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Crandon Mining Company

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Jerome D. Goodrich, Jr.
PRESIDENT

October 25, 1995

Mr. Bill Tans
Wisconsin Department of Natural Resources
Bureau of Environmental Analysis and Review
P.O. Box 7921
Madison, WI 53707

Mr. David L. Ballman, Ecologist
U.S. Army Corps of Engineers
St. Paul District
190 Fifth Street East
St. Paul, MN 55101

Dear Mr. Tans and Mr. Ballman:

Re: Crandon Project - *Tailings Management Area Feasibility Report/Plan of Operation*

Crandon Mining Company (CMC) is pleased to provide you with the enclosed updates to its *Tailings Management Area Feasibility Report/Plan of Operation*. The updates have been prepared on behalf of CMC by Foth & Van Dyke and Associates Inc. As noted on the attached distribution list, CMC has distributed the enclosed 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 to their appropriate staff members.

Also attached is a reference list with information indicating where to insert the revised pages in the three ring notebook for the *Tailings Management Area Feasibility Report/Plan of Operation* previously provided. The list has been prepared to both facilitate insertion of the additional information, and to serve as a log and reference identifying changes made to the document by CMC throughout the permitting process. If subsequent revisions are made, they will be added to the attached list in sequential order and the list will be forwarded with the changes.

If you or your staff have any questions regarding the updates, please contact me at (715) 365-1450.

Sincerely,

Jerome D. Goodrich, Jr.
President
Crandon Mining Company

JDG:mld2

MLD2\93C049\REPORTS\L-TMASB2\10000

Telephone: (715) 365-1451

FAX: (715) 365-1457

Crandon Mining Company

7 N. BROWN ST., 3RD FLOOR
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Jerome D. Goodrich, Jr.
PRESIDENT

May 30, 1995

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Wisconsin Department of Natural Resources
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Mr. David L. Ballman, Ecologist
U.S. Army of Corps of Engineers
St. Paul District
190 Fifth Street East
St. Paul, MN 55101

Dear Mr. Tans and Mr. Ballman:

Re: Crandon Project - Tailings Management Area Feasibility Report/Plan of Operation

Crandon Mining Company (CMC) is pleased to file the enclosed report titled *Tailings Management Area Feasibility Report/Plan of Operation* for the Crandon Project, together with supporting appendices. The report has been prepared on behalf of CMC by Foth & Van Dyke and Associates Inc. As noted on the attached distribution list, CMC has distributed the document 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.

The purpose of the report is to provide details relating to the plans and methods CMC intends to use to construct, operate and monitor the Crandon Project tailings management area. The report has been submitted to the WDNR and USCOE simultaneously with selected portions of the Project's Environmental Impact Report (EIR) and the Mine Permit Application (MPA). The report has been provided at this time to allow the WDNR and USCOE to begin their analysis of the proposed facility and to allow work on the preparation of the project's environmental impact statement to begin.

If you or your staff have any questions regarding the report, please contact me at (715) 365-1450.

Sincerely,

Jerome D. Goodrich, Jr.
President
Crandon Mining Company

JDG:lxb
Enclosures

Telephone: (715) 365-1451

FAX: (715) 365-1457

Crandon Mining Company

7 N. BROWN ST., 3RD FLOOR
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Jerome D. Goodrich, Jr.
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February 19, 1996

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Mr. David L. Ballman, Ecologist
U.S. Army Corps of Engineers
St. Paul District
190 Fifth Street East
St. Paul, MN 55101

Dear Mr. Tans and Mr. Ballman:

Re: Crandon Project - *Tailings Management Area Feasibility Report/Plan of Operation*

Crandon Mining Company (CMC) is pleased to provide you with the enclosed update to Section 10, Alternative Design, Location, and Operations Evaluations NR 182.08, of its *Tailings Management Area Feasibility Report/Plan of Operation* (TMA). The update has been prepared on behalf of CMC by Foth & Van Dyke and Associates Inc. As noted on the attached distribution list, CMC has distributed the enclosed 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 to their appropriate staff members.

The pages contained in this update need to be inserted into the TMA according to Items 12 through 19 on the attached reference list. This list serves as a log and reference identifying changes made to the EIR by CMC throughout the permitting process. Items 1 through 11 on the list were previously submitted in October 1995. If additional revisions are made, they will be added to the attached list in sequential order and the list will be forwarded with the changes.

If you or your staff have any questions regarding the update, please contact me at (715) 365-1450.

Sincerely,

Jerome D. Goodrich, Jr.
President
Crandon Mining Company

JDG:mld2

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Telephone: (715) 365-1451

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Crandon Mining Company

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May 15, 1997

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Mr. David L. Ballman, Ecologist
U.S. Army Corps of Engineers
St. Paul District
190 Fifth Street East
St. Paul, MN 55101

Dear Mr. Tans and Mr. Ballman:

Re: Crandon Project - *Tailings Management Area Feasibility Report/Plan of Operation*

Crandon Mining Company (CMC) is pleased to provide you with the enclosed update to Section 6.17, Design Performance Assessment, of its *Tailings Management Area Feasibility Report/Plan of Operation* (TMA). The update has been prepared on behalf of CMC by Foth & Van Dyke and Associates Inc. As noted on the attached distribution list, CMC has distributed the enclosed 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 to their appropriate staff members.

The pages contained in this update need to be inserted into the TMA according to Items 20 through 25 on the attached reference list. This list serves as a log and reference identifying changes made to the EIR by CMC throughout the permitting process. Items 1 through 19 on the list were previously submitted. If additional revisions are made, they will be added to the attached list in sequential order and the list will be forwarded with the changes.

If you or your staff have any questions regarding the update, please contact me at (715) 365-1450.

Sincerely,

D. E. Moe

Don Moe
Technical/Permitting Manager
Crandon Mining Company

MLD2\93C049\GBAPP\47387.61\10000

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Log of Updates and Additional Information

Entry Number	Date Revision Issued	Page(s)	Document/ Section Number	Description
1	10/25/95	2	Log of Updates	Addition (Place Behind Cover Letter)
2	10/25/95	3 - 8	Distribution List	Updated (Place Behind Log of Updates and Additional Information)
3	10/25/95	i	Title Page	Updated (Place Behind Distribution List)
4	10/25/95	xi & xiii	Table of Contents	Insert Updated Pages
5	10/25/95	4 & 5	Table 1-1	Insert Updated Pages
6	10/25/95	33	Section 4.1.1.1	Clarification Re: Waste Rock
7	10/25/95	75	Section 6.2.2.1	Clarification Re: Transport Methods to TMA
8	10/25/95	167, 167a & b	Section 6.17.1	Insert Updated Section
9	10/25/95	167c & d	Table 6.17-1	Insert New Table
10	10/25/95	168	Section 6.17.1	Insert New Page Which Deletes Last Two Paragraphs of May, 1995 Version of Section 6.17.1
11	10/25/95	N-1 - N-32	Appendix N	Delete May 1995 Version of Appendix N. Replace with Updated Version.

Crandon Project Tailings Management Area Feasibility Report/Plan of Operation
Log of Updates and Additional Information

Entry Number	Date Revision Issued	Page(s)	Document/ Section Number	Description
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8	10/25/95	167, 167a & b	Section 6.17.1	Insert Updated Section
9	10/25/95	167c & d	Table 6.17-1	Insert New Table
10	10/25/95	168	Section 6.17.1	Insert New Page Which Deletes Last Two Paragraphs of May, 1995 Version of Section 6.17.1
11	10/25/95	N-1 - N-32	Appendix N	Delete May 1995 Version of Appendix N. Replace with Updated Version.
12	02/19/96	2 - 3	Log of Updates	Updated
13	02/19/96	4 - 9	Distribution List	Updated
14	02/19/96	viii - xiii	Table of Contents	Updated
15	02/19/96	197	Section 10.1.4	Renamed Former Section 10.2
16	02/19/96	198 through 198g	Section 10.2	Insert New Section

Log of Updates *(Continued)*

Entry Number	Date Revision Issued	Page(s)	Document/ Section Number	Description
17	02/19/96	198h	Sections 10.3.1 and 10.3.2	Located Sections on New Page
18	02/19/96	—	Figures 10.2-1 through 10.2-4	Insert New Figures
19	02/19/96	—	Appendix P-1	Insert Appendix P-1 Update of Crandon Project Biological Function Model
20	5/15/97	3	Log of Updates	Updated
21	5/15/97	4-9	Distribution List	Updated
22	5/15/97	xiii	Table of Contents	Updated reference for Appendix N
23	5/15/97	167 through 167d	Section 6.17	Updated design performance evaluation
24	5/15/97	228	References	Added reference
25	5/15/97	—	Appendix N Title Sheet	Replace updated title sheet, remove entire appendix from document

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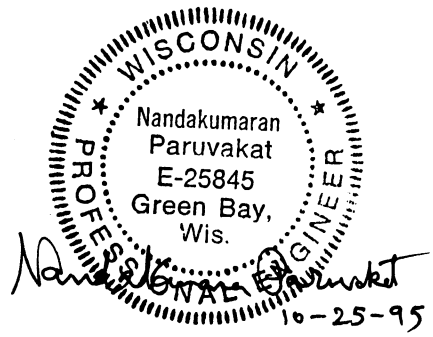
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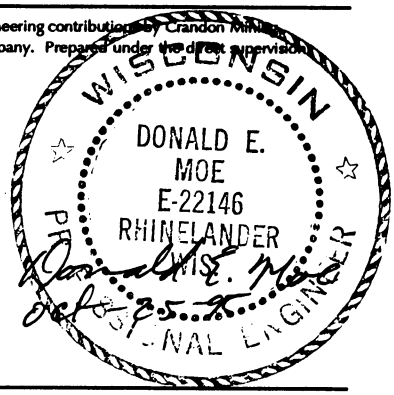
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Tailings Management Area Feasibility Report/ Plan of Operation for the Crandon Project

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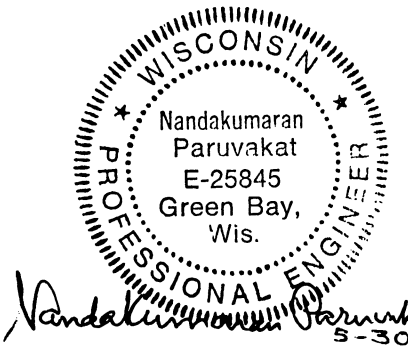
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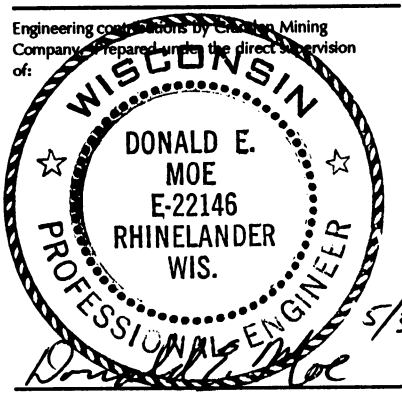
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Tailings Management Area
Feasibility Report/Plan of Operation
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Figure 10.3-1	Flow Chart Showing Approach to Acid Rock Drainage Control
Figure 10.3-2	Alternate Sites for Sub-aqueous (Underwater) Disposal

Drawings

Drawing 1	Title Sheet
Drawing 2	Existing Conditions - Crandon Project
Drawing 3	Existing Conditions - TMA
Drawing 4	Groundwater Table Contour Map October, 1994
Drawing 5	Geologic Cross Section Location Map
Drawing 6	Geologic Cross Section F"-F"'
Drawing 7	Geologic Cross Section G"-G"'
Drawing 8	Geologic Cross Section H"-H"'
Drawing 9	Geologic Cross Section J"-J"'
Drawing 10	Geologic Cross Section K"-K"'
Drawing 11	Geologic Cross Section L"-L"'
Drawing 12	Geologic Cross Section O"-O"'
Drawing 13	Site Sequencing Plan
Drawing 14	Subbase Grades
Drawing 15	Base Grades

Drawing 16	Piping Plan
Drawing 17	Top of Till Grades and Tailings Distribution Piping
Drawing 18	Final Grades
Drawing 19	TMA-1, Stage I Phasing Plan
Drawing 20	TMA-1, Stage II Phasing Plan
Drawing 21	TMA-1, Final Grades and TMA-2, Base Grades
Drawing 22	Drainage Plan
Drawing 23	Engineering Cross Sections
Drawing 24	Engineering Cross Sections
Drawing 25	Engineering Cross Sections
Drawing 26	Engineering Cross Sections
Drawing 27	Engineering Cross Sections
Drawing 28	Details
Drawing 29	Details
Drawing 30	Details
Drawing 31	Details

Appendices

Appendix A	Mine Waste Volume Calculation
Appendix B	Memoranda From Steffen Robertson, and Kirsten, USA Regarding Tailings Laboratory Physical Testing
Appendix C	Summary of Soil Laboratory Test Data from Exxon 1985 Feasibility Report
Appendix D	Geomembrane Design Calculations
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Appendix G	Perforated Leachate Pipe Strength Analysis
Appendix H	Leachate Sump Size Determination and Pump Sizing
Appendix I	Side Slope Riser (SSP) Pipe Strength Analysis
Appendix J	Leachate Volume Calculations for Post-Closure Conditions
Appendix K	Surface Water Management Design Calculations Including TR55 hydrology and POND 2 Models
Appendix L	Revised Universal Soil Loss Equation (RUSLE) for Final Cover and Exterior Berm Slopes
Appendix M	Revised Universal Soil Loss Equations for Interior Berms
Appendix N	Tailings Management Area Groundwater Quality Performance Evaluation
Appendix O	Tailings Management Area (TMA) Construction Quality Assurance Plan (CQA Plan)
Appendix P	Tailings Management Area and Appurtenances - Typical Assignment Operations Log
Appendix P-1	Update of the Crandon Project Biological Function Model
Appendix Q	Crandon Project Pyrite Marketing Update, April 1995, Market Access International

1 Introduction

The Crandon Mining Company (CMC), Crandon Project, *Tailing Management Area Feasibility Report and Plan of Operation* (FR) is submitted in compliance with Wis. Adm. Code NR 182, which regulates the location, design, construction, operation, maintenance, closure and long-term care of the Tailings Management Area (TMA) for CMC's proposed underground zinc-copper mine in Forest County, Wisconsin.

The TMA is designed to contain the wastes generated by the mining and milling of metallic ores from the Crandon deposit which cannot practically be returned to the underground mine as backfill. The TMA has been sited and designed, and will be operated to meet the performance objectives of providing an environmentally-secure facility which meets all applicable state and federal standards for groundwater, surface water, and air quality. The location of the TMA is chosen to avoid, minimize, and mitigate impacts to the surrounding environs. The proposed design applies contemporary technologies for mine waste containment facilities, mine waste handling and site reclamation.

Under §144.441(2)(c), CMC and its successors will have perpetual responsibility for the "long-term care" (defined as the "routine care, maintenance and monitoring") of the TMA. Also per §§107.30 - 107.32, CMC or its successors will be perpetually liable for any damages caused by environmental impacts from the TMA. The TMA has been sited and designed with these responsibilities in mind.

The FR is structured so that the sections, subsections, tables, figures, and appendices track specific sections of applicable regulations. Additionally, Table 1-1 contains a technical review checklist addressing applicable regulations. This FR is one of several documents being submitted to the Wisconsin Department of Natural Resources (WDNR) and the United States Army Corps of Engineers (USCOE) as part of the permitting of the Crandon Project.

This report provides the information necessary to satisfy the requirements of Wis. Adm. Code Chapters NR 182 and NR 132 and other regulations governing the siting, design, and operation of the TMA. These regulations are listed in Section 1.5 of this report. Some of the requirements of NR 182 and NR 132 are addressed in other documents being submitted concurrently with this document. Project documents including the Mine Permit Application (MPA), and the Environmental Impact Report (EIR) contain data referenced herein. In accordance with Wis. Adm. Code NR 132.08, the reclamation of the TMA facility is described in Section 5 of the project's MPA.

The public, municipalities, WDNR, and other interested parties have been informed through various meetings and previously submitted documents of the preliminary TMA design concepts, and their questions and concerns have been addressed in this report.

This report uses and references information from the December 1985 report titled *Crandon Project Mine Waste Disposal Facility, Feasibility Report, NR 182.08* (EMC, 1985), and other documents listed in the Notification of Intent to Collect Data and Detailed Scope of Study (NOI/SOS), prepared by Foth & Van Dyke, 1994.

Table 1-1

Wisconsin Administrative Code NR 182 Technical Review Checklist

Wisconsin Administrative Code Citation	Item/Description	Report Name/Location of Information (Section Figure or Table Number)
NR 182.02	Applicable standards	FR - Section 1.5
NR 182.05	Review fee	FR - Section 1.4.1
NR 182.06	General submittal requirements	FR - Section 1.4
NR 182.07	Location criteria	FR - Section 3.1
NR 182.075	Groundwater standards	-----
(1)	Groundwater quality	FR - Section 5.3, EIR - Section 3.6
(1)(a)	Compliance with groundwater standards	FR - Sections 5.3.2 and 6.17
(1)(a)	Alternative groundwater quality standards	EIR - Section 3.6.5
(1)(b)	Compliance boundary	FR - Section 6.17, EIR - Section 3.6.5
(1)(c)	Contingency plan	FR - Section 7.6, MPA - Sections 4.12 and 8
(1)(d)	Parameters for groundwater analysis and intervention	MPA - Section 7.2
(1)(e)	Non-compliance actions	FR - Section 7.6, MPA - Section 8
(2)	Groundwater quantity	FR - Section 5.3, EIR - Section 3.6.4
NR 182.08	Feasibility Report	-----
(1)	Determination of required approvals	FR - Section 1.4
(2)(a)	General facility information	FR - Section 1.7
(2)(b)	Waste characterization and analysis	FR - Section 4.0
(2)(b)3.	Chemical, radiologic and mineralogic analysis of the waste	FR - Section 4.2
(2)(b)3.	Physical properties of the waste	FR - Section 4.2
(2)(b)3.	Acid generation potential of the waste(s)	FR - Section 4.2
(2)(b)3.	Leaching potential of the waste(s)	FR - Section 4.2
(2)(b)3.	Alternative methods for disposal reuse, sale, recovery or processing of wastes	FR - Section 10.3
(2)(b)4.	Waste characterization related to handling storage and disposal	FR - Section 6.2
NR 182.08(2)(c)	Regional information	FR - Section 5.1
NR 182.08(2)(d)	Site specific information	FR - Section 5.2
(d)1.	Existing condition plan	FR - Drawing No. 3 and Section 5.2.2

Table 1-1 *(Continued)*

Wisconsin Administrative Code Citation	Item/Description	Report Name/Location of Information (Section Figure or Table Number)
(d)2,3,and 4.	Geotechnical investigation	FR - Section 5.2
(d)4 and 5	Field and laboratory testing	FR - Section 5.2.4
(d)6.	Groundwater table observation wells and piezometers	FR - Section 5.2.5
(d)7.	Geological cross-sections	FR - Section 5.2.6 and Drawings No. 5 through 12
(d)7.	Groundwater table map	FR - Section 5.27 and Drawing No. 4
NR 182.08(2)(d)8.	Site-specific environmental information	
(d)8	Environmental characterization	FR - 5.4.1, EIR - Section 3
(d)8	Baseline monitoring program	FR - 5.4.2, MPA - Section 7.2
(d)8	Land use mapping	FR - 5.4.3, EIR - Section 3.12
(d)8	Groundwater monitoring	FR - 5.4.2 and MPA - Section 7
(d)8	Existing surface water quality	FR - 5.4.3, EIR - Section 3.7
(d)8	Local climatological data	FR - 5.4.4, EIR - Section 3.4
NR 182.08(2)(e)	Proposed facility design	FR - Section 6.0
NR 182.08(2)(f)	Water budget	FR - Section 6.7
NR 182.08(2)(g)	Aesthetics	FR - Section 6.11, EIR - Section 3.13
NR 182.08(2)(h)	Dam safety factors	FR - Section 6.15
NR 182.08(2)(i) and NR 182.075(1)(c)	Contingency plan	FR - Section 7.6, MPA - Sections 4.12 and 8
NR 182.08(2)(j)	Closure and long-term care costs	FR - Section 8.1, MPA - Section 5.12
NR 182.08(2)(k)	Alternative design, location and operations	FR - Section 10.0
NR 182.08(2)(l)	Appendix	FR - Appendix A through Q
(2)(l)	References	FR - Section 11.0
NR 182.09(1)	Plan of operation	FR - Section 7.0
NR 182.09(2)(a)	Engineering drawings	
	• Title sheet	FR - Drawing No.1
	• Existing condition plan	FR - Drawing No. 2 and 3
	• Base grade plan sheet	FR - Drawing No. 15
	• Engineering modifications plan sheet	FR - Drawing No. 17
	• Final site topography	FR - Drawing No. 18
	• Phasing plan sheets	FR - Drawings No. 19, 20, and 21

Table 1-1 *(Continued)*

Wisconsin Administrative Code Citation	Item/Description	Report Name/Location of Information (Section Figure or Table Number)
	• Site monitoring plan	MPA - Section 7
	• Long-term care	MPA - Section 10
	• Site cross sections	FR - Drawings No. 5 through 12
	• Site engineering cross sections	FR - Drawings No. 23 through 27
	• Details and typical sections	FR - Drawings No. 28 through 31
NR 182.09(2)(b)	Operations manual	
(2)(b)2.	Specifications for site construction	FR - Section 7.2
(2)(b)2.	Specifications for site operation	FR - Section 7.3
	• Typical daily operations	FR - Section 7.3 and Appendix P
	• Development of subsequent phases	FR - Section 6.12
	• Site closing	FR - Section 7.3.9
	• Long-term care	FR - Section 7.3.10 and MPA Section 5.72
NR 182.08(2)(j) and NR 182.09(2)(b)2.g.	• Cost estimate for closure and long-term care	FR - Section 8.1 and MPA Section 5.12
(2)(c)	Design report	
(2)(c)	• Operating system	FR - Section 6.2
(2)(c)	• Liner design	FR - Section 6.3
(2)(c)	• Leachate management system	FR - Section 6.4
(2)(c)	• Embankment configuration	FR - Section 6.5
(2)(c)	• Final cover	FR - Section 6.6
(2)(c)	• Surface water management	FR - Section 6.9
(2)(c)	• Traffic routing	FR - Section 6.10
(2)(c)	• Security and screening	FR - Section 6.11, EIR - Section 3.13
NR 182.09(2)(c)	• Environmental monitoring	FR - Section 6.14, MPA - Section 7
(2)(b)	• Site development and phasing	FR - Section 6.12
(2)(c)	• Earthwork balance and stockpiling	FR - Section 6.12.3
NR 182.08(2)(e)	• Annual materials balance	FR - Section 6.13
NR 182.08(2)(d)	Contingency plan	MPA - Section 8
NR 182.09(2)(e)	Documentation and agreements	FR - Section 6.16
NR 182.10	Construction and completion reports	FR - Section 7.2
NR 182.11(1)	Minimum design and operation requirements	FR - Section 6.18

Table 1-1 *(Continued)*

Wisconsin Administrative Code Citation	Item/Description	Report Name/Location of Information (Section Figure or Table Number)
NR 182.11(2)	Application of parameters and concepts recommended by the department in NR 182.11(2)(a) through (g)	FR - Section 6.18
NR 182.11(3)	Priorities for mine, mill and disposal facility design and operation	FR - Section 6.18
NR 182.12	Inspections	FR - Section 7.5
NR 182.13	Monitoring	FR - Section 6.14, MPA - Section 7
NR 182.135	Requirements for certified or registered laboratory	MPA - Section 7
NR 182.14	Record keeping and reporting	FR - Section 7.4
NR 182.15	Closure and reclamation plan	FR - Section 8.0 and MPA - Section 5.7
NR 182.16	Financial responsibility for closure	FR - Section 9.1, MPA - Section 3
NR 182.17	Financial responsibility for long-term care	FR - Section 9.2, MPA - Section 3
NR 182.18	Waste management fund	FR - Section 9.3
NR 182.19	Exemptions and modifications	MPA - Section 2.3

EIR - Crandon Mining Company proposed Crandon Project - Environmental Impact Report, 1995, Foth and Van Dyke & Associates, Inc.

MPA - Crandon Mining Company: *Crandon Project - Mine Permit Application*, 1995, Foth & Van Dyke and Associates Inc.

FR - Crandon Mining Company: *Tailings Management Area Feasibility Report/Plan of Operation for the Crandon Project,* 1995, Foth & Van Dyke and Associates Inc.

Prepared by: REM
Checked by: NXP

1.1 General NR 182.06 Submittal Requirements

1.1.1 Review Fee NR 182.06(1)(a)

A plan review fee in the form of a check for the amount of $9,000 has been submitted by CMC to the WDNR under separate cover to satisfy the requirements of NR 182.05.

1.1.2 Desired Department Action and Distribution NR 182.06(1)(b) and (c)

The cover letter to this report outlines the requested action or response to this document from the WDNR. Also attached to the cover letter is a distribution list of parties to which copies of this report were forwarded.

1.1.3 Other Requirements NR 182.06(1) 1, 2 and 3

The report, appendices and drawings, submitted herein, were prepared under the supervision of Dr. Nandakumaran Paruvakat, a registered Professional Engineer in the State of Wisconsin. All investigative methods used are listed in the appropriate appendices, and nonstandard methods are explained in this report. The format for the report, drawings and appendices complies with the requirements of NR 182.06(1) 3.a. and b.

1.1.4 Cross Referencing of Reports and Applications NR 182.06(3)

Portions of the information required by NR 182, NR 132, and other administrative codes and/or statutes referenced therein is contained in other reports or applications which have been or will be submitted to the WDNR. The feasibility report and plan of operation are combined in this report and other reports or applications are cross-referenced without repetition of the required information. The referenced reports or applications are listed in Section 11 (References) of this report.

1.2 Applicable Standards

This report was prepared under Wis. Adm. Code NR 182 (Regulation of Metallic Mining Wastes) specifically to satisfy the requirements of NR 182.08 (Feasibility Report) and NR 182.09 (Plan of Operation). NR 182 incorporates references to other sections of the Wis. Adm. Code and Wisconsin State Statutes, including the following:

- Ch. NR 102-106 Wis. Admin. Code - Surface Water Classification and Protection
- Ch. NR 132 Wis. Admin. Code - Metallic Mineral Mining
- s.144.391 Statutes and Ch. 400-499 Wis. Admin. Code - Air Quality
- WPDES
- s.147.07 Statutes - Toxic and Pretreatment Effluent Standards
- s.144.855 Statutes and Ch. NR 140 Wis. Admin. Code - Groundwater Quality

Other regulations and standards governing aspects of the TMA include the following:

- U.S. Department of Transportation, Federal Aviation Administration (FAA) Order 5200.5A, waste disposal on or near airports.

- Environmental Protection Agency (EPA) 40 CFR, Part 122, storm water discharge associated with industrial activities.
- U.S. Army Corps of Engineers (USCOE), Section 404 of the Clean Water Act Dredge and Fill Permit.
- Clean Air Act 40 CFR 51.18.
- Various local governmental land-use regulations and zoning ordinances.

1.3 Technical Review Checklist

In order to expedite review, a checklist has been prepared to direct the reviewer to specific sections developed to address specific Wis. Admin. Code Sections.

Per NR 182.06(3) this FR cross-references other Crandon Project submittals, e.g. the MPA, EIR, etc. Table 1-1 has been tabulated to assist the reviewer in locating cross-references in these submittals. In addition Table 1-1 provides a checklist for the feasibility report and plan of operations requirements.

1.4 General Facility Information NR 182.08(2)(a)

- Project: Crandon Mining Company, Tailings Management Area (TMA)

- Primary Contact: Jerome D. Goodrich, Jr., President
  Crandon Mining Company
  7 North Brown Street, Third Floor
  Rhinelander, WI 54501-3161
  Phone: (715) 365-1450

- Owner: On September 13, 1993, subsidiaries of EMC and Rio Algom Limited formed Crandon Mining Company (CMC), a Wisconsin general partnership, with the intent to permit, develop, operate, and reclaim the Crandon Project.

- Site Location: The TMA is located primarily in Section 32, T35N, R13E, Town of Lincoln and in a small part of the south half of Section 29, T35N, R13E, Town of Lincoln. All affected property is either owned or under purchase option by CMC.

- Proposed Licensed Acreage: 219.5 acres within the TMA cells waste limits (355 acres total disturbance) as follows:
  TMA 1 - 53.5 acres (ac)
  TMA 2 - 70.7 ac
  TMA 3 - 45.9 ac
  TMA 4 - 49.4 ac

- Proposed Facility Site Life: Total TMA Site Life - 35 Years
  Initial Construction - 3 Years
  Active Life - 28 Years
  Final Reclamation Construction - 4 Years

- Estimated Waste Types (Refer to Section 4.1 for detailed descriptions):

  Finely-ground rock referred to as tailings
  Crushed/broken Type I and Type II waste rock
  Reclaim basin solids
  Wastewater Treatment Plant solids
  Laboratory wastes

- Disposal Capacity (tailings including a 13 to 15 percent contingency, refer to Section 4.1):

Zinc Tailings	9.75 Million (M) cubic yards (cy)
Copper Tailings	7.36 Mcy
Type I Waste Rock	0.03 Mcy
Type II Waste Rock	0.07-0.32 Mcy
Wastewater Treatment Solids	.70 Mcy
Laboratory Wastes	-- *
Contingency	2.3-2.55 Mcy
Total	20.46 Mcy

*Less than 300 cubic yards

- Mode of Operations: Tailings and related mine waste monofill. The tailings and the other solids will be pumped as a slurry by a pipeline to the TMA cells. Tailings will be transported and placed hydraulically into the lined basin by means of spigot lines located at the top of the active cell. Type II waste rock will be hauled by trucks and dumped in the facility or used as riprap for construction within the lined area. Type I waste rock will be hauled by trucks for use either inside or outside the lined area as required for construction purposes.

2 General Project Description

The main elements of the Crandon Project consist of an underground mine; ore concentrating facilities; water treatment facilities; a tailings management area; a water discharge pipeline and ancillary facilities such as an access road, a railroad spur line, and service and support facilities. An extensive description of the project is included in the Mine Permit Application. Following is a brief overview describing the entire project. The overview discusses the location of the mining facilities, the geology of the ore body, the mining process and the major project components which will be developed to operate and reclaim the proposed project in a manner which protects public health, safety, and the environment. The relative location of the project is shown on Figure 2-1.

The anticipated rate of production, project life and projected employment requirements for the project are shown in Table 2-1. As with any industrial operation, the life of the facility could change based on economic conditions.

Table 2-1

Anticipated Production and Operation Data

Daily Ore Production	5,500 tons
Annual Ore Production	2,000,000 tons
Total Ore Production	55,000,000 tons
Total Estimated Project Life	35 years
Preproduction	3 years
Mining	28 years
Reclamation	4 years
Production Schedule	7 days/week
Employment (estimates)	
Construction (Peak)	750
Operations	402-526

Prepared by: PAE
Checked by: JWS

Within this report, there are numerous references to the "project area", "mine site", "plant site" and the "tailings management area". These terms have specific meanings as follows.

- Project Area - The project area is defined by the boundaries delineated on Figure 2-2.

- Mine Site - The mine site is defined by the limits of disturbance of project facilities within the project area.

- Plant Site - The plant site is generally defined as the area within the mine site that includes all mining, processing, concentrating, water treatment, administrative, and storage facilities; portions of the railroad spur in the vicinity of the plant site; portions of the access road in the vicinity of the plant site; ventilation raises; and the project's water supply well and its accompanying pipeline corridor. The plant site also includes all surface water runoff and storage basins constructed in its vicinity.

- Tailings Management Area (TMA) - The TMA is defined as the area within the "mine site" that includes the project's four ta'ilings cells and berms, the reclaim pond, the tailings and reclaim water pipeline and access road corridors, and contiguous borrow and storage areas. The TMA also includes all surface water drainage facilities constructed in its vicinity.

Two additional areas located outside of the project area include the narrow corridor from the intersection of the site access road and State Trunk Highway (STH) 55 to the Wisconsin River in which the project's treated water discharge line is to be located, and the project's wetland mitigation site located off-site in Shawano and Oconto Counties. Design information for the discharge pipeline to the Wisconsin River is included as part of the water treatment system engineering report prepared pursuant to Wisconsin Administrative Codes. Design information relative to the wetland mitigation site is included as part of the Federal Clean Water Act Section 404 permit application. For completeness the description of the environmental aspects associated with these areas are included in the project's EIR.

The boundaries of the project area, plant site and TMA are shown on Figure 2-2. The plant site and the TMA are approximately 128 acres and 355 acres in size, respectively. The total area of disturbance, including the access road and railroad spur, is approximately 550 acres.

2.1 Site Location

The Crandon ore body is located in Forest County, Wisconsin. The civil land survey location is Section 25, Township 35 North, Range 12 East, Town of Nashville, and Section 30, Township 35 North, Range 13 East, Town of Lincoln. The project area is located five miles south of the City of Crandon and two miles east of STH 55 and the Mole Lake Indian Reservation. The plant site is approximately one-quarter mile north of Little Sand Lake and one mile south of Swamp Creek. The primary mine surface facilities, e.g., plant site, will be located north of the ore body. The proposed plant site layout is shown in Figure 2-3. Access to the plant site will be along a new access road from STH 55 northwest of the site. A railroad spur line serving the site will be connected northeasterly to the existing Wisconsin Central Limited Railroad. The project's TMA will be located approximately one mile southeast of the plant site.

The project area shown in Figure 2-2 includes those portions of property which CMC has purchased, leased, optioned for purchase or obtained by easements for use in the development of the plant site, TMA, access roads, railroad spur line and buffer areas.

2.2 Geology

The Crandon deposit is composed of two distinct mineralization types, zinc ore and copper ore. The minerals were deposited during the Precambrian era, about 900 to 2,500 million years ago. The deposit was formed at and just below the ocean floor by mineral bearing fluids of volcanic

origin. Some of the material deposited by this volcanic system was sulfide minerals which accumulated in low spots on the ocean floor. Continued accumulation of other volcanic materials and sediments occurred which buried the sulfide deposit. Deep burial, 33,000 to 50,000 feet, resulted in lithification and metamorphism which hardened and solidified both the host rock and the sulfide deposit over time. Later, a mountain building phase occurred in the region, tilting the volcanic layers and the deposit to a near vertical position. Thereafter, the deposit may have been covered by younger sediments, however weathering and erosion have removed these later rocks. The more recent geological process which has affected the deposit is related to Pleistocene glaciation which left the bedrock buried under unconsolidated glacial overburden deposits.

The Crandon ore body is long and tabular with an approximate width of 100 feet, north-south, and a strike length of 4,900 feet, east-west. Based on the results of drilling, the ore body extends to an approximate depth of 2,200 feet. The interpreted geologic stratigraphy and ore body configuration are shown on Figures 2-4 and 2-5 respectively.

The bedrock in the hanging wall and in the footwall of the ore body consists of a series of fragmental volcanic rocks, fine tuffs (solidified volcanic ash), debris flow (ocean floor and volcanic-derived sediments), breccia (blocky, angular particles), lapilli tuffs (gravel sized volcanic material) and flows. Overlying the bedrock is a sequence of unconsolidated glacial sands, clays, and gravels. The rock in contact with the unconsolidated glacial overburden is weathered to varying degrees. The amount of weathering ranges from simple staining to extreme weathering near the surface which reduced the rock to a clay-like material called massive saprolite. The glacial overburden consists of interbedded and co-mingled glacial till, which is material deposited directly by the glacier, and glacial outwash deposited by streams emanating from the glacier. These glacial deposits are found in various thicknesses in the area ranging from 75 feet to over 250 feet thick.

2.3 Key Project Elements

2.3.1 Mine Development

Access to the mine will be through a main production/service shaft located north of the ore body. The first of two ventilation shafts will be located east of this main production shaft. Underground lateral development drifts will access the ore body from the main shaft at 300 foot vertical intervals. These level development drifts are designed to provide access to the ore body. The lateral extent of a mine level at a given point in time will depend upon the need for access to mining blocks, ore passes, and ventilation raises. A schematic longitudinal section showing a typical main level plan is shown on Figure 2-6.

An underground ramp will also connect mine levels to allow for movement of mobile equipment, supplies, and personnel throughout the mine. This centrally-located ramp is also shown schematically on Figure 2-6.

Mine development will be divided into the following phases:

1) Site preparation and the sinking of the main production shaft and the east ventilation shaft. This phase is expected to take 18 months to complete.

2) The development of the underground ore handling and crushing system, the development of the dewatering system, lateral development into the ore horizons and development of the initial mining blocks (stopes). An internal mobile equipment access ramp will connect the main production ore levels. This phase of mine development is also expected to take 18 months to complete.

As shown on Figure 2-6, mine development and production will begin in areas chosen to avoid weathered bedrock which are expected to be the primary conduits for water inflow into the mine workings.

2.3.1.1 Phase I Development

Phase I development primarily includes simultaneous construction of two vertical shafts in the hanging wall rocks. Each shaft will be concrete lined through the overburden and the weathered subcrop rock. Collar construction through the glacial overburden will include stabilization and hydraulic control by ground freezing or other suitable techniques, followed by the excavation and concrete lining of the shaft into bedrock. If required, inert grout will be pumped under pressure through holes in the collar into the rock and glacial formations to provide a watertight seal. When the collar section of the main shaft and east ventilation shaft are completed, a headframe structure will be erected over each shaft. Conventional shaft sinking by drilling and blasting techniques will then commence at the main shaft and the east shaft concurrently.

During shaft development, it is estimated that the drainage water from both shaft sinkings will be controlled to less than ten gallons per minute (gpm) by grouting. All shaft water will be pumped to the surface water storage ponds.

2.3.1.2 Phase II Development

Because the east shaft is smaller than the main shaft, it will be completed sooner. Upon its completion, horizontal level development will consist of driving a horizontal opening in the hanging wall rock to connect the east shaft to the main shaft (Figure 2-6). After the two shafts are connected, level development can commence to access the ore body.

The underground ore handling facilities will be constructed near the main production shaft during this period. These facilities will consist of: (a) coarse ore and waste rock storage bins, (b) crusher facilities, (c) ore handling systems, and (d) a loadout facility.

2.3.2 Mine Operations

Level development from the main production shaft to the stoping areas will be driven at 300-foot vertical intervals. The primary mining method will be blasthole open stoping with delayed backfill. However, other mechanized variations, such as sublevel mining or cut-and-fill stoping methods may also be used. Stopes (Figure 2-7) will average approximately 300 feet high by 75 feet long, and will vary with the width of the ore body. Ore will be drilled in a stoping block, then blasted and removed. Top hammer or down-the-hole drills will be used to drill approximately four to six inch diameter blastholes on approximately 12-foot by 12-foot center spacing for production stope blasting. Broken ore will be removed from the drawpoints at the bottom of each stope using mechanized mining equipment which will then transfer the ore to the

crushing level below by means of ore pass raises. Primary crushed ore, at a top size of eight inches, will be conveyed to a skip loading pocket and hoisted to the surface.

A typical stope will contain approximately 250,000 tons of ore. At a 2,000,000-ton annual production rate, approximately eight stopes will be mined out each year, which exposes less than five percent of the footwall and hanging wall area of the ore body at any one time. Exact production parameters will be based on the grade of the ore in the mined stope; the mechanical characteristics of the rock in the stoping block; and the potential for inflow of water.

A permanent bridge, or crown pillar, of bedrock directly beneath the glacial overburden will be purposely excluded from mining activity. This bedrock barrier, averaging approximately 100 feet thick, along with the routine backfilling of mined-out stopes, will maintain surface stability and prevent subsidence.

In the uppermost mine levels where the ore and host rock may have been moderately weakened by surficial weathering, mechanized cut-and-fill mining may be employed. This method is commonly used by the industry and involves removal of horizontal lifts of ore of variable thicknesses. The void created by each horizontal mining pass is backfilled prior to mining the next upper lift. Less than 10 percent of the ore body may require use of this mining method.

The planned mining methods provide for backfilling all stopes following ore extraction. These practices, combined with the fact that five to 10 percent of the potentially minable ore will be left in place as pillars throughout the mine, will provide perpetual stability of the mine area bedrock and glacial overburden. Backfilling will also result in the reduction of pathways for water migration as mining progresses.

Mine backfilling will begin with the start of milling operations and after the first stope is depleted. The backfill will consist of mill tailings 10 to 15 micron or greater in size, supplemented with coarse waste rock retained underground. Uncemented tailings backfill will have a hydraulic conductivity on the order of 0.028 feet per day. The hydraulic conductivity of cemented tailings backfill will be lower. Backfill slurry containing approximately 60 to 70 percent solids will be pumped underground through boreholes fitted with distribution pipes. The backfilling operations will normally be conducted to coincide with the mining schedule.

Waste rock material from mine development will be used in the stope backfilling process and will be placed before or during the placement of hydraulic tailings fill. The hydraulically-placed fill will flow into and fill the voids between the rock fragments.

Cement will be added to the backfill when needed to provide stability so that the column of fill will stand unsupported and enable complete removal of the ore in the adjacent stope. Approximately one-third to one-half of the total backfill placed in the mine will contain cement.

2.3.3 Mine Dewatering and Groundwater Inflow Control

Groundwater inflow will vary during the different stages of mine construction and operation. The proposed mining plan for the Crandon Project avoids entry into weathered zones during the initial operations, therefore deferring maximum and steady state inflow rates. During the initial operations, groundwater inflow is expected to be minimal and localized, occurring through isolated bedrock fractures that have limited capacity to move water. During this period,

exploration holes will be advanced into the weathered bedrock areas to dewater them. The water removed in this fashion will primarily be stored water which will be withdrawn at rates that can be effectively managed at the project's water treatment plant. A more detailed description of this process follows.

As mine development progresses upward from the original mining areas (Figure 2-6), diamond drilling techniques will be used to identify active underground water courses prior to advancing the mine face. Diamond drill holes will be used throughout the mine to drain stored water. Flows encountered on the uppermost active mine level will be captured by interceptor drill holes and contained to avoid contamination by mining operations on levels below and to reduce pumping head. A conceptual cross-section of the groundwater interceptor system showing the collection methodology is presented in Figure 2-8. Standard rock grouting techniques, typically using neat cement, may also be used for local inflow control during the early mine years to limit total mine pumpage.

Typically, groundwater interception holes will form conical fans in the weathered rock above the development openings, thus increasing the radius of the drains. As is common practice in other mines, the drill hole collars will be fitted with valves to allow controlled water removal.

Groundwater collected from exploration drilling or other drill holes placed specifically for inflow interception will be routed directly to a clean water sump and pump station near the main shaft in the upper mine level. The collected groundwater will typically be pumped to the water treatment plant on the surface. However, a portion of the water may be retained underground for distribution as mine utility water.

Groundwater seepage that is not captured by the interceptor system will infiltrate the mine workings and ultimately be recovered in the main sumps along with the mine potable, utility and backfill drainage water. Normal mine drainage collection will begin on each mine level where groundwater seepage, utility water and backfill drainage will be ditched to small local sumps excavated in the drift wall. Decant water from the local mine level sumps will be piped or drained through boreholes or ditched to the main mine sumps located adjacent to the production shaft at the lowest level.

The main mine sumps and pump station will generally be arranged as indicated on Figure 2-9. Sumps will consist of downgrade excavations in the wall rock adjacent to the pump station. These will function as pumping reservoirs with an outlet end bulkhead containing the pump suction pipes.

2.3.4 Ore Processing

Ore mined from the Crandon deposit will be physically concentrated at the plant site by adding water to the crushed ore and grinding it to the size of fine sand particles. After grinding, the ore slurry will be pumped to a series of flotation circuits where reagents are added for separating metallic minerals from the ground-up ore. During this process, minerals will be selectively "floated" to the top of the flotation cells and removed. The remaining material, which is called tailings, will be either used as backfill in the mine or hydraulically transported to the TMA. Different flotation circuits require different reagents to concentrate specific individual minerals. A schematic of the ore processing circuits is shown on Figure 2-10. Separate concentrates of

zinc, copper and lead minerals will be recovered by the flotation process. The concentrate from these processes will be thickened and filtered to an eight to 10 percent moisture content.

The tailings will range in size from sand to very fine particles. The coarser tailings from the mineral separation circuits will be used to backfill the mined-out stopes. The finer fraction will be sent to the TMA.

In the TMA, the tailings will settle to the bottom of the lined basin. Excess water will then be pumped from the TMA basin to a reclaim pond for reuse in the ore processing facility. The ore concentration process, TMA, and reclaim pond are designed to operate as a closed circuit. The concentration process normally requires the continuous addition of "makeup" water. Water in this circuit will not require treatment because a discharge will not normally take place. The water treatment system will be designed to treat tailings cell waters for discharge, if necessary.

2.4 Infrastructure

Infrastructure features to support the mine and milling operations include a water treatment plant, ore and waste rock storage facilities, mining waste management facilities, access road, railroad spur line, electric power transmission lines, a natural gas pipeline, a treated water discharge pipeline, and ancillary buildings and storage facilities. A discussion of each follows.

2.4.1 Water Treatment

A water treatment plant will be constructed as part of the project facilities. It will treat mine water and, if needed, process water prior to discharge. Intercepted groundwater is expected to be representative of natural groundwater quality. If the monitoring of this groundwater indicates that the water quality is not suitable for direct discharge, it will be routed through the water treatment plant. Groundwater that bypasses the interceptor system and comes in contact with mining activities will be commingled with other mine drainage water, such as the water used to cool the drill bits while drilling the blast holes. All of these "contact waters" will be routed through the water treatment plant.

The water treatment plant will include a lime and sulfide precipitation system with filtration and pH adjustment. Treatment solids from this facility will be placed along with the ore processing tailings in the TMA. Mine water will be treated to meet WDNR Water Quality Standards before being discharged to the Wisconsin River via a discharge pipeline installed primarily along the U.S. Route 8 corridor (Figure 2-11).

The treatment system is designed with two holding ponds to retain the treated water so it can be sampled prior to discharge. This will ensure all water meets discharge standards, prior to discharge.

Sanitary water will also be generated at the facility. Sanitary water will be handled separately through a package sanitary water treatment plant. The treated effluent from this plant will be pumped to the TMA.

2.4.2 Mining Waste Management

Crandon Project mining wastes will include waste rock, tailings, refuse, water treatment plant solids, and laboratory wastes.

Over 50 percent of the waste rock generated by the project will be left in the mine to be used as backfill for mined-out stopes. Limited quantities of waste rock will be brought to the surface during preproduction and managed as discussed in Section 2.4.3 below. As discussed in Section 2.4.4 below, approximately 50 percent of the tailings generated by the project will be returned to the mine as backfill, with the remaining 50 percent placed in the TMA.

During the mining operations on-site laboratories will be used to conduct metallurgical testing for mining grade control and for production quality assurance testing related to milling operations. Wastes generated from the performance of these tests will be placed in the TMA. Approximately 900 cubic yards of general refuse such as office wastes will be generated at the facility each year during the 35 years of construction, operation and reclamation. The reclaimable portion of this waste will be recycled in accordance with state law. The remaining waste materials will be disposed of by a contractor in an approved off-site landfill.

Solids will be generated from the treatment of project generated waters. These solids will be placed in the TMA with the tailings.

2.4.3 Preproduction Ore/Waste Rock Storage Areas

Two storage areas will be located to the north of the main production shaft to store ore and waste rock hoisted to the surface during pre-production mine development. Prior to the commencement of underground crushing and the start of mill operations, approximately 1,050,000 tons of uncrushed ore and waste rock of a maximum size of 24-inches will be placed on the two separate areas. One storage facility which will be lined will be used to store approximately 350,000 tons of ore and about 100,000 tons of Type II waste rock. The remaining 600,000 tons of Type I waste rock will be deposited on an unlined area located east of the lined storage area. Type I waste rock is material that has a very low potential to leach, while Type II waste rock has a higher leaching potential. The lined area is designated as the preproduction ore storage area on Figure 2-3. The unlined area is referred to as the construction material storage area on the same figure. Both storage areas have been designed to accommodate the maximum potential amount of waste rock and ore hoisted from underground during the preproduction period and will occupy a total area of approximately 18 acres.

For the preproduction ore storage area a central ridge will divide the facility on its north-south axis. Each side of the lined pad will slope away from the center. The pad will be bounded by berms with runoff collection ditches which will route water to a lined water storage basin. The base of the pre-production ore storage area will consist of a compacted layer of existing soil overlain by a geomembrane liner. A till cushion will be placed over the geomembrane. Water from this area will be drained to a water storage basin. The location of the basin is sized to hold the volume of water from a 25-year, 24-hour storm event. Water from this basin will be either pumped to the TMA for use in ore processing or to the project's water treatment plant.

The base of the construction material storage area will consist of a compacted layer of existing on-site soil. The base will be sloped to drain toward another of the site's surface water runoff

basins. Water from this runoff basin will be discharged to natural site drainage ways. Following commencement of mill operations, the ore stored in the preproduction storage area will be processed. Type II waste rock stored in this area will be hauled by truck to the TMA for disposal or for use as riprap for TMA internal sidewalls. Type I waste rock will be used as construction material.

2.4.4 Tailings Management Area

All tailings produced by ore processing that are not used for mine backfill, hoisted Type II waste rock, water treatment plant solids and the small amount of laboratory wastes will be placed in the TMA. The TMA has been designed to provide long-term, environmentally-safe containment. Tailings and treatment plant solids will be pumped to the TMA through a high density polyethylene (HDPE) pipeline. Waste rock will be transported to the TMA by truck.

As shown on Figures 2-12 and 2-13, the TMA will consist of four cells, each of which will be lined and include a leachate collection system. The four cells, referred to as TMA1 through TMA4 will each be constructed and operated in two stages. TMA1 and TMA2 are designed to contain the tailings from processing the zinc ore. TMA3 and TMA4 will be used for the copper ore tailings. The approximate capacities and site lives for each cell are shown in Table 2-2.

Table 2-2

Approximate Tailings Management Area Capacity

TMA Cell	Capacity (in millions of cubic yards)	Approximate Site Life (years)
TMA 1	4.0	6
TMA 2	7.8	10
TMA 3	3.9	6
TMA 4	4.8	6
Total	20.5	28

Prepared by: PAE
Checked by: JWS

TMA cell construction and operation will first involve constructing and filling Stage 1 of TMA1. As the tailings in Stage 1 approach the design elevation, Stage 2 of TMA1 will be built. When approximately one to two years of capacity remain in TMA1, construction of Stage 1 for TMA2 will begin. When TMA1 is full, tailings placement in TMA2 will start. After consolidation, reclamation of TMA1 will begin, while filling in TMA2 progresses. The same process will continue for TMA3 and TMA4.

The tailings slurry will be transported from the concentrator building to the TMA through an approximate 16-inch inside diameter HDPE aboveground pipeline. The location of the pipeline is shown on Figure 2-2. The pipeline will lie above ground in a lined ditch. A 22-foot wide access road will be located next to the pipeline for service and maintenance. Pumps used for pumping the tailings slurry in the pipe will be located in the concentrator building. The pipeline ditch will be sloped to lined sumps located at the plant site and approximately midway between the plant site and the TMA to collect tailings and water in the event of percolation or to provide storage if the pipe must be drained.

The tailings slurry will be deposited in the active TMA cell using spigots. The spigot discharge point(s) will be regularly moved around the inner perimeter of the active cell to facilitate even distribution of tailings and to keep the tailings saturated. The excess water that drains from the slurry after the tailings have settled will flow to an area in the center of the cell and will be pumped to the reclaim pond. Water in the reclaim pond will be retained for a short time and then pumped to the mill for reuse in the process circuit. The tailings operating system is designed to maximize tailings density.

The TMA cells have been designed to meet the standards contained in applicable state statues and administrative codes which are written to protect the public health and welfare. Key TMA design features include:

- An average 43-foot separation from the base of the TMA to groundwater.
- A minimum 1,250-foot separation from the nearest lake or stream.
- A composite liner consisting of a low-permeability soil member and a geomembrane liner.
- A leachate collection system over the bottom of each cell and partially up the interior sidewalls of each cell.
- A reclaimed final cover consisting of the following components from top to bottom.
    - topsoil
    - rooting layer
    - drainage layer
    - geomembrane liner
    - low permeability soil liner
    - grading layer
- Surface water control structures designed to accommodate a 100-year, 24-hour storm event.

2.4.5 Access Roads

A site access road will be constructed from STH 55 to the plant site. A second access road will be constructed from the plant site to the TMA. The site access road will be approximately three miles long and consist of bituminous concrete with gravel shoulders. The treated water discharge line will be buried in the right-of-way of the site access road. The TMA access road will be

approximately one mile long and will be gravel-surfaced. Pipelines for tailings transport and reclaim water will be sited adjacent to the TMA access road in a lined ditch.

2.4.6 Railroad Spur

A 2.7-mile railroad spur line will be constructed from the plant site to the Wisconsin Central Limited Railroad located to the northeast. The spur line will consist of a single track along most of its corridor. A side track will be located near the point where the spur line connects with the main railroad line. The sidetrack will be used for switching and rail car staging. The spur line will be used to bring cement, lime and other materials to the plant, and to ship concentrates to market. Concentrate will be shipped in enclosed cars or containers.

2.4.7 Utilities

Electrical service to the project site will be provided by Wisconsin Public Service Corporation (WPSC) by an electric transmission line constructed between an existing substation near Monico, Wisconsin, and a new substation to be located at the plant site. The substation near Monico will be upgraded by WPSC as part of the extension of electrical power for the project.

The WPSC area distribution system, which will likely be located near the south end of Lake Metonga, will supply natural gas for the project via a pipeline installed to the plant site. The pipeline route will follow existing county roads, cross Swamp Creek north of the plant site, and then follow the main plant access road into the site.

2.4.8 Other Facilities

In addition to the project elements discussed above other site facilities as listed below will be constructed and used as part of the project:

Administrative offices	Surface maintenance shops
Changehouse facilities	Potable water supply and distribution system
Explosive storage areas	Fire protection systems
Gate house	Lubricant storage
Core logging and storage	Bulk fuel storage
Covered storage area	Lay-down areas
Truck weigh scale	Railroad weigh scale
Fencing	Mobile equipment fuel station
On-site roads	Parking areas
Area lighting	Material storage areas

2.4.9 Surface Water Controls

Precipitation falling within the limits of the plant site will be collected and directed to one of a number of water storage basins. Contact runoff will be directed to the water treatment plant or to the TMA. Non-contact runoff will be directed to existing natural drainage features after passing through runoff basins. Precipitation falling within the TMA will co-mingle with process water and become part of the water used in the mill circuit. Some of the surface water drainage originating from outside the active mining area will be intercepted by a series of drainage swales and directed to existing natural drainage features.

2.4.10 Wetland Mitigation

Although mine facilities have been designed to minimize impacts on wetlands, as part of project construction activities, approximately 29.5 acres of wetlands will be either excavated or filled. To compensate, CMC will develop replacement wetlands on a site located in Shawano and Oconto Counties. The selected site is in an area that was originally wetlands, but was converted to cropland. The establishment of the compensation site involves reconverting it from cropland back to wetlands.

2.4.11 Mine Reclamation

Topsoil will be salvaged and stored from all disturbed areas for use in reclamation activities. Reclamation of the mining site will occur on an ongoing basis during construction and operation, and as the final phase of the project. After mining, the area will be used for forestry and as open green space. During construction, disturbed soil areas will be revegetated on a continual basis such that wind and water erosion potential is significantly reduced. These areas will either be temporarily reclaimed or finally reclaimed depending upon their location relative to future construction activities.

Final reclamation of the plant site will begin after completion of mining. All open boreholes will be sealed in compliance with applicable regulations. Salvageable equipment from the mine will be brought to surface. Any equipment left underground will have potentially harmful fluids removed. The shafts to the mine will be sealed with reinforced concrete plugs. Surface facilities may be converted to other uses if possible. If other uses are not feasible, those facilities will be removed. The site area will be regraded and revegetated. Settling basins and ponds will be drained and the area reclaimed. Containment structures will be removed. Disturbed areas will be regraded and revegetated. The TMA will be reclaimed in phases during its lifetime. Final closure of the last cell of the TMA will occur late in the sequence of project reclamation.

The water treatment plant and associated pipelines will be removed after they are no longer required. Salvageable equipment will be transported off-site. Scrap and treatment solids will be placed in the TMA prior to closure of the final cell. Buried segments of pipelines will be purged and left in place. Above-grade pipelines will be removed. The water treatment plant area and pipeline routes will be graded and revegetated.

On-site roads, the plant site access road and the railroad spur line will be among the last items to be reclaimed. Reclamation of these features would be dependent upon the final site use. If no future use is anticipated, the construction materials will be removed. Bituminous pavement will be salvaged for use elsewhere, if possible, or placed in the TMA. Rail will be salvaged. The areas will be regraded and revegetated.

Utilities that service other customers along the route to the plant site will be left in place. The portion of the utilities that extend onto the plant site will be removed if above ground, or remain in service depending upon the final use of the site. Below ground piping will be flushed as required, capped and left in place, if no longer in service.

3 Location Criteria and Land Use

3.1 Location Standards NR 182.07(1) and NR 182.011(2)

The location standards of Wisconsin Administrative Code NR 182.07(1) and NR 182.011(2)(a) and (b) are listed in Table 3.1-1 along with the status of the TMA's compliance with each location standard. It is apparent from Table 3.1-1 that the TMA complies with all location standards.

3.1.1 Unique Features NR 131.03(22) and NR 132.03(25)

Section 3 of the project's EIR contains a detailed description of the environmental setting. There are no unique features as defined in NR 132.03(25)(b) located in the vicinity of the TMA.

3.1.2 Threatened and Endangered Species NR 182.07(1)(a)

Section 3.9 of the EIR contains descriptions of the flora and fauna baseline information in the vicinity of the TMA. The impacts of the TMA on the flora and fauna is detailed in Section 4 of the EIR. Mitigation of impacts is discussed in Section 5 of the EIR.

3.1.3 Minimization of Disturbance to Wetlands NR 182.07(2) and NR 132.06(4)

Wis. Admin. Code NR 182.07(2) requires that the TMA location comply with the standards and procedures in NR 132.06(4), relating to the minimization of disturbance to wetlands. Additionally, CMC must show that the facilities avoid, minimize, and mitigate disturbances to wetlands in compliance with USCOE 404 permit standards. The specific siting requirements of NR 132.06(4)(a) through (h) are analyzed in detail in the MPA and the EIR. This section addresses efforts made in the TMA design to minimize the acreage of wetlands impacted and the reasoning and rationale concerning efforts made in the design to minimize overall environmental impacts.

The TMA configuration and layout shown on Drawing No. 2 was selected based on the location standards of NR 182.07(1) and NR 182.07(2), i.e. minimizing the disturbance to wetlands to the extent practicable. During the process of laying out and designing the TMA cells the following operational, design and other considerations resulted in the minimization of wetland and overall impacts.

- Operational Considerations

  - use of sub-aerial deposition methods which provide an increased density when compared with subaqueous methods. The average density is increased to 97 pounds per cubic foot (pcf) as compared with 93 pcf for the subaqueous method.

  - use of managed tailings deposition method (i.e., ring discharge of tailings) which results in the efficient control of tailings deposition and minimization of the area on which there is a water pool.

Table 3.1-1

Summary of NR 182 Location Criteria For Tailings Management Area (TMA)

	Criteria	TMA Compliance	Reason or Evidence
182.07(1)	(a) Are excluded from areas defined by S.144.81(18). Stats. and NR 132.03(25) Wisconsin Administrative Code. The presence of endangered and threatened species will be considered.	Complies	EIR, 1995 Sections 3.9 and 4
	(b) Must be at least 1,000 feet from navigable lakes, ponds, and flowages.	Complies	Drawing No.2
	(c) Must be at least 300 feet from a navigable river or stream.	Complies	Drawing No. 2
	(d) Must be outside floodplains.	Complies	Drawing No. 2
	(e) Must be at least 1,000 feet from the right-of-way of a state trunk, interstate, or federal primary highway, parks, wild rivers, and other recreational and scenic areas unless visually screened to be aesthetically pleasing.	Complies	Drawing No. 2
182.07(1)	(f) Must be at least 1,200 feet from a water supply well.	Complies	There are no known water supply wells within the TMA compliance boundary.
	(g) Must not overlie known mineral resources likely to be mined and within 1,000 feet of the surface.	Complies	Drill holes have not indicated any mineral resource beneath the TMA.
	(h) Must be at least 200 feet from the property line.	Complies	See Figure 2.2 MPA
	(i) Must not be in an area where the Department finds there is a reasonable probability of surface water quality violations per NR 102 to 104, Wisconsin Administrative Code.	Complies	Refer to Section 6.9.

Table3.1-1 *(Continued)*

	Criteria	TMA Compliance	Reason or Evidence
	(j) Must not be in an area where the Department finds there is a reasonable probability of groundwater violations per NR 182.075, Wisconsin Administrative Code.	Complies	Refer to Section 5.0.
182.07(2)	Must comply with NR 132.06(4), Wisconsin Administrative Code regarding disturbance to wetlands.	Complies	Refer to Section 3.1.3.
NR 182.11(2)			
(a)	Should be located, where practicable, in the same watershed as the mining surface facilities.	Complies	All project facilities lie within the Wolf River watershed. The mining surface facilities are located in the Swamp Creek and Pickerel Creek watersheds. The TMA is located in the Pickerel Creek watershed and a small portion in the Hemlock Creek watershed. Other siting alternatives are not practicable and would cause greater environmental impacts.
(b)	Should be located so that tailings pipelines do not cross major water courses or wetlands.	Complies	The tailings pipeline route does not cross major water courses or wetlands.

Prepared by: REM
Checked by: NXP

- final deposition from the high point at the junction of center berms toward the low point on the opposite corner. This results in a final tailings depositional slope of approximately one-half percent and more efficient use of the lined area.

- after the final tailings slurry deposition, the tailings will be allowed to settle for one to two years prior to capping the cell. Tailings settlement expected during this time period is three to five feet. Additional tailings deposition could occur in settled areas to maximize the volume of each cell.

- minimizing water pond during operation. This results in a larger surface area which is available to dewater which results in an increased in-place density of the tailings.

- tailings and/or waste rock will be used when available to create the final slope inside each cell. This is a more efficient use of the available air space.

• Design Considerations

- 3:1 (H to V - Horizontal to Vertical) interior slopes are proposed which results in less land area disturbance, provides a more efficient use of lined area and a smaller overall footprint.

- a leachate collection system (LCS) is included which enhances the dewatering of the tailings. This provides greater tailings densities than without a LCS and results in a smaller overall TMA footprint.

- the two percent cell base slope is designed with a high point diagonally across the base of each cell. This increases the LCS efficiency and decreases potential percolation rates.

- the TMA cells are proposed to be located in the uplands portions of site 41 to minimize disturbances to wetlands.

- the interior and exterior berm crest widths are minimized which lessens the total footprint of the TMA. This smaller footprint has less potential overall environmental impacts.

• Other Considerations

- Approximately 50 percent of all the tailings are returned underground as mine backfill which minimizes the quantity of tailings stored on the surface and minimizes the TMA footprint.

- The four TMA cells are moved as far north as possible to avoid or minimize impacts on wetlands within the Pickerel Creek Drainage Basin.

- The proposed TMA footprint impacts approximately 22 acres of wetlands as compared to approximately 58 acres impacted by prior Site 41 designs.

- Avoidance of Burr Oak Swamp.

As discussed here, all locational, sizing, constructional and operational factors have been thoroughly evaluated to obtain the minimum practical impact on wetlands and the environment. By thus minimizing the disturbance to wetlands, the location criterion in NR 182.07 (2) of the Wis. Admin. Code is satisfied.

3.2 Land Use

A detailed discussion of land use in a 100 square-mile study area (Figure 3.2-1) surrounding the plant site and TMA is presented in Section 3.12 of the EIR. An overview of the results of that study are presented below.

3.2.1 Regional Land Use

Land use in Langlade County consists of approximately 70 percent forest lands and 24 percent farm/agricultural lands. The majority of the farmlands are located in the south central portion of the county, with the majority of the forest lands located in the northern half and northeastern portions. Langlade County has 867 lakes totalling more than 6,800 acres. A comprehensive land use plan for Langlade County forests (1985-1995) and an outdoor recreation plan are the only current county-wide land use plans available for Langlade County.

Land use in the Langlade County Town of Ainsworth, which comprises the southwestern quarter of the study area is shown in Table 3.2-1. As indicated in the table, more than 80 percent of the township is in forest and swamp lands, with residential and agricultural lands occupying less than 10 percent of the total land area in the town. The remaining 10 percent of the township includes tax-exempt lands such as public hunting/fishing grounds, Boy Scout lands, etc., which are also mostly forested.

Land use in Forest County consists of 84 percent forested lands, including national and county forest lands, and five percent agricultural lands. Surface water resources in Forest County include 824 lakes with a total surface area of 22,324 acres and over 317 streams with an excess of 700 miles of waterway. There are two land use plans available for Forest County. These include a ten-year (1985-1995) comprehensive land use plan for Forest County Forests and an outdoor recreation plan (1988).

Within Forest County, the Town of Lincoln comprises the northeastern portion of the study area. Land use in the Town of Lincoln is shown in Table 3.2-2. Approximately 87 percent of the town is forested, with about 10 percent of the lands being used for agricultural or residential land as indicated in the table.

Also within Forest County portions of the Town of Nashville comprise the northwestern and southeastern quarters of the study area. A breakdown of land use for the Town of Nashville is shown in Table 3.2-3. Similar to the Town of Lincoln, approximately 87 percent of the town is in forested lands with about 12 percent of the land area in agricultural or residential land.

Table 3.2-1

1994 Land Use, Town of Ainsworth, Langlade County

Land Use Category	Acres	Percent
Residential	1613	3.6
Commercial	45	0.1
Manufacturing	--	0
Agricultural	1,602	3.6
Swamp Lands	1,820	4.1
Forest Lands	35,027	78.8
Other[1]	4,330	9.8
Totals	44,437	100.0

Source: *Wisconsin Department of Revenue, 1994 Statement of Assessment.*

[1] Includes tax-exempt lands (i.e. public hunting/fishing grounds, lands owned by religious groups, Boy Scouts, etc.).

Prepared by: GMS
Checked by: RVS

Table 3.2-2

1994 Land Use, Town of Lincoln, Forest County

Land Use Category	Acres	Percent
Residential	1,664	4.6
Commercial	90	0.2
Manufacturing	--	--
Agricultural	2,047	5.6
Swamp Lands	279	0.8
Forest Lands	31,580	86.9
Other[1]	699	1.9
Totals	36,359	100.0

Source: *Wisconsin Department of Revenues, 1994 Statement of Assessment.*

[1] Includes tax-exempt lands, (i.e. public hunting, fishing grounds; lands owned by religious groups, Boy Scouts, etc.).

Prepared by: GMS
Checked by: RVS

Table 3.2-3

1994 Land Use, Town of Nashville, Forest County

Land Use Category	Acres	Percent
Residential	2,775	6.6
Commercial	76	0.2
Manufacturing	--	--
Agricultural	2,217	5.3
Swamp Lands	171	0.4
Forest Lands	36,503	86.5
Other[1]	440	1.0
Totals	42,182	100.0

Source: *Wisconsin Department of Revenue, 1994, Statement of Assessment.*

[1] Includes tax-exempt lands (i.e. public hunting, fishing grounds,land owned by religious groups, Boy Scouts, etc.).

Prepared by: GMS
Checked by: RVS

3.2.2 Study Area Land Use

Land use within the 100-square-mile study area has been divided into five categories, defined as follows:

Forestry - Land being used to produce forest products such as pulp wood, saw timber, wood chips and/or maple syrup. This includes all forested land capable of producing trees with a diameter at breast height of over 2.5 cm (one inch) and where canopy coverage is greater than 25 percent. It also includes clearcuts that will be left to revert to forest cover and old agricultural fields that are being allowed to revert to forest cover.

Agriculture - Land being used for agricultural purposes. This includes crop fields, hay fields, and pastures. Old agricultural fields that can potentially be used as pasture because they are fenced and are not reverting to forest cover are also included in this category.

Recreation - This land use includes those lands whose primary use is recreation. It does not include areas where recreation is a secondary use, such as Forest Crop Law lands. Lands included within this category are primarily parks and water bodies. However, this category also includes marshes and shrub swamps as recreation is probably their only major active use.

Residential/Institutional - This includes all residential areas, both permanent and seasonal, as well as any institutional uses such as churches, schools, nursing homes, and/or hospitals.

Transportation - Any lands used by transportation systems, such as roads, railroads and airports.

The distribution of the major land uses in the study area is shown on Figure 3.2-2. The percentage of each land use is shown on Table 3.2-4.

Table 3.2-4

Study Area Land Use

Land Use	Area in Square Miles	Percent of Total Area
Forestry	76.5	76.5
Recreation[1]	11.8	11.8
Residential	6.4	6.4
Agriculture	5.0	5.0
Transportation[2]	0.3	0.3
Total	100.0	100.0

[1] Includes water bodies/wetlands.
[2] Highways/railroads not included in acreage totals.

Source: *EMC, 1985.*

Prepared by: GMS
Checked by: RVS

The remainder of the land use discussion will focus on the major use categories found in the study area. These include: forest lands, agriculture, recreation, and residential. Commercial land use will not be addressed since it comprises a very minor proportion of the area.

3.2.2.1 Forestry

Approximately 77 percent of the study area is forest land that could conceivably be harvested, that is, used for forestry (Table 3.2-4 and Figure 3.2-2). In addition to providing resources for lumber, pulp and associated industries these forested lands provide numerous recreation opportunities, habitat for fish and wildlife and provide a stable environment along streams and other desirable resource areas.

Forest County owns over 10,800 acres of forest lands, 97 percent of which are classified as commercial forest. These lands, wholly located in the Town of Lincoln, are composed of the North Lincoln and South Lincoln units. South Lincoln is included in the study area. Aspen and northern hardwood are the two major cover types that make up over 80 percent of the total

county forest lands. The aspen cover type is composed, most commonly, of stands consisting of pure aspen or aspen mixed with white birch. The northern hardwood cover type consists of sugar maple, basswood, white ash, American elm, yellow birch, and red maple. Associated species include aspen, white birch, eastern hemlock, northern red oak, butternut hickory, butternut, and ironwood. Approximately 2,800 acres of Forest County-owned forest lands are included in the study area.

Langlade County owns approximately 126,500 acres of forest lands, of which 5,000 acres are located in the southwestern quarter of the study area, in the Town of Ainsworth. Approximately 93 percent of the Langlade County forest is considered commercial forest, with more than half of the total forest lands being composed of northern hardwood (33.2 percent) and aspen (31.9 percent).

According to the Forest and Langlade County plat books, approximately 35 percent (17,200 acres) of the forest land in the study area is owned by private corporations. Publicly owned state and county forest lands account for approximately 18 percent (9,000 acres), with the remaining 47 percent in private ownership (22,700 acres). Consolidated Papers, Inc. accounts for half of the corporately owned forest lands (8,750 acres) in the study area. Six other companies account for the remaining forested lands under corporate ownership, including CMC properties, of approximately 4100 acres.

3.2.2.2 Agriculture

Information on agricultural use in Forest and Langlade Counties was obtained from the Wisconsin Agricultural Statistics Service for the 1992-1993 seasons, and is shown in Table 3.2-5.

As indicated in the table, Forest County is consistently low in the rankings for agricultural products produced by Wisconsin counties. Even though Langlade County is ranked higher in agricultural production, the portion of Langlade County in the study area is much more like Forest County than the more agricultural portions of Langlade County to the south and west.

