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Nicolet Minerals
a Rio Algom company

Rhinelander Office:
7 N. Brown St., Third Floor
Rhinelander, WI 54501-3161
715.365.1450 tel
715.365.1457 fax

Crandon Office:
104 W. Madison St., P.O. Box 336
Crandon, WI 54520-0336
715.478.3393 tel
715.478.3641 fax
www.crandonmine.com

March 21, 2001

Mr. Bill Tans
Wisconsin Department of Natural Resources
Bureau of Integrated Science Services
101 South Webster Street
P.O. Box 7921
Madison, WI 53707-7921

Mr. Robert Whiting
U.S. Army Corps of Engineers
St. Paul District
190 5th Street East
St. Paul, MN 55101

Dear Mr. Tans and Mr. Whiting:

Re: Crandon Project - Mine Permit Application

Nicolet Minerals Company (NMC) is pleased to submit the enclosed updated pages for the Crandon Project's *Addendum No. 1 to the Mine Permit Application - Reflooded Mine Management Plan* (RMMP). The update has been prepared on behalf of NMC by Foth & Van Dyke and Associates, Inc. to eliminate potential confusion regarding the use of the term "contingency" in the plan.

NMC has distributed the information to appropriate state and federal agencies, to local officials, and to various interested parties. It is our understanding that the Wisconsin Department of Natural Resources (WDNR) and the U.S. Army Corps of Engineers (USCOE) will be responsible for distribution of the updates to their appropriate staff members.

The enclosed pages are considered to be an update to the project's RMMP according to Items 1 through 7 on the attached Log of Updates, which serves as a reference identifying changes made to the RMMP by NMC throughout the permitting process. If additional revisions are made, they will be added to the Log of Updates in sequential order and forwarded with the changes.

If you or your staff have any questions regarding these updates, please contact me at (715) 478-3393.

Sincerely,

Gordon Reid, P.E.
Manager of Engineering

GR:cer1

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Nicolet Minerals
a Rio Algom company

Rhinelander Office:
7 N. Brown St., Third Floor
Rhinelander, WI 54501-3161
715.365.1450 tel
715.365.1457 fax

Crandon Office:
104 W. Madison St., P.O. Box 336
Crandon, WI 54520-0336
715.478.3393 tel
715.478.3641 fax
www.crandonmine.com

December 1, 2000

Mr. Bill Tans
Wisconsin Department of Natural Resources
Bureau of Integrated Science Services
101 South Webster Street
P.O. Box 7921
Madison, WI 53707-7921

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Ms. Char Hauger
U.S. Army Corps of Engineers
St. Paul District
190 5th Street East
St. Paul, MN 55101-1638

Dear Mr. Tans and Ms. Hauger:

Nicolet Minerals Company (NMC) is pleased to submit its *Addendum No. 1 to the Mine Permit Application - Reflooded Mine Management Plan* (RMMP), which details the process and methods by which NMC will address the regulatory requirements with respect to the closure and reflooding of the proposed Crandon Mine.

This document has been prepared on behalf of NMC by Foth & Van Dyke and Associates, Inc. As noted on the attached distribution list, NMC has distributed the information to appropriate state and federal agencies, to local officials, and to various interested parties. It is our understanding that the Wisconsin Department of Natural Resources (WDNR) and the U.S. Army Corps of Engineers (USCOE) will be responsible for distribution of the document to their appropriate staff members.

If you or your staff have any questions regarding the RMMP, please contact either Ken Black or myself at (715) 478-3393.

Sincerely,

Gordon Reid, P.E.
Manager of Engineering

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1	Dr. David Blowes 315 Hiawatha Drive Waterloo, Ontario Canada N2L 2V9
1	Mr. Anders Andren 3326 Valley Creek Circle Middleton, WI 53562-1988
1	Mr. Tuncer B. Edil Department of Civil & Environmental Engineering University of Wisconsin - Madison Madison, WI 53706
1	Mr. Timothy J. Grundl 506 Green Bay Road Thiensville, WI 53092
1	Mr. Bezalel C. Haimson Geological Engineering Program Department Materials Science & Engineering University of Wisconsin - Madison 1509 University Avenue Madison, WI 53706-1595

Distribution *(Continued)*

1 Mr. Alphonse E. Zanoni
Marquette University
Department Civil & Environmental Engineering
1515 West Wisconsin Avenue
P.O. Box 1881
Milwaukee, WI 53201-1881

Crandon Project Reflooded Mine Management Plan
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Entry Number	Date Revision Issued	Page(s)	Document/ Section Number	Description
1	3/21/01	1-9	Distribution List	Updated (place behind cover letter)
2	3/21/01	1	Log of Updates	New (place behind Distribution List)
3	3/21/01	i - ii	Executive Summary	Updated - replace existing pages
4	3/21/01	v - vi	Table of Contents	Updated - replace existing pages
5	3/21/01	1 - 5	Sections 1 & 2	Updated - replace existing pages
6	3/21/01	46 - 74	Sections 8 - 12	Updated - replace existing pages 46 - 76
7	3/21/01	—	Figure 8-1	Updated - replace existing figure

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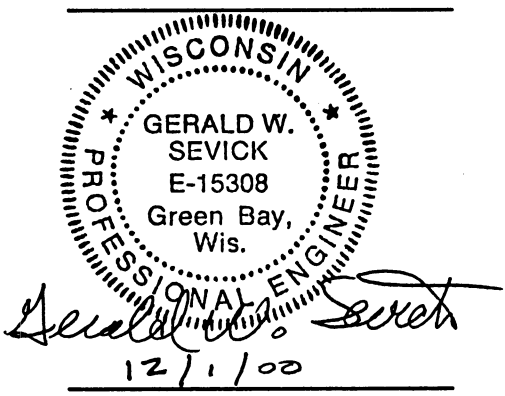
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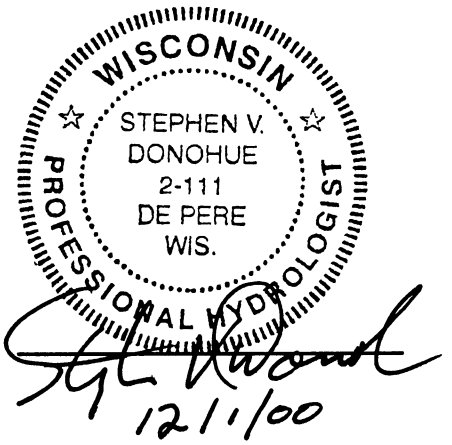
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Prepared for
Nicolet Minerals Company
7 North Brown Street, 3rd Floor
Rhinelander, WI 54501-3161

Prepared by
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December 1, 2000





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Reflooded Mine Management Plan

Executive Summary

The Wisconsin Administrative Code specifies that any proposed metallic mine must be designed, operated, and closed in such fashion as to protect Wisconsin's groundwater resources. This Reflooded Mine Management Plan (RMMP), incorporated as Addendum No. 1 to the project's *Mine Permit Application,* details the process and methods by which Nicolet Minerals Company (NMC) will address these requirements with respect to the closure and reflooding at the proposed Crandon Mine.

The RMMP focuses on those actions needed to protect groundwater quality after mine reflooding. It includes both a pre-closure and a post-closure component. The pre-closure component consists of features to be implemented during operations and closure. The post-closure component consists of measures that will be used to manage the reflooded mine by monitoring groundwater quality and implementing additional plan features as needed. Chapter NR 140 applies to each of these phases in different ways. With respect to the first, NR 140.22(1) requires that facilities be designed to minimize the concentration of substances in groundwater and to comply with NR 140 preventive action limits (PALs) to the extent technically and economically feasible. As to the second, the reflooded mine must be managed, through monitoring, in accordance with specific applicable NR 140 groundwater standards.

The underlying basis in the development of this RMMP is contained in NR 140.22(1), which states in part that "[f]acilities, practices, or activities regulated by the department . . . shall be designed to minimize the level of substances in groundwater and to comply with preventive action limits to the extent technically and economically feasible . . ." at the point of standards application. The RMMP details the steps NMC has taken to comply with the pre-closure and post-closure requirements of NR 140.

Technologies for minimizing the concentration of substances in groundwater were screened based on technical merit. Those technologies that were found technically feasible, or potentially feasible, were then screened again as to economic viability. Technologies that were deemed both technically and economically feasible were evaluated with a source concentration model and solute transport model to further assess each with respect to design objectives. Based on the results of this process, the following key design features were selected for implementation as part of the pre-closure component of the plan:

- Construction of drifts in the footwall and connecting the footwall drifts and hanging wall drifts at the western end of the mine. This passive hydraulic control will help in minimizing the concentration of solutes in the groundwater.

- Construction of oxygen barriers in the cross-cuts of each stope to reduce or eliminate the formation of oxidized-acidic backfill.

- Removal of such oxidized-acidic backfill, if present, at the end of mining.

- Cleaning the open access workings at the end of mining by physically removing localized areas of oxidized wall rock, physically removing spilled ore and muck, and by pressure washing the walls of the access workings.

- Accelerated reflooding of the mine to reduce the potential for additional oxidation of sulfide minerals.

After closure, the post-closure component of the plan will be implemented, consisting of monitoring groundwater quality to determine if, or when, either or both of the final two measures of the post-closure component of the plan are needed. These measures include a high-volume flushing system to remove solute mass emanating from sources in the open workings, and a hydraulic containment system to confine solute emanating from sources in the backfill.

An assessment of the resulting overall facility design was completed using the solute transport model. Results of that analysis indicate that, with the implementation of the pre-closure plan components, the solute concentration increases at the DMZ boundary will be below NR 140 PALs in the glacial overburden and strongly weathered bedrock. The glacial overburden and, to a lesser extent, the strongly weathered bedrock are the primary drinking water aquifers in the vicinity of the project. The glacial overburden is also the aquifer that directly interacts with surface waters in the vicinity of the project.

An assessment of the post-closure hydraulic containment system showed that, in the event post-closure monitoring determined hydraulic containment were needed, its implementation would control solute release such that solute concentrations in the moderately weathered, weakly weathered, and unweathered bedrock will be below PALs at the DMZ.

During the course of operations, the RMMP will be updated every five years based on an ongoing review of hydrogeologic data collected during operations and updated technical and economic assessments of design alternatives. The plan updates will be used to determine whether new technologies or methods yield better results at a manageable cost and, therefore, should become part of the then-current or final RMMP. The final RMMP will be submitted two years before the end of mining, and will include detailed evaluations of the then-available means and methods against the design requirements of NR 140.22(1), using the most current predictive models.

Reflooded Mine Management Plan

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1 Introduction

Nicolet Minerals Company (NMC) has evaluated and selected the best available means and methods for constructing, operating, closing, and monitoring its proposed underground mine in accordance with NR 140 and other applicable provisions of the Wisconsin Administrative Code. The purpose of the Reflooded Mine Management Plan (RMMP) is to document the design process used, and to present a comprehensive design for the reflooded mine. The RMMP encompasses activities to be performed during mine development, operations, closure, and during the post-closure period, resulting in the protection of groundwater resources. The plan accomplishes this by:

- Characterizing the reflooded mine with respect to potential impacts to groundwater quality at the site;
- Evaluating, selecting, and integrating the most effective currently available means and methods for minimizing such impacts;
- Reviewing applied and emerging technologies for potential future plan improvements;
- Monitoring and managing the mine after reflooding; and
- Implementing additional plan features as needed.

The Crandon Project *Mine Permit Application* (MPA) (Foth & Van Dyke, 1995/1998a) describes the underground mine development activities and reclamation plan for surface and underground mining facilities. The MPA also describes NMC's plan to combine pyrite rich tailings with cement and return the paste underground for use as backfill in mined-out stopes. Placement of the backfill in mined-out stopes will return those materials to the same environment, following reflooding, in which they existed for millions of years prior to mine development.

As groundwater is pumped from the mine, inward groundwater flow gradients will be established. Solutes cannot migrate out of the mine while those inward gradients exist. At the end of mining, groundwater will no longer be pumped from the mine and the access workings and backfilled stopes will resaturate as the water table recovers to pre-mining conditions. After water table recovery, local groundwater flow will return to a pattern very similar to pre-mining conditions.

An analysis completed by HSI GeoTrans, Inc. (HSI GeoTrans, 1999) indicates that under natural conditions, the water table in the vicinity of the mine will recover to within several feet of pre-mine conditions approximately four years after dewatering has ceased. The access workings themselves will likely resaturate within the first one to two years after the commencement of reflooding.

Chapters NR 132 and NR 182, Wis. Admin. Code, require the reflooded mine to comply with the groundwater quality provisions of NR 140. This report describes the measures NMC will undertake so that the reflooded mine will meet those requirements.

Given that reflooding of the mine will not occur until approximately 30 years after the commencement of mining, the reflooded mine design (design) presented in Section 8 of this report is largely conceptual in nature. NMC will submit updates to the design every five years after mining commences. In addition, two years prior to mine closure, NMC will submit, for Wisconsin Department of Natural Resources (WDNR) review, a detailed engineering design based upon the components discussed below. Once approved, the detailed engineering design will become the final RMMP for the Crandon Project.

The reflooded mine design encompasses a number of interrelated measures, including the control of solute generation during mining, the removal of soluble substances from various areas of the mine during closure, passive hydraulic controls (PHCs), and monitoring. The design also includes two features that will be implemented following closure if deemed necessary, based on groundwater monitoring results. The first feature is the removal of soluble substances from the mine workings. The second is hydraulic containment of soluble substances from the reflooded mine. In addition, the design includes provisions for ongoing evaluation of both site conditions and emerging technologies so that full advantage can be taken of information gained and techniques developed during the project's life.

The design is built upon several areas of scientific and engineering investigation. Major studies incorporated into this document include: (1) investigations of solute concentrations associated with various areas of the underground mine (SRK, 2000); and (2) solute transport modeling to estimate the potential migration of solutes to the boundary of the design management zone (DMZ) (HSI GeoTrans, 2000). The SRK and HSI GeoTrans reports referenced above address both prior written comments from the WDNR and issues raised during discussions at technical meetings with the WDNR in April and June 2000.

2 Regulatory Context

The Wisconsin Administrative Code requires, as a condition of permitting a new metallic mining operation, that the mine be designed, operated, closed, and monitored in such a way as to protect Wisconsin's groundwater resources. A 1998 amendment to Chapter NR 182 makes the groundwater quality standards of Chapter NR 140 applicable to such mining operations. The means by which the underground mine will be designed, operated, closed, monitored, and managed such that it will comply with the applicable provisions of NR 140 and NR 182 after reflooding are described in this RMMP. Compliance with other requirements (set out in Chapter NR 132 and elsewhere) regarding the development, operation, and management of the mine, including the specification of pertinent contingency measures, are dealt with in detail in the project's MPA (Foth & Van Dyke, 1995/1998a), *Environmental Impact Report* (EIR) (Foth & Van Dyke, 1995/1998b), and other permitting documents.

The reflooded mine design focuses on actions needed to protect groundwater quality after mine reflooding. It includes both a pre- and a post-closure component. The pre-closure component consists of those features and actions which will be implemented during the design, operation, and closure phases of the project to anticipate and minimize potential post-closure groundwater impacts. The post-closure component consists of measures that will be used to manage the reflooded mine by monitoring groundwater quality and implementing additional plan features, as needed. Chapter NR 140 applies to each of those components in somewhat different ways. With respect to the first, NR 140.22(1) requires that facilities be designed to minimize the levels of substances released to the groundwater and to comply with the Code's preventive action limits (PALs) to the extent technically and economically feasible. As to the second, the reflooded mine must be managed in accordance with specific applicable NR 140 groundwater standards through the monitoring required by NR 182.075.

2.1 Facility Design Under NR 140.22(1)

Section NR140.22 (1) provides in pertinent part that "[f]acilities, practices or activities regulated by the department . . . shall be designed to minimize the level of substances in groundwater and to comply with the preventive action limits to the extent technically and economically feasible . . ." at the point of standards application.

2.1.1 Initial Design and Management Plan

The primary design tools used to meet the objectives of NR 140.22(1) are an estimate of source concentrations and a solute transport model, described in more detail below. Those tools yield a measure of the effectiveness of design and management options. That information, in turn, allowed NMC to optimize the design by providing a basis for choosing those procedures and methods which best meet the requirements of NR 140. This methodology also provided NMC an important measure of technical feasibility for each option evaluated. Another component of the analysis was the estimation of projected costs for each technically workable option, and the

potential impacts on project viability. In combination, those steps provide a realistic selection of the best means available for meeting the design requirements of NR 140.22(1).

2.1.2 RMMP Updates

Mine reflooding will not occur until approximately thirty years after construction of the facility begins. NMC will continue the technical and economic assessment of facility design alternatives during that period. Such assessments will be detailed in RMMP updates which will be issued every five years throughout active mining operations. The updates will be used to determine whether new technologies or methods will yield better results at a manageable cost and, therefore, should become part of the then-current or final design.

2.1.3 Final Design

NMC will submit a final design two years before the end of mining. The final design will detail the means and methods selected for use during mine closure and the post-closure management period. In addition, the final design will include the results of the predictive modeling used to evaluate the then-available means and methods against the design requirements of NR 140.22(1).

2.2 Post-Closure Standards

Once reflooding begins, the focus of the design shifts to long-term monitoring and management, centered on the groundwater quality protection objectives of NR 140 (incorporated by reference in NR 182.075).

2.2.1 Monitoring

Chapter NR 140 sets out objective groundwater quality standards and the responses required in the event those standards are not met. The design details the post-closure monitoring program which will be used to evaluate the performance of the reflooded mine, and to determine if, or when, additional post-closure features of the plan need to be implemented.

2.2.2 Alternative Concentration Limits

The WDNR recognizes that in order to provide meaningful measures, groundwater standards may need to be adjusted to reflect background concentrations of the substances of interest (WDNR, 1994). The design anticipates that appropriate alternative concentration limits (ACLs) will be sought at the conclusion of pre-construction monitoring performed to establish baseline groundwater conditions. The RMMP also provides a means for determining whether an ACL should be established under NR 140.28 or NR 182.19 in connection with an exceedance disclosed by post-closure groundwater monitoring.

2.2.3 Compliance and Responses

Sections NR 140.24, 140.26, and 182.075 specify the required responses to groundwater quality standard exceedances in the event departures from expected conditions occur. As required by NR 182.075(1s), a contingency plan addressing the reflooded mine that identifies a range of appropriate response measures in the unlikely event such exceedances occur is included in Section 8 of the project's MPA (FVD, 1995/1998a).

3 Hydrogeologic Context

Groundwater in the vicinity of the proposed underground mine resides within two distinct hydrostratigraphic units: the glacial overburden and Precambrian bedrock. The glacial overburden, which overlies the Precambrian bedrock, consists of glacial till and outwash sand deposits, forming the primary drinking water aquifer within the study area. The glacial overburden, in general, is a highly permeable geologic formation that readily transmits appreciable quantities of meteoric water suitable for potable use, and is also the hydrostratigraphic unit that is directly connected to, and interacts most readily with, surface water resources. Surface waters in the vicinity of the project site act as either sources of recharge to the glacial overburden aquifer or as zones of groundwater discharge from that formation. Thus, protection of the groundwater in the glacial overburden, in accordance with the objective standards of NR 140, is of central importance to protecting the public health and welfare.

The Precambrian hydrostratigraphic unit consists of metavolcanic bedrock exhibiting a range of hydrogeologic characteristics. The hydraulic characteristics of the bedrock are primarily controlled by the degree of weathering which decreases with depth. As described in the *Environmental Impact Report* (EIR) (Foth & Van Dyke, 1995/1998b), the bedrock has been subdivided into four different units based on weathering intensity, and characterized as consisting of strongly weathered, moderately weathered, weakly weathered, and unweathered bedrock horizons. The transmissivity of the units declines with decreasing weathering intensity such that the strongly weathered bedrock is more transmissive than the deeper, less intensely weathered bedrock. In the vicinity of the project, the bedrock is not typically used as a source of drinking water, and does not directly interact with surface waters. Based on Figure 3.6-10A from the EIR (Foth & Van Dyke, 1995/1998b), the nearest potable wells penetrating the bedrock are over a mile away, located north of Swamp Creek and south of Mole Lake. It is improbable that those wells derive substantial water from any bedrock unit below the strongly weathered bedrock.

Water quality data from deep exploration holes at the site (Dames & Moore, 1978) suggest the deeper bedrock hosts areas of poor quality groundwater with naturally occurring elevated concentrations of total dissolved solids, chloride, and other solutes. Although that data is confined to the area around the Crandon formation, it also suggests that the water quality within the deeper bedrock naturally degrades with depth.

Overall, hydrogeologic data collected for the project indicates the following key points relevant to the reflooded mine design:

- The glacial overburden and, to a lesser degree, the strongly weathered bedrock are the principal hydrostratigraphic units used for potable water supplies.

- The glacial overburden is the only hydrostratigraphic unit directly connected to, and interacting with, area surface waters in the vicinity of the project.

- The deeper bedrock formations are not typically used as a source of potable water, and do not directly interact with surface waters in the vicinity of the mine.

- The quality of water hosted by the deeper bedrock units naturally degrades with depth.

4 Reflooded Mine Design Process

The development process for the RMMP is summarized in the steps shown in Figure 4-1. The specific steps are summarized as follows:

- Establish the design objective for the reflooded mine.
- The development of solute concentration and solute transport models to evaluate applicable engineering technologies that might be needed to meet the design objective.
- The identification and technical and economic screening of technologies that showed potential merit with respect to meeting the design objective.
- The testing and refinement of those technologies identified as having potential application to the reflooded mine design. Testing and refinement was completed using both the source concentration model and solute transport model. Based on the results of the testing and refinement process, the solute transport model was also refined to simulate the actual use of potentially applicable technologies.
- Finally, based on the results of the testing and model refinement step, technologies were selected that gave the best results with respect to meeting the design objective.

Each of these design steps are described in more detail in Sections 4.1 through 4.5.

4.1 Facility Design Objective

The objective of the facility design is to comply with NR 140.22(1): to identify the best available means and methods to minimize the levels of substances in groundwater and to comply with NR 140 PALs to the extent technically and economically feasible.

4.2 Application of Models to Design Process

In order to develop a design that meets the design objective, it was necessary to develop an understanding of the way in which solutes will be manifested in the reflooded mine, and how such solutes could be transported in groundwater. Once the geochemical and solute transport processes that control solute levels and transport were understood, it was possible to evaluate the merits of various potentially applicable technologies. Accordingly, a source concentration model and a solute transport model for the reflooded mine were developed.

The source concentration model provides estimates of potential solute concentrations associated with the reflooded mine. The solute transport model was constructed to represent the various solute source areas associated with the reflooded mine and to estimate the levels of solutes that could be expected in the groundwater downgradient from the reflooded mine. Results of the solute transport model were used to identify key source areas that affect the level of substances in

groundwater. From those results, key processes that affect solute concentration levels in the groundwater were identified and used to evaluate the applicability and potential effectiveness of various available technologies. For example, the source concentration and solute transport models indicated that a potential source of dissolved metals could be oxidized-acidic backfill resulting from the possible oxidation of backfill exposed to atmospheric oxygen in the cross-cuts. Accordingly, oxygen barriers in the cross-cuts and other control measures, such as removal of oxidized-acidic backfill, were evaluated as a means to minimize, or eliminate, solute contributions from this source. The solute transport model was also used to evaluate various hydraulic controls that could be used to minimize the concentration of solutes in groundwater.

4.3 Screening of Available Technologies

Various technologies available for potential incorporation into the design were screened in parallel to, and integrated with, the development of the source concentration and solute transport models. The first step in the screening process was the identification of potentially available technologies. The technical feasibility of each was assessed based on six criteria described in Section 5. The economic feasibility of the technologies surviving the technical screening was then evaluated. Those technologies identified as being both technically and economically feasible were carried forward for further assessment.

4.4 Technology and Model Testing and Refinement

Technologies deemed both technically and economically feasible were tested with respect to the relative effect they would have on meeting the design objective by using the source concentration and solute transport models. At this stage in the process, final refinements to the models were also made to appropriately represent the technologies being evaluated.

4.5 Selection of Technologies and Design Assessment

Based on results from the testing and model refinement process, technologies were selected that gave the best result with regard to minimizing and controlling the level of substances in groundwater and meeting PALs to the extent technically and economically feasible, as stipulated in NR 140.22(1). Once this process was completed, the overall plan was then evaluated with respect to its ability to meet long-term goals of protecting groundwater quality. The results of that evaluation are provided in Section 9.

5 Technology Evaluation

A key element in the development of the reflooded mine design was the identification and evaluation of technologies for reducing solutes at the source; removing them from solution; or controlling their movement. The process used to complete the evaluation involved identifying and categorizing technologies that had the best potential of meeting the design objective, assessing the technical merits of those technologies against a series of screening criteria, assessing the economics of those technologies that survived the technical screening, and, finally, selecting the technologies to be evaluated further in Sections 6, 7, and 8.

5.1 Identification of Technologies

The *Reflooded Mine Source Concentration Report* (SRK,2000) (Appendix A) evaluated four potential sources of solutes to the reflooded mine.

- Cemented pyrite concentrate used to backfill mined-out stopes;
- Exposed wall rock in the access workings;
- Unmined ore, including the crown pillar; and
- Combustion byproducts, including blasting residues.

The analysis determined that combustion byproducts are a de minimus source, warranting no additional consideration. To meet the design objective, it was deemed desirable to identify a series of technologies that had the potential to address, either individually or collectively, the solutes which could be generated by each of the remaining three sources.

Thirteen different technologies were identified as candidates for potential incorporation into the design. These technologies fall into three general categories, shown in Table 5-1. Source reduction technologies are those that could be employed to prevent the generation of solutes in the first instance, or to remove or immobilize them before they are placed underground. Source control technologies are those that could be implemented underground to remove soluble substances from the mine prior to reflooding, or to passively (i.e., *in situ)* remove them from solution after reflooding has occurred. Hydraulic controls are those technologies that could be used to: (1) control the movement of groundwater through the backfilled paste; (2) promote the mixing of water in the area from which solutes will move; (3) actively remove solutes from the access workings; or (4) actively contain solutes within a specified zone around the reflooded mine.

Table 5-1

Evaluated Technologies

Category	Technology
A. Source Reduction	Accelerated Reflooding Alternative Binders Chemical Precipitation Oxygen Barriers Backfill Pore Water Replacement
B. Source Control	Oxidized-Acidic Backfill Removal Biological Reduction Chemical Reduction Cleaning of Open Access Workings Reactive Barriers
C. Hydraulic Controls	Containment High-Volume Flushing of Workings Passive Controls

Prepared by: JWS
Checked by: SVD1

The evaluation completed as part of this report necessarily focused on those technologies which are sufficiently developed to permit thorough assessment. Because research in these fields is dynamic, new (and perhaps better) solutions may ultimately be found among emerging technologies yet to be fully developed. For example, studies are underway regarding various means to encapsulate pyritic tailings, and for using zeolite amendments in solid matrices to adsorb metallic ions. Emerging technologies with potential application to this plan will be evaluated as part of the continuing technology assessments included in the regular 5-year plan updates.

5.2 Screening Criteria

Table 5-2 lists the six criteria that were used to assess which of the identified technologies possessed sufficient technical merit to warrant being carried forward to the economic screening portion of the evaluation. The criteria address the state of the evolution of the technology (i.e., Proven); the ability of the technology to address all the substances of concern (i.e., Universal); the lack of secondary impacts associated with the implementation of the technology (i.e., Secondary Impacts); constraints on successful deployment of the technology in the field (i.e., Implementation Viability); the technology's ability to perform over the long-term once deployed (i.e., Long-Term Effectiveness); and the ability of the technology to perform over the long-term with little to no operation or maintenance (i.e., Long-Term Maintenance).

Table 5-2

Technical Screening Criteria

Criteria	Definition
A. Proven	Technology has been (or could easily be) applied at the production level, is not experimental, and would only require testing to define specific design parameters.
B. Universal	Technology would address design objectives with respect to all solutes of concern.
C. Secondary Impacts	Technology would not result in an increase of other parameters of concern or create unsafe working conditions.
D. Implementation Viability	Technology can be successfully applied to the Crandon Project to achieve the desired result.
E. Long-Term Effectiveness	Technology will be effective over the long-term.
F. Long-Term Maintenance	Technology requires little to no operation and/or maintenance over the long-term.

Prepared by: JWS
Checked by: SVD1

5.3 Description, Application and Screening

This section describes each of the technologies identified in Table 5-1, their potential application to the Crandon Project, and an evaluation of their ability to meet the screening criteria. This section concludes with a selection of those technologies that were carried forward for an evaluation of economic feasibility.

5.3.1 Accelerated Reflooding

5.3.1.1 Description and Application

The amount of time the access workings will be exposed to an oxidative environment can be reduced if reflooding of the mine were to occur at a faster rate. Reflooding will occur naturally as groundwater continues to flow into the mine after pumping ceases. To speed up the reflooding process, additional water could be delivered to the mine from the project's planned mitigation well via an access pipe constructed through the mine's east ventilation shaft.

5.3.1.2 Technical Screening

The matrix presented in Table 5-3 shows that accelerated reflooding meets or substantially meets all six of the screening criteria and, thus, the technology is deemed technically feasible. The technology is well developed and could be easily applied. Its effectiveness would cover all parameters of concern related to the oxidation of sulfides. There would be no secondary impacts related to the objectives of the design. Application of the technology would, however, result in greater volumes of water being extracted from the mitigation well than originally envisioned. Thus, a limitation on the concept is that water could only be delivered to the mine at rates that do not conflict with the need to provide mitigation water for other elements of the project.

The technology could be successfully deployed to shorten the reflooding period, thereby reducing the time that open workings are exposed to oxygen after closure activities were completed. It is estimated that application of the concept could reduce reflooding time for the workings by 50 percent, or to less than one year.

Since the objective of applying the technology will be to reduce the total mass of oxidized material available for dissolution upon reflooding, it would be effective in the long-term; although the estimated net effect is difficult to quantify. Finally, the technology would be applied over a short and finite time period and would not require any long-term operation or maintenance.

5.3.2 Alternative Binders

5.3.2.1 Description and Application

As described in the project's MPA (Foth & Van Dyke, 1995/1998a), NMC proposes to add Portland cement to the pyrite concentrate as a binder to provide structural stability within the backfilled stopes. This cement was identified as a potential contributor to the soluble sulfate levels measured in waste characterization work (SRK, 2000) performed for the project. There are several possible sources of sulfate in cement, including gypsum and various forms of potassium and sodium sulfate (Scheetz, 2000).

Per Scheetz (2000), gypsum in Portland cement performs two functions in the cementation process:

1. It is used as a grinding aid in the clinker grinding process; and,

2. It serves as a set regulator to control the hydration of tri-calcium aluminate (C_3A) by the formation of ettringite ($Ca_3Al_2O_6 \cdot 3CaSO_4 \cdot 32H_2O$).

Table 5-3

Technical Screening Matrix

	Proven	Universal	Secondary Impacts	Implementation Viability	Long-Term Effectiveness	Long-Term Maintenance
A. Source Reduction						
• Accelerated Reflooding	✓	✓	✓	✓	✓	✓
• Alternative Binders	—	O	—	—	✓	✓
• Chemical Precipitation	—	O	—	—	✓	✓
• Oxygen Barriers	✓	✓	✓	—	✓	✓
• Backfill Pore Water Replacement	✓	—	✓	—	✓	✓
B. Source Control						
• Oxidized-Acidic Backfill Removal	✓	✓	✓	✓	✓	✓
• Biological Reduction	O	O	—	—	O	✓
• Chemical Reduction	✓	O	—	—	O	O
• Cleaning of Access Workings	✓	✓	✓	—	✓	✓
• Reactive Barriers	O	O	—	—	O	✓
C. Hydraulic Controls						
• Containment	✓	✓	✓	✓	✓	O
• High-Volume Flushing of Workings	✓	✓	✓	✓	✓	—
• Passive Controls	✓	✓	✓	✓	✓	✓

Legend:
✓ = Technology meets or substantially meets criterion.
— = Insufficient evidence that technology meets the criterion.
O = Technology does not meet the criterion.

Prepared by: JWS
Checked by: SVD1

During manufacture, the sulfate content of Portland cement is optimized to produce the best 28-day strength and the least expansion in both air and water (ASTM C 563). Standard chemical requirements for Portland cement (ASTM C 150) limit the sulfate content to between 2.3 percent and 3.5 percent by weight, depending upon the type of Portland cement (Scheetz, 2000).

Sulfate can also be introduced into Portland cement in the form of alkalis such as potassium and sodium sulfate compounds (Scheetz, 2000). Such alkalis can be present in the feedstock used in the manufacturing of the cement. The amount of sulfate-rich alkalis present in Portland cement will vary depending upon their content in the feedstock.

If it were determined that gypsum- or sulfate-rich alkalis are likely to be significant contributors of sulfate to the backfill pore water, two alternative types of cements (i.e., blended or calcium aluminate based) might be used as alternative binding agents for the paste backfill.

Per Scheetz (2000), blended cements are widely used in construction for "hot" weather concrete placement. Although a 35 percent reduction in gypsum is possible in blended cements, some gypsum is still necessary for set control. Typically, slags used for the blending application come from the metal industry, and contain both elemental sulfur and sulfide phases, both having the potential to oxidize and release sulfate into pore fluids. Because gypsum is still present in this material, and because elemental sulfur and sulfide may also be contributed by such materials, it is unclear if blended cements would result in any substantial improvements over Portland cement.

As noted in Scheetz (2000), a class of calcium-aluminate based cements is manufactured in the United States as a low-sulfate alternative to Portland cement. This class of cement is manufactured with no addition of gypsum to the grinding process. Sulfates are reported for these cements in the range of 500 mg/kg to 2000 mg/kg, resulting from carryover from the raw feed material.

5.3.2.2 Technical Screening

The matrix in Table 5-3 shows that alternative binders could meet or substantially meet two of the six screening criteria. In regard to the proven criterion, low-sulfate cement is available on the market, but it is not clear whether the reduction in the sulfate content in a low-sulfate cement would be sufficient to meet plan objectives. In addition, further testing would be needed to determine if low-sulfate cement characteristics would meet the needs of the planned paste system from material handling, setup and strength standpoints.

An alternative binder would target only sulfate. The mass of other solutes of interest, including metals, would not be affected by the use of a low-sulfate cement. For this reason this technology does not meet the universality criterion.

It is unknown whether alternative binders would provide necessary strength and handling characteristics, or result in other undesirable secondary impacts. Implementation viability of

alternative binders is dependent upon their rheological and strength characteristics and is considered uncertain. If the desired sulfate reduction in the paste due to the use of an alternative binder were achievable, the technology would be effective over the long-term. The technology would be applied over a finite time frame and, therefore, would require no long-term operation or maintenance. Despite the uncertainties described above, this technology appeared to have sufficient potential technical merit to warrant further evaluation.

5.3.3 Chemical Precipitation

5.3.3.1 Description and Application

Selected solute concentrations in the paste backfill pore water might be reduced through chemical precipitation. The concept involves the addition of a selected reagent(s) to the paste to precipitate targeted solutes. Typically, a single reagent will precipitate one select or a limited number of solutes. Given the paste pore water matrix, multiple reagents may be needed if chemical precipitation were to be pursued for application as part of the design. For illustrative purposes, typical sulfate and metal precipitation processes are discussed below. Chemical precipitation will not be effective for highly soluble ions, such as chlorides, nitrates, and nitrites, since a practical reagent is not available to precipitate those substances.

Chemical precipitation of sulfate can be accomplished, for example, by adding barium in the form of a chloride, hydroxide, or oxide to the paste backfill. Sulfate in the backfill pore water will bond preferentially with barium over calcium, resulting in the formation of barium sulfate, which is far less soluble than calcium sulfate.

Some metals can be precipitated using a sulfide. Sodium sulfide could be added to the backfill paste to form a metal sulfide precipitate. The low solubility of such precipitates would reduce the amount of the metal ions available in the pore water matrix.

Each of the above chemical precipitation processes would require the addition of appropriate reagents either before or at the pug mill in the project's paste backfill plant. Equipment needs would consist of reagent storage tanks, solution tanks, chemical feed pumps, piping and controls.

While the reagents used in the precipitation process would reduce the concentration of the targeted solutes, they would also result in an increase in other solutes such as chloride or sodium. Barium hydroxide addition would increase the pH of the pore water matrix, possibly resulting in the dissolution of other soluble substances. In addition, the precipitation of targeted compounds can also be affected by other solutes present in the backfill pore water, or which may be added with other reagents.

Neither operational data nor pilot-scale data exist to determine whether these chemical precipitation processes, when applied to the paste matrix, could actually achieve desired solute reduction efficiencies. The technologies are considered developmental at this time, and would

require detailed geochemical analyses and testing at a production scale to verify their applicability to the project.

5.3.3.2 Technical Screening

The matrix in Table 5-3 shows that the screening results for the chemical precipitation technology are the same as those for alternative binders. While the chemistry of the precipitation processes is well understood, their application in a backfill pore water matrix is not. Therefore, it is uncertain if the technology meets the proven criterion. As with the alternative binder option, chemical precipitation is targeted at specific solutes and, therefore, would not be universal.

Chemical precipitation could also affect the concentration of other solutes of concern in the backfill pore water matrix. Therefore, it is unknown if the technology will meet the secondary impact criterion. Given the lack of field application data, it is unclear or uncertain whether the technology would meet the implementation viability criterion. If the technology were found to be applicable, it likely would be effective in the long-term. Finally, no long-term operation or maintenance would be required as part of its implementation, since the technology would be employed over a fixed time frame during mine operation. Despite the uncertainties described above, this technology appeared to have sufficient potential technical merit to warrant further evaluation.

5.3.4 Oxygen Barriers

5.3.4.1 Description and Application

A principal potential source of solutes in the mine is located in those areas where the paste backfill could be exposed to oxygen for long periods of time before reflooding. Such exposure is most likely to occur at the base of each backfilled stope where the cross-cut access to the stope intersects the paste backfill. To prevent or minimize the amount of oxygen that can come into contact with the exposed paste, a barrier to oxygen transport could be constructed in each cross-cut. To achieve maximum effectiveness, each barrier would need to be substantially airtight. The barriers could, for instance, be constructed of a wood frame with an impermeable cover designed to inhibit both air movement and diffusive transport of oxygen. Care would need to be taken in the construction of the barriers to effectively seal the contact between the barrier and the wall rock. It may also be necessary to seal the pore space in the wall rock adjacent to the barrier if the rock is highly fractured.

5.3.4.2 Technical Screening

As shown in Table 5-3, oxygen barriers meet five of the six technical screening criteria and, thus, the technology is deemed technically feasible. Since the technology is readily available, it meets the proven criterion. By preventing or limiting oxidation of the paste, this technology is expected to limit the amount of all solutes resulting from sulfide oxidation and, therefore, it meets the universality criterion. The technology has no known secondary impacts. Since the

technology has not been applied in the field as envisioned above, the ability to effectively deploy it is uncertain. Because the oxygen barriers are needed only until reflooding begins, they would likely be effective in the long-term and would require no long-term operation or maintenance.

5.3.5 Backfill Pore Water Replacement

5.3.5.1 Description and Application

The paste backfill will have a density of 84 percent total solids, with the remaining 16 percent consisting of pore water. Because the source of this pore water is the mill process water, it will contain concentrations of solutes similar to those present in the process water. It may be possible to reduce solute concentrations in the paste backfill pore water by replacing a sizeable volume of the residual pore water with water having significantly lower solute concentrations.

The mill will generate pyrite concentrate at 25 percent total solids by weight. Per the mine plan, a paste production system will be used to increase the solids content of the pyrite concentrate in the paste backfill plant. If pore water replacement were to be implemented, a different slurry thickening process would likely be required than that currently specified in the mine plan. Such a process would initially consist of increasing the solids content of the pyrite concentrate slurry using a gravity thickener. From the thickener, the slurry would be further concentrated by filtration. The pore water replacement would occur as the slurry is passed through the filtration step. Either a belt filter or a filter press could be used for the final concentration and water replacement stage. Both filters are capable of achieving the required final concentration of 84 percent total solids. The ultimate solute concentration reduction achievable in the filtration process would largely be a function of the amount of replacement water used and the initial solute concentration in the replacement water.

In order to minimize impacts to the water balance and to maximize reuse of the process water, a treatment system independent of the project's wastewater treatment facilities (WWTF) would be needed. That treatment system would consist of clarification, filtration, and reverse osmosis. The effluent from the clarifier would be directed to a sand filter, and then to the reverse osmosis system. The reverse osmosis permeate would be used as the replacement water, with the concentrate directed to the WWTF evaporator.

While the concentration of solutes in the backfill pore water would be reduced by replacement of the pyrite concentrate pore water, as described above, dissolution of some constituents existing in the solid phase may also occur as a result of the reduction of those solutes. For example, calcium and sulfate may be present as solids in the backfill because gypsum is an important component of normal Portland cement. Based on the project's waste characterization analyses, it appears that – in order to reduce the sulfate concentration in the backfill – it may be necessary to address the sulfate contribution from the cement used to produce the cemented paste backfill in addition to reducing the sulfate concentration in the pyrite concentrate pore water itself. Highly soluble substances such as chlorides, nitrates and nitrites are not expected to be present as solids in the backfill and, thus, could be expected to be removed by the backfill pore water replacement.

5.3.5.2 Technical Screening

The matrix in Table 5-3 shows that the backfill pore water replacement technology meets four of the six screening criteria. The individual processes involved are proven, although they have not been applied in the field as described above. Given the resolubilization issue, it is uncertain whether the technology is universal. As related to the design objectives, however, the technology appears to have no secondary impacts. Because of the resolubilization issue and the potential need to implement the technology in conjunction with an alternative binder, the viability of implementation is unclear or uncertain. Since the technology involves removing the mass of soluble substances from the system, it would be effective over the long-term. Finally, since the technology would not require operation or maintenance beyond the active mining period, it would meet the long-term maintenance criterion. Despite the uncertainties described above, this technology appeared to have sufficient potential technical merit to warrant further evaluation.

5.3.6 Oxidized-Acidic Backfill Removal

5.3.6.1 Description and Application

A principal potential source of solutes in the reflooded mine is any exposed paste backfill that has oxidized and becomes acidic prior to the initiation of the reflooding process. Section 5.3.4 above discussed a source reduction technology for preventing or minimizing such oxidation. A source control measure that could be employed is the removal of oxidized-acidic backfill from the mine prior to reflooding.

Such a removal process would commence at the end of mining. The first step would involve removing the oxygen barriers in the mine cross-cuts. The exposed paste backfill could then be tested to determine how much, if any, oxidized-acidic backfill is present. If such material is present, it could be removed using a number of methods, such as high-pressure washing or by mechanical means. If high-pressure washing is used, the wash water would be collected, pumped to the surface and directed to the project's WWTF. Solids generated by the removal process would also be brought to the surface, where they would either be processed at the mill or taken off-site to a licensed facility.

5.3.6.2 Technical Screening

As shown in the matrix contained in Table 5-3, removal of the oxidized-acidic backfill meets all six screening criteria and, thus, this technology is deemed technically feasible. The removal processes that would be employed are simple and proven. Removing the acidic material also targets all solutes of interest and, therefore, would be universal. There are no known secondary impacts related to the implementation of the technology. It is very likely the technology can be successfully deployed to achieve the desired results. Once the oxidized-acidic backfill has been removed from the mine, the mass of soluble substances associated with it is no longer available for dissolution and transport. Therefore, the technology would meet the long-term effectiveness

criterion. Since the technology would not require operation or maintenance beyond the closure period, it would also meet the long-term maintenance criterion.

5.3.7 Biological Reduction

5.3.7.1 Description and Application

Biological reactions could potentially be used to reduce, through *in situ* precipitation, soluble concentrations of selected anions and/or metals in the reflooded mine. The biological reduction process involves the use of sulfate-reducing bacteria to reduce sulfates and precipitate metals. Sulfate is commonly reduced in nature by two genera of bacteria - *Desulfovibrio* and *Desulfotomaculum* - under anaerobic conditions. Those bacteria oxidize organic material, using sulfate as an electron acceptor, to gain energy for cellular growth. The by-products of bacterial sulfate reduction are hydrogen sulfide and alkalinity. The hydrogen sulfide reacts *in situ* with dissolved metals to precipitate metal sulfides. Some metals may also precipitate as metal hydroxides as a result of the increased alkalinity that is needed for the process to function. The following elements are needed for bacterially mediated sulfate reduction to function:

- exclusion of oxygen;
- a source of sulfate;
- a source of organic carbon;
- a means by which to physically retain the metal sulfide precipitates; and
- a means to provide sufficient alkalinity to maintain a pH>5, so that the solubility of metal sulfide is decreased and the sulfate-reducing bacteria are not inhibited.

Providing a readily usable source of organic carbon and inoculating it with bacteria can encourage this natural process. Typical organic carbon sources are wood, sawdust, or hay. Those particular sources are used to provide organic compounds over longer time periods via dissolution and breakdown by bacteria. Researchers have used sediments from known sulfate reducing areas for inoculation.

Literature on biological sulfate reduction indicates that the following factors need to be considered:

- The supply and type of organic matter used will dictate the rate and longevity of bacterial sulfate reduction. There must be sufficient organic carbon and sulfate to promote bacterial growth.

- Sufficient residence time is required to achieve significant sulfate reduction. Studies to date involved well-mixed systems. In mining scenarios, sulfate water, bacterial populations, and organic carbon are unlikely to be as thoroughly mixed. Therefore, the rate at which organic carbon and/or sulfate diffuses through water to the bacteria (depending on the location of the bacteria - on the walls or in the organic substrate) will be a key factor influencing the rate of sulfate reduction.

- Temperature has a significant effect on sulfate reduction rates. The optimum temperature is approximately 30°C to 37°C. The sulfate reduction rate decreases by approximately a factor of 2 for every 10°C decrease in temperature.

- Extreme water quality, namely acidic pH and increased metals concentrations, can inhibit bacterial growth. Sulfate-reducing bacteria work optimally at neutral pH. Metals can be toxic to bacteria at sufficiently high concentrations. In extreme water quality conditions, mixed bacteria cultures will likely be needed to sustain significant sulfate reduction. Organic carbon (as organic acids) is toxic to sulfate-reducing bacteria in low pH conditions.

- Sulfate reduction with metal sulfide precipitation will reduce many, but not all, metals.

- If there are insufficient metals to balance the hydrogen sulfide produced by the bacteria, sulfides may remain in a soluble form and escape into the pore water.

The literature presents evidence indicating bacterial sulfate reduction is occurring in underground workings at the Lokken Mine in Norway (Arneson, et al., 1991). Sulfate and metal concentrations in the mine water have been decreasing, and two sulfate reducing bacterial species have been identified. No organic material was introduced to the mine, so it appears that timber used in mine support is the only carbon source sustaining this process.

An important practical consideration relating to biological reduction is the ability to mix sulfate and metal bearing water with organic material. If biological reduction were to be considered for inclusion in the design, this mixing issue would need to be resolved.

5.3.7.2 Technical Screening

The matrix in Table 5-3 shows that biological reduction meets only one of the six screening criteria. The technology is considered to be in the early stages of development as related to its potential application in the RMMP and, therefore, is not proven. The technology, while focused on sulfate reduction and metal precipitation, would not address all metals of interest; nor would it address highly soluble ions such as chlorides. Therefore, it is not universal in application. There do not appear to be any potential secondary impacts associated with the technology, but it has not been developed to the stage of certainty in that regard.

It is also uncertain if the technology can be successfully deployed to achieve the design objective. Given that the sulfate source in the reflooded mine is likely to exist for many years and that a carbon source will be required over the same period, it is unlikely that the technology will be effective over the long-term. The technology meets the long-term maintenance criterion since it requires no operation or maintenance during the post-closure period. For the foregoing reasons, this technology was deemed technically infeasible at this time.

5.3.8 Chemical Reduction

5.3.8.1 Description and Application

Chemical reduction involves the post-mining underground addition of reduced sulfur in a chemical form (i.e., hydrogen sulfide and/or solid sodium bi-sulfide) to promote the return of specific metals to the sulfide form. The resulting metal sulfides would remain stable under flooded conditions. The metal precipitation chemistry is similar to that described above for the biological reduction concept. However, the chemical reduction process does not rely on the bacterial formation of hydrogen sulfide from sulfate in solution. The chemical reactions that form metal sulfide precipitates are rapid, but sulfate is not reduced as part of the chemical precipitation process.

The following elements are needed for chemical reduction to be effective:

- directing groundwater flow through the treatment area;
- presence of adequate reagents;
- mixing of groundwater in the access workings with reagents; and
- adequate residence time.

As applied to the design, reagent addition and mixing with groundwater exiting the backfilled stopes is likely to take place in the longitudinal drifts in the hanging wall. Considerations regarding mixing and adequate flow into the longitudinal drifts are similar to those discussed for biological reduction.

5.3.8.2 Technical Screening

A review of the matrix in Table 5-3 shows that the chemical reduction technology meets two of the six screening criteria. The chemistry of the technology is well established, and therefore proven. Because the technology is focused on the control of metals; it would have no impact on sulfates or other highly soluble anions. Therefore, it is not universal. Because there is a potential for secondary impacts caused by unreacted reagents or secondary byproducts, it is uncertain if the secondary impact criteria would be met.

Given the potential difficulty of delivering reagents to the proper location in the proper amounts, it is uncertain that this technology meets the implementation viability criterion. For the same reasons, the technology is unlikely to meet the long-term effectiveness criterion. Finally, given the need for continual reagent addition, the technology would not meet the long-term maintenance criterion. For the foregoing reasons, this technology was deemed technically infeasible at this time.

5.3.9 Cleaning of Open Access Workings

5.3.9.1 Description and Application

Another likely source of solutes to the reflooded mine is the products from oxidation of sulfides exposed in the wall rock in the open access workings. Such oxidation would take place only while the mine is dewatered. The resulting oxidation products would be available for dissolution once the mine is allowed to reflood.

One technology that can be used to remove readily soluble products before reflooding is to wash the walls of the open workings with a high-pressure hose. Wash water would be directed to the mine sumps for delivery to the project's WWTF. One of the disadvantages of this technology is that not all oxidation products are readily soluble. To address this issue, the walls could be visually inspected and areas with significant staining could be targeted for physical removal of the oxidized material. Material so removed would be hoisted to the surface for processing in the mill or for placement in a licensed facility.

Shotcrete (or another sealant) is a technology that could be used to reduce the amount of oxidized wall rock that may be generated, and could be applied to the wall rock in localized areas of the open workings to inhibit oxidation from occurring.

5.3.9.2 Technical Screening

Table 5-3 shows that this technology meets all but one of the six screening criteria and, thus, is deemed technically feasible. High-pressure washing is an established technology. The washing and cleaning process would, in combination, target all oxidation products. Therefore, the technology would be universal. The technology meets the secondary impact criterion. Since the technology has not been applied in the field as envisioned above, its ability to be effectively deployed is uncertain. Once the soluble substances are removed from the mine workings, the mass of solutes associated with it will not be available for dissolution and transport. Therefore, the technology would meet the long-term effectiveness criterion. Since the technology would be employed and completed prior to reflooding, there would be no long-term operation or maintenance needed.

5.3.10 Reactive Barriers

5.3.10.1 Description and Application

Porous walls of reactive material can be placed in the path of groundwater flow and used to remove solutes. The concept involves the installation of a porous wall of predetermined thickness into an area perpendicular to groundwater flow. A reactive agent is added to the wall during construction. The type of solute to be removed determines the agent used. Groundwater flows through the wall, and the solutes of concern are removed via reduction or adsorption.

Solutes removed using this technology at the laboratory and/or field-scale level include sulfate, heavy metals, hydrocarbons, and chlorinated hydrocarbons. The reactive agents used include organic material for sulfate and heavy metals, and iron filings for hydrocarbons, arsenic and chromium.

Organic matter can be effective in removing metals by acting as a sulfate reductant and as a metal adsorbent. For this technique, organic matter such as leaf mulch, wood chips, cellulose, or combinations thereof, is mixed with a bacterial innoculant and pea gravel and placed such that groundwater will flow through it. An alternative to pea gravel is BIOFIX beads, where a non-living biomass is immobilized into a porous polysulfone bead (Bennett, et al., 1991). Suitable organic material can adsorb metals from very dilute solutions, and is reported to work best at a pH range between 3.0 and 8.0. Metals are adsorbed in preferential order, with calcium and magnesium eventually crowded off the adsorption sites by other metals such as copper, zinc, iron, and manganese. The permeability of the barrier is designed to be similar to or greater than the surrounding area.

The mechanism of sulfate and heavy metal removal is identical to that described for biological reduction. Namely, sulfate reducing bacteria use sulfate as an electron acceptor in the oxidation of the organic matter, producing hydrogen sulfide and alkalinity as by-products. Metals precipitate as metal sulfides.

Factors that need to be considered as part of system design are:

- the need to funnel groundwater flow through the wall with minimal short-circuiting around the wall;
- the need to maintain high permeability in the wall to prevent pore blockage or short-circuiting;
- difficulty in replacing reactive materials should they become depleted;
- addressing uncertainties with respect to the effective longevity of the barriers due to potential fouling by metal or other precipitates;
- the release of exchanged elements, depending on the reactive agent used; and
- a potential reduction in effectiveness should the organic material dry out prior to reflooding of the mine.

The technical feasibility of using porous reactive barriers containing organic matter for the passive treatment of mine impacted water is currently being tested in a field trial at the Nickel Rim Mine in Canada. The porous reactive barrier was installed in the aquifer downgradient from a surface tailings impoundment. Reported field data indicate that sulfate and heavy metals are being successfully removed from the groundwater (Blowes, et al., 1995).

The reactive barrier technology was developed for relatively shallow aquifers using cut and fill trenching from the ground surface. If the technology were to be applied as part of the design, construction techniques would need to be developed to address the placement of porous reactive walls within underground drifts. Particular attention would need to be paid to the placement and permeability of the reactive walls so that groundwater flows from the mine would travel through the reactive wall and that such walls would not act as plugs.

Use of iron filings in a porous reactive barrier would be similar to other reactive agents. The filings would be mixed with pea gravel to maintain sufficient permeability, and with calcite to maintain neutral pH for chromium and iron precipitation. The physical constraints discussed above relating to organic material would also be applicable to the use of iron filings. Unlike the use of organic matter, iron filings do not require constant anoxic conditions; therefore, installation of iron filing walls would not be time sensitive. Design issues regarding construction technique, location, and permeability are identical to those for porous reactive barriers with organic matter. Iron filings would not have an effect on sulfate or non-redox sensitive species.

5.3.10.2 Technical Screening

Table 5-3 shows that the reactive barrier technology meets only one of the six screening criteria. The technology is experimental in underground applications, and therefore unproven. Because the technology cannot target highly soluble anions, such as chloride, it would not be universal. While no secondary impacts are expected, it is (due to the lack of field- scale data) uncertain whether such impacts would occur.

It is also uncertain if the technology can be successfully deployed, given the scale of the Crandon Project and variable post-reflood groundwater flow directions. Given that reactive walls would contain a finite source of reactant materials and that the solute source will be present for many years, it is unlikely that the technology will be effective over the long-term. The technology, however, would not require long-term operation or maintenance. For the foregoing reasons, this technology was deemed technically infeasible at this time.

5.3.11 Containment

5.3.11.1 Description and Application

Hydraulic containment is a technology that could be used to prevent the movement of solutes from the reflooded mine toward the DMZ. A containment system would consist of a series of low-volume extraction wells installed after closure around the perimeter of the mine to capture water passing through the backfill and access workings. Captured water would be treated either on-site or hauled to an acceptable off-site treatment facility. The containment system's primary purpose is to keep solutes confined to within a specific capture zone, rather than to remove solute mass from the groundwater system. A containment system would need to be operated for many years, but would effectively target all solute sources in the reflooded mine.

Based on the assumption that the surrounding bedrock is interconnected to a degree that containment wells can effectively capture water from the reflooded mine, a containment system would likely consist of less than ten wells with an aggregate extraction rate of less than 35 gpm. The bedrock interconnection assumption is consistent with the parameters used to develop the groundwater flow model for the Crandon Project. A possibility does exist that the surrounding bedrock system has a greater level of heterogeneity than is currently understood, and that it may not be practical to extract water from wells located around the mine due to less conductive conditions than currently believed. If such conditions are found to be present during mining, the containment wells could be located in such a way as to connect the hydraulic network formed by the reflooded mine workings themselves. Extraction rates under this scenario would be similar to those mentioned above.

5.3.11.2 Technical Screening

Table 5-3 shows that the containment technology meets five of the six screening criteria and, thus, is deemed technically feasible. Well construction and extraction technology are well developed, and therefore considered proven. The technology would be universal in that it would target all solutes contained in the groundwater. There are no known secondary impacts from the system. Treated water requiring discharge would need to meet applicable discharge standards. Solids removed from the treated water would be placed in a facility licensed to accept such material.

It is likely the technology can be deployed to achieve desired results. Thus, containment meets the implementation viability criterion. If the bedrock is not interconnected as expected, the alternative approach of installing extraction wells to interconnected access workings is also viable. Once the system were installed and operating, it would be effective over the long-term. Since the technology would need to be employed for many years following site closure, however, it does not meet the long-term maintenance criterion.

5.3.12 High-Volume Flushing of Workings

5.3.12.1 Description and Application

High-volume flushing of workings is a method available to remove soluble substances that could dissolve in the groundwater that will fill the open access workings as part of the reflooding process. This technology requires the installation of a series of wells located at the west end of the mine to extract water from the workings, the treatment of the extracted water to remove solutes, and the installation of a piping system located on the east end of the workings for reintroduction of treated water into the mine. When in operation, water would circulate from east to west in the direction of normal groundwater flow. The system would collect solutes generated by oxidized sulfides exposed in the wall rock of the open workings, solutes flushed from fractures, and some of the solutes generated in the cross-cuts and backfilled stopes.

If implemented, high-volume flushing of the workings would operate at a relatively high extraction rate, likely in the range of 300 to 400 gpm. One advantageous feature of the high-volume flushing technology, when compared to containment, is that it would only need to be operated until the desired reduction in solute mass is achieved.

5.3.12.2 Technical Screening

The matrix in Table 5-3 shows that the high-volume flushing of the workings technology meets five of the six screening criteria and, thus, is deemed technically feasible. The pumping, water treatment, and water delivery systems involved are well developed, and therefore proven. The technology would be universal in that it would remove all solutes of interest contained in the water extracted by the system. There are no known secondary impacts associated with the technology. Solids produced by the treatment system would need to be placed in a facility licensed to accept them.

Although the technology could be successfully deployed to achieve desired results, it does have some limitations. First, it would be most effective in those drifts where water circulation would occur during the flushing process. In the cross-cuts, where little such circulation would occur, less solute capture and removal would take place. Nor would high-volume flushing have a substantial effect on solutes contained in the paste backfill pore water.

The system would be effective over the long-term, since solute mass would be permanently removed and, thus, would not be available for transport through the groundwater system. An advantageous feature of the technology is that it could be employed periodically during the post-closure period, if needed, to remove additional solutes that have been dissolved by the groundwater in the workings. However, it is uncertain if the technology meets the long-term maintenance criterion, since it could be necessary to use the process on more than one occasion during the post-closure period.

5.3.13 Passive Controls

5.3.13.1 Description and Application

After the mine has reflooded, the flux of solute from the mine and resulting groundwater quality at the DMZ boundary will be dependent on the flow of groundwater in the vicinity of the mine. By modifying the flow regime at targeted points around the mine it is possible to equalize the hydraulic head on each side of the backfill, and increase the flux of clean groundwater around the backfill. Such modifications, which could be installed as part of mine operations and/or during the closure of the mine, are referred to as passive hydraulic controls (PHCs).

NMC evaluated two potential PHCs identified as the "west end workings" and a "west end high-permeability zone". The west end workings could consist of wrapping drifts around the west end of the ore body and continuing them in the footwall rocks for some distance to the east. The west end high-permeability zone could consist of backfilling a series of stopes at the west end of the

mine with highly permeable material. After consideration of the engineering, effectiveness, and safety features of both of these options, NMC determined that the west end workings option has the best potential for application in the design.

The west end workings involve construction of drifts in the footwall and connecting both the hanging wall and footwall drifts at the west end of the mine. The connected workings on either side of the mine would equalize the hydraulic head on each side of the backfill. More significantly, this option would also increase the flux of clean groundwater to the western end of the mine to aid in reducing solute concentrations.

5.3.13.2 Technical Screening

The matrix in Table 5-3 shows that the passive control technology meets all six screening criteria and, thus, is deemed technically feasible. The technology could be easily applied at the production level and, therefore, meets the proven criterion. The technology is universal, since it would focus on minimizing the concentration of all solutes at the DMZ boundary. Implementation of the technology would result in the generation of a limited amount of additional waste rock, and it could be installed safely. Therefore, its secondary impacts would be minimal. Since the technology can be successfully deployed to achieve the desired results, it is consistent with the implementation viability criterion. Once installed, the technology will be effective over the long-term. Finally, since the technology will not require operation or maintenance after installation, it meets the long-term maintenance criterion.

5.3.14 Summary

Using the assessments summarized in Table 5-3, NMC has determined that accelerated reflooding, oxygen barriers, oxidized-acidic backfill removal, cleaning of open access workings, containment, high-volume flushing of workings, and passive hydraulic controls substantially meet the technical screening criteria and, therefore, warrant inclusion in the next stage of the evaluation process. Alternative binders, chemical precipitation, and backfill pore water replacement were, despite unresolved uncertainties, determined to have sufficient potential technical merit to also warrant inclusion in the next stage of the evaluation process. Because biological reduction, chemical reduction, and reactive barriers were deemed technically infeasible, NMC elected not to consider those technologies further at this time.

5.4 Economic Evaluation

The interplay of the technical feasibility and economic feasibility components of this analysis yields two categories of technologies suitable for summary decision making and one category requiring detailed analysis. The first category, which includes those technologies of high technical merit and relatively low projected economic impact, warrant evaluation as part of the facility design without substantial additional economic analysis. The second category includes technologies with technical merit so doubtful that they can be rejected without regard to their projected cost. The final category contains those technologies where some uncertainties are

present as to technical merit, economic feasibility, or both. The bulk of the discussion that follows deals with that last category.

Thirteen technologies were screened based on technical criteria in Section 5.3 above. Three of these candidate technologies were found to be technically infeasible and rejected. Seven of the remaining ten were found to have high technical merit and the ability to be implemented without adversely affecting project economics. Those seven (accelerated reflooding, oxygen barriers, oxidized-acidic backfill removal, cleaning of open access workings, passive hydraulic controls, containment, and high-volume flushing of the workings) are further assessed in Sections 6, 7, and 8 below as potential components of the facility design.

Because the technical case for the alternative binders, chemical precipitation, and backfill pore water replacement options is less clear; a closer scrutiny of the projected cost implications of each was undertaken to assess which, if any, warranted additional consideration. Table 5-4 contains projections of the present worth of the capital and operating costs for each of those three technologies. Capital cost estimates in the table include the cost of equipment, construction and installation, piping, controls, engineering, contingencies, etc. Operating costs include labor, materials, reagents, treatment, disposal, etc. All costs are incremental increases above what would normally be included in the project without application of the additional technologies.

Table 5-4

Technology Present Worth Analysis

Technology	Cost Estimate[1] Capital ($x1,000)	Cost Estimate[1] Annual Operating ($x1,000)	Year of Installation[2]	Years of Operation	Present Worth[3] ($x1,000)
Alternative Binders	—	20,200	3	28	417,000
Chemical Precipitation	460	322	2	28	7,430
Backfill Pore Water Replacement	6,800	717	3	28	21,600

1 In 2000 dollars.
2 From commencement of construction.
3 At 3 percent inflation and 5 percent discount rate.

Prepared by: JWS
Checked by: JJF1

Negligible capital costs would be required to implement the alternative binder system. Operating costs represent only the price differential between regular Portland cement and low-sulfate cements. Capital costs for a chemical precipitation system would principally relate to additional tankage, mixing, and reagent feed systems. Operating costs would include labor, reagents, maintenance, electricity, etc. The backfill pore water replacement system would involve the addition of filter presses, tankage, a water treatment system, additions to the WWTF, buildings, etc. The items included in the operating costs would be similar to those for the chemical feed system, plus added costs for water treatment and solids disposal.

Table 5-4 shows that the present worth cost of the alternative binder technology is $417,000,000, an expense deemed unjustifiable given the economics of the project and the uncertainties regarding the ultimate effectiveness of that technology. Consequently, backfill pore water replacement, which would likely be dependent upon the concurrent use of alternative binders for effective implementation, also cannot be justified.

The substantial projected cost of the chemical precipitation option, in combination with the fundamental uncertainties regarding its ultimate effectiveness, preclude adopting that technology as part of facility design at this point. Should those uncertainties be resolved by subsequent technical developments, the technical and economic feasibility of chemical precipitation will be reevaluated. This issue will be addressed in future RMMP updates.

6 Reflooded Mine Source Concentrations

Solute concentrations in the reflooded mine were estimated by SRK Consulting, Inc. The estimation methods were developed through a series of work plans, reports, and meetings with the WDNR dating back to 1996. Appendix A includes a report (SRK, 2000) describing the estimation methods used and the results obtained.

Four potential sources of solutes were considered:

- Cemented pyrite concentrate used to backfill mined-out stopes;
- Wall rock in the access workings;
- Unmined ore, including the crown pillar; and
- Combustion byproducts, including blasting residues.

Solute concentrations contributed by each source were originally estimated for an unmanaged case (i.e., without any source control measures). Both best engineering judgement (BEJ) and upper bound (UB) estimates were derived. Sensitivity analyses were then carried out to test the reliability of the resulting estimates. Finally, the effects of possible source control measures were estimated. The following sections summarize each step and present salient results.

6.1 Unmanaged BEJ Source Concentrations

BEJ estimates of the solute concentrations contributed by each source were first derived for an unmanaged case.

6.1.1 Pyritic Paste Backfill

NMC plans to use cemented pyrite concentrate to backfill mined-out stopes. The following steps were completed to estimate the solute concentrations that will arise in the pore space of the backfill when the mine is reflooded:

- The geochemical and mineralogical characteristics of the backfill were determined as part of the extensive waste characterization program reported in Foth & Van Dyke (1999/2000).

- The mining and backfilling methods proposed by NMC were reviewed to identify locations where the backfill is likely to be exposed to oxygen.

- Oxygen transport calculations were completed to estimate the amount of pyritic paste backfill that (a) could become oxidized-acidic backfill; (b) will oxidize but remain neutral; and (c) will not oxidize.

- Results of column tests on pyrite concentrate and pyritic paste backfill were used to derive estimates of solute concentrations in each of the (a) oxidized-acidic; (b) oxidized-neutral; and (c) unoxidized backfill.

The results of the geochemical and mineralogical characterization indicate that the backfill has the potential to become acidic if it is exposed to sufficient oxygen. However, the mining methods that NMC plans to use will limit such exposure.

Only a very small volume of backfill, exposed in the cross-cuts used to access each stope, will be accessible to oxygen long enough to potentially become acidic. The BEJ estimate of the extent of acidification within each cross-cut assumes that the only limitation to oxygen availability will be its slow rate of diffusion into the backfill itself. This assumption is conservative because the unmanaged case neglects the beneficial effect of the bulkheads that will be used to seal off the cross-cuts prior to and after the deposition of the backfill. Results of humidity cell tests on small cubes of pyritic paste backfill were used to estimate the rate of oxygen transport into the exposed backfill surface. The resulting oxidation was then assumed to be distributed in such a way that the maximum volume of oxidized-acidic material was produced. Solute concentrations within the oxidized-acidic material were then estimated from results of the humidity cell tests on the pyrite concentrate.

In the areas mined by cut and fill methods, the upper surfaces of each layer or cut will also be exposed to oxygen, but only for short periods of approximately six weeks. Those surfaces will oxidize slightly, but will remain at neutral pH. In areas mined by blasthole stoping methods, the walls of backfilled primary stopes may be exposed to oxidation during the mucking and backfilling of secondary stopes for a period of several weeks. The samples of backfill that were exposed to oxidation in the humidity cell tests remained neutral for over 60 weeks. Solute concentrations in the oxidized neutral backfill were estimated from results of those humidity cell tests.

Within each mined out stope, there will be no exposure to oxygen along any of the surfaces that abut wall rock. Therefore, most of the backfill will remain unoxidized. Solute concentrations in the unoxidized backfill were estimated from the results of saturated column tests carried out on pyritic paste backfill samples.

To complete the BEJ predictions of solute concentrations in the backfill, estimates were adjusted to account for solubility limitations. The geochemical equilibrium model MINTEQA2 was used to estimate concentrations of solutes that are expected to precipitate.

6.1.2 Wall Rock

To access and remove the ore, NMC will construct a series of shafts, vent-raises, drifts, cross-cuts, and ramps in the hanging wall (i.e., the "hanging wall workings"). Estimates of solute concentrations in water that will reflood the hanging wall workings were derived by a three-step procedure. First, geological descriptions of the hanging wall were reviewed and

compared to the mine plan to determine the surface area and duration of exposure of each formation.

Second, results from column tests of waste rock were used to estimate the mass of solutes that would be released from sulfides exposed to oxidation in each formation. Steady-state rates of solute release were estimated from cycles 10 through 54 of the column tests. The measured grain size distributions of the column samples were then used to convert the rates of solute release from each column to rates of release per unit surface area of rock. Those values were then multiplied by the estimated surface area and exposure time of each such formation in the hanging wall, to obtain estimates of the total masses of solute release.

Finally, the estimated masses of released solute were converted to solute concentrations by assuming that all of the released soluble substances would remain in the mine until closure, and then dissolve into the volume of water that would fill the hanging wall workings.

6.1.3 Unmined Ore

Mining will not remove all of the mineralized rock from the ore zone. A crown pillar with a minimum thickness of 100 feet will be left in place at the top of the mine to ensure ground stability. To assess the potential for the unmined areas to affect solute concentrations in the reflooded mine, a procedure similar to that described in the preceding section was followed. The geology and exposure of the residual ore zone components were reviewed, and potential sources of solutes were identified. The exposure time of the crown pillar base was estimated to be six weeks, i.e., the time needed for the last layer in each cut and fill stope below the crown pillar to be mined and backfilled. Constituent release calculations were then completed, using data from cycles 3 to 21 of a column test on zinc ore. Estimates of surface area were applied to the calculations of solute release from the crown pillar. The entire estimated mass of soluble substances was assumed to persist in the mine until closure, and then dissolve into the volume of water that would fill the assumed 2-foot wide gap between the base of the crown pillar and the backfilled stopes.

6.1.4 Combustion and Blasting By-Products

The combustion of natural gas for heating and diesel fuel for mobile equipment will result in the emission of particulate matter with adsorbed constituents, including metals. Most of the emissions will be completely removed by the mine ventilation system during operations. Additional particulate matter will be removed by water used within the mine during operations. Estimates of the combustion by-product metals remaining in the mine at closure indicated that this source is insignificant.

The primary explosive used in underground blasting will consist of a mixture of ammonium nitrate and fuel oil (ANFO). The great majority of the nitrogen contained in the ANFO will be converted to NO_x gas (providing the explosive effect) upon detonation, and removed from the mine by the ventilation system. However, some of the ANFO charges will not detonate, and will

become a potential source of nitrate, nitrite, and ammonia once the mine refloods. Residual nitrogen from blasting was estimated following the method suggested by Ferguson and Leask (1988). A water balance on the mill and mine system was used to estimate the amount of nitrogen that would be returned to the mine in the backfill pore water. To provide a conservative estimate, the removal of nitrogen likely to occur in the mill or TMA was neglected. The distribution of the nitrogen among nitrate, nitrite, and ammonia was also estimated from the Ferguson and Leask (1988) data.

6.1.5 Summary of Unmanaged BEJ Source Concentrations

Table 6-1 presents the BEJ estimates of solute concentrations and volumes of water associated with each of the significant sources for the unmanaged BEJ source concentrations. To reiterate, the estimates discussed in this section pertain to the unmanaged case and do not reflect any of the source control measures described in Section 8 of this report.

Table 6-1

BEJ Estimates of Solute Concentrations in Reflooded Mine Without Source Controls

	Pyritic Paste Backfill				Wall Rock						
					Mole Lake Formation						
Parameter	Acidic	Oxidized/ Neutral	Unoxidized	Overall	Lower	Upper	Rice Lake Formation	Skunk Lake Formation	Overall	Crown Pillar	Volume Average
Estimated Pore Vol. (ft^3)	$6.02x10^2$	$4.66x10^5$	$2.12x10^8$	$2.12x10^8$	$9.4x10^6$	$2.0x10^6$	$5.8x10^6$	$7.2x10^5$	$1.79x10^7$	$9.5x10^5$	$2.32x10^8$
Al (mg/L)	1896	0.0015*	0.025	0.030	0.61	0.14	1.40	0.22	0.80	0.0055	0.089
Sb (mg/L)	2.1	0.033	0.0012	0.0013	0.060	0.0084	0.051	0.069	0.052	0.0033	0.0052
As (mg/L)	650	0.24	0.0053	0.0076	0.026	0.0018	0.027	0.012	0.023	0.00036	0.0088
Ba (mg/L)	0.00071*	0.0031*	0.0057*	0.0057	0.0059	0.0012	0.0095	0.0062	0.0065	0.00044	0.0057
Be (mg/L)	0.082	0.0077	0.00033	0.00035	0.0051	0.00053	0.0043	0.0058	0.0043	0.00034	0.00065
B (mg/L)	0.029	—	0.0054	0.0054	0.0039	0.00071	0.0055	0.0050	0.0041	—	0.0052
Cd (mg/L)	26	0.0059	0.000065	0.00015	0.0051	0.00069	0.0043	0.0058	0.0044	0.0067	0.00050
Ca (mg/L)	471*	587*	954*	953	20	2.1	18	11	17	1.9	872
Cl (mg/L)	251	251	251	251	1.9	0.34	2.7	2.4	2.0	—	229
Cr (mg/L)	21	0.095	0.000070	0.00034	0.0085	0.00074	0.0048	0.0097	0.0065	0.00034	0.0008
Co (mg/L)	18	0.040	0.084	0.084	0.0089	0.00092	0.0076	0.0062	0.0075	0.0057	0.077
Cu (mg/L)	760	0.22	0.0027	0.0053	0.015	0.0030	0.039	0.032	0.022	0.066	0.0068
CN (mg/L)	—	—	—	0.010	—	—	—	—	—	—	0.0084
F (mg/L)	53	1.0*	—	0.0024	0.024	0.0043	0.034	0.030	0.025	—	0.0041

Table 6-1 *(Continued)*

Parameter	Pyritic Paste Backfill				Wall Rock						
					Mole Lake Formation						
	Acidic	Oxidized/ Neutral	Unoxidized	Overall	Lower	Upper	Rice Lake Formation	Skunk Lake Formation	Overall	Crown Pillar	Volume Average
Fe (mg/L)	59756	0.29	0.0080*	0.18	0.79	0.21	1.9	0.36	1.1	0.024	0.25
Pb (mg/L)	3.1*	0.27	0.0024*	0.0030	0.038	0.0036	0.021	0.030	0.028	0.0034	0.0049
Mg (mg/L)	2852	7.4	0.28	0.30	7.3	1.0	7.7	6.2	6.7	0.99	0.80
Mn (mg/L)	193	0.0053	0.0022	0.0028	0.066	0.0092	0.082	0.14	0.068	0.020	0.0078
Hg (mg/L)	0.27	0.00060	0.000080	0.000082	0.00031	0.000023	0.00019	0.00026	0.00024	0.000014	0.000093
Ni (mg/L)	3.3	0.039	0.000072*	0.00017	0.025	0.0041	0.033	0.045	0.026	0.0016	0.0022
NH_4-N (mg/L)	—	—	—	2.3	—	—	—	—	—	—	1.9
NO_3-N (mg/L)	—	—	—	20	—	—	—	—	—	—	17
NO_2-N (mg/L)	—	—	—	0.7	—	—	—	—	—	—	0.59
K (mg/L)	24	879	160	162	3.2	0.51	3.6	3.6	3.0	0.27	148
Se (mg/L)	7.1	0.51	0.031	0.032	0.0055	0.00091	0.0062	0.011	0.0054	0.0017	0.030
Ag (mg/L)	0.021*	0.021*	0.00023	0.00028	0.010	0.0010	0.0084	0.011	0.0084	0.0011	0.00090
Na (mg/L)	291	287	67	67	1.9	0.30	2.1	1.1	1.8	0.15	62
SO_4 (mg/L)	164,173*	2147*	1250*	1252	21	2.1	11	32	16	8.2	1145
SO_4 from S_2O_3 (mg/L)	686	686	686	686	0	0	0	0	0	0	626
Zn (mg/L)	9322	0.54	0.042	0.070	0.055	0.041	0.034	0.10	0.049	1.4	0.073

*Parameter concentration is solubility controlled.
— = No data.

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6.2 Unmanaged UB Source Concentrations

UB estimates of the solute concentrations contributed by each unmanaged source (i.e., without control measures) were also derived. The methods used for UB estimates were very similar to those described above for the BEJ estimates, differing only in the assumptions used in the two sets of estimates detailed below.

6.2.1 UB Assumptions

The UB assumptions that differed significantly from their BEJ equivalents were:

- In the UB calculation of the volume of backfill that might become acidic, it was assumed that acidification would cause the cemented paste matrix to deteriorate and, thus, there would be no limitation to oxygen diffusion into the paste itself. Under that assumption, the rate of oxidation would be limited only by the rate of oxygen diffusion from the bulkhead to the exposed backfill face. Available oxygen was assumed to be equal to atmospheric levels at a point just inside of the bulkhead, and completely consumed at the backfill face. The rate of oxygen diffusion through the waste rock between the bulkhead and the backfill was then estimated using Fick's Law.

- Solubility controls were not applied to solute concentrations estimated for the pyritic paste backfill.

- All 54 weeks of the wall rock column test data were used to estimate solute release rates in the hanging wall workings.

- All 21 cycles of zinc ore tests were used to estimate the rates of solute release from the crown pillar.

6.2.2 Summary of Unmanaged UB Source Concentrations

Table 6-2 presents the UB estimates of solute concentrations for the unmanaged case and the volume of water associated with each of the significant sources. Again, the effects of the source control measures described in Section 8 below were ignored.

The primary difference between the BEJ and UB estimates is in their respective volumes of oxidized-acidic backfill. The conservative assumptions used in the calculation of oxygen transport result in the UB volume estimate being more than two orders of magnitude greater than the BEJ estimate. As discussed below, one of the goals of the source control measures will be to control oxygen levels and, if necessary, remove oxidized-acidic material at the end of mining such that the actual volume of oxidized-acidic backfill is minimized.

Table 6-2

UB Estimates of Solute Concentrations in Reflooded Mine Without Source Controls

	Pyritic Paste Backfill				Wall Rock						
					Mole Lake Formation						
Parameter	Acidic	Oxidized/ Neutral	Unoxidized	Overall	Lower	Upper	Rice Lake Formation	Skunk Lake Formation	Overall	Crown Pillar	Volume Average
Estimated Pore Vol. (ft3)	$1.66x10^5$	$4.66x10^5$	$2.12x10^8$	$2.12x10^8$	$9.4x10^6$	$2.0x10^6$	$5.8x10^6$	$7.2x10^5$	$1.79x10^7$	$9.5x10^5$	$2.32x10^8$
Al (mg/L)	1896	13	0.025	1.5	1.3	0.32	5.7	1.1	2.6	0.072	1.6
Sb (mg/L)	2.1	0.33	0.0012	0.0036	0.059	0.008	0.051	0.068	0.051	0.0033	0.0072
As (mg/L)	650	1.1	0.0053	0.52	0.032	0.0019	0.027	0.013	0.026	0.00048	0.47
Ba (mg/L)	0.60	3.0	0.51	0.52	0.0072	0.0015	0.017	0.0087	0.010	0.00053	0.47
Be (mg/L)	0.082	0.028	0.00033	0.00045	0.0051	0.00054	0.0044	0.0059	0.0044	0.00035	0.00075
B (mg/L)	0.029	—	0.0054	0.0054	0.0039	0.00071	0.0055	0.0050	0.0041	—	0.0053
Cd (mg/L)	26	0.011	0.000065	0.020	0.0051	0.00067	0.0044	0.0062	0.0044	0.028	0.019
Ca (mg/L)	2602	716*	980*	981	20	2.0	17	12	17	2.0	897
Cl (mg/L)	251	251	1900	1895	1.9	0.34	2.7	2.4	2.0	—	1731
Cr (mg/L)	21	0.17	0.000070	0.017	0.0080	0.00079	0.0075	0.0098	0.0071	0.00035	0.016
Co (mg/L)	18	0.10	0.084	0.098	0.0091	0.00096	0.0089	0.0073	0.0080	0.014	0.090
Cu (mg/L)	760	0.58	0.0027	0.60	0.027	0.0047	0.10	0.071	0.051	0.55	0.55
CN (mg/L)	—	—	—	0.011	—	—	—	—	—	—	0.0092
F (mg/L)	53	8.1	—	0.060	0.024	0.0043	0.034	0.030	0.025	—	0.056

Table 6-2 *(Continued)*

Parameter	Pyritic Paste Backfill				Wall Rock						
					Mole Lake Formation						
	Acidic	Oxidized/ Neutral	Unoxidized	Overall	Lower	Upper	Rice Lake Formation	Skunk Lake Formation	Overall	Crown Pillar	Volume Average
Fe (mg/L)	59,756	1.5	0.037	47	2.0	0.46	8.2	1.7	3.8	0.30	43
Pb (mg/L)	520	0.30	0.038	0.44	0.038	0.0038	0.025	0.033	0.030	0.0069	0.41
Mg (mg/L)	2852	160	0.28	2.9	8.7	1.2	11	7.4	8.6	1.2	3.3
Mn (mg/L)	193	0.086	0.0011	0.15	0.092	0.013	0.22	0.19	0.13	0.029	0.15
Hg (mg/L)	0.27	0.0023	0.000080	0.00030	0.00031	0.000023	0.00019	0.00026	0.00024	0.000015	0.00029
Ni (mg/L)	3.3	0.14	0.027	0.030	0.027	0.0038	0.032	0.042	0.026	0.0029	0.029
NH_4-N (mg/L)	—	—	—	2.3	—	—	—	—	—	—	1.9
NO_3-N (mg/L)	—	—	—	20	—	—	—	—	—	—	17
NO_2-N (mg/L)	—	—	—	0.7	—	—	—	—	—	—	0.59
K (mg/L)	24	1125	160	162	4.6	0.66	4.9	5.1	4.3	0.25	148
Se (mg/L)	7.1	1.6	0.031	0.040	0.0058	0.00093	0.0060	0.013	0.0056	0.0022	0.037
Ag (mg/L)	0.37	0.71	0.00023	0.0021	0.0090	0.00094	0.0078	0.010	0.0078	0.0010	0.0025
Na (mg/L)	291	291	67	68	5.4	0.71	4.6	1.9	4.4	0.16	64
SO_4 (mg/L)	169,555	2659*	1317*	1451	28	2.5	13	38	21	17	1327
SO_4 from S_2O_3 (mg/L)	1371	1371	1371	1371	—	—	—	—	—	—	1252
Zn (mg/L)	9322	1.1	0.042	7.3	0.064	0.036	0.072	0.10	0.065	5.2	6.7

*Parameter concentrations are solubility controlled.
— = No data.

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6.3 Summary of Sensitivity Analysis

Sensitivity analyses were completed to determine the response of the BEJ and UB estimates of "unmanaged" solute concentrations to changes in input parameters for the following:

- number of stopes;
- surface area of backfill exposed in each cross-cut;
- exposure time;
- paste backfill NP;
- wall rock surface area; and
- early shutdown.

Results of individual sensitivity analyses are provided in the Reflooded Mine Source Concentration Report (Appendix A). The effect of each such change on individual sources is noticeable in most cases, and in all cases follows the pattern that would be expected. To give a better sense of the overall importance of each change, volume-weighted average concentrations were also calculated. In general, those results show that the predictions are largely insensitive to the changes. That is because the overwhelming volume of unoxidized backfill dominates all other sources, especially in the BEJ predictions.

In the UB predictions, changes that affect the volume of oxidized-acidic backfill have noticeable effects on the volume-weighted average concentrations of some solutes, notably iron, arsenic, copper, zinc, and others that would be present at high concentrations in such backfill. The reason for this sensitivity is that the volume of oxidized-acidic material estimated by the UB calculations is large enough that it has a significant impact on volume-weighted average concentrations.

6.4 Effects of Source Control Measures

Calculations were also carried out to investigate the effects of three source control measures:

- Construction of effective oxygen barriers in the cross-cuts during mining;
- Direct removal of oxidized-acidic backfill just prior to reflooding; and
- Cleaning of wall rock surfaces prior to reflooding.

The first two measures target the oxidized-acidic backfill. The oxygen barriers help prevent the generation of oxidized-acidic backfill in the first place, whereas direct removal eliminates such materials after they form. Since both measures result in minimizing the amount of oxidized-acidic backfill remaining in the mine when reflooding begins, their effects on solute concentrations are similar. Use of both the oxygen barriers and direct removal would have a compounding beneficial effect. As would be expected, cleaning of the wall rock surfaces prior to reflooding is also predicted to result in a significant reduction of solute concentrations in the mine from that source.

7 Reflooded Mine Solute Transport Evaluation

A three-dimensional numerical model was developed to predict increased concentrations of solutes at the DMZ boundary due to the migration of groundwater away from the sources in the reflooded mine. Appendix B contains a detailed report on the solute transport modeling analysis. The analysis involved construction of a numerical flow and transport model, prediction of DMZ boundary concentrations using hydrogeologic parameters consistent with the regional groundwater flow model (HSI GeoTrans, 1998), evaluation of the effect of passive hydraulic control options designed to promote mixing to lower solute concentrations, and testing of model sensitivity.

The effects of the solute source zones on estimated concentrations at the DMZ boundary from the unoxidized and oxidized-neutral paste backfill, from the oxidized-acidic backfill, from the exposed wall rock of the access workings, and from the base of the crown pillar, are analyzed in separate simulations. A nominal source concentration of 1.0 mg/L was used in each simulation. The simulated DMZ boundary concentrations were then expressed as a fraction of the source concentration. In that way, the model results presented in this section can be applied to all constituents of interest in a spreadsheet to calculate estimated concentration increases (above background levels) at the DMZ boundary. That application of the results is discussed in Section 9.

7.1 Model Development

The hydrogeologic setting of the mine is shown conceptually in Figure 7-1. That figure depicts the position of the cemented paste backfill and other source zones relative to the bedrock, crown pillar, and glacial aquifer units. After reflooding, groundwater flow will be generally from east to west, substantially as it is under current (pre-mining) conditions. The backfill, placed in the mined-out stopes, will have a very low hydraulic conductivity, which will limit groundwater flow through that source zone. Diffusion and dispersion will slowly transfer mass from the more concentrated pore water in the backfill to the area of lower concentrations in the water flowing around the backfill.

A three-dimensional model grid was constructed to represent the area around the reflooded mine. The model domain covers an area that extends just beyond the DMZ boundary, with the source near the center, as shown in Figure 7-2. Horizontal grid spacing varies from 1.5 feet at the edge of the backfill source zone to a default value of 37.5 feet. Vertically, the model domain extends from the water table in the glacial aquifer to a depth of approximately 1,200 feet, or about halfway down the mine. All layer elevations are constant, with a minimum thickness of 25 feet, and a maximum thickness of 85 feet (Figure 7-3). In each simulation, solute transport (advection, diffusion, and dispersion) was simulated for 10,000 years. In the unweathered bedrock, it takes about that long for the maximum concentration to be attained at the DMZ boundary.

Each of the model's cells is made up of one of the following hydrogeologic units: (1) glacial outwash; (2) glacial till (including massive saprolite); (3) the crown pillar; (4) the grouted portion of the crown pillar; (5) strongly weathered bedrock; (6) moderately weathered bedrock; (7) weakly weathered and unweathered bedrock; (8) cemented paste backfill; and (9) open access workings. The modeled layout of these units is a simplified representation of the hydrogeologic setting (Figure 7-3).

Open workings are modeled in the hanging wall and footwall (Figure 7-4), with access drifts and cross-cuts on every fourth layer in the mine (every 200 feet). Footwall workings were added as a passive hydraulic control measure for reducing DMZ boundary concentrations. The workings tend to equalize groundwater head between the hanging wall and footwall. More significantly, the workings also promote the flow of water around the backfill source, which helps to reduce solute concentrations as they move out from the source (see Appendix B, Section 5).

The model was defined using estimated values for hydraulic conductivity, porosity, dispersivity, and effective diffusion coefficient for each of the nine hydrogeologic units (Table 7-1). These values were based on prior analyses at the site, a review of applicable literature, and opinions of qualified experts (see Appendix B). An analysis of model sensitivity was conducted to assess the effect of uncertainty in these estimates on predicted results (Section 7.3). Derivation of the solute transport model is more thoroughly discussed in Appendix B.

7.2 Model Results

Separate simulations of the unoxidized and oxidized-neutral portions of the paste backfill source confirmed that it was appropriate to combine these two zones into a single source zone in the model. A continuous, or constant-concentration, source (1.0 mg/L) was used to simulate those constituents that are controlled by mineral solubility in the unoxidized and oxidized-neutral backfill (e.g., sulfate). Results of the 10,000-year simulation of the unoxidized and oxidized-neutral source are presented in Figure 7-5. The left side of that figure shows that the relative concentration-versus-time curves for representative cells on the DMZ boundary at different depth ranges. Those particular cells were chosen because they were where maximum concentrations occurred in the five depth ranges considered: (1) glacial aquifer; (2) strongly weathered bedrock; (3) moderately weathered bedrock; (4) weakly weathered bedrock; and (5) unweathered bedrock. In the glacial system, the maximum DMZ boundary concentration is predicted to be less than 3 percent of the unoxidized and oxidized-neutral source concentration. In the unweathered bedrock, the maximum DMZ boundary concentration is predicted to be about 35 percent of the source concentration. The right side of Figure 7-5 shows the maximum DMZ boundary concentration modeled in each layer. According to the simulation results, it would take approximately 300 years for the maximum concentrations to be attained in the glacial system, and approximately 10,000 years for the maximum concentrations to be attained in the unweathered bedrock.

Table 7-1

Parameter Values Used in the Solute Transport Model

	Conductivity E-W (ft/d)	Conductivity N-S (ft/d)	Conductivity Vertical (ft/d)	Porosity	Dispersivity Longitudinal (ft)	Dispersivity Transverse (ft)	Dispersivity Vertical (ft)	Diffusion (ft^2/d)
Outwash	40	40	4	0.1	50	5	0.5	1.6 E-04
Till (with Massive Saprolite)	2	2	0.075	0.1	50	5	0.5	1.6 E-04
Crown Pillar	8	8	8	0.1	50	5	0.5	1.6 E-04
Grouted Crown Pillar	0.3	0.3	0.3	0.01	50	5	0.5	1.6 E-04
SW Bedrock	0.1	0.1	0.1	0.05	50	5	5	1.6 E-04
MW Bedrock	0.02	0.02	0.02	0.01	50	5	5	1.6 E-04
WW/UW Bedrock	0.001	0.001	0.001	0.003	50	5	5	1.6 E-04
Backfill	0.0003	0.0003	0.0003	0.4	50	5	0.5	2.8 E-04
Open Workings	500	500	500	0.22	50	5	5	1.6 E-04

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Approved By: PFA

A separate simulation of the unoxidized and oxidized-neutral source zone was made for constituents of interest that are not solubility controlled (e.g., nitrate and nitrite). In this simulation, an instantaneous source of 1.0 mg/L was assigned throughout the entire backfill zone. This type of source representation assumes that the entire constituent mass is in solution from the beginning of the simulation. The maximum DMZ concentrations predicted in this simulation are lower than those predicted for the continuous source, and the concentration versus time curves have defined peaks that occur prior to the end of the simulation (Figure 7-6).

Simulation of the oxidized-acidic zone source at the interface of the paste backfill and cross-cuts was also made using an instantaneous source of 1.0 mg/L. This assumes that all of the constituent mass of the oxidized-acidic zone is dissolved into the groundwater at the start of the simulation. A conservative approach is taken whereby the source is assumed to be in the open working cell in the cross-cut just outside the backfill. Because of the small size of the oxidized-acidic zone source relative to the unoxidized/oxidized-neutral zone, the maximum DMZ boundary concentrations due to the oxidized-acidic zone source (Figure 7-7) are orders of magnitude lower than the predicted concentrations for the unoxidized and oxidized-neutral zone source (Figure 7-6). Still, if a particular substance of interest has an oxidized-acidic zone source concentration much higher than its averaged unoxidized and oxidized-neutral source concentration, then the oxidized-acidic zone may contribute significantly to the DMZ boundary concentration.

An instantaneous source was used in all open workings cells to simulate solute transport from the wall rock source. The resulting concentration versus time and concentration versus depth curves are shown in Figure 7-8. Like the oxidized-acidic source, this instantaneous source produces a peak concentration at all depths on the DMZ boundary during the first several hundred years of simulation.

The bottom (grouted) layer of the crown pillar was used to simulate the contribution of the crown pillar source zone. Again, the entire source mass was assumed to be instantaneously dissolved. Figure 7-9 shows the resulting concentration versus time and concentration versus depth curves. Note that using the 25-foot thick layer above the backfill (at 10 percent porosity) to simulate the assumed 2-foot thick (at 100 percent porosity) crown pillar zone, slightly overestimates the volume of this source.

The results for all of the source zone simulations are summarized in Table 7-2. In Section 9, these results are applied to the constituent-specific source concentrations derived in Section 6 to show the estimated increases in concentrations at the DMZ boundary.

Table 7-2

Summary of Model Simulations

Source	Maximum DMZ Boundary Relative Concentration				
	Glacial Aquifer	Strongly Weathered Bedrock	Moderately Weathered Bedrock	Weakly Weathered Bedrock	Unweathered Bedrock
Unoxidized/Oxidized Neutral Backfill (Continuous)	0.028	0.032	0.22	0.26	0.35
Unoxidized/Oxidized Neutral Backfill (Instantaneous)	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (In Cross-Cut)	0.0001	0.00010	0.00043	0.00041	0.00042
Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	0.0009	0.00017

Note: The relative concentrations shown are expressed as a fraction of the source concentration.

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7.3 Model Testing and Sensitivity Analysis

Nineteen simulations were made to test the significance of parameter uncertainty with respect to predicted DMZ boundary concentrations. In those simulations, different values were assigned for hydraulic conductivity, porosity, and dispersivity. Through this sensitivity analysis, it was found that changing parameter values, over the range discussed in Appendix B, results in changes to estimated DMZ boundary concentrations of less than a factor of two (significantly less than a factor of two in the unweathered bedrock).

Additional simulations were made to verify the appropriateness of the simplifying assumptions and simulation methods, and to test potential future scenarios. Those assumptions and methods, and the results of the simulations, are summarized in Appendix B.

7.4 Summary of Solute Transport Analysis

The solute transport model is a useful tool for estimating and evaluating the transport of solutes from the reflooded mine and to assist in the selection of engineering controls to be applied to achieve the reflooded mine design objective. The model includes additional access workings in the footwall to help minimize the level of solutes in groundwater per NR 140.22(1). The results of simulations with the model (Table 7-2) can be used in conjunction with source concentration estimates to project DMZ boundary concentration increases (above background levels) for all substances of interest.

8 Reflooded Mine Design

Based on the results of the design process outlined in Figure 4-1 and described above in Sections 4 through 7, the reflooded mine design includes the following major components:

- Engineering features to collectively minimize the level of substances in groundwater, including selected elements of source reduction, source control, and hydraulic control;
- Five-year updates based on assessments of hydrogeologic conditions encountered during operations and the evaluation of potentially applicable technological developments that have occurred over the preceding five years;
- Development and submittal, two years prior to closure, of a final reflooded mine design;
- Post-closure groundwater and reflooded mine performance monitoring, with the availability of two additional engineering control features, if deemed necessary.

The source reduction engineering features selected for inclusion in the design are oxygen barriers and accelerated reflooding. The selected source control features consist of the removal of oxidized-acidic material from the cross-cuts at the end of mining, and the cleaning of all open access workings. The hydraulic control engineering features selected for inclusion in the design are the west end workings passive hydraulic control, high-volume flushing, and containment. The design recognizes that more than 30 years will pass before some features of the design will be implemented. During this period, assessments of new and emerging technologies will be completed, and data will be collected to determine if hydrogeologic conditions remain consistent with those believed to exist when the design was originally developed.

Various aspects of the design will be implemented at different times during construction, operations, closure, and post-closure. Figure 8-1 contains a generalized depiction of the sequencing of the implementation of the design.

During pre-construction, and as part of the baseline data collection program conducted for the project as a whole, NMC will collect the data specified in the design and, where appropriate, seek alternative concentration limit (ACLs) in accordance with NR 140. In addition, during this period, the financial assurance component related to the reflooded mine will be detailed and integrated into the financial assurance package for the project as a whole.

During construction, NMC will establish the data base and methods to be used to track and evaluate advancements in technologies that could have a future application to the design. During the operations phase of the project, such technology evaluations will continue. Data regarding the hydrogeologic conditions encountered underground will be actively collected and analyzed. The primary engineering controls will be implemented. The performance of those features will be monitored throughout implementation, and design adjustments made as needed. Every five

years during operation, NMC will submit a review and update of the RMMP containing an evaluation of actual hydrogeologic conditions, an assessment of existing and emerging technologies potentially applicable to the design, recommendations for enhancements to the engineering controls implemented during operations, recommendations for new measures and methods, and an updated assessment of the design.

Two years prior to the end of mining, NMC will submit a final reflooded mine design detailing the engineering controls that were implemented during operations, and the final engineering controls that will be operationalized during the closure process. The final design will also contain a quantitative assessment of overall design performance.

During the closure period, oxidized-acidic backfill materials will be removed and the cleaning of the access workings will be completed. After all underground engineering controls have been applied, reflooding of the mine will be accelerated by pumping groundwater from the site's mitigation well into the access workings. Each closure step will be carefully documented so that its proper completion can be verified prior to reflooding.

During the post-closure period, groundwater monitoring will be conducted to provide data that can be used to evaluate the actual performance of the reflooded mine in comparison to that predicted in the final design submitted two years before the end of mining. Such monitoring will also be used to determine whether either, or both, of the remaining two engineering features of the plan need to be implemented, and to assess the long-term performance of the reflooded mine in relation to the requirements and standards of Wisconsin's groundwater quality laws.

8.1 Engineering Controls

The selected source reduction and source control measures focus on minimizing the levels of substances in the groundwater. The PHC has been designed to maximize the flux of ambient groundwater around the paste backfill and thereby minimize the concentrations of solute in the surrounding groundwater. High-volume flushing has been designed to remove soluble substances that are in the groundwater within the open access workings after reflooding. Hydraulic containment would prevent the migration of solutes from the reflooded mine toward the DMZ. Section 8.1.1 describes the source reduction and source control measures. Section 8.1.2 provides a conceptual design for each of the hydraulic control features included in the plan. Note that optimization of the controls to be applied during mining, and the final design of the measures to be implemented at closure and during the post-closure period, would be completed as part of the regular RMMP updates and the final reflooded mine design.

8.1.1 Source Reduction and Source Control

The methods that will be used by NMC to reduce solutes that could migrate from the mine include the installation of oxygen barriers in cross-cuts; the removal of oxidized-acidic backfill, if any, that may form behind the oxygen barriers; cleaning of the open access workings; and accelerated reflooding.

8.1.1.1 Oxygen Barriers

As shown on Figure 8-2, NMC plans to install a timber structure covered with impermeable plastic to close off the entire opening of each stope access cross-cut. As part of mining operations, NMC will evaluate the effectiveness of the proposed design by conducting oxygen monitoring behind barriers at the paste backfill/waste rock interface. Figure 8-3 shows a schematic representation of an oxygen probe that will be used to monitor oxygen levels behind the bulkheads. Oxygen measurements would be done on a quarterly basis until stable trends are documented, at which time the measurements would be completed on an annual basis to verify the continued effectiveness of the barrier.

The greatest potential for movement of oxygen past the oxygen barriers is likely to occur along the interface of the barriers and the wall rock. It is also possible that oxygen could flow around the barrier through fractures in the rock surrounding it. If it is found during monitoring that oxygen levels in contact with the paste backfill at the paste backfill/waste rock interface are greater than 5 percent of atmospheric oxygen levels, then design changes to enhance sealing effectiveness will be implemented.

Available design modifications include:

- The placement of organic material behind the barrier to act as an oxygen consumer through biodegration.
- Grouting or shotcreting the edges of the barrier to improve its seal with the walls of the cross-cut.
- Constructing a recessed cut in the walls of the cross-cut to permit grouting the edge of the bulkhead directly into the surrounding rock.
- Grouting of fractured wall rock around the barrier to cut off oxygen leakage around the barrier.
- Installation of a secondary bulkhead.

8.1.1.2 Removal of Oxidized-Acidic Material

In the event that the oxygen barriers do not prove fully effective at controlling oxygen transport into the sealed cross-cuts, an oxidized-acidic zone may form at the exposed surface of backfill immediately behind the waste rock. Based on the source concentration analysis (SRK, 2000), this acidic zone will not be widespread; rather, it will be localized to the exposed backfill face and is likely to penetrate no more than one inch to no more than several feet into that face. Given that the oxidized-acidic zone will be localized in the backfill face in accessible cross-cuts, it will be readily amenable to removal and treatment.

At the end of mining and prior to reflooding, the oxygen barriers and waste rock in each cross-cut will be removed. The exposed surface of the backfill will be tested for oxidation and acidity to determine the depth to which the acidic zone penetrates into the backfill. Where oxidized-acidic material is found, it will be removed.

Depending on the depth of oxidized-acidic zone penetration, several options exist for removing that material. In the event that the zone is constrained to the backfill surface, the primary means for removing the material would be high-pressure washing. Alternatively, the face of the exposed backfill could be manually scraped and the resulting solid material collected and removed from the cross-cut. Waste rock from the cross-cuts will be managed in a suitable fashion, such as being processed through the mill to remove oxidation products and to generate neutral paste backfill that will be returned underground.

If high-pressure washing is used, the rinse water and solubilized products would be directed to in-mine sumps. The water would be pumped from the sumps and directed to the project's wastewater treatment facilities (WWTF). Any residual sediment would be removed as part of the cleaning of the access workings described below.

In the event the oxidized-acidic zone penetrates beyond the surface of the exposed paste backfill, surficial and mechanical cleaning methods will be required. Mechanical methods could be used to remove the entire oxidized-acidic zone. It is likely that non-acidic material would also be removed during surficial cleaning of the paste backfill faces. A reasonable estimate is that there may be about a 30 percent increase in material due to dilution by non-acidic material. Overall, it is conservatively estimated that the volume of oxidized-acidic material would not exceed 6,000 yd^3 (based on data from Table 6-2, which is overstated because it does not account for the use of oxygen barriers). Given that estimate and a 30 percent increase due to dilution by non-acidic material, the total volume of material to be treated and disposed of is likely to approximate 8,000 yd^3. Because the oxidized-acidic backfill will be found only in discrete localized areas, professional judgement suggests that very nearly 100 percent of that material can be readily removed by excavation.

Several treatment and disposal alternatives exist if the oxidized-acidic zones are removed as solids. One such option is passing the material through the milling circuit, which would be kept operational until the mine is totally decommissioned prior to reflooding. The material would be passed through the circuit where it would be mixed with water to remove soluble oxidation products. The wash water would be routed to the WWTF and the washed solids would be mixed with cement and sent back underground. Another disposal alternative is shipment to an off-site disposal facility which is licensed to accept the material.

The selection of a final treatment and disposal method will be selected at the time the final design is prepared two years prior to mine closure. That is prudent because the closure activities (including the removal of the oxidized-acidic zones) will not begin for approximately thirty years, over which time new technologies will likely yield additional alternatives worthy of consideration.

8.1.1.3 Cleaning of Access Workings

Oxidation products that have accumulated on the walls of open drifts and cross-cuts and on surfaces of spilled ore and muck represent additional sources of solute. To minimize the contribution of solutes from these sources, NMC will both control (where feasible) the formation of oxidation products and remove (to the extent practicable) such oxidized materials.

To control the formation of oxidation products on the walls of the access workings, NMC will shotcrete the walls in those localized areas where oxidation is visibly apparent or obviously could occur. The shotcrete will be removed at the end of mining, and the oxidized wall rock that has formed will be removed and handled in a fashion similar to that described for the material removed from the cross-cuts.

NMC will also remove spilled ore and muck from all open access workings, and clean walls of the access workings and cross-cuts via high-pressure washing to remove soluble oxidation products that have formed. The spilled ore and muck will be processed through the mill, and the resulting neutral material returned to the mine as backfill. The water used for washing will be collected in the mine's existing drainage system, directed to the sumps, and pumped to the surface for treatment at the project's WWTF.

8.1.1.4 Accelerated Reflooding

After the completion of source removal measures and sealing of the mine shafts, pumping of water from the mine will cease and reflooding of the mine will begin. To speed the reflooding process, and to minimize the creation of new oxidized material within the mine after cleaning, water will be delivered from the project's mitigation well to the underground mine. Such water will be delivered to the mine only during those periods when it is not required elsewhere for mitigation.

8.1.2 Hydraulic Controls

The hydraulic control methods included in the design consist of passive controls, high-volume flushing of the workings, and containment. Passive controls would be installed prior to reflooding. High-volume flushing would be implemented if post-reflood groundwater quality monitoring were to indicate solutes (primarily metals) originating in the open access workings are likely to exceed regulatory limits at the DMZ. Hydraulic containment would be implemented if post-reflood groundwater quality monitoring were to indicate that solutes (primarily sulfate) originating in the backfill are likely to exceed regulatory limits at the DMZ.

8.1.2.1 Passive Hydraulic Controls

After the mine has reflooded, the flux of solute from the mine and the level of solutes in groundwater is largely dependent on the flow of groundwater in the vicinity of the mine. To assist in meeting the design objective of minimizing the level of solute in groundwater, it is

possible to increase the flux of ambient groundwater around the backfill by modifying the flow regime at targeted points around the mine using passive hydraulic controls (PHCs). One such PHC, referred to as the "west end workings", is shown as part of the solute transport modeling described in Section 7, above, to be capable of meeting that objective.

The conceptual design for the west end workings involves the construction of drifts in the footwall workings on all mining levels. Those drifts would be advanced from the hanging wall drifts on the western end of the ore body and then along the footwall horizon. The connected workings will equalize the head on each side of backfilled stopes. As currently envisioned, each footwall drift will be driven 40 feet from the stringer ore. Each footwall working would be approximately 2,500 feet in length. Including the connections between the footwall and hanging wall, it is estimated that approximately 65,000 yds^3 (or approximately 170,000 tons) of waste rock will be generated. The driving of footwall drifts will likely occur during the mining of the stringer ore. Even in the event of a premature shutdown, the west end workings would be completed prior to mine reflooding.

Drift development in the footwall will increase the total amount of waste rock produced over the life of the mine to 1.76 million tons, an increase of only about 10 percent over the original plan (Foth & Van Dyke, 1995/1998a). As such, the installation of the west end workings is not expected to significantly alter the project's waste rock management plan (Foth & Van Dyke, 1995/1998a). The additional waste rock will be managed in accordance with that plan.

8.1.2.2 High-Volume Flushing System

As described in Section 8.5, groundwater quality monitoring will be conducted on groundwater samples collected from the open access workings after reflooding. Those access workings are the principal source of soluble metals that could potentially migrate from the reflooded mine. To verify that water quality in the open access workings is consistent with predicted conditions used in developing the final reflooded mine design, NMC will sample the workings on a quarterly basis for 40 years after the mine refloods. The resulting data will be evaluated on an annual basis. If, after 40 years, the water quality data indicates the levels of dissolved metals in the workings remain consistent with the predicted conditions, monitoring of the access workings will cease. In the event that water quality data indicates that actual conditions are significantly different from predicted conditions, and that the actual conditions indicate that an exceedance of NR 140 standards at the DMZ boundary could occur at some time in the future, high-volume flushing as described below will be undertaken until such time that the concentrations of dissolved metals in the open workings decrease to targeted levels. Note that oxidation products within the open access workings are a finite source in a hydraulically accessible area and are, thus, amenable to the application of a high-volume flushing system.

The high-volume flushing system will include a series of wells located on the west end of the mine for extraction of water from the open access workings, and a piping system located on the east end of the mine which will be used to reintroduce that water into the access workings following treatment. The system will circulate water from east to west through the workings in

the general direction of the normal regional groundwater gradient. The circulation system will collect additional solutes originating from the walls of the workings, those flushed from fractures, and some contribution from backfilled stopes and cross-cuts. A detailed discussion of the conceptual design and operation of this system follows.

8.1.2.2.1 Circulation System

Each of the system's recovery wells, located on the downgradient (west) end of the mine, will penetrate one of the open drifts connecting the hanging wall workings and the footwall workings, and will be equipped with a pump. Water collected by the recovery wells will be delivered to the project's WWTF.

Treated water will be delivered back into the eastern end of the open access workings using a series of wells penetrating each of the open drifts in the hanging wall and footwall. It is anticipated that the rate of water circulation will be approximately 300 to 400 gpm. The flushing system can operate on either a continuous or intermittent basis, subject to the concentration of the targeted solute and the rate at which it is removed from the system. The final mode of operation would be determined at the time of implementation.

8.1.2.2.2 Water Treatment

Water removed from the mine as part of the high-volume flushing process will be treated at the project's existing WWTF. This system, described in the *Preliminary Engineering Report for Wastewater Treatment Facilities* (PER) (Foth & Van Dyke, 1995/1998c), will remain in operation until leachate volumes collected from the tailings management area (TMA) decrease to a point where it is no longer practical to operate. This is projected to occur approximately 16 years after closure of the TMA. Therefore, the period of post-closure operation of the WWTF coincides with the period during which any need for high-volume flushing would become apparent. As presented in the PER (Foth & Van Dyke, 1995/1998c), the project's WWTF will have sufficient capacity to handle the water collected by the recovery wells. As noted in the MPA (Foth & Van Dyke, 1995/1998a), solids generated by the WWTF following mine closure will be disposed of at an off-site licensed landfill. In the unlikely event the flushing system is required after decommissioning of the WWTF, NMC will evaluate alternative water treatment options available at that time and, with WDNR approval, implement the most suitable alternative.

8.1.2.3 Hydraulic Containment

In the event that post-closure monitoring data predicts that the backfill will generate problematic solute concentration at the DMZ boundary, hydraulic containment would be implemented. A conceptual design for the proposed containment system is discussed below.

The analysis prepared by HSI GeoTrans provided in Appendix C describes the process used to estimate the location of extraction wells and the total required extraction rate. The process

involved the use of the project's groundwater flow model (HSI GeoTrans, 1998) and MODPATH in a series of trial-and-error runs.

The analysis demonstrates that containment will be effective. Containment is defined as the creation of an area of inward gradients toward each containment well that encompass, in aggregate, all of the mine and open access workings. The analysis showed that 5 to 9 bedrock wells are needed for the system, and that the actual aggregate extraction rate needed to achieve containment ranges between 9 and 33 gpm. The final selection of the number of containment wells and the total extraction rate needed for the system, if required, would be made based on the actual site conditions encountered during mining. The performance of the containment system would be monitored to determine if additional wells would be needed or the extraction rate changed.

If implementation of the containment system is deemed necessary at some point during the post-closure phase, NMC would explore all applicable water treatment and disposal options available at that time and, with WDNR approval, implement the most suitable alternative. Water treatment and disposal alternatives could include contracting with a third party for water treatment, or the installation of an appropriately sized treatment system and soil absorption system.

8.2 Hydrogeologic/Technology Assessments and RMMP Updates

Another central component of the plan is the periodic reassessment of hydrogeologic conditions and design alternatives based on new and emerging technologies. Described below is a summary of the procedures that will be followed when performing such assessments, the results of which will be incorporated into updates to the RMMP at 5-year intervals.

8.2.1 Hydrogeologic Assessments

Hydrogeologic data has been collected from the site of proposed mining operations since the mid-1970s. Added to this historical data base will be the data collected and generated immediately before and during mining operations. Those data will be obtained through the groundwater monitoring program, the monitoring of groundwater inflow, implementation of the grouting program, and geologic mapping during development of the underground workings. Periodically, updated hydrogeologic data will be evaluated with specific regard to the design objective. Such assessments will allow changes in understanding of the hydrogeologic system to be incorporated into the regular updates of the reflooded mine solute transport model. The results of the updated solute transport model will be used, in turn, to reassess the continued ability of the design to meet the design objectives and, if needed, to generate appropriate design modifications to the plan.

8.2.2 Existing and Emerging Technology Assessments

Since it will be a number of years before backfilling of paste tailings will begin, and 30 years or more before reflooding will occur, it is probable that presently emerging technologies will become proven and new technologies will develop that may be of assistance in meeting the design objectives. As part of mine operations, NMC will monitor the progress of all such developments and thoroughly evaluate those showing the most promise.

NMC will search existing data bases on an annual basis to identify newly released publications that pertain to the design objective. Copies of relevant publications will be obtained and reviewed. Contacts will be made with the proponents of the most promising technologies, methods, or processes to further assess potential applicability. The publications and records of all contacts will be maintained by NMC in a data base specifically organized for the technology assessment.

Where warranted, NMC may conduct additional scientific and engineering evaluations to further assess the applicability of selected enhanced existing or emerging technologies to the plan. Scientific studies could consist of laboratory or field testing and modeling aimed at defining the value of the technology to the design. Engineering studies would focus on the possible application of the technology to the plan from the standpoint of construction, operation, maintenance, cost effectiveness, etc.

8.2.3 Five-Year Updates to RMMP

The ability to achieve the design objective will be influenced by: (1) the effectiveness of the engineering measures used to minimize the levels of solute in groundwater around the reflooded mine; and (2) the prevailing hydrogeologic conditions in the vicinity of the mine. As described above, an ongoing technology assessment will be conducted and additional data will be regularly collected to assess those hydrogeological conditions. Every five years NMC will report to the WDNR on the progress made in its ongoing hydrogeologic, scientific, and engineering evaluations. The information will be presented in report format and will address: (1) the status of site conditions relative to those believed to be present when the RMMP was first developed; (2) the status of the technology data base; (3) a description of all additional scientific and engineering studies completed; and (4) a reassessment of the continued applicability of planned features to meet the design objective. This information will be used to identify the need, if any, for potential modifications or enhancements to the design. Any such proposed modification or enhancement will be detailed, evaluated, and discussed in the updates.

8.3 Final RMMP

NMC will submit a final design for the reflooded mine two years prior to mine closure. The process used to develop the final design will roughly parallel the process presented in this document, and will rely on several sources of updated data. At the time the final plan is prepared, extensive data will be available from the monitoring of oxygen levels in the cross-cuts

behind the oxygen barriers, and through the water quality monitoring program described below. This additional data will be factored into an updated source concentration model for the reflooded mine. Additional insights gained regarding hydrogeologic conditions will also be factored into a revised solute transport model. Finally, new technologies that may yield better results at a reasonable cost will be assessed. All of that information will be used to update and optimize the final design.

The final RMMP will optimize all of the components of the plan prior to mine closure. Those components implemented during operations will be summarized in the final design document. Components chosen for implementation during the closure process will be identified and evaluated. Also, final design details for implementation will be identified. The final design document will specify the schedule for implementing the last steps of the plan and the final documentation process for closure.

8.4 Transition from the Closure to the Post-Closure Period

Following the completion of closure activities, management of the reclaimed mine will move into the post-closure phase. Closure will be deemed to be "complete" when the following steps have been performed and documented:

- Equipment that is left underground has been drained of operating fluids;
- Oxidized-acidic material, if any, has been removed from the cross-cuts;
- The open access workings have been cleaned;
- Other such measures as identified in the final design have been completed and documented; and
- The reflooding is complete.

8.4.1 Closure Documentation

As described above, NMC proposes to undertake a series of actions during mine operations and closure to minimize the level of solutes in groundwater around the reflooded mine. NMC proposes to complete the following steps to verify that the described actions have been completed to the extent required for closure certification per NR 132.13, Wis. Admin. Code, and the transition to post-closure monitoring can occur.

8.4.1.1 Applied Engineering Controls

Three of the four engineering controls proposed in Section 8.1 for use as part of NMC's source control program will be applied during the closure process. These are: (1) removal and treatment of oxidized-acidic material; (2) the physical removal of localized areas of oxidized sulfides in

exposed wall rock and spilled ore and muck, and high-pressure washing of exposed surfaces; and (3) accelerated reflooding. The oxygen barriers used during operations will be removed as part of the closure process.

The purpose of applying engineering controls during the closure process is to remove, to the extent technically and economically feasible, as many potential sources of solutes or the solutes themselves, prior to the reflooding process such that they are not available for potential migration from the reflooded mine. The ultimate effect of the engineering controls to be applied during closure will not be seen for several years after reflooding. Thus, it is desirable to develop a series of indirect criteria during the closure process to define when the desired effectiveness of each applied control has been achieved. Those criteria can then be used and logged by the closure team to verify that the control has been accomplished as planned. The criteria have one other significant purpose: they can be used by NMC and the WDNR to complete the certification process specified in NR 132.13, Wis. Admin. Code.

Given that it will likely be over 30 years before the closure process is actually implemented, the closure criteria NMC has developed should be considered conceptual. While the categories for such closure criteria may not themselves become obsolete over the next 30 years, it is very likely that specific details of how individual processes and tests would be performed will, in all likelihood, change. Therefore, the following discussion does not specify precise details of how the work will be done. Such details will be included in the final engineering design prepared two years prior to the commencement of the closure process.

8.4.1.1.1 Management of Oxidized-Acidic Material

To determine if oxidized-acidic cemented paste backfill is present at the face of the exposed backfill mass located behind the removed bulkheads, NMC will perform a series of paste pH and paste conductivity tests. Tests evenly distributed across the backfill face will be performed at ten points. If oxidized-acidic material is encountered, it will be removed. Once the removal process is complete, the test procedure will be repeated to verify that removal is complete.

8.4.1.1.2 Cleaning of Open Workings

The first step in cleaning the open access workings will be the physical removal of localized areas of visibly oxidized wall rock. Many of those areas will have been previously shotcreted during operations to inhibit oxidation. The shotcreted areas will be cataloged as the shotcrete is applied. The shotcrete will be removed at the end of mining. Any oxidized rock will be mechanically removed and the exposed surface will be examined by a designated inspector to verify that the zone is "clean". If additional material needs to be removed, a reinspection will be completed following removal of the additional material. All related records will be maintained by NMC. Prior to reflooding, NMC will inspect all of the open access workings in total to determine whether significant oxidation has occurred in areas that were not originally shotcreted. Those areas found to contain oxidized wall rock will be cleaned using the same procedures outlined for areas where shotcrete has been applied.

The second step in the cleaning process will be the removal of spilled ore and muck. The process will first divide the open workings into designated segments or sections. When a specific segment or section has been cleaned and is ready for inspection, the designated inspector will tour the area to verify that all reasonably accessible material has, in fact, been removed. Once an area has been cleared by the inspector, an appropriate entry will be made in an inspection log created specifically for this task. If it is determined that additional material needs to be removed, a reinspection will be conducted following the removal of the additional material. An entry into the log will be made of both the initial inspection and the reinspection. Prior to the commencement of reflooding, NMC will verify that every such segment or section of the mine has been inspected and found to be clear of spilled ore and muck. That final review will be detailed in entries to the log.

NMC will then pressure wash accessible walls, ceilings, and floors of open workings to remove readily soluble oxidation products. To verify that the washing is complete, NMC will test the rinse water for total dissolved solids (TDS). Washing will be considered complete when the TDS values in the rinse water reach stable levels. (Note that the washing of each section of open workings will be conducted at a constant wash rate to provide useful TDS measurements.) The washing process will use the same principals of segmentation or sectioning and log entries as those described above for the removal of spilled ore and muck. Final, detailed specifications on the washing program will be contained in the final RMMP.

8.4.1.2 Accelerated Reflooding

Reflooding will commence when the criteria specified in Section 8.4.1.1 are deemed complete for all sections of the mine. Because accelerated reflooding is subject to the availability of water from the project's mitigation well, it is not practical to establish a specific transition criteria for this element.

8.4.2 Closure Documentation Report

Following the completion of site closure activities, NMC will submit a report to the WDNR detailing the closure process and providing all data collected to document that closure has been completed in substantial conformance with the plan.

8.5 Monitoring

As part of the plan, NMC will undertake environmental monitoring and other data collection for the following purposes:

- To assess hydrogeologic conditions actually encountered during mining;
- To assist in the development of the 5-year RMMP updates and the final reflooded mine design; and

- To assess the long-term performance of the reflooded mine in relation to the requirements and standards of NR 140 and NR 182.075.

A discussion of each element of the monitoring plan follows.

8.5.1 Groundwater Monitoring

The project's MPA (Foth & Van Dyke, 1995/1998a) contains a detailed environmental monitoring plan for the Crandon Project. A subset of that plan describes the groundwater monitoring associated with the reflooded mine. A key component of the plan for the reclaimed mine is monitoring groundwater elevation and groundwater quality around the reflooded mine. The program discussed below is a summary of the groundwater monitoring program for the reflooded mine as presented in the MPA (Foth & Van Dyke, 1995/1998a), modified with respect to well locations based on the specific needs of the design. The final reflooded mine monitoring program will be detailed in the final environmental monitoring plan developed for the Crandon Project.

The groundwater monitoring program developed for the reflooded mine is designed to achieve the following objectives:

- To provide information sufficient to assess whether the actual groundwater quality within the reflooded access workings is reasonably consistent with estimated conditions used in the development of the design.

- To provide information to assess whether actual conditions at the DMZ and within the mandatory intervention boundary (MIB) are reasonably consistent with predictions developed at the time the design was prepared.

To achieve those objectives, the reflooded mine groundwater monitoring program incorporates the following key elements:

- Installation of groundwater monitoring wells at and within the MIB prior to mine construction.

- Installation of wells at and within the DMZ boundary prior to mine construction.

- Twelve consecutive months of groundwater monitoring prior to construction of the mine site to establish baseline conditions and applicable alternative concentration limits (ACLs) per NR 140.

- A broad assessment of groundwater quality conducted annually during operations.

- Quarterly monitoring of groundwater quality within the open access workings after reflooding.

- Quarterly monitoring of groundwater quality around the mine after mine reflooding.

The monitoring plan encompasses activities that will be done prior to construction, during operations, during the closure period, and during the post-closure period. Each of these components is described in more detail below.

8.5.1.1 Groundwater Monitoring Wells

Prior to mine construction, NMC will install groundwater monitoring wells at and within the MIB and at and within the DMZ per the requirements of NR 182.075(1u). Figure 8-4 shows the locations of the proposed groundwater monitoring wells around the mine site. Some of the wells, as shown in Figure 8-4, will be used only for monitoring groundwater elevation during and after mining operations. The wells for groundwater quality monitoring will be screened in the glacial outwash, Pre-to Early Wisconsinan Till, and strongly weathered bedrock. New wells constructed in the outwash ("OW") will be piezometers constructed with 5-foot screens at the base of the glacial outwash. Wells designated with a "P" are piezometers that will be constructed with 5-foot screens at the base of the Pre- to Early Wisconsinan Till (EWT). Wells designated as "PP" are bedrock wells (BR) that will be used to monitor groundwater quality within the bedrock. The monitoring intervals and construction techniques for the bedrock wells will be specified in the final monitoring plan and will be based on future discussions with the WDNR. Note that the proposed monitoring locations shown on Figure 8-4 are current as of the date of this document. Subsequent changes to the monitoring plan will be reflected in the revised environmental monitoring plan that will be developed in consultation with all appropriate regulatory agencies.

In addition to construction of the conventional groundwater monitoring wells, eight monitoring points will be constructed to measure water quality in the western quadrant of the hanging wall and footwall drifts at the 500, 700, 900, and 1100 foot levels. The approximate location of the four points are shown on Figure 8-4 as NMC-HW500 to HW1100 and NMC-FW500 to FW1100. The final location and construction details of these points will be specified as part of the final reflooded mine design. Because these wells are intended to provide water quality measurements in the access workings after reflooding, they will not be constructed until the end of mining. Data from those wells will be used in the decision process, described in Section 8.1.2.2.

8.5.1.2 Parameters and Frequency

Baseline monitoring will be performed on the designated groundwater quality monitoring wells shown in Figure 8-4 for the parameters listed in Table 8-1. The baseline monitoring program will consist of 12 consecutive months of sampling prior to mine construction. This data will be used to establish baseline groundwater conditions at each monitoring point. In instances where groundwater constituent concentrations exceed an NR 140 PAL or ES, ACLs will be established per the methodology described by the WDNR (1994).

During dewatering, groundwater flow paths in the immediate vicinity of the mine will be directed toward underground workings. There is no possibility for solute migration out of the mine during operations and no quarterly monitoring of the wells will be needed. However, to maintain a valid database against which future groundwater quality can be compared, the groundwater quality monitoring wells that do not go dry during operations will be monitored annually for all of the parameters listed in Table 8-2.

When reflooding begins, groundwater quality will again be monitored on a quarterly basis. At that time, sampling of the monitoring points in the hanging wall drifts will also begin. Quarterly monitoring will continue through the closure and long-term care periods. The parameters that will be analyzed and the frequency of analyses are listed in Table 8-2. Sample collection, preservation, and analysis will be conducted in accordance with the project's QA/QC plan that will be submitted for WDNR approval prior to commencement of baseline monitoring activities. Groundwater levels monitored as part of the program will be used to assess the rate at which water table recovery is occurring and the direction of groundwater flow. NMC proposes that monitoring of the wells in the access workings be terminated after 40 years because that amount of data will be sufficient to determine whether high-volume flushing of the access workings, as described in Section 8.1.2.2 is needed.

8.5.2 Backfill Pore Water Quality

During operations, NMC will monitor pore water quality within the backfill. Samples will be analyzed for relevant parameters set out in Table 8-2. The purpose of the test program will be to verify that the concentrations of the solutes within the backfill pore water are consistent with the concentrations used in the reflooded mine design.

Several options exist for collecting pore water samples. The first is to collect the samples by installing a monitoring probe, such as a suction cup lysimeter, from the adjacent drift into the paste backfill. The probe would be installed by drilling a hole into the backfill, inserting the probe into the hole, and sealing the probe into the backfill and surrounding rock. In the event that suction cup lysimeters prove unable to collect meaningful samples, backfill drainage water emanating from the cross-cuts could also be collected. NMC will test the merits of these and other options during the first two years of operations, and select (in consultation with the WDNR) the best available sampling methodology.

In the event the sampling probe technology is selected, probes will be installed in every tenth backfilled stope and sampled on an annual basis during mining operations. In the event that drainage water is sampled, water samples will be collected on a quarterly basis from every tenth cross-cut until such drainage ceases. The backfill pore water quality data will be analyzed and reviewed as part of each 5-year update to the RMMP.

Table 8-1

Baseline Groundwater Monitoring Analytical Parameters[1]

Parameter	
A. Public Health Groundwater Quality Standards	C. Indicator Parameters
Arsenic	Alkalinity, Total
Antimony	Calcium
Barium	Hardness, Total
Boron	Spec. Conductance (Field)
Beryllium	Magnesium
Cadmium	Temperature (Field)
Chromium, Total	pH (Field and Lab)
Cobalt	Sodium
Copper	Potassium
Cyanide	Total Dissolved Solids
Fluoride	Ammonia -N
Lead	
Mercury	D. Other Parameters
Nickel	Groundwater Level
Nitrogen, Nitrate+Nitrite	Silica
Selenium	Aluminum
Silver	Dissolved Oxygen
Thallium	Eh (Redox, Field)
B. Public Welfare Groundwater Quality Standards	
Chloride	
Iron	
Manganese	
Sulfate	
Zinc	

[1] Modified from MPA (Foth & Van Dyke, 1995/1998a) Environmental Monitoring Plan.

Prepared by: SVD1
Checked by: FJA

Table 8-2

Post-Baseline Groundwater Monitoring Parameter List[1,2]

Groundwater Level	Fluoride
Specific Conductance (Field)	Iron
pH (Field and Lab)	Lead
Eh (redox, field)	Magnesium
Total Dissolved Solids	Manganese
Alkalinity, Total	Mercury[3]
Aluminum	Nickel[3]
Ammonia-N[3]	Nitrogen, Nitrate+Nitrite
$Antimony_3$	Potassium
Arsenic	Selenium[3]
Barium[3]	Silver[3]
Beryllium[3]	Silica[3]
Boron[3]	Sodium
Cadmium[3]	Sulfate
Calcium	Thallium[3]
Chloride	Zinc
Chromium, Total[3]	Temperature (Field)
Cobalt	Dissolved Oxygen[3]
Copper[3]	Hardness, Total
Cyanide[3]	

[1] During operations, monitoring wells will be sampled annually for all parameters.
[2] List modified from MPA (Foth & Van Dyke, 1995/1998a) Environmental Monitoring Plan.
[3] After closure, parameter is analyzed on an annual basis only during the regularly scheduled quarterly sampling events.

Prepared by: SVD1
Checked by: FJA

9 Reflooded Mine Design Assessment

The RMMP's facility design objective is consistent with the requirements of NR 140.22(1): "to minimize the level of substances in groundwater and to comply with the preventive action limits to the extent technically and economically feasible." Sections 4 through 7 describe the process that was followed to arrive at the design elements described in Section 8. The purpose of this section of the report is to document how the resultant facility design meets the stated objective.

9.1 Design Evaluation

Based on the results of the source concentration estimates for the unmanaged case and the solute transport model, the unoxidized backfill and oxidized neutral backfill will be the primary sources of solutes, such as sulfate and nitrate-nitrite. The analysis also shows that, for the unmanaged case, the oxidized-acidic zone and open access workings will be the principal source of soluble metals. The facility design includes the following engineering controls that will be implemented before or during the closure process to minimize the level of those substances in groundwater to the extent technically and economically feasible:

- The west end access workings PHC;
- Oxygen barriers in the cross-cuts;
- Removal of oxidized-acidic backfill; and
- Cleaning of the open access workings.

In addition, the design includes means for high-volume flushing of the open workings and hydraulic containment either or both of which can be implemented, if deemed necessary based on post-closure monitoring, to minimize the migration of solutes from the reflooded mine.

The PHC will serve to reduce the concentration of substances in groundwater that could emanate from any of the solute sources within the reflooded mine. The oxygen barriers and removal of oxidized-acidic backfill will reduce the concentration of substances in groundwater that could emanate from this solute source. Similarly, the cleaning of open access workings and high-volume flushing will reduce the concentration of substances in groundwater that could emanate from the these workings. Hydraulic containment would serve to capture soluble substances emanating from the mine and open access workings before they could migrate toward the DMZ.

The effectiveness of the PHC is controlled by the hydrogeologic conditions in the footwall, hanging wall, backfill, and open access workings; and also the length of the PHC in the footwall. As part of the solute transport model analysis, the length of the PHC was established based on the current understanding of site hydrogeologic conditions. Based on the BEJ source concentration (Table 6-1), the following is required in order to minimize metals (such as iron) to levels that are below the PAL at the DMZ boundary:

- Oxygen barriers capable of reducing the estimated formation of oxidized-acidic material by 95 percent, or a combination of the oxygen barriers and oxidized-acidic

backfill removal sufficient to reduce the total amount of potential oxidized-acidic backfill by 95 percent; and

- A 65 percent reduction of solute load from the open access workings through a combination of preventive measures and cleaning.

Given the design, monitoring, and potential enhancements to the planned oxygen barriers detailed in Section 8.1.1.1 above, the oxygen barriers alone are likely to achieve the 95 percent threshold referenced above. Removal of oxidized-acidic backfill will, thus, go beyond what is required to meet the plan. When compared to the unmanaged case, a two-thirds reduction in the solute loading from the open access workings can be achieved through the implementation of NMC's plans, as specified in Section 8.1.1.3 above, for both oxidation controls and cleaning within the access workings.

The solute transport model was used to compare the effectiveness of those features of the RMMP to be implemented before or during the closure phase to the design criteria by assessing concentration increases at the DMZ boundary relative to NR 140 PALs. Appendix D describes the methodology used to arrive at predicted concentration increases at the DMZ boundary, and Table 9-1 compares those increases to NR 140 PALs.

