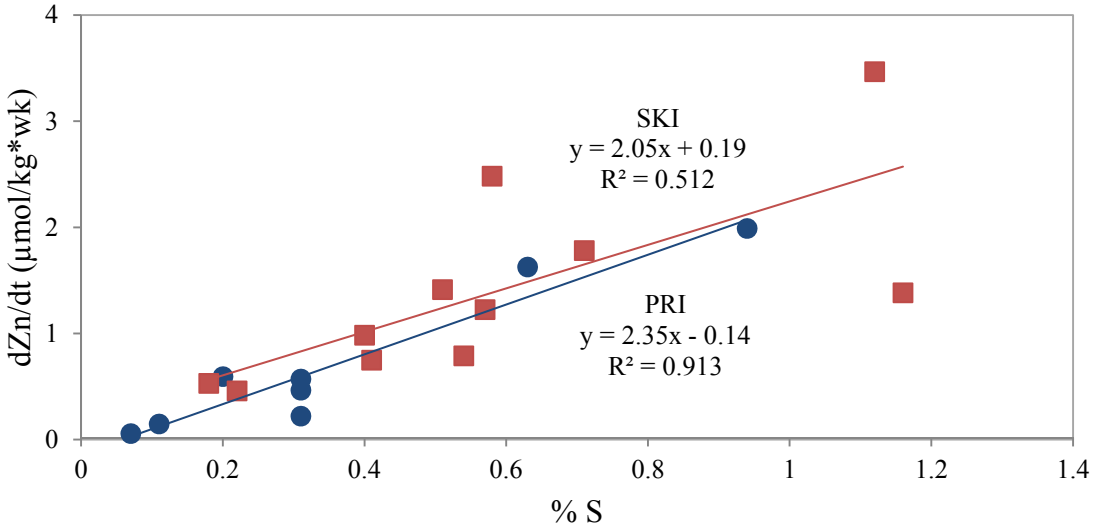

Attachment 17

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Duluth Complex Tailings Dissolution:
Ten-year laboratory experiment

June 2013

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Duluth Complex Tailings Dissolution:
Ten-year laboratory experiment

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0. EXECUTIVE SUMMARY

Environmental review of proposed mining operations in Duluth Complex rock requires evaluation of potential impacts of tailings on water quality. Sulfide minerals present in such mine tailings can react with oxygen and water to release acid and heavy metals. This reaction is affected by the water content of the tailings and the depth within the tailings. Laboratory dissolution tests are typically conducted by mining companies to aid in predicting the water quality resulting from tailings dissolution. The effectiveness of predictions based on such tests is dependent on their design and the extrapolation of the laboratory results to field conditions. Both of these aspects require an understanding of reactions occurring in tailings basins and the variables that affect these reactions. The present study contributes to this understanding and provides an extensive dissolution test data set for comparison with, and to inform interpretation of, test results submitted for environmental review.

In the present project laboratory dissolution tests were conducted on tailings produced by pilot-plant processing of Duluth Complex rock from the Babbitt prospect (also known as AMAX or Mesaba prospect) of the Partridge River Intrusion. Reaction of these tailings affects water quality, and the effects of 1) tailings water content and 2) bed depth on reactions were examined. Of particular interest were reactions affecting pH and release of copper, nickel, cobalt, and zinc. The present report addresses the characterization of tailings (bulk density, particle size distribution, chemistry, mineralogy, static tests) and 10 years of laboratory dissolution testing. Dissolution tests were conducted on 75-g samples in unsaturated (uncovered) and saturated (covered) reactors (both in triplicate) and in duplicate 1000-g humidity cells.

Key dissolution tests results are as follows.

1. All test produced circumneutral drainage pH, with the minimum of all values near 6.4.
2. During the first three years of testing, sulfide oxidation in saturated (non-evaporating) reactors was more rapid than that in unsaturated (evaporating) reactors, apparently due to the greater water retention.
3. The aforementioned accelerated oxidation yielded slightly lower drainage pH values in the saturated reactors.
4. The slightly lower pH values resulted in nickel concentrations in saturated reactor drainage that were roughly 25 times those in drainage from the unsaturated reactors.
5. Sulfide mineral oxidation rates (normalized for tailings mass) for the humidity cells were limited by oxygen diffusion, resulting in drainage pH values higher and trace metal concentrations lower than those for the smaller reactors.

Thus, although similar Partridge River intrusion tailings are unlikely to produce acidic drainage, nickel release will increase with pH decreasing into the sixes. Results further indicated that such decreases are more likely in areas of the basin in which tailings are saturated (but not submerged). The data generated also provide a basis for evaluating the variation of oxidation reactions with tailings depth and 10 years of drainage quality data for comparison with, and to aid interpretation of, future results.

1. INTRODUCTION

Teck-Cominco Inc. (Vancouver, B.C. Canada) processed a bulk sample of Duluth Complex rock collected in April 2001 from approximately the center of Section 29, T60N, R12W. The company sent 280 kg of the tailings generated from their pilot plant test to the Minnesota Department of Natural Resources Division of Lands and Minerals (MN DNR) in the late summer of 2001. These tailings were characterized (bulk density, particle size distribution, chemistry, mineralogy) and subjected to dissolution tests that began 19 February 2002. This is the second progress report on the project, and it presents drainage quality data through 7 February 2012, a total of 520 weeks of dissolution testing. Previously a 2003 report addressed the first 30 weeks of the experiment, and a 2009 paper compared results from the three different test methods over a period of about four years (Lapakko et al. 2003, Lapakko and Berndt 2009).

2. OBJECTIVES

In general, the present project intends to investigate the quality of drainage that would be generated by tailings produced by processing of Duluth Complex rock for recovery of base metals. Of particular interest are the potentials for release of acid, copper, nickel, cobalt, and zinc. In addition, comparison of laboratory dissolution methods for prediction of mine waste drainage quality was of interest. More specifically, the objectives of the study are as follows.

1. Describe the temporal variation of drainage pH and concentrations of alkalinity, sulfate, calcium and magnesium.

2. Determine the rates of sulfate, calcium, magnesium, sodium, and potassium release from tailings.

3. Determine concentrations of copper, nickel, cobalt and zinc in laboratory drainage from tailings.

4. Compare results from saturated and unsaturated small reactors containing 75 g tailings.

5. Compare results from small reactors with those from humidity cells containing 1000 g tailings.

3. METHODS

3.1. Materials

A tailings sample of approximately 280 kg was received by the MN DNR. The tailing sample was from the processing of a bulk sample of mineralized gabbro taken in 2001 from the Babbitt (Mesaba) prospect of the Partridge River Intrusion (PRI) near Babbitt, MN (see Figure 1 for flow chart). Three randomly selected samples were taken and analyzed at Lerch Brothers Inc. (Hibbing, MN) for total sulfur and CO_2 to determine the variability in tailings composition.

Total sulfur contents of the triplicate samples were 0.20, 0.19, and 0.20 percent, and the corresponding values for CO_2 were 0.28, 0.23, and 0.29 percent. The results indicated the tailings were reasonably homogeneous, but the CO_2 measurements were suspect based on previous experience with Duluth Complex rock. (As indicated in Results, subsequent analyses yielded considerably lower values.)

From the original bulk sample, a subsample weighing approximately 30 kg was randomly selected for additional testing. The sample was spread out on a table lined with paper to air dry. After drying a metal rolling pin was used to crush the tailings that were clumped together. The sample was then split using the Four Corners method (Scott 1942). Approximately one half (15 kg) of the sample was removed. The sample was then split again using the Four Corners method. Again approximately one half (7 kg) of the sample was removed. These two samples were archived for possible future testing.

The remainder of the sample (8 kg) was split using a Jones Splitter containing 0.75-inch chutes. After passing the sample through the splitter 3 times, one half (4 kg) of the sample was removed and archived. The remainder of the sample was split into aliquots of approximately 0.25 kg. Four of these aliquots were randomly selected to make up two 1 kg humidity-cell charges. Six aliquots were split one additional time to make up six 75-gram reactor charges. The remaining sample for each of the eight aliquots was archived for future analyses.

Dry and wet bulk densities of the tailing were determined at the MN DNR laboratory in Hibbing, MN. The dry density was determined by weighing 200 cm^3 of tailings. The wet bulk density was determined by weighing the same sample of tailings after saturating the sample with deionized water.

3.2. Apparatus

Small Reactors (75 g charge): The samples were placed onto a 1.6-micron Whatman GF/A glass fiber filter (5.5 cm diameter) resting on a perforated plastic plate in the upper section of a two-stage filter unit or reactor (Figure 2). The internal diameter was 7.0 cm at the top of the unit and 5.7 cm at the bottom (the top of the perforated plate). On 15 February 2002 the reactors were filled. A plastic weighing boat was placed on a scale and 75 grams was weighed out and added to the top section of the small reactors. The bed depth and volume were approximately 1.7 cm and 47.5 cm^3, respectively.

Humidity Cells (1 kg charge): Cylindrical cells (I.D. = 10.2 cm, h = 19.0 cm) constructed of acrylic plastic were used in the experiment. A 0.64-cm high acrylic sleeve was glued into the bottom of the cell to support a 0.16-cm thick PVC plate, which was perforated with 0.32-cm holes spaced 0.48 cm on center. The plate was covered with a polypropylene filter (National Filter Media Corporation, Polymax B, Style 226-075-2) which supported the solids. The bottom was sealed with an acrylic plate with a 0.85-cm outlet port in the center and a removable cover, with a 0.85-cm vent hole, was place on top of the cell (Figure 3). Prior to filling, the cells were acid washed with 10% HNO_3 and then rinsed three times with deionized water. The cells were weighed and the weight was recorded on the cell.

On 15 February 2002 the cells were each filled with 1000 g tailings, which were added in increments of approximately 250 grams. A plastic pan was placed on a scale and four of the

250-gram splits were added. The weight was generally about 50 grams over the desired 1000-gram sample size. A plastic spoon was used to randomly remove the excess and add it to the remaining 250-gram split. The 1000-gram sample was then poured into the cell. The samples were very dry and some of the very fine dust-like particles were lost due to suction from the exhaust hood. The plastic pan and spoon were rinsed with deionized water after each reactor was filled. The cells with dry solids were weighed and the tare weight recorded. The tailings bed depth was approximately 7.5 cm.

3.3. Experimental Procedure

A total of six small reactors and two humidity cells were used (Table 1). The small reactors were used to allow direct correlation of data with those generated using these reactors to examine the dissolution of Duluth Complex rock samples. To examine the effects of moisture content on dissolution, three small reactors were covered and three were uncovered. Two covered and two uncovered reactors were used to analyze drainage quality at a fairly high frequency. The third reactor in each set was to be analyzed less frequently and was intended to provide leached tailings for analysis prior to termination of the other reactors. The duplicate humidity cells were intended to provide a comparison of the small reactors and the ASTM D5744-96 method that uses these cells and to investigate the effect of tailings bed depth on drainage quality.

The solids in all units were rinsed three times at week zero to remove oxidation products that accumulated from the time they were processed until the beginning of the experiment. Deionized water (200 mLs) was slowly dripped into the small reactor from a buret and was allowed to drain freely from the cell. The reactors were subsequently rinsed weekly with one 200 mL volume of deionized water. Reactors 1-3 remained uncovered, and covers were placed on reactors 4-6 immediately after the water addition to inhibit evaporation of pore fluid.

The rinsing procedure for the humidity cells was to place a rubber cap on the outlet port and add 500 mLs of deionized water to each cell. After the week zero rinse with three 500-mL volumes of deionized water, humidity cells samples were rinsed weekly with a single 500 mL volume. The water was slowly dripped into the cell, with the outlet capped, from a separatory funnel. Covers were placed onto the cells immediately after the water addition. The water remained in the cell for ten minutes after the last one was filled, and then the cells were uncapped and allowed to drain into a collection vessel.

Effluent (or drainage) samples were weighed and analyzed for pH and specific conductance at the MN DNR laboratory. From the remaining drainage, samples were collected and filtered for subsequent determinations of sulfate and metal concentrations. After filtration, samples to be analyzed for metals were preserved with 0.2 mLs Baker Instr-Analyzed nitric acid per 50 mL sample. Samples were rinsed three times at the initiation of the dissolution experiment (week 0). The first rinse and a volume-weighted composite of rinses two and three were analyzed for SO_4, Ca, Mg, Na, K, Cu, Ni, Co, and Zn at the Minnesota Department of Agriculture (MDA, St. Paul, MN). Data for the initial rinses in the drainage quality tables represent a volume weighted composite of rinse 1 and the composite results of rinses 2 and 3 (see Appendix 3, Tables A3.2, A3.3 for initial rinse data).

3.4. Reaction Conditions

Between rinses the samples were retained in the cells to oxidize. The cells were stored in a room (8.5 x 10.5 x 9.5 ft) in which temperature and humidity were controlled by commercially available equipment. Temperature was maintained by a thermostatically controlled heater

(Lakewood 750) and humidity was maintained by a Holmes HM 3500 humidifier and a Sears Kenmore dehumidifier, both of which were controlled by Honeywell humidistats. Over the 520-week period of record temperature and relative humidity were measured three to four times a week with a Taylor wet-bulb, dry-bulb hygrometer, and readings were used to determine average weekly values. Each apparatus sat in the laboratory for the first 24 hours after the rinse water was added each week, after which they were returned to the constant temperature room.

Water retention in the small reactors and large cells was determined by weighing. During the first seven weeks of the experiment the units were weighed four to five times a week (Appendix 2, Table A2.3). Subsequently, weights were determined immediately before and one day after rinsing.

3.5. Analyses

3.5.1. Solid Phase Analyses

Particle size distribution (Table 2) was determined by Lerch Brothers Inc. using method ASTM E 276-93 (ASTM 2000). Triplicate samples were analyzed for sulfur, sulfate (sulfide was determined by difference), and evolved carbon dioxide by ACTLABS in Tucson, AZ using ASTM E-1915-97 (ASTM 2000). A 10% hydrochloric acid solution was used to solubilize the carbonate minerals, and the carbonate present was quantified as the difference between total carbon in the initial sample and that in the residue. The remaining solid-phase constituents were determined by ACTLABS in Ancaster, ON (Tables 3 and 4). Whole rock constituents were determined using a lithium tetraborate fusion modified from ASTM E886-94 (ASTM 2000) and analysis by inductively coupled plasma-atomic emission spectroscopy (ICP-AES) using a Thermo Jarrell-Ash ENVIRO II ICP. Concentrations of Ag, Cd, Cu, Ni, Pb, Zn, and Bi were determined using a total digestion modified from Crock et al. (1983), with analysis by ICP-AES. Other trace elements were determined using instrumental neutron activation analysis (Hoffman 1992).

Peter McSwiggen (McSwiggen & Associates, Minneapolis, MN) determined mineral content and chemistries. Tailings grains were mounted into a one-inch epoxy mount, which was then polished and carbon coated. One hundred twenty-three grains were randomly selected for analysis. Each grain was analyzed using a JEOL 8600 Electron Probe Microanalyzer (15 kV accelerating voltage, 20 namp beam current, 5 second dwell time, 5 μm beam diameter). Seven hundred sixty five tailings grains (nonsulfides) and seven sulfide mineral grains were also examined using the microprobe to estimate grain size distribution of these particles.

Paste pH and neutralization potential using the Sobek method (Sobek et al. 1978) were determined at the MN DNR lab on the three unleached tailings samples, and these samples were also analyzed by Lerch Brothers for sulfur and CO_2 (Appendix 2). For paste pH, 20 mLs of deionized water was added to 10 g of solids in a beaker. The slurry was stirred twice during the course of an hour. The pH of the slurry was then measured using an Orion model 720A meter with a Ross combination electrode (model 8165).

For NP determinations, modifications of the standard Sobek et al. (1978) procedure were used. A water bath was used in the digestion step rather than heating the sample with a flame. Three 2.0 g samples were weighed and placed into three 250 mL flasks. Based on a fizz rating of "none", 20 mLs of 0.1N HCL was added to each flask. The flasks were placed into a hot water bath of 90-95 ^{o}C and heated until the reaction was nearly complete**,** indicated when no further gas evolution was visible and particles settled evenly over the bottom of the flask. Deionized water

was added to make a total volume of 125 mLs and the contents were boiled for 1 minute. The contents were covered and cooled to room temperature and then titrated with 0.1N NaOH to the endpoints of pH 7.0 and pH 8.3.

3.5.2. Drainage Quality Analyses

Water samples were analyzed for specific conductance, pH, and alkalinity at the MN DNR laboratory. Specific conductance was analyzed using a Myron L conductivity meter, and an Orion SA720 meter, equipped with a Ross combination pH electrode (8165), was used for pH analyses. Alkalinity (for pH $\geq$ 6.3) and acidity were determined using standard titration techniques for endpoints of 4.5 and 8.3, respectively (APHA et al. 1992). The remaining sample was filtered for metals and sulfate analyses.

Metals and sulfate samples were analyzed at MDA. Ca, Mg, Na, and K were determined with a Varian 400 SPECTRAA; inductively coupled plasma mass spectrometry (ICP-MS, Hewlett Packard HP4500 Series, model #G1820A) was used for the remaining metals analyses. Sulfate concentrations were determined using a Lachat QuickChem 8000 or, for $[SO_4] < 5$ mg/L, a Dionex ion chromatograph.

3.6. Calculations

Weekly chemical mass release for each solute was calculated as the product of drainage volume and solute concentration. Cumulative release was calculated as the sum of weekly release values. For weeks that solute concentrations were not determined, the concentration was estimated as the average of previous and subsequent measured values.

Rates of solute release were calculated for every week drainage quality samples were analyzed for a specified solute. For each solute, the weekly rate of release was calculated as the product of the volume and solute concentration divided by the rock mass originally present. The weekly release rates were averaged over annual (52 week) periods. Standard deviations were calculated to provide a measure of rate variability during the periods. Annual average sulfate release per grams of sulfur remaining was calculated by dividing the weekly sulfate release by the calculated grams of sulfur for each reactor and humidity cell. The grams of sulfur remaining were calculated using the cumulative sulfate concentration (converted to grams of sulfur) and subtracted from the initial sulfur present based on solid-phase analysis.

4. RESULTS

4.1. Introduction

Tailings generated by processing of anorthositic gabbro bulk sample (Duluth Complex, Partridge River intrusion, Babbitt prospect) were characterized and subjected to dissolution testing in the laboratory. Characterization consisted of density determinations, particle size analysis, bulk chemistry, mineral content and chemistry, acid base accounting, and paste pH. Laboratory dissolution tests were conducted on the tailings under each of three conditions: triplicate unsaturated (uncovered) reactors containing 75 g of tailings, triplicate saturated (covered) reactors containing 75 g tailings (Figure 2), and duplicate 1000 g tailings in a covered humidity cell (Figure 3). Drainages from the third 75-g unsaturated reactor and third saturated reactor were analyzed less frequently than those from the other reactors. They will be used for analysis of leached solids prior to termination of the first two reactors in each set (Table 1). Dissolution tests are discussed in terms of temporal variation of drainage quality (pH, alkalinity, SO_4, Ca,

Mg, Na, K, Cu, Ni, Co, Zn), rates of solute release, and total solute release. The period of record presented in this report is 520 weeks, but replicated units for all three dissolution conditions remain in progress.

4.2. Solid-Phase Analyses

The dry and wet bulk densities of the tailings were 1.65 and 2.04 g/cm^3, respectively (Appendix 2, Attachment A2.1). Water content at saturation was determined to be 0.24 g water/g tailings. Based on the water content at saturation and associated tailings volume (80 and 200 cm^3), the tailings porosity was calculated as 40 percent. Sieve analysis of the tailings indicated approximately 80 % were finer than 100 mesh, 29 % finer than 270 mesh, and 16 % finer than 500 mesh (Table 2).

Analysis of images generated by electron microprobe was also used to describe particle size distribution. Mineral grain cross-sectional area was used as a surrogate for mass and calculated circular diameter ($d_{circ} = (4A/\pi)^{0.5}$) was used for particle diameter. Results using this approach agreed well with the sieve analysis for the larger fractions, yielding 84 % finer than 100 mesh, 28 % finer than 270 mesh (Appendix 1, Attachment A1.4). The mass in the -500 mesh fraction was only 7.5 %, or about half the value determined by sieving.

The tailings contained 0.2% sulfur, at least 90% of which was present as sulfide (Table 3). The carbon dioxide content was less than 0.05%, indicating an NP(CO_2) less than 1.1 g $CaCO_3$ equivalent (kg tailing)$^{-1}$. The major whole rock components were SiO_2 (47.5%), Al_2O_3 (15.6%), and Fe_2O_3 (15.2%). Calcium and magnesium contents as oxides were 7.8% and 7.0% (Table 3). Copper, nickel, cobalt and zinc contents were determined as 510, 360, 59 and 120 ppm (Table 4).