Approximately five percent, 3,200 acres, of the study area is in agricultural use which includes dairy farming, livestock production, and crops of oats, barley, wheat, alfalfa, hay and potatoes. Irregular slopes, stoniness, dryness on rises, wetness in depressions and short growing season limit agricultural use of soils in the study area.

The majority of the agricultural lands in the study area are primarily located in Forest County near Mole Lake and to the north along STH 55, along County Trunk Highway B near Nashville and south of Lake Metonga (Figure 3.2-2). Using the average size per farm listed in Table 3.2-5, and the number of acres in agricultural lands in the study area listed in Table 3.2-4, it is estimated that there are approximately 15 farms in the study area.

Table 3.2-5

Agricultural Statistics

	Forest County		Langlade County		1993 Statewide Rank
	1992	1993	1992	1993	Forest/Langlade
No. Farms	140	140	510	500	67/50
Ave. Size (acres)	214.3	221.4	282.4	284	
Crop Yields					
Corn bu./ac.	57.5	42.0	70.5	65.7	66/59
Oats bu./ac.	58.6	45.6	68.0	42.6	66/27
Barley bu./ac.	45.0	35.7	61.3	45.9	55/13
Wheat bu./ac.	28.0	--	33.1	43.2	--/27
Soybeans bu./ac.	--	--	18.1	18.0	--/59
Hay tons/ac.	2.1	2.3	1.9	2.9	68/30
Potatoes hundredweight/ acre	--	--	306	276	--
Milk Production lbs/cow	14,300	14,300	15,000	14,900	66/51

Source: *Wisconsin Agricultural Statistics - 1994, Wisconsin Agricultural Statistics Service, Madison, WI.*

Prepared by: GMS
Checked by: RVS

3.2.2.3 Recreation

The extensive amount of public forest lands, lakes and streams provide an abundance of outdoor recreational opportunities in the area. Figure 3.2-3 shows the recreational resources located in the study area. Approximately 9,000 acres of publicly owned state and county lands are available in the study area. Within Forest County alone there are a total of approximately 340,000 acres of publicly owned lands including U.S. Forest Service, county and state lands. These lands are open to hunting, fishing, picnicking, and camping. Other uses such as berry picking, hiking and bike riding are encouraged, although special regulations governing those uses have not yet been developed.

As shown in Figure 3.2-3, three campgrounds are located within the study area. A total of 110 campsites are available between the three campgrounds, with electricity, flush toilets and showers available at some of the sites. Public boat access is available at four lakes in the study area, as shown in Figure 3.2-3. These include Rolling Stone, Crane, Little Sand, and Lily Lakes. No other facilities are available at these sites. Suggested bicycle routes as identified in the Forest and Langlade County recreation plans are shown on Figure 3.2-4. The plans recommended

proper signage to caution drivers and paved shoulders on roadways designated as bike routes. The bicycle routes, however, have not been formally implemented.

Over 300 miles of groomed and signed snowmobile trails are available in Forest County. The specific trail segments included in the study area, and shown on Figure 3.2-4, are the 100 Mile Snow Safari Trail, Lumberjack Memorial Trail and Tombstone-Pickerel Trail.

Forest County Forestry Department developed and maintains an eight-mile all terrain vehicle trail (ATV) on Forest County forest land off of County Trunk Highway W at the end of Duff Road, shown on Figure 3.2-4. The trail is open for year round use. A loading dock is provided to assist with loading/unloading ATVs. A parking area is also available. No other services are provided.

Due to the abundance of lakes and streams in the area, the fishing pressure in Forest County is much less than what the available resource offers, however, fishing surfaces as the number one recreation experience. *The Forest County Comprehensive Outdoor Recreation Plan* referenced a special issues and actions report, prepared by the WDNR, that indicated protecting the existing water resources and addressing acid rain were two of the top three concerns expressed for WDNR's North Central Wisconsin District, which includes Forest County.

The *Forest County Comprehensive Outdoor Recreation Plan* includes a needs assessment for recreational facilities in Forest County for the period from 1986-1991. The needs assessment was determined by: (1) area based demand analysis; (2) trends in outdoor recreation activity participation; and (3) priority of facility needs. Through the needs assessment process priorities were established for the Forest County region, which also included Vilas and Oneida Counties.

From this analysis, the plan included recommendations for new and/or improved facilities. There were no new facilities recommended for development within the study area. However, the plan included several general recommendations for future facilities, including the following:

- additional river access sites for canoeing;
- designated bicycle routes along thoroughfares which are properly designed and signed;
- better fishing access accommodations/handicapped accessible fishing docks/off-road parking; and
- upgrade existing boat launch facilities/construct new launch facilities at lakes without launch facilities.

The Langlade County Outdoor Recreation Plan provided several recommendations for future recreational facilities in the county. No new facilities were recommended or identified in the study area. Among those included the following:

- development of a county/regional park in the eastern portion of the county (no site identified);
- additional improved public access to state and federal lands; and

- undertake a county-wide bicycle trail study/identify suggested bicycle routes with route signs.

Specific improvements recommended for the Town of Ainsworth facilities located at Rolling Stone Lake included:

- upgrade to be handicap accessible;
- finish boat launch with cement planks and gravel; and
- establishment of a picnic area with table/grill.

3.2.2.4 Residential/Institutional

Residential land use, estimated to include approximately 4,000 acres or six percent of the total land area in the study area, is concentrated around the larger lakes as shown in Figure 3.2-2. Permanent population totals for the Towns of Nashville, Lincoln and Ainsworth in 1990 were 871, 630, and 474, respectively. However, in noting the large percentage (50) of seasonal/ recreational housing units in Forest County, it can be assumed that the population in the Towns of Lincoln and Nashville increases during summer months.

Within the study area most permanent residences are located in Mole Lake, along Lakes Metonga and Lucerne and along STH 55, County Trunk Highway B, and Airport Road in the northern portion of the study area. Little Sand Lake, the closest lake to the ore body, is developed along the western, northwest and eastern shorelines. Many of these residences as well as others on lakes in the study area are one to two bedroom seasonal cottages.

There are only four institutional buildings located in the study area. These include a chapel, school and tribal building in Mole Lake and the Nashville Town Hall, which is located at the intersection of STH 55 and County Trunk Highway B.

3.2.2.5 Transportation

Transportation land uses in the study area include STH 55, County Trunk Highways "B","M", "Q", and "W", local town roads/forest roads, a Wisconsin Central Limited Railroad line and the Crandon Municipal Airport.

4 Waste Characterization and Analysis NR 182.08(2)(b)(1)

4.1 Estimated Waste Types and Quantities to be Stored NR 182.08(2)(3)(a)

4.1.1 Waste Types

Four major categories of wastes will be generated during the development of the mine, its operation, and concentration of ore. These wastes will include: 1) waste rock; 2) tailings; 3) wastewater treatment and reclaim pond solids; and 4) overburden. Tailings, treatment and reclaim pond solids, Type II waste rock and a portion of Type I waste rock will be placed in the TMA. In addition, a small amount of laboratory waste will also be placed in the TMA. This section contains a brief description of the wastes and projected waste quantities of the materials that are expected to be contained within the TMA.

4.1.1.1 Waste Rock

Waste rock will be generated during the preproduction and operational phases of the mine. The waste rock generated during the preproduction phase will be brought to the surface where it will be either used for construction purposes in an application where it will not be in direct contact with vehicular traffic or placed in the TMA, depending upon the results of leaching studies performed on these materials. The preproduction waste rock will be derived from the hanging wall rocks to the north of the ore body. It is expected that most of the preproduction waste rock will not produce acidic drainage and will leach only minute quantities of substances. Such waste rock is referred to as Type I waste rock. Some of the preproduction waste rock is expected to have the potential to produce acidic drainage and/or to leach greater quantities of substances. Such waste rock is referred to as Type II waste rock.

Type II waste rock will likely be generated during the preproduction and operating phases of the project. This material will be mainly generated during the development of the stope accesses. This material will be brought to the surface during the preproduction phase of mine development and will either be used in the construction of the TMA interior slopes, or will be contained within it.

4.1.1.2 Tailings

Tailings materials will be generated as a result of the primary beneficiation of ore at the site. Permanent placement of tailings will occur both in the mine and in the TMA. It is expected that about one-half of the tailings materials will be placed in the mine through the backfilling of stopes, and about one-half of the tailings materials will be placed in the TMA. Backfilling of depleted stopes within the mine will have priority for receiving tailings materials.

Current mine plans call for backfilling depleted stopes with a coarse tailings fraction consisting of material larger than 10 to 15 microns in diameter, and the placement of a fine fraction typically smaller than 10 to 15 microns in diameter in the TMA. During the time that backfilling is not occurring, whole tailings will be pumped to the TMA. The use of whole tailings as backfill is a potential backfilling option. Its application to the Crandon Project cannot be fully assessed until after operations begin. In the event that whole tailings can be used for backfilling, only whole tailings will be pumped to the TMA.

All tailings materials pumped to the TMA will be in the form of slurries with varying solids contents, depending upon the nature of the tailings pumped. The liquid portion of the slurries will primarily be process water.

4.1.1.3 Wastewater Treatment and Reclaim Pond Solids

Treatment of mine waters, using the planned lime and sulfide treatment processes, will produce solids. Both wastewater treatment and reclaim pond solids will be placed with tailings in the TMA.

Wastewater treatment solids are characterized in the project's wastewater treatment facility engineering report. These solids are expected to be produced and placed in the TMA on a continuous basis.

Reclaim pond solids will consist of fine particulate matter that has been transferred to the reclaim pond during the management of surface water levels within the TMA. The reclaim pond acts as a settling basin for this material, which would otherwise enter the recycled water returned to the ore process facilities. Reclaim pond solids will be similar to the fine tailings materials, with the exception that the reclaim pond solids will contain smaller proportions of pyrite and greater proportions of clay minerals and quartz than the fine tailings, due to the differential settleability of these minerals. Reclaim pond solids will be removed from the base of the pond, as needed, and placed in the TMA. For the purposes of the environmental investigation, reclaim pond solids have been considered equivalent to fine tailings, even though they will be less reactive than the tailings. No separate testing of reclaim pond solids was deemed necessary.

4.1.1.4 Laboratory Waste

Laboratory waste will primarily consist of crucibles of ground rock and will include a fraction of laboratory reagents that have been used in assaying and process bench testing. These wastes will be added to the process circuit where they will become part of the tailings transported to the TMA. It is expected that these materials will constitute only about one hundredth of one percent of the total volume of the TMA. Due to the insignificant volume of the laboratory waste, no separate testing of this material was deemed necessary.

4.1.2 Waste Quantities

The TMA waste quantities projections are based on the present estimate of the mine recoverable reserves from the zinc and copper ore, which total approximately 55 million tons. Table 4.1-1 contains waste quantity projections for the project based on the following assumptions. Approximately 20 percent of the total reserves is extracted as concentrates, 40 percent is returned underground as backfill and the remaining 40 percent is placed in the TMA cells. The zinc tailings and copper tailings fractions of the tailings are estimated at 57 percent and 43 percent, respectively.

Appendix A contains the waste volume calculations which are summarized on Table 4.1-2. It is estimated that a total of 20.46 million cubic yards of mine waste will be placed in the TMA. This quantity includes a 13 to 15 percent contingency to account for variations in the ore body, ore grade, volume of waste rock hoisted to the surface, etc. The contingency has been added to the final zinc and copper cells, TMA 2 and TMA 4 respectively.

Table 4.1-1

TMA Estimated Total Waste Quantities

Waste Type	Density (PCF)	Annual ($\times 10^6$) Tons	Generation ($\times 10^6$) cy	Total Generation ($\times 10^6$)[4] Tons	Total Generation ($\times 10^6$)[4] cy
Zinc Tailings	97[1]	.80	.61	12.77	9.75
Copper Tailings	97[1]	.80	.61	9.63	7.36
Type II Waste Rock	111.3[2]	--	--	.10	.07
Type I Waste Rock	111.3[2]	--	--	.05	.03
Wastewater Treatment Plant Solids	47.2[2]	--	--	.28	.70
Laboratory Waste[3]	NA	--	--		--
Total[4]				22.83	17.91[6]
Reclaim Pond Solids[5]	47.2[2]	--	--	.27	.42

(1) : 97 pcf (pounds per cubic foot) from SRK, December 1994, assumes subaerial deposition.
(2) : Densities from EMC, 1985.
(3) : Laboratory waste will be approximately 10.0 tons/year or approximately 280 tons for the total site life.
(4) : Total does not include the contingencies.
(5) : Reclaim Pond Solids is not included in the waste quantities total as this is fine tailings carryover that has already been accounted for in the tailings quantities.
(6) : An additional 250,000 cubic yards of waste rock may be mined and brought to the surface during operations. This waste rock would likely be placed in the TMA resulting in a reduction in the contingency to 13 percent.
cy : cubic yards
x 10^6 : million
NA : not applicable
-- : not calculated since volume is very small or waste stream does not occur on an annual basis

Prepared by: REM
Checked by: NXP

Table 4.1-2

Estimated Volumes and Site Life for TMA Cells and Stages

Tailings Cell	Stage	Tailings (x 10^6 cy)	Approximate Phase Life (yrs.)	Other Wastes (x 10^6 cy)	Contingencies (x 10^6 cy)	Calculated Need (x 10^6 cy)	Cell Design Total[4] (x 10^6 cy)	Capacity as Percent of Total
Zinc Ore Production								
TMA 1	I	1.83	3.0	.17	--	2.00	2.03	
TMA 1	II	1.83	3.0	.13	--	1.96	2.15	
Subtotal		3.66	6.0			3.96	4.18	35.6
TMA 2	III	3.36	5.5	0.1	--	3.46	3.39	
TMA 2[1]	IV	2.75	4.5	0.1	1.47	4.32[1]	4.17	
Subtotal		6.11	10.0			7.78	7.56	64.4
Zinc Total Tailings		9.77	16.0	0.5	1.47	11.74	11.74	100.0
Copper Ore Production								
TMA 3	V	1.22	2.0	0.1[2][3]	--	1.32	1.59	
TMA 3	VI	2.44	4.0	0.1[2][3]	--	2.54	2.65	
Subtotal		3.66	6.0	0.2	--	3.86	4.24	48.2
TMA 4	VII	1.83	3.0	0.1[2][3]	--	1.93	1.71	
TMA 4[1]	VIII	1.83	3.0	0.1[2][3]	1.10	3.03[1]	2.85	
Subtotal		3.66	6.0	0.2		4.86	4.56	51.8
Copper Total Tailings		7.32	12.0	0.4	1.10	8.72	8.80	100.0
Total Zinc & Copper		17.09	28.0			20.46	20.54	

(1) Final zinc and copper cell includes a 15 percent contingency calculated based on the total "expected" production of both zinc and copper tailings only.
(2) Other materials for copper include 1/4 of .30 x 10^6 cy of WTPS + 1/4 of the estimate for other wastes (.11 x 10^6 cy) for each cell or .075 + .0275 = 0.1025 million cy per stage.
(3) Other materials include Type I and Type II waste rock, wastewater treatment and reclaim pond solids, and laboratory wastes.
(4) Cell design total is the actual design volume based on model calculation performed by Foth and Van Dyke's Computer Aided Drafting and Design (CADD) system. The design total is within .5 percent of the required total.

cy - Cubic yards
yrs - years

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4.1.3 TMA Cell Volumes

The proposed design provides for two cells, TMA 1 and TMA 2, for zinc ore tailings and two cells, TMA 3 and TMA 4, for copper ore tailings. Each cell is proposed for construction and operation in two stages. For example, TMA 1 consists of Stage I and Stage II, TMA 2 consists of Stage III and Stage IV, etc. To account for unknowns concerning the ore body, size and volume, actual versus predicted recovery rates, and other factors, a contingency has been included in the calculation of the tailings quantities. Table 4.1-2 provides a summary of the estimated volume and life of each cell and cell stage. The staging of each TMA cell is based on staging the design and construction, such that about one-half of the TMA cell is constructed in the first stage. In addition, consideration was given to the need to have a construction schedule for each stage, which allows for at least one full year of no construction prior to the start of construction of subsequent phases. It is anticipated that each TMA stage will be constructed over a two-year period, major earthwork during the first year and the liner and leachate collection system construction during the following year. Sections 6.12 and 7.2 of this report provide additional details concerning phase development and construction, respectively.

4.2 Waste Characterization

Wisconsin Administrative Code (NR 182.08(2)(b)2) contains a provision that allows an applicant to perform a literature review of waste characterization results from similar existing facilities and to use this information in preparation of the waste characterization and analysis section of this report. CMC has opted to also perform rigorous testing to characterize the tailings materials that were generated during a metallurgical test of an actual zinc ore composite prepared from the Crandon Formation. Following is an overview of the waste characterization tests that have been performed to date. A detailed discussion of the waste characterization studies is contained within Section 3.5.5 of the EIR.

The conclusions reached as a result of the waste characterization program have been based on both existing data and studies, and the work performed during the current program. Due to the evolutionary nature of the reactions that occur, and the consequent length of time required for these reactions to be completed for some of the materials tested, some of the tests are on-going and have not been completed. The observations and conclusions presented in this section are therefore preliminary in nature. Those tests that have not reached desired end points will be continued, and an addendum will be prepared that will contain the final results.

It is expected that approximately 50 percent of the tailings produced during the mining operation will be returned to the mine to backfill the depleted stopes and the other 50 percent of the tailings will be pumped to the TMA. It is expected that three classes of tailings materials will be generated during mining/milling operations: coarse tailings, fine tailings and whole tailings. Current plans call for cycloning of whole tailings to produce the coarse tailings, as needed, for backfilling. Cycloning whole tailings will also produce fine tailings materials which will be pumped to the TMA. It is expected that 80 percent of the tailings materials that will be placed in the TMA will be fine tailings. The remaining 20 percent of the tailings that will be placed in the TMA will be whole tailings, which will be pumped directly to the TMA when coarse tailings are not needed for backfilling. Waste characterization studies of the tailings materials that will be placed in the TMA have therefore been limited to the fine and whole tailings materials.

4.2.1 Preparation of Tailings Materials from the Crandon Ore

Diamond drill core samples of the zinc ore collected during the late 1993 through early 1994 zinc ore confirmatory drilling program were shipped to Lakefield Research (Lakefield) of Lakefield, Ontario for metallurgical bench testing. A master ore composite was prepared from the recovered zinc ore samples to approximate the projected average mill head grade of 9.2 to 9.6 percent zinc. Lakefield processed the master ore composite to prepare zinc, copper and lead mineral concentrates during lock cycle flotation testing. A byproduct of the production of ore concentrates is the generation of tailings materials. Tailings materials generated during the flotation process were collected and separated into two size fractions: one containing materials typically coarser than 10 to 15 microns in diameter (referred to as coarse tailings, or sands), and one containing materials typically finer than 10 to 15 microns in diameter (referred to as fine tailings). Both size fractions were shipped in separate sealed containers to Foth & Van Dyke for waste characterization studies.

Upon receipt of the samples, the containers were weighed to determine the proportion of the whole tailings that each size fraction represented. Some of the waste characterization tests required the use of fine or coarse tailings materials and other tests required the use of whole tailings. Whole tailings were produced by Foth & Van Dyke, as required, by proportionately recombining the two tailings size fractions.

Testing of tailings materials produced from both zinc and copper ores were conducted in 1979 by Lakefield. A comparison of the bulk chemistry of the 1979 ore composites with the 1994 zinc ore composite shows that testing of the zinc ore alone will produce results that can be used to predict the reactions of the tailings from both the zinc and copper ores (see Section 3.5.5.2.2 of the EIR).

Studies of the physical, chemical, mineralogical, and radiological properties of the tailings materials have been conducted by several parties including: Dames and Moore, 1978; Golder, 1981; Colorado School of Mines Research Institute, 1982; Knight and Piesold, 1982; and, STS Consultants, 1983. To update the tailings data base and to provide specific data relating to the design and operation of the TMA, physical testing of the 1994 tailings materials was performed by Steffen, Robertson and Kirsten (SRK), and chemical testing of these materials was performed by Foth & Van Dyke.

4.2.2 Physical Properties and Testing of the Tailings Materials

The results of the testing conducted by SRK are contained in two memoranda; one pertaining to density testing; and one pertaining to moisture regime testing. Both of these memoranda are included in Appendix B. One of the memoranda contains laboratory data and interpretations relating to moisture contents of the tailings materials at various suction pressures and tailings densities. The results of these tests formed part of the input to the HELP model which is discussed in detail in Section 6.7 of this report.

The second memorandum contains laboratory data and interpretations relating to physical properties of the tailings. Table 4.2-1 presents a summary of some of the physical properties of the fine and whole tailings contained in the memorandum. The properties include: specific gravity, particle size distribution, and consolidation characteristics. SRK also completed large

Table 4.2-1

Summary of Tailings Properties[1]

Property	Description	Value/Units	
		Fine Tailings	Whole Tailings
Grain size	-75_u	100%	90%
	-20_u	92%	47%
	D_{85}	0.01 mm	0.07 mm
	D_{60}	0.006 mm	0.045 mm
	D_{50}	0.005 mm	0.026 mm
	D_{30}	0.0029 mm	0.0060 mm
	D_{15}	0.00148 mm	0.00270 mm
	D_{10}	--	0.00152 mm
	C_c	--	0.5248
	C_u	--	29.5121
	Percent Sand	0	10
	Percent Silt	100	90
	USCS Classification	ML	ML
Specific gravity		3.32	3.52
Hydraulic conductivity		1×10^{-6} cm/sec	2×10^{-6} cm/sec
Coefficient of consolidation		0.091 cm^2/sec	0.127 cm^2/sec
Dry density			
30 ft load		89 pcf	124 pcf
60 ft load		92 pcf	127 pcf
Recommend Design Density			
As Deposited, subaerial		97 pcf	
As Deposited, Subaqueous		93 pcf	
Final		100 pcf	

1. All data from Appendix B.

Prepared by: REM
Checked by: NXP

scale pan drying/self-weight consolidation measurements and column settled densities, which will be discussed in the operations sections of this report.

The grain size analysis of the fine tailings shows that 100 percent of the tailings particles are finer than 75um (microns) diameter (i.e. 100 percent passes the number 200 sieve). The fine tailings are nonplastic and are classified under the Unified Soil Classification System (USCS) as a silt (ML). Because of the fine nature, they have a low hydraulic conductivity in the range of 1×10^{-6} cm/sec.

The grain size analysis of the whole tailings shows that about 90 percent is finer than 75um (microns) (i.e. 90 percent passes the number 200 sieve). The whole tailings are nonplastic and classified classify under USCS as a silt, slightly sandy. Whole tailings also exhibit a low hydraulic conductivity being in the range of 2×10^{-6} cm/sec. The specific gravity of the fine and whole tailings is high (i.e., 3.32 and 3.52 respectively) due to the presence of iron pyrite, which has a specific gravity of approximately 5.0.

Based on these results, summarized in Table 4.2-1, SRK has calculated a time to 90 percent consolidation of approximately six months for the tailings mass that will consist of 80 percent fine tailings and 20 percent whole tailings assuming a 85-foot maximum tailings deposition thickness. Other analyzed tailings properties included in Appendix B are related to operation densities and design densities. These properties will be discussed respectively in the operation and design sections of the report.

4.2.3 Chemical Properties and Testing of the Tailings Materials

During operation of the mill, tailings materials will be mixed with process water and pumped to the TMA cells. The first tailings materials will be added to the cells subaqueously. Later, excess water will be drained from the surface of the accumulating cells, and from beneath the tailings by activating the leachate collection system located in the base of the cells.

In order to evaluate the reactions of the tailings stored in this environment, two sets of tests have been developed. In the first set of tests, composites of fine and whole tailings were emplaced in modified 30-gallon cylindrical polyethylene tanks and reacted first with process water, under saturated conditions, to simulate the reactions of the ponded tailings while saturated. These tailings were later drained and reacted with deionized water, under alternating wet and dry conditions, to simulate the conditions that may occur when excess water has been drained from the cells and if precipitative water and air entered the stored tailings in the unlikely event of a failure in the TMA cover system. As such, these tests are being conducted under worst case conditions, since they are maintained under warm, moist oxidizing conditions.

In the second set of tests, a composite of whole tailings and a composite of lime amended whole tailings were each emplaced in a modified 30-gallon cylindrical polyethylene tank and have been reacted with deionized water under alternating wet and dry conditions. These tests were designed to evaluate reactions between tailings materials added to the top of a tailings cell, during the final stage of cell use, to form a crown over which the TMA cover system will be installed. These tests are also being conducted under worst case conditions.

Whole tailings materials have maintained a pH in the leachate between 6 and 7 during the first six months of reacting either to continuous alternating wet and dry conditions or to saturated,

followed by alternating wet and dry conditions. Based on the available data collected to date, it appears that the whole tailings material that has been subjected to continuous wet and dry leaching conditions leaches more substances than the whole tailings material leached under saturated conditions.

The leaching of fine tailings under saturated conditions, followed by alternating wet and dry leaching, appears to be producing slightly acidic leachate after six months of testing. The concentrations of substances in solution during the saturated portion of the leaching of the fine tailings is similar to those produced during the saturated portion of the leaching of the whole tailings.

The pH of the leachate derived from the reacting of the lime amended whole tailings composite with deionized water under alternating wet and dry conditions has declined from 11 to about 7 during the first six weeks of testing. The addition of a small amount of lime to a whole tailings composite resulted in several differences in the composition of the leachate compared to the leachate derived whole tailings without lime, leached under the same conditions. Based upon the number of leachate analyses completed to date, the maximum amount of copper and lead leached from the lime amended whole tailings was about 10 times greater than that leached from the whole tailings composite without lime. However, the maximum sulfate leached from the lime amended whole tailings composite was about one-tenth of that leached from the whole tailings composite without lime. The maximum amount of calcium leached from both composites was about equal. The reduced leachability of the sulfate and calcium from the lime amended whole tailings composite may be partly the result of the greater development of gypsum on the top of the lime amended tailings, and thus a greater retention of these two substances.

The reader is referred to Section 3.5.5 of the Crandon Project EIR for more information regarding the waste characterization tests and for a presentation of the data collected to date.

4.2.4 Miscellaneous Testing of the Tailings Materials

4.2.4.1 Radiological Testing

Extensive radiological analyses were performed on the fine tailings, ore and waste rock during previous project environmental investigations. The results of the chemical and radiation tests, conducted and verified by several organizations, showed that these materials contain less uranium, thorium and radium, and have lower radioactivity, than nearby unmineralized crystalline rock. Although no additional radiological analyses are necessary to characterize these materials, uranium analyses were performed in 1994 and 1995 as part of the bulk chemical analyses, and during the first few leachate samples collected from each material being reacted. There were no definite detects of uranium in any of these samples. Definite detects are considered to be those that are three times greater than the instrumental detection limit for single digit detection limits.

4.2.4.2 Asbestiform Minerals

Thin sections of selected ore and waste rock samples were prepared during the previous project environmental investigations and examined by optical microscopy for the presence of asbestiform minerals. This work, which was verified by the Wisconsin Geological and Natural History Survey, found no asbestiform minerals in the analyzed samples. Samples of fine tailings material were examined by transmission electron microscopy by a laboratory recommended by WDNR. No

asbestiform minerals were found in the fine tailings material. No additional evaluations of asbestiform minerals have been performed.

4.2.4.3 Carbonate Minerals

Small and variable amounts of carbonate minerals (mainly calcite) have been found in the waste rock and ore from the site. The studies used chemical and petrographic techniques in determining that the carbonate mineral content of these materials varied from about one to four percent. Carbonate mineral content (as calcite) has been calculated in the present study from the results of the bulk chemical analyses. The calculated carbonate mineral content of the waste rock, ore and tailings varies from about one-half to four and one-half percent. It appears that the carbonate minerals have affected the acid generation potential of these materials. An evaluation in 1983 by J.A. Cherry of the acid generation potential of the Crandon tailings concluded that it is likely that these tailings will not develop acidic pore water due to the carbonate mineral content, particularly if the surface of the tailings is covered soon after impoundment. The current waste characterization program is designed to estimate the relationship between the time of exposure and oxidation rates.

5 Characterization of the Existing Environment

The characterization of the existing environment in the vicinity of the TMA is addressed in detail in Section 3 of the project's EIR. The data summarized below is relevant to the siting and design of the TMA. This data relates to geologic and hydrogeologic conditions, biology, wetlands, aesthetics, surface water and local climatological data.

The purpose of geologic and hydrogeologic investigations were to assess the existing conditions with respect to soil properties; bedrock properties; hydrogeologic properties that control recharge, discharge, and groundwater interaction with surface water bodies; and groundwater quality consistent with the requirements of NR 182. Included within this section are discussions of the regional geologic background information.

5.1 Regional Information NR 182.08(2)(c)

5.1.1 Regional Topography

The study area is located within the Northern Highlands physiographic province, a region of rolling terrain that reflects its glacial origins. Ground surface elevation in the area ranges from less than 1,500 feet MSL near Rolling Stone Lake, approximately three miles south of the ore deposit, to more than 1,750 feet MSL, approximately five miles northwest of the ore body.

The ground surface is variable, from nearly flat in floodplain and basin areas, to a grade of approximately 25 percent. Area topography is characterized by a general southwest trend of ridges and intervening valleys. This trend reflects the southwesterly advance of the most recent Woodfordian glacier, which reshaped the pre-existing topography. processes associated with the Woodfordian glacier deposited ne material on the reshaped uplands and valleys. This southwest trend is especially apparent in the upland areas five to ten miles northwest of the deposit where elongated elliptical ridges, or drumlines, have approximately 100 feet of vertical relief. The southwest trend is also apparent in the Swamp Creek valley and in the orientation of the prominent ridges south of Mole Lake and immediately to the east and west of the ore deposit.

The topography and landforms in the vicinity of the TMA are presented on the existing conditions map Drawings Nos. 2 and 3. The proposed TMA will be located on a north-south trending wooded upland area which is predominantly between elevation 1,610 and 1,770 feet. Elevations are as low as 1,600 feet near the lakes, streams, and wetlands which surround much of the site area. In the north portion of the facility, there are hills as high as 1,770 feet. The upland location of the TMA will provide setbacks from lakes, wetlands, and groundwater discharge areas. It will also provide for a large separation distance from the main groundwater aquifer with a sizable unsaturated zone beneath the facility.

5.1.2 Hydrology

The study area lies within the area of northeast Wisconsin that is drained by the Wisconsin, Wolf, Fox, Peshtigo, Oconto, and Menominee Rivers. The area is entirely within the northern headwaters of the Wolf River drainage basin. The TMA site is in the Wolf River - Fox River drainage basin. The Wolf River originates approximately 20 miles northwest of the site area and flows south to Lake Poygan in Winnebago County, which is approximately 100 miles south of the study area. The Fox River drains the Lake Poygan-Winnebago system in a northeasterly direction

to Green Bay. The Wolf River basin is bordered on the west by the Wisconsin River basin. The Wisconsin River flows south and then west to the Mississippi River. To the north and east of the Wolf River basin, drainage is provided by the Menominee, Peshtigo, and Oconto rivers, which all flow southeast to Green Bay. In the upper Wolf River basin, the relatively slow-flowing streams frequently pass through lakes and wetlands.

The highest monthly stream discharge on the Wolf River typically occurs in April when snowmelt runoff is highest, and is typically lowest in January and February. The streams transport only a small amount of sediment due to the forested nature of the area, granular nature of the soil, moderate slopes, low stream velocities, and the many lakes and wetlands which serve as catchment basins.

5.1.3 Regional Geology

On a regional scale, the geologic conditions around the study area consist of metavolcanic Precambrian bedrock, which hosts the ore body, and is overlain by deposits of glacial till and outwash. These components are described below.

5.1.3.1 Regional Bedrock Geology

The bedrock of northern Wisconsin is an extension of the Canadian Shield and is located in the Southern Province. This Province is one of seven Canadian Shield rock provinces consisting of rocks ranging in age from 960 to 3,200 million years as shown in Table 3.5-5 of the EIR.

Volcanic rocks are found within the Southern Province as irregularly shaped belts surrounded by granitic and gneissic rock. These belts are comprised of volcanic flows and pyroclastics with interbedded sedimentary rocks such as shale, sandstone and conglomerate. Locally, the volcanic belts are intruded by large masses of granite or granodiorite. It is within these volcanic belts that zinc and copper sulfide deposits may be found.

These sulfide deposits are generally a tabular shaped body consisting of more than 50 percent sulfide by volume. The principal sulfide mineral found in these deposits is pyrite (iron sulfide) with lesser amounts of sphalerite (zinc sulfide), chalcopyrite (copper sulfide), galena (lead sulfide), and trace amounts of gold and silver. Many zinc sulfide deposits are underlain by a stockwork of chalcopyrite veinlets that may contain quartz and pyrite.

In Wisconsin, three major ages of Precambrian rocks are exposed. Of these, the Middle Precambrian rocks are of interest in the study area and are described below.

1. Early Precambrian (Archean, older than 2,500 million years). Rocks in the Early Precambrian are among the oldest in North America. Most of the rock is granitic gneiss, 3,200 million years old, containing minor amounts of volcanic rock that have been folded and intruded by granite.

2. Middle Precambrian (Early Proterozoic, 1,600 to 2,500 million years). Beginning about 2,100 million years ago, sediments began to accumulate in northeastern Wisconsin. The most well-known sediments deposited during this period were the iron formations which host the iron ore of the Gogebic and Florence areas. To the south, flows and volcanic related sediments were deposited in a large trough extending

across Wisconsin from Ladysmith to Marinette. It is within this volcanic belt that the sulfide ore deposits at Ladysmith (Flambeau and Thornapple), Rhinelander (Lynn and Pelican) and Crandon are found. About 1,800 million years ago, during the Penokean Orogeny, this volcanic belt was faulted, folded and intruded by granite. Erosion followed and quartz-rich sandstone was deposited to the south and west in the vicinity of Rusk and Barron counties.

3. Late Precambrian (600 to 1,600 million years). During this period, quartz-rich sandstone continued to be deposited in western Wisconsin. About 1,500 million years ago, a large body of granite, the Wolf River batholith, was emplaced in east-central Wisconsin. The northernmost area of Precambrian rock in Wisconsin is composed of a sequence of basaltic flows with interbedded sedimentary rocks. These basalt flows accumulated in a large trough or rift, which formed about 1,100 million years ago and extended from Lake Superior into Kansas, a distance of over 1,000 miles. Deposits of copper occurred in some of these rocks in the Upper Peninsula of Michigan. Near White Pine, Michigan, copper deposits are found in shale.

There has been no major folding, faulting, or igneous activity in the Southern Province of the Canadian Shield since the Precambrian era.

5.1.3.2 Regional Pleistocene Geology

The majority of unconsolidated sediments in the study region, and across Northern Wisconsin, consist of glacial drift. Deposits from a number of major glacial advances have been recognized in Wisconsin. In the study region the majority of the unconsolidated sediments were deposited during the Wisconsinan Age. Sediments of Pre- to Early-Wisconsinan Age are present locally above the saprolite bedrock. The Pre- to Early-Wisconsinan sediments are generally buried by Wisconsin Age Sediments.

The unconsolidated Pre- to Early-Wisconsinan deposits of the Forest County area include undifferentiated sediments of the Marathon Formation. Pleistocene units younger than the Marathon Formation are Pre-to Early-Wisconsinan undifferentiated sediments of Lincoln Formation and the Wisconsinan Mapleview Member of the Horicon Formation. The youngest Pleistocene unit in Forest County is the Nashville Member of the Copper Falls Formation. The Nashville Member, deposited by the Langlade Lobe, is considered the stratigraphic equivalent of the Green Bay Lobe Mapleview Member. But in Forest County the Nashville Member overlies and truncates the Mapleview Member.

The Marathon Formation, a dark brown calcareous till, underlies the Lincoln Formation and Mapleview Member of the Horicon Formation. The Marathon and Lincoln Formations are the oldest glacial units of the region. The Marathon Formation contains till and associated sand and gravel. This unit generally thickens northward and the extent of the formation below the Copper Falls Formation and Horicon Formation is not known.

Overlying the Marathon Formation are discontinuous deposits of sediments belonging to the Lincoln Formation. In the Forest County region these sediments are undifferentiated. The Pre- to Early-Wisconsinan Lincoln Formation includes red-brown sandy loam till and associated sand and gravel. This formation also may directly overlie saprolite or bedrock. The Lincoln Formation in the area is generally greater than 40,800 ± 2,000 years BP, making the formation

Pre- to Early-Wisconsinan. The Lincoln Formation probably makes up the majority of sediments found below the Late Wisconsinan deposits.

The Mapleview Member of the Horicon Formation overlies the Marathon and Lincoln Formations. The Mapleview Member of the Horicon Formation was deposited by ice from the Green Bay Lobe in Late Wisconsinan time around 15,000 or 14,000 B.P. The Mapleview Member consists mainly of cobbly, pebbly, silty sand till with boulders and associated sand and gravel. According to Mickelson, et. al., 1984, the Mapleview Member is at least partly contemporaneous with the Nashville Member of the Copper Falls Formation deposited by the Langlade Lobe. In the vicinity of the site, the Mapleview Member is truncated and overlain by the Nashville Member of the Copper Falls Formation.

The majority of the surficial sediments in the Forest County Region belong to the Nashville Member of the Copper Falls Formation. The Nashville Member consists of a pebbly, cobbly sandy loam till with numerous sand lenses and associated sand and gravel. Till of the Nashville Member, in Forest County, is generally exposed at the surface in drumlins clustered on upland areas. The associated meltwater sands and gravels have filled in low areas around the uplands in pitted and unpitted outwash plains. The Nashville Member is of Late Wisconsinan Age and is locally overlain by post-Pleistocene sediments consisting of wind-blown silts and fine sand, organic deposits of grass-sedge, moss and woody peat, and modern alluvium.

5.1.4 Regional Hydrogeology

On a regional scale, the hydrogeologic conditions around the project area generally consist of saturated glacial till and outwash overlying Precambrian bedrock. This area is defined as the Northern Drift-Precambrian Hydrogeologic District by Zaporozec and Cotter (1985). According to NR 182.08(2)(c), the term regional is defined as "that area which may affect or be affected by a proposed site. In most cases this will be the proposed site and, the area within a radius of up to five miles from the site." Figure 3.6-10 of the EIR illustrates the five mile radial area surrounding the project area. NR 182.08(2)(c)(4) states that the regional hydrogeology discussion should include discussions on depth to groundwater, groundwater flow characteristics, recharge and discharge areas, groundwater divides, aquifers and groundwater use. Section 3.6 of the EIR, contains a more detailed description of the regional and site-specific groundwater characteristics. Each of these topics is discussed below.

5.1.4.1 Groundwater Aquifers

On a regional scale, groundwater in the vicinity of the project area occurs within two primary hydrostratigraphic units. These units consist of the Precambrian bedrock and the glacial overburden. The glacial overburden can be subdivided into glacial till and outwash. Regionally, the water table resides within the glacial deposits.

The bulk of the Precambrian bedrock, although locally fractured and showing secondary permeability features, typically yields only small amounts of water to wells. As such it does not fit any of the conventional definitions of an aquifer. Within a five mile radius of the site, there are no known uses of groundwater from the Precambrian bedrock, presumably due to the limited availability of water within the bedrock and the presence of more shallow water bearing glacial deposits.

The glacial aquifer consists of saturated deposits of glacial till and glacial outwash. The glacial till typically has a low permeability and typically would not be used as an aquifer except where embedded lenses of sand and gravel exist. Based on the work of Simpkins, *et al.*, (1987), the outwash deposits are relatively prolific on a regional scale. These outwash deposits are typically sand and gravel, and therefore are permeable enough to yield significant amounts of water. As such the outwash deposits form the primary regional aquifer for domestic and municipal water supplies.

The depth to groundwater within the glacial deposits is quite variable. This is attributed to the considerable topographic relief in the vicinity of the project area. Near groundwater discharge areas such as streams, lakes and wetlands, the depth to groundwater is on the order of several feet. In groundwater recharge areas located at topographic highs, or in upland areas, the depth to groundwater can be more than 100 feet. At the TMA site, the depth to groundwater ranges from 60 to 130 feet.

5.1.4.2 Regional Groundwater Flow System

Groundwater flow occurs primarily within the glacial till and outwash deposits with the majority of flow occurring within the more permeable outwash deposits. Flow within the outwash can typically be characterized as occurring under unconfined conditions. However, semi-confined conditions may occur locally within the outwash due to the presence of lacustrine clay lake beds overlying the outwash or in instances where the outwash deposits are overlain by less permeable glacial till.

On a regional scale, groundwater flows from areas of high hydraulic head to areas of low hydraulic head or from potentiometric highs to potentiometric lows. The potentiometric surface is defined as the height to which water will rise in a well tapping a confined aquifer and is the water table under unconfined conditions. Thus under unconfined conditions, areas of high hydraulic head correspond to topographic highs in the regional water table. Similarly, areas of low hydraulic head correspond to topographic lows in the regional water table. Under unconfined conditions the potentiometric surface and water table are the same except in areas where strong vertical gradients exist, such as would occur near discharge points and some lakes.

High water table elevations occur within upland areas and around upland lakes and wetlands, whereas low water table elevations occur in low land areas associated with streams, wetlands or lakes. Groundwater is replenished in recharge areas defined as an area where the net vertical component of flow is downward. Recharge areas are typically associated with upland areas, upland lakes and high water table elevations. Groundwater discharge areas are defined as areas where the net vertical component of flow is upwards. Groundwater discharge areas are typically restricted to a narrow band around streams, wetlands and or lakes.

In 1982 Golder Associates prepared a regional potentiometric surface map depicting the horizontal flow of groundwater within the glacial deposits. This potentiometric surface map, shown in Figure 3.6-11 of the EIR, was prepared from water levels taken in the study area in February 1982, and from regional groundwater elevations available from the USGS. The map covers a large geographic area encompassing Lakes Metonga and Lucerne to the north; Roberts and Lost Lakes to the east; Pickerel, Crane, and Rolling Stone Lakes to the south; and Bishop Lake to the west.

The potentiometric surface map is roughly equivalent to the regional water table, depicting the change in the water table elevation and the horizontal hydraulic gradient on a regional scale. Areas where the potentiometric surface map would not be equivalent to the water table, would be in the vicinity of upland lakes, such as Little Sand Lake, that are hydraulically connected to the regional groundwater system. In these localized areas a mounded water table exists due to the occurrence of low permeability lacustrine clay and silt lake beds which create semi-confined conditions in the underlying glacial till and outwash. The large scale potentiometric surface map does not reflect these small localized mounded systems. Rather, the regional potentiometric surface map represents the potentiometric surface in the semi-confined underlying glacial till and outwash.

Figure 3.6-11 of the EIR shows a well defined groundwater flow system constrained by a network of creeks and lakes. The controlling surface water features, moving clockwise from the north, are Lake Metonga, Swamp Creek, Hemlock Creek, Ground Hemlock Lake, Crane Lake, Pickerel Lake, Rolling Stone Lake, Pickerel Creek and Mole Lake, Rice Lake and Gliske Creek. Groundwater flows from water table highs located in upland areas to discharge areas controlled by the above referenced surface water features.

The regional groundwater high of about 1,640 feet is located in the vicinity of Lake Lucerne. The groundwater divide for the hydrologic basin is located just to the north of Lake Metonga and to the east of Lake Lucerne. South and east of this divide groundwater flows into Lake Metonga or Swamp Creek. South and west of Lake Metonga groundwater flows into Outlet Creek, Swamp Creek, Rice Lake or Gliske Creek. Outlet Creek, which drains Lake Metonga, is a tributary of Swamp Creek.

A second groundwater divide lies south of Swamp Creek in the vicinity of the proposed TMA. Flow north of this divide is to Swamp and Hemlock Creeks. South of this divide flow is southeasterly and southwesterly toward Crane, Pickerel, and Rolling Stone Lakes, and Pickerel Creek between Rolling Stone and Mole Lakes.

5.1.5 Regional Groundwater and Surface Water Quality

Information on regional water quality was obtained from Olcott (1968). The following sections describe generally the regional groundwater and surface water quality. More detailed descriptions of groundwater quality are contained in Section 3.6.3.3 and 3.6.5.2 of the EIR. More detailed descriptions of the surface water quality are contained in Section 3.7 of the EIR.

Groundwater in the Fox River-Wolf River Basin and Menominee River-Oconto River Basin is described as being of good quality, derived primarily from coarse grained glacial deposits. The groundwater is hard to very hard and of the calcium-magnesium-bicarbonate type. Total dissolved solids generally range from 200 to 300 mg/L, but can be as high as 400 mg/L. Iron can be high locally.

Surface water quality is generally good. Rapid movement of groundwater and runoff through Pleistocene sands and gravels and the generally low solubilities of the Precambrian rock results in rather low surface water mineralization (less than 200 mg/L). Surface water is generally moderately hard to very hard and of the calcium-magnesium-carbonate type. Surface water may also be moderately high in iron. Surface water, especially streams, in the region can be subject

to rapid changes in chemical quality due to sudden large quantities of overland flow resulting from precipitation events.

5.1.6 Regional Climatology-Meteorology

The climate of the region is continental, modified slightly by Lake Michigan, 80 miles to the east, and Lake Superior, 93 miles to the north. The region has moderately warm summers with prevailing westerly winds and long cold winters. Summer weather is generally mild. Temperatures rarely exceed 95 degrees Fahrenheit (°F) during the day, while night temperatures range between 50°F to 60°F. Winter temperatures range from 0°F to 25°F, and occasionally will fall below -30°F at night.

Historical monthly high and low temperatures, averaged from 1961 to 1990, were obtained from the State Climatologist for North Pelican Lake, approximately 15 miles west of the site. These temperatures are presented in Table 3.4-1 of the EIR. The highest average temperature, 77.2°F, occurs for the month of July. The lowest average temperature, 0.1°F, occurs for January. The highest actual temperature on record for North Pelican Lake (1950-1994) was 98°F in July 1955. In February 1967, the lowest actual temperature, -44°F, was recorded.

Average historical monthly precipitation at North Pelican Lake, as shown in Table 3.4-1 of the EIR ranges from approximately one to four inches. The greatest monthly average, 4.14 inches, occurs in the month of August. The least precipitation occurs in February with an average of 0.82 inches. In total, the historical average annual precipitation at North Pelican Lake is 30.36 inches. Snow fall in the region averages between 40 and 60 inches per year. Total monthly precipitation data collected from the meteorological station at Site No. 1 is also presented in EIR Table 3.4-1 for July 1994 through February 1995.

Wind speed and direction can be effectively presented in the form of wind rose diagrams. Wind roses are used to depict the frequency of occurrence of wind direction and speed in each of 16 compass directions. Four wind roses were prepared for the Crandon Project study area based on the data collected at the meteorological station at Monitoring Site No. 1. Figures 3.4-2, through 3.4-5 in the EIR show wind roses for the periods of July-September 1994, October-December, 1994, January-February, 1995 and July 1994-February 1995, respectively.

The wind roses show that over the period of record in the study area, winds typically come from the west, south and southwest, with a smaller component from the northwest. July through September winds over the period of record typically originate from the south, northwest, southeast and west. October through December winds originated most frequently from the south, west and southwest; while those during January and February typically originated from the west, northwest and southwest.

5.1.7 Regional Terrestrial Biology

A comprehensive regional ecology discussion is presented in Section 3.9.2 of the EIR. The regional study area which includes the TMA site has been defined as the land within an approximate 25 mile radius of the mine site. It is comprised of private land, national forest, county forest, and Native American reservation lands.

A vegetational transition zone crosses Wisconsin from the northwest to the southeast, separating two major vegetational provinces. The study area is located in the northern zone. This area is typically characterized by a mixture of coniferous and deciduous trees in upland areas, and conifer swamps in forested wetland areas. The two most abundant vegetation types are Northern Hardwood (including Aspen-Birch), and Swamp Conifer. The northern hardwood element is the most dominant, and is generally composed of maple, aspen, and birch. The swamp conifer is the minor element, and is generally composed of spruce, tamarack and cedar. The remainder of the region is comprised of shrub swamp, bog, marsh, urban or developed areas, old field and clear cut, and agriculture.

Three hundred eighty nine species of higher vertebrates spend some or all of their life in the upper Great Lakes region. The distribution and abundance of species found in an area will vary greatly depending on habitat types, as well as forest age class, canopy cover, ground cover, moisture, and time of the year.

Fifty-four species of amphibians and reptiles are known to be indigenous to Wisconsin. Of these, 32 species could be expected to occur in the general region. There are six species of salamanders, 11 frogs and toads, five species of turtles, nine snakes, and one lizard. The majority of these are associated with wetlands. They tend to take advantage of any cover ranging from leaf litter to hollowed out rotten logs.

Over 75 species of wild mammals range throughout the Great Lakes region. Of these, 55 species of mammals would be expected to be found in the general region. The list of potential species includes some mammals (e.g., moose, pine marten) which are represented by only occasional sightings or very local populations.

Glacial ice covered most of Canada and the upper midwestern United States, including the study area, between 18,000 to approximately 10,000 years B.P. The biotas that had previously occupied these areas were displaced by the ice; today's fauna consisted of species that have recolonized the region during geologically recent times. With the exception of highly mobile organisms such as birds, the diversity of mammals, herptiles, fishes and probably most groups of invertebrates is significantly lower in glaciated regions of North America than it is in southern regions which were not annihilated by glacial advances. Species ranges and ecological assemblages in the upper midwest probably are still in a state of flux as the ranges of different species expand and contract at different rates. Removal of important species (e.g., beaver, pine marten, fisher) by 18th and 19th century trappers, deforestation and subsequent growth of secondary forest during the early 1900s, and the elimination or near elimination of top predators (gray wolf, wolverine) has imposed additional dramatic changes in the fauna of the study area and nearby regions. As a result, today's assemblages of animals and plants in northern Wisconsin cannot realistically be viewed as highly integrated, ecologically distinct communities.

5.1.8 Regional Aquatic Biology

The Crandon Project study area is surrounded by two watersheds, the Swamp Creek and Pickerel Creek chains, both of which are within the Wolf River drainage basin. Some of the lakes located within the Crandon Project study area are connected directly via surface flow to one or the other chains, while other lakes are indirectly connected via subsurface flow or are isolated. Hemlock Creek which borders the project study area to the east, is directly connected to the Swamp Creek chain. Some lakes such as Duck Lake, Little Sand Lake, and Deep Hole Lake are at times

directly connected to the Pickerel Creek chain via Creek 12-9, however, only during periods of extremely high water is this a direct connection. In most cases, the outlet of Little Sand Lake is no more than recharge to adjacent wetlands. These physical connections are important from the standpoint of species distribution and migration up the Pickerel Chain, especially for species of fish and mussels. Other lakes which appear to have similar isolated conditions include Skunk Lake which indirectly drains toward the Pickerel Chain, and Oak Lake and Popple Pond which drain toward Swamp Creek. Another factor which restricts the migration of fish and mussels includes natural barriers such as beaver dams. This is especially true along Hemlock Creek and Swamp Creek above Rice Lake.

Biologically, the Swamp Creek watershed is connected only with the Wolf River watershed. There are no other surface water connections with any other watersheds via lakes or streams that would allow movements of fish, macroinvertebrates, or other forms of aquatic biota between watersheds. Aquatic insects, most of which are capable of dispersal flight in the adult stage, could conceivably colonize any suitable habitat within the project area, as physical/geographical barriers here are extremely slight. This is especially true for the Odonata. Differences in insect species composition at the different sample sites is much more dependant on specific habitat requirements.

The flow rate of tributary streams to Swamp Creek and Rice Lake is sufficient to support biological populations that could serve as sources for colonization of downstream water bodies. There is no linkage for aquatic species between the Pickerel Creek watershed and any adjacent watersheds except the Wolf River. No surface water connections exist with any other watersheds via lakes or streams that would allow movements of fish, macroinvertebrates, or other forms of aquatic biota between watersheds. The flow rate in tributary streams to Rolling Stone Lake is sufficient to support biological populations that could serve as sources for colonization of downstream water bodies.

Streams of the study area generally share similar physical characteristics. The streams are small, often not well defined, and of low gradient. Generally, the creeks have surface water drainage from and through wetland areas and the creek bottoms are composed of soft muck. However, small areas of firm sand and gravel substrate also exits in portions of most creeks. The Wolf River also drains wetlands but the bottom composition is generally more firm than the creeks. The extent of riffle habitat is minor in most of the streams with the only significant riffle area being found in Swamp Creek north of the project area. This is important for survey purposes, as riffles can support high biological diversity and a relatively high number of sensitive species.

5.2 Site Specific Geotechnical Information

5.2.1 General

This section is included to fulfill the requirement of NR 182.08(2)(d) for the presentation and discussion of the following site-specific topics: existing conditions, geotechnical investigation, laboratory soils and rock testing, installation of water table observation wells and piezometers, geologic cross-sections, groundwater contour maps and flow nets. These topics are addressed in the subsections which follow. Much of the field and laboratory data accumulated during the investigations conducted in the tailings management area are summarized in tables in this section. The field logs and data documentation are contained in the following appendices associated with the EIR.

Appendix 3.5-1	Logs for Dames & Moore 1977-1979 DMA, DMB, DMC, DMI, DMP, DMS, DW, and WW.
Appendix 3.5-2	Logs for Golder Associates 1979-1982 G40 and G41 Borings and Boring For TW-41 Test Well
Appendix 3.5-5	Logs for STS Consultants, Ltd. 1984 WP and EX Series Borings (WP-1 through 7 and EX-1 through EX-16)
Appendix 3.5-8	Logs for STS Consultants, Ltd. 1984 Duck, Skunk, Oak and Deep Hole Lake Borings (STS-DL-1, SL-1, OL-1 and DHL-1)
Appendix 3.5-9	Logs for STS Consultants, Ltd. 1985 Skunk and Duck Lake Borings (SL-3 through 5 and DL-2)
Appendix 3.5-10	Logs for WTD Environmental Drilling, Inc. 1994 Little Sand Lake; Skunk and Duck Lake Wetlands; Burr Oak Swamp; Swamp Creek Cedars and Wetlands Adjacent to Swamp Creek Borings
Appendix 3.5-12	Logs for WTD Environmental Drilling, Inc. 1994 Groundwater Monitoring Well Installation Program Borings
Appendix 3.5-14	Test Pit Logs
Appendix 3.5-16	Grain Size Curves from 1977-79 Dames & Moore "DM" Series Borings
Appendix 3.5-17	Grain Size Curves from 1979-82 Golder Associates "G40 and G41" Series Borings and Miscellaneous Test Pits
Appendix 3.5-19	Grain Size Curves from 1984 STS Consultants Ltd 1984 "Ex" Series Borings
Appendix 3.5-20	Grain Size Curves Reclaim Pond Borings from 1984 STS Consultants Ltd
Appendix 3.5-21	Grain Size Curves for STS Consultants Ltd 1984 Tests on Samples from the G40 and G41 Series Borings
Appendix 3.5-22	Grain Size and Proctor Curves for STS Consultants Ltd 1985 Tests on Samples from the Proposed MRDF Test Pits
Appendix 3.5-23	Grain Size Curves from 1994 Foth & Van Dyke Verification Groundwater Monitoring Well Installations
Appendix 3.5-27	Grain Size Curves and Permeability Test Results from STS Consultants LTD 1984 Duck, Skunk, Oak and Deep Hole Lake Borings
Appendix 3.5-28	Grain Size Curves and Permeability Test Results for 1994 Little Sand Lake; Skunk and Duck Lake Wetlands; Burr Oak Swamp; Swamp Creek Cedars and Wetlands Adjacent to Swamp Creek Boring Program
Appendix 3.5-30	Forest County Soil Data Sheets Source: USDA-SCS Section 11a Technical Guide, June 13, 1994, Draft
Appendix 3.6-1	Well Completion Records
Appendix 3.6-2	Dames & Moore Glacial Overburden Slug Test Results
Appendix 3.6-4	Golder Associates 1981 Pump Test Analyses, Crandon Project Waste Disposal System
Appendix 3.6-5	STS Consultants Ltd. 1984 Slug Test Results
Appendix 3.6-6	Foth & Van Dyke Slug Test Results for 1994 Lake Bed and Wetland Investigation
Appendix 3.6-8	Crandon Project Groundwater and Surface Water Elevation Data Base, Hydrographs Through April 1995 and Gradient Calculations

Appendix 3.6-10	Dames & Moore. 1978-1980 Groundwater Quality Data
Appendix 3.6-11	May 1994 to January 1995 Groundwater Quality Laboratory Results
Appendix 3.6-12	1994 and 1995 Groundwater Quality Database
Appendix 3.6-13	Statistical Analysis of May 1994 Through January 1995 Groundwater Quality Data
Appendix 3.6-14	Mann-Whitney U Test Results and Boxplots

5.2.2 Existing Site Conditions Map NR 182.08(2)(d)(1)

A detailed aerial topographic survey of the mine site area was completed on April 28, 1976 by Aero-Metric Engineering, Inc. of Sheboygan, Wisconsin. Borings, monitoring wells, piezometers, staff gauges and test pits were surveyed later using existing control points by Inman-Foltz and Associates, Inc. and Foth & Van Dyke and Associates, Inc. This existing conditions map was then digitized from a one inch equals 100-foot scale map prepared by Aero-Metric Engineering, Inc. A scale one inch equals 300 feet was used on the existing conditions map (Drawing No. 3). The contour interval for topography is five feet. The existing site conditions map does not meet the requirements of NR 180.08(2)(d)1 for a contour interval less than four feet nor for a scale no greater than one inch equal to 200 feet. This was due to the large area involved in this project. To meet the requirements of NR 182.08(2)(d)1 multiple sheets would have been required. For purposes of this report, the 1"=300' scale and five-foot contour interval is sufficient to show the required detail and that the option of using a smaller scale and smaller contour interval and splitting the map would result in added complexity to the review process by not showing all the facilities on one sheet.

5.2.3 Geotechnical Investigation NR 182.08(2)(3) 1, 2, 3, and 4a

5.2.3.1 General

NR 182.08(2)(d)1 requires a field investigation to define the site specific topography, soil types, depth to bedrock and groundwater characteristics. The geotechnical investigation of the tailings management site has been ongoing in phases since the late 1970s. The following subsections are a summary of the results of these investigations. The methods of investigation and original data interpretations are available in the reports which are referenced in this chapter and those chapters referred to in the EIR.

Geotechnical investigation at the TMA site began in 1977. The first phase and methods used during this portion of the TMA investigation are described in Section 3.5.1.2 of the EIR. This portion of the investigation was conducted under the supervision of the Dames & Moore. This phase was designed to provide a basic understanding of the geology and hydrogeology of the study area. This study was followed by an investigation conducted under the supervision of Golder for 1979 to 1982 as part of the geotechnical and hydrogeologic investigations for siting and design of the TMA. These investigations were conducted in two phases as described in Section 3.5.1.2.3 of the EIR. It was the second stage of this investigation which included the G41 series of borings that was directed at the area around the current proposed TMA location. Section 3.5.1.2.3 of the EIR describes the methods used during this portion of the TMA investigation.

In 1984, STS Consultants, Ltd. completed a series of borings, the EX-Series, in which a number of nested piezometers and wells were installed. The purpose of this investigation was to increase

the resolution of the hydrostratigraphic units within the study area. The methods used during this investigation are described in Section 3.5.1.2.5 of the EIR.

The most recent investigation of the TMA occurred during 1994 under the supervision of Foth & Van Dyke and Associates, Inc. This investigation involved the installation of groundwater monitoring wells. The methods and procedures are described in Section 3.5.1.4 of the EIR.

5.2.3.2 Boring/Test Pit Location, Drilling Depth and Details

Since the late 1970s, hundreds of soil borings, monitoring wells and test pits have been completed to aid in the design of the TMA. Drawing No. 3 shows the location of these soil borings, monitoring wells and test pits. Documentation of these borings, monitoring wells and test pits is provided in the EIR Appendices listed in Section 5.2.1 of this report. Summary tables of the borings and wells can be found in Tables 3.5-2 and 3.6-1 of the EIR.

5.2.4 Laboratory Soil Testing of Soil-Rock

During the many investigative phases of the TMA site, soil samples were selected from numerous borings at a number of various depths and tested for grain size, Atterberg Limits and laboratory permeability. A summary of the data collected during the latest phase of investigation is presented as Table 5.2-1. Summary results of soil testing from earlier investigations discussed in Section 5.2.3 are presented in Appendix C of this document. Individual soil laboratory data sheets are presented in the EIR Appendices listed in Section 5.2.1 of this report.

The TMA site geologic cross-sections presented in Drawings Nos. 5 through 12 contain summary tables with the soil samples analyzed from each boring and the test results.

The general and site specific geologic interpretations from previous and current investigations are described in detail in Sections 3.5 and 3.6 of the EIR. The following descriptions in this report are a summary of the general descriptions of each stratigraphic unit and the general physical properties of each unit.

5.2.4.1 General Descriptions of Unconsolidated Deposits

Unconsolidated deposits in the TMA site area have been separated into several stratigraphic and hydrostratigraphic units in the study area around the TMA. These are Late Wisconsinan till, coarse-grained outwash, fine-grained outwash, ice marginal sediments, lacustrine sediments, and Pre- to Early-Wisconsinan till. Massive saprolite was not delineated in the TMA as was done at the mine site since the extent of the saprolite below the TMA is not of great concern. Detailed information of the delineation of units in terms of properties, stratigraphic units and hydrostratigraphic units is contained in Sections 3.5 and 3.6 of the EIR.

The bedrock in the vicinity of the TMA is covered by 150 to 230 feet of Pleistocene glacial silt, sand, and gravel. These unconsolidated glacial deposits consist of undifferentiated sediments of the Marathon Formation, undifferentiated sediments of the Lincoln Formation, the Mapleview Member of the Horicon Formation, and the Nashville Member of the Copper Falls Formation. Each of these units consists of till and associated meltwater sand and gravels. The surficial sediments on the site belong mainly to the Copper Falls Formation. These are buried locally by wind blown silts, alluvium, peat, and lacustrine sediments.

Table 5.2-1

1994 Investigation Soil Grain Size Results from Proposed TMA Area

Boring ID (#)	Depth (ft)	Gravel (%)	Sand (%)	P200 (%)	Silt (%)	Clay (%)	USCS	Unit
CMC-01	82.5-84.5	8.2	69.1	22.7	-	-	SM	LWT
CMC-01R	53-55	15.3	61.6	23.1	-	-	SM	LWT
CMC-01P	143-145	3.1	83.9	13.0	-	-	SM	CO
CMC-02P	191-193	30	53.6	16.4	-	-	SM	CO
CMC-03RR	113-115	22.5	58.8	18.7	-	-	SM	CO
CMC-03P	190-191	33.6	54.1	12.3	-	-	SM	CO
CMC-04	108-110	2.5	86.6	10.9	-	-	SP-SM	CO
CMC-05	73.4-74.2	5.9	74.4	19.7	-	-	SM	LWT
CMC-06P	145-147	34.3	55.5	10.2	-	-	SP-SM	CO
CMC-07	40-42	9.8	71.5	18.7	-	-	SM	CO
CMC-BO-101A	14-16	0.4	23.3	73.3	53.6	22.7	CL	LA
CMC-BO-101B	4-6	0	53.7	46.3	32.5	13.8	SC-SM	IM
	12-14	0	4.4	95.6	73.3	22.3	CL	LA
	36-38	1.8	95.6	2.6	-	-	SP	CO
CMC-BO-102	16-18	17.6	62.4	20	-	-	SC	LWT
	35-37	0	82.9	17.1	-	-	SM	CO

LWT - Late Wisconsin Till
CO - Coarse Outwash
IM - Ice Marginal
LA - Lacustrine

Prepared by: EAS
Checked by: REM

The oldest Pleistocene deposits in the Crandon region belong to Pre- to Early-Wisconsinan Marathon Formation and the Pre- to Early-Wisconsinan Lincoln Formation. These units generally overlie saprolite or bedrock. The Marathon Formation is a brown loam till and the Lincoln Formation is a brown sandy loam till. These units have been combined on the TMA geologic cross-sections due to their similar physical properties and are not differentiated for the purposes of groundwater modeling (see Drawings No. 5 through 12). The Marathon-Lincoln Formation Complex has been called Pre- to Early-Wisconsinan till. In the TMA area this unit may also include some massive saprolite as earlier work in the area did not differentiate saprolite in all borings. In the 1984 STS report titled *Hydrogeologic Study Update for the Crandon Project*, this unit was termed the basal till. This unit generally is classified as SM, ML, or GM-GP. The percent passing the #200 sieve is generally less than 50 percent. A summary of grain size distributions from earlier investigations is included in Appendix C. The Pre- to Early-Wisconsinan till is fairly continuous across the TMA area.

Outwash associated with Pre- to Early-Wisconsinan till and the overlying Horicon Formation till has not been differentiated nor has outwash associated with the Horicon Formation been differentiated from outwash associated with the Copper Falls Formation. This was due to the fact that these deposits have similar physical characteristics and grain size distributions and these units are adjacent to one another and interfingered and function as similar hydrostratigraphic units. For modeling and cross-section purposes outwash deposits were separated into fine-grained and coarse-grained deposits.

The coarse outwash is mostly continuous across the site and generally occurs between the Horicon Formation till and the Pre- to Early-Wisconsinan till as well as between the Copper Falls Formation till and Horicon Formation till. Where Horicon Formation till is absent coarse outwash separates the Copper Falls Formation till from Pre- to Early-Wisconsinan till. The coarse outwash generally are classified as SP, SW, SM-SP, SM, GP-GM, GP, GW, or GM. A summary of grain size distributions from previous investigations is included as Appendix C of this document. Generally less than 20 percent of the samples passes the #200 sieve. In previous investigations this unit was termed coarse-grained stratified drift.

Fine-grained outwash is generally present above Pre- to Early-Wisconsinan till and below ice marginal deposits. The most continuous of these lies between the Horicon Formation till and Pre- to Early-Wisconsinan till. The fine outwash is classified as SP, SM-SP, or ML. A summary of grain size distributions from previous investigations is included in Appendix C of this document. Generally less than 40 percent of the sample passes the #200 sieve. This unit was termed fine-grained stratified drift in previous investigations.

The Mapleview Member of the Horicon Formation and the Nashville Member of the Copper Falls Formation were both deposited during Late Wisconsinan time. In the vicinity of the TMA site, till of the Nashville Member overlies till of the Mapleview Member. The two tills are often separated by coarse outwash but in some areas may be in direct contact. The Mapleview may also be absent in areas of the TMA. The Mapleview Member consist of a cobbly, gravelly, silty sand till with boulders. Till of the Nashville Member is a gravelly, cobbly, sandy loam. The two can be distinguished by the higher percentage of sedimentary rock clasts, primarily dolomite, present in the Mapleview Member. Due to their similar physical characteristics and contemporaneous stratigraphic position, these tills have been combined for the purposes of constructing cross-sections and modeling called Late Wisconsinan till. These tills are classified as

SM-SP, SM or SC. A summary of grain size distributions from previous investigations is included as Appendix C. Usually no more than 50 percent of the sample passes the #200 sieve.

In the vicinity of the TMA glacio-lacustrine deposits occur locally on the Copper Falls Formation tills and on Pre- to Early-Wisconsinan till. These deposits are generally fine grained and varved. The lacustrine deposits classify as SM-SP, SM, ML, and CL. A summary of the grain size distribution of these deposits from previous investigations is included as Appendix C.

5.2.4.2 General Physical Properties of Soil Units

The physical properties of the aforementioned unconsolidated sediments and comparisons of previously collected data to data collected during more recent investigations are contained below.

Appendix C presents the P200 distribution for the various stratigraphic units based on gradation curves from the EX-series of borings in the study area. Table 5.2-2 summarizes this information and provides comparison of this data to the soil results from samples collected during the 1994 monitoring well installation and wetland investigations.

Undifferentiated Late Wisconsinan Till - This till was termed Glacial Till during previous investigations. Eighty samples of this unit were analyzed during previous investigations and five were analyzed during the 1994 series of investigations.

Comparison of these results is summarized on Table 5.2-2. A composite gradation curve for samples from earlier investigations is included in Appendix C. Individual gradation curves for all samples are contained in Appendices 3.5-16, 3.5-17, 3.5-19 through 3.5-23 of the EIR. Typically, the samples fall under SM-SP, SM, or SC under the Unified Soil Classification System. The samples were described as slightly silty fine to silty fine to coarse sand with gravel and cobbles.

Undifferentiated Pre- to Early-Wisconsinan Till - An older till unit was encountered generally just above bedrock across much of the site. It was distinguishable from the Late Wisconsinan tills by color and texture. The Pre- to Early-Wisconsinan till, previously termed basal till, was generally dark brown to red-brown, had a higher P200 content and was more consolidated.

Eleven samples were analyzed from earlier investigations. These were described as silty fine to coarse silty sand, silty fine to coarse gravel or coarse sandy silt with clay and cobbles. A composite gradation curve is presented on Appendix C. No samples from borings in the TMA area were analyzed for properties of the Pre- to Early-Wisconsinan till from current investigations.

Table 5.2-2

Soil Sample Analysis Summary

Stratigraphic Unit	1994 Investigation				Previous Investigations			
	Number of Samples *	P200 Mean Percent	P200 Range Percent	USCS Classification	Number of Samples	P200 Mean Percent	P200 Range Percent	USCS Classification
Undifferentiated Late Wisconsinan Till	4	24	20.0-28.6	SM, SC	80	20	6-45	SM, SM-SP
Undifferentiated Pre- to Early-Wisconsinan Till	0	--	--	--	11	36	9-90	ML, SM, GM-GP
Coarse-grained Outwash	9	14	2.6-19.4	SM, SM-SP, SP	55	7	1-16	SM, SM-SP, SP
Fine-grained Outwash	0	--	--	--	30	11	2-51	SM, SM-SP, SP
Late Wisconsinan Lacustrine and Ice-Marginal Deposits	3	72	46.3-95.6	CL, SC-SM	19	66	9-98	SP, ML, CL

* Note: These results only include samples collected from monitoring wells and piezometers in the immediate vicinity of the TMA.

Prepared by: EAS
Checked by: REM

Coarse-grained Outwash - This unit consists of the coarse-grained glacial outwash sands and gravels found in the TMA. This unit was termed Coarse-Grained Stratified Drift in previous investigations. Fifty-five samples of this outwash were analyzed during previous investigations from the TMA area. Nine samples were analyzed from borings completed during 1994 investigations. Comparison of these results is presented on Table 5.2-2. A composite gradation curve is presented as Appendix C. The coarse outwash samples range from well-sorted silty fine sand to well-graded sand and gravel. The low P200 content and presence of stratification indicated fluvial deposition as the mechanism of sedimentation.

Fine-grained Outwash - This unit consists of fine-grained, well-sorted fine sand, silty fine sand or fine sandy silt. This unit was termed fine-grained stratified drift during previous investigations. Appendix C displays the composite gradation curve for 30 fine-grained outwash samples from the EX borings. The samples were generally classified as SP, SM-SP, SM or ML under the Unified Soil Classification System.

Lacustrine - The lacustrine deposits include both sediments which are Late Wisconsinan in age and Holocene in age. The fine-grained sediment are present below existing lake and wetlands and to a lesser extent depth separating the aforementioned units. Appendix C is a composite gradation of 19 samples analyzed from previous investigations. These are compared to three samples analyzed during the 1994 investigation. These samples were from lacustrine-ice marginal deposits located in wetlands adjacent to the proposed TMA. These samples were generally described as fine sands, silts and clays. Many of the samples were varved.