The comparison indicates that the concentration increases at the DMZ boundary in the glacial overburden and strongly weathered bedrock will be below NR 140 PALs for all parameters without either high-volume flushing or hydraulic containment. The NR 140 PALs are more stringent than state and federal drinking water standards. The glacial overburden and, to a lesser degree, the strongly weathered bedrock, are the main drinking water aquifers in the project vicinity. The glacial overburden is also the aquifer that, hydraulically, is directly connected to and interacts with surface waters in the vicinity of the project.

Concentration increases at the DMZ boundary in deeper bedrock units (moderately weathered, weakly weathered, and unweathered bedrock) were also examined. The deeper bedrock units are not typically used for water supply purposes and do not directly interact with surface water resources in the vicinity of the project site (Foth & Van Dyke, 1995/1998b). The reasons the deeper bedrock units are not typically used as a water supply resource are that they are not sufficiently transmissive, and that they host water of questionable quality (Dames & Moore, 1978) that can be expected to degrade with depth.

The analysis suggests that without the implementation of high-volume flushing or hydraulic containment projected concentration increases in the deeper bedrock units for antimony, cobalt, lead, nitrate-nitrite, and sulfate may exceed PALs for those substances. The predicted concentration increases should be considered in the context of potential ACLs where baseline concentrations are above NR 140 standards. The analysis in Appendix E calculates, based on available data, baseline concentrations in the bedrock and, where appropriate, calculates ACLs which would supersede the PALs. In the case of antimony and lead, when the predicted concentration increase is added to baseline data, the resulting concentrations are less than applicable ACLs, as summarized in Table 9-2.

Table 9-1

Comparison of Projected Concentration Increases at DMZ Boundary to NR 140 PALs

		Incremental Concentration Change at DMZ Boundary Due to Source Term[1]					
Parameter	Units	Base of Glacial Overburden	Strongly Weathered Bedrock	Moderately Weathered Bedrock	Weakly Weathered Bedrock	Unweathered Bedrock	NR 140 PAL[2]
Antimony	mg/L	0.00073	0.00069	0.0026	0.0023	0.0028	0.0012
Arsenic	mg/L	0.00040	0.00038	0.0018	0.0018	0.0024	0.005
Barium	mg/L	0.00025	0.00027	0.0016	0.0017	0.0023	0.4
Beryllium	mg/L	0.000063	0.000060	0.00024	0.00022	0.00028	0.0004
Boron	mg/L	0.00012	0.00012	0.00078	0.00093	0.0013	0.19
Cadmium	mg/L	0.000091	0.000079	0.00024	0.00020	0.00023	0.0005
Chloride	mg/L	2.9	3.0	27	36	53	125
Chromium	mg/L	0.000093	0.000088	0.00034	0.00031	0.00037	0.01
Cobalt	mg/L	0.0011	0.0011	0.0094	0.012	0.018	0.008
Copper	mg/L	0.00066	0.00055	0.0016	0.0015	0.0018	0.13
Cyanide	mg/L	0.00011	0.00012	0.0011	0.0014	0.0021	0.04
Fluoride	mg/L	0.00036	0.00035	0.0014	0.0013	0.0017	0.8
Iron	mg/L	0.017	0.016	0.061	0.053	0.062	0.15
Lead	mg/L	0.00047	0.00047	0.0020	0.0019	0.0024	0.0015
Manganese	mg/L	0.0010	0.00096	0.0036	0.0031	0.0037	0.025

Table 9-1 *(Continued)*

Parameter	Units	Incremental Concentration Change at DMZ Boundary Due to Source Term[1]					
		Base of Glacial Overburden	Strongly Weathered Bedrock	Moderately Weathered Bedrock	Weakly Weathered Bedrock	Unweathered Bedrock	NR 140 PAL[2]
Mercury	µg/L	0.0042	0.0041	0.020	0.021	0.029	0.2
Nickel	mg/L	0.00036	0.00034	0.0013	0.0011	0.0013	0.02
Nitrate+Nitrite	mg/L	0.24	0.25	2.2	2.9	4.4	2
Selenium	mg/L	0.00045	0.00046	0.0037	0.0048	0.0070	0.01
Silver	mg/L	0.00012	0.00011	0.00043	0.00038	0.00046	0.01
Sulfate	mg/L	43	49	352	423	584	125
Zinc	mg/L	0.0078	0.0058	0.011	0.010	0.013	2.5

[1] From Table 3 of Appendix D.
[2] NR 140 Preventive Action Limit.

Prepared by: SVD1
Checked by: JJF1

Table 9-2

Assessment of Projected Concentration Increases for Antimony and Lead Relative to Baseline Conditions and ACLs (mg/L)

	Incremented Concentration Increase[1]					
	Moderately Weathered Bedrock	Weakly Weathered Bedrock	Unweathered Bedrock	Baseline[2] Groundwater Quality	NR 140[3] PAL	NR 140[2] ACL
Antimony	0.0026	0.0023	0.0028	0.14	0.0012	0.20
Lead	0.002	0.0019	0.0024	0.044	0.0015	0.065

[1] From Table 3in Appendix D.
[2] From Table 1 in Appendix E.
[3] NR 140 Preventive Action Limit.
Note: For all cases the incremental concentration increase plus baseline groundwater quality is below NR 140 ACLs.

Prepared by: SVD1
Checked by: JJF1

The concentration increases for nitrate-nitrite and cobalt in the deeper bedrock are attributed to the backfill pore water and are below the NR 140 enforcement standards. The predicted concentration increases for sulfate are also driven by the backfill pore water, and, in the deeper bedrock, may exceed NR 140 enforcement standards.

The results of current solute transport modeling demonstrate that implementation of the specified pre-closure engineering controls will result in full compliance with NR 140's objective standards with respect to groundwater in the glacial overburden and strongly weathered bedrock. In the deeper bedrock, cobalt and nitrate-nitrite levels are not considered a threat to public health or welfare, given the hydrogeologic conditions and because the predicted increases are below NR 140 enforcement standards. The analysis suggests, however, that the application prior to reflooding of all technically and economically feasible technologies as required by NR 140.22(1) may not be sufficient to reduce sulfate concentrations below the applicable NR 140 enforcement standards in the deeper bedrock. Given that the deeper bedrock is not typically used for water supply purposes and is known to host groundwater of poor quality (Dames & Moore, 1978) that likely degrades with depth, the predicted sulfate increases in the deeper bedrock units are not expected to pose any threat to public health or welfare.

Nonetheless, in the event that an evaluation of post-closure groundwater monitoring data shows that concentrations of any regulated parameter will exceed an NR 140 standard at the DMZ boundary, either or both do the high-volume flushing or hydraulic containment measures would be implemented.

9.2 Sensitivity Analysis

As detailed in Sections 6 and 7, the assessment of concentration increases at the DMZ boundary is largely controlled by the estimated source concentrations. The main difference between the BEJ concentrations and UB concentrations is the amount of oxidized-acidic backfill. The generation of such material is addressed via management practices such as the incorporation of oxygen barriers and the removal, at the end of mining, of oxidized-acidic backfill that may form. Therefore, the above analysis incorporating UB source concentrations with management controls would yield results comparable to those set out in Table 9-1.

10 Implementation Process

Figure 10-1 is a flow chart describing the elements of the plan and decisions that will be made to secure its successful implementation. Implementation of the plan will commence with the collection of baseline groundwater data. Data from that program will be used to establish baseline concentrations for each well and, where applicable, ACLs. Operational groundwater quality monitoring will begin at the time of construction. During construction, localized areas of wall rock in the access workings that are observed to be oxidizing will be shotcreted to control the generation of additional oxidation products. Oxygen barriers will be installed in cross-cuts, beginning with the first backfilled stope. Each barrier will be tested for effectiveness and modified as required to achieve performance goals in accordance with the plan. Also, during operations, backfill pore water will be sampled and analyzed. The RMMP will be updated every five years based on pertinent information collected in accordance with the plan. The final design of the plan will be submitted to the WDNR for review and approval two years prior to closure.

During the closure phase, the oxygen barriers will be removed and the exposed backfill surfaces in the cross-cuts will be tested for oxidized-acidic material. If such material is present, it will be removed and processed in accordance with the plan. Localized areas of oxidizing wall rock will also be physically removed and processed during this phase. Spilled ore and muck will be removed from the mine and processed. The open access workings will then be cleaned. These closure steps will be documented in accordance with the procedures outlined in the RMMP, and a documentation report will be submitted to the WDNR. Reflooding of the mine will then commence by terminating groundwater withdrawal from the mine and accelerating the reflooding process by pumping clean groundwater into the access workings.

The post-closure period will commence with the implementation of the post-closure groundwater monitoring program. Groundwater monitoring data collected during the post-closure period will be evaluated annually. As illustrated in Figure 10-1, the evaluation process involves two major elements. The first relates to the quality of the water in the open access workings themselves. The second relates to the quality of water collected from the various groundwater monitoring wells to be used in the post-closure monitoring program.

In relation to the access workings, the decision process shown in Figure 10-1 focuses on determining if data from the access workings indicates a potential NR 140 PAL exceedance at the DMZ boundary. In that event, high-volume flushing would be implemented. If the data suggests no potential for an exceedance, routine monitoring will continue.

With respect to groundwater monitoring wells, the decision process focuses on determining whether the data show or suggest potential NR 140 ES and/or PAL exceedances at the point of standards application. In that event, the data would be evaluated within the context of final RMMP predictions and NR 140.28 to determine if there is a threat to public health and welfare. If no such threat exists, an ACL would be sought per NR 140.28 or NR 182.19. In the event a threat to public health or welfare exists, hydraulic containment will be implemented. If the data were to show or suggest that exceedances do not or will not occur, routine monitoring will continue.

11 Summary and Conclusions

The RMMP focuses on those actions needed to protect groundwater quality after mine reflooding. It includes both a pre-closure and a post-closure component. The pre-closure component consists of features that will be implemented during operations and closure. The post-closure component consists of measures that will be used to manage the reflooded mine by monitoring groundwater quality and implementing additional plan features, as needed. Chapter NR 140 applies to each of these phases in different ways. With respect to the first, NR 140.22(1) requires that facilities be designed to minimize the level of substances in groundwater and to comply with NR 140 PALs to the extent technically and economically feasible. As to the second, the reflooded mine must be managed in accordance with specific applicable NR 140 groundwater standards through monitoring.

Technologies for minimizing the concentration of substances in groundwater were screened based on technical criteria. Those technologies that were technically feasible, or potentially feasible, were then screened again as to economic viability. Technologies that were deemed both technically and economically feasible were evaluated with a source concentration model and solute transport model to further assess each with respect to design objectives. Based on the results of the evaluation process, the following pre-closure engineering features have been incorporated into the plan:

- Construction of drifts in the footwall and connecting the footwall drifts and hanging wall drifts at the western end of the mine. This passive hydraulic control will help in minimizing the concentration of solutes in the groundwater.

- Construction of oxygen barriers in the cross-cuts of each stope to reduce or eliminate the formation of oxidized-acidic backfill.

- Removal of such oxidized-acidic backfill, if present, at the end of mining.

- Cleaning the open access workings at the end of mining by physically removing localized areas of oxidized wall rock, physically removing spilled ore and muck, and by pressure washing the walls of the access workings.

- Accelerated reflooding of the mine to reduce additional oxidation of sulfide minerals.

An assessment of the facility design incorporating pre-closure engineering features was completed using the solute transport model. Results of the analysis indicate that, in the glacial overburden and strongly weathered bedrock, the solute concentration increases at the DMZ boundary will be below NR 140 PALs. The glacial overburden and, to a lesser extent, the strongly weathered bedrock are the main drinking water aquifers in the vicinity of the project. The glacial overburden is also the aquifer that directly interacts with surface waters in the vicinity of the project.

During the course of operations, the RMMP will be updated every five years based on an ongoing review of hydrogeologic data collected during operations and updated technical and economic assessments of design alternatives. The plan updates will be used to determine whether new technologies or methods yield better results at a manageable cost and, therefore, should become part of the then-current or final RMMP. The final RMMP will be submitted two years before the end of mining, and will include detailed evaluations of the then-available means and methods against the design requirements of NR 140.22(1), using the most current predictive models.

After closure, the post-closure component of the plan will be implemented, consisting of monitoring groundwater quality and the implementation of additional engineering features, if needed, to protect groundwater quality. These measures include a high-volume flushing system to remove solute mass emanating from sources in the open workings, and a hydraulic containment system to confine solute emanating from sources in the backfill.

12 References

Arneson, R.T., et al., 1991. *Monitoring Water Quality During Filling of the Lokken Mine: A Possible Role of Sulfate Reducing Bacterial in Metals Removal.* In: Proceedings of the Second International Conference on the Abatement of Acidic Drainage, Montreal, PQ, Canada. September 16-18, 1991.

Bennett, P.G., C.R. Ferguson, and T. Jeffers, 1991. *Biological Treatment of Acid Mine Waters - Case Studies.* In: Proceedings of the Second International Conference on the Abatement of Acidic Drainage, September 16-18, 1991, Montreal, Canada. pp. 283-299.

Blowes, D.W., C.J. Ptacek, J.G. Bain, K.R. Waybrant, and W.D. Robertson, 1995. *Treatment of Mine Drainage Water Using In Situ Permeable Reactive Walls.* In: Mining and the Environment Conference Proceedings, Sudbury, Ontario, Canada. May 28-June 1, 1995.

Dames & Moore, 1978. *Results of Permeability Tests and Analysis of Water Samples from Deep Exploration Holes, Crandon Ore Body, Crandon, WI.*

Ferguson and Leask, 1988. *The Export of Nutrients from Surface Coal Mines.* Canada Regional Program Report 87-12. March 1988.

Foth & Van Dyke, 1995/1998a. *Mine Permit Application.* Originally Issued May 1995. Updated December 1998.

Foth & Van Dyke, 1995/1998b. *Environmental Impact Report.* Originally Issued May 1995. Updated December 1998.

Foth & Van Dyke, 1995/1998c. *Preliminary Engineering Report for Wastewater Treatment Facilities.* Originally Issued September 1995. Updated November 1998.

Foth & Van Dyke, 1999/2000. *Tailings Management Area Groundwater Quality Performance Assessment.* Originally Issued in June 1999. Updated in March 2000.

HSI GeoTrans, Inc., 1998. *Addendum No. 2 to: Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Zinc and Copper Mine Near Crandon, Wisconsin.* August 1998.

HSI GeoTrans, Inc., 1999. Groundwater Recovery at the Crandon Site - Effect of Mitigation Pumpage. Technical Memorandum from Greg Council to Steve Dischler dated May 21, 1999.

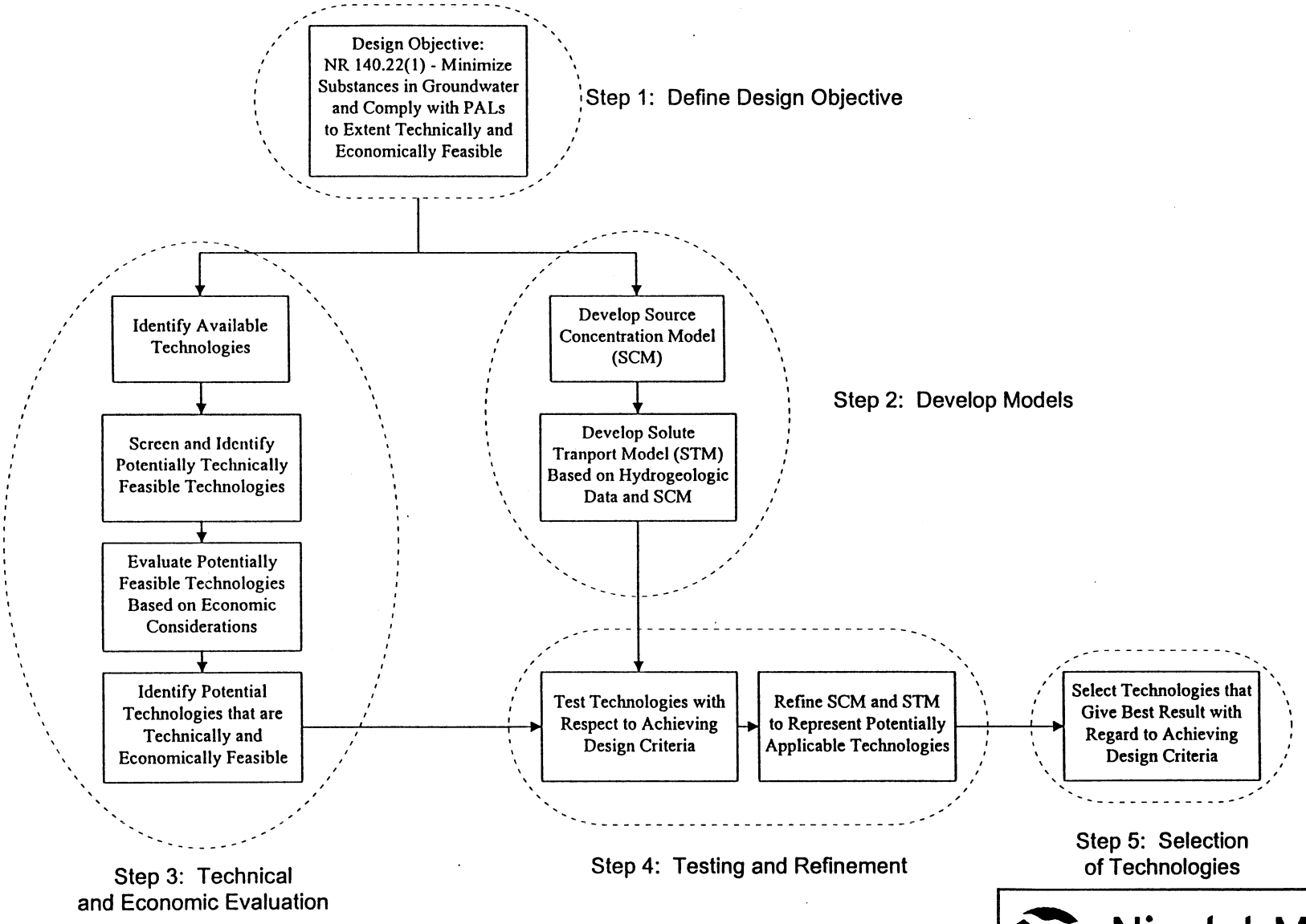
HSI GeoTrans, 2000. *Solute Transport Modeling of the Backfilled Crandon Mine After Reflooding.* November 15, 2000.

Scheetz, Barry, 2000. The Role and Control of Sulfate in Portland Cement. Memorandum to Mr. Ken Black, Nicolet Minerals Company, dated September 9, 2000.

SRK Consulting, 2000. *Reflooded Mine Source Concentrations, Crandon Project, Wisconsin.* November 2000.

Wisconsin Department of Natural Resources, 1994. *PAL/ACL Calculations Guidance for Solid Waste Facilities.* May 6, 1994.

FIGURES FOR CRANDON PROJECT REFLOODED MINE MANAGEMENT PLAN



Foth & Van Dyke				
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APPROVED BY:		SVDI	DATE:	NOV. 2000
APPROVED BY:		GWS	DATE:	NOV. 2000

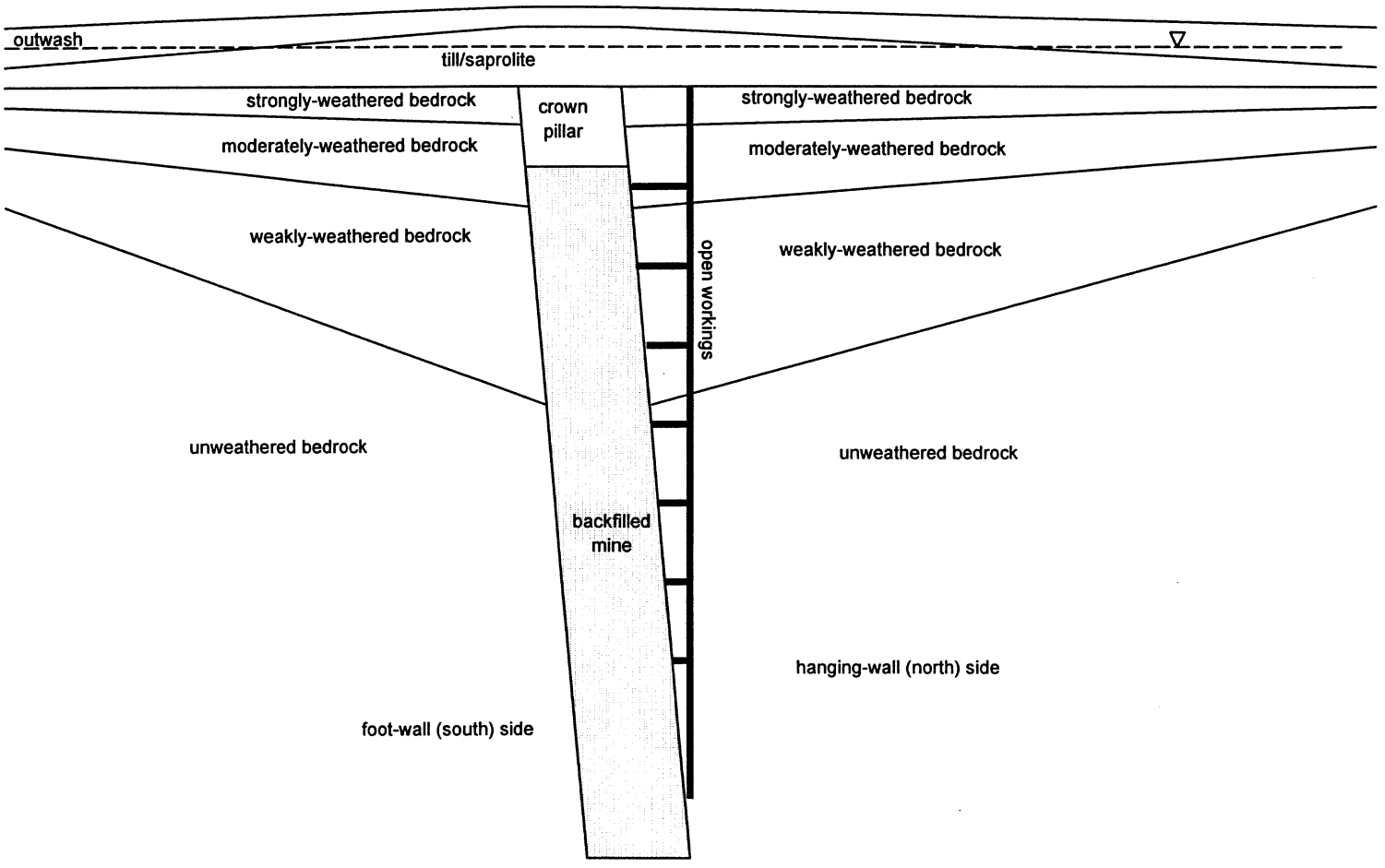
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FIGURE 4-1
FACILITY DESIGN PROCESS

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Prepared By: Foth & Van Dyke		By: JRB2	00C002

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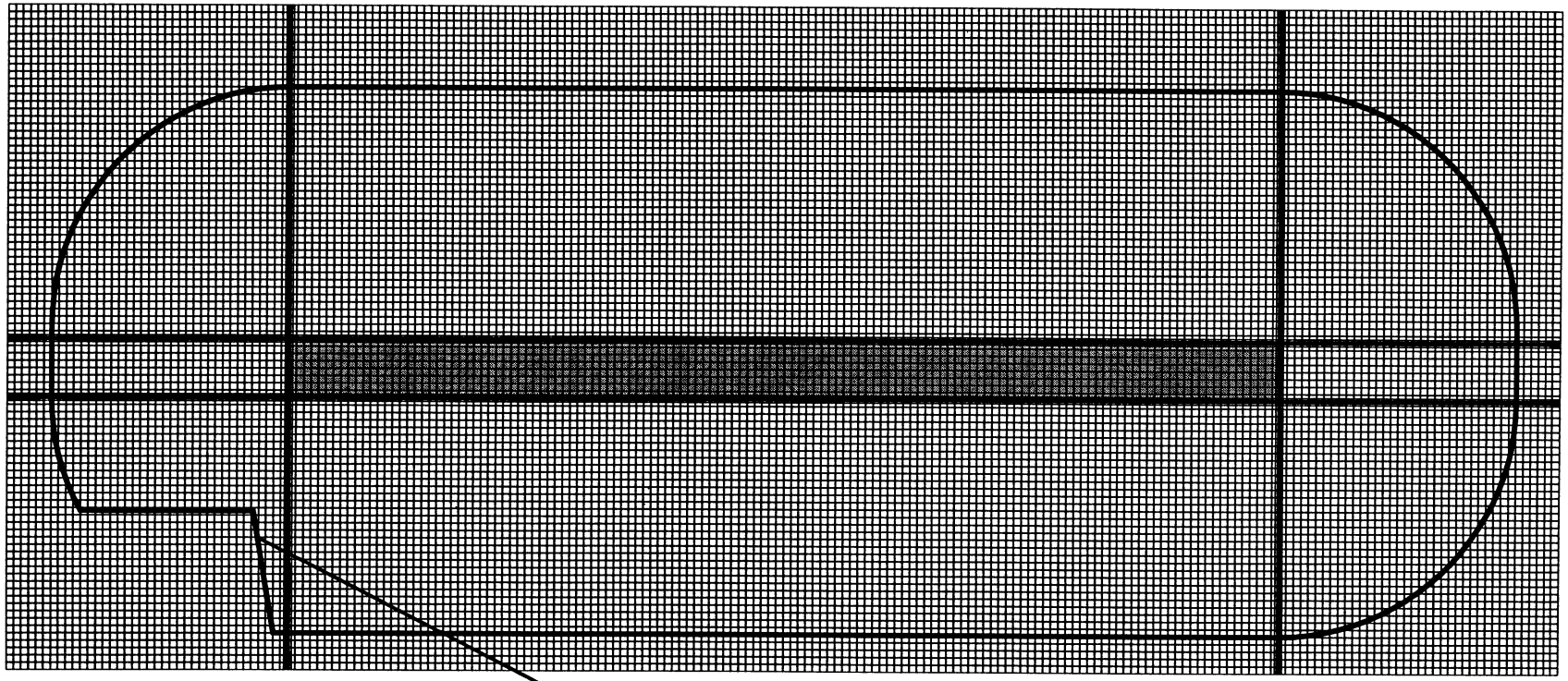
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APPROVED BY: PFA	DATE: 11/00
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Figure 7-1
Conceptual North-South Cross Section
Showing Hydrogeologic Units

Scale: None	Date: Nov 2000
Prepared By: HSI GEOTRANS	By: GWC

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Note: The distance to the DMZ boundary is less than 1200 ft in this area. The DMZ boundary in this area is the NMC property boundary.

Legend

- Design Management Zone (Approx.)
- Paste Backfill Zone

0 900 1800

Scale in Feet

N

HSI GEOTRANS

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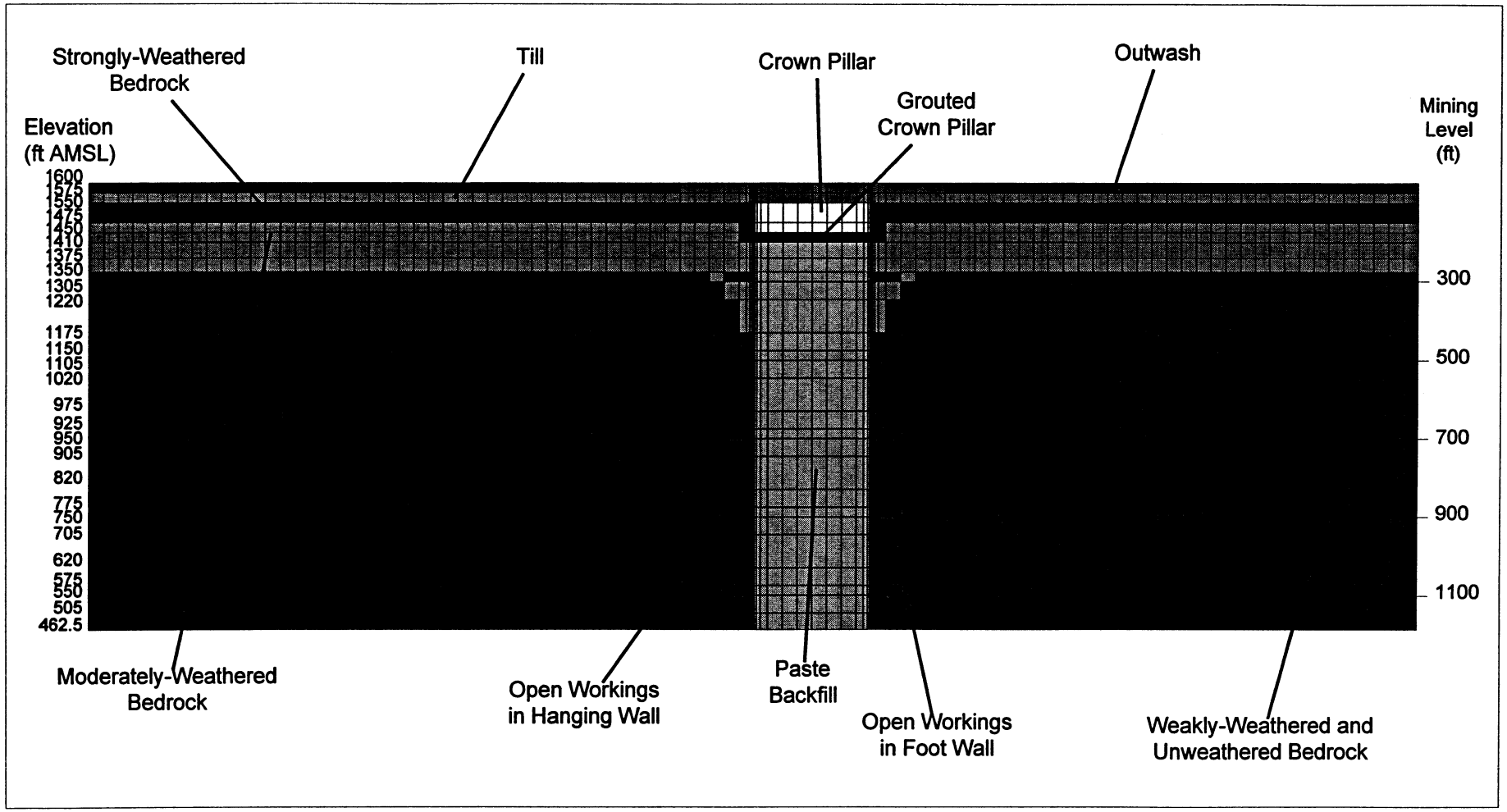
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APPROVED BY: PFA	DATE: Nov 2000
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Figure 7-2
Model Grid

Scale: 1 in. = 900 ft.	Date:November 2000
Prepared By: HSI GeoTrans	By: LMG

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0 400 800

Scale in Feet

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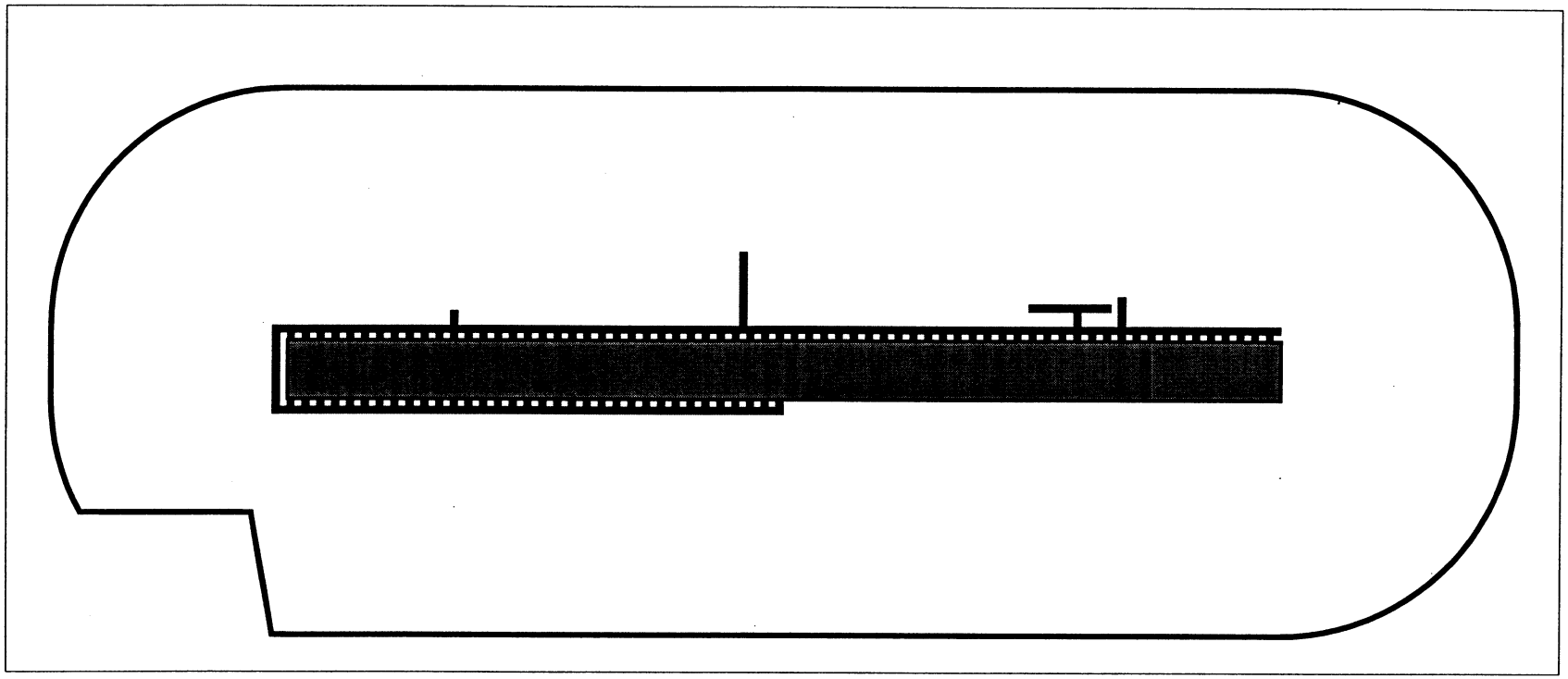
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APPROVED BY: PFA	DATE: Nov 2000
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Figure 7-3
Layer Elevations and Hydrogeologic Units, North-South Cross-Section View

Scale: 1 in. = 400 ft.	Date: Nov 2000
Prepared By: HSI GeoTrans	By: LMG

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Legend

- Cemented Paste Backfill
- Bedrock
- Open Workings

0 900 1800

Scale in Feet

N

Note: This configuration applies at every mining level (layers 8, 12, 16, 20, and 24).

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REVISED	DATE	BY	DESCRIPTION

CHECKED BY GWC DATE: Nov 2000

APPROVED BY PFA DATE: Nov 2000

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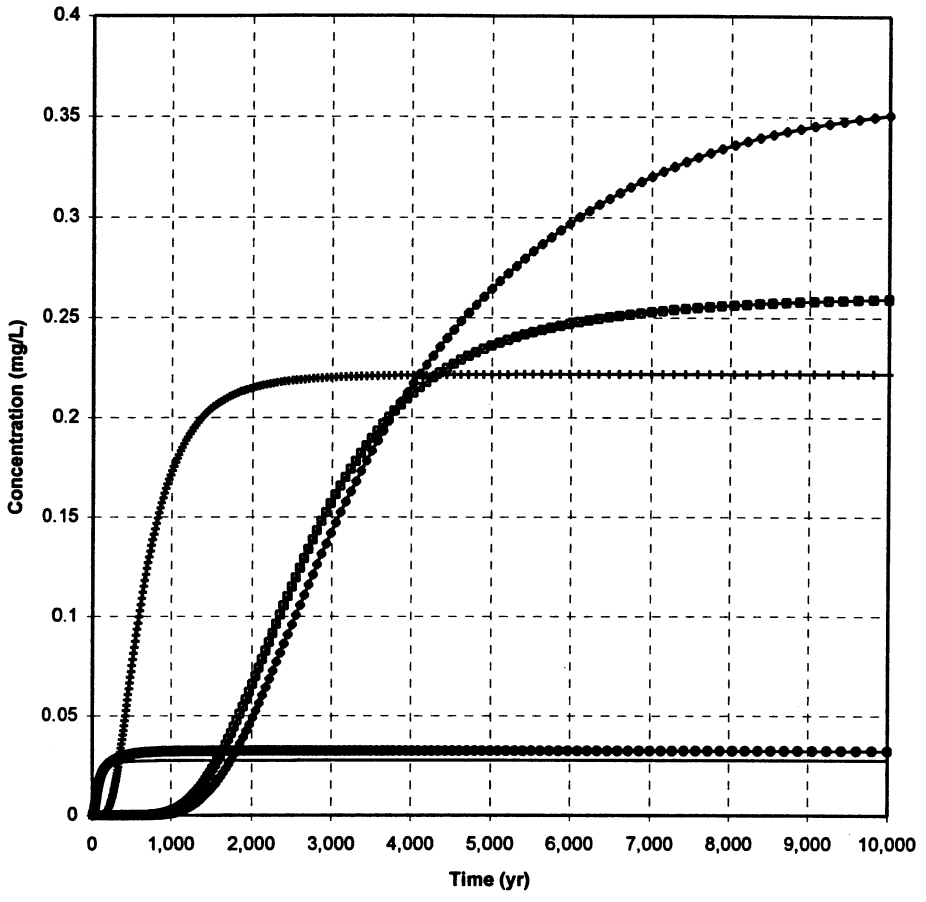
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Figure 7-4
Open Workings Configuration in the Solute Transport Model

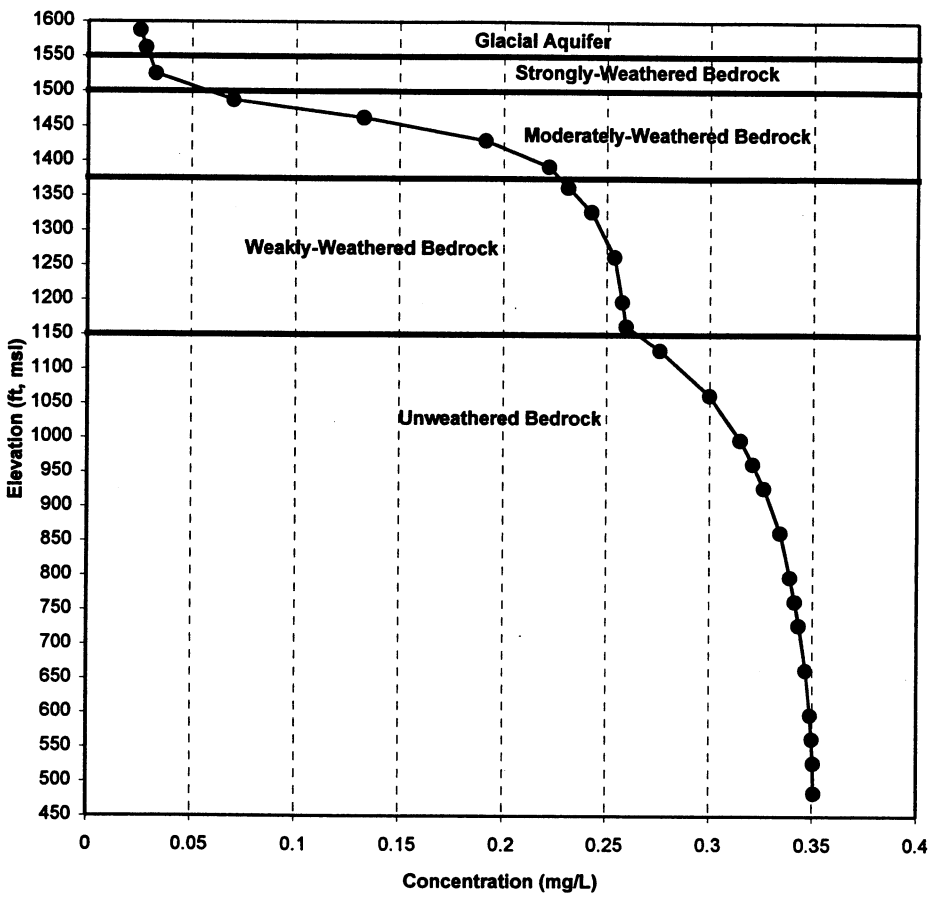
Scale: 1 in. = 900 ft. | Date: Nov 2000

Prepared By: HSI GeoTrans | By: LMG

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Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer evaluated independent of the simulation time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 40- 7)	0.028
	Strongly-Weathered	(3- 40- 7)	0.032
	Moderately-Weathered	(7- 85- 34)	0.22
	Weakly-Weathered	(12-101- 57)	0.26
	Unweathered	(26-101- 73)	0.35

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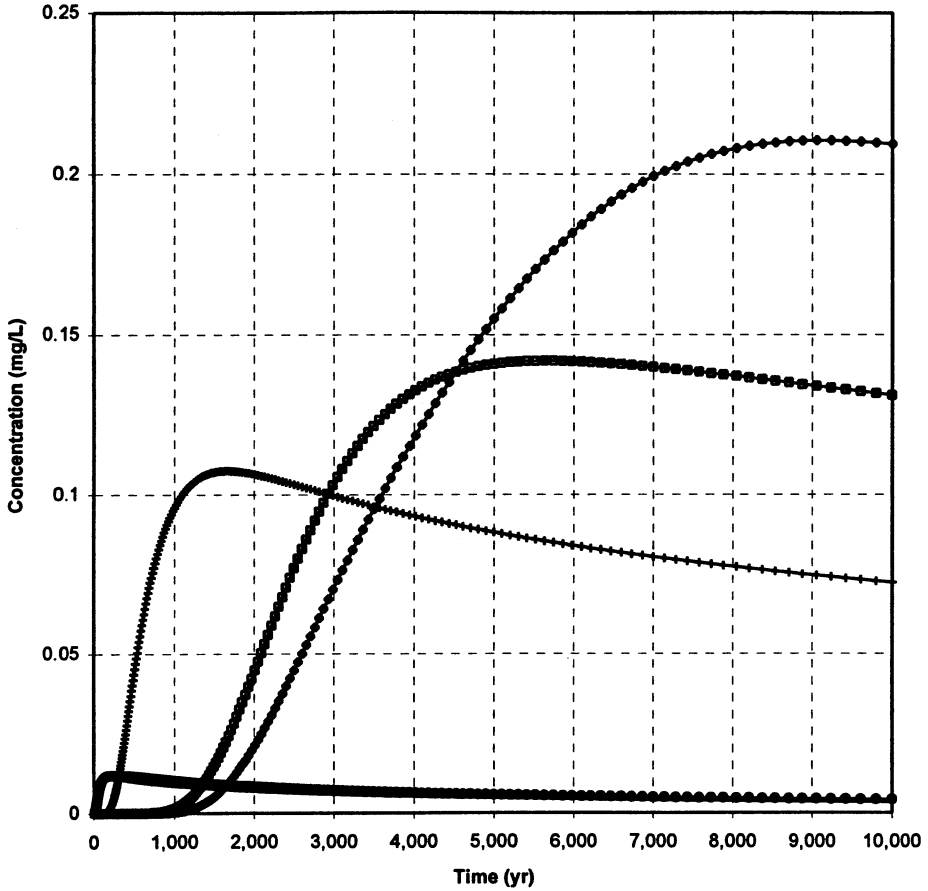
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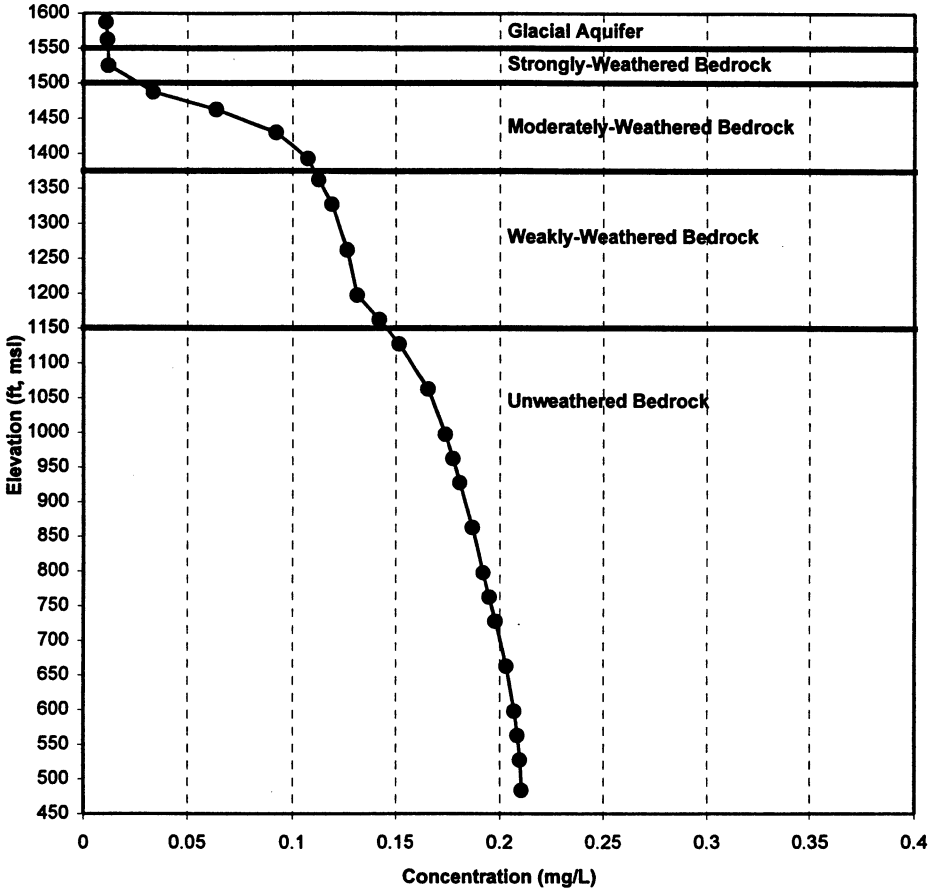
Figure 7-5
Concentration at the DMZ Boundary, Unoxidized Backfill Source

Scale: None	Date: November 2000
Prepared By: HSI GEOTRANS	By: LMG

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Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 38- 7)	0.011
	Strongly-Weathered	(3- 39- 7)	0.012
	Moderately-Weathered	(7- 85- 34)	0.11
	Weakly-Weathered	(12-101- 59)	0.14
	Unweathered	(26-101- 79)	0.21

HSI GEOTRANS

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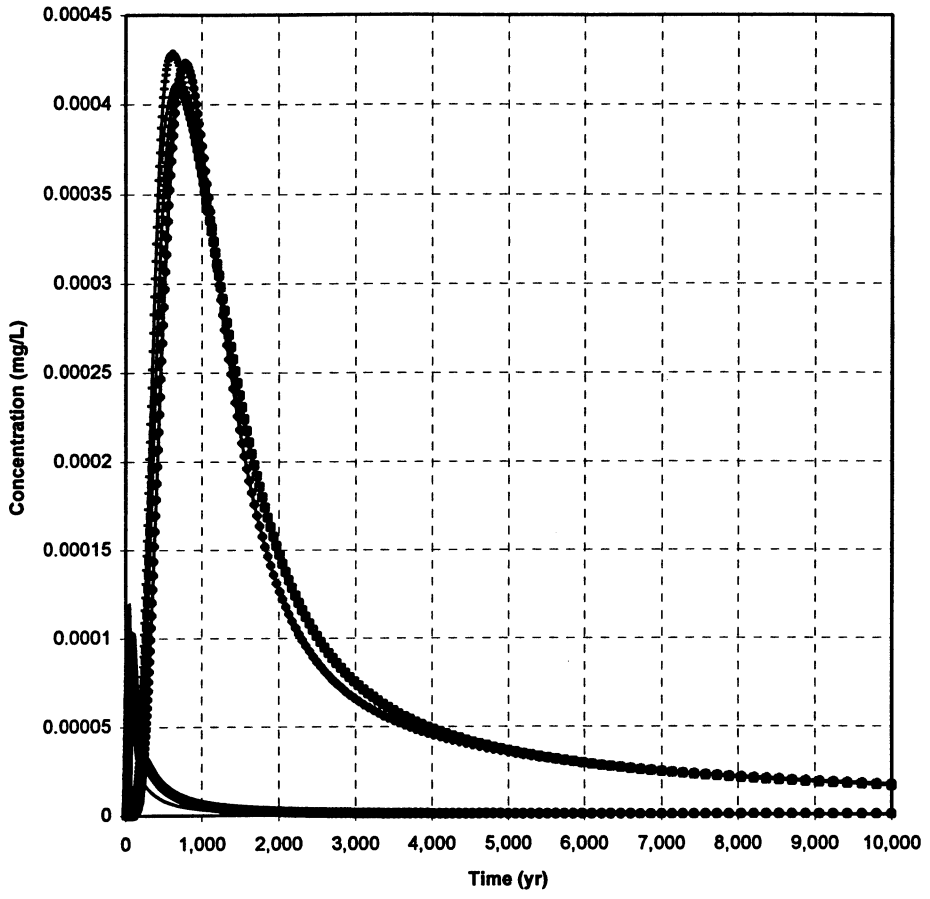
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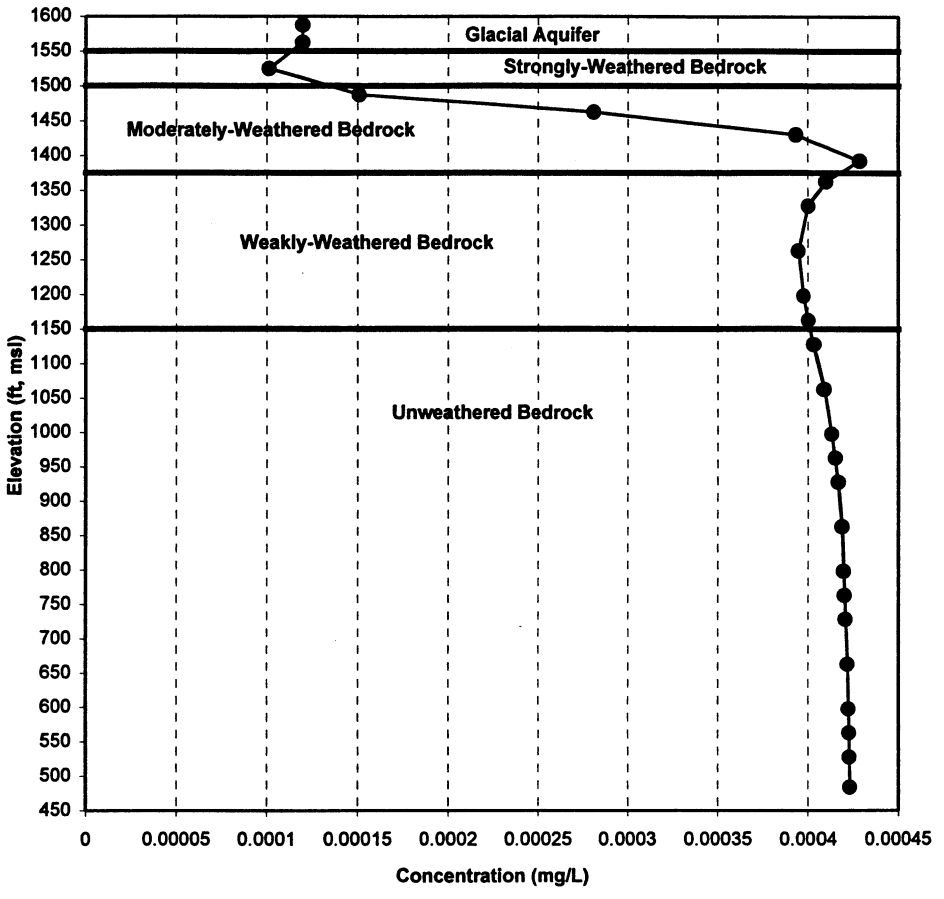
Figure 7-6
Concentration at the DMZ Boundary, Unoxidized Backfill Source (Instantaneous)

Scale: None	Date: November 2000
Prepared By: HSI GEOTRANS	By: LMG

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Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(1- 35- 8)	0.00012
	Strongly-Weathered	(3- 35- 8)	0.00010
	Moderately-Weathered	(7- 85- 34)	0.00043
	Weakly-Weathered	(8- 85- 34)	0 00041
	Unweathered	(26- 85- 34)	0 00042

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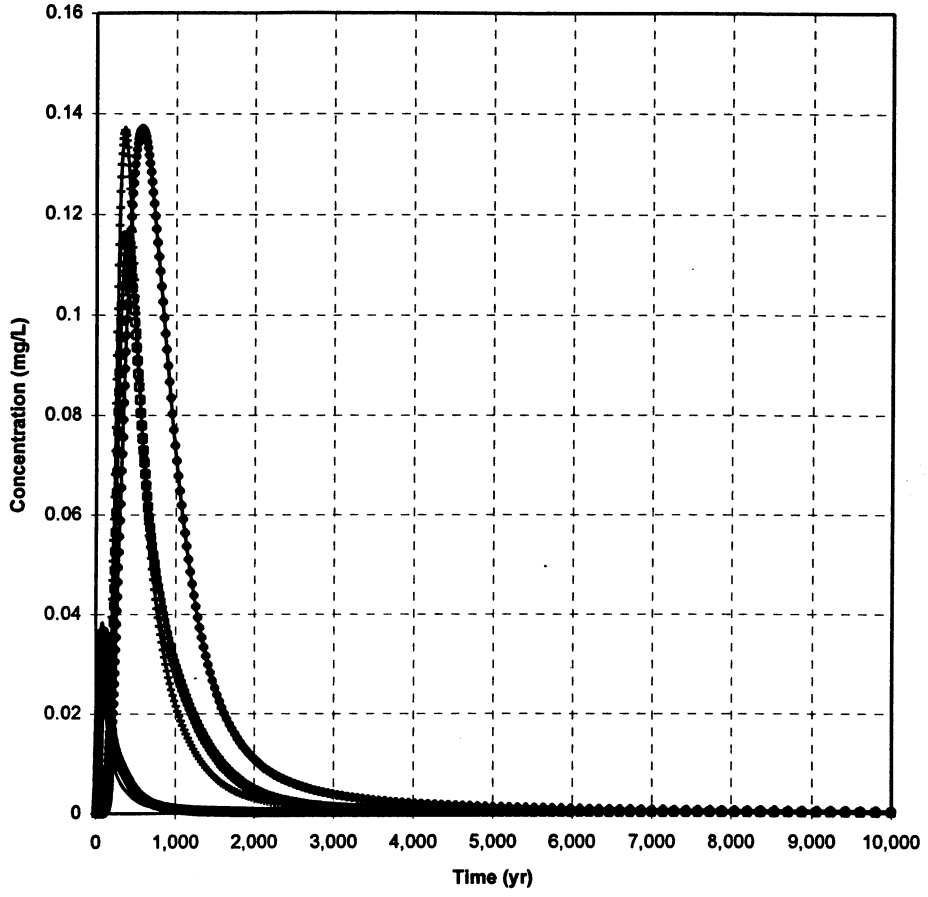
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Figure 7-7
Concentration at the DMZ Boundary, Acidic Backfill Source in Cross Cut

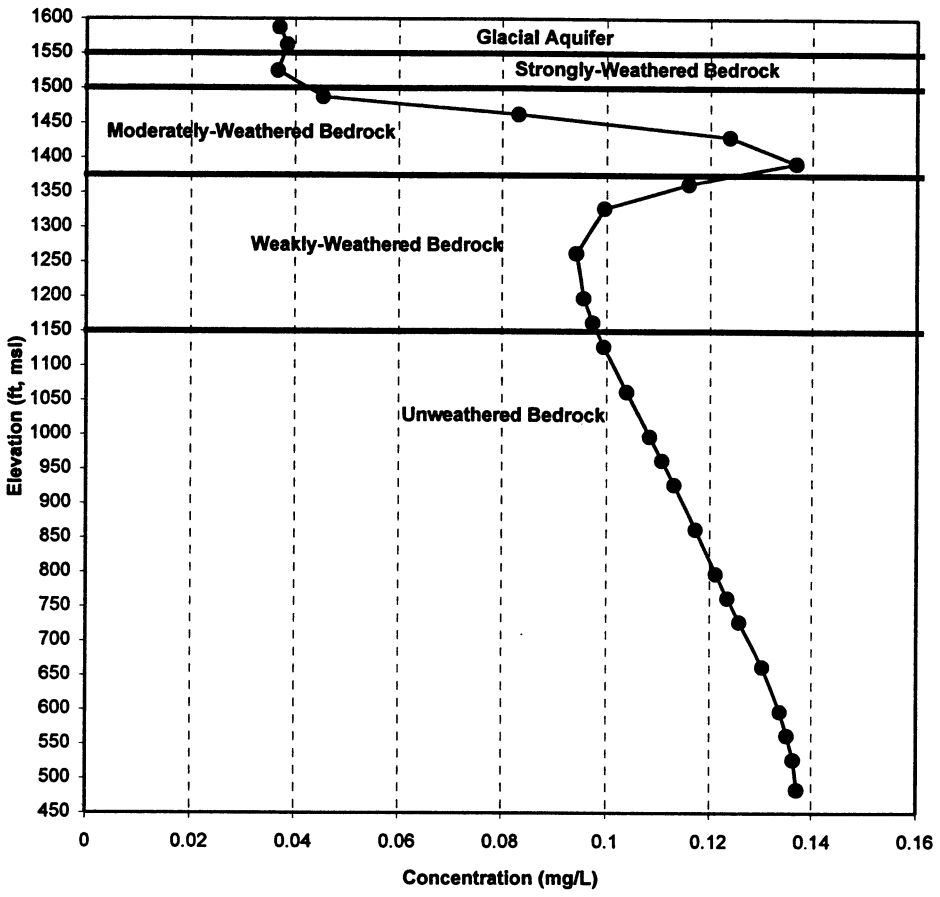
Scale: None | Date: November 2000

Prepared By: HSI GEOTRANS | By: LMG

p:\Crandon\Reflood3d\excel\rf151.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 44- 7)	0.038
	Strongly-Weathered	(3- 44- 7)	0.037
	Moderately-Weathered	(7- 56- 7)	0.14
	Weakly-Weathered	(8- 56- 7)	0.12
	Unweathered	(26- 85- 34)	0.14

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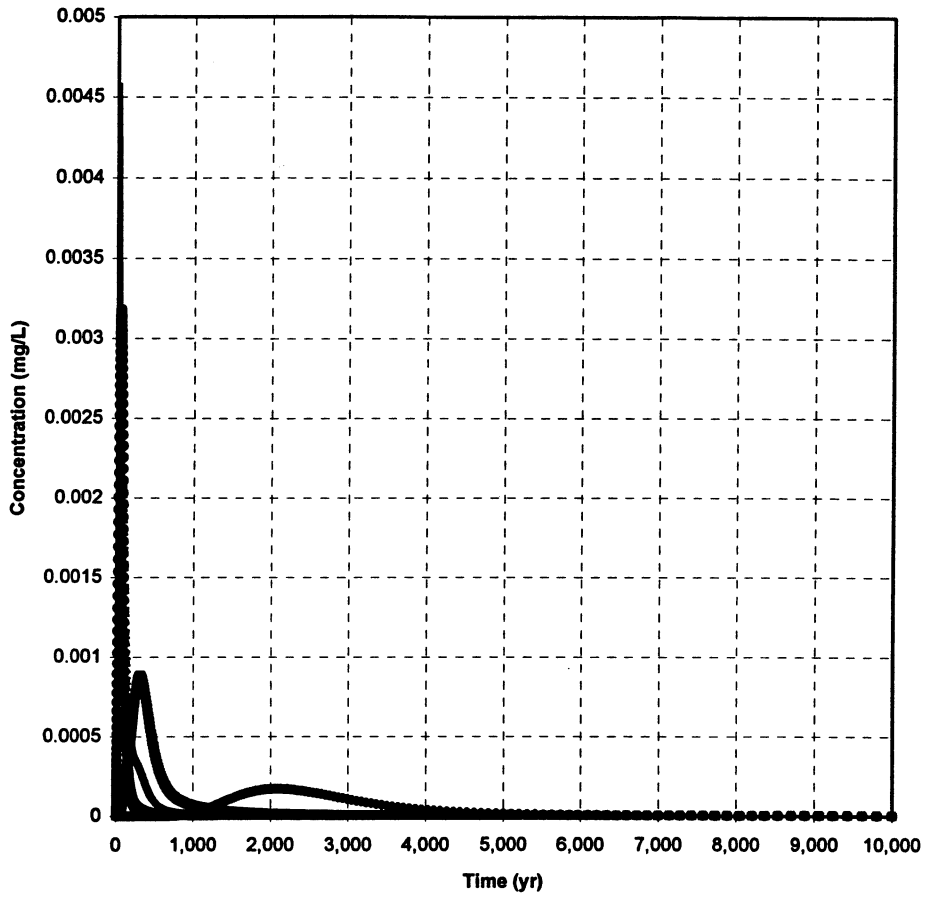
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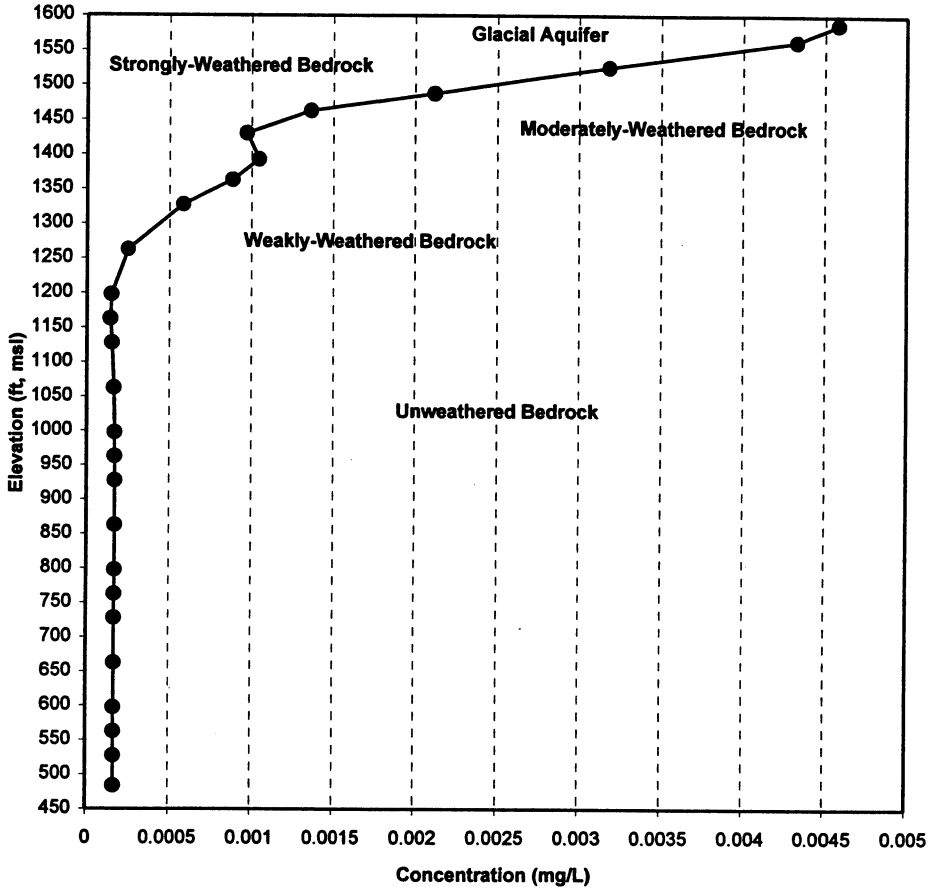
Figure 7-8
Concentration at the DMZ Boundary, Wall Rock Source

Scale: None	Date: November 2000
Prepared By: HSI GEOTRANS	By: LMG

p:\Crandon\Reflood3d\excel\rf152.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(1- 38- 7)	0.0046
	Strongly-Weathered	(3- 38- 7)	0.0032
	Moderately-Weathered	(4- 37- 7)	0.0021
	Weakly-Weathered	(8- 35- 8)	0.00088
	Unweathered	(18- 12- 83)	0.00017

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Figure 7-9
Concentration at the DMZ Boundary, Crown Pillar Source

Scale: None	Date: November 2000
Prepared By: HSI GEOTRANS	By: LMG

PROJECT PHASE	PERIOD OF ACTIVITY	ACTIVITY
A) PRE-CONSTRUCTION	1-YEAR	• COLLECT BASELINE GROUNDWATER QUALITY DATA. • ESTABLISH FINANCIAL RESPONSIBILITY.
B) CONSTRUCTION	3 YEARS	• TECHNOLOGY EVALUATION.
C) OPERATIONS	28 YEARS	• CONTINUE TECHNOLOGY EVALUATION. • ASSESS HYDROGEOLOGIC CONDITIONS. • IMPLEMENT INITIAL ENGINEERING CONTROLS SUCH AS OXYGEN BARRIERS. • MONITOR THE PERFORMANCE OF IMPLEMENTED FEATURES. • SUBMIT 5-YEAR UPDATES OF RMMP AT YEARS 5, 10, 15, 20, 25. • FINAL DESIGN ASSESSMENT. • SUBMIT FINAL DESIGN AT YEAR 26.
D) CLOSURE	2 YEARS	• IMPLEMENT PHYSICAL SOURCE CONTROLS AND OTHER MANAGEMENT CONTROLS AFTER MINING CEASES. • DOCUMENT CLOSURE PER TRANSITION CRITERIA. • ACCELERATE MINE REFLOODING. • NOTIFICATION TO WDNR OF COMPLETION OF RECLAMATION PLAN.
E) POST-CLOSURE	AFTER CLOSURE	• MONITORING. • HIGH VOLUME FLUSHING OF WORKINGS, IF NEEDED. • HYDRAULIC CONTAINMENT, IF NEEDED.

Foth & Van Dyke

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1	3/01	GWS	ADDED ACTIVITIES

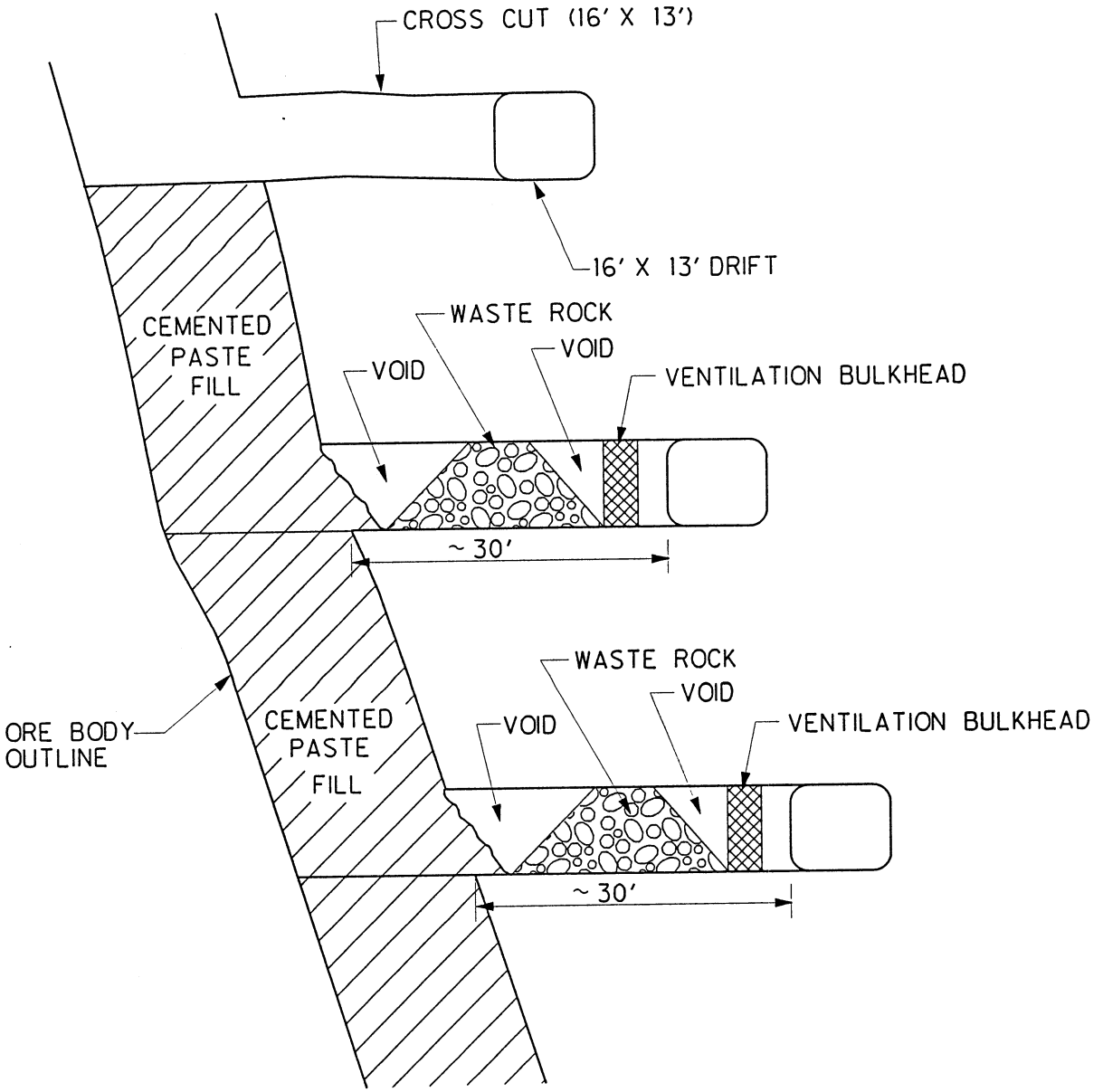
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APPROVED BY:	SVDI	DATE:	NOV. 2000
APPROVED BY:	JWS	DATE:	NOV. 2000

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FIGURE 8-1
IMPLEMENTATION TIME LINE

Scale: NOT TO SCALE		Date: NOVEMBER, 2000	
Prepared By: Foth & Van Dyke	By: DAT	01C002	

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SECTION

NOTES:

1. VENTILATION BULKHEAD WILL BE CONSTRUCTED WITH TIMBER FRAME AND PLASTIC SHEETING.
2. ENHANCEMENT MEASURES, IF NEEDED, COULD INCLUDE: PLACEMENT OF ORGANICS BEHIND BULKHEAD TO REDUCE O_2 LEVELS IN CROSS-CUT; SEALING TIMBER STRUCTURES AT INTERFACE WITH BEDROCK; KEYING STRUCTURE INTO WALL ROCK; GROUTING ROCK AROUND THE CROSS-CUT; AND CONSTRUCTION OF A SECONDARY BULKHEAD.

Foth & Van Dyke

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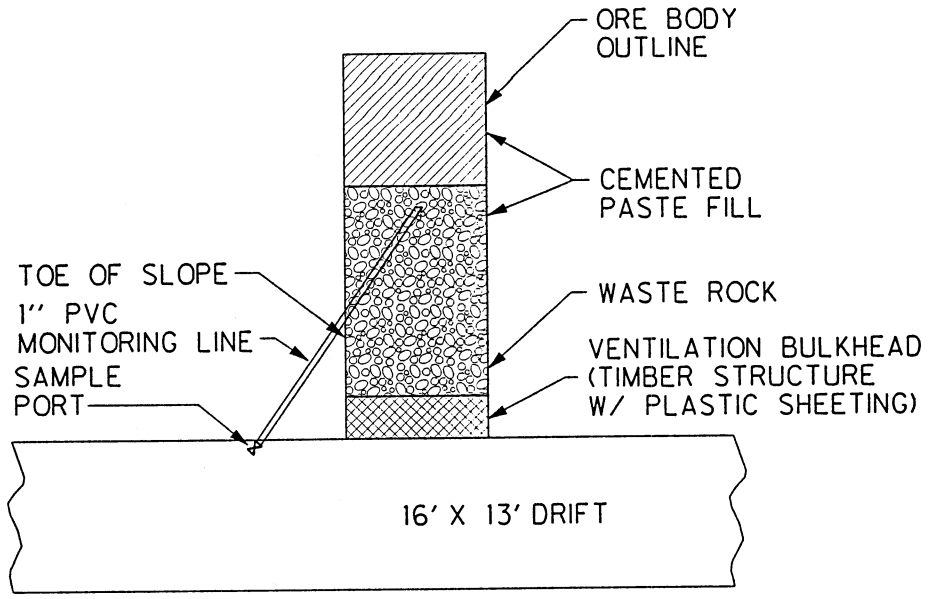
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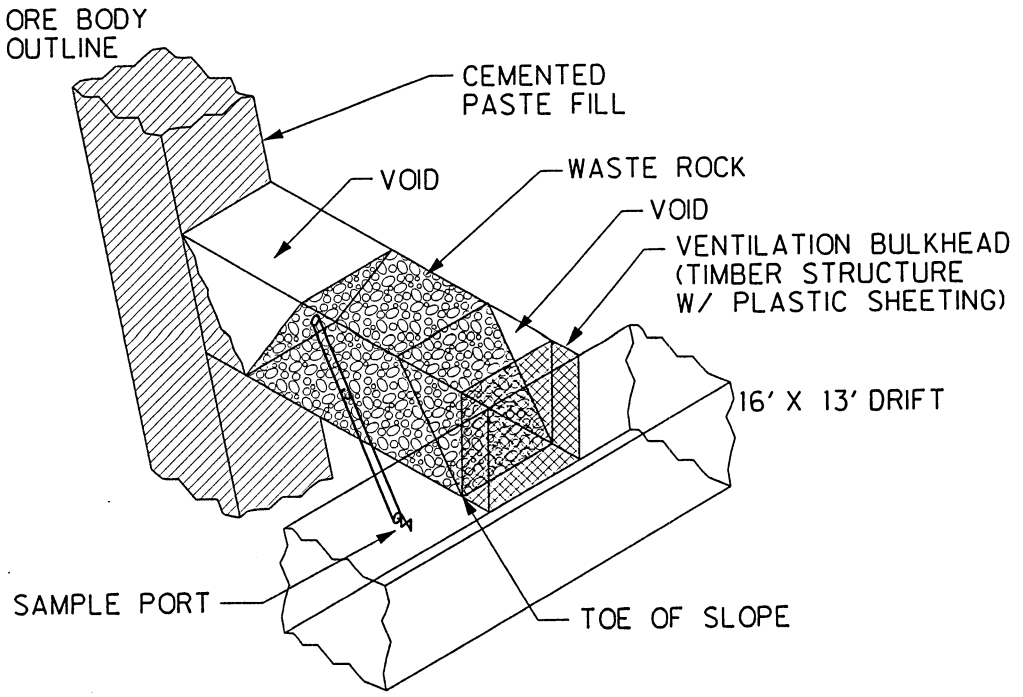
FIGURE 8-2
BULKHEAD LOCATION

Scale: NOT TO SCALE		Date: NOVEMBER, 2000	
Prepared By: Foth & Van Dyke		By: JRB2	00C002

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PLAN VIEW



ISOMETRIC VIEW

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JRB2	DATE:	NOV. 2000
APPROVED BY:	SVDI	DATE:	NOV. 2000
APPROVED BY:	JWS	DATE:	NOV. 2000

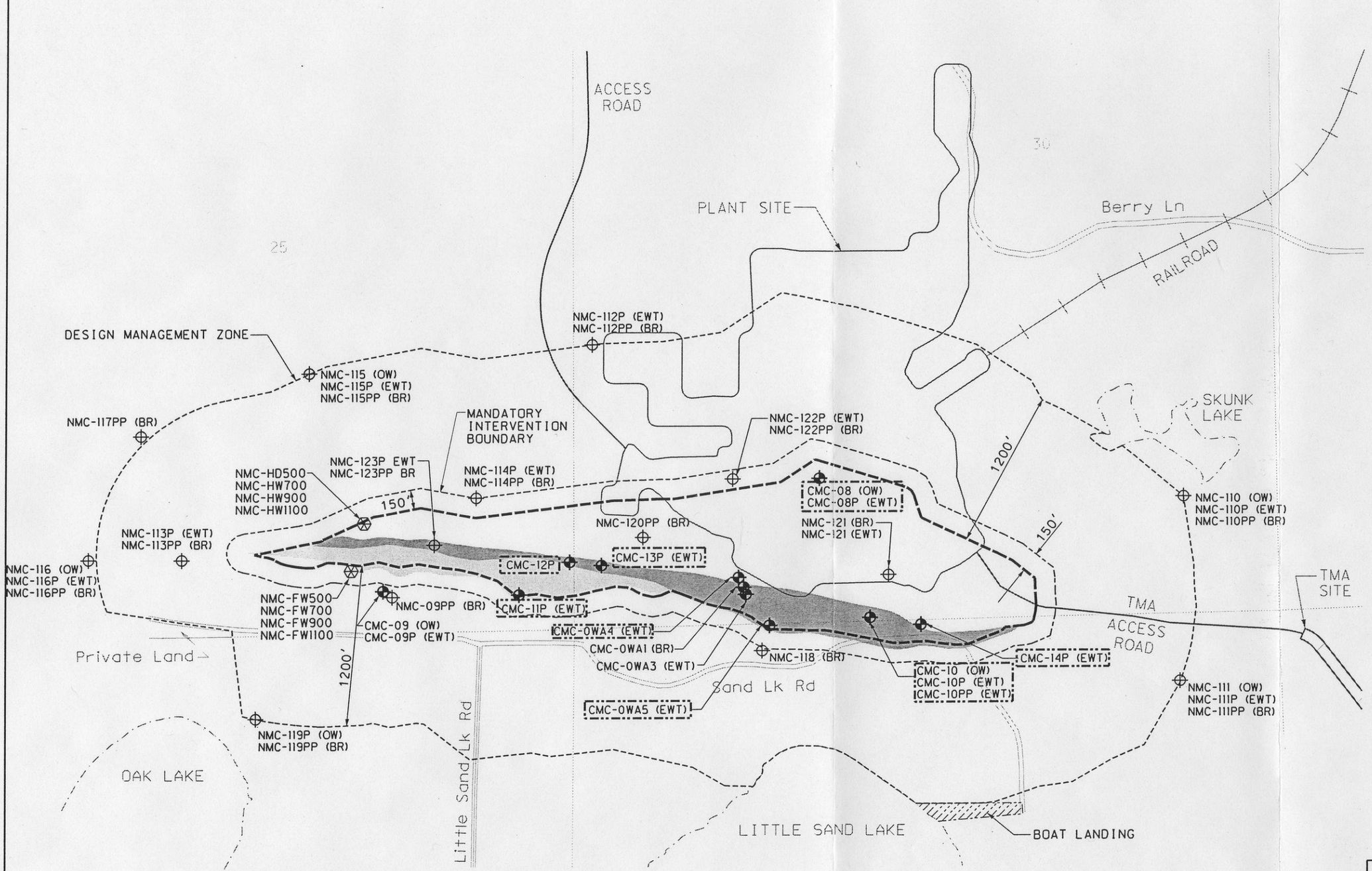
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FIGURE 8-3

OXYGEN PROBE MONITORING SETUP

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Prepared By: Foth & Van Dyke		By: GJB	00C002

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NOTES:

1. OW: OUTWASH WELL
2. EWT: PRE-TO EARLY WISCONSINAN TILL WELL
3. BR: WEATHERED BEDROCK WELL

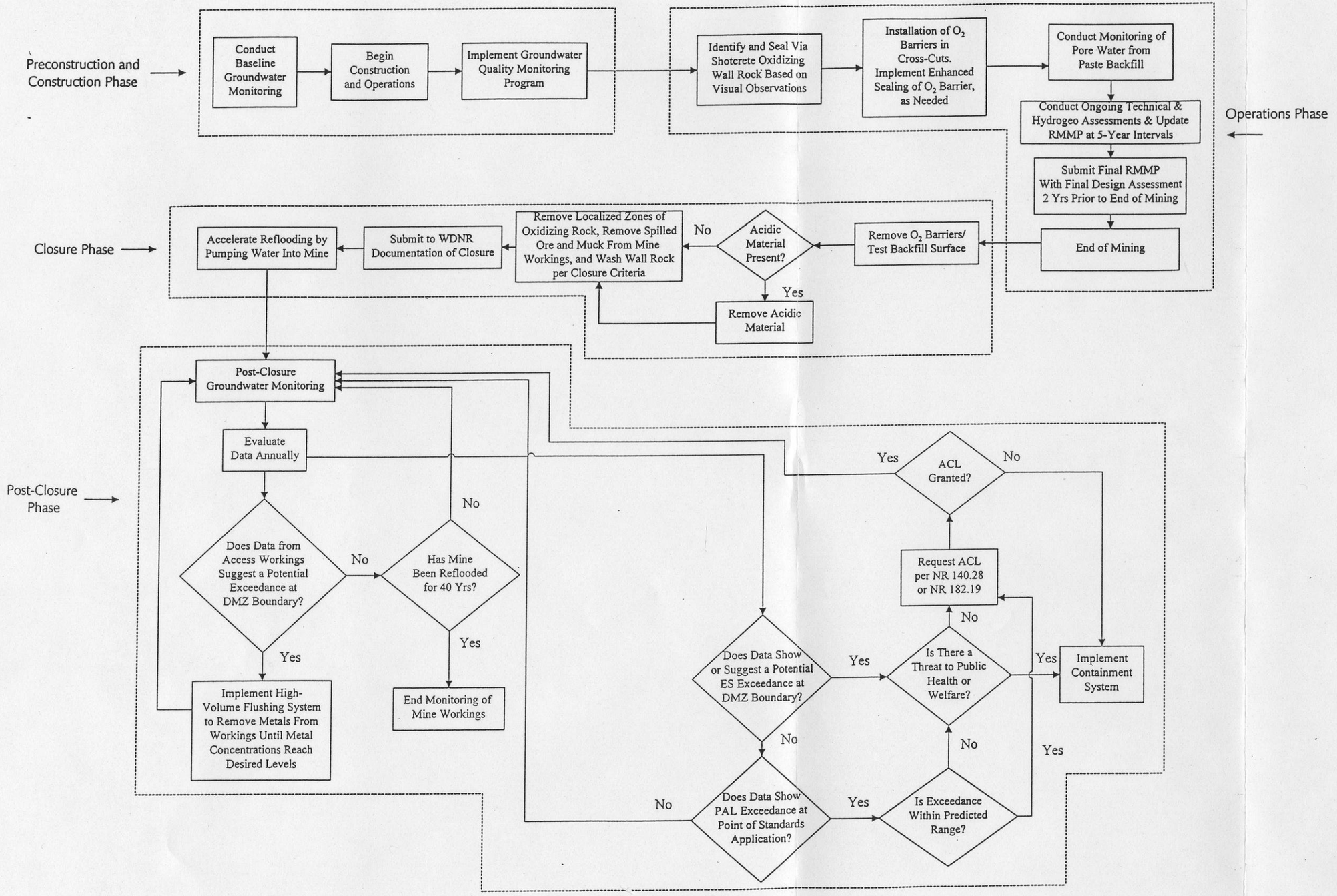
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APPROVED BY:	SVDI		DATE:	NOV. 2000	
APPROVED BY:	JWS		DATE:	NOV. 2000	

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FIGURE 8-4
REFLOODED MINE GROUNDWATER MONITORING WELL LOCATION MAP

Scale: 0 400' 800' | Date: NOVEMBER, 2000
Prepared By: Foth & Van Dyke | By: JOW | 00C002

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Foth & Van Dyke			
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CHECKED BY:	JRB2	DATE:	NOV. 2000
APPROVED BY:	SVDI	DATE:	NOV. 2000
APPROVED BY:	JWS	DATE:	NOV. 2000

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FIGURE 10-1
IMPLEMENTATION PROCESS FLOW DIAGRAM

Scale: 0 100' 200' | Date: NOVEMBER, 2000
Prepared By: Foth & Van Dyke | By: JRB2 | 00C002

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Appendix A

Reflooded Mine Source Concentrations Report

Foth & Van Dyke
consultants - engineers - scientists

1UF001.01

REFLOODED MINE SOURCE CONCENTRATIONS CRANDON PROJECT, WISCONSIN

SRK Consulting
Engineers and Scientists

1UF001.01

REFLOODED MINE SOURCE CONCENTRATIONS
CRANDON PROJECT, WISCONSIN

Prepared for:

FOTH & VAN DYKE
2737 South Ridge Road
Green Bay, WI, 54307-9012

Prepared by:
STEFFEN ROBERTSON AND KIRSTEN (CANADA) INC.
Suite 800, 580 Hornby Street
Vancouver, B.C. V6C 3B6
Tel: (604) 681-4196 • Fax: (604) 687-5532
E-mail: vancouver@srk.com Web site: www.srk.com

NOVEMBER, 2000

REFLOODED MINE SOURCE CONCENTRATIONS
CRANDON PROJECT, WISCONSIN

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ATTACHMENT D Memorandum from A.J. Erickson, Jr. to Mr. K. Black, dated February 6, 1997
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REFLOODED MINE SOURCE CONCENTRATIONS CRANDON PROJECT, WISCONSIN

1. INTRODUCTION

This report describes investigations directed towards estimating the concentrations of solutes in the reflooded mine. Four potential sources of solutes were considered:

- Pyritic paste backfill;
- Hanging wall rock;
- Unmined ore; and
- Fuel combustion and blasting residues.

The initial plan for the investigations reported herein was developed in 1996, and communicated to the Wisconsin Department of Natural Resources (WDNR) and the U.S. Army Corps of Engineers (Letter from Mr. Don Moe to Mr. Christopher P. Carlson and Mr. David L. Ballman, dated July 24, 1996). Upon receipt of comments, a letter outlining revisions to the initial plan was prepared (Letter from Mr. Don Moe to Mr. Christopher P. Carlson, dated October 14, 1996). The initial investigation was then carried out and a report prepared (SRK, March 1997). WDNR comments on the initial report were received in a letter dated January 13, 1998. SRK prepared responses to the comments, and revisions to the March 1997 report. The responses and revisions were provided in a letter and update dated April 23, 1998.

The project design underwent a modification in the spring of 1998, when Nicolet Minerals Company (NMC) committed to removing pyrite from the tailings that will be stored on surface, and sending the pyrite concentrate back underground as cemented paste backfill. A program to characterize the depyritized tailings, pyrite concentrate, and "pyritic paste backfill" was initiated. Interim results of the waste characterization program were used to update the estimates of solute concentrations associated with the backfill. The report on this interim update was completed in November 1998. WDNR comments on the interim update were received in a letter dated July 1, 1999. Results of the waste characterization program, including the tests on pyritic paste backfill,

were reported as part of the TMA Groundwater Quality Performance Evaluation (Foth & Van Dyke (FVD), 1999/2000).

A draft version of the current report, incorporating responses to the July 1, 1999 comments, was prepared in May 2000, and provided to WDNR and its reviewers for comment. The draft was discussed in technical meetings held in June 2000. The current report includes revisions to address questions raised at those meetings, as well as revised responses to the July 1, 1999 letter (Attachment K). To facilitate future reviews, the current report also incorporates relevant sections from all earlier reports and correspondence. Sections 2 to 6 below present the derivation of solute concentration estimates for each of the above-mentioned sources prior to implementing source control measures. Section 7 presents analyses to test the sensitivity of the estimates to various input assumptions. Section 8 presents further sensitivity analyses to investigate the effects of source control measures under consideration by NMC. Section 9 provides a summary of key conclusions.

2. PYRITIC PASTE BACKFILL

2.1 Approach

NMC is proposing to use cemented pyritic concentrate ("pyritic paste backfill") to backfill mined-out stopes. In addition to enhancing the stability of the underground workings, the backfilling will reduce both the volume and the sulfide content of tailings that need to be stored on surface and will result in favorable conditions for the long term stability of sulfide minerals. In essence, the sulfide minerals will be returned to the underground and submerged environment in which they have remained stable for millions of years.

However, pyritic paste backfill that is returned underground in the early years of mining may remain above the water table for up to thirty years. During this period there is a potential that sulfide minerals may oxidize. After the groundwater table is allowed to recover, oxidation products stored in the backfill could be released, contributing to elevated solute concentrations.

It is clear from the waste characterization reported in Foth and Van Dyke (1999/2000) that the pyrite concentrate has the geochemical potential to generate oxidation products and acidity. However, the resulting geometry of the backfilled stopes will lead to the isolation of the backfill from the ventilation system that will be supplying air to the remainder of the underground workings. The limitations on oxygen access will limit the extent of oxidation and acid generation within the backfill.

In order to predict the solute concentrations that will arise in the pyritic paste backfill when the mine is flooded, the following steps were completed:

- The geochemical and mineralogical characteristics of the pyritic paste backfill were determined as part of the extensive waste characterization program reported in Foth and Van Dyke (1999/2000).

- The mining and backfilling methods proposed by NMC were reviewed to identify locations where the backfill will be exposed to oxygen.

- Oxygen transport calculations were completed to estimate the amount of pyritic paste backfill that (a) could become acidic, (b) will oxidize but remain neutral and (c) will not oxidize.

- Results of column tests on pyrite concentrate and pyritic paste backfill were used to derive estimates of solute concentrations in each of the (a) acidic, (b) oxidized neutral and (c) unoxidized pyritic paste backfill.

2.2 Characteristics of Pyritic Paste Backfill

The geochemical characteristics of the pyritic paste backfill are discussed in detail in Appendix A1, A3 and A4 of FVD 1999/2000. Characterization of the pyritic paste backfill included mineralogical examination, bulk chemical analyses, acid base accounting, process water analyses, multiple step batch tests, saturated column tests and humidity cell tests. Key static test results are presented below.

2.2.1 Mineralogy

Mineralogical characterizations of the pyrite concentrate and pyritic paste backfill are summarized in Table 2.1 (Table 3.13 Appendix A1, FVD 1999/2000). Carbonate mineralogy is summarized in Table 2, Appendix A4 of the GWQPE (FVD 1999/2000). Approximately 43% of carbonates in the pyrite concentrate are $(Ca+Mg)CO_3$.

2.2.2 Bulk Chemical Analysis

The bulk chemical analysis of the pyritic paste backfill amended with 5% Portland cement is compared to separate analyses of the pyrite concentrate and the Portland cement in Table 2.2 (Table 3.19 Appendix A1, FVD 1999/2000). The bulk chemical analysis of the pyritic paste backfill (i.e. with cement) was used in the calculations below.

TABLE 2.1
Optical Mineralogy Results of Pyrite Concentrate and Pyritic Paste Backfill

Mineral	Specific Gravity	Pyrite Concentrate[1,2]		Pyritic Paste Backfill[1,2]	
		Number of Grains	(estimated weight %)	Number of Grains[3]	(estimated weight %)
Opaques:					
Pyrite	5	409	86	441	80
Other Sulfides					
Sphalerite	4	17	2.9	27	3.9
Arsenopyrite	6.1	5	1.3	5	1.1
Galena	7.5	6	1.9	4	1.5
Chalcopyrite	4.2	8	1.4	10	1.8
Pyrrhotite	4.6	0	<0.2	0	<0.2
Iron Oxyhyroxides	4.4	0	<0.2	0	<0.2
Hematite	5.1	0	<0.2	0	<0.2
Non Opaques					
Quartz	2.65	48	5.4	59	5.7
Muscovite/Illite	2.8	2	0.2	3	0.3
Chlorite	2.7	1	<0.2	1	0.2
Carbonates	2.8	4	<0.2	0	<0.2
Cement (calcium silicates)	0.4	n/a	n/a	450	6.5
TOTAL		500	99.4	1000	100

Notes: From GWQPE (Table 3.13, Appendix A1, FVD 1999/2000)

Prepared by: KSS
Checked by: CBL

[1] Estimates are based on point counts on 500 randomly selected mineral grains. The results are corrected for differences in specific gravity, but not particle size.

[2] Units are in percent abundance by weight

[3] Total of 7 day and 28 day samples.

"n/a"= not applicable (sample was not cemented)

TABLE 2.2
Bulk Chemistry of the Pyrite Concentrate and Pyritic Paste Backfill

Parameter	Units	Pyrite Concentrate Average	Pyritic Paste Backfill	Cement (LaFarge Type 10)*
Antimony	mg/kg	40	65.5	10
Arsenic	mg/kg	3,600	2,500	300
Barium	mg/kg	<20	20	360
Beryllium	mg/kg	<10	<20	<20
Cadmium	mg/kg	85	27.5	<5.0
Calcium	mg/kg	2,900	19,350	549,000
Chromium	mg/kg	92	185	55
Cobalt	mg/kg	83	100	12
Chloride	mg/kg	nd	nd	nd
Copper	mg/kg	995	1,000	46
Fluoride	mg/kg	nd	nd	nd
Iron (total)	mg/kg	376,000	-	14,400
Lead	mg/kg	4,200	3,750	<500
Magnesium	mg/kg	5,400	6,200	12,300
Manganese	mg/kg	340	265	320
Mercury	mg/kg	4.75	3.75	<0.3
Nickel	mg/kg	55.5	62.5	58
Selenium	mg/kg	105	95	<3
Silicon	mg/kg	52,850	56,050	-
Silver	mg/kg	29.5	25.85	<0.5
Sodium	mg/kg	235	50	2100
Strontium	mg/kg	7.85	26.5	300
Sulfur (T)	mg/kg	432,500	399,000	12,100
Zinc	mg/kg	12,100	10,900	500

Notes: From GWQPE, (Table 3.19, Appendix A1, FVD 1999/2000)
* The pyritic paste backfill contains approximately 5% cement (by weight).
nd = no data

Prepared by: KSS
Checked by: MDR

2.2.3 Acid Base Accounting

Table 2.3 presents the neutralization potential of the pyritic paste backfill after different curing times (information derived from Table 3.17, Appendix A1, FVD 1999/2000). The table reports modified Sobek-NP values for all samples. The NMC-1 samples, tested after 7 days, had anomalously high modified Sobek-NP values, indicating that the cement addition had not completely reacted. Two of the remaining samples, GPT #4 and NMC-2, have much lower modified Sobek-NP than the other samples.

The table also provides the measured CO_3-NP for each sample. As discussed in Appendix A4 of the GWQPE (FVD 1999/2000), the microprobe analyses indicate that the Ca+Mg fraction of the carbonate-NP in the pyrite concentrate is approximately 43%. However, CO_3-NP is not representative of the significant NP contributed by the oxide and hydroxide minerals derived from the addition of cement to the pyritic paste backfill. If all samples with curing times of 28 days or more are considered, the average modified Sobek-NP is 45.4 kg $CaCO_3$ eq./tonne and the average CO3-NP is 23.5 kg $CaCO_3$ eq./tonne. If the anomalous samples, GPT#4 and NMC-2, and the seven day samples NMC-1 are excluded, the remaining samples have an average modified Sobek-NP of 51.6 kg $CaCO_3$ eq./tonne, and an average CO_3-NP of 24.1 kg $CaCO_3$ eq./tonne. The difference between the two indicates an average non-carbonate-NP of 27.5 kg $CaCO_3$ eq./tonne.

Table 2.4 summarizes the estimates of NP availability derived from results of the multiple step batch tests on two pyritic paste backfill samples. These tests utilized a slightly acidic de-aerated leachate and an argon environment (Eh<0 mV) to assess NP availability. The results indicate that the non-carbonate-NP is completely available and the carbonate-NP is strongly available.

TABLE 2.3
NP Measurements of Pyritic Paste Backfill

Sample ID	Cement (%)	Curing time (days)	CO_3 (%)	Modified Sobek-NP (kg $CaCO_3$ eq./tonne)	CO_3-NP[1] (kg $CaCO_3$ eq./tonne)
GPT #1	5	28	1.29	49.8	21.5
GPT #1 Duplicate	5	28	1.35	52.1	22.5
GPT #2	5	28	1.47	52.4	24.5
GPT #2 Duplicate	5	28	1.42	52.6	23.7
GPT #1B	5	28	1.64	51.4	27.3
GPT #1B Duplicate	5	28	1.65	51.3	27.5
GPT #3	5	28	1.49	50.6	24.8
GPT #3 Duplicate	5	28	1.49	46.5	24.8
GPT #4	5	28	1.37	34.5	22.8
GPT #4 Duplicate	5	28	1.36	35.0	22.7
GPT #5	5	90	1.43	53.0	23.8
GPT #5 Duplicate	5	90	1.25	55.1	20.8
NMC-2	5	28	1.3	25.5	21.7
NMC-2 Duplicate	5	28	1.25	26.1	20.8
NMC-1	5	7	1.15	57.3	19.2
NMC-1 Duplicate	5	7	1.35	54.6	20.8
Average*				45.4	23.5

Notes: From GWQPE, (Table 3.17, Appendix A1, FVD 1999/2000) Prepared by: DEH
*NMC-1 samples were not included in determining average NP Checked by: JTC

[1] As discussed in Appendix A4, only 36% of the carbonate in the pyritic paste backfill is present as (Ca+Mg) CO_3-NP. Therefore, not all of the CO_3-NP is available for neutralization.

TABLE 2.4
NP Availability in Pyritic Paste Backfill

Parameter	Sample CPT-1	Sample CPT-2
Modified Sobek-NP from Solids Analyses		
Initial modified Sobek-NP	40.9 kg $CaCO_3$ eq/tonne	40.9 kg $CaCO_3$ eq/tonne
Residue modified Sobek-NP	2.5 kg $CaCO_3$ eq/tonne	3.3 kg $CaCO_3$ eq/tonne
Modified Sobek-NP Availability	94%	92%
CO_3-NP from Solids Analyses		
Initial CO3-NP	21.1 kg $CaCO_3$ eq/tonne	21.1 kg $CaCO_3$ eq/tonne
Residue CO3-NP	2.5 kg $CaCO_3$ eq/tonne	3.3 kg $CaCO_3$ eq/tonne
CO_3-NP Availability	88%	84%
Modified Sobek-NP Availability from acidity consumption above pH 6	95%	92%
Modified Sobek-NP Availability from Ca+Mg release above pH 6	88%	79%

Note: From GWQPE, (Tables 3.24 and 3.25, Appendix A1, FVD 1999/2000) Prepared by: DEH
Checked by: DDS/KSS

The fact that the modified Sobek-NP values reported in Table 2.3 vary by 50%, while available NP values are uniformly high, suggests that cement addition will play the most significant role in determining how much NP is available to counteract local oxidation. The amount of NP contributed by cement can be deduced from the average measured NP in pyrite concentrate and in pyritic paste backfill. As shown in Table 2.5, the modified Sobek-NP content of the Portland cement was calculated to be 600 kg $CaCO_3$ eq./tonne.

TABLE 2.5
NP Contributions to Pyritic Paste Backfill

	Modified Sobek-NP	CO_3-NP
Pyritic Paste Backfill NP (avg)* in 1 tonne of backfill with 5% cement	45.4 kg	23.5 kg
NP Contribution from Pyrite Concentrate (avg)** in 1 tonne of backfill (95% PyCon)	15.4 kg	15.1 kg
NP Contribution from Cement in 1 tonne of backfill†	30.0 kg	8.4 kg
NP of Cement‡	600 kg $CaCO_3$ eq./tonne	168 kg $CaCO_3$ eq./tonne

* See Table 2.3
** From GWQPE, (Table 3.16 Appendix A1, FVD 1999/2000)
NP contribution = NP(Pyrite Concentrate) * 0.95 = 16.15*0.95 = 15.4
† Cement NP contribution = Backfill NP – Pyrite Concentrate NP = 45.4 – 0.95x16.15
‡ Cement NP = NP Contribution from cement/% cement in backfill = 30/0.05

Prepared by: JTC
Checked by: DDS

Golder Paste Technology Ltd. (1998) demonstrated that 5% cement addition will result in a pyritic paste backfill with sufficient structural strength. However, this addition rate was not optimized and actual cement addition rates may vary slightly from 5%. The modified Sobek-NP of pyritic paste backfill with different cement contents can be estimated by adding the NP contribution of cement to the NP of pyrite concentrate. Table 2.6 summarizes the estimated modified Sobek-NP of pyritic paste backfill with varying amounts of cement. Section 7 below presents a sensitivity analysis to examine the effects of cement content variability on predicted solute concentrations in the backfill. As a conservative assumption, the average modified Sobek-NP of the two anomalously low samples (i.e. 30.3 kg CaCO3 eq./tonne, average of GPT#4 and NMC-2) was used for the estimate of available NP in the water quality calculations presented below.

TABLE 2.6
Estimated Modified Sobek-NP of Pyritic Paste Backfill with Varying Cement Content

Cement Content	Modified Sobek-NP* Provided by Cement	Modified Sobek-NP* Provided by Pyrite Concentrate	Estimated Modified Sobek-NP* of Pyritic Paste Backfill†
1%	6.0	16.0	22
3%	18	15.7	34
5%	30	15.4	45
7%	42	15.1	57

Note: * units are kg $CaCO_3$ eq/tonne pyritic paste backfill
† Estimated Backfill NP = Fraction Cement x 600 + (1-Fraction Cement) x 16.2

Prepared by: JTC
Checked by: DDS

2.3 Mining and Backfilling Methods

Oxygen transport into the pyritic paste backfill will be dependent on the geometry of the backfilled workings, which will in turn depend on the mining method. Two mining methods will be used: blasthole open stoping methods below the 300 foot elevation and cut and fill methods above the 300 foot elevation.

The mining and backfilling methods were described in the Mine Permit Application (FVD 1995/1998a). The following paragraphs highlight aspects of the two mining methods that will influence the transport of oxygen and hence the potential for sulfide oxidation.

2.3.1 Blasthole Open Stopes

Figure 2.1 show the sequence of mining and backfilling in the blasthole open stope method. The nominal dimensions of the blasthole stopes will be 75 feet along strike, 200 feet in height, and 100 feet in ore body width. Access to each stoping block will be accomplished by driving a 13' x 16' cross-cut from the development drift to the ore block. A typical blasthole stope will contain about 170,000 tons of ore. At the annual production rate of 2,000,000 tons of ore, about twelve stopes will be mined out each year.

Each mined out stope will be backfilled with pyritic paste backfill. In preparation for backfilling, a waste rock plug will be placed in the cross cut. The crosscut will then be sealed with a ventilation bulkhead. The pyritic paste backfill will be pumped into the top of the stope via a piping system.

2.3.2 Cut and Fill Stopes

Figure 2.2 shows the sequence of mining and backfilling in the cut and fill method. Each stope will be mined along strike in four cuts, each about 11 - 13 feet high. The width of the cut will be limited by the width of the ore and the location of rib pillars. Nominal dimensions of each cut and fill stope will be 120 feet along strike, 45 feet in height, and 100 feet in width. A typical cut and fill stope will contain about 60,000 tons of ore.

Once the ore is removed, each cut will then be backfilled with pyritic paste backfill. The top surface of the backfill then becomes the floor for the next cut. Preparation for backfilling of the final cut is similar to that described above. The cross cut entry will be plugged with waste rock. A ventilation bulkhead will be installed and the pyritic paste backfill deposited into the mined out volume.

2.3.3 Potential Oxidation Sites

Both mining methods will limit the exposure of sulfide minerals in the pyritic paste backfill to oxygen. Once the backfill is deposited into a mined out stope, there will be no exposure to oxygen along any of the surfaces that abut against wall rock or previously deposited backfill. Hence the great majority of the pyritic paste backfill will remain unoxidized.

The primary difference between the two mining methods is that in the open stoping method, the entire stope is mined out before the stope is backfilled. In the cut and fill method, the stope is mined and backfilled in horizontal lifts. The top surface of each backfilled blasthole stope will be exposed to air during mining the next stope above. In the cut and fill stopes, the top surface of each lift will be exposed during mining of the next lift. In all cases the exposure of the pyritic paste backfill surface will last for only a period of weeks. Hence the pyritic paste surfaces are exposed to oxidation but remain neutral.

In both mining methods pyritic paste backfill in the access cross-cuts will be exposed for a longer period. Oxygen access will be impeded by the waste rock plug and the ventilation bulkhead. However, the long period of exposure means there is a potential for some of the pyritic paste backfill in the access crosscuts to become acidic.

2.4 Oxygen Transport

Pyritic paste backfill will be exposed to oxygen in the access cross-cuts and at the top surfaces of blasthole stopes or cut and fill lifts. To estimate the extent of oxidation in each area, oxygen transport calculations were completed. A detailed description of the calculations is provided in Attachment A.

2.4.1 O_2 Transport Through Cross-Cuts

As shown in Figures 2.1 and 2.2, the plugging of the access cross-cuts will result in "pipes" of waste rock abutting the pyritic paste backfill. Oxygen will diffuse from the bulkhead, through the waste rock and into the pyritic paste backfill. On average, the pyritic paste backfill in the access cross-cuts will be exposed to oxidation for approximately 14 years. The calculation is detailed in Attachment B.

Two conceptual models of the oxygen transport process were developed to derive a best engineering judgement (BEJ) and upper bound (UB) estimates (see Figure 2.3).

The BEJ model assumes that diffusion into the pyritic paste backfill will be the rate limiting step. The oxygen flux into the surface of the pyritic paste backfill was calculated from data obtained from recently completed humidity cell tests. Table 2.7 shows the specific oxidation rates (i.e. oxygen uptake per unit surface area) derived from the humidity cell tests on small cubes of pyritic paste backfill. The oxidation rates derived by all calculation methods in Table 2.7 are in agreement and are tightly clustered about the average value. In the humidity cell tests, there was no limitation on the oxygen transport to the backfill (air was blown through the cells). Therefore, the observed low oxidation rates indicate that diffusion into the backfill itself is the rate-limiting step. The average value from the humidity cell tests, 1.6×10^{-8} mol O_2 $m^{-2}s^{-1}$ was therefore adopted as the BEJ estimate. Figure 2.3b shows the equivalent process as it would occur in the backfilled stopes. For a crosscut with nominal dimensions of 13 feet by 16 feet, or 19.3 m^2, the BEJ estimate of oxygen transport into the pyritic paste backfill is 3.1×10^{-7} mol O_2 s^{-1}.

TABLE 2.7
Summary of Specific Oxidation Rates in Humidity Cell Tests of Pyritic Paste Backfill

Calculated from sulfate release rates		
HC CPT-ST1	0 – 20 weeks	$1.4x10^{-8}$ mol O_2 m^{-2} s^{-1}
HC CPT-ST2	0 – 20 weeks	$1.3x10^{-8}$ mol O_2 m^{-2} s^{-1}
Calculated from initial and final HA extractable sulfate		
HC CPT-ST1	0-20 weeks	$1.4x10^{-8}$ mol O_2 m^{-2} s^{-1}
HC CPT-ST2	0-20 weeks	$1.3x10^{-8}$ mol O_2 m^{-2} s^{-1}
Calculated from initial and final residue sulfate		
HC CPT-ST1	0-20 weeks	$2.3x10^{-8}$ mol O_2 m^{-2} s^{-1}
HC CPT-ST2	0-20 weeks	$2.1x10^{-8}$ mol O_2 m^{-2} s^{-1}
All available data from intermediate test		
HC CPT-IT1	20-62 weeks	$1.3x10^{-8}$ mol O_2 m^{-2} s^{-1}
Average of all above values		**$1.6x10^{-8}$ mol O_2 m^{-2} s^{-1}**

* From GWQPE, (Table 4.3, Appendix A3, FVD 1999/2000) Prepared by: DEH
Checked by: JTC

The UB model assumed that diffusion into the pyritic paste backfill will be rapid, leaving only diffusion through the bulkhead and the waste rock plug as the rate-determining step. This conceptual model is illustrated in Figure 2.3c. The assumption of rapid diffusion into the pyritic paste backfill means that oxygen concentrations at the waste rock-backfill interface are very low. The flux of oxygen to the interface was estimated by applying Fick's Law across the waste rock. To obtain a conservatively high prediction of oxygen transport, it was further assumed that oxygen would be at atmospheric concentrations at the inside edge of the ventilation bulkhead. In other words the ventilation bulkhead was assumed to restrict air circulation, but not oxygen diffusion. Under these assumptions the rate of oxygen transport to the pyritic backfill is given by:

$$\text{Flux} = D\, C_o\, L^{-1}$$

where D is the diffusion coefficient (m^2s^{-1}), C_o is the oxygen concentration (mol O_2 m^{-3}) at the inside edge of the ventilation bulkhead, L is the length (m) waste rock plug behind the bulkhead, and the flux has units of mol O_2 $m^{-2}s^{-1}$. As discussed, the oxygen concentration was conservatively assumed to be in equilibrium with the atmosphere (i.e. C_o=8.9 mol O_2 m^{-3}) just behind the bulkhead and zero at the interface with the pyritic paste backfill. The diffusion coefficient through the waste rock was

estimated to be $6x10^{-6}$ m^2s^{-1}, equivalent to the upper end of the range of oxygen diffusion coefficients measured in waste rock and reported by Ritchie (1994). The length of the waste rock plug was estimated to be 40 feet, or 12.2 m. The resulting UB estimate of the oxygen flux was $4.4x10^{-6}$ mol O_2 $m^{-2}s^{-1}$. For a crosscut with nominal dimensions of 13 feet by 16 feet, or 19.3 m^2, the estimate rate of oxygen transport to the pyritic paste backfill is $8.5x10^{-5}$ mol O_2 s^{-1}.

The very substantial difference between the UB estimate of oxygen transport through a cross-cut to the pyritic paste backfill ($8.5x10^{-5}$ mol O_2 s^{-1}), and the BEJ estimate ($3.1x10^{-7}$ mol O_2 s^{-1}) is worthy of some discussion. The reason for the magnitude of the difference is that the two estimates assume that very different transport processes are dominant.

In the BEJ case, the pyritic paste backfill is assumed to remain intact, and therefore the slow rate of diffusion into the backfill minimizes the overall oxygen uptake. There is good evidence to suggest that this is a reasonable assumption. The pH measurements from the humidity cell tests show that, after an initial decline from strongly alkaline to neutral pH, geochemical conditions stabilize and remain stable for approximately sixty weeks. At neutral pH, one would expect the cemented backfill to remain completely intact. In fact the cubes in the humidity cell tests remained competent and oxidation was limited to a narrow rim around the outside of each surface. It was observed in the laboratory that 1 cm^3 pyritic paste backfill cubes remained intact even after 100 weeks of leaching.

Nonetheless, the UB estimates presented in the revised source term conservatively assume that the only limitation to oxidation rates will be the transport of oxygen to the backfill face. This is equivalent to assuming that the backfill structure is completely degraded, allowing oxygen to diffuse rapidly through the backfill itself. The ventilation bulkhead and waste rock plug then become the only limitations to oxygen transport, and a much higher estimate of the oxygen flux rate is obtained.

2.4.2 O_2 Transport Between Blasthole Stopes and Cut & Fill Layers

During mining immediately above a backfilled stope, the top surface of the backfilled stope will be exposed to air and the backfill will oxidize. In the cut and fill stopes, the top surface of each backfilled layer will be exposed to air while the layer above is being mined. The humidity cell tests on small cubes of pyritic paste backfill are

representative of this situation. In section 2.5 below, results of those tests are used to estimate solute concentrations in the "oxidized neutral" backfill.

Pyritic paste backfill cubes were retrieved from the short term humidity cell tests after 20 weeks of exposure. Mineralogical examination showed that the oxyhydroxide staining associated with oxidation extended less than 1 mm below intact surfaces (GWQPE, Appendix A3, Section 3.3.2.1, FVD 1999/2000). Secondary gypsum and calcite were also observed to accumulate and partially block the pores near the cube surface. After approximately 100 weeks of operation, the intermediate term humidity cell tests were stopped on August 17, 2000. The test was dismantled in September 2000. The laboratory report is in preparation. However, visual inspection of the cubes indicate that oxyhydroxide staining extended approximately 1 mm below the cube surfaces.

In calculating the solute concentrations described in Section 2.5, it was assumed that the "oxidized neutral" solute concentrations would be attained in all of the pyritic paste backfill within 50 mm (approximately 2 inches) of the exposed surface. That depth is a conservative value based on consideration of oxygen transport, together with evidence from the humidity cell residues.

In cut and fill stopes, where the exposure time between backfilling of each lift will typically be six weeks, it is reasonable to assume that the depth of oxidation will be similar to that observed in the pyritic paste backfill cubes. In blasthole stopes, where it may take up to one year to mine and backfill the above stope, exposure time will be longer. However, the longer period of exposure would to some extent be counteracted by the secondary precipitate filling of the pores, which would tend to decrease oxygen diffusion rates. Even neglecting the latter effect, and considering a period of sixty weeks exposure time, the oxidation would still penetrate less than three millimeters into the exposed pyritic paste backfill. Hence the assumption that oxidation will penetrate to a depth of 50 mm is conservative.

2.5 Solute Concentrations in Reflooded Pyritic Paste Backfill

Due to the geometry of the stopes, the mine plan and the physics of oxygen transport into the stopes, the degree of oxidation in the pyritic paste backfill will differ between the different areas within each stope. For predicting potential solute release to the reflooding mine water, the pyritic paste backfill can be divided into three "types" of

backfill – "unoxidized", "oxidized neutral" and "acidic". Calculations to estimate the volume of each "type" of backfill and its associated solute release are described below. A detailed description of the solute concentration calculations is provided in Attachment A.

2.5.1 Acidic Backfill

Each stope will be accessed by a cross-cut which is mined from the hanging wall drift. Once the stope has been mined, the cross-cut drift will be filled with waste rock and a ventilation bulkhead constructed at the entrance. Backfill will then be poured into the mined out stope behind the waste rock. Oxygen will diffuse through the ventilation bulkhead and the waste rock to the pyritic paste backfill. The oxygen diffusion will continue until the mine is reflooded. The average duration of this oxidation period is estimated to be 14 years (see Attachment B), which may be sufficient to produce acidic conditions in a small volume of the backfill.

To estimate the volume of acidic backfill that could form within each cross-cut, it was assumed that oxygen entering the backfill would be distributed such that the resulting acidity released from sulfide oxidation completely consumes the available NP. The calculations for the volume of acidic backfill are presented in Section 1c of Attachment A. Using BEJ oxygen transport estimates, the total volume of acidic backfill is estimated to be 43 m^3 (56 yd^3). The UB oxygen transport estimates lead to an estimated 11,700 m^3 (15,400 yd^3) of acidic backfill. (The wide difference between these two estimates is due to different assumptions regarding the limitations to oxygen transport.)

Solute concentrations in the reflooded acidic backfill were estimated by a series of calculations similar to those used in Section 2.5.2.1 of FVD (1999/2000) to estimate solute concentration in the oxidation zone of the Crandon TMA. The total sulfide oxidation in the acidified backfill was estimated from stoichiometry, i.e. one mole of sulfide was assumed to be oxidized for each mole of $CaCO_3$ eq. consumed. Table 2.8 presents the relative solute release rates estimated from the humidity cell tests on pyritic paste backfill and pyrite concentrate (uncemented). The relative solute release rates in Table 2.8 were then used to estimate how much of each solute would be released by the oxidation and related reactions. The total mass of each solute release was initially assumed to dissolve into the porewater that will fill the acidified backfill when the mine refloods.

The relative release calculations led to some solutes being estimated to be present at concentrations exceeding solubility limitations. MINTEQA2 modeling was then carried out to correct for solubility limitations. The results are summarized in Table 2.9. That table provides estimates calculated with relative solute release rates from both the pyritic paste backfill and the pyrite concentrate. Details of the calculations are provided in Attachment A and the results of the MINTEQA2 modeling are presented in Attachment C.

TABLE 2.8
Relative Solute Release Rates in Pyrite Concentrate and Pyritic Paste Backfill

	Pyritic Paste Backfill (mol Me/mol S_2)			**Pyrite Concentrate** (mol Me/mol S_2)		
Parameter	**HC-CPT-ST-1**	**HC-CPT-ST-2**	**Average**	**HC-UPT-IT-1**	**HC-UPT-IT-2**	**Average**
Al	3.29E-03	5.71E-02	3.0E-02	7.90E-02	8.02E-02	7.96E-02
Sb	6.79E-04	9.57E-04	8.2E-04	1.75E-05	2.24E-05	2.00E-05
As	1.36E-02	1.73E-02	1.5E-02	1.02E-02	9.44E-03	9.83E-03
Ba	1.35E-04	1.54E-04	1.4E-04	5.07E-06	4.85E-06	4.96E-06
Be	3.89E-04	4.42E-04	4.2E-04	9.79E-06	1.09E-05	1.03E-05
B	-	-	-	3.16E-06	2.86E-06	3.01E-06
Cd	4.21E-04	8.73E-04	6.5E-04	2.62E-04	2.64E-04	2.63E-04
Ca	2.58E+00	2.56E+00	2.6E+00	8.27E-02	6.44E-02	7.36E-02
Cl	2.8E+00	2.6E+00	2.7E+00	1.09E-02	9.73E-03	1.03E-02
Cr	1.86E-05	1.10E-04	6.4E-05	4.52E-04	4.47E-04	4.49E-04
Co	5.98E-05	1.17E-05	3.6E-05	3.60E-04	3.18E-04	3.39E-04
Cu	4.47E-05	1.62E-02	8.1E-03	1.42E-02	1.29E-02	1.36E-02
F	1.36E-02	8.85E-03	1.1E-02	3.14E-03	3.24E-03	3.19E-03
Fe	1.16E+00	3.10E+00	2.1E+00	1.16E+00	1.26E+00	1.21E+00
Pb	2.12E-04	1.17E-03	6.9E-04	4.16E-03	1.52E-03	2.84E-03
Mg	5.02E-02	3.30E-02	4.2E-02	1.40E-01	1.26E-01	1.33E-01
Mn	3.77E-03	9.80E-03	6.8E-03	3.99E-03	3.97E-03	3.98E-03
Hg	1.05E-07	1.13E-08	5.8E-08	9.25E-07	2.13E-06	1.53E-06
Ni	1.02E-03	1.61E-02	8.6E-03	7.01E-05	5.87E-05	6.44E-05
K	1.59E-01	2.21E-01	1.9E-01	6.55E-04	7.48E-04	7.02E-04
Se	5.22E-04	6.70E-04	6.0E-04	1.19E-04	8.44E-05	1.02E-04
Ag	3.16E-05	5.82E-05	4.5E-05	3.30E-06	4.55E-06	3.92E-06
Na	9.04E-02	1.25E-01	1.1E-01	1.29E-01	1.15E-01	1.22E-01
SO_4	2.00E+00	2.00E+00	2.0E+00	2.00E+00	2.00E+00	2.00E+00
Zn	1.05E-01	3.00E-01	2.0E-01	1.61E-01	1.62E-01	1.62E-01

Note: calculations were similar to those presented in the GWQPE (Appendix A3, Section 3.1.2.4 FVD 1999/2000). Data for calculations are from Appendix A3, Attachment C, Tables C2-4 and C3-4 FVD 1999/2000)

Prepared by: JTC

Checked by: DDS

HC-CPT-ST tests run for 20 weeks (intermediate term tests are in the process of being terminated)

HC-UPT-IT tests run for 58 weeks.

TABLE 2.9
Estimates of Solute Concentrations in Acidic Backfill

Parameter	Estimated Solute Concentrations (mg/L)				
	Initial		MINTEQA2 (files: acid_bf.in & -.out)	Final	
	Paste Backfill	Pyrite Concentrate		BEJ	UB
Al	718	1,896		1,896	1,896
Sb	88	2.1		2.1	2.1
As	1020	650		650	650
Ba	18	0.60	0.00071	**0.00071**	0.60
Be	3.3	0.082		0.082	0.082
B	-	0.029		0.029	0.029
Cd	64	26		26	26
Ca	90,916	2,602	471	**471**	2,602
Cl	83,236	323*		251**	251**
Cr	2.9	21		21	21
Co	1.9	18		18	18
Cu	456	760		760	760
F	188	53		53	53
Fe	105,187	59,756		59,756	59,756
Pb	126	520	3.1	**3.1**	520
Mg	892	2,852		2,852	2,852
Mn	329	193		193	193
Hg	0.01	0.27		0.27	0.27
Ni	443	3.3		3.3	3.3
K	6,570	24		24	24
Se	42	7.1		7.1	7.1
Ag	4.3	0.37	0.021	**0.021**	0.37
Na	2,187	2,481*		291**	291**
SO_4	169,555	169,555	164,173	**164,173**	169,555
Zn	11,679	9,322		9,322	9,322

Note: Because of the similarity in the initial solute concentrations, the MINTEQA2 run was not repeated for the paste backfill calculation

Prepared by: DDS
Checked by: CK

Bold values are solubility controlled values.

* = Initial concentration exceeds amount in solids.

** = Calculated from solids analyses GWQPE.

The results in Table 2.9 show that concentration estimates based on relative solute release rates from the pyritic paste backfill tests are similar to those derived from the pyrite concentrate tests for most elements (within a factor of 3 for 13 of 25 elements). Concentration estimates for Sb, Be, Cl, Ni, K and Se are significantly higher if the pyritic paste backfill data are used. Concentration estimates for Cr, Co and Hg are significantly higher if the pyrite concentrate data are used. The concentration estimates derived for the pyrite concentrate data are considered to better represent conditions expected in acidic backfill for two reasons. First, the pyrite concentrate became acidic within the first few cycles of the tests. In contrast, the pyritic paste

backfill tests remained neutral. Second, the paste backfill tests were terminated after 20 weeks of testing whereas the pyrite concentrate tests were continued for 58 weeks. Consequently, the rate of oxidation in the paste backfill tests was significantly lower than in the pyrite concentrate tests and the sulfate generated was considerably less. When sulfate release is low, metal release associated with sulfide oxidation is more variable. Therefore, the relative release rate calculation for the pyritic paste backfill would be more prone to error. The errors are exacerbated by the fact that metals such as Cd, Ni, Sb and Fe were below detection limit in leachates generated from the paste backfill tests. (The detection limit was used in the solute release calculation for analyses that were below detection.) The susceptibility to error is demonstrated by the variability between the two sets of pyritic paste backfill reported in Table 2.8.

2.5.2 Oxidized Neutral Backfill

Humidity cell tests of pyritic paste tailings remained neutral for over 60 weeks (FVD 1999/2000) and show that the pyritic paste backfill will not become acidic during the brief periods of oxidation expected when adjacent stopes or lifts are mined. The data from those tests also provide a good basis for estimating the solute release that could occur when the "oxidized neutral" surfaces of the backfill are flooded.

Table 2.10 presents the cumulative solute releases observed after five weeks and twenty weeks of the humidity cell tests. Solute release was converted to solute concentration using the following formula:

$$\text{Concentration} = \frac{\text{solute release (mg/kg) x backfill bulk density (kg/m}^3\text{)}}{\text{Porosity (m}^3_w/\text{m}^3_t\text{) x 1000 (L/m}^3\text{)}}$$

Solute concentrations are given in Table 2.11. The five-week values were used to derive BEJ estimates of solute concentrations in the oxidized neutral backfill. The twenty-week values were used to derive the UB estimates. Details of the calculations are provided in Attachment A.

The volume of "oxidized neutral" backfill generated during the mine life was estimated by multiplying the area of a surface by the depth of oxygen penetration and by the number of surfaces. The depth of oxygen penetration was calculated from the one-dimensional diffusion-reaction equation (see Section 2.4.2). Using an oxygen

penetration depth of 0.05 m, a total of volume of 33,000 m^3 (43,200 yd^3) of oxidized neutral backfill will be present underground at the end of mining.

TABLE 2.10
Cumulative Solute Release from Oxidized Neutral Backfill

Parameter		HC-CPT-ST1		HC-CPT-ST2		Average	
		To Cycle 5	To Cycle 20	To Cycle 5	To Cycle 20	To Cycle 5	To Cycle 20
Al	(mg/kg)	0.81	2.6	0.47	1.8	0.64	2.2
Sb	(mg/kg)	0.0067	0.074	0.0046	0.039	0.0057	0.056
As	(mg/kg)	0.05	0.21	0.031	0.17	0.040	0.19
Ba	(mg/kg)	0.23	0.56	0.17	0.49	0.20	0.52
Be	(mg/kg)	0.0012	0.0049	0.0014	0.0047	0.0013	0.0048
B	(mg/kg)	nd	nd	nd	nd	nd	nd
Cd	(mg/kg)	0.0018	0.0027	0.00023	0.001	0.001	0.0019
Ca	(mg/kg)	760	3100	640	2400	700	2800
Cl	(mg/kg)	713	2918	569	2099	641	2508
Cr	(mg/kg)	0.016	0.029	0.017	0.03	0.016	0.029
Co	(mg/kg)	0.0065	0.017	0.0073	0.016	0.0069	0.016
Cu	(mg/kg)	0.038	0.085	0.037	0.11	0.038	0.099
F	(mg/kg)	0.37	1.9	0.32	0.89	0.34	1.4
Fe (T)	(mg/kg)	0.087	0.39	0.014	0.14	0.05	0.26
Pb	(mg/kg)	0.057	0.063	0.034	0.041	0.046	0.052
Mg	(mg/kg)	1.7	36	0.83	19	1.3	27
Mn	(mg/kg)	0.00074	0.022	0.0011	0.0074	0.00091	0.015
Hg	(ug/kg)	0.11	0.55	0.095	0.25	0.10	0.40
Ni	(mg/kg)	0.0074	0.026	0.0061	0.022	0.0067	0.024
K	(mg/kg)	150	186	150	200	150	193
Se	(mg/kg)	0.087	0.30	0.088	0.25	0.087	0.27
Ag	(mg/kg)	0.027	0.11	0.036	0.14	0.031	0.12
Na	(mg/kg)	49	62	50	66	49	64
SO_4	(mg/kg)	1500	5736	1238	4430	1369	5736*
Zn	(mg/kg)	0.10	0.22	0.084	0.15	0.093	0.18

Note: *The maximum SO_4 value was used to provide an improved ion balance in MINTEQA2 Prepared by: JTC
Source: GWQPE, Appendix A3, Tables A-9.3 & A-10.3 (FVD 1999/2000) Checked by: DDS
nd = no data

TABLE 2.11
Estimated Solute Concentrations in Oxidized Neutral Backfill

Parameter	Initial Concentration		MINTEQA2 (files: Neutbej.in & -.out)		Final Concentration	
	BEJ	UB	BEJ	UB*	**BEJ**	**UB**
	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Al	3.7	13	0.0015		**0.0015**	13
Sb	0.033	0.33			0.033	0.33
As	0.24	1.1			0.24	1.1
Ba	1.2	3.0	0.0031		**0.0031**	3.0
Be	0.0077	0.028			0.0077	0.028
B	nd	nd			nd	nd
Cd	0.0059	0.011			0.0059	0.011
Ca	4,092	14,136	587	716	**587**	**716***
Cl	251**	251**			251	251
Cr	0.095	0.17			0.095	0.17
Co	0.040	0.095			0.040	0.095
Cu	0.22	0.58			0.22	0.58
F	2.0	8.1	1.0		**1.0**	8.1
Fe	0.29	1.5			0.29	1.5
Pb	0.27	0.30			0.27	0.30
Mg	7.4	160			7.4	160
Mn	0.0053	0.086			0.0053	0.086
Hg	0.00060	0.0023			0.00060	0.0023
Ni	0.039	0.14			0.039	0.14
K	879	1,125			879	1,125
Se	0.51	1.6			0.51	1.6
Ag	0.18	0.71	0.021		**0.021**	0.71
Na	287	291**			287	291
SO_4	7,982	33,443	2,147	2,659	**2,147**	**2,659***
SO_4 from S_2O_3	686	1,371			686	1,371
Zn	0.54	1.1			0.54	1.1

Bold values are solubility controlled parameters
nd = no data
Prepared by: DDS
Checked by: CK

* = Only gypsum precipitation was allowed in UB runs.

** = Initial concentration exceeded the amount in solids. The value presented in the table above is calculated from solids analyses presented in the GWQPE.

2.5.3 Unoxidized Backfill

2.5.3.1 Saturated Column Test and MINTEQA2 Results

The majority of the pyritic paste backfill will not be exposed to oxidation. Solute concentrations measured in a saturated column test of pyritic paste backfill (Appendix

A1, FVD, 1999/2000) were used to estimate the pore water quality associated with unoxidized pyritic paste backfill. Generally, the pore water equilibrated at a high pH and solute concentrations were found to be low.

The geochemical equilibrium model MINTEQA2 was used to determine if any of the concentrations observed in the porewater indicated supersaturation of secondary phases. The modeling simulated pH 11 and Eh of –170 mV, representative of the conditions observed in the saturated column tests. Chloride concentrations measured in the saturated column tests appear to be erroneous, as they exceeded the total chlorine content of the paste backfill. A chloride concentration of 650 mg/L provided a better ion balance and was therefore used in the modeling. Only iron and nickel were determined to be above saturation (with ferrihydrite and nickel hydroxide, respectively). Other phases that were close to equilibrium included gypsum, barite and lead hydroxide. Of the saturated phases, only the hydroxide solubilities are sensitive to changes in pH and the iron solubility is sensitive to changes in Eh. All MINTEQA2 results are provided in Attachment C.

2.5.3.2 Cyanide

The question of cyanide concentrations in the pyritic paste backfill has been raised in reviews of earlier versions of the source term prediction. Sodium cyanide is used in the selective flotation of lead-copper-zinc ores as a depressant for pyrite and to a lesser extent, sphalerite. Cyanide depresses pyrite flotation by reacting with the pyrite surface and inhibiting the binding of flotation reagents.

The surfaces of the pyrite particles are re-activated for the flotation of the pyrite concentrate by decreasing the pH from above 10 s.u. (where cyanide species are stable) to a pH of about 6 s.u.. At this pH cyanide is readily volatilized as HCN as follows:

$$H^+ + CN^- \leftrightarrow HCN\,(g)$$

Volatilization of HCN (g) reduces free cyanide concentrations in the porewater to very low levels. Actual measured cyanide concentrations in the pyrite flotation process water were <0.0034 mg/L. The decrease in the CN^- concentration leads to the dissociation of the dissolved $Fe(CN)_6]^{4-}$ complex, which, in turn, causes dissolution of the surface $Fe_4[(Fe(CN)_6]_3$ precipitate and renders the pyrite amenable to flotation

during depyritization. Thus most of the cyanide is lost to volatilization prior to pyrite flotation.

Residual cyanide precipitates remaining on the pyrite, and ultimately reporting to the pyritic paste backfill, are expected to be negligible and/or very stable. This expectation is supported by the observed total cyanide concentrations in the pyrite concentrate porewater. As summarized in Table 2.12, total cyanide increased from below detection to about 0.010 mg/L after a two month equilibration period. (See also GWQPE, Appendix A1, Attachment D, FVD 1999/2000).

TABLE 2.12
Cyanide Concentrations Measured in Porewater

Time	Total CN in Pyrite Concentrate Porewater* (mg/L)
Initial	<0.0034 / <0.0034
60 days	0.011/ 0.0091

* Data presented as Pyrite Concentrate 1/Pyrite Concentrate 2 Prepared by: DDS
Checked by: KSS

2.5.3.3 Thiosalts

Thiosalts are metastable sulfur oxyanions that form as intermediate products of the oxidation of sulfide to sulfate. They can be produced during the processing of sulfide ores, and therefore could be present in the paste backfill. Under the right conditions, the thiosalts can continue to oxidize, resulting in the production of additional sulfate. Attachment H presents a review of the thiosalt data from the Crandon project, and recommends that thiosalts be considered an additional source of 686 mg/L sulfate in the BEJ case and 1371 mg/L sulfate in the UB case.

2.5.3.4 Summary

Table 2.13 summarizes the solute concentrations from the saturated column test and the MINTEQA2 modeling, and presents the selected BEJ and UB estimates. The solubility controls predicted by MINTEQA2 are reflected in the BEJ estimates, but conservatively neglected in the UB estimates. The BEJ estimate for chloride assumed complete dissolution of the chloride content measured in the solids analyses. The UB estimate adopted the apparently erroneous measurement from the column test.

The bulk of the pyritic paste backfill will remain unoxidized. The volume of unoxidized pyritic paste backfill was estimated by subtracting the volume of "acidic" and "oxidized neutral" backfill from the total backfill volume (ie. the volume of the mineable ore reserve, see Section 1g of Attachment A). The resulting estimate for BEJ and UB is 1.50×10^7 m^3 (1.97×10^7 yd^3).