The mineral contents and chemistries were 57% labradorite ($Ca_{0.57}Na_{0.43}Si_{2.42}O_8$), 11% augite ($Ca_{0.83}Mg_{0.77}Fe_{0.35}Si_{1.92}O_{6.00}$), 9.8% olivine ($Mg_{1.10}Fe_{0.89}Si_{0.99}O_{4.00}$), 9.8% quartz (anomalous, apparently contamination), and 5.7% biotite ($K_{1.78}(Mg_{3.27}Fe_{1.79})(Al_{2.39}Si_{5.55}O_{17})(OH)_{1.72}$). Ilmenite, potassium feldspar, and orthopyroxene were also reported present at near 1% (Appendix 1, Attachment A1.3).

Chalcopyrite (approximately $CuFeS_2$) was the only sulfide mineral detected during the microprobe analysis. The chemical analyses (assuming 0.2 %S^{2-}) in conjunction with the chalcopyrite chemistry indicated that the sulfur content associated with chalcopyrite was roughly 0.05%. The NiO content of the olivine was 0.15 % which, considering the bulk nickel content of 360 ppm and olivine content of 9.8 %, implies that the nickel content associated with sulfur was 239 ppm. This yields an estimated sulfur content associated with pentlandite of 0.02 %. It is assumed that the remaining 0.13 %S (0.2 %S_{bulk} – 0.05 %S($CuFeS_2$) – 0.02 %S (($Fe_{4.5}Ni_{4.5})S_8$) = 0.13% S) is largely associated with pyrrhotite. Microprobe analysis indicated that 87 % of the sulfide minerals were liberated and, therefore, readily available for oxidation.

The 0.2 %S content indicated an acid production potential (AP) of 6.2 g $CaCO_3$ equivalent (kg tailing)$^{-1}$, assuming all sulfur were associated with iron. Accounting for the sulfur associated with copper and nickel yields 0.16 %S associated with iron and a refined AP of 5.0 g $CaCO_3$ equivalent (kg tailing)$^{-1}$, Neutralization potentials (Sobek et al. 1978) determined in triplicate averaged 19 g $CaCO_3$ equivalent (kg tailing)$^{-1}$. Using a backtitration endpoint of 8.3 for two of the three samples (rather than the 7.0 endpoint used in the technique of Sobek et al. 1978) yielded an average NP of 14 g $CaCO_3$ equivalent (kg tailing)$^{-1}$ for duplicate samples (Appendix

2, Table A1.4). These values were considerably higher than the NP(CO_2) of less than 1.1 g $CaCO_3$ equivalent (kg tailing)$^{-1}$. The paste pH of the tailings was 8.29.

4.3. Reaction Conditions

4.3.1. Temperature and Relative Humidity

Over the 520-week period of record, average weekly temperature ranged from 20.3 to 27.9 ^{o}C. Temperature averaged 23.7 ^{o}C with a standard deviation of 1.4 ^{o}C. Relative humidity ranged from 42 % to 75 %, averaging 58 % with a standard deviation of 4.3 % (Appendix 2, Table A2.2).

4.3.2. Water Retention

Water retention was determined the morning after the reactors were rinsed, roughly a 24-hour period. Retention in the duplicate unsaturated reactors was initially in close agreement, but values began to diverge after about three years. The discrepancy decreased through year 8, when the volume retained in reactor 1 was consistently about 5 mL less than that in reactor 2 (Appendix 2, Figure A2.1). Subsequently replication improved markedly, and for the period of record water retention in Reactors 1 and 2 averaged 14.5 and 16.8 mL, respectively. The reasons for these variations are not understood. In contrast, replication was consistently good for the duplicated saturated reactors and humidity cells (Appendix 2, Figures A2.2, A2.3).

Retention in all reactors during the initial 40 weeks typically ranged from 20-25 mL, somewhat higher than the 18 mL inferred by the saturated moisture content determination (see previous section). Subsequently retention in the unsaturated reactors declined to a typical range of 10-15 mL after week 300. Retention in the saturated reactors declined to a lesser degree, with values after week 400 reporting in the typical range of 18 to 25 mL. For the entire period of record retention in the unsaturated reactors was about 7 mL less than that in the three saturated reactors (Appendix 2, Table A2.5). The lower retention in the unsaturated reactors was the result of evaporation between the rinse and measurement of water retention approximately 24 hours later; the covers on the remaining reactors effectively inhibited evaporation during this period (and throughout the weekly cycle). Water retention in the unsaturated reactors varied seasonally, with values tending to be greater during summer months, when less evaporation would be expected due to humid conditions in the laboratory (section 3.4). Retention in all reactors tended to decrease slightly over the period of record (Appendix 2, Figures A2.1, A2.2).

Water retention in the duplicate humidity cells was highly reproducible (Appendix 2, Table A2.5, Figure A2.3). The average retention in the duplicate cells over the period of record was 240.1 and 243.3 mL, in excellent agreement with the 24 % saturated moisture content (section 4.2). The measured retention was about 20 % lower than the average retention per unit mass in the smaller saturated cells. The average evaporation during the weekly cycle was about 5.5 mL for the larger cells, slightly higher than that for the saturated reactors (Appendix 2, Table A2.5).

4.3.3. Leachate Volume

The drainage volume from the duplicate unsaturated reactors reflected, at least qualitatively, differences in moisture retention. Replication was initially very good but, over time, volumes from Reactor 2 tended to be slightly lower than those from Reactor 1 (Appendix 2, Figure A2.4). The difference averaged about 3.4 mL over the period of record (Appendix 2, Table A2.7). The

unsaturated reactors, R1 and R2, averaged 169.3±2.87 mL and 165.9±3.76 mL respectively. The saturated reactors, R4 and R5, averaged 191.0±4.08 mL and 186.8±7.77 mL respectively (Appendix 2, Figures A2.5, A2.6).

4.4. Drainage Quality and Solute Release

4.4.1. 75-g Reactors

Drainage quality for both unsaturated and saturated reactors replicated reasonably well (Figures 4-5, Tables 5-6), but some discrepancies were apparent. For the unsaturated reactors, reactor 1 sulfate concentrations tended to be lower than those from reactor 2 in year 1 and this ordering was reversed in year 2. Subsequently agreement was good (Figures 4 and 5). Concentrations of calcium, magnesium, and sodium from reactor 1 tended to be lower than those from reactor 2 after year 1. The net effect of these differences led to 5th percentile annual pH values from Reactor 1 that tended to be 0.05 to 0.1 units lower than those from Reactor 2. Similar differences were observed for the saturated reactors (Figures 4 and 5).

The initial rinse at week zero was consisted of three 200-mL additions of deionized water as opposed to the single 200 mL volume in subsequent weeks. The initial rinse typically contained the highest concentrations of solutes. This reflected the removal of reaction products accumulated between the time of sample collection and the beginning of the experiment. Subsequent rinses reflected reaction products generated during the previous week.

Drainage pH from the reactors was initially near 7.9 and reached minimums in the range of 6.4 to 6.6. Drainage pH from the saturated reactors reached the minimum range between weeks 50 to 150, as opposed to weeks 400 and 450 for the unsaturated reactors (Figure 6). Alkalinity concentrations from all reactors were initially elevated, with concentrations as high as 25 – 30 mg/L as $CaCO_3$ in the first weeks. The saturated reactors yielded higher concentrations in the initial weeks and declined to levels below detection (5 mg/L as $CaCO_3$) after 40 weeks. The decline for the unsaturated reactors was slightly slower, reaching the detection limit after 48 weeks (Figure 6, Appendix 3, Tables A3.4 – A3.9).

Sulfate concentrations decreased rapidly initially and tended to plateau or decrease slightly after year 6 (Figure 7). Concentrations from the unsaturated reactors were higher than those from the saturated reactors during the first 10 weeks. However, sulfate release from the unsaturated reactors decreased more rapidly, yielding second year sulfate release rates that were roughly 20 to 35 percent of first year rates as opposed to a corresponding range of roughly 50 to 70 percent for the saturated reactors (Figure 7). Sulfate release rates for all reactors after year 6 were around five percent of those in year 1 (Table 5). The fraction of solid-phase sulfur depleted during the 10-year period was roughly 46 % for the unsaturated reactors and 66 % for the saturated reactors (Table 8). In all reactors about two-thirds of the sulfur depletion occurred during the first two years.

Molar concentrations of major cations initially decreased in the order Ca > Mg > K > Na and the ordering subsequently evolved to Mg > Ca > Na > K. Concentrations of calcium and potassium tended to parallel those of sulfate (Figure 7; Appendix 3, Tables A3.4 - A3.9). Calcium and potassium release rates in year 10 were, respectively, roughly 5 and 10 % of those in year 1 (Table 5). In contrast, magnesium and sodium concentrations were less variable over the period of record (Figure 7), with release rates at year 10 averaging 18 and 60%, respectively, of year 1 rates (Table 5). Total molar release of calcium was the highest, measuring about 1.15 times that

of magnesium and roughly 2.5 times that of sodium and potassium. Total release of sodium was around 1.1 times that of potassium (Table 7).

It should be noted that unsaturated reactors 1 and 2 displayed unusual behavior with respect to potassium release (Figure 7). Potassium concentrations decreased to the detection limit (0.1 mg/L) by week 140 and remained below the detection limit through week 250, at which point concentrations increased to approximately 0.4 mg/L. Concentration decreased to 0.2 mg/L by Year 10, but always remained above the detection limit. A subset of the samples after week 250 was sent back to Minnesota Department of Agriculture (MDA) for re-analyses, and the initial results were confirmed.

Further examination of the drainage quality revealed that the Mg/K ratio for the unsaturated reactors was approximately 2 after week 250, and this approximates the corresponding stoichiometric ratio for biotite present in the tailings (Mg/K = 3.27/1.78 = 1.84, Appendix 1, Attachment A1.3). Based on this analysis, it appeared that the potassium concentrations determined at MDA was erroneous for weeks 100-250. A subset of these samples was sent to Pace Analytical laboratory and they confirmed potassium concentrations below the detection limit (0.2 mg/L).

It should be noted that this behavior in potassium was isolated to these two unsaturated reactors in this experiment and did not occur in any other experimental results conducted in the MNDNR Hibbing laboratory during this time frame. This, in conjunction with the confirmatory analyses described above, suggests that the anomalous behavior of potassium could not be readily attributed to analytical error or experimental methods. Furthermore, there is no obvious geochemical explanation for the relatively sudden decrease in potassium concentrations to levels below detection with a subsequent increase to previous levels (Appendix 3, Figures A3.17-18). Perhaps the wet-dry cycling (variation from saturation to essentially dry on a weekly basis) in these reactors physically affected biotite present in the samples and, consequently, potassium release in some unusual fashion. However, the authors are presently unaware of any publications on biotite dissolution that might suggest such a possibility.

Nickel contributed the majority of trace metal release, followed by zinc. Cobalt concentrations from all reactors and copper concentrations from the saturated reactors were typically near or below detection limits (Figure 8). Trace metal release tended to increase with the subtle decreases in drainage pH. As mentioned above, periods of relatively low pH occurred from the saturated reactors during years 2 and 3 and from the unsaturated reactors during years 8-10. Although the extent of pH depression was not large, with minimum values near 6.5, trace metal concentrations generally responded to these subtle changes.

Nickel and other heavy metal concentrations increased substantially over relatively small pH changes. From year 1 to year 2, drainage pH from the saturated reactors decreased from about 7 to 6.6, and, consequently, the rate of nickel release from the saturated reactors in year 2 was about eight times that in year 1 (Figure 9, Table 6). Nickel release rates from the saturated reactors averaged 20 and 6 times those from the unsaturated reactors in years 2 and 3, respectively, in response to pH differences of roughly 0.1 to 0.5 units (Figure 9, Table 6). Nickel release from the saturated reactors decreased to levels typically near or below detection after year 4. Cobalt concentrations from the saturated reactors paralleled these trends (Figure 10) and were only above detection during the period of pH depression. Drainage pH from the unsaturated

reactors decreased from 7.1 to 6.7 after year 6, and nickel release rates increased correspondingly. Zinc release responded to the pH declined for the saturated reactors at year 2 and for the unsaturated reactors after year 6 (Table 6).

Copper concentrations from the saturated reactors were typically near or below detection limits. Copper release rates from the unsaturated reactors during years 8-10, when drainage pH from these reactors was lowest, tended to be higher than the corresponding average rates (Table 6). Total release of copper was less than 0.5 % of that present initially and, correspondingly, zinc was less than 4 %. The fractions of nickel and cobalt depletion were intermediate to these values (Table 7).

4.4.2. 1000-g Humidity Cells

Drainage quality from the duplicate 1000 g humidity cells was in good agreement (Figure 11; Appendix 3, Tables A3.10-A3.11). Drainage pH was initially near 8, slowly decreased below 7 near week 200 and then, with the exceptions of occasional excursions into the lower sevens, typically ranged between 6.8 and 7 (Figure 6). These values were generally 0.1 to 0.3 units higher than those observed from the small reactors. Similarly, alkalinities from the humidity cells were higher than those from reactors. Concentrations were 50 mg/L as $CaCO_3$ at week 1 and declined over time, falling below 20 mg/L as $CaCO_3$ at week 40, below 10 mg/L as $CaCO_3$ at week 115, and typically below detection (5 mg/L as $CaCO_3$) after week 240 (Figure 6; Appendix 3, Tables A3.10-A3.11).

The initial rinses contained the highest concentrations of solutes. Sulfate concentrations ranged from 20 – 40 mg/L through week 5, decreased below 20 mg/L at week 40, and stabilized near 10 mg/L after week 100 (Figure 11). Although sulfate release decreased over time, the extent was mild relative to that for the reactors. The sulfate release rate in year 1 was about 115 µmol•(kg rock•wk)$^{-1}$ in year 1, decreased through year 4, and then stabilized near 50 µmol•(kg rock•wk)$^{-1}$ (Table 5). Total release during the 10-year period represented about 53 % of that originally present. After year 3 the depletion was relatively constant, annually averaging about 4.2 % of that originally present (Table 8).

Major cation molar release rates from all units, reflecting rates of acid neutralization, initially decreased in the order Ca > Mg > K > Na and subsequently evolved to Mg > Ca > K > Na. Major cation concentrations followed two distinct temporal trends. Calcium and potassium concentrations followed a trend similar to that of sulfate concentrations, with markedly elevated values during the initial weeks, declining exponentially, and then becoming relatively stable (Figure 12). The dissolution time prior to stabilization, however, was considerably longer than that for sulfate concentrations, on the order of 250 to 300 weeks (Figure 12). Year 10 release rates for calcium and potassium were roughly 10 and 20 percent of the respective year 1 rates (Table 5). The calcium and potassium releases observed represented about 2 and 4 %, respectively, of the amounts originally present in the tailings.

Releases of magnesium and sodium were more stable over time than those of sulfate, calcium, and potassium. Magnesium concentrations were slightly elevated initially, decreased through roughly week 60, and then increased through about week 400 and plateaued (Figure 12). Sodium concentrations were slightly elevated initially, then declined and stabilized near 0.3 mg/L after about 40 weeks of dissolution. For both solutes, release rates in year 10 were near 70 % of those in year 1.

Of the trace metals, nickel release was typically the highest. The magnitudes of copper, cobalt and zinc release did not vary greatly. Nickel concentrations at week 1 were near 0.05 mg/L, and then remained around 0.01 mg/L through week 30. Concentrations were near or below the detection limit from about week 60 through 200. Concentrations tended to increase slowly at this time, coincident with pH declining below 7, reaching values in the approximate range of 0.006 to 0.008 mg/L after about week 350. Zinc concentrations were typically below detection but displayed occasional excursions as high as 0.007 mg/L. However, even the occasional excursions yield values typically below 0.004 mg/L (Figure 13). Copper concentrations were 0.006 – 0.009 mg/L at week 0 then decreased and remained below detection after week 6. Cobalt concentrations were typically below detection throughout the test. Release rates and total metal release generally reflected these trends, although it should be noted calculations were affected by concentrations reported below detection (Table 6).

5. DISCUSSION

5.1. Introduction

This section presents a brief description of the mineral dissolution reactions that occur during chemical weathering of Duluth Complex tailings. This summary is intended to provide a conceptual structure to frame the discussion of results. It should be noted that detailed analysis of mineral dissolution reactions, such as calculation of mineral dissolution rates, is beyond the scope of this report.

Duluth Complex tailings were characterized (bulk density, particle size distribution, chemistry, mineralogy, static tests) and subjected to laboratory dissolution testing. Although the dissolution tests remain in progress, the following discussion presents the first 10 years of data. The dissolution tests conducted to examine the influence of moisture content and tailings depth on reactions rather than to simulate field drainage quality. Laboratory dissolution tests were conducted on 75-g samples in unsaturated and saturated reactors (both in triplicate) and in duplicate 1000-g humidity cells. Measurements of bed volume and water retention after the rinse were consistent with respective determinations of tailings density and saturated water content.

The three reaction conditions produced two general similarities in drainage quality. First, drainage pH from all tests initially ranged from 7.8 to 8 and declined for 100 (saturated reactors) to 400 weeks, reaching minimums of 6.4 to 6.6. Second, solute concentrations were highest at the beginning of the experiment and tended to decline and plateau over time. Concentrations of sulfate, calcium, and magnesium from the thin beds (~1.5 cm deep) generally decreased and plateaued after about six to eight years of dissolution. Similar behavior was observed in the deeper beds (~ 7.5 cm deep) after five to six years of dissolution.

Repeatability for all replicates was good for the period of record but was variable over time. Replication was generally better for saturated reactors and humidity cells than for the unsaturated reactors. Drainage pH values for all replicate pairs were virtually always within 0.1 units. Although discrepancies in concentrations were observed over time, the difference from the mean total mass release of sulfate, calcium, magnesium, sodium, and potassium for duplicates was less than four percent in 80 % of the cases (Table 7). The three instances in which the percent difference from the mean exceeded four percent were for magnesium, sodium, and potassium mass release from the unsaturated reactors (6.2, 6.0, and 14 %, respectively).

The unsaturated and saturated reactors provided respective reaction environments in which pore water evaporated and remained near saturation. This allowed assessment of the effects of moisture content on geochemical reactions. The tailings in the humidity cells also remained near saturation, and comparison with the saturated reactors allowed assessment of the influence of bed depth on reactions. In addition to differences in reaction environment, the solid-phase characterization aided interpretation of drainage quality data and solute release rates.

5.2. Solid-Phase Characteristics

Approximately 80 percent of the tailings mass was finer than 100 mesh, 29 percent finer than 270 mesh, and 16 percent finer than 500 mesh. Analysis of electron microprobe images yielded similar results for the -100 and -270 mesh fractions. However, the -270 fraction was approximately partitioned as 20 % coarser than 500 mesh and 8 % finer than 500 mesh. Thus, the sieving yielded a larger percentage in the finest fraction. This is somewhat puzzling because it seems that sieving would be more likely to under represent the fines due to adherence of these particles to coarser grains. Nonetheless, it is noteworthy that the agreement between the sieving and image analysis for the -100 and -270 mesh fractions was quite good.

The tailings contained 0.2% sulfur, at least 90% of which was present as sulfide, and less than 0.05% carbon dioxide. Assuming all sulfur is present as iron sulfide yields an acid potential (AP) of 6.2 g $CaCO_3$ equivalent (kg tailing)$^{-1}$. Accounting for the sulfur associated with copper and nickel (see below) yields 0.16 %S associated with iron and a refined AP of 5.0 g $CaCO_3$ equivalent (kg tailing)$^{-1}$. The CO_2 content of the sample (<0.05 %) implies a maximum neutralization potential of 1.1 g $CaCO_3$ equivalent (kg tailing)$^{-1}$, assuming all CO_2 is associated with calcium or magnesium. This is considerably lower than the 19 and 14 g $CaCO_3$ equivalent (kg tailing)$^{-1}$ neutralization potential determined using variants of the method of Sobek et al. (1978). This suggests that silicate mineral dissolution contributed to the latter determinations.