5.2.4.3 Bedrock Delineation

Bedrock in the TMA was delineated during the many phases of investigations. Six 984-foot bedrock diamond drill core holes (#'s 199, 200, 203, 204, 205 and 206) were placed in the area around the proposed TMA. Several meters of bedrock core were also taken from the top of six glacial overburden holes (G41-C15, G41-E13, G41-E17, G41-E19A, G41-F24 and G41-P24) as well as cuttings from glacial overburden hole CMC-01P to delineate bedrock geology. Plan views of the bedrock geology as well as cross-sections of the bedrock geology in the TMA area are included in Section 3.5.3.1 of the EIR. The bedrock is predominately a very fine-grained tuffaceous sequence with flows common found with interflow tuff beds. A laminated tuff sequence occurs in the central to southern portions of the TMA. South of the laminated tuff lies a crystal tuff sequence.

5.2.4.4 Field Hydraulic Conductivity and Laboratory Permeability Testing

A number of programs have been conducted since the late 1970s to evaluate the hydraulic conductivity of major hydrogeologic units that control regional groundwater flow and the interaction of groundwater between the glacial overburden units. These testing programs consisted of lab permeability tests, slug tests on piezometers, packer tests in open bedrock exploration holes and a pump test in the glacial drift. A brief description of each testing program is presented in Sections 3.6.1.6.1 through 3.6.1.6.5 of the EIR.

Soil permeability tests in the TMA study area were determined as part of the regional hydrogeologic investigation for the installation of DM series borings and piezometers. Information on *in situ* permeability was collected by conducting falling and rising head slug tests on selected DM series wells.

Rising head and falling head tests were conducted on eight piezometers in the area of the proposed TMA. These tests were conducted by changing the static water level in the well by adding or removing water. The subsequent return of the water level to the static condition is then recorded over time. Field data collected during the falling and rising head tests were analyzed using Hvorslev's relationship (1951), which is an exponential curve function that relates the change in head in a piezometer versus time to arrive at an estimate of *in situ* soil permeability. Figure 3.6-5, of the EIR shows the location of DM series wells that slug tests were performed on.

In 1981, Golder Associates performed a pump test in the vicinity of the TMA site to assess major hydraulic parameters of the glacial till and outwash deposits. The pump test was comprised of a test well, 13 primary observation wells, and 17 secondary observation wells. The test well was screened over the full depth of the saturated glacial deposits. Observation wells were completed at various distances from the test well and at various depths. The drawdown phase of the pump test lasted 24 days, from June 27 to July 21, 1981, and were conducted at an average pumping rate of 1,420 gpm. Pump test drawdown and recovery data was analyzed according to standard type curve analysis. Figure 3.6-7, of the EIR shows the location of the test well and observation wells used in the pump test. Summary of the results of this pump test is displayed on Table 5.2-3.

Field permeability (hydraulic conductivity) tests were conducted by STS in 51 EX piezometers, seven well points, and 27 wells from previous investigations. Where possible, both rising and falling head tests were conducted in each well. Falling head tests were only conducted in piezometers sealed into very low permeability formations, such as bedrock. In the small diameter BE series wells, rising head tests were only conducted by using compressed air to remove the water column and then the rate of water level recovery was manually recorded. Except for the small diameter wells, the field permeability tests were conducted using a pressure transducer and data logger.

The general procedures started with placing a 25 psi transducer approximately 20 feet below the static water level. The model PT-105V pressure transducer was connected to the Enviro Labs model DL120 data logger which has a microprocessor capable of being programmed to take and store transducer readings at any intervals from ten seconds to 24 hours. Once the data logger was programmed to begin logging, a solid one-inch diameter PVC slug approximately 15 feet long was dropped into the borehole to displace the column of water. Once the falling head test was completed, the slug was removed and a rising head test was conducted. After the test was completed, the data were printed out. Recovery data was analyzed according to methodology developed by the Department of Navy (1971), known as the DM-7 Method. The methods used during this investigation are described in more detail in Section 3.6 of the EIR.

Field permeability test data is provided in Appendices 3.6-2 through 3.6-6 of the EIR. The first sheet provides a summary of the well construction details and permeability results. The second sheet is the graphical plot of the head ratio versus time. The third sheet is a summary of the field data (transducer readings). Figure 3.6-5, of the EIR, shows the location of the EX and WP series wells that slug tests were performed on.

Table 5.2-3

Hydraulic Parameter Summary From 1981 Glacial Overburden[a]
24-Day Pump Test

Geologic Unit	Horizontal Hydraulic Conductivity Kh		Vertical Hydraulic Conductivity Kv		Specific Storage Ss	Specific Yield Sy
	ft/d	cm/s	ft/d	cm/s	ft^{-1}	Percent
Late Wisconsinan Till	0.8	2.8×10^{-4}	0.27	9.4×10^{-5}	4.6×10^{-6}	5.4
Outwash	37.2	1.3×10^{-2}	3.7	1.3×10^{-3}	4.6×10^{-6}	7.0
Pre- to Early Wisconsinan Till	0.8	2.8×10^{-4}	0.27	9.4×10^{-5}	4.6×10^{-6}	5.4

[a] Obtained from Golder, 1981.

Prepared by: SVD1
Checked by: XXC

Wells installed during the wetland investigation during 1994 were slug tested using methods similar to the 1984 series of slug tests. In general, a 25 psi transducer was placed in the well and connected to a Campbell Scientific Instrument data logger. Once the data logger was programmed to begin operating, a known volume of water was withdrawn using a bailer or a slug was inserted into the well and recording of well recovery was initiated. The results of these *in situ* hydraulic conductivity tests are summarized on Table 5.2-4.

The only laboratory permeability testing conducted in the immediate vicinity were on the thin wall tube samples of ice-marginal and lacustrine sediment from Duck Lake in 1984 and from Burr Oak Swamp in 1994. The Duck Lake lacustrine samples, one from 25.5-28 feet and one from 38-40 feet, yielded coefficients permeability of 1.3×10^{-6} cm/sec and 1.0×10^{-7} cm/sec, respectively. The ice-marginal and lacustrine samples from Burr Oak Swamp, one from 4-6 feet and one from 12-14 feet, yielded coefficients of permeability of 1.0×10^{-6} cm/sec and 2.9×10^{-9} cm/sec, respectively.

5.2.5 Groundwater Table Observation Wells and Piezometers

Groundwater monitoring wells and piezometers have been installed throughout an extended series of investigations as described in Section 3.6.1 of the EIR. This information is summarized in Sections 5.2.3 of this report. Construction details, location, formation monitored and the current status of the wells and piezometers in the TMA are included in Section 5.2.3 of this report on Table 5.2-5. Well construction reports from the drilling contractors are included in Appendix 3.5-12 of the EIR as are WDNR Soil Boring Log Information Form (Form 4400-122) for borings included in the latest round of investigation conducted by Foth & Van Dyke. Table 5.2-5 lists the active and temporarily abandoned wells located specifically within the compliance boundary. Currently, existing active wells are being monitored for groundwater elevations or background water quality.

5.2.5.1 Well Construction Reports

The well construction reports for the wells used to provide background data for the TMA are included in Appendices of the EIR. Table 5.2-5 contains a list of wells specifically located within the proposed TMA compliance boundary.

5.2.5.2 Well Development

The well development details for wells used in the collection of background information are contained in Appendices 3.5-1, 3.5-2 and 3.6-1 of the EIR, along with well construction details.

All wells located within the TMA study area and used to monitor background groundwater quality were developed multiple times as described in Section 3.6. of the EIR.

5.2.5.3 Water Level Measurements

Water level measurements from the wells and piezometers installed during the different phases are included in Appendix 3.6-8 of the EIR. Appendix 3.6-8 of the EIR also contains hydrographs of the recorded water level measurements for the wells referenced in this report. Procedures utilized in the measurement of water levels are described in Section 3.6 of the EIR.

Table 5.2-4

In Situ Aquifer Response Testing

Well ID	Test Type	Hydraulic Conductivity (cm/sec)	Formation
DMB-1A	FH	6.1×10^{-4}	FO/LWT
DMB-2	FH	5.8×10^{-6}	CO/LWT
DMB-3	FH	5.9×10^{-4}	CO
	RH	1.9×10^{-4}	CO
DMB-4	FH	1.2×10^{-4}	LWT/CO
DMB-5	FH	1.1×10^{-6}	CO/LWT
DMB-5A	FH	2.0×10^{-5}	LWT/CO
DMB-6	FH	1.1×10^{-6}	LWT
DMB-8	FH	2.6×10^{-5}	LWT
EX-7 AL	FH	9×10^{-4}	EWT
	RH	7×10^{-4}	EWT
EX-7 BL	FH	1×10^{-3}	EWT
	RH	1×10^{-3}	EWT
EX-7 BU	FH	1×10^{-3}	CO
	RH	1×10^{-3}	CO
EX-8 AL	FH	1×10^{-4}	EWT
	RH	7×10^{-5}	EWT
EX-8 AU	FH	2×10^{-4}	EWT
	RH	2×10^{-4}	EWT
EX-8 BL	FH	2×10^{-3}	FO
	RH	1×10^{-3}	FO
EX-8 BU	FH	1×10^{-3}	CO
EX-9 AL	FH	2×10^{-8}	UDB
EX-9 AU	FH	6×10^{-3}	FO
	RH	3×10^{-3}	FO
EX-9 BL	FH	1×10^{-4}	CO
	RH	2×10^{-3}	CO
EX-16 AL	FH	1×10^{-3}	FO
	RH	1×10^{-3}	FO
EX-16 AU	FH	4×10^{-3}	CO
	RH	6×10^{-3}	CO

Table 5.2-4 *(Continued)*

Well ID	Test Type	Hydraulic Conductivity (cm/sec)	Formation
EX-16 BL	FH	9×10^{-6}	LWT/FO
	RH	1×10^{-6}	LWT/FO
G41-E17	FH	1×10^{-3}	EWT
	RH	1×10^{-3}	EWT
G41-P16	FH	4×10^{-5}	LWT/CO
	RH	5×10^{-5}	LWT/CO
*CMC-BO-101A	FH	7.1×10^{-5}	IM

Prepared by: EAS
Checked by: REM

Formation Monitored:

LWT - Late Wisconsinan Till
EWT - Pre- to Early-Wisconsinan Till
CO - Coarse Outwash
FO - Fine Outwash
LA - Lacustrine
IM - Ice Marginal
UDB - Undifferentiated Bedrock
FH - Falling Head
RH - Rising Head

* In-situ aquifer response test, EIR, 1995.

Source: EMC, 1985a.

Table 5.2-5

Active or Temporarily Abandoned Monitoring Wells Located Within the TMA Compliance Boundary

Water Table Monitoring Well ID	Associated Piezometer ID	Status	Purpose	Location In Facility	Formation Monitored
DMB-1		TA-Dry	---	Berm	NA
CMC-07		A	BGWM	Berm	CO
	DMB-1A	A	BGWM	Berm	LWT/FO
	G41-E13	A	WL	Berm	FO/UDB
	G41-E13A	TA	---	Berm	FO
DMB-2	NA	TA	---	WIF	LWT/CO
DMB-4		A	WL	CB	LWT/CO
EX-10BU		TA	---	WIF	LWT
	EX-10BL	TA	---	WIF	LWT
	EX-10AU	A	WL	WIF	FO
	EX-10AL	A	WL	WIF	LA
EX-11CL		TA	---	CB	LWT
	EX-11BU	TA	---	CB	LWT
	EX-11BL	TA	---	CB	CO
	EX-11AU	TA	---	CB	FO
	EX-11AL	TA	---	CB	LA/CO
EX-12BU		A	WL	CB	LWT
	EX-12BL	A	WL	CB	CO
	EX-12AU	A	WL	CB	EWT
	EX-12AL	A	WL	CB	LA
EX-9BU		A	BGWM	WIF	LWT/CO
	EX-9BL	A	WL	WIF	CO
	EX-9AU	A	WL	WIF	FO
	EX-9AL	A	BGWM	WIF	UDB
G41-G12		TA	---	WIF	LWT
G41-G13		TA	---	WIF	LWT/CO
G41-G14C		TA	---	WIF	LWT
	G41-G14A	TA	---	WIF	LWT
	G41-G14B	TA	---	WIF	EWT
G41-G14E		TA	---	WIF	CO
	G41-G14D	TA	---	WIF	CO/FO
	G41-G14F	TA	---	WIF	EWT
G41-G15A		TA	---	Berm	CO
	G41-G15B	TA	---	Berm	LWT
	G41-G15	TA	---	Berm	EWT/UDB/LA
G41-H9		A	WL	Berm	LWT/CO/FO/ LA/EWT

Table 5.2-5 *(Continued)*

Water Table Monitoring Well ID	Associated Piezometer ID	Status	Purpose	Location In Facility	Formation Monitored
G41-K13B		TA-Dry	---	WIF	LWT
	G41-G13A	TA	---	WIF	LWT
	G41-G13	A	NA	WIF	CO/LWT
CMC-O1R		A	BGWM	CB	LWT
	CMC-01	A	NA	CB	LWT/CO
	CMC-01P	A	BGWM	CB	CO
CMC-02R		A	BGWM	WIF	LWT/CO
	CMC-02P	A	BGWM	WIF	CO
CMC-03RR		A	BGWM	CB	LWT/CO
	CMC-03P	A	BGWM	CB	CO
CMC-BO-101A		A	WL	CB	LA
	CMC-BO-101B	A	WL	CB	CO

Source: EIR, 1995.

Prepared by: EAS
Checked by: REM

A - Active
TA - To be abandoned
BGWM - Background groundwater monitoring well
WL - Water level observation point
WIF - Located within facility
CB - Located within compliance boundary/outside berm
Berm - Located in berm area
LWT - Late Wisconsinan Till
EWT - Early Wisconsinan Till
CO - Coarse-grained Outwash
FO - Fine-grained Outwash
LA - Lacustrine
UDB - Undifferentiated bedrock
NA - Not Available
--- - Not an active well

5.2.6 Geologic Cross-Sections

Geologic cross-sections illustrating topography, soil borings, soil classifications, soil properties, interpreted soil stratigraphy, bedrock, well and boring locations and constructions and stabilized water level readings have been constructed for the proposed TMA area. These cross-sections were constructed off the regional cross-sections presented in Section 3.5 of the EIR. The cross-sections have been modified to display the compliance boundary, final grades and detailed soil relationships. These cross-sections also contain individual soil samples and the laboratory soil test results.

Detailed descriptions of the individual stratigraphic units are contained in Section 5.2.4 of this report. Comprehensive descriptions of the geology and soils and groundwater for the study area are contained in Sections 3.5 and 3.6 respectively of the EIR.

5.2.7 Groundwater Table Maps

Groundwater levels have been monitored intermittently for several years in the area of the proposed TMA facility. These groundwater levels have been used to construct a groundwater contour map for the month of October, 1984 (Drawing No. 4).

The upland area in which the proposed TMA is to be located acts as a local recharge area. Horizontal gradients in the vicinity range from 0.0013 ft/ft on the west side and central portion of the site to 0.025 ft/ft on the east side of the proposed site near Hemlock Creek. Groundwater flow is generally radially outward from the proposed TMA site except in the vicinity of Duck Lake which also serves as a groundwater recharge area.

The groundwater contour maps portray the area around the site of the proposed TMA facility as a recharge area. This location is a groundwater divide with flow to the north and east toward Swamp and Hemlock Creeks and to the south and west toward Rolling Stone Lake.

5.3 Groundwater Quality Validation

Planning for the groundwater quality validation program included reviewing the cumulative hydrogeological data collected by the various consultants in the 1980s. Based on this review, a groundwater quality validation program was developed. The first step in this program was to select the depths within the overburden aquifer for monitoring. In the vicinity of the TMA, groundwater generally occurs deeper than 40 feet below the land surface. The logical choice, then, was to monitor groundwater at the water table, since this would be the groundwater first potentially affected by project activities. Monitoring of a deeper zone within the aquifer was also desired. The deeper zone selected for monitoring was the glacial outwash deposits, because the outwash would have a higher transmissivity than either of the till units above and below the outwash.

Once the planned depths for monitoring were chosen, existing wells in the TMA areas were reviewed for their location, whether the wells were temporarily abandoned or active, and for the screened interval of the well. An initial assessment was made and wells which were temporarily abandoned were reopened so that a field evaluation could be conducted for well continuity.

The field evaluation of the wells included all wells being actively monitored for groundwater levels and those temporarily abandoned wells which had been reopened. Each well was assessed by visual inspection, measurement of the water level, measurement of total depth, and recording of the well size, condition and total depth. In addition, one or two photographs of each well were taken for documentation of existing conditions.

Existing wells were evaluated for their potential use in the groundwater sampling program. Wells EX-9BU, EX-9BL, EX-13BU, G41-H18B, DMB-6 and DMB-1A were selected for the groundwater monitoring validation program based on their location and screened intervals. New wells were planned and installed so that nested pairs of wells existed at each chosen monitoring location. Table 5.3-1 lists the new and existing wells used for groundwater sampling for the 1994 program.

The existing wells were installed prior to the promulgation of NR 141. The existing wells had all been installed with screens, filter packs and impermeable annular space seals and, therefore, the wells are adequate for groundwater sampling.

Prior to using existing wells for groundwater sampling each well was redeveloped by surging and purging with a pump until the water was clear and readings for pH, temperature, and conductivity had stabilized.

5.3.1 Groundwater Quality Sample Collection Procedures

Sampling procedures were consistent with those published in the WDNR PUBL WR-153-87, *Groundwater Sampling Procedures Guideline*.

Prior to purging, the depth to water was first determined by an electronic water level indicator. The well was then purged such that the purging would "pull" water from the aquifer into the screened area of the well and up through the casing.

Prior to sample collection, each well was purged by removing four times the volume of water in the monitoring well or until the well is dewatered, whichever came first. The wells were purged with a dedicated PVC bailer or a Grundfos® stainless steel submersible pump. In the piezometers, water was drawn from above the screen in the uppermost part of the water column such that fresh water from the screen moved upward.

Subsequent to purging, samples were removed with either the dedicated PVC bailer or the Grundfos® submersible pump if appropriate for the parameter being collected. Sampling followed purging by no more than six hours for wells which recovered quickly. For slowly recovering wells, sampling was completed within ten hours of purging. In no case were wells sampled after ten hours had elapsed following the purging event.

Water samples were poured from the bailer, or from the pump tubing, into containers specifically prepared for the given parameter(s). Metal sample aliquots were filtered in the field prior to acidification. These samples were passed through a 0.45-micron membrane.

Table 5.3-1

Groundwater Quality Validation Wells TMA

Well ID	Function	Approximate Depth in Feet[a]	Screen Length in Feet	Existing (E) or New (N)[b]	Formation Monitored
CMC-01R	water table well	68	15	N	LWT
CMC-01P	piezometer	190	5	N	CO
EX-9BU	water table well	115	20	E	LWT/CO
EX-9BL	piezometer	187	3	E	CO
CMC-02R	water table well	120	15	N	LWT/CO
CMC-02P	piezometer	190	5	N	CO
CMC-03RR	water table well	128	15	N	LWT/CO
CMC-03P	piezometer	170	5	N	CO
CMC-04	water table well	110	15	N	CO
G41-H18B	piezometer	147	10	E	CO/LWT
CMC-05	water table well	90	15	N	LWT
EX-13BU	piezometer	141	3	E	CO
DMB-6	water table well	75.5	10	E	LWT
CMC-06P	piezometer	150	5	N	CO
CMC-07	water table well	50	15	N	CO
DMB-1A	piezometer	69	10	E	CO/LWT

[a] For existing wells, approximate depth is top of well screen.
[b] As of 1994.

Prepared by: LLA
Checked by: EAS

Formations
LWT - Late Wisconsinan Till
CO - Coarse-Grained Outwash
FO - Fine-Grained Outwash

Documentation of purging and sampling activities was accomplished by using the Groundwater Monitoring Field Form. When collecting samples for trace parameters and/or metals, all equipment used was cleaned and/or decontaminated. Blanks were collected to determine the adequacy of cleaning.

If fuel-powered units were used in groundwater sampling activities, those units were placed away from and downwind of any site activities (e.g., purging, sampling decontamination). Where possible the individual handling such equipment was a different individual than the one who actually conducted the sampling.

Dedicated equipment (except for dedicated pump systems that are never removed from the well) was stored off-site between sampling events. Decontamination activities were also performed on the equipment followed by storage as a precaution to minimize contamination between sampling events.

When duplicate samples were collected, they were collected by sampling from successively collected sample volumes. Split samples, when collected, were taken from consecutive sample volumes from the same device or by mixing in an intermediate vessel. This sample was portioned into two or more containers from a single sample container. Portioning assumes adequate mixing to assure "split samples" are identical.

5.3.2 Groundwater Quality

The results of the groundwater validation program were analyzed and compared to NR 809 drinking waster standards. These results provide groundwater characterization to support the TMA permitting activities. Design of the preconstruction baseline monitoring program is included in Section 7.2 of the MPA.

A complete discussion of groundwater analytical results from previous investigations as well as those from the 1994 investigations is included in Section 3.6 of the EIR. The individual laboratory data sheets are included in Appendices 3.6-10 through 3.6-13 of the EIR. Comparisons of the analytical results to NR 809 drinking water standards revealed several common parameters tended to exceed the standards consistently or intermittently during the studies.

5.4 Site Specific Environmental Information NR 182.08)(2)(d)(8)

A detailed specific characterization of the mine site and surrounding area has been conducted by various scientists and engineers over the period of 1977 to 1995. The studies that were performed are described in the project's May, 1995 EIR. The reader is referred to Section 3 of that document for information pertaining to the environmental characterization of the mine site.

6 Design Report NR 182.09(2)(c)

6.1 Tailings Management Area (TMA) - Performance Objectives and Facility Description

6.1.1 Performance Objectives

This Feasibility Report/Plan of Operation (FR) provides information and a design for the containment and management of mine wastes which are not returned to the mine. The information provided includes all required data concerning the physical characteristics of the site and the mine wastes which are related to the design and operation of the Tailings Management Area (TMA). This report section includes design rationale, description of design elements and calculations which support the proposed TMA design performance objectives. The primary performance objectives were developed based on a review and understanding of the Mine Environment Neutralization Drainage Program (MEND) tailings management criteria, applicable Wisconsin laws and regulations (e.g., Wisconsin Administrative Code Chapters NR 182 and NR 132) and CMC's operations, management and reclamation goals. The performance objectives, presented in a memorandum to WDNR on November 28, 1994 included the following:

- To design, construct and operate the facility in accordance with good industrial practices and in compliance with the standards prescribed by the State of Wisconsin.
- To design the facility to avoid and minimize impacts to wetlands, to eliminate impacts from surface water flows, and to achieve isolation from groundwater.
- To the extent practicable, minimize the amount of required long-term care activities through the application of effective design, operation and reclamation features.
- To limit tailings oxidation and solute release from the tailings mass through:
  - the placement of tailings at a high pH, with excess alkalinity to minimize the potential for acid generation during operation;
  - limiting of oxygen entry into tailings during the operating and post-closure period; and
  - the design of a composite base liner system to control percolation rates.
- To design a final cover which meets the following objectives:
  - limits oxygen entry into the tailings mass;
  - limits infiltration of precipitation into the tailings mass, using a hydraulic barrier of equal or less permeability compared to the TMA base liner;
  - provides a stable surface which can meet reclamation requirements; and
  - provides an erosion-resistant surface.

The approach taken in designing the TMA components focused first on evaluating the MEND tailings management criteria which contain operational and design alternatives for the control of acid generation potential (Primary Control Alternatives), acid mine drainage migration control (Secondary Control Alternatives) and collection and treatment (Tertiary Control Alternatives). These MEND alternatives for the TMA construction and operation are evaluated in Section 10 of this report. Alternative evaluations resulted in the selection of design and operational components which meet the performance objectives stated earlier. The selected components will meet the performance objectives of compliance with the groundwater standards by providing containment which prevents acid generation through operational and closure controls and maintains the long-term environmental security of the site by the final cover and reclamation design. The major design components and their primary design functions are summarized under the performance objectives.

- Containment for separation from surface waters and groundwater:
  - composite liner and composite cap as hydraulic barriers for containment;
  - leachate collection system for decreasing percolation potential; and
  - facility monitoring during operations and post-closure period to initiate remedial action should containment become less effective.
- Prevention of Acid Generation:
  - high pH of deposited tailings to maintain tailings stability;
  - managed tailings deposition system to prevent initiation of acid generation; and
  - prevention of oxygen entry by required maintenance of cap and sealing potential oxygen entry points.
- Long-Term Maintenance:
  - selection of components to achieve long-term stability and integrity of the final cover system;
  - design of reclamation with low erosion potential; and
  - monitoring to evaluate maintenance needs.

The reasoning and design logic for the TMA design and operational components are provided in the following sections.

6.1.2 Tailings Management Area - Facility Description

The purpose of Tailings Management Area (TMA) is to provide for the long-term environmentally safe containment of waste resulting from the mining and/or milling processes. The TMA consists of four cells designated TMA 1, TMA 2, TMA 3, and TMA 4. TMA 1 and TMA 2 are designed to contain the mine wastes resulting from the zinc ore production and

processing. TMA 3 and TMA 4 are designed to contain the waste resulting from the copper ore production and processing. The anticipated waste quantities for each cell are provided in Table 4.1-2 in this report. Each cell is constructed and operated in stages designated as Stage I through Stage VIII. Approximately 96 percent of the mine wastes contained in the TMA cells are tailings, about 80 percent of which are fine tailings and about 20 percent whole tailings. The physical and chemical characteristics of the type of wastes to be contained in the TMA are described in Sections 4.2.3 and 4.2.4, respectively of this report.

The subsections which follow provides the design criteria information and calculations to support the major design components included in the TMA operating systems and containment design. The major design components which are addressed include the following:

- TMA operating systems;
- TMA composite liner design elements description and design calculations;
- TMA leachate collection system design elements including leachate drainage, collection, removal and transfer systems;
- TMA embankment configuration and design including staging berms;
- TMA final composite cover design elements descriptions;
- TMA water budget calculation;
- TMA surface water management design;
- Other TMA design elements including: traffic routing , security and screening;
- TMA site phasing and development including material balances;
- TMA foundation and stability analysis; and
- TMA reclamation plan and environmental monitoring as described in detail in the MPA.

The final subsection will provide an assessment of the expected performance of the TMA as it relates to meeting the applicable groundwater standards.

The design sections of the report are supported by calculations and other design data found in referenced appendices and drawings which are provided at a reduced scale preceding the appendices.

6.2 TMA Operating System Design

6.2.1 General

Section 6.2 discusses the operating systems for the TMA including the reasoning and logic which are the basis for the selection of the operating systems. This section was prepared with assistance from the firm Steffen Robertson and Kirsten (U.S.) Inc. (SRK). Specifically, Messrs. Andy MacG Robertson, Rob Dorey and Dave Hallman of SRK prepared the preliminary design of the TMA operating systems. During final engineering and prior to construction slight changes to specific components of the operating system may be made. The key operating systems of the TMA are:

- tailings delivery system which includes reclaim water and tailings slurry pipelines;
- tailings placement system which includes tailings deposition in the TMA cell(s);
- tailings management systems to mitigate potential operational problems;
- tailings placement related to TMA cell closure;
- reclaim water management; and
- tailings management systems for adverse weather conditions.

The reclaim water management system is included in this section because it is part of the total tailings operating system. The reclaim pond feasibility, design and operation is not included in this report since it is considered part of the total water management system. Details concerning the reclaim water management system design and operation will be submitted to the WDNR Industrial Wastewater Section and USCOE as part of the plant wastewater treatment plant design report. The reclaim water management system discussion contained herein is confined to how the reclamation pond fits into the tailings operating system.

6.2.2 TMA Waste Delivery System

The waste characterization of the waste streams which will be placed in the TMA are described in detail in Section 4 of this report. The waste stream will generally include the following:

- fine and whole tailings;
- wastewater treatment plant solids;
- Type II waste rock (i.e., sulphur content greater than two to three percent);
- reclaim pond solids; and
- very small quantities of laboratory waste.

6.2.2.1 Truck Transport of Mine Wastes

Of the waste streams listed above, Type II waste rock will be transported to the TMA site by truck. Open dump trucks will most likely be used for this purpose. The trucks will travel via the access road between the plant site and the TMA as shown on Drawing No. 2. Laboratory waste and wastewater treatment solids will be transferred to the tailings pump box and hydraulically pumped to the TMA with the tailings. Reclaim pond solids removal will take place on an as needed basis. Either trucks or hydraulic transfer will be used to transport the solids to the TMA. The volume of solids that will be removed is estimated to be equivalent to one or two truckloads per year. If trucks are used for this purpose, they will also travel the TMA access road as shown on Drawing No. 2 and Figure 6.12-5. Hydraulic conveyance would be accomplished using a temporary pipeline located in the TMA pipeline corridor.

Type II waste rock will be generated during mine development and operation over a period of about 31 years. An estimated 100,000 tons or 70,000 cubic yards of Type II waste rock will be produced during the mine development period. An additional 300,000 tons of waste rock will be mined during the operating life. An estimated 350,000 tons will be hoisted to the surface and used as interior riprap in the TMA cell construction.

6.2.3 Tailings Delivery System

6.2.3.1 Tailings Slurry Characteristics

The mill tailings will be transported from the mill to the TMA as a slurry through a surface pipeline graded to drain to two low points. The tailings will be generated at a rate of approximately 180 to 210 tons of solids per hour of mill operation with the exact rate being a function of the hardness of the ore. A portion of the tailings will be used as underground backfill which affects the quantity of tailings that will be transported to the TMA at any given time.

Underground backfill operations will use about 50 percent of the tailings solids. Backfilling operations will use the coarse tailings fraction which will be separated by hydrocyclones at the mill. During periods when the mine cannot take tailings backfill, or when the backfill plant is not operating, it will be necessary to pump whole tailings to the TMA.

The fine tailings remaining after the coarse fraction is removed for backfill have a specific gravity of 3.32, and will be transported as a slurry which is approximately 12 percent solids by weight or 4.0 percent solids by volume. The whole tailings have a specific gravity of 3.52 and will be transported as a slurry which is approximately 25 percent solids by weight or 8.8 percent solids by volume.

6.2.3.2 Pipeline Route

The tailings delivery system will include a pipeline to transport slurried tailings from the mill to the TMA and a pipeline to transport return water from the reclaim ponds to the mill. The pipeline route is shown on Drawing No. 2. The route has been selected to avoid and minimize wetland disturbance, to minimize potential disturbance to threatened and endangered species, and to minimize visual impacts. In order to provide access to the TMA and access for pipe inspection and maintenance, an all-weather road will be constructed and maintained adjacent to the pipeline. Practical considerations for vehicle and equipment access during winter limits the maximum grade for the road and the pipeline to ten percent. In order to provide pipeline drainage under gravity, a minimum pipeline grade of one percent will be provided. Flatter grades may be necessary within the plant site area.

From the mill to the TMA, the tailings pipeline will have a length of approximately 4,800 feet. Within the TMA, the tailings delivery pipeline will be routed along the inside of the crests of the berms and the tailings will be ring-discharged into each cell. At the mill, the pipeline is anticipated to start at an elevation of approximately 1,650 feet, while the TMA crest elevation will be approximately 1,730 feet. Thus, the tailings pipeline will be operated as a pressure line with the slurry pumped from the mill.

In the event of an unplanned flow stoppage or a planned shutdown, tailings draindown facilities will be provided at the low point along the access road and at the mill to allow the tailings line to be drained. Similarly, the pipeline will be equipped with suction breaks at the high points.

6.2.3.3 Draindown and Spill Requirements

Although the pipelines will be designed to prevent spillage, the delivery system will be installed in such a way that any potential spillage is fully contained and recoverable. The pipelines will be placed in a lined ditch draining to the collection sumps at the system low, or draindown point and at the mill. The ditch will have a minimum bottom width of four feet and a minimum depth of approximately four feet. The ditch will be constructed adjacent to the pipeline access road. It will have an open channel flow capacity well in excess of the in-pipe slurry or return water flow rates, and a large capacity to contain the tailings solids should a leak occur. The ditch will be lined to prevent loss of tailings water in the unlikely event that a leak occurs. The proposed design includes erosion control measures required to stabilize the ditch for both anticipated surface water flows and potential pipeline leakage flows. A typical lined ditch system is shown on Figure 6.2-1. Both the tailings line and the return water line will be installed in this lined ditch.

A lined ditch system has the following advantages:

- allows visual pipe inspections on a regular basis;
- allows access for maintenance, repairs, and clearing blockages; and
- provides full containment and recovery even in case a significant volume of tailings should leak.

Draindown Basin (refer to Figure 6.2-2) will be provided both at the mill and at the low point in the pipeline system to collect tailings drained from the pipeline during periods of planned shutdown or when the pipe blocks, as well as unexpected spills that could occur due to leaks in the pipe. The sumps are sized to contain a 25-year, 24-hour storm event on the ditch and roadway, and two full tailings pipeline volumes. Ditch grades sufficient to ensure that spilled tailings will flow to the collection points are not practical, therefore some accumulation of solids within the ditch at a leak or spill location could be experienced. When this occurs, the accumulated materials will be removed by a backhoe operated from the access road and loaded onto a truck for transport to the TMA. Solids and liquids removed from the ditch will be placed in the TMA.

Experience at other tailings operations, with pipelines of the type selected, has demonstrated that the most likely location for leaks is at the flange connections. Therefore, the tailings pipeline will be covered with a half-round segment of pipe culvert at each flange, about 400 feet apart.

Direct precipitation into the pipeline ditch will flow to an emergency draindown basin from where it can be removed by a pump truck for transfer to the TMA. The ditch cross-section will include above grade berms, as shown on Figure 6.2-1, to prevent surface water run on. At drainage crossings, surface water flows will be passed over or beneath the pipeline ditch via culverts.

6.2.3.4 Pipe

The tailings slurry pipeline has been sized to handle the required range of flows and slurry densities resulting from the operation of the backfill plant and normal variations in milling rate and ore grades. For the purposes of the preliminary designs discussed below, the following two primary slurry streams have been considered:

- Whole tailings: approximately 180 to 210 tons per hour (tph) at 25 percent solids weight by weight (w/w) with a slurry density of 76.3 pounds per cubic foot;
- Fine tailings: approximately 90 to 105 (tph) of fine tailings at 12 percent solids w/w with a slurry density of 68.1 pcf.

Preliminary design studies indicate that the whole tailings will behave as a heterogeneous slurry. The whole tailings must be transported at a minimum velocity of approximately seven feet per second (fps) to prevent settling and the formation of a bedload. The fine tailings slurry stream is expected to behave as a homogeneous suspension at normal flow velocities and thus no settling of the solids particles is anticipated.

Initial calculations for both slurry streams have been based on the use of a 16-inch diameter high density polyethylene (HDPE) pipe for the tailings pipeline. This pipe has long-term safe operating pressures greater than those needed for this application. During final design, it is possible that two separate slurry pipelines for use with fine and whole tailings, respectively, may be specified. For the whole tailings the velocity through the single pipeline will be seven fps. Preliminary estimates of the friction head loss are 0.013 feet of slurry per foot of pipe. For the fine tailings the flow velocity will be 8.3 fps and head loss of approximately 0.016 feet of slurry per foot of pipe. These head loss estimates will be updated upon further evaluation of slurry rheology test results.

In order to pump the variable tailings flows from the mill to the TMA, a variable speed pump or series of pumps will be required. For pumping whole tailings, the pump(s) must be capable of delivering the tailings at a maximum pressure of approximately 150 pounds per square inch (psi). For pumping the fine tailings, the pump(s) must be capable of delivering the tailings at a pressure of 150 psi. Allowances will be necessary for some variations in flow rates and pressure. To reduce the possibility of overpressurizing the pipe when restarting the pump(s) following a planned shutdown, provisions will be necessary to allow clear water to be pumped through the pipeline for a short period prior to stopping and following start-up with a gradual transition to and from a slurry feed. If the system can be flushed with clear water or cleared by alternate means, draindown during shutdowns would not be required. However, during cold weather the water must be drained from the lines during shutdowns to prevent freezing. In the event of a short shutdown, the system need not be drained but should be restarted with clear water at gradually increasing flow rates.

The individual HDPE pipe sections will be heat fusion welded to create a continuous pipeline. Along the access road, the pipeline will be jointed at 400-foot centers with bolted flanges for ease in replacing damaged or worn pipe sections, for clearing blockages should such conditions arise, and for rotating the pipe to equalize the effects of wear. Along the tailings cell embankment, the pipeline will be jointed at 200-foot intervals to provide for ease of maintenance along the cell crests.

The tailings pipeline will be instrumented to measure flow rates and pressures at the mill and TMA. If these flow rates differ, the slurry pump at the mill will shut off automatically. When the flow of slurry is stopped, valves at the draindown facility will be opened to allow the pipeline to drain to the draindown sump under gravity. The pipeline will be visually inspected via a daily drive-by to monitor for leaks or other conditions requiring maintenance.

6.2.4 Tailings Distribution System

Tailings will be deposited hydraulically in lined cells designed as described in Section 6.2 of this report. The main tailings delivery pipe from the plant site will extend to the high point at the central common corner for all the cells. Distribution from this high point will be through a ring pipeline around each cell from which a system of spigot distribution pipes discharge tailings to the beaches in the individual cells. The slurried deposition system proposed for the TMA is commonly referred to as a subaerial method.

The subaerial deposition system was selected because this system offers the following advantages:

- higher tailings densities can be achieved which reduces the required TMA cell volume and the overall environmental impact of the project;
- a smaller water pond can be achieved over the tailings, which reduces the hydraulic head on the TMA liner, and thereby reduces the potential rate of seepage from the facility;
- subaerial deposition allows tailings to settle rapidly from the slurry and densify on the beaches. The increased density achieved during operation results in less overall settlement during closure and post-closure;
- partially dewatered tailings provide a more trafficable surface for the placement of the final cover; and
- operational problems related to glaciation and dusting can be mitigated by simple modifications to cell operational methods (refer to Section 6.2.6 for additional discussion.)

Subaerial deposition is typically accomplished by discharging the slurry in a "panel" from groups of adjacent spigots. The discharge "panel" is moved progressively around the cell by progressively closing spigots at one end of the panel while opening spigots at the other end of the panel. Each spigot consists of a length of flexible pipe which extends from the ring main through a pinch valve to the desired discharge level on the tailings beach. As slurry is discharged, it flows across the beach to the low point or cell. The tailings solids settle out and are deposited on the beaches. The group of discharge points or panel of spigots are operated sequentially to achieve

a beach shape which slopes at an average of approximately 0.5 percent to one percent toward the center of the TMA cell from all sides of the cell. This discharge of tailings from the periphery of the cell results in a water pool on the concave tailings surface near the center of the cell. The sequential deposition of tailings results in a relatively uniform elevation of tailings around the cell forcing the water pool to the center. Tailings transport water and process water from the pipeline slurry are liberated as the solid tailings particles settle out, and drain to the central pond.

Once a layer of tailings has been placed in any particular area, further tailings deposition will be directed towards other areas. This will allow the freshly deposited layer of tailings to settle and consolidate and the entrained water to drain to the water pool. Water will be continuously decanted from the water pool for re-use at the mill, resulting in a "beach" of exposed tailings sloping toward the central water pool. The areal extent of the tailings beach will be a function of the cell geometry, tailings deposition angles, and the operating pond level needed for decanting the accumulated water in the pool.

Although the beached tailings will be exposed to the atmosphere, the exposure time will be limited and they will be flushed with process water as overlying tailings are deposited. The process water is highly alkaline while tailings contain significant quantities of calcite, preventing initiation of the biologically induced acid generation processes. On-going laboratory studies will be used to evaluate the time-dependent processes by which acid generation is initiated. This data will be used to define the tailings spigotting cycle time necessary to prevent acid generation.

The beach tailings will also be exposed to evaporation which will reduce the moisture content of the tailings. As the moisture content is reduced, reduction in tailings volume results until the shrinkage limit is reached. Thus the subaerial method of tailings deposition will result in higher *in situ* densities, and thereby minimize the required capacity. Rates of tailings deposition will be such that there will not be sufficient evaporation to achieve the shrinkage limit. Design studies indicate that depositing the tailings in this fashion will result in an average tailings dry density of approximately 97 pounds per cubic foot (pcf).

The spigot points used to discharge tailings will be installed at 100 to 200-foot centers around the perimeter of the facility as shown in Drawing No. 17. Each spigot assembly will consist of a pinch valve and flexible rubber hose connected to the main tailings distribution line as shown in Drawing No. 29, Detail 3-29. Initially, the flexible rubber hose extends to the bottom of the cell along the sidewall of the embankment. The length of the spigot is shortened as the level of tailings rises so that the discharge is maintained on to the surface of the tailings.

The discharge rate of tailings has been estimated at 610,000 cubic yards per year while the plant is in production. For 365 days per year of operation this results in the deposition of approximately 1,700 cubic yards (2,200 tons) per day under normal operating conditions. This quantity will vary based on recovery rate, mill efficiency, operating down time of the mill and other factors.

6.2.5 TMA Water Reclaim System

During normal operation of the subaerial tailings deposition system, a water pool will be maintained near the center of the operating cell with a minimum operating depth of

approximately three to five feet. This central pond is maintained such that the pond water surface is located no closer than 150 to 200 feet from the interior berm. The system used to reclaim water from the central pool consists of a skid-mounted pump initially located near the base of the cell (refer to Drawing No. 30, Detail 3-30). This pump can be moved up the interior slope by winching from top of the berm as required as the tailings cell fills. A floating intake located in the center of the water pool will be connected to the pump. As the level of tailings increases the pump will be withdrawn up the side slopes of the cell. Water decanted from the cell will be pumped to the reclaim water pond. This will enable each cell to be filled to the maximum level possible with tailings while maintaining the required freeboard of five feet above the level of the free water pool and providing storage capacity for storm events.

6.2.5.1 Return Water Pipeline

Water reclaimed from the tailings cells will be pumped to a reclaim pond located adjacent to the TMA through a HDPE pipeline. From the reclaim pond, water will be pumped to the mill in a 12- to 16-inch diameter HDPE pipeline laid parallel to the tailings delivery pipe in the containment ditch. The return water pump at the reclaim basin and pipeline system will be capable of delivering the entire makeup water flow of approximately 3,000 gpm. An additional allowance will be required for higher peak flow demands that may be necessary on occasion. Both pumps may be variable speed types to accommodate the anticipated range of pressures and flows. The decant pump could also have a fixed discharge rate and be operated in an intermittent fashion to account for ranges in pressures and flows.

6.2.5.2 Reclaim Pond

In order to provide a greater water management flexibility, one lined reclaim pond will be constructed west of the TMA. The reclaim pond has been sized for a three day storage volume at a 3,500 gpm flow rate, or 15 million (M) gallons. Section 6 of the MPA contains the details of sizing of the reclaim pond taking into account the water balance for the entire facility.

6.2.6 TMA Operation in Adverse Weather Conditions

6.2.6.1 General

The access to the TMA is provided by an all weather road from the plant site to the TMA. This road will be kept clear of snow during adverse weather conditions so that access to the TMA is not restricted and TMA operation can continue under all weather conditions.

The following is a discussion of adverse weather conditions and how these conditions could impact the operation of TMA. For each weather-related operational problem a solution or prevention method is explained.

6.2.6.2 Warm Drying Conditions

Warm drying weather conditions will result in short term evaporative drying and low tailings moisture contents on tailings beaches. This evaporative drying and desiccation by sun-drying

results in densification of the tailings beaches. Because a subaerial (managed discharge) system is being used the tailings beaches will not be exposed for sufficient periods of time for acid generation to occur. Laboratory tests on tailings discussed in Section 4.2 of this report show that because of the high pH at deposition and calcite content, the beached tailings will remain neutral even if exposed to air for over six months. The continuous sequencing of spigots described earlier results in a fresh, high pH, wet tailings being deposited over the dry beach at intermittent intervals which prevent the initiation of acid generation.

Under very dry conditions, portions of the beaches dry to the extent that dusting may occur under high wind conditions. This can be prevented by discharging a veneer of fresh high moisture tailings on the dry beaches to keep them wet.

6.2.6.3 Continued Cold Conditions

Under extremely cold conditions, the thin layered tailings formed by a subaerial discharge can, and does, freeze. This results in the accumulation of substantial thickness of frozen tailings slurry (tailings fines at 12 percent solids by weight), often termed glaciation. Freezing of the tailings prevents consolidation and drainage of the tailings for as long as they remain frozen. If these glaciated zones are covered with subsequent layers of tailings, they are insulated and may not thaw until a number of years after burial. On thawing, they are subject to large consolidation settlements disrupting any covers placed over or on them. It is, therefore, necessary to prevent glaciation and the development of frozen zones in the impoundment. This is achieved by the following operational procedures.

The ring discharge around each cell is equipped with dump points at the corners of each cell where the full tailings stream can be discharged from a single dump point. As weather conditions cool, the panel width of the subaerial discharge is reduced, concentrating the heat energy in the "warm" tailings discharge over a smaller beach area to avoid freezing. During the coldest weather, discharge is made only from one or two dump points to minimize the amount of glaciation.

Following the onset of thawing conditions in the spring, the subaerial deposition is resumed.

6.2.6.4 Dry Cold Condition

Tailings beaches left exposed to dry and very cold conditions for extended periods of time may result in the surface "freeze drying" of tailings. These climatic condition can occur during prolonged high-pressure, low-temperature periods. National Oceanic and Atmospheric Administration, 1993 data indicates that the average temperature is typically below freezing for the months of November, December, January, February and March. However, in all of these months precipitation in the form of snow typically exceeds one inch of moisture. So only in certain climatic circumstances would the tailings surface be subjected to extended periods of cold, dry conditions without snow cover which would allow tailings to freeze-dry. In the event that these climatic condition occur concurrent with windy conditions, a condition referred to as "tailings dusting" may occur. If climatic conditions exist conducive to tailings dusting, the operational controls listed below can be implemented:

- spigot sequencing can be increased to place a veneer of fresh high moisture content tailings over freeze-dried areas;

- spigots around the cell can be used to direct water which will allow ice to develop on the freeze-dried areas; and

- other control measures like applying water or a chemical binder to the tailings surface

6.2.7 Control of Acid Generation during TMA Operations

6.2.7.1 General

This section describes ways to prevent and/or control the oxidation of the sulfide minerals in the tailings within the TMA, and the consequent generation of acid during the time that the TMA cells are operational. The primary ingredients for the oxidation and acid generation processes to proceed include the: 1) presence of sulfide minerals; 2) presence of oxygen; and 3) presence of water. All three of these ingredients must be present for the oxidation of sulfide minerals to occur. It is therefore possible to control sulfide mineral oxidation by controlling any one of the three ingredients.

The rate of sulfide mineral oxidation is significantly controlled by the pH of the interstitial pore fluid. The rate of inorganic sulfide mineral oxidation increases with decreasing pH. The rate of biochemical sulfide mineral oxidation (due to the action of lithotrophic bacteria) increases significantly with decreasing pH, especially if pH values decline below 4.5.

The oxidation of sulfide minerals in the tailings could lower the pH of the water. At lower pH, heavy metals, such as iron, zinc, copper and/or lead, are more soluble and the concentration of these metals in the water would increase. The quantity and availability of neutralizing minerals, such as calcite (calcium carbonate), determines if the pH of the water in the tailings will be lowered by the sulfide mineral oxidation. During sulfide mineral oxidation, the neutralizing reaction produces the mineral gypsum (hydrated calcium, sulfate) and water as end products, and in doing so, neutralizes the water. The solubility of some of the metals mobilized at a lower pH, such as iron, are pH dependant and form precipitates as the pH increases. Other metals, such as zinc, copper and lead, do not precipitate until the pH is raised. These metals tend to remain in solution in neutralized water and potentially may enter the environment within leachate that may percolate from the TMA area. Therefore, the design and operation of the TMA must prudently control the oxidation of sulfide minerals which in turn controls the amount and type of substances in solution within the TMA pore water.

Two major groups of controls exist for controlling the oxidation of the sulfide minerals in the tailings. One group consists of operational, or preventive, controls which would be implemented during the operational life of each TMA cell. The second group of controls are reactive controls that would be implemented only if conditions so warranted.

6.2.7.2 Operational Controls

The primary goal of operational controls is to prevent, or inhibit, the oxidation of the sulfide minerals in the tailings. These controls can be: 1) naturally occurring; and/or 2) engineered.

The most important natural control is the amount and availability, or reactivity, of the neutralizing substances within the tailings, such as the mineral calcite. These substances neutralize the water through their participation in irreversible chemical reactions. The calcite content of the Crandon Project tailings has been calculated to be about four percent by weight which will provide a significant amount of neutralization. However, oxidation must take place within the tailings before these naturally occurring substances can participate in the neutralization reactions. In designing and operating a TMA facility, it is therefore generally best to consider the presence of natural controls as a safety measure and to concentrate engineering efforts on the control of sulfide mineral oxidation, which will in turn control both lowering of pH and solubilization of metals.

Three engineered controls will be used in the operation of the TMA, two of which will control the oxidation of the sulfide minerals, and one of which will add additional neutralization to the tailings. These controls include the method of tailings deposition, the time of exposure of emplaced tailings and the pH of the process water used to slurry the tailings to the TMA.

The first two controls are part of the subaerial method of deposition as proposed by CMC. Earlier in the section the subaerial tailings deposition method proposed for the TMA was described in detail. This deposition method has advantages for the prevention or inhibition of sulfide mineral oxidation within the deposited tailings. The following is a list of these advantages along with a description of how they prevent or delay the on-set of oxidation.

- Subaerial deposition will result in a three to five-foot pool near the center of each cell during operations. This pool will cover as little as 50 percent to as much as 75 percent of the tailings beach area depending on the cell size, cell configuration, precipitation, and the water storage needs of the plant circuit. All tailings below this pool then are in a subaqueous deposition condition and are inhibited from oxidizing since they are below water and, because oxygen, one of the three components necessary to initiate oxidation has a very limited solubility and diffusion in water.

- Subaerial deposition, as described in this section, involves the sequential deposition of tailings through a ring of spigots located around the outside of the cell. A likely operating scenario would be for the operator to close a spigot at one end of a panel and open one spigot at the other end of the panel daily, continuing sequentially around the cell. The result of this practice is the deposition of layers of fresh tailings on a continuous basis. These fresh layers of tailings cover the tailings deposited earlier, resaturating them with high pH process water, thus maintaining a high pH as well as reducing oxygen entry.

- Due to the continuous deposition of tailings, and the slow rate of dewatering, the retained moisture within the tailings mass will act as a substantial control to inhibit oxidation. The small amounts of acid that may be generated are controlled during operations by the natural calcium carbonate content of the tailings and the high pH of

the frequently deposited layers of tailings. The laboratory testing of the rate of onset of acid generating conditions performed for Crandon tailings clearly demonstrates that the onset of acidic conditions will not occur for many months from the time of exposure.

These controls accomplish the primary objective of the operation designs which is to prevent or delay the onset of acid formation.

6.2.7.3 Available Measures to Control Initiated Sulfide Mineral Oxidation

The results of the waste characterization testing of the tailings indicate that the natural calcite content and high pH of the tailings slurry will prevent the lowering of pH due to the oxidation of pyrite and other sulfides from starting for many months. In the unlikely event that a lowering of the pH is observed during a period of prolonged shutdown or temporary closure, prevention and abatement techniques will be used. These techniques are described below.

The prevention and abatement techniques discussed above relied primarily on keeping the tailings in an environment that is not conducive to the oxidation of pyrite and other sulfides - an environment that excludes one or more of the primary components needed for sulfide oxidation, i.e., sulfide minerals, oxygen and water, or controls of the pH of the tailings interstitial water. This section discusses the control measures available if unusual or unexpected circumstances delay additional layer placement sufficiently long for the oxidation process to be initiated. The potential for sulfide mineral oxidation could be enhanced if a temporary mill shut down resulted in the suspension of tailings deposition for an extended period of time. Under these conditions, the tailings would be partially saturated and, therefore, partially exposed to oxygen. The initial results indicate that the natural buffering capacity of the tailings, especially the whole tailings, could inhibit the oxidation of the sulfide minerals in the tailings for six months or more. It is expected that once the buffering substances are consumed that oxidation would begin or would proceed at an increased rate thereby leading to the generation of lower pH interstitial pore fluids within the tailings. Under these conditions, the operator has the following simple operational techniques available to prevent or retard sulfide mineral oxidation.

- Spigot a limestone solution over the exposed tailings beach. This would provide for additional buffering capacity and prevent the initiation of sulfide mineral oxidation.

- If the shut down is for an extended period of time (e.g., six months or more) flooding the entire facility could be considered as an alternative way of preventing sulfide mineral oxidation. This could be accomplished by pumping water from the mill or reclaim pond or wastewater storage basin into the TMA cells until the entire tailings beach is flooded. However, if this control measure were undertaken it would require that riprap be placed on the cell slopes to prevent wave erosion of the liner cover. The source of the riprap could be either mine waste rock or cobbles and boulder screened from the till. Keeping the tailings underwater during a long shutdown period will significantly limit sulfide mineral oxidation.

6.3 TMA Liner Design NR 182.09(2)(c)1

6.3.1 General

The primary goal of the TMA liner design is to provide an effective hydraulic barrier beneath the tailings to contain the tailings pore water (leachate) and to allow for the effective removal of the leachate during operations and after closure. The design must meet these primary goals so that the performance objective of complying with the applicable groundwater standards can be met.

The composite liner and leachate collection system components consist of seven distinct layers for which the design rationale and function is explained in the subsections which follow. Two types of liner systems are proposed, one for the cell bottoms and one for the interior slopes of the cell. The liner on the cell bottoms (and partially up the interior side-slopes) is overlain by leachate drainage layers which are described in detail in Section 6.4 of this report. The liner system on the interior side slopes includes a leachate drainage layer to an elevation equal to the high point on the cell base. The height up the slope from the cell ranges from approximately nine feet in TMA 3 to approximately 17 feet in TMA 1. The two types of liner system have components from top to bottom as follows:

- Base liner system configuration (with leachate collection system) (refer to Drawing No. 29, Detail 1-29):

  - 18 inches riprap;
  - six inches of fines from the till processing (filter layer);
  - 18 inches glacial till (filter layer);
  - geotextile (filter);
  - 24 inches gravel (drainage layer);
  - geotextile (cushioning);
  - thermoplastic geomembrane;
  - 12 inches of low permeability soils; and
  - prepared subgrade consisting of compacted till fill or native till soils.

- Interior sloped configuration of composite liner (without leachate collection system) (refer to Drawing No. 28, Detail 6-28):

  - 18 inches riprap (only on interior slopes where required during final deposition);
  - 18 inches of glacial till (protective layer);
  - geotextile (cushioning);
  - textured thermoplastic geomembrane;
  - 12 inches of low permeability soil; and
  - prepared subgrade consisting of compacted till fill or native till soils.

The design rationale for each component is presented in the subsections which follow. The supporting calculations, where applicable, are found in the referenced appendices. The liner components are discussed from the bottom to the top, generally in the order the components are constructed.

6.3.2 Prepared subgrade NR 182.11(1)(r)

The purpose of preparing the subbase grade is to provide a stable surface on which to compact the low permeability soils. The liner soils are compacted to 95 percent of Standard Proctor maximum dry density (ASTM D-698), so it is imperative that a stable subgrade exists. In addition, the subgrade must support the composite liner and the live and dead loads occurring during the TMA filling processes. A thoroughly and uniformly compacted subgrade will minimize the possibility of differential settlement.

The TMA cell bottom and a portion of the interior slope are cut into the existing topography so that they are founded in natural till soils which are very dense (110-130 pounds per cubic foot - Golder, 1981). The cell bottom and portions of interior side slopes then will require only grading and proof rolling. The remainder of the side slopes will be formed by grade cell embankments constructed of compacted glacial till soil as described below.

The glacial till excavated within the TMA cell footprints is the primary source of soil for the TMA liner support materials (i.e. prepared subgrade). The glacial till at the TMA site can generally be characterized as a heterogeneous mixture of primarily sand, silt, and gravel with small amounts of clay (<10 percent), cobbles and boulders (<5 percent). A summary of some physical properties of the glacial till from the 80 samples tested are as follows:

- Unified Soil Classification System (USCS): SM -SP or SM, slightly silty fine to medium sand to silty fine to coarse sand.
- Percent passing the number 200 sieve: range 6-45 percent, average 20 percent, majority is in the 10-30 percent range.
- In situ coefficient of permeability: K_h (horizontal) 2.8×10^{-4} cm/s and K_v (vertical) 9.4×10^{-5} cm/s.
- Laboratory coefficient of permeability on recompacted sample on the order of 1×10^{6} cm/s.
- Proportion of cobbles and boulders subject to spatial variation estimated by Golder, 1982 as typically about five percent. The maximum cobble size is estimated at six to eight inches (150 to 200 mm) but occasionally large cobbles or boulders are encountered.
- In situ dry density of 110 to 138 pounds per cubic foot.
- Maximum dry density for the same samples of till are 124 to 137 pcf based on the standard Proctor Test (ASTM D-698).

The glacial till has physical properties which make it a good soil for embankment construction. NAVFAC DM-7.2, 1982 in a rating of the relative desirability of soils as compacted fill rates SP-SM with a rating of 4 as an embankment fill material where 1 is the best and 14 least desirable. The glacial till excavated from cut areas can be transported to fill areas, deposited in approximately 18-inch loose lifts and compacted to a maximum 12-inch compacted lift thickness. The only preparation required to this material prior to use as a compacted fill is the removal of

cobbles larger than eight inches. Generally in earthwork the soils used in a compacted fill should not have a maximum particle size which exceeds about two-thirds of the compacted lift thickness. Since these cobbles are only occasionally present, it is expected that they can be pushed to the embankment side by the construction equipment during the cutting and filling processes.

In addition to use as liner support soil, the unprocessed till material described above is proposed for constructing the following TMA components:

- berms (embankments);
- final cover - grading layer;
- final cover rooting layer;
- geomembrane liner protective layer; and
- miscellaneous construction fill (eg. reclaim and sedimentation pond embankments, access road fill etc.).

Details and specifications for the construction with glacial till soils are provided in Section 7.3 of this report.

6.3.3 Low-Permeability Soil Component

The design presented herein is based on the use of native clay soil as the low-permeability soil component of the clay liner. Section 10.8 of this report evaluates other alternative materials for the low-permeability soil component including fines amended till, bentonite amended soils and geosynthetic clay liners (GCLs). Based on the analyses in Section 10.8 the native clay liner was selected for the low-permeability soil component as the best soil component available for the composite liner at this time. For the purpose of this FR, a sufficient quantity of native clay has been defined for the soil component of the composite liner for TMA 1 construction and TMA 1 closure. Although a clay liner was selected for construction of the first cell, TMA 1, other soil liners types such as bentonite-amended soil, fines amended till or geosynthetic clay liner may be considered for future cell liners or final cover construction. Based on the field and laboratory testing and analysis performed for the 1985 Feasibility Report, CMC has selected a bentonite-amended soil as the primary alternative soil liner for the TMA construction. Other soil components may be considered as soil liner components for the construction and closure of TMA 2, 3, and 4. All low-permeability soils used will meet specifications necessary for groundwater protection, as shown in Table 6.3-1.

The low-permeability soil component (clay liner) is the only soil material which will be hauled in from off-site. Crandon Mining Company (CMC) proposes to retain a contractor to provide an adequate quantity and quality of clay soils to completely construct TMA 1, the initial TMA cell. CMC has entered into an agreement with ASHCO Inc. of Wausau, Wisconsin, to ensure a supply of approximately 181,000 cubic yards (in-place) of clay for the construction of the clay liner and clay cap components of TMA 1. The ASHCO agreement ensures a supply of clay with specifications as listed in Table 6.3-1.

Table 6.3-1

TMA Clay Specifications

Property	Value/Units Low-Permeability Soil Specifications	
Percent Passing the No. 200 Sieve	50%	
Clay Size Content (<.005mm)	20% or greater	
Saturated Hydraulic Conductivity	1×10^{-7} cm/sec or less	
	Average of all Borrow Samples	No Individual Sample Can be Below
Liquid Limit (LL)	25	20
Plasticity Index (PI)	12	10

These clay properties are better than the clay liner requirements of the 40 CFR, Part 258 Subpart D, Design criteria.

The clay liner component of the composite liner is a one-foot-thick liner compacted in two six-inch lifts. The one-foot thickness was selected because it is the thickness required to enable the combination of geomembrane and clay to function as a composite liner. Bonaparte, *et al.*, (1989) have shown that the effectiveness of a composite liner (expressed as the ratio between the percolation rates due to a geomembrane hole through a geomembrane alone and through a composite liner, for a given hydraulic conductivity of the soil component of the composite liner) is independent of the thickness of the soil layer as long as the layer thickness is larger than the head on the liner. Since the design of the leachate collection system will be done to limit the head on the liner to less than one foot, the optimum thickness of the soil liner is one foot.

As indicated earlier in this section CMC has negotiated an option with ASHCO, Inc. for the purchase of 181,000 cubic yards (in-place) of clay which is a sufficient volume for the clay liner and clay cap for TMA 1, Stages I and II and the closure of TMA 1. CMC has optioned the clay for only the initial cell and its closure for the following reasons:

- Geosynthetic clay liners (GCLs) are available, however they are not at present typically used for liners and/or final covers in Wisconsin. As technology advances and GCLs may become a more attractive option for the soil component of the composite liner, CMC may wish to modify the plan of operation for GCL use in the liner and/or cover system.

- Additional clay sources as provided in the Table 6.3-1 may be available to meet the project needs.

- Other low-permeable soil layers may become more attractive as alternatives to clay such as till or bentonite-amended till.

This information has been provided to appraise the Department concerning the potential source(s) of clay material from off-site. The TMA liners/caps will be constructed at various times over the operation period of approximately 32 years. During this time,improved liner and capping technology could emerge as new combinations of natural and manufactured materials are found to be more effective for containment of wastes. CMC desires to use the most effective, efficient and economical liner and capping system available for the TMA. To enable CMC to exercise the flexibility to select the best liner/capping components available, clay was secured from a supplier only for the TMA 1 construction and closure. Plan modification will address any changes to the type or source of low permeability soil material.

CMC understands that Wisconsin Administrative Code NR 182 does not address the need to characterize the quality and quantity of clay required. However, since the clay is an important component in the composite liner system, CMC proposes to request that the clay supplier (contractor) provide sufficient information to ensure the availability of sufficient quantity and quality of clay under the referenced agreement herein. Although, CMC has negotiated an option with ASHCO Inc., CMC intends to open to public bidding the supply of materials and construction of TMA liner and cap. It is apparent from Table 6.3-2 that other clay suppliers have clay sources with adequate quality and quantity of clay soils. CMC proposes to use a clay source which meets the regulatory and project requirements and is cost-effective in meeting CMC's needs.

6.3.3.1 Estimated Clay Quantities

Assuming that clay soil from an off-site source is used to construct the entire TMA facility, the amount of clay soil material and trucking effort required over the anticipated site life has been estimated. Table 6.3-3 contains a list of the operations years, the liner or capping component for which clay material is required along with an estimated number of truck loads required for each stage of construction. The in-place cubic yards required were increased by an estimated 30 percent to account for shrinkage due to the increase in density from the borrow area to the liner or cap and construction losses. The number of truck loads assumes 20 loose cubic yards per truck and the number of loads is rounded off to the nearest ten trucks. The purpose of this table is to provide the Department with the sequencing for off-site borrow activities and an estimate on the preliminary basis of the hauling effort required to bring the required amount of clay soil to the TMA site. This information is also used in the impact analysis contained in the EIR.

Table 6.3-2

Alternate Clay Liner/Cap Suppliers

Supplier	Source and Location	Estimated Quantity (in-place cubic yards)	Haul Distance (miles)
Hanz Contractors, Inc. 1801 Burek Ave. Wausau, WI 54402 715/675-2361	William J. Tomcek Property T30N-R10E, Section 32, NW¼	~ 150,000	~ 54
Musson Brothers, Inc. P. O. Box 638 Rhinelander, WI 54501 715/369-4828	Brown Brothers Property T35N-R10E, Section 30, SW¼, SW¼	142,000 to 340,000	~ 40
Somers Construction Co. 5474 U.S. Highway 10 East Stevens Point, WI 54481 715/344-9521	Lincoln County Clay lease T30W-R10E, Section 19, NE¼ and Jorgenson Site (held by clay lease location unknown)	150,000 to 500,000 (combination of the two sites)	NP
Krueger & Steinfest, Inc. 539 Forest Road Antigo, Wi 54409 715/627-7693 and Flannery Construction Co. 636 South Grandview Crandon, WI 54520 715/478-2415	Elroy Larsen Tract (held by clay lease) T30N, R10E, Section 5	150,000 to 500,000	NP
Schoone Construction Co. 1708 North 4th Street Tomahawk, WI 54487-2145 715/453-2015	NP	~ 150,000	~ 88
Superior Services, Inc. 10150 West National Ave. Suite 350 West Allis, WI 52227 414/788-4855	Milwaukee Area	~ 700,000	NP

NP - Not Provided

Prepared by: REM
Checked by: KB

Table 6.3-3

TMA Construction Clay Required from Off-Site

Operations Stage	Construction Year	Clay Quantities (cy)		Construction Activity or Component	Approximate Number of Trucks[2]
		In-Place (cy)	Trucked-In (cy)[1]		
Stage I	3	73,500	95,550	Stage I Liner	4,780
Stage II	5	19,900	25,870	Stage II Liner	1,290
Stage III	8	89,900	116,870	Stage III Liner	5,840
Closure TMA 1	12	87,530	113,780	TMA 1 Final Cover	5,690
Stage IV	13	33,000	42,900	Stage IV Liner	2,150
Stage V	19	54,000	70,200	Stage V Liner	3,510
Stage VI	21	26,900	34,970	Stage VI Liner	1,750
Closure TMA 2	23	115,430	150,100	TMA 2 Final Cover	7,510
Stage VII	24	59,900	77,870	Stage VII Liner	3,890
Stage VIII	27	27,100	35,200	Stage VIII Liner	1,760
Closure TMA 3	28	75,050	97,600	TMA 3 Final Cover	4,880
Closure TMA 4	34	80,770	105,000	TMA 4 Final Cover	5,250
Totals		742,980 cy	965,910 cy		48,300 Trucks

cy -cubic yards

[1]Trucked-in cy equals in-place cy plus 30 percent for compaction and construction losses.
[2]Assumes 20 cy/truck hauling "trucked-in" quantity.

Prepared by: REM
Checked by: NXP

6.3.4 Geomembrane Liner Design

Geomembranes are impermeable liquid or vapor barriers typically manufactured as continuous sheets which are flexible and resistant to adverse environments. Currently, thermoplastic polymers are the most frequently specified geomembranes for waste containment applications. The geomembranes are typically manufactured in sheets in the factory and are transported to the job site where placement and field seaming are performed to complete the job.

There are a number of thermoplastic polymer geomembranes currently in use for waste containment application including polyethylene (PE), polypropylene, polyvinyl chloride (PVC) and others. Geomembrane manufacture and design of new products has been evolving rapidly with new and improved geomembranes and other products appearing in the market frequently. CMC desires to provide a geomembrane for the TMA which provides a high level of environmental security at a reasonable cost. Because of CMC's desire to provide the best available product at the time of construction for the geomembrane, a geomembrane type or manufacturer is not specified at this time. The decision on the geomembrane will be made during the final design and bidding process so that the best product can be selected to meet the containment needs for this facility.

6.3.4.1 Materials

Geomembranes are made of one or more polymers along with a variety of other ingredients such as carbon black, pigments, fillers, plasticizers, processing aids, crosslinking chemical, stabilizers, etc. The polymeric material most frequently used for solid and hazardous waste containment today are made of polyethylene material which are crystalline thermoplastics. Examples of polyethylene liner are High Density Polyethylene (HDPE), medium density polyethylene (MDPE) and linear low density polyethylene (LLDPE). The EPA (1993) recommends a minimum thickness of 60 mils (1 mil = 0.001 inches) for polyethylene liners.

To enable CMC to select the best geomembrane available at the time of construction, a generic minimum 60 mil, thermoplastic geomembrane is specified in this report. However, for the purpose of design and stability calculations, it is assumed that a HDPE geomembrane is used for the liner and the geomembrane cap for the proposed facility. The final geomembrane material selection will be based on the best material economically available at the time of final design and construction. The design calculations will be reviewed at that time to take into account any minor modification which may be required. However, the minimum thicknesses of the geomembranes would be 60 mils and would have properties similar to or better than HDPE.

6.3.4.2 Design Calculations

The basic requirements for the geomembrane for this project are:

- chemical resistance to the types of leachate likely to be produced;
- ability to provide enough shear strength along the interfaces both above and below the geomembrane such that slippage of materials placed on the geomembrane will not occur;
- durability; and

- tensile strength properties to withstand the expected stress conditions.

The last item can be achieved by design such that stresses on the geomembrane are absent or minimal. Chemical resistance and durability are a function of the type of the thermoplastic. The interface friction required is a function of both the geometry and the materials and surfaces involved. The minimum required interface shear strength has been evaluated (Appendix D). In order to achieve the required stability, a textured geomembrane may be required on the interior side slopes of the TMA.

6.3.5 Geomembrane Protective Layer

Section 6.3 indicates that the composite liner has two different configurations; one for the bottom and partially up the interior slopes and another for the remainder of the interior side slope. Geomembranes in both configurations are provided with protection against potential puncture and abrasion during the operating period.

For both configurations of the composite liner, the geomembrane is protected from the two mechanical stresses that may act on the liner:

- abrasion due to rough objects in contact with the geomembrane; and
- puncture stress due to forces caused by construction and operational activities.

For puncture stress due to forces caused by compaction and operational activities, industry practices typically dictates that the cushioning geotextile should have a weight per unit area of 6 oz/yd^2. Puncture tests on geotextiles over various types of geomembrane have shown that a puncture resistance of about 75 pounds is obtained with a cushioning geotextile of 6 oz/yd^2. Therefore, a geotextile with a minimum unit weight of 6 oz/yd^2 is proposed for the geotextile cushioning layer. This geotextile cushioning layer will be placed above the geomembrane throughout the facility.

The EPA (1983) recommends a protective soil cover of at least 18 inches for protection of the geomembrane from mechanical damage. EPA (1983) also recommends that the material be placed with a low-ground pressure vehicle to limit stresses on the geomembrane. Puncture damage to the geomembrane is most likely to occur during placement of cover soil if it contains large rocks or sharp objects. In order to avoid this condition, the gradation of glacial till used for cover soils is specified as having 100 percent finer than the three-inch sieve size. This till soil will have a minimum thickness of 18 inches over the entire interior sideslope area except in areas near the bottom of the cell where a leachate collection system (LCS) drainage layer is placed.

Where the LCS drainage layer is placed the configuration of the protective soils is as shown in Drawing No. 29, Detail 1-29.. Detail 1-29 shows that the geomembrane is protected from abrasion by a geotextile as mentioned above and protected from potential puncture by the two-foot thick gravel drainage layer plus the two foot thick till layer. So, four feet of soil protect the geomembrane from puncture on the bottom rather than the minimum 18 inches recommended by EPA.

6.3.5.1 Geomembrane and Leachate Collection System Protection During Initial Filling

Initial start-up of each TMA cell involves putting a sufficient quantity of water in the TMA. Typically this will be a quantity sufficient to cover the leachate collection system drainage layer and allow for the reclaim pumps to operate efficiently. Prior to putting water in the cell, riprap will be placed in the water-filled area as shown on Drawing No. 29, Detail 1-29 During the construction of Stage I, a sufficient quantity of Type II waste rock will be available for this purpose. This waste rock varies in size from 24-inch nominal diameter to 6-inch diameter and should not require extensive processing prior to its use. Since an excess of mine waste rock will be available during TMA 1, Stage I construction for the thickness can exceed 18 inches. During later years of TMA construction, waste rock may not be available in sufficient quantities or not at all times. Screening the larger than three-inch cobbles from the till when processing both drainage layer and screened till (refer to Section 6.12), will make available sufficient quantity of cobbles and boulders for this purpose. The riprap sizing and quantity calculations in Appendix E show that between the mine waste rock and cobbles and boulders screened from the till, a sufficient quantity of riprap for the cell bottoms and for sideslope protection during final filling is available.