TABLE 2.13
Solute Concentrations in Unoxidized Paste Backfill

Parameter	Saturated Column Test†	MINTEQA2 Output (Files: UNOX.in/-.out)	Estimated Solute Concentrations in Unoxidized Paste Backfill	
			BEJ	UB
	(mg/L)	(mg/L)	(mg/L)	(mg/L)
Al	0.025		0.025	0.025
Sb	< 0.0012		< 0.0012	< 0.0012
As	0.0053		0.0053	0.0053
Ba	0.51	0.0057	**0.0057**	0.51
Be	0.00033		0.00033	0.00033
B	0.0054		0.0054	0.0054
Cd	< 0.000065		< 0.000065	< 0.000065
Ca	980	954	**954**	980
Cl	1900**		251***	1900
Cr	0.00007		0.00007	0.00007
Co	0.084		0.084	0.084
Cu	0.0027		0.0027	0.0027
F	n/a		n/a	N/a
Fe	0.037	0.0080	**0.0080**	0.037
Pb	0.038	0.0024	**0.0024**	0.038
Mg	0.28		0.28	0.28
Mn	0.0011		0.0011	0.0011
Hg	< 0.00008		< 0.00008	< 0.00008
Ni	0.027	0.000072	**0.000072**	0.027
K	160		160	160
Se	0.031		0.031	0.031
Ag	< 0.00023		< 0.00023	< 0.00023
Na	67		67	67
SO_4	1317		1250	1317
Zn	0.042		0.042	0.042
SO_4 from S_2O_3	1408#		1408#	1408#
pH (s.u.)	11			
Eh (mV)	-170*	-170	-170	

†Source: GWQPE (Table 3.26, Appendix A1, FVD 1999/2000)

Prepared by: DDS
Checked by: CK

Note: bold signifies solubility controlled solutes

n/m = not modeled

n/a = not analyzed

* = Measured in laboratory.

** = Reported concentration exceeds amount of chloride in solids.

*** = Calculated from solids analyses GWQPE.

= S_2O_3 calculated from ion balance and converted to SO_4 equivalent.

2.6 Overall Backfill Contribution to Reflooded Mine Water Quality

The predicted solute concentrations in each component of the pyritic paste backfill are summarized in Table 2.14.

TABLE 2.14
Overall Contribution of Pyritic Paste Backfill to Solute Concentrations in Reflooding Groundwater (without Source Control Measures)

Parameter	Best Engineering Judgement (BEJ)			Upper Bound (UB)		
	Acidic	Oxidized/ Neutral	Unoxidized	Acidic	Oxidized/ Neutral	Unoxidized
Volume (ft^3)	$1.50x10^3$	$1.17x10^6$	$5.30x10^8$	$4.15x10^5$	$1.17x10^6$	$5.29x10^8$
Porosity	0.40	0.40	0.40	0.40	0.40	0.40
Pore vol (ft^3)	$6.02x10^2$	$4.66x10^5$	$2.12x10^8$	$1.66x10^5$	$4.66x10^5$	$2.12x10^8$
Al mg/L	1896	0.0015	0.025	1,896	13	0.025
Sb mg/L	2.1	0.033	0.0012	2.1	0.33	0.0012
As mg/L	650	0.24	0.0053	650	1.1	0.0053
Ba mg/L	0.00071	0.0031	0.0057	0.60	3.0	0.51
Be mg/L	0.082	0.0077	0.00033	0.082	0.028	0.00033
B mg/L	0.029	nd	0.0054	0.029	nd	0.0054
Cd mg/L	26	0.0059	0.000065	26	0.011	0.000065
Ca mg/L	471	587	954	2,602	716†	980
Cl mg/L	251	251	251	323	251	1900
Cr mg/L	21	0.095	0.000070	21	0.17	0.000070
Co mg/L	18	0.040	0.084	18	0.10	0.084
Cu mg/L	760	0.22	0.0027	760	0.58	0.0027
F mg/L	53	1.0	nd	53	8.1	nd
Fe mg/L	59,756	0.29	0.0080	59,756	1.5	0.037
Pb mg/L	3.1	0.27	0.0024	520	0.30	0.038
Mg mg/L	2,852	7.4	0.28	2,852	160	0.28
Mn mg/L	193	0.0053	0.0022	193	0.086	0.0011
Hg mg/L	0.27	0.00060	0.000080	0.27	0.0023	0.000080
Ni mg/L	3.3	0.039	0.000072	3.3	0.14	0.027
K mg/L	24	879	160	24	1,125	160
Se mg/L	7.1	0.51	0.031	7.1	1.6	0.031
Ag mg/L	0.021	0.021	0.00023	0.37	0.71	0.00023
Na mg/L	291	287	67	2,481	291	67
SO_4 mg/L	164,173	2,147	1,250	169,555	2,659†	1,317
SO_4 from S_2O_3 mg/L	686	686	686	1,371	1,371	1,371
Zn mg/L	9,322	0.54	0.042	9,322	1.1	0.042

* based on individual source data without MINTEQA2 modelling

†Values are gypsum controlled

Prepared by: DDS
Checked by: CK

3. HANGING WALL WORKINGS

3.1 Approach

To access and remove the ore, NMC proposes to construct a series of shafts, vent-raises, drifts, cross-cuts, and ramps in the hanging wall. Estimates of solute concentrations in water that will reflood the hanging wall workings were derived by a three-step procedure. First, geological descriptions of the hanging wall rocks were reviewed and compared to the mine plan to determine the extent to which each formation will be exposed. Second, the results from column tests of waste rock from each formation, reported in Foth and Van Dyke (1995/1998b), were used to estimate the mass of solutes that would be released by oxidation of the exposed wall rock. Finally, the estimated masses of released solute were assumed to dissolve in the open volume of access workings.

3.2 Geology and Proposed Workings

The geological information was obtained from the following sources:

- Geological Summary Crandon Deposit by A.J. Erickson, Jr. and R. Cote, dated May 24, 1996. Institute on Lake Superior Geology Proceedings, 42nd Annual Meeting.

- Memo Re: Hanging Wall Sulfide Content Review, Crandon Deposit Updated, from A.J. Erickson, Jr. to Mr. D.E. Moe, CMC, dated July 10, 1996, Appendix C, Crandon Project Mine Permit Application (FVD, 1995/1998a).

- Memo Re: Hanging Wall Sulfide Estimates, from A.J. Erickson, Jr. to Mr. K. Black, CMC, dated February 6, 1997 (See Attachment D).

Proposed access developments are described in the Crandon Project Mine Permit Application, and include shafts, vent raises, drifts, cross-cuts and ramps.

- The main production shaft will be 22 feet to 26 feet in diameter and approximately 2,000 feet deep.

- A second shaft, approximately 16 feet in diameter, will be sunk on the eastern end of the orebody to a depth of 1,100 to 1,500 feet.

- A third ventilation shaft, approximately 16 feet in diameter, will be excavated in the western horizon of the orebody.

- A ramp, development crosscuts and drifts will be excavated to access the orebody. They will have a nominal cross section of 16 feet by 13 feet.

- An underground maintenance shop and refueling station will be excavated adjacent to the ramp.

Most of the access development will be contained in the stratigraphic hanging wall rocks to the north of the orebody. These rocks include the Mole Lake, Rice Lake and Skunk Lake Formations, in order of sequence away from the orebody.

The Mole Lake Formation is adjacent to the zinc-rich Crandon Formation. It is a chloritic tuff that contains minor coarse pyroclastics. The eastern half of the formation is a fine chlorite tuff that contains little sulfide. The western half is a sericite-chlorite tuff with chert interbeds that contains veins with up to 4% pyrite.

The Rice Lake Formation is a thick series of volcanic debris and banded pumice flows. These flows typically contain angular chloritic fragments in a light-colored matrix, and little sulfide.

The Skunk Lake Formation is the most distant from the orebody. It consists of chert, cherty tuff and tuff, and contains zones of up to 10% sulfide in some near-surface areas of stringer mineralization.

Estimates of the surface area of wall rock that will be exposed in the access developments are summarized in Table 3.1. The calculations are described in Attachment E. The acid base accounting of the waste rock samples tested in the laboratory investigation (FVD, 1995/1998b) are also provided in Table 3.1.

TABLE 3.1
Estimated Development Wall Rock Exposure Surface Area and Waste Rock Acid Base Accounting

Formation	Exposed Wall Rock	Composite	ABA			
			Sulfide (%)	AP	NP	NNP
Lower Mole Lake	2,617,000 ft^2	Master	0.27	8.4	46	38
Upper Mole Lake	566,000 ft^2	Master	0.1	3.1	48	45
Rice Lake	1,610,000 ft^2	Master	0.05	1.6	1.6	0.0
Skunk Lake	201,000 ft^2	Master	0.14	4.4	1	-3.4

Prepared by: KSS
Checked by: DDS

3.3 Solute Release from Exposed Hanging Wall Rock

3.3.1 Constituent Release Model

The mass of solute released by oxidation of the exposed hanging wall rock was estimated using a simple calculation method:

Constituent release (mg) = specific release (mg ft^{-2} yr^{-1}) x exposed surface (ft^2) x time (yr),

This method is similar to that used in studies of contaminant release from pit walls (e.g. Morin & Hutt, 1995). Although the calculation is quite simple, it does allow the importance of surface area, time, and material properties (as represented by the specific release) to be simulated.

3.3.2 Specific Release Rates

The rates of release of soluble constituents from a unit surface area of wall rock were estimated from results of the large scale humidity cell tests on waste rock samples (FVD 1995/1998b). The concentration and cumulative mass flux for each constituent are provided in Attachment F. The cumulative mass flux for each solute was converted to a weekly mass flux, as summarized in Tables 3.2 and 3.3. Test data over weeks 10 to 54 represent steady state release and were used for the BEJ estimate (Table 3.2). The total mass flux (weeks 1 to 54) was used to estimate the UB (Table 3.3). It should be noted that the database presented in Attachment F has been updated to include antimony, beryllium, cadmium, chromium, cobalt, lead, nickel and silver. These parameters were dropped from the list of analyses reported in Foth and Van

Dyke (1995/1998b) because they were consistently below the detection limits. However, on review of their archives, the testing laboratory (Pace) was able to recover the additional data. Boron, chloride and fluoride data were not available from the wall rock tests. Results from the humidity cell tests on depyritized tailings, which have a sulfide content similar to that of wall rock, were used for these parameters.

TABLE 3.2

Waste Rock Column Test Results - Weekly Flux Rates (mg/kg/week) - BEJ

Parameter	Lower Mole Lake WD-08	Upper Mole Lake WD-07	Rice Lake WD-06	Skunk Lake WD-05
Al	0.034	0.045	0.056	0.0097
Sb	0.0034	0.0026	0.0020	0.0030
As	0.0015	0.00055	0.0011	0.00052
Ba	0.00033	0.00036	0.00038	0.00028
Be	0.00028	0.00016	0.00017	0.00026
B**	0.00022	0.00022	0.00022	0.00022
Cd	0.00028	0.00021	0.00017	0.00026
Ca	1.1	0.64	0.71	0.47
Cl**	0.11	0.11	0.11	0.11
Cr	0.00048	0.00023	0.00019	0.00043
Co	0.00050	0.00028	0.00030	0.00028
Cu	0.00083	0.00094	0.0016	0.0014
F**	0.0013	0.0013	0.0013	0.0013
Fe	0.044	0.064	0.076	0.016
Pb	0.0021	0.0011	0.00083	0.0013
Mg	0.41	0.32	0.30	0.27
Mn	0.0037	0.0028	0.0032	0.0064
Hg	0.000017	0.0000071	0.0000075	0.000011
Ni	0.0014	0.0013	0.0013	0.0020
K	0.18	0.16	0.14	0.16
Se	0.00031	0.00028	0.00024	0.00047
Ag	0.00055	0.00032	0.00033	0.00049
Na	0.11	0.094	0.082	0.047
SO_4	1.2	0.64	0.42	1.4
Zn	0.0031	0.013	0.0013	0.0046

Notes: Calculations based on flux cycles over 10-54 Prepared by: DDS

** B, Cl and F data was not available for these tests. The values presented are from HC-DPT IT & LT (Depyritized tailings), humidity cells (GWQPE Appendix A3, Attachment A, Tables A3 and A4, FVD (1999/2000)). The humidity cell data were corrected for surface area calculated from the tailings size distribution. Checked by: CK

TABLE 3.3
Waste Rock Column Test Results - Weekly Flux Rates (mg/kg/week) - UB

Parameter	Lower Mole Lake WD-08	Upper Mole Lake WD-07	Rice Lake WD-06	Skunk Lake WD-05
Al	0.074	0.098	0.23	0.047
Sb	0.0033	0.0025	0.0020	0.0030
As	0.0018	0.00058	0.0011	0.00057
Ba	0.00040	0.00046	0.00068	0.00038
Be	0.00029	0.00017	0.00017	0.00026
B**	0.00022	0.00022	0.00022	0.00022
Cd	0.00029	0.00021	0.00018	0.00027
Ca	1.1	0.63	0.69	0.52
Cl**	0.11	0.11	0.11	0.11
Cr	0.00045	0.00024	0.00030	0.00043
Co	0.00051	0.00030	0.00035	0.00032
Cu	0.0015	0.0014	0.0041	0.0031
F**	0.0013	0.0013	0.0013	0.0013
Fe	0.11	0.14	0.32	0.076
Pb	0.0022	0.0012	0.0010	0.0015
Mg	0.49	0.36	0.45	0.33
Mn	0.0051	0.0040	0.0086	0.0085
Hg	0.000017	0.0000071	0.0000075	0.000011
Ni	0.0015	0.0012	0.0013	0.0018
K	0.26	0.20	0.20	0.23
Se	0.00033	0.00029	0.00024	0.00058
Ag	0.00051	0.00029	0.00031	0.00046
Na	0.30	0.22	0.18	0.086
SO_4	1.6	0.76	0.52	1.7
Zn	0.0036	0.011	0.0029	0.0046

Notes: Calculations based on flux cycles over 10-54 Prepared by: DDS
** B, Cl and F data was not available for these tests. The values presented are from Checked by: CK
HC-DPT IT & LT (Depyritized tailings), humidity cells (GWQPE Appendix A3, Attachment A, Tables A3 & A4, FVD (1999/2000)). The humidity cell data were corrected for surface area calculated from the tailings size distribution.

The column testing was completed on master composites characteristic of each formation. The samples had a size distribution of less than 1.5 inches diameter. Based on the size gradation analyses, the specific surface area of the rock samples tested is on the order of 18 to 20 ft^2/kg. The specific surface area calculations are provided in Attachment G. The rate of constituent leaching is expressed on a surface area basis in Table 3.4 (BEJ) and in Table 3.5 (UB). Boron, chloride and fluoride leaching rates, which were obtained from tests on depyritized tailings, were corrected to account for the higher tailings surface area.

TABLE 3.4
Column Test Results - Mass Leached/Surface Area - BEJ

Parameter	Lower Mole Lake	Upper Mole Lake	Rice Lake	Skunk Lake
	Specific Surface Area (ft^2/kg)			
Spec. S.A.*	19	19	18	20
	Specific Release Rate ($mg/ft^2/yr$)			
Al	0.094	0.12	0.16	0.025
Sb	0.0092	0.0071	0.0060	0.0078
As	0.0041	0.0015	0.0031	0.0013
Ba	0.00090	0.0010	0.0011	0.00070
Be	0.00078	0.00045	0.00050	0.00065
B**	0.00060	0.00060	0.00065	0.00056
Cd	0.00078	0.00058	0.00050	0.00065
Ca	3.1	1.8	2.1	1.2
Cl**	0.29	0.29	0.31	0.27
Cr	0.0013	0.00063	0.00056	0.0011
Co	0.0014	0.00078	0.00088	0.00070
Cu	0.0023	0.0026	0.0046	0.0036
F**	0.0037	0.0037	0.0039	0.0034
Fe	0.12	0.18	0.22	0.040
Pb	0.0058	0.0031	0.0024	0.0033
Mg	1.1	0.88	0.90	0.69
Mn	0.010	0.0078	0.0095	0.016
Hg	0.000048	0.000019	0.000022	0.000029
Ni	0.0039	0.0034	0.0039	0.0050
K	0.50	0.43	0.42	0.41
Se	0.00084	0.00077	0.00072	0.0012
Ag	0.0015	0.00086	0.00098	0.0013
Na	0.30	0.26	0.24	0.12
SO_4	3.3	1.8	1.2	3.6
Zn	0.0085	0.035	0.0039	0.012

* Specific Surface Area
Prepared by: DDS

** B, Cl and F data was not available for these tests.
Checked by: CK

The values presented are from HC-DPT IT & LT (Depyritized tailings), humidity cells (GWQPE Appendix A3, Attachment A, Tables A3&4, FVD (1999/2000)). The humidity cell data were corrected for surface area calculated from the tailings size distribution.

TABLE 3.5
Column Test Results - Mass Leached/Surface Area - UB

Parameter	Lower Mole Lake	Upper Mole Lake	Rice Lake	Skunk Lake
	Specific Surface Area (ft^2/kg)			
Spec. S.A.*	19	19	18	20
	Specific Release Rate ($mg/ft^2/yr$)			
Al	0.20	0.27	0.66	0.12
Sb	0.0091	0.0067	0.0059	0.0077
As	0.0050	0.0016	0.0032	0.0015
Ba	0.0011	0.0013	0.0020	0.00098
Be	0.00079	0.00045	0.00051	0.00067
B	0.00060	0.00060	0.00065	0.00056
Cd	0.00079	0.00057	0.00052	0.00070
Ca	3.1	1.7	2.0	1.3
Cl	0.29	0.29	0.31	0.27
Cr	0.0012	0.00066	0.00087	0.0011
Co	0.0014	0.00081	0.0010	0.00083
Cu	0.0042	0.0040	0.012	0.0080
F	0.0037	0.0037	0.0039	0.0034
Fe	0.31	0.39	0.95	0.19
Pb	0.0059	0.0032	0.0029	0.0038
Mg	1.3	0.99	1.3	0.83
Mn	0.014	0.011	0.025	0.022
Hg	0.000048	0.000019	0.000022	0.000029
Ni	0.0041	0.0032	0.0037	0.0047
K	0.70	0.56	0.58	0.58
Se	0.00090	0.00078	0.00070	0.0015
Ag	0.0014	0.00080	0.00091	0.0012
Na	0.83	0.60	0.53	0.22
SO_4	4.3	2.1	1.5	4.3
Zn	0.0098	0.031	0.0084	0.012

* Specific Surface Area

Prepared by: DDS
Checked by: CK

3.3.3 Exposed Wall Rock Surface Area

Initial estimates of the exposed wall rock surface area within each formation were calculated from waste rock volumes and nominal mine working dimensions of 16' by 13'. However, blasting associated with mine development and operation will fracture the wall rock immediately surrounding mine openings. The effect will be to expose

additional surface area to oxidation. The amount of additional surface area will depend on the:

- structural characteristics of the wall rock and mineralization;
- degree of weathering;
- confining pressures;
- explosive charge and characteristics;
- blasting pattern employed;
- mining/ground support methods; and,
- size and orientation of mine working relative to the primary structural planes of the wall rock.

A study carried out in three open pits measured reactive surface/nominal surface ratios of 27:1, 33:1 and 161:1 (Morin and Hutt, 1995). Because the lateral extent of mine wall fracturing associated with underground workings is generally less than that of surface workings (Morin, 1990), it is reasonable to assume that the reactive surface/nominal surface ratio for underground mine operations should lie near or below the lower end of the values measured in the open pit operations. For the constituent release calculations, it was assumed that the reactive surface/nominal surface ratio for the Crandon Project would be 30:1. A sensitivity analysis included in Section 7 considers the effects of higher reactive surface/nominal surface ratios.

3.3.4 Exposure Time

An average oxidation period for each rock formation was determined by NMC, based on the mine development plan. Development in most formations will occur early in the mine life to create access drifts. Most of the development in the Upper Mole Lake formation will occur early in the mine life due to excavation of a storage stope for excess paste backfill. It is assumed that this stope will be filled early in the mine life also. Therefore the exposure time for this formation is short. Exposure times of 22 years (Lower Mole Lake), 4 years (Upper Mole Lake), 29 years (Rice Lake) and 30 years (Skunk Lake) were assumed in the calculation.

3.3.5 Results

The resulting estimates of constituent release from the hanging wall rock are summarized in Tables 3.6 and 3.7.

TABLE 3.6
Estimated Constituent Release from Wall Rock in Development Workings - BEJ

Parameter	Lower Mole Lake	Upper Mole Lake	Rice Lake	Skunk Lake	Total Wall Rock
Area (ft^2)	2,617,000	566,000	1,610,000	201,000	4,995,000
Eff. Area (ft^2)*	78,511,000	16,986,000	48,314,000	6,039,000	149,850,000
Constituent Accumulation (kg)					
Al	162	8.3	229	4.5	404
Sb	16	0.48	8.4	1.4	26
As	7.0	0.10	4.4	0.24	12
Ba	1.6	0.067	1.6	0.13	3.3
Be	1.3	0.030	0.71	0.12	2.2
B	1.0	0.041	0.90	0.10	2.1
Cd	1.3	0.040	0.71	0.12	2.2
Ca	5278	120	2913	217	8528
Cl	501	20	436	49	1005
Cr	2.3	0.042	0.78	0.20	3.3
Co	2.4	0.053	1.2	0.13	3.8
Cu	4.0	0.17	6.4	0.65	11
F	6.3	0.25	5.5	0.62	13
Fe	209	12	315	7.3	543
Pb	10	0.21	3.4	0.61	14
Mg	1952	60	1255	126	3392
Mn	18	0.53	13	3.0	34
Hg	0.082	0.0013	0.031	0.0053	0.12
Ni	6.7	0.23	5.5	0.91	13
K	857	29	586	74	1547
Se	1.5	0.052	1.0	0.22	2.7
Ag	2.6	0.059	1.4	0.23	4.2
Na	512	17	339	22	890
SO_4	5688	120	1750	659	8216
Zn	15	2.4	5.5	2.1	25

* Effective Area = Wall Rock Surface Area * Reactive Surface Area/Wall Rock Surface Area Ratio (30:1)

Prepared by: DDS
Checked by: CK

TABLE 3.7
Estimated Constituent Release from Wall Rock in Development Workings - UB

Parameter	Lower Mole Lake	Upper Mole Lake	Rice Lake	Skunk Lake	Total Wall Rock
Area (ft^2)	2,617,000	566,000	1,610,000	201,000	4,995,000
Eff. Area (ft^2)*	78,511,000	16,986,000	48,314,000	6,039,000	149,850,000
Constituent Accumulation (kg)					
Al	349	18	929	22	1318
Sb	16	0.46	8.3	1.4	26
As	8.6	0.11	4.5	0.27	13
Ba	1.9	0.085	2.8	0.18	5.0
Be	1.4	0.031	0.72	0.12	2.2
B	1.0	0.041	0.90	0.10	2.1
Cd	1.4	0.039	0.73	0.13	2.2
Ca	5284	117	2847	240	8487
Cl	501	20	436	49	1005
Cr	2.1	0.045	1.2	0.20	3.6
Co	2.4	0.055	1.5	0.15	4.1
Cu	7.3	0.27	17	1.5	26
F	6.3	0.25	5.5	0.62	13
Fe	529	26	1338	35	1928
Pb	10	0.22	4.1	0.68	15
Mg	2317	68	1837	151	4372
Mn	24	0.74	35	4.0	64
Hg	0.082	0.0013	0.031	0.0053	0.12
Ni	7.1	0.22	5.2	0.85	13
K	1215	38	806	104	2163
Se	1.5	0.053	0.98	0.27	2.9
Ag	2.4	0.054	1.3	0.21	3.9
Na	1429	41	746	40	2255
SO_4	7461	141	2138	772	10512
Zn	17	2.1	12	2.1	33

* Effective Area = Wall Rock Surface Area * Reactive Surface Area/Wall Rock Surface Area Ratio (30:1)

Prepared by: DDS
Checked by: CK

3.4 Wall Rock Contribution to Reflooded Mine Water Quality

To relate the estimated constituent releases from the exposed hanging wall rock to solute concentrations in the reflooding water, it was assumed that the accumulated mass of each constituent (as shown in Table 3.6 and Table 3.7) will completely dissolve in the water that fills the workings. Under this assumption, the concentration estimates are obtained by dividing the masses accumulated in each formation by the volume of the waste rock development in each formation. The volume calculation is provided in Attachment E. The BEJ estimates are summarized in Table 3.8. UB estimates are summarized in Table 3.9.

TABLE 3.8
Estimated Contribution to Development Opening
Groundwater After Reflooding - BEJ

Parameter	Lower Mole Lake	Upper Mole Lake	Rice Lake	Skunk Lake	Total Wall Rock*
Estimated Vol. (ft^3)	$9.39x10^6$	$2.03x10^6$	$5.78x10^6$	$7.22x10^5$	$1.79x10^7$
Estimated Concentrations on Reflooding (mg/L)					
Al	0.61	0.14	1.4	0.22	0.80
Sb	0.060	0.0084	0.051	0.069	0.052
As	0.026	0.0018	0.027	0.012	0.023
Ba	0.0059	0.0012	0.0095	0.0062	0.0065
Be	0.0051	0.00053	0.0043	0.0058	0.0043
B	0.0039	0.00071	0.0055	0.0050	0.0041
Cd	0.0051	0.00069	0.0043	0.0058	0.0044
Ca	20	2.1	18	11	17
Cl	1.9	0.34	2.7	2.4	2.0
Cr	0.0085	0.00074	0.0048	0.0097	0.0065
Co	0.0089	0.00092	0.0076	0.0062	0.0075
Cu	0.015	0.0030	0.039	0.032	0.022
F	0.024	0.0043	0.034	0.030	0.025
Fe	0.79	0.21	1.9	0.36	1.1
Pb	0.038	0.0036	0.021	0.030	0.028
Mg	7.3	1.0	7.7	6.2	6.7
Mn	0.066	0.0092	0.082	0.14	0.068
Hg	0.00031	0.000023	0.00019	0.00026	0.00024
Ni	0.025	0.0041	0.033	0.045	0.026
K	3.2	0.51	3.6	3.6	3.0
Se	0.0055	0.00091	0.0062	0.011	0.0054
Ag	0.010	0.0010	0.0084	0.011	0.0084
Na	1.9	0.30	2.1	1.1	1.8
SO_4	21	2.1	11	32	16
Zn	0.055	0.041	0.034	0.10	0.049

* Total concentration based on weighted average and assumes complete mixing

Prepared by: DDS
Checked by: CK

TABLE 3.9
Estimated Contribution to Development Opening Groundwater After Reflooding - UB

Parameter	Lower Mole Lake	Upper Mole Lake	Rice Lake	Skunk Lake	Total Wall Rock*
Estimated Vol. (ft^3)	$9.39x10^6$	$2.03x10^6$	$5.78x10^6$	$7.22x10^5$	$1.79x10^7$
Estimated Concentrations on Reflooding (mg/L)					
Al	1.3	0.32	5.7	1.1	2.6
Sb	0.059	0.0079	0.051	0.068	0.051
As	0.032	0.0019	0.027	0.013	0.026
Ba	0.0072	0.0015	0.017	0.0087	0.010
Be	0.0051	0.00054	0.0044	0.0059	0.0044
B	0.0039	0.00071	0.0055	0.0050	0.0041
Cd	0.0051	0.00067	0.0044	0.0062	0.0044
Ca	20	2.0	17	12	17
Cl	1.9	0.34	2.7	2.4	2.0
Cr	0.0080	0.00079	0.0075	0.0098	0.0071
Co	0.0091	0.00096	0.0089	0.0073	0.0080
Cu	0.027	0.0047	0.10	0.071	0.051
F	0.024	0.0043	0.034	0.030	0.025
Fe	2.0	0.46	8.2	1.7	3.8
Pb	0.038	0.0038	0.025	0.033	0.030
Mg	8.7	1.2	11	7.4	8.6
Mn	0.092	0.013	0.22	0.19	0.13
Hg	0.00031	0.000023	0.00019	0.00026	0.00024
Ni	0.027	0.0038	0.032	0.042	0.026
K	4.6	0.66	4.9	5.1	4.3
Se	0.0058	0.00093	0.0060	0.013	0.0056
Ag	0.0090	0.00094	0.0078	0.010	0.0078
Na	5.4	0.71	4.6	1.9	4.4
SO_4	28	2.5	13	38	21
Zn	0.064	0.036	0.072	0.10	0.065

* Total concentration based on weighted average and assumes complete mixing

Prepared by: DDS
Checked by: CK

4. UNMINED CRANDON ORE

4.1 Approach

Mining will not remove all of the rock from the ore zone. A 100-foot crown pillar will be left at the top of the mine to ensure ground stability. To assess the potential for the unmined areas to affect solute concentrations in the reflooded mine, a procedure similar to that described in Section 3 was followed. The geology and exposure of the residual ore zone components were reviewed, and potential sources of solutes were identified. Constituent release calculations were then completed and solute concentrations estimated.

4.2 Components of Unmined Ore Zone

4.2.1 Unmined Stopes and Ore Zone Wall Rock

Blasthole open stoping will be the mining method used in ore horizons below the 300 foot level. Above that elevation cut and fill mining methods will be used in the ore below the crown pillar. With these methods, unmineable sill and rib pillars, low grade zones and other unrecoverable ore will be left in place.

These active surfaces will remain exposed to mine air for a relatively short period (weeks) prior to backfilling. Significant oxidation and metal release in the long term is therefore not anticipated. All surfaces within the orebody (with the exception of the crown pillar back that is discussed in the next section) will be contacted with alkaline water during backfilling. Most of the solutes released during mining will be carried into the mine drainage water, collected and pumped to the mine water treatment system. It was therefore concluded that unmined stopes and wall rock in the ore zone are unlikely to be significant sources of constituent release to the reflooding groundwater.

4.2.2 Crown Pillar

The crown pillar geology is also described in the references cited in Section 4.2.1. The crown pillar will vary in thickness from a minimum of about 100 feet to greater than 310 feet (Erickson and Coté, January, 1996). The highest point of the top of the crown pillar is about 100 feet below ground level.

The crown pillar is predominantly composed of:

- an upper iron oxide, silica-rich, gossan;
- a limited zone of supergene sulfide mineralization lying below the gossan; and,
- a lower, unmined sulfide zone.

The gossan has resulted from weathering of the sulfide mineralization, and it contains only trace quantities of residual sulfides. The gossan generally extends less than 100 feet below the subcrop, and is commonly only a few tens of feet thick. Very locally it extends in excess of 400 feet below the subcrop along subvertical lithologic horizons, or along subvertical contacts within and along the margin of the Crandon Formation. After mining, it is estimated that approximately 2.3 M tons of the oxidized zinc ore gossan and approximately 0.7 M tons of the oxidized copper ore gossan will remain. The very limited amount of sulfide mineral in the gossan zone, and the extensive weathering that has already occurred, suggest that the gossan is unlikely to be a significant source of constituent release to the reflooding groundwater.

A limited amount of supergene copper sulfide mineralization, including chalcocite, covellite, and minor bornite, is present beneath the gossan. Minor amounts of copper oxides have also been noted in the saprolite, located over the orebody and within the upper, strongly weathered foot wall and hanging wall rocks. The supergene copper enriched zone is, however, small, in part due to the low amount of copper in the primary mineralization and in part to copper removal through weathering and surface erosion.

Sulfide ore is encountered below the gossan zone and saprolite. The upper limit of mining will be in the sulfide zone. The unmineable sulfide ores in the crown pillar vary from approximately 10 to 180 feet thick. The estimated quantities are:

- 2.1 million tons of zinc ore, with an average sulfur content of 36% (sulfide mineral content of 72%); and,

- 1.9 million tons of copper ore, with an average sulfur content of 9% (sulfide mineral content of 17%).

The sulfide ores represent a potential source for constituent release to the reflooding groundwater. However, the crown pillar area is covered by glacial overburden, which

varies in thickness from 75 feet to over 250 feet. The overburden cover is expected to minimize the direct access of oxygen to the sulfidic parts of the crown pillar.

- Under the conservative assumptions that the entire mean annual infiltration of approximately 10 inches (GeoTrans, 1996) percolates through the glacial overburden to the crown pillar, and maintains an oxygen concentration at the theoretical maximum solubility of 12.8 mg/L (at sea level), the flux of oxygen to the crown pillar will be approximately $3x10^{-9}$ mol O_2 m^{-2} s^{-1}. For comparison, this flux is four orders of magnitude less than the oxygen fluxes estimated in Chapter 2 for direct contact of ventilation air with the backfill.

- Similarly, air phase diffusion of oxygen through an average 150 feet of overburden at a diffusion coefficient of $3 \times 10^{-6} m^2 s^{-1}$ (which assumes uniformly unsaturated conditions), would supply only 5×10^{-7} mol $O_2 m^{-2} s^{-1}$ to the top of the crown pillar. Diffusion rates through the gossan zone are likely to be at least one order of magnitude less, implying that the diffusive flux to the sulfidic portions of the crown pillar will also be insignificant.

Clearly, areas where ventilation air is in direct contact with the crown pillar, i.e. the stope backs, will see the most significant oxidation.

4.3 Solute Release from Exposed Crown Pillar

4.3.1 Solute Release Model

The model used in Section 3 to estimate solute release from the exposed wall rock was also applied to estimate solute release from the exposed lower surface of the crown pillar.

4.3.2 Specific Release Rates

The rates of release of solutes associated with the crown pillar were estimated using the results of a leaching program conducted on a sample of Crandon zinc ore (NMC, March 31, 2000). The purpose of the leaching program was to generate acidic wastewater containing significant amounts of solutes, particularly sulfate, arsenic and metals, for wastewater treatment bench testing. The ore sample used was therefore over-represented in metal sulfides. Although this testing protocol does not represent the conditions expected for the crown pillar, the results do provide a conservative estimate of solute release.

The mass of each solute leached from the column over cycles 3 to 21 (BEJ estimate) and all 21 cycles (UB estimate), and the specific release rates over the same period are shown in Table 4.1.

TABLE 4.1

Crown Pillar – Weekly Flux Rates and Specific Mass Leached

Parameter	Weekly Flux Rate		Specific Mass Leached	
	BEJ† (mg/kg/wk)	UB‡ (mg/kg/wk)	BEJ (mg/ft²/yr)*	UB (mg/ft²/yr)*
Al	0.024	0.32	0.090	1.2
Sb	0.014	0.014	0.054	0.054
As	0.0016	0.0021	0.0058	0.0079
Ba	0.0019	0.0023	0.0073	0.0086
Be	0.0015	0.0015	0.0056	0.0057
B**	nd	nd	nd	nd
Cd	0.029	0.12	0.11	0.46
Ca	8.1	8.8	30	33
Cl**	nd	nd	nd	nd
Cr	0.0015	0.0015	0.0056	0.0057
Co	0.025	0.059	0.093	0.22
Cu	0.29	2.4	1.1	9.1
F**	nd	nd	nd	nd
Fe	0.11	1.3	0.40	4.9
Pb	0.015	0.030	0.056	0.11
Mg	4.3	5.2	16	20
Mn	0.086	0.13	0.32	0.48
Hg	0.000060	0.000064	0.00022	0.00024
Ni	0.0068	0.012	0.026	0.047
K***	1.2	1.1	4.4	4.1
Se	0.0075	0.0094	0.028	0.035
Ag***	0.0047	0.0045	0.018	0.017
Na	0.63	0.71	2.4	2.7
SO_4	36	72	135	271
Zn	6.1	23	23	85

* Estimated specific surface area is approximately 13.9 ft² for the Crandon Ore Sample.

Prepared by: DDS
Checked by: CK

*** Total mass released from cycle 3 to 21 was similar to mass released over the entire testing period. When divided by the longer time period used for the UB, the UB concentrations for these parameters are slightly less than the BEJ case.

† Calculated from mass flux over cycles 3 to 21. This range was selected to reduce the influence of first flush.

‡ Calculated from mass flux over cycles 1 to 21.

nd = No data

4.3.3 Surface Area Estimates

The surface area of copper sulfide ore and zinc sulfide ore that would be exposed at the base of the crown pillar was estimated from the sections provided by Erickson (1996). The ratio of reactive surface area to wall area was assumed to be 30:1, for the reasons provided in Section 3.3.3.

4.3.4 Exposure Time

Mining directly below the crown pillar will be by cut and fill methods, and the backfill will be cemented. Tight-filling methods will be used to minimize the void space between the top of the backfill and the stope backs. The backfill will seal off the stope access point, and prevent subsequent access of ventilation air to the stope back. The oxidation will therefore be limited to the period from end of mining to backfilling, estimated to be approximately six weeks.

Exposure time in the grout drift will be considerably longer. However, NMC plans to remove the grout drift along with ore during the later stages of mining, so it will not be a source of solutes during reflooding.

4.3.5 Results

The resulting estimation of solute release from the lower surface of the crown pillar is shown in Table 4.2.

TABLE 4.2
Estimated Solute Release from Crown Pillar

Parameter	BEJ	UB
Area (ft^2)	475,000	475,000
Eff. Area (ft^2)	14,263,000	14,263,000
Solute Accumulation (kg)		
Al	0.15	1.9
Sb	0.088	0.089
As	0.0096	0.013
Ba	0.012	0.014
Be	0.0092	0.0095
B*	nd	nd
Cd	0.18	0.75
Ca	50	55
Cl*	nd	nd
Cr	0.0092	0.0095
Co	0.15	0.37
Cu	1.8	15
F*	nd	nd
Fe	0.66	8.0
Pb	0.091	0.19
Mg	27	32
Mn	0.53	0.78
Hg	0.00037	0.00040
Ni	0.042	0.077
K	7.2	6.8
Se	0.046	0.058
Ag	0.029	0.028
Na	3.9	4.4
SO_4	222	446
Zn	38	139

nd = No data

Prepared by: DDS
Checked by: CK

4.4 Crown Pillar Contribution to Reflooded Mine Water Quality

To estimate the influence of the crown pillar oxidation on the quality of the reflooding groundwater, it was assumed that all of the released solutes would dissolve in the volume of water that would fill the void between the crown pillar back and the top of the stope backfill. The void was assumed to be uniformly 2 ft. high across the entire

area of stope backs in the crown pillar. The resulting concentration estimates are shown in Table 4.3.

The results shown in Table 4.3 are conservatively high because they do not take into account solubility controls. Furthermore, the pyritic paste backfill is expected to act as a source of alkalinity that would neutralize much of the oxidation products from the adjacent crown pillar. Neglecting this effect makes the concentration estimates more conservative.

TABLE 4.3
Estimated Contribution to Crown Pillar Head Space Groundwater After Reflooding

Parameter	Estimated Concentration	
	BEJ mg/L	**UB** mg/L
Al	0.0055	0.072
Sb	0.0033	0.0033
As	0.00036	0.00048
Ba	0.00044	0.00053
Be	0.00034	0.00035
B	nd	nd
Cd	0.0067	0.028
Ca	1.9	2.0
Cl	nd	nd
Cr	0.00034	0.00035
Co	0.0057	0.014
Cu	0.066	0.55
F	nd	nd
Fe	0.024	0.30
Pb	0.0034	0.0069
Mg	0.99	1.2
Mn	0.020	0.029
Hg	0.000014	0.000015
Ni	0.0016	0.0029
K	0.27	0.25
Se	0.0017	0.0022
Ag	0.0011	0.0010
Na	0.15	0.16
SO_4	8.2	17
Zn	1.4	5.2

nd = no data

Prepared by: DDS
Checked by: CK

5. COMBUSTION BY-PRODUCTS AND BLASTING RESIDUES

5.1 Approach

During operation of the mine, there will be three sources of combustion products:

- Natural gas used for mine air heating;
- Fuel oil used for mobile equipment use; and,
- Blasting operations.

The potential for these sources to influence the quality of the reflooded groundwater was evaluated using simple mass balance calculations and a consideration of the geochemical and physical properties of the emission constituents. Metal releases associated with the combustion of natural gas and fuel oil are considered in Section 5.2. Nitrogen compounds resulting from blasting are considered in Section 5.3.

5.2 Metal Release and Accumulation

The primary combustion products from the mine air heating, mobile equipment and blasting are delineated in Table 5.1. The fate of these emissions will be driven by the air flow rate of 700,000 cfm and relatively high levels of humidity. It is expected that gaseous emissions such as sulfur dioxide, volatile organic compounds, carbon monoxide, and the majority of nitrogen oxides will be completely removed from the underground mine by the large air flow. Volatile aerosols including formaldehyde and any unburned fuel oil are expected to evaporate and/or biodegrade resulting in negligible residues in the mine.

Particulate matter with adsorbed constituents, e.g., metals and polycyclic aromatic hydrocarbons (PAHs), will be largely exhausted from the mine, however, a portion will remain. Based on combustion of 1,306,700 gallons per year of fuel oil by the underground equipment, an assumed composition of the fuel (Attachment I) and a 66% particle settling rate in the vertical rise, the annual accumulation of particulate matter that will remain in the mine and metals adsorbed on the particulates are as shown in Table 5.2. The table also includes the contribution of particulate matter and metals from natural gas mine air heating.

The trace metals found in No. 2 fuel oil will vaporize in the high temperature combustion zone of the mobile equipment engines and will eventually condense as the

flue gas temperature decreases. During this condensing phase, most of the metals will condense onto existing particulate matter emitted as part of the diesel exhaust. This occurs uniformly on all available particulate surface area. Studies have shown that a majority of the particles emitted from the combustion of fuel oil are in the submicron size category (Watts and Waytulonis, 1991). Since submicron particles contribute most of the available surface area, these particles will have a high concentration of volatile metals. This process is known as fine-particle enrichment (USEPA, April 1989; USEPA, 1993). As a result of the fine-particle enrichment process, most of the trace metal emissions will be attached to submicron particles.

The very small submicron particles will not settle out in the horizontal passageways within the mine as a result of gravity, but instead will remain suspended in the air stream and be transported to the West Ventilation Raise. As the particles rise, significant deposition will occur as a result of adiabatic air expansion and the condensation of moisture. This deposition process is discussed in "The Phenomenon of Dust Deposition in Exhaust Ventilation Shafts at Inco Mines" (Stachulak, *et al.*, 1991). Therefore, a majority of the trace metals will ultimately end up at the bottom of the west raise, with a portion of these metals likely transported to the project's wastewater treatment system with the mine drainage water that originates from the west raise.

Polycyclic aromatic hydrocarbons (PAH) that result from incomplete combustion are assumed to remain in the underground mine since they are not readily biodegradable or volatile. The resulting annual accumulation of PAHs from both mine air heating and mobile equipment use is 4.03 lb/yr. This is a very conservative estimate since some percentage of the PAHs will be exhausted by the large air flow through the mine.

TABLE 5.1
Primary Mine Related Emissions

Source	Primary Emissions
Mine Air Heating	Particulate Matter Sulfur Dioxide Carbon Monoxide Nitrogen Oxides Hydrocarbons
Mobile Equipment	Particulate Matter Sulfur Dioxide Carbon Monoxide Nitrogen Oxides Hydrocarbons Metals (Arsenic, Beryllium, Cadmium, Chromium, Copper, Lead, Manganese, Mercury, and Nickel)
Blasting	Sulfur Dioxide Carbon Monoxide Nitrogen Oxides

Source: Foth and Van Dyke (1995/1998c) Prepared by: KB
Checked by: SAS1

TABLE 5.2
Annual Particulate Matter and Metal Accumulation from Fuel Combustion from Mine Air Heating and Mobile Sources

Particulate Matter	0.55 ton/yr
Arsenic	0.26 lb/yr
Beryllium	0.15 lb/yr
Cadmium	0.67 lb/yr
Chromium	2.96 lb/yr
Copper	17.07 lb/yr
Lead	0.42 lb/yr
Manganese	0.86 lb/yr
Mercury	0.96 lb/yr
Nickel	10.40 lb/yr

Calculated based on data in Foth and Van Dyke (1995/1998c) Prepared by: SAS1
Checked by: JWS

In Table 5.3 the total mass of metal constituents, estimated to accumulate in the mine from combustion over the mine life, are compared to the estimated total mass of metals to be contained in the pyritic paste backfill. This comparison demonstrates that the contribution from the combustion sources is relatively small when compared to the backfill source on a total mass basis. The potential mass of any metal accumulated from combustion is orders-of-magnitude less than the mass in the backfill.

TABLE 5.3
Comparison of Total Metal Mass in Backfill vs.
Accumulated Combustion By-Product Emissions

Metal	Total Metal Mass in Pyritic Paste Backfill kg x 1000	Combustion By-Products kg x 1000
Arsenic	109, 000	0.004
Beryllium	639	0.003
Cadmium	2,970	0.009
Chromium	5,750	0.04
Copper	33,600	0.22
Lead	126,000	0.005
Manganese	185,000	0.01
Mercury	118	0.01
Nickel	1,690	0.13

Calculated based on data in Foth and Van Dyke (1995/1998c)

Prepared by: SAS1
Checked by: JWS

The total metal mass estimates presented in Table 5.3 do not represent the readily available mass of metals to groundwater. Geochemical and physical properties controlling the potential release of combustion constituents to groundwater include:

- DPM (diesel particulate matter) is comprised of submicron to micron sized particles with elemental carbon as the primary constituent, thus resulting in a high surface area particle with good constituent adsorption potential (Watts and Waytulonis, 1991).

- Specific DPM metal adsorption coefficients are not available in the literature as industry and academic experts have viewed the metal DPM fraction to be a negligible environmental concern (Johnson and Gangal, personal communication).

- A large percentage of DPM will be transported to the vertical exhaust shaft and be subsequently removed by agglomeration with shaft water particles. These heavier

particles of water and particulate will fall to the bottom of the shaft over time with the mine drainage water.

- High mine air flow will promote the discharge of the lighter fraction of the DPM.

- Reflooded mine groundwater will be high to neutral in pH with an elevated alkalinity, thereby dissolved metal concentrations will be further controlled by hydroxide secondary mineral phases.

The small potential contribution of constituents from combustion processes, combined with their geochemical and physical properties inhibiting or preventing releases to groundwater, suggest that combustion be viewed as a de minimus source, warranting no additional consideration.

5.3 Nitrogen Contribution from Blasting

The explosive used in the blasting of waste rock and ore will consist of a mixture of ammonium nitrate and fuel oil (ANFO). The great majority of the nitrogen contained in the ANFO will be converted to NO_x gas (providing the explosive effect) upon detonation, and removed from the mine in ventilation air. However, some of the ANFO charges will not detonate, and will become a potential source of nitrogen compounds in the reflooded mine.

Residual nitrogen in the reflooded mine was estimated following the method suggested by Ferguson and Leask (1988). The calculations are presented in Attachment J. Ferguson and Leask (1988) evaluated the export of nitrogen to the environment from coal mines in southeastern British Columbia, Canada. The following paragraphs describe the estimation of residual explosives content, the proportion remaining in the underground mine at closure, and the speciation of the resulting nitrogen solutes.

NMC estimates useage factors for explosives to be approximately 2.2 pounds of explosives per ton of waste mined and 1.0 pound per ton of ore mined. Ferguson and Leask (1988) estimated the proportion of unexploded charges based on the type of explosive used (ANFO or emulsion). Missed or partially missed charges were common at that time due to the very wet conditions in the open pit coal mines. For the residual N calculations, it was assumed that 10% of the explosives used would remain as residual ANFO and that ANFO would be 93% ammonium nitrate.

The amount of the residual N in the backfill returning to the mine was estimated by carrying out a mass balance that incorporates the following steps:

- It was assumed that the residual N in the ore and waste would be released to the mine water. Ore production is assumed to average 5,500 tpd and waste production is 100 tpd.

- The mine water would capture all of the residual N and distribute it among flows to the Mine Water Treatment Plant (90%), to mine drainage water (1%) and to moisture in the ore (9%).

- Nitrogen would be removed from circulation in concentrate shipped from the site, in interstitial porewater retained in the tailings, and by treatment and release of mine water.

- No credit was taken for ammonia stripping within the mill or TMA.

Ferguson and Leask (1988) found that the relative proportions of nitrogen as ammonium, nitrite and nitrate were relatively consistent for different open pit mines. Ammonium nitrate contains 23% nitrogen as ammonia and 77% nitrogen as nitrate. However, oxidation of ammonia to nitrite and nitrate resulted in 10% N-NH4, 3% N-NO_2 and 87% N-NO_3. This proportion was used to estimate the quantities of each form of nitrogen in the backfill.

The mass balance calculation indicated that about 67 mg/L of ammonium nitrate residue would be present in the paste backfill porewater. The speciation would be 2.3 mg/L NH_4-N, 20.4 mg/L NO_3-N and 0.70 mg/L NO_2-N.

It should be noted that the zinc flotation circuit is operated at a pH in excess of 9.5 at which ammonium will convert to ammonia, and that for the well-aerated conditions during flotation, ammonia would be volatilized from the process water. Therefore the NH_4-N concentration is likely to be lower than the above estimate.

6. SUMMARY OF ESTIMATED SOLUTE CONCENTRATIONS

Table 6.1 summarizes BEJ estimates of the solute concentrations and water volumes associated with each source in the reflooded mine. Table 6.2 shows the UB estimates. These estimates represent the "unmanaged" base case, i.e. they do not reflect the effect of the source control measures under consideration by NMC.

TABLE 6.1
BEJ Estimates of Solute Concentrations in Reflooded Mine without Source Controls

Parameter	Pyritic Paste Backfill				Wall Rock					Crown Pillar	Volume Average
	Acidic	Oxidized/ Neutral	Unoxidized	Overall	Mole Lake		Rice Lake	Skunk Lake	Overall		
					Lower	Upper					
Estimated Pore Vol. (ft^3)	$6.02x10^2$	$4.66x10^5$	$2.12x10^8$	$2.12x10^8$	$9.4x10^6$	$2.0x10^6$	$5.8x10^6$	$7.2x10^5$	$1.79x10^7$	$9.5x10^5$	$2.32x10^8$
Al (mg/L)	1,896	0.0015*	0.025	0.030	0.61	0.14	1.40	0.22	0.80	0.0055	0.089
Sb (mg/L)	2.1	0.033	0.0012	0.0013	0.060	0.0084	0.051	0.069	0.052	0.0033	0.0052
As (mg/L)	650	0.24	0.0053	0.0076	0.026	0.0018	0.027	0.012	0.023	0.00036	0.0088
Ba (mg/L)	0.00071*	0.0031*	0.0057*	0.0057	0.0059	0.0012	0.0095	0.0062	0.0065	0.00044	0.0057
Be (mg/L)	0.082	0.0077	0.00033	0.00035	0.0051	0.00053	0.0043	0.0058	0.0043	0.00034	0.00065
B (mg/L)	0.029	nd	0.0054	0.0054	0.0039	0.00071	0.0055	0.0050	0.0041	nd	0.0052
Cd (mg/L)	26	0.0059	0.000065	0.00015	0.0051	0.00069	0.0043	0.0058	0.0044	0.0067	0.00050
Ca (mg/L)	471*	587*	954*	953	20	2.1	18	11	17	1.9	872
Cl (mg/L)	251	251	251	251	1.9	0.34	2.7	2.4	2.0	nd	229
Cr (mg/L)	21	0.095	0.000070	0.00034	0.0085	0.00074	0.0048	0.0097	0.0065	0.00034	0.0008
Co (mg/L)	18	0.040	0.084	0.084	0.0089	0.00092	0.0076	0.0062	0.0075	0.0057	0.077
CN (mg/L)				0.010							0.0084
Cu (mg/L)	760	0.22	0.0027	0.0053	0.015	0.0030	0.039	0.032	0.022	0.066	0.0068
F (mg/L)	53	1.0*	nd	0.0024	0.024	0.0043	0.034	0.030	0.025	nd	0.0041
Fe (mg/L)	59,756	0.29	0.0080*	0.18	0.79	0.21	1.9	0.36	1.1	0.024	0.25
Pb (mg/L)	3.1*	0.27	0.0024*	0.0030	0.038	0.0036	0.021	0.030	0.028	0.0034	0.0049
Mg (mg/L)	2,852	7.4	0.28	0.30	7.3	1.0	7.7	6.2	6.7	0.99	0.80
Mn (mg/L)	193	0.0053	0.0022	0.0028	0.066	0.0092	0.082	0.14	0.068	0.020	0.0078
Hg (mg/L)	0.27	0.00060	0.000080	0.000082	0.00031	0.000023	0.00019	0.00026	0.00024	0.000014	0.000093
Ni (mg/L)	3.3	0.039	0.000072*	0.00017	0.025	0.0041	0.033	0.045	0.026	0.0016	0.0022
NH_4-N (mg/L)				2.3							1.9
NO_3-N (mg/L)				20							17
NO_2-N (mg/L)				0.70							0.59
K (mg/L)	24	879	160	162	3.2	0.51	3.6	3.6	3.0	0.27	148
Se (mg/L)	7.1	0.51	0.031	0.032	0.0055	0.00091	0.0062	0.011	0.0054	0.0017	0.030
Ag (mg/L)	0.021*	0.021*	0.00023	0.00028	0.010	0.0010	0.0084	0.011	0.0084	0.0011	0.00090
Na (mg/L)	291	287	67	67	1.9	0.30	2.1	1.1	1.8	0.15	62
SO_4 (mg/L)	164,173*	2,147*	1,250	1,252	21	2.1	11	32	16	8.2	1145
SO_4 from S_2O_3 (mg/L)	686	686	686	686	0	0	0	0	0	0	626
Zn (mg/L)	9,322	0.54	0.042	0.070	0.055	0.041	0.034	0.10	0.049	1.4	0.073

* Parameter concentration is solubility controlled.
nd = no data

Prepared by: DDS
Checked by: CK

TABLE 6.2
UB Estimates of Solute Concentrations in Reflooded Mine without Source Controls

Parameter	Pyritic Paste Backfill				Wall Rock					Crown Pillar	Volume Average
	Acidic	Oxidized/ Neutral	Unoxidized	Overall	Mole Lake Lower	Mole Lake Upper	Rice Lake	Skunk Lake	Overall		
Estimated Pore Vol. (ft^3)	$1.66x10^5$	$4.66x10^5$	$2.12x10^8$	$2.12x10^8$	$9.4x10^6$	$2.0x10^6$	$5.8x10^6$	$7.2x10^5$	$1.79x10^7$	$9.5x10^5$	$2.32x10^8$
Al (mg/L)	1,896	13	0.025	1.5	1.3	0.32	5.7	1.1	2.6	0.072	1.6
Sb (mg/L)	2.1	0.33	0.0012	0.0036	0.059	0.008	0.051	0.068	0.051	0.0033	0.0072
As (mg/L)	650	1.1	0.0053	0.52	0.032	0.0019	0.027	0.013	0.026	0.00048	0.47
Ba (mg/L)	0.60	3.0	0.51	0.52	0.0072	0.0015	0.017	0.0087	0.010	0.00053	0.47
Be (mg/L)	0.082	0.028	0.00033	0.00045	0.0051	0.00054	0.0044	0.0059	0.0044	0.00035	0.00075
B (mg/L)	0.029	nd	0.0054	0.0054	0.0039	0.00071	0.0055	0.0050	0.0041	nd	0.0053
Cd (mg/L)	26	0.011	0.000065	0.020	0.0051	0.00067	0.0044	0.0062	0.0044	0.028	0.019
Ca (mg/L)	2,602	716*	980	981	20	2.0	17	12	17	2.0	897
Cl (mg/L)	251	251	1900	1895	1.9	0.34	2.7	2.4	2.0	nd	1731
Cr (mg/L)	21	0.17	0.000070	0.017	0.0080	0.00079	0.0075	0.0098	0.0071	0.00035	0.016
Co (mg/L)	18	0.10	0.084	0.098	0.0091	0.00096	0.0089	0.0073	0.0080	0.014	0.090
CN (mg/L)				0.011							0.0092
Cu (mg/L)	760	0.58	0.0027	0.60	0.027	0.0047	0.10	0.071	0.051	0.55	0.55
F (mg/L)	53	8.1	nd	0.060	0.024	0.0043	0.034	0.030	0.025	nd	0.056
Fe (mg/L)	59,756	1.5	0.037	47	2.0	0.46	8.2	1.7	3.8	0.30	43
Pb (mg/L)	520	0.30	0.038	0.44	0.038	0.0038	0.025	0.033	0.030	0.0069	0.41
Mg (mg/L)	2,852	160	0.28	2.9	8.7	1.2	11	7.4	8.6	1.2	3.3
Mn (mg/L)	193	0.086	0.0011	0.15	0.092	0.013	0.22	0.19	0.13	0.029	0.15
Hg (mg/L)	0.27	0.0023	0.000080	0.00030	0.00031	0.000023	0.00019	0.00026	0.00024	0.000015	0.00029
Ni (mg/L)	3.3	0.14	0.027	0.030	0.027	0.0038	0.032	0.042	0.026	0.0029	0.029
NH_4-N (mg/L)				2.3							1.9
NO_3-N (mg/L)				20							17
NO_2-N (mg/L)				0.70							0.59
K (mg/L)	24	1,125	160	162	4.6	0.66	4.9	5.1	4.3	0.25	148
Se (mg/L)	7.1	1.6	0.031	0.040	0.0058	0.00093	0.0060	0.013	0.0056	0.0022	0.037
Ag (mg/L)	0.37	0.71	0.00023	0.0021	0.0090	0.00094	0.0078	0.010	0.0078	0.0010	0.0025
Na (mg/L)	291	291	67	68	5.4	0.71	4.6	1.9	4.4	0.16	64
SO_4 (mg/L)	169,555	2,659*	1,317	1,451	28	2.5	13	38	21	17	1,327
SO_4 from S_2O_3 (mg/L)	1,371	1,371	1,371	1,371							1,252
Zn (mg/L)	9,322	1.1	0.042	7.3	0.064	0.036	0.072	0.10	0.065	5.2	6.7

* Parameter concentrations are solubility controlled
nd = no data

Prepared by: DDS
Checked by: CK

7. SENSITIVITY ANALYSIS

7.1 Approach

The sensitivity of the predicted solute concentrations to various model assumptions was assessed by varying individual input parameters. The results are presented in the following sections.

7.2 Number of Stopes

The number of stopes in the mine could change for many reasons. First, changes in the details of stope design, perhaps based on improved understanding of the rock mechanical properties, could lead to individual stopes being larger or smaller. Second, changes in copper and zinc prices could lead to a change in the amount of ore that is economically extractable. Third, definition drilling could lead to the reclassification of some of the current ore into the uneconomic category or current mineralization into the economic category. The sensitivity analysis presented herein considers the first case, i.e. a change in the number of stopes without a change in the total tonnage of ore. The other possibilities are covered by the sensitivity analysis presented under "early closure" in Section 7.7.

The most probable change in stope dimensions would be the width of the stope (along strike) rather than the vertical height. An increase in the number of stopes would result in an increase in the number of access crosscuts that would be required. Conversely, a decrease in the number of stopes would decrease the number of cross-cuts. The only effect of such changes on the reflooded mine water quality would be a change in the volume of acidic backfill that will form within each cross-cut (see Section 2).

Table 7.1 shows the change in estimated volumes of acidic backfill for a 20% change in the number of stopes. Table 7.2 indicates the effects on volume-average estimates of solute concentrations in the paste backfill. Table 7.3 shows the effect on volume-average estimates of solute concentrations in the overall reflooded mine.

The results indicate that the number of stopes will have a small and generally insignificant effect on water quality. The reason is that the large volume of unoxidized backfill dominates any volume-average concentration estimates. Exceptions are the estimated concentrations of some metals in the UB case. Those

solutes are at sufficiently high concentrations in the acidic backfill that changes to the acidic backfill volume have a noticeable effect, even after volume averaging across the whole reflooded mine.

TABLE 7.1
Pore Volume of Acidic Paste Backfill for Different Number of Stopes

Case	BEJ Volume of Acidic Backfill (ft^3)	UB Volume of Acidic Backfill (ft^3)
20% Fewer Stopes	1.20E+03	3.31E+05
Base Case	1.50E+03	4.15E+05
20% More Stopes	1.81E+03	5.00E+05

Prepared by: DDS
Checked by: CK

TABLE 7.2
Effect of Number of Stopes on Volume Averaged Solute Concentrations in the Pyritic Paste Backfill

Parameter	Best Engineering Judgement (BEJ)			Upper Bound (UB)		
	20% Fewer Stopes	Base Model	20% More Stopes	20% Fewer Stopes	Base Model	20% More Stopes
No. Blasthole	255	318	384	255	318	384
No. Cut & Fill	78	98	118	78	98	118
Total Stopes	333	416	502	333	416	502
Al (mg/L)	0.029	0.030	0.031	1.2	1.5	1.8
Sb (mg/L)	0.0013	0.0013	0.0013	0.0033	0.0036	0.0039
As (mg/L)	0.0073	0.0076	0.0080	0.41	0.52	0.62
Ba (mg/L)	0.0057	0.0057	0.0057	0.52	0.52	0.52
Be (mg/L)	0.00035	0.00035	0.00035	0.00044	0.00045	0.00047
B (mg/L)	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054
Cd (mg/L)	0.00014	0.00015	0.00017	0.016	0.020	0.025
Ca (mg/L)	953	953	953	980	981	981
Cl (mg/L)	251	251	251	1895	1895	1895
Cr (mg/L)	0.00033	0.00034	0.00035	0.013	0.017	0.020
Co (mg/L)	0.084	0.084	0.084	0.095	0.098	0.10
Cu (mg/L)	0.0049	0.0053	0.0058	0.48	0.60	0.72
F (mg/L)	0.0024	0.0024	0.0025	0.051	0.060	0.068
Fe (mg/L)	0.14	0.18	0.21	37	47	56
Pb (mg/L)	0.0030	0.0030	0.0030	0.36	0.44	0.53
Mg (mg/L)	0.30	0.30	0.31	2.4	2.9	3.3
Mn (mg/L)	0.0026	0.0028	0.0029	0.12	0.15	0.18
Hg (mg/L)	0.000082	0.000082	0.000082	0.00025	0.00030	0.00034
Ni (mg/L)	0.00017	0.00017	0.00017	0.029	0.030	0.030
K (mg/L)	162	162	162	162	162	162
Se (mg/L)	0.032	0.032	0.032	0.039	0.040	0.041
Ag (mg/L)	0.00028	0.00028	0.00028	0.0020	0.0021	0.0021
Na (mg/L)	67	67	67	68	68	68
SO_4 (mg/L)	1,252	1,252	1,252	1,425	1,451	1,478
Zn (mg/L)	0.064	0.070	0.075	5.9	7.3	8.8

Prepared by: DDS
Checked by: CK

TABLE 7.3
Effect of Number of Stopes on Volume Averaged Solute Concentrations in the Reflooded Mine

Parameter	Best Engineering Judgement (BEJ)			Upper Bound (UB)		
	20% Fewer Stopes	Base Model	20% More Stopes	20% Fewer Stopes	Base Model	20% More Stopes
Total Stopes	333	416	502	333	416	502
Al (mg/L)	0.088	0.089	0.090	1.3	1.6	1.9
Sb (mg/L)	0.0052	0.0052	0.0052	0.0069	0.0072	0.0075
As (mg/L)	0.0084	0.0088	0.0091	0.38	0.47	0.57
Ba (mg/L)	0.0057	0.0057	0.0057	0.47	0.47	0.47
Be (mg/L)	0.00065	0.00065	0.00065	0.00074	0.00075	0.00077
B (mg/L)	0.0052	0.0052	0.0052	0.0053	0.0053	0.0053
Cd (mg/L)	0.00049	0.00050	0.00052	0.015	0.019	0.023
Ca (mg/L)	872	872	872	897	897	897
Cl (mg/L)	229	229	229	1,731	1,731	1,730
Cr (mg/L)	0.00080	0.00081	0.00082	0.013	0.016	0.019
Co (mg/L)	0.077	0.077	0.077	0.087	0.090	0.092
Cu (mg/L)	0.0064	0.0068	0.0072	0.44	0.55	0.66
F (mg/L)	0.0041	0.0041	0.0042	0.049	0.056	0.64
Fe (mg/L)	0.21	0.25	0.28	34	43	52
Pb (mg/L)	0.0049	0.0049	0.0049	0.33	0.41	0.48
Mg (mg/L)	0.80	0.80	0.80	2.9	3.3	3.7
Mn (mg/L)	0.0077	0.0078	0.0079	0.12	0.15	0.18
Hg (mg/L)	0.000093	0.000093	0.000093	0.00025	0.00029	0.00033
Ni (mg/L)	0.0022	0.0022	0.0022	0.029	0.029	0.030
K (mg/L)	148	148	148	148	148	148
Se (mg/L)	0.030	0.030	0.030	0.036	0.037	0.038
Ag (mg/L)	0.00090	0.00090	0.00090	0.0024	0.0025	0.0025
Na (mg/L)	62	62	62	62	62	62
SO_4 (mg/L)	1,145	1145	1,145	1,303	1,327	1,352
Zn (mg/L)	0.068	0.073	0.078	5.4	6.7	8.1

Prepared by: DDS
Checked by: CK

7.3 Exposed Surface Area

Both the BEJ and UB models of oxygen transport into the acidic paste backfill require an assumption about the surface area of the pyritic paste backfill that will be exposed within each cross-cut. Changes to the assumed surface area will change the resulting estimates of the volume of acidic backfill.

To examine this effect, the lineal dimensions of the cross-cut were assumed to vary by 10%. A 10% change in both dimensions (height and width) of the cross-cut results in the exposed surface area changing by approximately 20%. (The same change in

dimensions would effect a 10% change in the surface area of wall rock within the cross-cuts. However, that translates to less than a 2% change in the overall wall rock surface area, and so was not considered in the sensitivity analysis.)

Table 7.4 summarizes the changes in volume of acidic paste backfill for 10% changes in the cross-cut dimensions. Table 7.5 shows the effects on volume-average estimates of solute concentrations in the paste backfill. Table 7.6 shows the effect on volume-average estimates of solute concentrations in the overall reflooded mine. The effects in this case are the same as in Section 7.2, *i.e.* increases or decreases in the volume of acidic paste backfill are completely dominated by the much larger volume of unoxidized backfill. The only exceptions are again the UB estimates of solutes that are at high concentrations in the UB acidic backfill.

TABLE 7.4
Volumes of Acidic Paste Backfill for Changes in Crosscut Dimensions

Case	BEJ Volume of Acidic Backfill (ft^3)	UB Volume of Acidic Backfill (ft^3)
10% Smaller (11.7' x 14.4')	1.22E+03	3.36E+05
Base Case (13' x 16')	1.50E+03	4.15E+05
10% Larger (14.3' x 17.6')	1.82E+03	5.02E+05

Prepared by: DDS
Checked by CK

TABLE 7.5

Effect of Cross-Cut Dimension on Volume Averaged Solute Concentrations in the Pyritic Paste Backfill

Parameter	Best Engineering Judgement (BEJ)			Upper Bound (UB)		
	10% Smaller	Base Model	10% Larger	10% Smaller	Base Model	10% Larger
Dimensions	11.7' x 14.4'	13' x 16'	14.3' x 17.6'	11.7' x 14.4'	13' x 16'	14.3' x 17.6'
Al (mg/L)	0.029	0.030	0.031	1.3	1.5	1.8
Sb (mg/L)	0.0013	0.0013	0.0013	0.0033	0.0036	0.0039
As (mg/L)	0.0073	0.0076	0.0080	0.42	0.52	0.62
Ba (mg/L)	0.0057	0.0057	0.0057	0.52	0.52	0.52
Be (mg/L)	0.00035	0.00035	0.00035	0.00044	0.00045	0.00047
B (mg/L)	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054
Cd (mg/L)	0.00014	0.00015	0.00017	0.017	0.020	0.025
Ca (mg/L)	953	953	953	980	981	981
Cl (mg/L)	251	251	251	1,895	1,895	1,895
Cr (mg/L)	0.00033	0.00034	0.00035	0.013	0.017	0.020
Co (mg/L)	0.084	0.084	0.084	0.095	0.098	0.10
Cu (mg/L)	0.0049	0.0053	0.0058	0.48	0.60	0.72
F (mg/L)	0.0024	0.0024	0.0025	0.052	0.060	0.068
Fe (mg/L)	0.15	0.18	0.21	38	47	57
Pb (mg/L)	0.0030	0.0030	0.0030	0.37	0.44	0.53
Mg (mg/L)	0.30	0.30	0.31	2.4	2.9	3.3
Mn (mg/L)	0.0026	0.0028	0.0029	0.12	0.15	0.18
Hg (mg/L)	0.000082	0.000082	0.000082	0.00026	0.00030	0.00034
Ni (mg/L)	0.00017	0.00017	0.00017	0.029	0.030	0.030
K (mg/L)	162	162	162	162	162	162
Se (mg/L)	0.032	0.032	0.032	0.039	0.040	0.041
Ag (mg/L)	0.00028	0.00028	0.00028	0.0020	0.0021	0.0021
Na (mg/L)	67	67	67	68	68	68
SO_4 (mg/L)	1,252	1,252	1,252	1,426	1,451	1,479
Zn (mg/L)	0.064	0.070	0.075	5.9	7.3	8.9

Prepared by: DDS
Checked by: CK

TABLE 7.6

Effect of Cross-Cut Dimensions on Volume Average Solute Concentrations in the Reflooded Mine

Parameter	Best Engineering Judgement (BEJ)			Upper Bound (UB)		
	10% Smaller	Base Model	10% Larger	10% Smaller	Base Model	10% Larger
Dimensions	11.7' x 14.4'	13' x 16'	14.3' x 17.6'	11.7' x 14.4'	13' x 16'	14.3' x 17.6'
Al (mg/L)	0.088	0.089	0.090	1.3	1.6	1.9
Sb (mg/L)	0.0052	0.0052	0.0052	0.0069	0.0072	0.0075
As (mg/L)	0.0084	0.0088	0.0091	0.38	0.47	0.57
Ba (mg/L)	0.0057	0.0057	0.0057	0.47	0.47	0.47
Be (mg/L)	0.00065	0.00065	0.00065	0.00074	0.00075	0.00077
B (mg/L)	0.0052	0.0052	0.0052	0.0053	0.0053	0.0053
Cd (mg/L)	0.00049	0.00050	0.00052	0.016	0.019	0.023
Ca (mg/L)	872	872	872	897	897	897
Cl (mg/L)	229	229	229	1,731	1,731	1,730
Cr (mg/L)	0.00080	0.00081	0.00082	0.013	0.016	0.019
Co (mg/L)	0.077	0.077	0.077	0.088	0.090	0.093
Cu (mg/L)	0.0065	0.0068	0.0073	0.45	0.55	0.67
F (mg/L)	0.0041	0.0041	0.0042	0.049	0.056	0.064
Fe (mg/L)	0.22	0.25	0.28	35	43	52
Pb (mg/L)	0.0049	0.0049	0.0049	0.34	0.41	0.49
Mg (mg/L)	0.80	0.80	0.80	2.9	3.3	3.7
Mn (mg/L)	0.0077	0.0078	0.0079	0.12	0.15	0.18
Hg (mg/L)	0.000093	0.000093	0.000093	0.00025	0.00029	0.00033
Ni (mg/L)	0.0022	0.0022	0.0022	0.029	0.029	0.030
K (mg/L)	148	148	148	148	148	148
Se (mg/L)	0.030	0.030	0.030	0.036	0.037	0.038
Ag (mg/L)	0.00090	0.00090	0.00090	0.0024	0.0025	0.0025
Na (mg/L)	62	62	62	62	62	62
SO_4 (mg/L)	1,145	1,145	1,145	1,304	1,327	1,352
Zn (mg/L)	0.068	0.073	0.078	5.5	6.7	8.1

Prepared by: DDS
Checked by: CK

7.4 Exposure Time

Attachment B shows the length of time that backfilled stopes will be in place prior to reflooding, and indicates average "exposure times" of about 14 years for the blasthole and cut-and-fill stopes. Section 3.3.4 presents average exposure times for the wall rock. Changes to the mine plan could affect these assumed exposure times.

In most cases, changes to the mine plan will change only the exposure time of some stopes, and not significantly change the average exposure times. As a worst case assumption for this sensitivity analysis, the average exposure times were altered

upwards and downwards by 3 and 6 years, which would only be possible if there were significant delays or acceleration in the overall mine plan.

The effects of different exposure time on the estimated volume of acidic paste backfill are shown in Table 7.7, and the effects on overall pore water quality in the pyritic paste backfill are summarized in Table 7.8. These results are directly analogous to those presented in the preceding two sections.

The effect of different exposure times on solute release from the wall rocks are summarized in Table 7.9. In general, solute release from the wall rocks for both the BEJ and UB cases are directly proportional to the exposure time.

The combined effects on volume averaged solute concentrations are summarized in Table 7.10. In general, the solute concentrations are not strongly affected by changes in the assumed exposure time.

The effects of an early shutdown of the mine is discussed in Section 7.7.

TABLE 7.7
Volumes of Acidic Paste Backfill for Changes in Stope Exposure Time

Case	BEJ Volume of Acidic Backfill (ft^3)	UB Volume of Acidic Backfill (ft^3)
- 6 Years	$8.62x10^2$	$2.38x10^5$
- 3 Years	$1.18x10^3$	$3.27x10^5$
Base Case	$1.50x10^3$	$4.15x10^5$
+ 3 Years	$1.83x10^3$	$5.05x10^5$
+ 6 Years	$2.15x10^3$	$5.94x10^5$

Prepared by: DDS
Checked by : CK

TABLE 7.8
Effect of Exposure Time on Volume Averaged Solute Concentrations in the Pyritic Paste Backfill

Parameter Exposure Time	Best Engineering Judgment (BEJ)					Upper Bound (UB)				
	-6 yrs mg/L	-3 yrs mg/L	Base Model mg/L	+3 yrs mg/L	+6 yrs mg/L	-6 yrs mg/L	-3 yrs mg/L	Base Model mg/L	+3 yrs mg/L	+6 yrs mg/L
Al	0.028	0.029	0.030	0.031	0.033	0.90	1.2	1.5	1.9	2.2
Sb	0.0013	0.0013	0.0013	0.0013	0.0013	0.0029	0.0032	0.0036	0.0040	0.0043
As	0.0069	0.0073	0.0076	0.0080	0.0084	0.30	0.41	0.52	0.63	0.73
Ba	0.0057	0.0057	0.0057	0.0057	0.0057	0.52	0.52	0.52	0.52	0.52
Be	0.00035	0.00035	0.00035	0.00035	0.00035	0.00043	0.00044	0.00045	0.00047	0.00048
B	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054
Cd	0.00012	0.00014	0.00015	0.00017	0.00018	0.012	0.016	0.020	0.025	0.029
Ca	953	953	953	953	953	980	980	981	981	981
Cl	251	251	251	251	251	1,896	1,895	1,895	1,895	1,895
Cr	0.00031	0.00032	0.00034	0.00035	0.00036	0.010	0.013	0.017	0.020	0.024
Co	0.084	0.084	0.084	0.084	0.084	0.092	0.095	0.098	0.10	0.10
Cu	0.0044	0.0049	0.0053	0.0058	0.0063	0.34	0.47	0.60	0.73	0.85
F	0.0024	0.0024	0.0024	0.0025	0.0025	0.042	0.051	0.060	0.069	0.078
Fe	0.11	0.14	0.18	0.21	0.25	27	37	47	57	67
Pb	0.0030	0.0030	0.0030	0.0030	0.0030	0.27	0.36	0.44	0.53	0.62
Mg	0.30	0.30	0.30	0.31	0.31	1.9	2.4	2.9	3.3	3.8
Mn	0.0025	0.0026	0.0028	0.0029	0.0030	0.088	0.12	0.15	0.18	0.22
Hg	0.000082	0.000082	0.000082	0.000082	0.000082	0.00021	0.00025	0.00030	0.00034	0.00039
Ni	0.00016	0.00017	0.00017	0.00017	0.00017	0.029	0.029	0.030	0.030	0.031
K	162	162	162	162	162	162	162	162	162	162
Se	0.032	0.032	0.032	0.032	0.032	0.038	0.039	0.040	0.041	0.042
Ag	0.00028	0.00028	0.00028	0.00028	0.00028	0.0019	0.0020	0.0021	0.0021	0.0022
Na	67	67	67	67	67	68	68	68	68	68
SO_4	1,252	1,252	1,252	1,252	1,252	1,395	1,423	1,451	1,480	1,508
Zn	0.058	0.064	0.070	0.075	0.081	4.2	5.8	7.3	8.9	10

Prepared by: DDS
Checked by: CK

TABLE 7.9
Effect of Exposure Time on Estimated Wall Rock Solute Concentrations

Parameter / Exposure Time	Best Engineering Judgment (BEJ)					Upper Bound (UB)				
	-6 yrs mg/L	-3 yrs mg/L	Base Model mg/L	+3 yrs mg/L	+6 yrs mg/L	-6 yrs mg/L	-3 yrs mg/L	Base Model mg/L	+3 yrs mg/L	+6 yrs mg/L
Al	0.60	0.69	0.80	0.90	1.0	2.0	2.3	2.6	2.9	3.2
Sb	0.038	0.045	0.052	0.059	0.066	0.038	0.044	0.051	0.058	0.065
As	0.017	0.020	0.023	0.026	0.029	0.020	0.023	0.026	0.030	0.033
Ba	0.0049	0.0057	0.0065	0.0074	0.0082	0.0074	0.0086	0.010	0.011	0.012
Be	0.0032	0.0038	0.0043	0.0049	0.0055	0.0033	0.0038	0.0044	0.0050	0.0056
B	0.0031	0.0036	0.0041	0.0047	0.0052	0.0031	0.0036	0.0041	0.0047	0.0052
Cd	0.0032	0.0038	0.0044	0.0049	0.0055	0.0033	0.0038	0.0044	0.0050	0.0056
Ca	12	15	17	19	21	12	15	17	19	21
Cl	1.5	1.7	2.0	2.2	2.5	1.5	1.7	2.0	2.2	2.5
Cr	0.0048	0.0056	0.0065	0.0074	0.0082	0.0053	0.0062	0.0071	0.0080	0.0089
Co	0.0055	0.0065	0.0075	0.0085	0.0095	0.0060	0.0070	0.0080	0.0091	0.010
Cu	0.017	0.019	0.022	0.025	0.028	0.039	0.045	0.051	0.057	0.063
F	0.019	0.022	0.025	0.028	0.032	0.019	0.022	0.025	0.028	0.032
Fe	0.80	0.93	1.1	1.2	1.4	2.9	3.3	3.8	4.3	4.7
Pb	0.021	0.024	0.028	0.032	0.036	0.022	0.026	0.030	0.034	0.038
Mg	5.0	5.8	6.7	7.6	8.5	6.4	7.5	8.6	9.7	11
Mn	0.051	0.059	0.068	0.077	0.086	0.10	0.11	0.13	0.14	0.16
Hg	0.00017	0.00020	0.00024	0.00027	0.00030	0.00017	0.00020	0.00024	0.00027	0.00030
Ni	0.020	0.023	0.026	0.030	0.033	0.020	0.023	0.026	0.030	0.033
K	2.3	2.6	3.0	3.5	3.9	3.2	3.7	4.3	4.8	5.4
Se	0.0040	0.0047	0.0054	0.0061	0.0068	0.0042	0.0049	0.0056	0.0064	0.0071
Ag	0.0062	0.0073	0.0084	0.0095	0.011	0.0058	0.0067	0.0078	0.0088	0.0098
Na	1.3	1.5	1.8	2.0	2.2	3.3	3.8	4.4	5.0	5.7
SO_4	12	14	16	18	21	15	18	21	24	26
Zn	0.033	0.040	0.049	0.058	0.067	0.046	0.054	0.065	0.075	0.086

Prepared by: DDS
Checked by: CK

TABLE 7.10

Effect of Exposure Time on Volume Average Solute Concentrations in the Reflooded Mine

Parameter / Exposure Time	Best Engineering Judgment (BEJ)					Upper Bound (UB)				
	-6 yrs mg/L	-3 yrs mg/L	Base Model mg/L	+3 yrs mg/L	+6 yrs mg/L	-6 yrs mg/L	-3 yrs mg/L	Base Model mg/L	+3 yrs mg/L	+6 yrs mg/L
Al	0.072	0.080	0.089	0.098	0.11	1.0	1.3	1.6	1.9	2.2
Sb	0.0041	0.0046	0.0052	0.0057	0.0062	0.0055	0.0064	0.0072	0.0081	0.0089
As	0.0076	0.0082	0.0088	0.0094	0.0099	0.27	0.37	0.47	0.57	0.67
Ba	0.0056	0.0057	0.0057	0.0058	0.0059	0.47	0.47	0.47	0.47	0.47
Be	0.00057	0.00061	0.00065	0.00070	0.00074	0.00064	0.00070	0.00075	0.00081	0.00087
B	0.0052	0.0052	0.0052	0.0053	0.0053	0.0052	0.0052	0.0053	0.0053	0.0053
Cd	0.00039	0.00044	0.00050	0.00056	0.00062	0.011	0.015	0.019	0.023	0.027
Ca	871	871	872	872	872	896	896	897	897	898
Cl	229	229	229	229	229	1,731	1,731	1,731	1,730	1,730
Cr	0.00066	0.00073	0.00081	0.00089	0.00097	0.0092	0.012	0.016	0.019	0.022
Co	0.077	0.077	0.077	0.077	0.077	0.084	0.087	0.090	0.093	0.095
Cu	0.0056	0.0062	0.0068	0.0075	0.0081	0.32	0.44	0.55	0.67	0.79
F	0.0036	0.0039	0.0041	0.0044	0.0047	0.040	0.048	0.056	0.065	0.073
Fe	0.16	0.20	0.25	0.29	0.33	25	34	43	52	61
Pb	0.0043	0.0046	0.0049	0.0052	0.0055	0.25	0.33	0.41	0.49	0.57
Mg	0.66	0.73	0.80	0.87	0.94	2.2	2.8	3.3	3.8	4.3
Mn	0.0063	0.0070	0.0078	0.0086	0.0094	0.088	0.12	0.15	0.18	0.21
Hg	0.000088	0.000090	0.000093	0.000096	0.000098	0.00020	0.00025	0.00029	0.00033	0.00038
Ni	0.0017	0.0019	0.0022	0.0024	0.0027	0.028	0.029	0.029	0.030	0.031
K	148	148	148	148	148	148	148	148	148	148
Se	0.030	0.030	0.030	0.030	0.030	0.035	0.036	0.037	0.038	0.039
Ag	0.00074	0.00082	0.00090	0.0010	0.0011	0.0022	0.0024	0.0025	0.0026	0.0028
Na	62	62	62	62	62	62	62	62	62	62
SO_4	1,144	1,145	1,145	1,145	1,145	1,275	1,301	1,327	1,353	1,379
Zn	0.061	0.067	0.073	0.079	0.085	3.9	5.3	6.7	8.2	9.6

Prepared by: DDS
Checked by: CK

7.5 Pyritic Paste Backfill NP

As discussed in Section 2.2.3, the average NP of the two pyritic paste backfill samples that exhibited anomalously low NP was used in the base model, i.e 30.3 kg $CaCO_3$ eq./tonne. Table 2.6 shows that value is equal to the modified Sobek NP that would be contributed by the cement amendment alone, at an addition rate of 5% (wt/wt). Although the base case assumption is thought to be conservative, the sensitivity to changes in the assumed NP was investigated, to cover the possibility of significant variation in NP availability and/or cement content.

Technical advisers to the Wisconsin Department of Natural Resources stated that an available NP of 20 kg $CaCO_3$ eq./tonne would be a reasonable worst case assumption. The average modified Sobek-NP of the humidity cell samples is 45.4 kg $CaCO_3$ eq./tonne. Table 7.11 shows the effect of these NP values on the estimated volume of acidic paste backfill. The effect is inverse and linear, i.e. a 10% decrease in the assumed NP leads to a 10% increase in the predicted volume of acidic backfill.

However, changes in neutralization potential do not significantly change the estimated volume averaged concentrations in either the pyritic paste backfill or the overall reflooded mine. The reason is that changes to the volume of acidic backfill are counteracted by changes to the solute concentrations within the backfill. In other words, although a lower NP would lead to a greater volume of acidic backfill, the amount of solute within each unit volume of that acidic backfill would be proportionately lower.

This conclusion is strictly true for most solutes, for which changes to the assumed NP result in absolutely no changes in predicted concentrations. The exceptions are solutes that are solubility limited in the acidic backfill, namely Ba, Ca, Pb, and Ag. They remain at their solubility-determined concentration even when the volume of acidic material changes. However, as discussed in Section 7.2 and 7.3, the contribution of those solutes from the acidic backfill is minimal relative to the overwhelming volume of unoxidized backfill, and therefore no significant changes in overall concentrations result.

TABLE 7.11

Volumes of Acidic Paste Backfill for Changes in Pyritic Paste Backfill NP

Case	BEJ Volume of Acidic Backfill (ft^3)	UB Volume of Acidic Backfill (ft^3)
20 kg $CaCO_3$ eq/tonne	$2.28x10^3$	$6.28x10^5$
Base Case (30.3 kg $CaCO_3$ eq/tonne)	$1.50x10^3$	$4.15x10^5$
45.4 kg $CaCO_3$ eq/tonne	$1.00x10^3$	$2.76x10^5$

Prepared by: DDS
Checked by: CK

7.6 Wall Rock Surface Area

As discussed in Section 3.3.3, the reactive surface area of the wall rock is estimated by multiplying the nominal surface area by a factor of 30. That factor was selected from the values provided by Morin and Hutt (1995), based on tests in open pits. The low end of the range was selected because underground blasting techniques generally lead to less fracturing than would be expected in open pit mines.

Tables 7.12 and 7.13 show the results when the nominal surface area is increased by different factors. In addition to the base case, factors of 15 and 45 were tested. Solute concentrations from the wall rocks vary directly with the surface area, and therefore increase or decrease in proportion with the surface area factor. The same relationship is evident in the volume averaged concentrations for the entire reflooded mine (Table 7.13), with the exception of solutes such as Ca, Na, K, and SO_4 that are predominantly released by the paste backfill.

TABLE 7.12

Effect of Wall Rock Fracture Factor on Solute Concentrations from Wall Rock

Parameter	Best Engineering Judgment (BEJ)			Upper Bound (UB)		
Fracture Factor	Low End	Base Model	High End	Low End	Base Model	High End
	15	30	45	15	30	45
Al (mg/L)	0.40	0.80	1.2	1.3	2.6	3.9
Sb (mg/L)	0.026	0.052	0.078	0.025	0.051	0.076
As (mg/L)	0.012	0.023	0.035	0.013	0.026	0.040
Ba (mg/L)	0.0033	0.0065	0.0098	0.0049	0.010	0.015
Be (mg/L)	0.0022	0.0043	0.0065	0.0022	0.0044	0.0066
B (mg/L)	0.0021	0.0041	0.0062	0.0021	0.0041	0.0062
Cd (mg/L)	0.0022	0.0044	0.0065	0.0022	0.0044	0.0066
Ca (mg/L)	8.4	17	25	8.4	17	25
Cl (mg/L)	1.0	2.0	3.0	1.0	2.0	3.0
Cr (mg/L)	0.0032	0.0065	0.0097	0.0035	0.0071	0.011
Co (mg/L)	0.0037	0.0075	0.011	0.0040	0.0080	0.012
Cu (mg/L)	0.011	0.022	0.033	0.025	0.051	0.076
F (mg/L)	0.013	0.025	0.038	0.013	0.025	0.038
Fe (mg/L)	0.54	1.1	1.6	1.9	3.8	5.7
Pb (mg/L)	0.014	0.028	0.042	0.015	0.030	0.045
Mg (mg/L)	3.3	6.7	10	4.3	8.6	13
Mn (mg/L)	0.034	0.068	0.10	0.063	0.13	0.19
Hg (mg/L)	0.00012	0.00024	0.00035	0.00012	0.00024	0.00035
Ni (mg/L)	0.013	0.026	0.039	0.013	0.026	0.039
K (mg/L)	1.5	3.0	4.6	2.1	4.3	6.4
Se (mg/L)	0.0027	0.0054	0.0081	0.0028	0.0056	0.0084
Ag (mg/L)	0.0042	0.0084	0.013	0.0039	0.0078	0.012
Na (mg/L)	0.88	1.8	2.6	2.2	4.4	6.7
SO_4 (mg/L)	8.1	16	24	10	21	31
Zn (mg/L)	0.024	0.049	0.073	0.032	0.065	0.097

Prepared by: DDS
Checked by: CK

TABLE 7.13
Effect of Wall Rock Fracture Factor on Volume Averaged Solute Concentrations in the Reflooded Mine

Parameter	Best Engineering Judgment (BEJ)			Upper Bound (UB)		
	Low End	Base Model	High End	Low End	Base Model	High End
Fracture Factor	15	30	45	15	30	45
Al (mg/L)	0.058	0.089	0.12	1.5	1.6	1.7
Sb (mg/L)	0.0032	0.0052	0.0072	0.0052	0.0072	0.0092
As (mg/L)	0.0079	0.0088	0.0097	0.47	0.47	0.47
Ba (mg/L)	0.0055	0.0057	0.0060	0.47	0.47	0.47
Be (mg/L)	0.00048	0.00065	0.00082	0.00058	0.00075	0.00092
B (mg/L)	0.0051	0.0052	0.0054	0.0051	0.0053	0.0054
Cd (mg/L)	0.00032	0.00050	0.00068	0.019	0.019	0.019
Ca (mg/L)	871	872	872	896	897	897
Cl (mg/L)	229	229	229	1,731	1,731	1,731
Cr (mg/L)	0.00056	0.00081	0.0011	0.015	0.016	0.016
Co (mg/L)	0.077	0.077	0.078	0.090	0.090	0.090
Cu (mg/L)	0.0059	0.0068	0.0078	0.55	0.55	0.56
F (mg/L)	0.0032	0.0041	0.0051	0.055	0.056	0.057
Fe (mg/L)	0.20	0.25	0.29	43	43	43
Pb (mg/L)	0.0038	0.0049	0.0060	0.41	0.41	0.41
Mg (mg/L)	0.54	0.80	1.1	2.9	3.3	3.6
Mn (mg/L)	0.0052	0.0078	0.010	0.14	0.15	0.15
Hg (mg/L)	0.000084	0.000093	0.00010	0.00028	0.00029	0.00030
Ni (mg/L)	0.0012	0.0022	0.0032	0.028	0.029	0.030
K (mg/L)	148	148	148	148	148	148
Se (mg/L)	0.029	0.030	0.030	0.037	0.037	0.037
Ag (mg/L)	0.00058	0.00090	0.0012	0.0022	0.0025	0.0028
Na (mg/L)	62	62	62	62	62	62
SO_4 (mg/L)	1,144	1,145	1,145	1,326	1,327	1,328
Zn (mg/L)	0.068	0.073	0.078	6.7	6.7	6.7

Prepared by: DDS
Checked by: CK

7.7 Early Shutdown

To examine the sensitivity of the model predictions to the assumed duration of mining, an early shutdown case was modeled. The early shutdown case assumed that only the zinc ore would be mined. Under that assumption, only approximately 30 million tons of ore will be mined, and the duration of mining would be 16 years. Table 7.14 presents the BEJ and UB estimates of solute concentrations for the early shutdown case.

Table 7.15 compares the volume-averaged solute concentrations in the reflooded mine under the cases of 16 and 28 years of mining. Due to the reduction in possible oxidation time, solute concentrations after 16 years would generally be slightly lower than the concentrations after 28 years. The unoxidized backfill, which is the same in both cases, dominates the volume-averaged concentrations. A somewhat greater difference is seen in the UB predictions for solutes that are at high concentrations in the acidic backfill.

TABLE 7.14
Estimated Solute Concentrations for Mine Shutdown at Year 16

Parameter		BEJ				UB			
		Paste Backfill	Wall Rocks	Crown Pillar	Reflooded Mine	Paste Backfill	Wall Rocks	Crown Pillar	Reflooded Mine
Al	(mg/L)	0.028	0.47	0.0055	0.090	1.0	1.5	0.072	1.1
Sb	(mg/L)	0.0013	0.030	0.0033	0.0054	0.0030	0.030	0.0033	0.0068
As	(mg/L)	0.0070	0.014	0.00036	0.0078	0.33	0.016	0.00048	0.28
Ba	(mg/L)	0.0057	0.0038	0.00044	0.0054	0.52	0.0058	0.00053	0.43
Be	(mg/L)	0.00035	0.0026	0.00034	0.00066	0.00043	0.0026	0.00035	0.00074
B	(mg/L)	0.0054	0.0024	-	0.0049	0.0054	0.0024	-	0.0049
Cd	(mg/L)	0.00013	0.0026	0.0067	0.00052	0.013	0.0026	0.028	0.012
Ca	(mg/L)	953	10	1.9	802	980	10	2.0	824
Cl	(mg/L)	251	1.2	-	211	1,896	1.2	-	1,592
Cr	(mg/L)	0.00032	0.0038	0.00034	0.00082	0.011	0.0042	0.00035	0.010
Co	(mg/L)	0.084	0.0044	0.0057	0.071	0.093	0.0048	0.014	0.079
Cu	(mg/L)	0.0046	0.013	0.066	0.0062	0.39	0.030	0.55	0.33
F	(mg/L)	0.0024	0.015	-	0.0041	0.045	0.015	-	0.040
Fe	(mg/L)	0.12	0.63	0.024	0.19	30	2.2	0.30	26
Pb	(mg/L)	0.0030	0.017	0.0034	0.0049	0.30	0.018	0.0069	0.25
Mg	(mg/L)	0.30	3.9	0.99	0.82	2.1	5.1	1.2	2.5
Mn	(mg/L)	0.0026	0.040	0.020	0.0080	0.10	0.075	0.029	0.094
Hg	(mg/L)	0.000082	0.00014	0.000014	0.000089	0.00022	0.00014	0.000015	0.00021
Ni	(mg/L)	0.00016	0.015	0.0016	0.0023	0.029	0.015	0.0029	0.026
K	(mg/L)	162	1.8	0.27	136	162	2.5	0.25	136
Se	(mg/L)	0.032	0.0032	0.0017	0.027	0.038	0.0033	0.0022	0.032
Ag	(mg/L)	0.00028	0.0050	0.0011	0.00095	0.0020	0.0046	0.0010	0.0023
Na	(mg/L)	67	1.0	0.15	57	68	2.6	0.16	57
SO_4	(mg/L)	1,252	10	8.2	1,053	1,405	12	17	1,181
Zn	(mg/L)	0.060	0.026	1.4	0.065	4.7	0.036	5.2	4.0

Prepared by: DDS
Checked by: CK

TABLE 7.15
Effects of 16 Year and Base Case Mine Life on Volume Averaged Solute Concentrations in the Reflooded Mine

Parameter	Best Engineering Judgement (BEJ)		Upper Bound (UB)	
Mine Life	16 yrs mg/L	Base Case mg/L	16 yrs mg/L	Base Case mg/L
Al	0.090	0.089	1.1	1.6
Sb	0.0054	0.0052	0.0068	0.0072
As	0.0078	0.0088	0.28	0.47
Ba	0.0054	0.0057	0.43	0.47
Be	0.00066	0.00065	0.00074	0.00075
B	0.0049	0.0052	0.0049	0.0053
Cd	0.00052	0.00050	0.012	0.019
Ca	802	872	824	897
Cl	211	229	1,592	1,731
Cr	0.00082	0.00081	0.010	0.016
Co	0.071	0.077	0.079	0.090
Cu	0.0062	0.0068	0.33	0.55
F	0.0041	0.0041	0.040	0.056
Fe	0.19	0.25	26	43
Pb	0.0049	0.0049	0.25	0.41
Mg	0.82	0.80	2.5	3.3
Mn	0.0080	0.0078	0.094	0.15
Hg	0.000089	0.000093	0.00021	0.00029
Ni	0.0023	0.0022	0.026	0.029
K	136	148	136	148
Se	0.027	0.030	0.032	0.037
Ag	0.00095	0.00090	0.0023	0.0025
Na	57	62	57	62
SO_4	1,053	1,145	1,181	1,327
Zn	0.065	0.073	4.0	6.7

Prepared by: DDS
Checked by: CK

8. SOURCE CONTROL MEASURES

8.1 Approach

Further sensitivity analyses were carried out to investigate the effects of the source control measures under consideration by NMC. Parameters in the solute concentration calculations were varied to represent the effects of each control measure. The results are discussed below.

8.2 Oxygen Barriers

The amount of oxygen entering the backfilled stopes can be reduced by installing oxygen barriers within the cross-cuts. Barriers could be physical, such as bulkheads, or chemical (for example, oxygen consuming).

Tables 8.1 and 8.2 show the effects on volume averaged solute concentrations if the barriers were 90% - 99.99% effective, *i.e.* if they reduce oxygen entry to 10% - 0.01% of the base case values. The effect on the BEJ solute concentration estimates is minimal, because of the dominance of the unoxidized backfill. In the UB case however, the oxygen barriers are predicted to significantly decrease the volume averaged concentrations of As, Cd, Cr, Cu, Fe, Pb, Mn and Zn.

8.3 Acidic Backfill Removal

The removal for of-site disposal of acidic paste backfill, immediately prior to reflooding of the mine, will have the same effect as the use of oxygen barriers. For example, removal of 98 % of the paste backfill would lead to the same solute concentrations as decreasing the oxygen flux by 98%. Therefore the effects of backfill removal on solute concentrations can be seen by reference to Tables 8.1 and 8.2.

8.4 Wall Rock Flushing

Another source control measure that could be implemented immediately prior to reflooding of the mine would be to remove oxidation products from the wall rock, for example by flushing with high pressure water. Table 8.3 shows the effects of such a measure, under the assumption that 50% - 90% of the wall rock solutes could be removed. As expected, flushing would reduce the concentration of solutes that originate primarily from the wall rock, but would have no effect on solute concentrations contributed by the paste backfill.

TABLE 8.1
Effect of Oxygen Barriers on Volume Averaged Solute Concentrations in the Pyritic Paste Backfill

Parameter	Best Engineering Judgement (BEJ)				Upper Bound (UB)			
O_2 Flux (% of Base)	0.01%	1 %	10%	Base Case[1]	0.01%	1 %	10%	Base Case[2]
	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Al	0.025	0.025	0.025	0.030	0.053	0.068	0.20	1.5
Sb	0.0013	0.0013	0.0013	0.0013	0.0019	0.0019	0.0021	0.0036
As	0.0058	0.0058	0.0060	0.0076	0.0077	0.013	0.058	0.52
Ba	0.0057	0.0057	0.0057	0.0057	0.52	0.52	0.52	0.52
Be	0.00035	0.00035	0.00035	0.00035	0.00039	0.00039	0.00040	0.00045
B	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054	0.0054
Cd	0.000078	0.000079	0.000085	0.00015	0.000091	0.00029	0.0021	0.020
Ca	953	953	953	953	979	979	980	981
Cl	251	251	251	251	1,896	1,896	1,896	1,895
Cr	0.00028	0.00028	0.00028	0.00034	0.00045	0.00061	0.0021	0.017
Co	0.084	0.084	0.084	0.084	0.084	0.084	0.085	0.098
Cu	0.0032	0.0032	0.0034	0.0053	0.0040	0.0099	0.063	0.60
F	0.0023	0.0023	0.0023	0.0024	0.018	0.018	0.022	0.060
Fe	0.0086	0.010	0.026	0.18	0.045	0.51	4.7	47
Pb	0.0030	0.0030	0.0030	0.0030	0.039	0.043	0.079	0.44
Mg	0.30	0.30	0.30	0.30	0.63	0.65	0.85	2.9
Mn	0.0022	0.0022	0.0023	0.0028	0.0013	0.0028	0.016	0.15
Hg	0.000081	0.000081	0.000081	0.000082	0.000085	0.000087	0.00011	0.00030
Ni	0.00016	0.00016	0.00016	0.00017	0.027	0.027	0.028	0.030
K	162	162	162	162	162	162	162	162
Se	0.032	0.032	0.032	0.032	0.034	0.034	0.035	0.040
Ag	0.00028	0.00028	0.00028	0.00028	0.0018	0.0018	0.0018	0.0021
Na	67	67	67	67	67	67	68	68
SO_4	1,252	1,252	1,252	1,252	1,320	1,321	1,333	1,451
Zn	0.043	0.043	0.046	0.070	0.045	0.12	0.77	7.3

[1] Table 6.1 BEJ estimate of solute concentrations in reflooded mine without source control
[2] Table 6.2 UB estimate of solute concentrations in reflooded mine without source control

Prepared by: DDS
Checked by: CK

TABLE 8.2
Effect of Oxygen Barriers on Volume Averaged Solute Concentrations in the Reflooded Mine

Parameter	Best Engineering Judgement (BEJ)				Upper Bound (UB)			
O_2 Flux (% of Base)	0.01%	1 %	10%	Base Case[1]	0.01%	1 %	10%	Base Case[2]
	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Al	0.084	0.084	0.085	0.089	0.25	0.26	0.38	1.6
Sb	0.0052	0.0052	0.0052	0.0052	0.0057	0.0057	0.0058	0.0072
As	0.0071	0.0071	0.0073	0.0088	0.0091	0.014	0.055	0.47
Ba	0.0057	0.0057	0.0057	0.0057	0.47	0.47	0.47	0.47
Be	0.00065	0.00065	0.00065	0.00065	0.00070	0.00070	0.00070	0.00075
B	0.0052	0.0052	0.0052	0.0052	0.0052	0.0052	0.0052	0.0053
Cd	0.00043	0.00044	0.00044	0.00050	0.00054	0.00072	0.0024	0.019
Ca	872	872	872	872	896	896	896	897
Cl	229	229	229	229	1,732	1,732	1,732	1,731
Cr	0.00076	0.00076	0.00076	0.00081	0.00096	0.0011	0.0024	0.016
Co	0.077	0.077	0.077	0.077	0.077	0.078	0.079	0.090
Cu	0.0049	0.0049	0.0051	0.0068	0.0099	0.015	0.064	0.55
F	0.0040	0.0040	0.0040	0.0041	0.018	0.019	0.022	0.056
Fe	0.090	0.092	0.11	0.25	0.34	0.76	4.6	43
Pb	0.0049	0.0049	0.0049	0.0049	0.038	0.041	0.075	0.41
Mg	0.79	0.79	0.79	0.80	1.2	1.3	1.4	3.3
Mn	0.0073	0.0073	0.0074	0.0078	0.011	0.012	0.025	0.15
Hg	0.000092	0.000092	0.000092	0.000093	0.000096	0.00010	0.00012	0.00029
Ni	0.0022	0.0022	0.0022	0.0022	0.027	0.027	0.027	0.029
K	148	148	148	148	148	148	148	148
Se	0.030	0.030	0.030	0.030	0.032	0.032	0.032	0.037
Ag	0.00090	0.00090	0.00090	0.00090	0.0022	0.0022	0.0023	0.0025
Na	62	62	62	62	62	62	62	62
SO_4	1,144	1,144	1,144	1,145	1,207	1,208	1,219	1,327
Zn	0.049	0.049	0.051	0.073	0.067	0.13	0.73	6.7

[1] Table 6.1 BEJ estimate of solute concentrations in reflooded mine without source control
[2] Table 6.2 UB estimate of solute concentrations in reflooded mine without source control

Prepared by: DDS
Checked by: CK

TABLE 8.3
Effects of Wall Rock Flushing on Volume Averaged Concentrations in the Reflooded Mine

Parameter	Best Engineering Judgement (BEJ)			Upper Bound (UB)		
Oxidation Products Remaining	10 % Remaining	50 % Remaining	Base Case[1]	10 % Remaining	50 % Remaining	Base Case[2]
	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Al	0.034	0.058	0.089	1.4	1.5	1.6
Sb	0.0016	0.0032	0.0052	0.0037	0.0053	0.0072
As	0.0072	0.0079	0.0088	0.47	0.47	0.47
Ba	0.0053	0.0055	0.0057	0.47	0.47	0.47
Be	0.00035	0.00048	0.00065	0.00045	0.00059	0.00075
B	0.0050	0.0051	0.0052	0.0050	0.0051	0.0053
Cd	0.00020	0.00033	0.00050	0.019	0.019	0.019
Ca	870	871	872	896	896	897
Cl	229	229	229	1,731	1,731	1,731
Cr	0.00036	0.00056	0.00081	0.015	0.015	0.016
Co	0.077	0.077	0.077	0.089	0.090	0.090
Cu	0.0053	0.0060	0.0068	0.55	0.55	0.55
F	0.0024	0.0032	0.0041	0.055	0.055	0.056
Fe	0.17	0.20	0.25	43	43	43
Pb	0.0030	0.0038	0.0049	0.41	0.41	0.41
Mg	0.33	0.54	0.80	2.7	2.9	3.3
Mn	0.0031	0.0052	0.0078	0.14	0.14	0.15
Hg	0.000077	0.000084	0.000093	0.00027	0.00028	0.00029
Ni	0.00036	0.0012	0.0022	0.027	0.028	0.029
K	148	148	148	148	148	148
Se	0.029	0.029	0.030	0.037	0.037	0.037
Ag	0.00032	0.00058	0.00090	0.0020	0.0022	0.0025
Na	62	62	62	62	62	62
SO_4	1,144	1,144	1,145	1,326	1,326	1,327
Zn	0.070	0.071	0.073	6.7	6.7	6.7

[1] Table 6.1 BEJ estimate of solute concentrations in reflooded mine without source control
[2] Table 6.2 UB estimate of solute concentrations in reflooded mine without source control

Prepared by: DDS
Checked by: CK

9. CONCLUSIONS

This report has presented estimates of solute concentrations associated with pyritic paste backfill, wall rock, crown pillar rock, and combustion by-products and blasting residues in the reflooded Crandon Mine. The methods used to derive the estimates made use of data from the extensive waste characterization program and the current mine design, supplemented (where appropriate) by conservative assumptions from literature sources.

The Best Engineering Judgement (BEJ) estimates of the solute concentrations associated with each source, in the absence of source control measures, are presented in Table 6.1. Upper Bound (UB) estimates, again for each source without controls, are presented in Table 6.2. The major difference between the two sets of estimates is in the volume of acidic backfill. The very conservative assumptions used to derive the UB estimates of oxygen transport into the paste backfill lead to a high estimate of the volume of backfill that could become acidic.

Sensitivity analyses presented in Section 7 show that the BEJ and UB solute concentration estimates for each source vary predictably with changes in input assumptions. When volume-averaged concentrations are considered, it becomes clear that the BEJ estimates are insensitive to most changes in input parameters because the large volume of unoxidized paste backfill dominates all other sources. The volume-weighted average UB estimates are more sensitive to changes in input parameters, especially those that affect the (more significant) volume of acidic backfill.

Additional sensitivity analyses presented in Section 8 show the potential benefits of various source control measures. Measures to restrict oxygen access to the backfill, and measures to remove any backfill that might become acidic have strong effects on the volume-averaged UB solute concentrations. NMC should consider such measures as means to keep actual solute concentrations at or below the BEJ estimates. Measures to remove solutes released from the hanging wall rock have less effect on the overall volume-weighted average concentrations, but may be worthy of further consideration if the movement of solutes out of the hanging wall workings is more rapid than from other sources.

This report, **1UF001.01, Reflooded Mine Source Concentrations – Crandon Project, Wisconsin**, has been prepared by:

STEFFEN, ROBERTSON AND KIRSTEN (CANADA) INC.

Diana Sollner, P.Eng.
Environmental Engineer

Daryl Hockley, P.Eng.
Principal

10. REFERENCES

Erickson, A.J. Jr., 1996. Crandon Mining Company. Memo to file Re: Reserves/Permanent Crown Pillar, dated November 1996.

Erickson, A.J. Jr. and R. Cote, 1996. Geological Summary - Crandon Deposit. Institute on Lake Superior Geology Proceedings, 42nd Annual Meeting, Cable, WI, May 24, 1996.

Ferguson and Leask, 1988. The Export of Nutrients from Surface Coal Mines. Environment Canada Regional Program Report 87-12, March 1988.

Foth and Van Dyke, 1999/2000. Crandon Project TMA Groundwater Quality Performance Evaluation. Originally issued June 1999. Updated March 2000.

Foth and Van Dyke, 1995/1998a. Crandon Project Mine Permit Application. Originally issued May 1995. Updated December 1998.

Foth and Van Dyke, 1995/1998b. Appendix 3.5-33. Environmental Impact Report, Crandon Project. Originally issued May 1995. Updated December 1998.

Foth and Van Dyke, 1995/1998c. Air Pollution Control Permit Application. Originally issued August 1995. Updated December 1998.

GeoTrans, 1996. Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Zinc and Copper Mine near Crandon, Wisconsin.

Golder Paste Technology Ltd., 1998. Technical memorandum to NMC: Hydraulic Conductivity Testing of Paste Samples. Project 981-9905, Dated June 1, 1998.

Morin, K.A., 1990. Acid Drainage from Mine Walls: The Main Zone Pit at Equity Silver Mines. British Columbia Acid Mine Drainage Task Force.

Morin, K.A. and Hutt, N.M., 1995. A Technique for Predicting Water Chemistry in Open Pit and Underground Mines. In: Sudbury '95 Mining and the Environment Conference Proceedings.

Nicolet Minerals Company, 2000. Response to WDNR Comments on TMA GWQPE, March 31, 2000 (Incorporated into Crandon Project EIR as part of EIR update for Items 337-344).

Ritchie, A.J.M., 1994. Sulfide Oxidation Mechanism-Controls and Rates of Oxygen Transport. In: The Environmental Geochemistry of Sulfide Mine Wastes. D.W. Blowes and J.L. Jambor, eds. CANMET, 1994.

Stachulak, J.S., E.D. Dainty, and M.J. McPherson, November 1991. "The Phenomenon of Dust Deposition in Exhaust Ventilation Shafts at Inco Mines". *Journal of the Mine Ventilation Society of South Africa.*

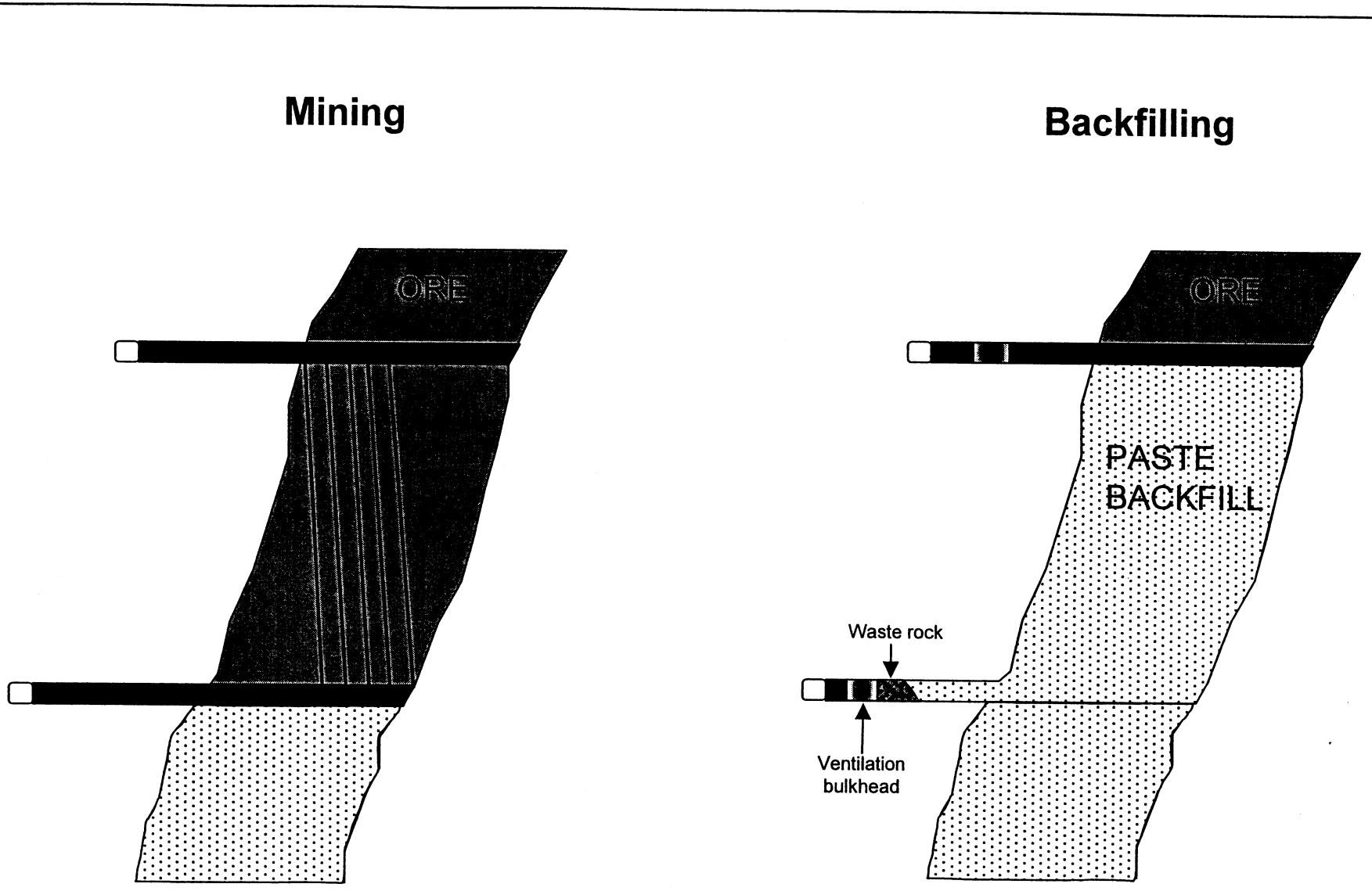
Steffen Robertson and Kirsten, 1997. Reflooded Mine Source Term, Crandon Mine – Final Report. Report F107107/2 to Foth & Van Dyke, March 1997.

U.S. Environmental Protection Agency, April 1989. Estimating Air Toxics Emissions from Coal and Oil Combustion Sources. Radian Corporation. EPA/450/2-89/001.

U.S. Environmental Protection Agency, September 1993. Locating and Estimating Air Toxic Emissions from Medical Waste Incinerators. Radian Corporation. USEPA Contract No. 68D90054. DCN #96/275/026/42/02.

Watts, W.F. and R.W. Waytulonis, 1991. "Why is Diesel Particulate in Mines an Issue and How Can It Be Controlled?" U.S. Department of the Interior, Bureau of Mines. Proceedings of the Symposium on Environmental Management for the 1990s.

FIGURES



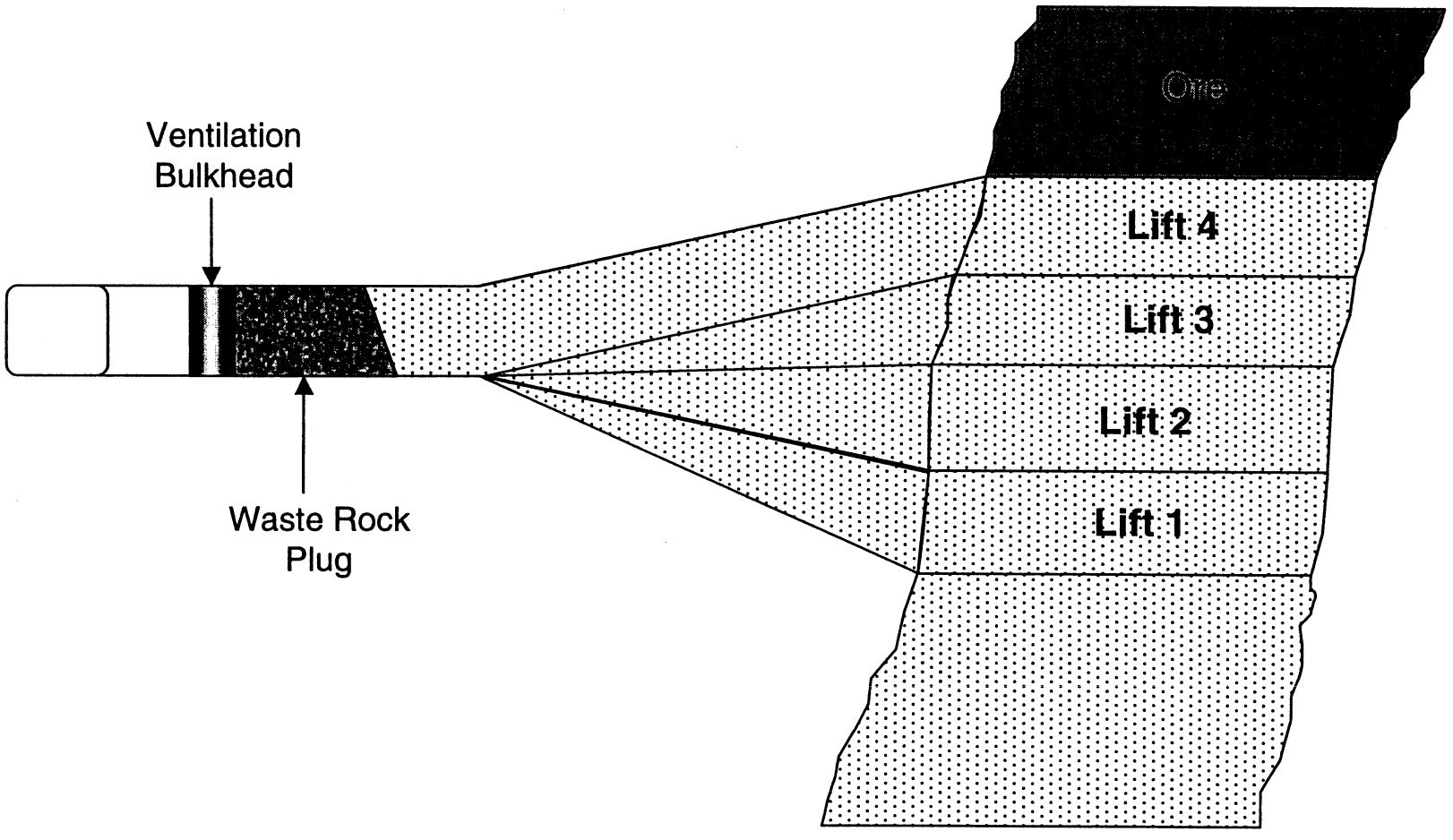
SRK Consulting
Engineers and Scientists

Foth & Van Dyke

Nicolet Minerals Company
Crandon Project

Mine & Backfill Sequence of Blasthole Open Stopes

PROJECT:	DATE:	APPROVED:	FIGURE:
1UF001.01	August 2000		2.1



SRK Consulting
Engineers and Scientists

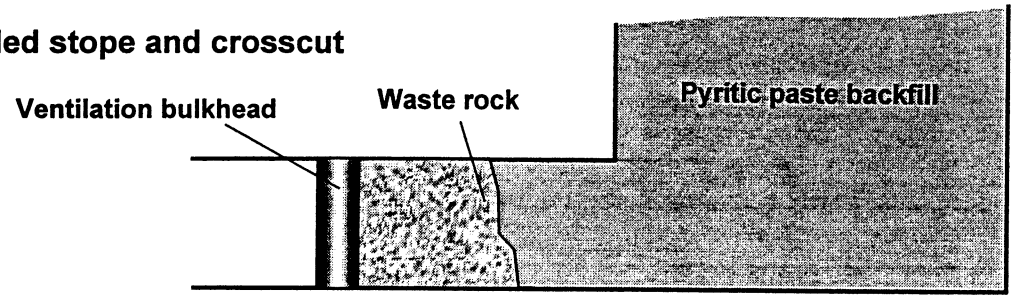
Foth & Van Dyke

Nicolet Minerals Company
Crandon Project

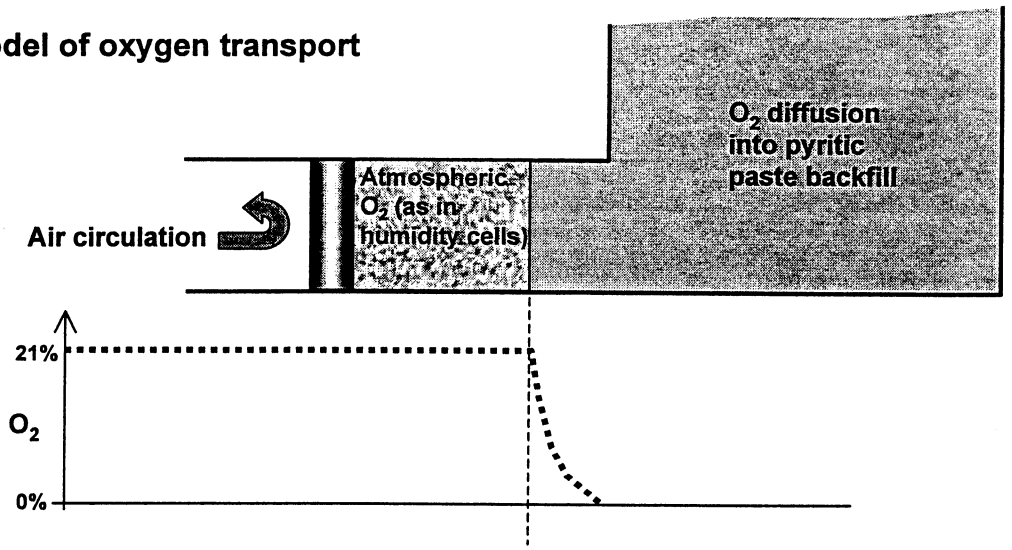
Mine & Backfill Sequence of Cut & Fill Stopes

PROJECT:	DATE:	APPROVED:	FIGURE:
1UF001.01	November 2000		2.2

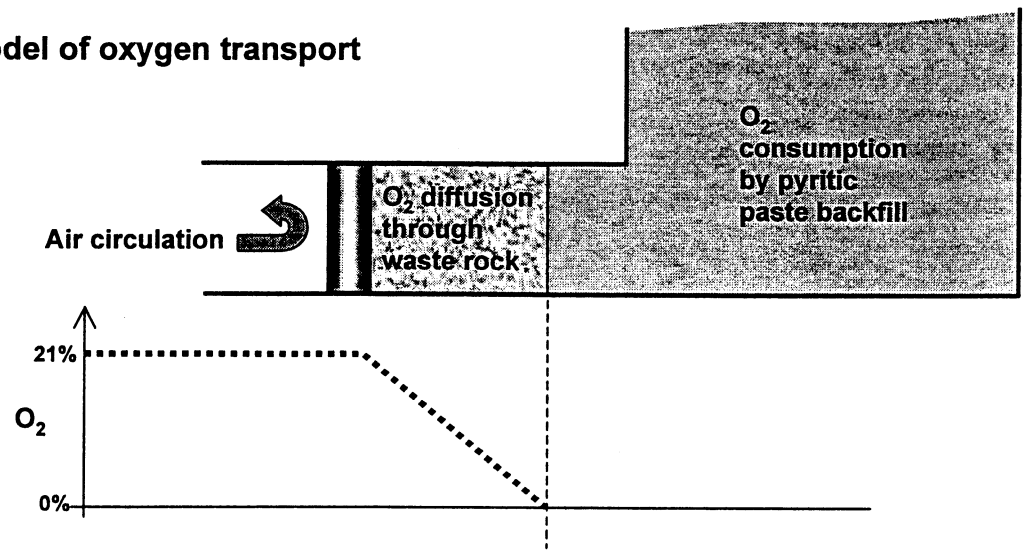
(a) Backfilled stope and crosscut



(b) BEJ Model of oxygen transport



(c) UB Model of oxygen transport



SRK Consulting Engineers and Scientists	Nicolet Minerals Company Crandon Project			
Foth & Van Dyke	Oxygen Transport into Backfill Stopes			
	PROJECT: 1UF001.01	DATE: November 2000	APPROVED:	FIGURE: 2.3

ATTACHMENT A

DESCRIPTION OF SOLUTE CONCENTRATION CALCULATIONS

ATTACHMENT A
DESCRIPTION OF SOLUTE CONCENTRATION CALCULATIONS

The following quantities were used throughout the calculations of O_2 transport:

- Dimensions of cross cuts allowing O_2 into stopes is 13 ft. x 16 ft. (3.96 m x 4.88 m)
- Distance between bulkhead and stope is 40 ft = 12.192 m.
- O_2 concentration in air is 8.9 mol/m^3.
- Pyritic paste backfill dry bulk density is 0.073 tons/ft^3 (2.332 tonnes/m^3); porosity is 0.4.

1. Pyritic Paste Backfill (PPB)

1a. BEJ estimates of O_2 Transport

The BEJ O_2 transport was estimated from the humidity cell tests of pyritic paste backfill, i.e. assuming that diffusion into the backfill itself is the rate limiting step (see Figure 2.3b):

O_2 per stope = O_2 flux (mol/m^2/s) * cross cut cross-sectional area (m^2) * exposure time (s)
= 135 mol (blasthole stope)
= 135 mol (cut & fill stope)

Where:
O_2 flux = $1.59 * 10^{-8}$ mol/m^2/s (from the humidity cell data)
Cross-sectional area = 3.96 m * 4.88 m
Exposure time = 14 yrs

1b. UB Estimates of O_2 Transport

In the UB case, O_2 transport is assumed to be by diffusion from the inside of the bulkhead and through the waste rock to the backfill face (see Figure 2.3c). The flux of O_2 is given by Fick's Law:

$$\text{Flux } O_2 = \frac{D^*C_o}{L}$$

Where:
D = diffusion coefficient = $6*10^{-6}$ m^2/s (Ritchie, 1994)
C_o = O_2 concentration at bulkhead = 8.9 mol/m^3
L = distance between bulkhead and stope = 12.192 m

The rate of O_2 entry to each stope is given by:

O_2 entry (mol/s)= O_2 Flux * cross-sectional area of cross-cut
$= 4.38*10^{-6} * 3.96 * 4.88$
$= 8.46*10^{-5}$ mol/s

The average stope will be exposed to O_2 for 14 years (both blasthole and cut & fill stopes) prior to reflooding. Therefore:

O_2 per stope = 8.46×10^{-5} mol/s * 14 years
= 3.73×10^{4} mol (blasthole stope)

Similarly,

O_2 per stope = 3.73×10^{4} mol (cut & fill stope)

1c. Volume of Acidic Paste Backfill

Pyrite oxidizes to iron hydroxide via the following reaction:

$$FeS_2 + 15/4O_2 + 7/2H_2O \rightarrow Fe(OH)_{3(s)} + 2SO_4^{2-} + 4H^+$$

Acidity is neutralized by calcite:

$$2CaCO_3 + 2H_2SO_4 \rightarrow 2CaSO_4 + 2CO_{2(g)} + 2H_2O$$

Each mole of O_2 produces 8/15 moles of $CaCO_3$ equivalent acidity.

Therefore, in each stope:

Acidity Produced (kg CaCO3 eq.) = O_2 per stope * 8/15 * 0.1 kg $CaCO_3$/mol $CaCO_3$

Assuming an average available NP of 30.3 kg $CaCO_3$ eq./tonne (see Section 2.2.3), the maximum mass of acidic tailings that could be produced in each stope is given by:

$$\text{Mass of Acidic Tailings} = \frac{\text{Acidity Produced}}{\text{NP concentration}}$$

Therefore, under the BEJ oxygen transport conditions the volume of acidic tailings is:

$$\frac{135 \text{ mol } O_2/\text{stope} \times 8/15 \text{ mol } CaCO_3/\text{mol } O_2 \times 0.1 \text{ kg } CaCO_3/\text{mol } CaCO_3}{30.3 \text{ kg } CaCO_3 \text{ eq/tonne} \times 2.332 \text{ tonne/m}^3}$$

$= 0.10$ m^3/stope (blasthole)
$= 0.10$ m^3/stope (cut & fill)

Given a total of 318 blasthole stopes and 98 cut & fill stopes (see Section 2.3.3), the total volume of acidic tailings is:

0.10 m^3/stope * 318 stopes = 33 m^3 (blasthole)
0.10 m^3/stope * 98 stopes = 10 m^3 (cut & fill)
Total = 43 m^3

Under the UB oxygen transport conditions the volume of acidic tailings in each stope is:

$$\frac{3.73\text{x}10^4 \text{ mol } O_2/\text{stope} * 8/15 \text{ mol } CaCO_3/\text{mol } O_2 * 0.1 \text{ kg } CaCO_3/\text{mol } CaCO_3}{30.3 \text{ kg } CaCO_3 \text{ eq/tonne} * 2.332 \text{ tonne/m}^3}$$

$$= 28.2 \text{ m}^3\text{/stope (blasthole)}$$
$$= 28.2 \text{ m}^3\text{/stope (cut \& fill)}$$

$28.2 \text{ m}^3\text{/stope} * 318 \text{ stopes} = 8{,}972 \text{ m}^3$ (blasthole)
$28.2 \text{ m}^3\text{/stope} * 98 \text{ stopes} = 2{,}769 \text{ m}^3$ (cut & fill)
Total = 11,741 m^3

1d. Solute Concentrations in Acidic Backfill

The assumption that oxygen will be distributed to obtain the maximum volume of acidic tailings also simplifies the estimation of solute concentrations. Throughout the acidic tailings, oxidation will have proceeded just sufficiently to consume the assumed available NP of 30.3 kg $CaCO_3$ eq./tonne, i.e.

$$\text{Oxidation} = \frac{30.3 \text{ kg } CaCO_3 \text{ eq./tonne}}{0.1 \text{ kg/mol } CaCO_3} = 0.303 \text{ mol } CaCO_3\text{/kg}$$

From the reactions given in Step 1c, the ratio of $CaCO_3$ consumption to sulfide release is 2:1. Therefore:

$$S_2 \text{ oxidized} = 0.151 \text{ mol } S_2\text{/kg}$$

The relative release rates of other constituents (i.e. the amount of each constituent released per mole of sulfide oxidized) were calculated from the humidity cell tests on pyrite concentrate.

Initial estimates of solute concentrations in porewater were then obtained by dividing the constituent releases into the porosity of the backfill. For example, the initial estimate of sulfate concentrations was given by:

$$\frac{0.151 \text{ mol } S_2/\text{kg} * 2332 \text{ kg/m}^3 * 2 \text{ mol } SO_4/\text{mol } S_2}{0.4 * 1000 \text{ L/m}^3} = 1.765 \text{ mol } SO_4\text{/L}$$

The initial concentration estimates were entered into the MINTEQA2 geochemical model to identify solubility controls. Details of the MINTEQA2 modeling are provided in Attachment C. The lesser of the initial concentrations and any solubility controlled concentration was used as the concentration estimate for each solute.

1e. Volume of Oxidized Neutral PPB

Oxygen will diffuse into a backfilled blasthole stope while the stope above or below is being mined and into the top surface of a backfilled lift in a cut and fill stope while the next lift is being mined. Upper surfaces of stopes will remain open for the time it takes to drill off the next stope above. Once blasting begins, the upper surface will be covered by broken ore, and exposure to oxygen will be intermittent for the approximately 170 days of ore removal. The acidity generated during these periods of exposure would not consume the modified Sobek-NP

in the backfilled tailings. Therefore, the backfilled tailings exposed to oxygen in this time period would remain neutral.

The volume of oxidized neutral PPB was calculated by multiplying the exposed stope surface by the depth of oxygen penetration. Using a depth of penetration of 0.050 m (see Section 2.4.2), the following volumes of oxidized neutral PPB were determined:

Blasthole stopes: 35 m^3/stope
11,095 m^3 for 318 stopes

Cut & fill stopes: 56 m^3/stope
21,915 m^3 for 98 stopes

1f. Solute Concentrations in Oxidized Neutral Backfill

The solute release during the brief oxidation period was estimated directly from solute releases measured in the humidity cell tests on pyritic paste tailings. Initial BEJ estimates of solute concentrations in the oxidized neutral backfill porewater were obtained by dividing the solute release measured over five weeks of the humidity cell tests by the porosity, i.e.

$$\frac{\text{Humidity Cell 5-Week Release (mg/kg)} * 2332\ kg/m^3}{0.4\ m^3/m^3 \times 1000\ L/m^3}$$

eg: Na = 49 mg/kg * 2332/(0.4 * 1000) = 287 mg/L (BEJ)

UB estimates were obtained by the same method, but using the entire 20 week release from the humidity cells.

The initial BEJ estimates were then input to MINTEQA2 to check for solubility constraints. For each solute, the lesser of the initial estimate and any MINTEQA2 solubility controlled concentration was adopted as the final BEJ concentration estimate.