Dissolution of the silicate mineral present plagioclase (57 wt. %) and lesser amounts of augite and olivine (both ~10 wt. %)) can neutralize acid after (and before) the trace amounts of carbonate minerals are depleted. The silicate mineral neutralization potential is kinetically limited and will not be effective in maintaining pH 6 drainage if the rate of acid production is too high (White et al. 1999, Lapakko 1994). The fine particle size of the tailings suggests that the minerals present largely occur as individual grains and are, therefore, exposed and available for reaction. That is, the fraction of minerals included within other minerals or interstitial to other minerals is small. This was generally supported by electron microprobe images, although multimineralic aggregates of very fine particles were observed (Appendix 1, Attachment A1.5). Although these aggregates were assumed to be artifacts of sample preparation, such aggregation might also occur under natural conditions.

Copper, nickel, cobalt and zinc contents were determined as 510, 360, 59 and 120 ppm. The chemical analyses (assuming 0.2 %S^{2-}) in conjunction with the chalcopyrite chemistry indicated that the sulfur content associated with chalcopyrite was roughly 0.05%. The NiO content of the olivine was 0.15 % which, considering the bulk nickel content of 360 ppm and olivine content of 9.8 %, implies that the nickel content associated with sulfur was 239 ppm. This yields an estimated sulfur content associated with pentlandite of 0.02 %. It is assumed that the remaining 0.13 %S (0.2 %S_{bulk} – 0.05 %S as ($CuFeS_2$) – 0.02 %S as (($Fe_{4.5}Ni_{4.5})S_8$) = 0.13% S) is largely associated with pyrrhotite. Microprobe analysis indicated that 87 % of the sulfide minerals were liberated and, therefore, readily available for oxidation. This estimate, however, was based on a

median sulfide mineral diameter of 53 μm determined by examination of seven sulfide grains. In contrast, the median silicate mineral diameter of 83 μm included examination of 765 grains.
The mass of tailings calculated based on bed dimensions in the reactors and humidity cells were in excellent agreement with the measured mass. The tailings mass calculated for the reactors was within four percent of the measured mass. The corresponding value for the humidity cells was less than one percent (Appendix 2, Attachment A2.2).

5.3. Effect of Moisture Retention on pH

The water retention anticipated based on moisture content measurements was somewhat lower than that in the reactors about 24 hours after rinsing but in good agreement with that correspondingly retained in the humidity cells. Subsequently, all but about 2 mL of pore water in the unsaturated reactors evaporated in three days, and essentially all pore water had evaporated by four days after rinsing. The average rate of drying in the unsaturated reactors between the three weekly weights performed was 6.2±2.3 mL (Appendix 2, Table A2.4). In contrast, water retained in the saturated reactors and humidity cells declined only slightly during the one-week cycle.

The effect of moisture content on reactions was assessed based on comparison of unsaturated and saturated reactors. It is assumed that the entire tailings mass present in the thin beds (~1.7 cm deep) was subjected to oxidizing conditions. That is, oxygen diffusion did not limit oxidation rates. Sulfate release from these reactors decreased over time, due largely to the depletion of sulfide minerals by oxidation.

Minimum pH values from thin beds in the reactors were in the neighborhood of 6.4-6.5. Drainage pH from the saturated reactors reached the minimum range between weeks 50 to 150, as opposed to weeks 400 and 450 for the unsaturated reactors. The more rapid pH decline for the saturated reactors is attributed to sulfate release rates during the first three years that were about 1.6 times those from the unsaturated reactors. Apparently the greater water retention in these units (section 4.3.2) was more conducive to sulfide oxidation.

The rate of acid production (as indicated by sulfate release) was higher under water-saturated conditions (covered reactors) than in an environment in which pore water was allowed to evaporate (uncovered reactors). The ratio of average annual rates from the saturated reactors to corresponding rates from the unsaturated reactors reached a maximum of 2.7 in year two and declined to 1.1 for years seven and eight (Table 5; Appendix 4, Figure A4.9). In subsequent years the sulfate release rates for the saturated reactors was roughly 80 to 90 % of that from the unsaturated reactors. However, it should be noted that the sulfur content of rock in the saturated reactors was less than two-thirds that in the unsaturated reactors, due to the more rapid oxidation during years one through eight (Table 8). Thus, normalizing based on mass sulfur present would yield higher rates for the saturated reactors in the final two years (Figure 14).

Lapakko and Berndt (2009) discussed the faster pyrrhotite oxidation under saturated conditions based on the initial 202 weeks of dissolution. During this period oxidation rates under saturated conditions averaged 1.7 times those under unsaturated condition. They recognized that the data did not allow confident presentation of a mechanistic foundation for the difference, but they did present a plausible explanation based on previous studies of pyrite oxidation. The following paragraphs are excerpted from Lapakko and Berndt (2009).

The pore water in the uncovered reactors evaporated over roughly 3.5 days and the covered reactors were near field saturation throughout the weekly oxidation cycle. The difference in sulfate release rates observed might be due to different mechanisms of pyrrhotite oxidation under conditions close to field saturation and in humid air. Lapakko and Antonson (2006) reported oxidation rates for pyrite in waste rock at near field saturation (with respect to water) were consistent with those presented by Williamson and Rimstidt (1994). However, rates presented by Jerz and Rimstidt (2004) appear more appropriate for reaction under the drier conditions observed in the uncovered reactors.

Recognizing that the mechanism of pyrrhotite oxidation might differ from that of pyrite, pyrite oxidation rates under the two test conditions were calculated to provide a comparison with observed sulfate release rates. For the uncovered reactors, conditions close to saturation were assumed for 3.5 days of the weekly cycle and sulfate release for this period was calculated using data from Williamson and Rimstidt (1994). Dry conditions (in moist air) were assumed for the remainder of the week and sulfate release was calculated using data from Jerz and Rimstidt (2004). The sulfate release calculated using Williamson and Rimstidt (1994) rates for the entire week was 1.7 times that using the Williamson and Rimstidt (1994) rates for 3.5 days and rates from Jerz and Rimstidt (2004) for 3.5 days. Thus, the ratio of uncovered: covered rates agrees well with this ratio. It is also possible, however, that rates were affected by differences in the activity of neutrophilic bacteria such as *Thiobacillus thioparus* (Gould and Kapoor, 2003). However, bacterial populations were not determined.

It is also interesting to note that sulfate release from the unsaturated reactors generally exceeded that from the saturated reactors during the first 10 weeks of dissolution (Figure 15). Subsequently, sulfate release rates (normalized for tailings mass) from the saturated reactors was greater through at least year 8 (Table 5; Appendix 4, Figure A4.9). It is conceivable that a ferric oxyhydroxide coating developed more rapidly on pyrrhotite in the unsaturated reaction environment. The lesser amount of water available for transporting reaction products from the mineral surface might accelerate this development. Analysis of sulfide coatings for leached samples from the two reaction environments will lend insight into this possibility.

The potential effect of drainage pH on sulfide mineral oxidation and the resultant sulfate release rate must be considered. For example, drainage pH from the saturated reactors was markedly lower than that from the unsaturated reactors during years two through five (Figure 6). It is conceivable that the lower pH during this period enhanced biological mediation of sulfide mineral oxidation. In particular the largest difference in drainage pH between unsaturated (unsaturated tailings) and saturated reactors (saturated tailings) occurred during year two (Figure 6). This is also the time when the ratio of sulfate release rate from the saturated tailings to that of the unsaturated tailings was highest (Table 5). However, drainage pH values from saturated and unsaturated tailings were similar after year five (week 260). Nonetheless, sulfate release from the saturated tailings remained higher during years six through eight, reflecting the dependence of sulfate release on water content. As noted above, even in years nine and ten the release from the saturated tailings was higher when normalized for sulfur mass present (Figure 14, Table 9).

5.4. Acid Neutralization

Acid neutralization occurred by silicate mineral dissolution throughout the experiment and was apparently augmented by dissolution of trace amounts of calcite in the initial phase of the experiment. Although the CO_2 content of the tailings was reported as less than 0.05 %, dissolution of trace amounts of carbonate mineral likely contributed to elevated calcium and alkalinity release at the beginning of the experiment.

It should be noted that the trace carbonate mineral present, assumed to be calcite, was depleted more rapidly from the saturated reactors. This is evidenced by the more rapid release of both alkalinity and calcium under saturated conditions (Figure 15). It should be further noted that the sulfate release from the saturated reactors during this period was slightly less than that from the unsaturated reactors. Thus the more rapid calcite dissolution was apparently the result of the saturated conditions rather than accelerated acid production resulting from sulfide mineral oxidation. A similar observation for silicate minerals was reported by Lapakko and White (2000).

The cumulative calcium release during the first 40 to 50 weeks was roughly 1.2 and 1.4 millimoles under unsaturated and saturated conditions, respectively (Figure 16). The maximum calcite content inferred from the reported CO_2 content of the tailings ($CO_2 < 0.05$ %) is 0.85 millimoles. This indicates that some of the calcium release during this period was from silicate minerals such as plagioclase and augite.

Once calcite was depleted, acid generated resulting from iron sulfide oxidation was neutralized by dissolution of silicate minerals, including labradorite. In the saturated reactors, calcium release was significantly higher than sodium release until approximately week 104, at which point the ratio of calcium to sodium approached the stoichiometric ratio of plagioclase in the solids (1.425) and remained constant (see section 4.2; Tables 5, 10; Appendix 4, Figures A4.20, A4.21). Future analysis should examine rates of silicate mineral dissolution (i.e. labradorite, augite, olivine, biotite) to quantify the neutralization rates by dissolution of the various components.

5.5. Effect of pH on trace metal release

A major consequence of the accelerated sulfide mineral oxidation and consequent pH depression was a large increase in nickel and cobalt concentrations. As pH decreased from 6.8 to 6.4 in the saturated reactors, nickel and cobalt concentrations increased by 35 and 8 times respectively. Increases in nickel and zinc release were also observed, to a lesser extent, in the unsaturated reactors when pH decreased to 6.4 by week 400. Release rates increased 2-3 times as a result of this small decrease in pH.

5.6. Comparison of tailings and waste rock release rates

A sample of 0.2% S waste rock from the Babbitt Prospect of the Partridge River Intrusion was subjected to dissolution under the same conditions to which the unsaturated tailings were exposed. The rock particles ranged from 0.053 to 0.149 mm in diameter and were coarser than the tailings, for which 29 percent were finer than 0.053 mm (Table 2). The plagioclase contents

of the waste rock and tailings were similar (50 and 57 percent, respectively), as were the calcium contents of the plagioclase present (Appendix 1, Table A1.7). Augite and biotite contents of the waste rock were reported to be slightly lower than those of the tailings (11 vs. 7.8% and 5.7 vs. 2.3%, respectively). In contrast, the olivine content of the waste rock was considerably higher than that in the tailings (26 vs. 9%). Respective concentrations of copper, nickel, and cobalt in the waste rock were 2.0, 1.2, and 1.2 times those in the tailings (Appendix 1, Table A1.6).

The pH of drainage from the waste rock sample was typically 0.8 to 1 unit lower than that of the tailings, with respective minimum values of 5.7 and 6.5. Both higher sulfide mineral oxidation rates and lower silicate mineral dissolution rates contributed to the lower pH values from the waste rock. The rates of sulfate release from unsaturated tailings reactors averaged about 60 μmol (kg rock·wk)$^{-1}$ over the first eight years of the experiment (Table 5), 80 percent of the 75 μmol (kg rock·wk)$^{-1}$ reported for the 0.20 %S waste rock over the same period (Lapakko et al. 2013). Sulfate release rates for the tailings and rock were very close in year 1 (260 vs 240 μmol (kg rock·wk)$^{-1}$) and year 8 (16 vs 15 μmol (kg rock·wk)$^{-1}$). Rates for the rock tended to be somewhat higher in the intermediate years.

Sulfate release rates area influenced by the sulfide mineral surface area exposed, the sulfide mineral chemistry, and the availability of oxygen. Considering the finer particle size of the tailings, and associated higher specific surface area, it would be expected that they would exhibit higher sulfate release rates. The lower observed sulfate release rates observed for the tailings are apparently the result of removal of "liberated" sulfide minerals during mineral processing. Consequently, relative to unprocessed waste rock, a greater fraction of sulfide minerals present in the tailings are included within other rock particles and, therefore, less available for reaction. This supported by mineralogical analyses that reported respective liberation probabilities of 100 and 87 percent for the rock and tailings (Appendix 1, Attachment A1.5).

It should be noted that there were also sulfide mineral content differences between the two samples. Respective copper contents of 0.1 and 0.051 percent for the waste rock and tailings were used to estimate the fraction of sulfur present with chalcopyrite. Assuming all copper was present as chalcopyrite indicated an estimated 50 percent of the sulfur in the waste rock and 25 percent of the sulfur in the tailings was associated with chalcopyrite. This infers that the pyrrhotite content of the tailings was about 1.5 times that of the waste rock. Pyrrhotite is reported to oxidize more rapidly than chalcopyrite, so this would tend to accelerate sulfate release from the tailings relative to the waste rock.

Finally, the finer grained tailings might have limited oxygen diffusion to sulfide mineral surfaces and thereby inhibited sulfide mineral oxidation. Both the finer grain size itself and increased water retention with finer grain size would tend to reduce diffusion. However the influence of oxygen diffusion might not be substantial due to the thin tailings beds used in the experiment.

Average rates of calcium and magnesium release from the tailings were on the order of twice those from the rock. This was apparently largely due to the finer size of the tailings, resulting in a greater surface area of silicate minerals. Plagioclase was the major calcium-bearing mineral in both the waste rock and tailings. There was little difference between the waste rock and tailings in plagioclase contents (50 vs. 57%) or the atom fractions of calcium in the plagioclase (0.54 vs. 0.57; Appendix 1, Table A1.7). Thus, mineralogical differences between the samples were small and any associated influences on calcium release would likely be minimal.

There were differences in magnesium-bearing contents of the waste rock and tailings. Relative to the tailings, the waste rock had less augite and biotite (7.8 vs. 11%, 2.3 vs. 5.7%, respectively) and more olivine (26% vs. 9.8%). The rates of calcium and magnesium release from the tailings

were roughly twice the corresponding rates from the waste rock. The difference in calcium release rates was attributed to differences in particle size, given the similarity in plagioclase contents and chemistries between the two samples. Particle size, and its associated influence on silicate mineral surface area, would be expected to have a similar effect on magnesium release. If this is so, mineralogical differences between the tailings and rock had little influence on magnesium release. This further suggests, given that the olivine content of the waste rock was more than 2.5 times that of the tailings, that magnesium release was dominated by dissolution of augite and biotite rather than olivine.

Respective annual rates of copper and nickel from the waste rock, on average, were 40 and 150 times greater than those from the tailings. Similarly annual rates of zinc and cobalt release from the waste rock averaged 4 and 8 times those from than the tailings, respectively. These rates tended to be highest during years 2 and 3. The differences are attributed to pH values from the waste rock that was near 1 unit lower than those from the tailings. Concentrations of copper, nickel, and cobalt in the waste rock were 2.0, 1.2, and 1.2 times those in the tailings, respectively (Appendix 1, Table A1.6). These differences probably contributed to a lesser degree.

5.7. Sulfide oxidation as function of tailings bed depth

In the deeper tailings beds in humidity cells, sulfide oxidation was limited by oxygen diffusion through the water saturated tailings. Consequently, the humidity cell tailings bed consisted of, in a simplified sense, an upper zone of oxidation and a lower zone that was essentially devoid of oxygen. Consequently, sulfide oxidation was limited to the upper zone of the bed. Preliminary calculations indicated that roughly one third of the tailings, representing an approximate depth of 2.5 cm, were in the oxidation zone in the initial stage of the experiment. Sulfate release rates from the humidity cells were relatively constant after year 2, reflecting the progression of the oxidation zone downwards into zones of unoxidized tailings. Thus, sulfide oxidation in the humidity cells during the 10-year experiment was limited by oxygen diffusion rather than the amount of reactant (sulfide minerals) present. Drainage pH levels from the deeper beds (~7.5 cm deep) were about 0.3 units higher in all cases. The experiments indicated both water content of thin tailings beds and tailings bed depth affected the quality of drainage from the tailings.

In the lower half of the humidity cells, acid neutralization reactions may have occurred. Release rates of calcium decreased significantly in the first 200 weeks and began to level off at the stoichiometric ratio of plagioclase in the solids by week 360 (Appendix 4, Figures A4.22, A4.23). Rates of potassium release also decreased significantly over the first 200 weeks and began to level off in a similar manner as calcium. Magnesium release rates reached a minimum by 100 weeks, but subsequently have increased in concentration by a factor of two.

Trends in trace metal release rates were much more erratic in the humidity cells. Nickel release rates tended to stay near the detection limit until pH went below 7 after 200 weeks, at which point nickel release approximately doubled. The majority of drainage quality for copper, cobalt, and zinc were at or below the detection limit and too sporadic, therefore no clear trend was observed over the period of record.

5.8. Comparison of saturated reactors and humidity cells

Rates of cation release from the saturated reactors were generally higher than those from the humidity cells. For example, rates of sodium and potassium release from the saturated reactors were typically about 3 and 1.5 times, respectively, those from humidity cells (Figure 12, Table 5). Cation release from the humidity cells was, however, higher in two cases. First, calcium release rates from the humidity cells in years 2 and 3 were 1.3 to 1.5 times that from the saturated reactors. This occurred despite the fact that sulfate release rates from the humidity cells were 0.3 to 0.7 times those from the saturated reactors during this period. The elevated calcium release from the humidity cells is attributed to calcite dissolution and this is supported by alkalinity data. Alkalinity from the saturated reactors decreased below detection after week 40, while concentrations in drainage from the humidity cells remained above detection until at least week 162 (Appendix 3, Tables A3.4-A3.11). The low alkalinity release from the saturated reactors after week 40 suggests that calcite content had been substantially reduced, perhaps near depletion. This is consistent with the elevated calcium release during year 1 (Table 5), when calcium release per gram tailing from the reactors was almost 2.5 times that from the humidity cells (see discussion on calcite depletion in section 5.4). The calcite dissolution from the humidity cells during years 2 and 3 was not driven entirely by sulfide mineral oxidation. This is supported by humidity cell sulfate release rates during years 2 and 3 that were 0.4 and 0.7 times, respectively, corresponding rates from the saturated reactors (Table 5). That is, some of the calcium release was due to calcite solubility.

Second, magnesium release rates from the humidity cells roughly equaled that from the saturated reactors in years 6 and 7 and clearly exceeded the latter rates in years 8-10. This was possibly influenced by higher sulfate release from the humidity cells during this period (Table 5).

A charge balance was calculated to determine the acid neutralization potential of the cation release. The annual average cation-anion ratio for the unsaturated and saturated reactors was 1.01±0.43 and 0.81±0.18, respectively, whereas the ratio for the humidity cells was 1.01±0.1 (Table 11). These differences between cations and anions can be attributed to the resolution in measured alkalinity. For the reactors, alkalinity reached values below the detection limit by the first year. Since half the detection limit value was used in the charge balance for alkalinity (0.05 meq/L), the difference between cations–anions should be ± 0.05, which it is in most cases. Better analytical resolution of alkalinity would allow for a more in-depth examination of the charge balance in the future. Alternatively, alkalinity could be calculated as the difference between measured cation and anion concentrations

Trace metal release rates from the saturated reactors were almost always higher than the humidity cells. The one exception being nickel release in the humidity cells was 10% higher during years 9 and 10, when pH levels were close to one another. Copper release on average was typically five times higher in the saturated reactors. Release of nickel, zinc, and cobalt were 25, 19, and 7 times higher in the reactors, respectively. Differences in trace metal release were highest between the saturated reactors and the humidity cells during year 2, when pH decreased from pH 6.8 to 6.4 in the saturated reactors (Figure 13).

6. SUMMARY

In the present project laboratory dissolution tests were conducted on tailings produced by processing of Duluth Complex rock from the Babbitt prospect (also known as AMAX and Mesaba prospect) of the Partridge River Intrusion for recovery of base metals. Reaction of these tailings affects water quality, and the effects of 1) tailings water content and 2) bed depth on reactions were examined. Of particular interest were reactions affecting pH and release of copper, nickel, cobalt, and zinc. This report describes the characterization of tailings (bulk density, particle size distribution, chemistry, mineralogy, static tests) and 10 years of laboratory dissolution testing. Dissolution tests were conducted on 75-g samples in unsaturated (uncovered) and saturated (covered) reactors (both in triplicate) and in duplicate 1000-g humidity cells.

The dry bulk density of the tailings was determined as 1.65 g cm-3. Approximately 80 percent were finer than 100 mesh, 29 percent finer than 270 mesh, and 16 percent finer than 500 mesh. The tailings contained 0.2% sulfur, at least 90% of which was present as sulfide, and less than 0.05% carbon dioxide. The major whole rock components were SiO_2 (47.5%), Al_2O_3 (15.6%), and Fe_2O_3 (15.2%). Calcium and magnesium contents as oxides were 7.8% and 7.0%. Copper, nickel, cobalt and zinc contents were determined as 510, 360, 59 and 120 ppm. Plagioclase (~An60Ab40) was the dominant mineral (57%), with augite, olivine, and quartz each comprising roughly 10 percent.