6.3.5.2 Geomembrane Protection During Final Filling

Section 7.3.9 of this report describes details concerning a typical final filling of a cell. During the final placement of tailings into the cell, tailings will be placed primarily from two sides of the cell which will force the water pond up against or in close proximity to the interior 3:1 (H to V) slopes of the outer embankments. As a result, wave action on the embankment slopes where the water pool is up against the embankment must be accounted for. The riprap will be installed on the interior slope of the outer embankment where the water pool could come in contact with the embankment slopes. For design purposes, the worst-case scenario for wave action and precipitation was selected. Appendix E contains the wave action calculations for significant wave height, maximum wave height, wave length and design run up based on a worst case fetch of 2,600 feet, an average four-foot pool depth and a maximum wind velocity of 90 miles per hour. These are very conservative assumptions. A conservative assumption was made that riprap will be placed in the upper ten vertical feet of the interior side slopes on each pond during final deposition as illustrated in Appendix E, page 11. Based on the analysis in Appendix E, it was determined that Wisconsin Department of Transportation Section 606 Medium Random riprap stone is adequate to protect the TMA interior slopes. The required riprap material can be processed from waste rock or cobbles and boulders left over from the processing of till soils. Calculations in Appendix E indicate that there appears to be an adequate quantity of these materials available to riprap the interior sideslopes as well as the cell bottoms if that becomes necessary.

Filter calculations in Appendix E show that a geotextile is required between the riprap and the glacial till to prevent migration of fines which might occur due to wave action on the embankment. The geotextile required should be a non-woven geotextile with an apparent size opening size (AOS) of 0.27 mm (~ the number 60 sieve) or less. Appendix E lists examples of geotextiles with adequate AOS for use as a filter between the till and the riprap.

6.4 Leachate Management System NR 182.11(1)(o)

6.4.1 Background Information

The waste characterization studies performed on the tailings and other wastes proposed for containment in the TMA is located in Section 4.2. Section 4.2, includes an evaluation of the leaching potential, including acid generation potential of the tailings. Additionally, the chemical and physical properties and testing of the tailings material is discussed.

6.4.2 Design Rationale for Leachate Collection System

The primary design function is accomplished by the four main components of the LCS which are:

- leachate drainage layer;
- leachate collection system piping;
- leachate removal system; and
- leachate transfer system.

The design of the LCS component will be discussed in the context of these performance goals. The LCS is designed to meet the following performance goals:

- to provide transmissivity so as to limit the head on the liner during operation and the post-closure period;
- to provide collection and removal such that the overall allowable percolation rates during the design period can be met; and
- to provide a plumbing system which can be cleaned during the operating and closure period.

The primary design functions of the LCS are to convey tailings pore water (leachate) to the collection points in the most efficient manner possible and to remove it as needed.

6.4.3 Leachate Drainage System

The purpose of the leachate drainage system (drainage system) is to minimize the head on the base liner and convey free liquids (tailings pore water) to the LCS piping.

6.4.3.1 Leachate Drainage Layer and Filter Layer Design

The gradation, hydraulic properties, etc. of the drainage layer, the filter layer, and the composite liner have been input to the HELP model. The drainage layer has been designed to inhibit the migration of tailings solids into the drainage layer and have sufficient hydraulic capacity for the design flow. The design rationale for the drainage layer and filter layer design relate to the physical filtration of solid particles which could potentially clog the drainage layer.

Appendix F contains the filter design calculations for the drainage and filter layers. Since the tailings are very fine and consist primarily of particles passing the number 200 sieve (P200=100 percent) and the drainage layer is coarse mainly passing two inches and retained on the number 40 sieve, a filter layer is necessary to prevent the physical migration of tailings to the drainage layer. Therefore, the glacial till will be processed to produce a material suitable for the drainage layer. The till soils will be screened to separate the plus two-inch size, and the undersize materials will be removed by screening and washing to produce a material which is 100 percent larger than the number 40 sieve. The resultant material, which will be used for the drainage layer grades primarily from two-inch size to number 40 size. The "wastes" from the production of the drainage layer is coarse gravel and cobbles (i.e., > two inches) and material passing the number 40 sieve (i.e., fine sand and silt) referred to as P40 soils. These P40 soils, available from the processing of drainage layer will be used as a filter between the tailings and the till. Appendix F contains the calculations which show that with the P40 layer between the tailings and the till, all filter criteria are met. Appendix F also contains filter criteria calculation for the unprocessed till as a base above the drainage layer (as the filter). In addition, a geotextile filter is required between the till and the drainage layer. The geotextile filter selected based on these calculations is a non-woven geotextile with an apparent opening size (AOS) of less than 0.160 mm. In addition, the calculations show that head loss between the two layers should be minimal and therefore drainage is not inhibited. The resultant design for the drainage layer and filter layers for the leachate collection system include the following from top to bottom below the tailings:

- six inches of P40 soils;
- 18 inches unprocessed till soil;
- geotextile; and
- 24 inches drainage layer (< 2", > number 40 sieve).

A 24-inch thick filter has been provided for additional protection from potential chemical precipitation. Likewise, the drainage layer is increased from what is typically a 12-inch layer to 24 inches to account for potential clogging.

6.4.4 Cell Base Configuration

The base of each TMA cell is designed with grades which result in a high point which in general run diagonally across each cell as shown on Drawing No. 15. From the high point the cell base slopes at a two percent grade in opposite directions toward sumps located in cell corners directly opposite of the high point. For example, in TMA 1 (southwestern most cell, see Drawing No. 15) the sumps are located in the southwest and northeast corners of the cell. The maximum leachate flow distance (i.e., distance from the central high point in the cell to the sump) varies from approximately 940 feet in TMA 1, to approximately 520 feet in TMA 3. The drainage layer extends up the interior 3:1 (Horizontal to Vertical (H to V)) slope on the low end of each cell a vertical height equal to the elevation of the base grade high point in the center of each cell. Drawing No. 16 shows the extent of the drainage layer in plan view and typical section views are provided on Drawing No. 29, Detail 1-29 and Drawing No. 28, Detail 6-28.

6.4.5 Leachate Collection System (LCS) Piping

The LCS piping is located along the toe of the interior 3:1 (H:V) slopes in each cell. Drawing No. 16 shows the location on LCS piping on the base grades while details of the pipe systems are provided on Drawing No. 29, Details 1-29 and 2-29. The pipe system consists of a perforated six-inch diameter polyethylene (PE) pipe with a standard dimension ratio (SDR) of 17. The pipe is designed with an open channel flow capacity which exceeds the predicted peak flows and with a strength which exceeds the stresses under anticipated loads. The pipes are designed against large deflections (> five percent of pipe diameter) under the maximum load expected after closure. The pipe bedding and the placement of bedding material around the pipe will cause a uniform loading on the pipe as well as reduction of load on the pipe due to soil-structure interaction of the bedding material completely surrounding the pipe and the strength of the tailings material itself. Pipe flow calculations and pipe strength analysis are provided in Appendix G of this report. Each LCS pipeline drains to a leachate collection system sump located opposite the high point in the base grade. The design of the leachate collection sumps is discussed in the section which follows. The six-inch diameter pipes are provided with clean-outs at each end to allow for cleaning.

6.4.6 Leachate Removal System

The base of the facility is designed with a minimum two percent slope toward leachate collection sumps located directly opposite of the high point in the base grades perpendicular to the liner system base configuration. These sumps are designed to be above the TMA liner so that there is no need for liner penetrations. Drawing No. 30, Detail 1-30 provides section and plan view of the proposed sumps and side slope risers (SSRs) which provide the mechanical means of evacuating leachate from the base collection point of the tailings cell. Sumps and pumps have been sized (see calculations in Appendix H) such that under maximum leachate flow, the leachate can be removed to an acceptable level. Note that the SSR's are sized and will be pumped such that a water trap exists at all times after pumping.

The SSR riser pipe will be an 18-inch PE pipe with perforations in the sump section. Inside the 18-inch pipe is a submersible pump designed for operation in adverse environments. The pump curve determination based on total dynamic head loss for each TMA sump pump is provided in Table 6.4-1. The pumps proposed for the TMA sumps are rated at 70-90 gpm, with a maximum total head of 85-125 feet. Each pump is actuated by a submersible liquid level transducer mounted over the submersible pump. The transducer will be programmed to provide an automatic pump response and alarm system as follows:

- Pump on - water level at 6½ feet above SSR invert at upslope edge of SSR sump (Drawing No. 30, Detail 1-30);
- Pump off - water level 4½ feet above water trap in SSR sump (Detail 1-30); and
- Pump alarm - water level greater than 6½ feet above than LCS invert upslope edge of SSR sump (Detail 1-30).

Programming the transducer in this manner ensures that a water trap remains in the sump even if the pump off switch malfunctions. Thus the opportunity for oxygen to come in contact with the tailings is minimized.

Table 6.4-1

Leachate Sump Evacuation Pump

Location	Design Flow (gal/min)	Static Head (ft)	Total Dynamic Head (ft)
TMA 1 - North Sump	25	85	86
	125	85	102
	225	85	136
TMA 1 - South Sump	25	90	95
	125	90	182
	225	90	364
TMA 2 - North Sump	25	125	131
	125	125	244
	225	125	480
TMA 2 - South Sump	25	115	116
	125	115	137
	225	115	180
TMA 3 - North Sump	25	110	114
	125	110	196
	225	110	365
TMA 3 - South Sump	25	110	115
	125	110	205
	225	110	394
TMA 4 - North Sump	25	115	121
	125	115	233
	225	115	465
TMA 4 - South Sump	25	110	114
	125	110	196
	225	110	365

Note: Frictional losses are due to the interior berm configuration pipe length, gate valves and air release valves placed at mid point of maximum length pipe runs with a pipe roughness coefficient of 120. (Refer to Appendix H for additional pump sizing calculations).

Prepared by: JBK
Checked by: NXP

All SSR pump alarms will report to the mill control room and signal the operator of a malfunction or other problem in the LCS pump(s).

Leachate is pumped to the surface through a flexible three-inch PE pipe SDR 7 (Drawing No. 30, Detail 1-30). The pump and transducer attached to the PE pipe are mounted on a transporter. The transporter is attached to two stainless steel cables extending to the surface which enables the operator to easily remove the pump and transducer for maintenance, repair or replacement. The second stainless steel cable is provided as a backup in the event the other cable were to break.

The 18-inch PE SSR pipe along with the six-inch PE LCS cleanout pipe extend up the cell interior slope in the same gravel filled trench. The trench bottom is lined with a cushioning geotextile to prevent puncture of the geomembrane liner by the pipe bedding material.

Appendix I contains pipe crushing calculations for the SSR which indicate that an 18-inch PE SDR11 pipe has sufficient strength to meet the anticipated loadings.

The LCS cleanouts are designed so that there are no liner penetrations. LCS cleanouts are located at each end of each LCS pipe to allow the LCS piping to be cleaned from two directions. The SSR pipe can be cleaned by removing the flexible leachate transfer pipe, pump with transducer.

6.4.7 Leachate Transfer System

Leachate will be pumped from the cell bottoms with a submersible pump to the surface via a three-inch diameter flexible PE pipe. This connects to a three-inch PE pipeline located next to the tailings pipeline and reclaim water return line in a notch in the berm as shown in Drawing No. 30, Detail 1-30. Spillage will be contained in the unlikely event that a breach would occur since this leachate transfer pipe is within the TMA cell lined area. All three-inch leachate transfer lines join the reclaim water forcemain at the junction of the TMA cell common berms. Leachate then leaves the TMA in the 12-inch PE reclaim pond pipe. The reclaim piping penetrates the liner system at six locations, between TMA 3 and 1, and TMA 4 and 2, and when exiting TMA 1 and 2. These penetrations are approximately one foot below the maximum free board elevation, so the likelihood of tailings or leachate attaining this height is remote. Drawing No. 29, Detail 5-29 provides a detail of the seal design for the liner penetration. After the reclaim water pipe exits the liner it is encased in a 18-inch diameter PE pipe for the entire distance to the discharge at the reclaim pond. The reclaim water pipe, once outside the lined area of the TMA, flows by gravity to discharge in the reclaim pond. In the process water pond the leachate and reclaim water will have a detention time of approximately three days, after which it will be recycled with the water decanted back to the concentrator as process water.

6.4.7.1 Leachate Storage System During Post-Closure

The reclamation activities detailed in the MPA indicate that the reclaim pond will remain for approximately 10 years until post-closure leachate quantities decrease. After this time, the reclaim pond may be reclaimed and a leachate storage tank maybe installed for operation of the leachate management system during the remainder of the post-closure period. The tank is proposed adjacent to the TMA access road as shown on Drawing No. 18. Details of the leachate tank, piping, load-out structure and spill containment area are not provided at this time since the

tank is not expected to be constructed until 10 years into the post-closure period, at which time new technology will likely be available.

The HELP model calculations indicate that a maximum leachate generation rate is conservatively estimated at 25 gpm (Appendix J) during the first ten years of post-closure. Based on this design criteria, the process water basin, which has a capacity of 15 million gallons is an adequate capacity for more than one year of leachate storage.

Removal of the leachate will be contained per an appropriate schedule. The site operator will be responsible for arranging the timely removal and hauling of the leachate. CMC will maintain the leachate collection system and leachate storage system during the forty-year post-closure care period as needed. If the leachate collection rate drops to a sufficiently low level, the operation of the LCS and wastewater treatment may be discontinued early. A discussion of the treatment of leachate at a wastewater treatment plant is provided in the MPA.

6.5 Tailings Management Area Embankment Configuration NR 182.11(1)(s) and (t)

6.5.1 General

The TMA embankments described herein include construction of the berm in two stages for each cell. To minimize earthwork and the area of disturbance, each of the four TMA cells share one common berms with two other cells. The materials proposed for embankment construction are described in detail in Section 6.3.2 (liner support materials) and specification for construction methods and materials are provided in Section 7.2 of this report.

6.5.2 Initial Staging Berm

The berms for each TMA cell are staged to allow for construction and operation in two vertical stages. For the initial stages the total depth of the cell will range from 45 feet to 50 feet. Two stage construction and operation was selected because it has the following advantages:

- less operational difficulties with a 45-50 feet cell depth;
- minimizes liner exposure to the elements;
- easier to close in the event of early closure or a long-term suspension of plant activity;
- less earthwork in preproduction phase;
- easier access to the cell bottom; and
- lower capital expenditures during the pre-production phase of plant operation.

Drawing No. 28, Detail 3-28 shows typical design features for the construction of TMA 1 Stage I. The important features of the design are listed below.

- Twelve-foot wide berm crest roadway constructed using granular material will provide all-weather access to the TMA operating system.

- The berm crest roadway will be sloped at two percent toward the TMA cell which prevent any spills or contact water from leaving the containment area.

- Three pipelines consisting of a tailings slurry, water reclaim, and a leachate pipe are located on a shelf on the inside crest of this berm as shown in Drawing No. 28, Detail 3-8. The shelf is within the lined containment area and is sloped toward the inside of TMA. Any breach in any of these three pipelines would result in complete containment of the spill within the TMA cell.

- The roadway is outside of the geomembrane anchor and so the geomembrane is protected from abrasion and potential damage by traffic on the berm.

- The clay component and geomembrane component of the composite liner are overbuilt at the first stage crest elevation to provide ease in splicing of the clay and geomembrane for the second stage of the cell.

- The geomembrane splice for the second stage will occur on a flat portion of the berm which minimizes stress on the splice.

As shown in Drawing No. 14 and Drawing No. 28, Detail 5-28 each of the TMA cells share a common berm on two sides. An all weather access road for operation and construction activities is provided at the crest. These common berms have a high point where the four cells meet and the slope is outward from this point at a 0.5 percent slope in all directions. This high point at the common corner of each cell provides the discharge point for the final slurried tailings deposition as described in Section 6.6.2.1 of this report. In addition, surface water drainage off of the final cover is facilitated by the outward 0.5 percent slope on these common berms. During the post-closure period the access road can be used to access the central portions of the TMA for inspections, monitoring and other post-closure activities.

6.5.3 Second Stage of Berm

The general berm configuration for the second stage of each cell is illustrated on Drawing Nos. 23 through 27, the Engineering sections. The berm is raised by compacting glacial till soils starting at the outside toe of the stage one berm. Upon completion of this structural fill construction the composite liner construction will be extended. Sequencing and description of construction is provided in more detail in Section 7.2 of this report. The all-weather road on this second stage berm will remain for access during the post-closure period.

In summary, the berms are proposed to be constructed in two stages for each TMA cell. All-weather access roads are provided during operation, closure and post-closure. Pipelines are located on a shelf on the inboard slope of each cell for spill containment.

6.5.4 Freeboard NR 182.11(1)(q)

Wisconsin Admin. Code NR 182.04(21) defines freeboard as "the height of the crest of the dam above the adjacent liquid surface within the impoundment". The "design freeboard" is defined as "the minimum freeboard which would occur during the design flood". NR 182.11(1)(q) requires that "sufficient freeboard measured from the inside crest shall be provided so as to contain the

100-year, 24-hour rainfall event and to prevent overtopping by waves during the design storm, or a minimum of five feet of freeboard."

The proposed TMA embankment design provides for five feet of freeboard. The freeboard as defined above is not of concern during normal operations because the tailings beaches will slope away from the inside berm slopes at a very flat 0.5 percent slope toward the center of the cell from all directions. The water pond which has an operating depth of three to five feet will exist at least 150-200 feet away from the interior side slope. The slope of the tailings below the water will be likely steeper at five percent or greater. The freeboard and overtopping have been considered during the final stages of slurried tailings filling when the freeboard is a minimum. The tailings filling sequence is described in Section 6.6.2.1 of this report. To analyze for overtopping the worst case scenario in TMA 2, the largest cell, was considered. Based on this analysis presented in Appendix E of this report, the required freeboard conditions are met.

6.6 TMA Final Cover Design NR 182.11(1)(m) and (n)

6.6.1 General

The general performance objectives for the design, construction, operation and closure of the TMA facility are in accordance with good industrial practice, and in compliance with standards prescribed by the State of Wisconsin.

To limit tailings oxidation and solute release from the tailings mass, the following features are incorporated:

- placement of tailings at a high pH, with excess alkalinity to minimize the potential for acid generation during operation;
- limiting of oxygen entry into tailings during the operating and post-closure period;
- design of a composite base liner system to control percolation rates; and
- design of a final cover which meets the following objectives:
  - limits oxygen entry into the tailings mass,
  - limits infiltration of precipitation into the tailings mass,
  - provides a stable surface which can meet reclamation requirements,
  - provides an erosion-resistant surface, and
  - has a permeability equal to or less than the TMA base liner system.

The selected final cover which meets the stated design criteria consists of a multi-layered cover system including the following components from top to bottom:

- topsoil and vegetative layer;

- rooting layer;
- drainage layer;
- thermoplastic geomembrane;
- low permeability soil layer; and
- grading layers and support layers where required.

Section 6.7 of this report provides data generated by modeling the final cover and liner systems using the HELP model, which shows that the final cover effectively limits infiltration from precipitation. Section 6.6.8 of this report discusses the effectiveness of the final cover in preventing oxygen intrusion during the post-closure period.

6.6.2 Final Cover Grades, Grading Layers and Support Layers

The final cover grades are achieved by the completion of two processes, the final tailings slurrying explained in Section 6.6.2.1 of this report and the placement of cover grading layers as described in Section 6.6.2.2.

The final cover grades are proposed to meet the requirements of Wisconsin Administrative Code NR 182.11(1)(m). The final grades provide for a minimum two percent slope as shown in Drawing No. 18 (Final Grades). Each TMA cell is graded and closed to the configuration shown and will drain radially from a high point near the center of the cell. Surface water management for the closed TMA cells is discussed in Section 6.9 of this report.

6.6.2.1 Tailings Deposition for Final Cover Grading

Section 7.3 of this report provides details concerning the tailings deposition which will occur during normal operation. Briefly, normal operations will consist of depositing tailings from spigots opened sequentially around the cell to allow for tailings beaching and development of a pond with a minimum depth of approximately three to five feet. Figure 6.6-1 shows this configuration (in section and plan view) at the point when this depositional method will stop and the reclamation shaping of the tailings will begin.

For purposes of discussing the reclamation cap, TMA 1 is used as an example of the final tailings shaping during reclamation. The timing and sequencing of the start of reclamation is tied to the completion of the adjacent TMA cell which in the case of TMA 1 is TMA 2 Stage III. TMA 2, Stage III then would be operational at the time tailings shaping for reclamation begins. When TMA 1 reaches a typical configuration as shown in Figure 6.6-2, reclamation shaping of the final tailings surface would begin and generally will aim at slurry filling to the extent possible before attempting placement of other materials to achieve the required final slopes. The reclamation activities then would consist of the following:

- reclamation shaping of the tailings surface would begin by operating the spigots primarily around two sides of TMA 1 refer to Figure 6.6-2;

- the resultant configuration will be an approximate 0.5 percent grade from northeast to southwest for TMA 1; and

- discharging in this manner results in a water pool forming against the southwest and south inside berms of TMA 1. The size of the water pool will be kept at a minimum by pumping accumulated water to the operating TMA cell or the reclaim pond.

The slurried tailings as shown in Figures 6.6-2 would have an average slope of 0.5 percent toward the southwest. At the pool the tailings slope below water may range from five to ten percent. The final tailings spigotting would proceed from the north and east sides of TMA 1 until the tailings level approach the top of the north and east berms.

The depression left after the water pool is pumped out to the next operating TMA cell is filled through a combination of techniques which may include spigotting (discharging) tailings for a short time from the west and south berms until the depression shrinks to the point where water can no longer be effectively decanted by the floating barge system. At this point excess water would be dewatered from the cell surface to the extent practicable, after which the filled cells would be allowed to settle for six to 12 months. Additional beach tailings could be placed in areas of settlement by excavating from the tailings beaches in the subsequent (now operational) cell. An attempt would also be made to cover any soft or spongy areas with waste rock to stabilize them, if possible. During the period of time the cell is allowed to settle, the tailings will be inspected daily for signs of the initiation of acidic conditions. This would be accomplished by performing a set of field paste pH tests to measure surface pH values. If significant oxidation is identified, the abatement techniques described in Section 6.2.7 of this report can be used to eliminate the problem. Typically, techniques such as spraying the impacted areas with a lime solution and/or bactericide will abate the oxidation process.

6.6.2.2 Grading for Final Cover

It is presently anticipated that reclamation grading for the final cover two percent slope will involve placing a minimum three-foot thick working mat of one of the following:

- Type I or Type II waste rock hauled by truck from the stockpile at the plant site;

- glacial till or other available soils hauled from stockpiles in the TMA area.

Placing these materials would likely require winter placement or that a geotextile be placed over the tailings in the former pool area to allow mechanical placement of tailings, mine waste rock or soils. Typically a high modulus woven geotextile is used to bridge over soft areas. A woven geotextile or similar product is provided to allow the required trafficability.

As indicated earlier, three to five feet of settlement is expected with the larger settlements occurring in the center of the cell where mainly fine tailings accumulate. It is anticipated that during the six- to 12-month period, a crust of dry tailings will develop within sufficient trafficability to allow placement of a three-foot thick lift of till soils. If the subgrade trafficability does not provide adequate support for construction traffic it is likely that a woven geotextile will be needed to improve the trafficability of the tailings. With this final grading layer in place, and the cell shaped to the correct grade at a minimum two percent slope, the final cover layers can begin to be placed.

Ideally the reclamation grading layer would be placed in the winter when the tailings surface is frozen, then allowed to settle over winter, spring and early summer. To determine the settlement characteristics of the tailings, one dimensional consolidation testing was completed by SRK and reported in Appendix B. The SRK consolidation tests simulated consolidation of tailings at depth within the TMA. Based on these tests the majority of the settlement in the tailings is expected to take place within one year of the dewatering of the leachate. If the tailings increase in density by six to eight percent as suggested by test results (Appendix B, Table 3), a consolidation (settlement) of the post-closure tailings surface of four to six feet is estimated. The final cover construction normally will not be started until eleven months had elapsed from the last slurried tailings deposition. During this time period some settlement would be expected resulting in the need to add material to re-establish the reclamation grading slope. This would be accomplished by filling from the outsides of the cell inward with mine waste rock or till soils. Once the final reclamation grades have been established at the two percent slope the cell would be ready for final cover placement.

6.6.2.3 Final Tailings Deposition for TMA 4

The above-described reclamation scenario is predicated on having the adjacent TMA cell constructed and in operation at the time when the final cover layers are being placed in a cell. This situation does not exist when slurry deposition in TMA 4 is completed. For this circumstance a slightly different reclamation shaping (grading) scenario is anticipated because the water in the pool will have to be pumped to the reclaim pond.

Instead of beach tailings, till soil, waste rock construction debris or a combination thereof could be used to fill in the corner sump area. The two percent reclamation grade also cannot be formed by tailings, and as a result the earthwork estimates for TMA 4 (refer to Section 6.12) assume that all materials above the level obtained with the final tailings slurried would be filled with borrow till or other soil and/or non-organic construction debris.

6.6.3 Low-Permeability Soil Layer

The clay capping layer is of similar design to the clay liner design described in Section 6.3.3 of this report. The material specification and functions are also the same.

The low-permeability soil capping layer, however, also is expected to assist the geomembrane in preventing oxygen entry into the tailings. Section 6.3.3 of this report provides material specifications for the clay capping soil which specifics among other things that the clay be compacted at or above the optimum moisture content as determined by the standard Proctor Test (ASTM D-698). A clay material compacted to an adequate density at this moisture range has a high moisture content with very few voids (degree of saturation >95 percent) through which oxygen diffusion could occur. Senes and Beak, 1988 using the RATAP model showed that oxygen entry by diffusion is reduced substantially for compacted, saturated till caps. Additional discussions and calculations for oxygen diffusion through the liner and cap are provided in Section 6.6.8.

6.6.4 Geomembrane Material

The geomembrane placed above the low permeability soil layer is extremely unlikely to be subjected to tensile stresses and undesirable differential settlements in view of the uniformity of the tailings. In addition the surface is relatively flat at two percent slopes and this eliminates slope stability concerns. Therefore, the most desirable features are durability and chemical resistance (to a much less extent compared to the liner) for the selected geomembrane. A final geomembrane selection will be made at the time of construction. Selection now would be limited to products available, such as HDPE, LLDPE, PP and PVC.

6.6.5 Drainage Layer Design

The drainage layer above the geomembrane serves to transmit the water percolating through the cover soils above the drainage layer so that ponding on top of the geomembrane and increased percolation through the hydraulic barrier can be avoided. The transmissivity of this layer has to be high and also the hydraulic conductivity. The material selected for the base drainage layer, a processed material from the local till (minus two-inch; plus No. 40 Size) has been tested and found to have a hydraulic conductivity of 0.3 cm/sec. With this high hydraulic conductivity, the head on the liner and the percolation through the cap have been found to be acceptably small, from HELP model simulations which are discussed in detail in Section 6.7.

6.6.6 Rooting Layer Design

The rooting layer overlies the drainage layer and the composite cap consisting of the geomembrane and clay capping layer. The rooting layer is designed to serve three functions as follows:

- provide sufficient depth on the reclamation cover to enable vegetation to establish itself;
- have adequate moisture holding capacity to sustain plant growth; and
- have an adequate thickness to prevent climatic and other damage to the composite cap components.

The rooting layer is proposed to have a minimum thickness of three feet which, when combined with the topsoil provides three and one-half feet of soil for vegetative soil rooting zone. Ayres, 1984 indicated that ... "99 percent of the roots of the expected species are located within 900mm (35.4 inches) of the ground surface." The three-foot rooting zone proposed then is an adequate thickness for the expected species to establish roots and also is of adequate thickness to prevent roots from damaging the composite capping components.

The rooting layer must have adequate moisture capacity to sustain plant growth under varying climatic conditions. The Soil Conservation Service (SCS) measures the moisture capacity of soils as the available water capacity. The available water capacity is defined as the difference between the field capacity and the wilting point expressed as inches of water per inch of soil. The non-organic horizon of the glacial till is proposed for use as the rooting layer in the final cover. SCS, Soil Survey of Forest County soil survey field sheet number 67, shows that the majority of the soil are the Wabeno-Goodwit series which, according the SCS have available water capacity

ranging from .08 to .18 inch per inch. Soils with available water capacities in this range are suitable for sustaining plant growth under most climatic conditions.

The rooting layer soil in combination with the topsoil layer and drainage layer must have an adequate thickness to prevent climatic conditions such as soil drying and freezing and thawing from negatively impacting the composite liner components. Soil drying is not of great concern since the total soil thickness above the geomembrane component is 54 inches (four and one-half feet) and soil cracking from drying typically does not extend to a depth greater than two feet.

The concern with frost action centers around the potential for damage from frost heave primarily to the clay component of the composite liner. Frost action is a concern for the clay component because frost heave caused by the formation and growth of ice layers and lenses in clay soil could result in increased hydraulic conductivity. The proposed composite final cover components are protected from frost heave by the clay component being covered with 54 inches of soil (i.e., six inches topsoil, 36 inches - rooting layer and 12 inches drainage layer). The clay is then below the maximum frost depth which is 54 inches as reported by Ayres, 1984.

Current frost depth and snow depth data for Wisconsin from the Wisconsin Agricultural Statistics Service show that frost depth is greatly reduced by snow cover, so much so that the frost depth across the state is more dependent on snow cover than latitude. Frost in Wisconsin between 1961 and 1995 in areas not associated with traffic had a maximum depth of 34.4 inches in 1981 with the state average for this period being about 18 inches. The 54-inch frost depth provided in Ayres, 1984 is conservative and frost heave in the clay component of the liner is unlikely.

6.6.7 Topsoil and Vegetative Layer

Details for the revegetation of the final cover are outlined in Section 5, Reclamation of the MPA.

6.6.8 Final Cover Performance in Preventing Oxygen Entry

Oxygen entry to the tailings may occur by three mechanisms; diffusion, convective currents and barometric pumping; and by three generalized pathways as illustrated in Figure 6.6-3, through the cover, through the embankments/liner and through the foundation soils/liner.

6.6.8.1 Convective Flow

Because of the extremely fine nature of the tailings (hence their very low air permeability), their encapsulation in fine grained soil liners and covers with very low permeability, and the buried location of the deposit, the potential for convective flow of air through the tailings is negligible.

6.6.8.2 Diffusion

While the geosynthetic membranes remain reasonably intact there will be an extremely low rate of diffusion through the geomembrane liner and this is assumed to be negligible. The upper geomembrane liner will be maintained to ensure its long-term effectiveness, thus diffusion through the cover (pathway (1) in Figure 6.6-3) will remain negligible. Diffusion through the sidewalls and foundation pathways [pathways (2) and (3)] will be slow even after hundreds of

years as a consequence of the long pathway lengths. Kinetic laboratory testing results are required to estimate the quantity of oxygen that must transfer into the tailings before significant oxidation will occur. These kinetic tests are presently underway as described in Section 4. These tests have, as yet, not resulted in the development of low pH conditions.

6.6.8.3 Barometric Pumping

Since the geomembrane in the cover will be maintained, barometric pumping through the cover will be negligible. If the geomembrane cap deteriorates changes in barometric pressure will result in airflow into and out of the airfilled void spaces in the tailings via pathways (2) and (3). Since the inward and outward flow of pore gas will be across a boundary with granular media on either side, it represents essentially the exchange of the same pore gas. Since the pore gas transferring over this boundary will be deficient in oxygen, barometric pumping will not add to the oxygen flux into the tailings.

6.6.9 Closure Schedule

The schedule for closure of each of the four cells is based on a 28-year operating life and ore production assumptions. The anticipated times for closing each of the four cells are detailed in Section 6.12 of this report.

6.6.10 Short-Term Shutdown and Premature Closure

6.6.10.1 Short-Term Shutdown

In the event that the mining and milling operations are suspended for two years or less, the mine waste materials would have to be protected from conditions which could lead to the development of acid conditions in the tailings. The initiation of acid generation is prevented by either of the following methods:

- for a short term shutdown (i.e., 3 months or less) the tailings surface could be treated with a lime spray and/or bactericide; and

- for a longer-term shutdown (i.e., up to two years) the entire tailings filled area could be flooded and maintained underwater for the duration of the shutdown. Prior to the flooding of the tailings however, riprap would have to be placed along the inside berm slope of the cell from the top of the existing tailings to a minimum of five vertical feet above the anticipated high water level. Properly sized and graded waste rock could be used for this purpose.

Either of these methods are effective in protecting the tailings from environmental impacts.

6.6.10.2 Premature Closure

Premature closure of the TMA might be necessary if the mining and milling operation is shut down for an extended period of time. Given this scenario the active TMA cell may require closure prematurely, that is, closure and final covering before the final design grades are met. For the purpose of this discussion it has been assumed that TMA 2, the largest cell has Stage III and IV constructed, Stage III is full and active filling is taking place in Stage IV of TMA 2. A

worst-case scenario exists when Stage IV is about half filled since the tailings deposition has not completely filled the cell and since operations have ceased and closure and final covering activities have to be initiated. To prepare the partially filled cell for receiving the final cover as described in Section 6.6.1 through 6.6.7, the following activities would be required:

- placement of all remaining mine wastes such as wastewater treatment plant solids, Type II waste rock, etc., in the open TMA cell;
- excavation of the exterior berm to an elevation a minimum of five feet above the existing tailings surface. Only berms not common to the other cells would be excavated since common berms are required to be maintained for the already developed and closed cells;
- removal of the composite liner (geomembrane and clay components) from the excavated portion of the berm and attachment with the hydraulic barrier of the final cover;
- changes in plumbing (SSRs clean outs, leachate conveyance pipes etc) as a result of lowering of the embankments; and
- placement of the excess soils excavated from the cell berms as part of the grading layer for the final cover.

The completion of the above-listed additional premature closure activities will allow the facility to be closed and final covered as described earlier in this section. Premature closure of a cell then is easily accomplished with these modifications to the closure procedures described above.

6.7 Water Budget for the TMA NR 182.08(2)(f)

NR 182.08(2)(f) requires preparation of water budgets for three time periods and three climatological situations to estimate amount and quality of seepage and discharge to surface and groundwater.

For the TMA the above requirements are met by the use of two separate analyses. First, the percolation through the liner is estimated considering precipitation, slurry water, evaporation, surface run off, evapotranspiration, soil and tailings moisture holding capacity and the rate at which fluids can conduct through the vertical profile of the TMA. The second analysis consists of determining the quality of discharge to the groundwater by a flow-transport model which considers the quality and quantity of percolation from the TMA, recharge in the areas surrounding the TMA and groundwater flow velocities and volumes. These analyses are performed continuously for the periods of pre-construction, operation and post-closure periods of the TMA such that the requirements of the three time periods and the three climatological situations are satisfied.

The first part of the analysis, i.e., estimation of the percolation through the liner, has been prepared using the water balance model HELP, Version 3.01 developed by Schroeder, *et al.*, 1994a, 1994b. All variables listed in the previous paragraph, except slurry water (i.e., precipitation, evaporation, surface runoff, soil and tailings moisture holding capacity and hydraulic conductivity of the materials) can be accounted for in the HELP model. For including

the effects of slurry water, a slight modification can be made as described later in Section 6.7.4.2 to obtain conservatively higher values of percolation through the liner. Thus, this approach is appropriate and, in general, conservative, provided the appropriate input parameters are selected. Besides the geometry of the cells, the key parameters for the HELP model application for the TMA are the properties of the tailings material and the hydraulic conductivities of the barrier layers. For the tailings materials, laboratory tests were conducted to obtain data on porosity, field capacity, wilting point and hydraulic conductivity such that estimates of these properties could be made for different consolidation conditions (Appendix B). For the barrier layers, the saturated hydraulic conductivity is the critical property and this will be specified in the construction specifications.

Once the percolation through the liner is estimated, flow and transport through the foundation soils are modeled. This transport modeling is not complete at this time and will be submitted as an addendum to this report. During construction, surface waters from precipitation will be routed through appropriate runoff basins prior to discharge. During operation, discharge to the surface waters will not occur from the portions within the waste limits of the TMA. After reclamation, surface precipitation is diverted off the top of the reclaimed tailings cells. The discharge to the surface waters is described in Section 6.9.

Thus, by the use of two separate models, i.e., HELP and Flow and Transport models, a conservative estimate of water balance is done. The details of the HELP model, the modification for the slurry water ponding and properties of the tailings used in the analyses follow after which the results are discussed.

6.7.1 HELP Model For Water Balance of Tailings Cells

The water balance for the TMA (one contiguous area consisting of four waste tailings cells) has been determined by the use of the HELP model Version 3.01 released on October 14, 1994. This computer program was developed by the USCOE Waterways Experiment Station, Vicksburg, Mississippi, for the U.S. EPA, Risk Reduction Engineering Laboratory , Cincinnati, Ohio, in response to needs in the Resource Conservation and Recovery Act (RCRA) and the Comprehensive Environmental Response, Compensation and Liability Act (CERCLA).

The HELP computer program is a quasi-two-dimensional hydrologic model of water movement across, into, through and out of solid waste management facilities. The computer program accepts weather, soil and design data. Version 3.01 uses solutions techniques that account for the effects of surface storage, snow melt, runoff, infiltration, evapotranspiration, vegetative growth, soil moisture storage, lateral subsurface drainage, leachate recirculation, unsaturated vertical drainage, and percolation though soil, geomembrane or composite liner. Waste storage facility designs consisting of various combinations of vegetation, cover soils, waste cells, lateral drainage layers, low-permeability barrier soils, and synthetic geomembrane liners may be modeled. The model facilitates a rapid estimation of the amounts of runoff, evapotranspiration, drainage, leachate collection, and liner leakage that may be expected to result from the waste containment facility.

The primary purpose of HELP model is to assist in the comparison of facility design alternatives as judged by their water balances. Since this model performs water balance for a layered system including vertical percolation layers, lateral drainage layers, barrier soil layers, and geomembrane

liners, the HELP model is an appropriate tool for determining the TMA's water balance in this project.

Weather data required are daily precipitation, temperature and solar radiation. Necessary soil data includes porosity, field capacity, wilting point, saturated hydraulic conductivity, and SCS runoff curve number for antecedent moisture condition II. Design specifications include parameters such as slope and maximum drainage distance for lateral drainage layer, layer thicknesses, surface cover characteristics and information on geomembranes utilized in the design.

6.7.2 HELP Model Program Overview

The program uses weather, soil, and design data to generate daily estimates of water movement across, into, through and out of the tailings cells. To accomplish this objective and compute a water balance, daily precipitation is partitioned into surface storage (snow), snow melt, interception, runoff, infiltration, surface evaporation, evapotranspiration from soil, subsurface moisture storage, liner percolation and subsurface lateral drainage to collection and removal.

For accomplishing the water balance analysis one needs the weather data on the site, soil data of all layers, and design data of the tailings cells.

6.7.2.1 Weather Data Requirements

The weather data required in the HELP model are classified into four groups: evapotranspiration, precipitation, temperature, and solar radiation data. Daily weather data are required for the four groups. The HELP model has four modes of compiling daily weather: default, synthetic, manual or importing other data bases. Default data is provided with the HELP Model. Default precipitation, temperature and solar radiation data are available in the form of five years of historical data for 102 U.S. cities. Daily synthetically generated precipitation data can be stochastically computed for up to one hundred years for 183 U.S. cities. The HELP model has weather coefficients and algorithms to stochastically compute one hundred years of evapotranspiration, temperature and solar radiation data for 183 U.S. cities. Evapotranspiration, precipitation, temperature and solar radiation data can also be input manually on a daily basis for the entire period for which water balance is required to be simulated. The final method of incorporating daily weather data is to import the weather data from other climatological data base sources.

6.7.2.2 Evapotranspiration Data

The required general climate data for determining evapotranspiration include the growing season, average annual wind speed, average quarterly relative humidities, normal mean monthly temperature, maximum leaf area index (LAI), evaporative zone depth and site latitude. Coefficients for the evapotranspiration parameters have been computed or developed from the *Climate of the States* and *Climatic Atlas of the United States* for 183 U.S. cities. Daily weather data requirements for evapotranspiration include precipitation, mean temperature and total global solar radiation. For this analysis, the HELP model's daily evapotranspiration has been determined stochastically based on daily weather coefficients and algorithms for Green Bay, Wisconsin. Green Bay, is the nearest city to Crandon for which the HELP model is able to generate synthetic daily evapotranspiration values.

6.7.2.3 Synthetic Precipitation Data Generation

The daily precipitation data for determining the water balance of the TMA has been synthetically generated based on the algorithms for Green Bay, Wisconsin. The synthetic generation mode was selected for determining the daily precipitation values because of the model's ability to generate from one to 100 years of daily values. Normal mean monthly precipitation values for the project site (instead of the precipitation data available in the program) are required to be input if the site is located more than a few miles from the city selected for the synthetic precipitation data generation. The TMA site in Crandon is located approximately 85 miles northwest of the city of Green Bay for which the program provides synthetic data generation algorithm. The North Pelican Lake (NPL) weather station which is the closest to the site is located approximately 15 miles northwest of the project area. The Wisconsin State Climatological Office, Madison, Wisconsin, provided the normal mean monthly precipitation values for the NPL weather station. The NPL normal mean monthly precipitation values have been determined for the thirty year average for the years 1961 to 1990. The NPL normal mean monthly precipitation values are provided in Table 6.7-1.

6.7.2.4 Synthetic Temperature Data Generation

The daily temperature data for determining the water balance of the TMA tailings cells has been synthetically generated based on the algorithms for Green Bay, Wisconsin. The synthetic generation was selected as the mode of determining the daily temperature values because of the model's ability to determine from one to 100 years of daily values. The user's guide for the HELP Model Version 3 advises the user to enter normal mean monthly temperature values if the project site is located more than 100 miles from the city selected for the synthetic temperature data generation or if the difference in elevation between the project site and the selected city is more than 500 feet. The selected city is Green Bay, Wisconsin, located approximately 85 miles southeast of Crandon, Wisconsin. However, Green Bay, Wisconsin is located at an elevation of approximately 600 feet above mean sea level (MSL) whereas, the project site is located at an elevation of approximately 1,600 feet above MSL. Therefore normal mean monthly temperature values for the closest reporting weather station at North Pelican Lake, Wisconsin was used. The North Pelican Lake (NPL) weather station is located approximately 15 miles northwest of the project area. The Wisconsin State Climatological Office, Madison, Wisconsin, provided the normal mean monthly temperature values for the NPL weather station. The NPL normal mean monthly temperature values have been determined for the thirty-year average for the years 1961 to 1990. The NPL normal mean monthly temperature values are provided in Table 6.7-1.

6.7.2.5 Synthetic Solar Radiation Data Generation

The daily solar radiation data for determining the water balance of the TMA tailings cells have been synthetically generated based on the algorithms for Green Bay, Wisconsin. The synthetic generation was selected as the mode of determining the solar radiation values because of the model's ability to determine from one to 100 years of daily values. The user's guide for the HELP Model Version 3 advises the user to enter the latitude if the project site is more than 50 miles north or south of the selected city. The project site is approximately 85 miles northwest of Green Bay, Wisconsin. Green Bay, Wisconsin is located at a latitude of approximately 44.29°N. The latitude of the TMA tailings cells are at approximately 45.29°N.

Table 6.7-1

Normal Mean Monthly Weather Data for North Pelican Lake (NPL) Weather Station

Month	Precipitation (in.)	Temperature (°F)	Comments
January	1.08	10.2	Crandon site is located at a latitude 45.29°N.
February	0.82	14.2	
March	1.62	26.0	
April	2.35	40.4	
May	3.18	53.2	
June	3.58	61.5	
July	3.73	66.1	
August	4.14	63.3	
September	3.99	54.9	
October	2.46	44.7	
November	1.96	30.5	
December	1.45	15.5	

Note: The data represents the thirty-year average for the years 1961 to 1990. The data was provided by Wisconsin State Climatological Offices, Madison, Wisconsin.

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6.7.3 Soil Data Requirements

HELP model Version 3.01 allows the user to enter soil/material data characteristics necessary for determining the water balance by one of three modes. The modes of soil/material characteristics consist of using the default soil/material textures option, the user-defined soil texture option or a manual option.

6.7.3.1 Data Used For This Project

The data requirements of a soil layer are porosity, field capacity, wilting point, initial soil water content, and saturated hydraulic conductivity values. HELP Version 3.01 has a built-in library of 42 default soil/material types. Since the default data does not include mine tailings as one of the materials, the mode used for this analysis was a combination of the default soil/material types option and the user-defined soil/material option. The properties of materials used are discussed in greater detail in Section 6.7.4.2.

6.7.3.2 Tailings Properties

The properties for the tailing layers have been based on laboratory tests data, literature data or engineering judgement. The tailings properties (porosity, field capacity, wilting point, and saturated hydraulic conductivity) were estimated taking into account the depth at which each layer is located.

6.7.3.2.1 Porosity, Field Capacity and Wilting Point

Laboratory tests conducted on the tailings samples (Appendix B) provide the specific gravity of the tailings materials to be 3.32 for the fine tailings and 3.52 for the whole tailings. For design purposes, a specific gravity of 3.36 was used to account for approximately 20 percent whole tailings which will be placed with the fine tailings in the TMA. Based on this specific gravity a relationship between the tailings dry density and porosity was established as shown in Figure 6.7-1. Appendix B also provided field capacity and wilting point for four samples at various dry densities (see Table 6.7-2). Based on these data, variation of field capacity and wilting point as functions of dry density of the tailings was plotted on Figure 6.7-1.

Table 6.7-2

Summary of Moisture Retention Test Results

Sample	Dry Density (pcf)	Field Capacity		Wilting Point	
		Percent Volume	Percent Dry Mass	Percent Volume	Percent Dry Mass
#1	61.0	43.5	44.5	27.16	27.8
#1 (R)	67.5	46.5	43.0	29.1	26.9
#2	69.9	47.4	42.3	28.1	25.1
#2 (R)	75.3	49.2	40.8	30.5	25.3

Source: Memorandum, Laboratory Test Results for Crandon Tailings, January 25, 1995, SRK Project No. 71501.

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Consolidation test data (Appendix B) was used to develop a relationship between the depth of tailings and resultant dry density as shown on Figure 6.7-2. Thus, once the depth of tailings is known (to the midpoint of tailings layer) the dry density can be estimated from Figure 6.7-2 and the corresponding porosity, field capacity, and wilting point values can be obtained from Figure 6.7-2. These values are further described in Section 6.7.4.2.

6.7.3.2.2 Saturated Hydraulic Conductivity

The saturated hydraulic conductivity (K_s) is used to describe flow though porous media where the void spaces are filled with a wetting fluid such as water. This value for tailings was estimated

from test data on the tailings (Appendix B). The recommended value of 1.0×10^{-6} cm/sec was assigned for the lowest layer of tailings to account for maximum consolidation. Higher values of hydraulic conductivity were assigned to the upper layers of tailings based on engineering judgment.

6.7.3.2.3 Unsaturated Hydraulic Conductivity and Wilting Point

Unsaturated hydraulic conductivity (K_u) is used to describe flow through a layer with void spaces that are filled with both wetting and non-wetting fluid such as water and air. The HELP model Version 3.01 computes the K_u based on the residual volumetric water content and pore-size distribution index constants of the Brooks-Corey equation. The Brooks-Corey equation relates volumetric water content to the soil matrix potential of capillary pressures and adsorptive forces.

The HELP model solves the Brooks-Corey equation simultaneously for different capillary pressures of field capacity and wilting point to determine the bubbling pressure and pore-size distribution index of the volumetric moisture content. Field capacity is the volumetric moisture content at a suction pressure of 0.33 bar. Wilting point is the volumetric moisture content at a suction pressure of 15 bar. With the porosity, field capacity and wilting point known, the Brooks-Corey equation is solved for the bubbling pressure and pore-size distribution index. In the HELP model this process is repeated for each soil layer to obtain the parameter for computing moisture retention and unsaturated drainage which is based on K_u.

6.7.4 Design Data

HELP models have been run on the two primary liner configurations of the TMA tailings cell: sideslopes and base liner configurations. Submodels of the primary HELP models to represent the operations and post-closure scenarios of the TMA tailings cell were also used in the water balance analyses. The submodels represent the required scenarios of water balance for the TMA tailings cells at different time periods.

6.7.4.1 Primary Models

Two primary models were used for water balance determination of the selected TMA design. These two primary HELP model designs are hence referred to as Sideslope and Base HELP Models. Both the Sideslope and Base HELP Models have the same final cover configuration. The Sideslope HELP Model bottom liner layer is sloped at 33 percent. The Base HELP Model has a different bottom liner configuration which is sloped at two percent from a high point in the center of each cell.

6.7.4.2 Submodels

The Sideslope and Base HELP Models have been further defined to better represent the primary configurations over time. The Sideslope HELP Model has been subdivided into several models to represent the filling, closure, and post-closure scenarios. The Base HELP Model has also been divided into submodels representing the same time periods to more accurately represent the water balance at different time periods. The primary Sideslope and Base HELP Models have been divided into the following submodels:

- Stages I, III, V, and VII (first stage of TMA cell filling, referred to as first stage);

- Stages II, IV, VI, and VIII (second stage of TMA cell filling) (referred to as second stages);

- Post-Closure Monitoring Period;

- Post-Closure Period With Lateral Drainage on the Sideslope and No Lateral Drainage on the Base (represents the case when leachate removal is discontinued); and

- Post-Closure Period With Lateral Drainage on the Sideslope and No Lateral Drainage on Base, and only the clay portion of the liner (represents the case when the geomembrane may no longer be functional after a very long period).

The above scenarios represent the TMA defined by construction, operation, and post-closure conditions.

The Sideslope and Base HELP Models for the first stages have been conducted based on the following generalized design specifications and soil characteristic input parameters:

Sideslope HELP Model: First Stages

- no cover;
- no surface water runoff;
- vertical profile of 22.5 feet of tailings (half of maximum tailings depth);
- 150-foot length of base drainage at 3H:1V slope;
- lateral drainage above the liner;
- composite liner; and
- soil and tailing waste layer data as provided in Table 6.7-3.

Base HELP Model: First Stages

- no cover;
- no surface water runoff;
- vertical profile of 45 feet of tailings;
- 940-foot length of base drainage at two percent slope;
- lateral drainage above the liner;
- composite liner; and
- soil and tailing waste layer data as provided in Table 6.7-4.

Table 6.7-3

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Sideslope - First Stages (I, II, V, VII)

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Tailings	30	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5803
2	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.5529
3	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.5445
4	Till/ Lateral Drainage	18	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
5	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
6	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270

(1) FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Note: Sideslopes modeled at one-half vertical depth of tailings above the base.

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Checked by: MDF

Table 6.7-4

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Base - First Stages (I, III, V, VII)

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Tailings	60	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5803
2	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.5529
3	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.5445
4	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.5385
5	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.5338
6	Till	24	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
7	Gravel/ Lateral Drainage	24	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0320
8	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
9	Clay	12	16	Barrier Layer		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270

(1) FML Flexible Membrane Liner - Thermoplastic Geomembrane.

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Checked by: MDF

The Sideslope and Base HELP Models for second stages have been conducted based on the following generalized design specifications and soil characteristic input parameters:

Sideslope HELP Model: Second Stages

- no cover;
- no surface water runoff;
- vertical profile of 45 feet of tailings (half of maximum tailings depth);
- 300-foot length of base drainage at 3H:1V slope;
- lateral drainage above the liner;
- composite liner; and
- soil and tailing waste layer data as provided in Table 6.7-5.

Base HELP Model: Second Stages

- no cover;
- no surface water runoff;
- vertical profile of 90 feet of tailings;
- 940-foot length of base drainage at two percent slope;
- lateral drainage above the base;
- composite liner; and
- soil and tailing waste layer data as provided in Table 6.7-6.

The Sideslope and Base HELP Models for Post-closure Monitoring Period have been conducted based on the following generalized design specifications and soil characteristic input parameters:

Sideslope HELP Model: Post-closure Monitoring Period

- final cover soils;
- 950-foot length of cover drainage at 2 percent slope;
- composite hydraulic barrier;
- vertical profile of 45 feet of tailings (half of maximum tailings depth);
- 300-foot length of base drainage at 3H:1V percent slope;
- lateral drainage above the liner;
- composite liner; and
- soil and tailing waste layer data as provided in Table 6.7-7.

Base HELP Model: Post-closure Monitoring Period

- final cover soils;
- 950-foot length of cover drainage at 2 percent slope;
- composite hydraulic barrier;
- vertical profile of 90 feet of tailings;
- 940-foot length of base drainage at 2 percent slope ;
- lateral drainage above the liner;
- composite liner; and
- soil and tailing waste layer data as provided in Table 6.7-8.

Table 6.7-5

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Sideslope - Second Stages (II, IV, VI, VII)

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Tailings	60	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5803
2	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.5529
3	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.5445
4	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.5385
5	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.5338
6	Till/ Lateral Drainage	18	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
7	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
8	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270

(1) FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Note: Sideslopes modeled at one-half vertical depth of tailings above the base.

Prepared by: JBK
Checked by: MDF

Table 6.7-6

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Base - Second Stages (II, IV, VI, VIII)

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Tailings	120	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5803
2	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.5529
3	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.5445
4	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.5385
5	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.5338
6	Tailings	120	0	---	---	0.5297	0.4930	0.3230	2.33×10^{-6}	0.5297
7	Tailings	120	0	---	---	0.5259	0.4915	0.3215	2.0×10^{-6}	0.5259
8	Tailings	120	0	---	---	0.5226	0.4905	0.3205	1.5×10^{-6}	0.5226
9	Tailings	120	0	---	---	0.5195	0.4900	0.3200	1.0×10^{-6}	0.5195
10	Till	24	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
11	Gravel/ Lateral Drainage	24	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0320
12	FML(1)	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
13	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270

(1) FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Prepared by: JBK
Checked by: MDF

Table 6.7-7

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Sideslope - Post-Closure Monitoring Period

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Topsoil	6	8	L	ML	0.4630	0.2320	0.1160	3.7×10^{-4}	0.3231
2	Till/ Rooting Zone	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2443
3	Gravel/ Lateral Drainage	12	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0320
4	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
5	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270
6	Till/ Grading Layer	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
7	Tailings	60	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5803
8	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.5529
9	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.5445
10	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.5385
11	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.5338
12	Till/Lateral Drainage	18	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
13	FML[1]	0.06	35	---	---	---	---	---	2×10^{-13}	---
14	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1×10^{-7}	0.4270

[1] FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Note: Sideslopes modeled at one-half vertical depth of tailings above the base.

Prepared by: JBK
Checked by: MDF

Table 6.7-8

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Base - Post-closure Monitoring Period

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Topsoil	6	8	L	ML	0.4630	0.2320	0.1160	3.7×10^{-4}	0.3231
2	Till/ Rooting Zone	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2443
3	Gravel/ Lateral Drainage	12	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0320
4	FML(1)	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
5	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270
6	Till/Grading Layer	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2240
7	Tailings	120	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5803
8	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.5529
9	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.5445
10	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.5385
11	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.5338
12	Tailings	120	0	---	---	0.5297	0.4930	0.3230	2.33×10^{-6}	0.5297
13	Tailings	120	0	---	---	0.5259	0.4915	0.3215	2.0×10^{-6}	0.5259
14	Tailings	120	0	---	---	0.5226	0.4905	0.3205	1.5×10^{-6}	0.5226
15	Tailings	120	0	---	---	0.5195	0.4900	0.3200	1.0×10^{-6}	0.5195
16	Till	24	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
17	Gravel/ Lateral Drainage	24	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0320
18	FML(1)	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
19	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270

(1) FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Prepared by: JBK
Checked by: MDF

The Sideslope and Base HELP Models for Post-closure period with lateral drainage on the sideslope and no lateral drainage on the base have been conducted based on the following generalized design specifications and soil characteristic input parameters:

Sideslope HELP Model: Post-closure Period With Lateral Drainage

- final cover soils;
- 950-foot length of cover drainage at two percent slope;
- composite hydraulic barrier;
- vertical profile of 45 feet of tailings (half of the maximum tailings depth);
- 300-foot length of base drainage at 3H:1V percent slope;
- lateral drainage above the liner;
- composite liner ; and
- soil and tailing waste layer data as provided in Table 6.7-9.

Base HELP Model: Post-closure Period With No Lateral Drainage

- final cover soils;
- 950-foot length of cover drainage at two percent slope;
- composite hydraulic barrier;
- vertical profile of 90 feet of tailings;
- composite liner; and
- soil and tailing waste layer data as provided in Table 6.7-10.

The Sideslope and Base HELP Models for Post-closure Period with lateral drainage on the sideslope and no lateral drainage and no geomembrane layer in the base have been conducted to represent the worst case assumption of a complete geomembrane failure. The following generalized design specifications and soil characteristic input parameters represent this worst case:

Sideslope HELP Model: Post-closure Period With Lateral Drainage and No Composite Liner

- final cover soils;
- 950-foot length of cover drainage at two percent slope;
- composite hydraulic barrier;
- vertical profile of 45 feet of tailings (half of the maximum tailings depth);
- 300-foot length of base drainage at 3H:1V slope;
- lateral drainage above the liner;
- liner consisting of soil component only; and
- soil and tailing waste layer data as provided in Table 6.7-11.

Base HELP Model: Post-closure Period With No Lateral Drainage and No Composite Liner

- final cover soils;
- 950-foot length of cover drainage at two percent slope;
- composite hydraulic barrier;
- vertical profile of 90 feet of tailings;
- liner consisting of soil component only; and
- soil and tailing waste layer data as provided in Table 6.7-12.

Table 6.7-9

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Sideslope - Post-closure Period With Lateral Drainage

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Topsoil	6	8	L	ML	0.4630	0.2320	0.1160	3.7×10^{-4}	0.3662
2	Till/ Rooting Zone	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2547
3	Gravel/ Lateral Drainage	12	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0321
4	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
5	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270
6	Till/ Grading Layer	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
7	Tailings	60	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5100
8	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.4880
9	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.4802
10	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.4745
11	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.4699
12	Till/Lateral Drainage	18	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2489
13	FML[1]	0.06	35	---	---	---	---	---	2×10^{-13}	---
14	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1×10^{-7}	0.4270

[1] FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Note: Sideslopes modeled at one-half vertical depth of tailings above the base.

Prepared by: JBK
Checked by: MDF

Table 6.7-10

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Base - Post-closure Monitoring Period With No Lateral Drainage

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Topsoil	6	8	L	ML	0.4630	0.2320	0.1160	3.7×10^{-4}	0.3662
2	Till/Rooting Zone	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2545
3	Gravel/ Lateral Drainage	12	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0341
4	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
5	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270
6	Till/Grading Layer	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
7	Tailings	120	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5055
8	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.4817
9	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.4739
10	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.4682
11	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.4636
12	Tailings	120	0	---	---	0.5297	0.4930	0.3230	2.33×10^{-6}	0.4594
13	Tailings	120	0	---	---	0.5259	0.4915	0.3215	2.0×10^{-6}	0.4559
14	Tailings	120	0	---	---	0.5226	0.4905	0.3205	1.5×10^{-6}	0.4530
15	Tailings	120	0	---	---	0.5195	0.4900	0.3200	1.0×10^{-6}	0.4503
16	Till	24	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
17	Gravel/ Lateral Drainage	24	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0320
18	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
19	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270

[1] FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Prepared by: JBK
Checked by: MDF

Table 6.7-11

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Sideslope - Post-closure Period With Lateral Drainage and No Composite Liner

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Topsoil	6	8	L	ML	0.4630	0.2320	0.1160	3.7×10^{-4}	0.3231
2	Till/ Rooting Zone	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2431
3	Gravel/ Lateral Drainage	12	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0340
4	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
5	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270
6	Till/ Grading Layer	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
7	Tailings	60	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5056
8	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.4818
9	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.4700
10	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.4683
11	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.4637
12	Till/Lateral Drainage	18	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2441
13	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1×10^{-7}	0.4270

[1] FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Note: Sideslopes modeled at one-half vertical depth of tailings above the base.

Prepared by: JBK
Checked by: MDF

Table 6.7-12

Soil, Waste and Geosynthetic Characteristics Used for Water Balance Model Base - Post-closure Monitoring Period With No Lateral Drainage and No Composite Liner

Layer #	General Description	Thickness (Inches)	Classification			Total Porosity (vol/vol)	Field Capacity (vol/vol)	Wilting Point (vol/vol)	Saturated Hydraulic Conductivity (cm/sec)	Initial Soil Water Content (vol/vol)
			HELP	USDA	USCS					
1	Topsoil	6	8	L	ML	0.4630	0.2320	0.1160	3.7×10^{-4}	0.3233
2	Till/ Rooting Zone	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2432
3	Gravel/ Lateral Drainage	12	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0398
4	FML[1]	0.06	35	---	---	---	---	---	2.0×10^{-13}	---
5	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270
6	Till/ Grading Layer	36	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
7	Tailings	120	0	---	---	0.5803	0.5100	0.3400	4.0×10^{-6}	0.5055
8	Tailings	120	0	---	---	0.5529	0.5066	0.3366	3.66×10^{-6}	0.4817
9	Tailings	120	0	---	---	0.5445	0.5033	0.3333	3.33×10^{-6}	0.4739
10	Tailings	120	0	---	---	0.5385	0.5000	0.3300	3.0×10^{-6}	0.4682
11	Tailings	120	0	---	---	0.5338	0.4965	0.3265	2.66×10^{-6}	0.4636
12	Tailings	120	0	---	---	0.5297	0.4930	0.3230	2.33×10^{-6}	0.4594
13	Tailings	120	0	---	---	0.5259	0.4915	0.3215	2.0×10^{-6}	0.4559
14	Tailings	120	0	---	---	0.5226	0.4905	0.3205	1.5×10^{-6}	0.4530
15	Tailings	120	0	---	---	0.5195	0.4900	0.3200	1.0×10^{-6}	0.4503
16	Till	24	10	SCL	SC	0.3980	0.2440	0.1360	1.2×10^{-4}	0.2440
17	Gravel/ Lateral Drainage	24	21	Gravel		0.3970	0.0320	0.0130	3.0×10^{-1}	0.0331
18	Clay	12	16	Barrier Soil		0.4270	0.4180	0.3670	1.0×10^{-7}	0.4270

(1) FML Flexible Membrane Liner - Thermoplastic Geomembrane.

Prepared by: JBK
Checked by: MDF

From Tables 6.7-3 through 6.7-6 it will be noted that the models do not include the ponding of the slurry water. The reason for not including the ponding is that the purpose of the water balance is to determine primarily the leachate generated during the post-closure monitoring period and the percolation from the site. Since the simulation starts with the case of initial saturation of all tailing layers, the effect of the ponding (which is to keep the tailing layers saturated) will be indirectly accounted for if the percolation through the liner for the first year of simulation is assumed to be prevailing throughout the operation period of the cell. After conducting the HELP model runs in this fashion, Dr. Paul Schroeder of the USCOE, Waterways Experiment Station, who is the primary author of HELP, was contacted for his comments on this approach. He suggested that the model may be operated to mimic the presence of ponding on the surface if the precipitation values were increased and surface runoff prevented. Accordingly, the four submodels for the open conditions were rerun with the precipitation values increased by a factor of two. The values obtained for leachate generation and percolation from the site for this case are very similar to those obtained earlier and provide an interesting aspect of the simulation, that is, the percolation through the base liner is impacted more by the saturation and hence the head on the liner rather than the rainfall intensity during any year. One would expect the same for a ponded condition since the rainfall variation will be neutralized by the relatively constant pond depths on the tailings surface.

For this analysis the geomembrane included both in the final cover and the base liner was considered to have one pinhole per acre due to manufacturing defects and four holes per acre due to installation defects. The contact between the membrane and the clay component of the composite liner was taken as "good" discarding the option of "excellent" contact in order to obtain conservative (higher) values of percolation through the composite liner. Per Schroeder, et al. 1994a, all the above conditions are either readily achievable by a good CQA program or more conservative (producing larger percolation) than what can be achieved. Thus, by considering different geometry, appropriate material properties, and techniques for simulating operation scenarios, HELP model has provided reasonably conservative values of leachate quantities and percolation from the site. These results are discussed in Section 6.7.5.

6.7.5 Summary of Results

6.7.5.1 Percolation Through the Liner and Leachate Generation

A summary of the results from the water balance study using the HELP models is shown in Table 6.7-13. The results shown pertain to the different submodels of both the Sideslope HELP Model and Base HELP Model. Thus different operation scenarios such as open case, closed case during post-closure care period, closed case after the discontinuation of leachate removal and closed case after a very long time period when the geomembrane in the base liner may no longer be effective, are covered. The percolation and leachate generation are given in terms of annual averages for the duration of the simulation period.

The yearly variation of percolation through the liner during the operation period for two cases, i.e., sideslope First Stage, base First Stage are given in Table 6.7-14. The percolation through the liner for Second Stage of the sideslope and base cases are shown on Table 6.7-15. From Tables 6.7-14 and 6.7-15, it will be noticed that the percolation quantities diminish somewhat through the years of operation. This is indicative of the gradual draining of the tailings and therefore does not account for the ponding on top of the cell. As described earlier in

Table 6.7-13

Results of HELP Models Water Balance Analyses

	Average Annual Totals for the Simulation Period					Peak Daily Values			
HELP Model ID	Water Balance Simulation Period Years	Percolation Through Cover[1] (in/yr); (Percent)	Lateral Drainage/Leachate Collected[1] (in/yr); (Percent)	Percolation Through Liner[1] in/yr; (Percent)	Average Head Across Liner (in)	Percolation Through Cover (in/day)	Lateral Drainage/Leachate Collected (in/day)	Percolation through Liner (in/day)	Average Head Across Liner (in)
Sideslope - First Stage (Table 6.7-3)	10	NA	5.12507 (16.28714)	0.05541 (0.17610)	51.222	NA	0.02650	0.000820	223.626
Base - First Stage (Table 6.7-4)	10	NA	6.52576 (20.73843)	0.00055 (0.00175)	0.530	NA	0.09082	0.000008	3.196
Sideslope - Second Stage (Table 6.7-5)	10	NA	3.97372 (12.62821)	0.72523 (2.30472)	403.713	NA	0.01302	0.002780	503.922
Base - Second Stage (Table 6.7-6)	10	NA	8.28679 (26.33484)	0.00065 (0.00208)	0.627	NA	0.03454	0.000003	0.955
Sideslope - Final Cover (Table 6.7-7)	40	0.00084 (0.00273)	0.81836 (2.66068)	0.05658 (0.18395)	35.576	0.000060	0.01415	0.002827	509.266
Base - Final Cover (Table 6.7-8)	40	0.00248 (0.00806)	1.91087 (6.21269)	0.00016 (0.00053)	0.145	0.000105	0.03423	0.000003	0.946
Sideslope - Final Cover (Table 6.7-9)	100	0.00084 (0.00273)	0.03395 (0.11016)	0.00016 (0.00052)	0.137	0.000071	0.00039	0.000002	0.576
Base - No Drainage (Table 6.7-10)	100	0.00246 (0.00799)	NA	0.00033 (0.00107)	0.296	0.000105	NA	0.000002	0.579
Sideslope - No Geomembrane at Base (Table 6.7-11)	100	0.00084 (0.00273)	0.00000 (0.00000)	0.00165 (0.00536)	0.000	0.000071	0.00001	0.003405	0.013
Base - No Geomembrane at Base (Table 6.7-12)	100	0.00247 (0.00800)	NA	0.00456 (0.01479)	0.000	0.000105	NA	0.003483	0.286

[1]These values are given in inches/year as well as a percentage of precipitation, the latter within parentheses.

Notes: HELP Model ID - Soil properties are provided in respective table listed under the HELP Model ID.
NA - Not applicable.

Prepared by: JBK
Checked by: MDF

Table 6.7-14

Percolation From TMA
Based on Maximum Average Annual Percolation Through Liner
During Operations Years of First Stages

Year	Sideslope (in/yr)	Base (in/yr)
1	0.187326	0.002003
2	0.149477	0.000396
3	0.112421	0.000441
4	0.044193	0.000342
5	0.005470	0.000290
6	0.013519	0.000327
7	0.006639	0.000404
8	0.012266	0.000315
9	0.017525	0.000489
10	0.005306	0.000491

Prepared by: JBK
Checked by: MDF

Table 6.7-15

Percolation From TMA
Based on Maximum Average Annual Percolation Through Liner
During Operations Years of Second Stages

Year	Sideslope (in/yr)	Base (in/yr)
1	0.817797	0.000816
2	0.844675	0.000956
3	0.788535	0.000956
4	0.683487	0.000959
5	0.554911	0.000751
6	0.687312	0.000356
7	0.666802	0.000505
8	0.659438	0.000288
9	0.818071	0.000356
10	0.731234	0.000604

Prepared by: JBK
Checked by: MDF

Section 6.7.4.2, one way to approximate the effects of ponding is to assume that the results for the first year (the maximum value) will continue to prevail throughout the time of ponding.

A second, perhaps more acceptable method is to increase the rainfall data by a factor of two and thus create excess water on the top tailings layer (as described in Section 6.7.4.2). The results of these analyses in terms of averages through the simulation period are given in Table 6.7-16. Yearly values of percolation through the liner for Stage I and Stage II are shown on Tables 6.7-17 and 6.8-18 respectively. These results show that the percolation through the liner does not diminish with time and also that average head on the liner (Table 6.7-16) is similar to peak daily head thereby indicating that the effects of ponding is possibly simulated by the technique used, i.e., increased rainfall.

Based on the above results it has been concluded that for the period when the TMA remains open and ponding takes place on top of the tailings, the leachate generation and percolation through the liner should be taken from the water balance study using two times rainfall data (Tables 6.7-16, 6.7-17 and 6.7-18). After closure of the TMA's when there will be no ponding and draining of the tailings is a reality, and therefore results from the analyses with normal rainfall are appropriate (Table 6.7-13).

6.7.5.2 Total Percolation from the TMA

The results of HELP model studies show that the quantity of percolation varies with time due to varying operation conditions (open, closed, geomembrane not effective, etc.). Also, the quantity of percolation varies due to the change in geometry (side slope, base, tailing thickness, etc.). Therefore, in order to estimate the variation of percolation through the liner with time, both the operation conditions and the proportioning of the area of percolation between side slopes and base need to be considered.

Assuming the first stage to be operative five years and the second stage for another five before a TMA cell is closed and further assuming that 57 percent of the TMA is sideslope and 43 percent is base, the average percolation for the first five years will be 0.23 in/yr (1). For the years five to ten the percolation will be 0.66 in/yr (2). For these estimates, the maximum percolation for the respective five year periods have been used.

For the remaining periods the estimated quantities of percolation are as shown on Table 6.7-19. It may be noted that this percolation is estimated as a typical case for TMA 1. However, since the assumptions are valid for other TMAs, the percolation rates are not expected to vary for practical purposes.