Solubility controls were not considered for the UB estimate, ie all solutes were assumed to remain in solution, in order to be conservative.

1g. Volume of Unoxidized Backfill

The volume of unoxidized backfill was calculated by difference:

$$V_{unox} = V_{ore\ body} - V_{acidic} - V_{ox_neut}$$

Where $V_{orebody}$ = tons Zn ore x specific volume + tons Cu ore x specific volume

$= 3.0 \times 10^7$ tons x 8.8 ft^3/ton + 2.5×10^7 tons x 10.67 ft^3/ton
$= 5.31 \times 10^8\ ft^3$

1h. *Solute Concentrations in Unoxidized Backfill*

Solute concentrations measured in the saturated column tests of pyritic paste backfill were used as the initial estimates of solute concentrations in the unoxidized backfill porewater. The initial estimates were the same for the BEJ and UB cases. MINTEQA2 calculations were completed to identify solubility controls in the BEJ case only (see Attachment C).

1i. *Overall Contribution from Pyritic Paste Backfill*

The overall concentration of each solute in the pyritic paste tailings porewater was estimated by taking a volume-weighted average of concentrations in the acidic, oxidized neutral and unoxidized tailings, i.e.:

$$C_{overall} = \frac{C_{acidic} * V_{acidic} + C_{ox\ neut} * V_{ox\ neut} + C_{unox} * V_{unox}}{V_{acidic} + V_{ox_neut} + V_{unox}}$$

where the C's represent concentrations and the V's represent volumes.

2. Wall Rock

2a. *Solute Concentration in Wall Rock*

Access drifts to the ore body will be located in the hanging wall. The hanging wall is comprised of four rock formations: Lower Mole Lake, Upper Mole Lake, Rice Lake and Skunk Lake. Each formation contains an amount of sulphides, predominantly pyrite. Humidity cell tests were conducted on an average composite to characterize the potential solute release from each formation. The solute concentration in the reflooded mine resulting from each formation was calculated as follows:

$$\text{Solute conc. (mg/L)} = \frac{\text{amount of solute released (kg)}}{\text{vol. of drift } (m^3)} \Rightarrow \text{converted to mg/L}$$

Where:

$$\text{Solute released (kg)} = \text{release rate } (mg/ft^2/yr) * \text{effective area } (ft^2) * \text{exposure time (yr)} \Rightarrow \text{converted to kg}$$

and

$$\text{Release rate } (mg/ft^2/yr) = \frac{\text{cumulative release in one year (mg/kg)}}{\text{specific surface of sample material } (ft^2/kg)}$$

The equation components are defined as follows:

Cumulative release = determined from humidity cell data

Specific surface area of sample material = determined from sieve tests on material used in the humidity cell tests

Effective area = the exposed surface area of drift walls multiplied by fracture factor 30 (to take into account solute released from fracture surfaces).

Exposure time = how long the drift will be open to ventilation prior to mine reflooding. Exposure time was determined from the development schedule.

2b. Overall Contribution from Wall Rock

The overall concentration of each solute in the access drifts was estimated by taking a volume-weighted average of concentrations in each of the four rock formations, i.e.:

$$C_{overall} = \frac{C_{L.Mole} * V_{L.Mole} + C_{U.Mole} * V_{UMole} + C_{Rice} * V_{Rice} + C_{Skunk} * V_{Skunk}}{V_{total\ access\ drift}}$$

where the C's represent concentrations and the V's represent volumes.

3. Unmined Ore

3a. Solute Concentrations in Unmined Stopes

Approximately 10% of the mineable ore will be left in place to provide structural support to the underground workings. Groundwater contacting this unmined ore will remain at background water quality. Therefore, the solute release from the unmined ore was set to zero for the purposes of the model.

3b. Solute Concentrations in the Crown Pillar

The solute concentrations released from the crown pillar were calculated similar to the wall rock, using leach data from ore samples.

ATTACHMENT B

ESTIMATED AVERAGE STOPE EXPOSURE TIME

ATTACHMENT B
DESCRIPTION OF CALCULATIONS ESTIMATING AVERAGE STOPE EXPOSURE TIME

Assumptions:

% of ore mined by blasthole		90%
Mineable Zinc Ore		30,000,000
Mineable Copper Ore		25,000,000
Zinc Production Period		16
Copper Production Period		12
Closure and Re-flooding Period		2
Average Blasthole Stope Dimensions	Length	75 Feet
	Width	100 Feet
	Height	200 Feet
	Insitu Density (Zinc Ore)	8.8 ft^3/ton
	Insitu Density (Copper Ore)	10.67 ft^3/ton
	Zinc Tons	170,455
	Copper Tons	140,581
Average Cut and Fill Stope Dimensions	Length	120 Feet
	Width	100 Feet
	Height	45 Feet
	Insitu Density (Zinc Ore)	8.8 ft^3/ton
	Insitu Density (Copper Ore)	10.67 ft^3/ton
	Zinc Tons	61,364
	Copper Tons	50,609

Zinc Ore Zone

Blasthole tons =		27,000,000
Cut and Fill tons =		3,000,000
Number of Blasthole Stopes (Accesses)		158
Number of Cut and Fill Stopes (Accesses)		49
Total Number of Access Crosscuts		207
Average Exposure Time	Zinc Production Period	8 years
	Copper Production Period	12 years
	Closure Period	1 years
	Total	21 years

Copper Ore Zone

Blasthole tons =		22,500,000
Cut and Fill tons =		2,500,000
Number of Blasthole Stopes (Accesses)		160
Number of Cut and Fill Stopes (Accesses)		49
Total Number of Access Crosscuts		209
Average Exposure Time	Zinc Production Period	0 years
	Copper Production Period	6 years
	Closure Period	1 years
	Total	7 years
Total Number of Access Crosscuts		417
Average Exposure Time		14 years

ATTACHMENT C

MINTEQA2 FILES

(files are provided on compact disc to selected reviewers)

ATTACHMENT D

MEMORANDUM from A.J. ERICKSON, Jr. TO MR. K. BLACK, DATED FEBRUARY 6, 1997

MEMORANDUM

EXXON COAL AND MINERALS COMPANY
Technical Department

February 6, 1997

TO: Ken Black - CMC

FROM: A. J. Erickson Jr

SUBJ: Addendum – Hanging Wall Sulfide Content Review, Crandon Deposit - Updated

---

The attached are summary and detail calculation sheets for work done in December 1996 on hanging wall sulfide estimates in key development areas of zinc mineralization at the Crandon, Wisconsin property.

A preliminary summary was faxed to K. Black, CMC, 12/16/96 with comments as indicated below. The data represents additional work carried out since 7/10/96, the date of the above titled report.

1. Basis was the same holes as in the 7/10/96 report.

2. Drill holes and sections used are representative of the area.

3. Data from 29 holes, locations as shown on the sections and plans in the 7/10/96 referenced report, were used in this update. These holes provide a representative sample of the ground in which mine development will take place.

4. Estimates on individual logs compiled 7/10/96.

5. Numerical values of estimates in the calculations are shown on the attached sheet SULSUM6.XLS. If the drill log showed trace (tr) sulfide content, the value 0.1% was used in the calculation. If the value of sulfide was estimated at 1.0 to 3.0%, the value of 2.0% was used in the calculations.

6. SULSUM5.XLS is a summary on a hole by hole basis.

7. No holes were included in the West Vent Shaft area as they do not sample the appropriate stratigraphy.

8. As shown on Stope 60 sections (shaft) the sulfides in the Skunk Lake are considerably north of the proposed shaft. Little sulfide is anticipated in the shaft, while the sand fill drift may contain some sulfide.

9. Individual sheets are available for each of the 29 holes.

4-2-13-278

CRANDON MINING COMPANY
Sulfide Distribution in Hanging Wall - By Formation

Footage Distribution								
	< 5% Sulfide				> 5% Sulfide			
	Ft	%Sulfide	Range(%)	Comment	Ft	%Sulfide	Range(%)	Comment
Sk	6843	0.6	0.0-4.0		226	6.2	4.8-15.0	
Rc	8606	0.4	0.0-3.0		12	6.0	6.0	
Ml	5776	0.5	0.0-2.0		3	26.7	25.0-30.0	
Lml	3395	0.5	0.0-5.0		26	5.0	5.0	

Percentage Distribution								
	< 5% Sulfide				> 5% Sulfide			
	Ft(%)	%Sulfide	Range(%)	Comment	Ft(%)	%Sulfide	Range(%)	Comment
Sk	96.8	0.6	0.0-4.0		3.2	6.2	4.8-15.0	
Rc	99.86	0.4	0.0-3.0		0.14	6.0	6.0	
Ml	99.95	0.5	0.0-2.0		0.05	26.7	25.0-30.0	
Lml	99.24	0.5	0.0-5.0		0.76	5.0	5.0	

% Sulfide	to	%Sulfur
0.1	equals	0.05
0.5	equals	0.27
0.7	equals	0.37
1.0	equals	0.53
1.5	equals	0.80
2.0	equals	1.06
2.5	equals	1.33
3.0	equals	1.59
3.5	equals	1.86
4.0	equals	2.12
4.5	equals	2.39
5.0	equals	2.65
5.5	equals	2.92
5.7	equals	3.00
6.0	equals	3.18

Value estimated Sulfide		
Log Estimate	% Sulfide Range	% Sulfide CalcValue
Nil	0.0	0.0
Nil to Tr	0 to 0.1	0.1
Tr	< 0.1	0.1
Wk	0.5	0.5
Tr to 1.0	0.5	0.5
< 1.0	< 1.0	0.7
< = 1.0	1.0	1.0
1.0 to 3.0	1.0 to 3.0	2.0
5.0 to 7.0	5.0 to 7.0	6.0

CRANDON MINING COMPANY

Sulfide Distribution in Hanging Wall - By Formation

Hole No.	Skunk Lake						Rice Creek						Mole Lake						Lower Mole Lake						Comment
	< 5% sulfide			>5 % sulfide			< 5% sulfide			>5 % sulfide			< 5% sulfide			>5 % sulfide			< 5% sulfide			>5 % sulfide			
	Ft	%	Range	Ft	%	Range	Ft	%	Range	Ft	%	Range	Ft	%	Range	Ft	%	Range	Ft	%	Range	Ft	%	Range	
9	220	1.3	0.5-2.0				370	0.7	0.0-0.7				60	0.6	0.5-0.7				126	0.7	0.5-1.0				
15	380	0.1	0.7-4.0				678	0.8	0.7-1.0				162	0.8	0.0-2.0				114	0.8	0.0-1.5				
20	230	0.1	0.0-0.1				241	0.1	0.0-0.1				78	0.1	0.0-0.1				115	0.5	1.0-1.5				
21	634	0.9	0.0-2.0	117	4.8	3.0-6.5	387	0.7	0.7-1.0				694	0.9	0.7-1.5				152	1.1	1.0-1.5				
				13	6.0	6.0																			
23	671	0.5	0.1-0.7	7	10.0	10.0	429	0.5	1.0-2.0				143	0.5	0.5				67	0.5					
24	854	1.1	0.0-3.0				315	0.5	0.5				174	0.2	0.2-0.7				111	0.1	0.1				
44	523	0.6	0.5-1.5	10	5.0	5	219	0.5	0.5-1.0				198	0.6	0.5-1.0				123	0.0	0.1-0.5				
46	88	0.1	0.1				715	0.3	0.1-2.0				297	0.3	0.1-2.0				123	0.5	0.1-2.0				
73	30	0.1	0.1				348	0.1	0.0-2.0				85	1.6	0.5-2.0				157	0.03	0.0-0.5				
74	254	0.6	0.5-0.7				282	0.1	0.0-0.5				116	0.0	0.0				117	0.5	0.0-0.7				
84	172	1.18	0.0-3.0	10	5.0	5.0	225	0.1	0.1				157	0.5	0.0-1.5	1	30.0	30.0	106	0.0	0.0				
																2	25.0	25.0							
85																									
91	497	0.4	0.1-3.0	10	5.0	5.0	211	0.1	0.0-0.1				73	0.4	0.0-2.0				83	1.5	1.0-2.0				NoDetail-936
92	241	0.5	0.5	57	9.0	9.0	182	0.5	0.1-1.0				87	0.6	0.1-0.7				133	0.5	0.0-1.0				
94	201	0.2	0.0-0.5				375	0.0	0.0-0.5				75	0.0	0.0-0.1				136	0.6	0.1-0.7				
98	612	0.6	0.0-2.0	2	15.0	15	273	0.1	0.0-0.7				238	0.03	0.0-0.3				123	0.8	0.1-1.0				
132													190	0.1	0.0-0.5				136	0.5	0.0-1.0				
133													200	0.1	0.01				128	0.4	0.0-1.0				
140	587	0.6	0.1-0.7				742	0.7	0.1-0.7				362	0.7	0.0-1.0				195	1.0	1.0				
143							67	0.0	0				143	0.0					120	0.3	0.1-0.5				
149							112	0.0	0				460	0.3	0.0-2.0				210	0.2	0.0-1.0				
154							106	0.9	0.7-1.5				207	1.0	0.7-1.0				162	0.0	0				
155	749	0.0	0.0-2.0				1353	0.2	0.0-6.0	1	6.0	6.00	528	0.3	0.0-6.0				145	0.9	1.0				
										11	6.0	6.00													
160							476	0.6	0.1-1.0				409	0.9	0.1-2.0				104	2.0	2.0				
170							250	0.2	0.0-3.0				80	0.0	0				96	0.0	0.0-5.0				
172							230	0.3	0.1-0.7																
174							20	0.0					150	0.1	0.0-0.1							26	5.0	5.0	
176													170	0.0	0				160	0.0	0.0-1.0				
214													240	0.2	0.2				151	0.5	0.0-1.5				
WtAvg	6843	0.6		226	6.2		8606	0.4		12	6.0		6776	0.6		3	26.7		3396	0.5		26	5.0		
%	96.8	0.6		3.2	6.0		99.86	0.6		0.1	6.0		99.95	0.6		0.05	26.7		99.24	0.6		0.76	5.0		

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278844	9	80	100	20	sk	0.5						
		100	131	31	sk	2.0						
		131	136	5	sk	1.0						
		136	144	8	sk	1.0						
		144	150	6	sk	1.0						
		150	161	11	sk	1.5						
		161	200	39	sk	1.0						
		200	300	100	sk	1.5		220	1.3			
		300	367	67	rc	0.7						
		367	670	283	rc	0.7		370	0.7			
		670	700	30	ml	0.7						
		700	730	30	ml	0.5		60	0.6			
		730	749	19	lml	0.5						
		749	751	2	lml	1.0						
		751	856	105	lml	0.7		126	0.7			
		856										

X

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

		Logs										
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278581	15	1200	1300	100	sk	1.0						
		1300	1358	58	sk	0.7						
		1358	1405	47	sk	0.7						
		1405	1418	13	sk	1.0						
		1418	1425	7	sk	4.0				Close		
		1425	1580	155	sk	0.7		380	0.9			
		1580	1720	140	rc	0.7						
		1720	1772	52	rc	0.7						
		1772	1923	151	rc	0.7						
		1923	2000	77	rc	1.0						
		2000	2209	209	rc	0.7						
		2209	2258	49	rc	1.0		678	0.8			
		2258	2278	20	ml	1.0						
		2278	2345	67	ml	0.7						
		2345	2353	8	ml	2.0						
		2353	2371	18	ml	0.7						
		2371	2375	4	ml	1.5						
		2375	2397	22	ml	0.7						
		2397	2409	12	ml	1.5						
		2409	2420	11	ml	0.0		162	0.8			
		2420	2438	18	lml	0.0						
		2438	2443	5	lml	1.0						
		2443	2460	17	lml	0.7						
		2460	2475	15	lml	1.0						
		2475	2500	25	lml	0.7						
		2500	2516	16	lml	1.0						
		2516	2534	18	lml	1.5		114	0.8			
		2534										
		0										
		0										
											x	

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

		Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278581	20	91	142	51	sk	0.1							
		142	176	34	sk	0.0							
		176	200	24	sk	0.0							
		200	321	121	sk	0.0		230					
		321	430	109	rc	0.0							
		430	480	50	rc	0.1							
		480	562	82	rc	0.0		241					
		562	592	30	ml	0.0							
		592	598	6	ml	0.1							
		598	640	42	ml	0.1		78					
		640	680	40	lml	0.1							
		680	720	40	lml	1.5							
		720	755	35	lml	1.5		115	0.5				
		755											
													x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278188	21	149	203	54	sk	0.0	oxide					
		203	246	43	sk	2.0						
		246	271	25	sk	1.0						
		271	300	29	sk	1.5						
		300	313	13	sk	1.0		164	1.0			
		313	323	10	sk	6.5		10		6.5		
		323	389	66	sk	4.5	cp	66		4.5		
		389	400	11	sk	4.0						
		400	421	21	sk	6.0			21.0	6.0		
		421	430	9	sk	3.0	cp	117		4.8		
		430	466	36	sk	0.7						
		466	615	149	sk	1.0						
		615	695	80	sk	0.7						
		695	756	61	sk	0.7						
		756	800	44	sk	0.7		370	0.8			
		800	813	13	sk	6.0	cp5 py1	13		6.0	664	1.7
		813	838	25	rc	1.0						
		838	872	34	rc	1.0						
		872	891	19	rc	0.7						
		891	1200	309	rc	0.7		387	0.7			
		1200	1274	74	ml	1.5						
		1274	1287	13	ml	1.0						
		1287	1300	13	ml	0.7						
		1300	1585	285	ml	1.0		694	0.9			
		1585	1709	124	lml	1.0						
		1709	1737	28	lml	1.5		152	1.1			
		1737										

x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278844	24	107	130	23	sk	0.0						
		130	400	270	sk	2.0						
		400	438	38	sk	2.0						
		438	463	25	sk	3.0						
		463	600	137	sk	1.0						
		600	700	100	sk	0.1						
		700	706	6	sk	0.2						
		706	716	10	sk	0.6						
		716	721	5	sk	0.2						
		721	756	35	sk	0.0						
		756	766	10	sk	0.4						
		766	961	195	sk	0.5		854	1.1			
		961	1276	315	rc	0.5		315	0.5			
		1276	1279	3	ml	0.3						
		1279	1286	7	ml	0.3						
		1286	1311	25	ml	0.7	>1 cp					
		1311	1326	15	ml	0.4						
		1326	1450	124	ml	0.1		174	0.2			
		1450	1561	111	lml	0.1		111	0.1			
		1561										

12/16/96 at 3:46 PM - AJEJr

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CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs													
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278581	23	155	203	48	sk	0.1		48	0.1			
		203	210	7	sk	10.0		7		10.0		
		210	330	120	sk	0.1						
		330	360	30	sk	0.5						
		360	444	84	sk	0.7						
		444	464	20	sk	0.7						
		464	474	10	sk	0.7						
		474	754	280	sk	0.5						
		754	764	10	sk	0.5						
		764	833	69	sk	0.5		623	0.5		678	0.5
		833	900	67	rc	0.5						
		900	944	44	rc	0.1						
		944	1043	99	rc	0.5						
		1043	1054	11	rc	0.7						
		1054	1096	42	rc	0.7						
		1096	1110	14	rc	2.0						
		1110	1262	152	rc	0.5		429	0.5			
		1262	1405	143	ml	0.5		143	0.5			
		1405	1472	67	lml	0.5		67	0.5			
		1472										

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Section	Logs Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278844	46	0	1100	1100	ak	0.0		0				
		1100	1186	86	ak	0.1		86		0.1		
		1186	1214	28	rc	0.1						
		1214	1293	79	rc	2.0						
		1293	1903	610	rc	0.1		715		0.3		
		1903	1945	42	ml	0.1						
		1945	1973	28	ml	2.0						
		1973	2200	227	ml	0.1		297		0.3		
		2200	2290	90	lml	0.1						
		2290	2300	10	lml	1.0						
		2300	2323	23	lml	2.0		123		0.5		
		2323										

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

		Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278188	44	105	170	65	sk	0.5							
		170	174	4	sk	1.0							
		174	390	216	sk	0.5							
		390	394	4	sk	0.7							
		394	424	30	sk	1.5							
		424	474	50	sk	1.0							
		474	494	20	sk	1.5			389	0.7			
		494	504	10	sk	5.0			10		5.0		
		504	514	10	sk	1.5							
		514	590	76	sk	0.5							
		590	638	48	sk	0.5			134	0.6		533	0.7
		638	829	191	rc	0.5							
		829	833	4	rc	1.0							
		833	857	24	rc	0.5			219	0.5			
		857	1014	157	ml	0.5							
		1014	1040	26	ml	1.0							
		1040	1055	15	ml	0.5			198	0.6			
		1055	1100	45	lml	0.5							
		1100	1178	78	lml	0.1			123	0.2			
		1178											

x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278188	74	106	200	92	sk	0.7	oxide					
		200	362	162	sk	0.5		254	0.8			
		362	400	38	rc	0.5						
		400	644	244	rc	0.0		282	0.1			
		644	760	116	ml	0.0		116	0.0			
		760	800	40	lml	0.0						
		800	810	10	lml	0.7						
		810	829	19	lml	0.7						
		829	877	48	lml	0.7		117	0.5			
											67	0.7
												x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2276581	73	980	990	30	ak	0.1		30	0.1			
		990	1050	60	rc	0.1						
		1050	1146	96	rc	0.0						
		1146	1154	8	rc	3.0						
		1154	1329	175	rc	0.0						
		1329	1334	5	rc	2.0						
		1334	1338	4	rc	2.0		348	0.1			
		1338	1400	62	ml	2.0						
		1400	1423	23	ml	0.5		85	1.6			
		1423	1430	7	lml	0.0						
		1430	1465	35	lml	0.0						
		1465	1496	31	lml	0.5						
		1496	1580	84	lml	0.0		157	0.03			
		1580										
		0										
		0										
		0										
		0										
		0										
		0										
												x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2277925	85	130	320	190	sk		no detail					
		320	771	451	rc		no detail					
		771	895	124	ml		no detail					
		895	936	41	lml		no detail					
		936	977	41	lml	2.0						
		977	1019	42	lml	1.0		83	1.5			
		1019										
												x

12/16/96 at 3:45 PM - AJEJr

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CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs												
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2279108	84	93	126	33	sk	0.0	Oxidized	s					
		126	184	58	sk	1.5		s					
		184	200	16	sk	3.0		s	107	1.3			
		200	210	10	sk	5.0		s	10		5.0		
		210	225	15	sk	0.0	Oxidized	s					
		225	247	22	sk	3.0		s					
		247	275	28	sk	0.0	Oxidized	s	65	1.0		182	1.4
		275	464	189	rc	0.1		r					
		464	480	16	rc	0.1		r					
		480	500	20	rc	0.0	Oxidized	r	225	0.1		225	0.1
		500	510	10	ml	1.5		m					
		510	526	16	ml	0.0	Oxidized	m	26	0.6			
		526	527	1	ml	30.0		m	1		30.0		
		527	532	5	ml	0.0	Oxidized	m	5				
		532	534	2	ml	25.0		m	2		25.0		
		534	542	8	ml	1.5		m					
		542	635	93	ml	0.5		m					
		635	660	25	ml	0.0		m	126	0.5		160	1.0
		660	768	108	lml	0.0	Oxidized	m	108	0.0		108	0.0
		768			cr								

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278712	91	73	151	78	sk	0.0	oxidized					
		151	158	7	sk	0.1	ptly oxidized					
		158	175	17	sk	0.5	ptly oxidized	102	0.1			
		175	185	10	sk	5.0		10		5.0		
		185	195	10	sk	0.5						
		195	280	85	sk	0.5						
		280	295	15	sk	1.0						
		295	422	127	sk	0.5						
		422	443	21	sk	0.0						
		443	445	2	sk	3.0						
		445	505	60	sk	0.7						
		505	520	15	sk	1.0						
		520	565	45	sk	0.5						
		565	581	16	sk	0.1		396	0.5		508	0.5
		581	700	119	rc	0.1						
		700	792	92	rc	0.0		211	0.1			
		792	800	8	ml	0.0						
		800	813	13	ml	2.0						
		813	865	52	ml	0.0		73	0.4			
		865	868	3	lml	0.0						
		868	900	32	lml	0.5						
		900	924	24	lml	0.1						
		924	954	30	lml	1.0						
		954	998	44	lml	0.5		133	0.5			
		998										

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278844	92	81	200	119	sk	0.5		s	119	0.5			
		200	257	57	sk	9.0			57		9.0		
		257	379	122	sk	0.5			122	0.5		298	2.1
		379	527	148	rc	0.5							
		527	532	5	rc	1.0							
		532	538	6	rc	0.5							
		538	546	8	rc	0.1							
		546	561	15	rc	0.7			182	0.5			
		561	571	10	ml	0.1							
		571	577	6	ml	0.7							
		577	593	16	ml	0.5							
		593	602	9	ml	0.1							
		602	630	28	ml	1.0							
		630	648	18	ml	0.7			87	0.6			
		648	663	15	lml	0.1							
		663	670	7	lml	0.1							
		670	750	80	lml	0.7							
		750	782	32	lml	0.7							
		782	784	2	lml	2.0			136	0.6			
		784	800	16	cr	2.0							
		800	809	9	cr	7.0							
		809	813	4	cr	3.0							
		813	821	8	cr	11.0							
		821	835	14	cr	0.5							
		835	851	16	cr	11.0							
		851											

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278319	94	99	210	111	sk	0.0	oxidized					
		210	300	90	sk	0.5		201	0.2			
		300	455	155	rc	0.0	oxidized					
		455	627	172	rc	0.0	oxidized					
		627	650	23	rc	0.5	ptly oxidized					
		650	675	25	rc	0.0	ptly oxidized	375	0.0			
		675	700	25	ml	0.0	ptly oxidized					
		700	750	50	ml	0.1	ptly oxidized	75	0.1			
		750	758	8	lml	0.1	ptly oxidized					
		758	825	67	lml	1.0						
		825	873	48	lml	0.7		123	0.8			
		873										
		0										
		0										
		0										
		0										
		0										
		0										
		0										
		0										
		0										
		0										

x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs													
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278188	98	138	235	97	sk	0.0							
		235	264	29	sk	1.0							
		264	303	39	sk	0.0							
		303	312	9	sk	1.0							
		312	338	26	sk	1.0							
		338	378	40	sk	2.0							
		378	400	22	sk	0.0							
		400	600	200	sk	0.7							
		600	710	110	sk	0.7			572	0.8			
		710	712	2	sk	15.0	cp		2		15.0		
		712	754	42	sk	0.7			42	0.7		616	0.7
		754	800	46	rc	0.7							
		800	1027	227	rc	0.0			273	0.1			
		1027	1143	116	ml	0.0							
		1143	1145	2	ml	3.0	cp						
		1145	1285	120	ml	0.0			238	0.03			
		1285	1300	35	lml	0.0							
		1300	1332	32	lml	0.7							
		1332	1341	9	lml	1.0							
		1341	1401	60	lml	0.7			136	0.5			
		1401											
													x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2276319	132	870	1000	130	ml	0.0	oxidized						
		1000	1030	30	ml	0.5	oxidized			0.1			
		1030	1060	30	ml	0.0			190	0.0			
		1060	1094	34	lml	0.0							
		1094	1138	44	lml	1.0							
		1138	1188	50	lml	0.1			128	0.4			
		1188											
		0											
		0											
		0											
		0											
		0											
		0											
		0											
		0											
		0											

x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2277925	133	900	1100	200	ml	0.1		200	0.1			
		1100	1160	60	lml	1.0						
		1160	1295	135	lml	1.0		195	1.0			
		1295										

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278581	140	1213	1400	187	sk	0.7						
		1400	1449	49	sk	0.7						
		1449	1500	51	sk	0.1						
		1500	1600	100	sk	0.7						
		1600	1638	38	sk	0.7						
		1638	1695	57	sk	0.7						
		1695	1800	105	sk	0.5		587	0.6			
		1800	1815	15	rc	0.7						
		1815	1829	14	rc	0.7						
		1829	1891	62	rc	0.7						
		1891	2535	644	rc	0.7						
		2535	2542	7	rc	0.1		742	0.7			
		2542	2622	80	ml	0.7						
		2622	2800	178	ml	0.7						
		2800	2815	15	ml	0.1						
		2815	2860	45	ml	1.0						
		2860	2900	40	ml	0.7						
		2900	2904	4	ml	0.0		362	0.7			
		2904	2920	16	lml	0.1						
		2920	2929	9	lml	0.1						
		2929	2942	13	lml	0.5						
		2942	2989	47	lml	0.1						
		2989	3024	35	lml	0.1		120	0.3			
		3024										
												x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

		Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278844	143	700	767	67	rc	0.0			67	0.0			
		767	910	143	ml	0.0			143	0.0			
		910	965	55	lml	0.0							
		965	1000	35	lml	0.5							
		1000	1100	100	lml	0.0							
		1100	1120	20	lml	1.0			210	0.2			
		1120											
		0											
		0											
		0											

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278319	149	1100	1212	112	rc	0.0		112	0.0			
		1212	1384	172	ml	0.0						
		1384	1390	6	ml	2.0						
		1390	1560	170	ml	0.0		460	0.03			
		1560	1722	162	lml	0.0		162	0.0			
		1722										

x

12/16/96 at 2:23 PM - AJEJr

SULS149.XLS

4.2-13-301

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278712	154	1202	1226	24	rc	1.5							
		1226	1300	74	rc	0.7							
		1300	1308	8	rc	1.5			106	0.9			
		1308	1354	46	ml	1.0							
		1354	1420	66	ml	1.0							
		1420	1442	22	ml	0.7							
		1442	1452	10	ml	1.0							
		1452	1500	48	ml	1.0							
		1500	1515	15	ml	1.0			207	1.0			
		1515	1588	73	lml	1.0							
		1588	1606	18	lml	1.0							
		1606	1660	54	lml	0.7			145	0.9			
		1660											
		0		0									
		0		0									
		0		0									
		0		0									
		0		0									

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2276188	155	111	200	89	sk	0.0	oxide					
		200	311	111	sk	0.0						
		311	353	42	sk	1.5						
		353	502	149	sk	0.5						
		502	514	12	sk	2.0						
		514	762	248	sk	0.5						
		762	860	98	sk	0.0		749	0.4			
		860	950	90	rc	0.0						
		950	1261	311	rc	0.0						
		1261	1280	19	rc	0.3						
		1280	1553	273	rc	0.0		693	0.0			
		1553	1554	1	rc	6.0		1		6.0		
		1554	1581	27	rc	0.2						
		1581	1583	2	rc	3.0						
		1583	1619	36	rc	0.0		65	0.2			
		1619	1652	33	rc	3.0		3		3.0		
		1652	1663	11	rc	6.0		11		6.0		
		1663	1940	277	rc	0.0						
		1940	1959	19	rc	0.1						
		1959	1965	6	rc	1.0						
		1965	2000	35	rc	0.1						
		2000	2022	22	rc	0.5						
		2022	2124	102	rc	0.1						
		2124	2183	59	rc	0.3						
		2183	2225	42	rc	0.0		562	0.1		1365	0.2
		2225	2470	245	ml	0.0						
		2470	2500	30	ml	0.5						
		2500	2680	180	ml	0.0						
		2680	2760	80	ml	1.0						
		2760	2830	70	ml	0.1						
		2830	2998	168	ml	0.0		528	0.3			
		2998										
												x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278168	160	1580	1600	20	rc	0.1							
		1600	1800	200	rc	0.7							
		1800	1995	195	rc	0.5							
		1995	2019	24	rc	1.0							
		2019	2056	37	rc	0.1			476	0.6			
		2056	2142	86	ml	0.1							
		2142	2254	112	ml	0.7							
		2254	2290	36	ml	0.7							
		2290	2330	40	ml	0.0							
		2330	2465	135	ml	2.0			409	0.9			
		2465	2569	104	lml	2.0			104	2.0			
		2569											
		0											

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs

Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278319	170	1000	1200	200	rc	0.0			0.0			
		1200	1205	5	rc	1.0						
		1205	1213	8	rc	3.0						
		1213	1250	37	rc	0.7		250	0.2		250	0.2
		1250	1256	6	ml	0.0						
		1256	1330	74	ml	0.0		80	0.0			
		1330	1395	65	lml	0.0						
		1395	1421	26	lml	5.0		26		5.0		
		1421	1452	31	lml	0.1		122	1.1			
		1452										

x

12/16/96 at 2:25 PM - AJEJr

1 of 1

4.2-13-305

SULS170.XLS

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Logs													
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes		Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278581	172	1500	1642	142	rc	0.1							
		1642	1730	88	rc	0.7			230	0.3			

x

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

Section	Logs Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2278844	174	700	720	20	rc	0.0		20	0.0			
		720	729	9	ml	1.0						
		729	870	141	ml	0.0		150	0.1			
		870	913	43	lml	0.0						
		913	928	15	lml	1.0						
		928	1000	72	lml	0.1						
		1000	1030	30	lml	0.0		160	0.1			
		1030										

12/16/96 at 2:24 PM - AJEJr

SULS174.XLS

4.2-13-307

CRANDON MINING COMPANY

Sulfide Distribution Preproduction Development

	Logs											
Section	Hole	Fr Ft	To Ft	Tk Ft	Fm	TS Est%	Sulfide Notes	Tk Ft	< 5%WtAvg	> 5%WtAvg	Tk Ft	Fm WtAvg
2276319	176	700	870	170	ml	0.0		170	0.0			
		870	900	30	lml	0.0						
		900	920	20	lml	1.0						
		920	985	65	lml	0.0						
		985	1021	36	lml	1.5		151	0.5			
		1021										

12/16/96 at 2:24 PM - AJEJr

SULS176.XLS

4.2-13-308

ATTACHMENT E

ESTIMATION OF WALL ROCK SURFACE AREA

ATTACHMENT E
ESTIMATION OF WALL ROCK SURFACE AREA

The amount of waste rock removed from each rock formation was estimated from the length of drift and cross cut development during mining and drift width and height. From these estimates, the wall rock surface areas and the drift voids were determined.

Wall rock surface area (ft^2) = 2 x {(wall height (ft) x length (ft)) + (back width (ft) x length (ft))}

Estimated volume (ft^3) = wall height (ft) x back width (ft) x length (ft)

The table below summarizes the calculation for each wall rock formation:

	Wall height ft.	Back width ft.	Length* ft.	Surface Area ft.^2	Est. Vol. ft.^3
L. Mole Lake Formation	13	16	45,100	2617000	$9.39x10^6$
U. Mole Lake Formation	13	16	9,800	566000	$2.03x10^6$
Rice Lake Formation	13	16	27,800	1610000	$5.78x10^6$
Skunk Lake Formation	13	16	3,500	201000	$7.22x10^5$

* Values have been rounded

ATTACHMENT F

SOLUTE CONCENTRATIONS AND MASS FLUX FROM COLUMN TESTS OF HANGING WALL ROCK

Table F-1.1

Parameters Measured Weekly
Skunk Lake Waste Rock Master Composite (WD-05)

Test: **Leached under alternating wet/dry conditions**
Water Sample: **Bottom seepage**

DATE	CYCLE	DAYS	VOLUME (L)	pH	CONDUCT. (uS/cm)	ALKALINITY (mg/L)	ACIDITY (mg/L)	SULFATE (mg/L) *	CUMULATIVE SULFATE (mg/kg)
25-Oct-94	1	0	5.725	6.2	316	44		**83**	6
2-Nov-94	2	8	5.470	6.3	294	33		**77**	11
9-Nov-94	3	15	4.055	5.7	240	33		66	14
16-Nov-94	4	22	4.930	6.9	192	30		55	17
21-Nov-94	5	27	4.460	6.0	186	34		**44**	19
28-Nov-94	6	34	5.560	6.1	128	27		39	22
5-Dec-94	7	41	4.250	6.5	121	29		33	24
12-Dec-94	8	48	4.930	6.0	126	30		28	25
19-Dec-94	9	55	4.320	7.7	126	31		**23**	26
27-Dec-94	10	63	4.460	7.7	131	28		24	28
3-Jan-95	11	70	4.970	7.6	131	31		25	29
9-Jan-95	12	76	3.740	7.3	108	26		27	30
17-Jan-95	13	84	4.290	7.3	112	26		28	32
23-Jan-95	14	90	4.200	7.6	123	24		29	33
30-Jan-95	15	97	4.580	7.5	123	26		30	35
6-Feb-95	16	104	0.500	7.1	126	23		32	35
13-Feb-95	17	111	8.160	8.3	147	23		**33**	38
20-Feb-95	18	118	4.700	7.1	132	26		33	40
27-Feb-95	19	125	4.100	7.4	131	29		32	41
6-Mar-95	20	132	4.780	7.5	141	24		32	43
13-Mar-95	21	139	3.610	7.1	113	200		32	45
20-Mar-95	22	146	4.550	7.0	124	20		31	46
27-Mar-95	23	153	4.420	6.8	134	18		31	48
4-Apr-95	24	161	5.080	6.8	129	16		30	50
10-Apr-95	25	167	4.960	6.9	138	20		30	52
17-Apr-95	26	174	4.080	6.7	151	16		30	53
24-Apr-95	27	181	3.940	6.6	130	14		29	54
1-May-95	28	188	4.060	6.7	123	24		29	56
8-May-95	29	195	3.580	6.7	120	17		29	57
15-May-95	30	202	4.400	6.9	120	20		28	58
22-May-95	31	209	3.950	7.1	116	16		28	60
30-May-95	32	217	4.840	6.9	113	18		27	61
5-Jun-95	33	223	4.320	6.7	114	18		**27**	63
12-Jun-95	34	230	4.600	7.0	96	22		26	64
19-Jun-95	35	237	5.740	7.5	107	14		26	66
26-Jun-95	36	244	3.980	7.5	103	14		26	67
3-Jul-95	37	251	4.280	7.3	106			25	68
10-Jul-95	38	258	5.260	7.4	110			25	70
17-Jul-95	39	265	0.000						
24-Jul-95	40	272	7.110	7.1	114			25	72
31-Jul-95	41	279	3.320	7.2	117			25	73
7-Aug-95	42	286	4.900	7.1	120			24	74
14-Aug-95	43	293	3.560	7.2	112			24	75
21-Aug-95	44	300	4.460	7.2	114			24	77
30-Aug-95	45	309	5.440	7.1	111			23	78
5-Sep-95	46	315	4.120	7.1	118	12		**23**	79
11-Sep-95	47	321	3.840	7.2	126	12		**24**	80
18-Sep-95	48	328	3.620	7.0	117	13		**35**	82
25-Sep-95	49	335	4.980	6.7	113	13		28	83
2-Oct-95	50	342	4.160	6.5	104	14		**20**	84
9-Oct-95	51	349	4.280	6.4	102	14		**20**	85
16-Oct-95	52	356	4.180	6.4	97	13		**39**	87
23-Oct-95	53	363	3.810	6.4	98	11		**26**	89
30-Oct-95	54	370	4.770	6.5	101	14		**19**	90

* Measured values of sulfate shown in bold.

Table F-1.2

Leachate Analysis
Skunk Lake Waste Rock Master Composite (WD-05)

Test: **Leached under alternating wet/dry conditions**
Water Sample: **Bottom seepage**

| | | LEACHATE ANALYSIS (mg/L) * | | |

Table F-1.3

Cumulative Mass Flux
Skunk Lake Waste Rock Master Composite (WD-05)

Test: **Leached under alternating wet/dry conditions**
Water Sample: **Bottom seepage**

		CUMULATIVE MASS FLUX (mg/kg sample)																			
CYCLE	DAYS	Aluminum Al	Antimony Sb	Arsenic As	Barium Ba	Beryllium Be	Cadmium Cd	Calcium Ca	Chromium Cr	Cobalt Co	Copper Cu	Iron Fe	Lead Pb	Magnesium Mg	Manganese Mn	Mercury Hg	Nickel Ni	Potassium K	Selenium Se	Silver Ag	Sodium Na	Zinc Zn
1	0	0.28	0.0034	0.001	0.001	0.000	0.000	1.4	0.000	0.0006	0.020	0.46	0.00	0.9	0.035	0.00001	0.001	0.9	0.002	0.000	0.7	0.01
2	8	1.06	0.0066	0.002	0.003	0.0007	0.001	2.6	0.001	0.0013	0.06	1.75	0.01	2.2	0.075	0.00003	0.003	1.6	0.004	0.001	1.3	0.02
3	15	1.46	0.0090	0.003	0.005	0.0009	0.001	3.4	0.002	0.0017	0.07	2.42	0.01	2.9	0.099	0.00004	0.004	2.1	0.005	0.001	1.6	0.03
4	22	1.73	0.012	0.004	0.006	0.0012	0.001	4.2	0.003	0.0023	0.09	2.87	0.01	3.6	0.119	0.00005	0.005	2.7	0.007	0.001	1.9	0.03
5	27	1.78	0.015	0.004	0.006	0.0015	0.002	4.7	0.003	0.0028	0.09	2.95	0.01	3.9	0.130	0.00006	0.006	3.1	0.008	0.001	2.0	0.03
6	34	1.85	0.018	0.005	0.007	0.0018	0.002	5.4	0.003	0.0035	0.09	3.07	0.01	4.4	0.144	0.00007	0.007	3.7	0.009	0.002	2.2	0.04
7	41	1.91	0.020	0.006	0.007	0.0020	0.003	5.9	0.003	0.0040	0.10	3.17	0.02	4.7	0.153	0.00008	0.008	4.1	0.010	0.002	2.3	0.04
8	48	2.00	0.023	0.006	0.008	0.0023	0.003	6.4	0.004	0.0046	0.10	3.29	0.02	5.1	0.164	0.00009	0.009	4.6	0.010	0.002	2.4	0.04
9	55	2.08	0.026	0.007	0.008	0.0026	0.003	6.9	0.004	0.0051	0.11	3.41	0.02	5.4	0.174	0.00010	0.010	5.0	0.010	0.003	2.5	0.04
10	63	2.15	0.029	0.008	0.009	0.0028	0.003	7.3	0.004	0.006	0.11	3.52	0.02	5.6	0.183		0.012	5.3	0.011	0.003	2.6	0.05
11	70	2.22	0.032	0.009	0.009	0.0031	0.004	7.8	0.005	0.006	0.11	3.62	0.02	6.0	0.193		0.013	5.7	0.011	0.003	2.6	0.05
12	76	2.27	0.034	0.010	0.010	0.0034	0.004	8.2	0.005	0.006	0.12	3.69	0.03	6.2	0.201		0.014	6.0	0.011	0.004	2.7	0.05
13	84	2.31	0.037	0.010	0.010	0.0036	0.004	8.7	0.005	0.007	0.12	3.75	0.03	6.5	0.210		0.016	6.3	0.011	0.004	2.8	0.05
14	90	2.34	0.040	0.011	0.010	0.0039	0.004	9.1	0.006	0.007	0.12	3.79	0.03	6.7	0.219		0.017	6.6	0.012	0.004	2.8	0.05
15	97	2.37	0.043	0.011	0.011	0.0041	0.005	9.6	0.006	0.008	0.12	3.83	0.03	7.0	0.228		0.019	6.9	0.012	0.005	2.9	0.06
16	104	2.37	0.043	0.012	0.011	0.0042	0.005	9.7	0.006	0.008	0.12	3.83	0.03	7.0	0.229		0.019	6.9	0.012	0.005	2.9	0.06
17	111	2.38	0.049	0.013	0.011	0.0046	0.005	10.6	0.007	0.008	0.13	3.84	0.03	7.5	0.245		0.023	7.4	0.013	0.006	3.0	0.06
18	118	2.38	0.052	0.013	0.012	0.0049	0.006	11.1	0.008	0.008	0.13	3.84	0.04	7.8	0.255		0.026	7.6	0.013	0.006	3.1	0.06
19	125	2.38	0.055	0.014	0.012	0.0052	0.006	11.5	0.008	0.009	0.13	3.85	0.04	8.0	0.262		0.027	7.8	0.013	0.007	3.1	0.06
20	132	2.39	0.058	0.014	0.012	0.0055	0.006	12.0	0.009	0.009	0.13	3.86	0.04	8.3	0.271		0.030	8.1	0.014	0.007	3.2	0.07
21	139	2.39	0.061	0.015	0.012	0.0057	0.006	12.4	0.009	0.009	0.13	3.86	0.04	8.6	0.278		0.031	8.3	0.014	0.008	3.2	0.07
22	146	2.40	0.064	0.015	0.013	0.0059	0.007	12.9	0.010	0.009	0.13	3.87	0.04	8.8	0.286		0.034	8.5	0.014	0.008	3.3	0.07
23	153	2.40	0.067	0.016	0.013	0.0062	0.007	13.4	0.010	0.010	0.13	3.88	0.04	9.1	0.293		0.036	8.7	0.015	0.009	3.3	0.07
24	161	2.40	0.071	0.016	0.013	0.0065	0.007	14.0	0.011	0.010	0.13	3.89	0.04	9.4	0.302		0.038	8.9	0.015	0.010	3.4	0.07
25	167	2.41	0.075	0.017	0.013	0.0068	0.007	14.5	0.012	0.010	0.14	3.90	0.05	9.7	0.310		0.040	9.1	0.016	0.010	3.4	0.08
26	174	2.41	0.077	0.017	0.014	0.0070	0.008	15.0	0.012	0.010	0.14	3.91	0.05	10.0	0.316		0.042	9.3	0.016	0.011	3.5	0.08
27	181	2.42	0.080	0.018	0.014	0.0073	0.008	15.4	0.012	0.011	0.14	3.91	0.05	10.2	0.322		0.044	9.5	0.017	0.011	3.5	0.08
28	188	2.42	0.083	0.018	0.014	0.0075	0.008	15.9	0.013	0.011	0.14	3.92	0.05	10.5	0.328		0.046	9.6	0.017	0.012	3.6	0.08
29	195	2.42	0.086	0.019	0.014	0.0077	0.008	16.3	0.013	0.011	0.14	3.93	0.05	10.7	0.333		0.048	9.8	0.018	0.012	3.6	0.09
30	202	2.43	0.089	0.019	0.015	0.0080	0.009	16.8	0.014	0.011	0.14	3.94	0.05	11.0	0.339		0.050	9.9	0.018	0.012	3.7	0.09
31	209	2.43	0.092	0.020	0.015	0.0082	0.009	17.2	0.014	0.012	0.14	3.95	0.06	11.2	0.344		0.052	10.0	0.019	0.013	3.7	0.09
32	217	2.44	0.095	0.020	0.015	0.0085	0.009	17.7	0.015	0.012	0.14	3.96	0.06	11.5	0.350		0.054	10.2	0.019	0.014	3.8	0.09
33	223	2.44	0.098	0.021	0.015	0.0088	0.009	18.2	0.015	0.012	0.15	3.98	0.06	11.8	0.355		0.056	10.3	0.020	0.014	3.8	0.10
34	230	2.45	0.101	0.021	0.016	0.0090	0.010	18.7	0.016	0.012	0.15	3.99	0.06	12.1	0.360		0.058	10.5	0.020	0.015	3.9	0.10
35	237	2.45	0.105	0.022	0.016	0.0094	0.010	19.4	0.017	0.013	0.15	4.00	0.06	12.5	0.367		0.061	10.6	0.021	0.015	3.9	0.10
36	244	2.46	0.108	0.022	0.016	0.0096	0.010	19.8	0.017	0.013	0.15	4.01	0.06	12.7	0.372		0.063	10.7	0.022	0.016	4.0	0.11
37	251	2.46	0.111	0.023	0.016	0.0099	0.010	20.3	0.017	0.013	0.15	4.02	0.07	13.0	0.377		0.065	10.8	0.022	0.016	4.0	0.11
38	258	2.47	0.115	0.023	0.017	0.0102	0.011	20.9	0.018	0.014	0.15	4.03	0.07	13.3	0.383		0.067	11.0	0.023	0.017	4.0	0.12
39	265																					
40	272	2.47	0.120	0.024	0.017	0.0106	0.011	21.7	0.019	0.014	0.15	4.05	0.07	13.8	0.391		0.071	11.1	0.024	0.018	4.1	0.13
41	279	2.48	0.122	0.025	0.017	0.0108	0.011	22.0	0.019	0.014	0.16	4.05	0.07	14.0	0.395		0.072	11.2	0.024	0.018	4.1	0.13
42	286	2.48	0.126	0.025	0.018	0.0111	0.012	22.6	0.019	0.014	0.16	4.06	0.07	14.3	0.401		0.075	11.3	0.025	0.019	4.2	0.14
43	293	2.48	0.128	0.026	0.018	0.0113	0.012	22.9	0.020	0.015	0.16	4.07	0.07	14.5	0.405		0.076	11.4	0.025	0.019	4.2	0.14
44	300	2.49	0.132	0.026	0.018	0.0115	0.012	23.4	0.020	0.015	0.16	4.07	0.07	14.8	0.410		0.078	11.5	0.026	0.020	4.3	0.15
45	309	2.49	0.135	0.027	0.018	0.0119	0.012	24.0	0.021	0.015	0.16	4.08	0.08	15.2	0.417		0.081	11.6	0.027	0.020	4.3	0.16
46	315	2.49	0.138	0.027	0.019	0.0121	0.013	24.4	0.021	0.016	0.16	4.09	0.08	15.4	0.421		0.083	11.7	0.027	0.021	4.4	0.17
47	321	2.50	0.141	0.028	0.019	0.0123	0.013	24.9	0.021	0.016	0.16	4.09	0.08	15.7	0.426		0.085	11.7	0.028	0.021	4.4	0.18
48	328	2.50	0.144	0.028	0.019	0.0125	0.013	25.2	0.022	0.016	0.16	4.10	0.08	15.9	0.430		0.087	11.8	0.028	0.022	4.4	0.18
49	335	2.50	0.147	0.029	0.019	0.0128	0.013	25.8	0.022	0.016	0.16	4.10	0.08	16.2	0.436		0.089	11.9	0.029	0.022	4.5	0.20
50	342	2.50	0.150	0.029	0.020	0.0131	0.014	26.2	0.022	0.016	0.17	4.10	0.08	16.5	0.441		0.091	11.9	0.029	0.023	4.5	0.21
51	349	2.51	0.153	0.030	0.020	0.0133	0.014	26.7	0.022	0.017	0.17	4.11	0.08	16.8	0.446		0.093	12.0	0.030	0.023	4.5	0.22
52	356	2.51	0.156	0.030	0.020	0.0136	0.014	27.1	0.023	0.017	0.17	4.11	0.08	17.0	0.451		0.095	12.1	0.030	0.024	4.6	0.23
53	363	2.51	0.159	0.030	0.020	0.0138	0.014	27.5	0.023	0.017	0.17	4.11	0.08	17.3	0.455		0.097	12.1	0.031	0.024	4.6	0.24
54	370	2.51	0.162	0.031	0.021	0.0141	0.015	28.0	0.023	0.017	0.17	4.12	0.08	17.6	0.461		0.099	12.2	0.032	0.025	4.6	0.25

Compiled by: RWL\MLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

Table F-2.1

Parameters Measured Weekly
Rice Lake Waste Rock Master Composite (WD-06)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

DATE	CYCLE	DAYS	VOLUME (L)	pH	CONDUCT. (uS/cm)	ALKALINITY (mg/L)	ACIDITY (mg/L)	SULFATE (mg/L) *	CUMULATIVE SULFATE (mg/kg)
25-Oct-94	1	0	6.490	7.0	345	130		**43.5**	2
2-Nov-94	2	8	4.785	7.4	315	110		**35.4**	4
9-Nov-94	3	15	3.760	5.7	251	100		30	4
16-Nov-94	4	22	7.160	7.5	288	110		24	6
21-Nov-94	5	27	3.400	7.0	238	91		**18.9**	6
28-Nov-94	6	34	2.710	7.4	215	91		17	7
5-Dec-94	7	41	2.480	7.8	214	92		16	7
12-Dec-94	8	48	6.650	7.3	169	71		**14.1**	8
19-Dec-94	9	55	5.400	8.1	195	80		**27.8**	8.9
27-Dec-94	10	63	3.880	8.1	243	93		26	10
3-Jan-95	11	70	3.900	8.3	233	87		25	11
9-Jan-95	12	76	3.340	8.2	209	89		23	11
17-Jan-95	13	84	3.400	8.2	202	83		21	12
23-Jan-95	14	90	3.820	8.3	210	52		20	12
30-Jan-95	15	97	4.030	8.4	206	51		18	13
6-Feb-95	16	104	0.500	8.2	210	82		17	13
13-Feb-95	17	111	7.480	8.2	217	86		**15**	14
20-Feb-95	18	118	4.460	8.2	199	83		15	14
27-Feb-95	19	125	4.320	8.4	202	82		15	15
6-Mar-95	20	132	4.880	8.2	204	81		14	15
13-Mar-95	21	139	3.820	8.2	174	78		14	16
20-Mar-95	22	146	4.660	8.1	183	76		14	16
27-Mar-95	23	153	4.600	8.1	202	78		14	17
4-Apr-95	24	161	5.740	8.1	181	75		13	17
10-Apr-95	25	167	5.160	8.1	204	73		13	18
17-Apr-95	26	174	3.770	8.2	167	73		13	18
24-Apr-95	27	181	4.400	8.1	166	39		13	19
1-May-95	28	188	4.380	8.1	170	80		12	19
8-May-95	29	195	4.960	8.1	168	71		12	20
15-May-95	30	202	5.100	8.1	164	74		12	20
22-May-95	31	209	3.220	8.0	159	68		12	20
30-May-95	32	217	3.780	7.9	157	68		11	21
5-Jun-95	33	223	3.720	7.9	165	70		**11**	21
12-Jun-95	34	230	4.700	8.1	153	70		11	21
19-Jun-95	35	237	5.470	8.3	162	70		10	22
26-Jun-95	36	244	5.100	8.2	156	70		10	22
3-Jul-95	37	251	4.820	8.2	158			10	23
10-Jul-95	38	258	5.150	8.2	154			10	23
17-Jul-95	39	265	0.000						
24-Jul-95	40	272	7.180	8.0	162			9	24
31-Jul-95	41	279	4.480	8.1	158			9	24
7-Aug-95	42	286	4.780	8.1	154			9	24
14-Aug-95	43	293	4.960	8.1	152			9	25
21-Aug-95	44	300	4.750	8.1	154			8	25
30-Aug-95	45	309	5.620	8.1	157			8	25
5-Sep-95	46	315	4.360	8.2	154	65		**8**	26
11-Sep-95	47	321	4.780	8.0	158	60		**8**	26
18-Sep-95	48	328	4.360	7.9	155	65		**8.4**	26
25-Sep-95	49	335	4.340	7.7	156	43		8	26
2-Oct-95	50	342	3.860	7.6	156	64		**7**	27
9-Oct-95	51	349	3.940	7.4	157	66		**8**	27
16-Oct-95	52	356	3.880	7.4	151	62		**8.7**	27
23-Oct-95	53	363	3.820	7.4	153	62		**11**	27
30-Oct-95	54	370	3.840	7.5	160	61		**8**	28

Compiled by: RWL\MLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

Table F-2.2

Leachate Analysis
Rice Lake Waste Rock Master Composite (WD-06)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

LEACHATE ANALYSIS (mg/L) *																														
CYCLE	DAYS	Aluminum Al		Antimony Sb		Arsenic As	Barium Ba		Beryllium Be		Cadmium Cd	Calcium Ca		Chromium Cr		Cobalt Co	Copper Cu	Iron Fe		Lead Pb	Magnesium Mg	Manganese Mn		Mercury Hg		Nickel Ni	Potassium K		Selenium Se		Silver Ag	Sodium Na	Zinc Zn
1	0	**76**	**<**	**0.050**		**0.049**	**0.12**	**<**	**0.005**	**<**	**0.005**	**20**		**0.054**		**0.039**	**0.85**	**110**		**0.065**	**66**	**2.3**		**0.0002**		**0.030**	**19.1**	**<**	**0.005**		**0.005**	**43**	**0.57**
2	8	**42**	**<**	**0.050**		**0.033**	**0.078**	**<**	**0.005**	**<**	**0.005**	**14**		**0.043**		**0.019**	**0.52**	**60**		**0.058**	**40**	**1.2**	**<**	**0.0002**		**0.028**	**17.4**	**<**	**0.005**	**<**	**0.005**	**30**	**0.34**
3	15	32.3		0.050		0.031	0.063		0.005		0.005	14		0.030		0.016	0.410	46.7		0.052	33.7	0.940		0.0002		0.025	14.90		0.0057		0.005	23.3	0.270
4	22	22.7		0.050		0.028	0.048		0.005		0.006	14		0.018		0.013	0.300	33.3		0.046	27.3	0.680		0.0002		0.023	12.40		0.0063		0.005	16.7	0.200
5	27	**13**	**<**	**0.050**		**0.026**	**0.033**	**<**	**0.005**		**0.006**	**14**	**<**	**0.005**	**<**	**0.010**	**0.19**	**20**	**<**	**0.040**	**21**	**0.42**	**<**	**0.0002**	**<**	**0.020**	**9.9**		**0.007**	**<**	**0.005**	**10**	**0.13**
6	34	11.0		0.050		0.028	0.036		0.005		0.006	16		0.005		0.010	0.303	17.0		0.040	18.3	0.467		0.0002		0.020	8.87		0.0063		0.005	8.53	0.180
7	41	9.0		0.050		0.031	0.040		0.005		0.005	17		0.005		0.010	0.417	14.0		0.040	15.7	0.513		0.0002		0.020	7.83		0.0057		0.005	7.07	0.230
8	48	**7.0**	**<**	**0.050**		**0.033**	**0.043**	**<**	**0.005**	**<**	**0.005**	**19**		**0.005**	**<**	**0.010**	**0.53**	**11**	**<**	**0.040**	**13**	**0.56**	**<**	**0.0002**	**<**	**0.020**	**6.8**	**<**	**0.005**	**<**	**0.005**	**5.6**	**0.28**
9	55	**19**	**<**	**0.050**		**0.022**	**0.037**	**<**	**0.005**	**<**	**0.005**	**16**		**0.013**	**<**	**0.010**	**0.23**	**27**	**<**	**0.040**	**23**	**0.62**	**<**	**0.0002**	**<**	**0.020**	**9.7**	**<**	**0.005**	**<**	**0.005**	**5.0**	**0.16**
10	63	16.7		0.051		0.023	0.034		0.005		0.005	16		0.012		0.010	0.213	23.8		0.038	21.3	0.556				0.023	9.29		0.0050		0.006	4.83	0.147
11	70	14.5		0.053		0.023	0.031		0.005		0.005	16		0.011		0.010	0.197	20.6		0.036	19.6	0.493				0.025	8.88		0.0050		0.006	4.65	0.134
12	76	12.2		0.054		0.024	0.028		0.005		0.005	16		0.010		0.010	0.180	17.4		0.034	17.9	0.429				0.028	8.46		0.0050		0.007	4.48	0.121
13	84	10.0		0.055		0.024	0.026		0.005		0.005	16		0.009		0.010	0.163	14.2		0.033	16.3	0.365				0.030	8.05		0.0050		0.008	4.30	0.109
14	90	7.7		0.056		0.025	0.023		0.005		0.005	16		0.008		0.010	0.146	11.0		0.031	14.6	0.301				0.033	7.64		0.0050		0.008	4.13	0.096
15	97	5.5		0.058		0.025	0.020		0.005		0.005	16		0.007		0.010	0.130	7.8		0.029	12.9	0.238				0.035	7.23		0.0050		0.009	3.95	0.083
16	104	3.2		0.059		0.026	0.017		0.005		0.005	16		0.006		0.010	0.113	4.6		0.027	11.2	0.174				0.038	6.81		0.0050		0.009	3.78	0.070
17	111	**0.99**	**<**	**0.060**		**0.026**	**0.014**	**<**	**0.005**	**<**	**0.005**	**16**	**<**	**0.005**	**<**	**0.010**	**0.096**	**1.4**	**<**	**0.025**	**9.5**	**0.11**			**<**	**0.040**	**6.4**	**<**	**0.005**	**<**	**0.010**	**3.6**	**0.057**
18	118	0.9		0.060		0.027	0.014		0.005		0.005	16		0.005		0.010	0.091	1.32		0.026	9.4	0.106				0.040	6.20		0.0052		0.010	3.50	0.055
19	125	0.9		0.060		0.027	0.013		0.005		0.005	16		0.005		0.010	0.085	1.24		0.026	9.3	0.102				0.040	6.00		0.0054		0.010	3.40	0.052
20	132	0.8		0.060		0.028	0.013		0.005		0.005	16		0.005		0.010	0.080	1.16		0.027	9.2	0.098				0.040	5.80		0.0056		0.010	3.30	0.050
21	139	0.8		0.060		0.028	0.013		0.005		0.005	17		0.005		0.010	0.074	1.08		0.027	9.1	0.095				0.040	5.60		0.0058		0.010	3.20	0.048
22	146	0.7		0.060		0.029	0.012		0.005		0.005	17		0.005		0.010	0.069	1.00		0.028	9.0	0.091				0.040	5.40		0.0059		0.010	3.10	0.045
23	153	0.7		0.060		0.029	0.012		0.005		0.005	17		0.005		0.010	0.063	0.92		0.028	8.9	0.087				0.040	5.20		0.0061		0.010	3.00	0.043
24	161	0.6		0.060		0.030	0.011		0.005		0.005	17		0.005		0.010	0.058	0.84		0.029	8.8	0.083				0.040	5.00		0.0063		0.010	2.90	0.041
25	167	0.6		0.060		0.031	0.011		0.005		0.005	17		0.005		0.010	0.053	0.77		0.030	8.8	0.079				0.040	4.80		0.0065		0.010	2.80	0.039
26	174	0.5		0.060		0.031	0.011		0.005		0.005	17		0.005		0.010	0.047	0.69		0.030	8.7	0.075				0.040	4.60		0.0067		0.010	2.70	0.036
27	181	0.5		0.060		0.032	0.010		0.005		0.005	17		0.005		0.010	0.042	0.61		0.031	8.6	0.071				0.040	4.40		0.0069		0.010	2.60	0.034
28	188	0.4		0.060		0.032	0.010		0.005		0.005	17		0.005		0.010	0.036	0.53		0.031	8.5	0.067				0.040	4.20		0.0071		0.010	2.50	0.032
29	195	0.4		0.060		0.033	0.010		0.005		0.005	18		0.005		0.010	0.031	0.45		0.032	8.4	0.064				0.040	4.00		0.0073		0.010	2.40	0.029
30	202	0.3		0.060		0.033	0.009		0.005		0.005	18		0.005		0.010	0.025	0.37		0.032	8.3	0.060				0.040	3.80		0.0074		0.010	2.30	0.027
31	209	0.3		0.060		0.034	0.009		0.005		0.005	18		0.005		0.010	0.020	0.29		0.033	8.2	0.056				0.040	3.60		0.0076		0.010	2.20	0.025
32	217	0.2		0.060		0.034	0.008		0.005		0.005	18		0.005		0.010	0.014	0.21		0.033	8.1	0.052				0.040	3.40		0.0078		0.010	2.10	0.022
33	223	**0.18**	**<**	**0.060**		**0.035**	**0.008**	**<**	**0.005**	**<**	**0.005**	**18**	**<**	**0.005**	**<**	**0.010**	**0.009**	**0.13**		**0.034**	**8.0**	**0.048**			**<**	**0.040**	**3.2**		**0.0080**	**<**	**0.010**	**2.0**	**<**	**0.020**
34	230	0.2		0.060		0.035	0.008		0.005		0.005	19		0.005		0.010	0.009	0.14		0.031	7.8	0.045				0.040	3.08		0.0080		0.010	1.91		0.020
35	237	0.2		0.060		0.034	0.008		0.005		0.005	20		0.005		0.009	0.008	0.14		0.030	7.7	0.044				0.040	3.01		0.0081		0.010	1.87		0.020
36	244	0.2		0.060		0.034	0.008		0.005		0.005	20		0.005		0.009	0.008	0.15		0.028	7.6	0.043				0.040	2.95		0.0081		0.010	1.83		0.020
37	251	0.2		0.060		0.034	0.008		0.005		0.005	21		0.005		0.009	0.008	0.15		0.027	7.5	0.041				0.040	2.89		0.0081		0.010	1.79		0.020
38	258	0.2		0.060		0.034	0.007		0.005		0.005	22		0.005		0.009	0.008	0.16		0.025	7.4	0.040				0.040	2.8		0.0081		0.010	1.74		0.020
39	265																																
40	272	0.2		0.060		0.034	0.007		0.005		0.005	22		0.005		0.008	0.008	0.16		0.024	7.3	0.039				0.040	2.77		0.0081		0.010	1.70		0.020
41	279	0.2		0.060		0.033	0.007		0.005		0.005	23		0.005		0.008	0.007	0.17		0.022	7.2	0.037				0.040	2.70		0.0082		0.010	1.66		0.020
42	286	0.2		0.060		0.033	0.007		0.005		0.005	24		0.005		0.008	0.007	0.17		0.021	7.1	0.036				0.040	2.64		0.0082		0.010	1.61		0.020
43	293	0.2		0.060		0.033	0.007		0.005		0.005	24		0.005		0.008	0.007	0.18		0.019	7.0	0.035				0.040	2.58		0.0082		0.010	1.57		0.020
44	300	0.2		0.060		0.033	0.007		0.005		0.005	25		0.005		0.007	0.007	0.18		0.018	7.0	0.033				0.040	2.52		0.0082		0.010	1.53		0.020
45	309	0.2		0.060		0.033	0.007		0.005		0.005	25		0.005		0.007	0.007	0.19		0.016	6.9	0.032				0.040	2.46		0.0082		0.010	1.49		0.020
46	315	0.2		0.060		0.033	0.007		0.005		0.005	26		0.005		0.007	0.007	0.19		0.015	6.8	0.031				0.040	2.40		0.0082		0.010	1.44		0.020
47	321	0.2		0.060		0.032	0.007		0.005		0.005	27		0.005		0.007	0.006	0.20		0.013	6.7	0.029				0.040	2.33		0.0083		0.010	1.40		0.020
48	328	0.2		0.060		0.032	0.007		0.005		0.005	27		0.005		0.006	0.006	0.20		0.012	6.6	0.028				0.040	2.27		0.0083		0.010	1.36		0.020
49	335	0.2		0.060		0.032	0.006		0.005		0.005	28		0.005		0.006	0.006	0.21		0.010	6.5	0.027				0.040	2.21		0.0083		0.010	1.31		0.020
50	342	0.2		0.060		0.032	0.006		0.005		0.005	29		0.005		0.006	0.006	0.21		0.009	6.4	0.025				0.040	2.15		0.0083		0.010	1.27		0.020
51	349	0.2		0.060		0.032	0.006		0.005		0.005	29		0.005		0.006	0.006	0.22		0.007	6.3	0.024				0.040	2.09		0.0083		0.010	1.23		0.020
52	356	0.2		0.060		0.031	0.006		0.005		0.005	30		0.005		0.005	0.005	0.22		0.006	6.2	0.023				0.040	2.02		0.0084		0.010	1.19		0.020
53	363	0.2		0.060		0.031	0.006		0.005		0.005	30		0.005		0.005	0.005	0.23		0.004	6.1	0.021				0.040	1.96		0.0084		0.010	1.14		0.020
54	370	**0.22**	**<**	**0.060**		**0.031**	**0.006**	**<**	**0.005**	**<**	**0.005**	**31**	**<**	**0.005**	**<**	**0.005**	**0.005**	**0.23**	**<**	**0.003**	**6.0**	**0.02**			**<**	**0.040**	**1.9**		**0.0084**	**<**	**0.010**	**1.1**	**<**	**0.020**

Compiled by: RWL\MLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

Table F-2.3

Cumulative Mass Flux
Rice Lake Waste Rock Master Composite (WD-06)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

CYCLE	DAYS	CUMULATIVE MASS FLUX (mg/kg sample)																				
		Aluminum Al	Antimony Sb	Arsenic As	Barium Ba	Beryllium Be	Cadmium Cd	Calcium Ca	Chromium Cr	Cobalt Co	Copper Cu	Iron Fe	Lead Pb	Magnesium Mg	Manganese Mn	Mercury Hg	Nickel Ni	Potassium K	Selenium Se	Silver Ag	Sodium Na	Zinc Zn
1	0	3.90	0.003	0.003	0.006	0.000	0.000	1.0	0.003	0.0020	0.044	5.64	0.003	3.4	0.118	0.00001	0.002	1.0	0.0003	0.0003	2.2	0.029
2	8	5.48	0.0045	0.004	0.009	0.0004	0.000	1.6	0.004	0.0027	0.06	7.91	0.006	4.9	0.163	0.00002	0.003	1.6	0.0004	0.0004	3.3	0.042
3	15	6.44	0.0059	0.005	0.011	0.0006	0.001	2.0	0.005	0.0032	0.08	9.29	0.007	5.9	0.191	0.00002	0.003	2.1	0.001	0.001	4.0	0.050
4	22	7.73	0.0088	0.006	0.014	0.0009	0.001	2.8	0.006	0.0039	0.09	11.18	0.010	7.4	0.230	0.00004	0.005	2.8	0.001	0.001	5.0	0.061
5	27	8.08	0.0101	0.007	0.015	0.0010	0.001	3.1	0.006	0.0042	0.10	11.72	0.011	8.0	0.241	0.00004	0.005	3.0	0.001	0.001	5.2	0.065
6	34	8.31	0.0112	0.008	0.015	0.0011	0.001	3.5	0.007	0.0044	0.10	12.08	0.012	8.4	0.251	0.00004	0.006	3.2	0.001	0.001	5.4	0.069
7	41	8.49	0.0122	0.008	0.016	0.0012	0.001	3.8	0.007	0.0046	0.11	12.35	0.012	8.7	0.261	0.00005	0.006	3.4	0.001	0.001	5.6	0.073
8	48	8.86	0.0148	0.010	0.018	0.0015	0.002	4.8	0.007	0.0051	0.14	12.93	0.014	9.4	0.290	0.00006	0.007	3.7	0.002	0.001	5.9	0.088
9	55	9.67	0.0169	0.011	0.020	0.0017	0.002	5.5	0.007	0.0056	0.15	14.08	0.016	10.4	0.317	0.00007	0.008	4.2	0.002	0.002	6.1	0.095
10	63	10.18	0.0185	0.012	0.021	0.0018	0.002	6.0	0.008	0.0059	0.16	14.81	0.017	11.0	0.334		0.009	4.4	0.002	0.002	6.2	0.099
11	70	10.63	0.0201	0.012	0.022	0.0020	0.002	6.5	0.008	0.0062	0.16	15.45	0.018	11.6	0.349		0.009	4.7	0.002	0.002	6.4	0.103
12	76	10.95	0.0215	0.013	0.023	0.0021	0.002	6.9	0.008	0.0064	0.17	15.91	0.019	12.1	0.360		0.010	4.9	0.002	0.002	6.5	0.107
13	84	11.22	0.0230	0.014	0.023	0.0023	0.002	7.3	0.009	0.0067	0.17	16.29	0.020	12.5	0.370		0.011	5.2	0.002	0.002	6.6	0.110
14	90	11.45	0.0247	0.014	0.024	0.0024	0.003	7.8	0.009	0.0070	0.18	16.62	0.021	13.0	0.379		0.012	5.4	0.003	0.003	6.7	0.112
15	97	11.63	0.0265	0.015	0.025	0.0026	0.003	8.3	0.009	0.0073	0.18	16.87	0.022	13.4	0.387		0.013	5.6	0.003	0.003	6.8	0.115
16	104	11.64	0.0268	0.015	0.025	0.0026	0.003	8.4	0.009	0.0074	0.18	16.89	0.022	13.4	0.388		0.013	5.6	0.003	0.003	6.9	0.115
17	111	11.70	0.0303	0.017	0.026	0.0029	0.003	9.3	0.009	0.0080	0.19	16.97	0.024	14.0	0.394		0.015	6.0	0.003	0.004	7.1	0.119
18	118	11.73	0.0324	0.018	0.026	0.0031	0.003	9.9	0.010	0.0083	0.19	17.02	0.025	14.3	0.398		0.017	6.2	0.003	0.004	7.2	0.121
19	125	11.76	0.0345	0.019	0.027	0.0032	0.003	10.5	0.010	0.0086	0.19	17.06	0.025	14.6	0.401		0.018	6.4	0.003	0.004	7.3	0.122
20	132	11.79	0.0368	0.020	0.027	0.0034	0.004	11.1	0.010	0.0090	0.20	17.10	0.027	15.0	0.405		0.020	6.7	0.004	0.005	7.4	0.124
21	139	11.82	0.0386	0.020	0.027	0.0036	0.004	11.6	0.010	0.0093	0.20	17.14	0.027	15.3	0.408		0.021	6.8	0.004	0.005	7.5	0.126
22	146	11.84	0.0408	0.022	0.028	0.0038	0.004	12.2	0.010	0.0097	0.20	17.17	0.028	15.6	0.411		0.022	7.0	0.004	0.005	7.7	0.127
23	153	11.87	0.0430	0.023	0.028	0.0039	0.004	12.8	0.011	0.0101	0.20	17.21	0.029	15.9	0.414		0.024	7.2	0.004	0.006	7.8	0.129
24	161	11.90	0.0457	0.024	0.029	0.0042	0.004	13.6	0.011	0.0105	0.21	17.24	0.031	16.3	0.418		0.026	7.5	0.005	0.006	7.9	0.131
25	167	11.92	0.0481	0.025	0.029	0.0044	0.004	14.3	0.011	0.0109	0.21	17.28	0.032	16.7	0.421		0.027	7.7	0.005	0.007	8.0	0.132
26	174	11.94	0.0499	0.026	0.030	0.0045	0.005	14.8	0.011	0.0112	0.21	17.30	0.033	16.9	0.424		0.029	7.8	0.005	0.007	8.1	0.134
27	181	11.95	0.0520	0.027	0.030	0.0047	0.005	15.4	0.011	0.0116	0.21	17.32	0.034	17.2	0.426		0.030	7.9	0.005	0.007	8.2	0.135
28	188	11.97	0.0541	0.028	0.030	0.0049	0.005	16.0	0.011	0.0119	0.21	17.34	0.035	17.5	0.428		0.031	8.1	0.006	0.008	8.3	0.136
29	195	11.98	0.0564	0.030	0.031	0.0051	0.005	16.7	0.012	0.0123	0.21	17.35	0.036	17.9	0.431		0.033	8.2	0.006	0.008	8.4	0.137
30	202	12.00	0.0589	0.031	0.031	0.0053	0.005	17.4	0.012	0.0127	0.21	17.37	0.037	18.2	0.433		0.035	8.4	0.006	0.008	8.4	0.138
31	209	12.00	0.0604	0.032	0.031	0.0054	0.005	17.8	0.012	0.0130	0.21	17.38	0.038	18.4	0.435		0.036	8.5	0.006	0.009	8.5	0.139
32	217	12.01	0.0622	0.033	0.031	0.0055	0.006	18.4	0.012	0.0133	0.21	17.38	0.039	18.6	0.436		0.037	8.6	0.007	0.009	8.6	0.139
33	223	12.02	0.0639	0.034	0.032	0.0057	0.006	18.9	0.012	0.0136	0.21	17.39	0.040	18.9	0.438		0.038	8.7	0.007	0.009	8.6	0.140
34	230	12.02	0.0662	0.035	0.032	0.0059	0.006	19.6	0.012	0.0139	0.22	17.39	0.041	19.2	0.439		0.039	8.8	0.007	0.010	8.7	0.141
35	237	12.03	0.0688	0.037	0.032	0.0061	0.006	20.5	0.013	0.0143	0.22	17.40	0.043	19.5	0.441		0.041	8.9	0.007	0.010	8.8	0.142
36	244	12.04	0.0712	0.038	0.033	0.0063	0.006	21.3	0.013	0.0147	0.22	17.40	0.044	19.8	0.443		0.043	9.0	0.008	0.010	8.9	0.142
37	251	12.05	0.0735	0.039	0.033	0.0065	0.007	22.1	0.013	0.0150	0.22	17.41	0.045	20.1	0.445		0.044	9.2	0.008	0.011	8.9	0.143
38	258	12.05	0.0759	0.041	0.033	0.0067	0.007	23.0	0.013	0.0154	0.22	17.42	0.046	20.4	0.446		0.046	9.3	0.008	0.011	9.0	0.144
39	265																					
40	272	12.06	0.0793	0.043	0.034	0.0070	0.007	24.2	0.014	0.0158	0.22	17.42	0.047	20.8	0.448		0.048	9.4	0.009	0.012	9.1	0.145
41	279	12.07	0.0814	0.044	0.034	0.0072	0.007	25.0	0.014	0.0161	0.22	17.43	0.048	21.1	0.450		0.050	9.5	0.009	0.012	9.1	0.146
42	286	12.08	0.0837	0.045	0.034	0.0073	0.007	25.9	0.014	0.0164	0.22	17.44	0.049	21.3	0.451		0.051	9.6	0.009	0.012	9.2	0.147
43	293	12.09	0.0861	0.046	0.034	0.0075	0.008	26.9	0.014	0.0167	0.22	17.44	0.050	21.6	0.452		0.053	9.7	0.010	0.013	9.3	0.147
44	300	12.09	0.0883	0.048	0.035	0.0077	0.008	27.8	0.014	0.0170	0.22	17.45	0.050	21.9	0.454		0.054	9.8	0.010	0.013	9.3	0.148
45	309	12.10	0.0910	0.049	0.035	0.0079	0.008	28.9	0.015	0.0173	0.22	17.46	0.051	22.2	0.455		0.056	9.9	0.010	0.014	9.4	0.149
46	315	12.11	0.0930	0.050	0.035	0.0081	0.008	29.8	0.015	0.0176	0.22	17.47	0.051	22.4	0.456		0.057	10.0	0.011	0.014	9.4	0.150
47	321	12.12	0.0953	0.051	0.036	0.0083	0.008	30.8	0.015	0.0178	0.22	17.47	0.052	22.7	0.457		0.059	10.1	0.011	0.014	9.5	0.150
48	328	12.13	0.0974	0.052	0.036	0.0085	0.009	31.8	0.015	0.0180	0.22	17.48	0.052	22.9	0.458		0.060	10.2	0.011	0.015	9.5	0.151
49	335	12.13	0.0994	0.054	0.036	0.0087	0.009	32.7	0.015	0.0182	0.22	17.49	0.053	23.1	0.459		0.062	10.3	0.012	0.015	9.6	0.152
50	342	12.14	0.1013	0.055	0.036	0.0088	0.009	33.6	0.015	0.0184	0.22	17.49	0.053	23.3	0.460		0.063	10.3	0.012	0.015	9.6	0.152
51	349	12.15	0.1031	0.056	0.036	0.0090	0.009	34.5	0.016	0.0186	0.22	17.50	0.053	23.5	0.461		0.064	10.4	0.012	0.016	9.7	0.153
52	356	12.15	0.1050	0.056	0.037	0.0091	0.009	35.4	0.016	0.0188	0.22	17.51	0.053	23.7	0.461		0.065	10.4	0.012	0.016	9.7	0.154
53	363	12.16	0.1068	0.057	0.037	0.0093	0.009	36.3	0.016	0.0189	0.22	17.51	0.054	23.9	0.462		0.066	10.5	0.013	0.016	9.7	0.154
54	370	12.17	0.1086	0.058	0.037	0.0094	0.0095	37.3	0.016	0.0191	0.22	17.52	0.054	24.1	0.463		0.068	10.6	0.013	0.017	9.8	0.155

Compiled by: RWL\MLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

Table F-3.1

Parameters Measured Weekly
Upper Mole Lake Waste Rock Master Composite (WD-07)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

DATE	CYCLE	DAYS	VOLUME (L)	pH	CONDUCT. (uS/cm)	ALKALINITY (mg/L)	ACIDITY (mg/L)	SULFATE (mg/L) *	CUMULATIVE SULFATE (mg/kg)
25-Oct-94	1	0	6.700	7.0	380	120		**48**	2.4
2-Nov-94	2	8	4.915	7.2	358	120		**44.7**	4.1
9-Nov-94	3	15	4.550	6.1	301	110		43	5.5
16-Nov-94	4	22	2.490	7.6	339	110		41	6.3
21-Nov-94	5	27	3.760	7.2	322	100		**39.8**	7.4
28-Nov-94	6	34	7.950	7.6	217	87		35	9.5
5-Dec-94	7	41	3.360	7.8	246	100		30	10.3
12-Dec-94	8	48	4.700	7.8	256	97		25	11.2
19-Dec-94	9	55	4.340	7.7	256	93		**20.6**	11.9
27-Dec-94	10	63	4.080	8.2	268	96		23	12.6
3-Jan-95	11	70	4.650	8.3	266	94		26	13.5
9-Jan-95	12	76	3.730	8.3	224	91		28	14.3
17-Jan-95	13	84	4.020	8.4	228	90		31	15.2
23-Jan-95	14	90	3.780	8.3	233	88		33	16.1
30-Jan-95	15	97	5.040	7.7	296	71		36	17.5
6-Feb-95	16	104	0.500	8.1	291	68		38	17.6
13-Feb-95	17	111	7.990	8.1	264	79		**41**	20.1
20-Feb-95	18	118	4.460	8.3	220	88		39	21.4
27-Feb-95	19	125	3.960	8.5	222	86		37	22.5
6-Mar-95	20	132	3.960	8.4	228	84		36	23.6
13-Mar-95	21	139	3.590	8.3	203	82		34	24.5
20-Mar-95	22	146	4.390	8.2	215	78		32	25.5
27-Mar-95	23	153	3.750	8.3	235	80		30	26.4
4-Apr-95	24	161	3.240	8.3	212	78		28	27.1
10-Apr-95	25	167	4.080	8.0	276	67		27	27.9
17-Apr-95	26	174	5.100	8.3	211	78		25	28.8
24-Apr-95	27	181	4.300	8.3	191	77		23	29.6
1-May-95	28	188	4.120	8.3	193	84		21	30.2
8-May-95	29	195	3.940	8.1	197	75		19	30.8
15-May-95	30	202	3.540	8.2	192	76		17	31.3
22-May-95	31	209	4.900	8.2	190	79		16	31.8
30-May-95	32	217	5.740	8.1	180	76		14	32.4
5-Jun-95	33	223	4.320	7.5	177	75		**12**	32.8
12-Jun-95	34	230	4.700	8.2	168	74		12	33.2
19-Jun-95	35	237	5.180	8.4	176	82		12	33.7
26-Jun-95	36	244	4.120	8.3	171	73		12	34.0
3-Jul-95	37	251	4.600	8.3	181			12	34.4
10-Jul-95	38	258	4.860	8.1	174			12	34.9
17-Jul-95	39	265	0.000						
24-Jul-95	40	272	7.230	8.1	182			11	35.5
31-Jul-95	41	279	4.080	8.2	175			11	35.8
7-Aug-95	42	286	4.670	8.2	172			11	36.2
14-Aug-95	43	293	4.450	8.3	172			11	36.6
21-Aug-95	44	300	4.700	8.2	170			11	37.0
30-Aug-95	45	309	5.100	8.2	173			11	37.4
5-Sep-95	46	315	3.960	8.3	176	70		**11**	37.8
11-Sep-95	47	321	3.900	8.2	184	67		**11**	38.1
18-Sep-95	48	328	4.340	8.1	176	69		**14**	38.5
25-Sep-95	49	335	4.800	7.9	182	68		12	39.0
2-Oct-95	50	342	5.160	7.8	176	71		**9**	39.3
9-Oct-95	51	349	4.730	7.6	172	70		15	39.8
16-Oct-95	52	356	4.700	7.7	167	69		**12**	40.2
23-Oct-95	53	363	4.600	7.6	168	69		**11**	40.6
30-Oct-95	54	370	5.020	7.7	183	55		**9**	41.0

* Measured values of sulfate shown in bold

Table F-3.2

Leachate Analysis
Upper Mole Lake Waste Rock Master Composite (WD-07)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

LEACHATE ANALYSIS (mg/L) *																																
CYCLE	DAYS	Aluminum Al		Antimony Sb		Arsenic As		Barium Ba		Beryllium Be		Cadmium Cd		Calcium Ca		Chromium Cr		Cobalt Co		Copper Cu	Iron Fe		Lead Pb		Magnesium Mg	Manganese Mn		Mercury Hg		Nickel Ni	Potassium K		Selenium Se		Silver Ag	Sodium Na	Zinc Zn
1	0	**20**	<	**0.050**		**0.022**		**0.045**	<	**0.005**	<	**0.005**		**13**		**0.015**		**0.0**		**0.22**	**30**	<	**0.040**		**20**	**0.51**	<	**0.0002**	<	**0.020**	**12.9**		**0.009**	<	**0.005**	**49**	**0.19**
2	8	**14**	<	**0.050**		**0.022**		**0.029**	<	**0.005**	<	**0.005**		**12**		**0.012**	<	**0.0**		**0.17**	**20**	<	**0.040**		**17**	**0.34**	<	**0.0002**	<	**0.020**	**16.2**		**0.012**	<	**0.005**	**36**	**0.14**
3	15	10.3		0.050		0.019		0.024		0.005		0.005		13		0.010		0.010		0.126	14.767		0.040		16	0.260		0.0002		0.020	14.7		0.011		0.005	30.00	0.108
4	22	6.5		0.050		0.015		0.020		0.005		0.005		15		0.007		0.010		0.081	9.533		0.040		14	0.180		0.0002		0.020	13.2		0.009		0.005	24.00	0.075
5	27	**2.8**	<	**0.050**		**0.012**		**0.015**	<	**0.005**	<	**0.005**		**16**	<	**0.005**	<	**0.010**		**0.037**	**4.300**	<	**0.040**		**13**	**0.100**	<	**0.0002**	<	**0.020**	**11.7**		**0.008**	<	**0.005**	**18**	**0.043**
6	34	5.1		0.050		0.019		0.019		0.005		0.005		17		0.006		0.010		0.055	7.475		0.040		14	0.153		0.0002		0.020	11.2		0.008		0.005	15.40	0.057
7	41	7.4		0.050		0.026		0.023		0.005		0.005		17		0.007		0.010		0.074	10.650		0.040		16	0.205		0.0002		0.020	10.7		0.008		0.005	12.80	0.071
8	48	9.7		0.050		0.033		0.026		0.005		0.005		18		0.007		0.010		0.092	13.825		0.040		17	0.258		0.0002		0.020	10.1		0.008		0.005	10.20	0.085
9	55	**12**	<	**0.050**		**0.015**		**0.030**	<	**0.005**	<	**0.005**		**18**		**0.008**	<	**0.010**		**0.110**	**17.000**	<	**0.040**		**18**	**0.310**	<	**0.0002**	<	**0.020**	**9.6**		**0.008**	<	**0.005**	**7.6**	**0.099**
10	63	10.7		0.051		0.015		0.029		0.005		0.006		18		0.004		0.010		0.105	15.200		0.038		17	0.299				0.023	9.3		0.008		0.006	7.19	0.262
11	70	9.4		0.053		0.014		0.028		0.005		0.007		19		0.005		0.010		0.099	13.400		0.036		17	0.288				0.025	8.9		0.007		0.006	6.78	0.424
12	76	8.1		0.054		0.014		0.026		0.005		0.007		19		0.005		0.010		0.094	11.600		0.034		16	0.276				0.028	8.6		0.007		0.007	6.36	0.587
13	84	6.8		0.055		0.013		0.025		0.005		0.008		20		0.005		0.010		0.088	9.800		0.033		15	0.265				0.030	8.3		0.007		0.008	5.95	0.750
14	90	5.5		0.056		0.013		0.024		0.005		0.009		20		0.005		0.010		0.083	8.000		0.031		14	0.254				0.033	7.9		0.006		0.008	5.54	0.912
15	97	4.2		0.058		0.012		0.023		0.005		0.010		20		0.005		0.010		0.077	6.200		0.029		14	0.243				0.035	7.6		0.006		0.009	5.13	1.075
16	104	2.9		0.059		0.012		0.021		0.005		0.010		21		0.005		0.010		0.072	4.400		0.027		13	0.231				0.038	7.2		0.005		0.009	4.71	1.237
17	111	**1.6**	<	**0.060**		**0.011**		**0.020**	<	**0.005**		**0.011**		**21**	<	**0.005**	<	**0.010**		**0.066**	**2.600**	<	**0.025**		**12**	**0.220**			<	**0.040**	**8.9**	<	**0.005**	<	**0.010**	**4.3**	**1.4**
18	118	1.5		0.063		0.012		0.019		0.005		0.011		21		0.005		0.010		0.062	2.443		0.027		12	0.207				0.040	6.7		0.005		0.010	4.17	1.315
19	125	1.4		0.066		0.012		0.018		0.005		0.010		21		0.006		0.010		0.059	2.287		0.029		12	0.194				0.040	6.6		0.006		0.010	4.04	1.230
20	132	1.3		0.069		0.013		0.017		0.005		0.010		21		0.006		0.010		0.055	2.130		0.031		11	0.181				0.040	6.4		0.006		0.010	3.91	1.145
21	139	1.2		0.073		0.013		0.017		0.005		0.010		21		0.006		0.010		0.051	1.973		0.034		11	0.169				0.040	6.3		0.006		0.010	3.78	1.060
22	146	1.1		0.076		0.014		0.016		0.005		0.009		20		0.007		0.010		0.047	1.817		0.036		11	0.156				0.040	6.1		0.007		0.010	3.64	0.974
23	153	1.1		0.079		0.014		0.015		0.005		0.009		20		0.007		0.010		0.044	1.660		0.038		11	0.143				0.040	5.9		0.007		0.010	3.51	0.889
24	161	1.0		0.082		0.015		0.014		0.005		0.008		20		0.007		0.010		0.040	1.503		0.040		11	0.130				0.040	5.8		0.007		0.010	3.38	0.804
25	167	0.87		0.085		0.016		0.013		0.005		0.008		20		0.008		0.010		0.036	1.347		0.042		10	0.117				0.040	5.6		0.008		0.010	3.25	0.719
26	174	0.78		0.088		0.016		0.012		0.005		0.008		20		0.008		0.010		0.032	1.190		0.044		10	0.104				0.040	5.4		0.008		0.010	3.12	0.634
27	181	0.69		0.091		0.017		0.011		0.005		0.007		20		0.008		0.010		0.029	1.033		0.046		10	0.091				0.040	5.3		0.008		0.010	2.99	0.549
28	188	0.60		0.094		0.017		0.010		0.005		0.007		20		0.008		0.010		0.025	0.876		0.048		10	0.078				0.040	5.1		0.008		0.010	2.86	0.464
29	195	0.51		0.098		0.018		0.010		0.005		0.007		20		0.009		0.010		0.021	0.720		0.051		9.5	0.066				0.040	5.0		0.009		0.010	2.73	0.379
30	202	0.41		0.101		0.018		0.009		0.005		0.006		19		0.009		0.010		0.017	0.563		0.053		9.2	0.053				0.040	4.8		0.009		0.010	2.59	0.293
31	209	0.32		0.104		0.019		0.008		0.005		0.006		19		0.009		0.010		0.014	0.406		0.055		9.0	0.040				0.040	4.6		0.009		0.010	2.46	0.208
32	217	0.23		0.107		0.019		0.007		0.005		0.005		19		0.010		0.010		0.010	0.250		0.057		8.8	0.027				0.040	4.5		0.010		0.010	2.33	0.123
33	223	**0.14**		**0.110**		**0.020**		**0.006**	<	**0.005**	<	**0.005**		**19**	<	**0.010**	<	**0.010**		**0.006**	**0.093**		**0.059**		**8.6**	**0.014**			<	**0.040**	**4.3**		**0.010**	<	**0.010**	**2.2**	**0.038**
34	230	0.14		0.103		0.020		0.006		0.005		0.005		19		0.010		0.010		0.006	0.096		0.054		8.5	0.014				0.040	4.1		0.010		0.010	2.10	0.037
35	237	0.14		0.101		0.020		0.006		0.005		0.005		19		0.009		0.009		0.006	0.097		0.051		8.4	0.013				0.040	4.0		0.010		0.010	2.06	0.037
36	244	0.14		0.099		0.020		0.006		0.005		0.005		19		0.009		0.009		0.006	0.098		0.048		8.4	0.013				0.040	3.9		0.010		0.010	2.01	0.037
37	251	0.14		0.096		0.020		0.006		0.005		0.005		19		0.009		0.009		0.006	0.099		0.046		8.3	0.013				0.040	3.8		0.010		0.010	1.96	0.036
38	258	0.14		0.094		0.019		0.006		0.005		0.005		19		0.009		0.009		0.006	0.101		0.043		8.2	0.013				0.040	3.7		0.010		0.010	1.91	0.036
39	265																																				
40	272	0.14		0.092		0.019		0.006		0.005		0.005		19		0.008		0.008		0.006	0.102		0.040		8.2	0.013				0.040	3.6		0.010		0.010	1.87	0.036
41	279	0.14		0.090		0.019		0.006		0.005		0.005		19		0.008		0.008		0.006	0.103		0.038		8.1	0.012				0.040	3.5		0.010		0.010	1.82	0.035
42	286	0.14		0.087		0.019		0.006		0.005		0.005		19		0.008		0.008		0.006	0.105		0.035		8.0	0.012				0.040	3.4		0.010		0.010	1.77	0.035
43	293	0.14		0.085		0.019		0.006		0.005		0.005		19		0.008		0.008		0.006	0.106		0.032		8.0	0.012				0.040	3.3		0.010		0.010	1.72	0.035
44	300	0.13		0.083		0.019		0.005		0.005		0.005		20		0.007		0.007		0.005	0.107		0.030		7.9	0.012				0.040	3.2		0.010		0.010	1.68	0.034
45	309	0.13		0.080		0.019		0.005		0.005		0.005		20		0.007		0.007		0.005	0.108		0.027		7.9	0.012				0.040	3.1		0.010		0.010	1.63	0.034
46	315	0.13		0.078		0.019		0.005		0.005		0.005		20		0.007		0.007		0.005	0.110		0.024		7.8	0.012				0.040	3.0		0.010		0.010	1.58	0.034
47	321	0.13		0.076		0.019		0.005		0.005		0.005		20		0.007		0.007		0.005	0.111		0.022		7.7	0.011				0.040	2.9		0.010		0.010	1.53	0.033
48	328	0.13		0.074		0.019		0.005		0.005		0.005		20		0.006		0.006		0.005	0.112		0.019		7.7	0.011				0.040	2.8		0.010		0.010	1.49	0.033
49	335	0.13		0.071		0.018		0.005		0.005		0.005		20		0.006		0.006		0.005	0.114		0.016		7.6	0.011				0.040	2.7		0.010		0.010	1.44	0.033
50	342	0.13		0.069		0.018		0.005		0.005		0.005		20		0.006		0.006		0.005	0.115		0.014		7.5	0.011				0.040	2.6		0.010		0.010	1.39	0.032
51	349	0.13		0.067		0.018		0.005		0.005		0.005		20		0.006		0.006		0.005	0.116		0.011		7.5	0.011				0.040	2.5		0.010		0.010	1.34	0.032
52	356	0.13		0.065		0.018		0.005		0.005		0.005		20		0.005		0.005		0.005	0.117		0.008		7.4	0.010				0.040	2.4		0.010		0.010	1.30	0.032
53	363	0.13		0.062		0.018		0.005		0.005		0.005		20		0.005		0.005		0.005	0.119		0.006		7.4	0.010				0.040	2.3		0.010		0.010	1.25	0.031
54	370	**0.13**	<	**0.060**		**0.018**	<	**0.005**	<	**0.005**	<	**0.005**		**20**	<	**0.005**	<	**0.005**	<	**0.005**	**0.120**	<	**0.003**		**7.3**	**0.010**			<	**0.040**	**2.2**		**0.010**	<	**0.010**	**1.2**	**0.031**

* Measured values shown in bold

Compiled by: RWL\WLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

Table F-3.3

Cumulative Mass Flux
Upper Mole Lake Waste Rock Master Composite (WD-07)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

		CUMULATIVE MASS FLUX (mg/kg sample)																			
CYCLE	DAYS	Aluminum Al	Antimony Sb	Arsenic As	Barium Ba	Beryllium Be	Cadmium Cd	Calcium Ca	Chromium Cr	Cobalt Co	Copper Cu	Iron Fe	Lead Pb	Magnesium Mg	Manganese Mn	Mercury Hg	Nickel Ni	Potassium K	Selenium Se	Silver Ag	Sodium Na	Zinc Zn
1	0	1.01	0.003	0.001	0.002	0.000	0.000	0.7	0.001	0.001	0.011	1.51	0.002	1.01	0.026	0.00001	0.0010	0.6	0.0005	0.0003	2.5	0.01
2	8	1.52	0.0044	0.002	0.003	0.0004	0.000	1.1	0.001	0.001	0.017	2.25	0.003	1.63	0.038	0.00002	0.0017	1.2	0.0009	0.0004	3.8	0.01
3	15	1.87	0.0061	0.003	0.004	0.0006	0.001	1.6	0.002	0.001	0.022	2.75	0.005	2.17	0.047	0.00002	0.0024	1.7	0.0013	0.0006	4.8	0.02
4	22	2.00	0.0070	0.003	0.005	0.0007	0.001	1.8	0.002	0.002	0.023	2.93	0.006	2.44	0.050	0.00003	0.0028	2.0	0.0014	0.0007	5.3	0.02
5	27	2.07	0.0084	0.003	0.005	0.0008	0.001	2.3	0.002	0.002	0.024	3.05	0.007	2.80	0.053	0.00003	0.0034	2.3	0.0017	0.0008	5.8	0.02
6	34	2.38	0.011	0.004	0.006	0.0011	0.001	3.3	0.002	0.002	0.027	3.50	0.009	3.65	0.062	0.00005	0.0046	3.0	0.0021	0.0011	6.7	0.02
7	41	2.57	0.013	0.005	0.007	0.0013	0.001	3.7	0.002	0.003	0.029	3.77	0.010	4.05	0.068	0.00005	0.0051	3.3	0.0023	0.0013	7.0	0.03
8	48	2.91	0.014	0.006	0.008	0.0014	0.001	4.3	0.003	0.003	0.033	4.25	0.012	4.64	0.077	0.00006	0.0058	3.6	0.0026	0.0014	7.4	0.03
9	55	3.30	0.016	0.007	0.009	0.0016	0.002	4.9	0.003	0.003	0.036	4.81	0.013	5.22	0.087	0.00006	0.0064	3.9	0.0029	0.0016	7.6	0.03
10	63	3.63	0.018	0.007	0.009	0.0018	0.002	5.5	0.003	0.004	0.039	5.27	0.014	5.75	0.096		0.0071	4.2	0.0031	0.0018	7.8	0.04
11	70	3.96	0.019	0.008	0.010	0.0019	0.002	6.1	0.003	0.004	0.043	5.74	0.015	6.33	0.11		0.0080	4.5	0.0034	0.0020	8.1	0.06
12	76	4.18	0.021	0.008	0.011	0.0021	0.002	6.6	0.003	0.004	0.045	6.07	0.016	6.77	0.11		0.0088	4.8	0.0036	0.0022	8.3	0.07
13	84	4.39	0.023	0.008	0.012	0.0022	0.002	7.2	0.003	0.005	0.048	6.36	0.017	7.22	0.12		0.0097	5.0	0.0038	0.0024	8.4	0.09
14	90	4.5	0.024	0.009	0.013	0.0024	0.003	7.8	0.004	0.005	0.050	6.59	0.018	7.63	0.13		0.011	5.2	0.0039	0.0026	8.6	0.12
15	97	4.7	0.026	0.009	0.013	0.0026	0.003	8.6	0.004	0.005	0.053	6.82	0.019	8.14	0.14		0.012	5.5	0.0041	0.0030	8.8	0.16
16	104	4.7	0.027	0.009	0.013	0.0026	0.003	8.6	0.004	0.005	0.054	6.84	0.019	8.18	0.14		0.012	5.6	0.0042	0.0030	8.8	0.17
17	111	4.8	0.030	0.010	0.015	0.0029	0.004	9.9	0.004	0.006	0.058	7.00	0.021	8.90	0.15		0.014	6.0	0.0045	0.0036	9.1	0.25
18	118	4.9	0.032	0.010	0.015	0.0030	0.004	10.6	0.004	0.006	0.060	7.08	0.022	9.30	0.16		0.016	6.2	0.0046	0.0039	9.2	0.29
19	125	4.9	0.034	0.011	0.016	0.0032	0.004	11.2	0.004	0.006	0.061	7.15	0.023	9.64	0.16		0.017	6.4	0.0048	0.0042	9.3	0.33
20	132	4.9	0.036	0.011	0.016	0.0033	0.005	11.8	0.005	0.007	0.063	7.21	0.023	9.98	0.17		0.018	6.6	0.0050	0.0045	9.4	0.36
21	139	5.0	0.038	0.011	0.017	0.0035	0.005	12.4	0.005	0.007	0.064	7.26	0.024	10.28	0.17		0.019	6.7	0.0052	0.0048	9.5	0.39
22	146	5.0	0.041	0.012	0.017	0.0036	0.005	13.1	0.005	0.007	0.066	7.32	0.026	10.64	0.18		0.021	7.0	0.0054	0.0051	9.7	0.42
23	153	5.0	0.043	0.012	0.018	0.0038	0.006	13.6	0.005	0.008	0.067	7.37	0.027	10.94	0.18		0.022	7.1	0.0056	0.0054	9.8	0.45
24	161	5.1	0.045	0.013	0.018	0.0039	0.006	14.1	0.005	0.008	0.068	7.41	0.028	11.20	0.19		0.023	7.3	0.0057	0.0057	9.8	0.47
25	167	5.1	0.048	0.013	0.018	0.0041	0.006	14.7	0.006	0.008	0.069	7.45	0.029	11.52	0.19		0.024	7.4	0.0060	0.0060	9.9	0.49
26	174	5.1	0.051	0.014	0.019	0.0042	0.006	15.5	0.006	0.009	0.071	7.49	0.031	11.90	0.19		0.025	7.6	0.0063	0.0064	10.1	0.52
27	181	5.1	0.054	0.014	0.019	0.0044	0.006	16.1	0.006	0.009	0.071	7.53	0.032	12.22	0.20		0.027	7.8	0.0065	0.0067	10.2	0.53
28	188	5.2	0.057	0.015	0.020	0.0046	0.007	16.7	0.006	0.009	0.072	7.55	0.034	12.52	0.20		0.028	8.0	0.0068	0.0070	10.2	0.55
29	195	5.2	0.060	0.015	0.020	0.0047	0.007	17.3	0.007	0.010	0.073	7.57	0.035	12.80	0.20		0.029	8.1	0.0071	0.0073	10.3	0.56
30	202	5.2	0.062	0.016	0.020	0.0048	0.007	17.8	0.007	0.010	0.073	7.59	0.036	13.05	0.20		0.030	8.2	0.0073	0.0075	10.4	0.57
31	209	5.2	0.066	0.016	0.020	0.0050	0.007	18.5	0.007	0.010	0.074	7.60	0.038	13.38	0.20		0.032	8.4	0.0076	0.0079	10.5	0.57
32	217	5.2	0.071	0.017	0.021	0.0052	0.008	19.4	0.008	0.011	0.074	7.62	0.041	13.76	0.21		0.033	8.6	0.0081	0.0083	10.6	0.58
33	223	5.2	0.074	0.018	0.021	0.0054	0.008	20.0	0.008	0.011	0.074	7.62	0.043	14.04	0.21		0.035	8.7	0.0084	0.0087	10.7	0.58
34	230	5.2	0.078	0.019	0.021	0.0056	0.008	20.7	0.008	0.011	0.075	7.62	0.045	14.33	0.21		0.036	8.9	0.0087	0.0090	10.7	0.58
35	237	5.2	0.082	0.019	0.021	0.0058	0.008	21.4	0.009	0.012	0.075	7.63	0.047	14.66	0.21		0.038	9.0	0.0091	0.0094	10.8	0.58
36	244	5.2	0.085	0.020	0.022	0.0059	0.008	22.0	0.009	0.012	0.075	7.63	0.048	14.92	0.21		0.039	9.2	0.0094	0.010	10.9	0.58
37	251	5.2	0.088	0.021	0.022	0.0061	0.008	22.7	0.009	0.012	0.075	7.63	0.050	15.21	0.21		0.040	9.3	0.010	0.010	10.9	0.59
38	258	5.2	0.092	0.021	0.022	0.0063	0.009	23.4	0.010	0.013	0.075	7.64	0.051	15.51	0.21		0.042	9.4	0.010	0.010	11.0	0.59
39	265																					
40	272	5.2	0.097	0.022	0.022	0.0066	0.009	24.4	0.010	0.013	0.076	7.64	0.054	15.95	0.21		0.044	9.6	0.011	0.011	11.1	0.59
41	279	5.3	0.100	0.023	0.022	0.0067	0.009	25.0	0.010	0.013	0.076	7.64	0.055	16.20	0.21		0.045	9.7	0.011	0.011	11.2	0.59
42	286	5.3	0.103	0.024	0.023	0.0069	0.009	25.7	0.011	0.013	0.076	7.65	0.056	16.48	0.21		0.047	9.9	0.011	0.012	11.2	0.59
43	293	5.3	0.105	0.024	0.023	0.0071	0.009	26.3	0.011	0.014	0.076	7.65	0.057	16.75	0.21		0.048	10.0	0.012	0.012	11.3	0.59
44	300	5.3	0.108	0.025	0.023	0.0072	0.009	27.0	0.011	0.014	0.076	7.66	0.058	17.03	0.21		0.049	10.1	0.012	0.012	11.4	0.59
45	309	5.3	0.111	0.026	0.023	0.0074	0.010	27.8	0.011	0.014	0.077	7.66	0.059	17.33	0.21		0.051	10.2	0.012	0.013	11.4	0.60
46	315	5.3	0.114	0.026	0.023	0.0076	0.010	28.4	0.012	0.014	0.077	7.66	0.060	17.56	0.21		0.052	10.3	0.013	0.013	11.5	0.60
47	321	5.3	0.116	0.027	0.024	0.0077	0.010	28.9	0.012	0.015	0.077	7.67	0.060	17.79	0.21		0.053	10.4	0.013	0.013	11.5	0.60
48	328	5.3	0.118	0.027	0.024	0.0079	0.010	29.6	0.012	0.015	0.077	7.67	0.061	18.04	0.21		0.054	10.5	0.013	0.014	11.6	0.60
49	335	5.3	0.121	0.028	0.024	0.0081	0.010	30.3	0.012	0.015	0.077	7.67	0.062	18.31	0.21		0.056	10.6	0.014	0.014	11.6	0.60
50	342	5.3	0.124	0.029	0.024	0.0083	0.011	31.1	0.012	0.015	0.078	7.68	0.062	18.60	0.21		0.057	10.7	0.014	0.014	11.7	0.60
51	349	5.3	0.126	0.029	0.024	0.0084	0.011	31.8	0.013	0.016	0.078	7.68	0.063	18.87	0.21		0.059	10.7	0.014	0.015	11.7	0.60
52	356	5.3	0.128	0.030	0.024	0.0086	0.011	32.5	0.013	0.016	0.078	7.69	0.063	19.13	0.21		0.060	10.8	0.015	0.015	11.8	0.60
53	363	5.3	0.130	0.031	0.025	0.0088	0.011	33.2	0.013	0.016	0.078	7.69	0.063	19.39	0.21		0.062	10.9	0.015	0.015	11.8	0.60
54	370	5.3	0.133	0.031	0.025	0.0090	0.011	33.9	0.013	0.016	0.078	7.70	0.063	19.66	0.21		0.063	11.0	0.016	0.016	11.8	0.61

Compiled by: RWL\WLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

Table F-4.1

Parameters Measured Weekly
Lower Mole Lake Waste Rock Master Composite (WD-08)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

DATE	CYCLE	DAYS	VOLUME (L)	pH	CONDUCT. (uS/cm)	ALKALINITY (mg/L)	ACIDITY (mg/L)	SULFATE (mg/L) *	CUMULATIVE SULFATE (mg/kg)
25-Oct-94	1	0	5.690	7.2	595	120		**115**	8
2-Nov-94	2	8	4.655	6.7	431	91		**81**	13
9-Nov-94	3	15	4.700	6.1	342	89		69	17
16-Nov-94	4	22	4.780	7.4	294	75		58	21
21-Nov-94	5	27	4.360	6.8	286	76		**46**	24
28-Nov-94	6	34	5.300	7.5	227	75		41	26
5-Dec-94	7	41	3.795	7.6	206	76		35	28
12-Dec-94	8	48	4.840	7.7	224	77		30	30
19-Dec-94	9	55	3.730	7.9	323	79		**24.5**	31
27-Dec-94	10	63	4.320	8.0	230	83		25	32
3-Jan-95	11	70	4.840	8.1	226	81		25	34
9-Jan-95	12	76	3.880	8.3	191	75		25	35
17-Jan-95	13	84	3.950	8.3	195	71		25	37
23-Jan-95	14	90	4.100	8.3	197	73		25	38
30-Jan-95	15	97	5.290	8.4	199	76		25	40
6-Feb-95	16	104	0.500	8.3	214	70		25	40
13-Feb-95	17	111	8.500	8.2	213	72		**25**	42
20-Feb-95	18	118	5.580	8.3	194	73		25	44
27-Feb-95	19	125	3.940	8.4	196	73		24	45
6-Mar-95	20	132	5.460	8.3	198	72		24	47
13-Mar-95	21	139	4.530	8.2	180	76		23	48
20-Mar-95	22	146	4.510	8.1	188	68		23	50
27-Mar-95	23	153	4.540	8.3	206	71		22	51
4-Apr-95	24	161	4.620	8.2	188	66		22	52
10-Apr-95	25	167	5.490	8.2	204	64		22	54
17-Apr-95	26	174	4.140	8.2	217	62		21	55
24-Apr-95	27	181	4.720	8.2	188	63		21	56
1-May-95	28	188	5.000	8.2	180	74		20	58
8-May-95	29	195	4.540	8.1	177	62		20	59
15-May-95	30	202	5.040	8.1	175	64		19	60
22-May-95	31	209	3.580	8.2	167	64		19	61
30-May-95	32	217	5.100	8.1	167	62		18	62
5-Jun-95	33	223	3.790	7.8	164	61		**18**	63
12-Jun-95	34	230	4.280	8.1	149	60		18	64
19-Jun-95	35	237	4.300	8.3	175	60		18	65
26-Jun-95	36	244	4.160	8.2	166	62		18	66
3-Jul-95	37	251	3.680	8.1	167			18	67
10-Jul-95	38	258	4.860	8.1	167			18	68
17-Jul-95	39	265	0.000						68
24-Jul-95	40	272	7.750	7.9	172			19	70
31-Jul-95	41	279	4.060	8.1	175			19	71
7-Aug-95	42	286	4.180	8.0	166			19	72
14-Aug-95	43	293	4.800	8.1	168			19	73
21-Aug-95	44	300	4.480	8.0	168			19	74
30-Aug-95	45	309	4.840	8.0	164			19	75
5-Sep-95	46	315	4.040	8.1	172	55		**19**	76
11-Sep-95	47	321	4.220	8.0	179	58		**21**	77
18-Sep-95	48	328	3.690	8.0	192	52		**25**	78
25-Sep-95	49	335	4.420	7.9	170	52		20	80
2-Oct-95	50	342	4.440	7.7	166	56		**15**	80
9-Oct-95	51	349	4.280	7.5	165	53		21	82
16-Oct-95	52	356	3.740	7.5	159	53		**26**	83
23-Oct-95	53	363	4.160	7.5	163	53		**25**	84
30-Oct-95	54	370	4.280	7.6	170	57		**14**	85

* Measured values of sulfate shown in bold

Compiled by: RWL\MLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

Table F-4.2

Leachate Analysis
Lower Mole Lake Waste Rock Master Composite (WD-08)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

LEACHATE ANALYSIS (mg/L) *																														
CYCLE	DAYS	Aluminum Al		Antimony Sb		Arsenic As		Barium Ba		Beryllium Be		Cadmium Cd		Calcium Ca		Chromium Cr		Cobalt Co		Copper Cu		Iron Fe		Lead Pb		Magnesium Mg		Manganese Mn		Mercury Hg		Nickel Ni		Potassium K		Selenium Se		Silver Ag		Sodium Na		Zinc Zn
		4.9	**<**	**0.050**		**0.110**		**0.017**	**<**	**0.005**	**<**	**0.005**		**26**		**0.005**		**0.006**		**0.110**		**7.900**	**<**	**0.040**		**20.0**		**0.240**		**0.0003**	**<**	**0.020**		**16.600**		**0.011**	**<**	**0.005**		**57**		**0.14**
1	0	**12.0**	**<**	**0.050**		**0.095**		**0.022**	**<**	**0.005**	**<**	**0.005**		**21**	**<**	**0.005**	**<**	**0.010**		**0.200**		**20.000**	**<**	**0.040**		**21.0**		**0.400**		**0.0002**	**<**	**0.020**		**13.400**		**0.009**	**<**	**0.005**		**36**		**0.20**
2	8	8.5		0.050		0.075		0.018		0.005		0.005		19		0.005		0.010		0.146		14.267		0.040		18.0		0.307		0.0002		0.030		12.000		0.008		0.005		27.7		0.148
3	15	5.1		0.050		0.055		0.013		0.005		0.005		18		0.005		0.010		0.091		8.533		0.040		15.0		0.213		0.0002		0.040		10.600		0.007		0.005		19.3		0.097
4	22	**1.6**	**<**	**0.050**		**0.035**		**0.009**	**<**	**0.005**	**<**	**0.005**		**16**	**<**	**0.005**	**<**	**0.010**		**0.037**		**2.800**	**<**	**0.040**		**12.0**		**0.120**	**<**	**0.0002**		**0.050**		**9.200**		**0.006**	**<**	**0.005**		**11**		**0.045**
5	27	1.8		0.050		0.034		0.009		0.005		0.005		16		0.005		0.010		0.038		2.950		0.040		11.5		0.125		0.0003		0.043		8.650		0.006		0.005		9.38		0.050
6	34	1.9		0.050		0.034		0.009		0.005		0.005		17		0.005		0.010		0.040		3.100		0.040		11.0		0.130		0.0004		0.035		8.100		0.006		0.005		7.75		0.055
7	41	2.1		0.050		0.033		0.008		0.005		0.005		17		0.005		0.010		0.041		3.250		0.040		10.5		0.135		0.0004		0.028		7.550		0.005		0.005		6.13		0.060
8	48	**2.2**	**<**	**0.050**		**0.028**		**0.008**	**<**	**0.005**	**<**	**0.005**		**17**	**<**	**0.005**	**<**	**0.010**		**0.042**		**3.400**	**<**	**0.040**		**10.0**		**0.140**		**0.0005**	**<**	**0.020**		**7.000**	**<**	**0.005**	**<**	**0.005**		**4.5**		**0.065**
9	55	2.1		0.051		0.028		0.008		0.005		0.005		17		0.006		0.010		0.039		3.163		0.038		9.8		0.140		0.0005		0.020		6.750		0.005		0.006		4.26		0.065
10	63	2.0		0.053		0.027		0.008		0.005		0.005		17		0.006		0.010		0.037		2.925		0.036		9.7		0.140		0.0004		0.020		6.500		0.005		0.006		4.03		0.065
11	70	1.8		0.054		0.027		0.008		0.005		0.005		17		0.007		0.010		0.034		2.688		0.034		9.5		0.140		0.0004		0.020		6.250		0.005		0.007		3.79		0.064
12	76	1.7		0.055		0.026		0.008		0.005		0.005		18		0.008		0.010		0.032		2.450		0.033		9.3		0.140		0.0004		0.020		6.000		0.005		0.008		3.55		0.064
13	84	1.6		0.056		0.026		0.007		0.005		0.005		18		0.008		0.010		0.029		2.213		0.031		9.1		0.140		0.0003		0.020		5.750		0.005		0.008		3.31		0.064
14	90	1.5		0.058		0.025		0.007		0.005		0.005		18		0.009		0.010		0.026		1.975		0.029		9.0		0.140		0.0003		0.020		5.500		0.005		0.009		3.08		0.064
15	97	1.3		0.059		0.025		0.007		0.005		0.005		18		0.009		0.010		0.024		1.738		0.027		8.8		0.140		0.0002		0.020		5.250		0.005		0.009		2.84		0.063
16	104	**1.2**	**<**	**0.060**		**0.024**		**0.007**	**<**	**0.005**		**0.005**		**18**	**<**	**0.010**	**<**	**0.010**		**0.021**		**1.500**	**<**	**0.025**		**8.6**		**0.140**	**<**	**0.0002**	**<**	**0.020**		**5.000**		**0.005**	**<**	**0.010**		**2.6**		**0.063**
17	111	1.1		0.060		0.024		0.007		0.005		0.005		18		0.010		0.010		0.020		1.418		0.027		8.5		0.133		0.0002		0.020		4.825		0.005		0.010		2.53		0.060
18	118	1.1		0.060		0.025		0.007		0.005		0.005		18		0.010		0.010		0.020		1.336		0.029		8.4		0.126		0.0002		0.020		4.650		0.005		0.010		2.46		0.058
19	125	1.0		0.060		0.025		0.007		0.005		0.005		19		0.010		0.010		0.019		1.254		0.030		8.2		0.120		0.0002		0.020		4.475		0.005		0.010		2.39		0.055
20	132	0.9		0.060		0.025		0.007		0.005		0.005		19		0.010		0.010		0.019		1.173		0.032		8.1		0.113		0.0002		0.020		4.300		0.005		0.010		2.33		0.052
21	139	0.9		0.060		0.026		0.006		0.005		0.005		19		0.010		0.010		0.018		1.091		0.034		8.0		0.106		0.0002		0.020		4.125		0.005		0.010		2.26		0.050
22	146	0.8		0.060		0.026		0.006		0.005		0.005		19		0.010		0.010		0.017		1.009		0.036		7.9		0.099		0.0002		0.020		3.950		0.005		0.010		2.19		0.047
23	153	0.8		0.060		0.026		0.006		0.005		0.005		19		0.010		0.010		0.017		0.927		0.037		7.7		0.092		0.0002		0.020		3.775		0.005		0.010		2.12		0.044
24	161	0.7		0.060		0.027		0.006		0.005		0.005		20		0.010		0.010		0.016		0.845		0.039		7.6		0.086		0.0002		0.020		3.600		0.005		0.010		2.05		0.042
25	167	0.6		0.060		0.027		0.006		0.005		0.005		20		0.010		0.010		0.015		0.763		0.041		7.5		0.079		0.0002		0.020		3.425		0.005		0.010		1.98		0.039
26	174	0.6		0.060		0.027		0.006		0.005		0.005		20		0.010		0.010		0.015		0.681		0.043		7.4		0.072		0.0002		0.020		3.250		0.005		0.010		1.91		0.036
27	181	0.6		0.060		0.027		0.006		0.005		0.005		20		0.010		0.010		0.014		0.599		0.044		7.2		0.065		0.0002		0.020		3.075		0.005		0.010		1.84		0.033
28	188	0.5		0.060		0.027		0.006		0.005		0.005		20		0.010		0.010		0.014		0.518		0.046		7.1		0.058		0.0002		0.020		2.900		0.005		0.010		1.78		0.031
29	195	0.4		0.060		0.028		0.006		0.005		0.005		20		0.010		0.010		0.013		0.436		0.048		7.0		0.051		0.0002		0.020		2.725		0.005		0.010		1.71		0.028
30	202	0.4		0.060		0.028		0.005		0.005		0.005		20		0.010		0.010		0.012		0.354		0.050		6.9		0.045		0.0002		0.020		2.550		0.005		0.010		1.64		0.025
31	209	0.3		0.060		0.028		0.005		0.005		0.005		21		0.010		0.010		0.012		0.272		0.051		6.7		0.038		0.0002		0.020		2.375		0.005		0.010		1.57		0.023
32	217	0.3		0.060		0.029		0.005		0.005		0.005		21	**<**	**0.010**	**<**	**0.010**		**0.011**		**0.190**		**0.053**		**6.6**		**0.031**	**<**	**0.0002**	**<**	**0.020**		**2.200**	**<**	**0.005**	**<**	**0.010**		**1.5**	**<**	**0.020**
33	223	**0.2**	**<**	**0.060**		**0.029**	**<**	**0.005**	**<**	**0.005**	**<**	**0.005**		**21**	**<**	**0.010**	**<**	**0.010**		**0.010**		**0.181**		**0.049**		**6.5**		**0.029**				**0.022**		**2.133**		**0.005**		**0.010**		**1.45**		**0.028**
34	230	0.2		0.060		0.028		0.005		0.005		0.005		21		0.010		0.010		0.010		0.177		0.048		6.5		0.028				0.023		2.100		0.005		0.010		1.42		0.031
35	237	0.2		0.060		0.028		0.005		0.005		0.005		21		0.009		0.009		0.010		0.172		0.047		6.5		0.027				0.024		2.067		0.005		0.010		1.39		0.035
36	244	0.2		0.060		0.028		0.005		0.005		0.005		21		0.009		0.009		0.010		0.168		0.045		6.4		0.026				0.025		2.033		0.005		0.010		1.37		0.039
37	251	0.2		0.060		0.028		0.005		0.005		0.005		21		0.009		0.009		0.009		0.163		0.044		6.4		0.026				0.026		2.000		0.005		0.010		1.34		0.043
38	258	0.2		0.060		0.027		0.005		0.005		0.005		21		0.009		0.009		0.009		0.159		0.043		6.4		0.025				0.027		1.967		0.006		0.010		1.31		0.047
39	265																																									
40	272	0.2		0.060		0.027		0.005		0.005		0.005		21		0.008		0.008		0.009		0.154		0.042		6.3		0.024				0.028		1.933		0.006		0.010		1.29		0.050
41	279	0.2		0.060		0.027		0.005		0.005		0.005		21		0.008		0.008		0.008		0.150		0.040		6.3		0.023				0.029		1.900		0.006		0.010		1.26		0.054
42	286	0.1		0.060		0.026		0.005		0.005		0.005		21		0.008		0.008		0.008		0.145		0.039		6.3		0.022				0.030		1.867		0.006		0.010		1.23		0.058
43	293	0.1		0.060		0.026		0.005		0.005		0.005		20		0.007		0.007		0.008		0.141		0.038		6.2		0.021				0.030		1.833		0.006		0.010		1.21		0.062
44	300	0.1		0.060		0.026		0.005		0.005		0.005		20		0.007		0.007		0.008		0.136		0.036		6.2		0.020				0.031		1.800		0.006		0.010		1.18		0.066
45	309	0.1		0.060		0.026		0.005		0.005		0.005		20		0.007		0.007		0.007		0.132		0.035		6.2		0.019				0.032		1.767		0.006		0.010		1.15		0.070
46	315	0.1		0.060		0.025		0.005		0.005		0.005		20		0.007		0.007		0.007		0.127		0.034		6.1		0.018				0.033		1.733		0.006		0.010		1.13		0.073
47	321	0.1		0.060		0.025		0.005		0.005		0.005		20		0.006		0.006		0.007		0.123		0.033		6.1		0.017				0.034		1.700		0.006		0.010		1.10		0.077
48	328	0.1		0.060		0.025		0.005		0.005		0.005		20		0.006		0.006		0.006		0.118		0.031		6.1		0.017				0.035		1.667		0.006		0.010		1.07		0.081
49	335	0.1		0.060		0.024		0.005		0.005		0.005		20		0.006		0.006		0.006		0.114		0.030		6.0		0.016				0.036		1.633		0.006		0.010		1.05		0.085
50	342	0.1		0.060		0.024		0.005		0.005		0.005		20		0.006		0.006		0.006		0.109		0.029		6.0		0.015				0.037		1.600		0.006		0.010		1.02		0.089
51	349	0.1		0.060		0.024		0.005		0.005		0.005		20		0.005		0.005		0.006		0.105		0.028		6.0		0.014				0.038		1.567		0.007		0.010		0.99		0.092
52	356	0.1		0.060		0.024		0.005		0.005		0.005		20		0.005		0.005		0.005		0.100		0.026		5.9		0.013				0.039		1.533		0.007		0.010		0.97		0.096
53	363	0.1		0.060		0.023		0.005		0.005		0.005		20		0.005		0.005		0.005		0.096		0.025		5.9		0.012				0.040		1.500		0.007		0.010		0.94		0.10
54	370	**0.1**	**<**	**0.060**		**0.023**	**<**	**0.005**	**<**	**0.005**		**0.005**		**20**	**<**	**0.005**	**<**	**0.005**	**<**	**0.005**		**0.096**	**<**	**0.025**		**5.9**		**0.012**				**0.040**		**1.500**		**0.007**	**<**	**0.010**		**0.94**		**0.10**

* Measured values shown in bold

Compiled by: RWL\MLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

Table F-4.3

Cumulative Mass Flux
Lower Mole Lake Waste Rock Master Composite (WD-08)

Test: **Leached under wet/dry alternating cycles**
Water Sample: **Bottom seepage**

		CUMULATIVE MASS FLUX (mg/kg sample)																				
CYCLE	DAYS	Aluminum Al	Antimony Sb	Arsenic As	Barium Ba	Beryllium Be	Cadmium Cd	Calcium Ca	Chromium Cr	Cobalt Co	Copper Cu	Iron Fe	Lead Pb	Magnesium Mg	Manganese Mn	Mercury Hg	Nickel Ni	Potassium K	Selenium Se	Silver Ag	Sodium Na	Zinc Zn
1	0	0.36	0.004	0.008	0.001	0.000	0.000	1.9	0.000	0.000	0.008	0.58	0.003	1.46	0.018	0.00002	0.001	1.2	0.001	0.000	4.2	0.01
2	8	1.08	0.007	0.014	0.003	0.0007	0.001	3.2	0.001	0.001	0.020	1.78	0.005	2.72	0.042	0.00003	0.003	2.0	0.001	0.001	6.3	0.02
3	15	1.59	0.010	0.018	0.004	0.0010	0.001	4.3	0.001	0.002	0.029	2.64	0.008	3.81	0.060	0.00005	0.004	2.7	0.002	0.001	8.0	0.03
4	22	1.90	0.013	0.022	0.004	0.0013	0.001	5.4	0.001	0.002	0.034	3.16	0.010	4.73	0.073	0.00006	0.007	3.4	0.002	0.001	9.2	0.04
5	27	1.99	0.016	0.024	0.005	0.0016	0.002	6.3	0.002	0.003	0.037	3.32	0.012	5.41	0.080	0.00007	0.010	3.9	0.003	0.002	9.8	0.04
6	34	2.11	0.019	0.026	0.006	0.0019	0.002	7.4	0.002	0.004	0.039	3.52	0.015	6.19	0.088	0.00009	0.013	4.5	0.003	0.002	10.4	0.04
7	41	2.21	0.021	0.028	0.006	0.0021	0.002	8.2	0.002	0.004	0.041	3.67	0.017	6.73	0.095	0.00011	0.014	4.9	0.003	0.002	10.8	0.05
8	48	2.33	0.025	0.030	0.006	0.0025	0.002	9.3	0.002	0.005	0.044	3.88	0.020	7.38	0.103	0.00013	0.016	5.4	0.004	0.002	11.2	0.05
9	55	2.44	0.027	0.031	0.007	0.0027	0.003	10.1	0.003	0.005	0.046	4.04	0.022	7.86	0.110	0.00016	0.017	5.7	0.004	0.003	11.4	0.05
10	63	2.56	0.030	0.033	0.007	0.0030	0.003	11.0	0.003	0.006	0.048	4.21	0.024	8.41	0.118	0.00018	0.018	6.1	0.004	0.003	11.7	0.06
11	70	2.68	0.033	0.034	0.008	0.0033	0.003	12.1	0.003	0.006	0.050	4.40	0.026	9.01	0.126	0.00021	0.019	6.5	0.004	0.003	11.9	0.06
12	76	2.77	0.036	0.036	0.008	0.0035	0.004	13.0	0.004	0.007	0.052	4.53	0.028	9.48	0.133	0.00023	0.020	6.8	0.005	0.004	12.1	0.06
13	84	2.85	0.039	0.037	0.009	0.0038	0.004	13.9	0.004	0.007	0.053	4.66	0.029	9.95	0.141	0.00025	0.021	7.1	0.005	0.004	12.3	0.07
14	90	2.94	0.041	0.038	0.009	0.0040	0.004	14.8	0.005	0.008	0.055	4.77	0.031	10.43	0.148	0.00026	0.022	7.4	0.005	0.005	12.5	0.07
15	97	3.04	0.045	0.040	0.009	0.0044	0.004	16.0	0.005	0.008	0.057	4.91	0.033	11.04	0.157	0.00028	0.024	7.8	0.006	0.005	12.7	0.07
16	104	3.04	0.046	0.040	0.009	0.0044	0.004	16.1	0.005	0.009	0.057	4.92	0.033	11.10	0.158	0.00028	0.024	7.8	0.006	0.005	12.7	0.07
17	111	3.18	0.052	0.043	0.010	0.0050	0.005	18.1	0.006	0.010	0.059	5.08	0.036	12.04	0.174	0.00030	0.026	8.4	0.006	0.006	13.0	0.08
18	118	3.26	0.057	0.044	0.011	0.0053	0.005	19.4	0.007	0.010	0.061	5.18	0.038	12.65	0.183	0.00032	0.028	8.7	0.006	0.007	13.1	0.09
19	125	3.31	0.060	0.046	0.011	0.0056	0.006	20.3	0.008	0.011	0.062	5.25	0.039	13.07	0.190	0.00033	0.029	8.9	0.007	0.008	13.3	0.09
20	132	3.38	0.064	0.047	0.012	0.0059	0.006	21.6	0.008	0.012	0.063	5.34	0.041	13.65	0.198	0.00034	0.030	9.3	0.007	0.008	13.4	0.09
21	139	3.44	0.067	0.049	0.012	0.0062	0.006	22.7	0.009	0.012	0.064	5.41	0.043	14.12	0.205	0.00035	0.031	9.5	0.007	0.009	13.6	0.10
22	146	3.49	0.071	0.050	0.012	0.0065	0.007	23.8	0.009	0.013	0.065	5.47	0.045	14.58	0.211	0.00037	0.032	9.7	0.008	0.009	13.7	0.10
23	153	3.54	0.074	0.052	0.013	0.0068	0.007	24.9	0.010	0.013	0.066	5.53	0.047	15.04	0.217	0.00038	0.033	10.0	0.008	0.010	13.8	0.10
24	161	3.58	0.078	0.053	0.013	0.0071	0.007	26.1	0.011	0.014	0.067	5.58	0.049	15.50	0.222	0.00039	0.035	10.2	0.008	0.011	14.0	0.10
25	167	3.63	0.082	0.055	0.013	0.0075	0.007	27.5	0.011	0.015	0.068	5.64	0.052	16.04	0.228	0.00040	0.036	10.5	0.009	0.011	14.1	0.11
26	174	3.66	0.085	0.057	0.014	0.0077	0.008	28.5	0.012	0.015	0.069	5.68	0.054	16.44	0.232	0.00041	0.037	10.6	0.009	0.012	14.2	0.11
27	181	3.70	0.089	0.058	0.014	0.0080	0.008	29.7	0.012	0.016	0.070	5.73	0.057	16.88	0.237	0.00043	0.038	10.8	0.009	0.012	14.3	0.11
28	188	3.73	0.093	0.060	0.014	0.0083	0.008	31.0	0.013	0.016	0.071	5.76	0.060	17.35	0.241	0.00044	0.040	11.0	0.009	0.013	14.4	0.11
29	195	3.76	0.096	0.062	0.015	0.0086	0.009	32.2	0.014	0.017	0.072	5.80	0.062	17.76	0.244	0.00045	0.041	11.2	0.010	0.014	14.5	0.12
30	202	3.78	0.100	0.064	0.015	0.0090	0.009	33.5	0.014	0.018	0.072	5.82	0.065	18.21	0.248	0.00046	0.042	11.4	0.010	0.014	14.7	0.12
31	209	3.80	0.103	0.065	0.015	0.0092	0.009	34.5	0.015	0.018	0.073	5.84	0.068	18.53	0.250	0.00047	0.043	11.5	0.010	0.015	14.7	0.12
32	217	3.81	0.107	0.067	0.016	0.0095	0.010	35.8	0.015	0.019	0.074	5.86	0.071	18.97	0.252	0.00049	0.044	11.6	0.011	0.015	14.8	0.12
33	223	3.82	0.110	0.068	0.016	0.0098	0.010	36.9	0.016	0.019	0.074	5.87	0.074	19.29	0.254	0.00050	0.045	11.8	0.011	0.016	14.9	0.12
34	230	3.83	0.113	0.070	0.016	0.010	0.010	38.0	0.016	0.020	0.075	5.88	0.076	19.65	0.255		0.047	11.9	0.011	0.016	15.0	0.12
35	237	3.84	0.117	0.071	0.017	0.010	0.010	39.2	0.017	0.020	0.075	5.89	0.079	20.01	0.257		0.048	12.0	0.011	0.017	15.1	0.12
36	244	3.85	0.120	0.073	0.017	0.011	0.011	40.3	0.017	0.021	0.076	5.90	0.082	20.36	0.258		0.049	12.1	0.012	0.018	15.1	0.13
37	251	3.86	0.123	0.074	0.017	0.011	0.011	41.3	0.018	0.021	0.076	5.90	0.084	20.66	0.259		0.050	12.2	0.012	0.018	15.2	0.13
38	258	3.87	0.126	0.076	0.017	0.011	0.011	42.6	0.018	0.022	0.077	5.91	0.086	21.06	0.261		0.052	12.3	0.012	0.019	15.3	0.13
39	265																					
40	272	3.88	0.132	0.079	0.018	0.012	0.012	44.6	0.019	0.023	0.078	5.93	0.091	21.70	0.264		0.054	12.5	0.013	0.020	15.4	0.13
41	279	3.89	0.135	0.080	0.018	0.012	0.012	45.7	0.020	0.023	0.078	5.94	0.093	22.03	0.265		0.056	12.6	0.013	0.020	15.5	0.14
42	286	3.90	0.139	0.081	0.018	0.012	0.012	46.8	0.020	0.023	0.079	5.95	0.095	22.37	0.266		0.057	12.7	0.014	0.021	15.6	0.14
43	293	3.91	0.142	0.083	0.019	0.012	0.012	48.1	0.021	0.024	0.079	5.95	0.098	22.75	0.267		0.059	12.8	0.014	0.021	15.6	0.14
44	300	3.92	0.146	0.084	0.019	0.013	0.013	49.2	0.021	0.024	0.080	5.96	0.100	23.11	0.269		0.061	12.9	0.014	0.022	15.7	0.15
45	309	3.93	0.150	0.086	0.019	0.013	0.013	50.5	0.021	0.025	0.080	5.97	0.102	23.50	0.270		0.063	13.1	0.015	0.023	15.8	0.15
46	315	3.93	0.153	0.087	0.020	0.013	0.013	51.6	0.022	0.025	0.081	5.98	0.104	23.82	0.271		0.065	13.1	0.015	0.023	15.8	0.16
47	321	3.94	0.156	0.089	0.020	0.014	0.014	52.7	0.022	0.025	0.081	5.99	0.106	24.15	0.272		0.067	13.2	0.015	0.024	15.9	0.16
48	328	3.94	0.159	0.090	0.020	0.014	0.014	53.6	0.022	0.026	0.081	5.99	0.107	24.44	0.273		0.068	13.3	0.016	0.024	16.0	0.16
49	335	3.95	0.162	0.091	0.020	0.014	0.014	54.8	0.023	0.026	0.082	6.00	0.109	24.79	0.274		0.070	13.4	0.016	0.025	16.0	0.17
50	342	3.96	0.166	0.093	0.021	0.014	0.014	55.9	0.023	0.026	0.082	6.00	0.111	25.13	0.274		0.072	13.5	0.016	0.025	16.1	0.17
51	349	3.96	0.169	0.094	0.021	0.015	0.015	57.1	0.023	0.027	0.082	6.01	0.112	25.46	0.275		0.074	13.6	0.017	0.026	16.1	0.18
52	356	3.97	0.172	0.095	0.021	0.015	0.015	58.0	0.024	0.027	0.083	6.02	0.114	25.75	0.276		0.076	13.7	0.017	0.026	16.2	0.18
53	363	3.97	0.175	0.096	0.021	0.015	0.015	59.1	0.024	0.027	0.083	6.02	0.115	26.07	0.277		0.078	13.8	0.017	0.027	16.2	0.19
54	370	3.98	0.178	0.098	0.022	0.015	0.015	60.2	0.024	0.028	0.083	6.03	0.116	26.39	0.277		0.080	13.8	0.018	0.027	16.3	0.19

Compiled by: RWL\WLD2
Checked by: JET
Updated by: JPS
Checked by: KSS

ATTACHMENT G

SPECIFIC SURFACE AREA CALCULATIONS FOR COLUMN TESTS OF HANGING WALL ROCK

ATTACHMENT G
SPECIFIC SURFACE AREA CALCULATIONS FOR COLUMN TESTS OF HANGING WALL ROCK

Each wall rock formation was submitted for sieve analysis. Specific surface area (SSA) was calculated as follows:

SSA (m^2/kg) = sum(SSA of material retained on each sieve (m^2/kg))

Where:

$$\text{SSA (material per sieve)} = \frac{\text{SSA per particle} * \text{weight retained on sieve (kg)}}{\text{total sample weight (kg)}}$$

And:

$$\text{SSA (per particle on sieve)} = \frac{\text{surface area of sphere}}{\text{mass of sphere}}$$

Definitions:

Surface area of sphere – assume all particles are spheres with diameter of the mean of the sieve on which the particle was retained and the sieve above.