The 0.2% S content indicated an acid production potential of 6.2 g $CaCO_3$ equivalent (kg tailing)$^{-1}$. A digestion modified from Sobek et al. (1978) and backtitration endpoints of 7.0 and 8.3 yielded respective neutralization potentials (NP) of 19 and 14 g $CaCO_3$ equivalent (kg tailing)$^{-1}$, considerably higher than the NP(CO_2), which was determined as less than 1.1 g $CaCO_3$ equivalent (kg tailing)$^{-1}$. The paste pH of the tailings was 8.29.

Drainage quality from all replicates was in good agreement and differences were observed among the three methods. Drainage pH from all units was initially near 8 and tended to decline subsequently. Minimum pH values for the reactors were near 6.5 as opposed to roughly 6.8 for the humidity cells. Drainage pH from the saturated reactors declined most rapidly, reaching minimum values after about 50 weeks, as opposed to 400 weeks for the unsaturated reactors and humidity cells.

The comparatively low pH for the saturated reactors persisted from roughly week 50 to week 150. The pH depression was due largely to sulfide mineral oxidation rates in the saturated reactors that were 2.7 and 1.7 times those from unsaturated reactors during years two and three, respectively. The increased rates were apparently influenced by greater water retention in the saturated cells (i.e. less evaporation). Sulfate release from all reactors decreased over time and the magnitude of decrease declined over time. This suggested that sulfide oxidation was a first order reaction with respect to solid-phase sulfur content and oxygen transport was not rate limiting. In contrast, sulfate release rates from the humidity cells was virtually constant after year 2, suggesting that oxygen transport limited the rate of sulfide mineral oxidation. Major cation molar release rates from all units, reflecting rates of acid neutralization, initially decreased in the order Ca > Mg > K > Na and subsequently evolved to Mg > Ca > Na ≥ K.

Trace metal concentrations in drainage from the reactors exhibited a dependence on the minor variations of pH within the circumneutral region. Concurrent with the relatively small pH decrease in the saturated reactors at week 50, nickel concentrations increased to near 0.1 mg L^{-1}, roughly 25 times those in the unsaturated reactors. Similar but less pronounced trends were observed for cobalt and zinc. Nickel, copper, and zinc concentrations also increased as drainage pH from the unsaturated reactors declined slightly after about five years of reaction. Slight increases in nickel concentrations were also observed as pH of drainage from the humidity cells decreased in the upper sixes after year 4.

Key results are as follows.

1. Under all test conditions the tailings produced circumneutral pH, with the minimum of all values near 6.4.
2. During the first three years of testing, sulfide oxidation in saturated (non-evaporating) reactors was more rapid than that in unsaturated (evaporating) reactors, apparently due to the greater water retention.
3. The aforementioned accelerated oxidation yielded slightly lower drainage pH values in the saturated reactors.
4. The slightly lower pH values produced nickel concentrations in drainage from the saturated reactors that were roughly 25 times those in drainage from the unsaturated reactors.
5. Sulfide mineral oxidation rates (normalized for tailings mass) for the humidity cells were limited by oxygen diffusion, resulting in drainage pH values higher and trace metal concentrations lower than those for the smaller reactors.

These results provide valuable insight for assessing the geochemical behavior of tailings during environmental review. They also provide an empirical benchmark for comparison with results of future dissolution tests on fairly low-sulfur Duluth Complex tailings.

7. ADDITIONAL WORK

The present studies should be continued to provide a long-term empirical data base for informing environmental review of proposed copper-nickel mining operations. Analysis of the present data should be continued to estimate dissolution rates of individual minerals present in the tailings. This is particularly important for the silicate minerals present because it would allow better prediction of the long-term acid-neutralizing potential of the solids. Sulfide mineral oxidation in tailings beds should also be modeled as a function of moisture content and tailings depth, using the experimental data for calibration. Analysis of leached solids and geochemical equilibrium modeling should be conducted to examine possible phases controlling trace metal concentrations.

8. ACKNOWLEDGMENTS

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Table 1. Description and numbering of reactors and humidity cells.

Reactor / Cell Number	Mass of Tailing (g)	Reactor / Cell Description
1	75	Uncovered small reactor[1]
2	75	Uncovered small reactor (duplicate)[1]
3	75	Uncovered small reactor (sampled on a limited basis)[2]
4	75	Covered small reactor[1]
5	75	Covered small reactor (duplicate)[1]
6	75	Covered small reactor (sampled on a limited basis)[2]
7	1000	Humidity cell[3]
8	1000	Humidity cell (duplicate)[3]

1 - Duplicate covered and uncovered small reactors designed to monitor drainage quality and the effect of different moisture conditions. These reactors will also enable us to tie into data from previous work with the Duluth Complex.

2 - Small reactors setup to analyze leached tailing at some point in time during the experiment. These reactors are sampled on a limited basis.

3 - Duplicate humidity cells to serve as a comparison using the DNR version of the ASTM method.

Table 2. Particle size distribution. Analysis by Lerch Brothers, Inc..

MESH SIZE	PERCENT
+100	19.70
-100 / +270	51.50
-270 / +500	12.85
-500	15.95

Table 3. Whole rock chemistry (percent). Analysis by ACTLABS, Inc.

Parameter	Duplicate 1	Duplicate 2	Duplicate 3
S	0.20	0.20	0.20
S^{2-} [1]	0.18	0.18	0.18
SO_4^{2-} as S	<0.016	0.020	<0.016
CO_2	<0.05	<0.05	<0.05
$A1_2O_3$	15.64	15.52	15.68
CaO	7.77	7.75	7.79
Fe_2O_3	15.22	15.84	15.60
K_2O	0.77	0.87	0.83
MgO	6.97	7.35	7.20
MnO	0.163	0.172	0.168
Na_2O	2.48	2.48	2.49
P_2O_5	0.18	0.17	0.18
SiO_2	47.51	48.10	48.28
TiO_2	1.521	1.672	1.591
LOI	0.67	0.48	0.66
TOTAL[2]	98.88	100.31	100.48

[1]Determined by difference.
[2]For parameters from Al_2O_3 through LOI2.

Table 4. Trace metal chemistry for concentrations >50 ppm. Analysis by ACTLABS, Inc.

Parameter	Duplicate 1	Duplicate 2	Duplicate 3
Ba	215	213	218
Co	59	61	64
Cr	152	160	162
Cu	511	516	508
Ni	355	363	366
Pb	106	81	79
Sr	264	261	264
V	137	148	146
Zn	116	114	115
Zr	83	109	99

Note: See Table A1.3 for complete trace metal analyses

Table 5. Page 1 of 3. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in umol*(kg rock*week)$^{-1}$. Period of record is 6 - 520 weeks for R1, R2, R4, R5, HC7, HC8. Period of record is 6 – 232 weeks for R3 and R6. During the first year, 11-16 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 6-8 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 1 75 grams -100/+270 unsaturated	1	7.04	294	135	313	129	114	20.5	22.5	7.27	120	50.2
	2	6.87	62.4	15.3	76.4	15.2	80.1	9.5	14.5	2.11	35.5	17.1
	3	6.72	45.4	10.5	42.4	7.07	60.8	10.8	17.1	5.03	16.9	20.2
	4	6.78	31	3.99	26.9	3.93	44.8	4.98	13.4	1.68	2.85*	0.023
	5	6.88	25.1	2.58	24.1	3.03	40.7	3.34	13.8	2.1	6.85	10.5
	6	6.65	19.5	2.71	19.9	1.98	37.4	5.14	13.4	1.44	11.9	8.07
	7	6.59	16.6	2.16	18.9	3.12	30.9	2.44	15.3	3	21.4	5.49
	8	6.50	15.3	1.82	19.1	2.96	28.5	3.08	17.1	3.18	18.7	2.97
	9	6.41	13.9	5.68	15.3	2.36	18.1	9.68	17.5	2.07	12.9	8.05
	10	6.64	13.9	4.12	13.8	5.22	20.8	4.04	12.8	7.69	12	3.22
	Total[2]	6.52	53.7	86	57	91.9	47.6	29.8	15.7	2.94	25.9	34.3
Reactor 2 75 grams -100/+270 unsaturated	1	7.20	232	77.4	300	109	107	14.3	22	7.49	84.8	38.4
	2	6.92	81.4	19.9	84.1	20	93.8	14.4	16	2.74	13.3	6.08
	3	6.84	48.4	10.6	45.8	5.49	67.6	8.39	19	4.32	2.79*	0.0255
	4	6.88	33.4	4.79	32.7	4.93	56.2	5.33	16	1.57	2.77*	0.0267
	5	6.88	23.9	2.26	29	2.14	48.8	3.37	16.5	2.35	5.06	4.07
	6	6.74	19	2.75	23.2	2.84	40.4	3.68	15.5	1.98	21.8	1.9
	7	6.71	16.9	3.42	24.6	3.22	38.6	4.74	21.2	5.11	18	2.54
	8	6.57	16.7	1.28	23.2	2.51	33.5	2.55	17.4	1.62	14.2	0.838
	9	6.45	22.4	10.1	21.9	2.63	30.8	3.34	18.7	2.49	11.7	1.51
	10	6.64	16.3	2.55	19.9	5.19	28.2	3.43	16.2	3.91	9.64	1.34
	Total[2]	6.60	51	66.6	60.4	86.3	54.4	27.1	17.9	2.31	18.4	24.1
Reactor 3 75 grams -100/+270 unsaturated	1	7.24	240	68.4	290	70.7	106	11.2	21.1	2.92	98.5	20
	2	6.99	89.9	28.1	96.5	27.8	98.8	13	14	1.71	16.2	8.64
	3	6.81	55.8	6.17	53.1	4.23	78.6	12.5	21.7	3.6	4.6	3.03
	4	6.90	34.1	6.25	32	4.69	53.6	6.28	15.5	2.63	2.84*	0.0247
	5	7.00	26.4	2.39	29.7	1.39	51	1.9	16.6	2.74	3.97	4.29
	6	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	7	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	8	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	9	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	10	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	Total[2]	6.93	89.3	87.8	100	110	77.7	25.3	17.8	3.42	25.2	41.3

[1] 5th percentile value

[2] Average and standard deviation for annual average release rates

* Majority of concentrations during this time interval were below detection limit. One half the detection limit was used to calculate the annual average release rate.

Table 5. Page 2 of 3. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in umol*(kg rock*week)$^{-1}$. Period of record is 6 - 520 weeks for R1, R2, R4, R5, HC7, HC8. Period of record is 6 – 232 weeks for R3 and R6. During the first year, 11-16 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 6-8 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 4 75 grams -100/+270 saturated	1	6.81	371	161	385	192	158	29.8	27.7	4.59	91.5	38.7
	2	6.48	180	61.6	75.7	19.6	105	30.2	33.3	4.72	41.3	11.2
	3	6.59	74.7	16.3	39.6	4.64	51.1	5.14	26.8	5.48	20.2	5.2
	4	6.60	66.5	15.5	32.3	4.12	49	6.64	21.2	2.31	19.6	2.23
	5	6.82	38.3	3.87	27	2.62	39.4	2.08	21.6	4.17	17.3	3.26
	6	6.72	25.9	3.83	20.5	1.97	32.9	2.02	17.7	4.6	12.6	1.76
	7	6.80	18.9	1.25	21.6	2.54	33.3	2.72	19.5	4.6	14.2	2.43
	8	6.61	18.1	1.93	22.6	2.3	30.6	2.12	21.6	3.25	13.5	0.757
	9	6.65	12.1	4.74	18.4	3.91	23.2	4.99	20.3	2.41	10.4	1.19
	10	6.67	13.2	4.95	18.7	5.27	23.8	2.6	18.7	6.11	9.86	1.59
	Total[2]	6.59	81.9	114	66.1	113	54.7	43.5	22.8	4.9	25	25
Reactor 5 75 grams -100/+270 saturated	1	6.79	311	108	372	160	163	31.1	32.3	5.47	85.4	31.4
	2	6.41	212	78.3	84.9	25.9	115	37.8	38.1	5.02	38.3	11.1
	3	6.55	89.2	16.4	41.4	3.06	54.3	6.69	29.8	5.17	24.5	9.32
	4	6.52	54.2	11.2	29.2	4.1	41.2	3.79	22.2	2.61	17.1	1.78
	5	6.78	33.9	4.88	25	2.31	36.2	1.74	20.3	3.04	16.9	5.38
	6	6.64	24.1	3.43	18.5	1.93	29.6	3	17.1	1.61	12.4	1.14
	7	6.79	19.2	1.17	21.7	3.26	31.3	3.08	21.3	6.51	13.9	3.22
	8	6.67	18.4	2.77	22.4	2.2	28.9	1.49	20.7	2.1	13.9	1.29
	9	6.51	15.2	12.7	17.8	2.08	20.1	3.61	20.8	2.35	9.4	1.29
	10	6.59	15.8	2.88	16.1	4.47	22.2	3.71	19.2	8.9	8.84	1.01
	Total[2]	6.50	79.3	101	64.9	110	54.2	47.1	24.2	6.8	24.1	23.2
Reactor 6 75 grams -100/+270 saturated	1	6.86	306	135	358	149	157	21.4	26.6	3.93	80.5	21.4
	2	6.55	187	71.8	85.7	23.7	113	27	34.3	3.4	39.8	7.86
	3	6.49	108	10	52.8	9	69.8	5.53	34.1	2.74	30	8.57
	4	6.44	62.9	22.9	29	6.57	41.1	7.06	28.3	9.48	16.6	3.09
	5	6.76	34.4	4.29	24.3	1.04	34.2	1.07	25.4	6.23	15.4	3.47
	6	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	7	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	8	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	9	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	10	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	Total[2]	6.51	140	109	110	141	83	51.7	29.7	4.18	36.5	26.6

[1] 5th percentile value

[2] Average and standard deviation for annual average release rates

* Majority of concentrations during this time interval were below detection limit. One half the detection limit was used to calculate the annual average release rate.

Table 5. Page 3 of 3. Minimum pH and average annual rates of release for sulfate, calcium, magnesium, sodium, and potassium in umol*(kg rock*week)$^{-1}$. Period of record is 6 - 520 weeks for R1, R2, R4, R5, HC7, HC8. Period of record is 6 – 232 weeks for R3 and R6. During the first year, 11-16 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 6-8 per year.

ID	Year	min pH[1]	Sulfate		Calcium		Magnesium		Sodium		Potassium	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 7 1000 grams -1/4 inch	1	7.30	115	25.5	157	28.6	47.8	22	10.3	3.44	35.2	9.52
	2	7.38	71.8	13.3	103	18.6	17.9	1.16	6.81	0.691	23.2	4.36
	3	7.15	55.6	6.61	59.8	12.6	25.9	2.84	6.71	1.21	16.1	3.32
	4	6.87	49.2	4.59	34.6	4.11	30.2	2.56	6.63	0.503	12.8	1.22
	5	6.88	49.5	5.63	26.5	2.64	34.8	3.11	7.13	0.998	11.7	1.43
	6	6.82	51.8	5.69	19.2	1.74	35.1	3.72	6.27	0.427	9.4	0.751
	7	6.79	49.6	3.43	16.1	1.13	35.1	3.47	7.24	1.47	9.37	0.479
	8	6.74	51.6	3.72	17.2	0.999	40.3	3.25	7.89	0.705	10.5	2.51
	9	6.69	51	2.81	15.9	1.87	39.6	3.37	8.03	0.491	8.8	0.765
	10	6.82	50.9	5.23	14.5	2.69	38.9	4.93	7.18	1.57	8.57	1.52
	Total[2]	6.78	59.7	20.7	46.4	47.9	34.6	8.34	7.42	1.15	14.6	8.53
Humidity cell 8 1000 grams -1/4 inch	1	7.48	116	23.5	154	23.5	45.6	21.7	11.6	3.89	40.4	7.56
	2	7.39	76.5	13	109	16.5	16.2	1.01	6.79	0.635	23	3.7
	3	7.24	61.8	6.5	64.9	14.6	26.2	3.47	6.78	0.727	15	1.41
	4	6.97	53.2	5.97	35.3	5.82	30.7	2.47	6.71	0.703	12.9	1.66
	5	6.87	48.4	5.94	25.1	3.59	32	2.71	7.04	0.763	11.4	1.46
	6	6.87	50.2	4.88	18.8	2.47	32.7	3.22	6.34	0.416	9.25	0.724
	7	6.76	49.4	4.96	16.1	0.927	33.5	3	7.85	1.67	9.48	0.684
	8	6.64	51	4.62	17	1.06	36.4	2.75	8.21	0.583	10.1	1.73
	9	6.69	50.9	2.56	16.1	0.845	37.8	2.2	8.44	0.387	8.78	0.279
	10	6.74	48	6.39	14.4	2.95	33.7	5.1	7.56	2	7.9	1.25
	Total[2]	6.74	60.6	21.4	47.1	48.3	32.5	7.64	7.73	1.53	14.8	10

[1] 5th percentile value

[2] Average and standard deviation for annual average release rates

* Majority of concentrations during this time interval were below detection limit. One half the detection limit was used to calculate the annual average release rate.

Table 6. Page 1 of 3. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in umol*(kg rock*week)$^{-1}$. Period of record is 6 - 520 weeks for R1, R2, R4, R5, HC7, HC8. Period of record is 6 – 232 weeks for R3 and R6. During the first year, 11-16 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 6-8 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 1 75 grams -100/+270 unsaturated	1	7.04	294	135	0.0543	0.0443	0.0883	0.0504	0.0381*	0.000469	0.12	0.0506
	2	6.87	62.4	15.3	0.0582	0.0282	0.132	0.0377	0.0378*	0.000219	0.0771	0.0404
	3	6.72	45.4	10.5	0.0351*	0.000336	0.0794	0.0362	0.0378*	0.000362	0.0593	0.0275
	4	6.78	31	3.99	0.064	0.0432	0.0802	0.0226	0.0378*	0.000296	0.0911	0.0489
	5	6.88	25.1	2.58	0.0441	0.0266	0.082	0.0269	0.0379*	0.000216	0.0596	0.0326
	6	6.65	19.5	2.71	0.0428*	0.0186	0.165	0.0373	0.0385*	0.000309	0.0476*	0.0341
	7	6.59	16.6	2.16	0.0666	0.0262	0.284	0.024	0.0386*	0.000555	0.191	0.109
	8	6.50	15.3	1.82	0.115[3]	0.0629	0.289	0.0621	0.0388*	0.000163	0.142	0.0965
	9	6.41	13.9	5.68	0.0557	0.0241	0.369	0.0603	0.0389*	0.000426	0.369	0.0711
	10	6.64	13.9	4.12	0.0706	0.0287	0.337	0.0715	0.0387*	0.000585	0.262	0.0815
	Total[2]	6.52	53.7	86	0.0606	0.0221	0.191	0.117	0.0383	0.000462	0.142	0.104
Reactor 2 75 grams -100/+270 unsaturated	1	7.20	232	77.4	0.0415	0.0174	0.0679	0.0371	0.0377*	0.00113	0.0943	0.0386
	2	6.92	81.4	19.9	0.0459	0.0218	0.147	0.025	0.0378*	0.000239	0.112	0.0228
	3	6.84	48.4	10.6	0.0345*	0.000476	0.104	0.053	0.0372*	0.000514	0.0788	0.0503
	4	6.88	33.4	4.79	0.0469	0.0222	0.125	0.0317	0.0368*	0.000355	0.109	0.0243
	5	6.88	23.9	2.26	0.0392	0.0148	0.0937	0.0149	0.0368*	0.000956	0.0559	0.0299
	6	6.74	19	2.75	0.0348*	0.000321	0.113	0.0148	0.0375*	0.000346	0.0662	0.045
	7	6.71	16.9	3.42	0.0342*	0.00212	0.178	0.0401	0.0369*	0.00229	0.178	0.0484
	8	6.57	16.7	1.28	0.0661	0.0399	0.183	0.047	0.0379*	0.000206	0.135	0.12
	9	6.45	22.4	10.1	0.0633	0.0389	0.359	0.0916	0.044	0.0153	0.293	0.0992
	10	6.64	16.3	2.55	0.0843	0.0408	0.453	0.0816	0.0469	0.0217	0.313	0.0778
	Total[2]	6.60	51	66.6	0.0491	0.0168	0.182	0.125	0.039	0.00351	0.144	0.091
Reactor 3 75 grams -100/+270 unsaturated	1	7.24	240	68.4	0.0429	0.0205	0.0442	0.016	0.038*	0.000833	0.126	0.0233
	2	6.99	89.9	28.1	0.035*	0.000509	0.123	0.0153	0.0378*	0.000549	0.107	0.0537
	3	6.81	55.8	6.17	0.0353*	0.000291	0.0787	0.0483	0.038*	0.000314	0.0801	0.0613
	4	6.90	34.1	6.25	0.0467	0.0189	0.0803	0.0169	0.0415*	0.0121	0.11	0.0722
	5	7.00	26.4	2.39	0.035*	0.000123	0.0577	0.0229	0.0377*	0.000132	0.137	0.0619
	6	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	7	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	8	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	9	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	10	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	Total[2]	6.93	89.3	87.8	0.039	0.00546	0.0767	0.0299	0.0386	0.00164	0.112	0.0215

[1] 5th percentile value

[2] Average and standard deviation for annual average release rates

[3] Two of the highest copper values in the period of record occurred during year 8, along with few values below the detection limit.