(1) $$(0.401 \times 0.57) + (0.002 \times 0.43) = 0.23$$

(2) $$(1.159 \times 0.57) + (0.000959 \times 0.43) = 0.66$$

Table 6.7-16

HELP Model Results: Simulation of Ponding

HELP Model ID	Water Balance Duration Years	Average Annual Totals for Model Duration: Lateral Drainage/Leachate Collected (in/yr)	Average Annual Totals for Model Duration: Percolation Through Liner (in/yr)	Average Annual Totals for Model Duration: Average Head Across Liner (in)	Peak Daily Values: Lateral Drainage/Leachate Collected (in/day)	Peak Daily Values: Percolation Through Liner (in/day)	Peak Daily Values: Average Head Across Liner (in)
Sideslope First Stages (Table 6.7-3)	10	7.95554	0.38090	262.805	0.02356	0.001182	288.000
Base - First Stages (Table 6.7-4)	10	24.18747	0.00205	2.183	0.09100	0.000008	3.208
Sideslope - Second Stages (Table 6.7-5)	10	5.09729	1.13119	537.443	0.01444	0.003269	558.000
Base - Second Stages (Table 6.7-6)	10	12.23701	0.00094	0.926	0.03401	0.000003	0.940

HELP Model ID - Soil properties are provided in respective table listed under the HELP Model ID.

Prepared by: JBK
Checked by: MDF

Table 6.7-17

Percolation From TMA Based on Twice Precipitation Values During Operations Years of First Stages

Year	Sideslope (in/yr)	Base (in/yr)
1	0.296209	0.002108
2	0.401257	0.001499
3	0.394566	0.002211
4	0.391485	0.002127
5	0.383456	0.002100
6	0.388407	0.001990
7	0.387811	0.002158
8	0.382444	0.001945
9	0.395112	0.002166
10	0.388289	0.002145

Prepared by: JBK
Checked by: MDF

Table 6.7-18

Percolation From TMA Based on Twice Precipitation Values During Operations Years of Second Stages

Year	Sideslope (in/yr)	Base (in/yr)
1	0.968434	0.000816
2	1.159335	0.000956
3	1.152525	0.000956
4	1.152098	0.000959
5	1.141516	0.000956
6	1.148169	0.000956
7	1.144952	0.000956
8	1.145495	0.000959
9	1.153419	0.000956
10	1.145948	0.000956

Prepared by: JBK
Checked by: MDF

Table 6.7-19

Percolation Through TMA 60 mil Thermoplastic Geomembrane Bottom Liner

Year from Placement of Final Cover for TMA	Sideslope (in/yr)	Sideslope[1] Proportional (in/yr)	Base (in/yr)	Base[1] Proportional (in/yr)	Combined Sideslope & Base Proportional (in/yr)
1	0.825931	0.467642	0.000816	0.000344	0.467996
2	0.627056	0.355039	0.000956	0.000415	0.355454
3	0.404497	0.229026	0.000956	0.000415	0.229441
4	0.245185	0.138824	0.000718	0.000311	0.139135
5	0.103574	0.058644	0.000243	0.000105	0.008749
6	0.010155	0.005750	0.000211	0.000092	0.005842
7	0.002831	0.001603	0.000190	0.000082	0.001685
8-15	0.002498	0.001414	0.000175	0.000076	0.001490
16-25	0.001702	0.000964	0.000109	0.000047	0.001011
26-35	0.001112	0.000630	0.000074	0.000032	0.000662
36-40	0.000733	0.000415	0.000003	0.000001	0.000416
41-50	0.000593	0.000336	0.000027	0.000012	0.000348
51-60	0.000396	0.000224	0.000026	0.000011	0.000235
61-70	0.000274	0.000155	0.000030	0.000013	0.000168
71-80	0.000191	0.000108	0.000042	0.000018	0.000126
81-90	0.000132	0.000075	0.000049	0.000021	0.000096
91-115	0.000092	0.000052	0.000072	0.000031	0.000083
116-140	0.000039	0.000022	0.000096	0.000042	0.000064
141	0.024768	0.014024	0.032713	0.014191	0.028215
142-165	-	-	-	-	-
166-175	-	-	0.000546	0.0010237	0.000237
176-185	-	-	0.000864	0.000375	0.000375
186-195	-	-	0.000970	0.0030421	0.000421
196-205	-	-	0.000653	0.000283	0.000283

Table 6.7-19 *(Continued)*

Year from Placement of Final Cover for TMA	Sideslope (in/yr)	Sideslope[1] Proportional (in/yr)	Base (in/yr)	Base[1] Proportional (in/yr)	Combined Sideslope & Base Proportional (in/yr)
206-215	-	-	0.000572	0.000248	0.000248
216-225	-	-	0.000813	0.000353	0.000353
226-235	-	-	0.000778	0.000337	0.000337
236-240	-	-	0.000479	0.000208	0.000208

[1]Sideslope Surface Area of TMA 1-4 is 4,843,780 ft^2 (56.62 percent of Total Surface Area).
Base Surface Area of TMA 1-4 is 3,710,950 ft^2 (43.38 percent of Total Surface Area).
- Not Calculated.

Prepared by: JBK
Checked by: MDF

6.7.5.3 Leachate Generated

Tables 6.7-20 and 6.7-21 show leachate quantities for first and second stage filling of a TMA cell. The results are shown for the case where ponding is simulated (i.e., two times precipitation). After closure of the unit, the leachate quantities are as shown on Table 6.7-13.

Table 6.7-20

Lateral Drainage/Leachate Collected[1] First Stage

Year	Sideslope (in/yr)	Base (in/yr)
1	7.0826	24.9838
2	8.1704	18.4111
3	8.0969	26.0176
4	8.0732	25.0234
5	7.9752	24.5570
6	8.029	23.6666
7	8.0229	25.3258
8	7.9744	23.0697
9	8.1028	25.6709
10	8.0281	25.1485

(1) Table based on results from HELP model runs with twice normal mean monthly precipitation values.

Notes:

Proportionalizing Year 3 Leachate to Sideslope and Base Surface Areas of TMA 1-4, the maximum leachate collected is 15.8709 in/yr for first stage conditions (for year 3 when maximum occurs).

Prepared by: JBK
Checked by: MDF

Table 6.7-21

Lateral Drainage/Leachate Collected[1]
Second Stages

Year	Sideslope (in/yr)	Base (in/yr)
1	4.6617	10.5703
2	5.1740	12.4146
3	5.1542	12.4146
4	5.1576	12.4487
5	5.1224	12.4146
6	5.1407	12.4146
7	5.1326	12.4146
8	5.1378	12.4487
9	5.1566	12.4146
10	5.1352	12.4146

[1]Table based on results from HELP model runs with twice normal mean monthly precipitation values.

Notes:

- Proportionalizing Year 4 Leachate to Sideslope and Base Surface Areas for TMA 1-4, the Leachate Collected is 8.32048 in/yr for Stage II Conditions.

- Sideslope Surface Area of TMA 1-4 is 4,843,780 ft^2 (56.62 percent of Total Surface Area)

- Base Surface Area of TMA 1-4 is 3,710,950 ft^2 (43.38 percent of Total Surface Area)

Prepared by: JBK
Checked by: MDF

6.8 Compatibility of the Waste and the Liner System NR 182.11(1)(p)

6.8.1 General - Geomembrane

NR 182.11(1)(p) requires that "a waste site shall be located, designed, constructed and operated so that any liner system or naturally occurring soil barrier is compatible with all disposed or stored mining waste." It is stated in Section 6.3.4 that the geomembrane selection will be made as part of final design and during the preparation of construction plans and specifications. It is CMC's desire to select the best material available at that time. Various thermoplastic geomembranes commercially available today are suitable for the liner of the TMA. These include High Density Polyethylene (HDPE), Lower Density Polyethylene (MDPE, LLDPE), Polypropylene (PP), Polyvinyl Chloride (PVC), Ethylene Alloys (XR-5). All the above materials have proven resistance against the type of chemicals contained in the leachate anticipated to be produced at the Crandon TMA. Therefore compatibility as required by NR 182.11(1)(P) Wis. Adm. Code will be achieved by the use of any of the thermoplastics identified.

6.9 Surface Water Control System - Tailings Management Area NR 182.11(1)(b) and (e)

This discussion pertains to surface water management during operations and after closure of the TMA site. Stormwater from the top of the active TMA cell berm will be directed into the TMA cell. Stormwater from the external slopes and other non-contact areas will be managed with the surface water control system. Stormwater runoff for the various phases of operation and disturbed areas will be collected and routed through the required stormwater management facilities. Areas disturbed also include temporary borrow activities and material stockpile areas as well as the access road. Runoff from these areas will all be properly managed before being discharged.

During operation activities in any area, runoff from the disturbed landscape will be routed to the runoff basin planned for that area. After completion of the final reclamation for each area, runoff will continue to be routed to the runoff basins until vegetation is established, at which time the runoff basins will be reclaimed and drainage will follow natural patterns.

The site drainage system has three purposes:

- to collect and treat all stormwater runoff with potential for pollutant contact;
- to collect, route and discharge stormwater runoff while minimizing ponding and subsequent infiltration into the tailings management area; and
- to control erosion and sediment transport.

The components of the surface water management system include drainage diversion berms, downslope swales, perimeter drainage ditches, culverts, and runoff basins. The surface water management system is illustrated on Drawing No. 31, Details 1-31 through 4-31.

To control surface runoff over the final cover of the reclamation site, surface water diversion berms will be provided as needed to direct flows to match existing condition hydrology of the various wetlands. A perimeter drainage ditch conveys the surface water runoff to runoff basins,

provided at various points around the facility. The runoff basins will be constructed prior to construction and maintained until after reclamation. Design calculations for all components of the surface water management system are provided in Appendix K of this report. The erosion control plan given in Section 4.10 of the MPA describes the erosion control features of the plan during various operational phases.

All drainageways and swales are designed to safely accommodate the runoff from the 100-year, 24-hour precipitation event without scouring or eroding the bottom and sides of the receiving channels. Manning's equation and Soil Conservation Service (SCS) grassed swale design procedures are used to develop ditch dimensions adequate for the design discharge. Factors included in this analysis include channel configuration, type of soil, channel vegetation, peak flow, longitudinal slope, and channel runoff velocity. Velocities are checked for erosion and sediment deposition problems. Various erosion control materials are specified for the swales, berms, and channels to help stabilize the bare ground surface until vegetation is established. Culverts are designed to pass the peak runoff with no road overtop or erosion at the inlet or outlet points.

To minimize the risk of sediment transport from the site, treatment facilities (runoff basins) will receive all runoff from the site. These facilities are designed with adequate capacity to settle out at least silt-sized (10 micron) particles during storm events of up to a 100-year, 24-hour frequency for the TMA. Typical sections of runoff basins located around the site are shown on Drawing No. 31, Detail 4-31. The sedimentation basin discharges into uplands that border the site, ultimately flowing to various wetland areas. In addition to sediment control, the runoff basins provide runoff detention which will reduce peak discharges from the site thereby minimizing any impact to the surrounding areas. The storage and outlet capabilities of the runoff basins are such that peak runoff from the 100-year, 24-hour precipitation event is stored without reaching the emergency spillway elevation of the basin, thus meeting the requirements of NR 182.11. Appendix K contains the TR-55 hydrology and POND2 (Haestad Methods routing program) documentation showing the adequacy of the design. Table 6.9-1 provides a comparison of actual storage area with required storage area for the runoff basins at the site, along with other more specific routing information.

Based on this comparison, the actual storage areas for the runoff basins exceed the required storage areas, and adequate sedimentation will be provided. Also, these storage volumes are conservative in that additional storage volume is available for sediment storage which is considered dead storage for these routing calculations.

The runoff basins are equipped with a bypass outlet (emergency spillway). As documented in Appendix K, storage is available to handle the peak flow of runoff from the 100-year, 24-hour precipitation event without damage to the treatment facility or surrounding structures. Riprap or equivalent erosion protection is provided on the discharge side of the spillway to minimize damage to the receiving areas.

All discharge structures are designed to dissipate the flow discharge with perforated header pipe system. This will spread the outflow over a wide area to better match existing conditions.

Table 6.9-1

Runoff Basin Design Details for 100-year 24-Hour Event

Basin No.	Peak Inflow (cfs)	Peak Discharge (cfs)	Maximum Water Elevation (ft)	Storage Volume (ac-ft)	Required[1] Surface Area (sf)	Actual Water Surface Area (sf)[2]
Runoff Basin 8	52	9.8	1,628.7	3.14	19,870	42,690
Runoff Basin 9	45	7.8	1,694.3	2.47	15,915	82,764
Runoff Basin 10	42	9.2	1,650.9	2.12	18,720	35,284
Runoff Basin 11	60	10.5	1,656.7	3.60	21,400	39,204
Runoff Basin 12	31	8.5	1,701.0	1.42	17,300	32,234

[1] Based on silt-sized (10 micron) particle sedimentation (2,030 sq ft per cfs out).
[2] Water surface area at maximum water level in the pond for the 100-year, 24-hour precipitation event.

Notes:
- cfs - cubic feet per second
- ac-ft - acre feet
- sf - square feet
- ft - feet

Prepared by: MDL
Checked by: SRB

The surface water control structures are an integral part of the design and function of the final cover system. As such, the surface water control system will be operated and maintained during the entire design period of the facility.

Runoff from undisturbed areas is diverted away from the site drainage facilities as the contours from the undisturbed areas flow away from the perimeter ditching. Undisturbed runoff is not anticipated to become commingled with runoff from disturbed areas.

6.10 Traffic Routing

Traffic to and from the TMA is limited to travel on the access road between the plant site and the TMA (refer to Drawing No. 2). Details concerning the location and design of the access road are provided in the MPA. The vehicular traffic during operations will in general include the following trips from the plant to the TMA and a return trip:

- two to three trips per shift of a one-half ton pickup truck for the purpose of inspecting and operating the TMA;
- approximately five trips per day during the initial three-year construction period for 20-ton truck hauling Type II waste rock during the construction and operating period;
- approximately 35 truck trips per day hauling clay liner and capping soils from off-site (refer to Table 6.3-2 for details) for construction of base liner or cap;
- an unknown number of unscheduled trips for any maintenance to the process water pond pipeline and the TMA required during operations; and
- an unknown number of unscheduled truck trips related to construction of the TMA.

During the operating period, the amount of traffic between the TMA and the plant site will be principally mill operations and environmental personnel who are responsible for inspecting the facility. Construction of the tailings cells will result in periods of increased traffic.

6.11 TMA Security and Screening, NR 182.11(1)(c) and NR 132.17(1)(a) to (d)

6.11.1 TMA Security

NR 182.11(1)(c) requires that "access to the facility, particularly the active area, shall be restricted through the use of fencing, natural barriers or other methods approved by the Department."

Access to the TMA and associated facilities will be controlled by erecting a fence and other devices around active areas which will restrict access by unauthorized personnel.

6.11.2 TMA Screening, NR 132.17(1)(a) Through (d)

NR 132.17(1)(a) through (d) contain criteria related to the visual and aesthetic qualities of the proposed TMA and appurtenant features. The aesthetic are described in Section 3.13 of the EIR. The impacts of the preferred alternative for the TMA and appurtenant features are discussed in Section 4 of the EIR.

6.12 Site Development and Phasing, NR 182.08(2)(e)

6.12.1 General

The site development and phasing described in this section is based on the expected plant production used in sizing the facility as per Section 4 of this report. Mine production rates may change during the mine life depending on changes in ore grade, conditions encountered during mining, and other factors.

The earthwork balance provided is based on mine production rates. Any changes in the site development plans and phasing would impact the earthwork required for a particular phase.

This section provides the information required by NR 182.08(2)(e) including the following:

- description of site development;
- description of site phasing;
- waste balance and projected conditions;
- earthwork balance and stockpile sizes;
- processing of on-site soils for the TMA; and
- discussion of soil materials purchased from off-site.

6.12.2 Description of Site Development

The TMA consists of four cells designated TMA 1, TMA 2, TMA 3 and TMA 4. TMA 1 and 2 are designed to contain the mine wastes resulting from the zinc ore production which is estimated in Table 4.1-2 at 11.74 million cubic yards. This figure includes tailings, Type II waste rock, wastewater treatment solids , plus a 15 percent contingency. TMA 3 and 4 are designed to contain the mine wastes resulting from copper ore production which is estimated in Table 4.1-2 at 8.72 million cubic yards which includes all the mine wastes listed above plus a 15 percent contingency.

Each TMA cell is operated and constructed in two stages. For example, TMA 1 has Stage I and Stage II, TMA 2, Stage III and Stage IV, etc. More detailed information concerning the anticipated capacity of each cell and stage is provided in Section 4.1 and Table 4.1-2 of this report.

The site development scenario which follows portrays, to the extent possible, the development of the TMA. This site development scenario is based on an average annual production rate of 730,000 cubic yards of mine waste. Table 6.12-1 provides a summary of the expected rate of filling for the four TMA cells assuming this average mine waste filling rate. Table 6.12-1 also provides an indication as to whether a particular cell or stage is operating (O), full (F), closed (C), cell under construction (CUC) and/or final cover under construction (FCC) along with the cumulative mine waste volume based on the 730,000 cubic yards per year production rate. Refer to Table 16.13-1 for the definition of the terms listed above. This operating scenario, was developed within the following assumptions:

- Construction of each stage would occur over approximately two construction seasons (i.e., two years). During the first year, the major earthwork is completed and the liner and leachate collection system are constructed during the second year.

- Each cell will operate until it reaches its design capacity. The final year of cell operation (except TMA 4) will have two cells open. This is necessary since a portion of the final tailings deposition may be from beach tailings from the next stage rather than as a slurry.

- Each cell will remain open for one year or less prior to the placing of the final cover components. During this time period the tailings are expected to settle as and mine waste rock, beach tailings or coarse tailings can be placed and graded to shape the cell to the minimum 2 percent grades required prior to the placement of the final cover components.

- The placement and construction of the final cover is expected to be completed during one construction season after the final mine waste or soil grading layer are placed.

Table 6.12-1

TMA Staging Volumes[1] and Stage Life

Operating Year	TMA 1		TMA 2		TMA 3		TMA 4	
	Stage I (cy x 10^6)	Stage II (cy x 10^6)	Stage III (cy x 10^6)	Stage IV (cy x 10^6)	Stage V (cy x 10^6)	Stage VI (cy x 10^6)	Stage VII (cy x 10^6)	Stage VIII (cy x 10^6)
2	CUC							
3	CUC							
4	.73	CUC	-	-	-	-	-	-
5	1.46	CUC	-	-	-	-	-	-
6	F/2.03	.16	-	-	-	-	-	-
7	N	.89	CUC	-	-	-	-	-
8	N	1.62	CUC	-	-	-	-	-
9	N	F/2.15	0.20	-	-	-	-	-
10	N	F	0.93	-	-	-	-	-
11	N	FCC	1.66	-	-	-	-	-
12	NC	FCC	2.39	CUC	-	-	-	-
13	NC	C	3.12	CUC	-	-	-	-
14	NC	C	F/3.39	0.46	-	-	-	-
15	NC	C	N	1.19	-	-	-	-
16	NC	C	N	1.92	-	-	-	-
17	NC	C	N	2.65	-	-	-	-
18	NC	C	N	3.38	CUC	-	-	-
19	NC	C	N	4.11	CUC	-	-	-
20	NC	C	N	F/4.17	0.67	CUC	-	-
21	NC	C	N	F	1.40	CUC	-	-
22	NC	C	N	FCC	F/1.59	0.54	-	-
23	NC	C	NC	FCC	N	1.27	CUC	-
24	NC	C	NC	C	N	2.00	CUC	-
25	NC	C	NC	C	N	F/2.65	0.08	-
26	NC	C	NC	C	N	F	0.81	CUC
27	NC	C	NC	C	N	FCC	1.54	CUC
28	NC	C	NC	C	NC	FCC	F/1.71	0.56
29	NC	C	NC	C	NC	FCC	N	1.29
30	NC	C	NC	C	NC	FCC	N	2.02

Table 6.12-1 *(Continued)*

Operating Year	TMA 1		TMA 2		TMA 3		TMA 4	
	Stage I (cy x 10^6)	Stage II (cy x 10^6)	Stage III (cy x 10^6)	Stage IV (cy x 10^6)	Stage V (cy x 10^6)	Stage VI (cy x 10^6)	Stage VII (cy x 10^6)	Stage VIII (cy x 10^6)
31	NC	C	NC	C	NC	C	N	F/2.85
32	NC	C	NC	C	NC	C	N	F
33	NC	C	NC	C	NC	C	N	FCC
34	NC	C	NC	C	NC	C	N	FCC
35		C		C		C		C

(1) Staging volume refers to total tailings volume consumed of at the end of the year.

Conditions are:

C : Closed.

F : Full - cell has been filled to its maximum capacity but final cover is not in place. Assumes cell will remain open for one year or less prior to final covering.

FCC : Final cover construction - cell is graded to the required final grades, the final cover is constructed and the other reclamation activities for that cell are completed.

CUC : cell is "under construction".

cy x 10^6 : in-place cubic yards x 10^6 (millions) based on an annual deposition rate of approximately .73 x 10^6 cy for mine wastes. Assumes no interruptions of ore production and/or milling.

- : indicates cell or stage is not constructed at that time.

N : Next stage above being filled.

NC : Next stage above closed.

Prepared by: REM
Checked by: NXP

6.12.2.1 Site Phasing

Detailed phasing plans for TMA 1 and TMA 2, along with the earth construction quantities of TMA cells, are shown on Drawings 19, 20 and 21. These drawings summarize the construction material requirements for each stage of development of the TMA. Figures 6.12-1 through 6.12-5 show the stage construction of the TMA facility. The overall site sequencing plan is shown on Drawing 13. Piping and other miscellaneous construction quantities are provided only for TMA 1 and 2. The phasing of TMA 3 and 4 would follow a pattern similar to that for TMA 1 and 2. Design optimizations and other operating factors could result in some reconfiguration in the last two cells. Therefore, the detailed phasing and construction quantities of TMA 3 and 4 are not provided at this time.

Figures 6.12-1 through 6.12-5 shows the phasing of the construction and closure of TMA 1 through TMA 4 along with phasing of the construction staging area and surface water drainage. The construction staging areas generally include topsoil, till soil and mulch stockpiles, soil stockpiles for the soils used for processing, the soil processing plant and related facilities and an area for storing construction materials and construction equipment. A description of the construction or closure, shown along with a description of construction staging area and surface water management, is as follows:

- Figure 6.12-1, TMA 1 Stage I construction shows the condition after completion of TMA 1, Stage I construction including the following:
  - The earthwork for TMA 1, Stage I balances, therefore, no soil excavation is required outside of TMA 1.
  - Only the soil required for processing are stockpiled in the construction staging area shown north of TMA 1.
  - The reclaim pond, TMA access road, and runoff basins 8 and 9 are also constructed at this time.
  - Surface water drainage swales and ditches will be established to route water to runoff basins 8 and 9.
  - TMA 1, Stage I base grades are depicted.
  - A small quantity of soil from TMA 2 excavation is stockpiled in the southwest corner of TMA 4 along with soil and mulch required for processing for TMA 2 construction and TMA 1 closure.
  - Runoff basin 11 is constructed to receive drainage from the TMA 2 construction and stockpiling activities in the southwest corner of TMA 4.
  - TMA 2, Stage IV berm base grades are deposited since, as indicated, a significant portion of Stage IV berms is constructed during this period.
  - TMA 1 final grades are depicted.

- Figure 6.12-2, TMA 1, Stage II construction shows the conditions after completion of TMA 1, Stage II construction, including the following:
  - The earthwork for TMA 1, Stage II requires excavation from TMA 2.
  - Only the soil required for processing is stockpiled in the construction staging area shown north of TMA 1.
  - Runoff basin 10 and runoff ditching are constructed to receive drainage from the construction activities occurring in TMA 2, the approximate limits of excavation for which are shown.
  - TMA, Stage II berm base grades are depicted.
  - Access road to TMA 1 Stage II is constructed.
- Figure 6.12-3, TMA 1 closure and TMA 2 construction shows after completion TMA 2, Stage III and IV construction and final reclamation grades for TMA 1.
  - TMA 2 earthwork filling for both Stages III and IV is completed during the excavation of TMA 2 to eliminate double handling of excavated soil.
  - TMA 1 closure earthwork is completed by excavating and TMA 3 construction are stockpiled.
  - Runoff basin 12 is constructed to receive drainage from excavation and stockpiling areas in TMA 4 and TMA construction activities.
  - TMA 3, Stage VI berm grades are depicted since, as indicated, a significant portion of Stage VI berms is constructed during this period.
  - TMA 2 final grades are depicted.
- Figure 6.12-4, TMA 2 closure and TMA 3 construction, including the following:
  - TMA 3 earthwork filling for both Stages V and VI is completed during the excavation of TMA 3 to eliminate double handling of excavated soils.
  - TMA 2 closure earthwork is completed from the excavation of TMA 3 and additional excavation from TMA 4.
  - Only soils required for processing for TMA 2 closure are stockpiled.
- Figure 6.12-5, TMA 3 closure and TMA 4 construction, including the following:
  - TMA 4 earthwork filling for both Stages VII and VIII is completed during the excavation of TMA 4 to eliminate double handling of excavated soils.
  - TMA 3 closure earthwork is completed from the excavation of TMA 4.

- Excess soils from TMA 4 excavation are stockpiled north of TMA 2 along with soils required for processing for TMA 3 closure and TMA 4 construction.

- TMA 4, Stage VIII berm grades are depicted since, as indicated, a significant portion of Stage VIII berms is constructed during this period.

- TMA 3 final grades are depicted.

- Surface water drainage of the construction staging and borrow are routed to runoff basin 11.

- Drawing 18 and Drawing 22 show final grades and drainage plans, respectively, for all TMA cells at final reclamation grades, including the following:

  - Stockpile north of TMA 2 is removed. This soil is for TMA 4 closure.

  - The final grades for the borrow area north of TMA 4 are shown. This borrow area is required for TMA 4 closure.

  - TMA 4 final grades and final grades for the other three cells are shown.

6.12.3 TMA Earthwork Balance and Stockpile Sizing

The present design concept requires that only clay soils for lining and capping the proposed cells are brought in from off-site. All other soil materials required for the earthwork construction of the TMA cells are excavated from the cells themselves or are from Type I waste rock from the mining operations. Materials excavated from the TMA cells will either be used as unprocessed tills, i.e., as they come out of the excavation with minor modifications or as processed tills, i.e., screened, washed and/or crushed to meet specific gradation requirements. The earthwork balance and the processing of the required soil materials as described in the sections which follow. In addition the procurement of clay soils for lining and capping is discussed. These sections show that:

- the TMA earthwork balances and all materials can be excavated on-site, with the exception of low-permeability soils used for clay liner capping;

- the borrow outside of the proposed TMA footprint is kept to a minimum, thus limiting overall environmental impacts; and

- the progression of stockpiling during the site life results in a need for an on-site borrow area outside the boundaries of the TMA cells only for the final cover construction for TMA 4. The quantity required from the borrow area may be reduced depending on the amount of available Type I rock during the operating life of the mine.

6.12.3.1 Unprocessed Till Soil

Glacial till excavated from within the TMA footprint is the primary source of soil for the TMA embankments and other features. The glacial till at the Crandon Project can generally be

characterized as a heterogeneous mixture of primarily silt, sand, and gravel with small amounts of clay, (<10 percent) and cobbles and boulders (<5 percent). Section 6.5 of this report contains a detailed description of the in-situ and recompacted properties of the till soils. The unprocessed till material is proposed for constructing the following TMA components:

- TMA cell berms;
- TMA cell final cover - grading layer;
- TMA cell final cover - rooting layer; and
- miscellaneous construction fill such as reclaim pond, runoff ponds, embankments, access road fill, etc.

Typically in earthwork construction a shrinkage and swelling factor is used to determine whether the soil volume will shrink or swell from the borrow area to the fill site. This is typically estimated by comparing the compacted dry density determined by the laboratory tests to the *in situ* dry density of the in-place soils. Golder, 1982 indicated the *in situ* dry densities of the till soils ranged from 110 to 138 pounds per cubic foot while dry densities as determined by a percent compaction of 95 percent based on the standard Proctor test ranged from 118 to 130 pounds per cubic foot. It is apparent from these density figures that the *in situ* dry density of the till is very similar to the proposed compacted dry density. It is important to note that as per Golder, 1982, the *in situ* test data results are from test pits up to 20 feet deep while the tailings cell may be excavated to depths of up to 100 feet. Since it is well known that the till density increases with depth, the average *in situ* density may be slightly more than the 110 to 138 pounds per cubic foot referenced above. So it is most likely that there will be a slight earth work swell, i.e., a greater available volume in-place when compared with the volume required in the embankments. However, since the differences are likely to be small and some wasting of material may occur, a shrink/swell factor is not used in the earthwork balance calculations.

6.12.3.2 Mine Waste Rock

Mine waste rock will become available during the first three years of the mine development and some mine waste rock may be reactive while a large percentage will contain little or no sulfur bearing minerals. This Type II waste rock will be used only within the TMA. Other waste rock from mine development with less than two to three percent sulfur may be used as construction aggregate. It is estimated that approximately 600,000 tons of Type I rock will be available for construction purposes. In addition, it is estimated that approximately 100,000 tons of the Type II waste rock will be mined during the pre-production period and will be placed in the TMA. In addition, it is estimated that approximately 350,000 tons of rock material will be hoisted to surface for placement within the TMA facility. This rock, however can be used only inside the waste limits of the TMA because of its sulphur content. The mine waste rock materials brought to the surface are typically eight inches to 24 inches in diameter with some fines. The loose density of this rock as brought to the surface is approximately 111 pounds per cubic foot. The potential use and processing required for mine waste rock in the TMA and ancillary facilities development are as follows:

- Type II waste rock (100,000-450,000 tons estimated);

- crushed and screened for riprap used only on the interior slopes and cell bottoms of the TMA;
- unprocessed for construction of a causeway for the decant system if required; and
- unprocessed for the grading layer of the final cover.

- Type I Mine Waste Rock (600,000 tons estimated)
  - crushed and screened for riprap for stormwater and erosion control structures;
  - crushed and screened for base course or finish course surfacing such as railroad ballast, railroad subballast;
  - crushed for base courses or finish course for gravel roads, etc.; and
  - unprocessed for the grading layer of the final cover.

6.12.3.3 Processed Glacial Till

The TMA facility is designed such that with the exception of clay soils from off-site, all earthwork is constructed with glacial till from excavation of the cells, from a borrow area or from waste rock from the mine. Table 6.12-2 contains the general soil material gradation, a list of the process(es) required to produce the soil, the final use of the processed soil and the use of any by-products. "By-products" here is used to describe materials which are left over after processing the glacial till to make a specific soil gradation. The by-products can generally be added to other soil process lines and/or used for other purposes. Table 6.12-2 contains a listing of the soil types processed from the on-site glacial tills and the possible use for by-products. The quantities of materials required from processed glacial till are inflated by 15 percent to account for the variability in the gradation of the glacial till and potential construction losses.

6.12.3.4 Earthwork Balance for TMA Cell Construction and Closure

One of the primary goals of the TMA layout and design is to limit to the extent practicable the overall impacts of the TMA on the local environs. To accomplish this the earthwork balance for the TMA includes consideration of the following:

- attempting to balance the earthwork within the TMA footprint;
- balancing earthwork such that a borrow area is not required until the closure of the final cell (TMA 4);
- balancing the earthwork so that stockpiling is kept to a minimum, realizing that stockpiling must occur for each construction event involving the liner and leachate collection system and final cover since soil gradation processed from till must be provided for each of these construction events; and
- balancing the cut and fill for TMA 1, Stage I, so that initial disturbance is minimized.

Table 6.12-2

Summary of Soil Types Processed from Till

Soil Description/ Processing Required	Processed Soil Gradation	Processed Soil Use	By-Product/ Estimate Percent of Total	By-Product Use(s)
Till filter layer/screen all cobbles and boulders (i.e., all > 3")	3" (76 mm) to fines	Filter layer between tailings and drainage layer	5 percent cobbles, boulders greater than 3 inches	Add to unprocessed till or process for use as riprap
Liner protective layer/screen off all cobbles, boulders, (i.e., all > 3")	3" (76 mm) to fines	Protective layer on interior sideslopes over geomembrane	5 percent cobbles, boulders > 3" diameter	Add to unprocessed till or process for use as riprap
Drainage layer/wash to remove P_{200}, and screen to required size	Passing 2 " (51 mm) to coarser than No. 40 sieve (0.42 mm)	Drainage layer for leachate collection system and final cover	5 percent > 2 " and 53 percent portion passing the No. 40 sieve	• > 2" diameter to riprap • P40 to filter layer between tailings and till or liner protective layer
Crushed gravel base course/crush to 1" maximum size and remove content greater than approximately 10 percent	1" or 1½" to P_{200} (WisDOT gradation No. 1, section 304, for crushed gravel)	Road gravel, base course for pavements	Fines up to 10 percent of the till processed	Add to unprocessed till or liner protective layer
Riprap/crush cobbles and boulders from till	Riprap - 6" to 16" with not < 50% > 10" (WisDOT Section 606 Riprap)	Slope protection, erosion control	None	Not applicable

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Drawings 19, 20 and 21 show the phasing of the TMA for cells TMA 1 and TMA 2 and for the Site Preparation and Construction quantities for all four TMA cells. To achieve an earthwork balance including the needs for both processed and unprocessed glacial till soils for the four TMA cells, the considerations were the following

- TMA 1 earthwork balanced and the double handling of soils would not be necessary. The top of berms are placed at similar elevations and the base elevations adjusted such that TMA 1 and TMA 2 together would hold the zinc tailings.

- The earthwork for the remainder of cells (TMA 3 and TMA 4) are balanced such that borrowing outside the TMA footprint is kept to a minimum.

- The materials required for construction after cell 4 excavation is complete (i.e., cover materials for TMA 3 and TMA 4) are obtained in equal quantities from previous stockpiles and a borrow area. Thus both borrow quantities and disturbed area for stockpiles are kept to a minimum.

Table 6.12-3 provides the waste quantities and earthwork balance for the construction and closure of the four TMA cells. Table 6.12-3 provides the quantities of till soil needed for processing as well as soils needed for other uses as mentioned earlier. By-products from processing (i.e., typically coarser or finer soil material removed from the till) can and will be used in the embankment fills or final cover where appropriate or used for further processing into other required soil gradations. The quantity of stockpile soils is important from the standpoint of planning their location and providing the required surface water management and erosion control features. From Table 6.12-3, Table 6.12-4 was developed which lists required stockpiles by year and indicates the purpose for the stockpile and the approximate duration in years. Note that Table 6.12-4 assumes that when required in the construction sequencing excess soil excavation from a subsequent cell will be used to construct berms for the second phase of the active cell. Constructing in this fashion eliminates the double handling of excavated soils and minimizes or eliminates stockpiled quantities. In addition, the earthwork is accomplished in a more efficient manner.

For the shape and configuration of the stockpiles, it is assumed that soil will be excavated and hauled by wheel tractor-scrapers and deposited in a stockpile.

Figures 6.12-1 through 6.12-5 show the sequence of excavation, construction and closure of the four cells along with an indication of where stockpiles of excess material and stockpiles of the soil required for processing would be located. Table 6.12-3 and 6.12-4 give a detailed accounting of excavation quantities and stockpile sizes. The construction sequence which corresponds to the referenced figures is explained in the bulleted items which follow:

- Construction years 2 through 3: TMA Stage I Construction (refer to Figure 6.12-1). TMA 1, Stage I is constructed from soil excavated from Stage I, only the soil required for processing needs to be stockpiled.

Table 6.12-3

Waste Volumes and Earthwork Balances for TMA Cells

Year	Cell	Stage/Activity (cy x 10^6)[1]	Waste Volume[2]	Cut[3] (cy x 10^6)[1]	Fill[3]	Construction Material (cy x 10^6)		Borrow[6] (cy x 10^6)[1]	Extra Cut[7] (cy x 10^6)[1]	Comments
						For Processing	Other Uses[5]			
2 to 3	1	Stage I/Construction	---	1.339	.911	.319	Access Road Ramp - .109	---	-0-	Cut entirely from Cell 1 Stage I, balances with fill required for Stage I construction
3	1	--	---	---	---			---		Construct Stage I liner and LCS
4	1	Stage I/Operation	.73	---	---	---	---	---	---	---
5	1	Stage I/Operation and Construction Stage II	1.46	0.0001	1.807	.037		1.844	---	Borrow from TMA 2
6	1	Stage I and II Operation and Construction Stage II	I - 2.03 II - .16	---	---	---	---	---	---	Construct Stage II liner and LCS
7	1	Stage II/Operation	.89	---	---	---	---	---	---	---
8	2	Stage II/Operation and Stage III/Construction	1.62	4.039 - 1.844 = 2.195	.146	.365	---	---	1.684	Borrow from TMA 2
9	1	Stage II and III/Operation and Stage III/Construction	2.15	---	--	---	---	---	1.684	Stage III liner and LCS and construct TMA 2 Stage IV berms in advance
10	2	Stage III/Operation	.93	---	---	---	---	---	---	---
11	2	Close TMA 1 and Stage III/Operation	1.66	0.0012	.935	.155	---	---	0.5952	---
12	2	Stage III/Operation	2.39	---	---	---	---	---	---	---
13	2	Stage III/Operation and Stage IV/Construction	3.12	---	1.617	.061	---	1.083	---	Borrow from TMA 3

Table 6.12-3 *(Continued)*

Year	Cell	Stage/Activity (cy x 10^6)[1]	Waste Volume[2]	Cut[3] (cy x 10^6)[1]	Fill[3]	Construction Material (cy x 10^6)		Borrow[6] (cy x 10^6)[1]	Extra Cut[7] (cy x 10^6)[1]	Comments
						For Processing	Other Uses[5]			
14	2	Stage III and IV/Operation and Stage IV/Construction	III - 3.39 IV - 0.46	---	---	---	---	---	---	Construct Stage IV liner and LCS
15	2	Stage IV/Operation	1.19	---	---	---	---	---	---	---
16	2	Stage IV/Operation	1.92	---	---	---	---	---	---	---
17	2	Stage IV/Operation	2.65	---	---	---	---	---	---	---
18	2	Stage IV/Operation	3.38	---	---	---	---	---	---	---
19	2	Stage IV/Operation and Stage V/Construction	4.11	2.198 - 1.083 = 1.115	0.007	.207	---	---	0.901	---
20	3	Stage IV and V/ Operation and Stage V/Construction	IV - 4.17 V - .67	---	---	---	---	---	---	Construct Stage V liner and LCS
21	2	Close TMA 2	---		1.34	.203		0.642	---	Borrow from TMA 4
	3	Stage V/Operation and Stage VI/Construction	1.4	---	.614	.049		1.305	---	Borrow from TMA 4
22	3	Stage V and VI/Operation and Stage VI/Construction	1.59	---	---	---	---	---	---	Construct Stage VI liner and LCS
23	3	Stage VI/Operation	1.27	---	---	---	---	---	---	---
24	3	Stage VI/Operation and Stage VII/Construction	2.00	3.435 - 1.305 = 2.13	0.0007	.228	---	---	1.901	---
25	3	Stage VI and VII/Operation and Stage VII/Construction	2.65	---	---	---	---	---	---	Construct Stage VII liner and LCS
26	4	Stage VII/Operation and Close TMA 3	.81	---	.788	.133	---	---	0.980	

Table 6.12-3 *(Continued)*

Year	Cell	Stage/Activity (cy x 10^6)[1]	Waste Volume[2]	Cut[3] (cy x 10^6)[1]	Fill[3]	Construction Material (cy x 10^6)		Borrow[6] (cy x 10^6)[1]	Extra Cut[7] (cy x 10^6)[1]	Comments
						For Processing	Other Uses[5]			
27	4	Stage VII/Operation and Construction Stage VIII	1.54	.045	.178	.053	---	---	0.794	
28	4	Stage VII and VIII/Operation and Stage VIII/Construction	VII - 1.71 VIII - .56	---	---	---	---	---	---	Construct Stage VIII liner and LCS
29	4	Stage VIII/Operation	1.29	---	---	---	---	---	---	---
30	4	Stage VIII/Operation	2.02	---	---	---	---	---	---	---
31	4	Stage VIII/Operation	2.75	---	---	---	---	---	---	---
32	4	Stage VIII/Operation	2.85	---	---	---	---	---	---	---
33	4	Close Cell 4			.846	.143	---	.195		---
		Contingency @ 15%[8]			0.586	0.293	---	1.074		Borrow from a borrow area north of TMA 4

Notes:

1 (cy x 10^6) = cubic yards in millions.
2 Waste volumes are cumulative and include all mine wastes. A constant waste generation rate of .73 x 10^6 cy annually is assumed.
3 Cut/fill in millions of cubic yard for TMA cell excavation/berms generated by CADD.
4 Construction materials for processing quantities in millions of cubic yards is the quantity of material required for processing. The in-place quantities for processed soils are provided on Drawings 19, 20 and 21.
5 Construction materials for other uses in millions of cubic yards is the quantity of soil available which can not be used at that time for embankment, final cover or other fill needs.
6 Borrow is the quantity of soil in millions of cubic yards borrowed to construct the TMA stage or cell closure.
7 The extra cut will be used for future berm construction.
8 15 percent contingency for construction materials including fill for closing TMAs.
LCS - Leachate Collection System.
--- indicates that no borrow or stockpiling activities are ongoing.

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Table 6.12-4

TMA Construction - Stockpile Sequence[1]

Approximate Time Period (years)	Stockpile Size (cy x 10^6)	Purpose	Comments
2 - 3	.319	Till for Stage I Process Soils	Stage I soils and aggregates will be stockpiled after processing
5 - 6	.037	Till for Stage II Process Soils	Process soils placed after processing
8 - 9	.365	Till for Stage III Process Soils	Stage III soils and aggregates will be stockpiled after processing
10 - 11	0.67	Till for TMA 1 Closure	Extra excavation from TMA 2 stockpiled for TMA 1 Closure
11	.155	Till for TMA 1 Closure	Closure soils and aggregates will be stockpiled after processing
12 - 13	.061	Till for Stage IV Process Soils	Stage IV soils and aggregates will be stockpiled after processing
18 - 19	.207	Till for Stage V Process Soils	Stage V soils and aggregate will be stockpiled after processing
20 - 21	.252[2]	Till for TMA 2 Closure plus Stage VI construction	TMA 2 Closure and Stage VI Construction soils will be stockpiled after processing
23 - 24	1.351[3]	Till for Stage VII Process Soils plus TMA 4 excess excavation	Aggregates will be stockpiled after processing
26	.211	Till stockpile reduced for Cell 3 closure	Remaining stockpile size given
27	.018	Till stockpile for Stage VIII construction plus remaining stockpile	Stage VIII process soils and aggregates stockpiled after processing
28	.018	Remaining stockpile for TMA 4 closure	

1 This scenario assumes that excess excavation from subsequent cells is used to construct cells berms prior to when they are required.

2 0.203 + 0.049 = 0.252

3 1.901 - contingency 1.901 - .341 - .209 = 1.351

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- Construction years 4 through 5: TMA 1 Stage II Construction (refer to 6.12-2). TMA Stage II constructed with soil excavated from TMA 2. Only the soil required for processing needs to be stockpiled.

- Construction years 7 through 13: TMA 2 Stage III and IV Construction and TMA 1 closure (refer to Figure 6.12-3). TMA 2 Stage III and IV constructed with soil excavated from TMA 2. Berms for Stage IV constructed in advance during Stage III Construction so that stockpiling and double handling of soil are kept to a minimum. Soils required for processing are stockpiled in the southwest corner of TMA 4. The soils required for TMA 1 closure are excavated from TMA 3.

- Construction years 18 through 21, Stages V and VII Construction and TMA 2 closure (refer to Figure 6.12-4). The excavation of TMA 3 is completed during the construction of TMA 3, Stage V. Stage VII embankments are constructed during Stage V Construction to minimize double handling. The soil required for processing is stockpiled in the southwest corner of TMA 4. The soil required for TMA 2 closure is also borrowed from the TMA 3 excavation.

- Construction years 23 through 27: TMA 4 Cells VII and VIII Construction and TMA 3 closure (refer to Figure 6.12-5). The excavation of TMA 4 is completed during the construction of TMA 4. Stage VII and VIII embankments are constructed during Stage VII construction to minimize double handling. The soil required for processing and excess soils excavated from TMA 4 are stockpiled north of TMA 2 (Figure 6.12-5). The soil required for TMA 3 closure are obtained from this stockpile, which is exhausted upon completion of TMA 3 closure.

- Construction years 33 and 34: TMA 4 closure including the soils required for processing are excavated from the hill directly north of TMA 4. Refer to Drawing No. 18 for the final grades for the TMA cells and for the proposed borrow area.

Topsoil stockpiles are proposed to be located in construction staging areas shown in Figures 6.12-1, 6.12-2, 6.12-3, 6.12-4, and 6.12-5. These areas will be within the limits of the TMA. As construction progresses, topsoil will be stripped from TMA cells and added to the stockpile. As portions of the stockpile are added to or utilized, disturbed areas will be reseeded consistent with the procedures outlined in the MPA (Section 4.10). The excavation of topsoil is based on the removal of a nine-inch layer of topsoil. In addition, a portion of the nonmarketable wood waste, peat humus soils and hydric soils excavated from wetlands within the TMA footprint would be utilized as topsoil substitutes. Since an off-site wetland mitigation/compensation site has been selected, special provisions to maintain hydric soils are not presently planned. The procedures for topsoil renewal and preservation at the TMA would be consistent with those procedures described in the MPA Section 4.10, Erosion Control.

6.13 Typical Annual Waste Material Balances, NR 182.08(2)(e) 6., a. through d.)

NR 182.08(2)(e)6. requires the preparation from the best available information of a material balance for the time periods listed in NR 182.08(2)(e) 6. a. through d. The information which follows is taken from Table 6.12-1 and supplies the required material balances. The narratives designated a. through d. which follows provides the requested information.

NR 182.08(2)(e)6. a. A typical production year will result in the placement in the TMA of approximately 730,000 cubic yards of mine wastes which may include fine tailings, whole tailings, wastewater treatment plant solids , Type II waste rock and laboratory wastes.

NR 182.08(2)(e)6. b. Examples of possible changes in operation are a shutdown of the plant at a point in time when a cell is partially full or temporary shutdown, if the plant is scheduled to reopen within some time period. Section 6.6.10 of this report describes two possible scenarios for what is referred to as premature closure. Section 6.2.7 describes the steps which need to be taken to maintain the tailings' chemical stability given two possible time periods for a plant shutdown. For both cases it is assumed that the active cell is partially filled. As described in Section 6.2.10 in the case of a long-term shutdown (i.e., 2 years or more), it is recommended that a temporary water cover be placed on any tailings areas not final covered. Reopening the facility then would simply involve pumping the water down to the required operating depth and restarting tailings deposition as before.

NR 182.08(2)(e)6. c. The projected conditions existing at the end of operations as proposed herein is that TMA cells TMA 1, TMA 2, TMA 3 and TMA 4 would be filled. All mine wastes including tailings, mine waste rock, wastewater treatment plant solids and laboratory wastes would be placed in the TMA cells. It is estimated that this would amount to approximately 20.46 million cubic yards which includes a 13 to 15 percent contingency factor.

NR 182.08(2)(e)6. d. At the end of reclamation final cover would have been placed on the TMA cells and no additions to the waste balance is anticipated after the end of reclamation.

6.14 Environmental Monitoring, (NR 182.13)

An intensive monitoring program including a quality assurance plan is being developed for the construction and operation phase and the long-term care and maintenance phase of the TMA. Details of the monitoring program will be outlined in Section 10 of the MPA. The construction and operation phase of the monitoring program will run until closure of the site. This occurs when final site grading and initial revegetation is complete. At this time the long-term care and maintenance portion of the monitoring program will begin.

The construction and operation phase of monitoring includes programs for groundwater quality; groundwater elevations; surface water monitoring including water quality, lake levels, stream flows, sediments, fish, macroinvertebrates, habitat characteristics, and wetland surface flows; terrestrial ecology, meteorology, mine inflows, air, TMA leachate and reclamation cover subsidence. The details of each of these programs will be described in Section 7 of the MPA. The long-term care and maintenance monitoring begins after final grading and revegetation of the TMA. This monitoring program includes groundwater elevations, groundwater quality, vegetation and ground surface monitoring. MPA also contains details of testing microbiological and radiological samples satisfying NR 182.135.

6.15 Stability of TMA Embankments

6.15.1 General

The TMA consists of four cells each surrounded by embankments of compacted till. The cells share the interior embankments. The embankments will be constructed partly by excavation and partly by filling and have slopes of three (horizontal) to one (vertical). The impoundment (inside) slopes also have a stabilizing berm. The liquid in the impounded tailings is isolated from the embankment by the use of a hydraulic barrier. The hydraulic barrier consists of a geomembrane overlying a layer of low-permeability earthen material. Thus, the effect of the impoundment is to provide a stabilizing load and increase the stability of the slopes during all stages of TMA operation and the post-closure period, even if the tailings were subjected to a highly improbable scenario of seismically induced liquefaction. For the outside slopes, the stability of the embankments is likely to be lower in the absence of the stabilizing force described above. Also, since the slopes are the same for the full length of the embankments, the least factor of safety against slope failure will occur where the slope height is a maximum. Therefore, if the outside slope at the maximum height is analyzed and found to have factors of safety (static and seismic) indicative of satisfactory stability, then it can be concluded that the whole structure is stable. This is because all other sections and slopes of the structure will have higher factors of safety than the analyzed section. Besides the slopes, analyses need also to be performed to evaluate the stability of the foundations under both static and seismic conditions.

6.15.2 Geology

The geological sections developed for the TMA site indicate that the foundation soils consist of dense glacial till and dense outwash granular soils. Both these soils behave similarly under stress (being granular soils) and based on prior studies by Golder Associates (1981) for the Crandon Project have similar strength characteristics also. Therefore, for stability analyses, the foundation soils need only be modeled as a single layer with uniform properties and with no weak zones where potential failure surfaces may lie.

6.15.3 Methods of Analyses

The method of analysis for both slope stability and foundation stability is the same because the least stability of the foundations will exist near the toe of the slopes of the embankments and can be determined using limit equilibrium methods which are used for slope stability analyses.

The section used for analyses was 106 feet high. This is an approximately SW-NE section of the southwest embankment of TMA 1. As described earlier, the compacted till, in situ till as well as outwash deposits are granular soils and their properties used for stability analyses as shown on Table 6.15-1.

The slope stability analyses were performed using a PC-based program PCSTABL6 adapted for PC by Peter Bosscher of the University of Wisconsin, Madison, from the original main frame version STABL developed at Purdue University by Ronald A. Siegel. The Modified Bishop Method was used for the analyses. For analyzing seismic stability of the slopes and foundations a pseudostatic method was used wherein the earthquake induced forces are accounted for by the use of an "equivalent" lateral load. This lateral load is taken as the product of a seismic coefficient and the weight of the failure mass (the mass above the potential failure surface). If

Table 6.15-1

Soil Properties for Stability Analyses

Soil Type	Unit Weights* pcf		Strength Characteristics	
	Moist	Saturated	c′ psf[1]	ϕ′, degrees[2]
Compacted Till	133	140	0	35
Foundation Till	133	137	0	35
Foundation Outwash	127	137	0	35

(1) c′ - Cohesion
(2) ϕ′ - Angle of internal friction (degrees)

*Note Unit weights calculated with the following data:
Dry unit weight of compacted till is 125 pcf and dry unit weights of other soils is 121 pcf each; both from Golder (1981). Assumed moisture content for compacted till is 7 percent, *in situ* till, 10 percent and outwash, 5 percent.

Prepared by: NXP
Checked by: REM

the seismic coefficient is assumed as the peak ground acceleration expected at the site divided by acceleration due to gravity, the analysis will normally result in a very conservative result due to several reasons. The primary reason is that the peak pulse of the earthquake motion lasts only for a very short time and is followed by accelerations of smaller magnitude, many of which act in the opposite direction providing stabilizing forces. Thus, if the maximum horizontal destabilizing force caused during the design earthquake is considered to act all the time in spite of the fact that it acts only for a fraction of a second, a much smaller factor of safety than the actual factor of safety would be computed.

In order to conduct a more realistic analysis of the seismic behavior of earth embankments and slopes a displacement analysis is preferable. Several approaches for this are available in the literature such as Newmark method and its derivatives, Seed method and its variations. These methods account for not only the magnitude of the earthquake-induced forces, but also the duration and direction of these forces. However, if the strength properties of the embankment materials do not deteriorate due to earthquake-induced vibrations, a pseudostatic factor of safety in excess of one will indicate that the design earthquake will not cause any permanent deformation of the structure. This is valid for a seismic coefficient corresponding to the peak design ground acceleration (or the peak effective acceleration in the structure if amplification of acceleration is likely to result when the waves propagate through the embankment).

On the other hand, if the computed factor of safety falls below unity for the seismic coefficients described above, some amount of slipping along the potential failure surface can be expected. The magnitude of these slip - displacements will depend on the magnitude of the accelerations and the durations for which they act. The sum total of all displacements caused by the different pulses during the earthquake will be the permanent deformation suffered by the embankment. If this deformation is of little or no consequence to the structural integrity and intended performance of the embankment, the design must be deemed safe in spite of the fact the pseudostatic factor of safety was less than one, a value considered in the past to be the best indicator of earthquake stability of embankments.

In summary, an embankment is stable if the engineering design and analyses yield the static factor of safety and seismic displacements within acceptable limits. For the static factor of safety, the Mine Safety and Health Administration does not specify minimum values. Therefore, the required factor of safety proposed to be used for this site is based on recommendations of other regulatory agencies.

6.15.4 Results of Analyses

A slope stability analysis was performed for four different cases (Figure 6.15.1). The cases and the factors of safety (F.S.) are:

1. static case of embankment stability; F.S.= 2.186
2. seismic case of embankment stability F.S.= 1.816 with a seismic coefficient of 0.06
3. static case of foundation stability F.S.= 2.339
4. seismic case of foundation stability F.S.= 1.932 with a seismic coefficient of 0.06

6.15.5 Minimum Recommended Factors of Safety

NAVFAC DM-7.2 recommends that for embankments the minimum factor of safety for permanent or sustained loading conditions should be 1.5; for foundation of structures a factor of safety of no less than 2.0 is desirable, and for earthquake loads safety factors as low as 1.2 or 1.15 may be tolerated.

USEPA (1993) recommends embankment factors of safety to account for:

- uncertainty of strength measurements (small or large);
- danger to human life or major environmental impact if slope fails (imminent or not imminent); and
- loading conditions (static or seismic).

For the worst case (large uncertainty of strength measurements and imminent danger to life and environment) the recommended values by USEPA are 2 or greater (static) and 1.7 or greater (seismic).

Based on the two above recommendations and considering the conditions at the site, the proposed design values for factors of safety for the TMA cells are:

Slope stability - 1.5 (static); 1.2 (seismic)
Foundation stability - 2.0 (static); 1.5 (seismic)

As in all cases, the calculated factors of safety exceed the proposed design factors of safety for all conditions, thereby indicating a stable structure.

All the analyses described earlier were completed with circular failure surfaces. Since the embankment material is considered cohesionless, the lowest theoretical values of static and seismic factors of safety for the embankment can be computed from:

Static factor of safety of embankment

$$F.\ S.\ Static = \frac{\tan \phi}{\tan \beta}$$

Seismic factor of safety of embankment

$$F.\ S.\ Seismic = \frac{\tan \phi}{\tan (\beta + \tan^{-1} k)}$$

where ϕ = *angle of internal friction of the embankment material* = 35°

$\tan\beta$ = *slope of the embankment* = 1/3

k = *seismic coefficient* = 0.06

The static and seismic factors of safety are calculated as 2.1 and 1.745 respectively. For a seismic coefficient of 0.08, the seismic factor of safety is 1.65. It will be seen that these computed values are larger than the values proposed for the site.

6.15.6 Summary of Safety Factors for the TMA Embankments

The factors of safety of the TMA embankments have been estimated using a conservative and well-accepted method of stability analysis, viz. The Modified Bishop Method. The cases analyzed addressed the geology of the foundation soils and the materials and methods of construction of the embankments. The strength characteristics of the embankment materials (which is the most important parameter along with the geometry), have been obtained from earlier laboratory test results. The use of a hydraulic barrier consisting of a geomembrane overlying a layer of low permeability material will isolate the liquids in the impounded tailings from the embankment material. Therefore no pore pressures are likely to be generated in the embankment. Through time, changes to these conditions affecting any aspect of embankment or foundation stability are not likely to occur. The proposed factors of safety are conservative for the conditions expected at this TMA site. Finally, the computed factors of safety both under static and seismic conditions are much higher than the target values and therefore the embankments are designed as stable structures.

6.15.7 Erosion Potential of the Embankment Slopes

The embankment slopes are fairly steep at three to one (3H to 1V) and long. This combination could cause some erosion of the soils from the slope surfaces. The outer slopes will be vegetated after construction and in the course of time will be invaded by trees, shrubs, and mature grasses. Appendix L contains calculations using the Revised Universal Soil Loss Equation (RUSLE) which show that once vegetation is established soil loss is within an acceptable range at 2.24 tons per acre per year. The USDA recommends that soil loss be less than 2.0 tons per acre per year for cultivated land. The soil lost from the embankments is directed through the surface water management features described in Section 6.9 of this report, so siltation of waterways around the TMA is not of concern. The methods used to stabilize the slopes include seeding, fertilizing, and mulching the embankment slopes as soon as practicable after construction is completed. The reclamation of these slopes is detailed in Section 5.7 of the MPA.

For the inner slopes, however, the erosion potential of the till cover over the TMA liner will be smaller since the slopes will be exposed only for a limited period of time, typically four years or less. Therefore, the erosion potential for the inside slopes were estimated using RUSLE. These calculations are presented in Appendix M. The estimated erosion/soil loss is the equivalent of about four inches of the till cover in four years. This is relatively insignificant compared to the design thickness of the till, 18 inches. It may be noted that this large thickness of 18 inches was

selected primarily because of the concern for liner damage due to vehicular traffic during the construction period. Moveover, the estimated soil loss of four inches in four years when the slopes are exposed is a conservative value because of two aspects. First, only the higher elevations of the slope (separately for the two stages in each cell) will be exposed for four years, the lower reaches being covered earlier by the tailings placed in the cells. Secondly, the tailings placement will lead to the till being covered by the tailings thus increasing the effective thickness of the cover.

In view of the above, CMC does not propose to vegetate the inside slopes of the TMA. In the unlikely event of a deep erosion due to directed flow, immediate repair measures will be undertaken.

6.16 Documentation/Agreements

This section includes references to the data required to be submitted by NR 182.09(2)(e) in the appendices of the report. Some of the required information is contained in other reports referenced herein.

Section 2.1 of the MPA contains a legal description of the land owned or leased by CMC.

Section 6.3.3 of this report provides details concerning an option which CMC has with ASHCO, Inc. This option covers the procurement of clay soils for lining and capping of TMA 1 of the proposed facility. This option is considered proprietary since it contains financial terms.

Section 2.3 of the MPA references the requirement to secure land use and zoning approvals on local agreements with local government units impacted by the mine. These approvals/agreements have not been finalized at this time. The documentation of agreements, leases, etc. will be provided as required when they become available.

6.17 Design Performance Assessment

6.17.1 Performance Evaluation - Groundwater Criteria

The TMA has been designed with engineering features such that the percolation from the site is of acceptable quality and quantity to comply with groundwater standards. In order to evaluate the performance of the system design, a comparison of the estimated groundwater quality at the TMA's compliance boundary to potential groundwater quality standards was performed. The analysis is discussed in Section 4.2.5.10 of the project's *Environmental Impact Report*. A detailed report describing the comparison is contained in Appendix 4.2-12 of the *Environmental Impact Report*. The comparison shows that in all cases, the estimated concentration for all parameters for both the best engineering judgement and the practical worst case scenarios fall below the potential standard or an alternative concentration limit established where baseline concentrations exceed potential standards. Therefore it can be concluded that the TMA will not cause an exceedance of anticipated groundwater standards at the compliance boundary. An overview of the analysis is presented below.

The analysis first involved the determination of the quality (source concentrations) of the leachate that may percolate from the TMA. This was completed using data both from waste characterization studies completed in the 1980s, and the waste characterization studies currently

underway for the project. The methodology for determining the source concentrations and the results of the analysis are discussed in Appendix 4.2-12 of the project's *Environmental Impact Report*.

The analysis also involved solute transport modeling. This modeling was performed to establish the incremental concentration at the compliance boundary as a function of time for substances that potentially may percolate from the TMA. The modeling work described in Section 4.2.5.10 and Appendices 4.2-10 and 4.2-12 of the project's *Environmental Impact Report* was completed for both a best engineering judgement scenario and a practical worst case scenario. TMA percolation rates used in the model were based on the HELP model calculations described in Section 7 of *Addendum No. 3 to the Tailings Management Area Feasibility Report/Plan of Operation* (Foth & Van Dyke, 1997a).

Using the information from the source concentration derivation and solute transport modeling, the final steps of the compliance demonstration were completed as described below.

Groundwater quality associated with the TMA is regulated under Chapter NR 182, Wis. Admin. Code. According to this code, groundwater quality standards at the compliance boundary are to be based on three key criteria:

- Primary and secondary Maximum Contaminant Levels (MCLs) as promulgated by state or national drinking water standards;
- The baseline concentration where that concentration exceeds the MCL; and
- The need to protect public health, safety and welfare.

The following discussion is based on the groundwater quality regulatory framework in place in the State of Wisconsin at the time this section of the report was written. It should be noted that at that time potential revisions to NR 182 as related to groundwater quality were under consideration by the WDNR. If revisions to the groundwater quality provisions of NR 182 are promulgated in the future, the following discussion may need to be updated.

For the State of Wisconsin, the MCLs are addressed in Chapter NR 809 Wis. Admin. Code entitled "Safe Drinking Water." Standards relating to the protection of the public health, safety and welfare are addressed in Chapter NR 140, Wis. Admin. Code entitled "Groundwater Quality" where Enforcement Standards (ESs) and Preventive Action Limits (PALs) are specified. In most cases the numerical values for the MCLs of NR 809 and for the ESs of NR 140 are the same or very similar. Any exemptions granted by the WDNR under NR 182 must not exceed the levels required to protect public health, safety and welfare.

The term "baseline concentration" is defined in NR 182 as "the concentration of a substance in groundwater . . . as determined by monitoring before mining operations." The same regulation defines the term "background concentration" as "the concentration of a substance in groundwater . . . as determined by monitoring at locations which are not to be affected by the mining site."

In addition, NR 182 also defines "statistically significant change" as "an amount of change determined by the use of statistical tests for measuring significance at the 95 percent confidence level." This term is used in conjunction with the code's requirement that a mining waste site

"shall not cause concentrations of substances in groundwater at or beyond the compliance boundary in excess by a statistically significant amount of the groundwater quality standard" established per NR 182.

Given the above discussion, in order to accomplish the groundwater compliance assessment for the TMA, the following four steps had to be completed. The first was to define the source concentrations for the TMA. The second was to calculate estimated incremental groundwater changes at the compliance boundary due to the TMA without considering baseline groundwater concentrations. The compliance boundary is expected encompass the TMA for a distance of 1,200 feet from the edge of the facility.

The third step was to evaluate existing groundwater quality data within the proposed TMA compliance boundary. This analysis was performed to determine if any baseline concentrations exceed MCLs, which would then require the establishment of an alternative concentration limit (ACL) for those parameters. NR 182.075(1)(a)2 b states that "Where the baseline concentration of a substance subject to a state or national drinking water standard exceeds the MCL set by state or national drinking water standards, the groundwater quality standard shall be the baseline concentration of that substance . . . ." Since groundwater quality data typically varies over time, a statistical analysis of the data is used to define the baseline concentration. For the purposes of the TMA compliance demonstration, baseline concentrations statistically derived per NR 182.075(1)(a)2 b are referred to as ACLs.

The fourth step of the analysis involved the comparison of estimated parameter concentrations at the TMA compliance boundary to the MCLs, and ACLs where appropriate, to determine if the TMA as designed will meet potential groundwater standards.

In accordance with NR 182.075, the WDNR is to establish the groundwater quality standards for the TMA as part of the master hearing process. To assist in the completion of the TMA performance assessment, it was assumed that these standards will be the NR 809 primary and secondary MCLs, with ACLs established where appropriate. Using the NR 809 list for this analysis is valid since the metals of importance in the TMA source term are included in NR 809.

A detailed description of the calculations and evaluations performed for the four steps outlined above are contained in Appendix 4.2-12 of the project's *Environmental Impact Report*. Table 6.17-1, which is reproduced from Appendix 4.2-12, shows the results of the TMA compliance performance assessment. A review of the table shows that in all cases, the predicted concentration for all parameters for both the best engineering judgement and the practical worst case scenarios fall below the standard or an alternative concentration limit established where baseline concentrations exceed potential standards. Therefore it can be concluded that the TMA will not cause an exceedance of anticipated groundwater standards at the compliance boundary.

Table 6.17-1

Tailings Management Area Compliance Performance Assessment

Parameter	Units	Incremental Concentration Change At Compliance Boundary Due to Source Term[1]		Average Baseline Groundwater Quality[2]	Estimated Parameter Concentration at Compliance Boundary[3]		MCL[4]	ACL[5]
		Best Engineering Judgement	Practical Worst Case		Best Engineering Judgement	Practical Worst Case		
Aluminum	mg/L	0.000016	0.0059	<0.200	<0.200	<0.200	0.2[6]	
Antimony	mg/L	0.000072	0.00046	<0.050	<0.050	<0.050	0.006	0.070
Arsenic	mg/L	0.00016	0.038	<0.005	<0.005	0.040	0.05	
Barium	mg/L	0.0000062	0.000084	0.025	0.025	0.025	2	
Beryllium	mg/L	0.00087	0.0027	-[8]	-	-	0.004	
Cadmium	mg/L	0.00000035	0.0000039	<0.001	<0.001	<0.001	0.005	
Chromium	mg/L	0.00057	0.0021	<0.005	<0.005	<0.005	0.1	
Copper	mg/L	0.00010	0.00031	<0.030	<0.030	<0.030	1.3	
Cyanide[7]	ug/L	0.43	1.3	<20	<20	<20	200	
Fluoride	mg/L	0.00052	0.0017	0.098	0.099	0.100	4	
Iron	mg/L	0.00069	0.0026	<0.100	<0.100	<0.100	0.3[6]	
Lead	mg/L	0.00032	0.0011	<0.002	<0.002	0.0021	0.015	
Manganese	mg/L	0.000029	0.00059	0.011	0.011	0.012	0.05[6]	1.5
Mercury	ug/L	0.00063	0.0024	<0.2	<0.2	<0.2	2	
Nickel	mg/L	0.000014	0.000046	<0.030	<0.030	<0.030	0.1	

Table . *(Continued)*

Parameter	Units	Incremental Concentration Change At Compliance Boundary Due to Source Term[1]		Average Baseline Groundwater Quality[2]	Estimated Parameter Concentration at Compliance Boundary[3]		MCL[4]	ACL[5]
		Best Engineering Judgement	Practical Worst Case		Best Engineering Judgement	Practical Worst Case		
Selenium	mg/L	0.00042	0.0012	<0.005	<0.005	<0.005	0.05	
Silver	mg/L	0.00014	0.00042	<0.010	<0.010	<0.010	0.1[6]	
Sulfate	mg/L	1.9	7.6	13	15	21	250[6]	
Zinc	mg/L	0.00072	0.0055	<0.020	<0.020	<0.020	5[6]	

[1] From Table 3-2 of Section 3 of the *Crandon Project Tailings Management Area Groundwater Quality Performance Evaluation* (Foth & Van Dyke, 1997b).

[2] Value is upper 95 percent confidence limit of the mean.

[3] Equals the incremental source concentration at the compliance boundary plus the upper 95 percent confidence limit of the mean. For confidence limits below detection the incremental source concentration was added to one-half the detection limit. For those cases where the estimated parameter concentration was still less than the detection limit, the estimated parameter concentration was set at the detection limit.

[4] NR 809 MCL.

[5] Alternative Concentration Level calculated statistically using baseline water quality data.

[6] Value is secondary MCL.

[7] The source concentration is for total cyanide. The cyanide MCL is for free cyanide. Since the source concentration is appreciably lower than the MCL, a determination of the source concentration's free cyanide is not necessary.

[8] No baseline data available.

Prepared by: JWS
Checked by: SCL

6.17.2 Wetlands

Section 3.1.3 addresses the operational and design considerations related to the minimization of disturbance to wetlands. These considerations are required by NR 182.07(2) and the standards and procedures of NR 132.06(4). In addition, operational impacts which could result from the construction and operation of the TMA and appurtenant facilities must be considered. Section 4 of the EIR addresses the impacts the TMA may have on wetlands in and around the proposed TMA cells. Section 5 of the EIR also addresses the mitigation efforts proposed to replace the wetlands which are within the TMA footprint.

6.17.3 Surface Water Criteria

Section 6.9 of this report provides the design rationale and details concerning the surface water management system proposed for the TMA and appurtenant facilities. Section 4 of the EIR assesses the performance of the TMA as related to surface waters.

6.18 Minimum Design Requirements and Application of Parameters and Concepts Recommended by the Department Under NR 182.11(2)

6.18.1 Minimum Design Requirements NR 182.11(1)

NR 182.11(1)(a) through (u) contains minimum design requirements for mine waste management facilities, surface water drainage control facilities, operations, liners, leachate management systems, and other features. These design requirements, where applicable, are incorporated into the design of the major facility components contained herein. The proposed design components comply with all the design standards listed in NR 182.11.

6.18.2 Application of Parameters and Concepts Recommended by the Department Under NR 182.11(2) and (3)

NR 182.11(2)(a) requires that when practical, a mine waste site should be located in the same watershed as the mining surface facilities. The Crandon Project including the plant site and the TMA are located within the Wolf River watershed. However, because of the numerous lakes, streams and wetlands in the project area, the plant site and TMA are located in two different basins of tributaries of the Wolf River; Swamp Creek drainage and the Pickerel Creek drainage.

NR 182.11(2)(b) requires that tailings pipelines not cross water courses or wetlands where such crossings would be inconsistent with Wis. Admin. Code NR 132. Section 4 of the EIR provides an analysis of alternatives for the tailings pipeline and access road route. The route has been selected such that the intent of NR 182.11(2)(b) is met.

NR 182.11(2)(c) requires that the TMA be located so that upstream rainfall catchment areas are minimized. The TMA is located over several ridges which generally trend north-south. The TMA design provides for embankments above the existing topography except in the northwest and southeast corners of the TMA footprint. In these areas, the berms are formed by a cut slope as shown in Drawing No. 15. In both the northwest and southeast corners the drainage area above the TMA cell is very small and surface water will be diverted from the TMA cell because the top of the berm slopes away from the TMA and will form a ditch where it meets the cut slope.

NR 182.11(2)(d) Drawing No. 28, Details 1-28, 3-28 and 5-28 show that the staging berms, common berms (berms shared by two TMA cells), and final berms. Each of these berms has a shelf on which the tailings reclaim water and leachate pipes are located. The shelves are sloped at 2 percent toward the containment so that any discharge can not leave the site.

NR 182.11(2)(e) requires that where practicable, the waste disposal facility should consider staged reclamation. The reclamation of each TMA cell will occur as each cell is filled. Section 7.3 of this report provides details concerning the operational procedures for closing a TMA cell.

NR 182.11(2)(f) requires the Department to make specific findings of fact and conclusions of law regarding the marketability of the wastes. Section 10.3 of this report provides information concerning potential reuses of the tailings materials. The first involves using tailings for backfilling mined slopes. This reuse reduces the size of the TMA facility. The other options reviewed include the marketability of removing pyrite from the tailings. The details of these findings are provided in Appendix Q. The capital cost of constructing a smelter and acid plant to recover sulfur as sulfuric acid is not economically feasible, notwithstanding the additional impacts that may result to the environment. Therefore, it is requested the Department make a finding of fact and conclusion of law that the wastes are non-marketable.