Mass of sphere – assumes the density of the sphere is 2.5 kg/m^3

An example calculation using the Lower Mole Lake formation data is presented in the table below:

Sieve Fraction m	Median Diam. m	Wt. Retained lbs	Wt. Retained kg	Vol (1 grain) m^3	SA (1 grain) m^2	Mass(D=2.5) kg	m^2/kg	Distrib m^2
0.0381	0.14605	0	0	0.0016	0.067	4.1	0.016	0
0.0254	0.03175	1.0	0.45	1.67E-05	0.0032	0.042	0.076	0.00040
0.009525	0.017463	94	43	2.79E-06	0.00096	0.0070	0.14	0.076
0.0047	0.007113	31	14	1.88E-07	0.00016	0.00047	0.34	0.061
0.0017	0.0032	19	8.8	1.71E-08	3.22E-05	4.29E-05	0.75	0.085
0.001	0.00135	8.8	4.0	1.29E-09	5.73E-06	3.22E-06	1.8	0.091
0.0002	0.0006	11	5.1	1.13E-10	1.13E-06	2.83E-07	4.0	0.26
0.000074	0.000137	2.4	1.1	1.35E-12	5.9E-08	3.37E-09	17	0.25
0.000001	0.000038	2.5	1.1	2.76E-14	4.42E-09	6.90E-11	64	0.94
Total		171	78					1.8 (m2/kg)
						Specific Surface Area		**19 (ft2/kg)**

ATTACHMENT H

MEMORANDUM RE: THIOSALTS IN PASTE BACKFILL

SRK Consulting
Engineers and Scientists

Steffen Robertson and Kirsten (Canada.) Inc.
800, 580 Hornby Street
Vancouver, BC.
V6C 3B6

email: vancouver@srk.com
URL: http://www.srk.com
Tel: 604.681.4196
Fax: 604.687.5532

MEMORANDUM

DATE: November, 2000

TO: File 1UF001.01 Task 18

FROM: Daryl Hockley

RE: Thiosalts in Paste Backfill

1. INTRODUCTION

Thiosalts are defined as metastable sulfur oxyanions containing S-S bonds. The term "thiosalts" covers thiosulfate ($S\text{-}SO_3^=$ or $S_2O_3^=$) and polythionates ($SO_3\text{-}S_n\text{-}SO_3^=$). Thiosalts are intermediate products formed during conversion of sulfide to sulfate, and are therefore produced in varying quantities during processing of sulfide ores. When solutions containing thiosalts are stored under oxidizing conditions, for example in tailings or holding ponds, subsequent oxidation of thiosalts generate sulfate and acidity.

The remainder of this memorandum reviews data from the Crandon Project, and recommends a range of estimates of thiosalt concentrations for use in the reflooded mine management plan.

2. THIOSALTS IN CRANDON PROCESS WATER

2.1 Literature

A bibliography of literature on thiosalts has recently been compiled (Canmet 2000). The literature includes reports on studies at several base metal mines. In brief, the studies indicate that the milling processes such as grinding, conditioning, flotation and dewatering contribute to thiosalts generation. More specifically, the following factors have been found to promote the generation of thiosalts: high sulfide content of ore; finer grain size; alkaline grinding solution; higher dissolved oxygen in grinding solution; longer retention time; higher temperature; lower pulp density; increased agitation rate during flotation; flotation reagents and SO_2 additions. It is also known that thiosalts are removed naturally from effluent in holding ponds at warmer temperatures and extended retention times.

Two problems limit the direct application of literature values to the Crandon Project. First, recent studies on methods of thiosalt analysis suggest that much of the older data, and even some of the newer data, is questionable. Second, the many possible combinations of the above factors result in significant variability in thiosalt concentrations, both between different mines and within individual circuits.

2.2 Project Documents

In the Crandon metallurgical and environmental studies, only limited testing was conducted with respect to thiosalt generation, and several of the reports pertain to the project as it was proposed in 1986. The results are summarized here because many of the processes in the 1986 proposal are included in the current project. However, the high rates of recycle in the current process configuration will mean that thiosalt concentrations throughout the mill, TMA and backfill will represent some average of those measured in the various individual processes.

2.2.1 Lakefield Study of Thiosalt Degradation

In one study, (Lakefield Research Progress Report #16), a water sample was collected from the zinc ore tailings after zinc flotation. The sample was analyzed for both sulfate and thiosulfate immediately after flotation and then at two-week intervals. The analytical method is described in Lakefield Report #31 and was stated to not distinguish between thiosulfate and polythionates. Results are shown in Table 1.

TABLE 1
Thiosulfate Concentrations Measured in Lakefield Test of Thiosalt Degradation

Parameter (mg/L)	1 Hour	2 Weeks	4 Weeks	6 Weeks	8 Weeks	10 Weeks
$SO_4^=$	216	254	262	299	310	345
$S_2O_3^=$	52	10.6	7.2	4.9	26	10

2.2.2 Lakefield Study of Thiosalt Influence on Zinc Recovery

Testing was also conducted (Lakefield Research Progress Report #38) to determine the effect of increased thiosulfate concentration on metal recovery. The flotation test work was with water that contained thiosulfate added as $Na_2S_2O_3 \times 5H_2O$. The levels of $S_2O_3^=$ tested were 1600 mg/L, 3200 mg/L and 6400 mg/L, respectively. Results indicated that all of these concentrations resulted in depression of sulfide flotation and therefore loss of recovery of zinc.

2.2.3 Lakefield Pilot Plant Study

Water analyses were taken during the 1986 pilot plant test program on Crandon zinc ore at Lakefield (Progress Report No. 2, Lakefield Project No. L.R. 3106). Results are shown in Table 2.

TABLE 2
Thiosulfate Concentrations Measured in Lakefield 1986 Pilot Test Program

Parameter (mg/L)	Comb Zn Tail Water	Pb Conc Water	Lakefield Water	Zn Conc Water	Cu Conc Water	Pb Thick. O/F Water
$SO_4^=$	98.8	32.9	16.5	29.6	44.5	543
$S_2O_3^=$	54.7	9	2	9.9	12.5	78

2.2.4 CH2M Estimates for 1986 EIS

Thiosulfate concentrations were included in estimates of water quality reported in the November, 1985 Water Treatment Facility Engineering Report for the Crandon Project by CH2M Hill. The milling process at that time included physical separation of high pH zinc flotation tailings to produce coarse material for hydraulic backfill and fine material for TMA disposal. The concentration estimates for various streams are shown in Table 3.

TABLE 3
Thiosulfate Concentrations Estimated for 1986 EIS

Parameter (mg/L)	Mine Seepage	Utility Water	Backfill Drainage	Total Mine Water	Reclaim Pond Water
$SO_4^=$	200	9	1400	390	1400
$S_2O_3^=$	<1	<1	560	85	460

2.3 Estimates Derived from Waste Characterization Program

Appendix A of the Crandon Mine TMA Groundwater Quality Performance Evaluation (Foth and Van Dyke 2000) presents results from the testing of depyritized tailings, pyrite concentrate and cemented pyritic paste backfill. Although no thiosalt analyses were completed, it is possible to calculate estimates of thiosalt concentrations from the available data.

2.3.1 Depyritized Tailings

Analyses of process water and pore water samples from the depyritized tailings consistently showed a mass balance difference between total sulfur and sulfate analyses. The presence of thiosalts is one explanation for the difference. Table 4 shows the thiosulfate concentration estimates that result when it is assumed that all of the difference between measured total sulfur and measured sulfate is associated with thiosulfate. (n.b. the numbers are not directly additive due to the stoichiometric corrections.)

Humidity cell tests on the depyritized tailings showed a much smaller difference between total sulfur and sulfate, so the data cannot be used to estimate thiosulfate.

TABLE 4
Thiosulfate Concentrations Estimated from Depyritized Tailings Testing

Parameter (mg/L)	Depyritized Tailings	
	Process Water	Equil. Pore Water
Sulfate	490	901
Sulfur	219	343
$S_2O_3^=$ (calc)	97	75

2.3.2 Pyrite Concentrate Process Water and Equilibrated Pore Water

Analyses of process water and pore water samples from the pyrite concentrate also showed a difference between total sulfur and sulfate analyses. Table 5 shows the thiosulfate concentration estimates that result when it is assumed that the difference between measured total sulfur and measured sulfate sulfur is accounted for by thiosulfate.

TABLE 5
Thiosulfate Concentrations Estimated from Pyrite Concentrate Testing

Parameter (mg/L)	Pyrite Concentrate 1		Pyrite Concentrate 2	
	Process Water	Equil. Pore Water	Process Water	Equil. Pore Water
Sulfate	490	650	620	790
Sulfur	208	228	282	291
$S_2O_3^=$ (calc)	78	20	132	48

2.3.3 Cemented Paste Backfill

Figure 1 shows the results of similar calculations used to estimate thiosulfate concentrations in the humidity cell test on cemented pyritic paste backfill. The highest estimated thiosulfate concentration is 762 mg/L, in the first cycle of test CPT-IT1. The estimates from other tests show initial thiosulfate concentrations of approximately 400 mg/L, with an apparent increase to approximately 500 mg/L in subsequent cycles, and a sharp decrease thereafter. The pattern agrees with observations of thiosulfate production and degradation in the literature.

There will be no excess water associated with the cemented pyritic paste backfill, therefore no backfill "process water" was produced in the testing program. Samples of cemented pyritic paste backfill were crushed and leached in a saturated column test CPT-SC, to produce "equilibrated pore water". Results from the test showed very significant differences between total sulfur and sulfate analyses, suggesting analytical errors. Hence it is not reasonable to estimate thiosulfate concentrations by the method used in the above sections. An alternative method is to calculate the amount of thiosulfate needed to achieve a good ion balance. This method is made difficult by the chloride analysis, which was clearly erroneous. However, if the reported concentration of 1900 mg/L chloride is replaced by an estimate of 188 mg/L derived from the solids analyses (which showed excellent repeatability), the ion balance becomes more reasonable. In that case the amount of thiosulfate needed to achieve a complete charge balance is about 1157 mg/L.

2.4 Recommended Estimates of Thiosalts in Process Water

The above data and estimates show a wide variability. There are a number of results showing very low thiosulfate concentrations (less than 100 mg/L), one result showing a high concentration (1157 mg/L from the ion balance on test CPT-SC), and a number of results showing thiosulfate concentrations near 500 mg/L (the 1986 EIS estimates and the humidity cell tests on cemented pyritic paste backfill).

Such variability is in fact quite common in the thiosulfate literature. However, some of the variability in this case is certainly due to the fact that many different individual processes have been tested. In the Crandon Project mill and TMA system, the high rates of recycle will serve to smooth out some of the variability. It is therefore reasonable to expect that neither the extreme high values nor the extreme low values in the above data will be characteristic of average thiosulfate concentrations in the full scale process water.

We therefore recommend the intermediate value of 500 mg/L as a BEJ estimate of the average thiosulfate concentration in process water.

As an upper bound estimate of average thiosulfate concentration in process water we recommend 1000 mg/L, approximately halfway between the two highest values in the data set (1157 mg/L from test CPT-SC, and 762 mg/L from the humidity cell test CPT-IT1). This value is approaching the 1600 mg/L at which the Lakefield Report #38 indicated impairment of zinc recovery. Hence there will be an economic incentive for NMC to keep thiosalt levels within the suggested range.

3. THIOSALTS IN BACKFILL

Some of the water from the mill process will be contained in the paste backfill that will be placed in the underground mine. That water will contain thiosalts. The subsequent reactions that the thiosalts will undergo will depend on the conditions in the backfill and the surrounding geologic media.

3.1 Reactions in Backfill

The backfill itself will be an alkaline medium with a low redox potential and an excess of solid phase neutralizing potential. The low redox potential will make it unlikely that any thiosalt deposited with the backfill will oxidize rapidly. However, given the very long time frames involved, it is not possible to rule out conversion of the thiosalts to more stable phases. That conversion may consist of (very slow) oxidation to sulfate, reduction to elemental sulfur or sulfide, or some combination of the two.

If it is assumed that some portion of the thiosulfate in the backfill can oxidize, the production of acidity and sulfate will be produced. The acidity is not a cause for concern because it will be neutralized by the excess NP. In contrast, any sulfate produced from thiosulfate oxidation could conceivably contribute to the sulfate load to the surrounding groundwater. Some of the new sulfate will be removed by the precipitation of gypsum, which is predicted to be present in the backfill. It can be shown that, in the presence of calcium-based NP and gypsum, the oxidation of one mole of thiosulfate results in less than one mole of soluble sulfate.

Reduction of the thiosalts to elemental sulfur or sulfide is not expected to have any deleterious effects on water quality in the backfill.

3.2 Reactions in Bedrock

Some portion of the thiosalts will diffuse or be leached out of the backfill and into the surrounding bedrock. Like the backfill, the bedrock groundwater will also have a low redox potential, suggesting that any oxidation of the thiosalts will be very slow.

The neutralizing potential of the bedrock is variable. For example, waste rock test showed the Mole Lake formations to have relatively high NP's of approximately 45 kg $CaCO_3$ equiv./tonne, while the Skunk Lake and Rice Lake formations had NP's of less than 2 kg $CaCO_3$ equiv./tonne. Even with the lower NP values, the bedrock will contain enough NP to neutralize any acidity that can be produced by thiosalt oxidation.

Any sulfate produced from thiosalt oxidation in the bedrock will remain in solution. Under these conditions, the oxidation of one mole of thiosulfate will produce two moles of sulfate.

3.3 Reactions in Glacial Overburden

Any thiosalts that are transported into the glacial overburden will likely be exposed to more oxidizing conditions than are present in the backfill or bedrock. Oxidation of the thiosalts will occur, although the reactions are likely to remain slow. The acidity produced by the oxidation of thiosalts will be consumed by alkalinity in the groundwater, and/or by reactions with soil minerals. The sulfate is likely to remain in solution. Under these conditions, oxidation of one mole of thiosulfate will produce two moles of sulfate.

4. RECOMMENDED ADDITION TO BEJ AND UB SOURCE TERMS FOR BACKFILL

The current estimate of sulfate concentration in the pyritic paste backfill is between 1,250 mg/L and 1,317 mg/L for BEJ and UB conditions, respectively. As mentioned in Section 2.4, our recommended BEJ estimate of thiosalt concentration in Crandon project process water is 500 mg/L thiosulfate, and our recommended UB estimate is 1000 mg/L thiosulfate.

As discussed in Section 3, there are a number of scenarios whereby the thiosalt may oxidize to produce sulfate. Therefore, we recommend that the current estimates of sulfate concentration in the backfill be increased to account for the potential thiosalt contribution. Although the thiosalts will not necessarily convert to sulfate, this approach will lead to conservative estimates of sulfate concentrations in the groundwater down-gradient of the reflooded mine. The recommended estimates are shown in Table 6.

TABLE 6
Recommended Sulfate in Pyritic Paste Backfill

	Units	BEJ	UB
[1]Initial porewater $SO_4^=$	mg/L	1250	1317
[2]Estimates of $S_2O_3^=$ in process water	mg/L	500	1000
[3]Dilution of backfill process water into reflooded pore water	unitless factor	0.8	0.8
[4]Resulting $S_2O_3^=$ in backfill pore water	mg/L	400	800
[5]Maximum SO_4 that can be formed from $S_2O_3^=$	mg/L	686	1371
[6]Total potential $SO_4^=$	mg/L	1936	2688

Prepared by: DEH
Checked by: DDS

Footnotes to Table 6:

1. *From Table 2.13 of Reflooded Mine Source Concentration.*

2. *From Section 2.4 of this memorandum.*

3. *Assumes backfill placed at 16% water content and reflooded to 20% water content:*
   16% / 20% = 0.8

4. *Dilution by reflooding water:*
   500 mg/L x 0.8 = 400 mg/L

5. *Conservatively assumes 1 mole of thiosulfate (molar weight of 112) produces 2 moles of sulfate (molar weight of 96 g):*
   (2 x 96 g / 112 g) x 400 mg/L = 686 mg/L

6. *Sum of 1 and 5:*
   1250 mg/L + 686 mg/L = 1936 mg/L

ATTACHMENT I

FUEL OIL METAL COMPOSITION

Fuel Oil Metal Composition

Constituent	Concentration (ppmw)
Arsenic	0.085
Beryllium	0.05
Cadmium	0.21
Chromium (Total)	0.95
Copper	5.6
Lead	0.18
Manganese	0.28
Mercury	0.06
Nickel	3.4

ppmw = parts per million weight

Source: *Estimating Air Toxics from Coal and Oil Combustion*, EPA-450/2-89/001.

ATTACHMENT J

RESIDUAL NITROGEN MASS BALANCE CALCULATION

ATTACHMENT J
RESIDUAL NITROGEN MASS BALANCE CALCULATIONS

Ammonium Nitrate Balance for Crandon Mine

Assumptions

Conversions

1 lb =	0.4536	kg
NH4NO3 to inorg N	0.35	

Nitrogen Speciation (Ferguson and Leask)

NH4-N	10%
NO3-N	87%
NO2-N	3%

Explosives Use

ANFO use in Waste	2.2	lb/ton
ANFO use in Ore	1	lb/ton
AN content	93%	
Waste Rock	2.046	lb/ton
Ore	0.93	lb/ton

Unexploded ANFO 10%

Calculations

NH4NO3 Residual

Waste Rock	100	ton/day	0.205 lb/ton	20.5 lb/day	0.006 kg/min
Ore	5500	ton/day	0.093 lb/ton	511.5 lb/day	0.161 kg/min
Total					0.168 kg/min

Distribution of AN		USGPM	L/min	Dist.	kg AN/min	mg AN/L
Moisture in Ore	MLG	38	144	8.6%	0.0145	101
To Minewater Treatment	MNH	397	1,503	90.2%	0.1512	101
Mine Drainage Water	MNJ	5	19	1.1%	0.0019	101
		440	1,666	100.0%	0.1676	101

Loading to the underground

AN in Backfill Water	67	mg/L
Inorganic N in Backfill Water	23.4	mg/L
NH4-N	2.3	mg/L
NO3-N	20.4	mg/L
NO2-N	0.70	mg/L
NO3-N + NO2-N	21.1	mg/L

Water Balance

Node Name	Input / Output	Stream Name	Stream ID	Stream flow USGPM	Stream flow L/min	Amm. Nitrate concentration mg/L	Amm. Nitrate mass flow kg/min
Mill	Input	Moisture in Ore	MLG	38	144	**101**	0.014
	Input	Reagent Water	MLF	52	197	67	0.013
	Input	Seal/Gland Water	MLE	260	984	67	0.066
	Input	Process Water	MLO	3,546	13,423	67	0.894
	Output	Loss to Zinc Concentrate	MLA	26	98	67	0.007
	Output	Loss to Pyrite Concentrate	ML?	92	348	**67**	0.023
	Output	Thickener Overflow	MLC	1,411	5,341	67	0.358
	Output	**Flotation Tailings**	MLB	**2,367**	**8,960**	**67**	**0.600**
Tailings Prep	Input	Flotation Tailings	MLB	2,367	8,960	67	0.600
	Input	Sewage Liquid Waste	PPB	9	34	0	0.000
	Input	Service Building Liquid Waste	PWB	6	23	0	0.000
	Input	Pipeline Containment Drainage	TPE	8	30	0	0.000
	Input	W.T. Sludge & Filter Backwash	WTC	58	220	**101**	0.022
	Output	**Flow to Tailings Pond**	TPA	**2,448**	**9,267**	**67**	**0.622**
Tailings Pond	Input	Flow to Tailings Pond	TPA	2,448	9,267	67	0.622
	Input	Precipitation Input	TPB	98	371	0	0.000
	Output	NO_3 removal (precip./denitrification)					0.000
	Output	Evaporation Output	TPC	73	276	0	0.000
	Output	Retained Moisture (pore water)	TPD	34	129	66	0.009
	Output	**Flow to Reclaim Pond**	PSA	**2,439**	**9,233**	**66**	**0.613**
Reclaim Pond	Input	Flow to Reclaim Pond	PSA	2,439	9,233	66	0.613
	Input	Precipitation Input	PSB	8	30	0	0.000
	Output	Evaporation Output	PSC	5	19	0	0.000
	Output	**Bulk Reclaim Water**	PSE	**2,442**	**9,244**	**66**	**0.613**
Bleed to Treament	Input	Bulk Reclaim Water	PSE	2,442	9,244	66	0.613
	Output	Bleed to Water Treatment	PSG ?	0	0	66	0.000
	Output	**Recycled Reclaim Water**	PSF	**2,442**	**9,244**	**66**	**0.613**
Process Water Prep	Input	Recycled Reclaim Water	PSF	2,442	9,244	66	0.613
	Input	Thickener Overflow	MLC	1411	5,341	67	0.358
	Input	Mine Drainage Water	MNJ	5	19	**101**	0.002
	Output	Reagent Water	MLF	52	197	67	0.013
	Output	Seal/Gland Water	MLE	260	984	67	0.066
	Output	**Process Water**	MLO	**3,546**	**13,423**	**67**	**0.894**

1 US Gallon = 3.7854 litres

ATTACHMENT K

RESPONSES TO JULY 1, 1999 COMMENTS BY WISCONSIN DEPARTMENT OF NATURAL RESOURCES

ATTACHMENT K
RESPONSES TO JULY 1, 1999 COMMENTS BY WISCONSIN DEPARTMENT OF NATURAL RESOURCES

Comment 1: Please revise the reflooded mine source term derivation (Appendix 4.2-17) to be consistent with the changes in the mine plan involving grouting and changes in the expected time frame for reflooding, taking into account the grout and potential mitigation water pumping. As a part of that work, please reassess the assumed 15-year average time of exposure for the wall rock. This assumption had not been adequately justified with reference back to the mine plan before these project changes.

Response 1: *HSI GeoTrans (1999) completed an analysis evaluating mitigation pumpage effects on the recovery of the groundwater system after mining. The analysis shows that mitigation pumping has negligible effects on groundwater recovery. The groundwater system is predicted to be substantially recovered in approximately four years. Most of this time is attributed to the time it takes to resaturate the glacial overburden. The bedrock and mine will be saturated in a much shorter time frame, likely within two years. Note that the analysis was based on Nicolet Minerals Company's (NMC) practical worst case (PWC) model (HSI GeoTrans, 1998) which incorporates grouting.*

With respect to the previously assumed 15-year average time of exposure for the wall rock, NMC has adjusted this based on the mine development schedule in the different formations. Exposure times of 22 years (Lower Mole Lake), 4 years (Upper Mole Lake), 29 years (Rice Lake) and 30 years (Skunk Lake) were assumed in the calculation. Please see Section 3.3.4 of this report.

Comment 2: The reflooded mine source term derivation (Appendix 4.2-17) assumes that the neutralization potential (NP) not utilized in the oxidation of the backfill will react with any remaining oxidation products during the reflooding process. It is not clear that flow paths through the reflooded mine would ensure that those reactions would take place. Please revise the narrative to discuss the implications on the estimated water chemistry if the oxidation products do not have an opportunity to react with the remaining NP in the backfill.

Response 2: *The reflooded mine source concentration does not assume that the residual NP will react with oxidation products during reflooding. On the contrary, the derivation conservatively assumes that any solutes transported from the acidic backfill will remain in solution, even if the acidic water mixes with neutral groundwater and/or alkaline water from the unoxidized backfill.*

Comment 3: Please revise the geochemical modeling presented in the reflooded mine source term derivation (Appendix 4.2-17) to include the revisions to the thermodynamic database requested in our last comment letter on waste characterization/GWQPE, dated 3/16/99.

Response 3: *The new MINTEQA2 modeling in the Reflooded Mine Source Concentration Report (October 2000) uses the revised thermodynamic database.*

Comment 4: Please provide information on the speciation of the carbonate expected to be present within the backfilled tailings. Revise the appropriate calculations in the reflooded mine source term derivation (Appendix 4.2-17) to take the carbonate speciation into account in the determination of the available NP used in assessing the extent of oxygen penetration and the oxidation products produced.

Response 4: *See Response 17.*

Comment 5: The copper phase used in the geochemical modeling presented in the reflooded mine source term derivation (Appendix 4.2-17), tenorite, may not be reasonable for the materials and conditions expected in the mine. Please provide justification for the use of tenorite as a controlling phase or revise the model runs to eliminate tenorite.

Response 5: *The copper phase tenorite was excluded from the new MINTEQA2 runs in the Reflooded Mine Source Concentration Assessment (October, 2000). The phase brochantite was allowed to form but was not supersaturated in any of the current runs.*

Comment 6: The antimony hydroxide phase used in the geochemical modeling presented in the reflooded mine source term derivation (Appendix 4.2-17), may not be reasonable for the materials and conditions expected in the mine. Please provide

justification for the use of solid antimony hydroxide as a controlling phase or revise the model runs to eliminate antimony hydroxide.

Response 6: *The antimony hydroxide phase was not allowed to form in the new MINTEQA2 runs presented in the Reflooded Mine Source Concentration Assessment (October, 2000).*

Comment 7: Please revise Section 2.7 of the reflooded mine source term derivation (Appendix 4.2-17) to include an explanation of the basis for the selection of the oxidation states for iron and manganese.

Response 7: *In the earlier modeling runs, Fe^{3+} and Mn^{3+} were entered at very low concentrations to accelerate convergence. In the revised runs, the concentrations for these parameters were estimated with MINTEQA2. For the simulations of the acidic backfill, the Eh was set to 0 mV. For the simulations of the oxidized neutral backfill, Eh was set to –50 mV (the average Eh from the humidity cell tests of pyritic paste backfill). For the simulations of the unoxidized backfill, the Eh was set to -170 mV (the average Eh measured in the pyritic paste backfill pore water).*

Comment 8: Please revise the reflooded mine source term derivation (Appendix 4.2-17) to include a discussion of the fate of the cyanide used in the flotation process. If it is expected to report with the pyrite concentrate, please address the effect of the cyanide on the water quality in the reflooded mine. Include in the discussion any effect the cyanide may have on the behavior of other constituents of concern.

Response 8: *This comment is addressed in Section 2.5.3. That discussion is reproduced here.*

Sodium cyanide is widely used in the selective flotation of lead-copper-zinc ores as a depressant for pyrite and, to a lesser extent, sphalerite. Cyanide depresses pyrite flotation by reacting with the pyrite surface and inhibiting the binding of flotation reagents.

The surfaces of the pyrite particles are re-activated for the flotation of the pyrite by decreasing the pH from above 10 s.u. (where cyanide species are stable) to a pH of about 6 s.u. At this pH cyanide is readily volatilized as HCN as follows:

$$H^+ + CN^- \leftrightarrow HCN\,(g)$$

Volatilization of HCN (g) reduces free cyanide concentrations in the porewater to very low levels. Actual measured cyanide concentrations in the pyrite flotation process water were <0.0034 mg/L. The decrease in the CN^- concentration leads to the dissociation of the dissolved $Fe(CN)_6]^{4-}$ complex, which, in turn, causes dissolution of the surface $Fe_4[Fe(CN)_6]_3$ precipitate and renders the pyrite amenable to flotation during depyritization. Thus most of the cyanide is lost to volatilization prior to pyrite flotation.

Residual cyanide precipitates remaining on the pyrite (reporting to the pyrite concentrate) are expected to be negligible and/or very stable. This expectation is supported by the equilibrated pore water data in Appendix A1 of the GWQPE (Table D-1), which shows that for Pyrite Concentrate 1 after equilibration the concentration is only 0.011 mg/L.

Comment 9: Please revise the reflooded mine source term derivation (Appendix 4.2-17) to include a comparison of the available neutralization potential a) implied by solid-phase characterization (mineralogical, chemical, and static tests) and b) determined in the multiple step batch testing with that observed in the corresponding short-term and intermediate-term humidity cells for the pyrite concentrate and the pyritic paste backfill.

Response 9: *See Response 17.*

Comment 10: Please revise the reflooded mine source term derivation (Appendix 4.2-17) to include a comparison of trace metal concentrations derived for the reflooded mine source term and those observed in the humidity cells for the pyrite concentrate and the pyritic paste backfill.

Response 10: *Table 2.9 in Section 2.5.1 of this report summarizes the predicted concentrations in the acidic backfill pore water and the concentrations observed in humidity cell tests of the pyrite concentrate and pyritic paste backfill. Direct comparison is difficult due to the dilution that occurs in the humidity cell tests. However, results of the laboratory tests were used to derive the relative rates of solute release (GWQPE Appendix A3, March 2000) used in the estimates of water quality.*

The concentration estimates derived from the pyrite concentrate data are considered to better represent conditions expected in the acidic backfill for two reasons. First, the pyrite concentrate became acidic within the first few cycles of the tests. In contrast, the pyritic paste backfill tests remained neutral. Second, the paste backfill tests were terminated after 20 weeks of testing whereas the pyrite concentrate tests were continued for 58 weeks. Consequently, the rate of oxidation in the paste backfill was significantly lower than in the pyrite concentrate tests and the sulfate generated was considerably less. When sulfate release is low, metal release associated with sulfide oxidation is more variable. Therefore, the relative release rate calculation for the pyritic paste backfill would be more prone to error.

Comment 11: The fairly rapid decline of pH in the humidity cell testing of the pyritic paste backfill suggests that oxidation is occurring quickly and that the neutralization potential of this material may be limited. Given the susceptibility of concrete to sulfide mineral oxidation, we are concerned about the structural and hydrologic/gas transport integrity of the paste backfill. Please revise the reflooded mine source term derivation (Appendix 4.2-17) to include a discussion of the short-term and long-term integrity of the paste backfill in relation to the results from the humidity cell testing.

Response 11: *The currently available data shows a "rapid decline of pH" from alkaline to near neutral only, after which it stabilized. In fact, after 60 weeks of testing, leachate from the humidity cell test on pyritic paste backfill remained above pH 6.*

Examination of the pyritic paste backfill blocks after 20 weeks of testing indicated no deterioration of the block structure. In fact, the cubes remained competent and oxidation was limited to a narrow rim around the outside of each block. Intermediate term humidity cell tests were dismantled on September 21, 2000 after 100 weeks operation. The report is in preparation. It was observed in the laboratory that the cubes remained intact even after 100 weeks of leaching. There is no evidence that the "structural and hydrologic/gas transport integrity of the paste backfill" is in question. On the contrary, the laboratory data suggest that the integrity of the paste backfill limits the rate at which oxygen can reach the oxidation sites.

Nonetheless, the UB estimates presented in the revised source concentration conservatively assume that the only limitation to oxidation rates will be the transport

of oxygen to the backfill face. This is equivalent to assuming that the backfill structure has completely degraded, allowing oxygen to diffuse rapidly through the backfill itself.

Comment 12: Please revise the response to Comment 17 in the 4/23/98 SRK memo (Appendix 4.2-13) addressing potential nitrogen contaminants from blasting to include an analysis of the nitrogen mass balance in the mine. In general, it is probably reasonable to assume that all of the nitrogen in the explosives ends up as either ammonia or nitrate and would be available to be dissolved in water during operation or reflooding or to be volatilized and moved out of the mine through the ventilation system during operation.

Response 12: *This comment is addressed in Section 5.3 of this report. That discussion is reproduced here.*

The explosive used in the blasting of waste rock and ore will consist of a mixture of ammonium nitrate and fuel oil (ANFO). The great majority of the nitrogen contained in the ANFO will be converted to NO_x gas (providing the explosive effect) upon detonation, and removed from the mine in ventilation air. However, some of the ANFO charges will not detonate, and will become a potential source of nitrogen compounds in the reflooded mine.

Residual nitrogen in the reflooded mine was estimated following the method suggested by Ferguson and Leask (1988). The calculations are presented in Attachment I of the Reflooded Mine Source Concentration Report. Ferguson and Leask (1988) evaluated the export of nitrogen to the environment from coal mines in southeastern British Columbia, Canada. The following paragraphs describe the estimation of residual explosives content, the proportion remaining in the underground mine at closure, and the speciation of the resulting nitrogen solutes.

NMC estimates total explosives use at the Crandon Mine will be approximately 2.2 lbs/ton in waste and approximately 1.0 lb/ton in ore. Ferguson and Leask (1988) estimated the proportion of unexploded charges based on the type of explosive used (ANFO or emulsion). Missed or partially missed charges were common at that time due to the very wet conditions in the open pit coal mines. For the residual N calculations, it was assumed that 10% of the explosives used would remain as residual ANFO and that ANFO would be 93% ammonium nitrate.

The amount of the residual N in the backfill returning to the mine was estimated by carrying out a mass balance that incorporates the following steps:

It was assumed that the residual N in the ore and waste would be released to the mine water. Ore production is assumed to average 5,500 tpd and waste production is 100 tpd.

The mine water would capture all of the residual N and distribute it among flows to the Mine Water Treatment Plant (90%), to mine drainage water (1%) and to moisture in the ore (9%).

Nitrogen would be removed from circulation in concentrate shipped from the site, in interstitial porewater retained in the tailings, and by treatment and release of mine water.

No credit was taken for ammonia stripping within the mill or TMA.

Ferguson and Leask (1988) found that the relative proportions of nitrogen as ammonium, nitrite and nitrate were relatively consistent for different open pit mines. Ammonium nitrate contains 23% nitrogen as ammonia and 77% nitrogen as nitrate. However, oxidation of ammonia to nitrite and nitrate resulted in 10% N-NH4, 3% N-NO_2 and 87% N-NO_3. This proportion was used to estimate the quantities of each form of nitrogen in the backfill.

The mass balance calculation indicated that about 67 mg/L of ammonium nitrate residue would be present in the paste backfill porewater. The speciation would be 2.3 mg/L NH_4-N, 20.4 mg/L NO_3-N and 0.70 mg/L NO_2-N.

It should be noted that the zinc flotation circuit is operated at a pH in excess of 9.5 at which ammonium will convert to ammonia, and that for the well-aerated conditions during flotation, ammonia would be volatilized from the process water. Therefore the NH_4-N concentration is likely to be lower than the above estimate.

Comment 13: The oxygen transport modeling for the reflooded mine source term (presented in Appendix 4.2-13) assumes diffusion is the only mechanism for transport through stope bulkheads and the crosscut between the bulkhead and the backfilled tailings. As we have discussed previously, the bulkheads are not planned to be

airtight. With the air flow and air pressure changes that will occur in the mine, we believe that the assumption of only diffusive transport through the bulkhead to the backfill may be too conservative. Please provide oxygen transport analyses that assume atmospheric oxygen concentrations are present at the air/waste rock "plug" interface during the period of mining through reflooding or provide analyses assuming a lower oxygen concentration at the air/waste rock plug interface and justification for using a lower concentration.

Response 13: *The comment indicates a misunderstanding of the oxygen transport model. In fact, the model assumes that oxygen will be at atmospheric concentrations on the inside face (i.e., the backfill side) of the bulkhead. This assumption is equivalent to assuming that the bulkhead presents no restriction to oxygen diffusion. As discussed in the project's Reflooded Mine Management Plan, NMC is now committed to constructing high efficiency, engineered bulkheads that are as airtight as possible. Hence, the model assumption is conservative.*

The humidity cell tests on pyritic paste backfill provide further evidence that the oxygen transport model is conservative. As discussed in Section 2.4 of the Reflooded Mine Source Concentration report, oxidation rates in the humidity cells are more than two orders of magnitude lower than those derived from the oxygen transport model.

Comment 14: Please discuss in more detail the assumption in the oxygen transport work (presented in Appendix 4.2-13) that the oxidation occurs approximately uniformly in a zone immediately adjacent to the backfill/waste rock plug interface. Under this assumption, an effective front of oxidation and complete NP utilization would be established during the period between the placement of backfill and the reflooding of the mine. Please discuss the implications on the results if oxidation/NP utilization occurs less completely in the near interface zone and oxygen penetrates farther in the backfill.

Response 14: *The assumption that oxidation occurs uniformly in a zone immediately within the backfill simplifies the calculations. The simplifying assumption only affects concentration estimates when an equilibrium solubility constraint is encountered. In such cases, the volume of the acidic zone and the solubility control determine the total mass of solute released. As discussed in Section 2.5 of the Reflooded Mine Source Concentration report, only barium, calcium, lead, silver and sulfate are predicted to reach solubility controls in the acidic backfill. If oxidation was assumed to occur over*

a larger volume of backfill, it is conceivable that a greater mass of those solutes could come into solution. However, pH within the additional oxidation area would be neutral rather than acidic, and solubility controls would be imposed on other elements such as copper, iron, and zinc.

Comment 15: Section 2.4 of the Reflooded Mine Source Term report, revised April 1998 (Appendix 4.2-13), details a series of steps used in making the calculations of pore water solute concentrations in acidified tailings. The pyritic paste backfill source term report, dated November 1998 (Appendix 4.2-17), indicates that the same procedure was used to make the new calculations. Please provide a more detailed explanation of the process used and the rationale for each step.

Response 15: *Attachment A of the Reflooded Mine Source Concentration report now details all the steps in the calculation of pore water solute concentrations in the backfill.*

Comment 16: The total pore volume of the post-mining mine void space is estimated to be 1.2 x 10^8 ft^3 in the groundwater flow analysis presented in the updated Reflooded Mine Source Term report, dated April 1998 (Appendix 4.2-13). However, in the original report, dated March 1997, the total pore volume of the post-mining mine void space was estimated to be 2.2 x 10^8 ft^3. Please explain the change in post-mining mine void space between the two versions of the report. In addition, please indicate whether either number includes the grouting drift or the waste stopes and associated access ways.

Response 16: *The total pore volume of the post-mining void space is estimated to be 2.32 x 10^8 ft^3. The estimate includes approximately 1.79 x 10^7 ft^3 associated with the access workings (shafts, adits, crosscuts, etc.) and an assumed 2 ft gap between the uppermost backfilled stopes and the crown pillar. The grouting drift and waste stopes are not included, however a small pore volume in the crown pillar is included. The remaining 2.12 x 10^8 ft^3 of void space is associated with the backfill.*

The total volume of backfill was estimated by multiplying the mineable zinc reserve of 30 million tons by the in situ ore density of 8.8 ft^3/ton and adding the copper ore reserve of 25 million tons multiplied by the in situ density of 10.67 ft^3/ton to obtain a total volume of 5.31 x 10^8 ft^3. Tests on samples of pyritic paste backfill indicate a dry

bulk density of 2,332 kg/m^3. The pyrite concentrate has a specific gravity of 4,500 kg/m^3. Using the relationship:

Dry bulk density = specific gravity x (1 – porosity)

these values suggest that the porosity of the backfill could be as low as 0.32. However, the addition of cement and its subsequent hydration will reduce the specific gravity below that of the pyrite concentrate alone. Hence, the actual porosity may be higher than 0.32. A conservatively high value of 0.40 was assumed in the reflooded mine source concentration. Multiplying 0.40 by the total ore volume of 5.31 x 10^8 ft^3 gives the estimated 2.12 x 10^8 ft^3 of void space within the backfill.

Comment 17: In Section 2.1 of the reflooded mine source term derivation (Appendix 4.2-17), a Sobek NP of 30.5 kg/t $CaCO_3$ was used as a minimum value for paste tailings after 28 days of curing. However, a value of 26 kg/t $CaCO_3$ was reported for two duplicate NMC samples in Table 3.20 of the characterization report (Appendix 4.2-15). Please resolve this apparent discrepancy.

Responses 4, 9, 17, and 18: *NP measurements on the pyritic paste backfill are presented and discussed in Section 2.2.3 of the Reflooded Mine Source Concentration report. The results therein indicate an average modified Sobek-NP of 45.4 kg $CaCO_3$ eq./tonne for samples cured for 28 days or longer. The subsequent water quality calculations conservatively assume an available NP of 30.3 kg $CaCO_3$ eq./tonne.*

As discussed in Appendix A4 of the GWQPE (FVD 1999/2000), the microprobe analyses indicate that the Ca+Mg fraction of the carbonate-NP in the pyrite concentrate is approximately 43%. However, CO_3-NP is not representative of the significant NP contributed by the oxide and hydroxide minerals derived from the addition of cement to the pyritic paste backfill. Analyses of residues from the multiple step batch extraction tests on pyritic paste backfill indicated that the non-carbonate NP was completely consumed, and the CO_3-NP was 84% to 88% reacted.

To provide additional context for the data presented in Section 2.2.3, it is useful to point out how the assumed NP value is used in the reflooded mine source concentration. As explained in Attachment A of the report, the total oxygen transport to the backfill is divided by the assumed NP to estimate the volume of backfill that could become acidic. If the assumed NP value is reduced, the volume of backfill

estimated to become acidic increases. However, there is a corresponding decrease in the amount of solute released per unit volume of acidic backfill. In terms of total solute load, the two influences cancel each other out. Hence, the final reflooded mine water quality predictions are not sensitive to the assumed NP.

Comment 18: Section 2.3 of the reflooded mine source term derivation (Appendix 4.2-17), indicates that more than 80% of the carbonates in the pyritic paste backfill are expected to be Ca+Mg carbonates. However, the chemistry for 8 grains of the pyritic tailings presented in Table 3.18 of the characterization report (Appendix 4.2-15) indicates calcium and magnesium carbonates comprise only about 68% of the carbonates in the pyritic tailings. Please resolve this apparent discrepancy.

Response 18: *See Response 17.*

Comment 19: Please revise the reflooded mine source term derivation (Appendix 4.2-17) to clearly take into account the proposed use of the wastewater treatment plant brine in the production of the paste backfill. In addition, please provide detailed information on the effects that the use of the brine may have on the short-term and long-term integrity of the backfill.

Response 19: *The November (1998) Reflooded Mine Source Term Assessment for Pyritic Paste Backfill did take into account the wastewater treatment brine. However, NMC has now decided not to use the brine in the production of backfill. The change is reflected in the Reflooded Mine Source Concentration report.*

Appendix B

Reflooded Mine Solute Transport Model Report

HSI GEOTRANS
A TETRA TECH COMPANY

Solute Transport Modeling of the Backfilled Crandon Mine After Reflooding

November 15, 2000

Prepared by:

HSI GeoTrans, Inc.
1080 Holcomb Bridge Road
Building 100, Suite 190
Roswell, Georgia 30076
(770)642-1000
www.hsigeotrans.com

Prepared For:

Foth & Van Dyke, Inc.
2737 South Ridge Road
Green Bay, Wisconsin 54304
(920)497-2500

Project No. N015-211

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B Technical Memorandum from TRC regarding "Long Term Durability of Proposed Grouting in the Crown Pillar"

C Letter from Agapito Associates regarding "Long Term Drift Stability"

1 Introduction

Nicolet Minerals Company (NMC) intends to mine the Crandon deposit in northern Wisconsin and fill mined stopes with a paste backfill. Once the mining has been completed, NMC will allow the mined-out area to be reflooded. The model that is presented in this report is used to simulate transport of constituents from the paste backfill and other mine areas to the Design Management Zone (DMZ) boundary. The analysis is used in the design of an overall Reflooded Mine Management Plan (RMMP) for the project.

1.1 Background

The analysis presented in this report supercedes earlier analyses of solute transport from the reflooded mine (e.g., HSI GeoTrans, 1999a and 1998b). In the earlier analyses, particle tracking was used to estimate the advective travel times for solutes moving out of the paste backfill to the DMZ boundary in the glacial aquifer. In those analyses, "Best Engineering Judgement for Transport (BEJT)" and "Practical Worst Case for Transport (PWCT)" scenarios were presented using a range of values for uncertain parameters such as hydraulic conductivity and porosity. Using simple calculations of mixing ratios, the earlier analyses presented estimates of concentration reduction at the DMZ boundary in the glacial aquifer for the BEJT and PWCT scenarios.

This report presents a more sophisticated analysis of transport from the reflooded mine using solute transport modeling to predict concentration instead of a combination of particle tracking and assumed mixing depths. Importantly, the current method allows consideration of diffusive and dispersive mass flux from the source, as well as advective mass flux from the source. Additionally, in this analysis the source configuration is considered in more detail by including different zones within the paste backfill (i.e. unoxidized, oxidized-neutral, acidic), and additional sources outside the backfill (i.e. wall-rock, crown pillar). Finally, whereas the prior analyses focused on mixing in the glacial system, the current analysis also considers mixing in the weathered and unweathered bedrock system.

1.2 Overview of The Analysis

The purpose of the modeling analysis presented in this report is to predict solute concentrations at the DMZ boundary, due to sources in the reflooded mine. The model is also used to analyze the effectiveness of "passive hydraulic controls" or engineered underground structures in reducing solute concentrations.

The first steps in model analysis are conceptual model development (section 2) and construction of the numerical model (section 3). Once developed, an "Expected Case" model is used for simulations of solute transport from the reflooded mine (section 4). The Expected Case model includes constructed open workings in the footwall (in addition to the open workings in

the hanging wall used to access the mine) designed to reduce solute concentration at the DMZ boundary. The design of this arrangement of open workings is discussed in section 5. Predictive uncertainty in the model is addressed through sensitivity analysis, and model assumptions are verified through model testing, presented in section 6. Finally, section 7 gives an overall assessment of the model and describes its practical application. The model simulation results presented here can be applied to source concentration estimates for each constituent of interest (or COI, as defined by SRK, 2000) to estimate the future concentration of these solutes at the DMZ boundary.

2 Conceptual Model

2.1 Hydrogeologic Setting

The zinc ore zone that would be mined is in the Crandon Formation. The copper ore lies below and adjacent to the zinc ore (Figure 1). The Crandon Formation extends from the top of the bedrock (approximately 100 ft below ground surface, or about 1550 feet, msl) to a depth of about 2100 ft (-450 feet, msl), dipping slightly to the north. For structural stability, the uppermost 100 feet of the Crandon Formation will be left unmined. This area above the mined ore, called the crown pillar, is weathered and has a moderate to high hydraulic conductivity (HSI GeoTrans, 1998a). The sparsely-fractured bedrock to the north of the zinc ore (the hanging wall) and the sparsely-fractured bedrock to the south of the zinc ore (the footwall) are characterized by relatively low hydraulic conductivities (HSI GeoTrans, 1998a). The upper, more-weathered bedrock exhibits a higher conductivity than does the lower, less-weathered bedrock. Near the Crandon Formation, the bedrock is generally more weathered than it is far away from the Crandon Formation (HSI GeoTrans, 1998a).

After mining, the blast-hole stopes in the zinc and copper ore will be filled with a paste backfill (Foth & Van Dyke, 1995/1998b), having a very low hydraulic conductivity (Golder Paste Technology, 1998). The shafts and drifts in the bedrock, used to access the mine, will remain open after mining. At the end of mining, all shafts that are open to the atmosphere at ground surface during mining will be sealed shut at the top of the bedrock with a concrete plug.

An unconfined aquifer of glacial outwash and glacial till overlies the bedrock. The high-conductivity glacial outwash provides most of the water-bearing capacity at the site. Directly above the orebody, the relatively lower conductivity glacial till is thick, and the overlying glacial outwash is unsaturated, giving a lower aquifer transmissivity in this area. Figure 2 is a conceptual, cross-sectional diagram of the hydrogeologic setting of the backfilled mine.

2.2 Description of Flow and Transport Processes

Once the zinc and copper ore zones in the bedrock have been mined and backfilled, and after the underground reclamation is completed, pumping from the mine will cease and, eventually, the entire mined area and underground open workings will reflood with water. After about two years (GeoTrans, 1996 and HSI GeoTrans, 1999b) the pre-mining flow field will be re-established, and an east-to-west flow pattern will again prevail at the site. The paste backfill will contain water-soluble constituents, including sulfate and some metals (SRK, 2000). Despite the low conductivity of the paste backfill (approximately 3E-4 ft/d or 1E-7 cm/s, Golder Paste Technology, 1998) a small amount of groundwater will flow through pore spaces in the backfill, and will carry solutes away from the backfilled areas. Additionally, molecular diffusion and hydrodynamic dispersion will influence the transfer of solutes from high-concentration water in the pore spaces within the paste backfill to the lower-concentration water flowing around the backfill.

Solute-carrying water emanating from the source zone will slowly migrate toward the boundary of the Design Management Zone (DMZ), which is a regulatory boundary 1200 feet (or less) outside the area of underground development. As the water flows westward with the prevailing hydraulic gradient, hydrodynamic dispersion and molecular diffusion will decrease the maximum concentration of constituents at the DMZ boundary relative to the concentration of constituents near the source.

2.3 Source Description

Within the backfilled stopes, 3 geochemical zones may be present prior to reflooding. In addition to unoxidized backfill, there may be zones of oxidized-neutral backfill and acidic backfill (SRK, 2000). Oxidation on the walls of the open mine workings will potentially create a separate source zone. Finally, oxidation at the base of the crown pillar will potentially create another source area. All of these source areas must be considered to fully analyze solute transport from the underground mine following reflooding.

By far the largest source zone, by volume and mass of solute, is the unoxidized paste backfill (SRK, 2000). Some constituents (e.g., sulfate) will be present in this zone in sufficient mass such that mineral solubility will control the constituent concentration in the pore-space water in the backfill (SRK, 2000). As water moves slowly through the unoxidized backfill zone, it will continually obtain an equilibrium concentration determined by mineral solubility, until the mass of the soluble mineral is depleted. Thus, it is appropriate (and conservative) to assume that the unoxidized backfill zone will behave as a continuous solute source for some constituents such as sulfate.

If a constituent is not solubility controlled, it is appropriate and conservative to assume that all of the constituent mass is immediately present in the pore water of the source zone. This approximation, which is consistent with calculations of source concentration made by SRK

(2000), can be represented by an instantaneous source. The instantaneous-source approximation is applicable for constituents in the wall-rock and crown-pillar sources, as well as the acidic and oxidized-neutral zones of the backfill. Constituents in the unoxidized zone of the backfill that are not solubility controlled are also more appropriately modeled as instantaneous sources.

3 Model Construction

3.1 Simulation Methodology

Solute migration in groundwater depends on several factors, including groundwater velocity, the coefficient of molecular diffusion, and the dispersivity (length scale of hydrodynamic dispersion). To predict solute concentrations in groundwater, as a function of space and time, an analyst generally seeks a solution to the advection-dispersion-diffusion differential equation, which is derived and described in detail in many texts on groundwater hydrology. If groundwater velocity is uniform, and the source configuration is well-described by a point, line, or plane of mass inflow, then analytical techniques can be used to predict concentration at points down-gradient of the source as a function of time. However, in non-uniform flow fields a numerical simulation may be required.

For the present analysis, many factors contribute to the development of a flow field around the source area and out to the boundary of the DMZ that cannot be accurately approximated as a uniform (spatially homogeneous) flow field. The juxtaposition of the open workings and the low-conductivity backfill creates localized hydraulic gradients not in line with the overall flow pattern from east to west. Also, the varying conductivities among the glacial and bedrock units leads to some vertical flow, which could introduce solute mass into the glacial aquifer. Because the groundwater flow field is not uniform, a numerical simulation approach is used for this analysis.

A combined groundwater flow and solute transport model is used in this analysis. The steady-state, post-mining hydraulic head field and velocity field is first determined, then used to solve for solute concentration at all model nodes at all simulated times. For this analysis, it is reasonable and practical to assume that the fractured bedrock system behaves as a porous medium, allowing the use of a porous-medium numerical modeling approach rather than a discrete-fracture model, or another type of model (Smith, 2000, Attachment A). The MODFLOW-T program (HydroSolve and HSI GeoTrans, 1998) was used for the modeling presented here. This MODFLOW-based modeling program efficiently and accurately simulates groundwater flow and solute transport in three dimensions. MODFLOW-T uses a finite-difference formulation of the transport equation to solve for concentration, with user options for central- or backward-difference approximations of space and time derivatives.

3.2 Domain and Grid

A regional model of groundwater flow (HSI GeoTrans, 1998a) was modified to include a zone for the low-conductivity backfill and a zone for open workings (high conductivity cells in the hanging wall, adjacent to and north of the backfill zone) to simulate steady-state, post-mining conditions. A map of modeled hydraulic head in the bedrock (regional model layer 7) is shown in Figure 3. A smaller model domain around the backfilled mine was chosen for solute transport modeling (also shown on Figure 3), with the regional model results used to provide boundary conditions for the smaller model. The model domain for the transport analysis, which is 7,875 feet by 3,375 feet, extends about 1500 feet from the mine boundary. Vertically, the model domain was chosen to extend from the water table (approximately 1600 feet, msl) to an elevation of 462.5 feet, msl, which is halfway between the 1100-ft mining level (depth) and the 1300-ft mining level. The model domain includes the upper half of the mine workings. It is assumed that groundwater flow and solute transport at greater depths is similar to flow and transport at the deepest modeled depths, and that the source below the modeled depth does not significantly impact predictions of DMZ-boundary concentration at the modeled depths (the model-depth assumption is tested in section 6).

The model grid spacing was initially set to a uniform 37.5 feet in the horizontal dimensions, yielding 90 rows and 210 columns. This allowed representation of 75-ft stopes in exactly two columns, and provided the discretization necessary to place open workings with relative accuracy. During model testing (section 6), it was determined that further discretization around the paste backfill produced more accurate results. Therefore, the grid was refined to a minimum 1.5-foot spacing at the edge of the backfill (Figure 4). The final grid, depicted in Figure 5, has 106 rows and 226 columns. For simplicity, the backfilled ore zone was represented as a rectangular zone 300 feet wide by 5025 feet long, taking up 16 rows and 142 columns near the center of the model domain.

Vertically, 26 layers are used, with each layer having a constant thickness of between 25 and 85 feet (Figure 6). The top two layers represent the glacial aquifer, layers 3-5 represent the crown pillar and surrounding bedrock, and the remaining layers contain the backfilled stopes and surrounding bedrock. For simplicity, the northerly dip of the orebody and the changing shape of the orebody are not taken into account. The total number of cells in the model is 622,856. The model domain size and discretization were chosen with consideration of available computing resources, and practical simulation times (the model requires about 350 MB of RAM and takes about 15 hours to execute with a 600 MHz computer processor).

3.3 Hydrogeologic Units

Hydrogeologic properties are represented in the numerical model by assigning a hydrogeologic unit to each model cell, and then ascribing hydrogeologic parameter values to each unit. In this model, the following 9 hydrogeologic units are used: 1) glacial outwash, 2) glacial till (including massive saprolite), 3) crown pillar (strongly-weathered Crandon

Formation), 4) grouted crown pillar (conductivity reduced by grouting during mining), 5) strongly-weathered bedrock, 6) moderately-weathered bedrock, 7) weakly-weathered and unweathered bedrock, 8) paste backfill, and 9) open mine workings. The distribution of these units in the model are shown in Figures 6 through 16. Note that layer 2 is uniformly glacial till, and layers 12-26 are similar to layer 11, except for the position of the open mine workings. This property zonation follows the conceptual model of the hydrogeologic setting (section 2.1), and is based on the configuration of units documented in previous subsurface characterization reports (Foth & Van Dyke, 1995 and 1995/1998a). Note that, for simplicity, stratigraphic layering is not followed in exact detail, the hanging wall and footwall are not split into separate units, and the weakly-weathered and unweathered bedrock are lumped together into one unit. Also note that grouting is assumed to lower the conductivity of the 25-ft layer immediately above the backfill zone, consistent with the grout thickness used in prior models (e.g., HSI GeoTrans, 1998a). The open mine workings shown in the figures include drifts in both the hanging wall and footwall (see below).

3.4 Representation of Open Workings

The open workings are modeled as a geologic unit that would logically have a very high hydraulic conductivity and a high porosity due to the presence of a large open space within the cell. Figure 17 shows how a 16-ft by 13-ft (nominal dimensions) drift would fit into a 37.5-ft by 25-ft model cell. The resulting porosity in the cell is about 22% (assuming the porosity of the rock is much smaller). A very high hydraulic conductivity (orders of magnitude larger than any bedrock conductivity) is used for the cells containing open workings. Where two open workings cells are adjacent, the assigned high conductivity results in a high cell-to-cell conductance, which is appropriate for a pipe-like connection. Where a lower-conductivity bedrock cell abuts an open workings cell, the harmonic averaging of conductances used in MODFLOW-T means that the cell-to-cell conductance will be based primarily on the lower conductivity of the surrounding rock, which is appropriate for flow into or out of the open workings.

The configuration of the open workings in the model (see Figures 6-16) generally follows NMC's plans for development, including drifts and cross-cuts on each mining level (200-ft increments) in the hanging wall, three vertical shafts, and a spiraling ramp. Additionally, the model includes open workings on the footwall side of the mine, added to reduce concentrations reaching the DMZ boundary. These open workings in the footwall (and the connection on the west end to the open workings in the hanging wall) make up a passive hydraulic control (PHC) that NMC would construct prior to mine closure. In section 5, the need for this PHC is described in more detail, along with a description of why this particular PHC configuration was chosen.

3.5 Boundary Conditions for Groundwater Flow and Solute Transport

In order to establish the proper groundwater flow field, precipitation recharge is applied to the top of the model, and constant head conditions are assigned at cells on all model sides, in each layer (Figure 5). Based on regional analysis of groundwater flow (HSI GeoTrans, 1998a),

the recharge rate is approximated to be 10 in/yr (2.3E-3 ft/d) over the entire top surface of the model. For simplicity, no consideration is given to the effects that surface water features may have on model recharge. The constant heads along the model sides were chosen to closely replicate the model-edge conditions obtained by running a post-mining simulation of groundwater flow with a regional model (Figure 3). Because modeled vertical gradients at the model edges are very low in the regional model, each layer has the same arrangement of constant head values. The bottom boundary of the model is a no-flow boundary. This neglects flow to and from lower portions of the bedrock and backfill.

At the boundary of the model, where groundwater exits the domain, a third-type concentration boundary (or zero concentration gradient boundary) is applied. Where flow is into the model domain, a concentration of zero is assigned to inflowing water.

3.6 Method of Modeling the Source

In some simulations of the unoxidized portion of the paste backfill, the entire backfill area is treated as a continuous source with a nominal concentration of 1.0 mg/L. The assumption of a continuous source is consistent with the behavior of a solubility-controlled constituent such as sulfate. Using a nominal unit concentration allows for easy interpretation of results. For instance, if a model simulation shows a 0.01 mg/L concentration on the DMZ boundary, this indicates a 100-fold concentration reduction due to mixing between the source and DMZ boundary. The "relative concentration" of 0.01 (unitless, expressed as a fraction of the source concentration) can be multiplied by any constituent's source concentration to estimate the DMZ-boundary concentration for that constituent (due to the unoxidized backfill source).

Instantaneous-source simulations (initialized concentration in the source zone of 1.0 mg/L) are carried out to show what happens if all of the solute mass for a constituent is immediately dissolved into the groundwater. This approximation is more appropriate for chemical species not controlled by mineral solubility. Instantaneous source simulations are carried out to estimate DMZ-boundary concentrations that would result from sources in the acidic backfill, oxidized neutral backfill, open workings wall rock, and crown pillar. Additionally, an instantaneous source is used to model constituents in the unoxidized backfill that are not controlled by mineral solubility.

3.7 Assumptions and Simplifications

In order to keep the model as simple and easy to use as possible, while still incorporating the features that are important for prediction of solute transport, a number of simplifying assumptions are made. Many of these assumptions have already been mentioned: assumption of an equivalent porous medium in the bedrock, simplified orebody shape, lumping of footwall and hanging-wall units, lumping of weakly-weathered and unweathered units, and uniform recharge. The distribution of hydrogeologic units has been simplified in the model. The early-time transport of solutes is ignored by starting with a steady-state, post-mine flow field (early-time

flow directions, which would be more toward the mine, might lead to greater dispersive mixing and lower concentrations, especially for an instantaneous source). The modeled constituents are also assumed to be conservative tracers — that is, sorption, geochemical reactions, and microbiological transformation are not considered. The layout of the open mine workings is simplified, and the sources of solutes deeper than the model bottom (which is about halfway down the backfill zone) are assumed to have a negligible effect on predicted concentrations within the model domain. All of the layers in the model are assumed to have constant elevations, and time-invariant transmissivities (based on full layer thickness). In the bedrock, each unit is assumed to be characterized by isotropic conductivity. Many of these assumptions were tested for significance as part of the sensitivity and verification analysis that is presented in section 6.

4 Expected Case Model

The "Expected Case" model for solute transport, presented below, represents what is considered to be the best predictor of future solute concentrations at the DMZ boundary. The model was constructed as described in section 3, with appropriate best-estimate values assigned for each parameter and each hydrogeologic unit.

4.1 Selection of Best-Estimate Parameter Values

For steady-state flow and transient solute transport, the following parameters are defined for each hydrogeologic unit: hydraulic conductivity (horizontal in x and y directions, and vertical), effective porosity, dispersivity (longitudinal, horizontal transverse, and vertical), and effective diffusion coefficient. Using information from prior characterization studies (e.g., Foth and Van Dyke, 1995 and 1995/1998a), and prior modeling analyses (e.g., HSI GeoTrans, 1998a), and through consultation with NMC's other technical consultants (see Smith, 2000, in Attachment A and SRK, 2000), best-estimate values for each parameter and each hydrogeologic unit were assigned for the Expected Case model. Table 1 lists these parameter values.

Many of the conductivity values were based on prior regional modeling (HSI GeoTrans, 1998a). For combined units (such as the combined footwall and hanging-wall strongly-weathered zone), a geometric mean (of BEJ values from HSI GeoTrans 1998a) was used to assign conductivities, rounding to one significant digit, and assuming isotropy. The stated opinions of Dr. Leslie Smith (2000, Attachment A), the Wisconsin Department of Natural Resources (2000a), and SRK were taken into account when assigning dispersivity, diffusion coefficient, and porosity values. The conductivity of the paste backfill was taken from Golder Paste Technology (1998). The grout horizon was assumed to degrade over the long time periods simulated in this analysis (thousands of years), but even after degradation, the grout horizon would still have a lower conductivity than the ungrouted crown pillar (TRC, 2000, Attachment B). A conservatively high value, about 20 times greater than the value suggested by TRC (2000), was used in the simulations.

All of the assigned parameter values have associated uncertainty that translates into uncertainty in the predicted solute concentrations. The sensitivity analysis in section 6 addresses the importance of parameter value uncertainty.

4.2 Expected Case Model Simulations for Each Source Area

The Expected Case model simulations begin with a predicted steady-state, post-mining flow field. Figures 18 and 19 show the resulting hydraulic head contours and velocity vectors. Some important features of the flow field are clear from these figures. Overall, flow is from east to west. Generally, there is a downward component of flow on the eastern side of the model domain and an upward component of flow on the western side. The gradient-flattening effect of the open workings is apparent from these results (contour lines spread out in the vicinity of the drifts, for instance). In the west-to-east section on Figure 19, the 1580-ft head contour line bends due to the effect of the open workings at 200-ft depth intervals. Water flowing through the paste backfill zone moves more slowly than the water flowing around this zone. Most of the groundwater flowing away from the paste backfill passes through some segment of open workings (by design, see section 5).

The Expected Case model is used to predict concentrations from the various source zones that could develop during mining, filling, and reflooding. In the paste backfill, 3 source zones could be present: unoxidized backfill, oxidized-neutral backfill, and acidic backfill (SRK, 2000). The wall rock of the open mine workings would also be a source of constituents. Finally, oxidation at the bottom of the crown pillar would create another source zone.

4.2.1 Unoxidized Paste Backfill

By far the largest volumetric portion of the paste backfill (over 99.7%) will remain unoxidized (SRK, 2000). This portion of the backfill will be the primary long-term source for some constituents, including sulfate.

Two transport simulations are made to predict DMZ-boundary concentrations relative to the concentration within the unoxidized backfill. In the first, the entire backfill area is treated as a continuous source of concentration at 1.0 mg/L, representing a constituent that is solubility-controlled, such as sulfate. The simulation models 10,000 years of solute transport from the unoxidized backfill. Concentration versus time curves are shown for a few model cells on the DMZ boundary in Figure 20 (left side). Because the source is continuous, these curves eventually reach a steady-state maximum concentration. The five curves shown in this plot represent locations in different depth ranges on the DMZ boundary: glacial aquifer (model layers 1 & 2), strongly-weathered bedrock (layer 3), moderately-weathered bedrock (layers 4-7), weakly-weathered bedrock (layers 8-12) and unweathered bedrock (layers 13-26). These particular model cells were chosen for the figure because they had the highest final concentration for a depth range. Note that it takes about 300 years to obtain a steady-state concentration at the DMZ boundary in the glacial aquifer, but that the steady-state concentration is not quite obtained

at the DMZ boundary in the unweathered bedrock after 10,000 years. Also on Figure 20, the maximum DMZ-boundary concentrations at different model elevations are shown. Note that this figure is a composite profile of maximum concentrations, and it does not represent a concentration-depth profile that would be observed at any particular location on the DMZ boundary. It is clear from this figure that concentrations are greater in the deeper bedrock, where less mixing occurs. In the glacial aquifer, which is important because of its capacity to provide a potable water source and because of its interaction with surface water, the maximum DMZ-boundary concentration is about 3% of the source concentration. In the unweathered bedrock, where less mixing occurs, the maximum DMZ-boundary concentration is predicted to be about 35% of the source concentration. Figures 21 and 22 show the predicted (relative) concentration of solute throughout the model domain after 10,000 years. The shape of the 0.3 relative concentration contours in Figure 22 show some effect of the open workings at 200-ft intervals.

In the second simulation of the unoxidized backfill source, the entire backfill area is an instantaneous source (concentration again equal to 1.0 mg/L), representing complete dissolution of the entire constituent mass at the beginning of the simulation. The resulting concentration versus time curves (Figure 23) have a different shape, with a defined peak concentration that occurs prior to the end of the simulation. As expected, the peak concentrations are somewhat lower than the steady-state concentrations shown for the continuous source in Figure 20. Note that the time scale for solute transport to the DMZ boundary in the unweathered bedrock is thousands of years in this simulation. Many constituents will likely be released in a manner that is between the continuous source and instantaneous source. Even solubility-controlled constituents will be limited by the amount of mass available in the paste backfill, and as the mass is depleted near the edges of the backfill, the rate of mass flux from the source will be diminished.

4.2.2 Oxidized-Neutral Paste Backfill

A thin layer of oxidized-neutral backfill may develop at the top of each mining stope. This layer will be only a few inches thick (SRK, 2000), which is much smaller than the thinnest model layer. Nevertheless, a model run was made with the 25-foot thick layers at the bottom of each mining level as active sources in the backfill (layers 8, 12, 16, 20, 24). This source is a substantial overestimation of the size of the actual oxidized-neutral zone. The source is assumed to be instantaneous, representing complete dissolution of the constituent in the oxidized-neutral zone. For this simulation, the source in the unoxidized paste backfill was not simulated. The resulting concentration versus time curves are shown in Figure 24.

Note that the concentrations at the DMZ boundary for this simulation are about 1/8 of the concentrations produced by the instantaneous-source unoxidized zone (Figure 23). The modeled size of the oxidized-neutral zone is also 1/8 of the total backfill zone size (25 feet of every 200 feet). This parallel occurs because of the regular pattern of the oxidized-neutral zone within the paste backfill. The result indicates that it is appropriate to treat the oxidized-neutral and unoxidized zones as a single source. The appropriate source concentration for a constituent

would be a volumetrically-weighted average of the oxidized-neutral and unoxidized concentrations. The resulting average concentration would be used with results of the previous section (4.2.2) to show the combined effect of the oxidized-neutral and unoxidized backfill source zones. The previous section's results are applicable because the entire backfill zone is considered to be a source in those simulations.

4.2.3 Acidic Paste Backfill

A very small portion of the backfill in each mining block may become acidified at the interface with the cross cut. A Best Engineering Judgement, or BEJ, estimate of the volume of this acidified material is 3.6 ft^3 per cross cut, and an upper-bound (UB) estimate of the volume is 990 ft^3 per cross cut (unmanaged scenario, SRK, 2000). At each cross cut, the volume of the model grid cell at the edge of the backfill is 1406 ft^3 (1.5 ft x 25 ft x 37.5 ft).

A simulation of the acidic zone was first made by making all of the backfill cells adjacent to cross cuts instantaneous-source cells (Figure 25). The resulting concentration versus time curves are shown in Figure 26. This simulation over-estimates the size of the source zone, especially for the BEJ source-size estimate.

An alternative representation of the acidic zone was made by assuming that all of the acidic-zone mass was instantaneously dissolved into the water in the edge of the cross-cut. In a second acidic-zone simulation, the first open workings cell in the cross cut was made to be the source rather than the first cell in the backfill (Figure 25). The results are shown in Figure 27.

If the mass of a constituent in the acidic zone is known, it is more conservative to distribute that mass within in the first cross-cut cell, and assume immediate removal from the paste backfill, than to distribute the same mass within the cell inside the backfill, where it takes some time to be released from the backfill[1]. Therefore, the recommended procedure for handling the acidic zone is to use the simulation where the source is in the cross-cut.

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[1]Consider a constituent that has a mass of 1 kg at each cross cut. Mixing the mass into the 15,900 liters of water in the backfill cell (1406 ft^3 with 40% porosity) results in a source concentration of 63 mg/L. Using the results of Figure 26, the DMZ-boundary concentrations for this constituent would be from 0.011 mg/L in the glacial aquifer to 0.045 mg/L in the unweathered bedrock. Alternatively, placing the 1 kg source in the 8,800 liters of water in the cross cut (1406 ft^3 with 22% porosity) results in a source concentration of 110 mg/L. Using Figure 27, the DMZ-boundary concentrations in this case would be from 0.013 mg/L (glacial) to 0.046 mg/L (unweathered), which are slightly higher than in the first alternative. It is therefore more conservative to initiate the source in the cross-cut cell outside the backfill.

4.2.4 Open Mine Workings Wall Rock

Oxidation on the walls of the open mine workings may produce another source of constituents. In estimating source concentrations from this source, SRK (2000) assumed that the mass instantaneously mixes with the water in the open workings. This is consistent with an instantaneous source at each model cell used to represent open mine workings. The results of this simulation are shown in Figure 28. Since the source is not in the low-conductivity backfill, the maximum DMZ-boundary concentrations all occur in the first several hundred years. The glacial aquifer maximum DMZ-boundary concentration is predicted to be about 4% of the source concentration, and the unweathered maximum DMZ-boundary concentration is predicted to be about 14% of the source concentration.

4.2.5 Crown Pillar

Oxidation at the bottom of the crown pillar may produce another source zone. SRK (2000) estimated that this source zone would be a 2-foot thick zone of water above the backfill (free water, or 100% porosity). The grouted crown pillar zone, which is a 25-foot thick layer of 10% porosity (effective water thickness of 2.5 feet), was used to simulate the crown pillar source. Note that this zone slightly overestimates the source volume. An instantaneous source is assumed. The results are shown in Figure 29. Unlike the other sources, which are present throughout the depth of the mine, the crown pillar source produces the greatest concentration response at the DMZ boundary in the glacial aquifer (DMZ-boundary maximum concentration about 0.5% of the source concentration). The peak concentration at the DMZ boundary also occurs more quickly than in other simulations (after only 23 years in the glacial aquifer).

4.3 Combined Effect of All Source Zones in the Expected Case Model

The simulations for different source areas were carried out independently for convenience. The DMZ-boundary concentration of a given constituent due to the presence of the constituent in a given source zone can easily be estimated using the appropriate Expected Case model results and an estimate of the source concentration for that constituent in that source zone. Table 2 summarizes the results of the Expected Case simulations of the different source zones.

Using the principle of superposition, the constituent concentration at any point on the DMZ boundary at any point in time will be the summation of the concentrations predicted for that point in space and time due to each source zone (plus the background concentration). As a conservative approximation, the maximum DMZ-boundary concentrations (at *any* point on the boundary and at *any* time) due to each source can be added to estimate the overall DMZ-boundary concentration increase (above background). While this method ignores the fact that different sources result in peak DMZ-boundary concentrations that occur in different locations and at different times, it is a good approximation when one of the source zones dominates the concentration prediction for a particular constituent. Calculations of the total DMZ-boundary

concentration increases, using the results summarized in Table 2 and the source concentrations derived by SRK (2000), are presented in a separate memorandum (HSI GeoTrans, 2000b).

5 Passive Hydraulic Control

The open workings in the footwall were included in the Expected Case model because they were found to have a beneficial effect — they led to a reduced DMZ-boundary concentration in the unweathered bedrock. This configuration of the open workings was one of several "passive hydraulic control", or PHC, options that were tested during model development. A PHC was defined, for this analysis, as an engineered structure that could be constructed during or after mine operations for the purpose of reducing concentration at the DMZ boundary. The controls are "passive" because these engineered facilities do not require continual operation or maintenance. A prior analysis (HSI GeoTrans, 2000a) demonstrates that an "active" containment system could be reasonably constructed to completely contain flow from the source area.

Below, a model without any PHCs is presented to demonstrate the need for PHCs, and the method of choosing a practical and effective PHC option is discussed.

5.1 Simulation Results for the No-PHC-Case

The Expected Case model includes open workings on the footwall side that are designed to reduce the concentration at the DMZ boundary in the deeper bedrock. A similar model, but without these additional workings (Figure 30) produces a flow field in which groundwater flows from the southwestern edge of the backfill zone to the compliance boundary without passing near any open workings (Figure 31). Also, more groundwater moves through the backfill zone when the footwall workings are not included. When the continuous unoxidized backfill source zone is simulated, concentrations in the unweathered bedrock at the DMZ boundary are similar in magnitude to the source concentration (relative concentration 0.97, Figure 32), and much higher than in the Expected Case model. Figure 33 shows that the highest DMZ concentrations in the unweathered bedrock result from the southwestern portion of the backfill source.

5.2 Effects of Open Mine Workings and High-Conductivity Zones Near the Source

The Expected Case model and the No-PHC-Case model show that open workings around the backfill provide an effective structure for reducing solute concentration. The reduction occurs because there is a relatively large flow of water in these workings, which results from the lack of flow resistance in the workings. Solute from the backfill zone that is transported into the open workings mixes into the relatively large flow of water there, reducing solute concentration.

Extending this understanding, it is desirable for all groundwater moving out of the source area to be directed toward a zone of high mixing, in order to reduce the maximum DMZ-

boundary concentration. In the No-PHC-Case the area of solute with the greatest concentration on the DMZ boundary did not intersect such a mixing zone.

The modeling indicates that by constructing a zone or zones of mixing, NMC could influence, to some extent, the maximum concentration of solutes at the DMZ. Two types of PHCs for creating enhanced mixing were shown to be effective in the model: 1) additional open workings and 2) zones of high-permeability clean fill within the mined area.

5.3 Scoping Simulations for Selection of Most Practical PHC Option

As the Expected Case model was being developed, simulations of various layouts for PHCs were also conducted. In various simulations, open workings were extended to different lengths along the footwall side of the backfilled stopes, open workings were placed through the backfill and around the western end of the backfill, and open workings were extended further north in the hanging wall, and south into the footwall, in order to capture more water for mixing. Also, a high-permeability clean fill was simulated (instead of paste backfill) in various stopes, and along the south edge of the backfill zone on the western end of the mine.

The simulations of these various PHC layouts were made as the model was still under development. As such, the simulations do not have a consistent basis for direct comparison and detailed results from individual simulations are not presented here. However, the conclusions gained from the analysis were as follows: 1) further reduction in solute concentration at the DMZ boundary in the deeper bedrock can be attained either by addition of open workings or by using high-conductivity clean fill in place of paste backfill; 2) the most effective open workings configurations involve open workings on the footwall side with connections to the hanging-wall workings off the western tip of the backfill zone; 3) the most effective location for a zone of clean fill is in the few stopes at the western end of the mine, 4) the additional-workings PHC scenarios and the clean-fill PHC scenarios have similar effectiveness; and 5) continued increase in the amount of open workings, or the size of the clean-fill zone further reduces DMZ concentrations, but with diminishing marginal effectiveness.

After evaluating the PHC layouts and simulation results, it was concluded that the most practical PHC would involve footwall workings along the south side of the western half of the backfill zone, as incorporated in the Expected Case model and shown in Figure 13. This configuration applies at each mining level in the model (layers 8, 12, 16, 20, and 24), and additional internal ventilation raises are included in the intervening layers on the footwall side to vertically connect the footwall workings (see Figures 14-16). The alternative clean-fill PHCs were not selected because these would reduce the space available for the paste backfill, and because of concerns about the structural stability and hydraulic conductivity of the clean fill. Other configurations with additional open workings resulted in concentrations that were only slightly lower than the concentrations predicted for the selected PHC option. Thus, the addition of open workings beyond those of the selected PHC would provide little benefit at significantly increased construction cost.

6 Model Testing and Sensitivity Analysis

During the course of model development, simulations were made to evaluate the effect of parameter uncertainty and the simplifying assumptions that were made. These simulations were made before establishing an Expected Case model (in fact, the information gained from the simulations helped in establishing the Expected Case model).

Using a model similar to the Expected Case model but with a uniform grid spacing, many simulations of the continuous backfill source were made to verify that accurate results were being obtained, and to gain an understanding of the effect of parameter value uncertainty on predictions of DMZ-boundary concentration. Many of the simplifying assumptions outlined in section 3 were tested for importance as part of this analysis. Also, consideration was given to possible future scenarios, including collapse of the open workings and early shutdown of the mine.

6.1 Reference Simulation for Model Testing and Sensitivity

The model testing and sensitivity analysis simulations presented in this section were based on a simulation that is very similar to the Expected Case, except that the grid spacing is uniformly 37.5 ft in the horizontal plane. This uniform grid, with no refinement along the edges of the backfill source, was the original model grid (the Expected Case was developed later). The uniform grid configuration has 21% fewer model cells and runs in about half as much time. The results of the simulation are slightly lower DMZ-boundary concentrations, as compared to the Expected Case (reference simulation results are summarized at the top of each of the Tables 3-6). The faster simulation times allowed for more efficient completion of the large number of simulations used in this analysis. Note that the reference simulation used for sensitivity analysis and model testing does include the PHC described for the Expected Case (footwall workings).

6.2 Parameter Uncertainty

The simulation results are dependent on the values chosen for the model parameters such as hydraulic conductivity, porosity, and dispersivity. The values chosen for the reference simulation are thought to be the most reasonable estimates (see section 4.1), but it is recognized that there is uncertainty in these values. Nineteen simulations were made with varied parameter values to assess parameter-value sensitivity. The results are summarized in Table 3. Note that the simulations were made by applying different factors to one or more of the parameter values listed in Table 1. An attempt was made to simulate a wide range of uncertainty. For a few simulations, the chosen values are considered to be outside the reasonable range of uncertainty (as discussed below).

6.2.1 Hydraulic Conductivity

As stated in section 4.1, the reference simulation hydraulic conductivities for all glacial and bedrock units were based on prior BEJ estimates (HSI GeoTrans, 1998a). Sensitivity simulations (Table 3) show the relative significance of uncertainty in these estimates for prediction of DMZ concentration. When the hydraulic conductivity of all bedrock units in the footwall and hanging wall is increased by one order of magnitude, the resulting DMZ-boundary concentrations are similar to, but generally higher than, the reference simulation results. Conversely, lowering the bedrock conductivities by one order of magnitude results in slightly lower concentrations, generally. Changes in the ungrouted crown pillar conductivity also have modest impacts on predicted DMZ concentrations. Changes to glacial system conductivities have a more substantial effect. When the glacial conductivities are raised by a factor of five from the reference simulation, more flow makes its way down to the bedrock producing lower DMZ-boundary concentrations. Conversely, with lower glacial conductivities much less flow enters the bedrock from above, producing higher DMZ-boundary concentrations in the bedrock. However, these parameter value changes to the glacial conductivities are outside the reasonable range of uncertainty for glacial conductivity (in the latter, the vertical conductivity of the till is 0.015 ft/d and the horizontal outwash conductivity is 8 ft/d). Also, the constant head and recharge boundary conditions were not modified along with the conductivity changes. For a more complete assessment, the boundary conditions would be updated when the glacial conductivity values are changed significantly.

The WDNR has suggested a set of high-end estimates for hydraulic conductivity (Wisconsin Department of Natural Resources, 2000b, U.S. Geological Survey, 1999). Using their high-end estimate of vertical conductivity of the glacial till (0.6 ft/d) has very little effect on predicted concentrations. However, changing the bedrock conductivities based roughly on the suggested high-end estimates does lead to an increase in the predicted DMZ-boundary concentration, especially within the weakly-weathered bedrock. In this simulation, the strongly-weathered bedrock conductivity is 0.1 ft/d, the moderately-weathered bedrock conductivity is 0.07 ft/d, and the weakly-weathered/unweathered bedrock conductivity is 0.01 ft/d. Using higher bedrock conductivities increases the velocity of water flowing around the edge of the backfill which increases the dispersive flux from the backfill zone.

The hydraulic conductivity value for the paste backfill is 0.0003 ft/d (Golder Paste Technology, 1998) in the reference simulation. A higher paste backfill conductivity leads to more flow through the source and higher concentrations at all depths. A lower paste backfill conductivity has little effect at shallow depths, but does lead to a flow field change in the deeper bedrock that allows a portion of the solute zone to make it to the DMZ boundary with less mixing in the open workings, leading to a higher DMZ-boundary concentration in the unweathered bedrock. Note that, in regional model simulations of post-mining flow, it was found that changing the hydraulic conductivity of the paste backfill had almost no effect on predicted hydraulic head values at locations corresponding to the edges of the reflooded mine

transport model. Therefore, changes can be made to the paste backfill conductivity without necessitating a change in the model boundary conditions.

Uncertainty in the hydraulic conductivity of the grout horizon is addressed in additional sensitivity simulations. Note that the grout horizon conductivity used in the reference simulation assumes significant degradation of the grout (section 4.1 and TRC, 2000, Attachment B). Even though the grout would likely maintain a lower conductivity for the 10,000 years of simulation (as suggested by TRC, 2000), the DMZ-boundary concentration would be little changed. However, if degradation were to cause a return to pre-grouting crown-pillar conductivity (and porosity), higher concentrations would result at the DMZ boundary, especially at shallow depths, as a result of increased vertical dispersion from the top of the paste backfill (the vertical dispersive mass flux from the backfill is proportional to the velocity of the water just above the backfill). A scenario wherein the hydraulic conductivity of the grout blanket returns to the pre-grouting condition is not envisioned by TRC (2000, Attachment B).

6.2.2 Porosity

The main effect of changing porosity is to change the time at which the concentration at the DMZ boundary reaches a steady-state value. The effect on predicted steady-state concentrations is generally small. Reducing the porosity of all units by a factor of 2 doubles the velocity of groundwater moving toward the DMZ boundary which causes steady-state to be attained twice as fast (Figure 34). But the change also doubles the dispersive mixing, resulting in an insignificant change in predicted steady-state DMZ concentration. When all porosities are doubled, the concentrations after 10,000 years are not yet at steady-state in the lower model layers (Figure 35). In the upper layers, the porosity change produces no change in DMZ concentrations. Based on ranges of uncertainty for the porosity of the weakly-weathered and unweathered bedrock suggested by Smith (2000, Attachment A), additional simulations were made to test low-end (0.001) and high-end (0.01) porosity estimates (reference simulation porosity is 0.003). To allow concentrations to reach steady-state, the simulation with the high-end estimate was extended to 30,000 years. The results of these simulations indicate no significant sensitivity to porosity with respect to the magnitude of concentration. In general, porosity only has an effect if velocities are lowered to the point where diffusion and dispersion are of similar importance for solute spreading. Dispersion is the dominant spreading mechanism over the range of porosity tested in this analysis.

6.2.3 Dispersivity

The values assigned for dispersivity affect the amount of solute spreading and the magnitude of dispersive mass flux leaving the backfill source. The reference simulation incorporates a longitudinal dispersivity of 50 feet, which is less than one-twentieth of the transport length scale (roughly 1200 feet from backfill to DMZ boundary). The horizontal transverse dispersivity is one order of magnitude less, or 5 feet. In the glacial units, crown pillar, and paste backfill, the vertical transverse dispersivity is further reduced by one order of

magnitude to 0.5 feet. In the fractured bedrock, the vertical transverse dispersivity is set equal to the horizontal transverse dispersivity. These values were suggested by Smith (2000, Attachment A) based on a review of literature, a study of the history of transport modeling at the site, and professional experience. Smith (2000) also noted a reasonable range of uncertainty in these estimates, which was tested in sensitivity simulations. These simulations produce small changes in predicted DMZ concentrations. In the unweathered bedrock, the high (reference simulation times 4.0) dispersivity results in slightly higher concentrations, and the low (reference simulation times 0.4) dispersivity results in slightly lower concentrations. This result is interesting because, in general, higher dispersivity leads to more spreading and lower concentration, but in this model the increase in dispersive mass flux from the source is important. In another sensitivity simulation, the bedrock units were assigned a vertical transverse dispersivity of 0.5 ft to match that of the glacial units, resulting in only a small change in DMZ-boundary concentrations.

6.2.4 Summary of Parameter Sensitivity Simulations

This test of model sensitivity shows that large changes in parameter values, over a broad range of uncertainty, leads to DMZ concentrations that typically vary by less than a factor of two. In the glacial aquifer, the DMZ-boundary concentration would be roughly doubled by an order-of-magnitude higher value of bedrock conductivity or paste backfill conductivity. Other changes, such as a lower grout conductivity (certainly reasonable according to TRC, 2000), or a lower bedrock vertical dispersivity would cause predictions of DMZ-boundary concentration to go down slightly in the glacial aquifer. In the unweathered bedrock, the maximum effect on DMZ-boundary concentrations would be somewhat less than a factor of two for reasonable parameter changes (ignoring the simulations that involved large changes in glacial conductivities, which are outside the reasonable range of uncertainty). It is interesting to note that the different estimates of hydraulic conductivity indicated for the reference simulation and WDNR high-end conductivity simulation, which produce quite different results for mine inflow in regional model simulations (WDNR, 2000b), produce very similar results for DMZ-boundary concentration in this analysis.

Porosity has little effect on the maximum DMZ concentrations obtained, but does influence the time to reach steady-state. Uncertainty in dispersivity is generally less important than uncertainty in hydraulic conductivity.

6.3 Importance of Assumptions and Simplifications

The model was designed to be a simple representation of the area around the reflooded mine. Simplifying assumptions were made during the course of model development. Some of these were discussed in section 3.7. An appropriate simplifying assumption makes the model simpler to use and explain, while providing results that are similar to results that would be obtained without the simplification. In this section, several simulations are discussed that test the validity of these simplifying assumptions. The results are summarized in Table 4.

6.3.1 Model Depth

A simulation was made with the model extended 400 feet deeper into the bedrock (two additional mining levels). The number of layers is increased to 34 for this simulation (Figure 36). The result is a small increase in predicted DMZ concentrations (Figure 37). However, the change in predicted concentrations are less than the changes produced by many of the parameter sensitivity simulations in section 6.2, and the practical cost of increasing the model depth is great. Much more memory is required, which limits the number of available computers that can be used for the simulation, and simulation time is drastically increased. Therefore, it was noted that a deeper model would show slightly greater concentrations, but the model depth was not changed for future runs. Note that the effect of a deeper model on concentrations at the DMZ boundary is very small in the glacial aquifer and strongly-weathered bedrock.

6.3.2 Source Size and Shape

Changes in the orebody width, from 300 feet in the reference simulation, to 375 feet and 225 feet in testing simulations, were made to assess the potential effect of simplifying the size and shape of the backfill zone. These simulations were made by adding and subtracting, respectively, 2 model rows in the middle of the backfill. The concentration changes at the DMZ were fairly small, and the simplifying assumption of a 300-ft-wide rectangular backfill zone was shown to be acceptable.

6.3.3 Constant-Head Boundary Condition

Another simulation shows whether an alternate constant-head pattern on the boundary of the model domain produces different results. Instead of assuming no vertical gradient on the model boundary, a vertical gradient was imposed. In this simulation, constant heads were set such that there is a downward head gradient of 2 feet over the entire depth of the model on the eastern side and an upward head gradient of the same magnitude on the west side. Linear interpolation was used to set heads at intermediate depths and along the north and south model edges. The results of this simulation show negligible changes in predicted DMZ concentrations.

6.3.4 Open Workings Representation

The hydraulic conductivity of the open workings was chosen to be an arbitrarily high value in the reference simulation (500 ft/d). In theory, the hydraulic conductivity should be high enough such that the conductivity of the rock outside the open workings controls the flow field. Another simulation, with an even higher open-workings conductivity, was used to test whether the value was set high enough in the reference simulation. The results show that increasing the assigned conductivity of the open workings by an order of magnitude has almost no effect on predicted DMZ concentrations, validating the original value chosen.

6.3.5 Bedrock Anisotropy

The assumption of isotropy in the bedrock was tested in another simulation. In regional groundwater modeling, anisotropy has been assumed for the moderately-weathered, weakly-weathered, and unweathered bedrock (HSI GeoTrans, 1998a) such that the east-west (along-strike) conductivity is about 3 times greater than the vertical (base) conductivity, and the north-south conductivity is about 0.3 times the vertical conductivity. This ratio of anisotropy is used in the testing simulation, resulting in modestly different DMZ concentrations. In the unweathered bedrock, anisotropy causes concentrations at the boundary of the DMZ to be slightly lower.

6.3.6 Combination of Hanging-Wall and Footwall Units

As stated in section 3, the model was made simpler by combining the footwall and hanging wall bedrock zones. Another simulation shows what happens if the zones are split, as shown in Figure 38. In this simulation, the footwall conductivities are 0.2 ft/d (strongly-weathered), 0.03 ft/d (moderately-weathered), and 0.002 ft/d (weakly-weathered/unweathered), and the hanging-wall conductivities are 0.05 ft/d (strongly-weathered), 0.01 ft/d (moderately-weathered), and 0.001 ft/d (weakly-weathered/unweathered). The resulting DMZ concentrations are slightly larger than the reference simulation concentrations.

6.4 Simulation Methodology

Many choices are available to the analyst for obtaining a numerical solution to the flow and transport equations with known parameter values and boundary conditions. The analyst can choose the discretization method (i.e., finite-difference or finite-element) and level of refinement, the solution technique (e.g., method of characteristics, upstream-weighting, implicit, explicit), the approximation algorithm (e.g., pre-conditioned conjugate gradient method, orthagonalization-minimization method) and the various parameters that define convergence. While testing each choice would be a lengthy and impractical exercise, a few tests of different simulation methods are warranted (Table 5).

6.4.1 Method of Transport Solution

It is important to know if the choice of code or method of solving transport introduces any error, such as "numerical dispersion", into the solution. To test this, a check was performed early in the development of the model. The check was made by simulating an early rendition of the model using both MODFLOW-T and MT3D (Zheng, 1996). The MT3D simulation was made using the method of characteristics to remove numerical dispersion, an option not available in MODFLOW-T. The drawback of the method of characteristics is that simulations generally take many times longer to execute than with other techniques. The two simulations produced nearly identical results, lending credence to the results produced by the more efficient MODFLOW-T model.

6.4.2 Grid Spacing

An important aspect of the solution methodology is grid spacing. Especially where concentration gradients are large, near the edges of source zones, grid cells need to be closely spaced for accuracy. The horizontal grid spacing and vertical grid spacing were tested separately in two additional simulations. The test of horizontal grid spacing uses the refined grid of the Expected Case model (Figures 4 and 5) rather than the uniform grid of the reference simulation. The resulting concentrations are significantly larger than in the reference simulation, especially in the glacial aquifer and strongly-weathered bedrock where concentrations are more than double the reference simulation (Table 5). As a result of this simulation, the more refined spacing was adopted for the Expected Case model discussed in sections 3 and 4. A similar decrease in vertical grid spacing at the top of the backfill zone (Figure 39) had relatively little impact on results and was not carried forward to the Expected Case model.

6.4.3 Time-Step Length

A simulation was made to confirm that the time-step length was sufficiently small. In the reference simulation, 384 time steps were used with a time step multiplier of 1.02, giving an initial (minimum) step size of 36 days, and a final (maximum) step size of 200 years. This same time discretization was used in the Expected Case model. In the test simulation, 4000 time steps were used with a time step multiplier of 1.002, giving an initial (minimum) step size of 2.5 days, and a final (maximum) step size of 20 years. The results were no different, indicating that the initial choice of temporal discretization was appropriate.

6.5 Possible Future Scenarios

Simulations were made to assess the effect of possible future scenarios on the predicted DMZ concentrations. The results are summarized in Table 6.

6.5.1 Premature Closure

A simulation was made to consider the effect of early mine closure. It was assumed that only the zinc portion of the backfill zone was mined and backfilled (most of the eastern half, and smaller portions on the north edge of the western half, Figure 40). All of the PHC footwall workings were included in this simulation, except no cross-cuts were simulated on the footwall side. Since the source is further from the DMZ in this simulation, concentrations at the DMZ boundary are lower than in the reference simulation.

6.5.2 Collapse of Open Workings

Agapito Associates (2000, Attachment C) assessed the long term integrity of the open mine workings and the likely failure modes after mine closure. It is their opinion that a realistic range for hydraulic conductivity in a completely collapsed segment of the mine workings is

between 30 ft/d and 3000 ft/d. A model simulation was made to show what would happen if the entire open workings zone conductivity is reduced to 100 ft/d. The changes are small in the shallow depth ranges, but concentrations are increased in the unweathered bedrock. This simulation very likely overstates the effects of collapsed workings because a hydraulic conductivity on the low end of the range was used and because the entire system of open workings was assumed to completely collapse. In reality, it is expected that the open workings will continue to provide little flow resistance (conductivity > 500 ft/d over most of the length) indefinitely. Complete collapse would be less likely in the deeper bedrock where the presence of the open workings is more critical. Also, in this simulation, no credit was taken for the fact that the collapse of a segment of workings would result in an expansion of the size of the high-conductivity zone above the workings (height of collapsed zone would be as large as 56 feet, compared to original drift height of 13 feet, see Attachment C).

6.5.3 Long-Term Wet or Dry Periods

Recharge was varied up and down by 50% in two additional simulations (these could be future scenarios, representing extended very wet and very dry periods, or parameter sensitivity simulations if recharge rate is considered to be a parameter). The effect of greater recharge in the glacial system is to increase the amount of solute-free water for mixing, thereby lowering the maximum concentration. However, greater recharge does decrease the upward flow gradient on the west side, keeping more of the solute deep, and leads to a slight increase in the prediction of DMZ-boundary concentration in the unweathered bedrock. The effect of less recharge is the opposite. Note that these simulations may overstate the effect of such recharge changes because boundary heads were not adjusted to reflect regional flow changes that would result from the recharge changes.

6.6 Conclusions from Model Testing and Sensitivity Analysis and Implied Uncertainty in the Expected Case Model

Parameter uncertainty is an important factor in predictions of concentration at the DMZ boundary. Changing hydraulic conductivity values to upper and lower limits of their reasonable ranges results in changes to the model-predicted DMZ-boundary concentrations (by less than a factor of two). The simplifying assumptions and simulation methods chosen for the model are shown to have only small effects on predicted DMZ-boundary concentrations.

A complete analysis of sensitivity using the refined grid of the Expected Case model was not carried out for this report. However, it is expected that parameter sensitivity on the refined grid of the Expected Case model would be very similar to the sensitivity demonstrated for the uniform grid (Table 3). That is, the percentage increases and decreases in DMZ-concentrations inferred from Table 3 are also applicable to the Expected Case model predictions, as an approximation of model sensitivity.

7 Model Assessment and Conclusions

The Expected Case model presented in section 4 provides a predictive tool for analyzing solute transport from the reflooded mine to the DMZ boundary. The model indicates that in the glacial aquifer and shallow bedrock the DMZ-boundary concentrations resulting from the backfill source zone will be no greater than about 3% of the concentration in the source. In the deep, unweathered bedrock, the DMZ-boundary concentrations could be as much as 35% of the source concentration (unoxidized/oxidized-neutral, continuous source). This indicates much lower concentrations in the water-bearing aquifer units than in the deeper bedrock units.

Uncertainty in parameter value estimates leads to uncertainty in predicted concentrations. Based on the sensitivity analysis of section 6, it is estimated that the maximum glacial aquifer DMZ-boundary concentration is predicted to within a factor of two in the Expected Case model, and model uncertainty in the prediction of unweathered bedrock concentration is significantly less than a factor of two.

While prior analyses (e.g., HSI GeoTrans, 1999a and 1998b) included both BEJT and PWCT predictions of solute migration and mixing, only an Expected Case is presented for this analysis. Model uncertainty is recognized, however, in the results of the sensitivity analysis. The uncertainty in this analysis (less than a factor of two) is less than the range of uncertainty indicated in prior BEJT and PWCT predictions. Furthermore, the ranges in source concentration predictions (SRK, 2000, BEJ and UB estimates) are thought to be a more important factor to consider for predicting DMZ-boundary concentrations of individual constituents.

Table 2 shows maximum Expected Case concentrations at the DMZ boundary that result for the various source areas in the reflooded mine (assuming a source concentration of 1.0 mg/L). These results can be applied to estimates of source concentration for individual constituents in each source zone to arrive at estimates of DMZ-boundary concentrations. This exercise is carried out in a separate memorandum (HSI GeoTrans, 2000b).

8 References

Agapito Associates, 2000. Letter from Margaret Asgian and M.P. Hardy to Ken Black of NMC regarding "Long-Term Drift Stability Letter Report, Crandon Mine, Forest County, Wisconsin." November 2000. Included as Attachment C.

Foth & Van Dyke, 1995. *Bedrock Hydrogeology at the Crandon Site.*

Foth & Van Dyke, 1995/1998a. *Environmental Impact Report, Crandon Project.* Originally issued May 1995. Updated December 1998. (Sections 3.5 and 3.6).

Foth & Van Dyke, 1995/1998b. *Mine Permit Application.* Originally issued May 1995. Updated December 1998.

GeoTrans, 1996. *Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Mine Near Crandon, Wisconsin.* August 1996.

Golder Paste Technology, 1998. Technical Memorandum from Bill Cincilla and Phil Newman to Ken Black of NMC regarding "Hydraulic Conductivity Testing of Paste Samples." June 1998. (Attachment C of HSI GeoTrans, 1998).

HSI GeoTrans, 1998a. *Addendum No. 2 to: Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Mine Near Crandon, Wisconsin.* August 1998.

HSI GeoTrans, 1998b. Memorandum from Peter F. Andersen to Jerry Sevick of Foth & Van Dyke regarding "Reflooded Mine Source Term." November 1998.

HSI GeoTrans, 1999a. *Update No. 2 to: Numerical Simulation of Potential Solute Transport at the Proposed Zinc and Copper Mine Near Crandon, Wisconsin.* January 1999.

HSI GeoTrans, 1999b. Memorandum from Greg Council to Steve Dischler of Foth & Van Dyke regarding "Groundwater Recovery at the Crandon Site — Effect of Mitigation Pumpage." May 1999.

HSI GeoTrans, 2000a. Technical memorandum from Greg Council and Pete Andersen to Steve Donohue of Foth & Van Dyke regarding "Crandon Project, Reflood Containment Modeling." November 2000.

HSI GeoTrans, 2000b. Technical memorandum from Greg Council to Steve Donohue of Foth & Van Dyke regarding "Crandon Project, Calculation of Concentration Increases on the DMZ Boundary for Individual Constituents of Interest." November 2000.

HydroSolve and HSI GeoTrans, 1998. *MODFLOWT: A Modular Three-Dimensional Groundwater Flow and Transport Model, Users Manual.* Version 1.22, October 1998.

Smith, Leslie, 2000. Memorandum to Steve Donohue of Foth & Van Dyke regarding "Mine Reflood Analysis." October 2000. Included as Attachment A.

SRK, 2000. *Reflooded Mine Source Concentration, Crandon Project.* November 2000.

TRC, 2000. Technical Memorandum from V. Straskraba, J. Wright, and S. Effner regarding "Long Term Durability of Proposed Grouting in the Crown Pillar." November 2000. Included as Attachment B.

U.S. Geological Survey, 1999. Technical memorandum from Daniel Feinstein to Chris Carlson of WDNR regarding "Calibration of 213, 211, and Massive Saprolite (PWAR) Pumping Tests." December 1999.

Wisconsin Department of Natural Resources, 2000a. Letter from Christopher P. Carlson to Gordon Reid of NMC regarding "Review of NMC Tailings Management Area Solute Transport Model." July 2000.

Wisconsin Department of Natural Resources, 2000b. Letter from Christopher P. Carlson to Gordon Reid of NMC regarding "Update on Groundwater Modeling and Provisional Mine Inflow Estimates for the Proposed Crandon Mine." April 2000.

Zheng, C., 1996. *MT3D-96: Modular Three-Dimensional Transport Model.* Version 2.0.

Table 1. Best-Estimate Parameter Values

	Conductivity E-W (ft/d)	Conductivity N-S (ft/d)	Conductivity Vertical (ft/d)	Porosity	Dispersivity Longitudinal (ft)	Dispersivity Transverse (ft)	Dispersivity Vertical (ft)	Diffusion (ft^2/d)
Outwash	40	40	4	0.1	50	5	0.5	1.6 E-04
Till (with Massive Saprolite)	2	2	0.075	0.1	50	5	0.5	1.6 E-04
Crown Pillar	8	8	8	0.1	50	5	0.5	1.6 E-04
Grouted Crown Pillar	0.3	0.3	0.3	0.01	50	5	0.5	1.6 E-04
SW Bedrock	0.1	0.1	0.1	0.05	50	5	5	1.6 E-04
MW Bedrock	0.02	0.02	0.02	0.01	50	5	5	1.6 E-04
WW/UW Bedrock	0.001	0.001	0.001	0.003	50	5	5	1.6 E-04
Backfill	0.0003	0.0003	0.0003	0.4	50	5	0.5	2.8 E-04
Open Workings	500	500	500	0.22	50	5	5	1.6 E-04

Created By: LMG
Checked By: GWC
Approved By: PFA
Date: October 19, 2000

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Table 2. Summary of Expected-Case Model Simulations

Source	Maximum DMZ-Boundary Concentration				
	Glacial Aquifer	Strongly-Weathered Bedrock	Moderately-Weathered Bedrock	Weakly-Weathered Bedrock	Unweathered Bedrock
Unoxidized\Oxidized Neutral Backfill (continuous)	0.028	0.032	0.22	0.26	0.35
Unoxidized\Oxidized Neutral Backfill (instantaneous)	0.011	0.012	0.11	0.14	0.21
Acidic Backfill (In Cross-Cut)	0.00012	0.00010	0.00043	0.00041	0.00042
Workings/ Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	0.00088	0.00017

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Checked By: GWC
Approved By: PFA
Date: October 18, 2000

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Table 3. Parameter Value Sensitivity Analysis, Summary of Simulation Results

Simulation	Maximum DMZ-Boundary Concentration				
	Glacial Aquifer	Strongly-Weathered Bedrock	Moderately-Weathered Bedrock	Weakly-Weathered Bedrock	Unweathered Bedrock
Reference Simulation	0.013	0.015	0.14	0.19	0.28
Bedrock Conductivity x 10	0.025	0.054	0.086	0.17	0.41
Bedrock Conductivity x 0.1	0.010	0.010	0.13	0.21	0.25
Crown Pillar Conductivity x 5	0.008	0.010	0.14	0.23	0.31
Crown Pillar Conductivity x 0.1	0.014	0.017	0.14	0.19	0.28
Glacial Conductivity x 5	0.009	0.009	0.084	0.13	0.14
Glacial Conductivity x 0.2	0.014	0.050	0.13	0.28	0.80
Till Vertical Conductivity = 0.6 ft/d	0.013	0.014	0.12	0.18	0.27
WDNR High-End Conductivities	0.018	0.021	0.15	0.32	0.33
Paste Backfill Conductivity x 10	0.025	0.029	0.22	0.27	0.35
Paste Backfill Conductivity x 0.1	0.012	0.014	0.13	0.19	0.36
Grout Conductivity x 0.1	0.011	0.013	0.14	0.19	0.27
Grout Conductivity & porosity = Crown Pillar Conductivity & porosity	0.062	0.066	0.15	0.21	0.30
All porosities x 0.5	0.013	0.015	0.14	0.20	0.28
All porosities x 2	0.013	0.015	0.14	0.17	0.22
WW/UW Porosity 0.01	0.013	0.015	0.14	0.19	0.27
WW/UW Porosity 0.001	0.013	0.015	0.14	0.20	0.28
High Dispersivity (200 ft / 20 ft / 2 ft)	0.019	0.020	0.079	0.20	0.32
Low Dispersivity (20 ft / 2 ft / 0.2 ft)	0.014	0.021	0.17	0.19	0.26
Bedrock Vertical Dispersivity = 0.5 ft	0.011	0.025	0.20	0.22	0.29

Created By: LMG
Checked By: GWC
Approved By: PFA
Date: October 18, 2000

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Table 4. Testing of Simplifying Assumptions, Summary of Simulation Results

Simulation	Maximum DMZ-Boundary Concentration				
	Glacial Aquifer	Strongly-Weathered Bedrock Conductivity	Moderately-Weathered Bedrock Conductivity	Weakly-Weathered Bedrock Conductivity	Unweathered Bedrock Conductivity
Reference Simulation	0.013	0.015	0.14	0.19	0.28
Deeper model, 2 more levels, bottom = 62.5 ft msl	0.014	0.017	0.16	0.25	0.32
Ore Width + 75 ft	0.012	0.014	0.14	0.21	0.31
Ore Width - 75 ft	0.014	0.016	0.14	0.18	0.25
Vertical gradient for Constant Head	0.013	0.016	0.14	0.21	0.27
Access Workings Conductivity x10	0.013	0.016	0.13	0.19	0.28
Anisotropic Bedrock	0.015	0.020	0.12	0.14	0.18
Split Hanging Wall and Foot Wall into separate zones	0.014	0.018	0.16	0.31	0.36

Created By: LMG
Checked By: GWC
Approved By: PFA
Date: October 18, 2000

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Table 5. Testing of Simulation Methods, Summary of Simulation Results

Simulation	Maximum DMZ-Boundary Concentration				
	Glacial Aquifer	Strongly-Weathered Bedrock	Moderately-Weathered Bedrock	Weakly-Weathered Bedrock	Unweathered Bedrock
Reference Simulation	0.013	0.015	0.14	0.19	0.28
Tighter Horizontal Grid at Backfill Edges	0.028	0.032	0.22	0.26	0.35
Tighter Vertical Grid at Backfill Top	0.018	0.032	0.13	0.20	0.29
Shorter time step	0.013	0.015	0.14	0.19	0.28

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Approved By: PFA
Date: October 18, 2000

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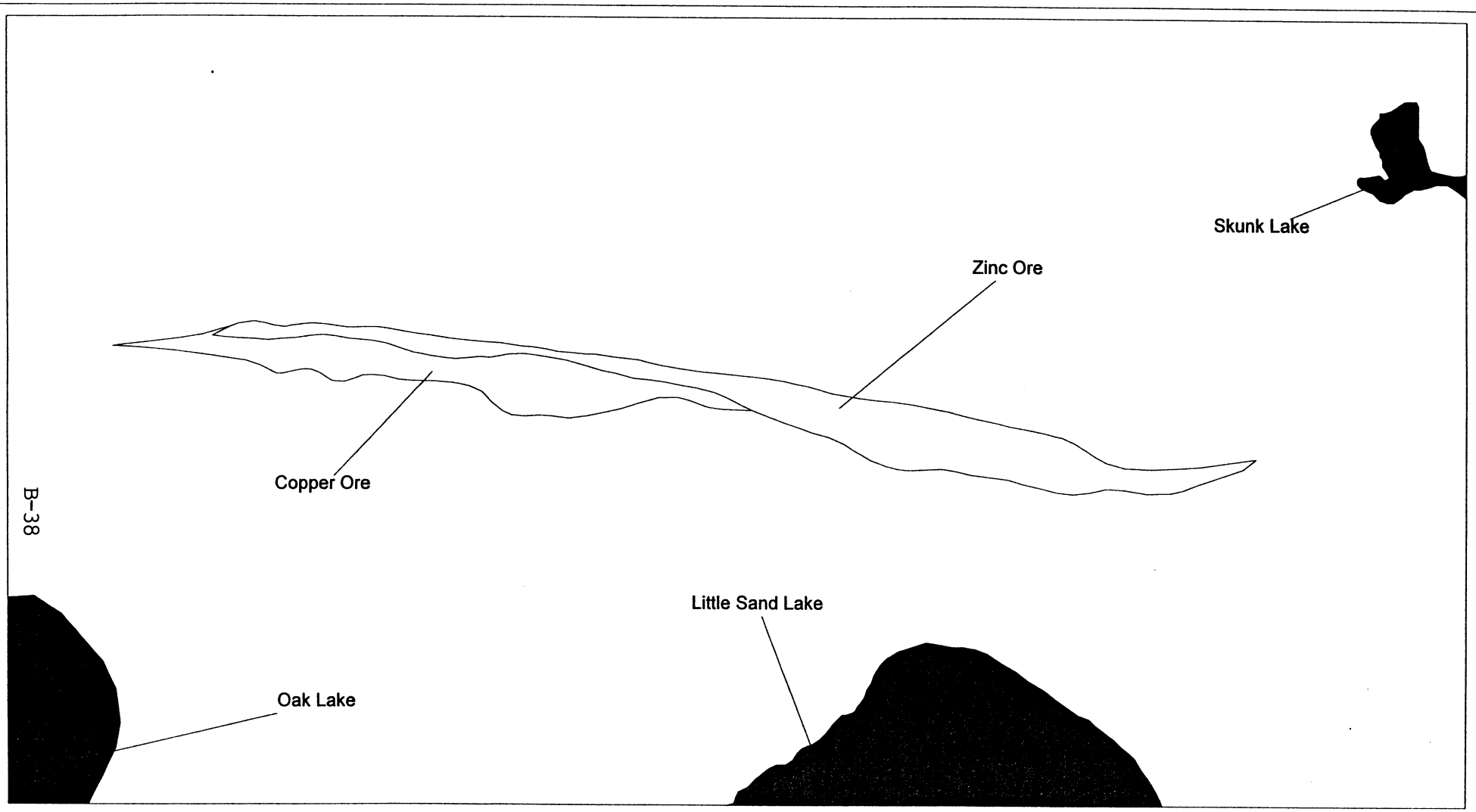
Table 6. Possible Future Scenarios, Summary of Simulation Results

Simulation	Maximum DMZ-Boundary Concentration				
	Glacial Aquifer	Strongly-Weathered Bedrock	Moderately-Weathered Bedrock	Weakly-Weathered Bedrock	Unweathered Bedrock
Reference Simulation	0.013	0.015	0.14	0.19	0.28
Premature closure	0.009	0.010	0.061	0.10	0.18
Workings K = 100 ft/d	0.012	0.014	0.16	0.29	0.45
Recharge x 1.5	0.009	0.011	0.12	0.23	0.46
Recharge x 0.5	0.022	0.024	0.12	0.16	0.19

Created By: LMG
Checked By: GWC
Approved By: PFA
Date: October 18, 2000

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0 750 1500

Scale in Feet

N

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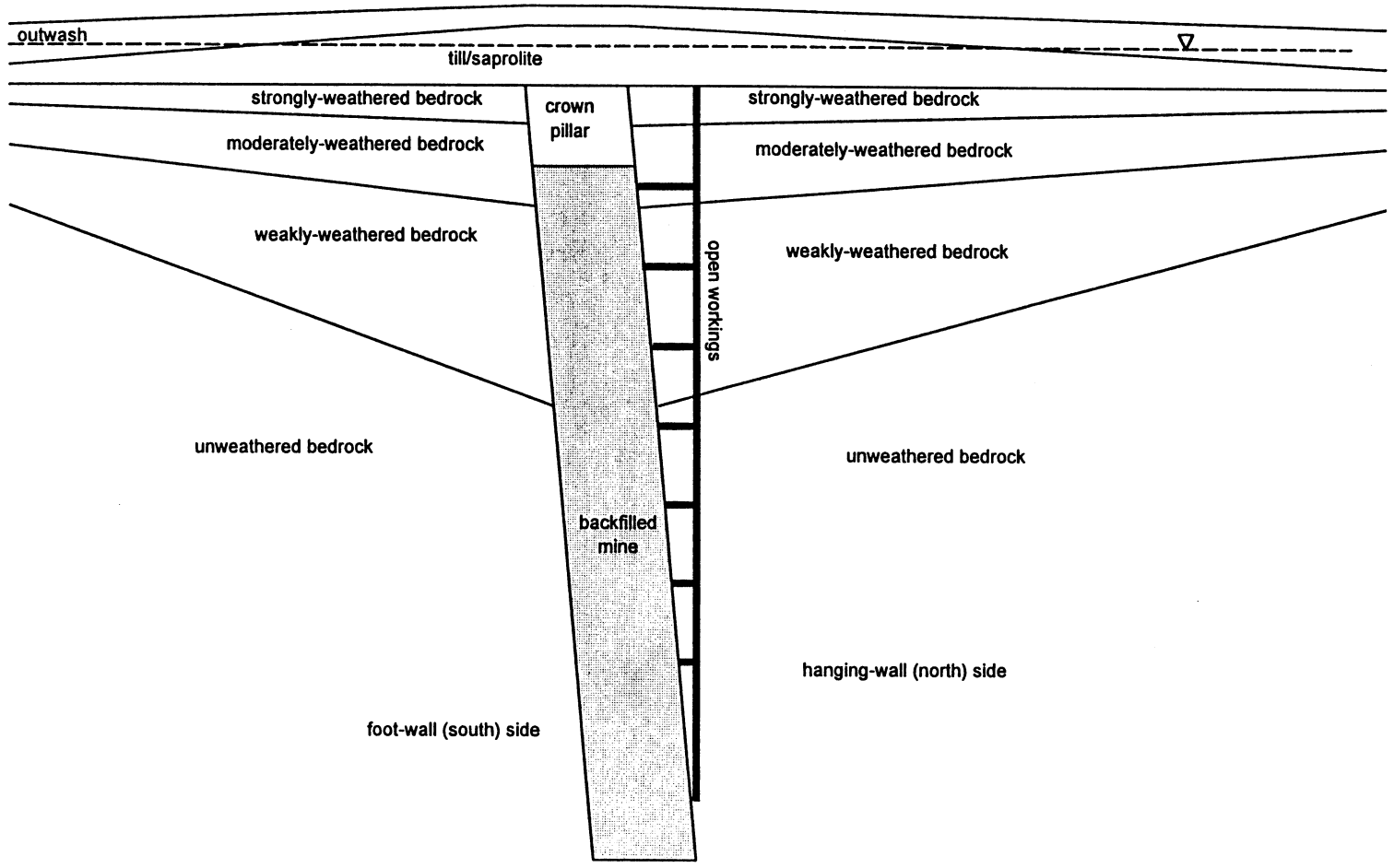
Nicolet Minerals
a Rio Algom company

Figure 1
Zinc and Copper Ore at Subcrop, Plan View

Scale: 1 in. = 750 ft. | Date: October 2000

Prepared By: HSI GeoTrans | By: LMG

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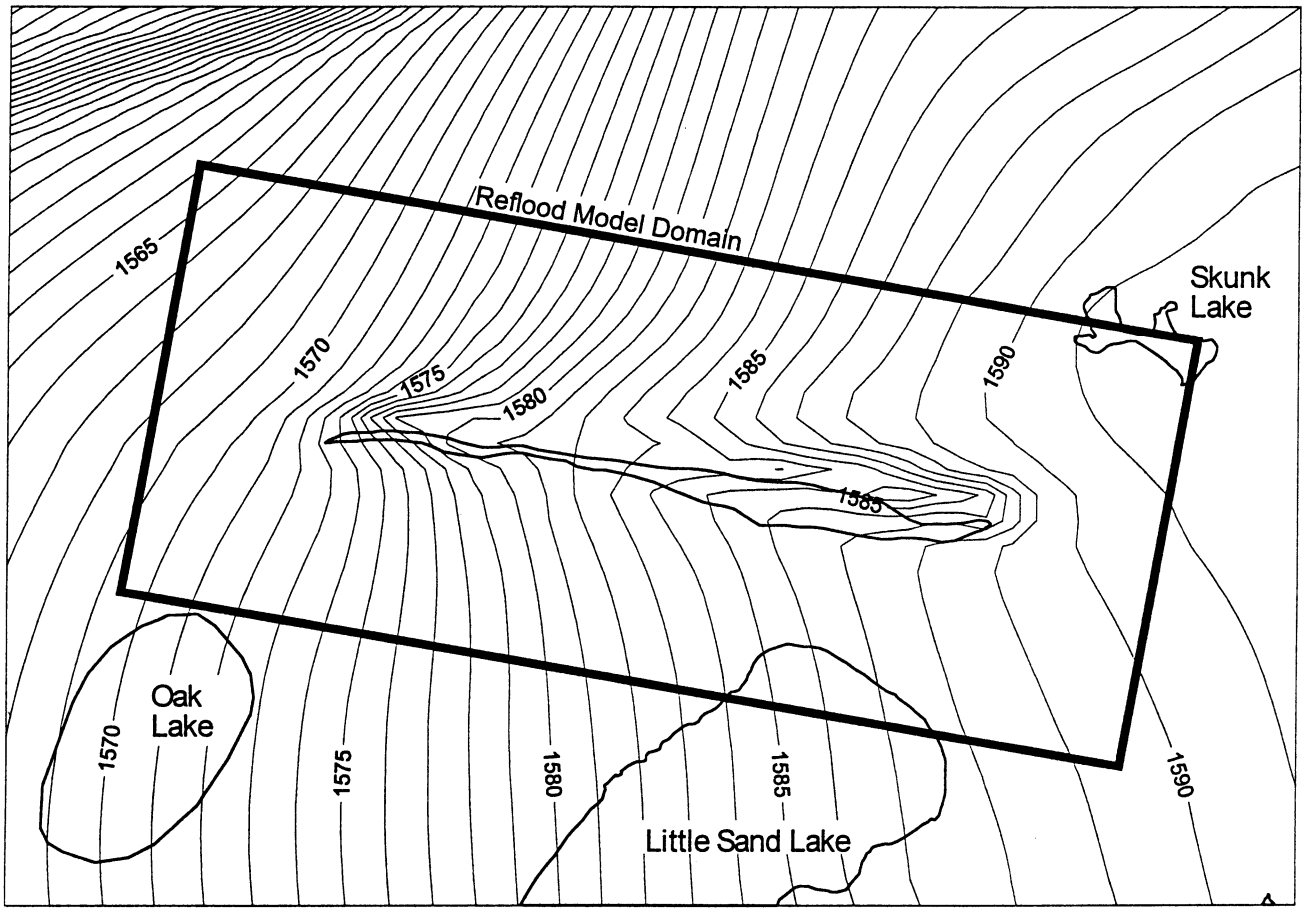


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CHECKED BY: LMG			DATE: 10/00
APPROVED BY: PFA			DATE: 10/00
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Figure 2
Conceptual North-South Cross Section
Showing Hydrogeologic Units

Scale: None	Date: Oct 2000
Prepared By: HSI GEOTRANS	By: GWC



Note: Modeled Head in Regional Model Simulation 49T7, Layer 7. Contour Interval 1 foot.

0 1500 3000

Scale in Feet

N

HSI GEOTRANS					
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CHECKED BY:	LMG		DATE:	1/00	
APPROVED BY:	PFA		DATE:	1/00	
APPROVED BY:		B-40	DATE:		

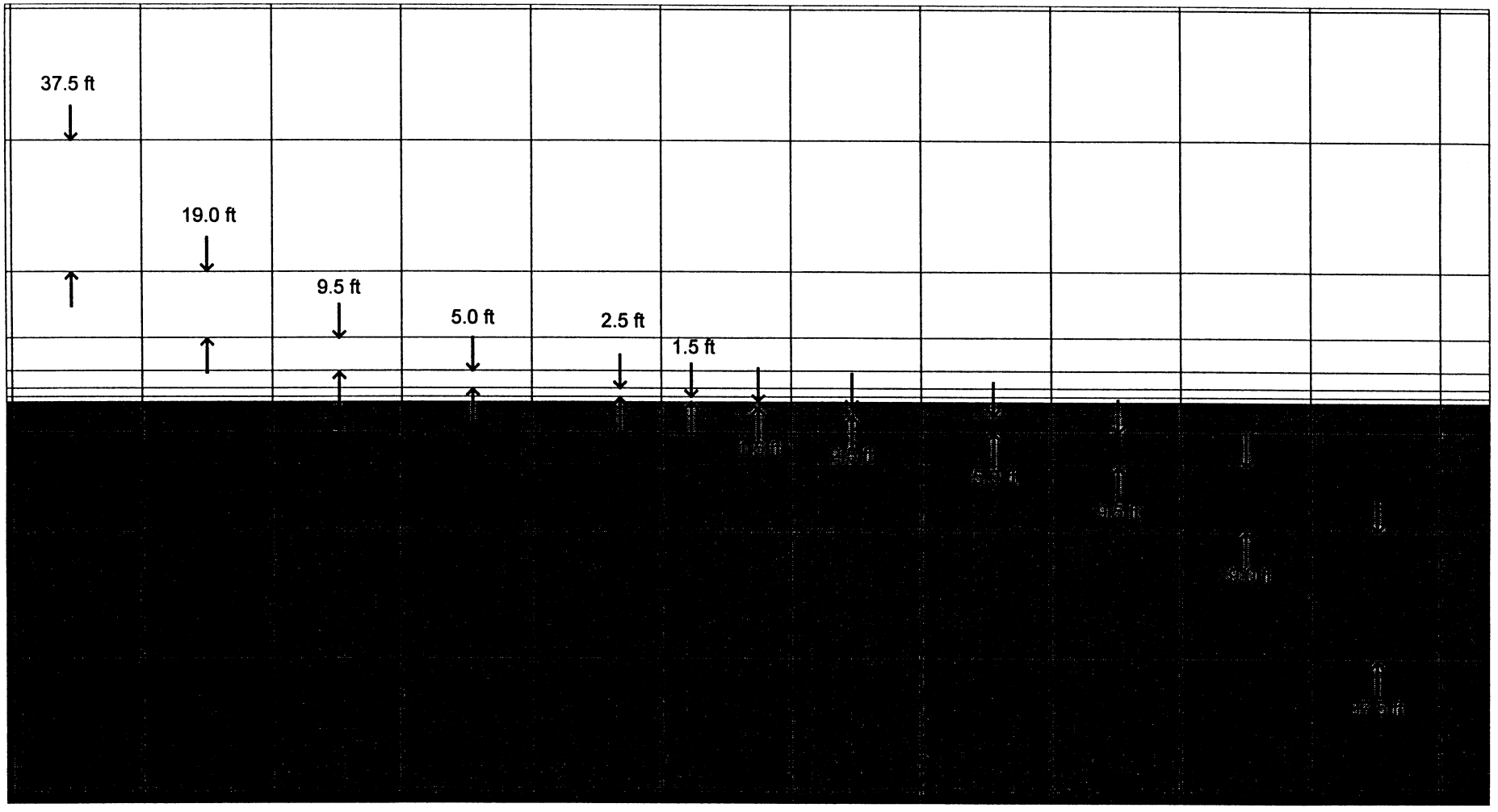
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Figure 3
Regional Groundwater
Flow Post-Mining

Scale: 1 in. = 1500 ft.	Date: Oct 2000
Prepared By: HSI GEOTRANS	By: GWC

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B-41

Legend

Grid Line

Backfill Zone

0 45 90

Scale in Feet

N

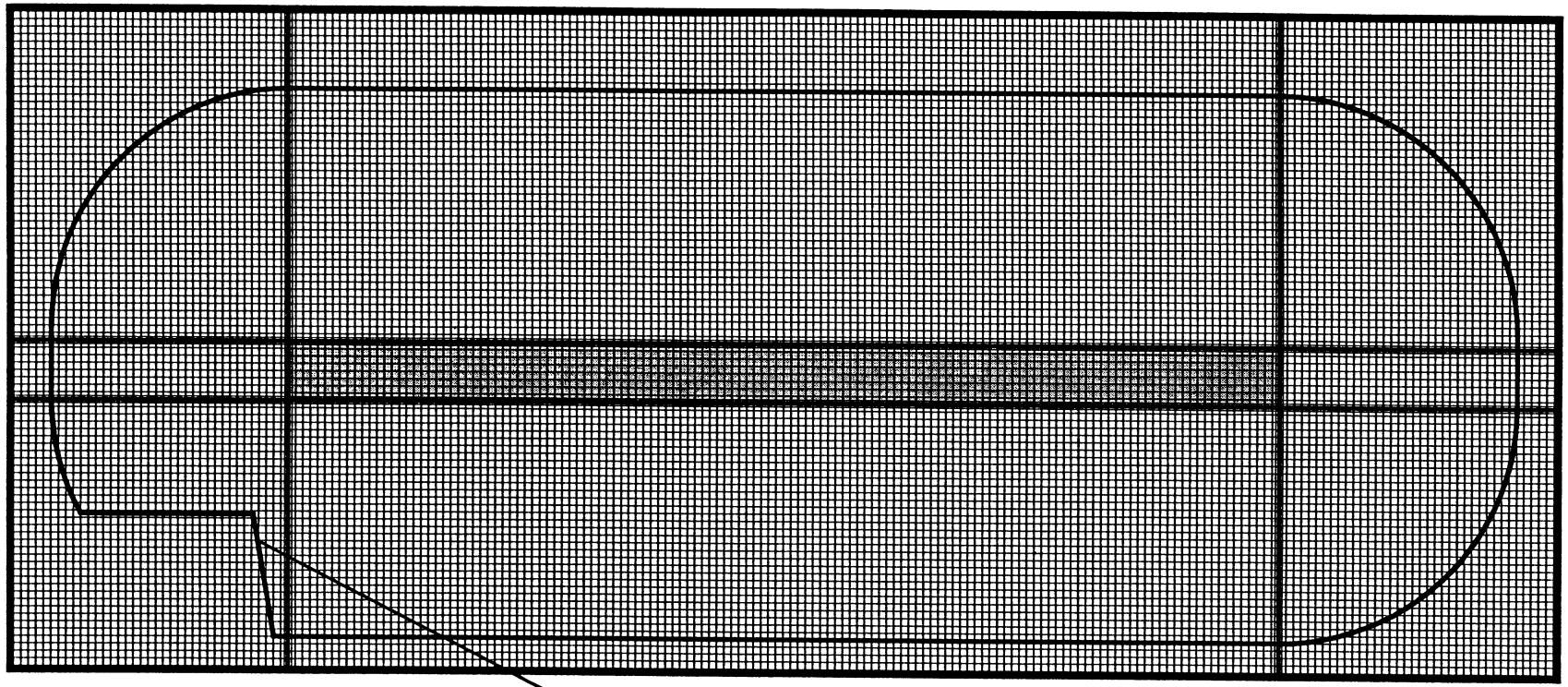
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REVISED	DATE	BY	DESCRIPTION	
CHECKED BY: GWC				DATE: Oct 2000
APPROVED BY: PFA				DATE: Oct 2000
APPROVED BY:				DATE:

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Figure 4
Grid Refinement at the Edge of the Backfill

Scale: 1 in. = 45 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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Note: The distance to the DMZ boundary is less than 1200 ft in this area. The DMZ boundary in this area is the NMC property boundary.