* Majority of concentrations during this time interval were below detection limit. One half the detection limit was used to calculate the annual average release rate.

Table 6. Page 2 of 3. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in umol*(kg rock*week)$^{-1}$. Period of record is 6 - 520 weeks for R1, R2, R4, R5, HC7, HC8. Period of record is 6 – 232 weeks for R3 and R6. During the first year, 11-16 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 6-8 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Reactor 4 75 grams -100/+270 saturated	1	6.81	371	161	0.0468*	0.0242	0.234	0.494	0.0506	0.0231	0.223	0.163
	2	6.48	180	61.6	0.0404*	0.000536	2.16	0.5	0.149	0.0312	0.283	0.04
	3	6.59	74.7	16.3	0.0403*	0.00103	0.534	0.371	0.0533	0.0242	0.134	0.105
	4	6.60	66.5	15.5	0.0403*	0.000445	0.135	0.0393	0.0434*	0.000479	0.177	0.0641
	5	6.82	38.3	3.87	0.0402*	0.000362	0.0595	0.0315	0.0434*	0.000379	0.19	0.0745
	6	6.72	25.9	3.83	0.0405*	0.000817	0.0438*	0.000885	0.0436*	0.000881	0.131	0.0508
	7	6.80	18.9	1.25	0.0398*	0.00744	0.0433*	0.000669	0.0429*	0.000802	0.197	0.0563
	8	6.61	18.1	1.93	0.0451*	0.0144	0.0433*	0.00115	0.0432*	0.00115	0.12	0.096
	9	6.65	12.1	4.74	0.0398*	0.000937	0.0431*	0.00101	0.0429*	0.00101	0.248	0.0229
	10	6.67	13.2	4.95	0.0395*	0.000446	0.0428*	0.000483	0.0426*	0.000481	0.246	0.0482
	Total[2]	6.59	81.9	114	0.0413	0.0025	0.334	0.659	0.0555	0.0331	0.195	0.0556
Reactor 5 75 grams -100/+270 saturated	1	6.79	311	108	0.0397*	0.00286	0.45	0.743	0.0597	0.0402	0.17	0.0805
	2	6.41	212	78.3	0.0497	0.0199	3.5	1.26	0.237	0.0903	0.41	0.106
	3	6.55	89.2	16.4	0.0402*	0.000731	0.516	0.294	0.0516*	0.0197	0.139	0.11
	4	6.52	54.2	11.2	0.0489*	0.0295	0.114	0.074	0.0538*	0.0352	0.218	0.0896
	5	6.78	33.9	4.88	0.0399*	0.000729	0.0432*	0.000789	0.043*	0.000786	0.176	0.0954
	6	6.64	24.1	3.43	0.0379*	0.00269	0.0411*	0.00291	0.0409*	0.0029	0.154	0.0473
	7	6.79	19.2	1.17	0.0391*	0.00115	0.0592*	0.0472	0.0422*	0.00124	0.251	0.103
	8	6.67	18.4	2.77	0.039*	0.000672	0.0474*	0.0144	0.042*	0.000725	0.313	0.0917
	9	6.51	15.2	12.7	0.0385*	0.00134	0.0418*	0.00138	0.0416*	0.00137	0.22	0.111
	10	6.59	15.8	2.88	0.0381*	0.00116	0.0412*	0.00125	0.041*	0.00125	0.158	0.0427
	Total[2]	6.50	79.3	101	0.0411	0.0044	79.3	101	0.0653	0.0606	0.221	0.0851
Reactor 6 75 grams -100/+270 saturated	1	6.86	306	135	0.0406*	0.000512	0.0728	0.0269	0.0436*	0.000487	0.143	0.106
	2	6.55	187	71.8	0.0401*	0.000859	1.71	0.595	0.084	0.0493	0.306	0.0608
	3	6.49	108	10	0.0404*	0.00068	0.784	0.211	0.0678	0.0283	0.34	0.0856
	4	6.44	62.9	22.9	0.04*	0.000826	0.229	0.106	0.0431*	0.000891	0.21	0.0787
	5	6.76	34.4	4.29	0.04*	0.00275	0.0716	0.0331	0.0431*	0.000296	0.154	0.0232
	6	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	7	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	8	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	9	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	10	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
	Total[2]	6.51	140	109	0.0402	0.000291	0.573	0.698	0.0563	0.0188	0.23	0.0887

[1] 5th percentile value

[2] Average and standard deviation for annual average release rates

* Majority of concentrations during this time interval were below detection limit. One half the detection limit was used to calculate the annual average release rate.

Table 6. Page 3 of 3. Minimum pH and average annual rates of release for sulfate, copper, nickel, cobalt, and zinc in umol*(kg rock*week)$^{-1}$. Period of record is 6 - 520 weeks for R1, R2, R4, R5, HC7, HC8. Period of record is 6 – 232 weeks for R3 and R6. During the first year, 11-16 concentrations were sampled for each parameter. Sampling frequency for subsequent years decreased to approximately 6-8 per year.

ID	Year	min pH[1]	Sulfate		Copper		Nickel		Cobalt		Zinc	
			rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.	rate	s.d.
Humidity cell 7 1000 grams -1/4 inch	1	7.30	115	25.5	0.00907	0.0038	0.0592	0.0301	0.00805*	0.000329	0.011	0.00959
	2	7.38	71.8	13.3	0.0111	0.00715	0.0228	0.0104	0.00822*	0.000145	0.00966	0.00441
	3	7.15	55.6	6.61	0.00763*	0.0000549	0.0121	0.00604	0.00823*	0.0000592	0.0119	0.011
	4	6.87	49.2	4.59	0.00771*	0.0000996	0.0225	0.0058	0.00831*	0.000107	0.0148	0.0162
	5	6.88	49.5	5.63	0.00764*	0.0000995	0.0308	0.00392	0.00824*	0.000107	0.0108	0.00693
	6	6.82	51.8	5.69	0.00763*	0.000126	0.0292	0.0131	0.00823*	0.000136	0.0111	0.00634
	7	6.79	49.6	3.43	0.0104	0.00708	0.0414	0.00957	0.00826*	0.00012	0.0214	0.0165
	8	6.74	51.6	3.72	0.0102	0.00495	0.0417	0.00445	0.0082*	0.0000863	0.0116	0.0118
	9	6.69	51	2.81	0.00764*	0.0000493	0.0428	0.00624	0.00824*	0.0000531	0.012	0.00597
	10	6.82	50.9	5.23	0.00765*	0.000111	0.038	0.00622	0.00825*	0.00012	0.0171	0.0177
	Total[2]	6.78	59.7	20.7	0.00866	0.00139	0.034	0.0134	0.00822	0.0000669	0.0131	0.00364
Humidity cell 8 1000 grams -1/4 inch	1	7.48	116	23.5	0.00906	0.00399	0.058	0.036	0.00814*	0.000188	0.0085*	0.00389
	2	7.39	76.5	13	0.00765*	0.000116	0.00954*	0.00368	0.00825*	0.000125	0.00743*	0.000113
	3	7.24	61.8	6.5	0.00771*	0.0000558	0.00835*	0.0000604	0.00831*	0.0000602	0.0119	0.0107
	4	6.97	53.2	5.97	0.0077*	0.0000508	0.00924*	0.00298	0.0083*	0.0000548	0.0101	0.00644
	5	6.87	48.4	5.94	0.00764*	0.0000556	0.0192	0.00481	0.00824*	0.00006	0.00841	0.00295
	6	6.87	50.2	4.88	0.00765*	0.0000826	0.0248	0.00465	0.00825*	0.0000891	0.00742*	0.0000808
	7	6.76	49.4	4.96	0.00765*	0.0000616	0.0404	0.00994	0.00825*	0.0000664	0.013	0.00715
	8	6.64	51	4.62	0.00762*	0.0000432	0.042	0.0134	0.00822*	0.0000466	0.0075*	0.000264
	9	6.69	50.9	2.56	0.00767*	0.0000793	0.0613	0.00381	0.00827*	0.0000855	0.0135	0.00692
	10	6.74	48	6.39	0.00764*	0.0000533	0.0557	0.0133	0.00824*	0.0000575	0.0111	0.00888
	Total[2]	6.74	60.6	21.4	0.0078	0.000446	0.0329	0.0212	0.00825	0.0000474	0.00987	0.00236

[1] 5th percentile value

[2] Average and standard deviation for annual average release rates

* Majority of concentrations during this time interval were below detection limit. One half the detection limit was used to calculate the annual average release rate.

Table 7. Total mass release percent depletion for weeks 0-520.

Reactor /HC	Total mass release after 520 weeks, mmol/kg rock									Trace metals, % remaining after 520 weeks			
	SO_4	Ca	Mg	Na	K	Cu	Ni	Co	Zn	Cu	Ni	Co	Zn
Unsaturated 75-g Reactors													
1	28.3	30.1	26.4	9.7	14.7	0.032	0.0994	0.02	0.0746	99.6%	98.4%	98.0%	95.9%
2	28.2	32.4	29.9	10.9	11.1	0.0251	0.094	0.0201	0.0737	99.7%	98.5%	98.0%	96.0%
3[1]	23.7	26.1	20.3	5.8	7.51	0.00954	0.0204	0.00914	0.0256	NA	NA	NA	NA
Saturated 75-g Reactors													
4	41.9	33.8	29.7	13.4	14	0.0217	0.172	0.0289	0.0992	99.7%	97.2%	97.1%	94.6%
5	40.9	33.4	29.4	13.7	13.6	0.0213	0.237	0.0326	0.112	99.7%	96.1%	96.7%	93.9%
6[1]	35.9	28.2	21.4	8.82	10.2	0.00940	0.151	0.0134	0.0565	NA	NA	NA	NA
1000-g Humidity Cells													
7[2]	34.3	25.5	19.5	5.38	8.07	0.00464	0.0184	0.00432	0.0066	99.9%	99.7%	99.6%	99.6%
8[2]	31.9	25.8	18.4	5.5	8.27	0.00418	0.0175	0.00433	0.0049	99.9%	99.7%	99.6%	99.7%

[1] Reactors 3 and 6 were terminated after 232 weeks
[2] Incorrect values have been updated for humidity cell total mass release, 6/17/2014

Table 8. Annual percent sulfur remaining for reactors and humidity cells.

Year	Unsaturated 75-g Reactors			Saturated 75-g Reactors			1000-g Humidity Cells	
	R1	R2	R3[1]	R4	R5	R6[1]	HC7	HC8
1	74.6%	77.7%	76.8%	68.1%	72.0%	71.0%	84.7%	89.3%
2	69.8%	71.2%	70.4%	54.9%	56.9%	57.8%	79.1%	83.3%
3	66.1%	67.3%	65.9%	49.0%	49.7%	49.6%	74.6%	78.2%
4	63.5%	64.5%	63.0%	43.3%	45.1%	43.8%	70.5%	73.8%
5	61.4%	62.5%	NA	40.1%	42.3%	NA	66.3%	69.8%
6	59.8%	60.9%	NA	38.0%	40.3%	NA	62.0%	65.5%
7	58.4%	59.5%	NA	36.4%	38.7%	NA	57.8%	61.4%
8	57.1%	58.1%	NA	34.9%	37.2%	NA	53.5%	57.2%
9	55.9%	56.2%	NA	33.9%	35.7%	NA	49.2%	52.9%
10	54.7%	54.8%	NA	32.8%	34.5%	NA	45.0%	48.9%

[1] Reactors 3 and 6 were terminated after 232 weeks

Table 9. Annual median pH, average sulfate release, and average sulfate release as a function of mass of sulfur remaining for 75 gram reactors.

Year	Unsaturated	Saturated	Unsaturated	Saturated	Unsaturated	Saturated
	Median pH		Avg SO_4 release rate (µmol/kg*wk)		Avg SO_4 release/g S remaining (µmol/g S*wk)	
1	7.48	7.55	262.9	341.2	152.6	202.3
2	7.12	6.64	71.9	195.9	48.6	152.6
3	7.02	6.70	46.9	81.9	34.1	77.6
4	6.95	6.75	32.2	60.4	24.6	64.8
5	7.07	6.92	24.5	36.1	19.4	42.3
6	6.92	6.90	19.2	25.0	15.7	31.1
7	6.81	6.92	16.7	18.9	14.0	24.8
8	6.76	6.83	16.0	18.2	13.7	24.8
9	6.57	6.70	18.2	13.6	16.0	19.2
10	6.74	6.82	15.1	14.5	13.6	21.1

Table 10. Estimated calcium release from plagioclase, $CaCO_3$ and other silicates. Release in mmol.

ID	Duration[1]	Total Ca[2]	Total Na[3]	Ca (plag)[4]	Ca w/ $CaCO_3$[5]	Ca from silicates[6]
Unsaturated, 75 gram reactors						
R1	275	1.949	0.442	0.63	0.83	0.49
R2	266	2.001	0.474	0.68	0.83	0.50
Saturated, 75 gram reactors						
R4	107	1.772	0.362	0.52	0.83	0.43
R5	101	1.754	0.363	0.52	0.83	0.41
1000 gram humidity cells						
HC7	362	23.03	4.18	5.96	11.00	6.07
HC8	367	23.36	4.30	6.13	11.00	6.23

[1] Time at which Ca to Na ratio equals the stoichiometric ratio for plagioclase (57/40=1.425)

[2, 3] Total Ca, Na mass release by the designated time period

[4] Estimated mass release of calcium associated with plagioclase, based the stoichiometric ratio

[5] Maximum amount of Ca with $CaCO_3$, (assumes 1.1 g $CaCO_3$/kg based on the maximum possible CO_2 content and assumption that all CO_2 is associated with $CaCO_3$)

[6] Remaining Ca that is associated with dissolution of other silicates (e.g. augite)

Table 11. Charge balance using average annual concentrations. Values reported in meq/L. For concentrations measured below the detection limit, half the detection limit was used.

Reactor/HC	Year	Ca	Mg	Na	K	SO4	Alk	Cations	Anions	Cations-Anions	Cations/Anions
Avg R1&R2 unsaturated reactors	1	0.273	0.098	0.010	0.046	0.234	0.148	0.427	0.382	0.045	1.12
	2	0.072	0.078	0.007	0.011	0.065	0.053	0.168	0.118	0.050	1.44
	3	0.040	0.058	0.008	0.004	0.042	0.050	0.110	0.092	0.018	1.20
	4	0.027	0.046	0.007	0.001	0.029	0.050	0.081	0.079	0.002	1.02
	5	0.024	0.041	0.007	0.003	0.022	0.061	0.074	0.083	-0.009	0.90
	6	0.019	0.035	0.006	0.008	0.017	0.055	0.068	0.072	-0.004	0.94
	7	0.020	0.031	0.008	0.009	0.015	0.050	0.068	0.065	0.003	1.05
	8	0.019	0.027	0.008	0.007	0.014	0.053	0.061	0.067	-0.006	0.91
	9	0.016	0.022	0.008	0.005	0.016	0.053	0.051	0.069	-0.018	0.74
	10	0.015	0.022	0.006	0.005	0.013	0.050	0.048	0.063	-0.015	0.76
Avg R4&R5 saturated reactors	1	0.295	0.127	0.012	0.035	0.269	0.204	0.468	0.473	-0.004	0.99
	2	0.063	0.087	0.014	0.016	0.155	0.050	0.180	0.205	-0.025	0.88
	3	0.032	0.041	0.011	0.009	0.064	0.050	0.093	0.114	-0.021	0.81
	4	0.024	0.035	0.009	0.007	0.047	0.050	0.075	0.097	-0.022	0.77
	5	0.020	0.030	0.008	0.007	0.028	0.053	0.065	0.081	-0.016	0.80
	6	0.016	0.025	0.007	0.005	0.020	0.055	0.053	0.075	-0.023	0.70
	7	0.017	0.026	0.008	0.006	0.015	0.054	0.057	0.069	-0.012	0.83
	8	0.018	0.024	0.008	0.005	0.015	0.050	0.055	0.065	-0.009	0.86
	9	0.015	0.017	0.008	0.004	0.011	0.050	0.044	0.061	-0.017	0.72
	10	0.014	0.019	0.008	0.004	0.012	0.050	0.044	0.062	-0.017	0.72
Avg HC7&HC8 humidity cells	1	0.648	0.194	0.023	0.079	0.483	0.490	0.944	0.973	-0.029	0.97
	2	0.437	0.070	0.014	0.048	0.307	0.212	0.569	0.519	0.050	1.10
	3	0.256	0.107	0.014	0.032	0.241	0.155	0.408	0.396	0.012	1.03
	4	0.143	0.124	0.014	0.026	0.210	0.067	0.307	0.276	0.031	1.11
	5	0.106	0.138	0.015	0.024	0.202	0.065	0.282	0.267	0.015	1.06
	6	0.078	0.140	0.013	0.019	0.210	0.060	0.250	0.271	-0.021	0.92
	7	0.066	0.141	0.016	0.019	0.204	0.053	0.242	0.257	-0.015	0.94
	8	0.071	0.159	0.017	0.021	0.212	0.050	0.267	0.262	0.005	1.02
	9	0.066	0.159	0.017	0.018	0.210	0.050	0.260	0.260	0.000	1.00
	10	0.060	0.149	0.015	0.017	0.203	0.050	0.241	0.253	-0.012	0.95

Figure 1. Concentrator Flow sheet

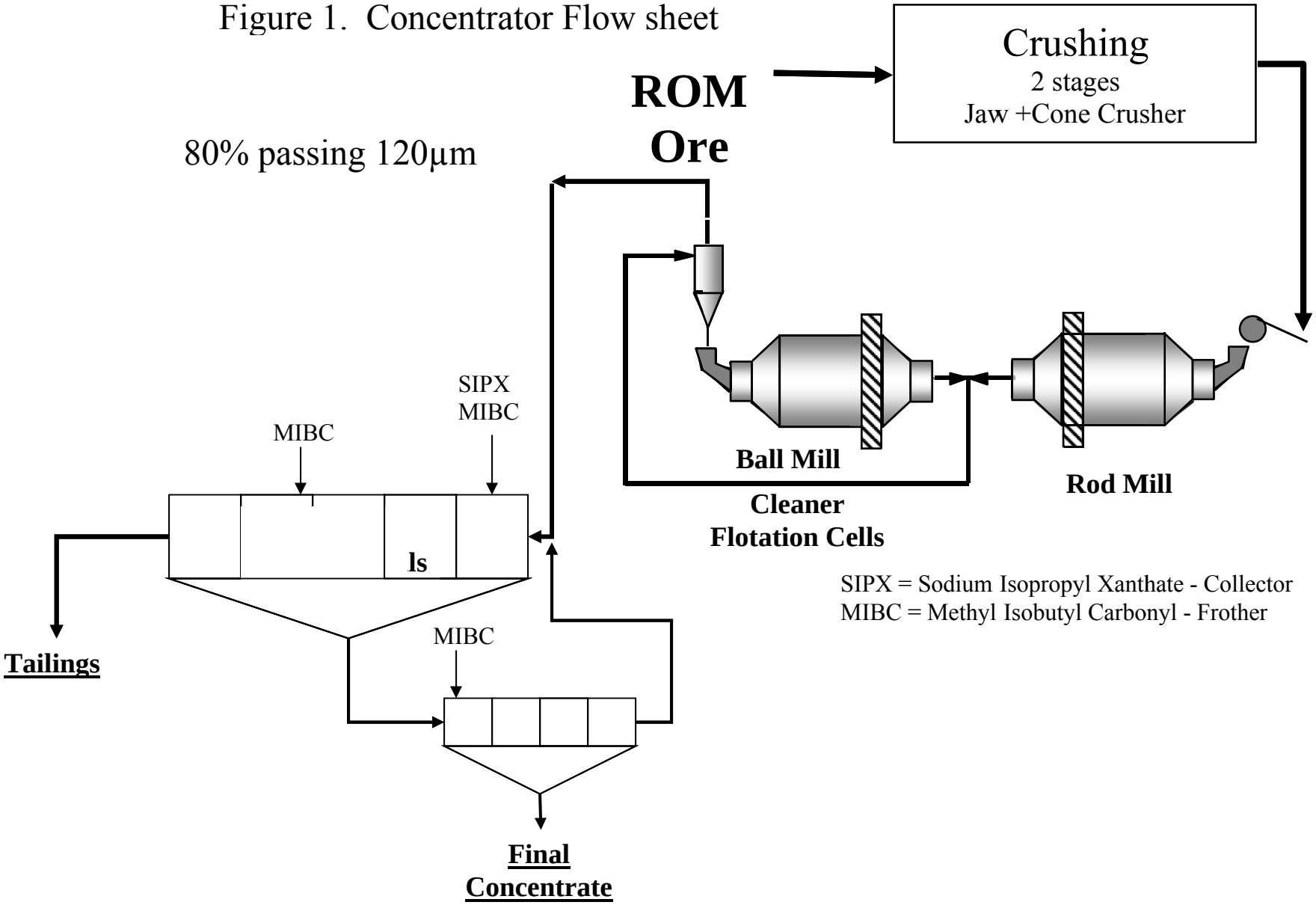


Figure 2. Schematic of small reactor.