NR 182.11(2)(g) requires that the mine waste facility minimize the discharge of environmental pollutants to the groundwater of the state. Section 6.17 of this report contains a preliminary assessment of the performance of the TMA cells in meeting the groundwater standards. This assessment indicates that the facility will meet all applicable groundwater standards at the compliance boundary during operation, and during the long-term care period. Final assessment of TMA performance and further details on compliance with the groundwater standards will be furnished in the groundwater modeling report which will be submitted at a later date.

NR 182.11(3) requires that a high priority be given to the selection of design and operating procedures for the mine, mill and the tailings management area which will provide for the reclamation of all disturbed areas and avoiding, minimizing and mitigating the risks of environmental pollution. NR 182.11(3) suggests when practicable, facilities and practices be selected which meet the requirements of NR 182.11(3)(a), (b) and (c). The following indicates how these objectives are met:

- NR 182.11(3)(a). The production of mine waste is minimized by returning tailings to the mines for backfilling of the stopes and returning mine waste rock to the mine. Section 4.9 of the MPA provides details concerning the tailings and mine waste rock returned to the mine.

- NR 182.11(3)(b). Only sulfide-bearing rock is milled to tailings so that all tailings will contain sulfides. The rock which will be mined for access to the ore body will be segregated into sulfide (Type I) and non-sulfide (Type II) bearing rocks based on the waste characterization testing which is discussed in Section 4 of this report. Type II waste rock will be placed in the TMA, and Type I waste rock will be processed and used as aggregate on the plant site and for the TMA construction.

- NR 182.11(3)(c). Section 4.1 estimates that approximately 25 percent of the total tailings will be mixed with portland cement and returned to the mine for use as cemented backfill. In addition, up to another 25 percent of the total tailings will be placed underground as uncemented tailings. At the completion of mining, it is estimated that approximately 90 percent of the mine void would have been backfilled.

The Type I and Type II mine waste rock as described in Section 4 of this report will be used as riprap material placed in the TMA cell, or will become part of the grading layer below the reclamation cover. Section 6.12 of this report describes some of the construction uses for the Type I waste rock excavated from the mine workings.

7 Operation Manual

7.1 General

This section contains the operation manual as required by NR 182.09(2)(b). The operation manual includes general project information, specification for site operations and construction, a construction quality assurance plan, a description of procedures for construction documentation, a general description of typical daily operations, phase development, site closure and long-term care. This section also addresses the inspections, record keeping and reporting requirements of NR 812.12 and 812.14 respectively. The general information required by NR 182.09(2)(b)1 is provided in Section 1.4 of this report.

Detailed construction specifications are not provided in this report. However, the construction materials for the major construction components are described, including specifications for soil materials. These specifications are provided to the level of detail possible and practicable at the time of submittal. Materials such as the geomembrane, geotextiles and others are purposely described with general specifications. It is CMC's opinion that such information is sufficient to analyze the environmental performance of the TMA facility components. CMC desires to use the best product available considering performance, availability and cost. Since some of the TMA construction materials especially the geosynthetics have seen rapidly changing technologies only general information is provided now and more detailed and specific data will be provided when the final engineering is completed and construction documents are prepared.

7.2 Specifications for Site Construction NR 182.09(2)(b)2.

The following is a listing of the general sequencing of construction which includes a description of the construction activity and an indication of the specification number and title which provides details concerning the quality control standards, materials and execution. The construction sequence described is for the initial site preparation and construction for TMA 1, Stage I. Each major cell construction event such as TMA 1, Stage I will occur over two construction seasons generally as follows:

Construction Year 2

- Clearing and grubbing of:
  - TMA 1 footprint;
  - construction storage area; and
  - soil and mulch stockpiling area.
- Site preparation, such as:
  - salvage marketable timber;
  - cut and pile unmarketable timber, stumps, and brush; and
  - cut and chip unmarketable timber and brush and stockpile in a mulch storage area within the construction staging area.

- Install temporary and permanent erosion control and surface water diversion structures as detailed in the MPA, Section 4.10, Erosion Control, in the general areas of:

  - downslope from TMA 1 construction area; and
  - downslope from construction staging area and stockpiles.

- Excavate and grade access road between plant and TMA and install culvert pipes and erosion control measures.

- Grub stumps, and major roots.

- Excavate and stockpile topsoil in stockpile area.

- Excavate and stockpile upper till soil in the construction staging area in the area designated for processing on-site soils.

- Excavate, place and compact till soils from TMA 1, Stage I in embankments (berms).

- Grade and shape to required subbase grades.

- Process and stockpile soils and aggregates for:

  - till filter layer;
  - drainage layer;
  - till protective layer;
  - riprap; and
  - crushed road base.

- Topsoil outside slopes of TMA berms.

- Seed, fertilize and mulch outside slopes of TMA berms.

- Install fencing, gates and other access limiting measures.

Construction Year 3

- Continually inspect, repair and/or replace as required all temporary and permanent erosion control structures as per Wisconsin Construction Site Best Management Practices.

- Inspect and repair all eroded areas on interior and exterior TMA 1 berm slopes.

- Prepare subbase grade for clay liner material by removing boulders and cobbles, regrading to subbase grade elevations, recompacting and proof rolling.

- Clay material contractor to load and haul and place clay material:

  - place clay on interior 3H:1V slopes and compact to meet project requirements;

- place clay on TMA base and compact to meet project requirements; and
- maintain clay moisture until geomembrane is placed.

- Geomembrane contractor to start preparation for geomembrane installation and:

  - preparation of geomembrane storage area in construction staging area;
  - unloading and storing of geomembrane in construction staging area;
  - deploying of geomembrane panels as per panel layout plan;
  - welding of geomembrane panels;
  - completion of required Quality Assurance (QA) and Quality Control (QC) activities including destructive and non-destructive testing and completion of required QA/QC reporting;
  - prepare construction documentation report as per department plan of operation review letter; and
  - cover constructed geomembrane with geotextile and processed till soils as shown in Drawing No. 28, Detail 6-28 and No. 29, Detail 1-29.

- Install leachate collection system including:

  - install leachate drainage layer from on-site stock pile except in area of leachate collection system piping and sideslope riser;
  - install perforated leachate collection system pipe;
  - install cleanout risers for leachate collection system piping;
  - construct and install Sideslope Riser (SSR) and sideslope riser sump;
  - backfill leachate collection system piping, SSR's and cleanouts; and
  - place riprap over area covered by the leachate collection system.

- Install tailings distribution pipelines on shelf on interior tailings berm slope including:

  - tailings distribution lines including all required valves, spigots and spigot pipes;
  - reclaim water return pipeline including required valves and fittings;
  - leachate transfer pipeline including required valves and fittings; and
  - a lined ditch or double encased pipelines where pipes are outside of the lined area.

- Construct required electric power service at TMA

- Construct, install and commission reclaim basin barge and pump

- Install and commission side slope riser pump and control

- Startup and commission all mechanical equipment including:

  - SSR pumps;
  - SSR liquid level transducers;
  - return water decant pump and controls; and
  - valves and controls.

Upon completion of the activities outlined above, and the submission and approval by the Department of the Construction Documentation Report the facility is ready to operate. The first step in starting TMA 1 is to fill the cell with a sufficient quantity of water to start the mill. This is approximately the amount of water needed to cover the high point in the center of a TMA 1 cell. Water would be pumped to TMA through the tailings pipeline and distributed in the cell by opening the two valves at the corners of the cells which are designated for end dumping (refer to Drawing No. 17). After the cell has been filled with water, tailings can be deposited in the cell by alternating the spigot lines as described in Section 6.2 of this report. Section 7.3 which follows provides a description of daily site operation.

7.2.1 Construction Quality Assurance

NR 182.09(2)(b)2.b. requires that a certification plan for initial site preparation be submitted to document that the in-field conditions encountered were the same as reported in the feasibility report and to document that the site was constructed according to the engineering plans and specifications submitted for department approval.

It is typical at the present time to prepare a Constructions Quality Assurance Plan (CQA Plan) for the construction of tailings cell and/or waste impoundments. A CQA plan typically consists of a plan for implementing inspection and testing activities including visual observations and photographic documentation, field testing and measurements, and laboratory testing. The CQA plan includes evaluation of test and other data for conformance with the project plans and specifications and the Department approvals. Appendix O contains a draft CQA Plan for the TMA construction and closure work. This CQA plan will undergo further development in the final engineering report.

Appendix O contains a CQA Plan for construction and closure of the TMA. CQA Plan addresses the following TMA components:

- all soil components of embankments, liners, drainage layers, erosion protection and control, measures;
- typical geosynthetic components including geomembranes, and geotextiles;
- typical piping components; and
- pumps and other mechanical equipment.

7.3 Specifications for Site Operation NR 182.09(2)(b)2

7.3.1 General

This section contains the specifications for the operation of TMA cells related to the deposition of tailings, tailings water management system, closure and long-term care and maintenance.

Details on closure, monitoring programs, long-term care and maintenance, etc. are contained in other documents, such as the EIR or MPA.

7.3.2 Tailing Distribution System

Section 6.2 of this report provides details concerning the operation of the sub-aerial tailings deposition system proposed for the TMA. The operating log provided as part of Appendix P of this report contains a list of actions required for operation of the tailings distribution system along with a list of items and/or conditions which must be checked during each shift.

7.3.3 TMA Decant Water System Operation

Section 6.2 of this report describes the general operation of the TMA decant water system. This system is designed with a skid mounted pump located on the interior berm slope above the tailings and a barge mounted suction line. The pump will operate automatically based on a signal from a pressure transducer located on the barge. The pump will operate to maintain a water level of from three to five feet in the free water pool in the approximate center of the TMA cell.

7.3.4 Leachate Collection and Disposal System

The design of the leachate collection and disposal system are located in Section 6.4 of this report. Operation of the leachate collection system is for the most part automatic and requires little interaction from the operator to ensure the system operates properly except for planned maintenance on the leachate extractions pumps and liquid level transducers in the Side Slope Risers (SSR). Details concerning the operation of these items are not known at this time since the operation and maintenance will depend on the type and manufacture of the equipment selected. The equipment, including the submersible pump, liquid level transducer, air-relief valves, check valves, sampling valves, etc., will be designed and selected in the final engineering report. Appendix P contains an operating log which contains a check list of operating procedures, checks and observations which must be made on the leachate collection system during each shift.

7.3.5 Surface Water Management System

Development of the operation of the surface water management system initially requires the identifying and assessing the existence of and potential for sources of contamination to surface water runoff. The TMA, as proposed herein, is designed in such a manner as to prevent or mitigate the opportunity for potential pollutants coming in contact with surface water which is not contained within the TMA. The design and operation of surface pollution controls include the following:

- containment structures for all pipelines carrying tailings or reclaimed process water (refer to Section 6.2.3);
- design of the TMA containment structure and operation which results in all tailings and Type II wastes rock being stored in a secure, lined facility. Details concerning the TMA liner and final cover systems are outlined in Section 6.3 and 6.6; and
- operations of the facility in a manner which prevents dusting by depositing the tailings in thin layers to maintain moist tailings beaches (refer to Section 6.2.6).

To implement the surface water management and surface water pollution prevention plan, Best Management Practices (BMP's) have been selected. The BMP's selected are processes, procedures, schedules of activities, prohibitions of practices and management practices which prevent or mitigate potential surface water contamination. The best management practices for the TMA include the following:

- good housekeeping, namely:
  - prompt cleanup of tailings spills outside the containment area of the TMA; and
  - regular cleaning of sediments from ditches, sedimentation runoff basins and other drainage structures.
- preventive maintenance, namely:
  - repair and maintenance of surface water management devices when necessary;
  - identification of surface water management systems which must be inspected on a regular basis; and
  - documentation of preventive maintenance through a record keeping system; example of which is provided in Appendix P.
- periodic visual inspections of surface water management systems and features which could impact surface water including:
  - access road and ditch between plant site and TMA;
  - pumps, pipelines, and pipeline containment areas, draindown sumps;
  - TMA berms and piping systems, spigot delivery and reclaim water system; and
  - TMA surface water management systems including features such as the tailings pool and reclaim pond, ditches, culverts, erosion control devices, sediment traps, sedimentation ponds, etc.
- development of a Spill Prevention Control and Countermeasures Plan (SPCC Plan). The SPCC plan will be submitted as part of the permit application package submitted later in the permitting process. The SPCC plan will include action plans to prevent and respond to spills in and around the TMA, which have the potential to impact surface waters;
- sediment and Erosion Control issues are addressed in the MPA which includes traditional surface water management practices and BMP's as listed in WDNR, 1989, (Wisconsin Construction site Best Management Practices Handbook, Publication WR-222-89);

- recordkeeping and reporting is discussed in Section 7.4 of this report. Appendix P includes a typical operation assignment log for the TMA. This log includes inspection of the surface water management systems; and

- employee training as it relates to tailings operation/maintenance will be undertaken by CMC during manpower build-up, prior to TMA operation.

The TMA will be regulated and permitted under Wisconsin Administration Code NR 216 and Federal Regulation 40 CFR, Part 122. These regulations required completion of the storm water discharge permitting process which includes submittal of the following plans:

- storm water pollution prevention plans;

- storm water management plan; and

- construction site erosion control plan.

The listed plans are anticipated to be submitted after completion of the final Tailing Management Area Engineering Report later in the permitting process.

7.3.6 Monitoring Systems Operation

A description of the environmental monitoring system operation is contained in Section 7 of the MPA.

7.3.7 Development of Subsequent Phases (TMA Stages)

Section 6.12 of this report describes the sequencing of the vertical stages of the TMA cells. The typical construction of these phases is described in Section 7.2.

7.3.8 Typical Daily Operations

The description of the typical daily operations for the TMA will be described in more detail when the Operation and Maintenance Manual is prepared for review by the WDNR. In general, typical daily operation information related to the TMA includes the following:

- the timetable for TMA cell development is presented in Table 6.12-3 of this report;

- the methods for containment of mine wastes including mine waste rock, tailings, wastewater treatment plant solids and laboratory waste are contained in Section 6.2 of this report;

- the hours of operation for the TMA are 24 hours per day, 365 days per year except for planned shut downs for maintenance and repairs and other contingencies;

- traffic routing is discussed in Section 6.10 of this report;

- drainage and erosion control are discussed in Section 4.10 of the MPA;

- TMA operation in adverse weather conditions are found in Section 6.2.6 of this report;

- fire protection equipment required for the safe operation of each piece of construction or maintenance equipment will be fitted. The TMA operation does not pose a special fire hazard, since the tailings are non-flammable and do not support combustion;

- the TMA operations will be manned on a shift basis by mill operating staff. These personnel will inspect the operating TMA cell on a regular basis. Operational adjustments to the spigots will be performed during those inspections. Equipment operating status will be monitored in the mill control room;

- potential dusting from the tailings during facility operation and construction will be prevented in the active TMA cell as detailed in Section 6.2.6.4;

- closure sequencing of filled area is discussed in detail in Section 7.3.9. The closure construction scenario is discussed in Section 7.3.9 of this report; and

- leachate system operation is discussed in Section 7.3.4 of this report.

7.3.9 Operational Requirements for Site Closure, NR 182.09(2)(b)2.e

Section 6.6.2 contains a detailed discussion of the sequence of events for closure of a TMA cell. The following is a listing of the general sequencing of the closure construction events which includes a description of the construction activity. Appendix O contains the construction quality assurance plan (CQA Plan) which provides details concerning the quality control (QC) standards, quality testing methods and execution required to document the closure work. The closure sequence which follows is for TMA 1, however other cell closures are expected to follow a similar sequence. As indicated in Section 6.6.2, to allow for tailings settlement a two-year closure construction scenario is anticipated as follows:

Closure Construction - Year 1

- TMA 1 has been filled to capacity through the ring discharge method of tailings placement. Tailing deposition will then switch to a central discharge as follows:

  - spigot tailings from north and east berms to form beaches sloping at 0.5 percent toward the southwest, driving entire pond to the southwest;

  - when spigotted tailings have reached the maximum fill elevation in the southwest corner pump free pond water to TMA 2 or reclaim pond; and

  - fill southwest cell area from south and west berms with beached tailings from TMA 2, mine waste rock, coarse tailings or glacial till.

- Allow tailings in the cell to settle for approximately 6 to 12 months with the following:

  - monitoring tailings surface for signs of oxidation; and

- treat tailings surface with a lime solution and/or bactericide if required to control sulfide oxidation.

Closure Construction - Year 2

- Place an approximate 3-foot grading layer of waste rock and on-site glacial till, which include the following:
  - place woven geotextile in areas with lower trafficability;
  - place additional fill listed above in areas with settlement; and
  - fill to required grades.
- Place, grade and compact grading layer to a minimum two-percent slope as shown on Drawing 28, Detail 2-28.
- Place, grade and compact the one (1)-foot thick clay capping layer, i.e.:
  - clay meeting the requirements of Section 6.6.3 will be hauled in from off-site; and
  - required Quality Assurance (QA) and Quality Control (QC) Activities as per CQA Plan (Appendix O) will be completed.
- Furnish, install and test geomembrane, including required QA and QC activities as per CQA Plan (Appendix O).
- Place, grade and test one (1)-foot thick drainage layer, including required QA and QC activities as per CQA Plan (Appendix O).
- Place, grade, install and test the three (3)-foot cover layer.
- Place and grade the six (6)-inch topsoil layer in accordance with Section 5, Reclamation Plan, of the MPA.
- Seed, fertilize and mulch closed area in accordance with the Reclamation Plan.

Upon completion of the above listed items the TMA 1 cell closure is completed. Following this, long-term care activities detailed in the MPA must be completed at the frequencies required.

7.3.9.1 Closure Sequencing and Schedule

Section 6.12 (Site Development and Phasing) describes the TMA development, phasing and closure of the four (4) TMA cells and eight (8) TMA stages. Table 6.12-1 provides the best estimate of the full dates and years of closure for the four TMA cells. The following is a summary of the closure sequence and closure completion year of the four TMA cells.

Cell	Operating Year	
	Start Closure Activities	Complete Closure Activities
TMA 1	Year 11	Year 12
TMA 2	Year 22	Year 23
TMA 3	Year 27	Year 28
TMA 4	Year 33	Year 34

Table 6.12-1 summarizes the operations and construction sequencing of each stage of the TMA cells.

7.3.10 Long-Term Care Information NR 182.09(2)(b)2.f.

The details of the long-term care information required by NR 182.09(2)(b) 2.f. and NR 132.08 (Reclamation Plan) are provided in Section 10 of the MPA.

7.4 Recordkeeping and Reporting, NR 182.14

7.4.1 General

NR 182.14 requires that CMC maintain an operating log which is open for inspection by Department personnel. This operating log will be available at the plant site office. The operating log will be completed each shift by the mill operators who will inspect the tailings pipeline, TMA cell(s) and reclaim pond every shift.

7.4.2 Operating Log, NR 182.14(1)(a)

The operating log for the TMA and appurtenant facilities will be recorded once for each shift during mill production. Appendix P contains a typical operating log which includes inspection of the following facilities:

- TMA operating cell;
- tailings delivery pipeline between mill and TMA, including drain down sumps;
- tailings delivery and distribution pipelines;
- TMA reclaim water decant system;
- TMA to process water pond decant water pipeline;
- reclaim pond;

- reclaim pond to mill return water pipeline; and
- roadways.

This operating log follows the order in which the inspection is expected to occur. Details based on the operation and maintenance required for the equipment selected in the final design will be added later.

7.4.3 Permanent Construction and Operating Records NR 182.14(1)(d)

CMC will maintain, at site, a permanent file for the TMA which includes the following:

- operations assignment log from each shift;
- operations and maintenance manuals for all mechanical equipment for the TMA, reclaim ponds and pipelines;
- as-built drawings and specifications for all major construction;
- daily construction observation reports;
- construction documentation reports for each phase of construction (refer to Appendix O for the contents of this document);
- plan of operation approval letter and all subsequent regulatory approvals;
- environmental monitoring results;
- operating and discharge permits;
- boring logs (WDNR Form 4400-122), Monitoring Well Construction Forms (WDNR Form-113B), Borehole Abandonment Forms (WDNR Form 3300-5W) and Well Information Form (WDNR Form 4400-89);
- records of any deviations or variations from the drawings, specification or conditions of approval for operation;
- records for any incidents which impacts operations (e.g., fires, weather events, etc.) and how the event was responded to;
- the current contingency plan;
- the current reclamation plan; and
- documentation that any spills or leaks were reported to the WDNR by phone immediately followed by a written report of the incident within five days.

As part of recordkeeping and reporting CMC will provide the WDNR the following information in an Annual Report:

- documentation of any spills or leaks;
- copies of quarterly environmental monitoring data results;
- an annual report which contains statistical summaries of annual and cumulative project data including:
  - waste characteristics;
  - leachate characteristics;
  - effluent characteristics;
  - baseline and background groundwater data;
  - baseline and background surface water data;
  - other environmental monitoring data;
  - results of verification procedures;
  - analysis of data trends; and
  - results for any changes in standards or parameters.
- any information required to be submitted by the operations approval letter or current operating permit.

This annual report shall be submitted within 90 days of the end of each operating year for the TMA.

7.5 Inspections, NR 182.12

Routine inspections of the closed and operating TMA cells may be undertaken by personnel or agents of the WDNR. CMC will designate a qualified representative to accompany the WDNR representative on these inspections. In addition, on a weekly basis the CMC representative will visually inspect where possible the TMA for the following:

- active TMA embankments for signs of erosion and condition of vegetation;
- pipeline from the plant to the TMA for wear and leaks;
- TMA internal piping for wear and proper operations of valves and spigots;
- leachate collection system side slope riser (SSR) for proper operation including:
  - normal operation of pump liquid level sensor;
  - normal on-off cycling of the pump in the SSR;
  - leaks in the SSR and associated piping;
  - exterior damage to protective casings on SSRs, cleanouts etc.; and
  - integrity of leachate transfer piping.

- the active and closed TMA cells will be inspected for the following:

  - condition of the surficial soils on the outside of the embankment to include an area 50 feet outside to of the embankment toe for signs of leakage or vegetative stress;

  - composite liner anchor and other liner areas not covered by tailings for signs of cover soil erosion, soil slumping or settlement;

  - condition of the pipeline bench within the TMA for evidence of cover soil disturbance, pipeline movement, pipeline wear and proper operation of valves; and

  - liquid level in the active TMA for compliance with the freeboard requirements.

- surface water management and erosion control facilities shall be visually inspected for the following:

  - accumulation of sediment in ditches, erosion control devices and sedimentation ponds;

  - integrity of erosion control devices, including riprap, erosion control blankets, silt fences, culverts, vegetative covers, etc.;

  - integrity of runoff basin structural components including inlet structures, outfall structures, containment berms, riprap at inlet and outlet etc.;

  - erosion of surface water management or erosion control structure;

  - culverts, let down structures, etc. for erosion at inlet or outlet, settlement, blockage and siltation;

  - runoff basin pond outlets for evidence of siltation, erosion or scouring above and below outlet structure;

  - after significant rainfall, inspect receiving water body for evidence of impacts from surface water discharge;

  - buffer area around the TMA embankments for evidence of impacts from surface water flows and discharges; and

  - buffer area around the TMA embankments, borrow and stockpile areas for evidence of impacts due to construction or operations activities.

The CMC representative shall record the results of his visual inspection on a form similar to the example provided in Appendix P. Dated photographs shall be supplied when appropriate to show defects and corrective actions.

7.5.1 Reporting and Correction of Defects, NR 182.12 (6) and (7)

The owner's representative shall notify the department of any potential defective conditions identified during an inspection at the earliest possible date. The owners shall document such defective conditions in his weekly operating report which is kept on file at the plant office. CMC will correct the identified defect at the earliest possible date. Upon completion, inform the department in detail as to how the corrective action taken was implemented, and what will be done to prevent recurrence of the incident.

7.6 Contingency Plan, NR 182.08(2)(i) and NR 182.075(1)(c)

CMC is preparing a contingency plan to avoid, minimize or mitigate human health or environmental damage in the unlikely event of an accidental or emergency discharge from the TMA facility. This contingency plan specifies the remedial actions which will be taken in the case of an accidental or emergency discharge, including remedial actions or interventions which will be taken if an analysis of groundwater indicates with a reasonable probability that any applicable standard is not being met. The contingency plans for the risks associated with the TMA will be discussed in Section 4.13 of the MPA. Section 4.13 will include an evaluation of the probability of a release occurring, a description of possible contingency measures which could be implemented and an assessment of the risk for each hazard.

8 Closure and Reclamation Plans, NR 182.15 and NR 182.09(2)(b)(2)(g)

The performance objectives of the closure design proposed herein are listed in Section 6.6 of this report. The closure design goals are met by the proposed design along with the project objective which is to close the TMA cells in a manner which protects human health and the environment.

8.1 Cost Estimate for Site Closure

The cost estimate for site closure required by NR 182.09(2)(b)2.g. is provided in Section 5.12 of the MPA. An estimate is provided in the MPA for each cell based on the design, closure scenario and cover materials proposed in this report. It has been conservatively assumed in these cost estimates that all materials deposited in a TMA after the final slurried tailings materials are soil materials borrowed from within the TMA site boundaries. This is a conservative assumption since, as stated earlier, mine waste rock could be used as part of the initial grading layer if any of these materials are available at the time a TMA cell is closed.

8.2 Closure Plans, NR 182.15

NR 182.15 requires that the closure requirements be incorporated in and made a part of the mine reclamation plan submitted pursuant to ch. NR 132 and referenced in the plan of operation. This closure plan contains a description of the sequencing of closure events, sequencing of closure for the four cells, construction quality standards, construction material specifications and construction methods for closure of a TMA cell. This report does not contain detailed information concerning the final reclamation activities related to reestablishing vegetation on the final cover, long-term care requirements or costs for closure or long-term care. Details concerning reestablishment and development of the finished surface for the TMA cells are found in Section 5.7 of the MPA.

8.2.1 Securing of Deposited Wastes

The proposed design as described in Section 6 of this report results in the complete encapsulation of all sulfide mine wastes in a composite liner system. This composite liner system consists of a thermoplastic geomembrane over a one-foot thick clay liner for both the facility liner and final cover. Means for protecting both the liner and the cover composites from environmental and other effects are incorporated into the design by providing a thick protective layer during operation and after closure. Refer to Section 6 of this report for details. The composite liner and thick soil cover will prevent human or animal contact with the mine waste and also to protect the environment.

8.2.2 TMA Cell Closure and Certification, NR 182.15(6) and (7)

Section 7.2.1 and Appendix O contain the CQA Plan for the initial site construction and closure construction. The CQA Plan includes a list of inspection activities that includes visual observations, field testing and measurements, laboratory testing and the evaluation of test data and measurements. The CQA Plan includes a list of planned observations, tests and measurements for each major component of the closure construction which when completed will show that the constructed reclamation cap meets the project specifications. After the construction is completed and all the testing required by the CQA Plan has been completed and evaluated, a construction documentation report is prepared. The construction documentation

report contains all information compiled during construction. This report, to be prepared under the seal of a professional engineer registered in the State of Wisconsin, will be submitted to the Department for the purpose of obtaining an operating permit for the TMA. Construction documentation reports typically include documentation that the construction work was constructed in substantial compliance with the department's plan approval and the project specifications.

The construction documentation report will include surveys to verify that the required final cover layer thickness is met and that the approved final grades have been achieved. Refer to Appendix O of this report for details concerning the CQA plan for the TMA closure.

9 Financial Assurance, NR 182.16 and NR 182.17

9.1 Financial Responsibility for Closure

To meet the requirements of NR 182.16, financial responsibilities for closure are outlined in Section 3 of the MPA.

9.2 Financial Responsibility for Long-term Care

To meet the requirements of NR 182.17, long-term care financial responsibilities are outlined in Section 3 and 10 of the MPA.

9.3 Waste Management Fund, NR 182.18

NR 182.18 requires that all licensed mine waste facilities pay the Department a tonnage fee, for each ton of waste received and placed in the facility, or a minimum waste management base fee of $100. CMC will provide the Department with the proof of financial responsibility described in S. 144.443(4) and (8). and therefore the waste management fee is based on NR 182.18(3)(c).

9.3.1 Waste Management Fund

It is CMC's intent to establish the annual mine waste volume by field measurement of the active tailings cell. The volume then would be converted to weight by multiplying the volume by the average density of 97 pounds per cubic foot (2619 pounds per cubic yard). Based on NR 182.18(3)(c) and Table 2 the following calculation is representative of a typical annual contribution to the Waste Management Fund. This figure will vary based on factors such as mining rate, ore production rate, percent recovery, etc. For tailings with a density of 97 pounds per cubic foot, the Waste Management Fund would be calculated for the average annual tailings generation rate as follows:

$$\frac{0.61 \times 10^6 cy}{year} \times \frac{27\ cf}{cy} \times \frac{97\ lbs}{cf} \times \frac{1\ ton}{2000\ lbs} \times \frac{\$0.015}{ton} \times 1.25 = \frac{14{,}977}{year}$$

or approximately $15,000 (1995 $) annually.

Assuming 28 years at a constant production rate of 0.64 million cubic yards and cubic yards (cy) per year and constant fee cost, the total fee for the 28 year site life would be:

$$\frac{\$15{,}000}{year} \times 28\ years = \$420{,}000\ (1995\ \$)$$

Type II waste rock likely will be placed in the facility primarily during the first three years of TMA operation. This estimate assumes the costs early in the facility life.

Total Estimated Waste Rock

$$100{,}000\ tons\ x\ \frac{1}{3\ yrs}\ x\ \frac{0.05}{ton}\ x\ 1.25\ =\ \$2{,}083.33\ (1995\ \$)$$

or approximately $2,100 annually for the first three years and approximately $800 annually for the remaining 28 years of production.

Wastewater treatment plant solids production is conservatively estimated at approximately 446,000 tons for the life of site (28 years). The annual waste management fee for solids then would be:

$$\frac{446{,}000\ tons}{28\ years}\ x\ \frac{\$.01}{ton}\ x\ 1.25\ =\ \$199.10\ (1995\ \$)$$

or approximately $200 annually.

The total waste management fee for the TMA during the initial three-year period is estimated at:

Tailings		Mine Waste Rock		WWTP Solids		
$15,000/year	+	$2,100/year	+	$200/year	=	$\frac{\$17{,}300}{year}$ (1995 $)

During the mine operations it is expected to be approximately $16,000 (1995 $).

CMC can expect to contribute on a typical year approximately $15,900 to $18,000 to the waste management fund.

9.3.2 Groundwater Fund

Additionally, §144.441(7) stats. assess a groundwater fee of one cent per ton for every ton of mine waste including waste rock, tailings and wastewater treatment solids. Based on the assumed mine wastes this fee could be calculated as follows for the first three years.

Tailings: (3 years)

$$3\ yrs\ x\ \frac{0.61\ x\ 10^6\ cy}{year}\ x\ \frac{27\ cf}{cy}\ x\ \frac{97\ lbs}{cf}\ x\ \frac{1\ ton}{2000\ lbs}\ x\ \frac{0.01}{ton}\ =\ \$25{,}200\ (1995\ \$)$$

Type II Waste Rock (3 Years)

$$100{,}000\ tons\ x\ \frac{0.01}{ton}\ =\ \$1{,}000\ (1995\ \$)$$

WWTP Solids (3 Years)

$$3\ yrs\ x\ \frac{446{,}000\ tons}{28\ yrs}\ x\ \frac{\$0.01}{ton} = \$500\ (1995\ \$)$$

The total for the first three years then would be:

$$25{,}200 + 1{,}000 + 500 = \$26{,}700\ or\ \$8{,}900/yr\ (1995\ \$)$$

For the remaining 25 years the groundwater fee could be calculated as follows:

Tailings

$$\frac{.61\ x\ 10^6\ cy}{year}\ x\ \frac{27\ cr}{cy}\ x\ \frac{97\ lbs}{cf}\ x\ \frac{1\ ton}{2{,}000\ lbs}\ x\ \frac{\$0.01}{ton}\ x\ 25\ years = \$200{,}000\ (1995\ \$)$$

WWPT Solids

$$25\ yrs\ x\ \frac{446{,}000\ tons}{28\ years}\ x\ \frac{\$0.01}{ton} = \$4{,}000\ (1995\ \$)$$

$\$200{,}000 + \$4{,}000 = \$204{,}000$ *or approximately* $\$8{,}160/yr$ (1995 $) *for the final 25 years.*

9.3.3 Summary

The total estimated fees for the Waste Management Fund (S 182.18) and Groundwater Fund are as follows:

First Three Years

Waste Management Fund)	+	Groundwater Fee	=	Total
$51,900	+	$26,700	=	$78,600 (1995 $)

The total estimated fees for the Waste Management Fund and Groundwater Fee for the final 25 years are:

Waste Management Fund	+	Groundwater Fee	=	Total
$400,000	+	$204,000	=	$604,000 (1995 $)

Total fees then are:

$451,900	+	$230,700	=	$682,600 (1995 $)

10 Alternative Design, Location, and Operations Evaluations NR 182.08

10.1 Evaluation of Alternative Site Locations for TMA

This section summarizes the information used in locating an acceptable TMA site. The approach used generally consisted of eliminating environmentally unacceptable land areas on the basis of location criteria in NR 182, and considering engineering and economic factors. The sections which follow address the history of the review of alternative locations for the TMA.

10.1.1 TMA Locations

Significant and exhaustive screening activity of potential TMA sites occurred over a five year period from 1978 to 1983. The environmental criteria applied in the 1980s are very similar to those used today and a comprehensive application of the same considerations results in the same conclusions as documented in Schuff (1986) and WDNR (1986). In the previous work, Areas 40 and 41 were selected as the final two TMA sites and Area 41 was judged to be the area of least overall environmental impact. Additional data collected during 1994 on biological resources expanded the knowledge base of the project study area and therefore required a re-evaluation of Areas 40 and 41 based on these new findings.

A summary review of the pre 1986 TMA site screening activity is provided in the following section. The additional biological data is then incorporated into the re-evaluation of Areas 40 & 41 in order to reassess which area presents the alternative with the least overall environmental impact.

10.1.2 Site Screening - Pre-1993 Evaluations

Site selection studies for the TMA occurred over a five year period (1978-1983), and included extensive environmental and engineering analyses. Primary screening criteria for site selection were facilities capacity, NR 182 and Federal wetlands criteria, NR 182 location criteria and setbacks and distance from cities or Native American communities.

Siting studies were conducted in a phased manner as shown on Figure 10.1-1 and included independent third-party review. The first phase started in 1978 with an evaluation of potential areas within 25 miles of the ore body. An early conclusion of this phase was that further efforts should be confined to an area within a 12-mile radius of the ore deposit because the probability of locating a more suitable area beyond the 12-mile radius that was both economically justifiable and would minimize overall environmental disturbance was judged to be negligible because of a longer pipeline and roadway corridors. Therefore, the Phase I studies were completed by concentrating efforts within the 12-mile radius of the ore deposit. The objective of Phase I work was to select the potential area(s) within a 12-mile radius of the ore body where a TMA could be developed and operated with the least potential effect on the environment and which was feasible from a construction and operating perspective.

The first step in the Phase I study was the identification of potential areas within the 12-mile radius of the ore deposit regardless of impacts to wetlands. Areas within 1,000 feet of lakes were eliminated. Ideally, a facility should be developed on an area using topographic containment to eliminate the need to construct large earth embankments to contain tailings. Such areas are in valleys and generally contain wetlands or streams which should be avoided, where possible.

Therefore, several areas were selected where topographic containment was not possible and where earth embankments would provide the containment volume.

The total number of areas investigated during the multiphased evaluation process was 42. Of the 42 areas, seven were eliminated after preliminary screening was completed leaving 35 areas for detailed screening. Table 10.1-1 and Figure 10.1-2 present information and the location of the areas considered in the overall evaluation.

In the second step of Phase I, several additional physical criteria were applied to the potential areas to further narrow the number for additional evaluation. The criteria were deliberately general so as not to eliminate potentially viable areas for detailed review. Areas falling into the following categories were eliminated.

- those less than 10.40 million cubic yards in potential capacity;
- those less than 0.5 mile from cities or towns; and,
- those less than 0.5 mile from Native American communities.

When the above criteria were applied, the screening process left 20 areas for further evaluation.

The third step in the Phase I work involved gathering additional site specific information for the remaining 20 areas and ranking them. Information was collected pertaining to environmental factors, land use, and hydrogeology. The areas were surveyed from the air and, in most cases, in the field. Soil borings were completed at most areas. A comprehensive scoring and ranking system was developed using environmental and engineering considerations. Phase I evaluations concluded that Areas 30, 40, and 41 be given closer scrutiny because they represented the areas with the smallest potential for environmental impacts and were feasible from an engineering and economic perspective. All three areas were sites where impacts to wetlands were minimized. None of the original 42 areas could be developed in a way to totally avoid wetlands.

The second phase of site screening for the TMA began in 1979 and overlapped the Phase I work. Phase II work involved an examination of the same geographical areas reviewed during the Phase I work. Like the Phase I work, the Phase II examination also involved an environmental review, but included a higher level of engineering analysis than that applied in Phase I. The first step in Phase II involved elimination of non-viable areas based on similar criteria used in Phase I. Areas that were known at the time to include lakes or be less than 1,000 feet from a lake; all areas below elevation 1,600 feet MSL, because these are predominantly wetlands; areas known at the time to contain special habitats or endangered species; and Native American lands and Nicolet National Forest lands were eliminated.

Several larger geographic areas were also eliminated from further consideration because of tailings pipeline corridor stream crossings, land use, undesirable topographic features, and distance from the ore deposit. Phase II studies concluded that Areas 40 and 41 were the best alternatives for location of the TMA from an environmental standpoint. A third area, designated as 30, was also evaluated further, but was judged less desirable than 40 or 41 due to proximity to STH 55 and the Mole Lake Indian Reservation, and the need for longer pipelines.

Table 10.1-1

Characteristics of Potential TMA Areas

Site Number	Distance from Mine (miles)	Total Area Evaluated (acres)	Estimated Storage Capacity yd^3 (million)
1	1.4	300	12
2	2.8	350	22
3	2.1	250	4
4	2.0	600	15
5	1.2	200	4
6	2.4	500	19
7	5.9	650	10
8	4.8	650	22
9	6.0	250	10
10	3.6	350	13
11	4.1	150	7
12	5.0	200	7
13	5.9	600	7
14	7.1	300	10
15	6.4	100	1
16	6.3	350	9
17	6.4	250	5
18	7.6	250	4
19	7.2	500	15
20	3.9	450	8
21	2.2	200	1
22	4.2	300	7
23	7.4	450	15
24	6.1	600	20
25	3.5	200	4
26	6.4	200	5

Table 10.1-1 *(Continued)*

Site Number	Distance from Mine (miles)	Total Area Evaluated (acres)	Estimated Storage Capacity yd^3 (million)
27	5.2	200	22
30	2.3	600	22
31	3.9	1,000	22
32	6.6	650	22
33	6.8	500	22
34	8.1	650	22
40	1.3	650	22
41	1.6	650	22
42	3.6	650	22

Source: WDNR; 1986

Prepared by: TJW
Checked by: GWS

At this point in the TMA site selection process, Areas 40 and 41 were judged to be essentially similar in overall acceptability. Detailed engineering proceeded for Area 41 because it was preferred from an engineering standpoint. Design studies which followed proposed a smaller, more-clearly defined site layout. The design concept was presented in 1985 and became the preferred project alternative in the 1986 WDNR Environmental Impact Statement. Independent assessments of environmental impacts and feasibility of all potential TMA sites performed by WDNR concluded that Area 41 met the "least overall environmental impact requirement".

An additional TMA siting study in the 1980s considered dispersing small, independent tailings cells throughout Areas 40 and 41 with the sole siting criteria of reducing impacts to wetlands. The proposed layout considered eight separate independent, but smaller sites, that would provide the necessary storage capacity for the tailings. This option, referred to as "Site 50", actually increased the area of direct impact by 26 percent over Area 41, not including additional roads and runoff basins that would be required to support these separate sites. The difficulties associated with constructing, permitting, monitoring and operating eight independent sites at eight separate locations, together with the significantly greater amount of land disturbed, and the smaller, perched wetlands that would be impacted regardless, caused the elimination of the "Site 50" alternative from further consideration.

10.1.3 Recent TMA Site Evaluation

In 1993 and 1994 CMC conducted additional biological and wetlands studies to update the baseline inventory and evaluation of the region around the project site. The results of this additional work are discussed in the EIR, Sections 3.9 and 3.10. The study area for terrestrial and wetland species was approximately 30 square miles. While field efforts focused on the

proposed plant site, TMA (Area 41), and the other areas potentially affected by the proposed project, baseline information was gathered from a much larger study area including the previously screened potential TMA site referred to as Area 40.

During the recent re-evaluation of Areas 40 & 41, the layout of the facilities on Area 41 was optimized to reduce the overall land area required, to further minimize impacts to wetlands and sensitive habitats, and also to improve operations. These activities reduced the overall area affected by approximately 40 acres, reduced the total wetland area affected by the TMA from approximately 58 to approximately 22 acres, and minimized the effect on the dense canopy maple upland forest where a state endangered species, the goblin fern, occurs. The reduction of acreage resulted primarily from the elimination of the on-site refuse disposal sites and a reduction in the size of the reclaim water ponds.

After reviewing the results of the recent programs, it was decided that a more rigorous comparison of Areas 40 and 41 would better summarize the collective current and historical environmental assessment information. Such a comparison of the two areas using updated information will also assess whether the conclusion made in the 1980's by the WDNR, that Area 41 was the TMA area with the least overall environmental impact, was consistent with an updated analysis.

Other sites listed on Table 10.1-1 were not reconsidered because the current estimated storage capacity requirement of 20- to 21-million cubic yards is similar to the 22-million cubic yard capacity requirement used in the prior evaluations.

Table 10.1-2 presents a summary of the comparative impacts between Areas 40 and 41. In order to be as definitive as possible in the comparison, a similar proposed layout (footprint) for the TMA on Area 41 was transposed onto Area 40 with the appropriate setbacks required by NR 182 from streams, lakes and private drinking water wells. Regarding Area 40, the two small streams flowing south from Area 40 to Rolling Stone Lake may be navigable, thereby possibly requiring exemptions under NR 182, if Area 40 were pursued further.

The total comparative score calculated in Table 10.1-2 clearly demonstrates that Area 41 is the TMA site with the least overall impact on the environment. The environmental parameters used in the comparative analysis are the same parameters used to describe the Environmental Setting (Section 3 of the EIR) so that all relevant factors are covered. The weighing of each parameter was selected to best represent the comparative importance of the parameter to the other parameters. If the relative importance was judged to be significant, a weight of three was assigned. If it was moderate, a weight of two was assigned, and so on. Ratings were assigned based on a relative comparison of Area 40 to Area 41, with the area having the least environmental impact for a given parameter assigned a rating value of zero. The site having the greater environmental impact relative to the other was assigned a rating of -1,-2 or -3 if the respective degree of difference was minor, moderate or significant. As the analysis illustrates, weighing judgements are essentially irrelevant because the total comparative score clearly demonstrates that Area 41 is the TMA location with the least overall impact on the environment even when weighing of all parameters is equal.

Table 10.1-2

Comparative Impact Assessment
Alternative TMA Areas 40 & 41

Environmental Parameter	Comparative Assessment	Parameter Weight	Rating 40	Rating 41	Total 40	Total 41
Historical-Archeology-Cultural Resources	• 40 visible from Spirit Hill, 41 is not • 40 is 3700 feet from Nii Win Spiritual Center at Oak Lake, 41 is 10,500 feet away	3	-2	0	-6	0
Transportation & Utilities	• 40 and 41 are the same distance from the plant site • road and pipeline will cross Sand Lake Road for 40	1	-1	0	-1	0
Climatology - Air Quality	• potential impact from dust from roads is greater at 40 due to proximity of residences and recreation waters	1	-1	0	-1	0
Geology - Soils - Groundwater	• soil permeabilities are lower and attenuation potential is greater at 41 than 40; private wells are much closer to site 40 • 40 needs greater borrow soils than 41	1	-1	0	-1	0
	• separation from groundwater is up to 20 percent greater at 41 than 40 • private drinking water wells at Little Sand Lake are 1200 feet from 40; 41 separated from wells at Ground Hemlock Lake by Creek and 5000 feet	3	-3	0	-9	0
Surface Water	• 40 must infringe on streams to south (navigability setback exemption may be required) • 41 is more consistent with code, in same drainage basin as plant site, 40 is not	2	-2	0	-4	0
Biology	• number of State Endangered Species 40 - 1 species, 1 location 41 - 1 species, 5 locations	3	0	-2	0	-6
	• number of other species of concern 40 - 0 species 41 - 9 species • affects on plants important to Native Americans is essentially the same for both sites	2	0	-2	0	-4
Wetlands	• lost acres to be compensated 40 - 42 acres 41 - 22 acres	3	-2	0	-6	0

Table 10.1-2 *(Continued)*

Environmental Parameter	Comparative Assessment	Parameter Weight	Rating		Total	
			40	**41**	**40**	**41**
Noise	• at 40 construction activity and operations will be 1200 feet from residences at Little Sand Lake and recreation waters at Oak and Little Sand Lakes; at 41 these minimum distances are 3 times farther to Little Sand Lake and Ground Hemlock Lake	2	-2	0	-4	0
Land Use - Zoning	• 40 has closer proximity to recreation lands/waters • Town of Ainsworth Resolution prohibits tailings disposal, Forest County does not • 40 crosses town and county boundaries; 41 does not	2	-2	0	-4	0
Aesthetics	• 41 will be completely screened from roads and residences; 40 more visible from Little Sand Lake especially during winter	1	-1	0	-1	0
Socioeconomics	• no comparative differences between sites	0	0	0	0	0
Native American Communities	• distances from tribal lands and community 40 - 8,000 feet 41 - 16,000 feet	2	-1	0	-2	0
	Total Score				**-39**	**-10**

Notes:

1. Environmental Parameters Consistent with Section 3.0.
2. Rating is comparative between Sites 40 and 41 only
   0 = baseline (site with least impact)
   -1 = minor difference
   -2 = moderate difference
   -3 = significant difference
3. Parameter Weight x Ratings = Total Score

Sources:
Dames & Moore, 1979
Golder, 1979
Golder, 1980
EMC, 1985a
EMC, 1985b
Foth & Van Dyke, 1994 field work
Forest & Langlade County Maps
WDNR, 1986

Prepared by: TJW
Checked by: GWS

Special consideration has been given to the biological segment of the evaluation. It has been noted that the goblin fern (state endangered) has been located within Area 41. However, Area 40 also contains favorable habitat for the goblin fern and one population was confirmed in that area. If a "super weighing" were to be assigned the biological parameter in the overall comparative assessment, it would have to be raised to an unreasonable level (to 18 from 3). This would mean that the biological function would be of more importance than the combined impact of Cultural Resources, Geology-Soils-Groundwater Protection, Surface Water Protection, Wetlands, Land Use-Zoning, and Native American Communities. Since both Areas 40 and 41 would disturb local populations of the goblin fern but would not endanger the continued survival of the species (other goblin fern populations have been identified) it does not seem appropriate to assign such a large weighing factor. The conclusion of the analysis is that Area 41 represents the area with the least overall impacts.

10.1.4 Advantages of Area 41

The work completed by Dames & Moore, 1979 and Golder and Associates, 1979 reviewed potential upland sites relative to meeting location standards and criteria such as depth to groundwater and bedrock, major soil types, etc. The general area referred to as Area 41 was selected as the site with the least total overall environmental impact. This is the same general area that the current TMA occupies.

Based on the above referenced work, the characteristics of Area 41 which make it advantageous over the other alternatives can be summarized as follows:

- Thick mantle of glacial till with a greater depth than other areas reviewed, i.e. till thickness = ranges from 60-200 feet.
- Low conductivity of glacial till soils. Hydraulic Conductivity has a range of 10^{-4} to 10^{-6} cm/sec.
- Greater distances from the groundwater table. Depth to groundwater ranges from 60 to 130 feet.
- Nearly flat groundwater table gradient.
- Greater length of groundwater flow path. Area 41 is located on a ground high with water movement to the west, north and east. Infiltration to the groundwater in Area 41 is expected to migrate in these three directions with the majority of the movement to the west.
- The depth to bedrock varies from 250 feet to over 300 feet.
- The average carbonate content of the glacial till soils is in excess of four percent. The till depths of 60-200 feet and the four-percent carbonate content provide additional attenuation capabilities to any potential affects of seepage from the TMA.
- Pipelines and roads leading to and from Area 41 can be constructed with minimum disturbance to wetlands.

- Earthwork requirements for Area 41 are such that a balance can be attained for the TMA. The overall earthwork requirements for Area 41 are less than Areas 30, 31, 40 and 42.

- Relative to land use issues, Area 41 is:

  - about one mile to the east and one mile to the west from Ground Hemlock lake and Little Sand Lake respectively which are the most prominent recreational areas in the vicinity; and

  - about 3.5 miles southeast of the Sokaogon Mole Lake Community and approximately five miles southwest of land owned by the Potawatomi Community.

Based on the review of the data collected earlier and the above listed characteristics, Area 41 is the best site for locating the TMA. The site selection work summarized above was reviewed by the WDNR in 1984, with the Department concluding that Area 41 "is a viable site which can preferably be developed into an environmentally acceptable disposal facility if adequate engineering design and construction of the site is accomplished."

10.2 TMA Location Within Area 41

As part of the finalization of the TMA design, consideration of a number of TMA siting criteria was required to select the TMA footprint with the least overall environmental impact within Area 41. This section presents the process used to select the TMA footprint and outlines the siting criteria used, the TMA footprint configurations considered during the development process, and summarizes a comparative impact assessment for the alternative TMA footprints within Area 41. This section also presents findings resulting from the 1993, 1994, and 1995 additional biological and wetland studies performed to update the baseline inventory of the region around the project site. In particular, the biological data base developed through recent studies has been summarized and included for consideration since this information was used during the development of the final location of the TMA within Area 41. An additional comparative impact assessment of a modified version of the currently proposed TMA footprint has also been prepared. This alternative was also assessed since this footprint would result in a reduction of the disturbance in the F15 watershed.

10.2.1 TMA Siting Criteria

As presented in Section 3, the TMA has been sited considering the location standards of NR 182.07(1) and NR 182.011(2)(a) and (b), Wis. Admin. Code, as listed in Table 3.1-1. Siting criteria used during the footprint development process were similar and included the following:

- Maintaining a 1,000 foot separation of the TMA disposal site from navigable lakes, ponds, or flowages, and a 300 foot separation from navigable streams or rivers.

- Maintaining a 1,200 foot separation from groundwater discharge location for an adequate compliance boundary.

- Minimization of direct and indirect wetland disturbance.

- Avoidance/minimization of disturbance within sensitive lake watersheds.

- Avoidance of rare, endangered and threatened species locations.
- Maintaining maximum groundwater separation.
- Minimizing TMA site exposure and using natural screening to the greatest extent possible.

10.2.2 Alternatives Evaluation

Following the confirmation of Area 41 as the least overall impact location for the TMA, additional biological investigations of wetlands and water bodies in the region around Area 41 were carried out and used to select the preferred TMA location within Area 41. During the process of evaluating TMA footprints, consideration of the TMA siting criteria listed above in Section 10.2.1 resulted in the selection of the TMA footprint with the least overall environmental impact. In particular, avoidance of rare, endangered and threatened species, avoidance of direct wetland disturbance, and minimization of impacts to sensitive lake watersheds were considered. Since there were numerous wetlands and several lake watersheds surrounding Area 41, the resulting TMA footprint had to balance impacts in order to select a site with the least overall environmental impact. The following subsections present a discussion of the impacts of the TMA footprint alternatives to wetland and biological resources, considering both original and updated information including that obtained during recent investigations summarized in Appendix P-1.

10.2.2.1 1986 TMA Footprint

The original TMA footprint resulting from the previous permitting efforts is shown on Figure 10.2-1. This TMA footprint would result in a total of 395 acres of disturbance, including 57.7 acres of direct wetland disturbance. Based on current biological data, this alternative would directly impact six known sites containing the state endangered *Botrychium mormo*. A total of five state special concern butterflies and one state special concern small mammal would also be affected. The proposed state special concern four-toed salamander was also observed within this alternative's footprint. The disturbance within lake watersheds would be approximately 183 acres for Deep Hole Lake, 130 acres for Duck Lake, and 37 acres for Skunk Lake. Approximately 56 acres of the Hemlock Creek watershed would also be disturbed. The F15 watershed disturbance would be approximately 27 acres.

10.2.2.2 Alternative TMA Footprints

Through a reevaluation of the quantity of recoverable ore, ore processing modifications as part of updated mining plans indicated that the TMA facility could be reduced in size by approximately 17 percent. This provided an opportunity to reduce the overall size of the 1986 TMA footprint and also provided an opportunity to consider its location within the envelope of the larger 1986 footprint. Three alternative footprints have resulted and are discussed in the following paragraphs. These footprints were developed based on the consideration of the previously mentioned siting criteria. As the envelope within which these footprints are located in what was already determined to be the most favorable location for the TMA based on a large-scale analysis, site specific locational criteria became more important. The alternative analysis that was completed looked at balancing impacts to one lake watershed versus another, direct impacts to wetlands versus indirect impacts to wetlands, and the protection of one habitat type versus another.

10.2.2.2.1 TMA South Alternative

The TMA South Alternative is shown in Figure 10.2-2. This alternative would result from initial modifications to the 1986 footprint to reduce impacts in the Deep Hole Lake watershed. This TMA footprint resulted in a total of 350 acres of disturbance, including 40 acres of direct wetland disturbance. Impacts to state protected plant and animal species would be the same as the previous alternative. The disturbance within the lake watersheds would be approximately 154 acres for Deep Hole Lake, 170 acres for Duck Lake, and 13 acres for Skunk Lake. Approximately 13 acres of the Hemlock Creek watershed would also be disturbed. The F15 watershed disturbance would be approximately 13 acres.

10.2.2.2.2 TMA North Alternative

The TMA North Alternative is shown in Figure 10.2-3. This alternative would result from modifications to avoid or minimize direct wetland disturbance and minimize disturbance within the Deep Hole and Duck Lake watersheds. This TMA footprint would result in a total of 354 acres of disturbance, including 23 acres of direct wetland disturbance. This alternative would result in impacts to five known sites containing *Botrychium mormo*. The same list of five state special concern plants as described previously would also be affected by this alternative. However, this alternative only affects one state special concern butterfly, and impacts to the wetlands known to have been occupied by the state special concern small mammal, butterfly, and the four-toed salamander have been minimized. The disturbance within the lake watersheds would be approximately 56 acres for Deep Hole Lake, 130 acres for Duck Lake, and 100 acres Skunk Lake. Approximately 68 acres of the Hemlock Creek watershed would also be disturbed. The F15 watershed disturbance would be approximately 100 acres.

10.2.2.2.3 F15 Setback Alternative

The F15 Setback Alternative is shown in Figure 10.2-4. This alternative is a modified version of the TMA North Alternative created to reduce indirect disturbance to wetland F15. This TMA footprint would result in a total of 360 acres of disturbance, including 28 acres of direct wetland disturbance. Direct impacts to state protected plant and animal species would be similar to the previous alternative. The disturbance within the lake watersheds would be approximately 71 acres for Deep Hole Lake, 135 acres for Duck Lake, and 77 acres for Skunk Lake. Approximately 77 acres of the Hemlock Creek watershed would also be disturbed. The F15 watershed disturbance would be approximately 77 acres.

10.2.3 Comparative Impact Assessment

This section presents a comparative impact assessment for the TMA alternatives within Area 41. The 1986 TMA alternative is not analyzed since it was eliminated during the TMA siting evaluation process. The environmental parameters used in the comparative analysis are the same parameters used to describe the environmental setting (Section 3 of the EIR) so that all relevant factors are covered. The weighing of each parameter was selected to best represent the comparative importance of the parameter to the other parameters. If the relative importance was judged to be significant, a weight of 3 was assigned. If it was moderate, a weight of 2 was assigned, and so on. Ratings were assigned based on the relative comparison of the TMA North and TMA South alternatives, with the alternative having the least environmental impact for a given parameter assigned a rating value of zero. The alternative having the greater

environmental impact relative to the other was assigned a rating of -1, -2, or -3 if the respective degree of difference was minor, moderate, or significant.

10.2.3.1 Comparative Impact Assessment TMA North and TMA South

In order to compare the TMA North and TMA South alternatives using updated information and evaluating all the environmental parameters needed to consider overall environmental impacts, a comparative impacts assessment has been prepared. Table 10.2-1 presents a comparative summary of the impacts between the TMA North and TMA South alternatives. The environmental parameters used are similar to the criteria used in the evaluation performed for TMA Areas 40 and 41 as presented in Section 10.1.

The presence of a State Threatened plant species in Duck Lake and a number of State Special Concern species in wetlands F66, F65, and F64 indicated that the final TMA footprint within Area 41 should be selected to minimize the impact on these resources (refer to Appendix P-1). The TMA North Alternative accomplishes this goal. This alternative reduces the impact to Duck Lake by 40 acres which indirectly also reduces the potential for impact to the State Threatened plant observed in the lake. Further, the TMA North Alternative reduces the impacts to wetlands F63, F64, F65, and F66 by approximately 12.1 acres. This savings of direct wetland impact was weighed against potential indirect impacts to wetlands to the north (i.e., wetland F15) that would be avoided by the TMA South Alternative. It was concluded that potential changes in surface water runoff volumes to wetland F15 of less than 15 percent (see February 12, 1996 Updated Wetland Surface Water Hydrology Evaluation Water Balance Modeling) resulting from the TMA North Alternative would be less of an impact than an additional 13 acres of direct wetland loss resulting from the TMA South Alternative. This is important from both a wetland acreage standpoint as well as from the standpoint of the protection of biological resources.

When the results of the updated biological data and the results of increased impact to lake watersheds are considered, TMA North is clearly the alternative with the least overall environmental impact. As the analysis illustrates in Table 10.2-1, weighting judgements are essentially irrelevant because the total comparative score even without weightings demonstrates that the TMA North alternative results in the least overall impact on the environment even when weighing all the parameters is equal.

10.2.3.2 Comparative Assessment of TMA North and F15 Setback Alternative

Although not considered during development of the TMA footprint, a comparative assessment of the currently proposed TMA footprint (TMA North) and an alternative TMA footprint which would provide a 200-foot setback from wetland F15 has been completed. The F15 Setback Alternative, is shown in Figure 10.2-4. The F15 Setback Alternative results in slightly less impact (i.e., 23 percent less) within the wetland F15 watershed at the expense of five additional acres of direct wetland impact. In addition to increased direct wetland impact, the F15 Setback Alternative increases the impact within the watersheds of wetlands F66, F65, and F64. State Special Concern species have been observed in each of these wetlands while no rare plant or animal species have been observed to the north in wetland F15. The comparative analysis parameters used to evaluate TMA alternatives TMA North and TMA South were also used for this comparative assessment. The total comparative score calculated in Table 10.2-2, with or without the application of weighting factors, demonstrates that, when compared to the F15 Setback Alternative, the TMA North Alternative is still the TMA footprint with the least overall impact on the environment.

Table 10.2-1

Comparative Impact Assessment
Alternatives TMA North and TMA South

Environmental Parameter	Comparative Assessment	Parameter Weight	Rating		Total	
			N	S	N	S
Historical-Archeology-Cultural Resources	• No known significant difference.	3	--	--	--	--
Transportation & Utilities	• No significant difference.	1	--	--	--	--
Climatology - Air Quality	• No significant difference.	1	--	--	--	--
Geology - Soils - Groundwater	• Soil permeabilities lower at TMA North.	1	0	-1	0	-1
	• Groundwater separation greater at TMA North.	3	0	-1	0	-3
	• TMA North requires less external borrow area disturbance and borrow materials.	1	0	-1	0	-1
Surface Water	• TMA North results in less significant impact on Duck Lake watershed and Deep Hole Lake watershed.	2	0	-2	0	-4
Biology	• TMA South increases disturbance of F66, I8, F65, F64 which have greater biological diversity.	3	0	-2	0	-6
	• TMA North increases indirect disturbance of F15 containing bur oak community.	2	-3	0	-6	0
	• Number of state endangered species: TMA North - 1 species TMA South - 1 species	3	--	--	--	--
	• Number of other species of concern: TMA North - 6 species TMA South - 8 species	2	0	-1	0	-2

Table 10.2-1 *(Continued)*

Environmental Parameter	Comparative Assessment	Parameter Weight	Rating N	Rating S	Total N	Total S
Wetlands	• TMA South increases direct disturbance to wetlands: TMA North - 23 acres TMA South - 40 acres	3	0	-2	0	-6
	• TMA North increases F15 watershed disturbance.	3	-3	0	-9	0
Noise	• TMA South increases proximity to Duck and Deep Hole Lakes.	2	0	-1	0	-2
Land Use - Zoning	• No significant difference.	2	--	--	--	--
Aesthetics	• Increased TMA exposure for TMA South.	1	0	-1	0	-1
Socioeconomics	• No significant difference.	1	--	--	--	--
Native American Communities	• No significant difference in proximity to Native American communities.	2	--	--	--	--
	Total Score				**-15**	**-26**

Notes:

1. Environmental Parameters Consistent with Section 3.0.
2. Rating is comparative between TMA North (N) and TMA South (S) only
   -- = equal impact for both alternatives
   0 = baseline (site with least impact)
   -1 = minor difference
   -2 = moderate difference
   -3 = significant difference
3. Parameter Weight x Ratings = Total Score

Prepared by: PAE
Checked by: RFS

Table 10.2-2

Comparative Impact Assessment
Alternatives TMA North and F15 Setback

Environmental Parameter	Comparative Assessment	Parameter Weight	Rating		Total	
			N	F15	N	F15
Historical-Archeology-Cultural Resources	• No significant difference.	3	--	--	--	--
Transportation & Utilities	• No significant difference.	1	--	--	--	--
Climatology - Air Quality	• No significant difference.	1	--	--	--	--
Geology - Soils - Groundwater	• Soil permeabilities - no significant difference.	1	--	--	--	--
	• Groundwater separation - no significant difference.	3	--	--	--	--
	• TMA North requires less external borrow area and borrow materials.	1	0	-1	0	-1
Surface Water	• TMA North results in less significant impact on Duck Lake watershed and Deep Hole Lake watershed.	2	0	-2	0	-4
	• F15 Setback decreases disturbance in F15 watershed.	3	-2	0	-6	0
Biology	• F15 increases disturbance of F66, F65, I8, F27 which have greater biological diversity.	3	0	-2	0	-6
	• TMA North increases indirect disturbance within F15 watershed containing bur oak community.	3	-2	0	-6	0
	• Number of state endangered species: TMA North - 1 species F15 Setback - 1 species	3	--	--	--	--
	• Number of other species of concern: TMA North - 6 species F15 Setback - 6 species	2	--	--	--	--
Wetlands	• F15 increases direct disturbance to wetlands. TMA North - 23 acres F15 Setback - 28 acres	3	0	-2	0	-6
Noise	• No significant difference.	2	--	--	--	--

Table 10.2-2 *(Continued)*

Environmental Parameter	Comparative Assessment	Parameter Weight	Rating		Total	
			N	F15	N	F15
Land Use - Zoning	• No significant difference.	2	--	--	--	--
Aesthetics	• Increased TMA exposure for F15.	1	0	-1	0	-1
Socioeconomics	• No significant difference.	1	--	--	--	--
Native American Communities	• No significant difference.	2	--	--	--	--
	Total Score				-12	-18

Notes:

1. Environmental Parameters Consistent with Section 3.0.
2. Rating is comparative between TMA North (N) and F15 Setback (F15) only
   -- = equal impact for both alternatives
   0 = baseline (site with least impact)
   -1 = minor difference
   -2 = moderate difference
   -3 = significant difference
3. Parameter Weight x Ratings = Total Score

Prepared by: PAE
Checked by: RFS

10.3 Alternatives To Control Oxidation Rates

10.3.1 General

The objective of evaluating alternatives was to select a site and a set of design criteria which best meet environmental criteria in the long-term with minimal maintenance and reasonable costs. The alternatives were evaluated taking into account the Acid Mine Drainage Control criteria as proposed by the Mine Environmental Neutral Drainage (MEND) Program.

Figure 10.3-1 is a reproduction from a MEND Document which has been adapted to the specifics of the Crandon TMA project. Figure 10.3-1 lists the Primary, Secondary and Tertiary Acid Mine Drainage (AMD) controls available for the TMA. Although listed as controls for AMD, these controls are also controls for sulfide mineral oxidation.

The sections which follow discuss the alternatives available for each control level and recommend the most suitable control alternatives.

10.3.2 Primary Controls Alternatives

The primary control technologies for acid generation control provide for the control of the acid generation processes and include the following:

- sulfide removal by waste segregation and blending;
- exclusion of oxygen through sub-aqueous deposition and final cover and seals to exclude oxygen;
- alkaline additives; and
- bactericides.

These acid generation primary control technologies are evaluated in the sections which follow taking into account the CMC tailings characteristics and specific site conditions. Primary controls are desirable in that they prevent the oxidation of sulfide minerals and minimize or eliminate the need for either secondary or tertiary control measures.

10.3.2.1 Sulfide Mineral Removal

A review of the bulk chemical analyses of the tailings composites used in the waste characterization studies (see Section 3.5.5 of the project's EIR) shows that the tailings are primarily composed of the minerals pyrite (iron disulfide) and quartz (silicon dioxide). The tailings also contain small amounts of clay, ferromagnesian and carbonate minerals.

Microscopic analyses of several tailings samples confirmed the assertion that pyrite is the most abundant sulfide mineral in the tailings, comprising more than 45 percent by volume of the tailings. Trace amounts of sphalerite (zinc sulfide), chalcopyrite (copper-iron disulfide) and galena (lead sulfide) were also detected in the tailings.

Due to the abundance of sulfide minerals in the tailings, segregation of sulfide minerals (primarily pyrite) is not a practical alternative for the prevention of acid generation within the TMA. The following provides an evaluation of the technology and economics of pyrite removal.

Market Access International complete a report titled: "Crandon Project Pyrite Marketing Update" in April, 1995 (See Appendix Q). This report considered information presented in the 1981, Davy McKee Pyrite study as well as the 1984, Trevino critique, and the 1986 Coefield update. This re-examination of current market conditions focused on the most viable products identified in earlier studies; sulfuric acid, iron pellets, and diammonioum phosphate, but also reviewed earlier marginal product alternatives in view of the time lapse since initial work. The evaluation considers both processing and marketing of the pyrite fines and the total pyrite expected to be generated.

It was found that the conclusions of earlier studies remain valid with regard to the lack of viable markets for pyrite concentrates, elemental sulfur, liquid sulfur dioxide, phosphoric acid, and byproduct gypsum.

The pertinent microeconomic market conditions for sulfuric acid from by-product pyrites have improved from the time of the original work, and the viability of marketing this product improves if it could be processed at the Crandon plant site rather than a remote processing center. This option would require construction of a plant and installation of a roaster and wet gas cleaner to produce sulfuric acid.

Capital cost for on-site production of sulfuric acid and by-product iron pellet plant were estimated at $169 million (1980 dollars) in the Davy McKee report. The Gross National Production Implicit Price Deflator was applied to this estimate, resulting in a 1994 capital cost to construct the plant of approximately $320 million.

Generally, the environmental impacts of on-site pyrite processing would:

- include more land disturbance to locate the roaster, wet gas cleaner, and sulfuric acid and pelletizing plant, including more potential for wetland disturbance;

- include the new facility being classified as a major air emitter;
- not eliminate waste disposal needs (e.g., still have potentially reactive slag, gypsum, etc.); and
- increase railroad and highway traffic to transfer raw materials and waste products to market or disposal sites.

Based on the environmental impacts and capital cost requirements, on-site pyrite processing and sulfuric acid production are not considered to be viable options and have been eliminated from further consideration a part of this project.

10.3.2.2 Exclusion of Oxygen Through Subaqueous Deposition

An alternative to land-based disposal of tailings and mine waste rock is placement below water referred to as subaqueous placement. This method includes placement of tailings into natural waters (eg. lakes and flooded stream valleys), flooded mine works or man-made impoundments. Available evidence indicates that subaqueous placement is currently a promising abatement measure available for the control of sulfide oxidation and consequent acid generation. The Crandon project will return approximately 50 percent of all the tailings back underground and below the water table to fill mined out areas.

10.3.2.3 Review of Potential Flooded Sites

Past work by Dames & Moore, 1979; Golder, 1979, and others focused on the review of potential upland sites. However, since this work was completed, the literature concerning the siting of tailings facilities has pointed out many advantages of sub-aqueous tailings placement. As a result, a review of the potential flooded sites in the vicinity of the CMC mine/mill was made. The lakes in the immediate area of the plant were not considered because Wisconsin state statutes prohibit the destruction or alteration of such waters. The review of potential sites for flooded tailings was therefore restricted to wetland and lowland areas where the water table is at or near the ground surface. Figure 10.3-2 illustrates two potential areas where a flooded TMA could be constructed. Based on a review of these sites, the wetland area between Mole Lake and Rolling Stone Lake along upper Pickerel Creek was the most suitable potential site for a flooded tailings site because it was surrounded by uplands to the southwest and northeast and the valley could be dammed on both ends to create an impoundment for tailings. Based on the physical characteristics of the site, it was concluded that it is possible to design and construct a tailings management facility in this area with present design and construction technology. However, this option was judged impractical and unacceptable for the following reasons:

- construction would result in the destruction of a large track of wetlands (approximately 400 acres) located in a corridor between two lakes, Mole Lake and Rolling Stone Lake;
- Wisconsin Administrative Code NR 182 requires selecting of the site which has the least impact to wetlands and adjacent water bodies;
- U.S. Army Corps of Engineers guidelines on wetland impact minimization would not be met; and

- the Town of Ainsworth has passed a resolution which prohibits siting mine waste disposal facilities within the town boundaries.

Although flooded placement is an effective option for preventing acid generation in the long-term, the available site(s) suitable for flooded placement are judged impractical and unacceptable. Flooded placement is eliminated from consideration because suitable sites do not fit into the regulatory frame work for permitting mine waste disposal sites in the State of Wisconsin and the environmental consequences of doing so are unacceptable.

10.3.2.4 Man-Made Flooded Impoundment

Since available evidence indicates that water cover provides a secure method of acid generation control consideration was also given for the construction of a man-made flooded impoundment at an upland site. Therefore, the present site 41 was considered in the evaluation of the alternative. The design for a man-made flooded impoundment has as its primary objective keeping the tailings covered with water in perpetuity. The other major design features may include the following:

- leachate collection system part way;
- composite liner;
- riprap covering the interior slopes;
- composite cap; and
- wetland cover over the composite cap.

A man-made flooded tailings impoundment may have the same advantages as those discussed in Section 10.3.2.2. Long-term sulfide mineral oxidation is controlled, in fact virtually eliminated, so long as the water level (flooding) is maintained. Since sulfide mineral oxidation is strongly inhibited in the impoundment, acidic drainage and metal solubilization do not occur, thus practically eliminating the requirement for short- or long-term leachate migration control;

This design consideration presents a number of disadvantages including:

- process water seepage must still be controlled;
- increased percolation from the site due to flooding may lead to increased migration of sulfate and metals from the process water;
- a reliable water source must be available to provide a continuous water cover over the tailings;
- for the proposed TMA site (Site 41), water must be pumped from wells or surface water sources to maintain the continuous water cover required to maintain the subaqueous conditions and long-term maintenance of pumping facilities is required to maintain the integrity of the system;
- subaqueous tailings placement results in lower tailings density resulting in:
  - approximately 20 percent more land area required for the tailings impoundments; and

- an approximate 20-percent increase in construction costs.