Legend

- Constant Head Boundary
- Design Management Zone (Approx.)
- Paste Backfill Zone

0 900 1800

Scale in Feet

N

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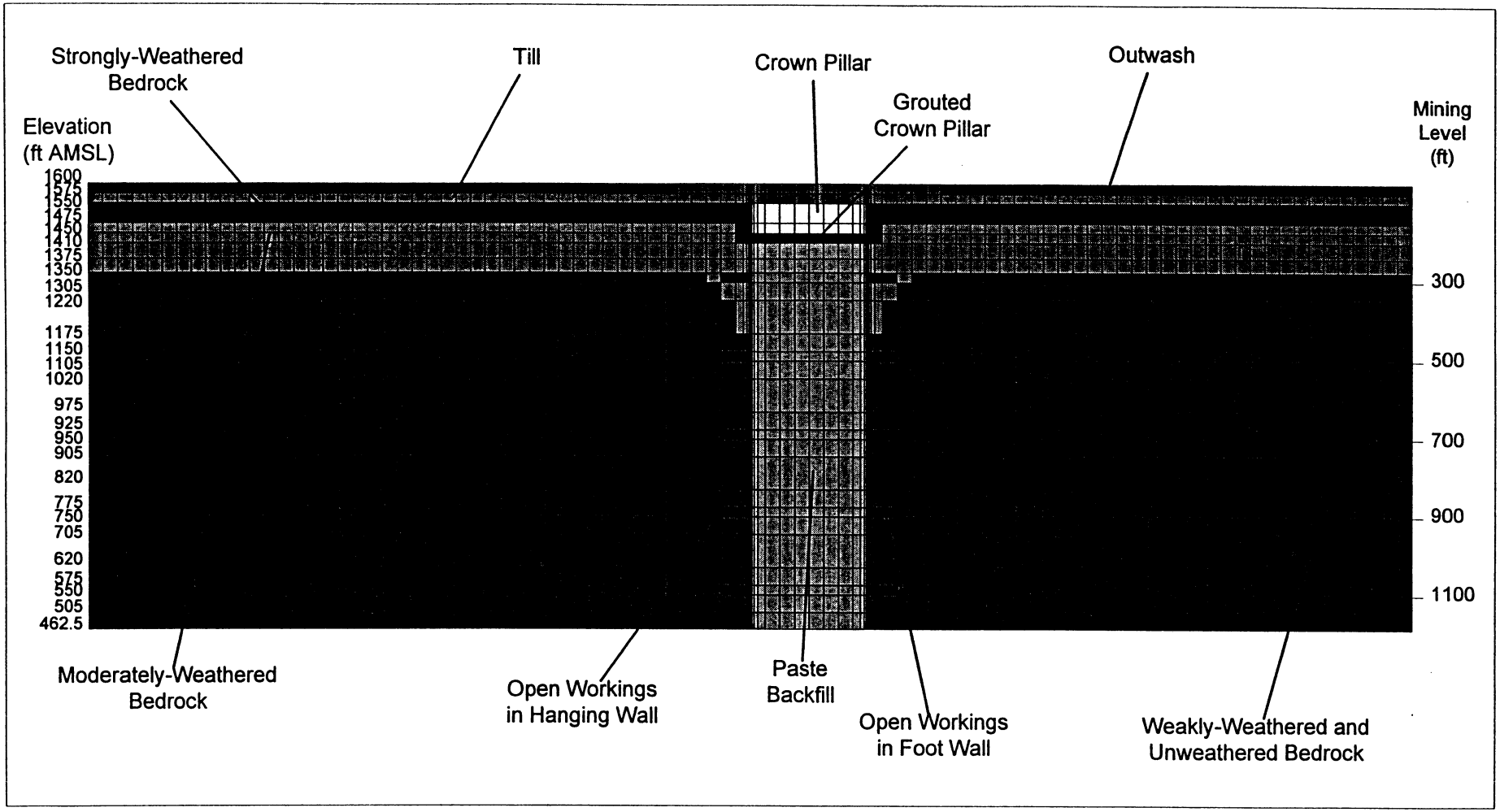
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 5
Model Grid and Constant-Head Boundary Conditions

Scale: 1 in. = 900 ft.	Date: October 2000
Prepared By: HSI GeoTrans	By: LMG

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B-43

0 400 800

Scale in Feet

HSI GEOTRANS

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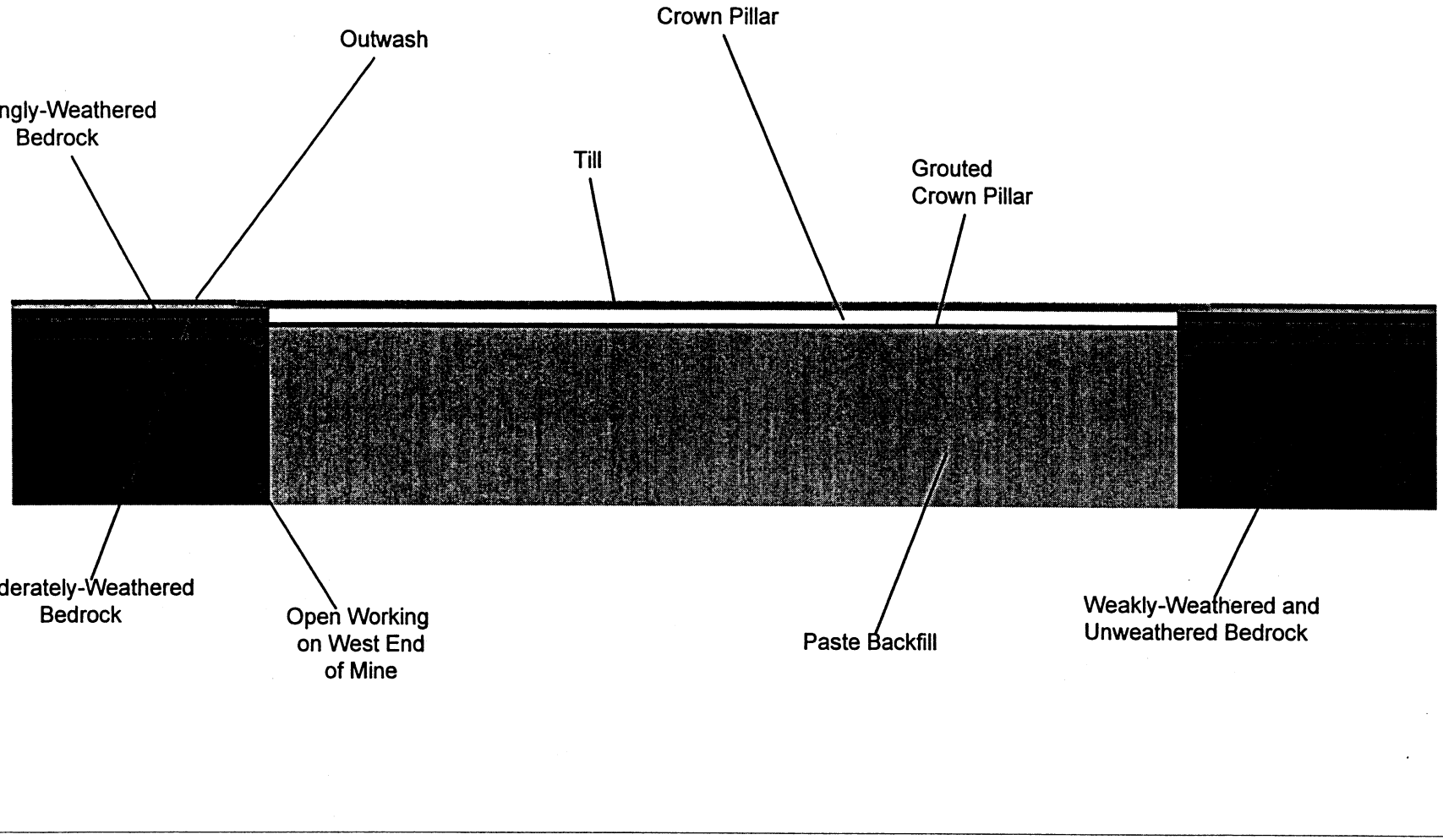
CHECKED BY: GWC	DATE: Apr 2000
APPROVED BY: PFA	DATE: Apr 2000
APPROVED BY:	DATE:

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Figure 6
Layer Elevations and Hydrogeologic Units
North-South Cross-Section View

Scale: 1 in. = 400 ft.	Date: Apr 2000
Prepared By: HSI GeoTrans	By: LMG

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0 900 1800

Scale in Feet

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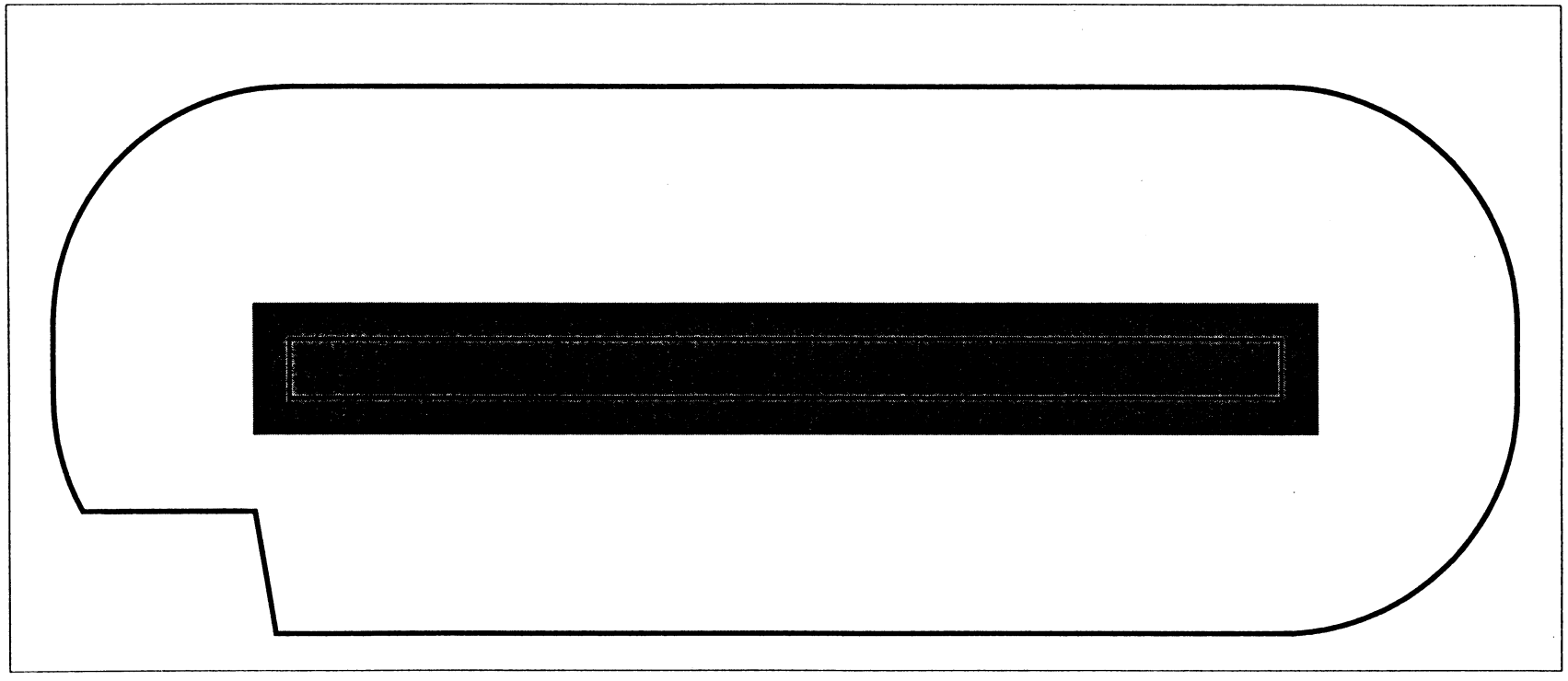
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 7
Hydrogeologic Units
East-West Cross-Section View

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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B-45

Legend

- Glacial Outwash
- Glacial Till

0 900 1800

Scale in Feet

N

HSI GEOTRANS

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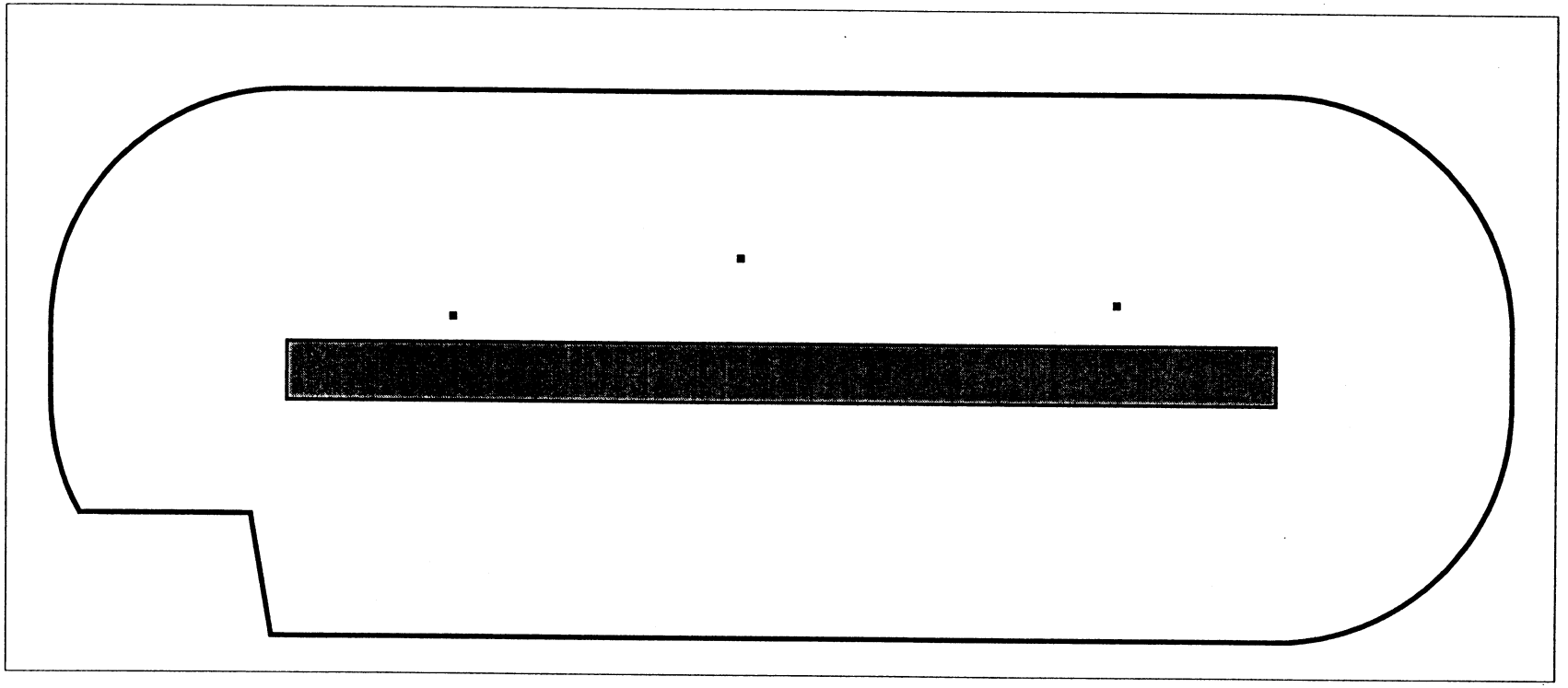
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APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 8
Hydrogeologic Units in Layer 1

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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B-46

Legend

- Crown Pillar
- Strongly-Weathered Bedrock
- Open Workings (shafts)

0 900 1800

Scale in Feet

N

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

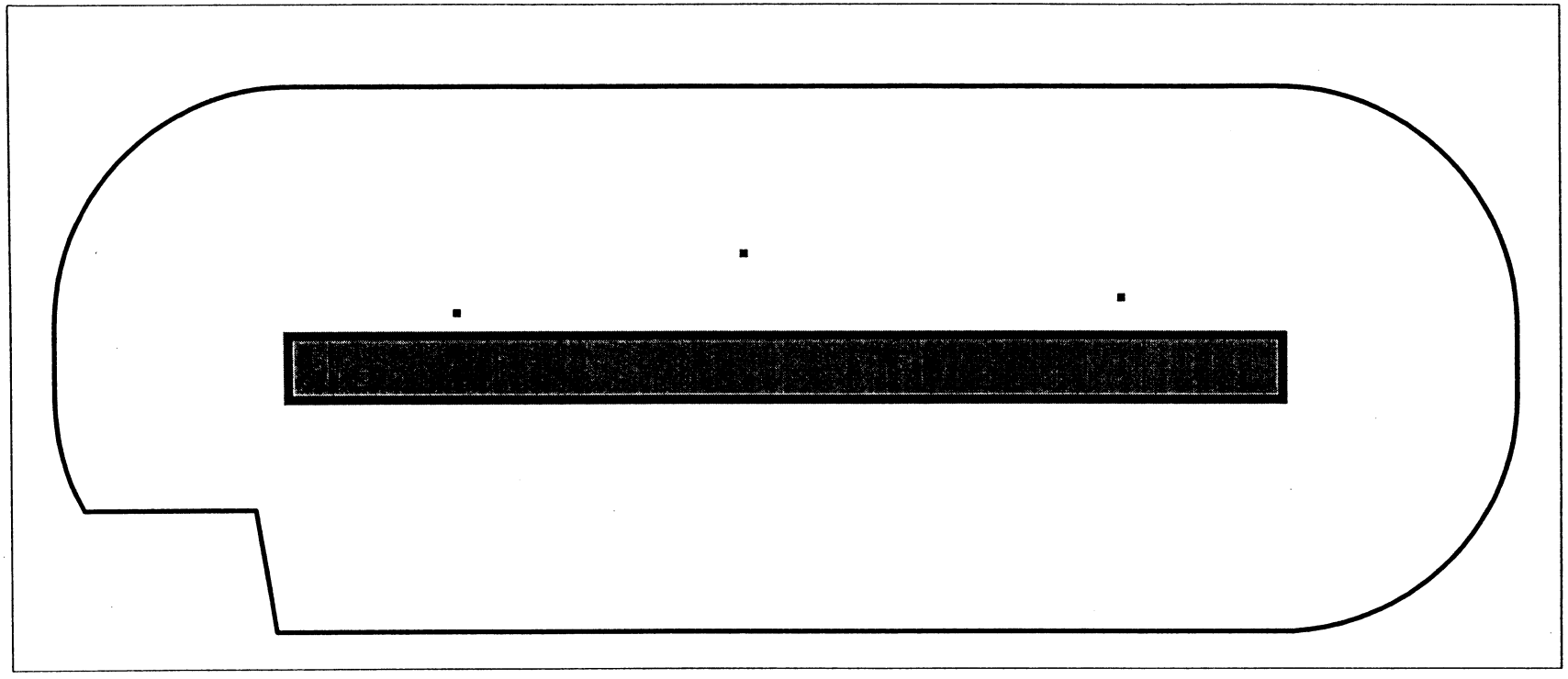
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 9
Hydrogeologic Units in Layer 3

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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B-47

Legend

- Crown Pillar
- Strongly-Weathered Bedrock
- Moderately-Weathered Bedrock
- Open Workings (shafts)

0 900 1800

Scale in Feet

N

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

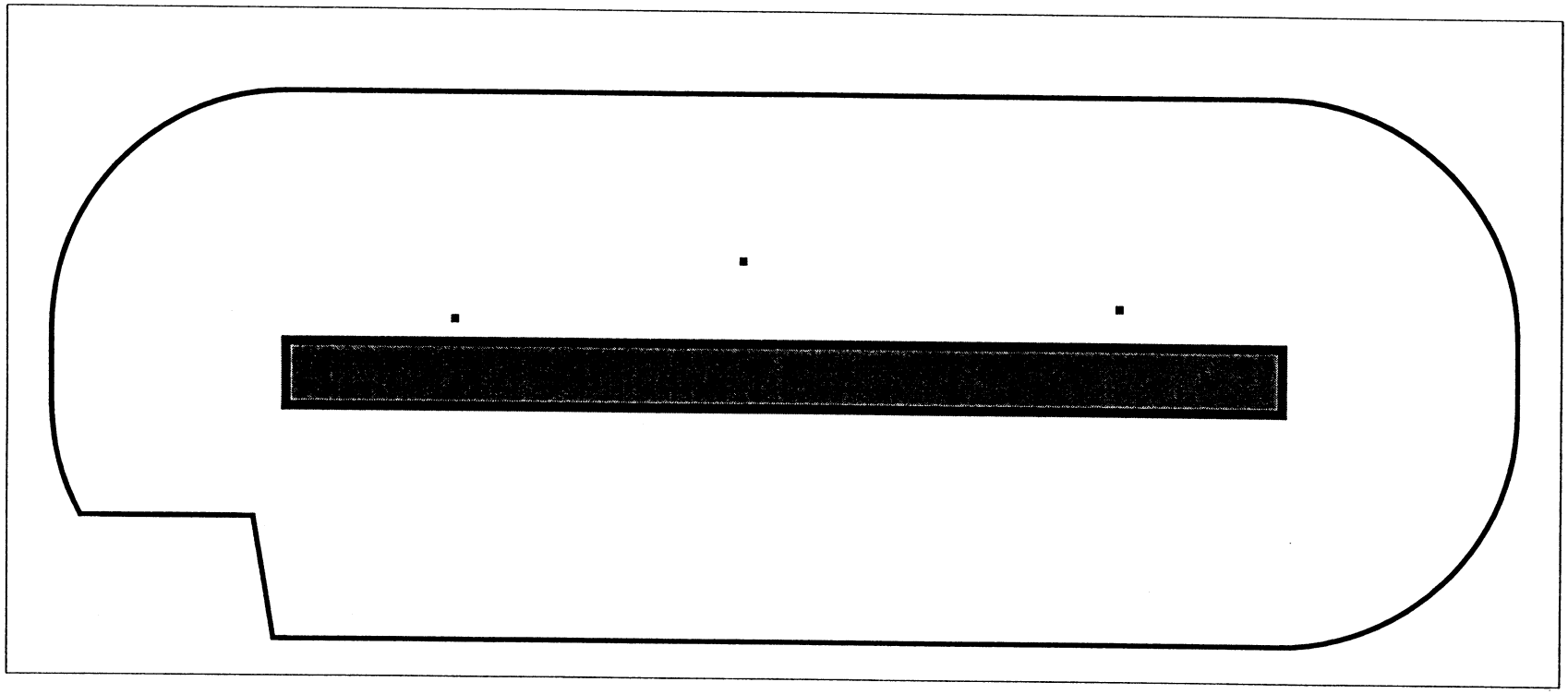
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APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 10
Hydrogeologic Units in Layer 4

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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B-48

Legend

- Strongly-Weathered Bedrock
- Moderately-Weathered Bedrock
- Open Workings (shafts)
- Grouted Crown Pillar

0 900 1800

Scale in Feet

N

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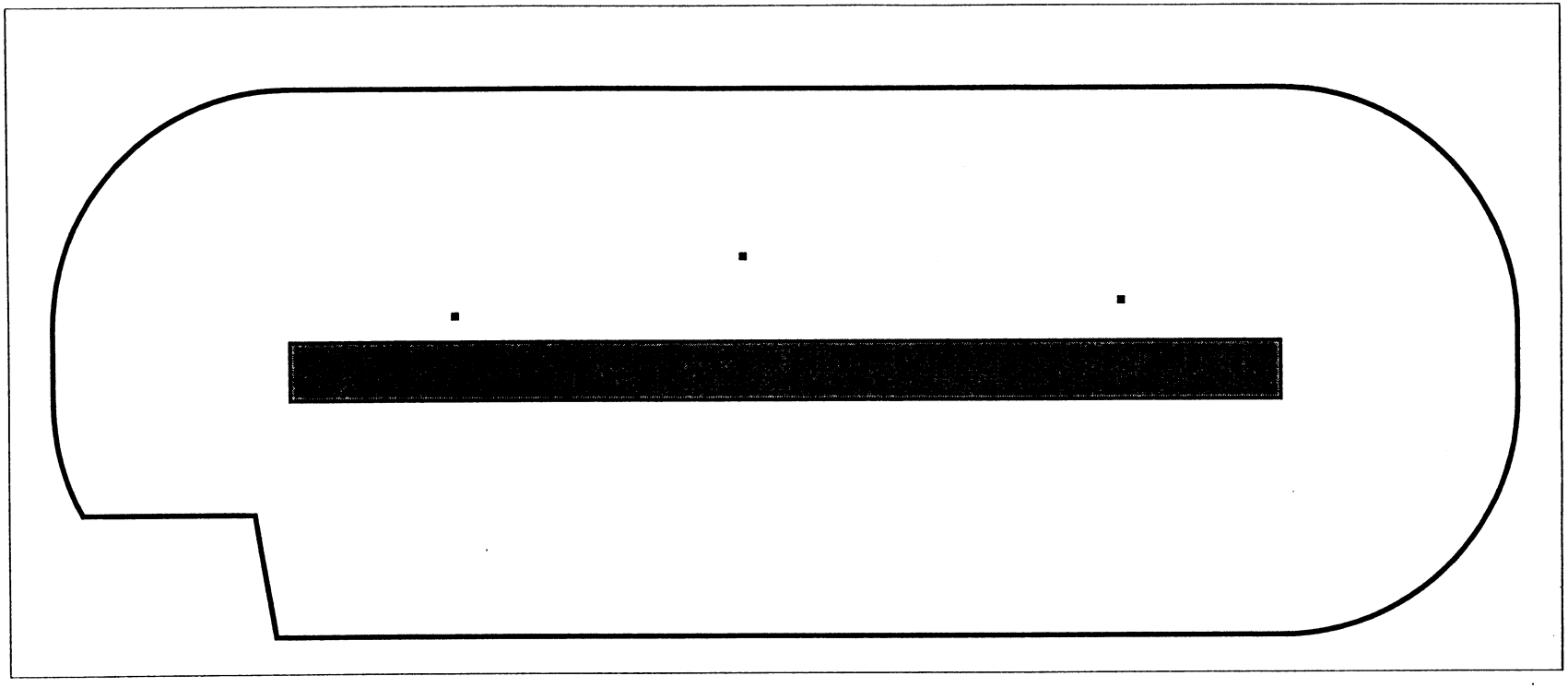
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APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 11
Hydrogeologic Unit in Layer 5

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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B-49

Legend

- Cemented Paste Backfill
- Moderately-Weathered Bedrock
- Open Workings (shafts)

0 900 1800

Scale in Feet

N

HSI GEOTRANS

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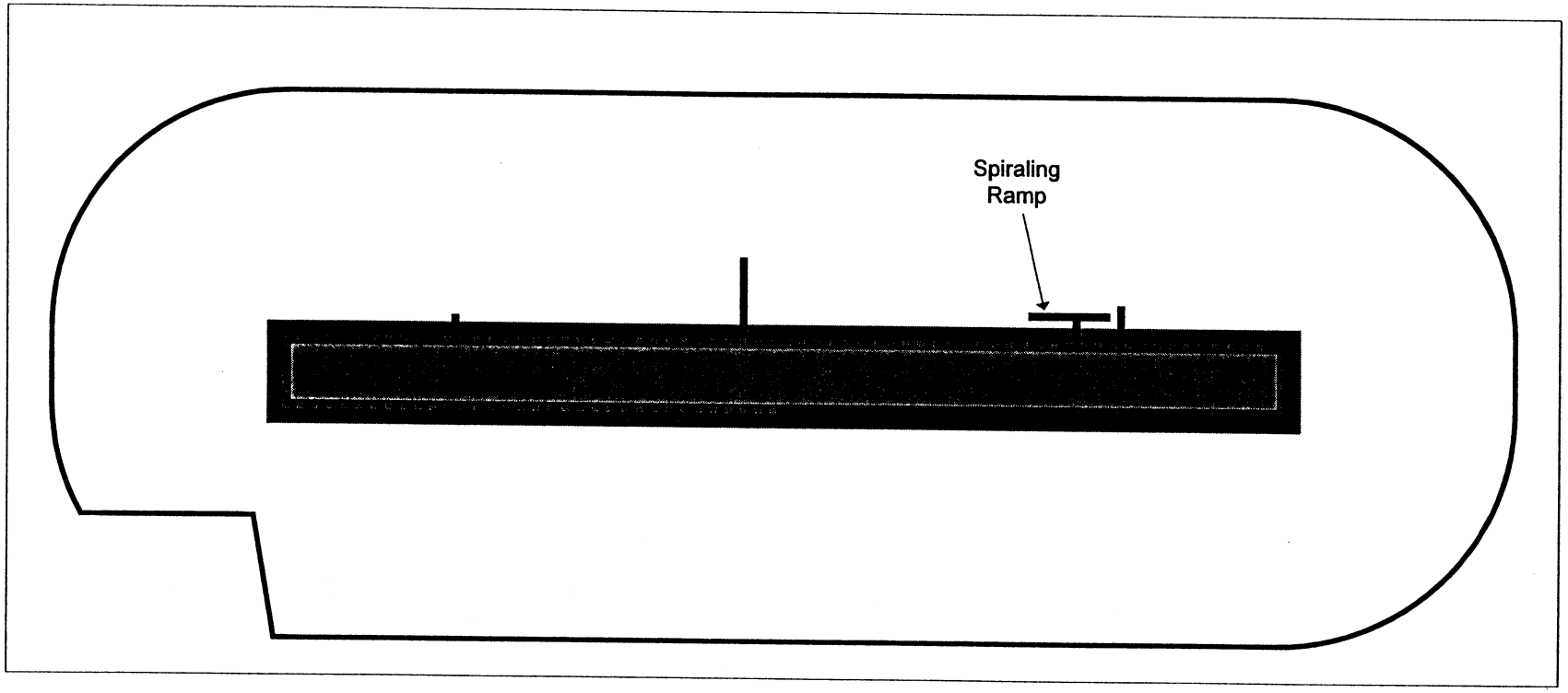
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APPROVED BY: PFA	DATE: Oct 2000
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Figure 12
Hydrogeologic Units in Layers 6 and 7

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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B-50

Legend

- Cemented Paste Backfill
- Moderately-Weathered Bedrock
- Weakly and Unweathered Bedrock
- Open Workings

0 900 1800

Scale in Feet

N

HSI GEOTRANS

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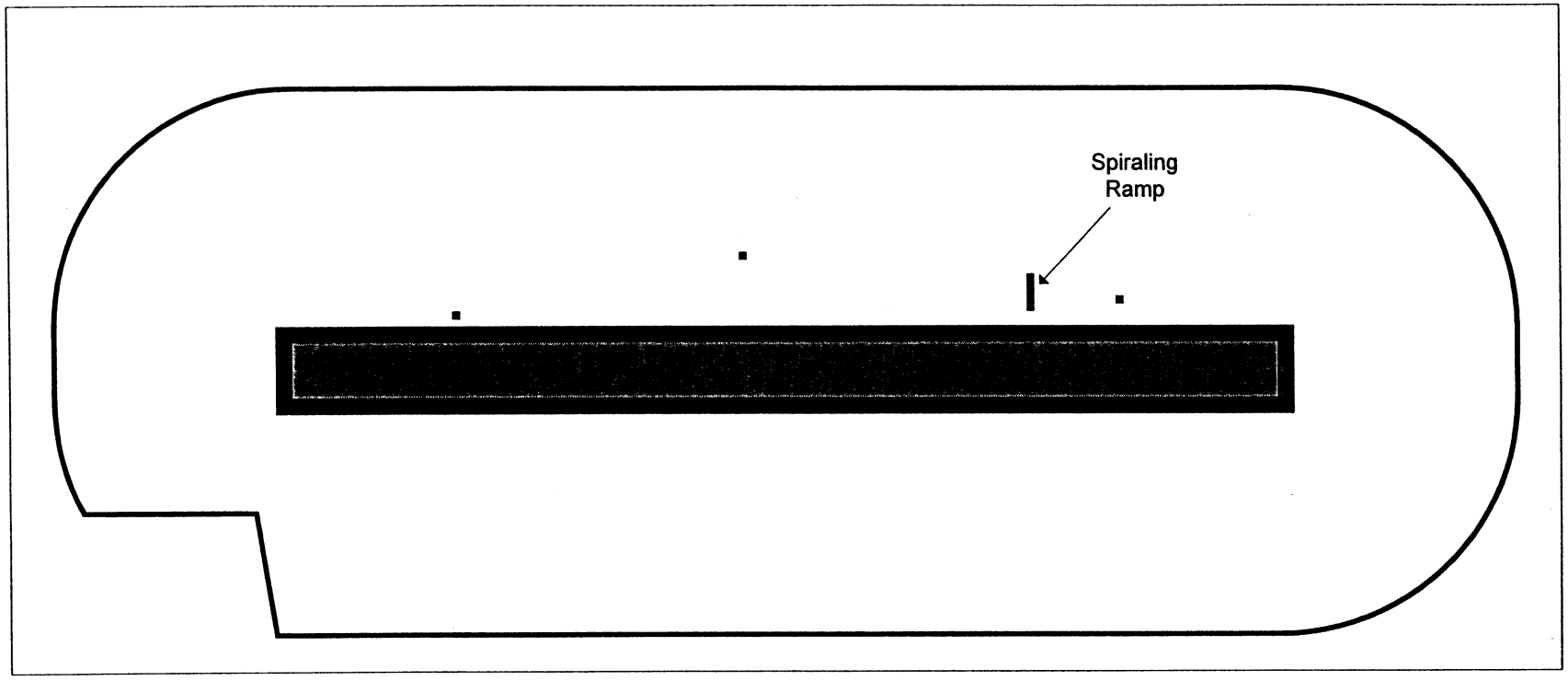
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 13
Hydrogeologic Units in Layer 8

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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Legend

- Cemented Paste Backfill
- Moderately-Weathered Bedrock
- Weakly and Unweathered Bedrock
- Open Workings

0 900 1800

Scale in Feet

N

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REVISED	DATE	BY	DESCRIPTION

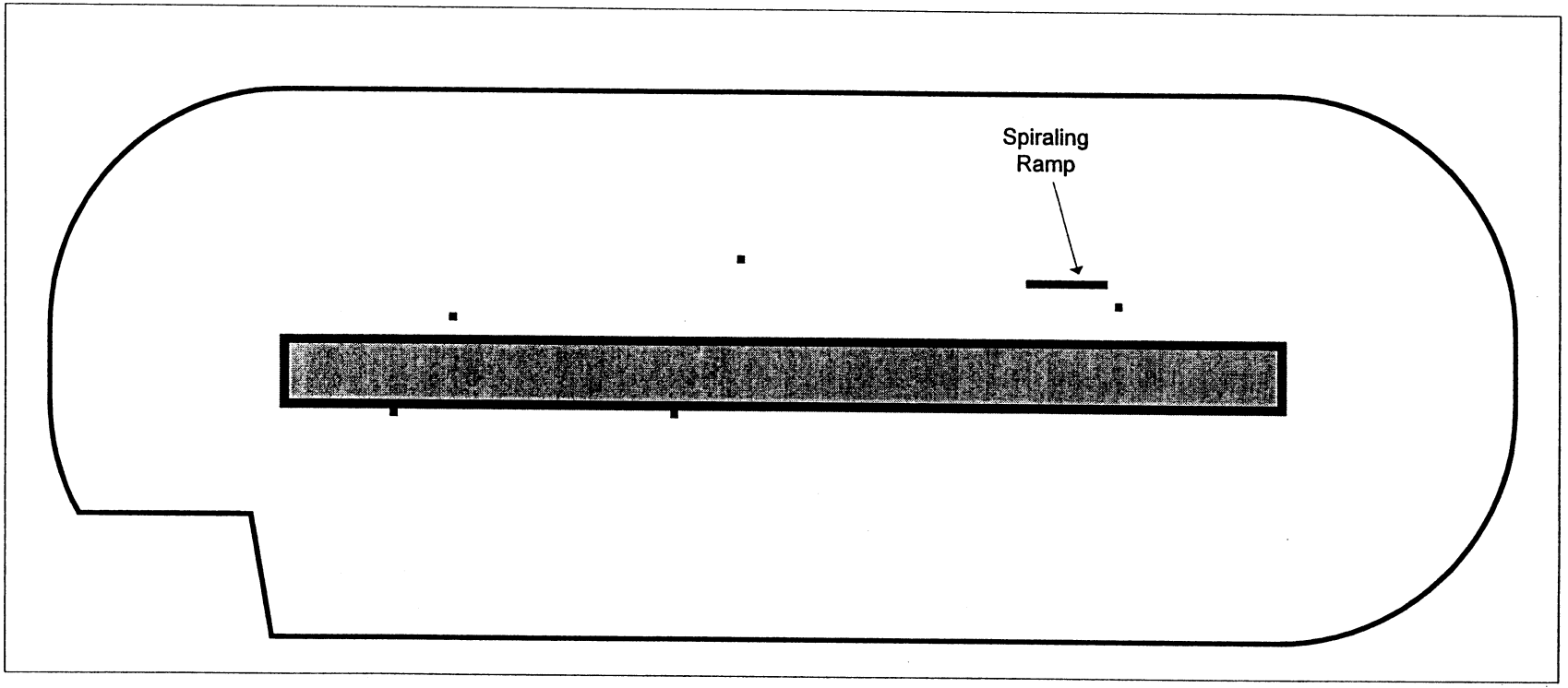
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APPROVED BY:	DATE:

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Figure 14
Hydrogeologic Units in Layer 9

Scale: 1 in. = 900 ft.	Date: Oct 2000
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B-52

Legend

- Cemented Paste Backfill
- Moderately-Weathered Bedrock
- Weakly and Unweathered Bedrock
- Open Workings

0 900 1800

Scale in Feet

N

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REVISED	DATE	BY	DESCRIPTION

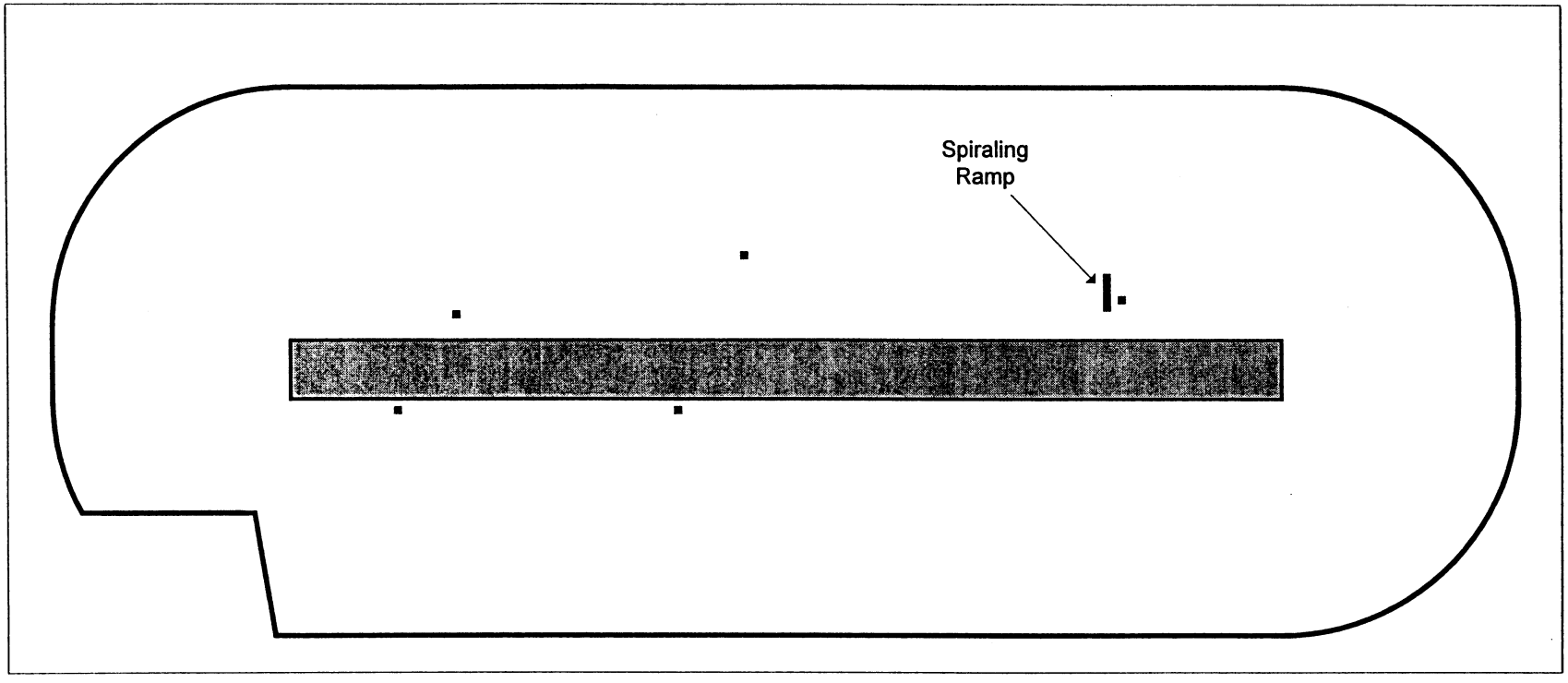
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APPROVED BY: PFA	DATE: Oct 2000
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Figure 15
Hydrogeologic Units in Layer 10

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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Legend

- Cemented Paste Backfill
- Weakly and Unweathered Bedrock
- Open Workings

0 900 1800

Scale in Feet

N

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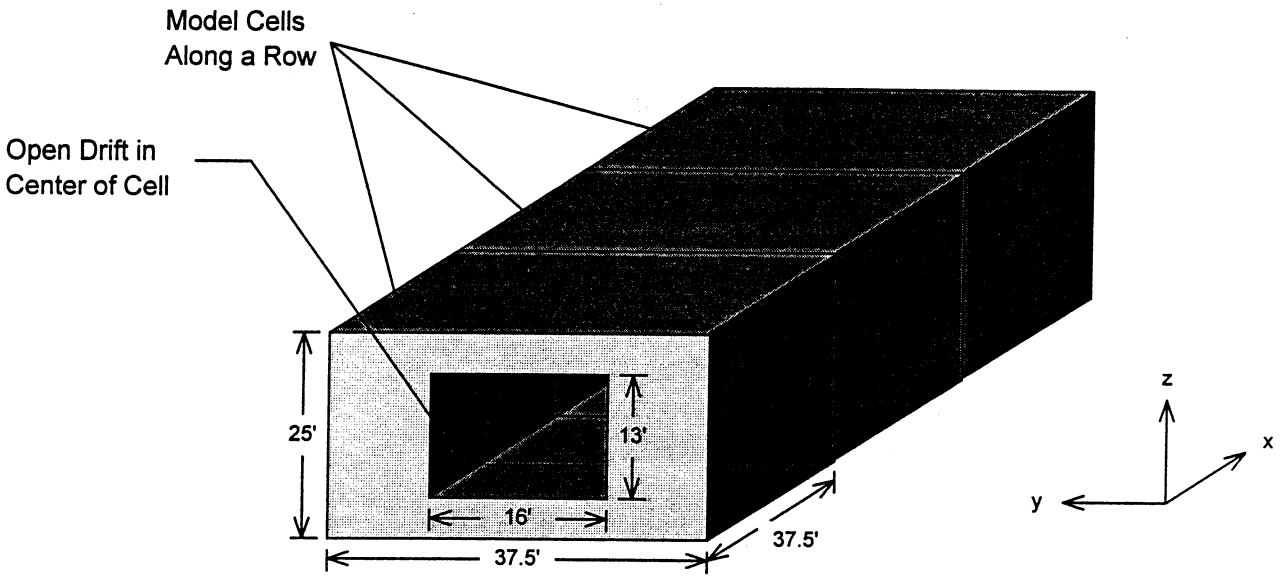
REVISED	DATE	BY	DESCRIPTION

CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 16
Hydrogeologic Units in Layer 11

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG



Calculation of Porosity:

Volume of Cell = 37.5 ft x 25 ft x 37.5 ft = 35,156 ft^3

Volume of Drift in Cell = 16 ft x 13 ft x 37.5 ft = 7800 ft^3

If rock porosity < 1%, Cell Porosity ~ Vol Drift / Vol Cell = **22%**

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CHECKED BY: LMG	DATE: Nov 2000
APPROVED BY: PFA	DATE: Nov 2000
APPROVED BY:	DATE:

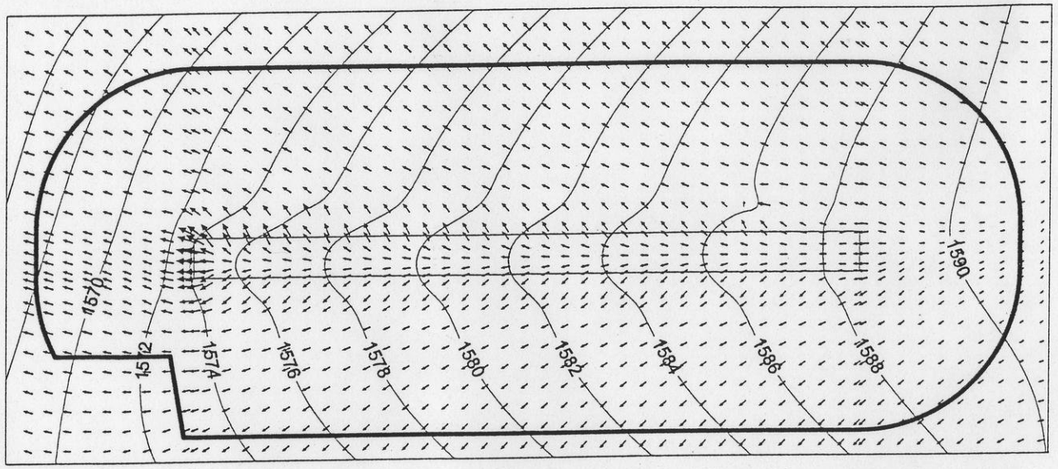
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Figure 17
Representation of Open Workings

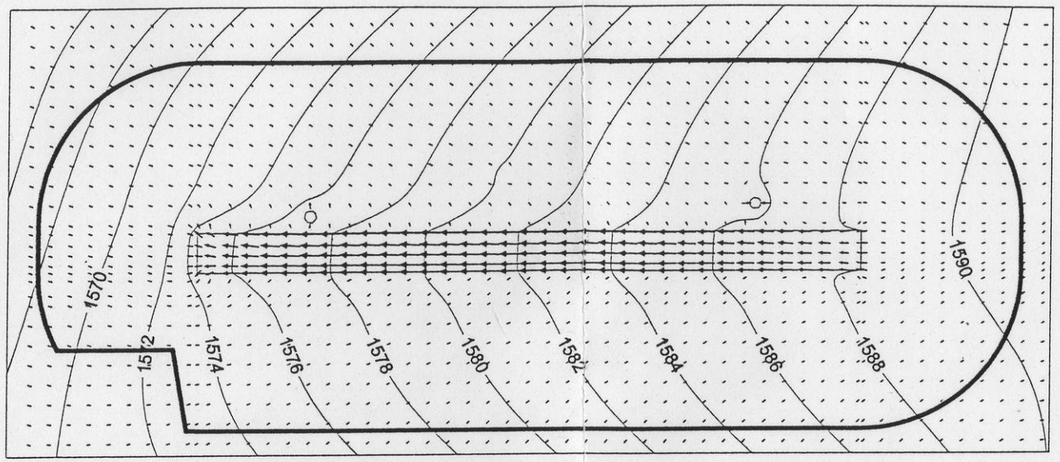
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Prepared By: HSI GEOTRANS	By: GWC

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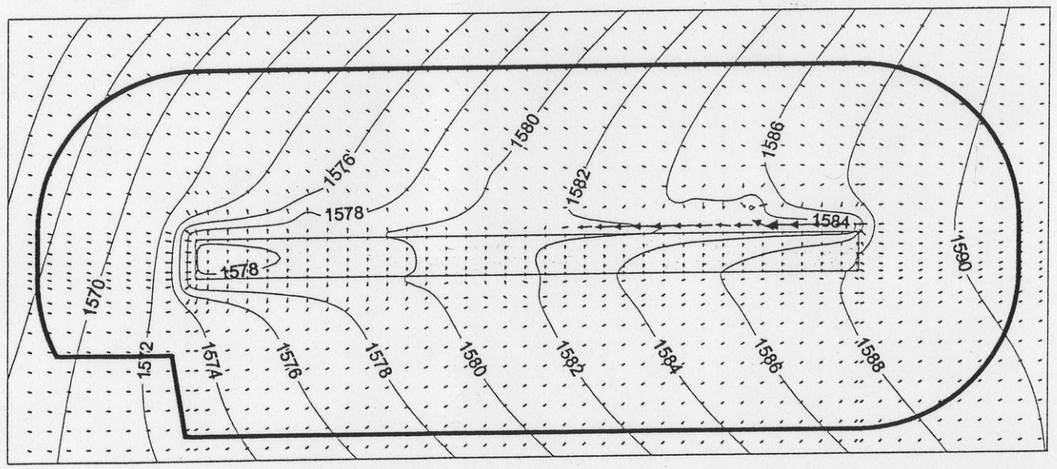
Layer 2 (Glacial Till)



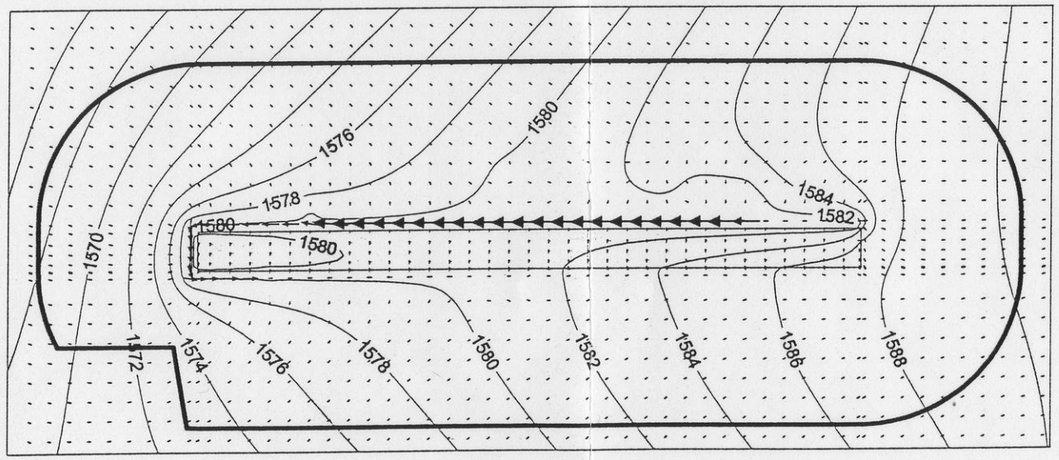
Layer 3 (Strongly-Weathered Bedrock and Crown Pillar)



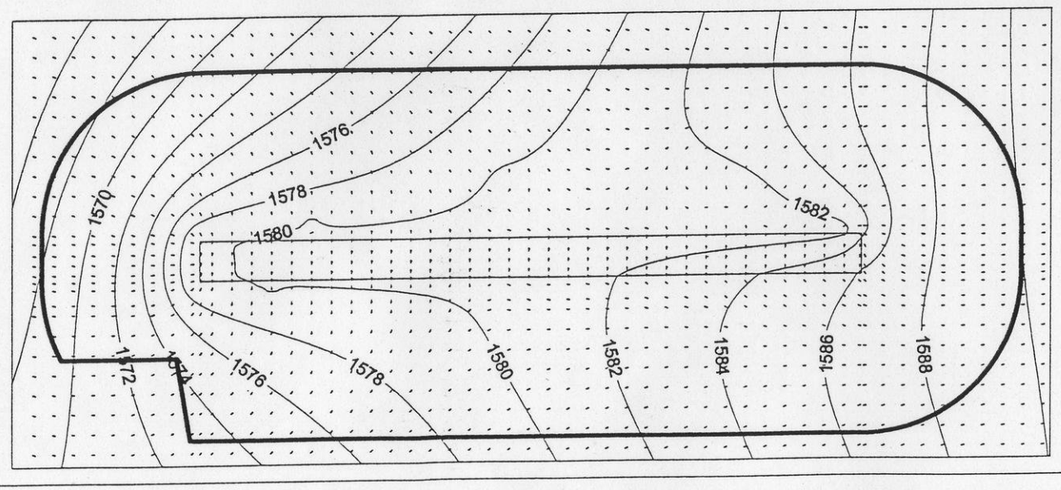
Layer 7 (Moderately-Weathered Bedrock and Top of Backfill)

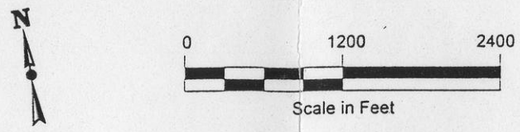


Layer 12 (Weakly-Weathered Bedock and Backfill at 500 ft Level)



Layer 26 (Unweathered Bedrock and Backfill Below 1100 ft Level)





Note: Contour Interval is 2 ft.
Vector length is proportional to groundwater velocity magnitude.

HSI GEOTRANS

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Simulation Rfl147	CHECKED BY: GWC	DATE: Oct 2000
	APPROVED BY: PFA	DATE: Oct 2000
	APPROVED BY:	DATE:

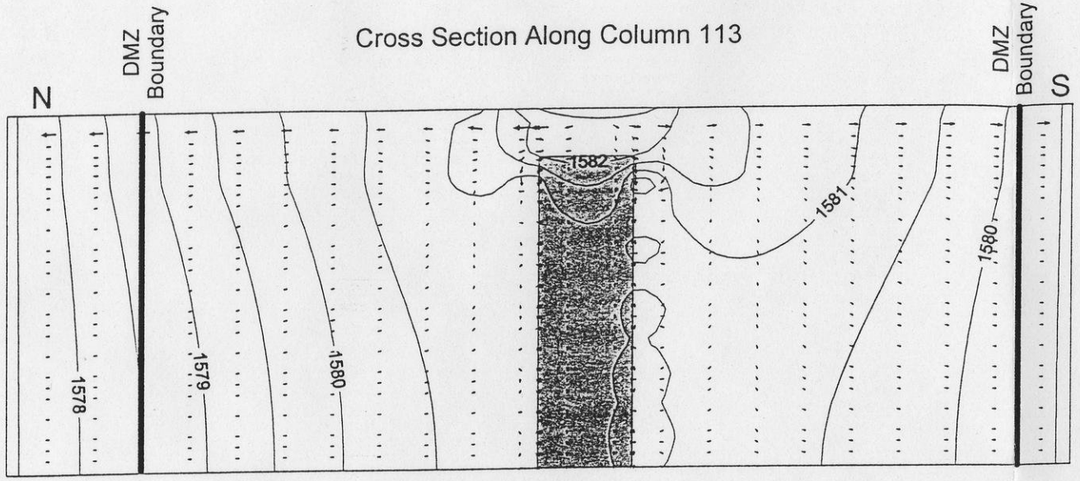
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Figure 18
Expected-Case Flow Field, Plan Views

Scale: 1 in. = 1200 ft.	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

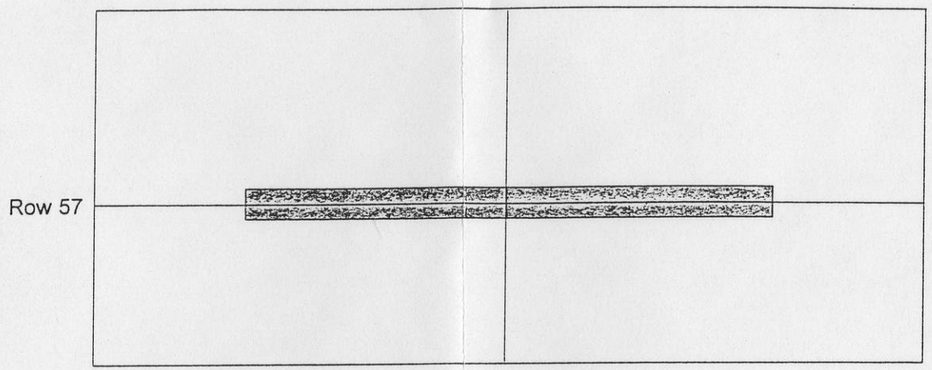
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Cross Section Along Column 113



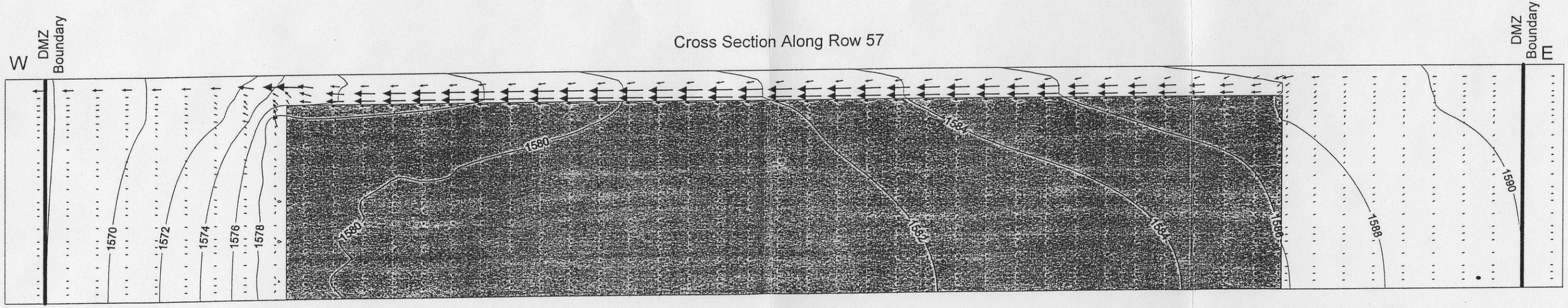
Note: Contour interval = 0.5 ft.

Locations of Cross-Sections



Column 113
Scale: 1"=1500'

Cross Section Along Row 57



Note: Contour interval = 2 ft.

0 500 1000
Scale in Feet

Note: Contour Interval is 2 ft.
Vector length is proportional to groundwater velocity magnitude.

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CHECKED BY: GWC			DATE: Oct 2000
APPROVED BY: PFA			DATE: Oct 2000
APPROVED BY:			DATE:

Simulation Rfl147

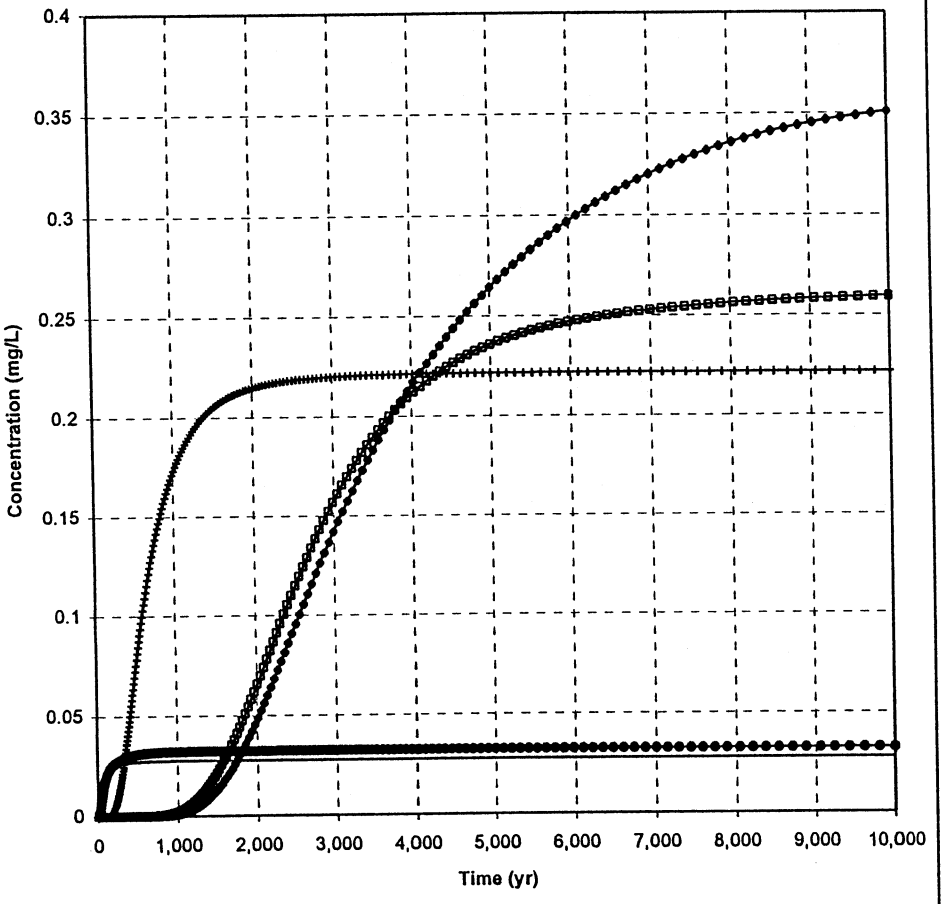
Nicolet Minerals
a Rio Algom company

Figure 19
Expected-Case Flow Field, Cross Section Views

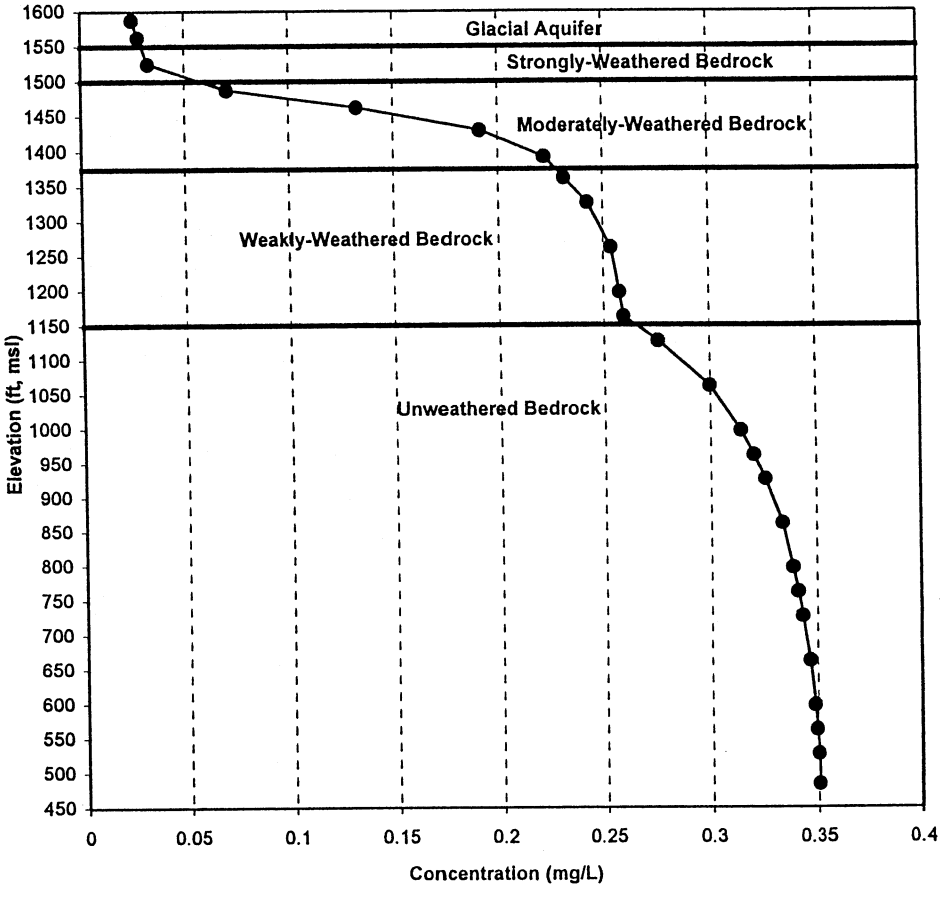
Scale: 1 in. = 500 ft.	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

P:\Crandon\Reflood3d\Surfer\Figures\rfl147cross.srf

p:\Crandon\Refiood3d\excel\rf147.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer evaluated independent of the simulation time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 40- 7)	0.028
	Strongly-Weathered	(3- 40- 7)	0.032
	Moderately-Weathered	(7- 85- 34)	0.22
	Weakly-Weathered	(12-101- 57)	0.26
	Unweathered	(26-101- 73)	0.35

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REVISED	DATE	BY	DESCRIPTION

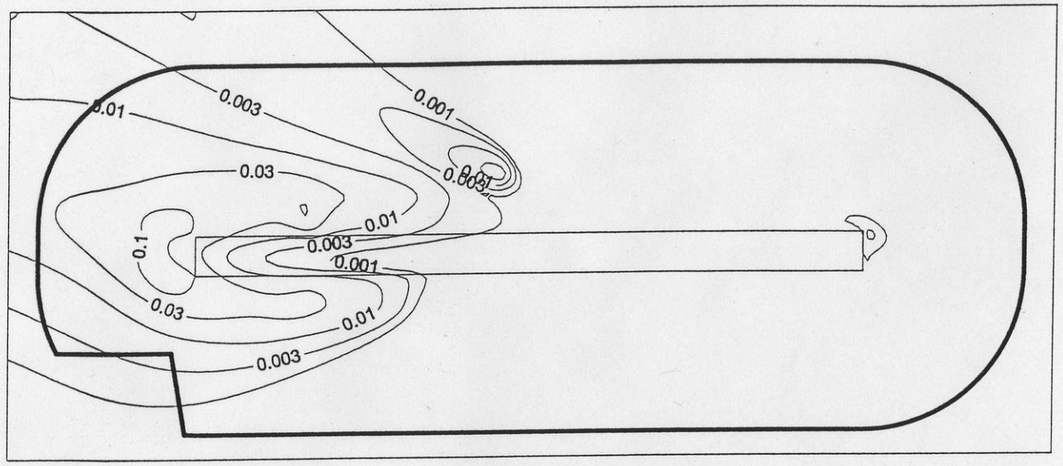
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

Nicolet Minerals
a Rio Algom company

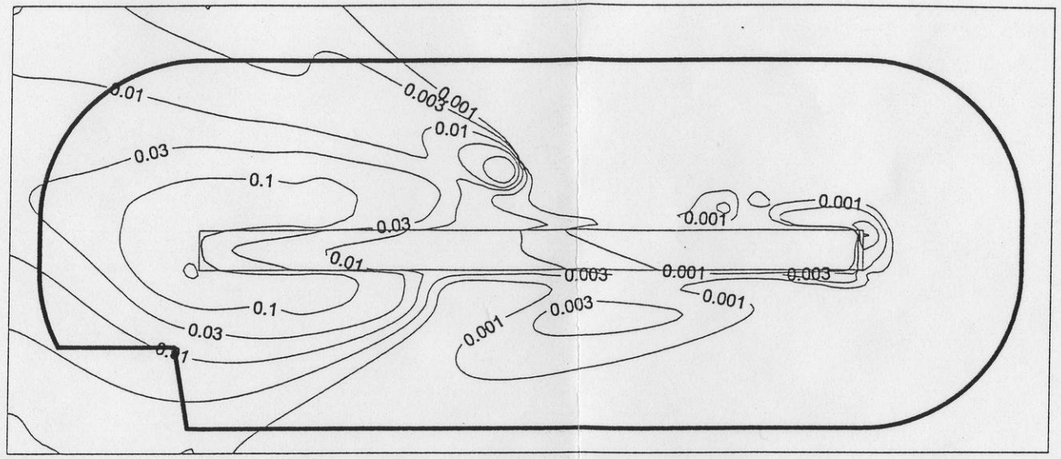
Figure 20
Expected-Case Concentration at the DMZ Boundary, Unoxidized Backfill Source

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

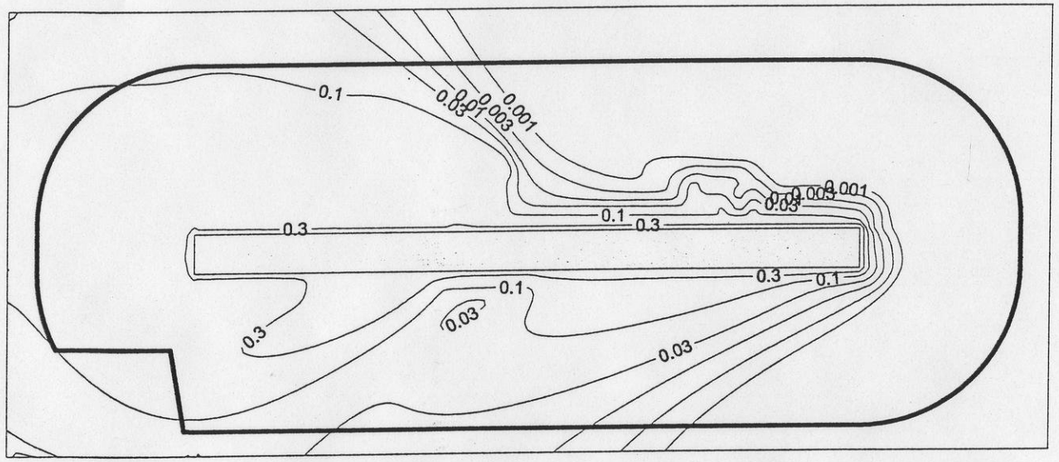
Layer 2 (Glacial Till)



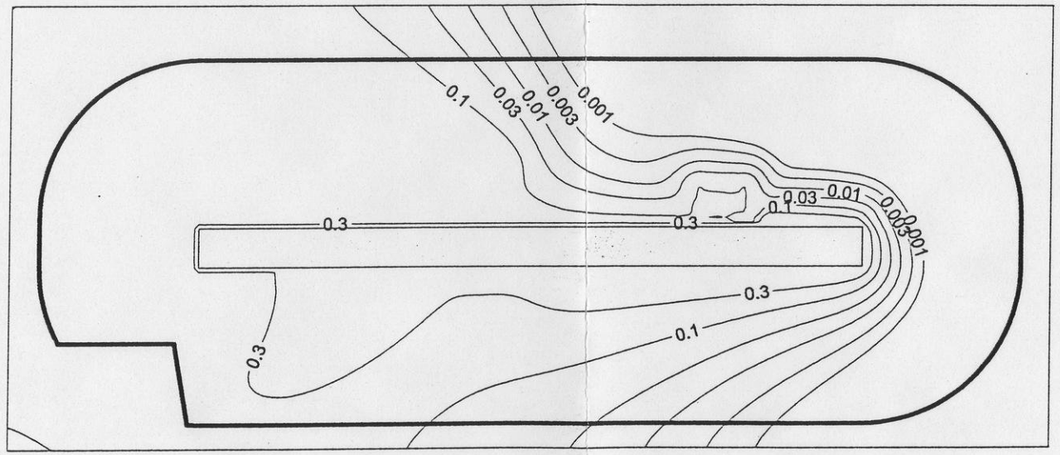
Layer 3 (Strongly-weathered Bedrock and Crown Pillar)



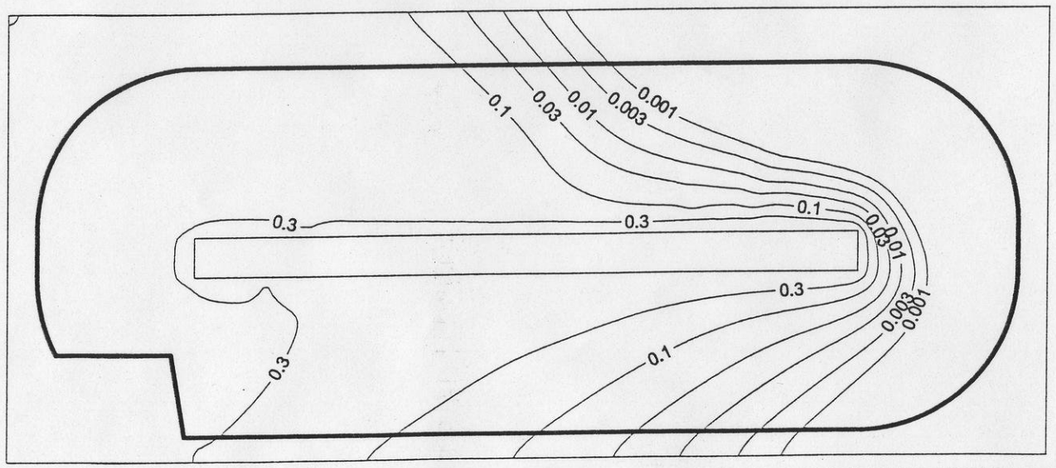
Layer 7 (Moderately-Weathered Bedrock and Top of Backfill)



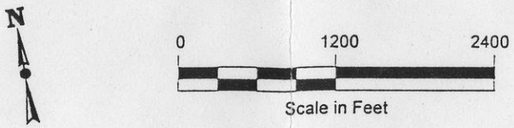
Layer 12 (Weakly-Weathered Bedrock and backfill at 500 ft Level)



Layer 26 (Unweathered Bedrock and Backfill Below 1100 ft Level)



Note: Relative concentration, expressed as a fraction of the source concentration, is shown on these plots.
Note: The concentration after 10,000 years of simulation time is shown.



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REVISED	DATE	BY	DESCRIPTION

Simulation Rfl47	CHECKED BY: GWC	DATE: Oct 2000
	APPROVED BY: PFA	DATE: Oct 2000
	APPROVED BY:	DATE:

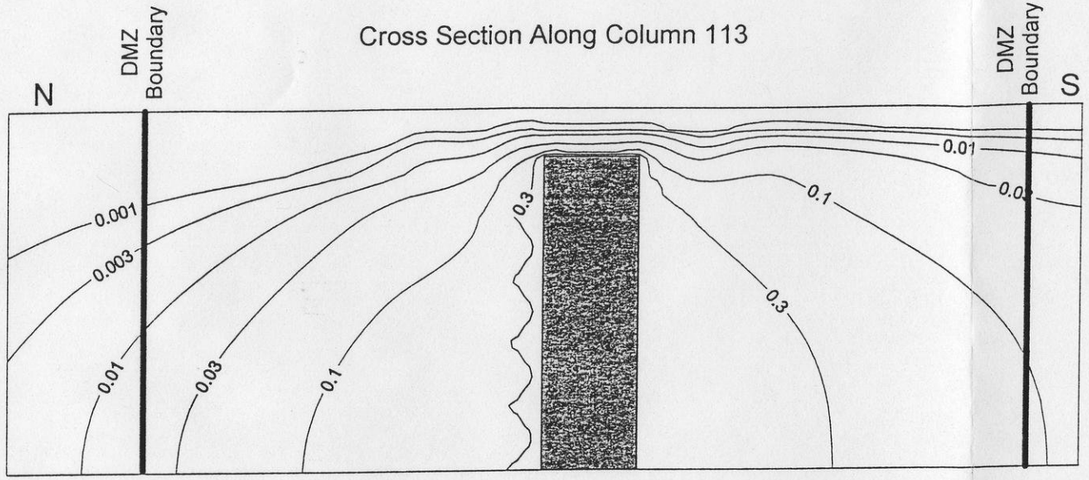
Nicolet Minerals
a Rio Algom company

Figure 21
Expected-Case Concentration Field, Unoxidized Backfill Source, Plan Views

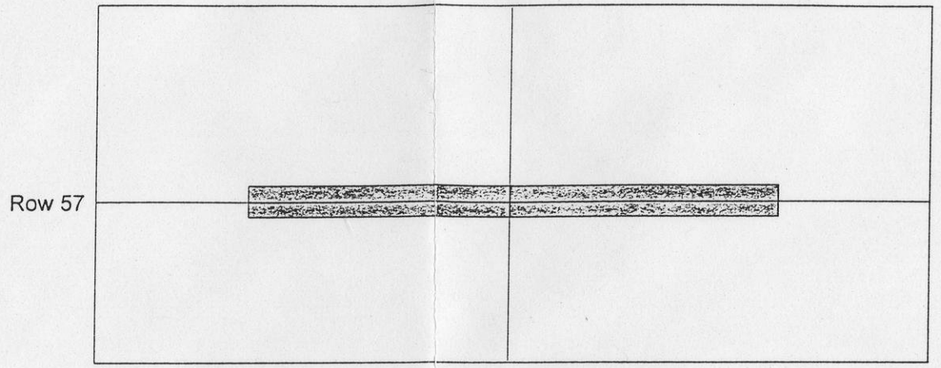
Scale: 1 in. = 1200 ft.	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

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Cross Section Along Column 113

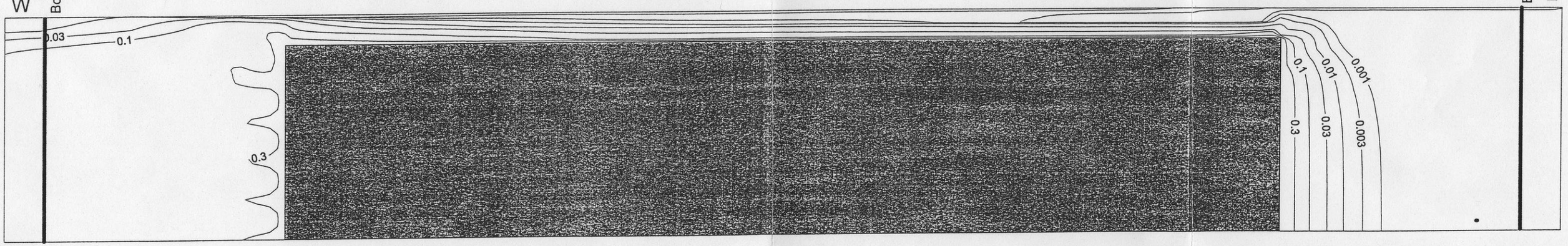


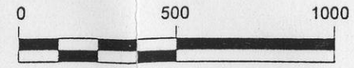
Locations of Cross-Sections



Scale: 1"=1500'

Cross Section Along Row 57





Note: Relative concentration, expressed as a fraction of the source concentration, is shown on these plots.
Note: The concentration after 10,000 years of simulation time is shown.

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REVISED	DATE	BY	DESCRIPTION

Simulation Rfl147	CHECKED BY: GWC	DATE: Oct 2000
	APPROVED BY: PFA	DATE: Oct 2000
	APPROVED BY:	DATE:

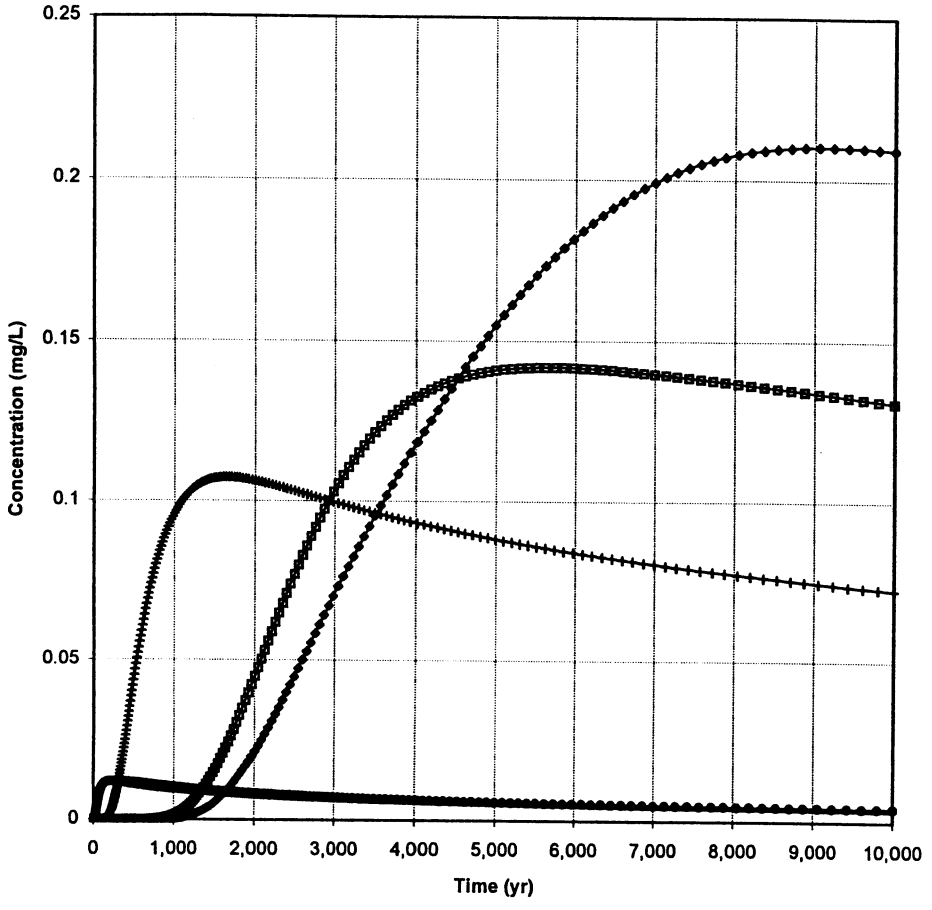
Nicolet Minerals
a Rio Algom company

Figure 22
Expected-Case Concentration Field, Unoxidized Backfill Source, Cross-Section Views

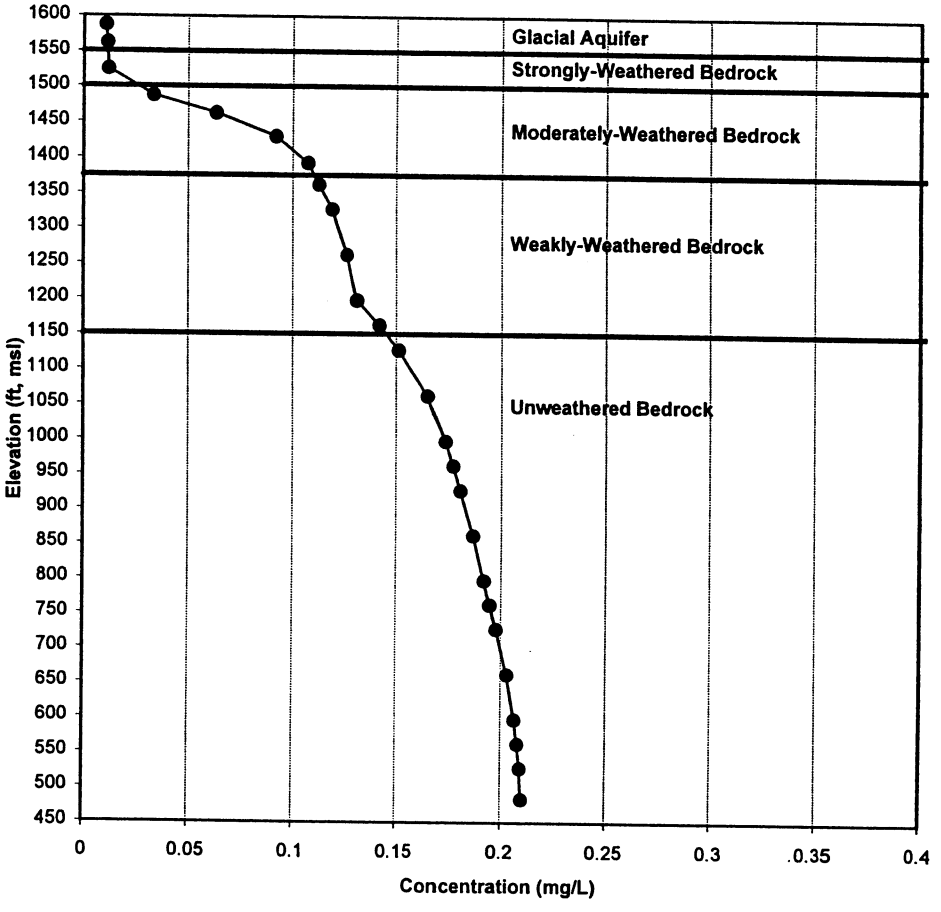
Scale: 1 in. = 500 ft.	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

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p:\Crandon\Reflood3d\excel\vf153.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 38- 7)	0.011
	Strongly-Weathered	(3- 39- 7)	0.012
	Moderately-Weathered	(7- 85- 34)	0.11
	Weakly-Weathered	(12-101- 59)	0.14
	Unweathered	(26-101- 79)	0.21

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

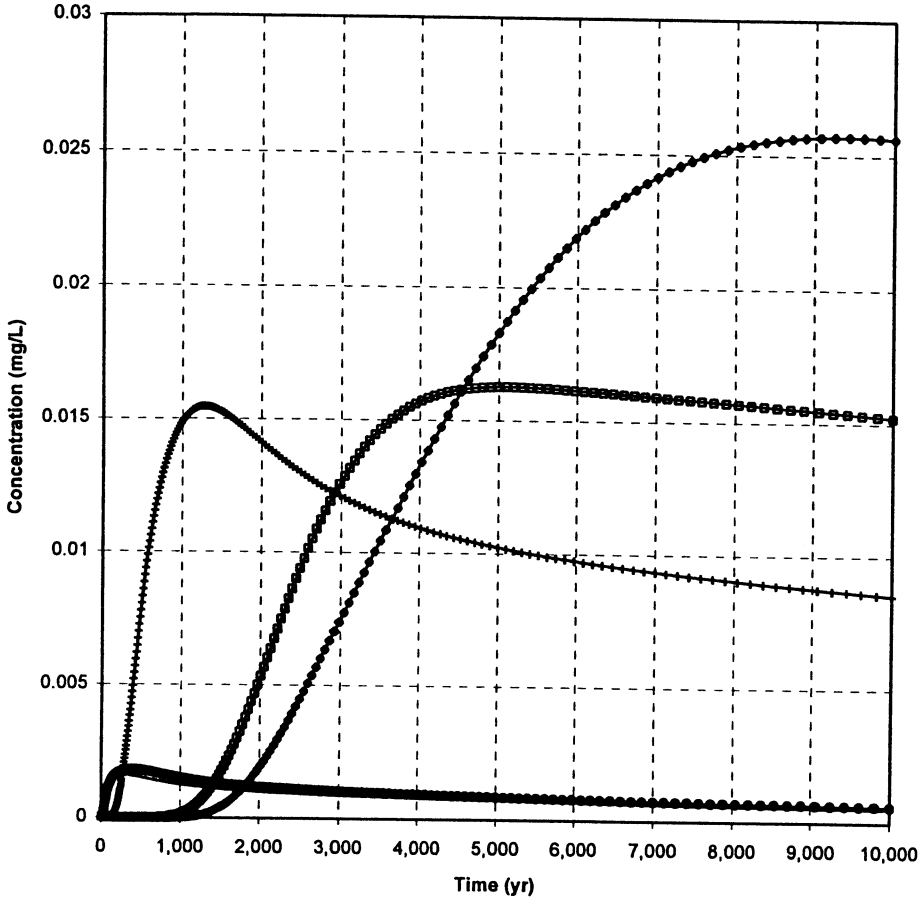
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

Nicolet Minerals
a Rio Algom company

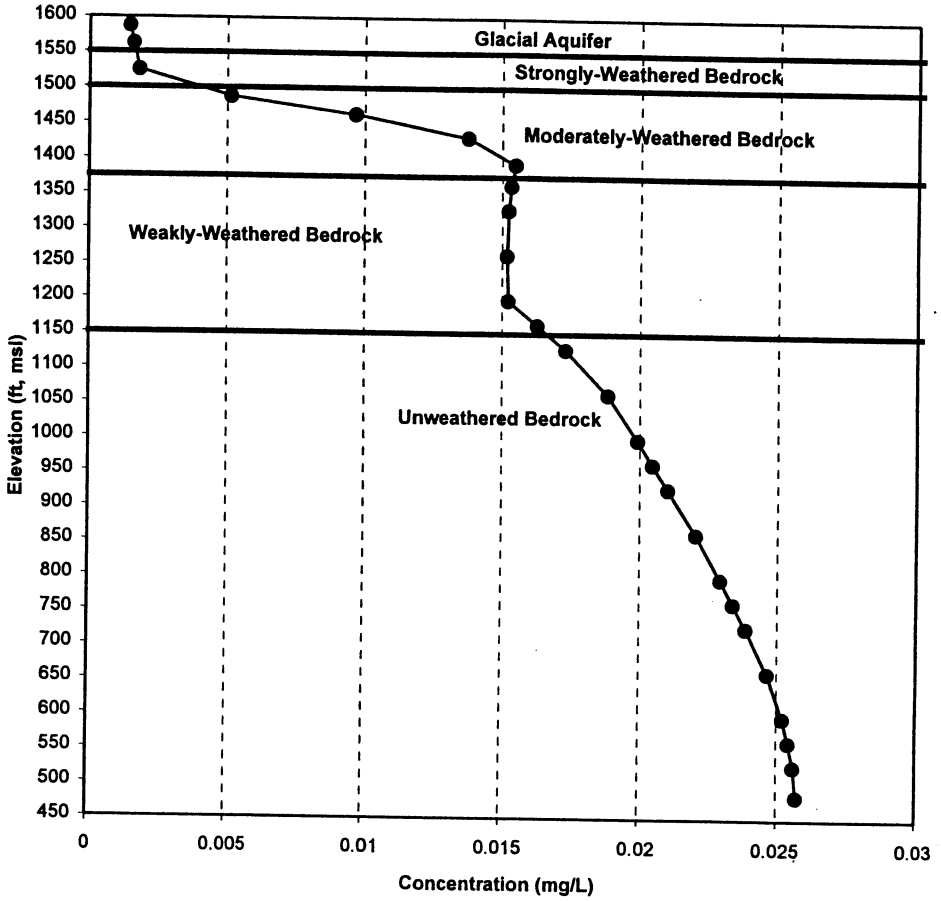
Figure 23
Expected-Case Concentration at the DMZ Boundary, Unoxidized Backfill Source (Instantaneous)

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

p:\Crandon\Reflood3d\excel\rf150.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 43- 7)	0.0016
	Strongly-Weathered	(3- 40- 7)	0.0018
	Moderately-Weathered	(7- 85- 34)	0.015
	Weakly-Weathered	(12-101- 60)	0.016
	Unweathered	(26-101- 81)	0.026

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

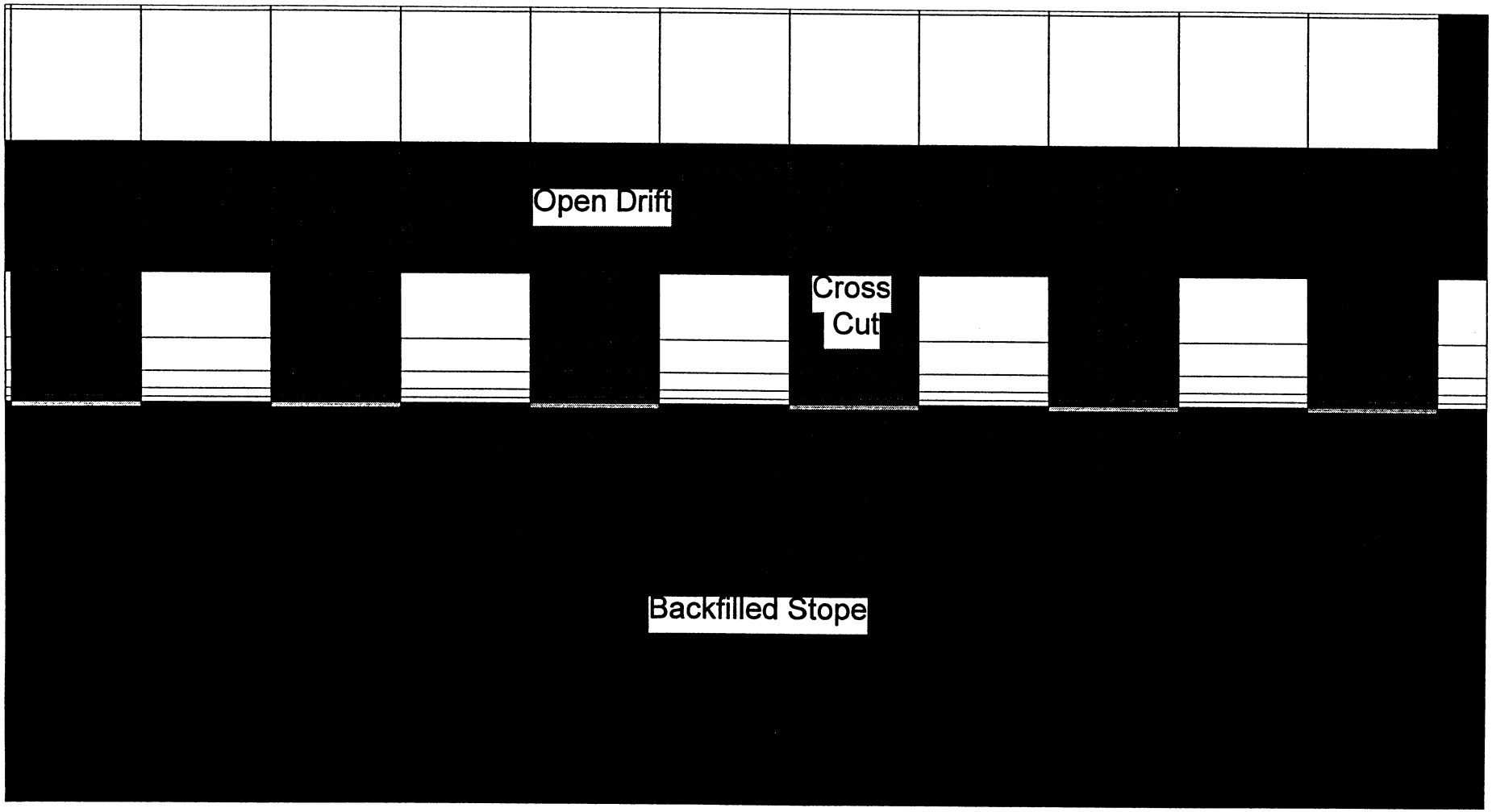
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

Nicolet Minerals
a Rio Algom company

Figure 24
Expected-Case Concentration at the DMZ Boundary, Oxidized-Neutral Backfill Source

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

P:\Crandon\Reflood3D\GIS\workspace\zoomprp.wor



Legend

- Acidic Zone Modeled in Backfill Cell
- Acidic Zone Modeled in Cross-Cut Cell
- Cemented Paste Backfill
- Open Workings
- Bedrock

0 45 90

Scale in Feet

N

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

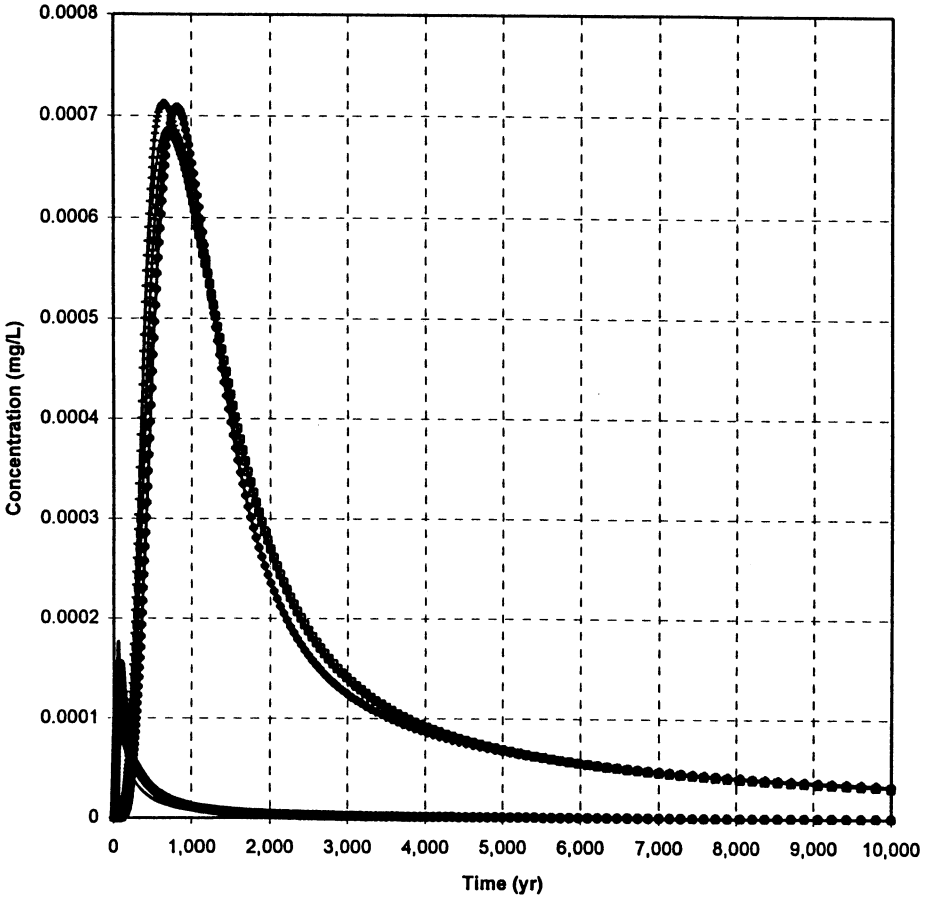
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

Nicolet Minerals
a Rio Algom company

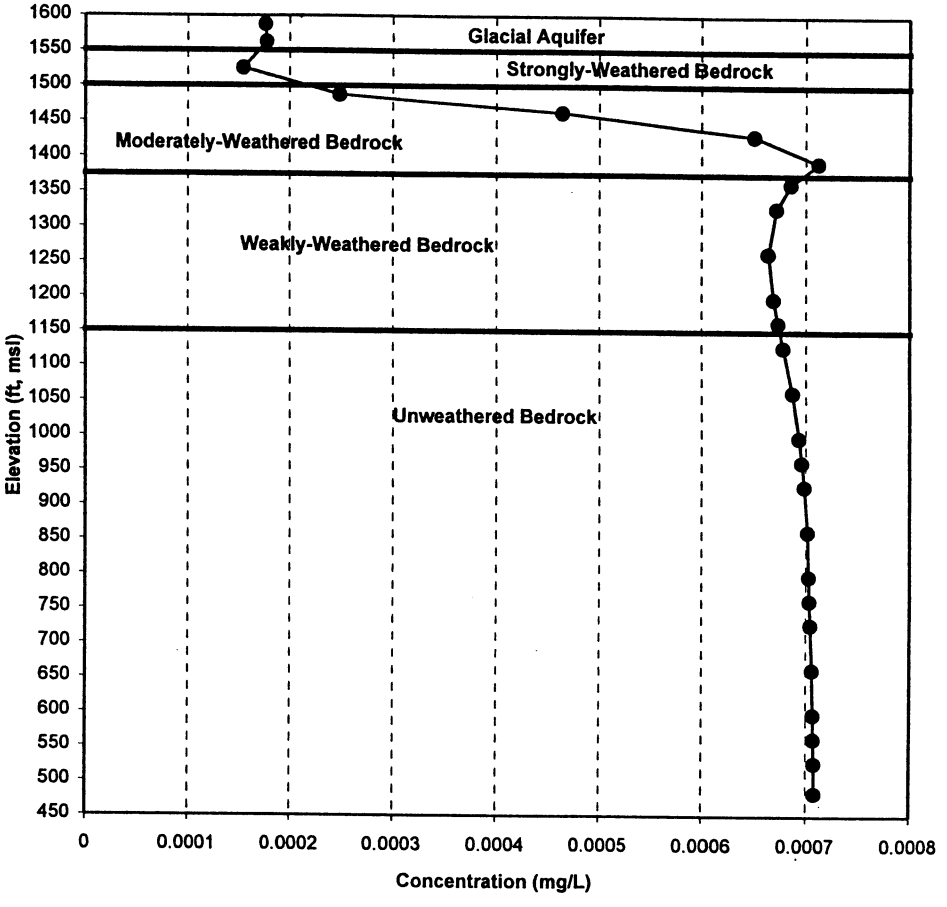
Figure 25
Location of Cells Used to Model the Acidic Backfill Source Zone

Scale: 1 in. = 45 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

p:\Crandon\Reflood3d\excel\rf148.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 35- 8)	0.00018
	Strongly-Weathered	(3- 35- 8)	0.00015
	Moderately-Weathered	(7- 85- 34)	0.00071
	Weakly-Weathered	(8- 85- 34)	0.00069
	Unweathered	(26- 85- 34)	0.00071

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

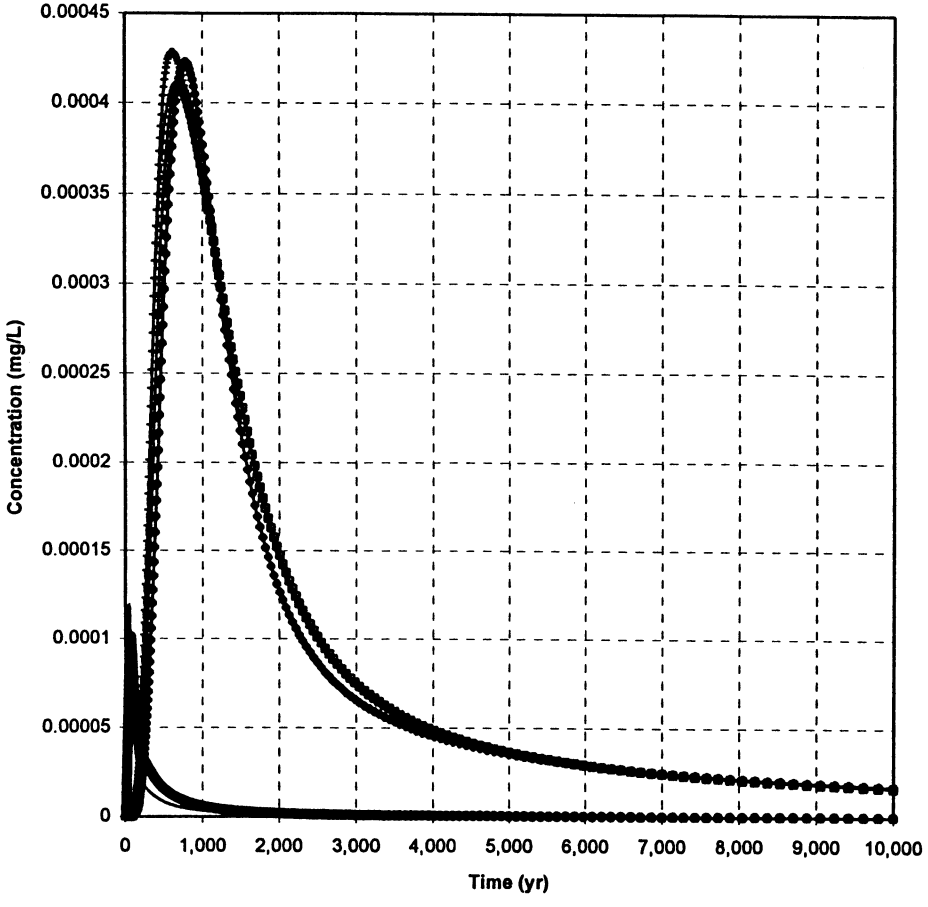
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

Nicolet Minerals
a Rio Algom company

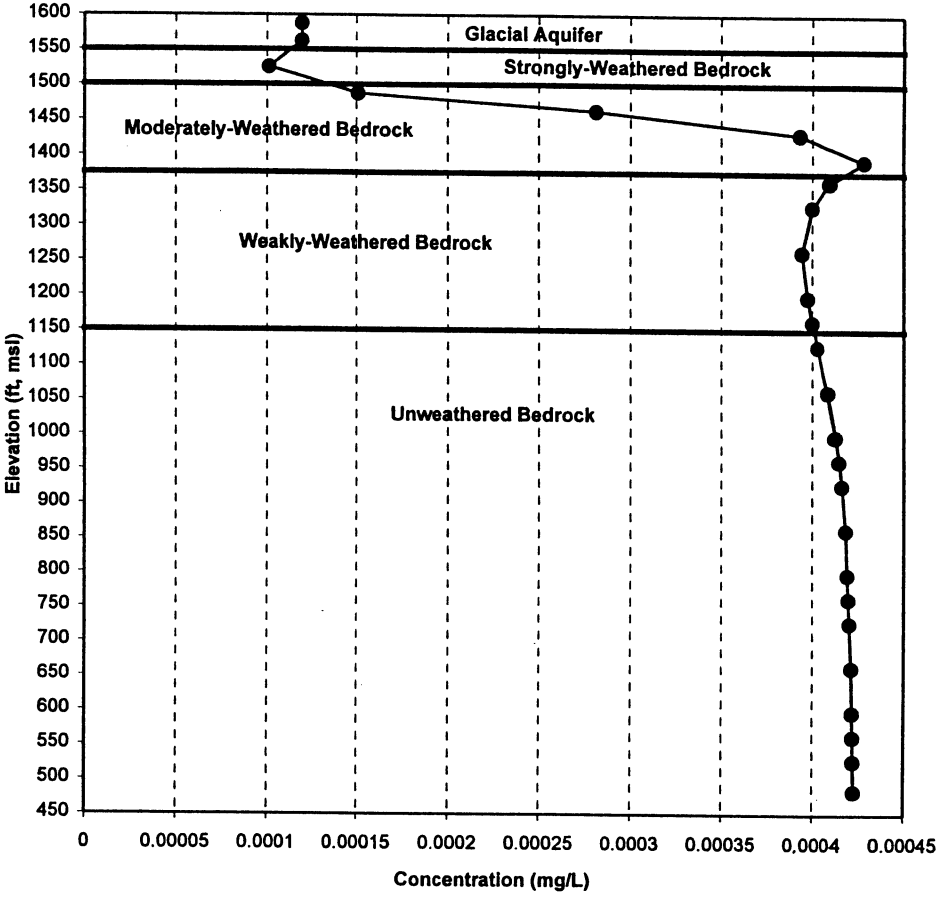
Figure 26
Expected-Case Concentration at the DMZ Boundary, Acidic Source in Backfill

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

p:\Crandon\Reflood3d\excel\rf149.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(1- 35- 8)	0.00012
	Strongly-Weathered	(3- 35- 8)	0.00010
	Moderately-Weathered	(7- 85- 34)	0.00043
	Weakly-Weathered	(8- 85- 34)	0.00041
	Unweathered	(26- 85- 34)	0.00042

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REVISED	DATE	BY	DESCRIPTION

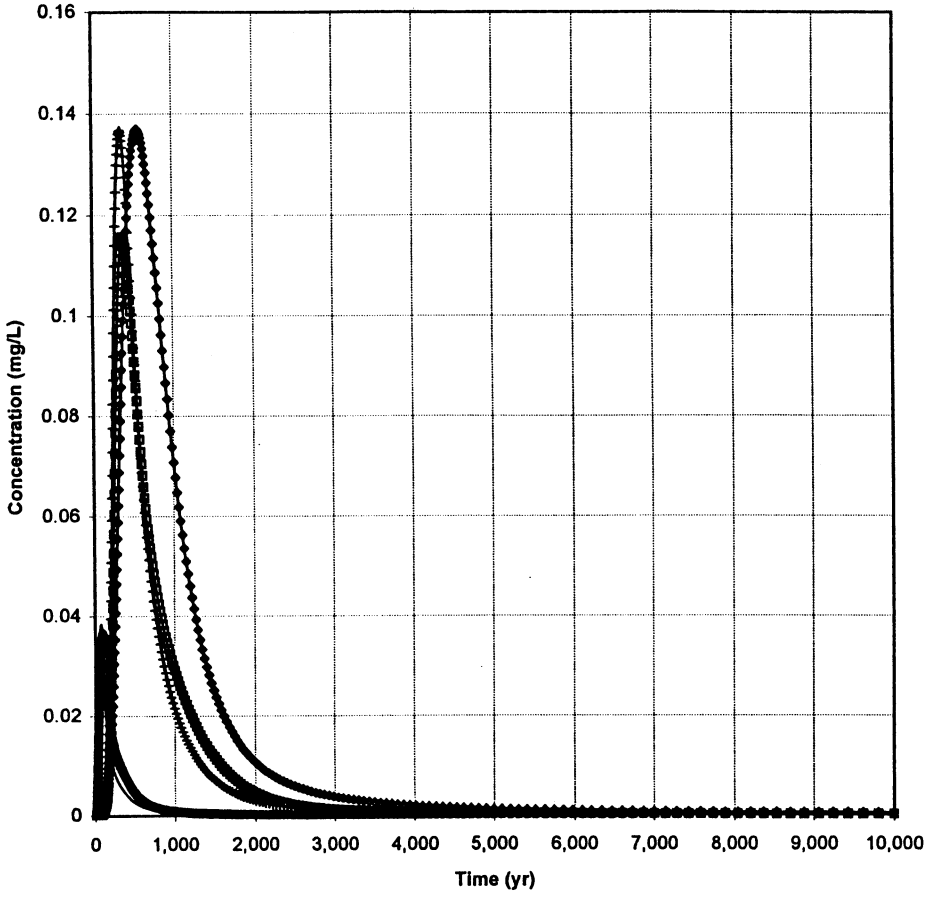
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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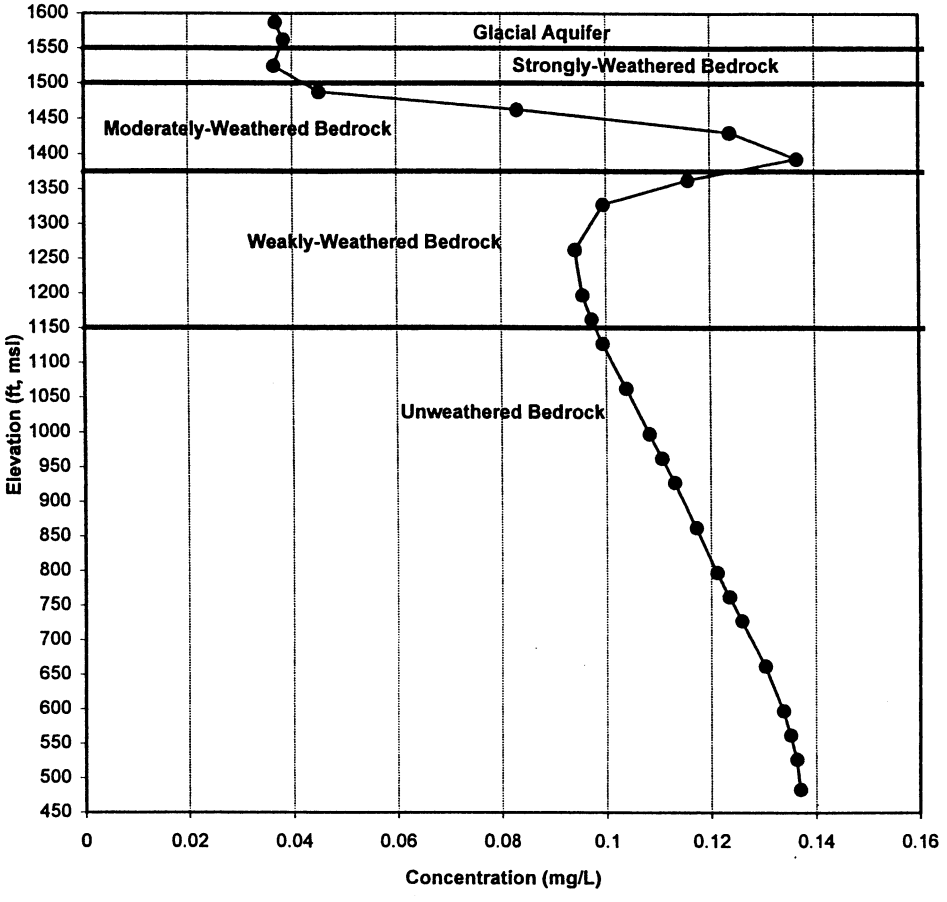
Figure 27
Expected-Case Concentration at the DMZ Boundary, Acidic Source in Cross Cut

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

p:\Crandon\Reflood3d\excel\rf151.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer at any time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 44- 7)	0.038
	Strongly-Weathered	(3- 44- 7)	0.037
	Moderately-Weathered	(7- 56- 7)	0.14
	Weakly-Weathered	(8- 56- 7)	0.12
	Unweathered	(26- 85- 34)	0.14

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

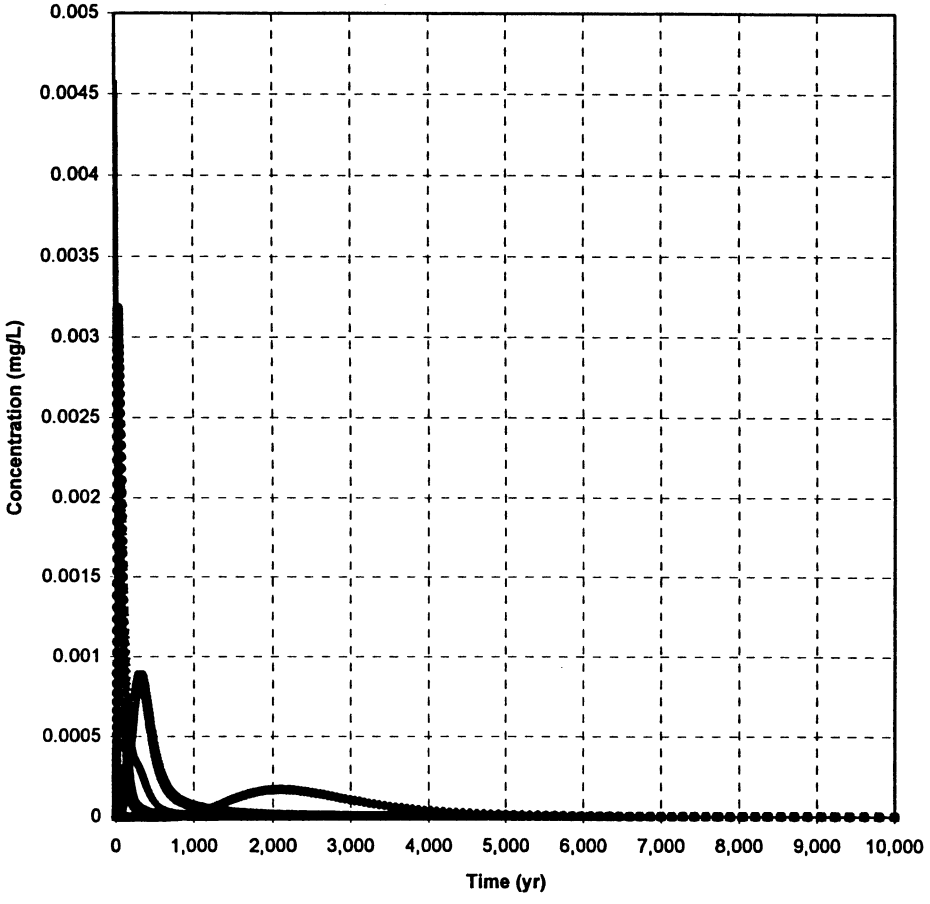
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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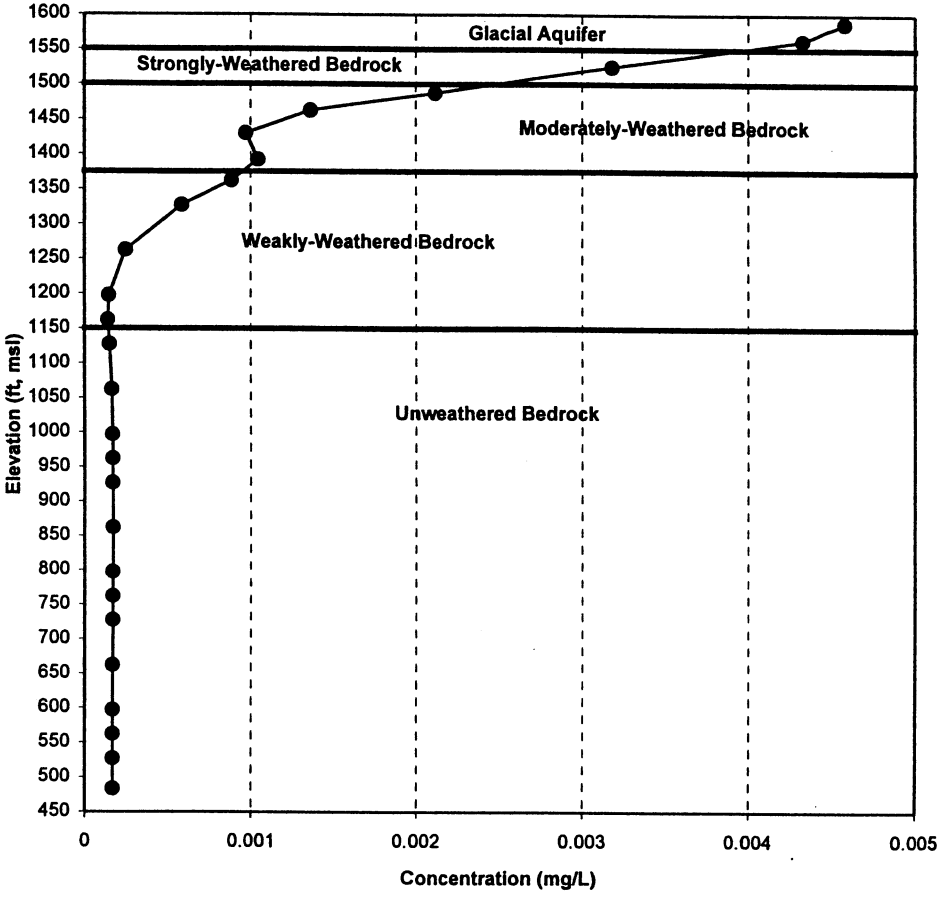
Figure 28
Expected-Case Concentration at the DMZ Boundary, Wall Rock Source

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

p:\Crandon\Reflood3d\excel\rf152Fig.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer evaluated independent of the simulation time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(1- 38- 7)	0.0046
	Strongly-Weathered	(3- 38- 7)	0.0032
	Moderately-Weathered	(4- 37- 7)	0.0021
	Weakly-Weathered	(8- 35- 8)	0.00088
	Unweathered	(18- 12- 83)	0.00017

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

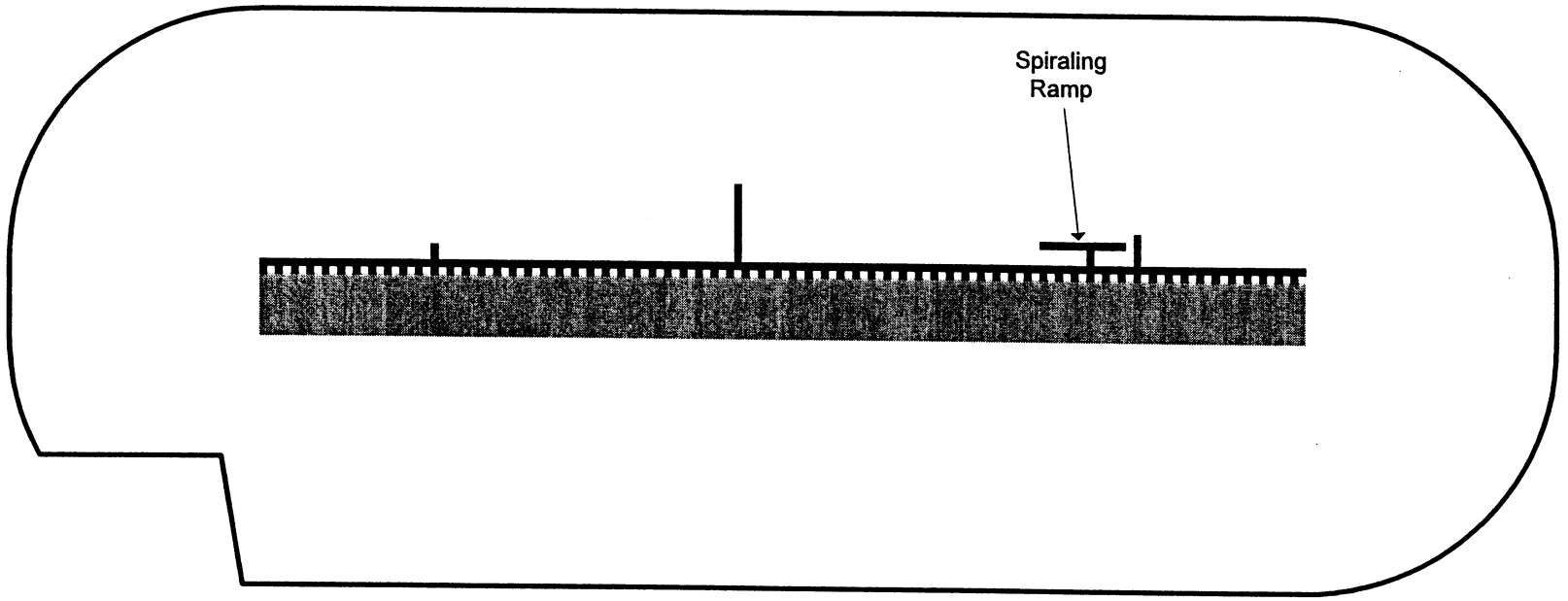
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 29
Expected-Case Concentration at the DMZ Boundary, Crown Pillar Source

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

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Legend

- Cemented Paste Backfill
- Weakly and Unweathered Bedrock
- Open Workings

0 900 1800

Scale in Feet

N

Note: This layout of open workings applies to all mining levels (layers 8,12,16,20, and 24)

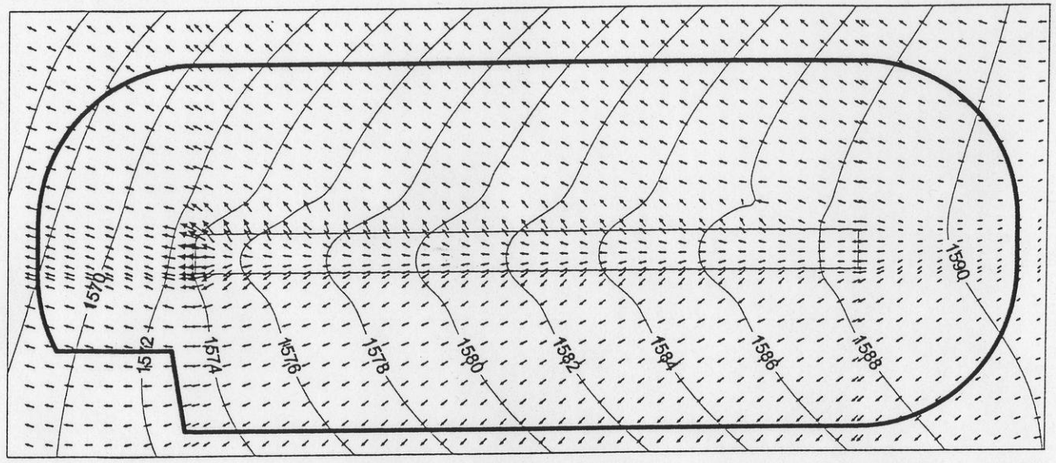
HSI GEOTRANS			
REVISED	DATE	BY	DESCRIPTION
CHECKED BY: GWC			DATE: Oct 2000
APPROVED BY: PFA			DATE: Oct 2000
APPROVED BY:			DATE:

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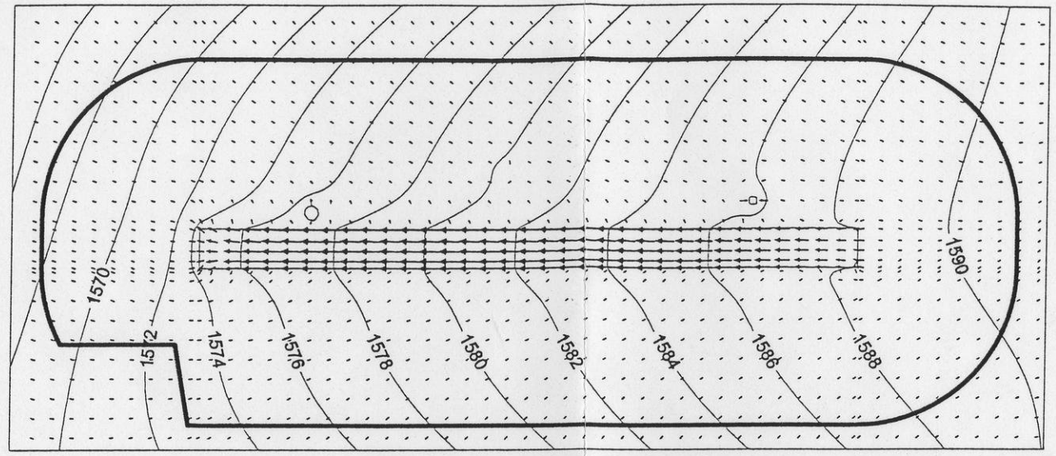
Figure 30
No-PHC-Case Open Workings Layout

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

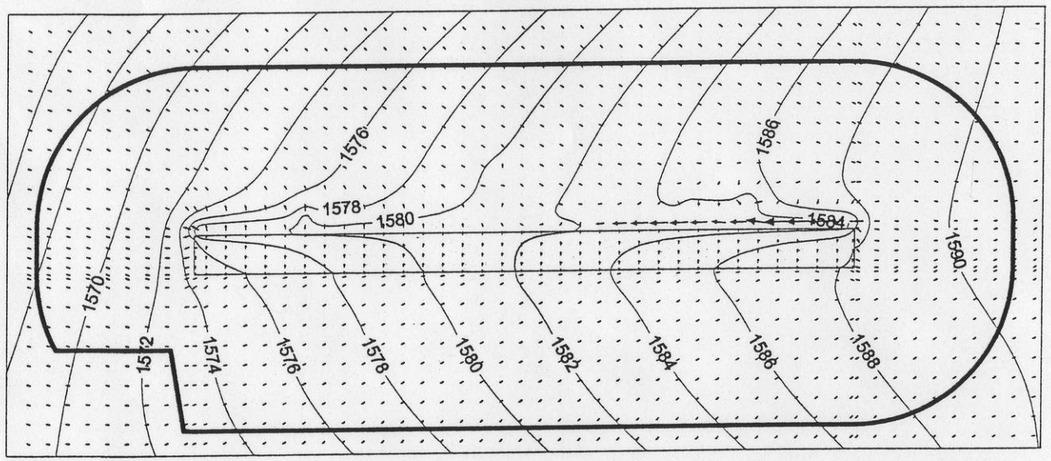
Layer 2 (Glacial Till)



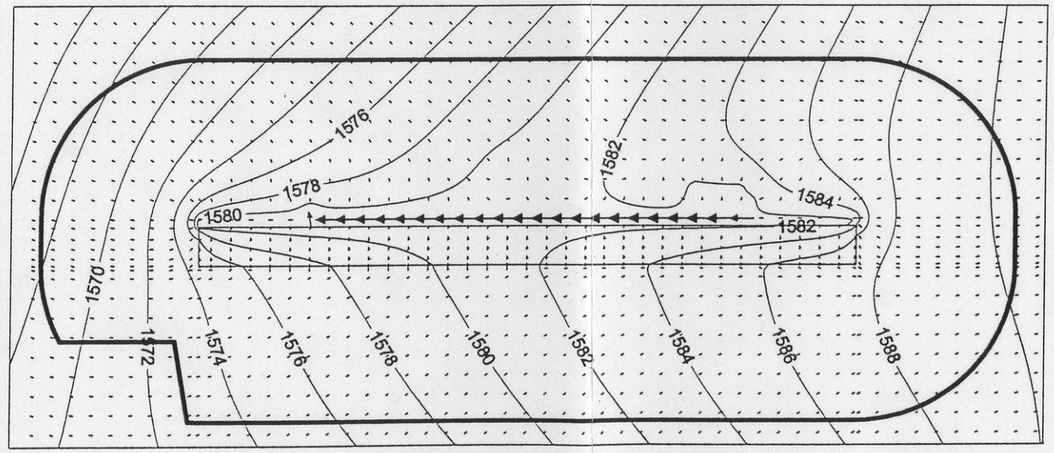
Layer 3 (Strongly-Weathered Bedrock and Crown Pillar)



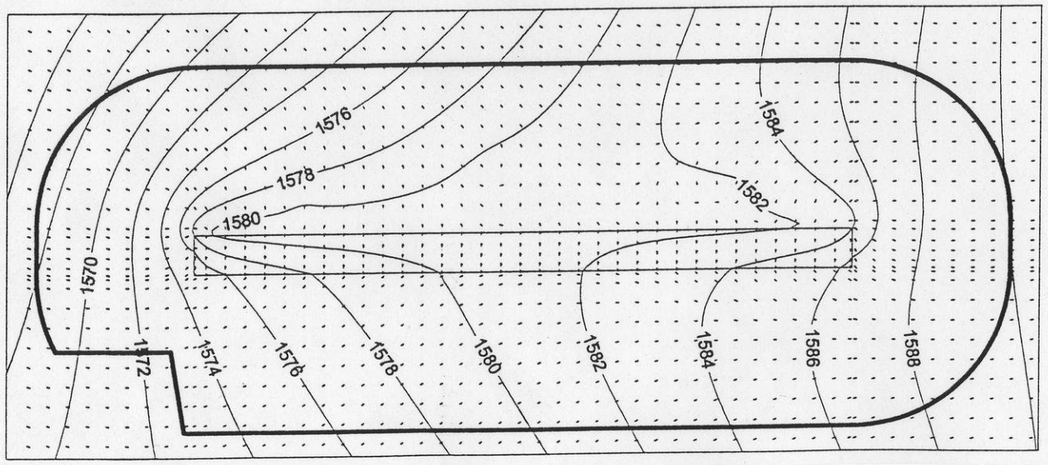
Layer 7 (Moderately-Weathered Bedrock and Top of Backfill)

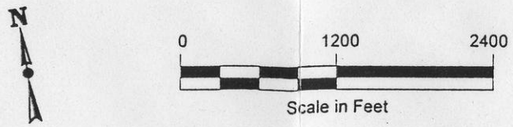


Layer 12 (Weakly-Weathered Bedock and Backfill at 500 ft Level)



Layer 26 (Unweathered Bedrock and Backfill Below 1100 ft Level)





Note: Contour Interval is 2 ft.
Vector length is proportional to groundwater velocity magnitude.