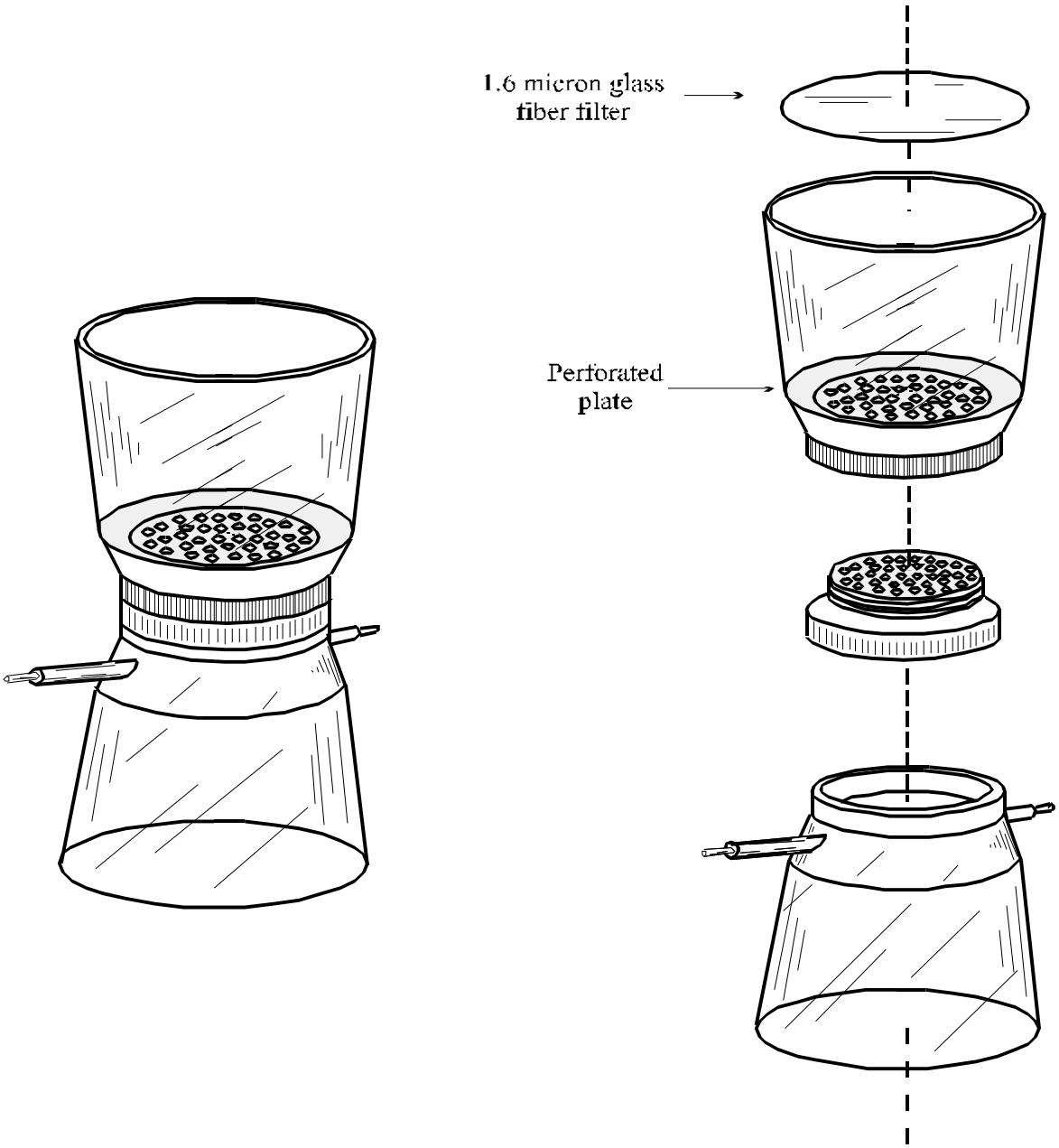


Figure 3. Schematic of humidity cell. All humidity cell materials are acrylic except the perforated plate (polyvinyl chloride) and the outlet pipe (high density polyethylene).

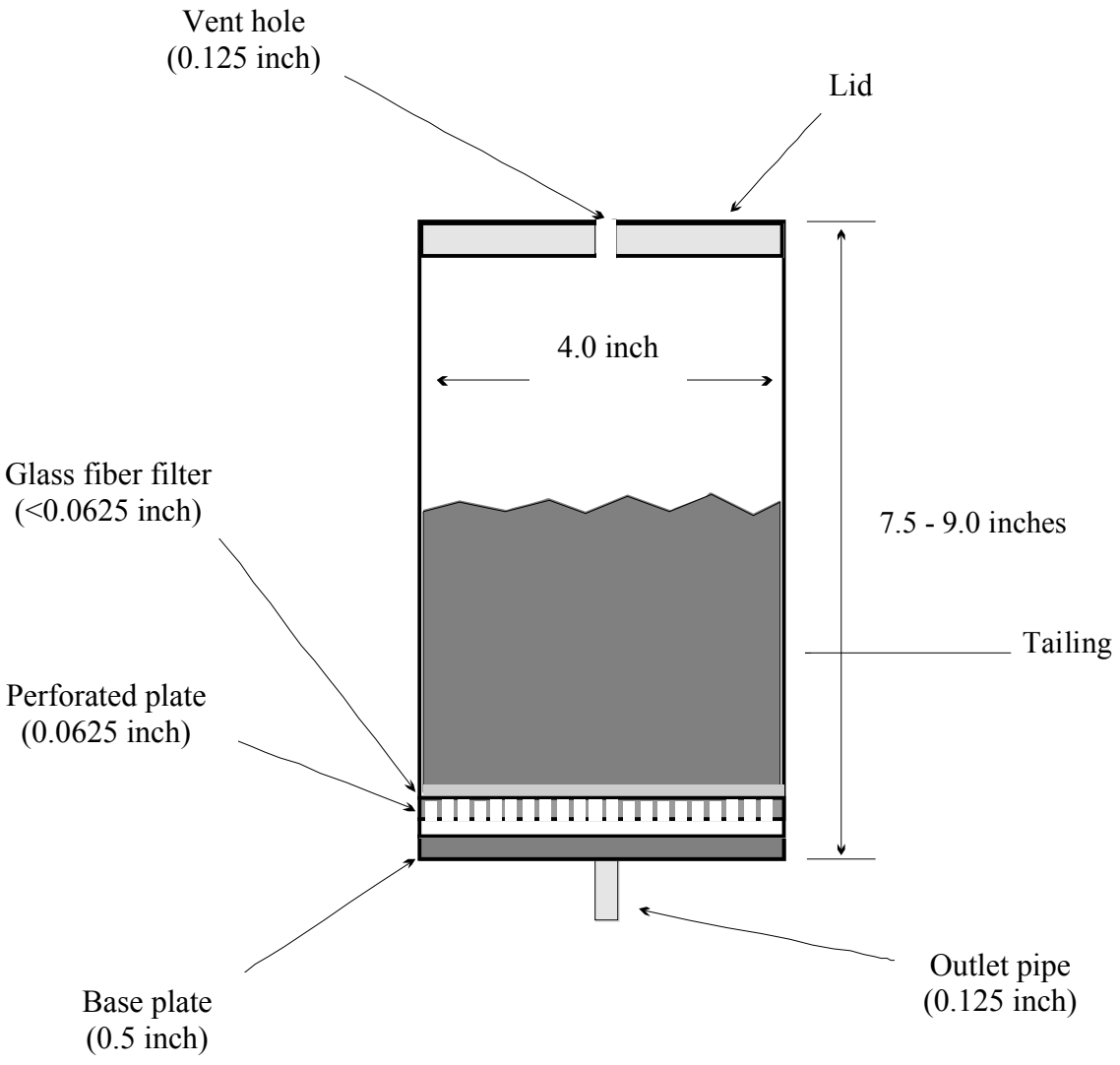


Figure 4. Drainage quality from the 75 g unsaturated tailings reactors (1, 2) replicated well. Weeks 0-5 were excluded to improve resolution.
Red = reactor 1, Blue = reactor 2

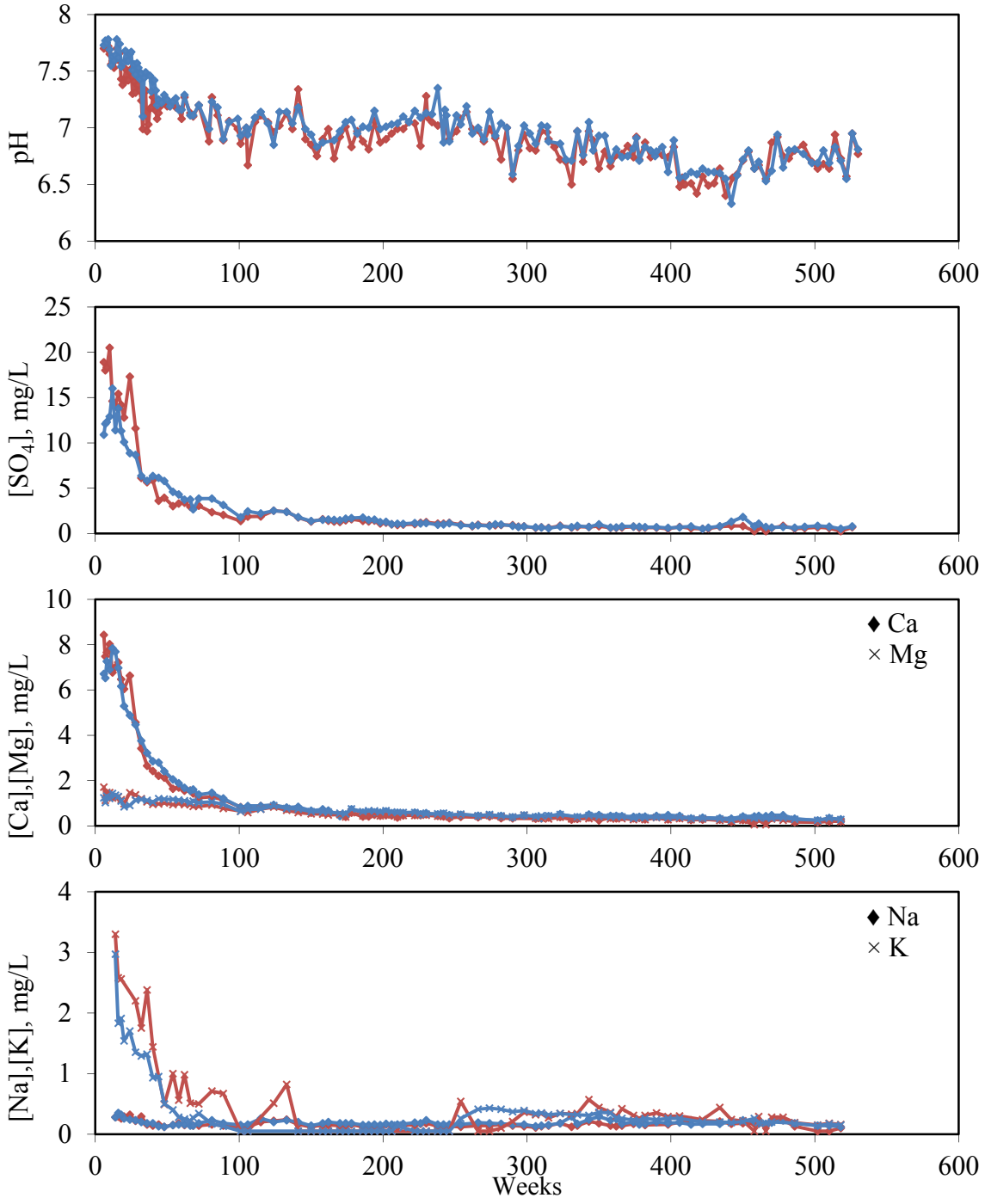


Figure 5. Drainage quality from the 75 g saturated tailings reactors (4, 5) replicated well. Weeks 0-5 were excluded to improve resolution.
Red = reactor 4, Blue = reactor 5

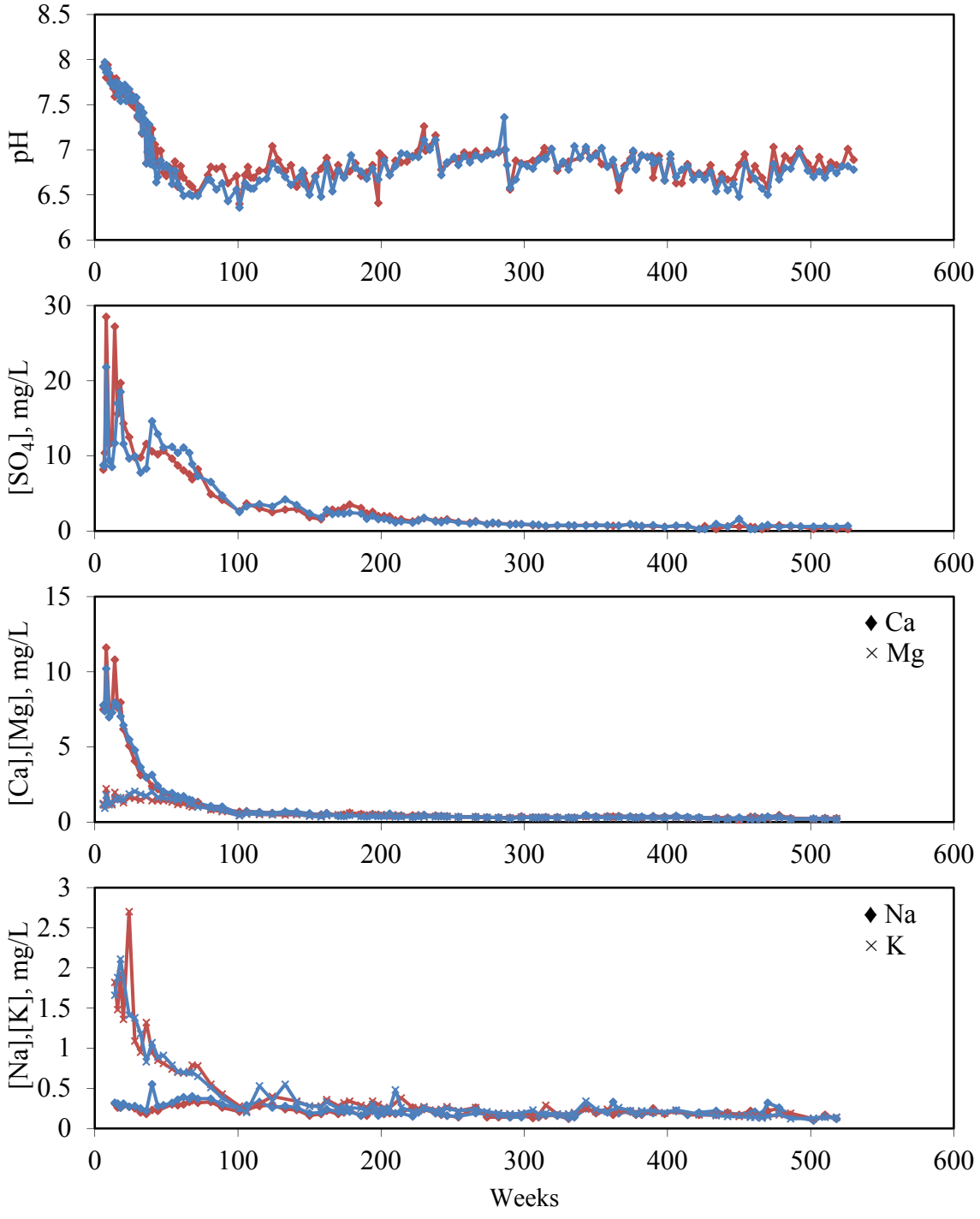
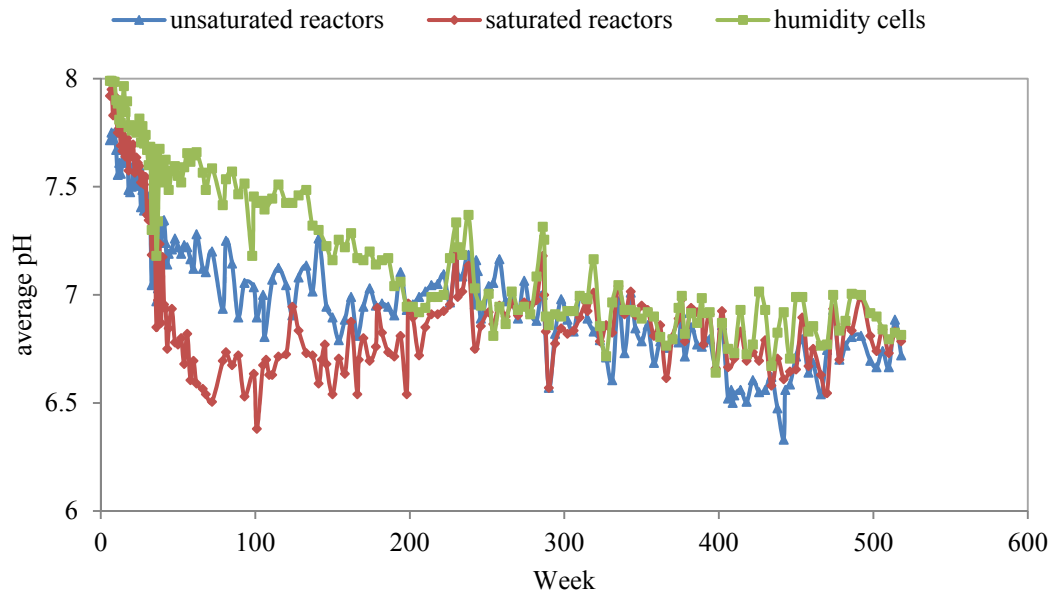


Figure 6. pH, alkalinity vs. time for 75 g reactors and 1000 g humidity cells. Weeks 0-5 were excluded to improve resolution.