- additional operating and long-term care costs result from the operation and maintenance of pumping equipment.

Based on the maintenance requirements and costs associated with maintaining the groundwater pumping system in perpetuity, this alternative was deemed impracticable.

10.3.2.5 Bactericides

Bactericides have been used to prevent bacteria from enhancing the inorganic acid generation rate. Bactericides are typically applied to tailings as a liquid by using hand or equipment-mounted sprayers, depending on the trafficability of the tailings. Although bactericides have been shown to be effective in control of the biological oxidation process, they are not effective in controlling the inorganic oxidation process and are a relatively short-term solution. Such agents in a rubberized pellet form provides a slow release of the bactericide, which may be effective for a few years. Because of their limited life, bactericides do not provide a long-term solution for controlling acid generation and are not considered an effective primary control mechanism. However, bactericides may be used to reduce oxidation rates for temporary control of acid generation initiation during short-term (six month) shut-downs.

10.3.2.6 Waste Conditioning (Base Additives)

The tailings that will be generated during the operation of the mine have the potential to become acid-generating. The rate of oxidation of sulfide tailings can be controlled by the addition of acid-consuming alkaline minerals or neutralizing materials such as finely crushed limestone (calcium carbonate) or lime (calcium hydroxide). These base materials are typically used in a solid form and mixed with rock or tailings prior to final deposition. The effectiveness of the addition of base materials, as stated in BC Acid Mine Task Force, 1989, depends primarily on:

- the movement of water through the system;
- nature of contact of acid rock or water with neutralizing additive (degree of mixing);
- the proportion of excess neutralizing material; and
- the type and purity of the neutralizing (base) material.

For application to tailings, the base materials typically are applied having a grain size which is similar to but slightly coarser than the tailings. Mixing can be accomplished either by mechanical means after tailings deposition or by adding the material into the slurry prior to pumping the tailings to the tailings management facility. Regardless of the method, proper mixing is required to ensure close proximity of base material to the acid-generating reaction site.

In order to neutralize all of the potential acid that theoretically could be produced by oxidizing all of the sulfide sulfur within the tailings, would require the addition of 65 pounds of crushed limestone per one percent sulfide sulfur per ton of tailings (65 lbs of limestone/1% S/ton tailings). Since the tailings contain more than 20 percent sulfide sulfur, this would require the addition of more than 1,300 lbs of crushed limestone per ton of tailings. This alternative would result in the doubling of the TMA cell volumes. Therefore, the addition of finely ground limestone sufficient to neutralize all of the acid that could potentially be produced as a consequence of oxidation of all of the sulfide sulfur is neither economical nor practical.

The Crandon tailings contain approximately one to four percent calcite, which is sufficient to neutralize initial acid product production resulting from sulfide oxidation. This neutralization is observed in the behavior of the tailings in the kinetic test, where the onset of acidic conditions has, at the time of writing, been prevented for a period of seven months under the moderately accelerated acid generating conditions existing in the laboratory. The addition of more neutralizing capacity in the form of finely washed limestone to the outer tailings zones where oxidation will initiate will assist in avoiding or reducing the potential for acid generation.

10.3.2.7 Oxygen Exclusion by Dry Cover

The primary option selected by CMC for acid generation control includes a dry cover, which will exclude oxygen entry from above by incorporating a geomembrane. Since the tailings deposit is perched above the water table it is also possible that oxygen could enter the tailings through the exterior berms, and to a much lower extent, the base of the tailings. This will be prevented by the geomembrane in the liner. If the geomembrane cover and liner deteriorate, it is possible to maintain or repair the cover geomembrane. Further, if any oxidation of the underside of the tailings occurs yielding contaminants to the seepage water, this contaminant load will be very small. This is because total flux of oxygen to the underside of the tailings will be very low, resulting in a low rate of oxidation. Thus the long-term control of solute migration will include additional secondary controls, such as a leachate collection system.

10.4 Secondary Control Alternatives

10.4.1 General

Earlier in this report, the three elements and the pH environment which are necessary for rapid acid generation and high contaminant concentrations to occur are discussed. The elements necessarily include the presence of sulfide-bearing rock or tailings, water and oxygen and a depressed pH for rapid acid generation, and high contaminant solubility. Sulfide removal from the tailings mass is not a practicable solution. Lime addition to the tailings will, along with the natural buffering minerals in the ore avoids or reduces the potential for acid generation, but will not prevent ultimate acid generation. Oxygen entry cannot be completely controlled over the long-term.

10.4.2 Final Covers and Seals to Exclude Infiltration and Oxygen

Control of infiltration of water (Figure 10.3-1) by means of low permeability covers or caps can be used to reduce the rate of contaminant migration. For this reason, CMC has decided that a primary design goal is to provide a final cover system which has a hydraulic conductivity which is less than or equal to that of the facility liner.

A composite final cover has been selected as a practicable alternative for assisting in a) the delay of the onset of acid generation and limit the contaminant generation rate by limiting oxygen entry into facility, and b) the reduction in contaminant migration rate by reducing infiltration and through flow. Alternative cover configurations are discussed in Section 10.4 of this report, and the selected final cover performance and design are discussed in Section 6.6.

10.4.3 Diversion of Surface Waters

Diversion of surface water can be accomplished by construction of diversion berms or ditches to re-route surface water, or selection of an upland site which is not subject to high run on.

In the site selection process, primary consideration was given to upland sites. The current site was selected as the alternative with the least overall environmental impact. This site is located over a group of northeast-southwest trending ridges which are generally high points relative to the surrounding topographic features. In addition, the design calls for exterior berms that will provide for the diversion of surface water from cut slopes. The location of the TMA on an upland area and the proposed exterior berm design are effective in diverting surface water runoff from the TMA and as a result the secondary control for surface water diversion is satisfied.

10.4.4 Diversion and Isolation From Groundwater

The objective of diversion and isolation from groundwater is to minimize groundwater contact with potential acid generating material or water which contains other products of sulfide oxidation. Diversion and isolation from groundwater can be accomplished by intercepting of groundwater before it contacts the waste and site selection to avoid the potential for groundwater to contact the waste.

In the site selection process primary consideration was given to selecting a site which had the maximum separation distance from the groundwater. The current site was selected because it has a groundwater separation averaging 43 feet and a thick mantle of till 60 to 200 feet below the TMA.

10.5 Tertiary Controls Alternatives

The final level of control (tertiary control) involves the collection and treatment of contact drainage before it impacts the environment, which is primarily groundwater around the TMA site. Collection and treatment is widely applied as a remediation measure when acid drainage has exfiltrated a tailings impoundment, waste rock pile, etc. Treatment measures can either be active systems (which require continuous operation) such as chemical treatment or passive systems intended to function with minimal intervention by man. An example of an active system is a mechanical chemical treatment plant, which includes a treatment process such as lime neutralization to treat acid drainage prior to discharging it to the environment. Active treatment systems typically require constant human intervention for plant, operation and maintenance. An example of a passive treatment system could be a wetland into which acid drainage is released and natural processes result in the amelioration of effluent water quality. Amelioration of water quality in a wetland occurs as a result of a combination of natural process including:

- settlement of solids due to low velocity sheet flow which is typical of waste flow in wetlands;
- sulfate reduction by sulfate reducing bacteria releasing hydrogen sulfide(H_2S) to precipitate metals, and bicarbonate (HCO_3) to neutralize the effluent; and
- metals absorption and fixing by organic matter.

As stated earlier, tertiary control measures are typically used after leachate has escaped from the impoundment. The primary goals of the proposed design are to minimize the potential for short-

and long-term sulfide mineral oxidation, and to prepare a facility design which will provide for the long-term containment of the waste. Accordingly, tertiary control measure are not considered as desirable options in the design of the TMA. However, tertiary controls are considered as potential options for remediation in the unlikely event that seepage of unacceptable quantity and/or quality occurs. The soils underlying the TMA facility area are such that an efficient and effective interception system could be installed, if needed.

10.6 Summary of Control Measures

Table 10.6-1 contains a summary of the control measure evaluated in the previous subsections along with summary comments on the suitability of each control measure, how they will be implemented and a qualitative description of their long-term performance. In addition, those measures which will be included in the TMA are referenced as to the section in this report where the design rationale and details are found.

10.7 Alternatives for Major Design Components NR 182.08

10.7.1 General

The major design components were evaluated relative to their technical suitability for use in the design of the TMA. The components evaluated in this section include the following:

- composite liner, consisting of soil component, geomembrane component, and protective layer;
- composite cap, consisting of soil component and geomembrane component; and
- leachate management, consisting of leachate collection and removal system or no leachate collection systems.

10.7.2 Liner and Cap Alternatives

In the evaluation of the liner/capping alternative, consideration was given to current liner/capping technologies which provide a high level environmental protection. The basic components of the liner/capping system, which are assumed for all alternatives reviewed, include the following:

- composite liner system include from top to bottom:
  - 18-inch protective soil layer;
  - 60-mil geomembrane liner;
  - 12-inch low permeability soil component; and
  - prepared subgrade.

Table 10.6-1

Acid Generation and Collection, and Treatment Technologies for TMA

Controls	Control Measures Description	Suitability	How Control Will be Implemented/Referenced in Report	Long-Term Performance[1]
Primary Controls	Sulfide removal by waste segregation and blending.	Unsuitable due to the economics of pyrite removal and additional environmental impacts associated with construction of a smelter and acid plant.	NA	May be suitable for long-term control. Not directly vulnerable to long-term stability.
	Exclusion of oxygen through subaqueous deposition.			Suitable for long-term control. Man-made impoundments dependent on long-term stability.
	- Subaqueous deposition in flooded site.	Unsuitable under of Wisconsin statutes and due to environmental concerns.	NA	
	- Subaqueous deposition in man-made impoundment.	Unsuitable due to need for long-term maintenance of water levels.	NA	
	Waste conditioning (base additive).	Complete prevention is uneconomic. Partial application to Crandon Project.	See Operating Plan, Section 7.3	Suitable for partial control. Uneconomic for long-term control.
	Bactericides	Unsuitable for long-term acid drainage control, effectiveness limited to 2-3 years.	NA	Suitable for partial, short-term control.
	Covers and seals to exclude oxygen.	Suitable cover designed to substantially reduce oxygen.	Design final cover to exclude oxygen from tailings mass. Refer to Section 6.6.8.	Suitable for partial long-term control. Effectiveness entirely dependent on long-term stability.
Secondary Controls	Covers and seals to exclude infiltration.	Suitable design cover to substantially reduce infiltration.	Cover design will have hydraulic conductivity less than the facility liner. Refer to Section 6.6.	Suitable for long-term control. Effectiveness dependent on long-term stability.
	Divert surface water.	Suitable.	Selected site located predominantly in an upland area for which surface water diversion is not a concern. Refer to Section 5.1.	Suitable for short-and long-term control. Maintenance required in long-term for sites with considerable run-on potential.
	Intercept or isolate from groundwater.	Suitable.	Selected site located in an upland area with groundwater separation of 60-130 feet. In addition, a composite liner is provided for isolation of waste from groundwater.	Suitable for short-and long- term control. Maintenance required in long-term for sites where wastes are placed below groundwater.
Tertiary Controls	Collection and treatment of acid mine drainage. - Active - Passive	Not recommended as part of primary prevention, considered as a contingency remedial measure.		Active - Suitable but not ideal for long-term control due to high operation and maintenance costs. Vulnerable to long-term disruptive forces. Passive - May be suitable for long-term control. Vulnerable to long-term disruptive forces.

[1] After BC Acid Mine Drainage Task Force Technical Guide, Volume 1, Table 9.1-1.

Prepared by: REM
Checked by: AMR

- composite capping system including from top to bottom:

  - 42-inch protective cover soil;
  - geotextile;
  - 12-inch soil drainage layer;
  - 60-mil geomembrane liner;
  - 12-inch low permeability soil component;
  - minimum 36-inch soil grading layer; and
  - tailings layer.

The above-listed components are standards to all the alternatives reviewed, however the thicknesses may vary depending on the design needs. For the liner/capping system, alternatives were analyzed for the soil component of the composite liner and alternatives for drainage layer (in the final cover only). The selection rationale for each component is as follows.

10.7.2.1 Composite Liner Components

- 18-inch protective layer

  - USEPA (1988) recommends a protective soil cover of at least 18 inches for protection of a flexible membrane liner (FML or Geomembrane) from mechanical damage. The 18-inch soil layer is also considered adequate to protect the geomembrane from damage from wildlife and human traffic which may occur in the lined area.

- 60-mil geomembrane (Liner)

  - The chemical compatibility of the synthetic membrane with the waste leachate must be determined. Koerner (1986) provided a chart indicating the chemical resistance of common synthetic membranes. Polyethylene (PE) and other thermoplastic geomembranes show generally good resistances to inorganic bases, heavy metals and salts. The 60-mil thickness was selected because it has become the standard for municipal solid waste and hazardous waste cell liners in Wisconsin and other states. Additionally, the 60-mil thickness has been found to be better suited for field application than thinner or thicker materials. Field welding is relatively easy and it is possible to maintain a high level of confidence in welded joints, which is of great importance in assuring containment within the liner area.

- 12-inch low permeability soil component (Liner)

  - The 12-inch thickness for the soil component of the composite was selected based on meeting the facility groundwater performance standards required by the Wisconsin Administrative Code. Four different alternatives were considered for the soil component which are evaluated later in this section.

- Prepared Subgrade

  - The prepared subgrade is proposed to provide a stable surface on which to compact the low-permeability soil component. A thoroughly and uniformly compacted subgrade will minimize the possibility of differential settlement. Additional details concerning the specification for subgrade preparation and contained in Section 6.3.2.

10.7.3 Composite Cap Components Alternative

- Protective Soil - The function of the protective layer is to protect the clay and geomembrane components of the cap from environmental effects. For the purpose of this discussion the drainage layer is included since it is a part of the soil above the geomembrane. The protective soil layer is 54 inches thick which includes six inches of topsoil, a 36-inch rooting zone, and a 12-inch drainage layer.

- The 36-inch rooting soil layer was selected because this thickness offers optimal conditions for moisture storage, availability and drainage which reflect naturally occurring conditions on similar soils in undisturbed areas of the project site. The total 54-inch thickness meets the regional depth of frost penetration, which protects the geomembrane and soil component from physical weathering caused by freeze/thaw and wet/dry cycles. Refer to Section 6.6 for design details.

- One-Foot Drainage Layer - The one-foot drainage layer was selected to remove percolating water that has infiltrated through the protective soil layer after surface run-off and evaportranspiration losses. By removing water which may contact the low permeability layers, the potential for leachate generation is further diminished. Section 6.7 of this report contains quantitative information concerning the effectiveness of the selected cap.

- 60-mil geomembrane - The 60-mil geomembrane was selected for the same reasons stated earlier.

- 12-inch Low-Permeability Soil Component - The 12-inch low-permeability soil component was selected to provide a natural material barrier to minimize infiltration and serve as a backup to the geomembrane.

10.7.3.1 Analysis of Alternative

Early in the design process CMC completed an analysis of alternatives for the soil component of the composite liner system. The four alternatives considered are as follows:

- bentonite amended soil;
- fines-amended glacial till soils;
- geosynthetic Clay Liners (GCLs); and
- native-clay soils.

Bentonite-amended soils consist of on-site glacial till mixed with bentonite. The till soil is processed to remove oversize material (i.e., greater than three-quarters inch), then mixed with

two percent or less (by weight) bentonite in a mixing chamber at a moisture content wet of optimum. The bentonite soil mixture can then be placed and compacted.

Fines-amended soils consist of on-site glacial till soils processed to removed oversize material (i.e., greater than three-quarters inch). These processed soils are then mixed with fines [material passing the number 200 sieve (P200)] obtained from screening and washing of till soils. It is estimated that the P200 of the fines-amended till would need to be approximately 30 percent, or greater, to obtain the desired hydraulic conductivities. Upon completion of mixing of the fines with the processed till, this soil mixture could be placed, moisture conditioned, and compacted.

Geosynthetic Clay Liners (GCLs) are manufactured liners typically consisting of a bentonite clay component encapsulated in woven or nonwoven geotextiles. The GCLs may be needle punched, stitched, or bonded with adhesive. GCLs typically come in roll lengths ranging from 100 to 200 feet and widths ranging from 13 to 17.5 feet. GCLs have been used for many years as the primary or secondary means of containment for landfills, lagoons, tanks, etc. GCLs have also been used as the primary hydraulic barrier in landfill liners or as the clay component in a composite cap.

Native-clay soils consist of in-situ clays from deposits within 40 to 50 miles of the TMA. This clay material would meet the current WDNR requirements for clay proposed for construction of composite liners.

These four alternatives for the soil components of the composite liner/cap were analyzed as to the advantages and disadvantages of each. Each soil component was also ranked based on technical aspects such as hydraulic conductivity, constructability, permeability, erodibility, etc.

In addition the cost of each alternative was calculated given a typical cell construction configuration. Based on an analyses of this information, native-clay soil was selected as the primary alternative for the soil component of the composite liner/capping system. Bentonite-amended soil was selected as the backup soil component should native clay become unavailable or impractical.

Based on the analysis outlined above native clay was selected as the primary alternative for the soil component of the composite liner for reasons as follows:

- design with clay as a component of a composite liner system is a proven technology;
- construction methods are well known and require no specialized equipment;
- construction quality assurance and quality control (QA/QC) uses common methods and good QA/QC ensures liner performances;
- hydraulic conductivity will meet or exceed project requirements;
- attenuation potential is high;
- this engineering/design technology is commonly permitted for containment design in the State of Wisconsin;

- longevity with respect to chemical and physical weathering is good; and
- the cost of a one-foot-thick clay liner is reasonable when compared to the alternatives analyzed.

For these reasons native clay was selected as the primary alternative for the soil component of the composite liner/cap. Bentonite-amended soil was selected as a design alternative, because sufficient testing was completed in conjunction with the previous feasibility report to demonstrate that bentonite-amended soil is a viable alternative for meeting project design requirements.

10.8 Evaluation of Preliminary Design Concepts

10.8.1 General

During the design process several meetings were held with WDNR and other interested parties to discuss preliminary design concepts for the Crandon Project TMA. At a September 14, 1994, meeting CMC presented and discussed the following four preliminary design concepts for the Crandon Project TMA.

- Option 1A - Underwater Placement
- Option 1B - Underwater (Created Wetland) In Lined Upland Facility
- Option 1B-CC - Composite Cover with Created Wetland in Lined Upland Facility
- Option 2A - Drained Impoundment with Collection System
- Option 2B - Drained Impoundment without Collection System

10.8.2 Description of TMA Design Options

10.8.2.1 Option 1A

Option 1A would involve subaqueous deposition, which is presently considered a successful method in preventing generation of acid in tailings. CMC did not feel this option was feasible in Wisconsin since it would involve deposition of tailings in surface waters, and would result in impacts on a wetland area of approximately 300 to 400 acres. (Refer to 10.2.2)

10.8.2.2 Option 1B

Option 1B would involve subaqueous deposition of tailings in a lined facility located on an upland site, with the final cover system consisting of a created wetland. Key design features for Option 1B include:

- A composite liner system consisting from top to bottom:
  - 60-mil HDPE geomembrane; and
  - one-foot soil component with hydraulic conductivity $\leq 1 \times 10^{-7}$ cm/sec.

- A leachate collection system (LCS) directly above the composite liner with:

  - LCS drainage layer above the geomembrane at the base of the facility;
  - one LCS trench per cell with collection pipe along the toe of the slope;
  - two-percent bottom slope toward LCS; and
  - LCS discharging to a sideslope riser which returns leachate to the tailings cells.

- Maintaining a water cover over the deposited tailings.

- A final cover system consisting of from top to bottom:

  - manmade wetland; and
  - twelve inches of native soil enhanced with organic materials.

- After closure, recirculation of collected leachate to assist in maintaining a wetland on the reclaimed tailings surface.

- Installation of a groundwater well(s) to assist in supplying water to maintain wetland conditions on the final reclaimed TMA surface.

10.8.2.3 Option 1B-CC

Option 1B-CC would involve deposition of tailings in a lined facility located on an upland site, with the final cover system consisting of a composite cover with a created wetland over the composite cover. Key design features for Option 1B-CC include:

- A composite liner system consisting of from top to bottom:

  - 60-mil geomembrane; and
  - one-foot soil component with hydraulic conductivity $\leq 1 \times 10^{-7}$ cm/sec.

- A leachate collection system (LCS) directly above the composite liner with:

  - LCS drainage layer above the geomembrane at the base of the facility;
  - (one) LCS trench (per cell) with collection pipe along the toe of the slope;
  - two-percent bottom slope toward LCS trench; and
  - LCS discharging to (a) sideslope riser(s) which returns leachate to the tailings cells.

- A final cover system consisting of from top to bottom:

  - manmade wetland;
  - twelve inches of native soil enhanced with organic materials
  - minimum 54-inch soil protective cover;
  - 60-mil (thermoplastic) geomembrane;
  - one-foot soil component with hydraulic conductivity $\leq 1.0 \times 10^{-7}$ cm/sec; and
  - a minimum one-foot grading layer.

- After closure, recirculation of collected leachate to assist in maintaining a wetland on the reclaimed tailing surface.

10.8.2.4 Option 2A

Option 2A would include subaerial deposition of tailings in a lined facility located on an upland site, with a final cover system designed to minimize oxygen entry and percolation into the tailings mass. Key design features for Option 2A include:

- A composite liner system consisting of from top to bottom:
  - 60-mil geomembrane; and
  - one-foot soil component with hydraulic conductivity $\leq 1.0 \times 10^{-7}$ cm/sec.
- A leachate collection system (LCS) directly above the composite liner with:
  - LCS drainage layer above the geomembrane at the base of the facility;
  - one LCS trench per cell with collection pipe along the toe of the slope;
  - two-percent bottom slope toward LCS; and
  - LCS discharging to a sideslope riser which returns leachate to the reclaim ponds.
- Maintaining a water cover over the deposited tailings during operations.
- A final cover system consisting of from top to bottom:
  - vegetative layer;
  - minimum 42-inch soil protective cover;
  - 12-inch sand or gravel drainage layer;
  - 60 mil geomembrane;
  - 12-inch soil component with hydraulic conductivity $\leq 1.0 \times 10^{-7}$ cm/sec; and
  - a minimum 36-inch grading layer.

10.8.2.5 Option 2B

Option 2B would be the same as Option 2A except that the leachate collection system directly above the composite liner was to be eliminated. This option was eliminated from consideration early in the process since initial HELP model runs indicated that the estimated allowable percolation rates could not be met by this option.

10.8.3 Design Considerations

The primary design consideration for the TMA is to protect the environment by meeting groundwater water quality standards. This requires that the facility be designed to control the discharge of solute so that an exceedance of groundwater quality standards at the compliance boundary is not experienced. Solute control can be achieved during the operating life of the facility by minimizing the potential for acid generation and through the installation of a composite cover and by installing a composite liner with a leachate collection system to prevent contaminant migration. Given these operation control methods, percolation through the composite liner for a great many years will consist primarily of process water. Based on past

project work, it is anticipated that sulfate will be the controlling parameter relating to groundwater quality impacts. Under neutral conditions, such as in process water, the sulfate concentration will be controlled by gypsum solubility to no more than approximately 2250 ppm. Sulfate is not toxic to humans or other organisms at those concentrations.

Solute control following TMA closure would be achieved by the inherent carbonate content of the ore and the lime additions during the beneficiation process, in combination with one or more engineering techniques. The engineering techniques applied depend upon which of options are implemented. For Option 1B, acid generation would be prevented since the tailings mass would be maintained in a saturated condition. For Option 2A, acid generation potential would be controlled by preventing oxygen from coming into contact with the tailings mass via the engineered final cover system. Post-closure solute percolation control for Option 1B would be accomplished by the composite liner and leachate collection system. In Option 2A, post-closure solute percolation control would be accomplished primarily by the final cover system.

10.8.4 Method of Analysis For Screening

The method used to evaluate the ability of a given design concept to meet groundwater quality standards was to model final cover and liner percolation rates and to then evaluate the impact of the released solutes on the groundwater at the facility's compliance boundary. With a few appropriate modifications, USEPA's Hydrologic Evaluation of Landfill Performance (HELP) model was used to perform a preliminary water balance analysis for Options 1B and 2A to estimate the percolation rates for each option under varied conditions. The evaluation of the impact of released solutes on groundwater quality was accomplished through a preliminary analytical modeling.

The analytical modeling chosen to define the preliminary allowable range of percolation rates for the TMA is described in Section 10.8. The HELP model was used to determine the percolation rates for both the Option 1B and 2A designs. These analyses were performed only for the closed conditions with the final cover in place. During operations the percolation rates are likely to be higher. A comparison of allowable rates to predicted rates was then used to identify a preferred design option. A discussion of this approach follows.

10.8.4.1 Preliminary Estimates of Allowable Percolation

To assist in the evaluation of the ability of the liner and leachate collection systems and final cover systems to meet applicable groundwater standards, it was necessary to define a range of percolation values, which, if permitted to percolate from the base of the TMA, is not likely to cause an exceedance at the compliance boundary 1,200 feet away. This value was to be used as a screening mechanism to evaluate some of the preliminary conceptual designs on the basis of potential percolation. Therefore the range, described above was defined as the preliminary estimate of allowable percolation. A one-dimensional model was used for the purpose of defining this range.

Based on the analysis, the estimated allowable percolation from the TMA ranges from 0.244 to 1.26 gallons per acre per day. This estimate was considered conservative because of several factors. The first was that groundwater flow beneath the TMA is essentially radial and, therefore, dispersion would actually occur, resulting in an approximate 50 percent decrease in sulfate concentration at the compliance boundary versus that calculated without dispersion.

Also, the current evaluation did not include the dilution effects associated with the infiltration of the runoff from the final cover. The degree of conservatism for the above analysis was appropriate given the preliminary nature of the overall liner and final cover analysis being performed at this time.

10.8.4.2 HELP Model

As previously discussed, the HELP model version 2.05 which was used for this study is a quasi-two-dimensional deterministic water budget model that performs a sequential daily analysis to determine runoff, evapotranspiration, barrier layer percolation and lateral drainage with daily climatologic data as input. (Please note that a later version of HELP model, version 3 was used for studies detailed in Section 6.7.) The Option 2A scenario for the TMA was easily modelled by this program. Option 1B with a wetland cap and maintenance of saturated conditions was more difficult to simulate. To facilitate this latter simulation, an assumption was made that if the initial simulation period of 20 years does not cause the tailings to significantly dewater, the calculated percolation values would be close to those under wetland/saturation conditions. This meant that the results of the 20-year simulation would be equivalent to the results of all future simulations, thus eliminating the need for simulations beyond 20 years.

10.8.4.3 Detailed Description of the HELP Models Used and Cases Studied

For Option 1B, the model primarily consisted of, from top to bottom, a 12-inch wetland vegetative layer, followed by six layers of tailings, each 168 inches thick. The liner and leachate collection layers were modelled by using a 12-inch lateral drainage layer and a barrier layer consisting of a composite liner (geomembrane and 12-inch low-permeability soil). This model is referred to as Option 1B-Case 1 in Section 10.8.2.2.

For Option 2A, the model primarily consisted of, from the top a 66-inch soil cover layer, a 12-inch lateral drainage layer, a composite barrier layer (geomembrane and 12-inch low-permeability soil), a grading layer, followed by six 168-inch thick tailings layers. The liner and leachate collection layers were modelled by using a 12-inch lateral drainage layer and a barrier layer consisting of a composite liner (geomembrane and 12-inch low-permeability soil). This model is referred to as Option 2A-Case 1 in Section 10.8.2.

During a December 15, 1994 meeting conducted to provide a technical exchange regarding the TMA, Archie Wilson and Bob Grefe of the WDNR requested further HELP Model analysis on a variation of Option 1B. The variation on Option 1B requested by the WDNR involved placement of a composite cover on the completed TMA prior to the construction of the wetland initially envisioned. This variation will be referred to as Option 1B-CC.

For Option 1B-CC, the model primarily consisted of, from top to bottom, a 12-inch saturated vegetative layer, 60-inch saturated cover soil layer, a composite barrier layer (geomembrane and 12-inch low-permeability soil layer), followed by approximately six 168-inch thick tailings layers. A protective soil layer is located immediately below the tailings and is underlain by a leachate drainage layer with a hydraulic conductivity of 1×10^{-2} cm/sec and a drainage length of 100 feet and a barrier layer consisting of a composite liner (geomembrane and 12-inch low-permeability soil).

The tailings were subdivided for both options into six layers to investigate how the drainage of pore water progressed. Further, because of the increased densities and decreased hydraulic conductivities as a result of increased consolidation with the increased depth of tailings, realistic estimates require accounting for the changes in these properties. Accordingly, subdivisions of the tailings into six layers with three sets of properties (two layers each with the same properties) is appropriate. The densities and hydraulic conductivities for the three sets were adapted from the preliminary consolidation test results reported by Steffen Robertson and Kirsten. Porosities were estimated on the basis of densities at low, medium, and high consolidation pressures within the facility. Field capacities for all cases were estimated on the basis of desiccated moisture content.

In order to estimate the impact of potential geomembrane deterioration on percolation rates, two separate HELP Model runs were performed. The run for Option 2A assumed that the liner system consisting of a geomembrane and one-foot low permeability soil component performed as a composite liner for 100 years after closure. At year 101 after closure, the run assumed that the geomembrane portion of the liner was no longer functional, and that the liner would consist of the one-foot low permeability soil. For Option 1B and 1B-CC, a similar analysis was performed, except since steady state is assumed to exist at closure, the analysis is not time based. These analyses are referred to as Option 2A-Case 2, Option 1B-Case 2, Option 1B-CC, respectively.

The Option 1B and 1B-CC analyses were each run for 20 years based on the assumption that steady-state conditions exist at closure due to the maintenance of the wetland and saturated conditions. After evaluating the initial model runs using the design parameters outlined in earlier the design was modified to incorporate multiple leachate collection pipes in each cell with 100-foot pipe spacing and a liner drainage layer with a hydraulic conductivity of $\geq 10^{-2}$ cm/sec to decrease the hydraulic head on the liner and to reduce the liner percolation rates. All Option 1B HELP model results discussed in this memorandum are based on this revised design.

For Option 2A, the model was run for 120 years to make sure that steady-state percolation conditions were reached. During the simulation period, percolation through the final cover system, and from the site, was obtained in 20-year increments. A summary of selected HELP Model run variables for Options 1B and 2A are shown in Table 10.8-1.

10.8.5 Discussion of Results

10.8.5.1 Option 1B

Table 10.8-2 contains the results of the HELP model analyses for Option 1B (created wetland). The table shows that Option 1B-Case 1 percolation rates are lower than the estimated allowable rate. The table also shows that if the geomembrane in the composite liner were to deteriorate after site closure (Option 1B-Case 2), the percolation rate would increase to values above the estimated allowable rate. Also included in the table is the maximum head on the composite liner as calculated by the HELP model for the given inputs. A reduction in the head can be achieved as part of the final design if Option 1B is pursued. It is unlikely, though, that a head reduction would improve the percolation rate for Case 2 to the degree necessary to fall below the allowable rate.

Table 10.8-1

Summary of Selected HELP[4] Model Run Variables

	Composite Liner		Composite Cap		Leachate Collection System	
Option	Geomembrane	Low Permeability Soil (K)[3]	Geomembrane	Low Permeability Soil (K)[3]	Pipe Spacing (ft)	Drainage Layer (K)[3]
1B - Case 1[1]	Yes	1×10^{-7}	wetland		100'	1×10^{-2}
1B - Case 2[1]	No	1×10^{-7}	wetland		100'	1×10^{-2}
1B-CC - Case 3[1]	Yes	1×10^{-7}	Yes (wetland)	1×10^{-7}	100'	1×10^{-2}
1B-CC Case 4[1]	No	1×10^{-7}	Yes (wetland)	1×10^{-7}	100'	1×10^{-2}
2A - Case 1[2]	Yes	1×10^{-7}	Yes	1×10^{-7}	760'	1×10^{-3}
2A - Case 2[2]	Yes	1×10^{-7}	Yes	1×10^{-7}	760'	1×10^{-3}

1 Option 1B runs were completed for a 20 year time frame since steady state is assumed to have occurred at closure. Case 1 was evaluated with a composite liner system. Case 2 was evaluated with a low permeability soil liner.

2 Option 2A
Case 1 - 1-120 years with composite liner system.
Case 2 - 1-100 years with composite liner system, 101-120 years with a low permeability soil liner.

3 (K) - Hydraulic conductivity in cm/sec

4 HELP Model Version 2.05

Prepared by: REM
Checked by: GWS

Table 10.8-2

Comparison of Option 1B and Option 1B-CC - HELP Model Output

Options	Case No.	Leachate Collected (g/a/d)	Time to Pass Three Pore Volumes (years)	Maximum Head on Bottom Liner (feet)	Base Liner Percolation Rate (g/a/d)[5]
1B[1]	Case 1	244	509	23.5	0.026
1B[2]	Case 2	150	509	23.5	93.8
1B-CC[3]	Case 3	29.3	924,000[6]	22.1	0.023
1B-CC[4]	Case 4	18.4	924,000	22.1	11.0

[1] Case 1 - Option 1B = composite liner with wetland cap (presented in November 28, 1994 memorandum also).
[2] Case 2 - Option 1B = low permeability soil liner with wetland cap (presented in November 20, 1994 memorandum also).
[3] Case 3 - Option 1B-CC = composite liner with composite wetland cap.
[4] Case 4 - Option 1B-CC = low permeability soil liner with composite wetland cap.
[5] Estimated allowable percolation rate per preliminary analytical model ranges from 0.244 to 1.26 g/a/d. For this analysis, assume the lower value of .244 g/a/d applies.
[6] Recirculation is equal to the leak through the cap only.

g/a/d = gallons per acre per day.

Prepared by: REM
Checked by: GWS

Since Option 1B will involve maintaining the tailings mass in a saturated state, any water percolating through the liner will be process water moderately high in sulfates. A scenario could exist where sufficient flushing of the tailings mass could occur through recirculation, such that the sulfate content of the percolated water would be reduced, so that the higher percolation rates that would occur if the geomembrane deteriorated would not adversely impact groundwater quality. This concept would need to be implemented with a wetland design involving the creation of an environment conducive to sulfate-reducing bacteria so that the sulfate content of the recirculated leachate would be significantly reduced before the water reentered the tailings mass.

To evaluate the above concept, it was assumed that no more than three pore volumes would need to pass through the tailings mass to accomplish the needed flushing. In reality, the concentrations would reduce considerably after passing one pore volume. If this option were to be pursued further, verification of the flushing rate would be needed. Using the leachate recirculation rates and the percolation rates in Table 10.8-2, and an assumed tailings porosity of 55 percent, the time required to pass three pore water volumes was calculated.

A review of Table 10.8-2 shows that the time to pass three pore volumes of water is over 500 years. Given this fact, this concept does not appear feasible because it would be impractical to expect that the operation of the recirculation system would continue for that period of time.

10.8.5.2 Option 1B-CC

Table 10.8-2 contains a summary of the HELP model results for Option 1B-CC along with the results from Option 1B discussed previously. As done with Option 1B a Case 3 (with a geomembrane in the liner and a Case 4 (without a geomembrane in the liner) were analyzed.

Case 1 and Case 2 data (Option 1B) are presented in Table 10.8-2 so that the performance of these two scenarios can be compared to Case 3 and Case 4. Note that the only difference between Cases 1 and 3 is that Case 3 has a geomembrane component in the final cover. Similarly, the only difference between Cases 2 and 4 is that Case 4 has a geomembrane component in the final cover. As a result, the comparison should focus on the differences resulting from the composite cap, which in review of Table 10.8-2 are as follows:

- For Case 1 and 3, the base liner percolation rate is lower than the estimated allowable percolation rate.
- For Case 3 and 4, lateral drainage from layer 11 of the HELP model version 3.01 decreases almost one order of magnitude when compared with Cases 1 and 2. This can be partly attributed to decreased percolation through the final cover resulting from the composite cap.
- For Case 3 and 4, the time to pass three pore volumes when compared with Case 1 and 2 is extremely large since the recirculation through the tailings will be limited to the percolation through the cap.
- For Case 2 and Case 4, the base liner percolation rate exceeds the estimated allowable leachate rate.

10.8.6 Option 2A

Table 10.8-3 contains a comparison of projected HELP Model Option 2A percolation rates to the preliminary projected estimated allowable rates. The table shows that the projected percolation rates through the base liner will be well below the allowable rates throughout the post-closure care period.

The table shows that for Option 2A-Case 1, percolation from the final cover system during the first 20 years is 0.122 gallons per acre per day. A steady-state rate of 0.141 gallons per acre per day occurs in the next 20 years. Percolation through the liner, for Option 2A-Case 1, is projected to be 0.068 gallons per acre per day in years 0-20. A steady-state rate of 0.023 is projected to occur in years 41-60.

Table 10.8-3 shows that for Option 2A-Case 2, HELP Model output is the same as for Option 2A-Case 1 through the first 100 years. For years 101-120, the final cover system percolation rate remains the same while the percolation rate through the base liner changes to 0.173 gallons per acre per day. The 0.173 gallon/acre/day value is below the lower end of the estimated allowable rate. If the Option 2A-Case 2 analysis was carried out beyond year 120, the percolation rate from the base liner would progressively decrease to a steady-state value below the year 101-120 rate. This analysis shows that control of base liner percolation rates is maintained by the TMA's final cover during the post-closure period.

10.8.7 Analysis of Results - Option 2A, 1B, 1B-CC

Based on the evaluation of Options 1B and 2A the following conclusions can be drawn:

- Liner percolation rates for Option 1B were shown to be lower than the conservative estimated allowable rates.
- The concept of a wetland cover, as proposed for Option 1B, will result in greater-than-allowable liner percolation rates if the geomembrane component of the liner deteriorates in less than 500 years.
- A tailings facility which is kept drained from the base during operation and for a period after closure such as that proposed for Option 2A will have liner percolation rates that are below the conservative estimated allowable rates.
- Deterioration of the geomembrane component of the Option 2A composite liner 100 years after closure will not result in liner percolation rates in excess of the conservative estimated allowable rates.
- For Option 1B, Case 2 and Option 1B-CC Case 4 base liner percolation rates are likely to exceed allowable percolation rates.

Table 10.8-3

Comparison of Projected Percolation Rates for Option 2A Cases 1 and 2 to the Preliminary Estimated Allowable Rate[1]

	Percolation Through Final Cover (g/a/d)	Percolation Through Base Liner (g/a/d)	Allowable[3] (g/a/d)
Case 1[1]			
0-20	0.122	0.068	0.244-1.26
21-40	0.141	0.025	0.244-1.26
41-60	0.141	0.023	0.244-1.26
61-80	0.141	0.023	0.244-1.26
81-100	0.141	0.023	0.244-1.26
101-120	0.141	0.023	0.244-1.26
Case 2[2]			
0-20	0.122	0.068	0.244-1.26
21-40	0.141	0.025	0.244-1.26
41-60	0.141	0.023	0.244-1.26
61-80	0.141	0.023	0.244-1.26
81-100	0.141	0.023	0.244-1.26
101-120	0.141	0.173	0.244-1.26

g/a/d - Gallons per acre per day, assumes an ~ 9.6 x 10^6 ft^2 lined area.

1 Case 1 of Option 2A is a drained case after facility closure.

2 Case 2 of Option 2A is a drained case assuming a low permeability soil liner from year 101 after facility closure.

3 For this analysis, assume that the lower value applies.

Prepared by: REM
Checked by: GWS

It is apparent from this analysis that Option 1B-CC (Case 4) and Option 1B (Case 2) will result in greater-than-allowable liner percolation rates. In comparison, the HELP model results of Option 2A as presented in Table 10.8-3 show that the tailings facility, which is kept drained from the base during operation and for a period after closure, will have liner percolation rates that are below the conservative estimated allowable rates. It may also be noted that the difference between Option 1B-CC and Option 2A Case 1 is that a wetland is maintained above the cap. Comparison of the percolation through the hydraulic barrier in this cap for the two cases show almost identical rates. Therefore the leachate collected and percolation through the liner should be same for the two cases.

A tailings facility which is kept drained from the base during operation and for a period after closure, such as that proposed for Option 2A, will have liner percolation rates that are below at conservative estimated allowable rates and as a result, meets the performance standards for the facility.

The establishment of a wetland cap over the TMA may also prevent the final cover maintenance activities necessary to maintain the integrity of the capping system and as a result a wetland cap would lead to even greater percolation out of the TMA facility than what is now predicted by the HELP model results. (The performance of the TMA is best maintained by Option 2A.)

10.8.8 Suggested Alternative

Based on the analysis presented in this section, the performance of the TMA is best maintained by design and implementation of Option 2A.

10.9 Alternative for Operational Components NR 182.08

This section reviews the alternative for the major operational components of the TMA which include:

- containment for the tailings delivery system;
- tailings deposition method;
- tailings decant water recovery method; and
- tailings dewatering alternatives (leachate collection system).

The operational components reviewed are limited by the selection of the wet method of operation (slurried tailings) as the primary operating concept. The 1985 EMC Feasibility Report considered other alternatives and the wet method was selected because it was and still is a proven system for tailings management.

10.9.1 Dry Tailings Deposition

The use of dry tailings handling and disposal systems has been found to be advantageous in those cases where the tailings solutions are highly contaminated. In this case, the removal of the tailings solution reduces the heads on liners and the potential for increasing percolation. These conditions do not apply to the Crandon tailings, where the original process waters associated with the tailings contain relatively-low concentrations of contaminants. The dewatering (production of dry tailings) of the Crandon tailings immediately after mineral processing is not economical.

There are also a number of substantial disadvantages to the production and handling of dry tailings at Crandon:

- 80 percent of the Crandon tailings to be placed in the TMA will be tailings fines with 90 percent of the particles less than 10 μm. Such tailings are difficult and costly to dewater.
- Belt, disk or drum filtering of such fine tailings are likely to yield "sloppy" tailings which are difficult to handle by conventional earth moving equipment resulting in high placement and handling costs.
- The sloppy tailings would be unusable and will not support traffic loads. Thus they will have to be placed in, and be contained by, dikes similar to those required for wet tailings.
- The high-moisture content of such sloppy tailings results in ample pore water being available for seepage release, hence an underdrainage leachate collection system will till be required, as for the current wet tailings design. A material reduction in the potential for percolation through the liner is not anticipated.
- The low-placed densities of the sloppy tailings offers no significant advantage with regard to long-term settlement or cover performance.
- The increased potential for oxygen access and oxidation of the tailings is a disadvantage with respect to potential acid generation.

Given the increased costs, increased handling and operational difficulties associated with the management of the Crandon fine tailings as a "dry tailings" system, it was concluded that a wet tailings systems was preferable for each of the cost, operational and environmental protection considerations.

10.9.2 Tailings Delivery System

The tailings delivery line will be constructed to transport tailings from the mill to the point common to all four proposed TMA cells. The tailings delivery system will generally consist of a 16-inch diameter tailings pipeline to deliver tailings to the TMA cell. In addition a parallel 12-inch diameter pipeline will reclaim process water from the TMA cell decant structure to the process water pond for return to the mill.

The tailings pipeline proposed is a pressure pipe. Operating experience would indicate that HDPE pipe with an standard dimension ratio of 15 should be appropriate for the gravity pipe and with an SDR of 11 for the pressure pipe (pressures to about 150 psi). All pipes should be jointed at 400-foot centers with bolted flanges. Negative-pressure vents are required at high points and gravity drainage outlets at low points.

10.9.3 Containment for the Tailings Delivery System

The selected alternative for the tailings pipeline is to install it in a lined channel or alternative in a "utilidor" such that any spillage is contained and channelled down to a collection pond at the

low points. Both the tailings line and the return water line will be installed in such a lined contained area.

At the low point(s) a pipe drain and spill/drainage collection pond will be required to drain the pipe if:

- there are unexpected leaks in the pipe;
- the pipe becomes blocked and must be drained; and
- pumping capability is interrupted and the pipe must be drained to prevent freezing.

Such drainage may not always occur at the low points and it must therefore be feasible to drain the pipes from any joint such that flow occurs along the lined drainage ditch to the collection pumps at the low points.

The lined ditch must be installed at a grade such that any spilled tailings will flow to a lined collection pump at the low points from where it can be removed by mechanical excavation or pumping. Grades of ditches must be sufficient to cause the spilled tailings to flow to the collection point. This requires a grade of about one percent. Given the long, flatter sections of the tailings delivery line, it may be necessary to create this grade by appropriate route selection and ditch grading to minimize the number of low spots.

10.9.3.1 Alternative to Pipelines in a Ditch

Utilidor on Surface

A "utilidor" (such as is used at Key Lake mine for uranium tailings), may be considered. This has the advantage that, since it can be closed, it provides better physical and weather protection. The disadvantage is the extra cost and the small flow area, increasing the likelihood of blockage during a spill.

Buried Pipeline

A buried pipeline has the advantage that it is not subject to the temperature variations of a surface line and is more protected from damage. It has the disadvantage that it cannot be inspected and leaks may go undetected for some time. It is also not possible to easily lift the pipe to rotate or replace worn pipe, repair damaged pipe or clear blocked pipes.

10.9.4 Tailings Deposition Alternative

As indicated at the beginning of this section based on present practice and experience the wet tailings delivery method was selected. Two deposition methods for wet tailings deposition evaluated include: Subaerial deposition and subaqueous deposition. The following is a general description of the two deposition methods.

- Subaqueous deposition involves discharges of tailings under water in an impoundment. Deposition is uncontrolled in that the tailings settle as a soft-bottom sediment and the tailings solids surface is generally undulating.

- Subaerial deposition (sometimes referred as managed deposition) is typically accomplished by discharging the slurry from several points around the perimeter of the impoundment area, as the slurry drains toward the low point in the impoundment. The solids spread out to form beaches and free water drains and concentrates at the low point. The discharge point or points is moved around the impoundment to allow the tailings to dewater and solidify. The subsequent layers are deposited and the cycle is repeated.

The subaerial deposition system was selected because maintaining a subaqueous system on an upland site is not practicable for the long-term, as shown by the analysis contained in Section 10.3.

10.9.5 TMA Process Water System

Four alternatives were considered for the TMA reclaim water system. These systems, in general, decant excess water from the slurried tailings and pump it to the process water pond where it is detained for approximately three days. The alternatives considered included central decant barge, rock fill central intake, decant tower and side-point decant. These alternative are briefly described below.

- A central barge decant would involve a central barge located in the middle of the cell with a floating access and reclaim water line support ramp. Barges have the advantage of greater flexibility in pool location. However, water level must be maintained at a depth of 3' to 5'.

- The rockfill central intake consists of an intake tower installed on top of the liner. The tower would be supported by rock fill from the mine (which provides a location for disposal of Type II waste rock) which also provided the access to the tower. The access and support ramp would be raised in six- to ten-foot increments within the tailings disposal area as the tailings beaches are build up. The tower intake would be constructed of circular concrete segments similar to many tailings penstock intakes, and would be raised in segments. Rock waste placed around the intake would control flow to the intake. Water would be removed from the decant tower by lowering a pump suspended from a skid mounted frame into the decant tower.

- A self-supporting decant tower could be installed to avoid the necessity to install waste rock access and support ramp. Such a tower may have its water intake controlled by stoplogs or holes which can be plugged as the water level rises. The water entering the decant tower could discharge to a conventional penstock through the embankment, which involves an undesirable penetration of the cell liner. Alternatively the tower could be built on top of the liner and conduct water via a penstock installed above the liner to a riser near the cell embankment from where water can be pumped.

- Side pond decant would necessitate discharging tailings from three sides of a TMA cell to force the pond to the opposite corner from which discharge is occurring. The pond then would be up against one interior side slope of the cell and water could be decanted from this side pond by a fix inclined riser. To protect the liner

and its protective layers during operation it is necessary to install riprap for the entire slope distance in the area where the pool can contact the interior berm. This is of concern after closure because the riprap provides a potential entire point of oxygen after closure.

The selected system is a skid mounted pump which is winched or pulled up the interior side slope and a barge with a suction intake for decanting water from a central pond. The proposed decant pond is described in more detail in Section 7.3 of this report.

10.9.6 Need for Leachate Collection System

Given the tailings properties and the operational scenario described above, it was determined that a tailings dewatering system in the form of a Leachate Collection System (LCS) needed to be incorporated into the design. For tailings management facilities there are advantages and disadvantages of installing a LCS. For the TMA facilities, a TMA cell with a full LCS to a cell with a partial LCS has been compared. The full LCS includes a high permeability drainage layer on the entire length of the interior slopes. The partial LCS includes a drainage layer only partially up the interior slope of the first stage constructed.

The advantages of a full LCS include the following:

- more efficient dewatering of tailings results in greater tailings densities which means that the total cell volume is decreased slightly; and
- reduced head on the interior slopes of the liner, which results in reduced percolation through the liner.

However a full LCS has some potential disadvantages which have significant environmental consequence. These disadvantages include:

- dewatering of the drainage layer and tailings adjacent to the drainage layer will result in pulling oxygen into the tailings mass through the high permeability drainage layer, leading to the following:
  - with oxygen present, tailings may start to form acids by chemical reactions with available oxygen; and
  - once these chemical reactions are started they will increase in speed with time and continue to occur over the long-term care period.
- the leachate produced with a full LCS will likely have a lower pH, higher sulfate and a significant metals content if the tailings have started to produce acid. Leachate strength may be several orders of magnitude higher with acid generating tailings.

Although the full LCS offers advantages related to increased LCS efficiency and high tailings densities, the major disadvantages like initiation of acid generation precludes selecting it as a suitable design alternative.

To address the disadvantages of a full LCS, the selected alternative proposes a partial leachate collection system (partial LCS). This partial LCS extends up the interior sideslope a distance equal in vertical height to the high point in the cell base. This ranges from approximately 9 feet to 16 feet in the TMA cells as proposed. The partial LCS provides the following advantages:

- leachate collection system is sealed off from contact with oxygen during operations and closure which minimizes the potential for acid generation to occur;
- since the exclusion of oxygen allows little potential for the tailings to generate acid, leachate should stabilize at a moderate pH (7-10) and sulfate and metals concentrations should be low; and
- the HELP model analyses indicate that a cell with a partial LCS will meet groundwater standards during operation, closure and post-closure years.

A partial LCS has some disadvantages, however those disadvantages have no significant environmental consequences. The disadvantages of the partial LCS include less efficient dewatering of the tailings results in lower tailings density, and therefore a slightly larger cell volume. Preliminary estimates show that the tailings density differences result in an approximate ten percent increase to TMA size and increased head on the interior slopes results compared to full LCS. However, refer to Section 6.7 of this report which discusses the HELP model runs results showing that groundwater standards can be met during operation, closure and post-closure conditions.

Because of the advantages of a partial LCS in maintaining the tailings in a non-oxidized state, it was selected as the preferred design option. The design rationale and detailed description of this system are found in Section 6.4 of this report.

11 References

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Figures for the *TMA Feasibility Report/Plan of Operation*

CRANDON
LINCOLN
LAONA
PLANT SITE NASH-
TMA
VILLE
FREEDOM
AINSWORTH
ONEIDA CO.
FOREST CO.
LANGLADE CO.
FOREST CO.
LANGLADE CO.
Crandon
Laona
Nashville
Mole Lake
Blackwell
Padus
Wabeno
MOLE LAKE INDIAN RESERVATION
POTAWATOMI INDIAN RESERVATION
POTAWATOMI INDIAN RESERVATION
POTAWATOMI INDIAN RESERVATION
Rice L.
Whiteye L.
Clear L.
Peshtigo L.
Bug L.
Otter Spgs.
Dawson L.
Bobs L.
Scattered Rice L.
Ut. Birch L.
Birch L.
Langer L.
Purdy L.
Wablkon L.
Ut. Pine L.
Silver L.
Ut. Riley L.
Indian L.
Metonga
Lake Lucerne
Wyman L.
Bradley L.
Horseshoe L.
Skunk L.
Ground Hemlock
Oak L.
Duck L.
Lost L.
Spring L.
Deep Hole L.
Kimberly L.
Walsh L.
Clark L.
Cook L.
Jungle L.
St. Johns L.
Crawford L.
Rolling Stone L.
Berendsen L.
Agnes L.
Crane L.
Spiders L.
Lost L.
Pickerel Lake
Hollister L.
Deadman L.
Tekakwitha L.
Horseshoe L.
Crystal L.
Arbutus L.
Grass L.
Mayflower L.
Midget L.
Ut. Star L.
Eds L.
Hardwood L.
Popple L.
Ut. Popple L.
Lundington L.
Himley
Roberts L.
Finnerty L.
Woods L.
Round L.
Bass L.
Richardson
Spring L.
Zarling L.
Ut. Long L.
Shoe L.
Roxy L.
Mud L.
Deer L.
Bogbrook Bk.
Spring Pd.
Kohlhoff L.
Windmill L.
Glade L.
Br. Lily R.
Range Line
Range Line
Kazimer L.
Ham L.
Trump L.
Eugene L.
Shabo
Story
Swanson
Camp 20
Honey
Rat R.
Mud R.
Wolf R.
Swamp
Hemlock
Squaw Cr.
Gliske Cr.
Logan
Mud Cr.
Swamp Cr.
Pickerel
Campbell Cr.
Grumen Cr.
Whiting
N.B.N.
Camb
Otter
Oconto
Torpee
Torpee
Swamp
Swamp
Mole L.
Sand L.
Jinnan L.
Ch
Ut.
Ut.
Ut.

2,270,000 E
2,275,000 E
2,280,000 E
2,285,000 E
125,000 N
120,000 N
115,000 N
110,000 N

SHERMAN LN
CREEK
SWAMP
OUTLET
SWAMP
WISCONSIN CENTRAL LIMITED RAILROAD
KEITH SIDING RD
HEMLOCK CREEK
HEMLOCK CREEK
PROJECT AREA BOUNDARY
PROPOSED RAILROAD SPUR
PROPOSED ACCESS ROAD
TMA BORROW SITE
PLANT SITE
SKUNK LAKE
RECLAIM POND
TMA
TMA ACCESS ROAD AND TAILINGS PIPELINE ROUTE
OAK LAKE
LITTLE SAND LAKE
DUCK LAKE
DEEP HOLE LAKE
DEEP HOLE LAKE
SAND LAKE CREEK
RD

1 2 3 4 5 6 19 20 24 29 30 31 32 33 35 36

N

LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- 1675 EXISTING CONTOUR
- X 1692.0 EXISTING SPOT ELEVATION
- SECTION LINE
- ORE BODY
- PROPOSED ACCESS ROAD
- PROPOSED TMA ACCESS ROAD AND TAILINGS PIPELINE ROUTE
- PROPOSED RAILROAD SPUR
- PROPOSED FACILITIES

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1" = 1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS 25 FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. ORE BODY OUTLINE IS REPRESENTATIVE OF THE SUBCROP AT THE BASE OF THE OVERBURDEN.

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	PAE	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

Crandon Mining Company

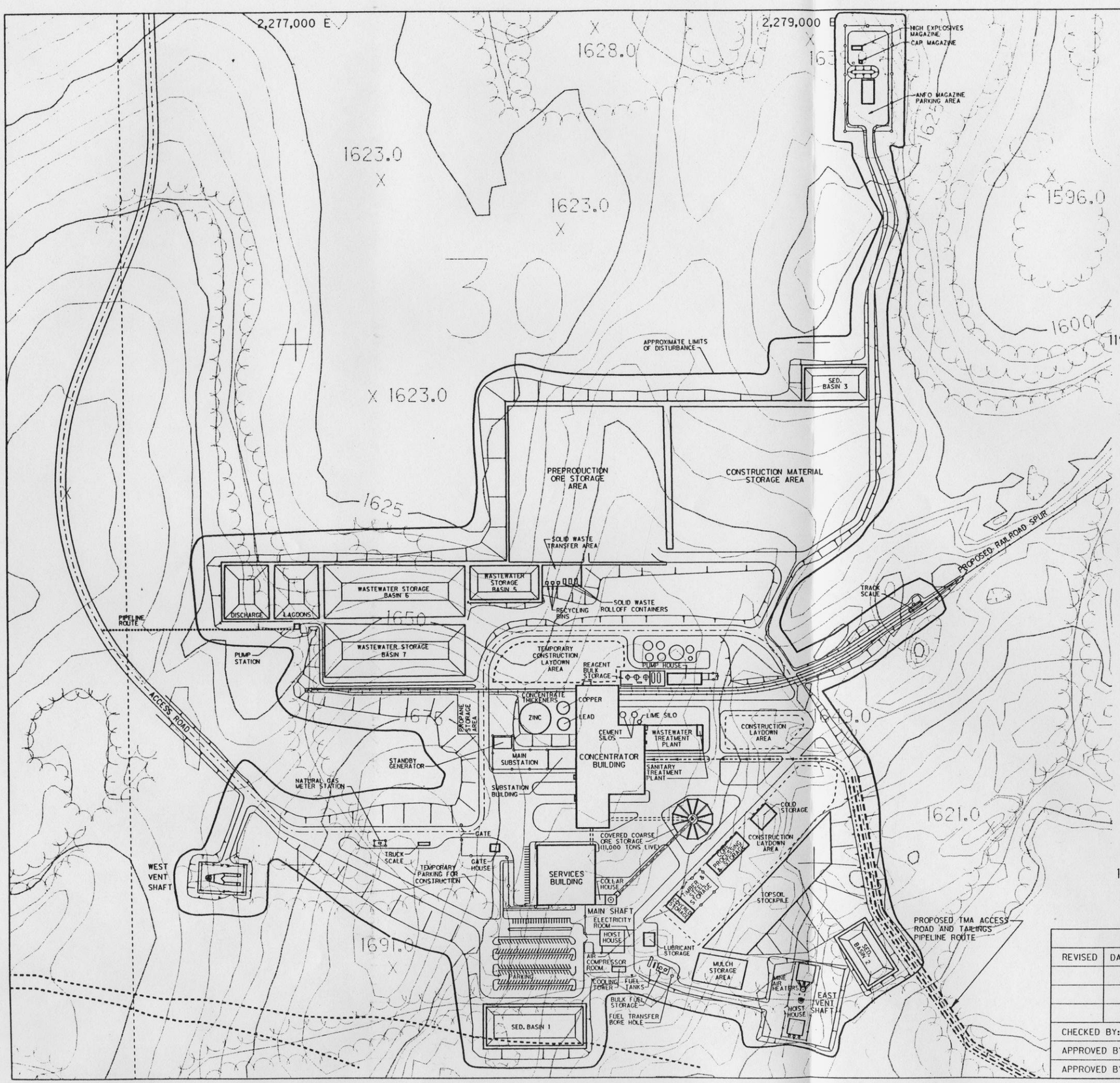
FIGURE 2-2
PROJECT AREA

Scale: 0 1000' 2000' | Date: MAY, 1995

Prepared By: Foth & Van Dyke | By: JRB

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93C049



N

LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- —1675— EXISTING CONTOUR
- X 1692.0 SPOT ELEVATION
- SECTION LINE
- ORE BODY

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

SCALE
0 200' 400'

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

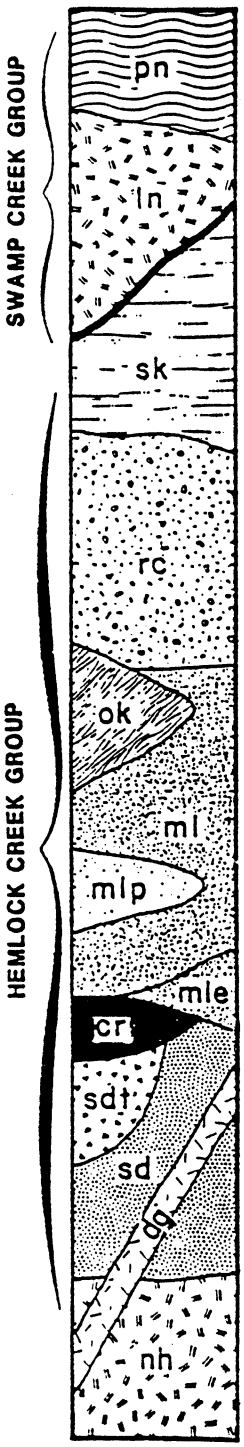
CHECKED BY:	JKSI	DATE: MAY.'95
APPROVED BY:	PAE	DATE: MAY.'95
APPROVED BY:	JWS	DATE: MAY.'95

Crandon Mining Company

FIGURE 2-3
PLANT SITE LAYOUT

Scale: SEE BAR SCALE | Date: MAY, 1995
Prepared By: Foth & Van Dyke | By: JRB2

93C049



PINE FORMATION (pn)
Cherty tuff and argillite.

LINCOLN FORMATION (ln)
Quartz porphyritic rhyolite flows with minor interflow tuff, chert and argillite.

SKUNK LAKE FORMATION (sk)
Predominantly fine to coarse ash chert tuff, some cherty and very minor argillite.

RICE LAKE FORMATION (rc)
Volcanic debris flows (blocky chloritic and minor siliceous lapilli and breccia size debris) and eutaxitic ash flows, some chert.

OAK LAKE FORMATION (ok)
Sequence of cherty tuff and sericitic tuff.

MOLE LAKE FORMATION (ml)
Predominantly mafic ash tuff.

PROSPECT MEMBER (mlp)
Volcanic debris flow consisting of siliceous, lapilli size debris.

EAGLE MEMBER (mle)
Volcanic greywacke.

CRANDON FORMATION (cr)
Laminated, bedded & replacement sulfides (zinc ore) interbedded with pyritic argillite, pyritic felsic tuff and chert.

SAND LAKE FORMATION (sd)
Sequence of fine felsic tuffs and minor felsic debris & lava flows.

TOWNSHIP MEMBER (sdt)
Volcanic vent breccia affected by multiple stage hydrothermal alteration and sulfide enrichment.

NASHVILLE FORMATION (nh)
Feldspar porphyritic mafic flows.

DUCK LAKE GABBRO (dg)
Fresh, 2 pyroxene gabbro.
Cross cuts nh and sd.