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

Simulation Rfl154	CHECKED BY: GWC	DATE: Oct 2000
	APPROVED BY: PFA	DATE: Oct 2000
	APPROVED BY:	DATE:

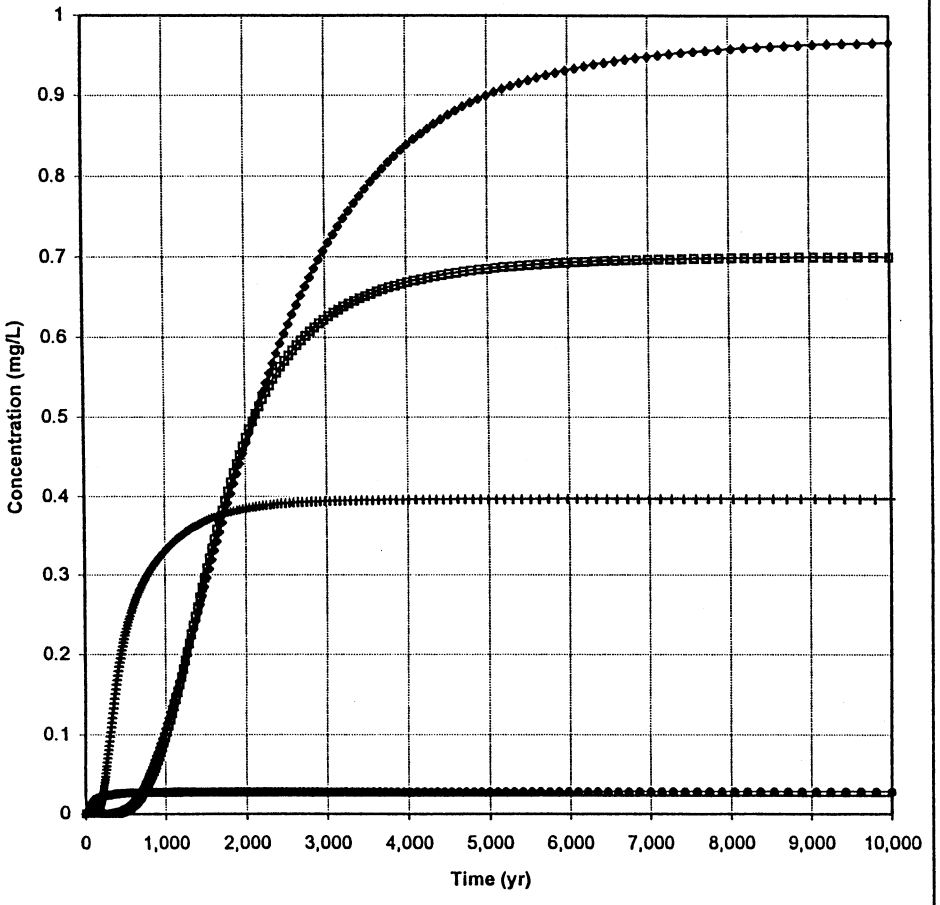
Nicolet Minerals
a Rio Algom company

Figure 31
No-PHC-Case Flow Field, Plan Views

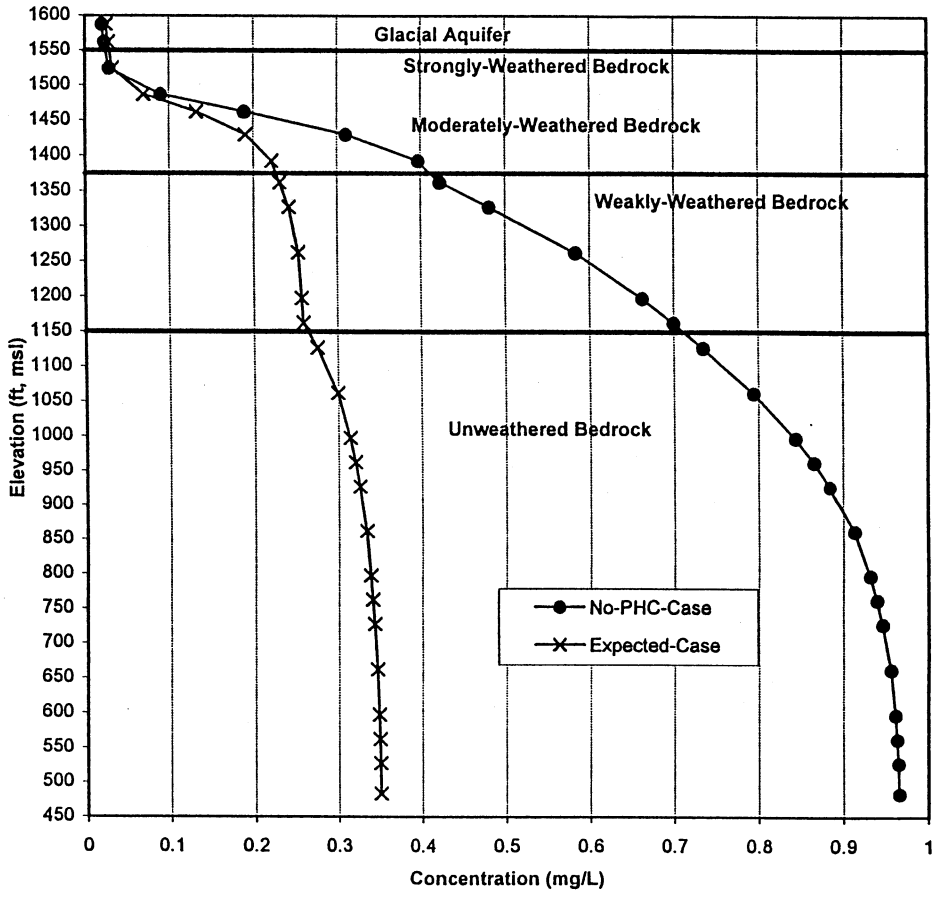
Scale: 1 in. = 1200 ft.	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

P:\Crandon\vellood3d\Surfer\Figures\rl154ft.srf

p:\Crandon\Reflood3d\rf154.xls, Figure 2



Concentration vs. Time: For the cells having the maximum DMZ-boundary concentration in each depth range (see legend below).



Concentration vs. Depth: Maximum DMZ-boundary concentration in each layer evaluated independent of the simulation time.

LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2- 35- 8)	0.023
	Strongly-Weathered	(3- 35- 8)	0.028
	Moderately-Weathered	(7- 44- 7)	0.40
	Weakly-Weathered	(12- 58- 7)	0.70
	Unweathered	(26- 85- 34)	0.97

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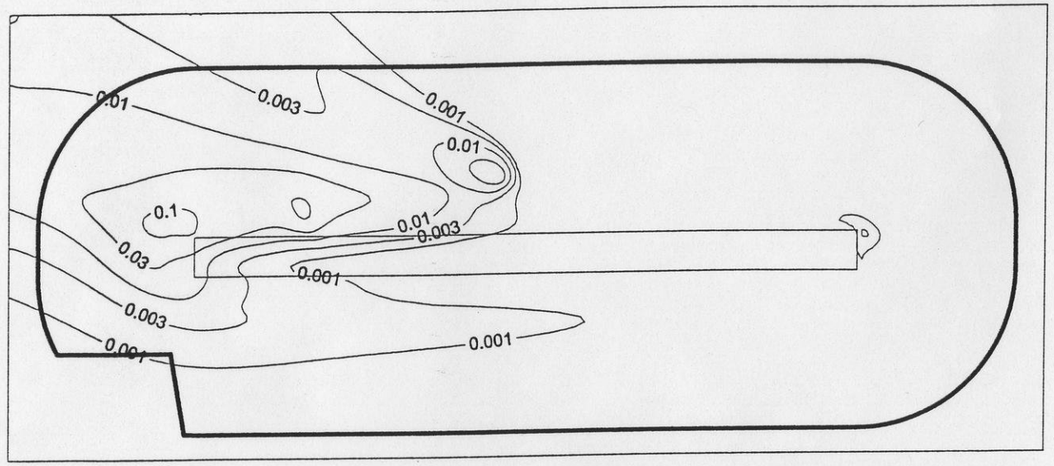
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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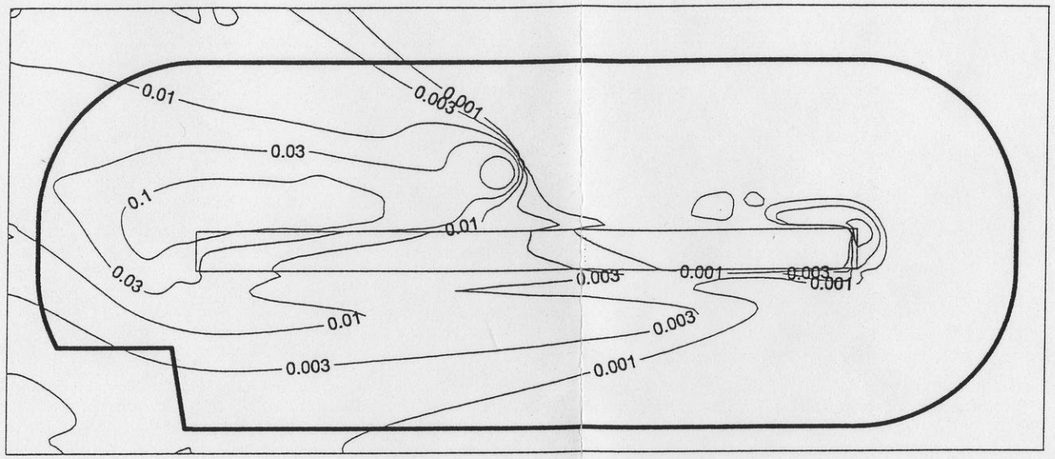
Figure 32
No-PHC-Case Concentration at the DMZ Boundary, Unoxidized Backfill Source

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

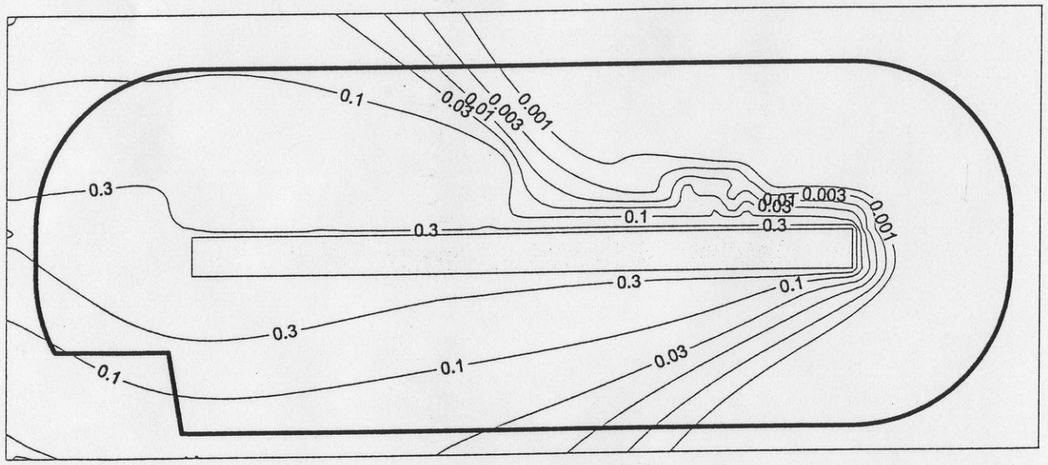
Layer 2 (Glacial Till)



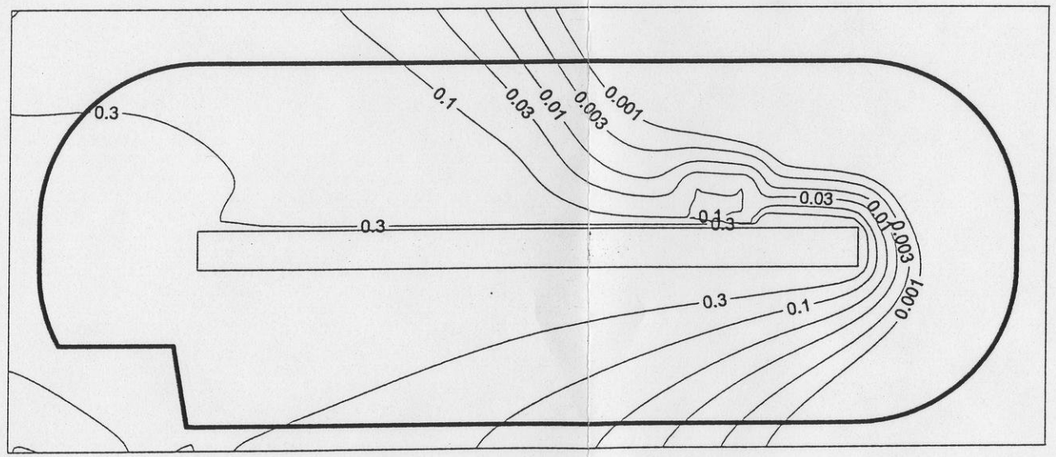
Layer 3 (Strongly-weathered Bedrock and Crown Pillar)



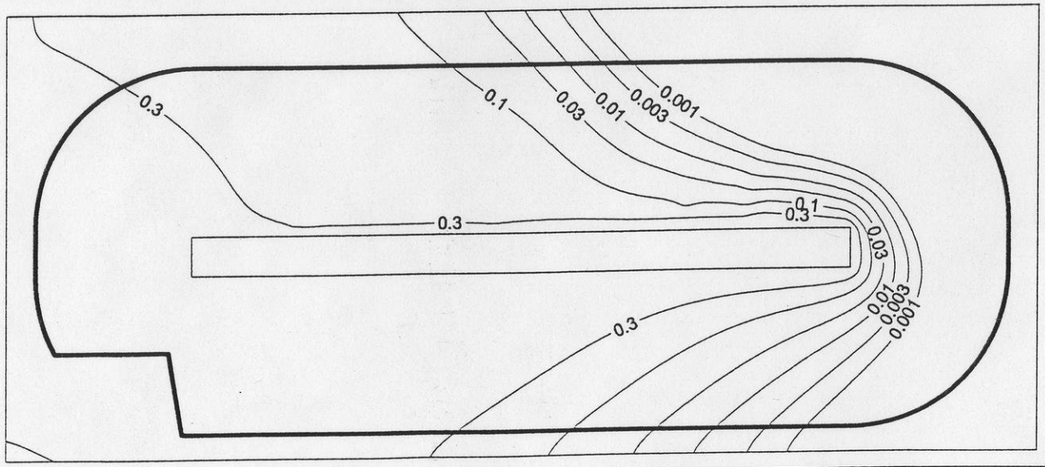
Layer 7 (Moderately-Weathered Bedrock and Top of Backfill)



Layer 12 (Weakly-Weathered Bedrock and backfill at 500 ft Level)



Layer 26 (Unweathered Bedrock and Backfill Below 1100 ft Level)



Note: Relative concentration, expressed as a fraction of the source concentration, is shown on these plots.
Note: The concentration after 10,000 years of simulation time is shown.

N

0 1200 2400

Scale in Feet

HSI GEOTRANS			
REVISED	DATE	BY	DESCRIPTION

Simulation Rfl54	CHECKED BY: GWC	DATE: Oct 2000
	APPROVED BY: PFA	DATE: Oct 2000
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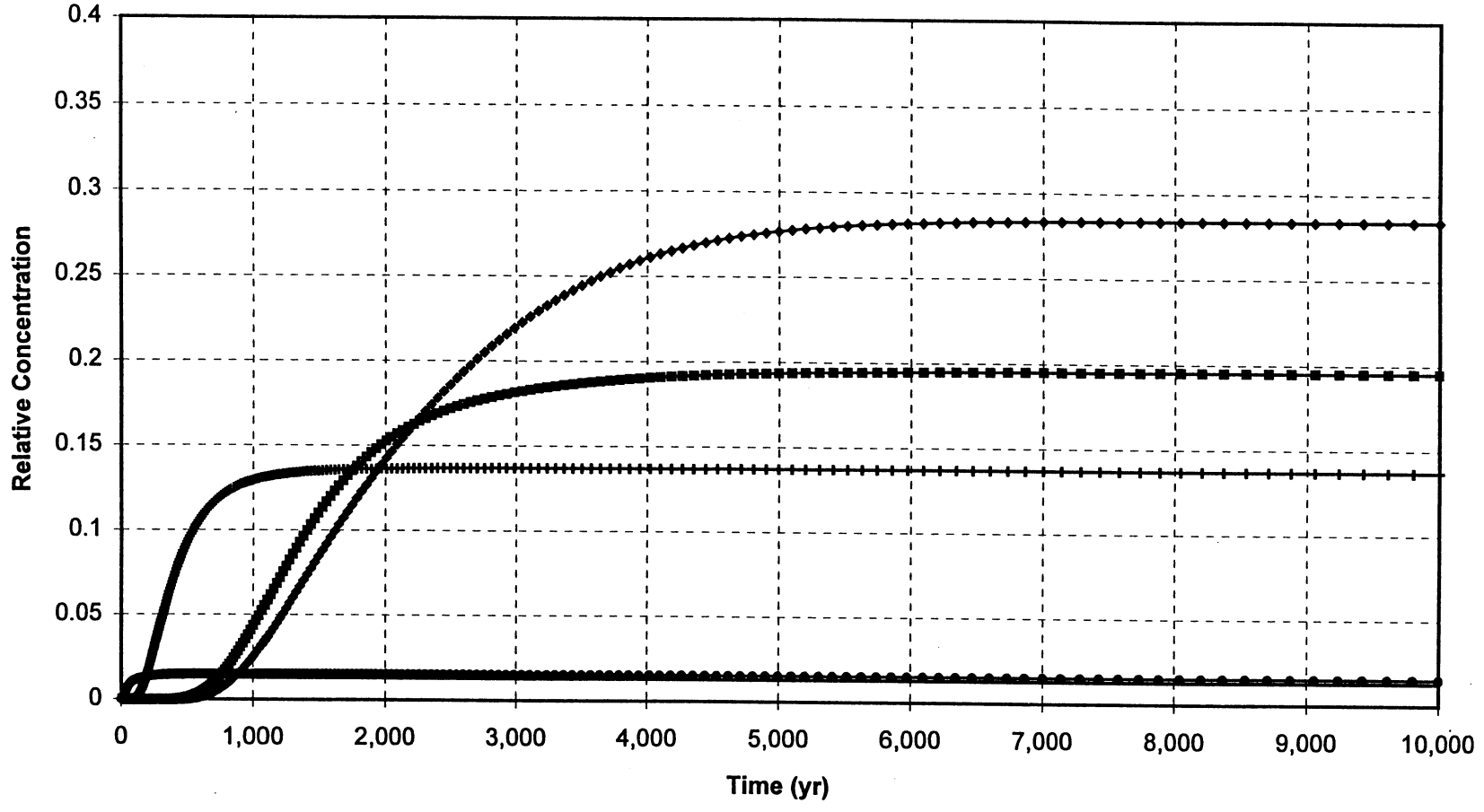
Nicolet Minerals
a Rio Algom company

Figure 33
No-PHC-Case Concentration Field, Unoxidized Backfill Source, Plan Views

Scale: 1 in. = 1200 ft.	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

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p:\Crandon\Reflood3d\Excel\rf122.xls, Figure



LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2-40-7)	0.013
	Strongly-Weathered	(3-40-7)	0.015
	Moderately-Weathered	(7-70-34)	0.14
	Weakly-Weathered	(12-85-52)	0.20
	Unweathered	(26-85-69)	0.28

Note: Relative Concentration is expreesed as a fraction of the source concentration.

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

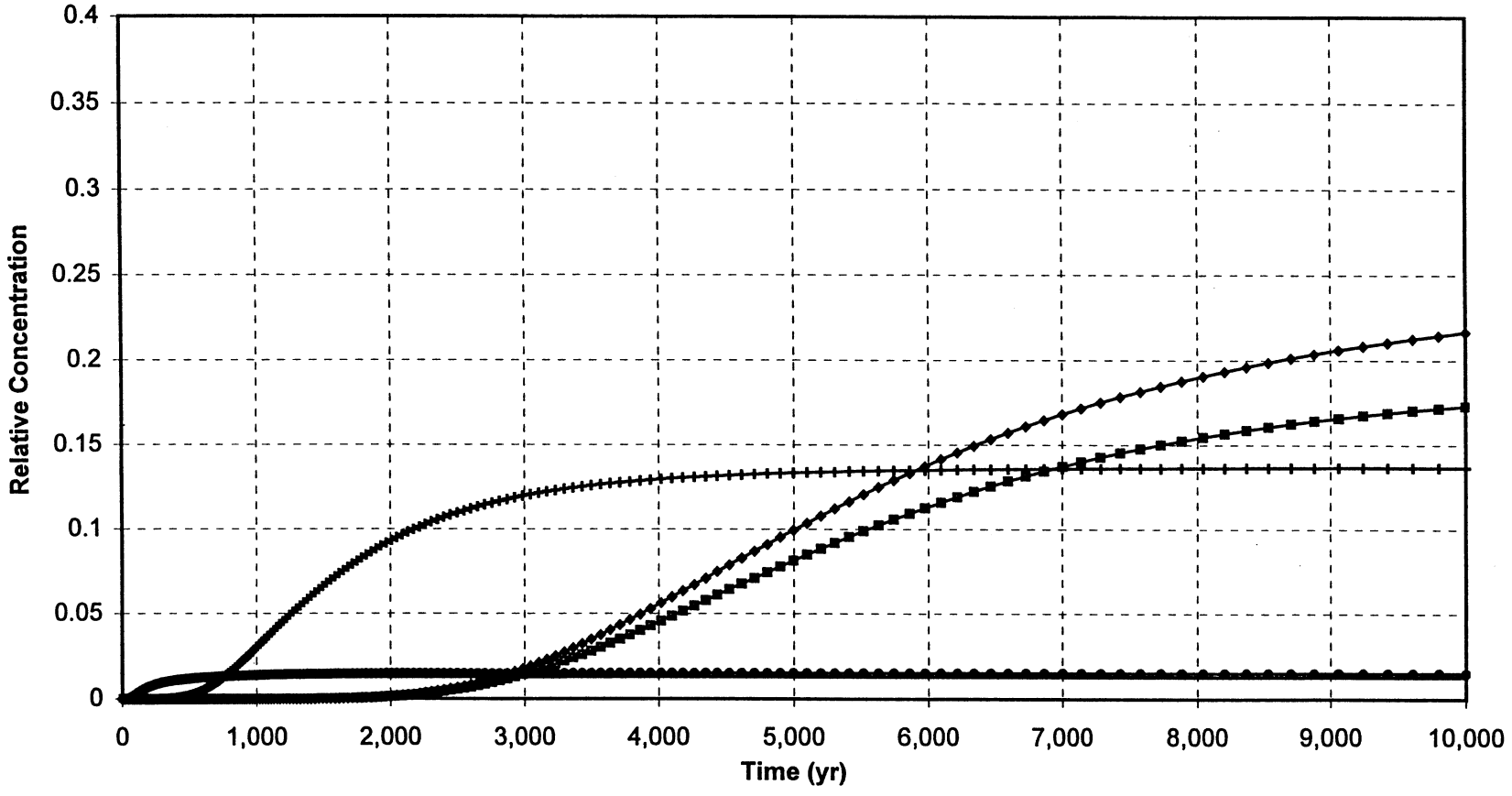
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 34
Concentration vs. Time at the DMZ Boundary, Low Porosity Simulation

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

p:\Crandon\Reflood3d\vf121.xls, Figure



LEGEND

Symbol	Depth Range	Model Cell	Max. Conc.
	Glacial Aquifer	(2-40-7)	0.013
	Strongly-Weathered	(3-40-7)	0.015
	Moderately-Weathered	(7-70-34)	0.14
	Weakly-Weathered	(12-85-50)	0.17
	Unweathered	(20-85-56)	0.22

Note: Relative Concentration is expreesed as a fraction of the source concentration.

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

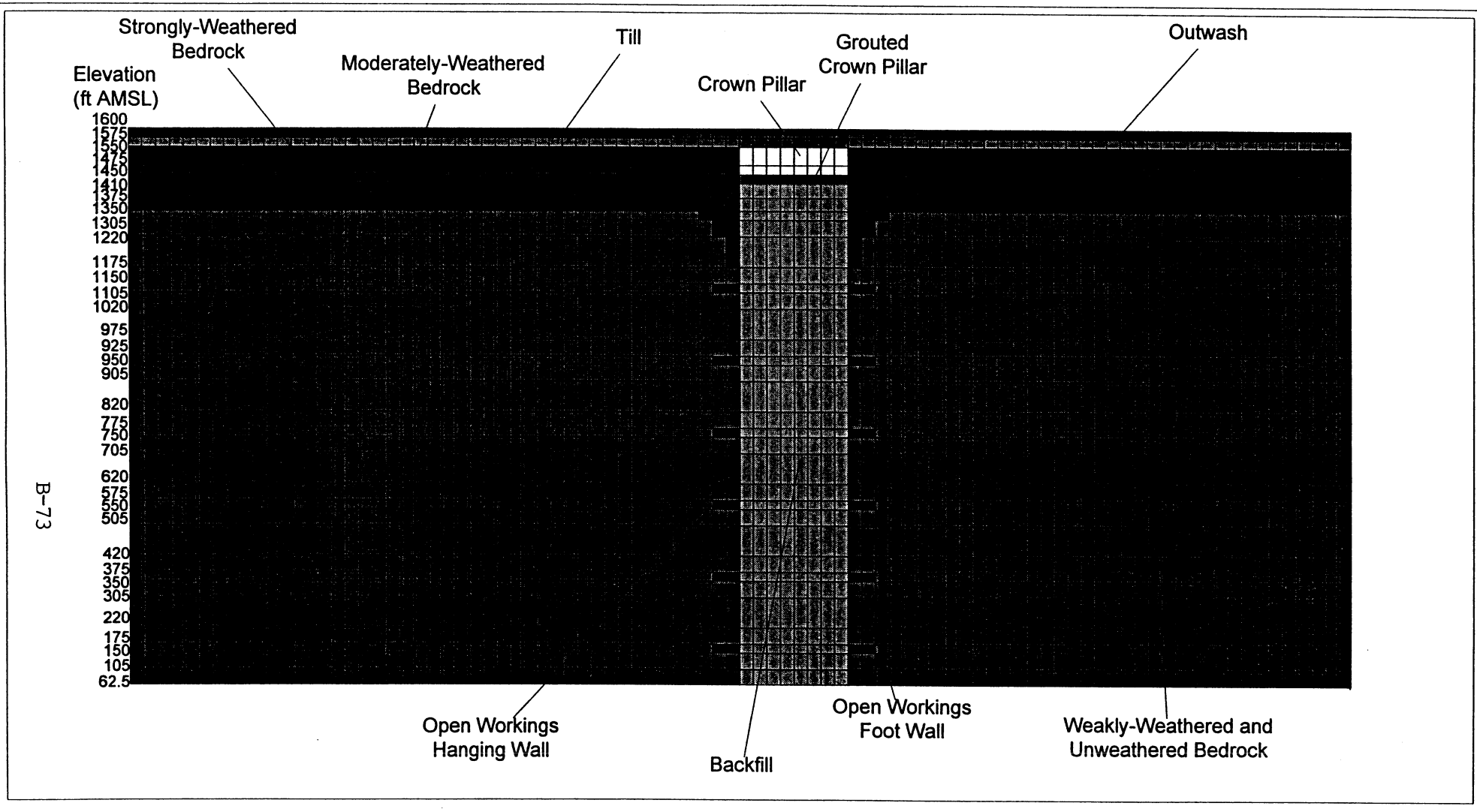
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 35
Concentration vs. Time at the DMZ Boundary, High Porosity Simulation

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

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B-73

0 400 800

Scale in Feet

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

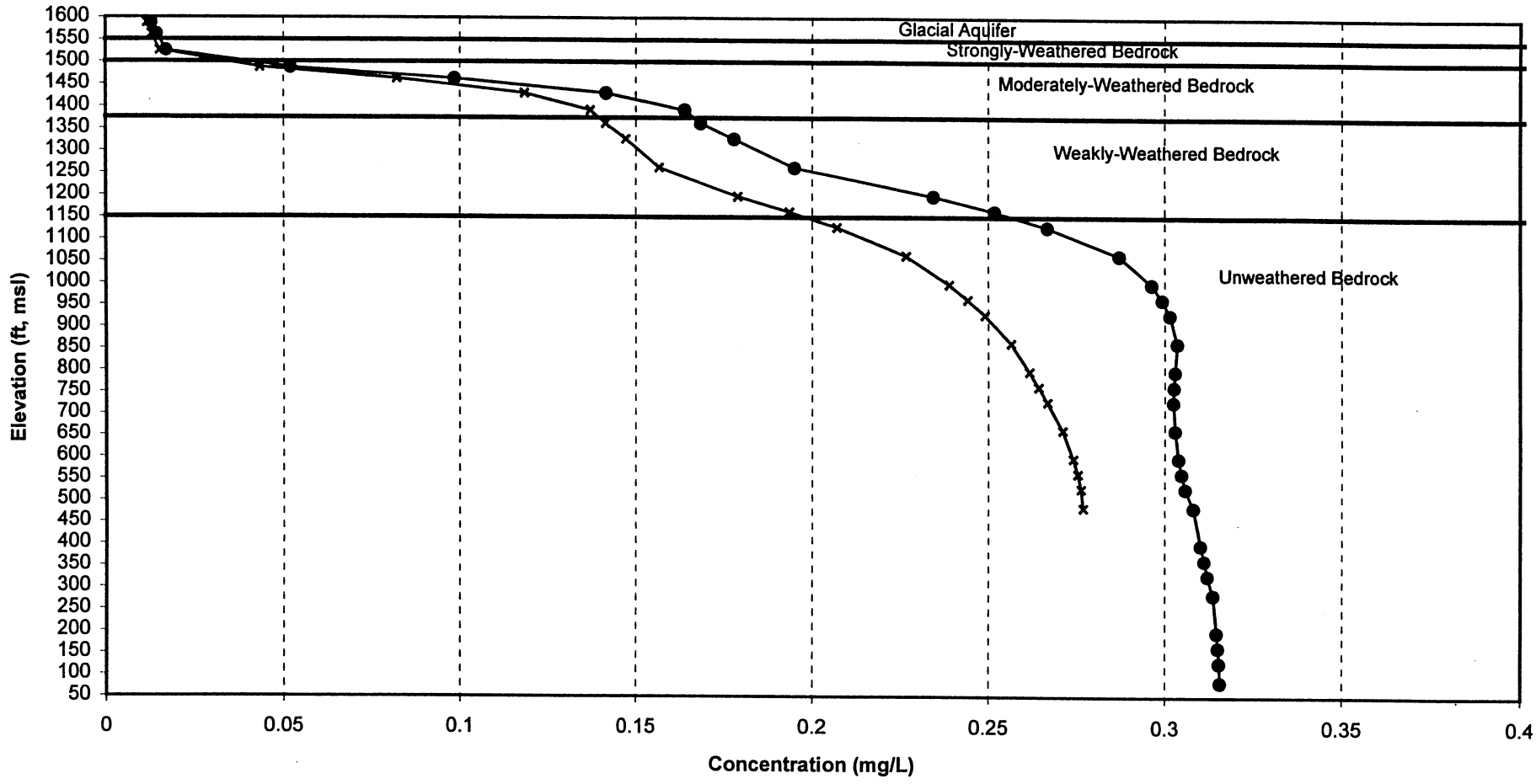
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 36
Model Cross Section for the Deeper Model

Scale: 1 in. = 400 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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Maximum DMZ Concentration

- Deeper Model
- Reference Simulation

Note: Relative Concentration is expressed as a fraction of the source concentration.

HSI GEOTRANS			
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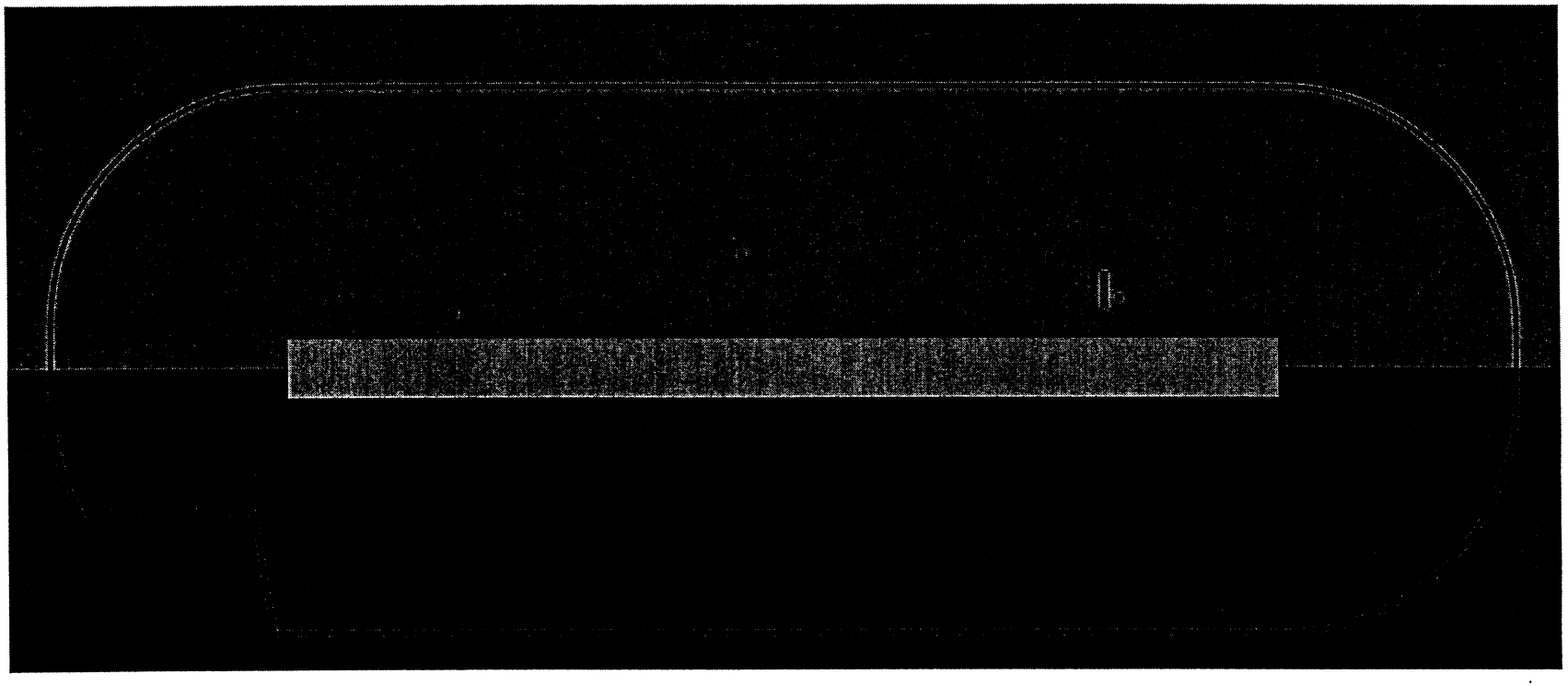
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 37
Concentration vs. Depth at the DMZ Boundary for the Deeper Model

Scale: None	Date: October 2000
Prepared By: HSI GEOTRANS	By: LMG

P:\Crandon\Reflood3D\GIS\workspace\hangfoot.wor



B-75

Legend

- Foot-Wall Zone
- Hanging-Wall Zone
- Backfill Zone
- Open Workings

0 900 1800

Scale in Feet

N

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

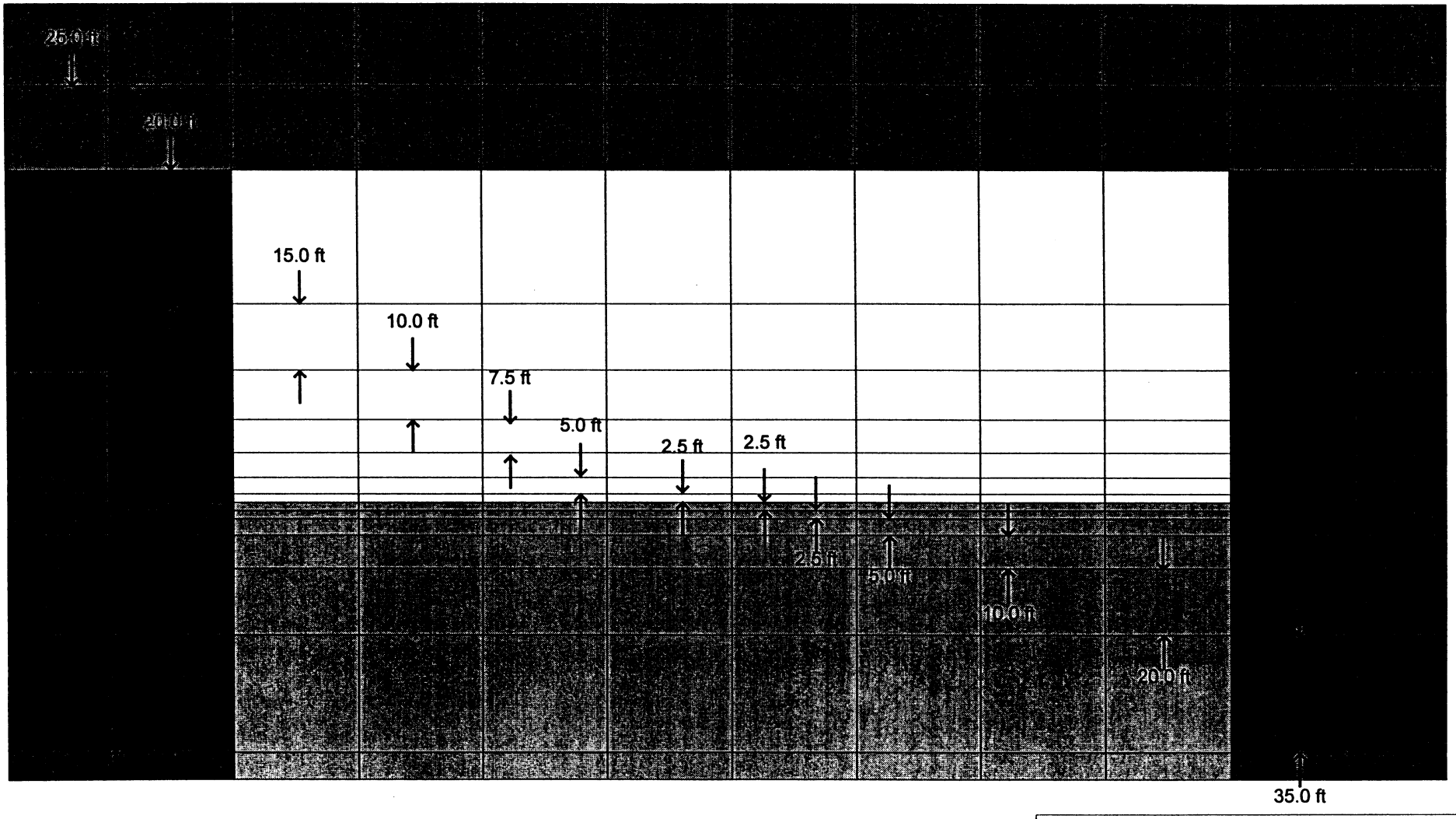
CHECKED BY: GWC	DATE: Oct 2000
APPROVED BY: PFA	DATE: Oct 2000
APPROVED BY:	DATE:

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Figure 38
Separated Hanging-Wall and Foot-Wall Zones

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

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B-76

0 45 90

Scale in Feet

HSI GEOTRANS			
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CHECKED BY: GWC			DATE: Oct 2000
APPROVED BY: PFA			DATE: Oct 2000
APPROVED BY:			DATE:

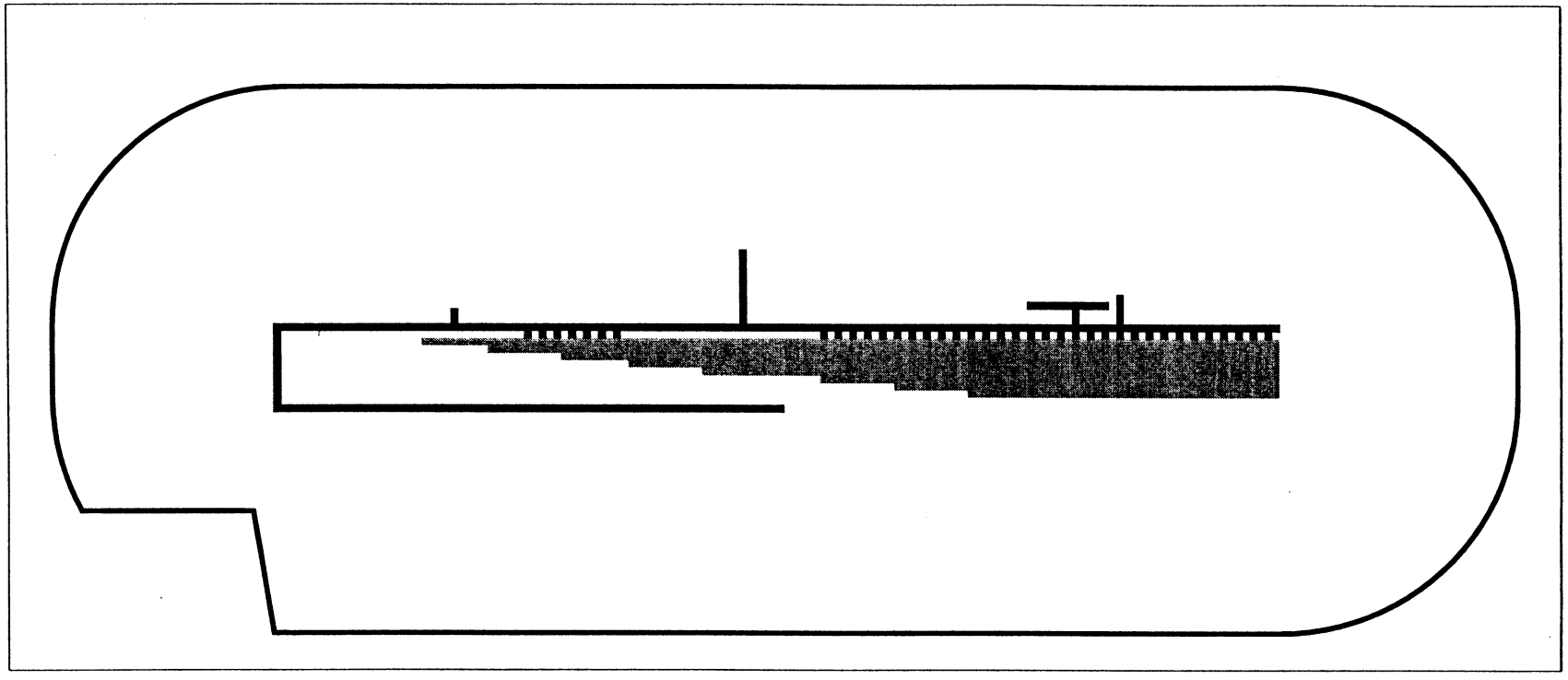
Nicolet Minerals
a Rio Algom company

Figure 39
Vertical Grid Refinement
at the Top of the Backfill

Scale: 1 in. = 45 ft. | Date: Oct 2000

Prepared By: HSI GeoTrans | By: LMG

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Legend

- Cemented Paste Backfill
- Weakly and Unweathered Bedrock
- Open Workings

0 900 1800

Scale in Feet

N

HSI GEOTRANS			
REVISED	DATE	BY	DESCRIPTION
CHECKED BY: GWC			DATE: Oct 2000
APPROVED BY: PFA			DATE: Oct 2000
APPROVED BY:			DATE:

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Figure 40
Backfill zone in Layer 12
Premature Closure Simulation

Scale: 1 in. = 900 ft.	Date: Oct 2000
Prepared By: HSI GeoTrans	By: LMG

Attachment A. Letter from Leslie Smith, PhD., regarding the "Mine Reflood Analysis"

Leslie Smith
Hydrogeological Analysis • Groundwater Contamination Studies

To: Mr. Steve Donohue
Foth & Van Dyke
Green Bay, Wisconsin

Cc: Mr. Gordon Reid, NMC

From: Leslie Smith, Ph.D.

Date: October 10, 2000

Re: Mine Reflood Analysis

At your request, in this memorandum I provide my opinion on three issues that arise in the mine reflood analysis when developing an estimate of future solute concentrations at the boundary of the design management zone (DMZ). These issues are:

(1) The appropriateness of using a continuum (porous medium) model to represent fluid flow and solute transport through the bedrock units at the Crandon site.
(2) The estimation of representative values for longitudinal and transverse dispersivities, for both the overburden and bedrock units at the Crandon site, and for the paste backfill.
(3) The context in which model predictions should be interpreted when the time scale for a transport analysis extends thousands of years into the future.

Porous Medium Model

Approaches to modeling solute transport in fractured rock systems fall into 4 broad classes: (1) deterministic continuum models, (2) stochastic continuum models, (3) discrete network models, and (4) hybrid methods. A description of these methods, and a discussion of their strengths and limitations, is provided by the U.S. National Research Council (1996). It is my opinion that a continuum approximation is the most sensible approach to adopt for the mine reflood analysis. This viewpoint is based on consideration of the geologic properties of the deeper bedrock units, the types of field data that are available, and the transport distances of interest.

The strength of a continuum model is that it reduces the complexity of the transport process into a mathematical form that is straightforward to implement. The parameters that are used in a continuum model apply at a scale where the domain of interest is sufficiently large that solute transport is not influenced to any significant degree by any individual fracture or its interconnections with other fractures within the rock mass. Individual fractures are not treated explicitly in a continuum model, except

4640 Mahood Drive, Richmond, British Columbia, Canada V7E 5C4
Office: (604) 822 4108 Fax: (604) 822 6088 Home: (604) 271 2799

where they occur on a scale large enough to be considered as a separate hydrogeologic unit. The U.S. National Research Council (1996) concluded that for most applications that are encountered in practice, some type of continuum approach remains the preferred alternative. It should be noted that I was a member of the expert committee that produced this report.

Two factors restrict the adoption of a discrete network modeling approach at the proposed Crandon site. First, discrete network models require a statistical database on fracture geometry and fracture transmissivity that does not exist, nor could it be readily obtained at the proposed Crandon site prior to construction of the underground workings. I see no way in which a discrete network model could be constructed that would provide a more reliable prediction of solute concentrations at the boundary of the DMZ than is currently provided by a conventional continuum model. Second, my experience with solute transport simulations using a state-of-the-art, three-dimensional discrete network model (FRACMAN/MAFIC, Golder Associates) indicates that the domain of interest at the Crandon site (~ 1200 – 5000 feet in the horizontal plane, ~ 1500 feet in the vertical plane) far exceeds current capabilities of readily-available computer facilities. Techniques for extending discrete network models to these larger domains are still under development within the research community.

If there were structural features of large lateral or vertical extent that significantly influence the patterns of groundwater flow within the bedrock system, such as a higher-permeability fracture zone, then it would be important to include these zones as discrete features in a continuum model. It is my understanding that no such fracture zones have been identified at the Crandon site.

An important issue when using a porous medium model to represent transport in a rock mass with fracture permeability is the specification of a value for effective porosity. The effective porosity is the volume of the connected void space (fractures, microcracks, grain-boundary porosity) through which solutes can move by advection. The groundwater velocity varies inversely with the effective porosity, as seen in the equation:

$$v_l = \frac{K}{\theta}\frac{dh}{dl}$$

where v_l is the average linear groundwater velocity, θ is effective porosity, K is the hydraulic conductivity, and dh/dl is the hydraulic gradient. With the hydraulic gradient determined from the regional scale model, and given the estimated values for the hydraulic conductivity of each unit, the groundwater velocity estimates will increase with lower values of the effective porosity.

No site-specific data are available at the Crandon site to develop estimates of the effective porosity of the bedrock units; therefore, reliance must be placed on representative values of effective porosity for crystalline rock that have been published in

the literature. In Table 1, I suggest a set of values for effective porosity that I consider reasonable for the rock types at the Crandon site.

Table 1
Suggested Values of Effective Porosity for Bedrock Units

Unit	Effective Porosity
Strongly weathered Crandon Formation	0.10
Strongly weathered bedrock	0.05
Moderately weathered bedrock	0.01
Weakly /Unweathered bedrock	0.003

The value for the weakly weathered and unweathered bedrock corresponds to the value for effective porosity of sparsely fractured rock below a depth of 300 m, adopted by Atomic Energy of Canada (Davison et al., 1994) in their EIS of a geologic repository in crystalline rock of the Canadian Shield. These estimates of effective porosity were based on measurements on rock cores. For moderately fractured rock within 150 m of the ground surface, they adopted an effective porosity of 0.005. In my view, it is not practical to carry out a tracer experiment at the Crandon site with the intent of obtaining a field-scale estimate of the effective porosity of the deeper bedrock. The logistics of such a study would likely make it infeasible. Rather, the impact of uncertainty in the effective porosity of the bedrock units should be assessed in a sensitivity study as part of the mine reflood analysis. For the unweathered and weakly weathered bedrock, I consider an effective porosity range from 0.001 to 0.01 to be reasonable for this sensitivity study. It may be possible during the mining operations to update the estimate of the effective porosity if, in response to dewatering operations, there is a recognizable perturbation in the subsurface distribution of the naturally-occurring solutes in the bedrock. Relevant measurements could be collected from the groundwater monitoring wells at the mine site.

Dispersivity

Concentrations of nonreactive solutes at the DMZ boundary which originate from within the paste backfill will be determined to a large degree by three processes: (1) the mass flux out of the paste backfill into the surrounding bedrock or mine workings, (2) mixing in the mine workings for a subset of the flow paths, and (3) dispersive mixing in the rock mass and sediments between the reflooded mine and the DMZ boundary. The dispersivity values assigned to the paste backfill and the bedrock / glacial deposits will influence the extent to which processes (1) and (3) act to reduce the maximum solute concentrations reaching the DMZ boundary.

Dispersivity is a scale-dependent parameter. A widely-quoted, but informal guideline that relates the magnitude of a dispersivity estimate to the scale of the transport prediction suggests a longitudinal dispersivity value (α_l) that is 10 – 20 times smaller than the transport distance of interest, a transverse lateral dispersivity (α_{th}) that is one

order of magnitude smaller than α_l, and a transverse vertical dispersivity (α_{tv}) that is one to two orders of magnitude smaller than α_{th}. The data base from sites with differing geology and flow regimes indicates that, for a given transport distance, a substantial range of dispersivity values is possible about this guideline (Gelhar et al., 1992). Therefore, the guideline should not be interpreted too literally. Gelhar et al. (1992) also concluded that the data base they examined did not support a different range of values for sites underlain by granular porous media or a rock mass with fracture-dominated permeability. It is important to note that the data base of reliable estimates of transverse dispersivity is not as extensive as that for longitudinal dispersivity.

Table 2 lists the set of dispersivity values that I recommend be adopted in the mine reflood analysis. Following the discussion of these values, I suggest a range for the dispersivity values that should be considered in a sensitivity study to examine how the solute concentrations at the boundary of the DMZ may vary with the assumed values for dispersivity.

Table 2 Recommended Dispersivity Values

Unit	Longitudinal Dispersivity (ft)	Transverse Lateral Dispersivity (ft)	Transverse Vertical Dispersivity (ft)
Glacial deposits – outwash and till/saprolite	50	5	0.5
Strongly weathered Crandon formation	50	5	0.5
Bedrock units (all weathering types)	50	5	5
Paste backfill	50	5	0.5

The values of dispersivity I recommend be adopted for the glacial deposits (50 ft, 5 ft and 0.5 feet for α_l, α_{th}, α_{tv}, respectively) are consistent with the values DNR has proposed in their review of transport modeling at the TMA (Carlson, July 13, 2000). To model solute migration in the bedrock units, I suggest NMC proceed under the assumption that the longitudinal dispersivity applied to the glacial deposits also characterizes longitudinal spreading in the bedrock (ie. an α_l of 50 ft.). I see no basis that would support an attempt to assign different dispersivity values to the various weathering units in the bedrock, although it is certainly possible to speculate how the different degrees of weathering may influence dispersive mixing.

It is my opinion that in fractured rock systems, it is more likely that the transverse vertical dispersivity will be of similar magnitude to the transverse lateral dispersivity. To my knowledge, there are no reliable data in the literature to distinguish the relative magnitudes of α_{th} and α_{tv} for a fracture-dominated system. The relative degree of spreading will depend, in part, on the orientation of the fracture sets, fracture densities,

and fracture connectivity. At Crandon, the bedrock contains both a near-vertical fracture set and less-frequent, near-horizontal fractures. Thus, I expect local perturbations in the mean flow regime both in the transverse lateral and vertical directions. Simulations of plume migration in discrete (orthogonal) fracture networks that we have carried out in research programs here at the University of British Columbia are strongly suggestive of significant transverse dispersion in both planes orthogonal to the mean hydraulic gradient (Parney, 1999; Burns, 2000). On this basis, I support the concept that α_{tv} be assigned the same value as α_{th} (ie. 5 ft.) for the bedrock units. It will be necessary in the sensitivity study to assess the impact on predicted solute concentrations of assigning a smaller value to α_{tv} (ie. 0.5 ft.)

Identifying representative dispersivities for the strongly weathered Crandon Formation in the region of the crown pillar is perhaps the most challenging issue with respect to this parameter set. The character of this unit likely changes with depth; behaving more like a granular porous medium at shallower depths where weathering is more intense, with a stronger imprint of fracture pathways at greater depths. I favor the approximation where it is assigned the same dispersivity values as the overlying till and massive saprolite. The main consequence of this assumption is to assign a smaller value to the transverse vertical dispersivity (0.5 ft.) than is applied in the deeper bedrock units. This parameter selection will influence both the mass flux from the paste backfill into the crown pillar, and the extent of vertical spreading through the crown pillar.

To my knowledge, there are no data in the literature on the field-scale dispersive properties of paste backfill. Given the granular nature of the tails, and the method of emplacement in the stopes, I would expect the paste backfill to have properties similar to a sedimentary deposit, with a small degree of vertical transverse spreading (α_{tv}) across any stratification that may develop in the paste backfill. Thus, I recommend that the same dispersivity values assigned to the glacial deposits be adopted for the paste backfill.

As noted above, it will be important to carry out a sensitivity analysis on the dispersivity values to determine to what extent the predicted concentrations at the boundary of the DMZ are influenced by the values that have been selected. In my opinion, a reasonable range within which to vary the longitudinal dispersivity value for all units is from 20 feet to 200 feet, given the transport distances involved and the nature of the geologic materials. Values at the high end of this range are less likely to occur (ie. the range should not be viewed as a uniform distribution). Corresponding values for the transverse lateral dispersivity are 2 feet to 20 feet, and for the transverse vertical dispersivity, 0.2 feet to 2 feet. In the bedrock units, I suggest a range for the transverse vertical dispersivity of 0.5 to 20 feet be considered.

Time Scale For Transport

Uncertainties in model predictions inevitably increase as calculations are extended hundreds to thousands of years into the future. On a time scale of thousands of years, it is not possible to state with great assurance how well current conditions will

represent the geologic and hydrologic features of a watershed at some future date, especially with respect to the near-surface conditions. Nor is there an accepted formalism for quantifying this type of prediction uncertainty as it requires assigning probabilities to future events, some of which will not be reasonably constrained by the historical record, and a few of which may not even be foreseeable.

The most in-depth analysis of the utility of model predictions involving solute migration in deep bedrock systems is likely found in the literature on geologic disposal of high-level nuclear waste. I am most familiar with this issue in the Canadian context, but a similar perspective is evident in U.S. regulations. The Atomic Energy Control Board of Canada (Regulatory Document R-104, 1987) states "The time period for demonstrating compliance with the individual risk requirements using predictive mathematical models need not exceed 10,000 years. Where predicted risks do not peak before 10,000 years, there must be reasoned arguments that beyond 10,000 years the rate of radionuclide release to the environment will not suddenly and dramatically increase". They acknowledge that selecting a time period of 10,000 years is a matter of judgement since there does not appear to be "any objective way of limiting the assessments in a scientifically satisfying manner". For a high-level nuclear waste repository, they conclude that 10,000 years is a reasonable maximum period for assessments of individual risk.

Atomic Energy of Canada, in their EIS submission for geologic disposal of nuclear waste in crystalline rock of the Canadian Shield, state the opinion that groundwater flow in the bedrock system can be expected to be relatively constant until the onset of the next glaciation (Goodwin et al., 1994). This includes the fluid pressures, the permeability of the rock mass and major fracture zones within it, the chemistry of the groundwater, and the chemistry of the rock mass. They recognize that when there is another glaciation, substantial changes in the flow patterns within the bedrock, and the permeability of fracture zones, can be anticipated.

In my view, when using long-term model predictions to guide decisions on the likelihood of future impacts, the primary information that can be drawn from the model calculations will be two-fold:

(1) Insight can be gained into the processes controlling system behavior and the magnitude of future impacts, for the current conceptualization of the hydrogeology. It is possible to infer whether or not these impacts will meet legislated standards should the present-day condition persist.
(2) It is likely that long-term concentration trends will be apparent, providing the basis for inferring whether or not sudden changes in concentration can be anticipated as steady state is approached.

The reality is that conclusions on long-term impacts must be based on sound judgement, guided by the model sensitivity analyses, rather than by strict adherence to numeric concentration values provided by a simulation model.

The transport time scale in the mine reflood analysis is itself an uncertain parameter. In the deeper bedrock, the time at which steady state concentrations are approached at the boundary of the DMZ is determined primarily by the assigned values of hydraulic conductivity and effective porosity. The predicted time to reach steady state in the deeper bedrock is likely to vary in an approximate linear fashion with the assumed value for effective porosity. As suggested earlier, the uncertainty in the effective porosity of the deeper bedrock may reasonably be expected to span a range between 0.001 and 0.01, with a corresponding uncertainty in the time for a steady state concentration profile to develop.

References

Atomic Energy Control Board of Canada, Regulatory Document R-104, Ottawa, 1987.

Atomic Energy of Canada Limited, The Disposal of Canada's Nuclear Fuel Waste, Postclosure Assessment of a Reference System

Burns, S., Contaminant Transport Through Abandoned Boreholes in Fractured Rock, M.A.Sc. Thesis, University of British Columbia, 2000.

Carlson, C., Review of NMC Tailings Management Area Solute Transport Model, July 13, 2000.

Davison, C. C. et al., The disposal of Canada's nuclear fuel waste: The geosphere model for postclosure assessment, Atomic Energy of Canada Ltd., AECL- 10719, 1994.

Gelhar, L, et al., A critical review of data on field-scale dispersion in aquifers, Water Resources Research, 28(7), 1992.

Goodwin, B.W., et al., The disposal of Canada's nuclear fuel waste: Postclosure assessment of a reference system, Atomic Energy of Canada Limited, AECL-10717, 1994.

National Research Council, Rock Fractures and Fluid Flow – Contemporary Understanding and Applications, National Academy Press, 1996.

Parney, R., Statistical Continuum Modeling of Mass Transport Through Fractured Media in Two and Three Dimensions, Ph.D. Thesis, University of British Columbia, 1999.

Attachment B. Technical Memorandum from TRC regarding "Long Term Durability of Proposed Grouting in the Crown Pillar"

TRC

Customer-Focused Solutions

TECHNICAL MEMORANDUM

TO: Stephen Donohue – Foth & Van Dyke

CC: Gordon Reid – Nicolet Minerals Company

FROM: Vladimir Straskraba – TRC V.S.
Jeff Wright – TRC
Scott Effner – Whetstone Associates

DATE: November 13, 2000

SUBJECT: Long Term Durability of Proposed Grouting in the Crown Pillar,
(revised 11/13/00)

Introduction

At the request of Foth & Van Dyke, this memo has been prepared to address the long term durability and potential changes in permeability that can be expected in the grout blanket at the Crandon Project over a period of approximately 10,000 years. Of greatest interest are the potential changes in permeability and porosity that could occur as the grouted horizon ages.

The following sections present brief descriptions of observed case histories of grouting, the mining method and grouting plan for the Crandon Project, and an analysis of the physical and chemical stability of the grouts that will be used to construct the grout blanket.

Grouting Experience at Other Mining and Engineering Projects

Bridges and aqueducts constructed in ancient Roman times (> 2,000 years ago) are some of the best examples that illustrate the durability of cement grouts. Cementitious materials composed of natural pozzolans (calcinated shale, pumicite, diatomite, and silica fume) were used to construct many features of Roman architecture that still exist today. Many of these structures are currently in very good condition in southern Europe, Italy and Spain and attest to the durability of the grout.

Portland cements have only been used for about 150 years. Numerous studies and experiments, however, attest to the longevity of cement based grouts. Factors which influence the durability of cement grout include the environment in which it is placed and the composition and relative proportions of the constituents which form the grout (Fernandez and Richardson, 1994).

The durability of cement grouts has been demonstrated on many mining projects. Vertical shafts penetrating water bearing strata in Europe were typically sealed by the application of cement based grouts. Many of these shafts were constructed over 50 years ago, and the grout seals have proven to be durable over the period of record.

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The potential for changes in the porosity and permeability of the proposed grouting materials over a period of approximately 10,000 years at the Crandon Project are discussed in the following sections.

Crandon Project Mining and Grouting Plan

Blasthole open stoping and cut-and-fill mining techniques will be used to produce ore at the Crandon Project. Blast hole open stoping will be used to mine the majority of the deposit. Cut-and-fill mining will be used below the crown pillar to minimize the disturbance of the overlying bedrock. After mining, stopes will be backfilled using cemented paste backfill. Stopes at the base of the crown pillar will be "tight" filled to minimize the potential for surface subsidence (Agapito, 1996). Any remaining pore or void space will be allowed to reflood at the end of mining.

A grout blanket, approximately 25 feet thick, will be constructed in the crown pillar between the 225 and 250 feet mine elevations. The grout blanket will be keyed into low permeability rocks in the hanging wall and footwall. Grouting will be performed using Portland cement based grouts. Microfine cement grouts will also be used to supplement traditional cement grouts. Cement and microfine grouts will be amended using stability and durability enhancing additives including silica fume, fly ash, and bentonite clay (TRC, 1998a).

Chemical grouts may be used on a limited basis to control some inflows. These materials, however, would be applied in special circumstances only, and would comprise a very small percentage of the grout blanket. Because of their potentially limited application, the durability of chemical grouts are not considered in this analysis.

It is anticipated that grouting will decrease the permeability of the crown pillar by about 3 orders of magnitude. The ungrouted hydraulic conductivity of the crown pillar is estimated to be about 4 to 5 ft/day (1 to 2 x 10^{-3} cm/sec). The hydraulic conductivity of the grout blanket at the completion of construction will be in the low 10^{-3} ft/day range (low 10^{-6} cm/sec range).

Potential Mechanisms for Deterioration of the Grout Blanket

Three mechanisms are identified which could cause the deterioration of the grout blanket over a period of approximately 10,000 years:

1. Physical erosion and removal of grout by flowing ground water
2. Physical disruption caused by movement or subsidence
3. Chemical reactions with ground water or bedrock

Potential for Erosion and Removal of Grout by Flowing Ground Water

Physical erosion of the grout blanket could occur provided that a strong hydraulic gradient exists across the grouted horizon. Based on the current mining plan, a strong gradient across the grouted horizon will only occur during active mining operations. During this time, the mine will be dewatered and a maximum vertical gradient of about 6 ft/ft will exist from the glacial overburden into the mine. The grout blanket will be maintained during active mining operations

and deterioration of the grout will not be allowed to occur. At the end of mining, the backfilled mine will be reflooded and the ground water system will equilibrate back to a state similar to the pre-mining condition. Under these conditions, a horizontal gradient of about 0.003 ft/ft will be established to the west, and vertical gradients of about 0.01 ft/ft will exist (HSI Geotrans 1996, 1998a, 1998b, 1998c). Vertical gradients will be upward at the western end of the deposit and downward at the eastern end (HSI Geotrans, 2000).

Under the reflooded condition, physical erosion of the grout blanket is not expected to occur because of the lack of driving head. If a porosity value of 1% is assumed for the grouted horizon, an average linear ground water velocity of 0.3 ft/year to the west can be calculated using the equation:

$$V_x = \frac{KI}{n_e}$$

Where:

V_x is the average linear velocity;

K is grouted hydraulic conductivity ($\approx$ 0.003 ft/day);

I is the gradient; and,

n_e is effective porosity.

The average linear velocity of ground water in a vertical direction is calculated to be 1.1 ft/yr. Given the very low velocity of ground water under the flooded mine conditions, physical erosion or scouring of grout is not expected to contribute to the deterioration of the grout blanket.

Potential for Disturbance of Grout by Movement or Subsidence

The use of cut-and-fill mining techniques below the crown pillar will minimize the potential for instability of the rock mass that could disturb the grout blanket and impact permeability. Grouting will also add strength to the crown pillar and further decrease the potential for instability. The majority of the deposit will be mined using blasthole open stoping with delayed backfill. At any given time, less than 5% of the total hanging wall of the ore body will be exposed and unsupported. This mining method will distribute open voids into different areas of the deposit and minimize the potential for impacts to the crown pillar (Agapito, 1996).

The stopes below the crown pillar will be "tight" filled to minimize long-term subsidence impacts to permeability and hydrologic stability (Agapito, 1996). Tight filling of stopes will leave unfilled voids below the crown pillar of about 2 feet in height. This will produce an overburden thickness to void height ratio greater than 40:1. Values greater than 40:1 provide a high level of confidence that long-term subsidence will not extend upward into the glacial aquifer (Agapito, 1996).

Agapito Associates (1996) also evaluated the factors of safety for shear movement (joint shear strength / by joint shear stress) along the five dominant fracture orientations in the upper bedrock. Their analysis indicated that four of the five most abundant fracture orientations would not undergo shear movement, and that the stress caused by mining would result in either fracture

closure or very little change. Closure of the fractures would result in reduced fracture flow rates or no change along the affected orientations. The factor of safety analysis for the least frequent of the five fracture sets indicated that shear movement may occur which could increase permeability along this orientation (64° NW dipping 25° NE).

Agapito's evaluation of fracture deformation in the crown pillar indicated that east-west trending fractures with high angle dips (the most abundant orientations), will close due to mining induced stress changes resulting in reduced hydraulic conductivity in east-west and vertical directions. Based on pumping test data (Foth & Van Dyke 1995; TRC 1998b) this fracture orientation has the highest natural hydraulic conductivity. The least dominant fracture set (64° NW dipping 25° NE) was projected to undergo shear failure and dilation, with a potential increase in hydraulic conductivity. However, when the pressure gradient on the individual fracture sets and their relative numerical occurrence are considered, Agapito's analysis of the hydrologic stability of the crown pillar indicated that no increase in the net permeability of the crown pillar will occur because of mining-induced stress changes (Agapito, 1996). A follow up study that involved a three-dimensional elastic analysis of the crown pillar (Agapito, 1997) supported the conclusions of the original 1996 study.

Based on the analyses performed by Agapito Associates (1996; 1997), movement or subsidence of the crown pillar is unlikely to result in large scale deterioration or disturbance of the grout blanket. Grouting in the crown pillar will increase the overall strength of the rock mass that will further decrease the potential for subsidence and movement that could result in large increases in hydraulic conductivity. It is reasonable to assume, however, that limited subsidence of the lower portion of the crown pillar after the cessation of mining could result in localized increases in permeability of up to one-half order of magnitude (estimated by TRC based on professional judgment). If any subsidence of the crown pillar were to occur, it would occur relatively soon after closure of the mine and the rock mass of the crown pillar would be expected to be stable over the longer span of approximately 10,000 years.

Chemical Stability of Grout

Chemical reactions with ground water and bedrock could affect the permeability of the grout blanket over a time span of approximately 10,000 years. The following sections summarize the existing chemistry of ground water and rocks that will be in contact with the grout, along with a detailed discussion of the chemistry and changes in cement composition that can be expected with time.

Ground water in the glacial overburden is a well buffered calcium bicarbonate water with near neutral pH (range 6.0 to 8.1). Calcium content averages 34 mg/l and sulfate concentrations are generally low averaging about 21 mg/l. Analyses for total dissolved solids have a mean concentration of 202 mg/l with bicarbonate being the main constituent present in the water. (Foth & Van Dyke 1995/1998). Water quality samples collected from pumping tests in the shallow bedrock (Foth & Van Dyke 1995; TRC 1998b) indicate that ground water quality in the shallow bedrock is reasonably similar to that in the glacial overburden.

Studies performed by Steffen Robertson & Kirsten (SRK, 2000) indicate that water in the reflooded mine will have near neutral pH and moderate to high concentrations of total dissolved solids. Sulfate is expected to be the major contributor to total dissolved solids.

The bedrock that will be grouted is composed of metamorphosed pyroclastic and sedimentary rocks with variable sulfide mineralogy. Sulfide mineralogy includes pyrite, chalcopyrite, and sphalerite. In some areas, the original sulfide minerals have been weathered to form oxides of iron. Observation of cores during the Pilot Grouting Program (TRC, 1998b) did not identify minerals with abundant sulfate content.

Unhydrated cement is composed of four major compounds:

1. Tricalcium silicate (C_3S)
2. Dicalcium silicate (C_2S)
3. Tetracalcium ferroaluminate (C_4AF)
4. Tricalcium aluminate (C_3A)

Upon mixture with water, the compounds listed above react to form hardened cement composed of hydrates of calcium silicate and calcium aluminate. Lime ($Ca(OH)_2$) is also generated by the cement reaction, and although it is relatively insoluble in water, lime is the most soluble product produced during cement hydration. Hydrated calcium silicates and calcium aluminates have very low solubilities and are not subject to significant dissolution in water (Neville, 1996).

Cement grouts for the Crandon Project will use pozzolanic admixtures[1] (i.e. fly ash and silica fume) that will have the beneficial effect of reducing the lime content of the hardened cement while reducing permeability and increasing strength (Johnson et al., 1994). Superplasticizers will also be used to allow the placement of grouts with lower water to cement ratios (i.e. higher cement contents). Grouts with lower water to cement ratios have higher strengths, better durability and lower hydraulic conductivity than thinner mixtures (Neville, 1996).

The hydraulic conductivity and porosity of cement are important characteristics that control the rate of water flow through the grout and its durability. Hydraulic conductivity in cement is largely controlled by connected capillary porosity, which decreases with decreasing water to cement ratios (Neville, 1996). The use of pozzolans further decreases the hydraulic conductivity of hardened cements (Johnson et al., 1994).

The porosity, pore size distribution and hydraulic conductivity of the grout will change with time. Changes in porosity and hydraulic conductivity will result from crystallization of amorphous phases, changes in habit of the crystalline phases, reaction of hydration products with ions diffusing through the grout, dissolution of soluble products such as $Ca(OH)_2$, and the progressive hydration of cement (Johnson et al., 1994). Examination of historic cements have almost always revealed the presence of unhydrated materials even when the cements are known to have been created using thin mixtures (Johnson et al., 1994). Common reactions that occur in cement in contact with ground water include carbonation of $Ca(OH)_2$, and reaction between soluble sulfates and C_3A to form ettringite.

The reaction of hardened cement with soluble sulfates is a destructive process that can cause spalling and decrepitation of the cement. The formation of ettringite causes expansion and decrepitation of the hardened cement by solid-state crystal growth as dissolved sulfate reacts with C_3A (Mather, 1984). Factors that affect the severity of ettringite expansion include the porosity and hydraulic conductivity of the cement, cement composition and concentration of soluble

sulfate in soils and water. Insoluble sulfates and other sulfur compounds do not participate in the formation of ettringite (Neville, 1996). The aggressiveness of water and soils containing sulfate are classified in Table 1.

[1] Pozzolans are finely divided siliceous or aluminous materials that, in themselves, have little cementitious value, but will react with calcium hydroxide in the presence of water to form compounds with cementitious properties.

Table 1

Classification of Severity of Sulfate Environment According to American Concrete Institute Specification ACI 201.2R-92

Exposure	Water Soluble SO4 Concentration in Soil	SO4 Concentration in Water
Mild	<0.1 %	< 150 mg/l
Moderate	0.1 to 0.2 %	150 to 1,500 mg/l
Severe	0.2 to 2.0 %	1500 to 10,000 mg/l
Very Severe	>2.0 %	>10,000 mg/l

Based on the classification presented in Table 1, the aggressiveness of ground water and soils in the project area would be classified as mild to moderate. Type II cement that has moderate resistance to sulfate attack has been specified for the grouting program. Given the expected sulfate environment, this material should provide adequate protection from sulfate attack to insure that the grout blanket is not disrupted by ettringite expansion.

Carbonation of $Ca(OH)_2$ in cement does not cause deterioration, per se, but may have other important effects. With respect to durability, the reaction of lime with carbon species in water can result in the dissolution of $Ca(OH)_2$ and the precipitation of calcium carbonate ($CaCO_3$) in fractures and pore spaces. This effect has been observed in concretes used by ancient civilizations, and in Portland cement based concretes that have been used since the mid 19th century (Johnson et al., 1994). The presence of unhydrated cement and the process of carbonation can cause cement to self seal fractures, and is responsible for the observed reduction in average pore size that occurs with increasing age (Johnson et al., 1994). It is expected that carbonation and cement hydration will decrease the porosity and permeability of the grout blanket with increasing time.

Available data on the longevity of cement grouts indicate that with the use of appropriate quality control measures in rock masses where hydraulic gradients are small and ground water flow is slow, cement based sealants with low hydraulic conductivity and porosity can persist for hundreds of thousands of years (Johnson et al., 1994). This estimate for longevity was developed for the International Stripa Project and is based on numerical modeling (Alcorn et al. 1992), experimental data (Onofrei et al., 1992), and theoretical evaluations of the thermodynamic processes occurring with time in cement based materials in contact with granite rock and ground waters typically encountered in the Canadian Shield (Pusch, 1992; Gray, 1993). The results of these studies

suggest that the hydraulic conductivities of grouts will remain constant for 10,000 to 100,000 years (Johnson et al., 1994).

In summary, cement grouts used for the Crandon Project will be mixed with a number of durability enhancing additives that will result in placement of high quality grouts with low water to cement ratios, low porosity and permeability, and good resistance to sulfate attack. The environment in which the grout blanket will be constructed will have mild to moderately aggressive water and rocks with regard to sulfate content. Over the period of interest ($\approx$ 10,000 years) the grout blanket is expected to be stable without significant increases in porosity and permeability caused by dissolution, ettringite expansion or other chemical processes. In addition, the processes of carbonation and the progressive hydration of cementitious materials may result in self healing of fractures and reduction of pore space in the grout blanket with increasing time. This would result in an actual reduction in permeability of the grout curtain with increasing age.

Conclusion

An evaluation of the potential mechanisms for degradation of the grout blanket did not indicate a process that would result in significant deterioration of the hydraulic barrier over a period of about approximately 10,000 years. After reflooding, hydraulic gradients will be low in the area of the grouted mine and the movement of ground water will be too slow to cause physical erosion of the grout barrier.

Modern grouting techniques and materials will be used to construct the grout curtain including the use of pozzolanic admixtures and superplasticizers that will result in the placement of chemically stable grouts with low porosity and hydraulic conductivity. The environment in which the grout blanket will be constructed will have mild to moderately aggressive water and rocks with regard to sulfate content, and grout is expected to be stable without significant increases in porosity and permeability caused by dissolution, ettringite expansion or other chemical processes. In addition, the processes of carbonation and the progressive hydration of cementitious materials may result in self healing of fractures and reduction of pore space in the grout blanket with increasing time. This would result in an actual reduction in the porosity and permeability of the grout curtain with increasing age.

The use of cut-and-fill mining techniques below the crown pillar and "tight" backfilling of the stopes with high strength cemented paste backfill will minimize the potential for subsidence and instability of the crown pillar. Construction of the grout blanket will also add additional strength and stability to the rock mass. Although some limited subsidence may occur after the end of mining, analyses by Agapito Associates indicate that the disturbance will be minimal and will not compromise the hydraulic stability of the crown pillar. Some disturbance of the grout blanket could occur, however, and it is estimated that this disturbance has the potential to increase the hydraulic conductivity of the grout blanket by about one half order of magnitude. If limited portions of the grout blanket are disturbed by subsidence, the process of carbonation may result in the healing of fractures with calcium carbonate and subsequently reduce the permeability of the grout blanket to near the original condition of about 3×10^{-3} ft/day (1×10^{-6} cm/sec).

Historical experience with grouting supports the conclusion that the grout curtain will remain intact without significant increases in permeability and porosity for an extended period of time. Several examples exist in which cement grouts were used to control water inflow to mine shafts and have not required repair for periods of more than 50 years. It is notable that these grout

barriers did not incorporate modern grouting methods or materials and have performed exceptionally over the period of record. Grouting technology and materials used at the Crandon Project will be state of the art, and should result in the construction of a hydraulic barrier superior in performance and durability to those constructed in the early 1900's. It is TRC's opinion that the grout blanket should maintain its hydraulic integrity and low permeability of approximately 3 x 10^{-3} ft/day (1 x 10^{-6} cm/sec) over a 10,000 year period barring disturbance by human activities or seismic events. The likelihood of seismic activity in the project area is considered to be extremely low (Merritt et al., 1995; and Foth & Van Dyke 1995/1998a).

References

Agapito Associates, Inc., (1996). Hydrologic Stability of the Crown Pillar, Crandon Deposits.

Agapito Associates, Inc., (1997). Numerical Analysis Results for Evaluation of the Influence of Mining on the Crown Pillar, Crandon Deposit.

Alcorn, S.R., Coons, W.E., Christian-Frear, T.L., and Wallace, M.G., (1992). Theoretical Investigations of Grout Seal Longevity – Final Report. Stripa Project Report, TR-92-23, Swedish Nuclear Fuel and Waste Management Company, Stockholm.

American Concrete Institute (1994). ACI 201.2R-92, Guide to Durable Concretes, ACI Manual of Concrete Practice.

Foth & Van Dyke, (1995). Investigation and Evaluation: Saprolite at the Crandon Site, Crandon Project, Crandon, Wisconsin. A report prepared for Crandon Mining Company by Foth & Van Dyke and Associates, Inc., Green Bay, Wisconsin.

Foth & Van Dyke, (1995/1998a). Environmental Impact Report for the Crandon Project. A report prepared for Crandon Mining Company by Foth & Van Dyke and Associates, Inc., Green Bay, Wisconsin.

Fernandez, J.A., Richardson, A.M., (1994). Yucca Mountain Characterization Project - A Review of the Available Technology for Sealing a Potential Underground Nuclear Waste Repository at Yucca Mountain, Nevada. Prepared by Sandia National Laboratories, Albuquerque, New Mexico.

Gray, M.N. (1993). OECD/NEA International Stripa Project Overview, Volume III: Engineered Barriers. Swedish Nuclear Fuel and Waste Management Company, Stockholm

HSI GeoTrans, (1996). Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Zinc and Copper Mine Near Crandon Wisconsin. A report prepared for Crandon Mining Company by GeoTrans, Inc., Roswell, Georgia.

HSI GeoTrans, (1998a). Addendum No.1 to Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Zinc and Copper Mine Near Crandon, Wisconsin. A report prepared for Crandon Mining Company by GeoTrans, Inc., Roswell, Georgia.

HSI GeoTrans, (1998b). Errata and Clarification to Addendum No.1 to Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Zinc and Copper Mine Near Crandon, Wisconsin. A report prepared for Crandon Mining Company by GeoTrans, Inc., Roswell, Georgia.

HSI GeoTrans, (1998c). Addendum No.2 to Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Zinc and Copper Mine Near Crandon, Wisconsin. A report prepared for Crandon Mining Company by GeoTrans, Inc., Roswell, Georgia.

HSI GeoTrans (2000). Personal Communication between Greg Council of HSI GeoTrans and Scott Effner of Whetstone Associates on September 13, 2000.

Johnson, L.H., Tait, J.C., Shoesmith, D.W., Crosthwaite, J.L., and Gray, N.M., (1994). The Disposal of Canada's Nuclear Fuel Waste: Engineered Barriers Alternatives. AECL Research Paper, AECL-10718; COG-93-8,

Mather, (1984). A discussion of the Paper "Theories of Expansion in Sulfoaluminate-Type Expansive Cements: Schools of Thought." by M.D. Cohen, Cement and Concrete Research, 14 pp. 603-609.

Merritt, F.S., Loftin, M.K., Ricketts, J.T., (1995). Standard Handbook for Civil Engineers, Fourth Edition. Published by McGraw-Hill.

Neville, A.M., (1996). Properties of Concrete. Published By John Wiley and Sons.

Onofrei, M., Pusch, R., Gray M.N., Borgesson, L., Karnland, O., Shenton, B., and Walker, B., (1992). Sealing Properties of Cement-Based Grout Materials. Stripa Project Report, TR-92-28, Swedish Nuclear Fuel and Waste Management Company, Stockholm.

Pusch, R., (1992). Executive Summary and General Conclusions of the Rock Sealing Project. Stripa Project Report, TR-92-27, Swedish Nuclear Fuel and Waste Management Company.

Steffen Robertson & Kirsten (2000). Reflooded Mine Source Concentration, Crandon Mine. A report prepared for Foth & Van Dyke by Steffen Robertson & Kirsten of Vancouver, British Columbia.

TRC Hydro-Geo Consultants, (1998a) Water Control and Grouting Plan, Crandon Project. A report prepared for Nicolet Minerals by Hydro-Geo Consultants Inc., Lakewood, Colorado.

TRC Hydro-Geo Consultants, (1998b). Description of Methods and Results: Pilot Grouting Program, Crandon Project. A report prepared for Nicolet Minerals by Hydro-Geo Consultants Inc., Lake

Attachment C. Letter from Agapito Associates regarding "Long Term Drift Stability"

AGAPITO ASSOCIATES, INC.
CONSULTING ENGINEERS

GRAND JUNCTION OFFICE

November 14, 2000 235-10

Gordon Reid
Nicolet Minerals Company
104 W. Madison Street
Crandon, WI 54520-0336

RE: **Long-term Drift Stability Letter Report, Crandon Mine, Forest County, Wisconsin**

Dear Mr. Reid:

This letter report describes our evaluation of the long-term drift stability at the proposed Crandon Mine in Forest County, Wisconsin. This evaluation reviewed the expected ground conditions in the mine development drifts in the extended period beyond mine closure.

The mine developments of concern are the eleven, 16-ft-wide drifts to be driven in the hanging wall in the volcanic host rock offset 40 ft from the Crandon Formation, as shown in Figures 1 and 2. The drifts between the 300 and 2300 levels will be driven with 200-ft vertical spacings. They will be located between elevations +1350 ft above mean sea level (MSL) to 650 ft below MSL, inclusive. All of the drifts are planned to be nominally 13-ft high with an arch back. The footwall drifts planned in the western portion of the orebody are of lesser concern because of the higher rock strengths and rock quality in the footwall.

The expected ground conditions in the development drifts have been characterized in reports prepared for mine design and permitting purposes (AAI, February and May 1996, and September 2000). The methodology adopted here was to review this data with specific reference to the spatial location of the development drifts. This data was used to establish the expected range of ground conditions (using Barton's Q method, Barton and Grimstad 1994) in the development drifts, the range of *in situ* and post-mining stresses at the location of the drifts, and the expected ground support (rockbolts, shotcrete, etc.) that would be installed during mining and maintained during mining operations. After closure of the mine, the water table is expected to recover, drifts are expected to flood, and the support is expected to deteriorate over time.

The degree of weathering of the host rock affects its mechanical strength and, hence, its ability to sustain stable drift openings. The degree of weathering of the host rock at a vertical cross section which passes through the mine coordinate x = 2,777,800E is shown in Figure 1. The degrees

of weathering in the hanging wall for all of the eleven drifts, at five cross sections along the strike of the ore body, are tabulated in Table 1. All of the 300 Level drifts in the five sections are in the moderately-weathered zone. All of the drifts below the 500 Level in all five sections are in the weakly-weathered or unweathered zones of the hanging wall.

Barton's Q system was used to assess the long-term drift stability at the proposed Crandon Mine. The Q system takes into account the weathering, the rock joint characteristics, RQD[1], and the rock strength-to-stress ratio. The Q method of rating stability is an empirical one based on case histories of many tunnels for various rock types, conditions, tunnel support systems (e.g., rock bolts), and tunnel sizes.

The range in site-specific parameters, which describe the Crandon Mine rock quality rating for the drift stability for the hanging wall drifts according to the Barton Q method, is listed in Table 2. Details of the Crandon rock quality parameters of Table 2 are given in Tables 3 and 4. For the conditions and rock around the hanging wall drifts, the rock mass quality Q varies between 0.23 and 1.67 as shown on Barton's stability chart of Figure 3. Also shown on the vertical axis of Figure 3 is the range of the opening dimension divided by the Engineering Support Ratio (ESR). The ESR reflects construction practice for the degree of safety and support demanded by an excavation. The range in ESR for the hanging wall drifts after mine closure is between 3 and 5 (values of ESR recommended by Barton and Grimstad (1994) for temporary mine openings). For 16-ft (4.8 meters) wide drifts, for the range in ESR and for the range in rock mass quality Q, Barton's stability chart indicates that support will be required in most drifts. Rio Algom's ground control policy is to install pattern bolting and mesh in all development drifts. A significant length of the drifts in the unweathered hanging wall with a Q value of ≥ 0.5 could require no ground support. For rock with $Q < 0.5$, the proposed ground support will be functional. For the poorer quality rock, the unsupported standup time is of the order of hours as per the methodology proposed by Bieniawski (1984). Ground support cannot be relied upon for the poorer quality rock in the very long term because of deterioration of the ground support components, and hence, collapse is likely.

In the footwall, the Q is estimated to be higher than 1.0 based on the higher rock strength and better RQD. The rock strengths for the footwall rocks are approximately three times the strength of the hanging wall rocks, and the RQD ranges from 72% to 100%. For Q's greater than 1.0 and drift dimensions of 16 ft width, no ground support is required based on the Barton stability chart, see Figure 3. Hence, we believe that deformation over time for the footwall drift will be less than for the hanging wall drifts.

[1]See Table 3 for definition of RQD.

After mine closure, some hanging wall drifts are likely to collapse when their ground support deteriorates. The failure in the hanging wall is expected to be gradual and result in blocky ground fallout with the failure controlled by the jointing. The maximum extent of this fallout will be limited by the bulking, or swell, of the fallout material. The broken fallout rock will eventually fill the void of the drift and fallout area in the roof. When the void is full, the rubble will resist further roof fall. The swell of the rubble that falls out of the roof could be up to 80% with a resulting porosity of 44% (Pfleider 1968), but may be as low as 30% if the fallout material stacks and does not rotate during fallout (this is equivalent to a porosity of 23%). The maximum upwards extent of the roof above the floor is equal to the drift height divided by the porosity, and it is independent of drift width[2]. Thus, it is anticipated that the failure zones will extend upwards approximately 16 to 43 ft above the original drift roofs, corresponding to porosities of 44% to 23%, respectively. Figure 4 shows schematically the expected long-term drift geometries ranging from stable drifts to drifts with 16- to 43-ft high collapsed zones. An intermediate condition with local roof fallout resulting in reformation of a stable arch is expected in the majority of the openings in the unweathered drifts. Little or no roof fallout is expected in the footwall drifts.

For the expected range in collapsed rock geometry (i.e., blocky failure with porosities ranging between 23% and 44%), the hydraulic conductivity of the collapsed rock could range from that of gravel to that of a moderately permeable to highly permeable Karst limestone, i.e., 10^{-2} to 1 cm/sec or 30 to 3000 ft/day (Freeze and Cherry 1979). For comparison purposes, the best engineering judgement (BEJ) pre-mining hydraulic conductivities in the moderately-weathered and weakly-weathered hanging wall range between 3×10^{-4} and 4.4×10^{-2} ft/day (HSI GeoTrans 1998) in the east-west, north-south and vertical directions. Thus, it is estimated that the hydraulic conductivity of the collapsed drifts will be orders of magnitude higher than the surrounding rock. For the drifts in the unweathered hanging wall at depth, it is expected that some drifts will remain open with only minor roof fallout and, hence, will have very high hydraulic conductivities— orders of magnitude higher than the surrounding rock.

In summary, we have evaluated the long-term stability of the hanging wall drifts at the proposed Crandon Mine. Our evaluation is based on the existing pre-mining database, upon our experience, and upon the experience of others as cited in the literature, especially the experience collated and classified by Barton and Grimstad (1994). It is our assessment that some drifts will collapse in the very long term, after their structural support systems (e.g., rockbolts and mesh) have corroded/weathered, etc. The collapse will propagate slowly upwards and eventually the fallout will fill the void of the drift. The collapse will stabilize at a height which depends on the porosity of fallout material. We estimate the porosity of the roof fallout material could range between 23% and 44%, and that the roof fallout could extend up to 43 ft above the original roof. We estimate that the hydraulic conductivity of the collapsed rock will be orders of magnitude higher than the surrounding

[2]Total height of fallout above floor = original drift height ÷ porosity.

rock. In many areas in the unweathered hanging wall rocks and in the footwall drifts, only minor roof fallout is expected in the long term.

Sincerely,

Margaret Asgian
Associate

M. P. Hardy
Principal

Attachments: References
Tables 1–4
Figures 1–4

cc: S. Donohue, Foth & Van Dyke (w/attachments)

MPH/MIA/klg

REFERENCES

Agapito Associates, Inc. (2000), "Post-Mining Induced Stresses, Crandon Mine, Forest County, Wisconsin," prepared for Nicolet Minerals, October 27.

Agapito Associates, Inc. (1996), "Rock Mechanics Review of Proposed Stope Dimension Areas 2511, 2011LC and 2011E Crandon Deposit," report submitted to Crandon Mining Company, Rhinelander, Wisconsin, February 27.

Agapito Associates, Inc. (1996), "Hydrologic Stability of the Crown Pillar, Crandon Deposit," report submitted to Crandon Mining Company, May 7.

Barton, N. and E. Grimstad (1994), "Rock Mass Conditions Dictate Choice Between NMT and NATM," *Tunnels & Tunnelling,* October.

Bieniawski, Z. T. (1984), *Rock Mechanics Design in Mining and Tunneling*, A. A. Balkema, Rotterdam.

Brady, B. H. G. and E. T. Brown (1985), *Rock Mechanics for Underground Mining,* George Allen & Unwin, London.

Crandon Mining Company (1995), Five figures labeled "Geosolution Resources Inc. Assay Order ZN, CU for Sections 275025E, 275925E, 276975E, 277800E, and 279000E," November 13.

Deere, D. U. (1968), "Geological Considerations," *Rock Mechanics for Engineering Practice,* K.G. Stagg and O.C. Zienkiewicz (eds.), London: Wiley, pp. 1–20.

Foth & Van Dyke Associates, Inc. (1995), "Bedrock Hydrogeology at the Crandon Site, Crandon Project, Crandon, Wisconsin," report prepared for Crandon Mining Company.

Freeze, R. Allan and John A. Cherry (1979), *Groundwater,* Prentice-Hall, Inc., Englewood Cliffs, New Jersey.

HSI GeoTrans Inc. (1998), Addendum No. 1 to "Numerical Simulation of the Effect of Groundwater and Surface Water on the Proposed Zinc and Copper Mine near Crandon, Wisconsin."

Pfleider, E. P. (1968), *Surface Mining*, American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., Maple Press Company, York, Pennsylvania.

Table 1. Degree of Weathering[1] in the Vicinity of the Hanging Wall Drifts[2]

Level (ft)	North–South Section x = 2,275,025E	North–South Section x = 2,275,925E	North–South Section x = 2,276,975E	North–South Section x = 2,277,800E	North–South Section x = 2,279,000E
300	MWI	MWI	MWI	MWI	MWI
500	WWI	WWI	MWI	WWI	MWI
700	WWI or UW	UW	WWI	UW	WWI
900	WWI or UW	UW	WWI	UW	WWI
1100	WWI or UW	UW	WWI or UW	UW	WWI
1300	WWI or UW	UW	WWI or UW	UW	UW
1500	WWI or UW	UW	WWI or UW	UW	UW
1700	WWI or UW	UW	WWI or UW	UW	UW
1900	WWI or UW	UW	WWI or UW	UW	UW
2100	WWI or UW	UW	WWI or UW	UW	UW
2300	WWI or UW	UW	WWI or UW	UW	UW

[1] Tabulated from Crandon Mining Company, five figures dated November 13, 1995, labeled "Geosolution Resources, Inc. Assay Order ZN, CU for Sections 275025E, 275925E, 276975E, 277800E and 279000E.
[2] All drifts are in the hanging wall.
SWI = Strongly-weathered Intensity
MWI = Moderately-weathered Intensity
WWI = Weakly-weathered Intensity
UW = Unweathered

Table 2. Site-Specific Parameters Describing Rock Mass Quality *Q* for the Hanging Wall Drift Stabilities

Engineering Rock Quality Parameter*	Range in Values	Comments
RQD (rock quality designation) RQD for moderately, weakly and unweathered rock	35–100	Refer to Table 3.
J_n (joint set number)	6	Two joint sets with spacing ≤ drift span, plus random joints at larger spacings (refer to Foth & Van Dyke (1995) for joint set descriptions).
J_r (joint roughness number)	2	Joint roughness characterized as smooth, undulating (refer to p. 4-4 of AAI, February 1996 report).
J_a (joint alteration number)	1	Joint alteration was characterized as unaltered, surface staining only (refer to p. 4-4 of AAI, February 1996 report).
J_w (joint water reduction factor)	0.1	Long-term water pressure in all drifts expected to be ≥ 9.2 kg/cm^2 = 130 psi.
SRF (stress reduction factor)	2–5	Refer to Table 4.

$$*Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

Refer to Barton and Grimstad (1994) for details and maximum possible ranges in each dimensionless rock quality parameter.

Q = 0.23 to 1.67 for the above values.

Table 3. RQD[1] as a Function of Weathering in the Hanging Wall

Degree of Weathering	Moderately-weathered[2]	Weakly-weathered[2]	Unweathered[3]
Mean RQD	54	66	—
Range in RQD	35–67	61–77	35–100

[1] RQD = Rock Quality Designation (as defined by Deere (1968) and cited by Brady and Brown (1985) is equal to a percentage of core in a drill run which has a length ≥ 10 cm.
[2] From AAI, May 1996 report, Table B-1, p. B-2.
[3] From AAI, February 1996 report, Appendix D. The range in RQD is 0 to 100 in the hanging wall. However, a lower cutoff of RQD = 35 has been made as we attribute the relatively infrequent, low RQDs to lost core and to zones of weakness. A substantial number of drill runs in the unweathered hanging wall have an RQD ≥ 90.

Table 4. Stress Reduction Factors (SRF) in Hanging Wall Rock Drifts

Weathering Intensity	Average Compressive Strength[1] σ_c	Range of Post-mining Maximum Effective Compressive Stress[2] σ'_1	Ratio of Strength to Stress σ_c/σ'_1	SRF Used
Moderate	5995 psi	870 to 2040 psi (300 to 500 levels)	2.9 to 6.9	2 to 5
Weak	5995 psi	800 to 2510 psi (500 to 1100 levels)	2.4 to 7.5	2 to 5
Unweathered	9690 psi	800 psi to 3480 psi (700 to 2300 levels)	2.8 to 12.1	2 to 5

[1] The average compressive strength σ_c for moderately-weathered and unweathered rock is from Table 3-1, p. 3-13, in AAI's May 1996 report. σ_c for weakly-weathered rock is assumed to be equal to that of moderately-weathered rock. Within a given weathering unit, there is a wide range in compressive strengths. For example, in the moderately-weathered hanging wall rock, σ_c varies between 1,865 and 11,857 psi in boreholes 171, 188, 191 and 195 (refer to Table B-4, pp. B-11 to B-16, AAI, May 1996 report).

[2] The maximum post-mining effective normal stress $\sigma'_1 = (\sigma^0_1 + \Delta\sigma_1) - p$ where $\sigma^0_1 = 8.1 + 0.03D$ and is in MPa, depth D is in meters, $\Delta\sigma_1$ is the mining-induced normal stress, and p is equal to the pore pressure assumed equal to the hydrostatic pressure with the water table at the ground surface, elevation = 1650 ft above MSL. Premining maximum compressive stress σ^0_1 is from p. C-2 in AAI, May 1996 report. $\Delta\sigma_1$ was taken from AAI (2000) to be equal to the mining-induced horizontal stress.

The mining-induced stresses vary within a given cross-section and they vary from cross-section to cross-section. For the four vertical cross-sections (x = 2,275,625E; x = 2,276,075E; x = 2,277,800E; and x = 2,278,550E), the mining-induced normal stresses on planes parallel to the orebody at Crandon are predicted to range between −1,200 psi (mining-induced tensile stresses) and +600 psi (mining-induced compressive stresses) at the hanging wall drifts. The predicted mining-induced normal stresses tend to be tensile in regions close to the mined ore and they tend to be compressive in regions close to the unmined ore.

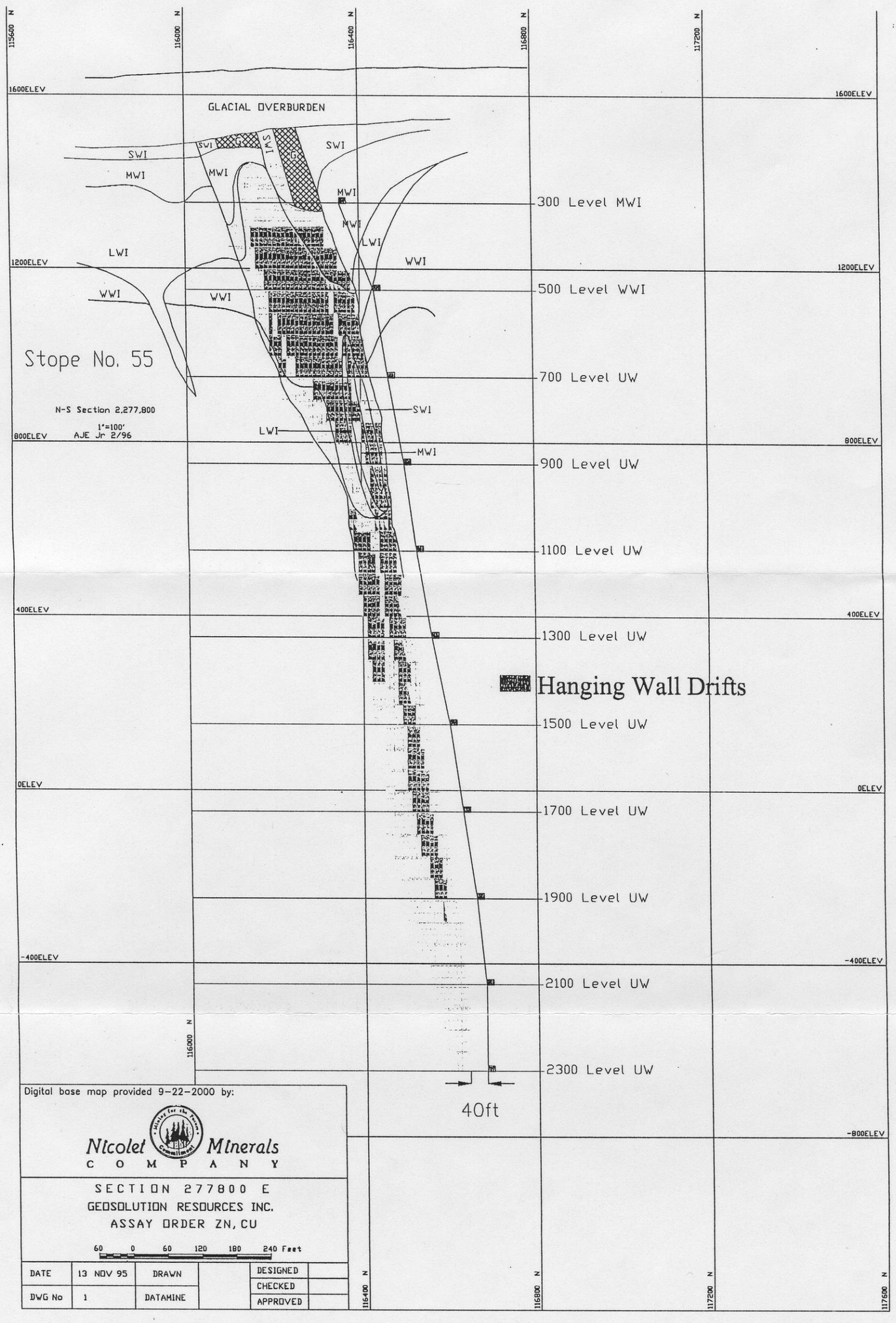


Figure 1. Hanging Wall Drifts in North-South Section 2,277,800 E

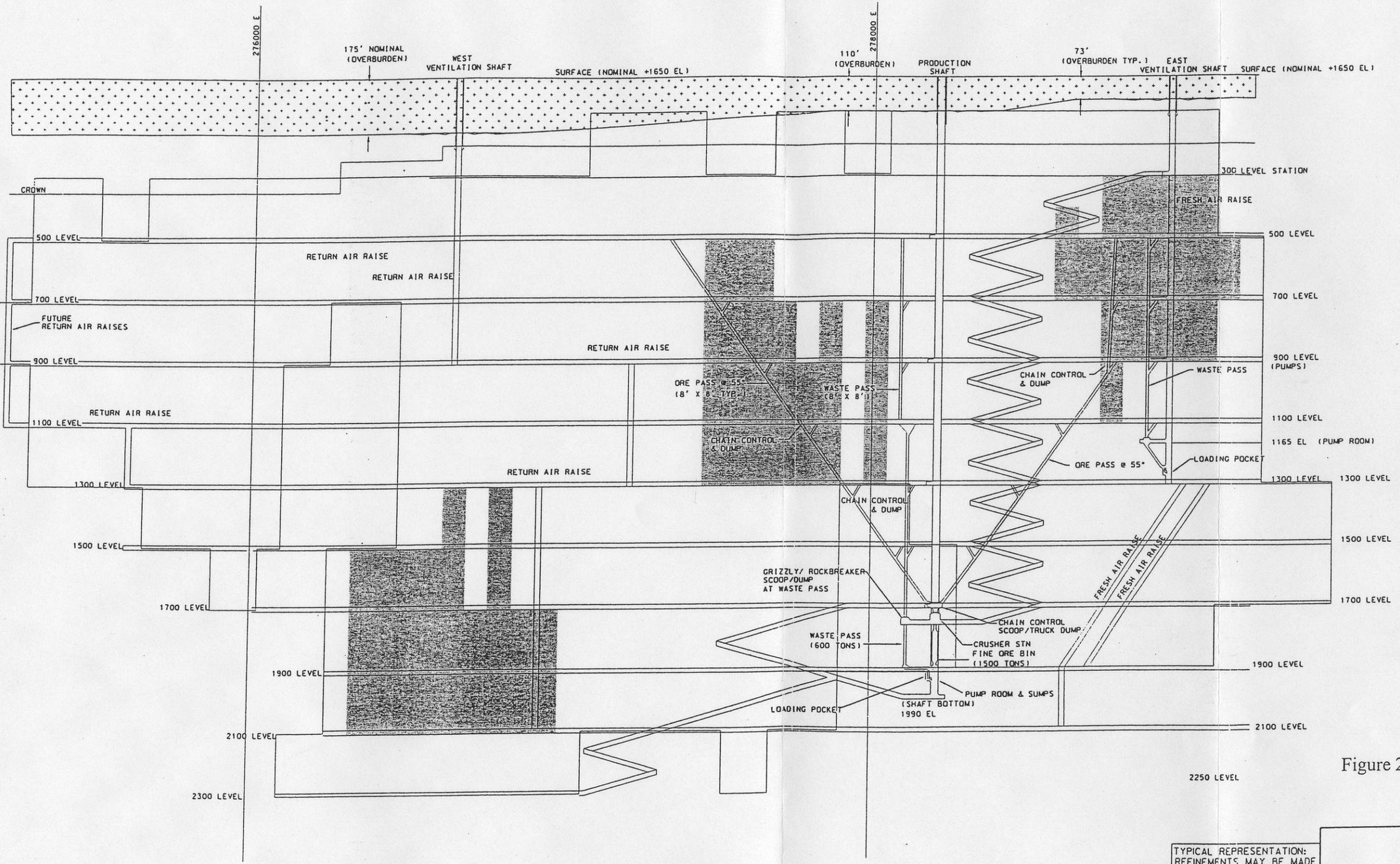


Figure 2. Depths of Hanging Wall Drifts at the Proposed Crandon Mine

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LEGEND

EARLY PRODUCTION AREAS

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	TWS	DATE:	DEC. '98
APPROVED BY:	SAD2	DATE:	DEC. '98
APPROVED BY:	GWS	DATE:	DEC. '98

Nicolet Minerals COMPANY

FIGURE 4-6

SCHEMATIC LONGITUDINAL SECTION (LOOKING NORTH)

Scale:	NOT TO SCALE	Date:	DECEMBER, 1998
Prepared By: Foth & Van Dyke		By: BSH	93C049

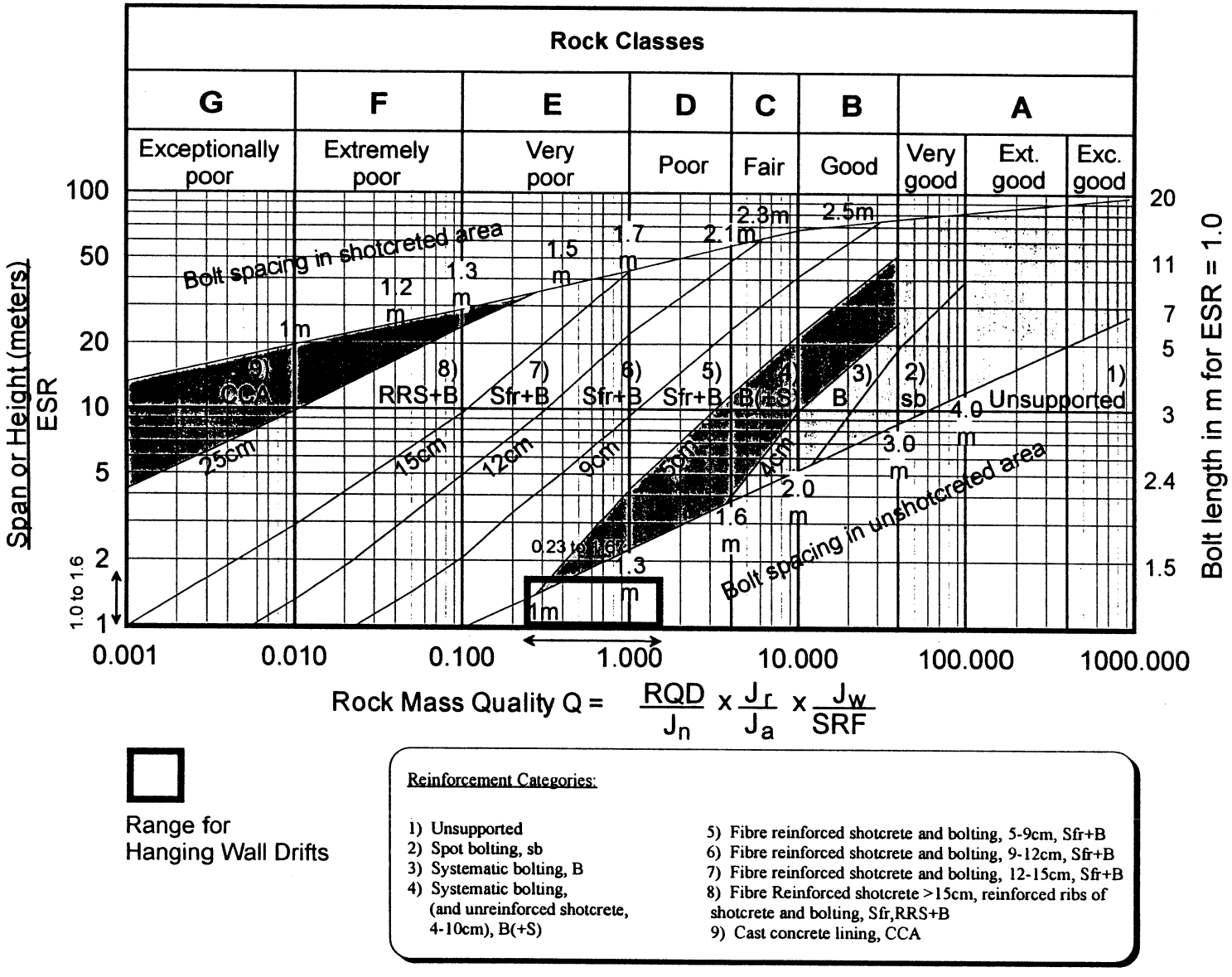


Figure 3. Stability of the Hanging Wall Drifts at the Proposed Crandon Mine, as Rated by Barton's Q Method (Stability Chart Redrawn from Barton and Grimstad, 1994)

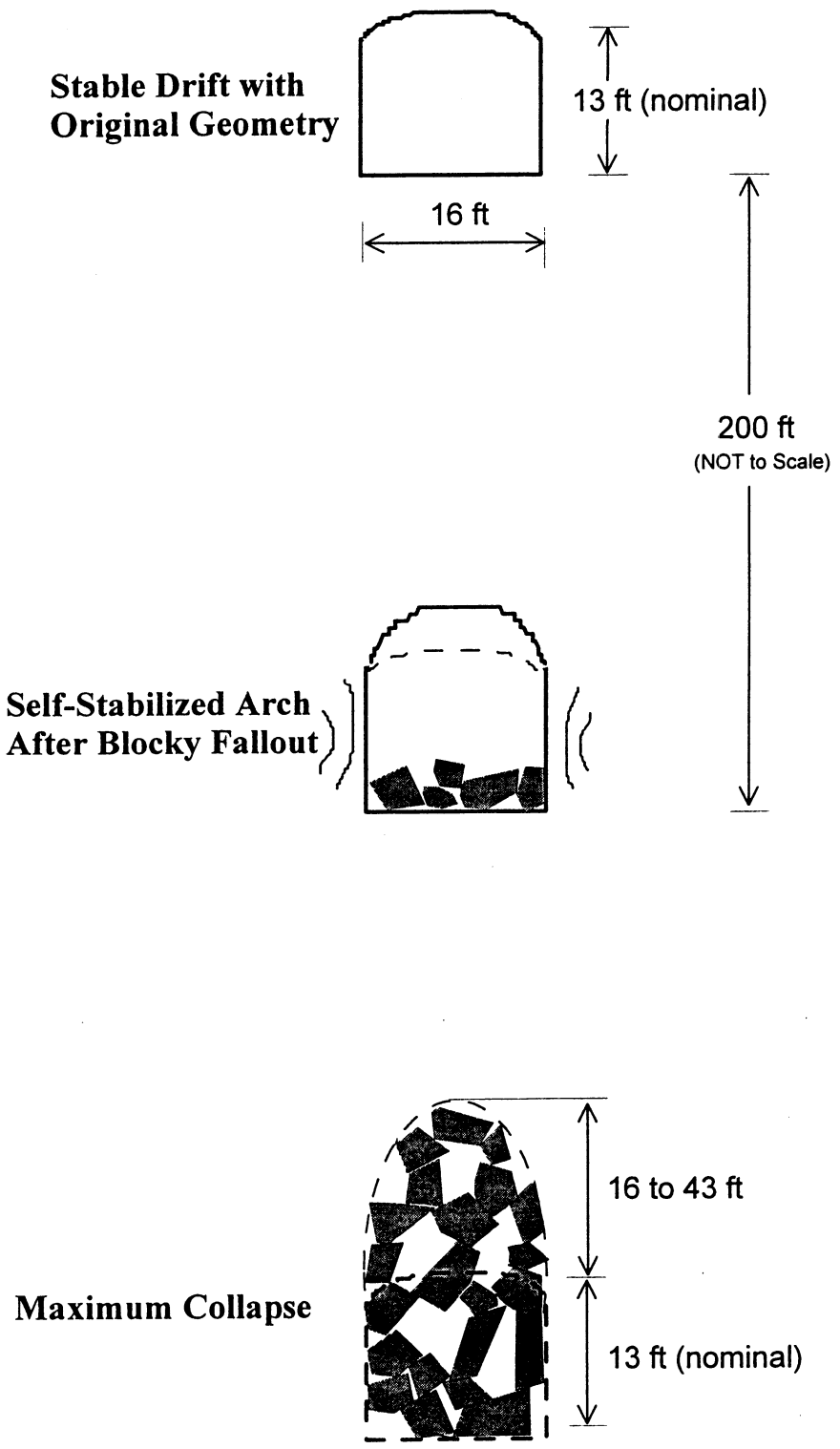


Figure 4. Expected Range in Drift Geometry in the Long Term after Mine Closure when the Ground Support (e.g., Rock Bolts and Mesh) Has Deteriorated

Appendix C

Reflooded Mine Containment System Modeling

HSI GEOTRANS
TECHNICAL MEMORANDUM

TO: Steve Donohue, Foth & Van Dyke

FROM: Greg Council and Pete Andersen, HSI GeoTrans

DATE: November 13, 2000

SUBJECT: Crandon Project, Reflood Containment Modeling
HSI GeoTrans Project No. N015-211

Introduction

The purpose of this memorandum is to evaluate, based on different conceptualizations of the hydraulic characteristics of the bedrock system, a range of hydraulic containment scenarios that could be employed for the Crandon Project reflooded mine to control the migration of solute.

The envisioned containment system would involve several extraction wells in the bedrock that would maintain an inward gradient from all mine workings and backfill toward the wells. In this memorandum, trial-and-error placement of containment wells in the regional groundwater flow model (HSI GeoTrans, 1998) is used to determine the feasibility of the envisioned containment system and to estimate the total extraction rate that would be required.

Methodology

MODFLOW and MODPATH were used to determine the most appropriate placement of the containment wells. Particles were placed in layers 5-13 in all active cells within the development area, which includes the mine and underground workings. Multiple model parameter sets were used — including the current BEJ model (HSI GeoTrans, 1998), a modified (more conservative) version of the PWC model (HSI GeoTrans, 1998), and the USGS high-range conductivity model (USGS, 1999) — to test containment feasibility under a full range of likely conditions, and to determine a range of total extraction that may be required. Additionally, a simulation to determine if containment could be achieved by extraction only in the upper portion of the bedrock was made, assuming that the unweathered bedrock has a very low conductivity. Finally, a sensitivity simulation was made to see whether preferential flow within open workings in the hanging wall would have an effect on the necessary extraction rate. The following five specific simulations were made:

- Simulation A, using the current BEJ model including a low-conductivity grout layer in the upper half of layer 6, and with a low conductivity assigned to all mined cells to represent the cemented backfill (unweathered bedrock inactive),

- Simulation B, using the PWC model with modifications that resulted from a recent recalibration to the bedrock pump tests (HSI GeoTrans, 1999), including activation of the unweathered bedrock at a conductivity of 0.00094 ft/d (as a conservative assumption, no reduction in conductivity was made for grout or cemented backfill),

- Simulation C, using the high-range conductivities of the USGS (1999), again with active unweathered bedrock (at a conductivity of 0.005 ft/d) and no conductivity reduction for grout or backfill,

- Simulation D, using the same BEJ model as Simulation A, but with extraction allowed only in layers 5-7 (above elevation 1250 ft, msl), and

- Simulation E, using the same BEJ model as Simulation A, but with a zone of high conductivity used as a simplified representation of open hanging wall workings.

Except in Simulation D, extraction was permitted in bedrock layers 5-11 (down to elevation 350 ft, msl). Simulation A was made to determine an expected value for required extraction to achieve containment, and Simulations B and C were for determination of a worst-case or upper-bound extraction rate. Simulation D was for evaluation of a potential strategy of only shallow extraction (assuming it was determined, during mining, that the unweathered bedrock was not transmissive). Simulation E was made to ensure that open hanging wall workings would not have an important effect on simulation results.

Wells were added in Simulation A until all particles were captured. Extraction rates were then reduced, in 0.26 gpm (50 ft^3/d) increments, to minimize extraction rates while still collecting all of the particles. Once a set of well locations was established for Simulation A, that set of wells was used for Simulation B. Extraction rates were increased and wells were added until all particles were captured. For Simulation C, the Simulation B wells were used and the extraction rates were modified until a minimum rate was found. Simulation D started with the Simulation A well locations in layers 5-7 only. Wells were added until all particles were captured. Likewise, the Simulation A wells were used for Simulation E, with locations and extraction rates altered until capture was achieved. In Simulation E, an arbitrarily high hydraulic conductivity of 5 ft/d was assigned for a row of cells in the hanging wall in all mining layers (layers 7-13).

Results

Well locations and extraction rates per cell are tabulated in Tables 1-5 for Simulations A-E. Figure 1 shows the locations of the wells and total extraction rates for Simulation A. The modeled particle capture zones for Simulation A, indicating complete capture within the development boundary, are shown in Figure 2. Figures 3, 4, 5 and 6 show the locations of the wells and total extraction rates for Simulations B, C, D and E, respectively. A total extraction of 9.6 gpm was required for Simulation A, while a total of 32.7 gpm was required for Simulation B. A total of 26.0 gpm was required for Simulation C, while 21.3 gpm was required for Simulation D. Simulation E required an extraction of 13.8 gpm. The well locations determined by this analysis are conceptual and the final locations would be based on site conditions as determined during mining operations. For instance, if the unweathered bedrock was found to be effectively impermeable, thus negating the need for deep containment wells, the containment system would target the weathered bedrock. The horizontal location of the wells would be optimized prior to construction based on revised estimates of the hydraulic characteristics of the bedrock obtained during the mining operations.

These simulations indicate that reflooded mine source containment is feasible. It can be achieved with 5-9 bedrock wells extracting between approximately 10 gpm and 35 gpm in aggregate. Simulation D shows that it may be feasible to extract only water from the upper bedrock and still achieve containment. This will be possible if it is determined (during mining) that the till/massive saprolite leakance is low and the unweathered bedrock conductivity is very low, as specified in the BEJ model.

References

HSI GeoTrans, 1998. Addendum No. 2 to: Numerical Simulation of the Effect on Groundwater and Surface Water of the Proposed Zinc and Copper mine near Crandon, Wisconsin.

HSI GeoTrans, 1999. October 7, 1999 Technical Memorandum from Greg Council (HSI GeoTrans) to Steve Donohue (Foth & Van Dyke) regarding "Crandon project, Model calibration for the 213 and 211 Pump Tests".

USGS, 1999. December 21, 1999 Technical Memorandum from Daniel Feinstein (USGS) to Chris Carlson (WDNR) regarding "Calibration of 213,211, and Massive Saprolite (PWAR) Pumping Tests".

Table 1. Simulation A (BEJ) well locations and extraction rates (gpm).

Well	1	2	3	4	5	Total
Row, Column	56,29	59,26	61,51	54,50	68,63	
Layer 5		0.52	0.26			0.78
Layer 6		0.52	0.26			0.78
Layer 7		0.52		0.26	0.52	1.30
Layer 8		0.52		0.26	0.52	1.30
Layer 9	1.56	0.52	0.52	0.78		3.38
Layer 10		0.52	0.52	1.04		2.08
Total Rate	**1.56**	**3.12**	**1.56**	**2.34**	**1.04**	**9.6**

Table 2. Simulation B (PWC) well locations and extraction rates (gpm).

Well	1	2	3	4	5	6	7	Total
Row, Column	56,29	59,26	61,51	54,50	68,63	61,37	55,64	
Layer 5			1.04					1.04
Layer 6			1.04					1.04
Layer 7		1.56		1.04	0.52			3.12
Layer 8	1.30	1.56		1.04	0.52	1.04	1.04	6.50
Layer 9	1.30	1.56	1.04			1.04	1.04	5.98
Layer 10	1.30	1.56	1.04	2.08		1.04	1.04	8.06
Layer 11	1.30	1.56		2.08		1.04	1.04	7.02
Total Rate	**5.20**	**7.80**	**4.16**	**6.24**	**1.04**	**4.16**	**4.16**	**32.7**

Table 3. Simulation C (USGS High Range) well locations and extraction rates (gpm).

Well	1	2	3	4	5	6	7	Total
Row, Column	56,29	59,26	61,51	54,50	68,63	61,37	55,64	
Layer 5			1.04					1.04
Layer 6			1.04					1.04
Layer 7		1.04		1.04	0.52			2.60
Layer 8	1.04	0.78		1.04	0.52	1.04	0.78	5.20
Layer 9	1.04	0.78	0.78			1.04	0.78	4.42
Layer 10	0.78	0.78	0.78	2.08		1.04	0.78	6.24
Layer 11	0.78	0.78		2.08		1.04	0.78	5.46
Total Rate	**3.64**	**4.16**	**3.64**	**6.20**	**1.04**	**4.16**	**3.12**	**26.0**

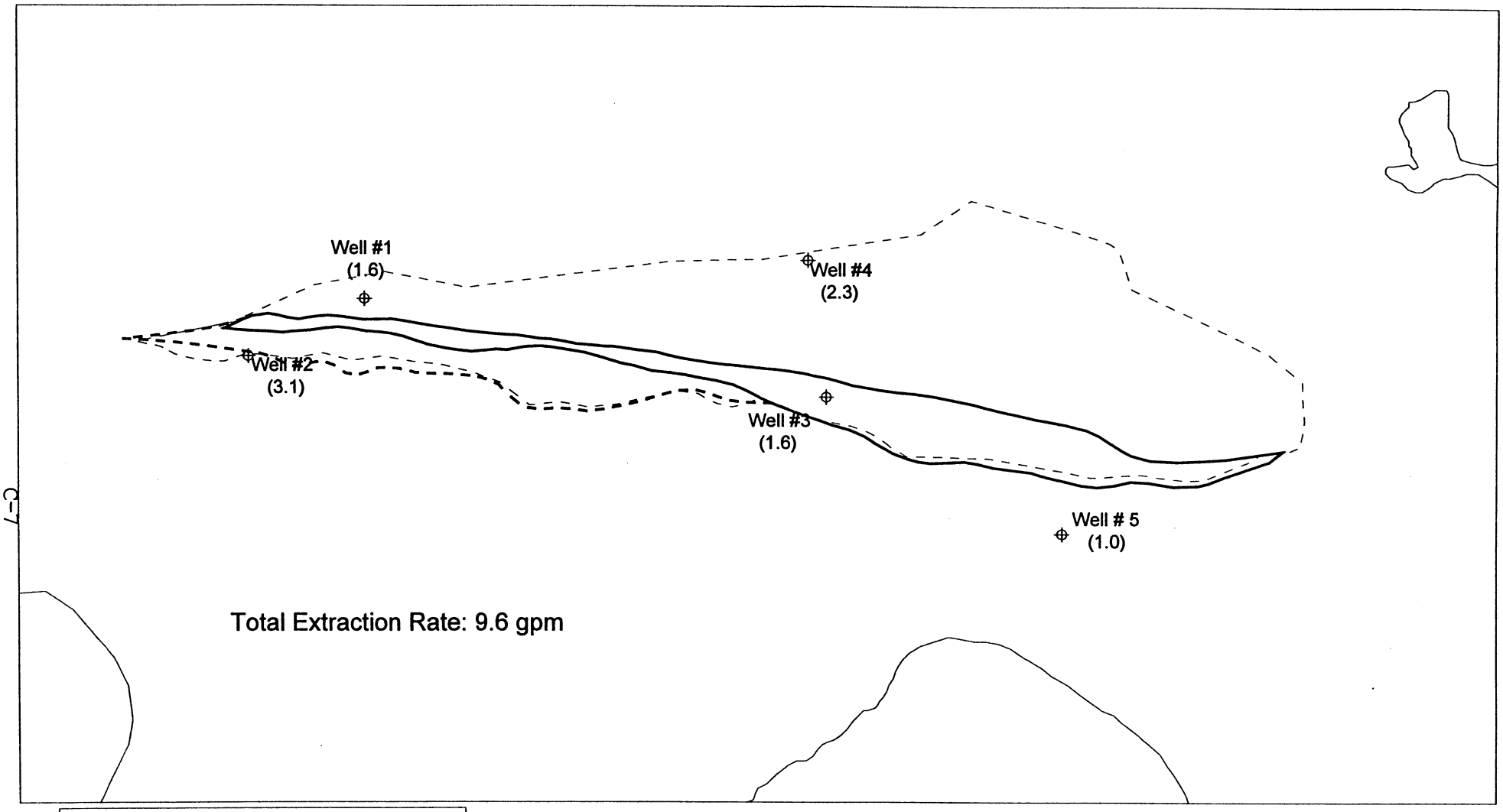
Table 4. Simulation D (BEJ, Upper Bedrock Extraction) well locations and extraction rates (gpm).

Well	1	2	3	4	5	6	7	8	9	Total
Row, Column	56,26	59,24	61,51	54,50	55,44	56,34	65,55	61,39	52,62	
Layer 5	1.30	1.30	1.30	0.78	1.04	0.52	1.04	1.30	0.52	9.1
Layer 6	1.30	1.30	1.30	0.26	1.04	0.52	1.04	1.30	0.52	8.58
Layer 7		1.30		0.26	0.52	0.52	1.04			3.64
Total Rate	**2.60**	**3.90**	**2.60**	**1.30**	**2.60**	**1.56**	**3.12**	**2.60**	**1.04**	**21.3**

Table 5. Simulation E (BEJ, workings) well locations and extraction rates (gpm).