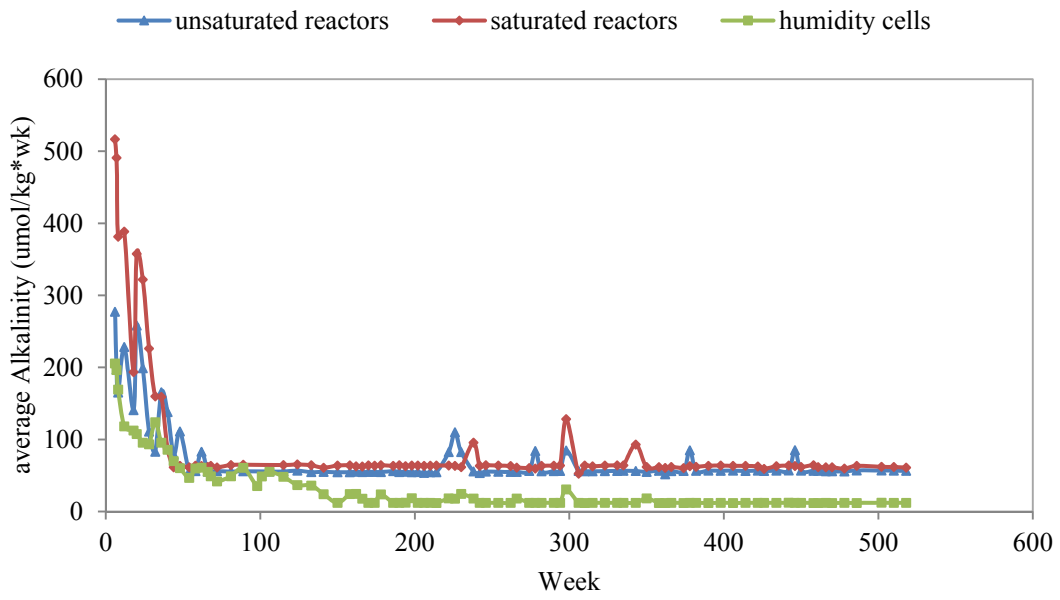


Figure 7. Average pH and SO_4, Ca, Mg, Na, K concentrations for unsaturated and saturated reactors. Weeks 0-50 were excluded to improve resolution. Similar comparison graphs for weeks 6-520 are located in Appendix 3 (Figure A3.33).

Red = saturated reactors, Blue = unsaturated reactors

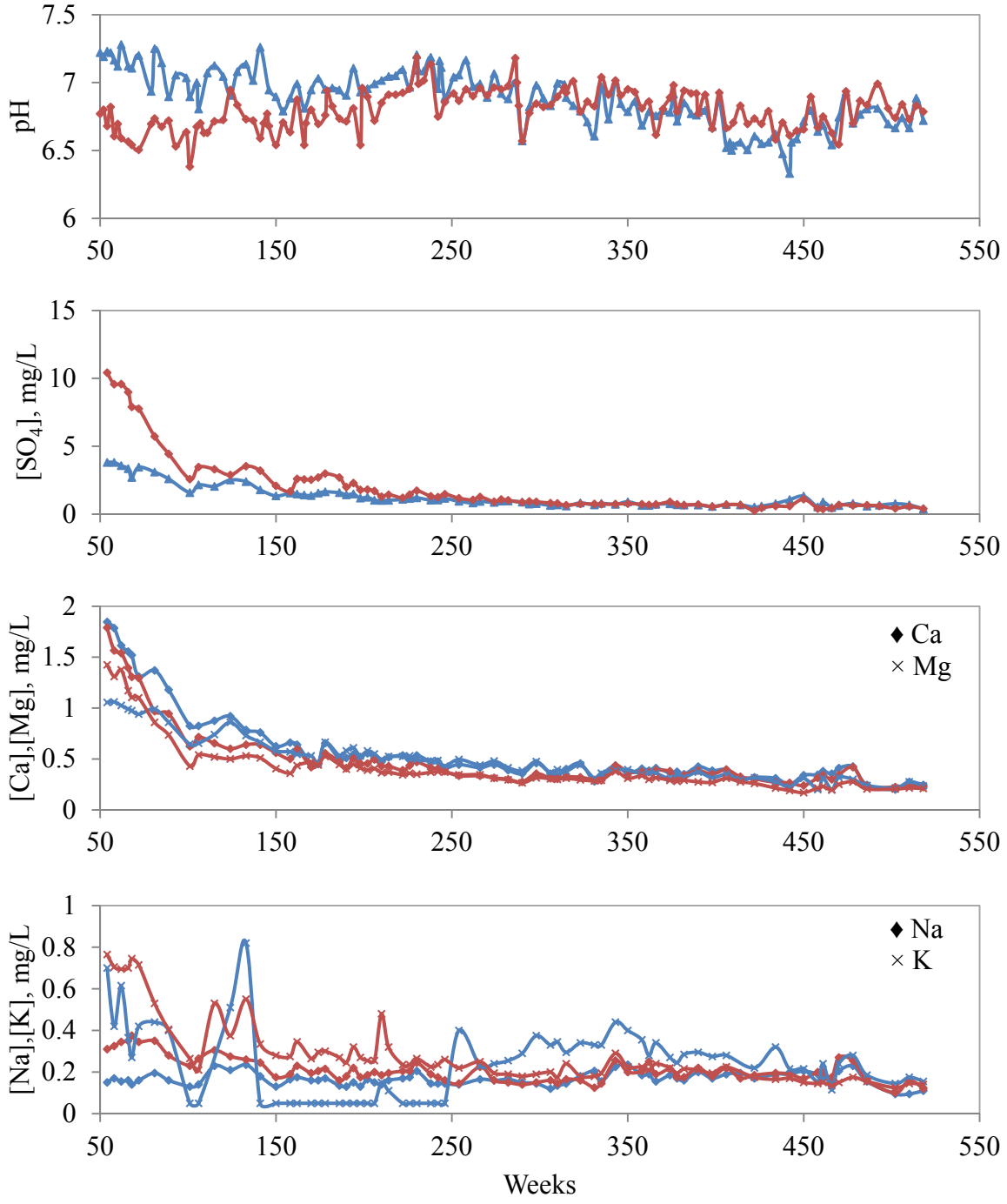


Figure 8. Average pH and SO_4, Cu, Ni, Co, Zn concentrations for unsaturated and saturated reactors. Weeks 0-50 were excluded to improve resolution. Similar comparison graphs for weeks 6-520 are located in Appendix 3 (Figure A3.34).

Red = saturated reactors, Blue = unsaturated reactors

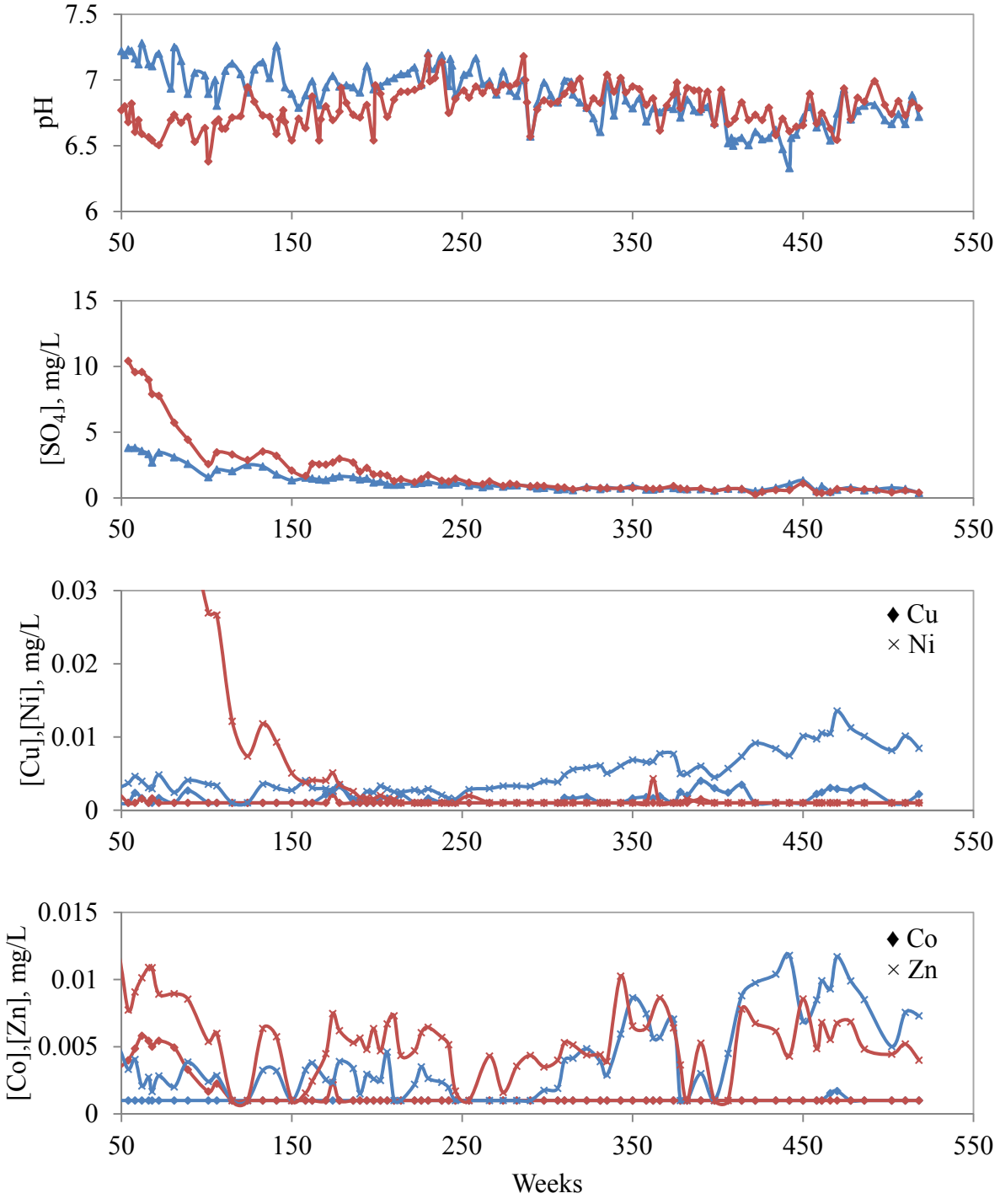


Figure 9. Nickel and pH vs. time for the 75 g unsaturated reactor 1 and saturated reactor 4. Weeks 0-5 were excluded to improve resolution.
Red = reactor 1 unsaturated, Blue = reactor 4 saturated

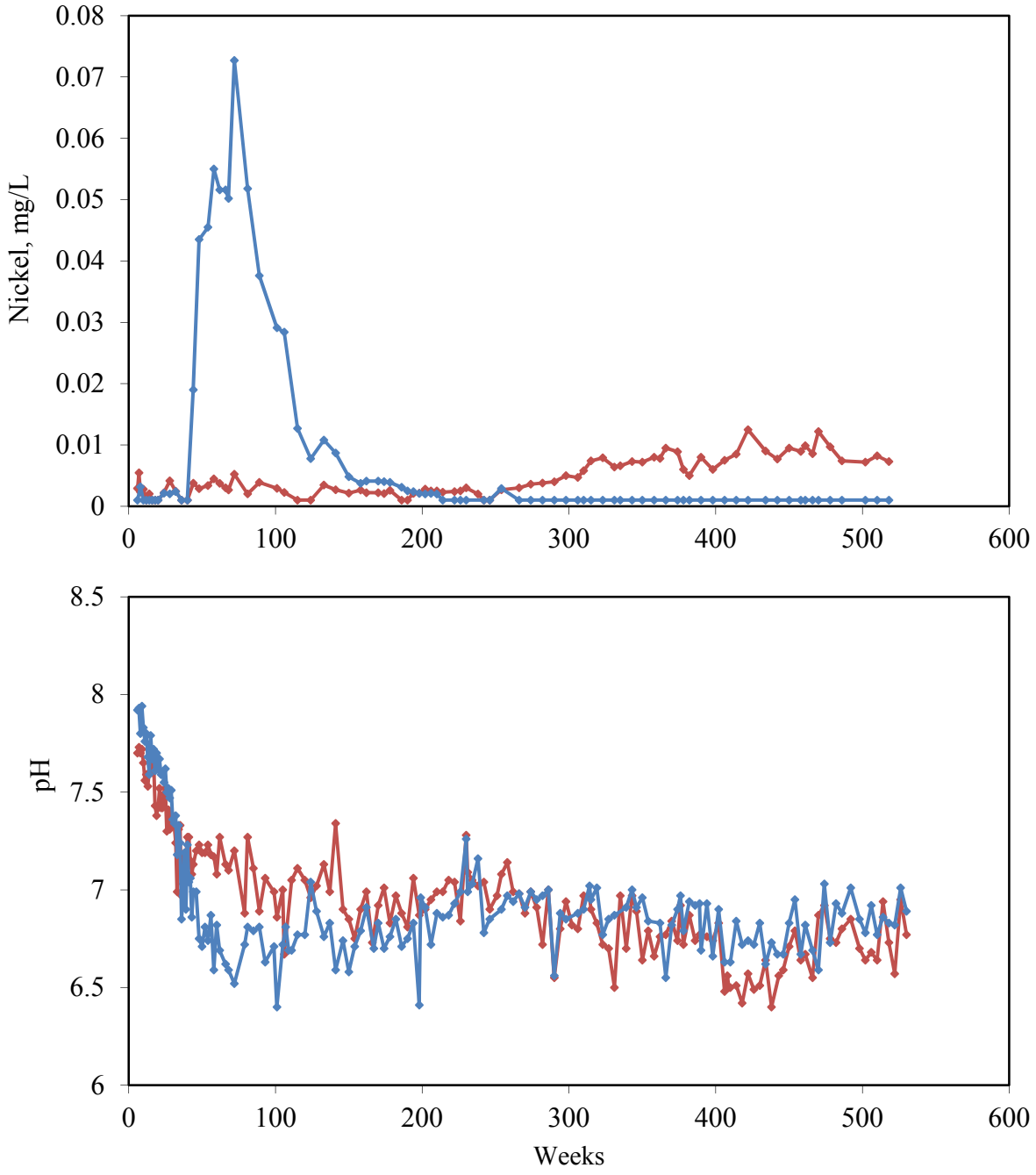


Figure 10. Cobalt and pH vs. time for the 75 g tailings unsaturated reactor 1 and saturated reactor 4. Weeks 0-5 were excluded to improve resolution.
Red = reactor 1 unsaturated, Blue = reactor 4 saturated

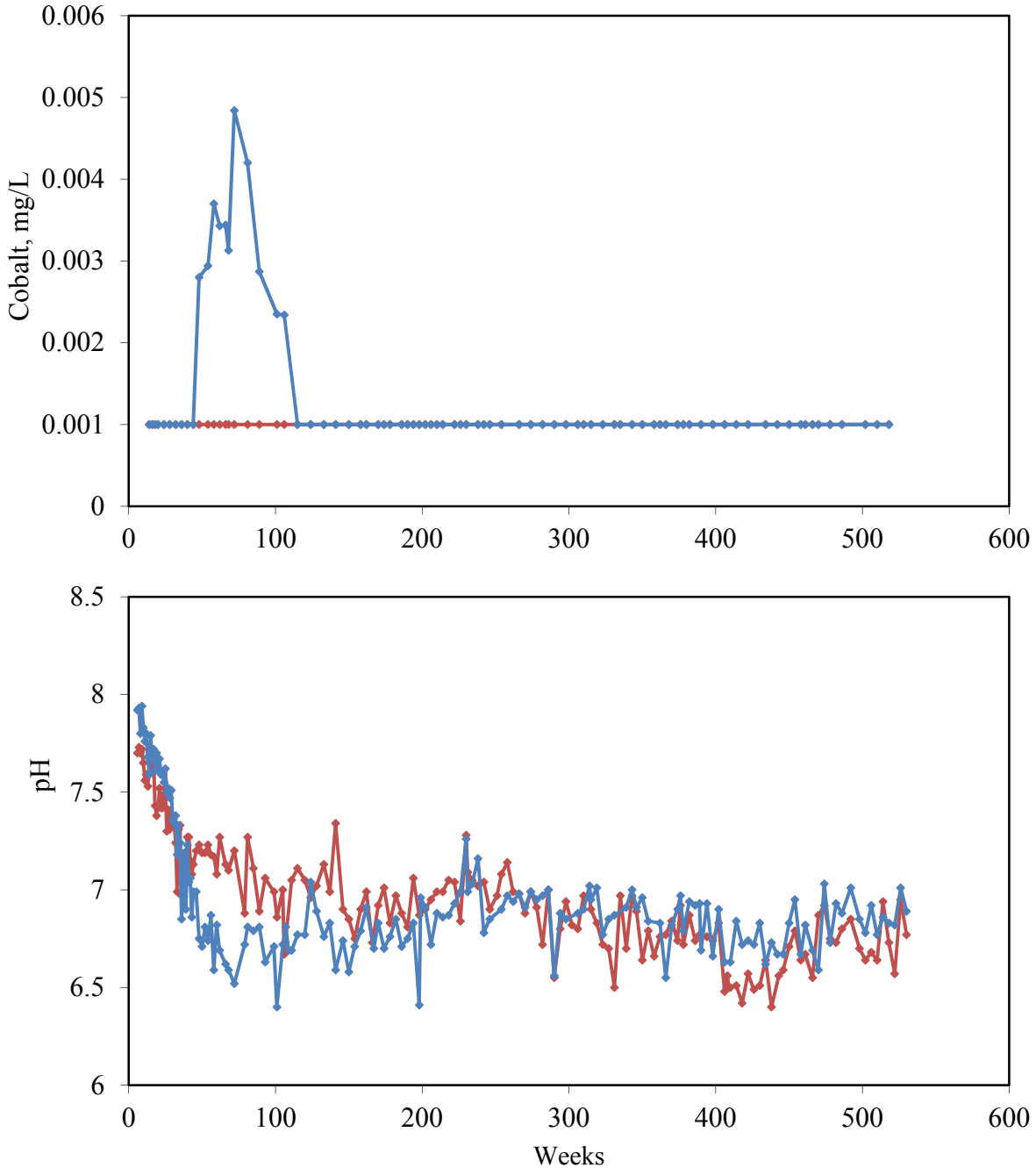


Figure 11. Drainage quality from the 1000 g tailings humidity cells (7, 8) replicated well. Weeks 0-5 were excluded to improve resolution.
Red = humidity cell 7, Blue = humidity cell 8

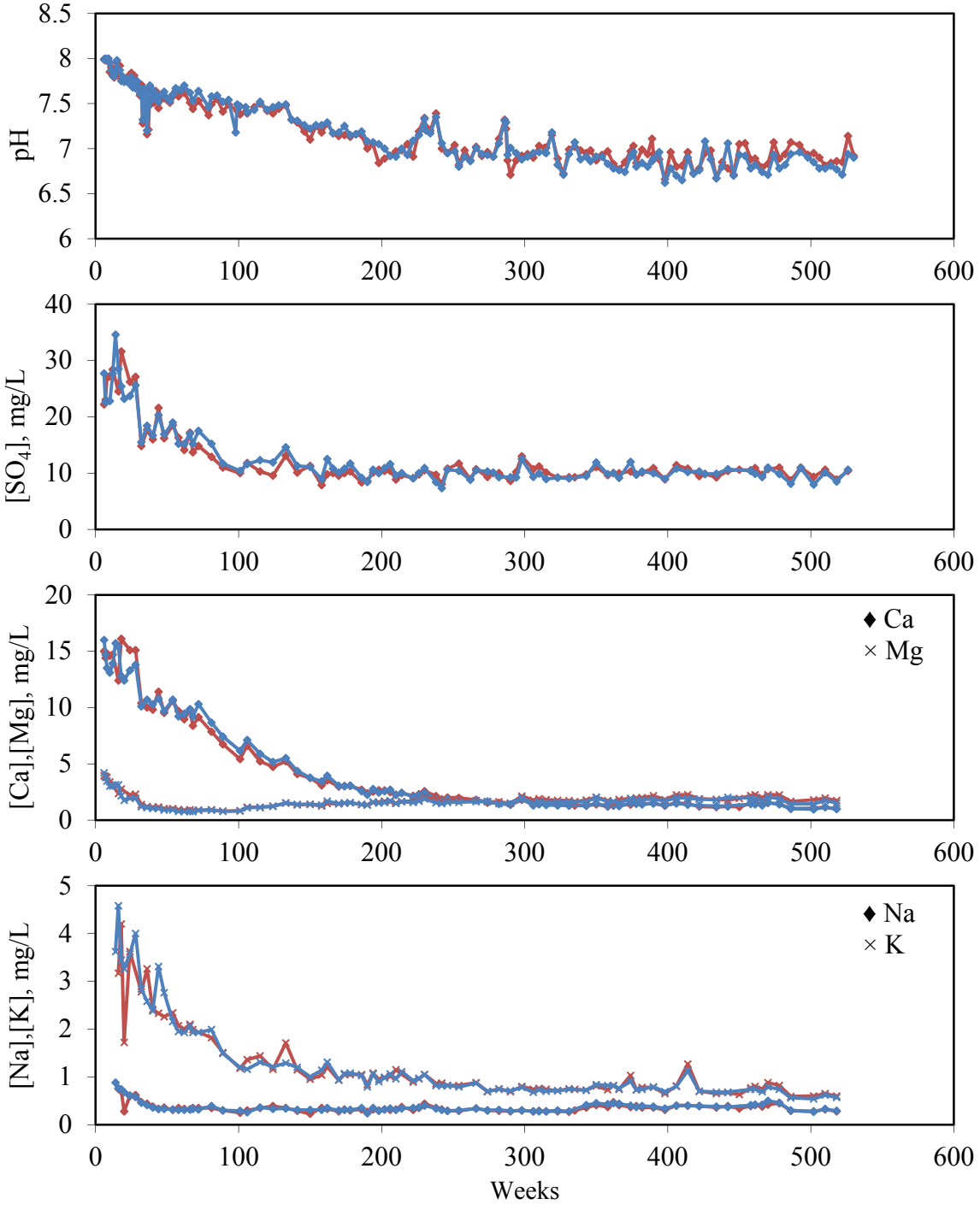


Figure 12. Average pH, SO_4, Ca, Mg, Na, K vs. time for saturated reactors and humidity cells. Weeks 0-50 were excluded to improve resolution. Similar comparison graphs for weeks 6-520 are located in Appendix 3 (Figure A3.35).