Well	1	2	3	4	5	6	Total
Row, Column	56,29	59,26	61,51	54,50	68,63	55,28	
Layer 5		1.04	0.26			0.52	1.30
Layer 6		1.04	0.26			0.52	1.30
Layer 7		0.52		0.26	0.52	0.52	1.82
Layer 8	1.56	0.52		0.26	0.52		2.86
Layer 9	1.56	0.52	0.52	0.78			3.38
Layer 10		0.52	0.52	1.04			2.08
Total Rate	**3.12**	**4.16**	**1.56**	**2.34**	**1.04**	**1.56**	**13.8**

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C-7

Legend

- Well Location (Extraction rate in gpm)
- Crandon Formation at Subcrop
- Development Boundary

0 750 1500

Scale in Feet

N

HSI GEOTRANS

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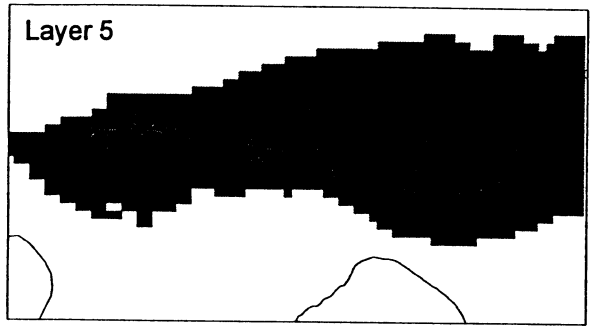
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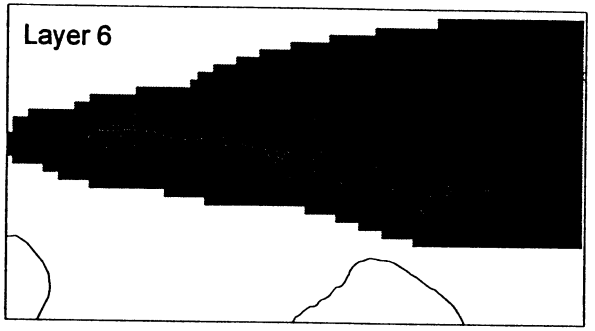
Nicolet Minerals
a Rio Algom company

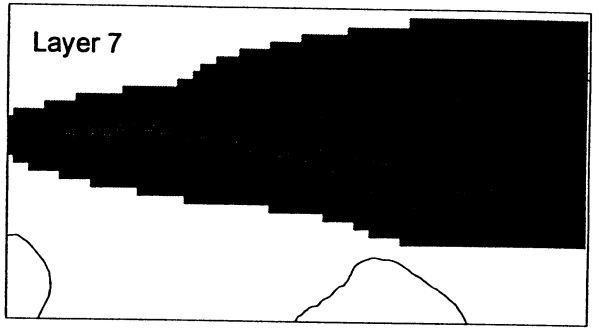
Figure 1 Reflood Containment Well Locations and Extraction Rates Simulation A (BEJ)

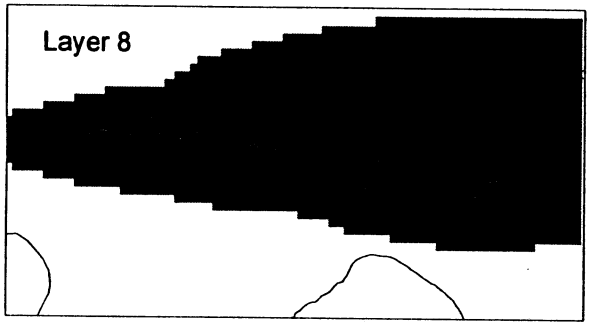
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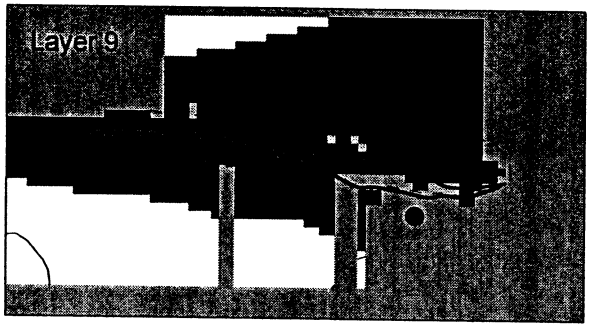
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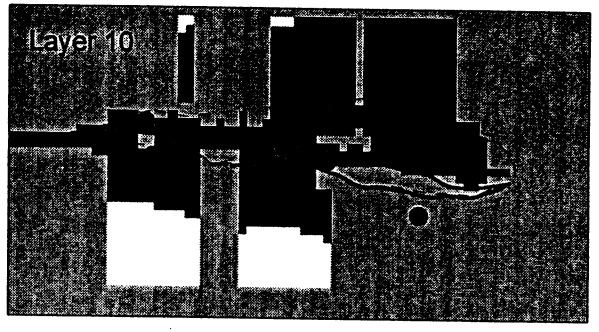


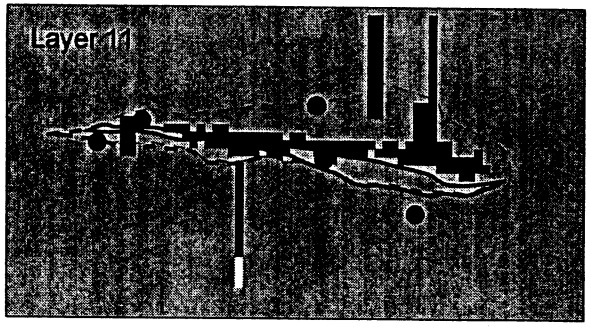


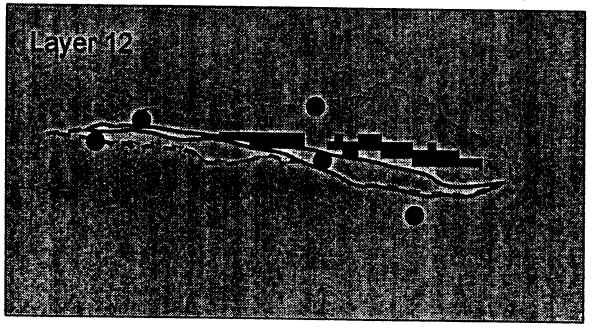


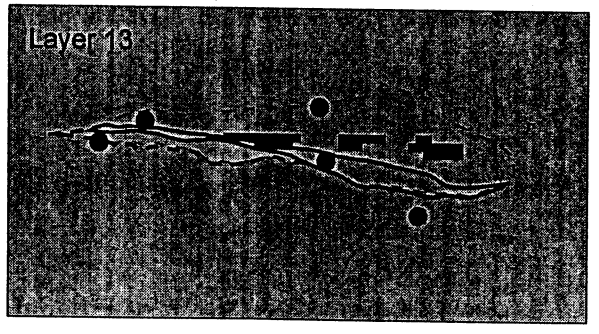












C-8

Legend

- Well Location
- Zone Captured By Wells
- Inactive Zone
- Active Zone Not Captured by Wells
- Develpment Boundary
- Crandon Formation at Subcrop

0 2500 5000

Scale in Feet

N

HSI GEOTRANS

REVISED	DATE	BY	DESCRIPTION

CHECKED BY: GWC	DATE: Apr 2000
APPROVED BY: PFA	DATE: Apr 2000
APPROVE	DATE:

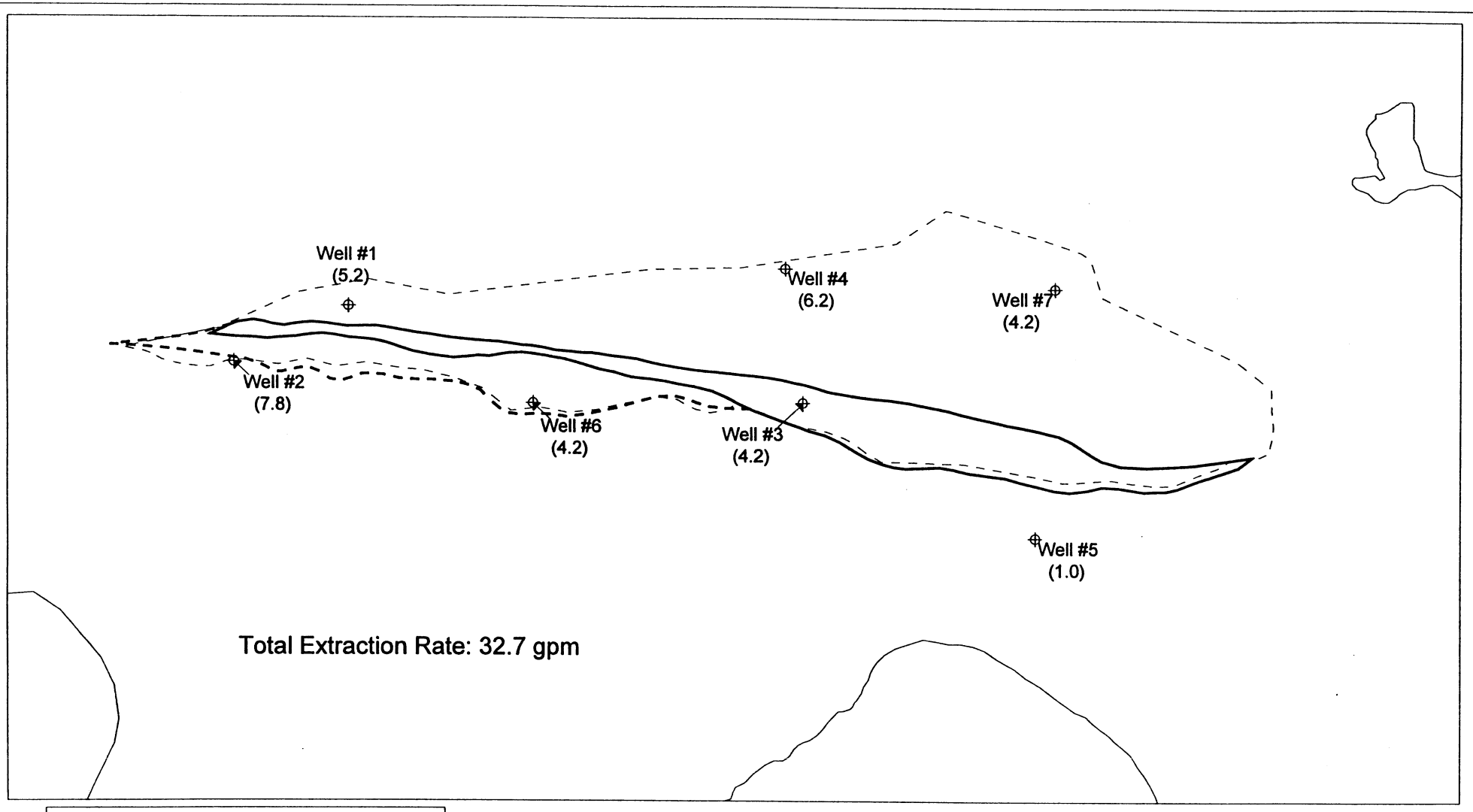
Nicolet Minerals
a Rio Algom company

Figure 2 Reflood Containment Capture Zones Simulation A (BEJ)

Scale: 1 in. = 2500 ft. | Date: April 2000

Prepared By: HSI GeoTra | By: LMG

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C-9

Legend

- Well Location (Extraction rate in gpm)
- Crandon Formation at Subcrop
- Development Boundary

0 750 1500

Scale in Feet

N

HSI GEOTRANS

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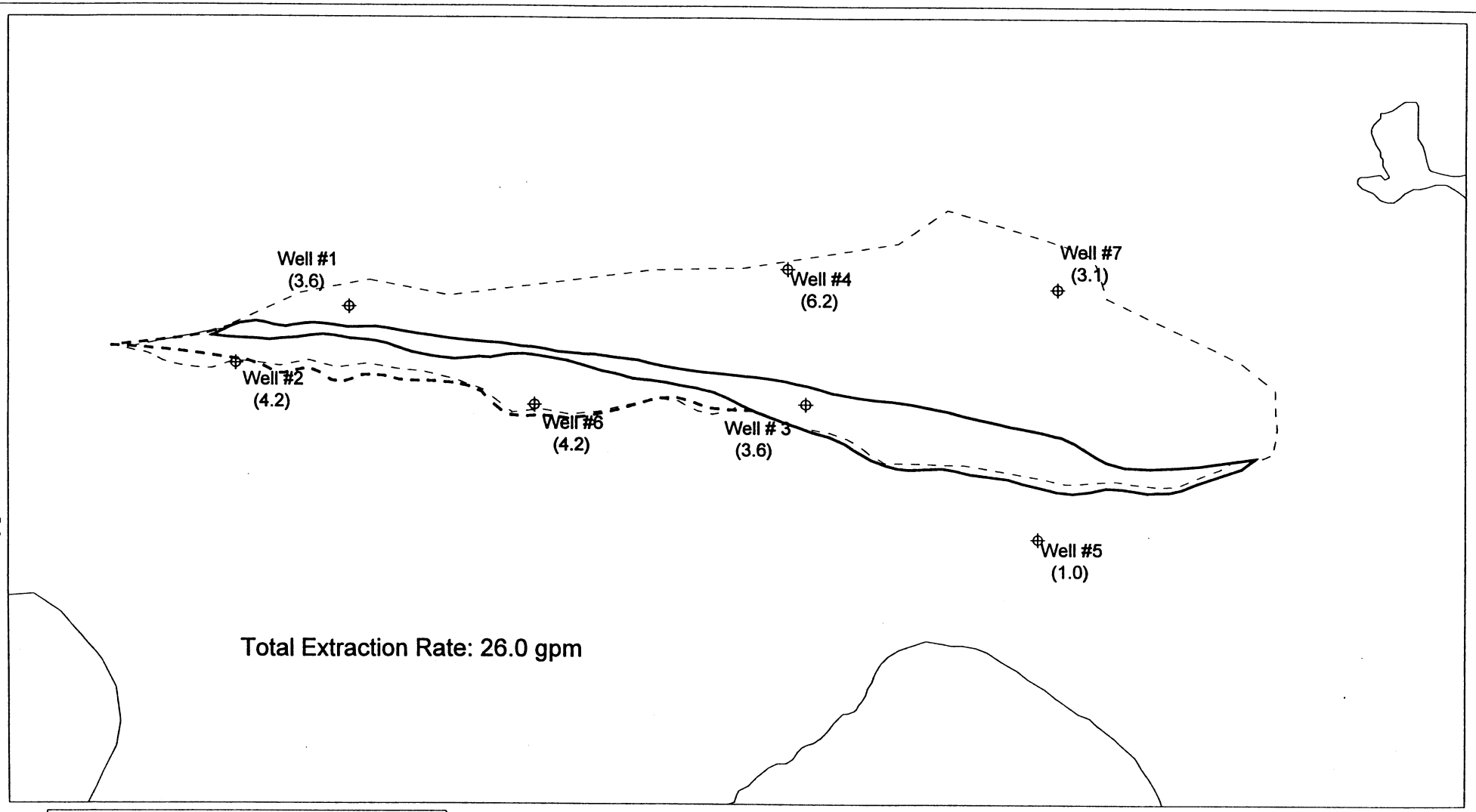
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Figure 3 Reflood Containment Well Locations and Extraction Rates Simulation B (PWC)

Scale: 1 in. = 750 ft.	Date: Feb 2000
Prepared By: HSI GeoTrans	By: LMG

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Legend

- Well Location (Extraction rate in gpm)
- Crandon Formation at Subcrop
- Development Boundary

0 750 1500

Scale in Feet

N

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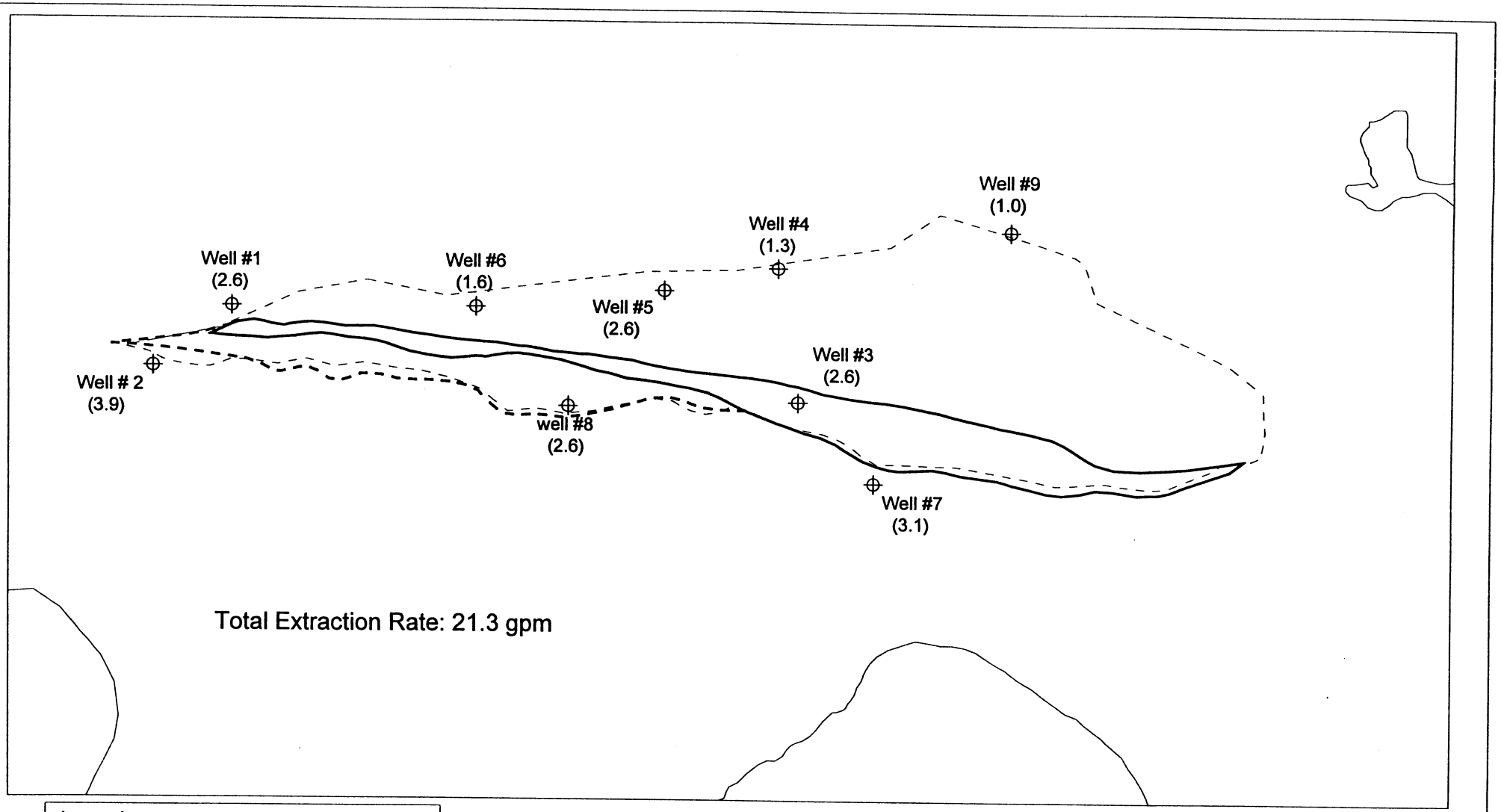
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Figure 4 Reflood Containment Well Locations and Extraction Rates Simulation C (USGS High Range Case)

Scale: 1 in. = 750 ft.	Date: Feb 2000
Prepared By: HSI GeoTra[illegible]	By: LMG

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C-11

Legend

- Well Location (Extraction rate in gpm)
- Crandon Formation at Subcrop
- Development Boundary

0 750 1500

Scale in Feet

N

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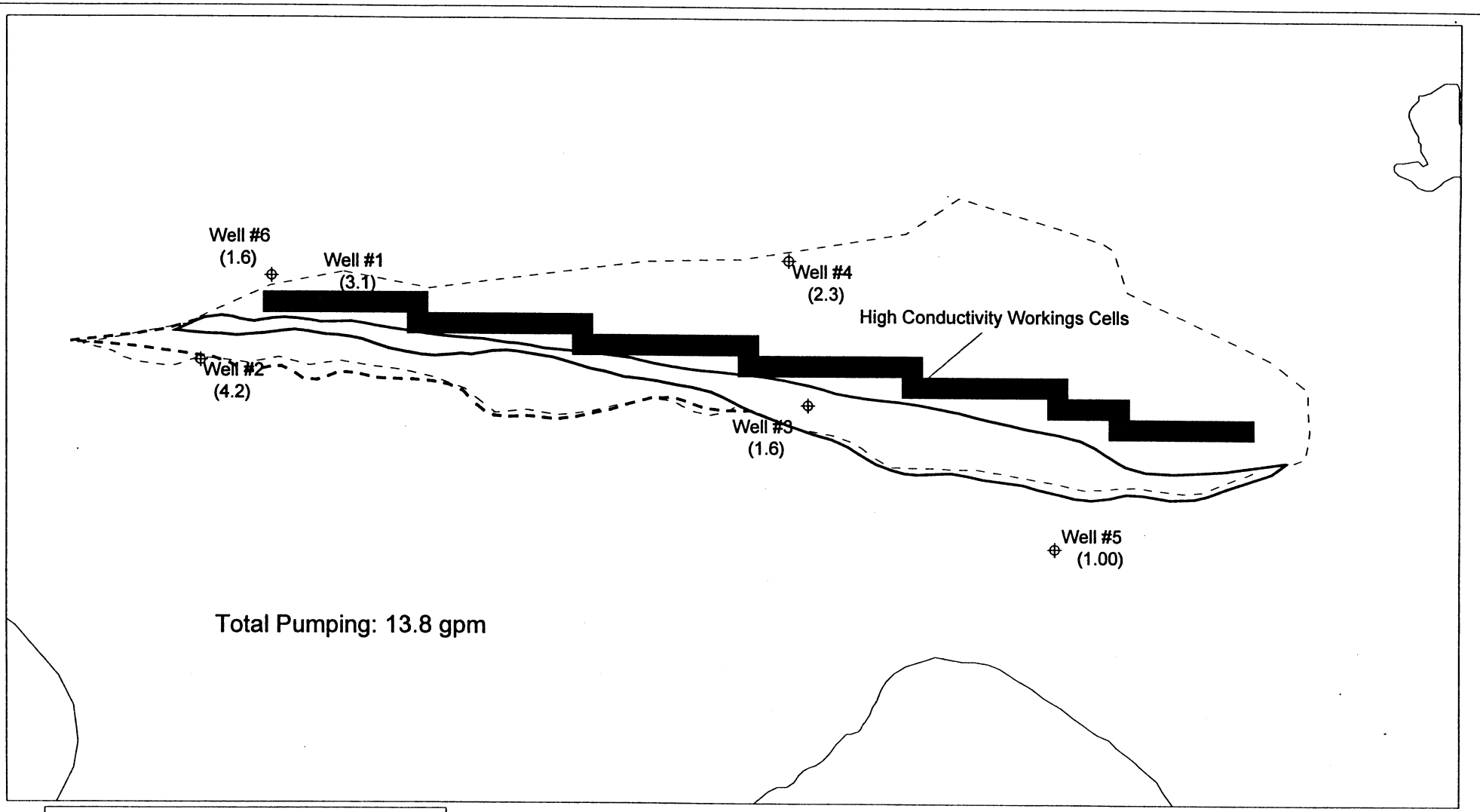
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Figure 5 Reflood Containment Well Locations and Extraction Rates Simulation D (BEJ-Shallow)

Scale: 1 in. = 750 ft.	Date: Feb 2000
Prepared By: HSI GeoTrans	By: LMG

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C-12

Legend

- Well Location (Pump rate in gpm)
- Crandon Formation at Subcrop
- Development Boundary

0 750 1500

Scale in Feet

N

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REVISED	DATE	BY	DESCRIPTION

CHECKED BY: GWC	DATE: Apr 2000
APPROVED BY: PFA	DATE: Apr 2000
APPROVH	DATE:

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Figure 6 Reflood Containment Well Locations and Pump Rates Simulation E (BEJ, Workings)

Scale: 1 in. = 750 ft. | Date: Apr 2000

Prepared By: HSI GeoTran | By: LMG

Appendix D

Estimated Concentration Increases at DMZ Boundary

HSI GEOTRANS
TECHNICAL MEMORANDUM

TO: Steve Donohue, Foth & Van Dyke

CC: Gordon Reid, Nicolet Minerals Company

FROM: Greg Council, HSI GeoTrans

DATE: November 28, 2000

SUBJECT: Crandon Project, Calculation of Concentration Increases at the DMZ Boundary for Individual Constituents of Interest
HSI GeoTrans Project No. N015-211

Introduction

SRK (2000) has provided estimates of constituent concentrations for the reflooded mine source components. HSI GeoTrans (2000) has modeled solute transport from the reflooded mine to the Design Management Zone (DMZ) boundary, giving estimates of DMZ boundary concentration as a fraction of source concentration. In this memorandum, the SRK (2000) analysis and the HSI GeoTrans (2000) analysis are combined to predict maximum concentration increases (above background) at the DMZ boundary for each constituent of interest (COI) for the reflooded mine.

As described in SRK (2000) and HSI GeoTrans (2000), the source is comprised of 5 distinct zones: 1) the unoxidized zone in the backfill, 2) the oxidized-neutral zone in the backfill, 3) the oxidized-acidic zone in the backfill, 4) the access workings wall rock, and 5) the base of the crown pillar. In the analysis presented below, each of these zones is considered, and a maximum total concentration increase at the DMZ boundary is predicted for each COI.

In this analysis, predictions are first made assuming no engineering measures are taken during or following mining to reduce the initial mass of constituents (the "unmanaged case"). Subsequently, the effects of wall rock cleaning and oxygen barriers are applied (the "managed case"). The final result of this analysis is an estimate of DMZ boundary concentration increases for all COIs.

The pertinent calculations for this analysis are carried out in worksheets (numbered 1 through 24), which are attached to this memorandum. Separate worksheets are provided for each COI. The calculations and sources are discussed below.

Source Concentrations and Model Results

Table 1 lists BEJ estimates of source concentration for each COI in each of the five source zones considered (from SRK, 2000). HSI GeoTrans (2000) showed that the unoxidized and oxidized-neutral portions of the backfill can be combined into a single volumetrically-averaged source zone. This averaged unoxidized and oxidized-neutral source concentration is also listed in Table 1. The unoxidized concentration is weighted much more heavily in this calculation because it is a much larger zone. Finally, SRK (2000) determined that barium, iron, lead, nickel, and sulfate concentrations in the unoxidized backfill would likely be controlled by mineral solubility, as noted in Table 1.

HSI GeoTrans (2000) presented model results that predict the maximum relative concentration increase of a constituent at various depths on the DMZ boundary, relative to the concentration of the constituent in each of the source zones (Table 2). For solubility-controlled constituents, a continuous source was simulated in the unoxidized and oxidized-neutral backfill zone. For other constituents, the results of the instantaneous source simulation are applicable. Only instantaneous sources were used for the oxidized-acidic zone, the wall-rock zone, and the crown pillar zone. The instantaneous-source representation assumes that all of the constituent mass is in solution at the beginning of the simulation.

In the attached worksheets, the section labeled "Source Zone Concentrations" lists the source concentrations from Table 1, and the section labeled "Model Results" lists the pertinent relative concentrations, for each source zone in each depth range of interest.

Note that an adjustment is made to the oxidized-acidic zone source concentration on each worksheet (under "Source-Cell") in order to maintain mass conservation. HSI GeoTrans (2000) showed that it is conservative to assume that the mass in the oxidized-acidic portion of the backfill is instantaneously dissolved into the water in the cross cuts just outside the backfill edge. The first access-working cell of each cross cut has dimensions of 1.5 feet by 37.5 feet by 25 feet, with 22% of the volume taken up by open water in the cross cut, giving a water volume of 310 ft^3, or 8,800 L. According to SRK (2000, unmanaged BEJ estimate), the oxidized-acidic zone at each cross cut would have a bulk volume of 3.6 ft^3, and a porosity of 40%, giving a pore water volume of 1.4 ft^3, or 41 L. Since the volume of water in the oxidized-acidic zone is 0.0047 times as large as the water volume in the cells used as the source in the oxidized-acidic zone simulation (HSI GeoTrans, 2000), the oxidized-acidic zone source concentrations are multiplied by 0.0047 before applying the relative concentration factors from the model. This way, the mass of each COI in the oxidized-acidic zone is conserved. For example, since antimony has an estimated oxidized-acidic zone source concentration of 2.1 mg/L (Table 1), and since the water volume of the oxidized-acidic zone at each cross cut is 41 L, the mass of antimony at each cross cut is 86 mg. Under the assumption that this mass is instantaneously dissolved into the 8,800 L of the first access-working cell, that source cell has an initial concentration of 0.0098 mg/L, which is 0.0047 times the reported oxidized-acidic zone concentration of 2.1 mg/L.

Unmanaged Case, Calculation of DMZ Boundary Concentrations

For each COI, the effect of each source zone on the DMZ boundary concentration, absent management controls, is determined by multiplying the unmanaged source concentration (Table 1) by the relative concentrations in the model (Table 2). Those concentration increases due to each source are subsequently added to estimate the total concentration increase due to all sources. These calculations are shown under the heading "Unmanaged Case, Predicted DMZ Concentrations" on the attached worksheets. Note that for convenience it is assumed that all of the sources produce a maximum concentration increase at the same locations on the DMZ at the same future time. This is an accurate assumption when one of the source zones produces much larger increases than do the others. Where two or more source zones produce similar increases, this assumption conservatively overestimates the maximum DMZ boundary concentration increase.

Managed Case, Calculation of DMZ Boundary Concentrations

The Reflooded Mine Management Plan (RMMP) calls for management practices that will reduce the amount of COI mass in the source area. Specifically, for the oxidized-acidic backfill zone, oxygen barriers will be constructed to reduce the amount of oxidized-acidic backfill. Acidic backfill that is present at the end of mining will be removed. In the access workings, localized oxidation of wall rock will be controlled via shotcreting and the wall rock will be thoroughly cleaned prior to reflooding, resulting in a substantial reduction of the COI mass in that zone.

It is estimated that the oxygen barriers in the cross cuts will result in at least a 95% reduction in the size and COI mass of the oxidized-acidic zone. This translates into a 95% reduction in the predicted DMZ boundary concentrations due to the oxidized-acidic backfill zone for each COI. Note that the RMMP also calls for physical removal of the acidified area if necessary. That removal, which would further reduce the DMZ concentrations due to the oxidized-acidic zone, is not reflected in the calculations presented here.

Based on the RMMP, it is assumed that when compared to the unmanaged case, a two-thirds (approximately 65%) reduction in solute loading from the access workings can be achieved. This translates into a 65% reduction in the predicted DMZ boundary concentrations due to the wall rock zone.

On each worksheet, under the heading "Managed Case, Predicted DMZ Concentrations," the managed case concentration increases are calculated. These calculations are the same as the unmanaged case, except that the oxidized-acidic zone concentration increases are reduced by 95% and the wall rock concentration increases are reduced by 65%. Table 3 lists these results for each COI. Note that these calculations are based on BEJ source concentrations.

In assessing the concentration of sulfate at the DMZ, the effect of thiosalts in the backfill are considered (SRK, 2000). Some portion of the thiosalts may convert to sulfate during transport to the DMZ boundary, which would effectively increase the total concentration of sulfate at the DMZ boundary. To be conservative, this analysis assumes that the thiosalts completely convert to sulfate at the DMZ boundary. In the last attached worksheet (number 24), the contribution of the thiosalts is added to the contribution from (unaltered) sulfate to arrive at predictions for "total sulfate." These results are also listed in Table 3.

References

HSI GeoTrans, 2000. *Solute Transport Modeling of the Backfilled Crandon Mine After Reflooding*. November 2000.

SRK, 2000. *Reflooded Mine Source Concentration, Crandon Project*. November 2000.

Table 1. BEJ Estimated Source Concentrations (from SRK, 2000) (mg/L).

COI	Unoxidized Backfill	Oxidized-Neut. Backfill	Oxidized-Acidic Backfill	Wall Rock	Crown Pillar	Average Unox. & Ox.-Neut. Backfill[1]	Solubility Controlled
Antimony	0.0012	0.033	2.1	0.052	0.0033	0.0013	—
Arsenic	0.0053	0.24	650	0.023	3.6E-04	0.0058	—
Barium	0.0057	0.0031	7.1E-04	0.0065	4.4E-04	0.0057	Yes
Beryllium	3.3E-04	0.0077	0.082	0.0043	3.4E-04	3.5E-04	—
Boron	0.0054	ND	0.029	0.0041	ND	0.0054	—
Cadmium	6.5E-05	0.0059	26	0.0044	0.0067	7.8E-05	—
Chloride	251	251	251	2.0	ND	251	—
Chromium	7.0E-05	0.095	21	0.0065	3.4E-04	2.8E-04	—
Cobalt	0.084	0.040	18	0.0075	0.0057	0.084	—
Copper	0.0027	0.22	760	0.022	0.066	0.0032	—
Cyanide	0.010	0.010	ND	ND	ND	0.010	—
Fluoride	ND	1.0	53	0.025	ND	0.0022	—
Iron	0.0080	0.29	59,756	1.1	0.024	0.0086	Yes
Lead	0.0024	0.27	3.1	0.028	0.0034	0.0030	Yes
Manganese	0.0022	0.0053	193	0.068	0.020	0.0022	—
Mercury	8.0E-05	6.0E-04	0.27	2.4E-04	1.4E-05	8.1E-05	—
Nickel	7.2E-05	0.039	3.3	0.026	0.0016	1.6E-04	Yes
Nitrate+Nitrite	21	21	ND	ND	ND	21	—
Selenium	0.031	0.51	7.1	0.0054	0.0017	0.032	—
Silver	2.3E-04	0.021	0.021	0.0084	0.0011	2.8E-04	—
Sulfate	1,250	2,147	164,173	16	8.2	1,252	Yes
Zinc	0.042	0.54	9,322	0.049	1.4	0.043	—
Sulfate from Thiosalts	686	686	686	ND	ND	686	—

Notes

ND Not detected

[1] Unoxidized volume = 2.1E8 ft^3, Oxidized-Neutral Volume = 466,000 ft^3.

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Table 2. Solute Transport Model Results (from HSI GeoTrans, 2000).

Simulation Description	Maximum Relative Concentration at the DMZ Boundary				
	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Continuous	0.028	0.032	0.22	0.26	0.35
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Notes

Relative Concentration is expressed as a fraction of the source concentration.

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Table 3. Estimated Concentration Increases at the DMZ Boundary (mg/L).

COI	Maximum Predicted Concentration Increase				
	Glacial	SWB	MWB	WWB	UWB
Antimony	7.3E-04	6.9E-04	0.0026	0.0023	0.0028
Arsenic	4.0E-04	3.8E-04	0.0018	0.0018	0.0024
Barium	2.5E-04	2.7E-04	0.0016	0.0017	0.0023
Beryllium	6.3E-05	6.0E-05	2.4E-04	2.2E-04	2.8E-04
Boron	1.2E-04	1.2E-04	7.8E-04	9.3E-04	0.0013
Cadmium	9.1E-05	7.9E-05	2.4E-04	2.0E-04	2.3E-04
Chloride	2.9	3.0	27	36	53
Chromium	9.3E-05	8.8E-05	3.4E-04	3.1E-04	3.7E-04
Cobalt	0.0011	0.0011	0.0094	0.012	0.018
Copper	6.6E-04	5.5E-04	0.0016	0.0015	0.0018
Cyanide	1.1E-04	1.2E-04	0.0011	0.0014	0.0021
Fluoride	3.6E-04	3.5E-04	0.0014	0.0013	0.0017
Iron	0.017	0.016	0.061	0.053	0.062
Lead	4.7E-04	4.7E-04	0.0020	0.0019	0.0024
Manganese	0.0010	9.6E-04	0.0036	0.0031	0.0037
Mercury	4.2E-06	4.1E-06	2.0E-05	2.1E-05	2.9E-05
Nickel	3.6E-04	3.4E-04	0.0013	0.0011	0.0013
Nitrate+Nitrite	0.24	0.25	2.2	2.9	4.4
Selenium	4.5E-04	4.6E-04	0.0037	0.0048	0.0070
Silver	1.2E-04	1.1E-04	4.3E-04	3.8E-04	4.6E-04
Sulfate	35	41	279	325	440
Zinc	0.0078	0.0058	0.011	0.010	0.013
Sulfate from Thiosalts	7.8	8.1	74	97	144
Total Sulfate	43	49	352	423	584

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Worksheet 1
Constituent **Antimony**
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0013 [2]	0.0013
Oxidized-Acidic Backfill	2.1	0.0098 [3]
Open Workings Wall Rock	0.052	0.052
Crown Pillar	0.0033	0.0033

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	1.5E-05	1.5E-05	1.4E-04	1.8E-04	2.7E-04
Oxidized-Acidic Backfill	1.2E-06	9.9E-07	4.2E-06	4.0E-06	4.1E-06
Open Workings Wall Rock	0.0020	0.0019	0.0071	0.0060	0.0071
Crown Pillar	1.5E-05	1.0E-05	7.0E-06	2.9E-06	5.6E-07
Total Increase, All Sources	0.0020	0.0019	0.0073	0.0062	0.0074

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	1.5E-05	1.5E-05	1.4E-04	1.8E-04	2.7E-04
Oxidized-Acidic Backfill	5.9E-08	5.0E-08	2.1E-07	2.0E-07	2.1E-07
Open Workings Wall Rock	7.0E-04	6.7E-04	0.0025	0.0021	0.0025
Crown Pillar	1.5E-05	1.0E-05	7.0E-06	2.9E-06	5.6E-07
Total Increase, All Sources	**7.3E-04**	**6.9E-04**	**0.0026**	**0.0023**	**0.0028**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 2
Constituent **Arsenic**
Case **BEJ**

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0058 [2]	0.0058
Oxidized-Acidic Backfill	650	3.0 [3]
Open Workings Wall Rock	0.023	0.023
Crown Pillar	3.6E-04	3.6E-04

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	6.7E-05	6.9E-05	6.2E-04	8.3E-04	0.0012
Oxidized-Acidic Backfill	3.6E-04	3.1E-04	0.0013	0.0012	0.0013
Open Workings Wall Rock	8.8E-04	8.4E-04	0.0031	0.0027	0.0032
Crown Pillar	1.6E-06	1.1E-06	7.6E-07	3.2E-07	6.1E-08
Total Increase, All Sources	0.0013	0.0012	0.0051	0.0047	0.0057

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	6.7E-05	6.9E-05	6.2E-04	8.3E-04	0.0012
Oxidized-Acidic Backfill	1.8E-05	1.5E-05	6.5E-05	6.2E-05	6.4E-05
Open Workings Wall Rock	3.1E-04	2.9E-04	0.0011	9.3E-04	0.0011
Crown Pillar	1.6E-06	1.1E-06	7.6E-07	3.2E-07	6.1E-08
Total Increase, All Sources	**4.0E-04**	**3.8E-04**	**0.0018**	**0.0018**	**0.0024**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.
[2] Not a solubility-controlled constituent, instantaneous-source results apply.
[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 3
Constituent Barium
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0057 [1]	0.0057
Oxidized-Acidic Backfill	7.1E-04	3.3E-06 [3]
Open Workings Wall Rock	0.0065	0.0065
Crown Pillar	4.4E-04	4.4E-04

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Continuous	0.028	0.032	0.22	0.26	0.35
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	1.6E-04	1.8E-04	0.0013	0.0015	0.0020
Oxidized-Acidic Backfill	4.0E-10	3.4E-10	1.4E-09	1.4E-09	1.4E-09
Open Workings Wall Rock	2.5E-04	2.4E-04	8.9E-04	7.5E-04	8.9E-04
Crown Pillar	2.0E-06	1.4E-06	9.3E-07	3.9E-07	7.5E-08
Total Increase, All Sources	4.1E-04	4.2E-04	0.0022	0.0022	0.0029

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	1.6E-04	1.8E-04	0.0013	0.0015	0.0020
Oxidized-Acidic Backfill	2.0E-11	1.7E-11	7.1E-11	6.8E-11	7.0E-11
Open Workings Wall Rock	8.7E-05	8.3E-05	3.1E-04	2.6E-04	3.1E-04
Crown Pillar	2.0E-06	1.4E-06	9.3E-07	3.9E-07	7.5E-08
Total Increase, All Sources	**2.5E-04**	**2.7E-04**	**0.0016**	**0.0017**	**0.0023**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 4
Constituent **Beryllium**
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	3.5E-04 [2]	3.5E-04
Oxidized-Acidic Backfill	0.082	3.8E-04 [3]
Open Workings Wall Rock	0.0043	0.0043
Crown Pillar	3.4E-04	3.4E-04

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	4.0E-06	4.1E-06	3.7E-05	4.9E-05	7.3E-05
Oxidized-Acidic Backfill	4.6E-08	3.9E-08	1.6E-07	1.6E-07	1.6E-07
Open Workings Wall Rock	1.7E-04	1.6E-04	5.9E-04	5.0E-04	5.9E-04
Crown Pillar	1.6E-06	1.1E-06	7.2E-07	3.0E-07	5.8E-08
Total Increase, All Sources	1.7E-04	1.6E-04	6.3E-04	5.5E-04	6.6E-04

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	4.0E-06	4.1E-06	3.7E-05	4.9E-05	7.3E-05
Oxidized-Acidic Backfill	2.3E-09	1.9E-09	8.2E-09	7.8E-09	8.1E-09
Open Workings Wall Rock	5.8E-05	5.5E-05	2.1E-04	1.7E-04	2.1E-04
Crown Pillar	1.6E-06	1.1E-06	7.2E-07	3.0E-07	5.8E-08
Total Increase, All Sources	**6.3E-05**	**6.0E-05**	**2.4E-04**	**2.2E-04**	**2.8E-04**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.
[2] Not a solubility-controlled constituent, instantaneous-source results apply.
[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 5
Constituent Boron
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0054 [2]	0.0054
Oxidized-Acidic Backfill	0.029	1.4E-04 [3]
Open Workings Wall Rock	0.0041	0.0041
Crown Pillar	0	0

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	6.2E-05	6.4E-05	5.8E-04	7.6E-04	0.0011
Oxidized-Acidic Backfill	1.6E-08	1.4E-08	5.8E-08	5.5E-08	5.7E-08
Open Workings Wall Rock	1.6E-04	1.5E-04	5.6E-04	4.7E-04	5.6E-04
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	2.2E-04	2.1E-04	0.0011	0.0012	0.0017

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	6.2E-05	6.4E-05	5.8E-04	7.6E-04	0.0011
Oxidized-Acidic Backfill	8.1E-10	6.9E-10	2.9E-09	2.8E-09	2.9E-09
Open Workings Wall Rock	5.5E-05	5.2E-05	2.0E-04	1.7E-04	2.0E-04
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	**1.2E-04**	**1.2E-04**	**7.8E-04**	**9.3E-04**	**0.0013**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 6
Constituent Cadmium
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	7.8E-05 [2]	7.8E-05
Oxidized-Acidic Backfill	26	0.12 [3]
Open Workings Wall Rock	0.0044	0.0044
Crown Pillar	0.0067	0.0067

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	8.9E-07	9.2E-07	8.4E-06	1.1E-05	1.6E-05
Oxidized-Acidic Backfill	1.4E-05	1.2E-05	5.2E-05	5.0E-05	5.1E-05
Open Workings Wall Rock	1.7E-04	1.6E-04	6.0E-04	5.1E-04	6.0E-04
Crown Pillar	3.1E-05	2.1E-05	1.4E-05	5.9E-06	1.1E-06
Total Increase, All Sources	2.2E-04	2.0E-04	6.8E-04	5.8E-04	6.7E-04

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	8.9E-07	9.2E-07	8.4E-06	1.1E-05	1.6E-05
Oxidized-Acidic Backfill	7.2E-07	6.1E-07	2.6E-06	2.5E-06	2.6E-06
Open Workings Wall Rock	5.9E-05	5.6E-05	2.1E-04	1.8E-04	2.1E-04
Crown Pillar	3.1E-05	2.1E-05	1.4E-05	5.9E-06	1.1E-06
Total Increase, All Sources	**9.1E-05**	**7.9E-05**	**2.4E-04**	**2.0E-04**	**2.3E-04**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 7
Constituent Chloride
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	251 [2]	251
Oxidized-Acidic Backfill	251	1.2 [3]
Open Workings Wall Rock	2.0	2.0
Crown Pillar	0	0

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	2.9	3.0	27	36	53
Oxidized-Acidic Backfill	1.4E-04	1.2E-04	5.0E-04	4.8E-04	4.9E-04
Open Workings Wall Rock	0.077	0.073	0.27	0.23	0.27
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	2.9	3.1	27	36	53

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	2.9	3.0	27	36	53
Oxidized-Acidic Backfill	7.0E-06	5.9E-06	2.5E-05	2.4E-05	2.5E-05
Open Workings Wall Rock	0.027	0.026	0.096	0.081	0.096
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	**2.9**	**3.0**	**27**	**36**	**53**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 8
Constituent Chromium
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	2.8E-04 [2]	2.8E-04
Oxidized-Acidic Backfill	21	0.098 [3]
Open Workings Wall Rock	0.0065	0.0065
Crown Pillar	3.4E-04	3.4E-04

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	3.2E-06	3.3E-06	3.0E-05	3.9E-05	5.9E-05
Oxidized-Acidic Backfill	1.2E-05	9.9E-06	4.2E-05	4.0E-05	4.1E-05
Open Workings Wall Rock	2.5E-04	2.4E-04	8.9E-04	7.5E-04	8.9E-04
Crown Pillar	1.6E-06	1.1E-06	7.2E-07	3.0E-07	5.8E-08
Total Increase, All Sources	2.7E-04	2.5E-04	9.6E-04	8.3E-04	9.9E-04

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	3.2E-06	3.3E-06	3.0E-05	3.9E-05	5.9E-05
Oxidized-Acidic Backfill	5.9E-07	5.0E-07	2.1E-06	2.0E-06	2.1E-06
Open Workings Wall Rock	8.7E-05	8.3E-05	3.1E-04	2.6E-04	3.1E-04
Crown Pillar	1.6E-06	1.1E-06	7.2E-07	3.0E-07	5.8E-08
Total Increase, All Sources	**9.3E-05**	**8.8E-05**	**3.4E-04**	**3.1E-04**	**3.7E-04**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 9
Constituent Cobalt
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.084 [2]	0.084
Oxidized-Acidic Backfill	18	0.084 [3]
Open Workings Wall Rock	0.0075	0.0075
Crown Pillar	0.0057	0.0057

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	9.6E-04	1.0E-03	0.0090	0.012	0.018
Oxidized-Acidic Backfill	1.0E-05	8.5E-06	3.6E-05	3.4E-05	3.5E-05
Open Workings Wall Rock	2.9E-04	2.7E-04	0.0010	8.7E-04	0.0010
Crown Pillar	2.6E-05	1.8E-05	1.2E-05	5.0E-06	9.7E-07
Total Increase, All Sources	0.0013	0.0013	0.010	0.013	0.019

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	9.6E-04	1.0E-03	0.0090	0.012	0.018
Oxidized-Acidic Backfill	5.0E-07	4.3E-07	1.8E-06	1.7E-06	1.8E-06
Open Workings Wall Rock	1.0E-04	9.6E-05	3.6E-04	3.0E-04	3.6E-04
Crown Pillar	2.6E-05	1.8E-05	1.2E-05	5.0E-06	9.7E-07
Total Increase, All Sources	**0.0011**	**0.0011**	**0.0094**	**0.012**	**0.018**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 10
Constituent Copper
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0032 [2]	0.0032
Oxidized-Acidic Backfill	760	3.5 [3]
Open Workings Wall Rock	0.022	0.022
Crown Pillar	0.066	0.066

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	3.6E-05	3.8E-05	3.4E-04	4.5E-04	6.7E-04
Oxidized-Acidic Backfill	4.2E-04	3.6E-04	0.0015	0.0015	0.0015
Open Workings Wall Rock	8.4E-04	8.0E-04	0.0030	0.0025	0.0030
Crown Pillar	3.0E-04	2.1E-04	1.4E-04	5.8E-05	1.1E-05
Total Increase, All Sources	0.0016	0.0014	0.0050	0.0045	0.0052

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	3.6E-05	3.8E-05	3.4E-04	4.5E-04	6.7E-04
Oxidized-Acidic Backfill	2.1E-05	1.8E-05	7.6E-05	7.3E-05	7.5E-05
Open Workings Wall Rock	3.0E-04	2.8E-04	0.0011	8.9E-04	0.0011
Crown Pillar	3.0E-04	2.1E-04	1.4E-04	5.8E-05	1.1E-05
Total Increase, All Sources	**6.6E-04**	**5.5E-04**	**0.0016**	**0.0015**	**0.0018**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 11
Constituent Cyanide
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.010 [2]	0.010
Oxidized-Acidic Backfill	0	0 [3]
Open Workings Wall Rock	0	0
Crown Pillar	0	0

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	1.1E-04	1.2E-04	0.0011	0.0014	0.0021
Oxidized-Acidic Backfill	0	0	0	0	0
Open Workings Wall Rock	0	0	0	0	0
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	1.1E-04	1.2E-04	0.0011	0.0014	0.0021

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	1.1E-04	1.2E-04	0.0011	0.0014	0.0021
Oxidized-Acidic Backfill	0	0	0	0	0
Open Workings Wall Rock	0	0	0	0	0
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	**1.1E-04**	**1.2E-04**	**0.0011**	**0.0014**	**0.0021**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.
[2] Not a solubility-controlled constituent, instantaneous-source results apply.
[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 12
Constituent Fluoride
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0022 [2]	0.0022
Oxidized-Acidic Backfill	53	0.25 [3]
Open Workings Wall Rock	0.025	0.025
Crown Pillar	0	0

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	2.5E-05	2.6E-05	2.4E-04	3.1E-04	4.6E-04
Oxidized-Acidic Backfill	3.0E-05	2.5E-05	1.1E-04	1.0E-04	1.0E-04
Open Workings Wall Rock	9.6E-04	9.1E-04	0.0034	0.0029	0.0034
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	0.0010	9.7E-04	0.0038	0.0033	0.0040

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	2.5E-05	2.6E-05	2.4E-04	3.1E-04	4.6E-04
Oxidized-Acidic Backfill	1.5E-06	1.3E-06	5.3E-06	5.1E-06	5.2E-06
Open Workings Wall Rock	3.4E-04	3.2E-04	0.0012	0.0010	0.0012
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	**3.6E-04**	**3.5E-04**	**0.0014**	**0.0013**	**0.0017**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 13
Constituent Iron
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0086 [1]	0.0086
Oxidized-Acidic Backfill	59,756	279 [3]
Open Workings Wall Rock	1.1	1.1
Crown Pillar	0.024	0.024

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Continuous	0.028	0.032	0.22	0.26	0.35
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	2.4E-04	2.8E-04	0.0019	0.0022	0.0030
Oxidized-Acidic Backfill	0.033	0.028	0.12	0.11	0.12
Open Workings Wall Rock	0.042	0.040	0.15	0.13	0.15
Crown Pillar	1.1E-04	7.6E-05	5.1E-05	2.1E-05	4.1E-06
Total Increase, All Sources	0.076	0.069	0.27	0.24	0.27

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	2.4E-04	2.8E-04	0.0019	0.0022	0.0030
Oxidized-Acidic Backfill	0.0017	0.0014	0.0060	0.0057	0.0059
Open Workings Wall Rock	0.015	0.014	0.053	0.045	0.053
Crown Pillar	1.1E-04	7.6E-05	5.1E-05	2.1E-05	4.1E-06
Total Increase, All Sources	**0.017**	**0.016**	**0.061**	**0.053**	**0.062**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 14
Constituent Lead
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0030 [1]	0.0030
Oxidized-Acidic Backfill	3.1	0.014 [3]
Open Workings Wall Rock	0.028	0.028
Crown Pillar	0.0034	0.0034

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Continuous	0.028	0.032	0.22	0.26	0.35
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	8.3E-05	9.7E-05	6.6E-04	7.7E-04	0.0010
Oxidized-Acidic Backfill	1.7E-06	1.5E-06	6.2E-06	5.9E-06	6.1E-06
Open Workings Wall Rock	0.0011	0.0010	0.0038	0.0032	0.0038
Crown Pillar	1.6E-05	1.1E-05	7.2E-06	3.0E-06	5.8E-07
Total Increase, All Sources	0.0012	0.0011	0.0045	0.0040	0.0049

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	8.3E-05	9.7E-05	6.6E-04	7.7E-04	0.0010
Oxidized-Acidic Backfill	8.6E-08	7.3E-08	3.1E-07	3.0E-07	3.1E-07
Open Workings Wall Rock	3.8E-04	3.6E-04	0.0013	0.0011	0.0013
Crown Pillar	1.6E-05	1.1E-05	7.2E-06	3.0E-06	5.8E-07
Total Increase, All Sources	**4.7E-04**	**4.7E-04**	**0.0020**	**0.0019**	**0.0024**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 15
Constituent Manganese
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.0022 [2]	0.0022
Oxidized-Acidic Backfill	193	0.90 [3]
Open Workings Wall Rock	0.068	0.068
Crown Pillar	0.020	0.020

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	2.5E-05	2.6E-05	2.4E-04	3.1E-04	4.6E-04
Oxidized-Acidic Backfill	1.1E-04	9.1E-05	3.9E-04	3.7E-04	3.8E-04
Open Workings Wall Rock	0.0026	0.0025	0.0093	0.0079	0.0093
Crown Pillar	9.2E-05	6.4E-05	4.2E-05	1.8E-05	3.4E-06
Total Increase, All Sources	0.0028	0.0027	0.0100	0.0086	0.010

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	2.5E-05	2.6E-05	2.4E-04	3.1E-04	4.6E-04
Oxidized-Acidic Backfill	5.4E-06	4.6E-06	1.9E-05	1.8E-05	1.9E-05
Open Workings Wall Rock	9.1E-04	8.7E-04	0.0033	0.0028	0.0033
Crown Pillar	9.2E-05	6.4E-05	4.2E-05	1.8E-05	3.4E-06
Total Increase, All Sources	**0.0010**	**9.6E-04**	**0.0036**	**0.0031**	**0.0037**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 16
Constituent **Mercury**
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	8.1E-05 [2]	8.1E-05
Oxidized-Acidic Backfill	0.27	0.0013 [3]
Open Workings Wall Rock	2.4E-04	2.4E-04
Crown Pillar	1.4E-05	1.4E-05

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	9.3E-07	9.6E-07	8.7E-06	1.2E-05	1.7E-05
Oxidized-Acidic Backfill	1.5E-07	1.3E-07	5.4E-07	5.2E-07	5.3E-07
Open Workings Wall Rock	9.2E-06	8.8E-06	3.3E-05	2.8E-05	3.3E-05
Crown Pillar	6.4E-08	4.5E-08	3.0E-08	1.2E-08	2.4E-09
Total Increase, All Sources	1.0E-05	9.9E-06	4.2E-05	4.0E-05	5.0E-05

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	9.3E-07	9.6E-07	8.7E-06	1.2E-05	1.7E-05
Oxidized-Acidic Backfill	7.5E-09	6.4E-09	2.7E-08	2.6E-08	2.7E-08
Open Workings Wall Rock	3.2E-06	3.1E-06	1.1E-05	9.7E-06	1.2E-05
Crown Pillar	6.4E-08	4.5E-08	3.0E-08	1.2E-08	2.4E-09
Total Increase, All Sources	**4.2E-06**	**4.1E-06**	**2.0E-05**	**2.1E-05**	**2.9E-05**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 17
Constituent Nickel
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	1.6E-04 [1]	1.6E-04
Oxidized-Acidic Backfill	3.3	0.015 [3]
Open Workings Wall Rock	0.026	0.026
Crown Pillar	0.0016	0.0016

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Continuous	0.028	0.032	0.22	0.26	0.35
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	4.4E-06	5.1E-06	3.5E-05	4.1E-05	5.5E-05
Oxidized-Acidic Backfill	1.8E-06	1.6E-06	6.6E-06	6.3E-06	6.5E-06
Open Workings Wall Rock	1.0E-03	9.5E-04	0.0036	0.0030	0.0036
Crown Pillar	7.3E-06	5.1E-06	3.4E-06	1.4E-06	2.7E-07
Total Increase, All Sources	0.0010	9.6E-04	0.0036	0.0031	0.0036

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	4.4E-06	5.1E-06	3.5E-05	4.1E-05	5.5E-05
Oxidized-Acidic Backfill	9.2E-08	7.8E-08	3.3E-07	3.2E-07	3.3E-07
Open Workings Wall Rock	3.5E-04	3.3E-04	0.0012	0.0011	0.0012
Crown Pillar	7.3E-06	5.1E-06	3.4E-06	1.4E-06	2.7E-07
Total Increase, All Sources	**3.6E-04**	**3.4E-04**	**0.0013**	**0.0011**	**0.0013**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 18
Constituent Nitrate+Nitrite
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	21 [2]	21
Oxidized-Acidic Backfill	0	0 [3]
Open Workings Wall Rock	0	0
Crown Pillar	0	0

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	0.24	0.25	2.2	2.9	4.4
Oxidized-Acidic Backfill	0	0	0	0	0
Open Workings Wall Rock	0	0	0	0	0
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	0.24	0.25	2.2	2.9	4.4

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	0.24	0.25	2.2	2.9	4.4
Oxidized-Acidic Backfill	0	0	0	0	0
Open Workings Wall Rock	0	0	0	0	0
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	**0.24**	**0.25**	**2.2**	**2.9**	**4.4**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.
[2] Not a solubility-controlled constituent, instantaneous-source results apply.
[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 19
Constituent Selenium
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.032 [2]	0.032
Oxidized-Acidic Backfill	7.1	0.033 [3]
Open Workings Wall Rock	0.0054	0.0054
Crown Pillar	0.0017	0.0017

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	3.7E-04	3.8E-04	0.0034	0.0045	0.0067
Oxidized-Acidic Backfill	4.0E-06	3.4E-06	1.4E-05	1.4E-05	1.4E-05
Open Workings Wall Rock	2.1E-04	2.0E-04	7.4E-04	6.3E-04	7.4E-04
Crown Pillar	7.8E-06	5.4E-06	3.6E-06	1.5E-06	2.9E-07
Total Increase, All Sources	5.9E-04	5.9E-04	0.0042	0.0052	0.0075

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	3.7E-04	3.8E-04	0.0034	0.0045	0.0067
Oxidized-Acidic Backfill	2.0E-07	1.7E-07	7.1E-07	6.8E-07	7.0E-07
Open Workings Wall Rock	7.3E-05	6.9E-05	2.6E-04	2.2E-04	2.6E-04
Crown Pillar	7.8E-06	5.4E-06	3.6E-06	1.5E-06	2.9E-07
Total Increase, All Sources	**4.5E-04**	**4.6E-04**	**0.0037**	**0.0048**	**0.0070**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 20
Constituent Silver
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	2.8E-04 [2]	2.8E-04
Oxidized-Acidic Backfill	0.021	9.8E-05 [3]
Open Workings Wall Rock	0.0084	0.0084
Crown Pillar	0.0011	0.0011

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	3.2E-06	3.3E-06	3.0E-05	3.9E-05	5.8E-05
Oxidized-Acidic Backfill	1.2E-08	9.9E-09	4.2E-08	4.0E-08	4.1E-08
Open Workings Wall Rock	3.2E-04	3.1E-04	0.0011	9.7E-04	0.0012
Crown Pillar	5.0E-06	3.5E-06	2.3E-06	9.7E-07	1.9E-07
Total Increase, All Sources	3.3E-04	3.1E-04	0.0012	0.0010	0.0012

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	3.2E-06	3.3E-06	3.0E-05	3.9E-05	5.8E-05
Oxidized-Acidic Backfill	5.9E-10	5.0E-10	2.1E-09	2.0E-09	2.1E-09
Open Workings Wall Rock	1.1E-04	1.1E-04	4.0E-04	3.4E-04	4.0E-04
Crown Pillar	5.0E-06	3.5E-06	2.3E-06	9.7E-07	1.9E-07
Total Increase, All Sources	**1.2E-04**	**1.1E-04**	**4.3E-04**	**3.8E-04**	**4.6E-04**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 21
Constituent **Sulfate**
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	1,252 [1]	1,252
Oxidized-Acidic Backfill	164,173	766 [3]
Open Workings Wall Rock	16	16
Crown Pillar	8.2	8.2

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Continuous	0.028	0.032	0.22	0.26	0.35
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	35	41	278	325	439
Oxidized-Acidic Backfill	0.092	0.078	0.33	0.31	0.32
Open Workings Wall Rock	0.61	0.59	2.2	1.9	2.2
Crown Pillar	0.038	0.026	0.017	0.0072	0.0014
Total Increase, All Sources	35	41	280	327	442

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	35	41	278	325	439
Oxidized-Acidic Backfill	0.0046	0.0039	0.016	0.016	0.016
Open Workings Wall Rock	0.22	0.20	0.77	0.65	0.77
Crown Pillar	0.038	0.026	0.017	0.0072	0.0014
Total Increase, All Sources	**35**	**41**	**279**	**325**	**440**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 22
Constituent Zinc
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	0.043 [2]	0.043
Oxidized-Acidic Backfill	9,322	43 [3]
Open Workings Wall Rock	0.049	0.049
Crown Pillar	1.4	1.4

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	4.9E-04	5.1E-04	0.0046	0.0061	0.0091
Oxidized-Acidic Backfill	0.0052	0.0044	0.019	0.018	0.018
Open Workings Wall Rock	0.0019	0.0018	0.0067	0.0057	0.0067
Crown Pillar	0.0064	0.0045	0.0030	0.0012	2.4E-04
Total Increase, All Sources	0.014	0.011	0.033	0.031	0.034

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	4.9E-04	5.1E-04	0.0046	0.0061	0.0091
Oxidized-Acidic Backfill	2.6E-04	2.2E-04	9.3E-04	8.9E-04	9.2E-04
Open Workings Wall Rock	6.6E-04	6.3E-04	0.0023	0.0020	0.0023
Crown Pillar	0.0064	0.0045	0.0030	0.0012	2.4E-04
Total Increase, All Sources	**0.0078**	**0.0058**	**0.011**	**0.010**	**0.013**

Notes

1 Solubility-controlled constituent, continuous-source results apply.

2 Not a solubility-controlled constituent, instantaneous-source results apply.

3 Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet 23
Constituent **Sulfate from Thiosalts**
Case BEJ

Source Zone Concentrations (mg/L)

Source Zone	From SRK	Source-Cell
Unoxidized/Oxidized-Neutral Backfill	686 [2]	686
Oxidized-Acidic Backfill	686	3.2 [3]
Open Workings Wall Rock	0	0
Crown Pillar	0	0

Model Results: Maximum Relative Concentrations on the DMZ Boundary

Simulation Description	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral, Instantaneous	0.011	0.012	0.11	0.14	0.21
Oxidized-Acidic Backfill (Cross-Cut Source Cells)	1.2E-04	1.0E-04	4.3E-04	4.1E-04	4.2E-04
Open Workings Wall Rock	0.038	0.037	0.14	0.12	0.14
Crown Pillar	0.0046	0.0032	0.0021	8.8E-04	1.7E-04

Unmanaged Case, Predicted DMZ Concentrations (mg/L) (Source-cell conc. times relative conc.)

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	7.8	8.1	74	97	144
Oxidized-Acidic Backfill	3.8E-04	3.2E-04	0.0014	0.0013	0.0014
Open Workings Wall Rock	0	0	0	0	0
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	7.8	8.1	74	97	144

Managed Case, Removal Efficiency

Source Zone	Efficiency	Measures Taken
Unoxidized/Oxidized-Neutral Backfill	0%	
Oxidized-Acidic Backfill	95%	Oxygen Barriers
Open Workings Wall Rock	65%	Cleaning
Crown Pillar	0%	

Managed Case, Predicted DMZ Concentrations (mg/L) (Unmanaged conc. times (1 - removal efficiency))

Source Zone	Glacial	SWB	MWB	WWB	UWB
Unoxidized/Oxidized-Neutral Backfill	7.8	8.1	74	97	144
Oxidized-Acidic Backfill	1.9E-05	1.6E-05	6.9E-05	6.6E-05	6.8E-05
Open Workings Wall Rock	0	0	0	0	0
Crown Pillar	0	0	0	0	0
Total Increase, All Sources	**7.8**	**8.1**	**74**	**97**	**144**

Notes

[1] Solubility-controlled constituent, continuous-source results apply.

[2] Not a solubility-controlled constituent, instantaneous-source results apply.

[3] Adjusted by a factor of 0.0047 to account for different water volume in acidic zone and model source cells.

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Worksheet **24**

Constituent **Total Sulfate**

Case BEJ

Managed Case, Predicted DMZ Concentrations (mg/L)

	Glacial	SWB	MWB	WWB	UWB
Increase Due to Sulfate Source	35	41	279	325	440
Increase Due to Thiosalt Source	7.8	8.1	74	97	144
Total Increase	**43**	**49**	**352**	**423**	**584**

Setup/Checked: GWC

Checked/Approved: PFA

Date: Nov 2000

Appendix E

Assessment of Bedrock Baseline Groundwater Quality and ACL

Foth & Van Dyke
Memorandum

November 20, 2000

TO: Steve Donohue, Foth & Van Dyke

CC: Gordon Reid, Nicolet Minerals Company
Master File

FR: Steve Lehrke, Foth & Van Dyke SL

RE: Crandon Project - Weathered Bedrock Groundwater Baseline Concentrations and PAL/ACL Calculations

Between November 1 and 16, 1994, a pump test was conducted to assess the hydraulic characteristics of the weathered bedrock and saprolite/glacial till at the Crandon site. The test well was screened in the weathered bedrock. During the test, groundwater quality samples were collected from the test well on seven distinct dates. Data from these sampling events were presented in the report titled *Investigation and Evaluation: Saprolite at the Crandon Site* (Foth & Van Dyke, 1995). Samples collected from the pump test well were sent to PACE and Northern Lake Services, Inc. (NLS) laboratories. The samples analyzed by NLS were collected on November 1 (two samplings), 2 (two samplings) 3, 4, and 10. The samples analyzed by PACE were collected on November 2, 9, and 16.

In this memo, the bedrock groundwater quality data collected between November 1 and 16, 1994 is used to construct baseline groundwater concentrations as well as preventive action limits (PALs) and alternative concentration limits (ACLs). Since the bedrock data is limited to a small number of samples from a single test well, it is first statistically compared to the background data collected from the overburden aquifer in the proposed mine area, to determine for what parameters the overburden data can be used as a surrogate. Using the overburden results for calculation of baseline concentrations and PALs/ACLs would be advantageous given the increased sample sizes, (i.e., a total of eight background samples collected from each of six wells). Baseline concentrations, PALs and ACLs of the overburden are reported in a Foth & Van Dyke memo concerning the tailings management area and reflooded mine area ACLs contained in Appendix F of Foth & Van Dyke, 1999/2000.

For the purpose of this analysis, the samples collected on November 2, 9, and 16 (analyzed by PACE), along with the samples collected on November 1, 3, 4, and 10 (analyzed by NLS), will be combined to create the background groundwater data set for the bedrock aquifer. This data set then consists of seven sampling events. Note that the two samples collected on November 1 and analyzed by NLS were too close together to be considered independent, and therefore were averaged to represent a single sample. Also, the PACE analytical data collected on November 2 was included in the analysis rather than the November 2 NLS analytical data.

If no significant difference exists between the bedrock and overburden data sets, or if the bedrock parameter concentrations are statistically less than in the overburden, the overburden data will be used as a surrogate to construct baseline concentrations and PALs/ACLs. If the bedrock data is statistically greater than in the overburden, baseline concentrations and PALs/ACLs will be constructed based on the available bedrock results. The bedrock and overburden sample results are included in Attachment 1.

The bedrock and overburden results of each parameter were statistically compared utilizing two methods. The first method was the nonparametric Mann-Whitney U test. This test compares the overall ranks of both data sets, testing for significant differences between the two. The second method was to compare each result of the bedrock data to the upper 80th percentile of the overburden data of the corresponding parameter. If a large number of bedrock results exceeded the overburden's 80th percentile, evidence would exist that concentrations between the two aquifers significantly differ. The calculations and results of these tests are given in Attachment 2.

The results of these two methods for determining concentration differences between the two aquifers matched exactly. The parameters illustrating no significant difference were aluminum, chloride, chromium, cobalt, copper, magnesium, manganese, mercury, molybdenum, nickel, silver, and sulfate. Parameters in which bedrock concentrations statistically exceeded overburden concentrations were antimony, arsenic, cadmium, iron, lead, selenium, and zinc. Barium was the only parameter for which bedrock concentrations were statistically less than overburden concentrations.

Based on the results of the statistical tests, the appropriate data set was selected for baseline and ACL calculations. Baseline values and PALs/ACLs for each parameter, along with the data set used (bedrock or overburden), are given in Table 1.

The average baseline concentration for each parameter was conservatively calculated as the upper 95% confidence limit of the mean (or median) for each parameter. Calculation of the upper 95% confidence limits followed the guidance given in USEPA (1989) and USEPA (1992). In the case of the bedrock data, all parameters have sample sizes considered to be small (less than 30). Therefore, normality tests were first performed with the Shapiro-Wilk's W test to determine if an upper confidence limit based on the Student's t distribution was appropriate. In the case of the overburden data, all sample sizes are large, and therefore an upper confidence limit based on the normal distribution can be used regardless, unless the number of non-detects is too high.

The method for adjusting the mean and standard deviation based on the percentage of non-detects is also discussed in USEPA (1989) and USEPA (1992). This criteria is given below:

- If at least 85% of the baseline data are above detection, all non-detects are replaced with one-half the detection limit and an upper confidence limit based on the normal distribution is calculated.

- If at least 50% but less than 85% of the baseline data are above the detection limit, the mean and standard deviation are adjusted using Cohen's adjustment and an upper confidence limit based on the normal distribution is calculated.

- If less than 50% of the baseline data are above the detection limit, a non-parametric upper confidence limit on the median is used.

Normality tests for the bedrock data, along with the upper 95% confidence limit calculations, are given in Attachment 3.

PAL and ACL calculations contained in Attachment 3 are performed based on the 1994 WDNR guidance document "PAL/ACL Calculations Guidance for Solid Waste Facilities." According to the guidance document, if one NR 140 ES or two NR 140 PAL exceedances are observed in the baseline data, or if the average of the baseline data exceeds the NR 140 PAL, an ACL is calculated for that parameter. The ACL is calculated as the average baseline groundwater quality plus two standard deviations. PALs are constructed for indicator parameters as the average baseline quality plus three standard deviations. All PALs and ACLs are "rounded up" to two significant digits, and all non-detects are replaced with one-half the detection limit.

The calculated PALs and ACLs are presented in Table 1. Similar to the baseline concentrations, the data set which was used in the construction of the PAL/ACL are again indicated. ACLs using the overburden data were constructed for sulfate, silver and mercury. ACLs using the bedrock data were constructed for arsenic, iron, selenium, antimony, cadmium and lead.

References

Foth & Van Dyke, 1995. *Investigation and Evaluation: Saprolite at the Crandon Site.* February 23, 1995.

Foth & Van Dyke, 1999/2000. *Crandon Project Tailings Management Area Groundwater Quality Performance Evaluation (Depyritized Tailings).* Originally Issued June 1999. Updated March 2000.

USEPA, 1989. *Statistical Analysis of Groundwater Monitoring Data at RCRA Facilities - Interim Final Guidance.* Office of Solid Waste, Washington D.C. PB89-151047.

USEPA, 1992. *Statistical Analysis of Groundwater Monitoring Data at RCRA Facilities - Addendum to Interim Final Guidance.* Office of Solid Waste, Washington D.C. EPA530-R-93-003.

Table 1

Bedrock and Surrogate Overburden Data
Calculated Baseline and ACLs

Parameter	Units	Baseline[1]	NR 140 PAL	NR 140 ES	ACL	Data Set
Antimony	mg/L	0.136	0.0012	0.006	0.2	Bedrock
Arsenic	mg/L	0.0986	0.005	0.05	0.11	Bedrock
Barium	mg/L	0.047	0.4	2	—	Overburden
Beryllium	mg/L	—	0.0004	0.004	—	—
Boron	mg/L	—	0.19	0.96	—	—
Cadmium	mg/L	0.014	0.0005	0.005	0.024	Bedrock
Chloride	mg/L	2	125	250	—	Overburden
Chromium	mg/L	<0.005	0.01	0.1	—	Overburden
Cobalt	mg/L	<0.010	0.008	0.04	—	Overburden
Copper	mg/L	<0.03	0.13	1.3	—	Overburden
Cyanide[2]	mg/L	<0.02	0.04	0.2	—	Overburden
Fluoride[2]	mg/L	0.17	0.8	4	—	Overburden
Iron	mg/L	0.331	0.15	0.3	0.4	Bedrock
Lead	mg/L	0.0440	0.0015	0.015	0.065	Bedrock
Manganese	mg/L	0.12	0.025	0.05	0.33	Overburden
Mercury	µg/L	<0.2	0.2	2	0.5	Overburden
Nickel	mg/L	<0.03	0.02	0.1	—	Overburden
Nitrate+Nitrite[3]	mg/L	1.12	2	10	2.5	Overburden
Selenium	mg/L	0.0153	0.01	0.05	0.027	Bedrock
Silver	mg/L	<0.01	0.01	0.05	0.015	Overburden
Sulfate	mg/L	58.8	125	250	160	Overburden
Zinc	mg/L	0.49	2.5	5	—	Bedrock

1 Calculated as Upper 95% Confidence Limit

Prepared by: SGL
Checked by: SVD1

2 Bedrock groundwater samples were not analyzed for fluoride. Glacial overburden data used as a surrogate.

3 See Attachment 4 for derivation of baseline concentrations which are based on data from glacial overburden. Bedrock samples were not analyzed for Nitrate-Nitrite. Glacial overburden data used as surrogate.

Attachment 1

Bedrock and Overburden Sample Results

Bedrock Water Quality Results

Location	Date	Sulfate mg/L	Chloride (dissolved) mg/L	Arsenic (dissolved) mg/L	Barium (dissolved) mg/L	Iron (dissolved) mg/L	Manganese (dissolved) mg/L	Molybdenum (dissolved) mg/L	Selenium (dissolved) mg/L	Silver (dissolved) mg/L	Chromium (dissolved) mg/L	Cobalt (dissolved) mg/L	Nickel (dissolved) mg/L	Antimony (dissolved) mg/L	Magnesium (dissolved) mg/L	Aluminum (dissolved) mg/L	Cadmium (dissolved) mg/L	Copper (dissolved) mg/L	Lead (dissolved) mg/L	Mercury (dissolved) ug/L	Zinc (dissolved) mg/L
Bedrock	11/01/94			0.09	0.02	0.24	0.016		0.0255	<0.0019	<0.00069	0.0045	<0.0012	0.2	14	<0.043	0.023	0.0065	0.061		0.67
Bedrock	11/02/94	6.7	2	0.096	0.022	0.31	0.031	<0.020	0.0081	<0.010	<0.0050	<0.010	<0.030	0.1	17	<0.050	0.014	<0.030	0.042	<0.20	0.51
Bedrock	11/03/94			0.082	0.017	0.29	0.015		0.01	<0.0019	<0.00069	0.0031	<0.0012	0.1	14	<0.043	0.0075	<0.0016	0.04		0.38
Bedrock	11/04/94			0.09	0.016	0.25	0.015	<0.020	0.014	<0.0019	<0.00069	0.0014	<0.0012	0.085	14	<0.043	0.005	<0.0016	0.016	<0.20	0.29
Bedrock	11/09/94	6.3	1	0.1	0.023	0.34	0.025	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	0.085	17	<0.050	0.0019	<0.030	0.0237	<0.20	0.25
Bedrock	11/10/94			0.098	0.015	0.27	0.014		0.0028	<0.0019	<0.00069	0.002	<0.0012	0.076	14	<0.043	0.0018	<0.0016	0.02		0.16
Bedrock	11/16/94	11	2	0.1	0.02	0.37	0.02	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	0.085	16	<0.050	0.0012	<0.030	0.022	0.22	0.25
Overburden 80th Percentile		48	2	<0.0050	0.066	<0.100	0.189	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.20	34	<0.200	0.0001	<0.030	<0.0020	0.3	<0.020

Indicates where sample result exceeds upper 80th percentile of overburden results.

Overburden Water Quality Results

Location	Date	Sulfate mg/L	Chloride (dissolved) mg/L	Arsenic (dissolved) mg/L	Barium (dissolved) mg/L	Iron (dissolved) mg/L	Manganese (dissolved) mg/L	Molybdenum (dissolved) mg/L	Selenium (dissolved) mg/L	Silver (dissolved) mg/L	Chromium (dissolved) mg/L	Cobalt (dissolved) mg/L	Nickel (dissolved) mg/L	Antimony (dissolved) mg/L	Magnesium (dissolved) mg/L	Aluminum (dissolved) mg/L	Cadmium (dissolved) mg/L	Copper (dissolved) mg/L	Lead (dissolved) mg/L	Mercury (dissolved) ug/L	Zinc (dissolved) mg/L
CMC-08	05/14/94	34.0	2	<0.005	0.050	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	25.0	<0.200	<0.001	<0.030	<0.001	0.4	<0.020
CMC-08	06/12/94	28.0	2	<0.005	0.076	<0.100	0.054	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	27.3	<0.200	<0.001	<0.030	0.002	0.3	<0.020
CMC-08	07/10/94	28.0	2	<0.005	0.079	<0.100	0.011	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	29.3	<0.200	<0.001	<0.030	0.006	0.2	<0.020
CMC-08	08/13/94	38.0	2	<0.005	0.065	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	30.1	0.249	<0.001	<0.030	<0.001	<0.2	<0.020
CMC-08	09/10/94	30	2	<0.0050	0.068	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.005	<0.010	<0.030	<0.20	32	<0.050	0.0005	<0.030	<0.0020	<0.2	<0.020
CMC-08	10/08/94	35	2	<0.0050	0.065	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.20	32	<0.050	0.00010	<0.030	<0.0020	<0.2	<0.020
CMC-08	12/10/94	30	2	<0.0050	0.066	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	33	<0.050	0.00010	<0.030	<0.0020	<0.20	<0.020
CMC-08	01/07/95	31	2	<0.0050	0.074	<0.050	<0.010	0.088	<0.0050	0.016	0.014	0.018	<0.030	<0.0050	32	<0.050	<0.0001	<0.030	<0.0020	<0.20	0.038
CMC-08P	05/14/94	53.0	7	<0.005	0.030	<0.100	0.043	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	20.7	<0.200	<0.001	<0.030	<0.001	0.3	<0.020
CMC-08P	06/12/94	48.0	7	<0.005	0.065	<0.100	0.046	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	25.0	<0.200	<0.001	<0.030	0.005	0.7	<0.020
CMC-08P	07/10/94	43.0	7	<0.005	0.081	<0.100	0.030	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	29.2	<0.200	<0.001	<0.030	0.002	0.4	<0.020
CMC-08P	08/13/94	43.0	5	<0.005	0.071	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	30.7	<0.200	<0.001	<0.030	<0.001	<0.2	<0.020
CMC-08P	09/10/94	35	5	<0.0050	0.080	0.11	0.011	<0.020	<0.0050	<0.010	<0.005	<0.010	<0.030	<0.20	34	<0.050	0.0002	<0.030	<0.0020	<0.2	<0.020
CMC-08P	10/08/94	41	4	<0.0050	0.077	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.20	34	<0.050	0.00038	<0.030	<0.0020	<0.2	<0.020
CMC-08P	12/10/94	30	2	<0.0050	0.076	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	36	<0.050	0.00042	<0.030	<0.0020	<0.20	<0.020
CMC-08P	01/07/95	31	4	<0.0050	0.077	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	36	<0.050	0.00038	<0.030	<0.0020	<0.20	<0.020
CMC-09	05/14/94	11.0	2	<0.005	0.037	<0.100	0.015	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	19.7	<0.200	<0.001	<0.030	<0.001	0.6	<0.020
CMC-09	06/12/94	9.00	2	<0.005	0.038	0.120	0.011	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	21.9	<0.200	<0.001	<0.030	<0.001	0.3	<0.020
CMC-09	07/10/94	10.0	2	<0.005	0.031	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	21.1	<0.200	<0.001	<0.030	0.002	0.5	<0.020
CMC-09	08/13/94	13.0	2	<0.005	0.026	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	22.6	<0.200	<0.001	<0.030	0.001	<0.2	<0.020
CMC-09	09/10/94	13	1	<0.0050	0.033	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.005	<0.010	<0.030	<0.20	22	<0.050	<0.0001	<0.030	<0.0020	<0.2	<0.020
CMC-09	10/08/94	16	2	<0.0050	0.032	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.20	23	<0.050	<0.0001	<0.030	<0.0020	<0.2	<0.020
CMC-09	12/10/94	12	2	0.0050	0.026	<0.050	<0.010	<0.020	<0.0050	0.032	<0.0050	<0.010	<0.030	<0.0050	19	<0.050	<0.0001	<0.030	<0.0020	<0.20	0.023
CMC-09	01/07/95	12	2	<0.0050	0.025	<0.050	0.039	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	19	<0.050	<0.0001	<0.030	<0.0020	<0.20	<0.020
CMC-09P	06/12/94	51.0	2	<0.005	0.028	<0.100	0.294	0.023	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	5.83	<0.200	<0.001	<0.030	<0.001	0.2	<0.020
CMC-09P	07/10/94	38.0	2	<0.005	0.028	<0.100	0.261	0.048	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	5.33	<0.200	<0.001	<0.030	0.003	0.3	<0.020
CMC-09P	08/13/94	31.0	2	<0.005	0.027	0.100	0.256	0.022	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	5.93	0.218	<0.001	<0.030	<0.001	<0.2	<0.020
CMC-09P	09/10/94	25	1	<0.0050	0.031	0.32	0.24	0.026	<0.0050	<0.010	<0.005	<0.010	<0.030	<0.20	5.6	0.083	<0.0001	<0.030	<0.0020	<0.2	0.032
CMC-09P	10/08/94	28	2	<0.0050	0.031	0.21	0.22	0.027	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.20	6.2	<0.050	<0.0001	<0.030	<0.0020	<0.2	0.025
CMC-09P	12/10/94	18	2	0.0053	0.030	0.38	0.22	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	6.8	<0.050	0.00030	<0.030	<0.0020	<0.20	0.021
CMC-09P	01/07/95	17	2	<0.0050	0.029	<0.050	0.22	0.022	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	7.3	<0.050	0.00027	<0.030	<0.0020	<0.20	0.021
CMC-09P	02/06/95	12	2	0.0062	0.027	<0.050	0.18	<0.020	<0.0050	<0.010	<0.0050	<0.010	0.016	<0.005	6.9	<0.050	0.0015	<0.030	<0.0020	<0.20	0.025
CMC-10	05/14/94	200	1	<0.005	0.039	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	43.2	<0.200	<0.001	<0.030	<0.001	0.4	<0.020
CMC-10	06/12/94	93.0	1	<0.005	0.034	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	42.8	<0.200	<0.001	<0.030	<0.001	0.4	<0.020
CMC-10	07/10/94	170	1	<0.005	0.026	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	40.7	<0.200	<0.001	<0.030	0.001	<0.2	<0.020
CMC-10	08/13/94	160	1	<0.005	0.023	<0.100	<0.010	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	35.8	0.236	<0.001	<0.030	<0.001	<0.2	<0.020
CMC-10	09/10/94	120	<1	<0.0050	0.021	0.14	<0.010	<0.020	<0.0050	<0.010	<0.005	<0.010	<0.030	<0.20	35	<0.050	<0.0001	<0.030	<0.0020	<0.2	<0.020
CMC-10	10/08/94	150	<1	<0.0050	0.018	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.20	33	<0.050	<0.0001	<0.030	<0.0020	<0.2	<0.020
CMC-10	12/10/94	160	<1	<0.0050	0.023	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	41	<0.050	<0.0001	<0.030	<0.0020	<0.20	<0.020
CMC-10	01/07/95	200	1	<0.0050	0.022	<0.050	<0.010	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	38	<0.050	<0.0001	<0.030	<0.0020	<0.20	<0.020
CMC-10P	05/14/94	8.00	1	<0.005	0.032	<0.100	0.182	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	14.2	<0.200	<0.001	<0.030	<0.001	<0.2	<0.020
CMC-10P	06/12/94	7.00	1	<0.005	0.032	<0.100	0.189	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	15.2	<0.200	<0.001	<0.030	<0.001	0.4	<0.020
CMC-10P	07/10/94	5.40	1	<0.005	0.030	<0.100	0.172	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	14.3	<0.200	<0.001	<0.030	0.003	0.4	<0.020
CMC-10P	08/13/94	10.0	1	<0.005	0.030	<0.100	0.189	<0.020	<0.005	<0.010	<0.005	<0.010	<0.030	<0.050	14.9	0.258	<0.001	<0.030	<0.001	<0.2	<0.020
CMC-10P	09/10/94	11	<1	<0.0050	0.030	<0.050	0.19	<0.020	<0.0050	<0.010	<0.005	<0.010	<0.030	<0.20	16	<0.050	0.0003	<0.030	<0.0020	<0.2	<0.020
CMC-10P	10/08/94	9.7	<1	<0.0050	0.027	<0.050	0.17	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.20	15	<0.050	<0.0010	<0.030	<0.0020	<0.2	<0.020
CMC-10P	12/10/94	6.6	<1	<0.0050	0.032	<0.050	0.48	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	15	<0.050	<0.0001	<0.030	<0.0020	<0.20	<0.020
CMC-10P	01/07/95	6.4	2	<0.0050	0.031	<0.050	0.40	<0.020	<0.0050	<0.010	<0.0050	<0.010	<0.030	<0.0050	15	<0.050	<0.0001	<0.030	<0.0020	<0.20	0.031

Overburden Water Quality Results (Cont.)

Location	Date	Cyanide (total) mg/L	Fluoride mg/L
CMC-08	05/15/94	<0.020	0.121
CMC-08	06/13/94	<0.020	0.136
CMC-08	07/12/94	<0.020	0.123
CMC-08	08/15/94	<0.020	0.12
CMC-08	09/13/94	<0.020	0.1
CMC-08	10/11/94	<0.020	0.1
CMC-08	12/14/94	<0.020	0.1
CMC-08	01/09/95	<0.020	0.1
CMC-08P	05/15/94	<0.020	0.134
CMC-08P	06/13/94	<0.020	0.156
CMC-08P	07/12/94	<0.020	0.139
CMC-08P	08/16/94	<0.020	0.144
CMC-08P	09/13/94	<0.020	0.2
CMC-08P	10/11/94	<0.020	0.1
CMC-08P	12/14/94	<0.020	0.1
CMC-08P	01/09/95	<0.020	0.1
CMC-09	05/18/94	<0.020	<0.1
CMC-09	06/16/94	<0.020	<0.1
CMC-09	07/14/94	<0.020	<0.1
CMC-09	08/17/94	<0.020	<0.1
CMC-09	09/14/94	<0.020	<0.1
CMC-09	10/12/94	<0.020	0.1
CMC-09	12/15/94	<0.020	1
CMC-09	01/10/95	<0.020	<0.1
CMC-09P	06/17/94	<0.020	0.302
CMC-09P	07/15/94	<0.020	0.29
CMC-09P	08/17/94	<0.020	0.257
CMC-09P	09/14/94	<0.020	0.3
CMC-09P	10/12/94	<0.020	0.2
CMC-09P	12/15/94	<0.020	0.3
CMC-09P	01/10/95	<0.020	0.2
CMC-09P	02/06/95	<0.020	0.2
CMC-10	05/17/94	<0.020	<0.1
CMC-10	06/16/94	<0.020	<0.1
CMC-10	07/13/94	<0.020	<0.1
CMC-10	08/18/94	<0.020	<0.1
CMC-10	09/13/94	<0.020	<0.1
CMC-10	10/14/94	<0.020	0.1
CMC-10	12/12/94	<0.020	<0.1
CMC-10	01/10/95	<0.020	<0.1
CMC-10P	05/17/94	<0.020	<0.1
CMC-10P	06/16/94	<0.020	0.105
CMC-10P	07/13/94	<0.020	0.102
CMC-10P	08/18/94	<0.020	0.109
CMC-10P	09/13/94	<0.020	0.1
CMC-10P	10/14/94	<0.020	0.1
CMC-10P	12/12/94	<0.020	<0.1
CMC-10P	01/10/95	<0.020	0.1

Attachment 2

Mann-Whitney U Test Results

Mann-Whitney U Test Results

Location	Date	Sulfate mg/L	Chloride (dissolved) mg/L	Arsenic (dissolved) mg/L	Barium (dissolved) mg/L	Iron (dissolved) mg/L	Manganese (dissolved) mg/L	Molybdenum (dissolved) mg/L	Selenium (dissolved) mg/L	Silver (dissolved) mg/L	Chromium (dissolved) mg/L	Cobalt (dissolved) mg/L	Nickel (dissolved) mg/L	Antimony (dissolved) mg/L	Magnesium (dissolved) mg/L	Aluminum (dissolved) mg/L	Cadmium (dissolved) mg/L	Copper (dissolved) mg/L	Lead (dissolved) mg/L	Mercury (dissolved) ug/L	Zinc (dissolved) mg/L
Mann-Whitney U Test																					
Rank Sum Group 1		20	75.5	364	36.5	352	216	92	316	189	192.5	286.5	192.5	364	107.5	178.5	363	220	364	94.5	364
Rank Sum Group 2		1306	1250.5	1176	1503.5	1188	1324	1286	1224	1351	1347.5	1253.5	1347.5	1176	1432.5	1361.5	1177	1320	1176	1283.5	1176
U		14	69.5	0	8.5	12	148	82	48	161	164.5	77.5	164.5	0	79.5	150.5	1	144	0	84.5	0
Z		-2.32	-0.10	-4.24	-4.03	-3.94	-0.51	-0.48	-3.03	-0.18	-0.09	-2.29	-0.09	-4.24	-2.23	-0.44	-4.22	-0.61	-4.24	-0.39	-4.24
p-level		0.020	0.920	0.000	0.000	0.000	0.614	0.631	0.002	0.860	0.930	0.022	0.930	0.000	0.025	0.659	0.000	0.544	0.000	0.693	0.000
N Group 1		3	3	7	7	7	7	4	7	7	7	7	7	7	7	7	7	7	7	4	7
N Group 2		48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48

Attachment 3

Baseline and ACL Calculations

Bedrock Background and ACL Calculations

Location Date	Arsenic (dissolved) mg/L	Iron (dissolved) mg/L	Selenium (dissolved) mg/L	Antimony (dissolved) mg/L	Cadmium (dissolved) mg/L	Lead (dissolved) mg/L	Zinc (dissolved) mg/L
# Samples	7	7	7	7	7	7	7
% Detections	100.0%	100.0%	71.4%	100.0%	100.0%	100.0%	100.0%
Minimum	0.082	0.24	<0.005	0.076	0.0012	0.018	0.16
Maximum	0.1	0.37	0.0255	0.2	0.023	0.061	0.67
Average (1)	0.09371	0.29571	0.00934	0.10443	0.00777	0.03239	0.359
St. Dev.(1)	0.00668	0.04756	0.00838	0.04304	0.00808	0.01588	0.177
Background (Upper 95% C.L. On Mean (2))	0.0986 Normal	0.331 Normal	0.0153 Normal (Cohen's)	0.136 Normal	0.0137 Normal	0.0440 Normal	0.489 Normal
NR 140 PAL	0.005	0.15	0.01	0.0012	0.0005	0.0015	2.5
# PAL Exceedances	7	7	3	7	7	7	0
NR 140 ES	0.05	0.3	0.05	0.006	0.005	0.015	5
# ES Exceedances	7	3	0	7	4	7	0
Min. Increase							
Calculated PAL (3)	-	-	-	-	-	-	-
ACL (3)	0.11	0.40	0.027	0.20	0.024	0.065	-

(1) All non-detected values replaced with one-half the detection limit as per the May 6, 1994 WDNR guidance document "PAL/ACL Calculations Guidance For Solid Waste Facilities"

(2) Calculated as per the Feb. 1989 USEPA interim guidance document, and July 1992 USEPA addendum to interim guidance document "Statistical Analysis of Groundwater Monitoring Data at RCRA Facilities." If at least 85% values were above the detection limit, the average was calculated replacing non-detects with one-half the detection limit. If between 50% and 85% of the values were above the detection limit, the average was calculated using Cohen's adjustment. If less than 50% of the values were detected, the median is given.

(3) Calculated as per the May 6, 1994 WDNR guidance document "PAL/ACL Calculations Guidance For Solid Waste Facilities"

Overburden Background and ACL Calculations

Location Date	Sulfate mg/L	Chloride (dissolved) mg/L	Arsenic (dissolved) mg/L	Barium (dissolved) mg/L	Iron (dissolved) mg/L	Manganese (dissolved) mg/L	Molybdenum (dissolved) mg/L	Selenium (dissolved) mg/L	Silver (dissolved) mg/L	Chromium (dissolved) mg/L	Cobalt (dissolved) mg/L	Nickel (dissolved) mg/L	Antimony (dissolved) mg/L	Magnesium (dissolved) mg/L	Aluminum (dissolved) mg/L	Cadmium (dissolved) mg/L	Copper (dissolved) mg/L	Lead (dissolved) mg/L	Mercury (dissolved) ug/L	Zinc (dissolved) mg/L
# Samples	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48	48
% Detections	100.0%	87.5%	6.3%	100.0%	14.6%	52.1%	14.6%	0.0%	4.2%	2.1%	2.1%	2.1%	0.0%	100.0%	10.4%	22.9%	0.0%	18.8%	31.3%	16.7%
Minimum	5.4	<1	0.005	0.018	0.05	0.01	0.02	0.005	0.01	0.005	0.01	0.03	0.005	5.33	0.05	0.0001	0.03	0.001	0.2	0.02
Maximum	200	7	0.0062	0.081	0.38	0.48	0.088	0.005	0.032	0.014	0.018	0.016	0.2	43.2	0.258	0.0015	0.03	0.006	0.7	0.038
Average (1)	46.065	2.104	0.00269	0.04227	0.06104	0.08829	0.01388	0.00250	0.00579	0.00274	0.00527	0.01502	0.03766	23.616	0.07383	0.00036	0.015	0.00119	0.190	0.01283
St. Dev.(1)	53.480	1.631	0.00075	0.02068	0.07043	0.12038	0.01289	0	0.00418	0.00166	0.00188	0.00014	0.03758	11.313	0.06223	0.00026	0	0.00107	0.153	0.00682
Background (Upper 95% C.L. On Mean (2))	58.8 Normal	2 Non-Parametric	<0.005 Non-Parametric	0.0472 Normal	<0.1 Non-Parametric	0.117 Normal (Cohen's)	<0.020 Non-Parametric	<0.005 Non-Parametric	<0.01 Non-Parametric	<0.005 Non-Parametric	<0.010 Non-Parametric	<0.03 Non-Parametric	<0.05 Non-Parametric	26.3 Normal	<0.2 Non-Parametric	<0.001 Non-Parametric	<0.03 Non-Parametric	<0.002 Non-Parametric	<0.2 Non-Parametric	<0.02 Non-Parametric
NR 140 PAL	125	125	0.005	0.4	0.15	0.025	-	0.01	0.01	0.01	-	0.02	0.0012	-	-	0.0005	0.13	0.0015	0.2	2.5
# PAL Exceedances	6	0	3	0	3	21	-	0	2	1	-	0	0	-	-	2	0	7	15	0
NR 140 ES	250	250	0.05	2	0.3	0.05	-	0.05	0.05	0.1	-	0.1	0.006	-	-	0.005	1.3	0.015	2	5
# ES Exceedances	0	0	0	0	2	17	-	0	0	0	-	0	0	-	-	0	0	0	0	0
Min. Increase	-	-	-	-	-	-	-	-	-	-	-	-	-	25	-	-	-	-	-	-
Calculated PAL (3)	-		-	-	-	-	-	-	-	-	-	-	-	58	-	-	-	-	-	-
ACL (3)	160	-	0.0042	-	0.21	0.33	-	-	0.015	-	-	-	0.12	-	-	0.00088	-	0.0034	0.50	-

(1) All non-detected values replaced with one-half the detection limit as per the May 6, 1994 WDNR guidance document "PAL/ACL Calculations Guidance For Solid Waste Facilities"

(2) Calculated as per the Feb. 1989 USEPA interim guidance document, and July 1992 USEPA addendum to interim guidance document "Statistical Analysis of Groundwater Monitoring Data at RCRA Facilities." If at least 85% values were above the detection limit, the average was calculated replacing non-detects with one-half the detection limit. If between 50% and 85% of the values were above the detection limit, the average was calculated using Cohen's adjustment. If less than 50% of the values were detected, the median is given.

(3) Calculated as per the May 6, 1994 WDNR guidance document "PAL/ACL Calculations Guidance For Solid Waste Facilities"

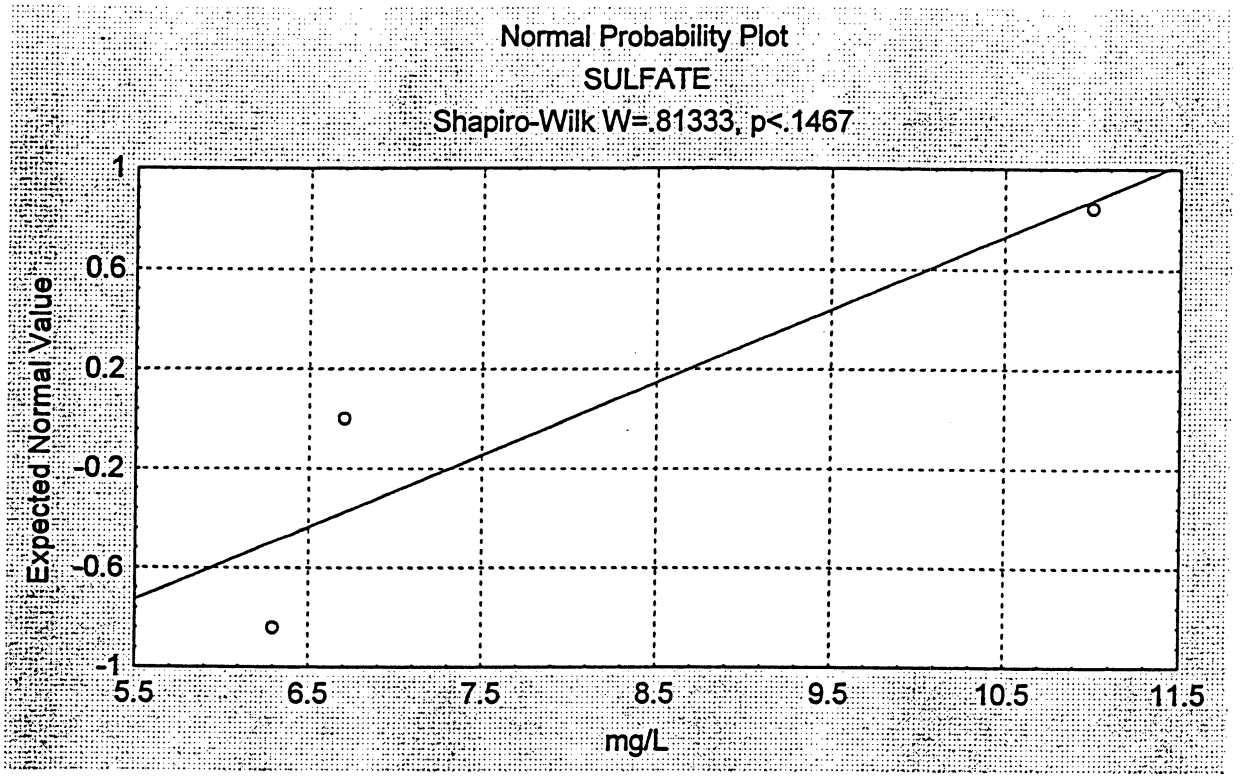
Overburden Background and ACL Calculations (Cont.)

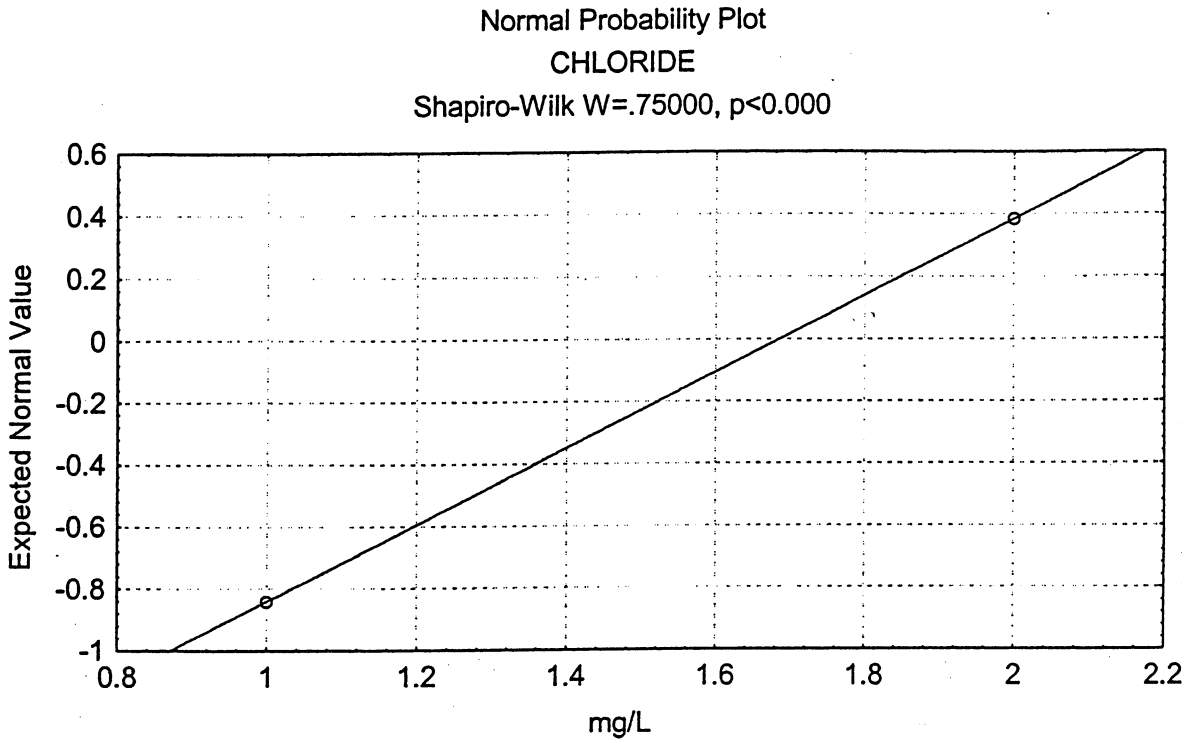
Location Date	Cyanide (total) mg/L	Fluoride mg/L
# Samples	48	48
% Detections	0	68.8%
Minimum	<0.020	<0.1
Maximum	<0.020	1
Average (1)	<0.020	0.137
St. Dev.(1)	0	0.147
Background (Upper 95% C.L. On Mean (2))	<0.02	0.17
NR 140 PAL	0.04	0.8
# PAL Exceedances	0	1
NR 140 ES	0.2	4
# ES Exceedances	0	0
Min. Increase		
Calculated		
PAL (3)	-	-
ACL (3)	-	-

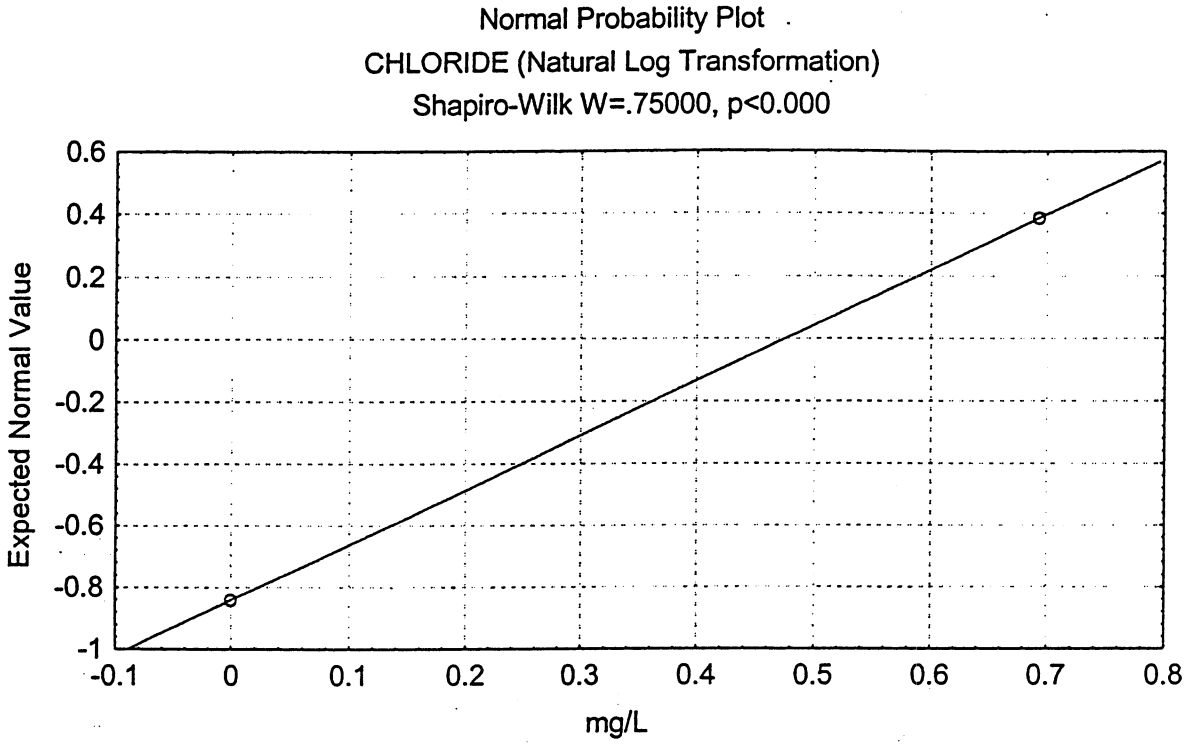
(1) All non-detected values replaced with one-half the detection limit as per the May 6, 1994 WDNR guidance document "PAL/ACL Calculations Guidance For Solid Waste Facilities"

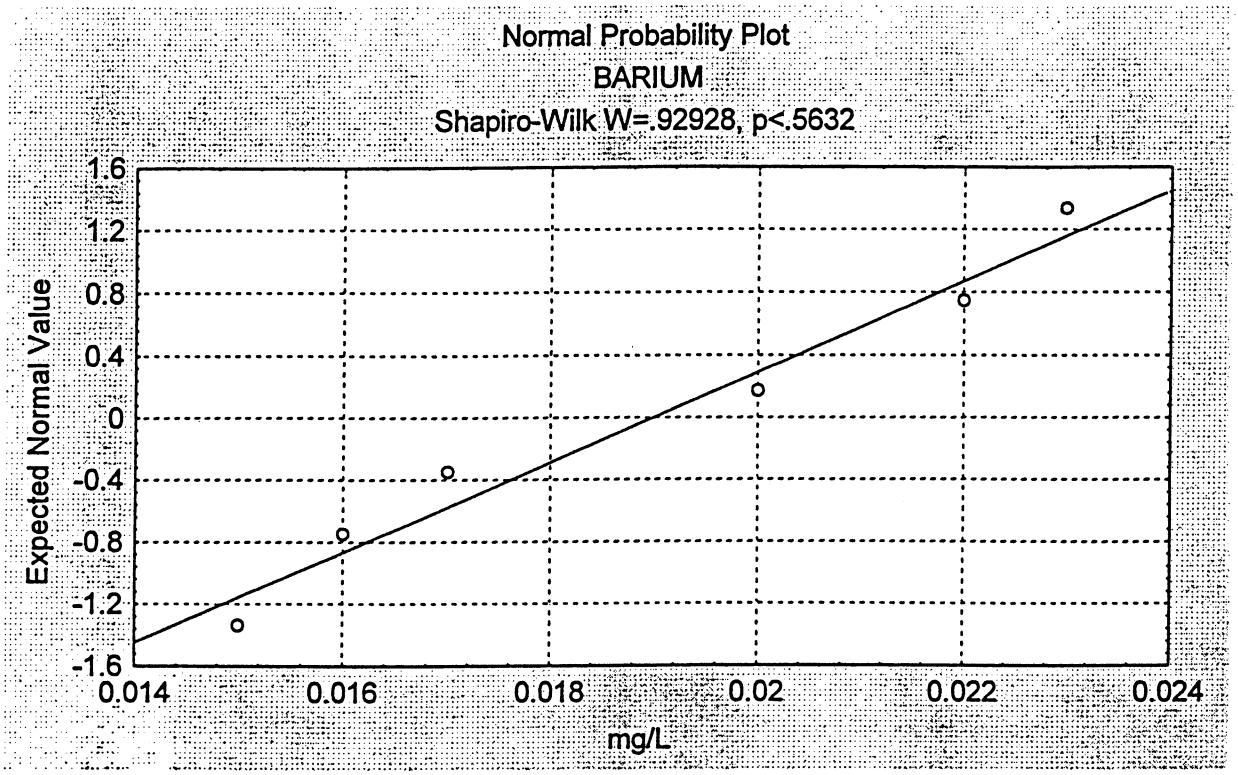
(2) Calculated as per the Feb. 1989 USEPA interim guidance document, and July 1992 USEPA addendum to interim guidance document "Statistical Analysis of Groundwater Monitoring Data at RCRA Facilities." If at least 85% values were above the detection limit, the average was calculated replacing non-detects with one-half the detection limit. If between 50% and 85% of the values were above the detection limit, the average was calculated using Cohen's adjustment. If less than 50% of the values were detected, the median is given.

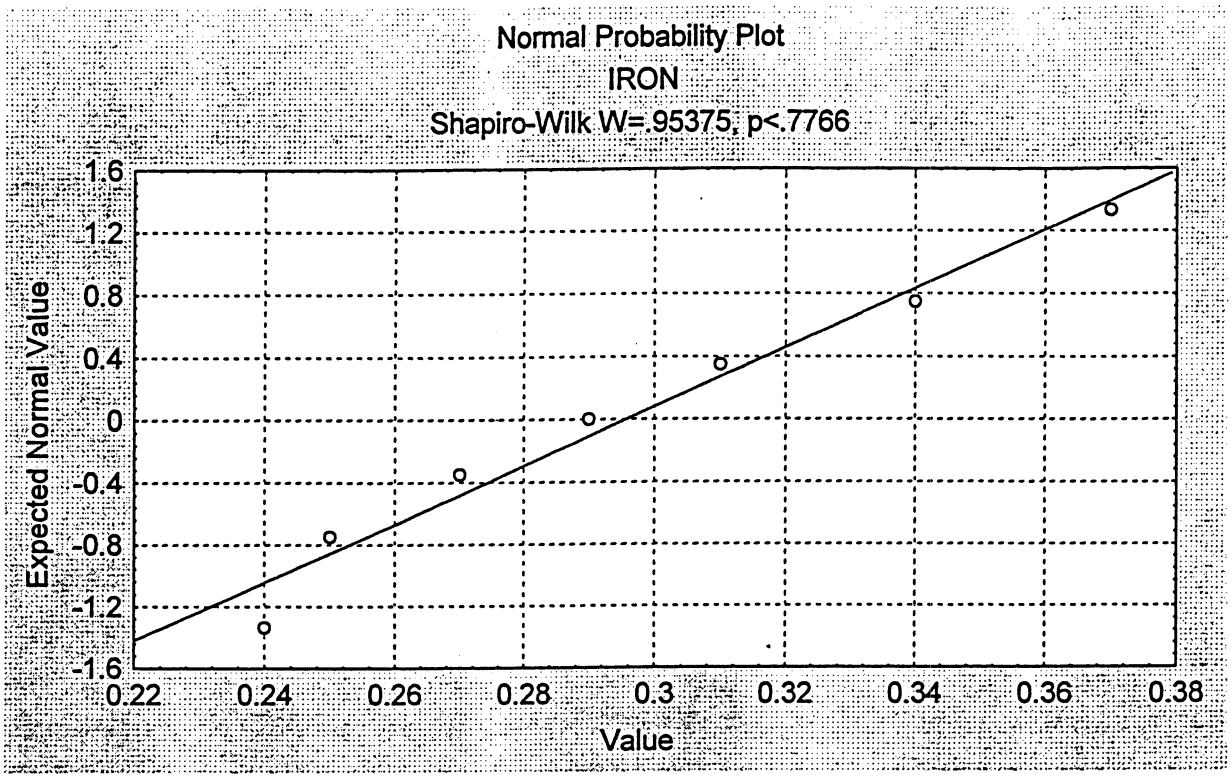
(3) Calculated as per the May 6, 1994 WDNR guidance document "PAL/ACL Calculations Guidance For Solid Waste Facilities"

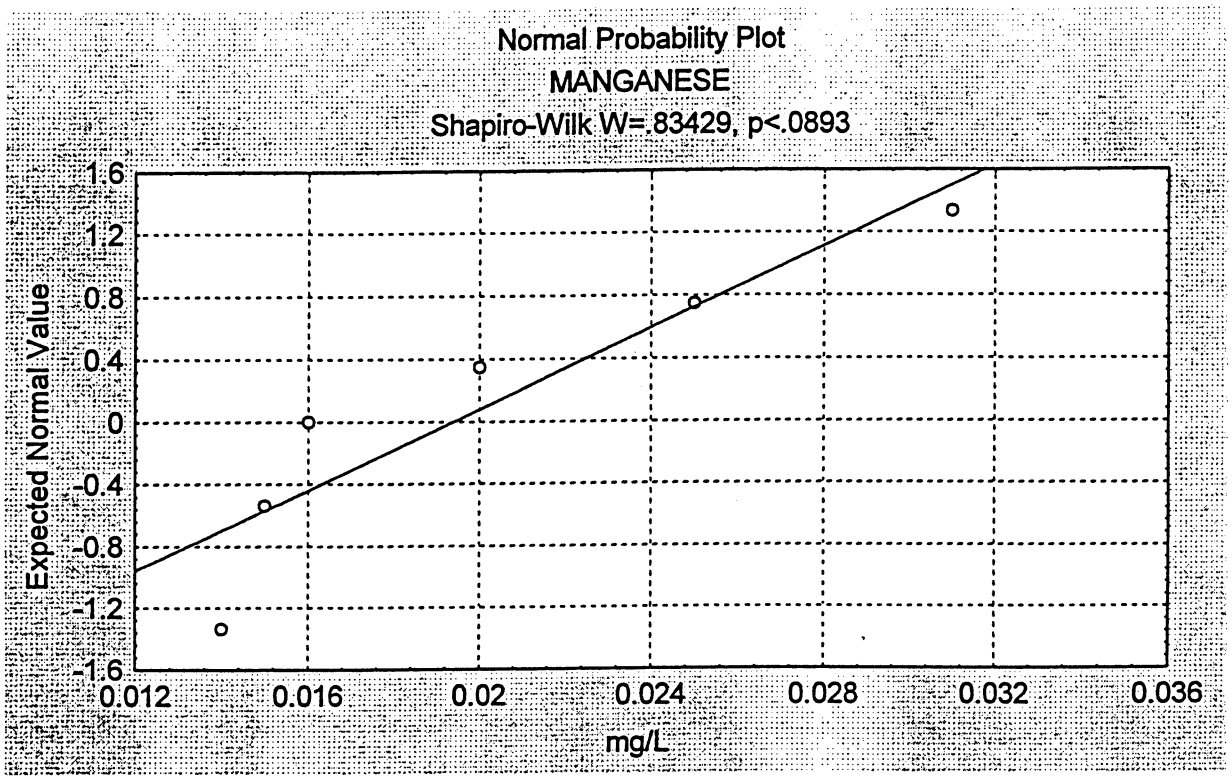

Normal Probability Plot
SULFATE
Shapiro-Wilk W=.81333, p<.1467
Expected Normal Value
1
0.6
0.2
-0.2
-0.6
-1
5.5
6.5
7.5
8.5
9.5
10.5
11.5
mg/L

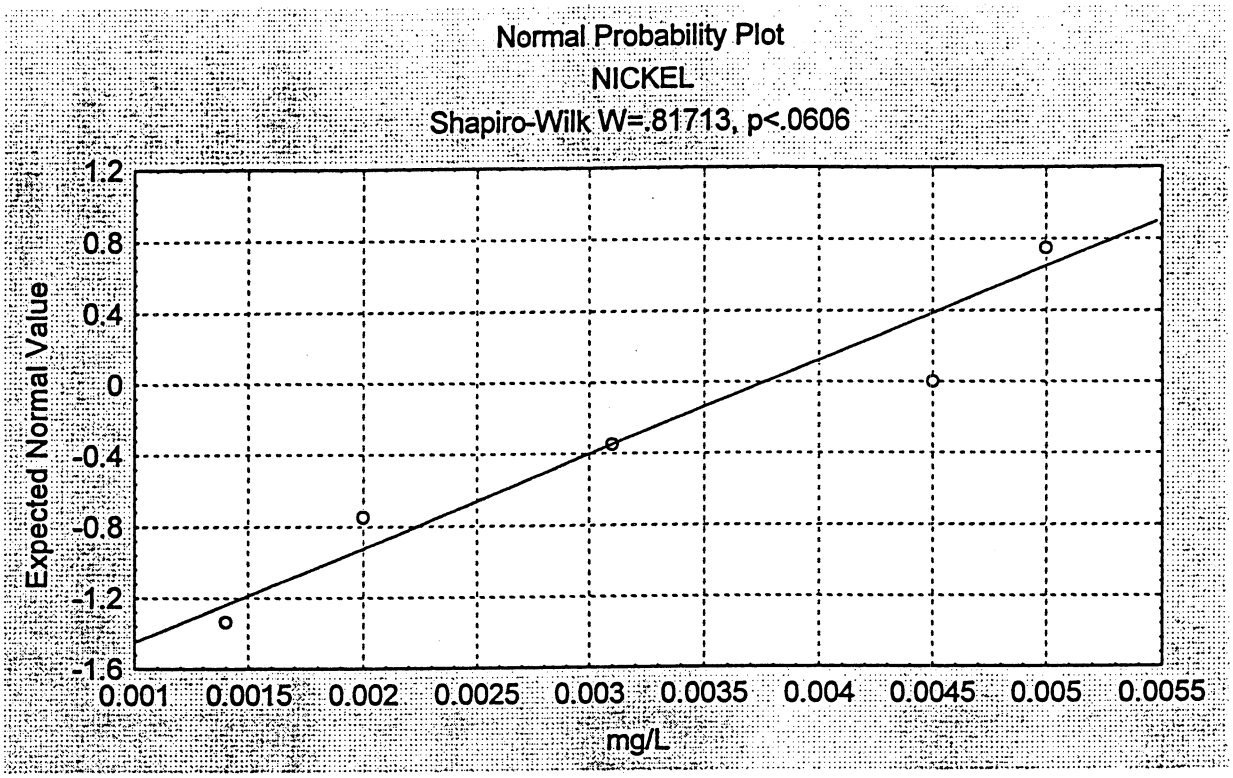

Normal Probability Plot
CHLORIDE
Shapiro-Wilk W=.75000, p<0.000
Expected Normal Value
0.6
0.4
0.2
0
-0.2
-0.4
-0.6
-0.8
-1
0.8
1
1.2
1.4
1.6
1.8
2
2.2
mg/L

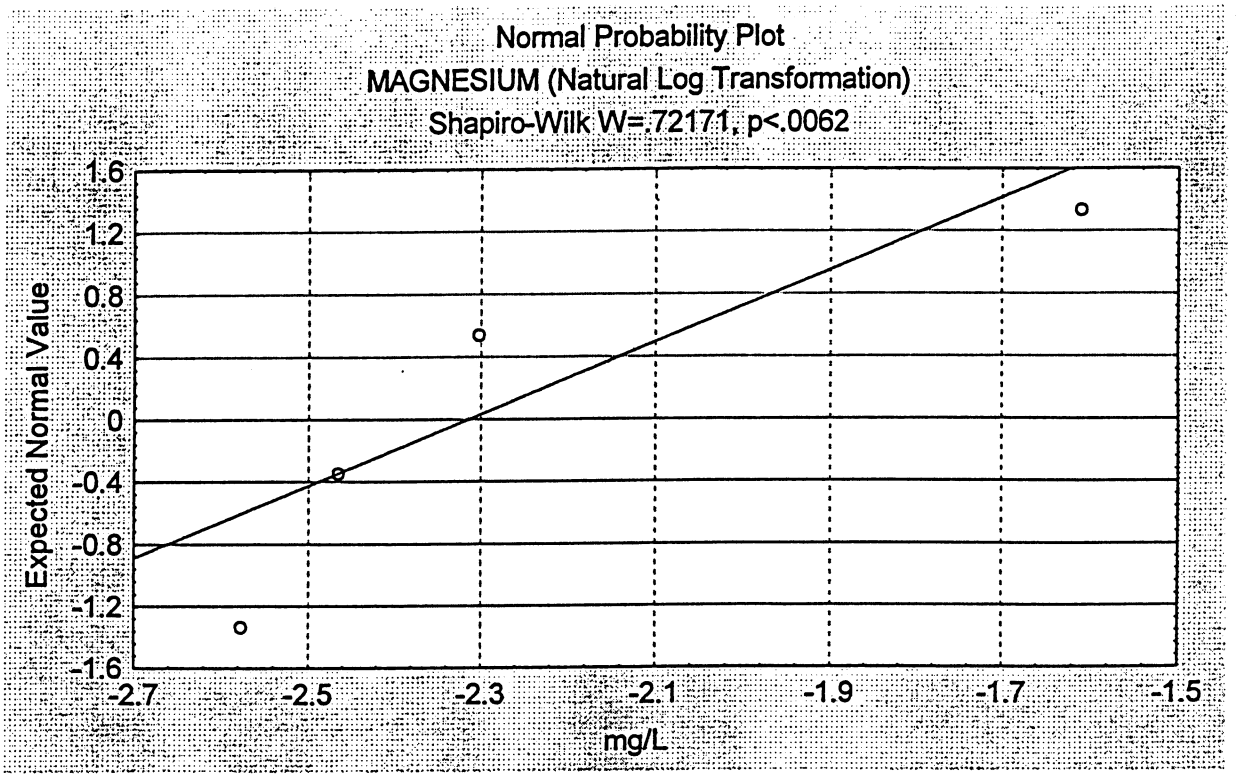

Normal Probability Plot
CHLORIDE (Natural Log Transformation)
Shapiro-Wilk W=.75000, p<0.000
Expected Normal Value
0.6
0.4
0.2
0
-0.2
-0.4
-0.6
-0.8
-1
-0.1
0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
mg/L

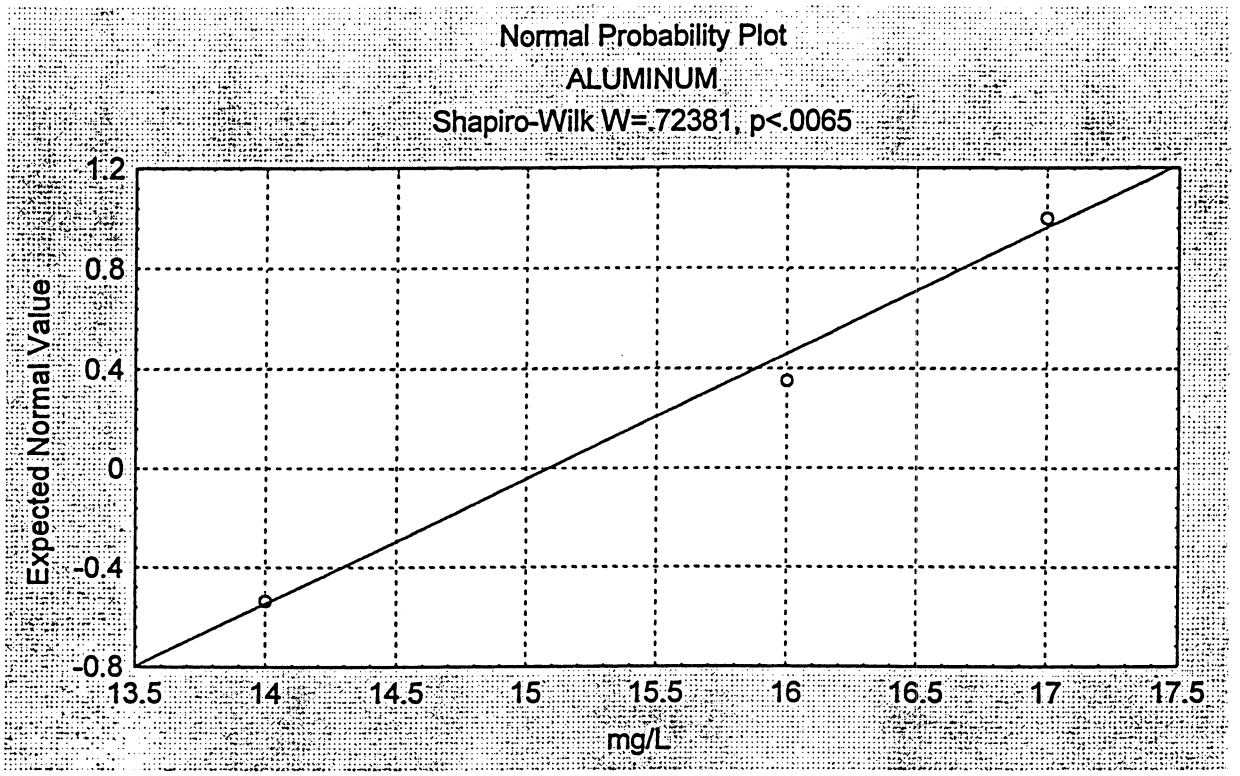

Normal Probability Plot
BARIUM
Shapiro-Wilk W=.92928, p<.5632
Expected Normal Value
1.6
1.2
0.8
0.4
0
-0.4
-0.8
-1.2
-1.6
0.014
0.016
0.018
0.02
0.022
0.024
mg/L

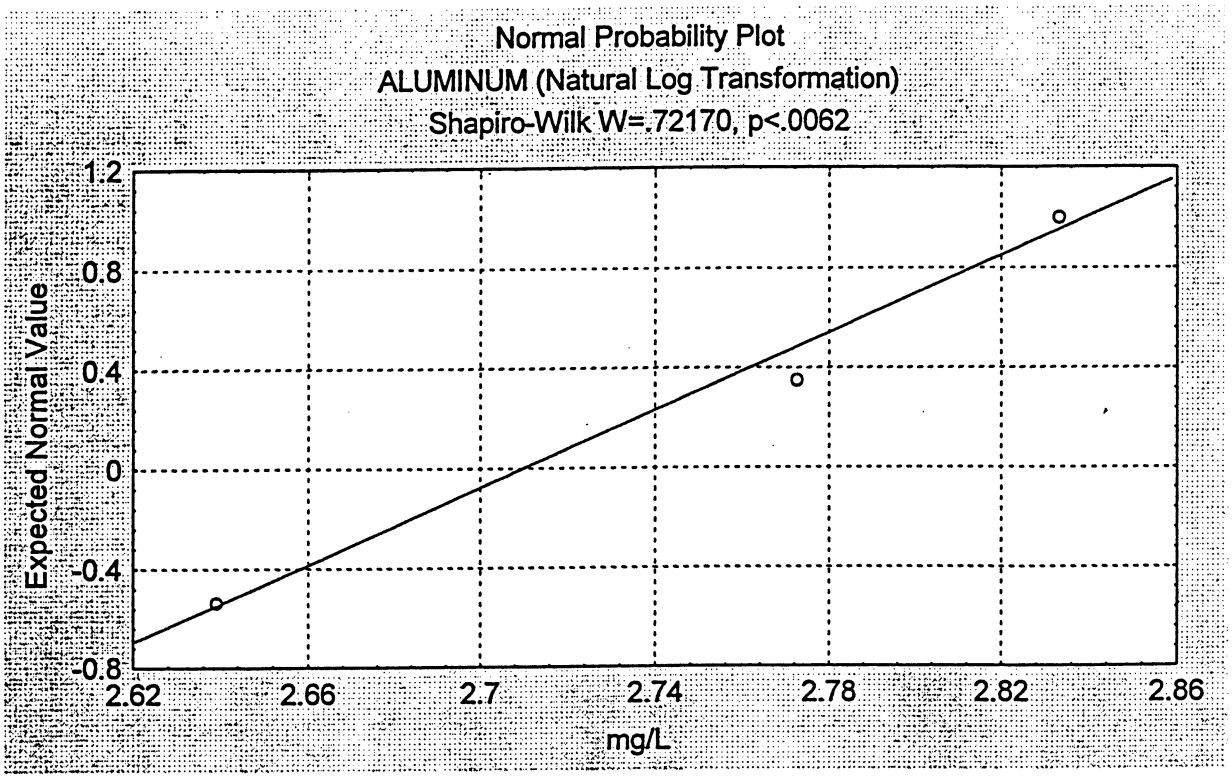

Normal Probability Plot
IRON
Shapiro-Wilk W=.95375, p<.7766
Expected Normal Value
1.6
1.2
0.8
0.4
0
-0.4
-0.8
-1.2
-1.6
0.22
0.24
0.26
0.28
0.3
0.32
0.34
0.36
0.38
Value

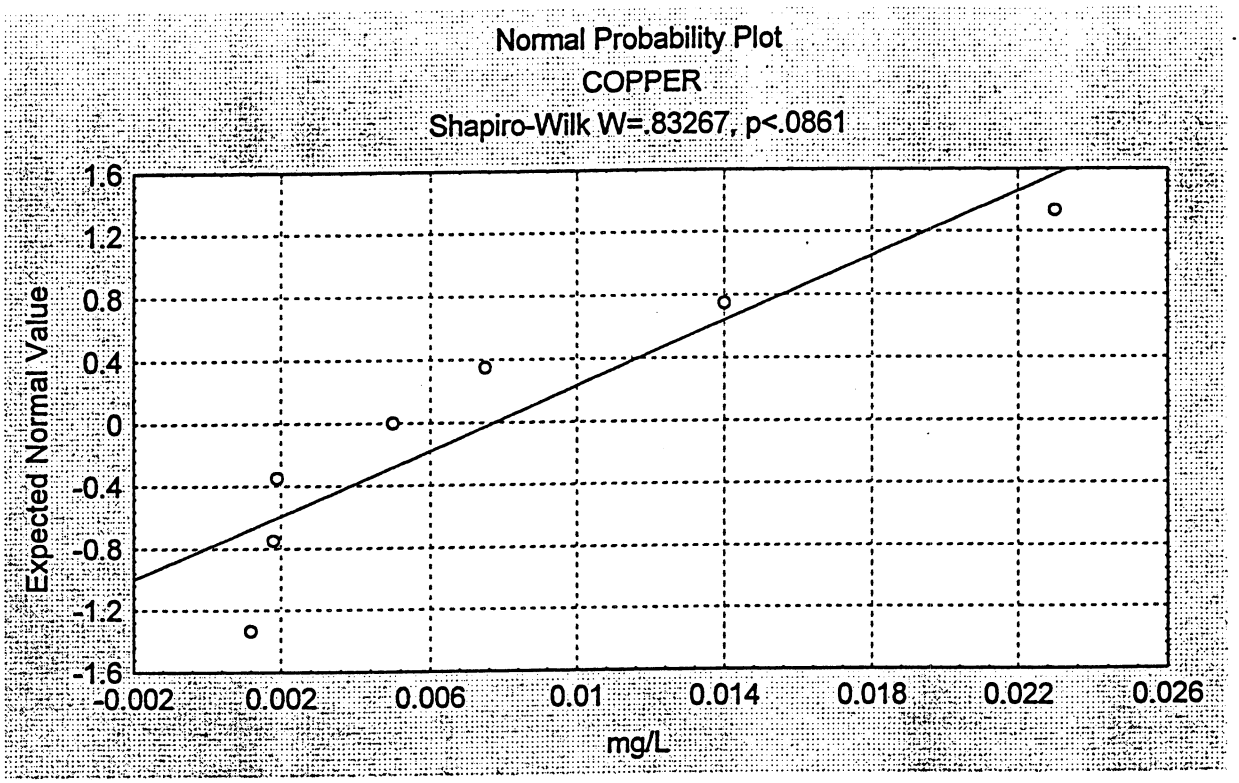

Normal Probability Plot
MANGANESE
Shapiro-Wilk W=.83429, p<.0893
Expected Normal Value
1.6
1.2
0.8
0.4
0
-0.4
-0.8
-1.2
-1.6
0.012
0.016
0.02
0.024
0.028
0.032
0.036
mg/L

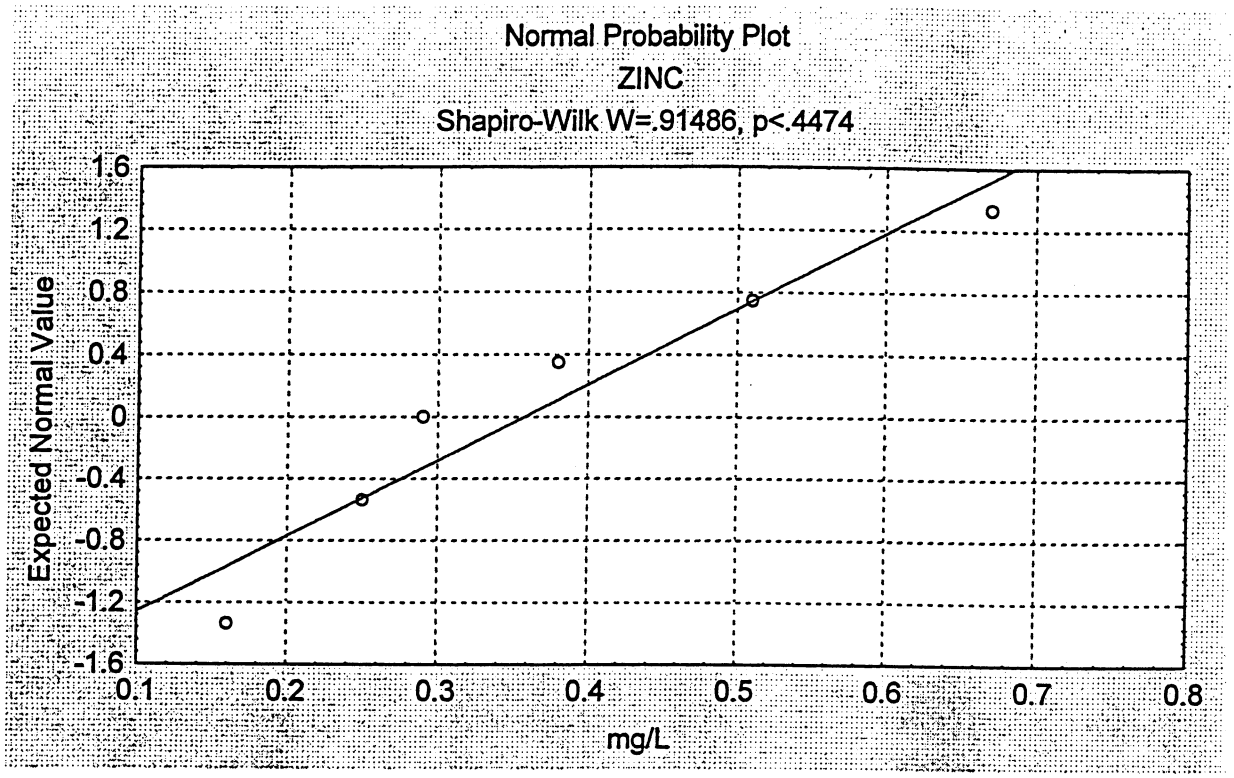

Normal Probability Plot
NICKEL
Shapiro-Wilk W=.81713, p<.0606
Expected Normal Value
1.2
0.8
0.4
0
-0.4
-0.8
-1.2
-1.6
0.001
0.0015
0.002
0.0025
0.003
0.0035
0.004
0.0045
0.005
0.0055
mg/L


Normal Probability Plot
MAGNESIUM (Natural Log Transformation)
Shapiro-Wilk W=.72171, p<.0062
Expected Normal Value
1.6
1.2
0.8
0.4
0
-0.4
-0.8
-1.2
-1.6
-2.7
-2.5
-2.3
-2.1
-1.9
-1.7
-1.5
mg/L


Normal Probability Plot
ALUMINUM
Shapiro-Wilk W=.72381, p<.0065
Expected Normal Value
1.2
0.8
0.4
0
-0.4
-0.8
13.5
14
14.5
15
15.5
16
16.5
17
17.5
mg/L


Normal Probability Plot
ALUMINUM (Natural Log Transformation)
Shapiro-Wilk W=.72170, p<.0062
Expected Normal Value
1.2
0.8
0.4
0
-0.4
-0.8
2.62
2.66
2.7
2.74
2.78
2.82
2.86
mg/L


Normal Probability Plot
COPPER
Shapiro-Wilk W=.83267, p<.0861
Expected Normal Value
1.6
1.2
0.8
0.4
0
-0.4
-0.8
-1.2
-1.6
-0.002
0.002
0.006
0.01
0.014
0.018
0.022
0.026
mg/L


Normal Probability Plot
ZINC
Shapiro-Wilk W=.91486, p<.4474
Expected Normal Value
1.6
1.2
0.8
0.4
0
-0.4
-0.8
-1.2
-1.6
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
mg/L

Attachment 4

Baseline and ACL Calculations for Nitrate + Nitrite

Foth & Van Dyke
Memorandum

November 20, 2000

TO: Steve Donohue

CC: Master File

FR: Steve Lehrke SL

RE: Crandon Project - Calculated ACL for Overburden NO_2+NO_3

The background concentration and alternative concentration limit (ACL) has been calculated for nitrate+nitrite in the overburden wells. This well list includes CMC-08, CMC-08P, CMC-09, CMC-09P, CMC-10, and CMC-10P. The methodology for calculating the ACL and background concentration is the same as that provided in the November 20, 2000 Foth & Van Dyke memorandum regarding the Crandon Project weathered bedrock groundwater baseline concentrations and PAL/ACL calculations.

The ACL for NO_2+NO_3 in the overburden wells is 2.5 mg/L. The background concentration (calculated as the upper 95% confidence limit) is 1.12 mg/L. Historical sample data for the overburden wells, along with the statistical results, are presented in Attachment A.

Attachment A

Historical Data and Statistical Results

Overburden Water Quality Results
Nitrate + Nitrite (mg/L)

Location	Date	Result
CMC-08	05/15/1994	1.47
CMC-08	06/13/1994	1.31
CMC-08	07/12/1994	1.37
CMC-08	08/15/1994	1.66
CMC-08	09/13/1994	1.2
CMC-08	10/11/1994	1.4
CMC-08	12/14/1994	1.1
CMC-08	01/09/1995	1.3
CMC-08P	05/15/1994	1.94
CMC-08P	06/13/1994	1.97
CMC-08P	07/12/1994	2.08
CMC-08P	08/16/1994	2.39
CMC-08P	09/13/1994	2.3
CMC-08P	10/11/1994	2.3
CMC-08P	12/14/1994	2.5
CMC-08P	01/09/1995	2.6
CMC-09	05/18/1994	1.19
CMC-09	06/16/1994	1.18
CMC-09	07/14/1994	1.19
CMC-09	08/17/1994	1.18
CMC-09	09/14/1994	0.8
CMC-09	10/12/1994	0.8
CMC-09	12/15/1994	0.5
CMC-09	01/10/1995	0.5
CMC-09P	06/17/1994	< 0.5
CMC-09P	07/15/1994	0.734
CMC-09P	08/17/1994	< 0.5
CMC-09P	09/14/1994	< 0.5
CMC-09P	10/12/1994	< 0.5
CMC-09P	12/15/1994	< 0.5
CMC-09P	01/10/1995	< 0.5
CMC-09P	02/06/1995	< 0.5
CMC-10	05/17/1994	0.872
CMC-10	06/16/1994	0.901
CMC-10	07/13/1994	0.916
CMC-10	08/18/1994	0.908
CMC-10	09/13/1994	< 0.5
CMC-10	10/14/1994	< 0.5
CMC-10	01/10/1995	< 0.5
CMC-10P	05/17/1994	0.728
CMC-10P	06/16/1994	0.731
CMC-10P	07/13/1994	0.757
CMC-10P	08/18/1994	0.982
CMC-10P	09/13/1994	< 0.5
CMC-10P	10/14/1994	< 0.5
CMC-10P	12/12/1994	< 0.5
CMC-10P	01/10/1995	< 0.5

Overburden Statistical Results
Nitrate + Nitrite (mg/L)

# Samples		47
% Detections		70.2%
Minimum	<	0.5
Maximum		2.6
Average (1)		1.006
St. Dev.(1)		0.710
Background (Upper 95% C.L. On Mean (2))		1.12
NR 140 PAL		2
# PAL Exceedances		6
NR 140 ES		10
# ES Exceedances		0
Min. Increase		
Calculated ACL (3)		2.5

(1) All non-detected values replaced with one-half the detection limit as per the May 6, 1994 WDNR guidance document "PAL/ACL Calculations Guidance For Solid Waste Facilities"

(2) Calculated as per the Feb. 1989 USEPA interim guidance document, and July 1992 USEPA addendum to interim guidance document "Statistical Analysis of Groundwater Monitoring Data at RCRA Facilities." If at least 85% values were above the detection limit, the average was calculated replacing non-detects with one-half the detection limit. If between 50% and 85% of the values were above the detection limit, the average was calculated using Cohen's adjustment. If less than 50% of the values were detected, the median is given.

(3) Calculated as per the May 6, 1994 WDNR guidance document "PAL/ACL Calculations Guidance For Solid Waste Facilities"