Red = saturated reactors, Blue = humidity cells

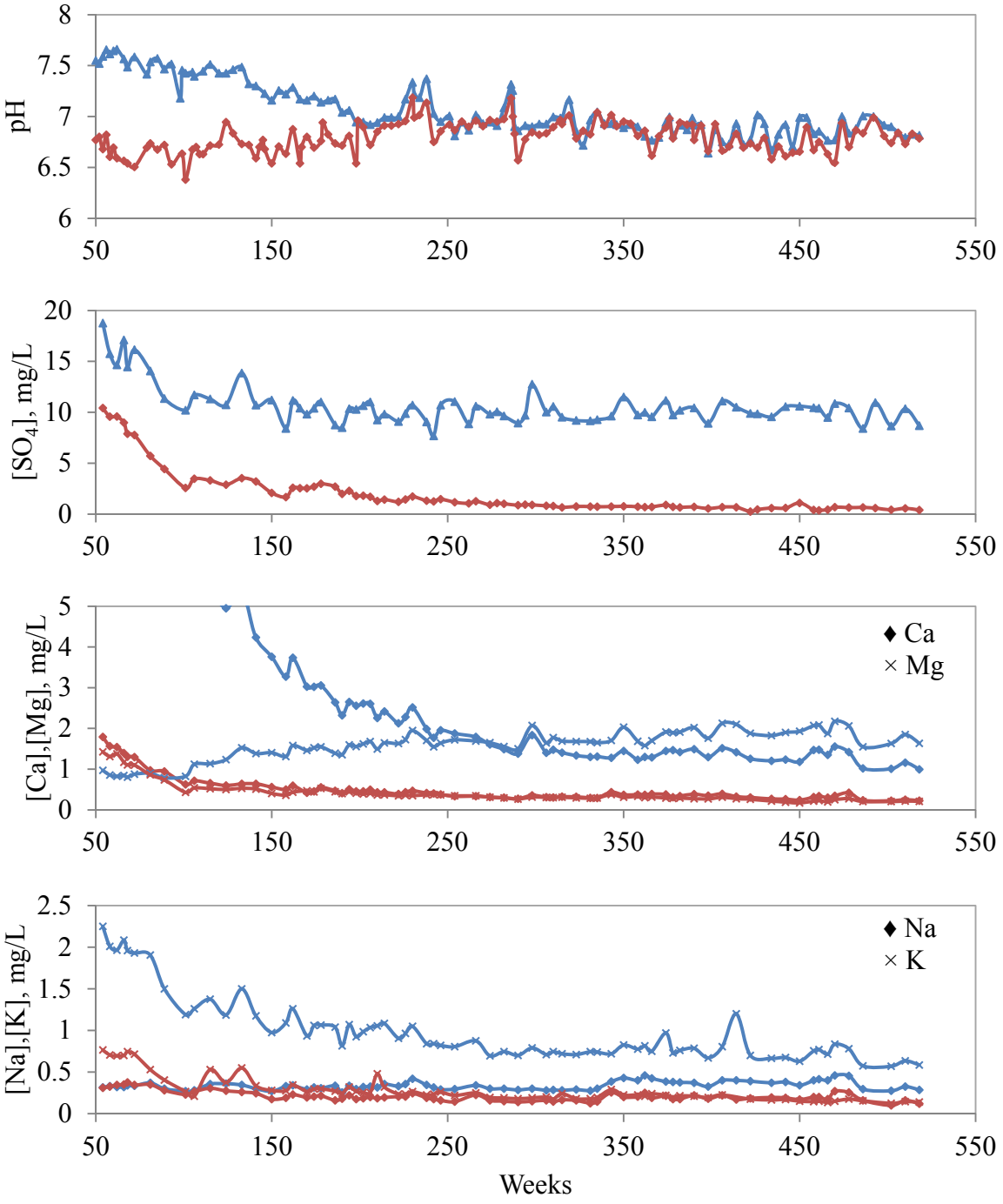


Figure 13. Average pH, SO_4, Cu, Ni, Co, Zn vs. time for saturated reactors and humidity cells. Weeks 0-50 were excluded to improve resolution. Similar comparison graphs for weeks 6-520 are located in Appendix 3 (Figure A3.36).

Red = saturated reactors, Blue = humidity cells

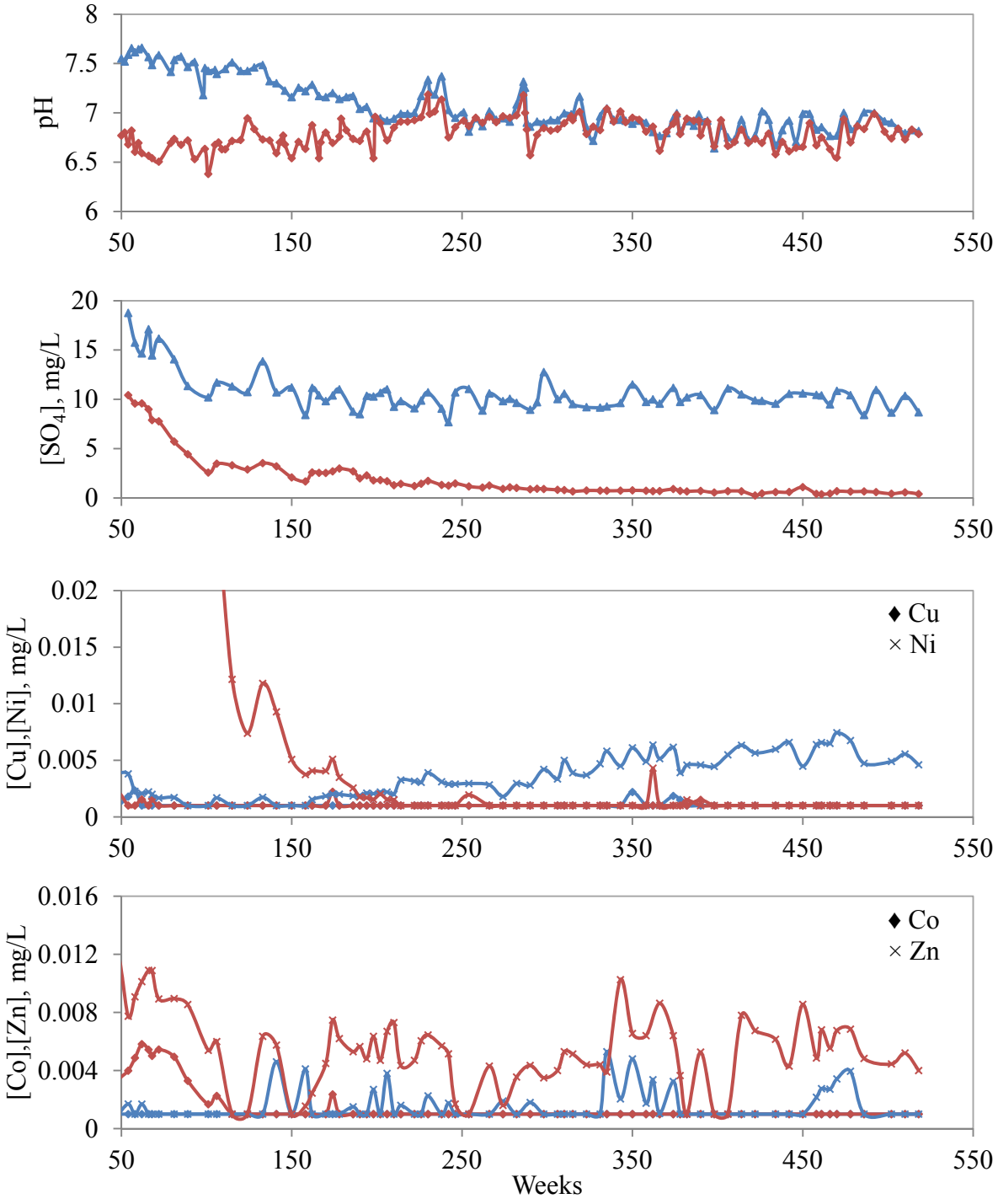


Figure 14. Sulfate release rates per gram sulfur remaining vs. time for reactors and humidity cells. Weeks 0-5 were excluded to improve resolution.

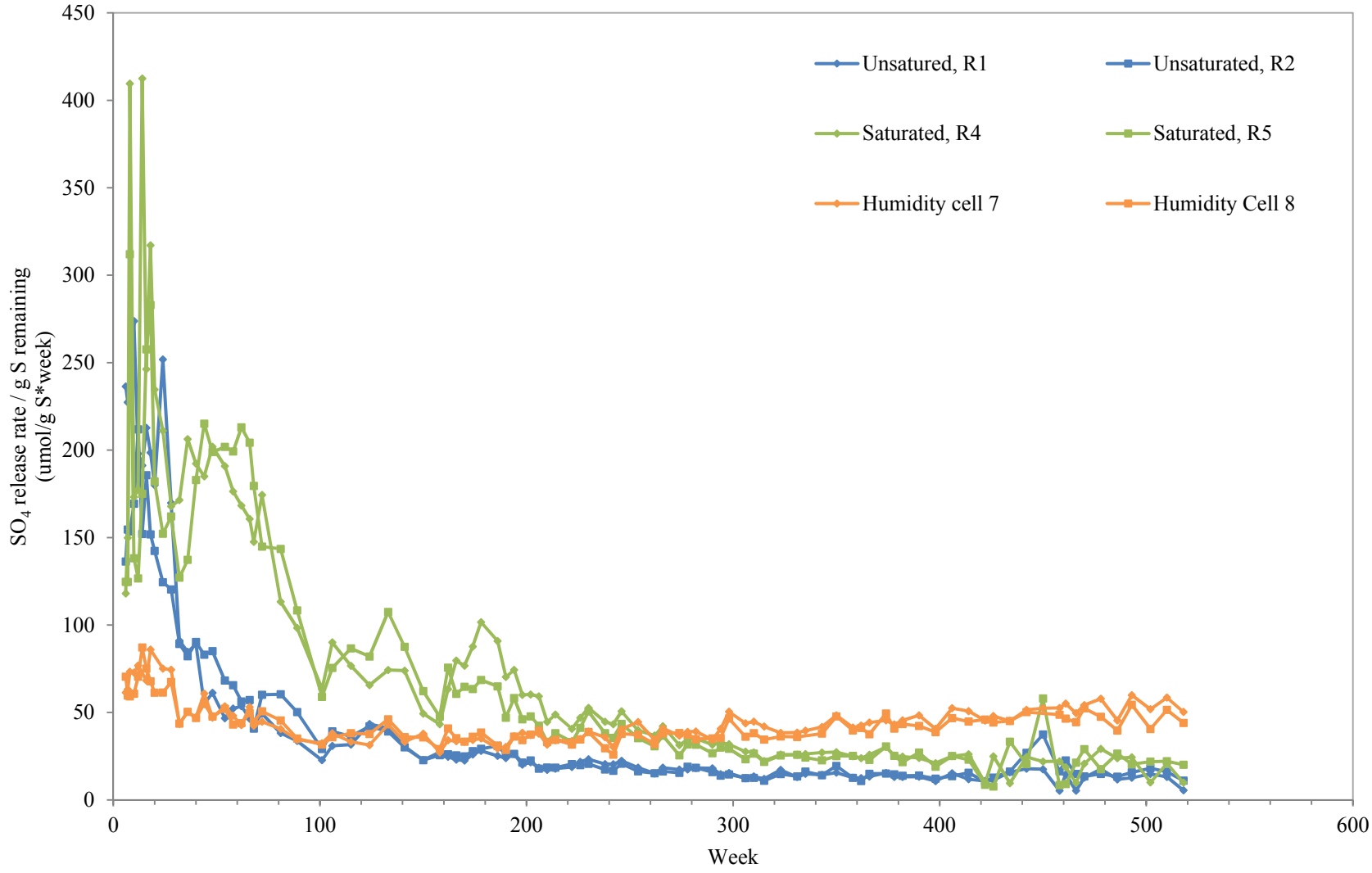


Figure 15. Calcium, alkalinity, and sulfate release rates vs. time for reactors, weeks 0-50.

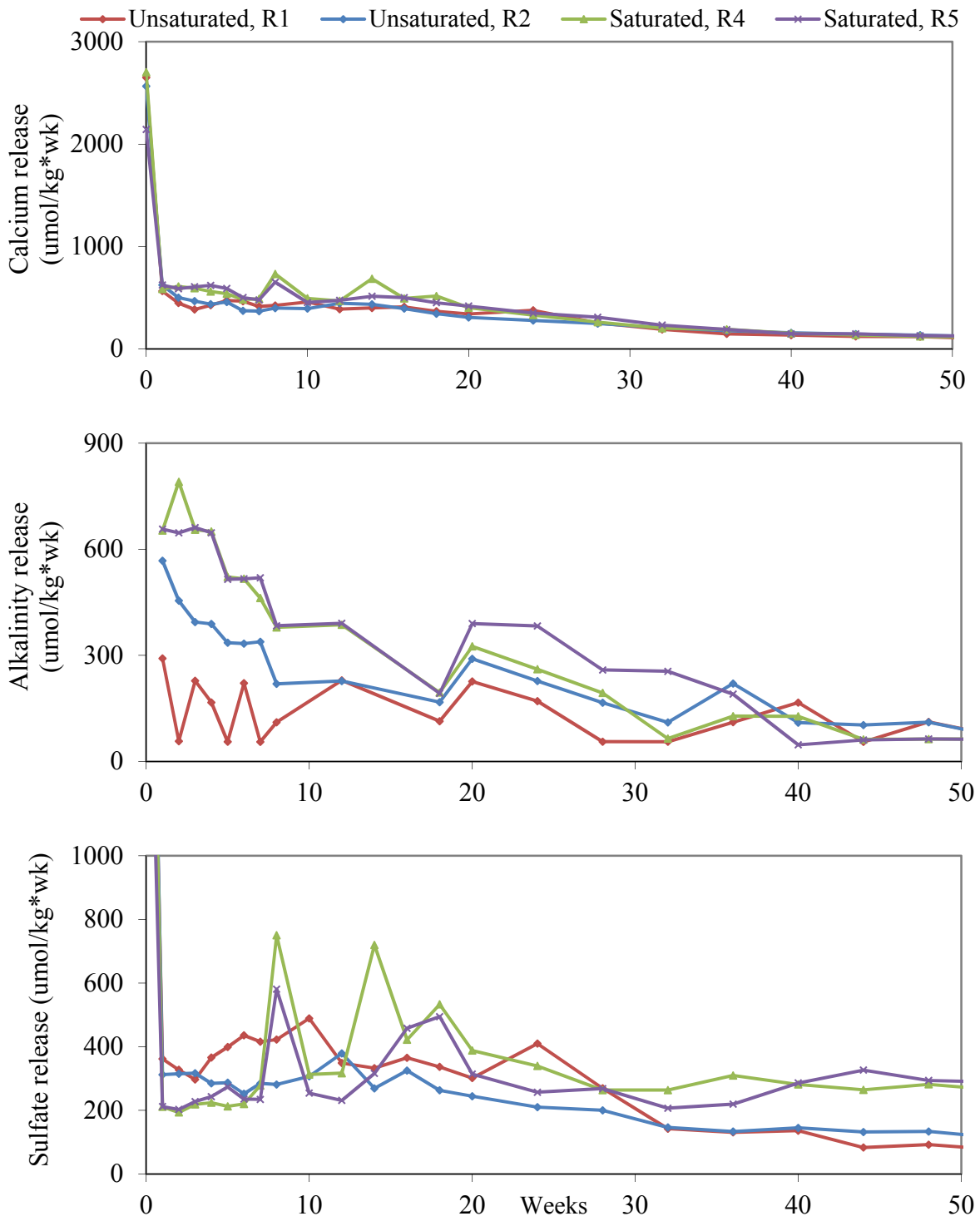
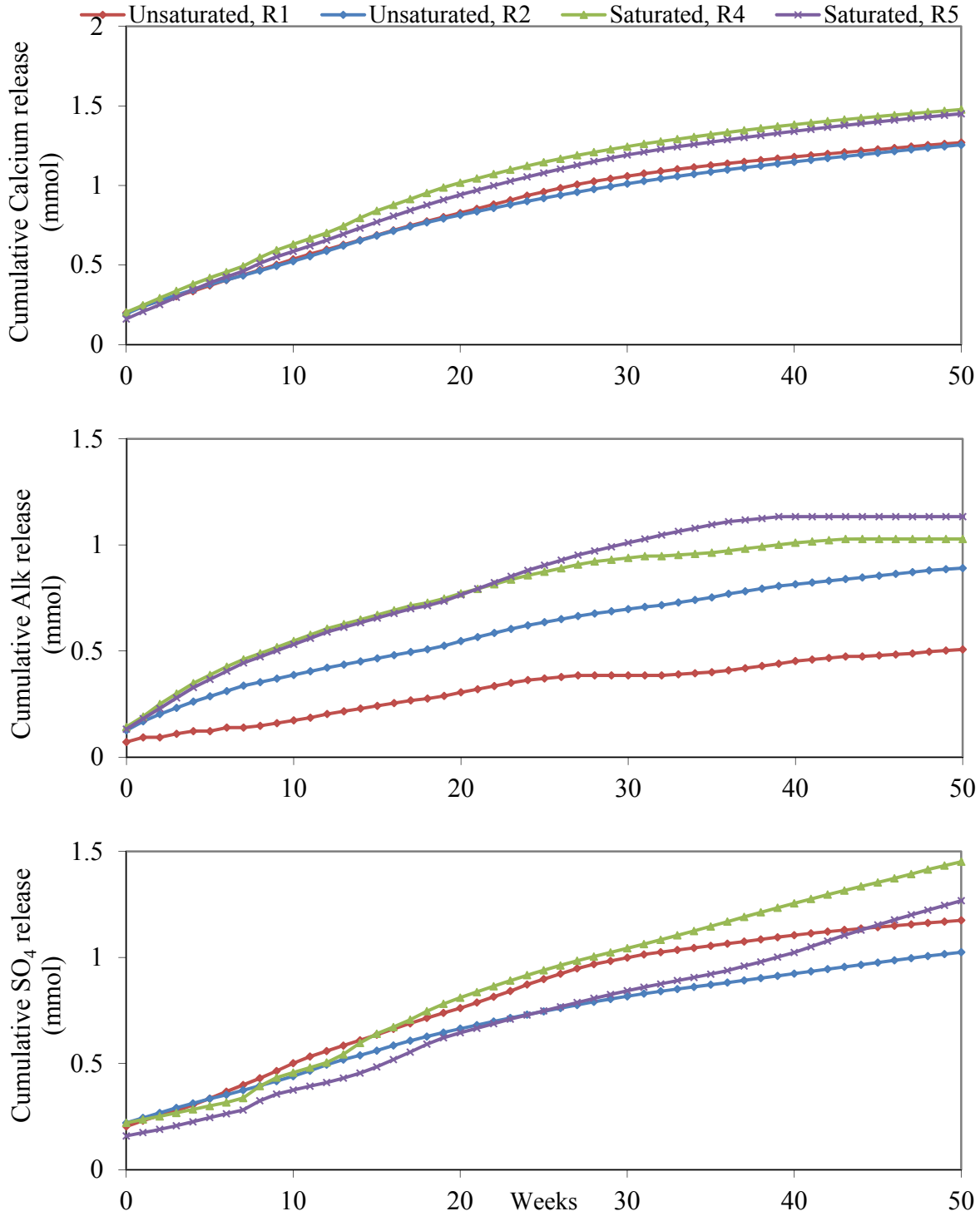



Figure 16. Calcium, alkalinity, and sulfate cumulative mass release vs. time for the 75 gram reactors, weeks 0-50. Assumed alkalinity equaled zero once it reached detection limit between weeks 40-50.