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE:	APR.'95
APPROVED BY:	PAE	DATE:	APR.'95
APPROVED BY:	GWS	DATE:	APR.'95

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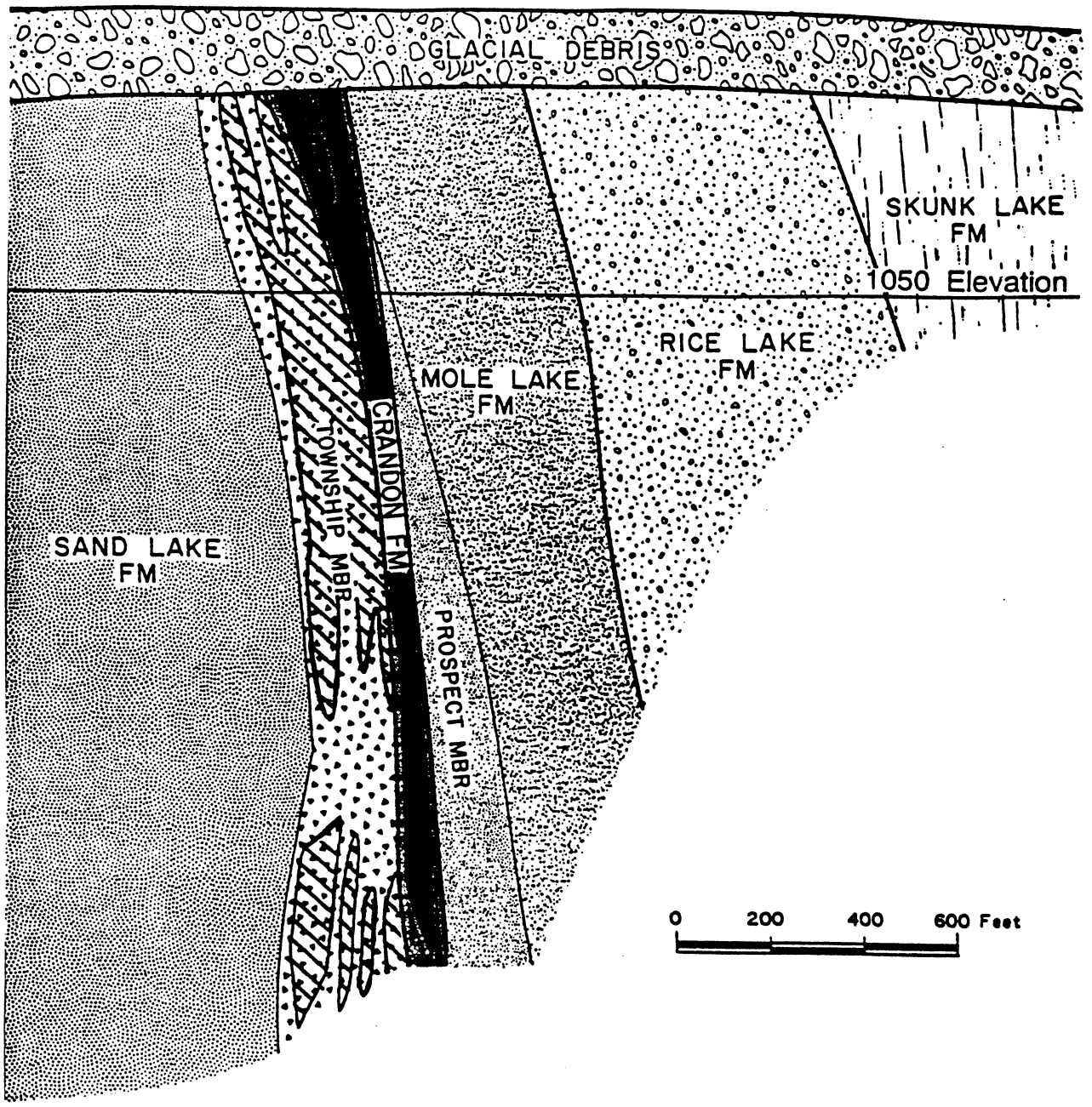
FIGURE 2-4
STRATIGRAPHIC COLUMN

Scale:	NOT TO SCALE	Date:	MARCH, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

/d3/dl/users/jprl/3c49m5.dgn

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LOOKING WEST



ZINC ORE

COPPER ORE

Foth & Van Dyke

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APPROVED BY:	GWS	DATE:	APR.'95

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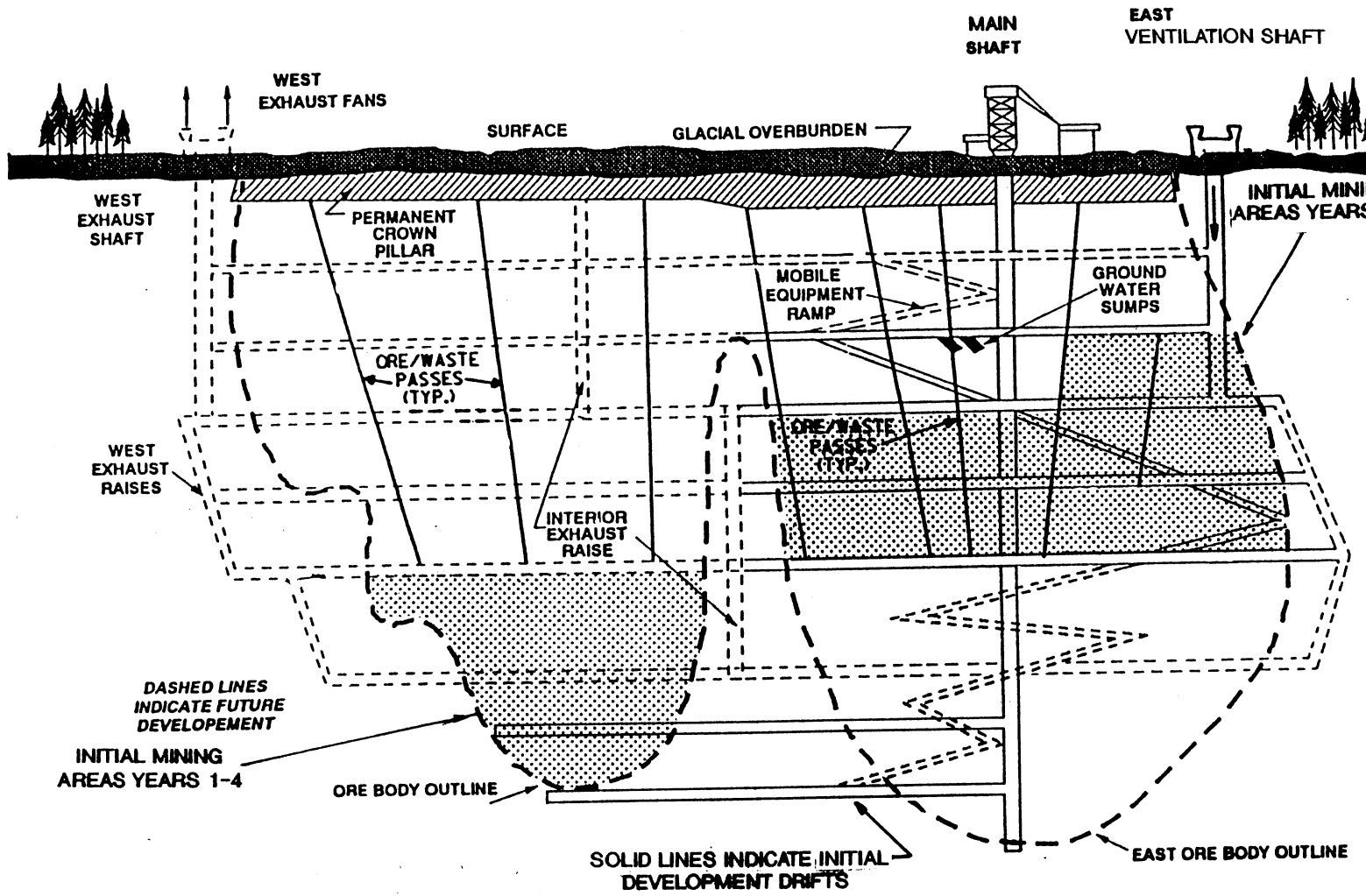
FIGURE 2-5

GENERALIZED GEOLOGIC CROSS SECTION C-C'

Scale:	AS SHOWN	Date:	MARCH, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

/d3/dl/users/jpr1/3c49m5.dgn

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APPROVED BY:	GWS	DATE:	APR.'95

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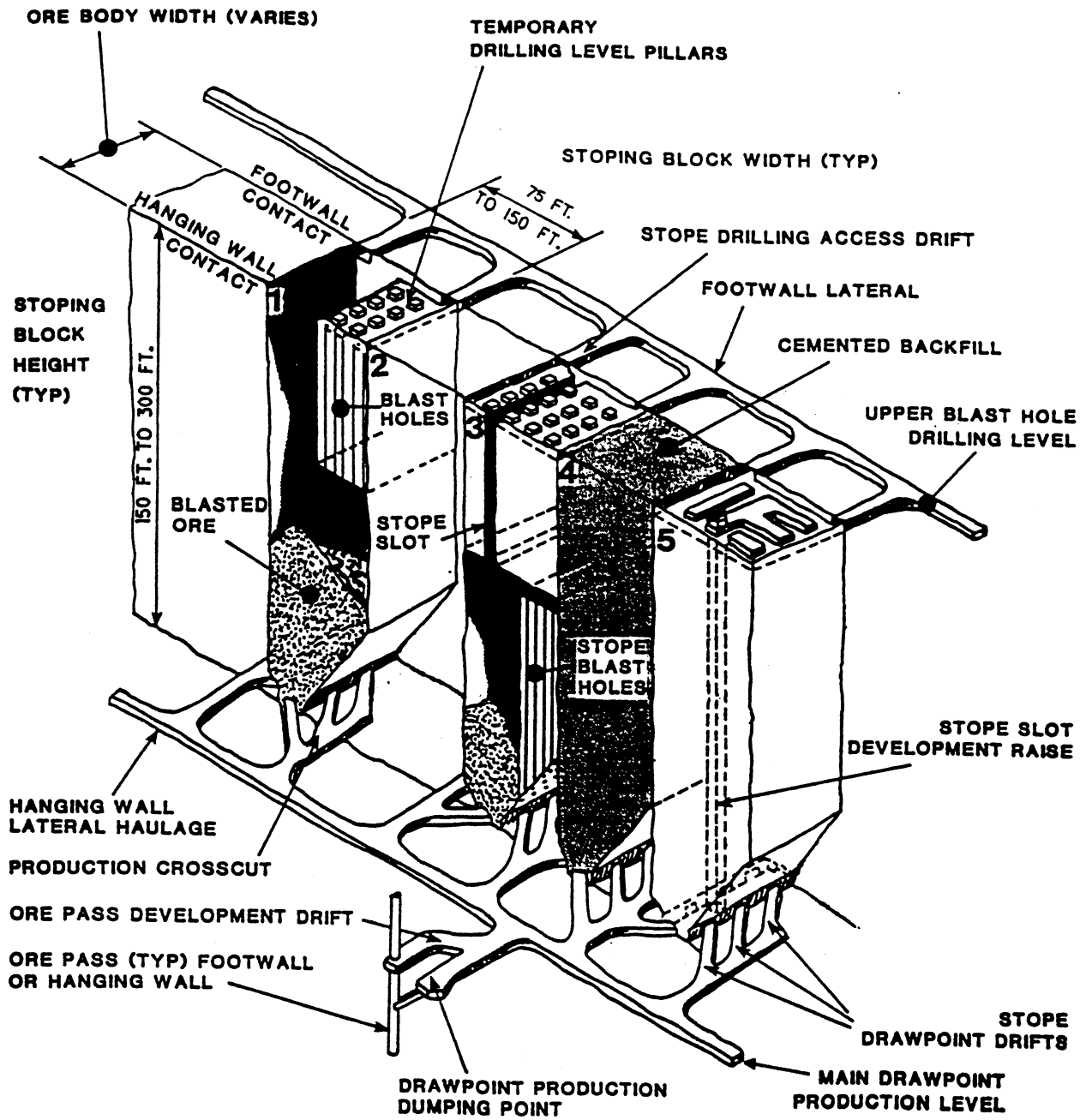
FIGURE 2-6

SCHEMATIC LONGITUDINAL SECTON (LOOKING NORTH)

Scale:	NOT TO SCALE	Date:	MARCH, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

/d3/dl/users/jpr1/3c49m5.dgn

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STOPING BLOCK NO. 1-ACTIVE PRODUCTION-STOPE BLASTING NEARING COMPLETION

STOPING BLOCK NO. 2-ALTERNATE STOPING BLOCK/PILLAR-TO BE MINED AFTER BACKFILLING ADJACENT STOPES

STOPING BLOCK NO. 3-ACTIVE PRODUCTION-INITIAL STAGES OF STOPE BLASTING

STOPING BLOCK NO. 4-DEPLETED STOPE BLOCK-CEMENTED BACKFILL IN PLACE

STOPING BLOCK NO. 5-STOPE BLOCK PARTIALLY DEVELOPED

Foth & Van Dyke

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CHECKED BY:	JKSI	DATE:	APR.'95
APPROVED BY:	PAE	DATE:	APR.'95
APPROVED BY:	GWS	DATE:	APR.'95

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FIGURE 2-7

CONCEPTUAL STOPING SEQUENCE

Scale:	NOT TO SCALE	Date:	MARCH, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

/d3/dl/users/jpr1/3c49m5.dgn

93C049

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY: JKSI DATE: APRIL '95

APPROVED BY: PAE DATE: APRIL '95

APPROVED BY: GWS DATE: APRIL '95

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FIGURE 2-8

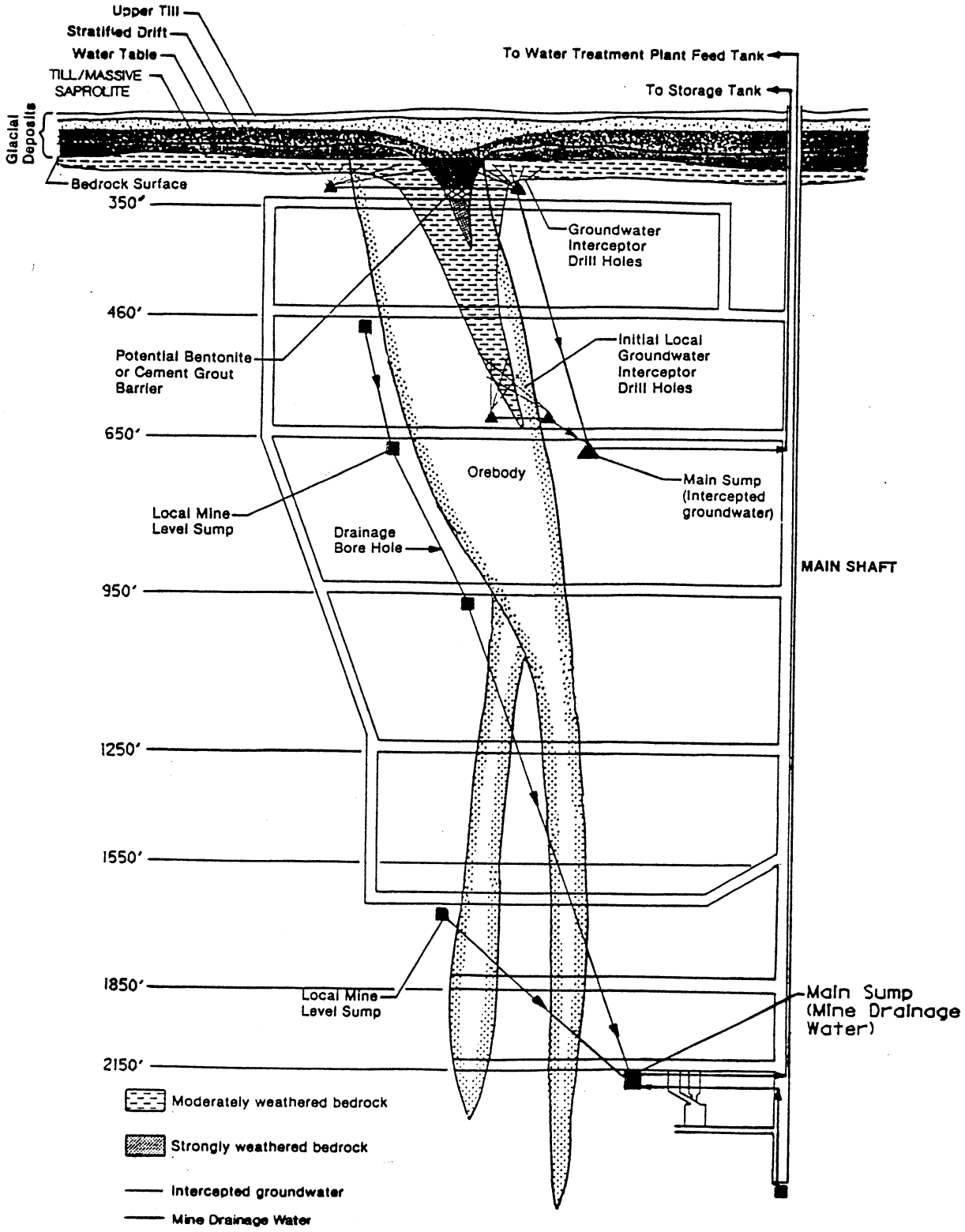
GROUNDWATER INTERCEPTOR SYSTEM
(CONCEPTUAL CROSS SECTION)

Scale: AS SHOWN Date: MARCH, 1995

Prepared By: Foth & Van Dyke By: BSH

c:\cadwork\3c49m5.dgn

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CHECKED BY:	JKSI	DATE:	APR.'95
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APPROVED BY:	GWS	DATE:	APR.'95

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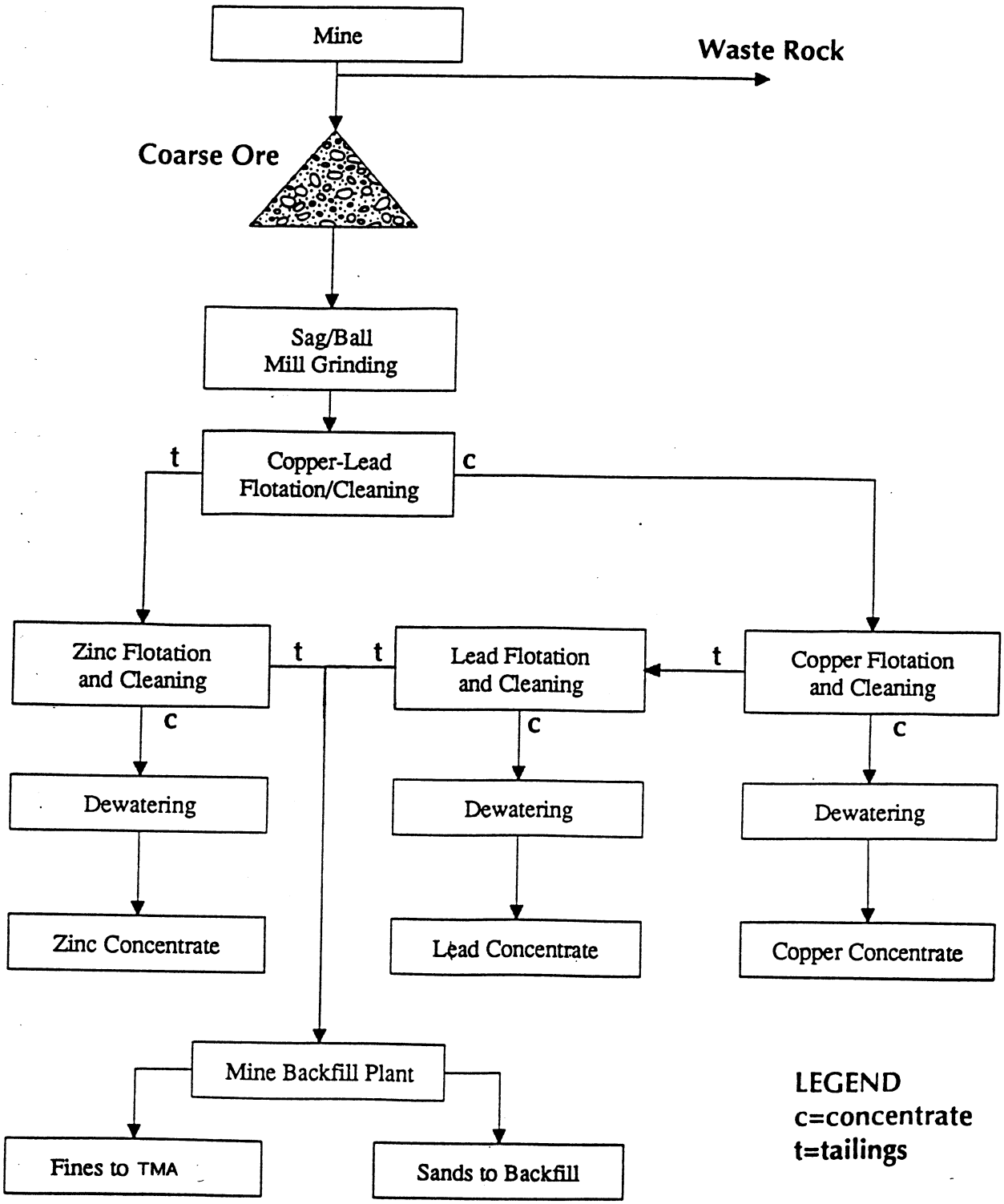
FIGURE 2-9

MINE DRAINAGE SCHEMATIC

Scale:	NOT TO SCALE	Date:	MARCH, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

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CHECKED BY:	JKSI	DATE:	APR.'95
APPROVED BY:	PAE	DATE:	APR.'95
APPROVED BY:	GWS	DATE:	APR.'95

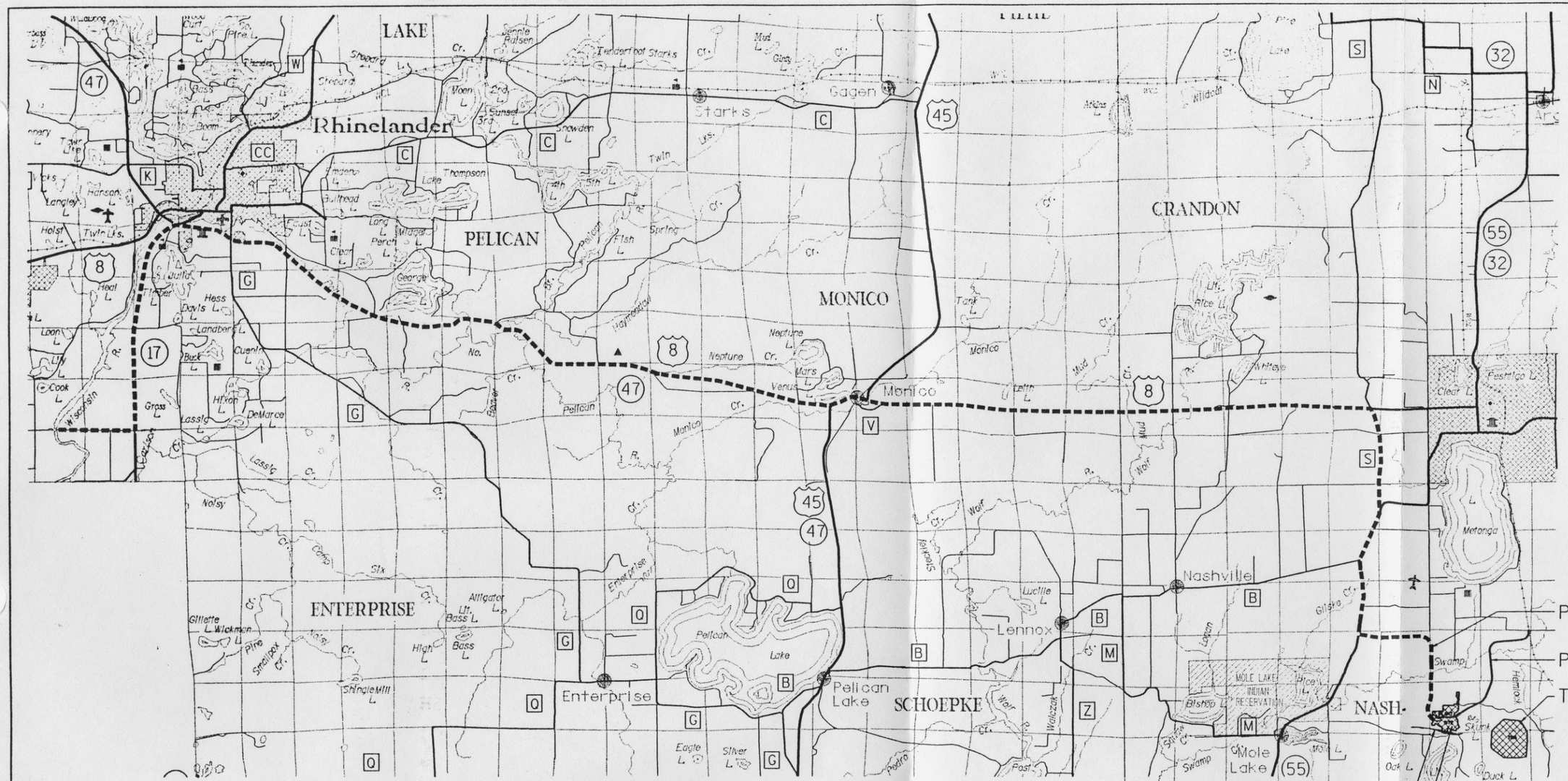
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FIGURE 2-10
CONCENTRATOR PROCESS FLOWSHEET

Scale:	NOT TO SCALE	Date:	MARCH, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

/d3/dl/users/jpr1/3c49m5.dgn

93C049



LEGEND

- U.S. OR STATE HWY
- COUNTY TRUNK ROAD
- TOWN ROAD
- SECTION LINE
- (8) U.S. HWY NO.
- (47) STATE HWY NO.
- [W] COUNTY HIGHWAY LETTER
- PROPOSED WASTEWATER PIPELINE

NOTES:

1. BASE MAP DERIVED FROM COUNTY MAPS PREPARED BY THE WISCONSIN DEPARTMENT OF TRANSPORTATION.
2. ORE BODY OUTLINE IS REPRESENTATIVE OF THE SUBCROP AT THE BASE OF THE OVERBURDEN.

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

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CHECKED BY:	JKSI	DATE: MAY.'95
APPROVED BY:	PAE	DATE: MAY.'95
APPROVED BY:	GWS	DATE: MAY.'95

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FIGURE 2-11

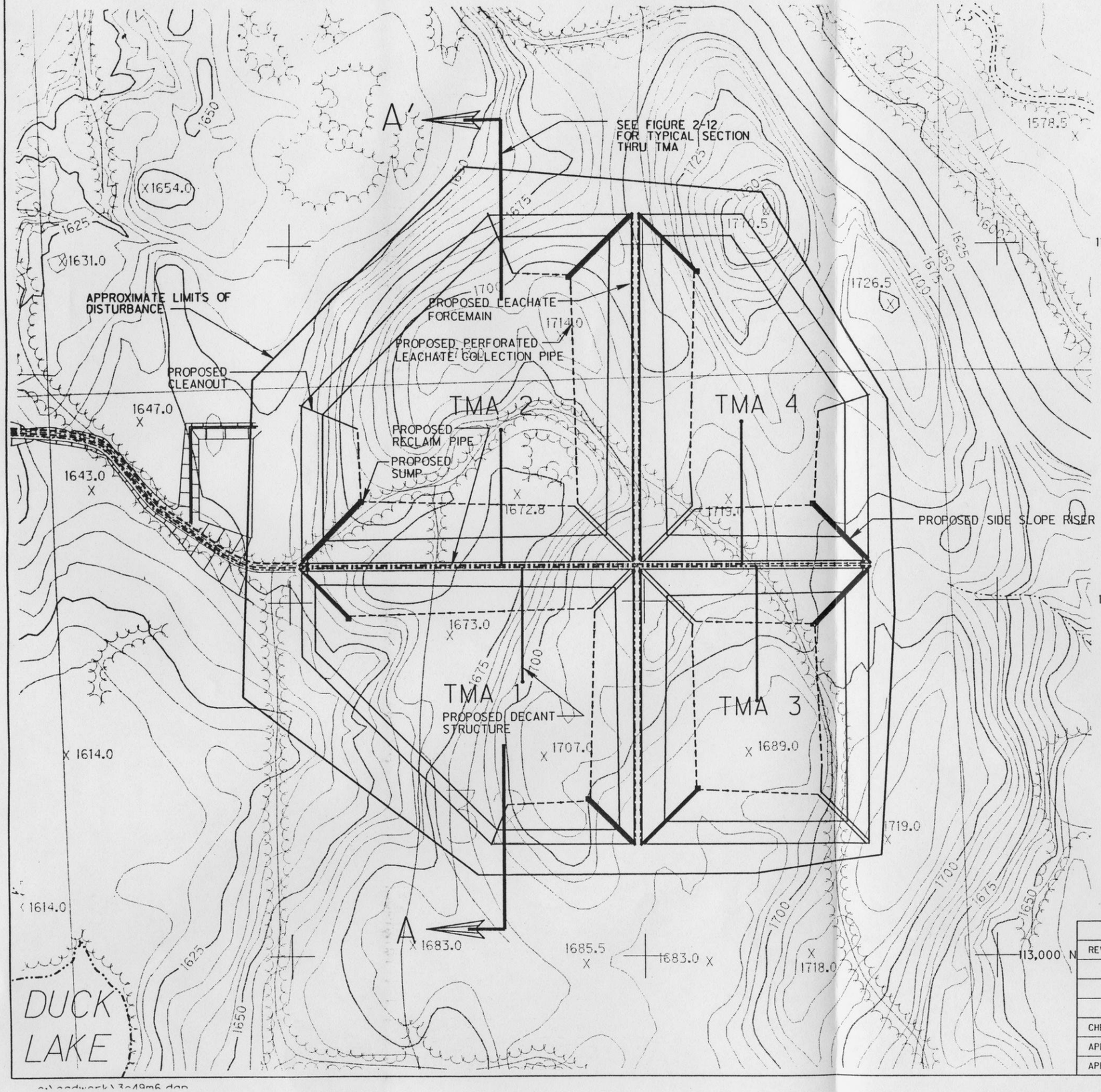
PROPOSED PIPELINE ROUTE FOR WISCONSIN RIVER DISCHARGE

Scale: 0 6000' 12,000' Date: MAY, 1995

Prepared By: Foth & Van Dyke By: GAM

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93C049



N

LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- 1692.0 SPOT ELEVATION
- SECTION LINE
- TMA APPROXIMATE LIMITS OF DISTURBANCE

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

SCALE
0 300' 600'

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

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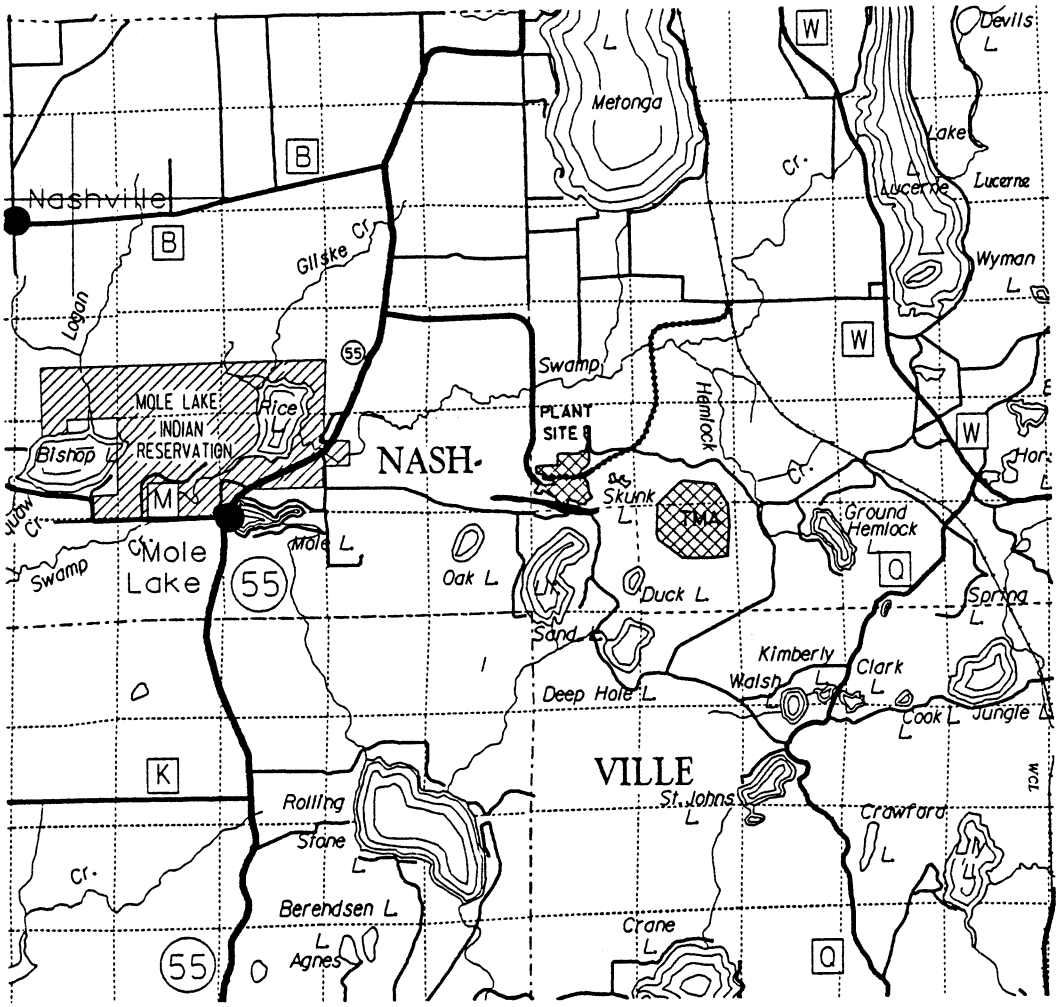
Crandon Mining Company

FIGURE 2-12
PROPOSED TMA LAYOUT

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	MRS

c:\cadwork\3a19m6.dgn

93C049



N

LEGEND

- U.S. OR STATE HWY
- COUNTY TRUNK ROAD
- TOWN ROAD
- COUNTY BOUNDARY
- CIVIL TOWN BOUNDARY
- SECTION LINE
- (8) U.S. HWY NO.
- (47) STATE HWY NO.
- [W] COUNTY HWY LETTER
- RESERVATION LANDS
- UNINCORPORATED VILLAGE
- ORE BODY
- PROPOSED ACCESS ROAD
- PROPOSED RAILROAD SPUR
- PROPOSED FACILITIES

NOTES:

1. BASE MAP DERIVED FROM COUNTY MAPS PREPARED BY THE WISCONSIN DEPARTMENT OF TRANSPORTATION.
2. ORE BODY OUTLINE IS REPRESENTATIVE OF THE SUBCROP AT THE BASE OF THE OVERBURDEN.

Foth & Van Dyke

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APPROVED BY:	GMS	DATE: MAY.'95
APPROVED BY:	GWS	DATE: MAY.'95

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FIGURE 3.2-1

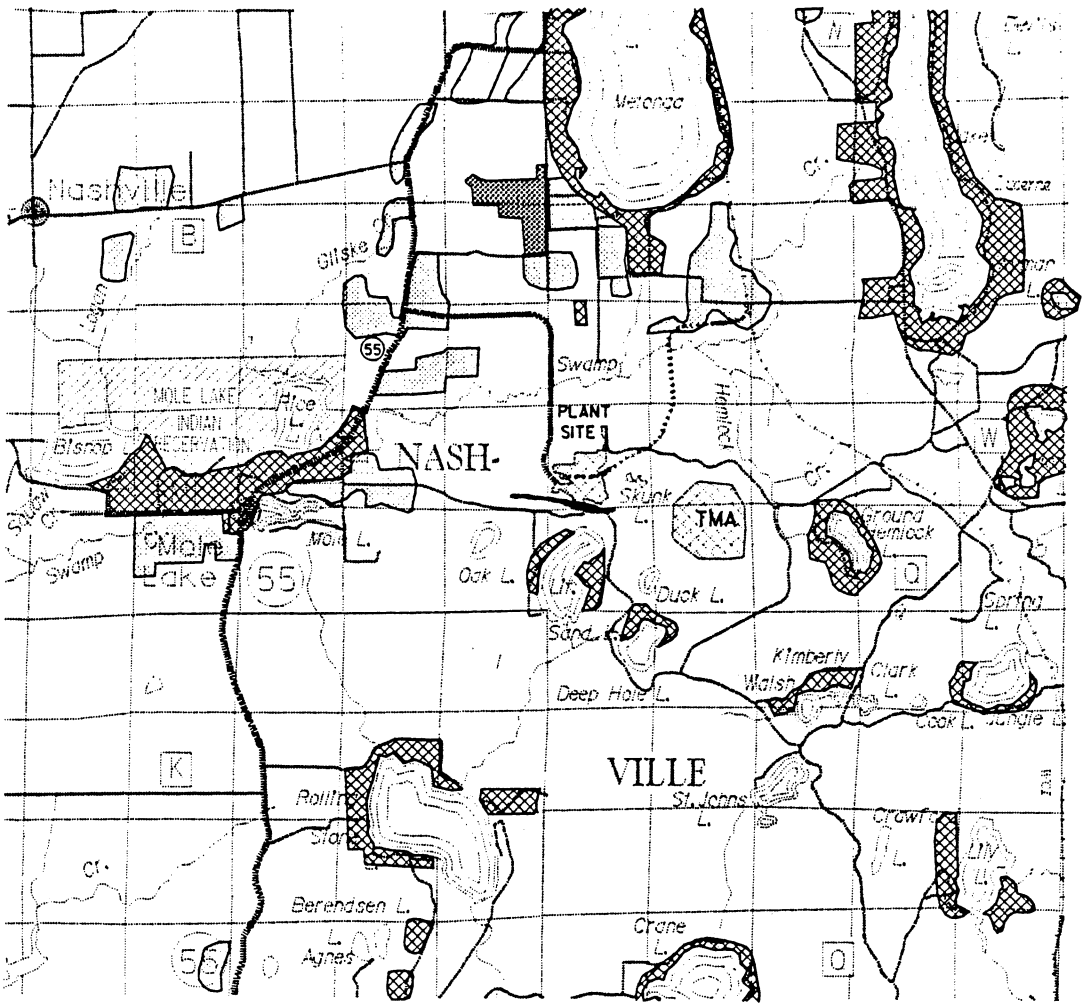
LAND USE STUDY AREA

Scale: 0 10,000' 20,000' Date: MAY, 1995

Prepared By: Foth & Van Dyke By: JOW

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93C049



LEGEND

- U.S. OR STATE HWY
- COUNTY TRUNK ROAD
- TOWN ROAD
- COUNTY BOUNDARY
- CIVIL TOWN BOUNDARY
- SECTION LINE
- (8) U.S. HWY NO.
- (47) STATE HWY NO.
- W COUNTY HWY LETTER
- RESERVATION LANDS
- UNINCORPORATED VILLAGE
- ORE BODY
- PROPOSED ACCESS ROAD
- PROPOSED RAILROAD SPUR
- PROPOSED FACILITIES
- AGRICULTURE
- FOREST LANDS/WETLANDS
- RESIDENTAL/INSTITUTIONAL
- TRANSPORTATION

NOTES:

1. BASE MAP DERIVED FROM COUNTY MAPS PREPARED BY THE WISCONSIN DEPARTMENT OF TRANSPORTATION.
2. ORE BODY OUTLINE IS REPRESENTATIVE OF THE SUBCROP AT THE BASE OF THE OVERBURDEN.
3. MINIMUM LAND USE AREA DEPICTED 40 ACRES.

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE:	MAY.'95
APPROVED BY:	GMS	DATE:	MAY.'95
APPROVED BY:	GWS	DATE:	MAY.'95

Crandon Mining Company

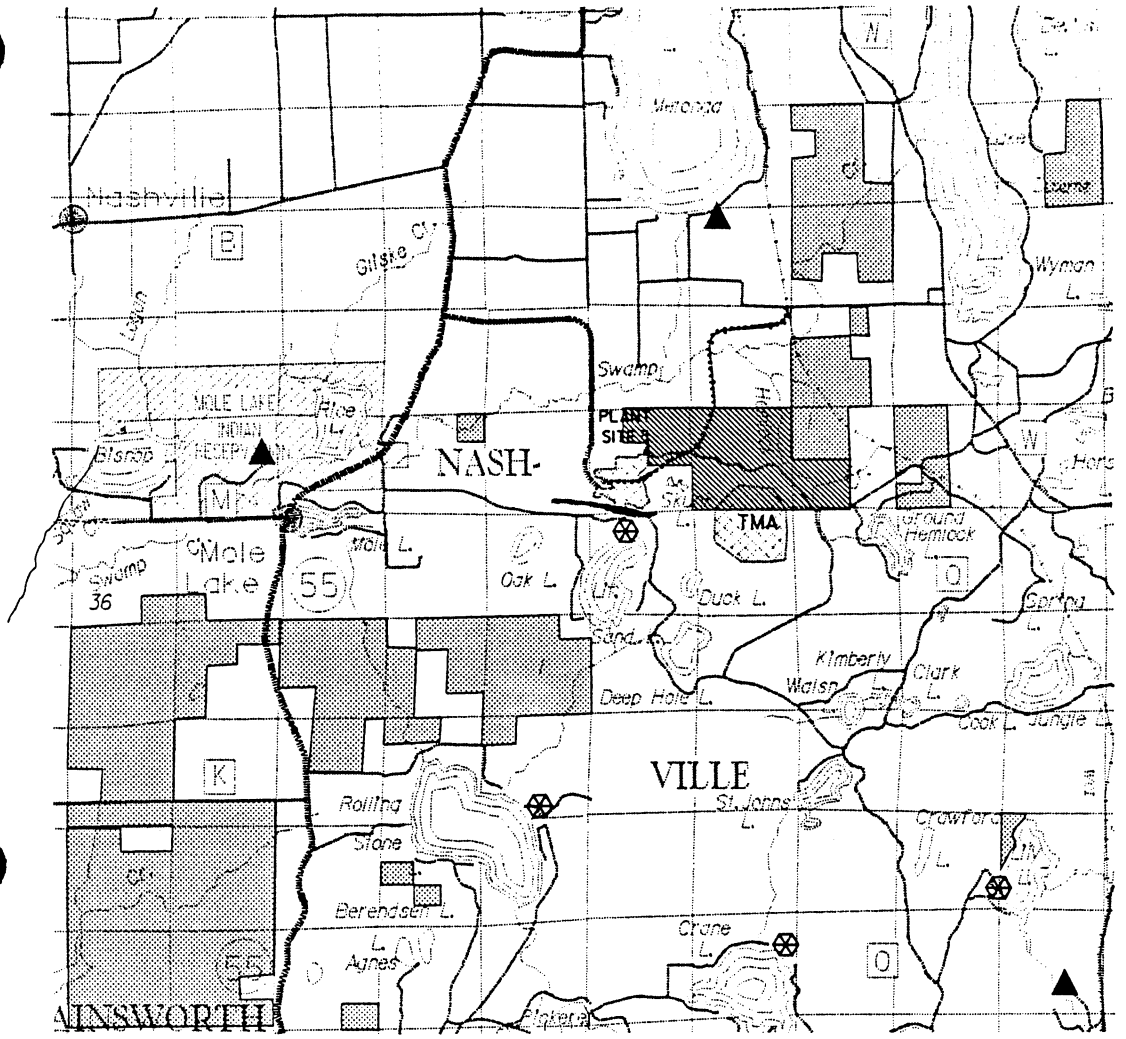
FIGURE 3.2-2
LAND USE IN THE STUDY AREA

Scale: 0 10,000' 20,000' Date: MAY, 1995

Prepared By: Foth & Van Dyke By: JOW

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930049



LEGEND

- U.S. OR STATE HWY
- COUNTY TRUNK ROAD
- TOWN ROAD
- COUNTY BOUNDARY
- CIVIL TOWN BOUNDARY
- SECTION LINE
- (8) U.S. HWY NO.
- (47) STATE HWY NO.
- [W] COUNTY HWY LETTER
- RESERVATION LANDS
- UNINCORPORATED VILLAGE
- ORE BODY
- PROPOSED ACCESS ROAD
- PROPOSED RAILROAD SPUR
- PROPOSED FACILITIES
- BOAT ACCESS
- PARK/CAMPGROUND
- PUBLIC LANDS
- PUBLIC LANDS OPTIONED BY CMC

NOTES:

1. BASE MAP DERIVED FROM COUNTY MAPS PREPARED BY THE WISCONSIN DEPARTMENT OF TRANSPORTATION.
2. ORE BODY OUTLINE IS REPRESENTATIVE OF THE SUBCROP AT THE BASE OF THE OVERBURDEN.
3. SOURCE: NORTH CENTRAL WISCONSIN REGIONAL PLANNING COMMISSION; LANGLADE COUNTY OUTDOOR RECREATION PLAN 1991-1995, JANUARY 1991; FOTH & VAN DYKE, COMPREHENSIVE OUTDOOR RECREATION PLAN, FOREST COUNTY FORESTRY DEPARTMENT, NOVEMBER, 1988; FOREST COUNTY RECREATION MAP, 1994.

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY.'95
APPROVED BY:	GMS	DATE: MAY.'95
APPROVED BY:	GWS	DATE: MAY.'95

Crandon Mining Company

FIGURE 3.2-3

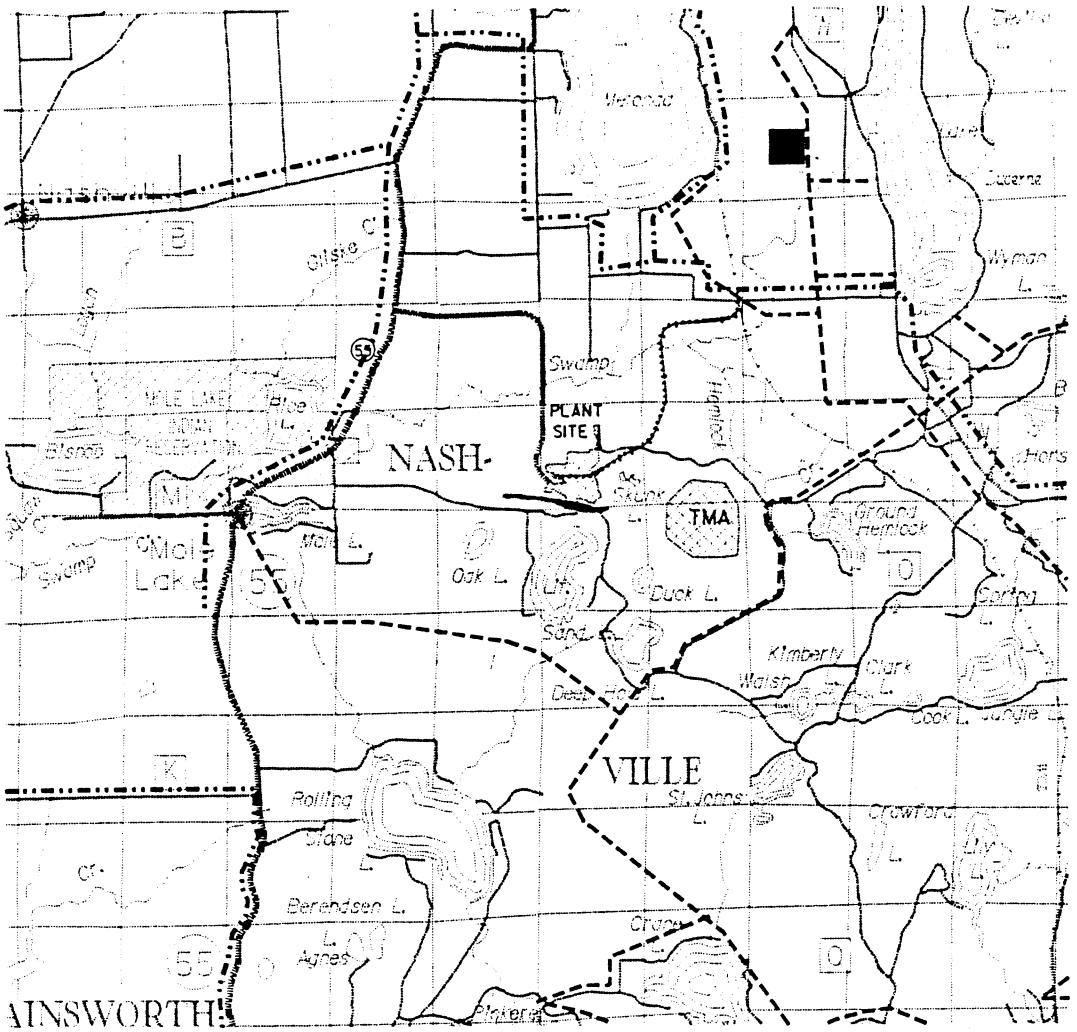
RECREATIONAL RESOURCES IN THE STUDY AREA

Scale: 0 10,000' 20,000' Date: MAY, 1995

Prepared By: Foth & Van Dyke By: JOW

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93C049



LEGEND

- U.S. OR STATE HWY
- COUNTY TRUNK ROAD
- TOWN ROAD
- COUNTY BOUNDARY
- CIVIL TOWN BOUNDARY
- SECTION LINE
- U.S. HWY NO.
- STATE HWY NO.
- COUNTY HWY LETTER
- RESERVATION LANDS
- UNINCORPORATED VILLAGE
- ORE BODY
- PROPOSED ACCESS ROAD
- PROPOSED RAILROAD SPUR
- PROPOSED FACILITIES
- ALL TERRAIN VEHICLE (ATV) TRAILS
- SNOWMOBILE TRAILS
- SUGGESTED BICYCLE ROUTES

NOTES:

1. BASE MAP DERIVED FROM COUNTY MAPS PREPARED BY THE WISCONSIN DEPARTMENT OF TRANSPORTATION.
2. ORE BODY OUTLINE IS REPRESENTATIVE OF THE SUBCROP AT THE BASE OF THE OVERBURDEN.

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APPROVED BY:	GMS	DATE: MAY,'95
APPROVED BY:	GWS	DATE: MAY,'95

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FIGURE 3.2-4

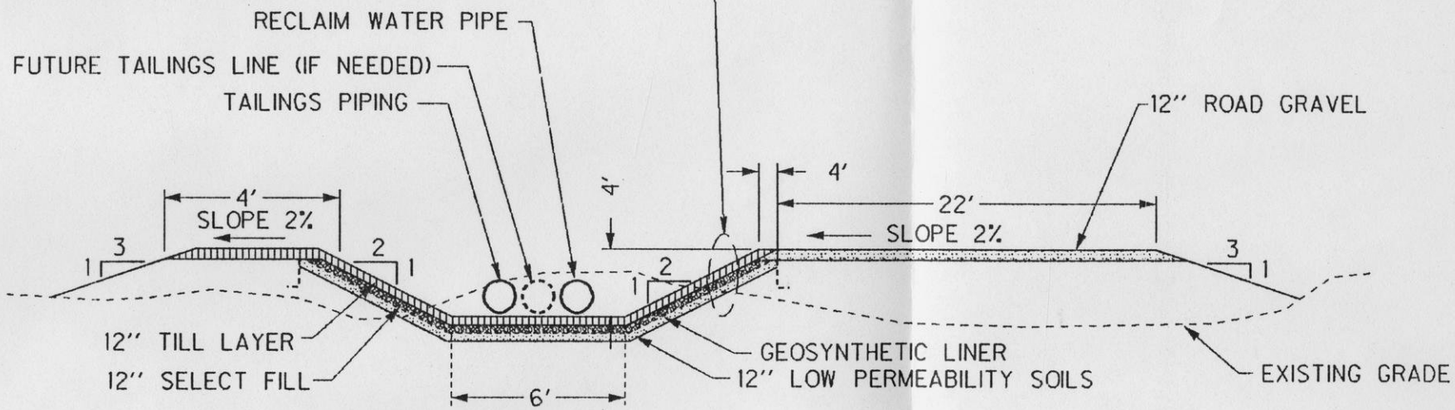
RECREATIONAL TRAILS IN THE STUDY AREA

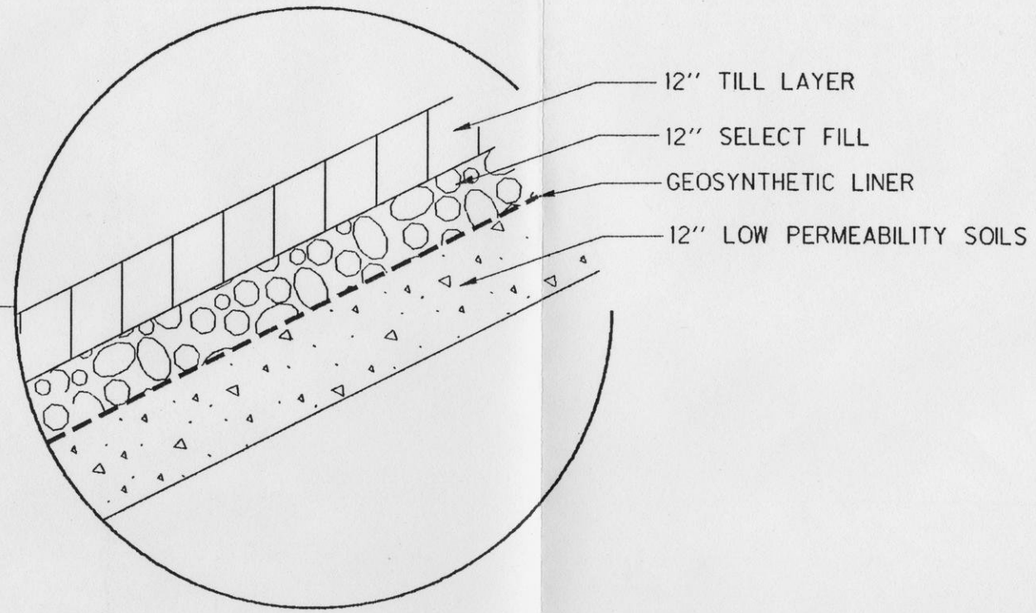
Scale: 0 10,000' 20,000' Date: MAY, 1995

Prepared By: Foth & Van Dyke By: JCW

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TYPICAL SECTION

NOT TO SCALE

Foth & Van Dyke

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CHECKED BY:	JKSI	DATE: MAY '95
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APPROVED BY:	PAE	DATE: MAY '95

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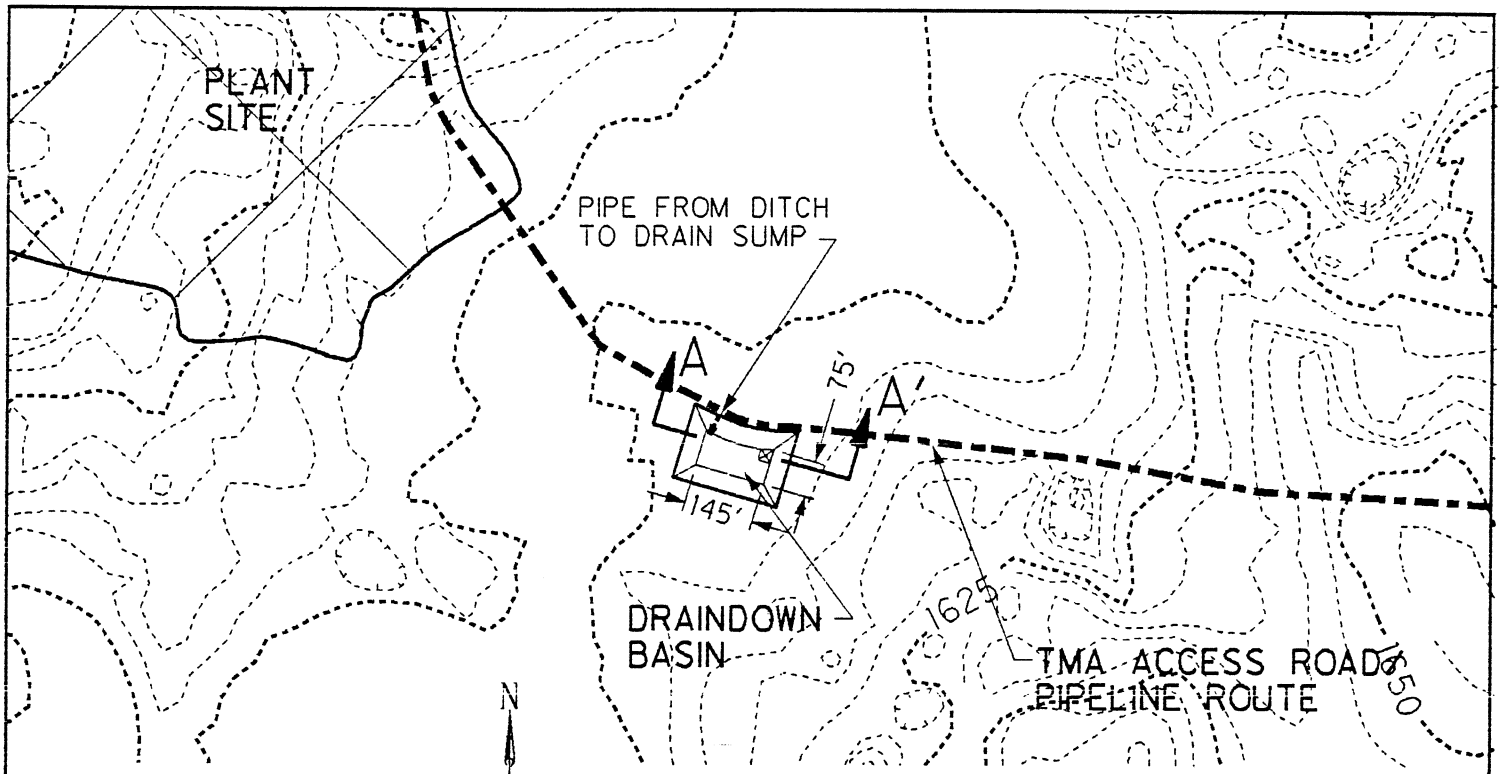
FIGURE 6.2-1

TYPICAL SECTION THROUGH
TMA ACCESS ROAD / PIPELINE CORRIDOR

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JOW

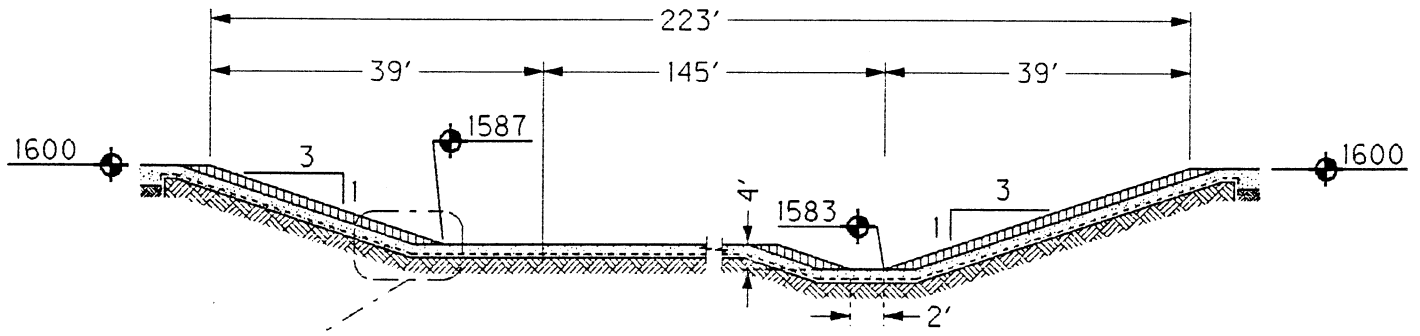
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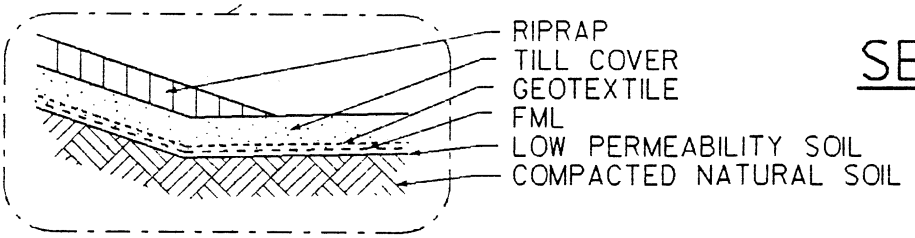
93C049



PLAN

SCALE: 1" = 400'





SECTION A - A'

NOT TO SCALE

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	NXP	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

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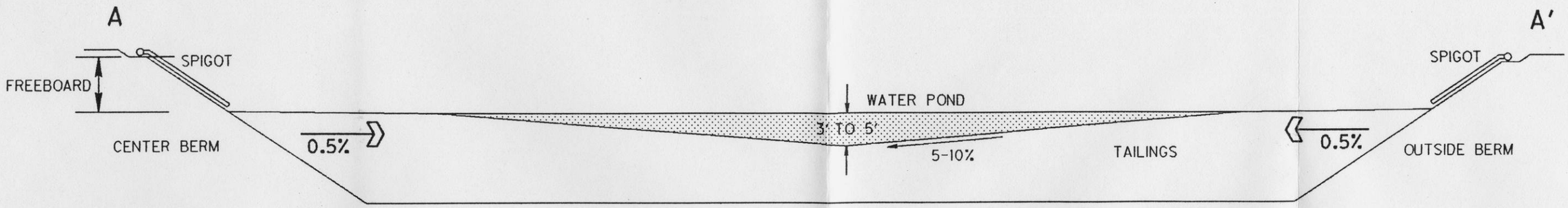
FIGURE 6.2-2

DRAINDOWN BASIN
PLAN AND SECTIONS

Scale:	AS SHOWN	Date:	MAY 1995
Prepared By:	Foth & Van Dyke	By:	ARJ

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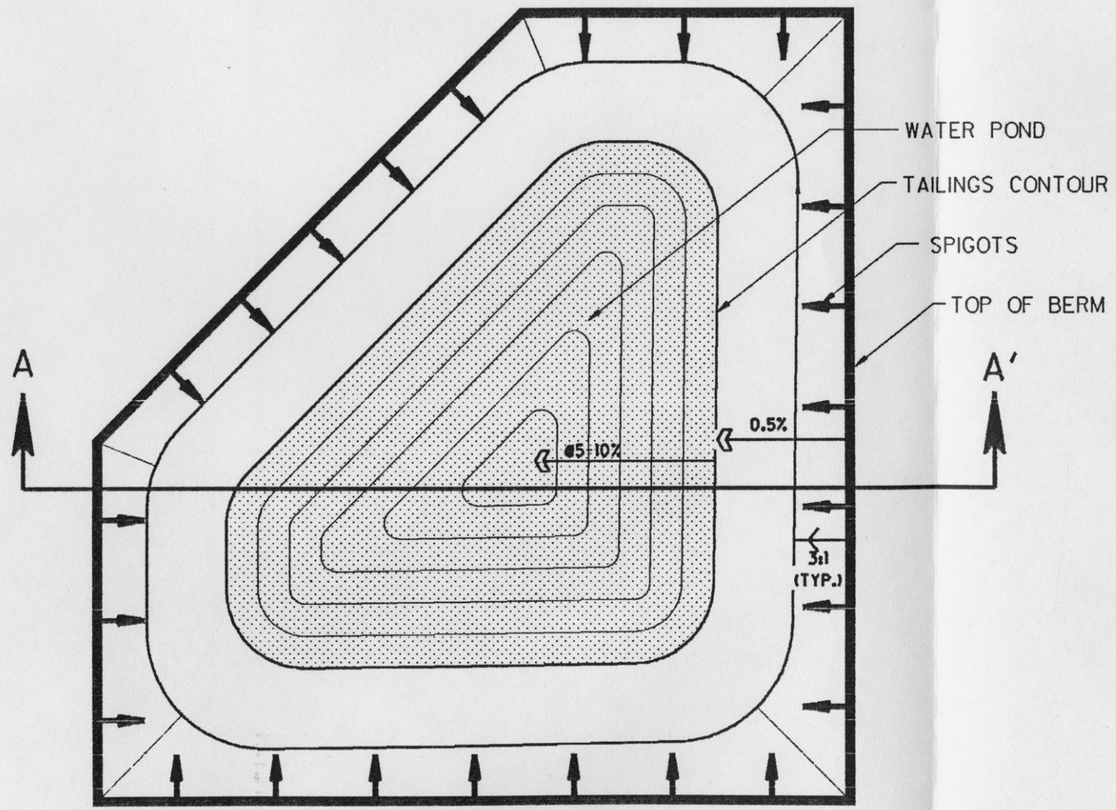
94C054



SECTION A-A'

INITIAL FILLING

NOTES: TAILINGS DEPOSITION IS SUBAERIAL RING DISCHARGE RESULTING IN A CENTRAL WATER POND.



PLAN VIEW

Foth & Van Dyke				
REVISED	DATE	BY	DESCRIPTION	
CHECKED BY:	JKSI		DATE: MAY '95	
APPROVED BY:	REM		DATE: MAY '95	
APPROVED BY:	GWS		DATE: MAY '95	

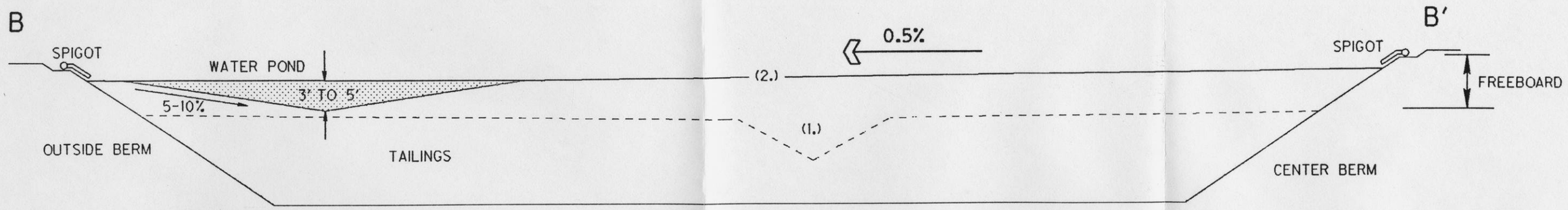
Crandon Mining Company

FIGURE 6.6-1
PLAN VIEW OF TYPICAL POND CONFIGURATION DURING FINAL TAILINGS DEPOSITION

Scale:	NOT TO SCALE	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	GAM

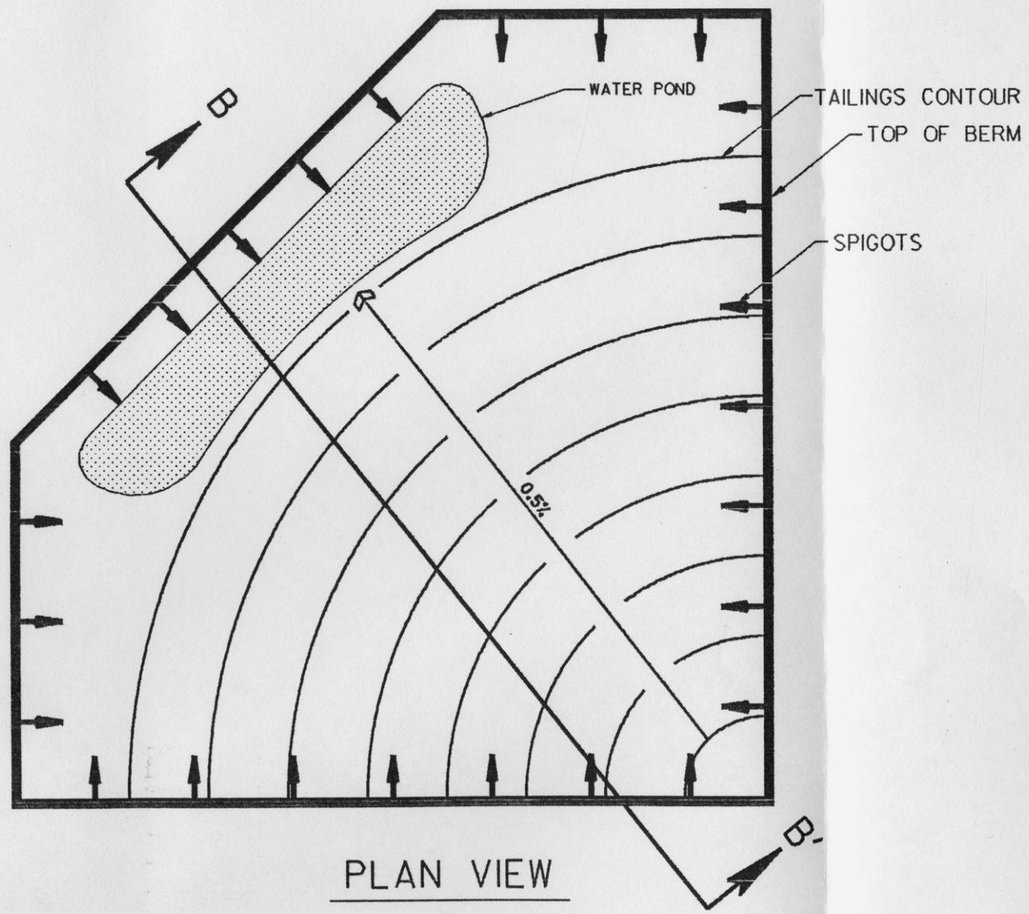
93C049

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SECTION B-B'
AFTER FINAL TAILINGS DISCHARGE

NOTES: (1.) TAILINGS SURFACE AT AN APPROXIMATE GRADE OF 0.5% TOWARD CENTER OF CELL
(2.) TAILINGS DEPOSITED PRIMARILY FROM CENTER BERM RADIALLY ACROSS THE POND TOWARD THE OUTSIDE BERM



PLAN VIEW

Foth & Van Dyke				
REVISED	DATE	BY	DESCRIPTION	
CHECKED BY:		JKSI		DATE: MAY '95
APPROVED BY:		REM		DATE: MAY '95
APPROVED BY:		GWS		DATE: MAY '95

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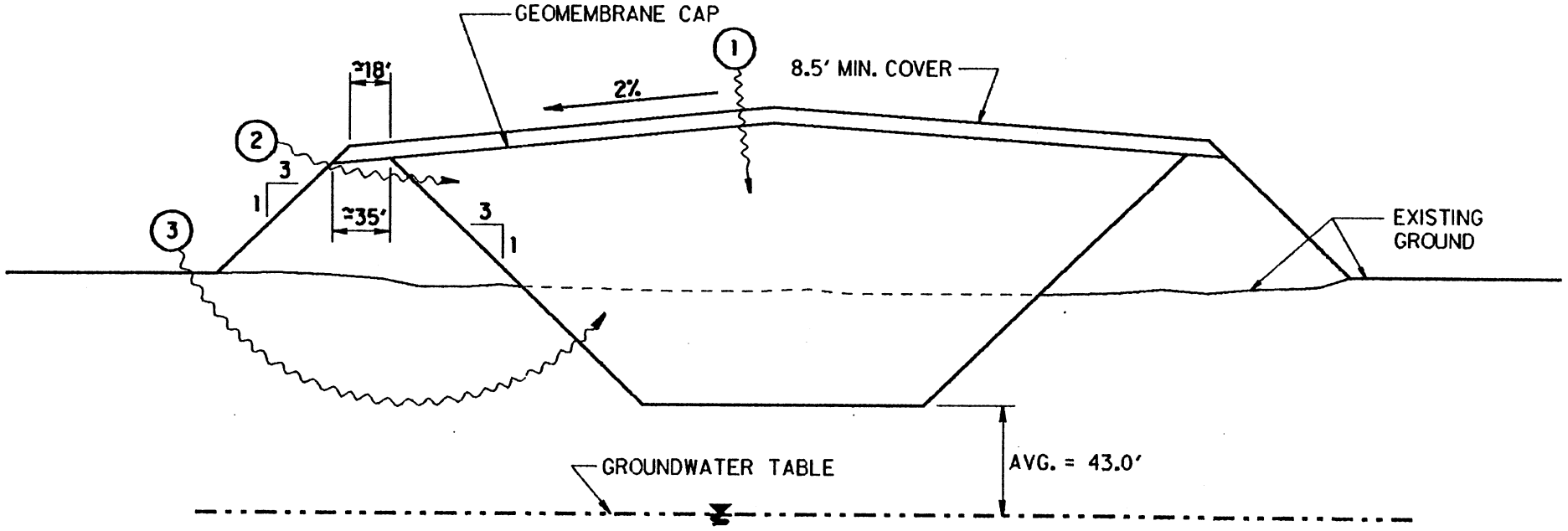
FIGURE 6.6-2
PLAN VIEW OF TYPICAL POND CONFIGURATION DURING FINAL DEPOSITION

Scale:	NOT TO SCALE	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	GAM

93C049

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c:\cadwork\4c54f4.dgn



OXYGEN ENTRY PATHWAYS

(1) THROUGH FINAL COVER

(2) THROUGH EMBANKMENT

(3) THROUGH FOUNDATION SOILS

Foth & Van Dyke			
REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	REM	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

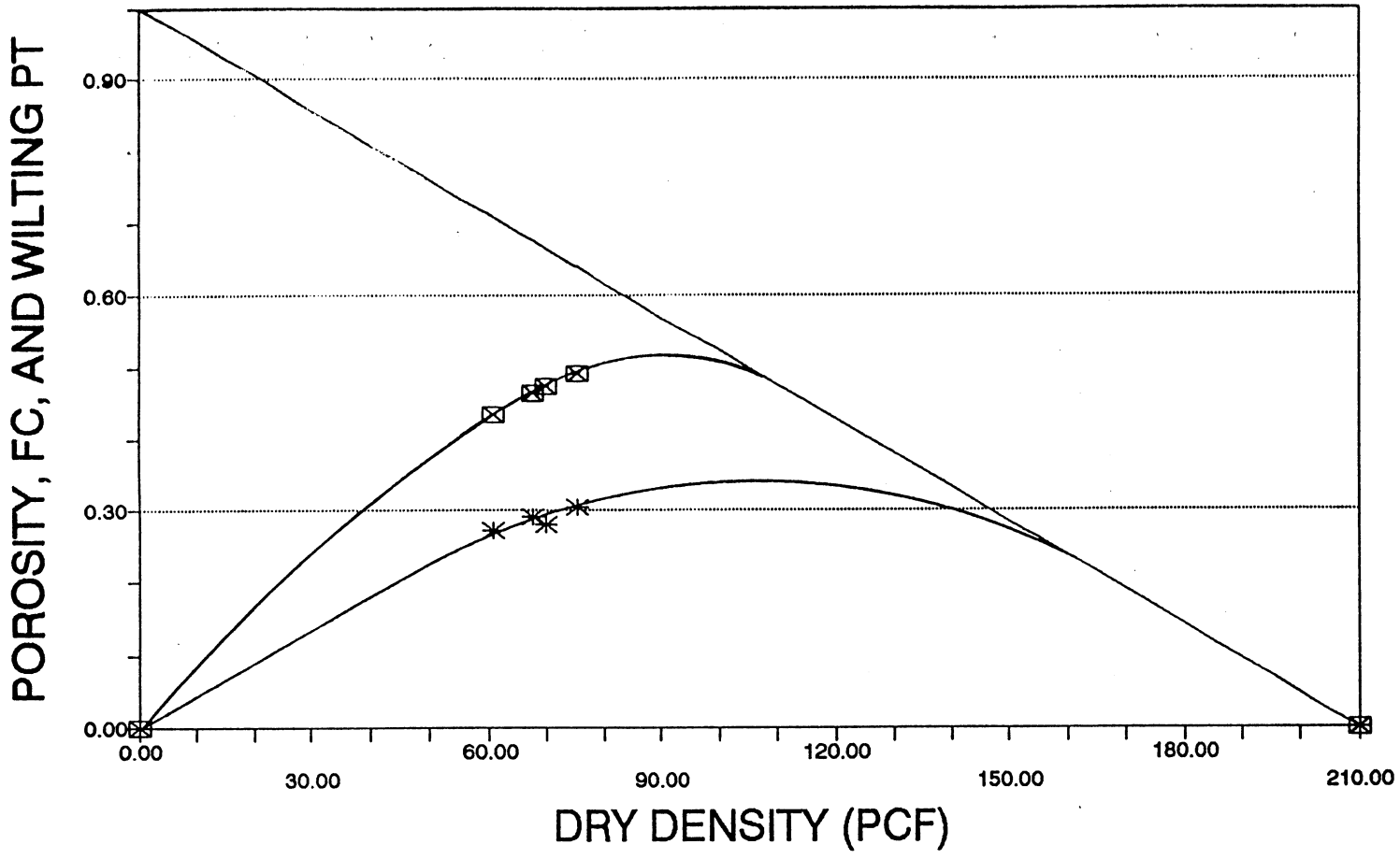
Crandon Mining Company

FIGURE 6.6-3

POTENTIAL PATHWAYS FOR OXYGEN ENTRY TO TAILINGS

Scale:	NOT TO SCALE	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JPRI

93C049



Foth & Van Dyke

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APPROVED BY:	REM	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

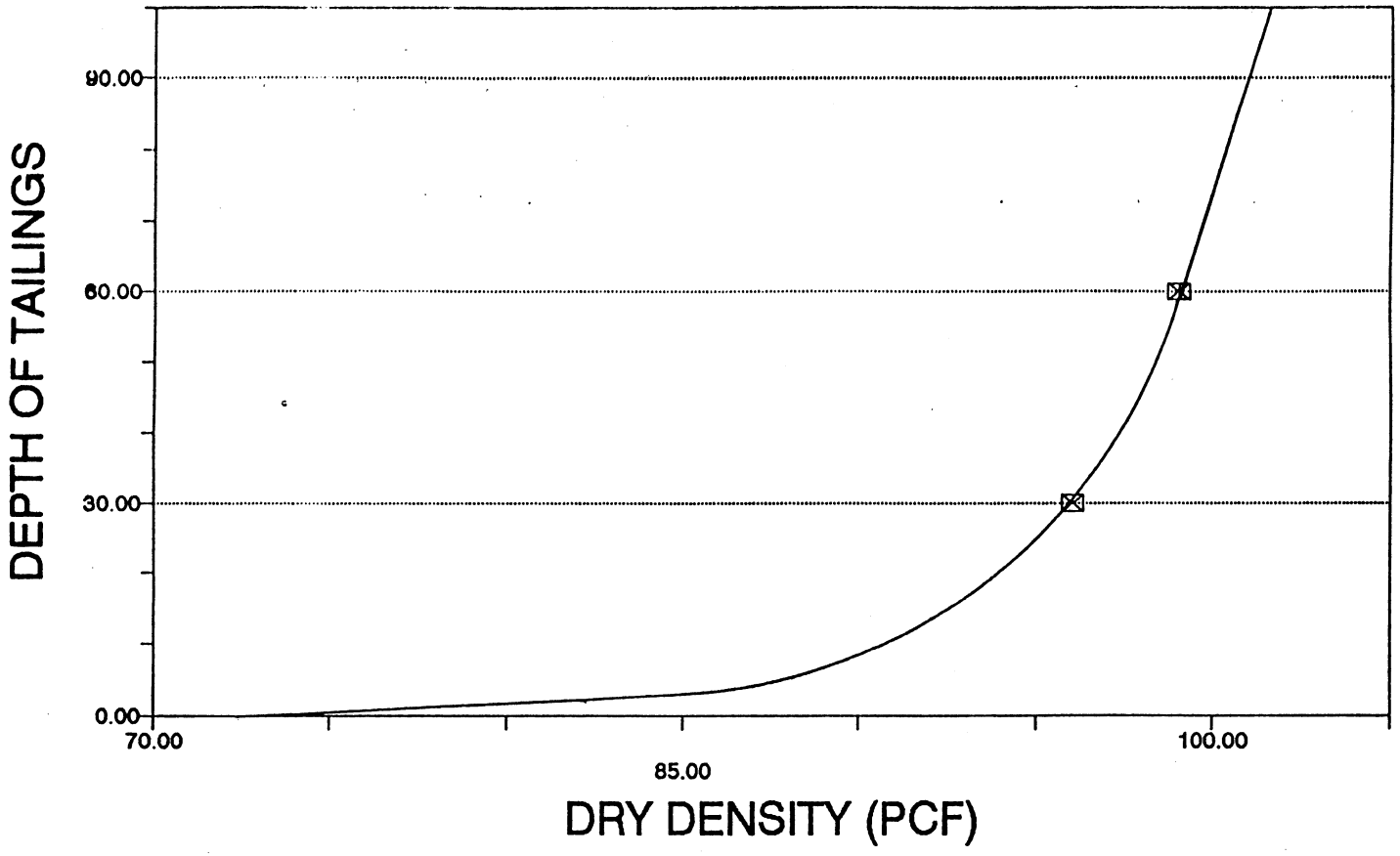
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FIGURE 6.7-1

TMA WATER BALANCE POROSITY, FIELD CAPACITY AND WILTING POINT VS. DENSITY

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

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REVISED	DATE	BY	DESCRIPTION

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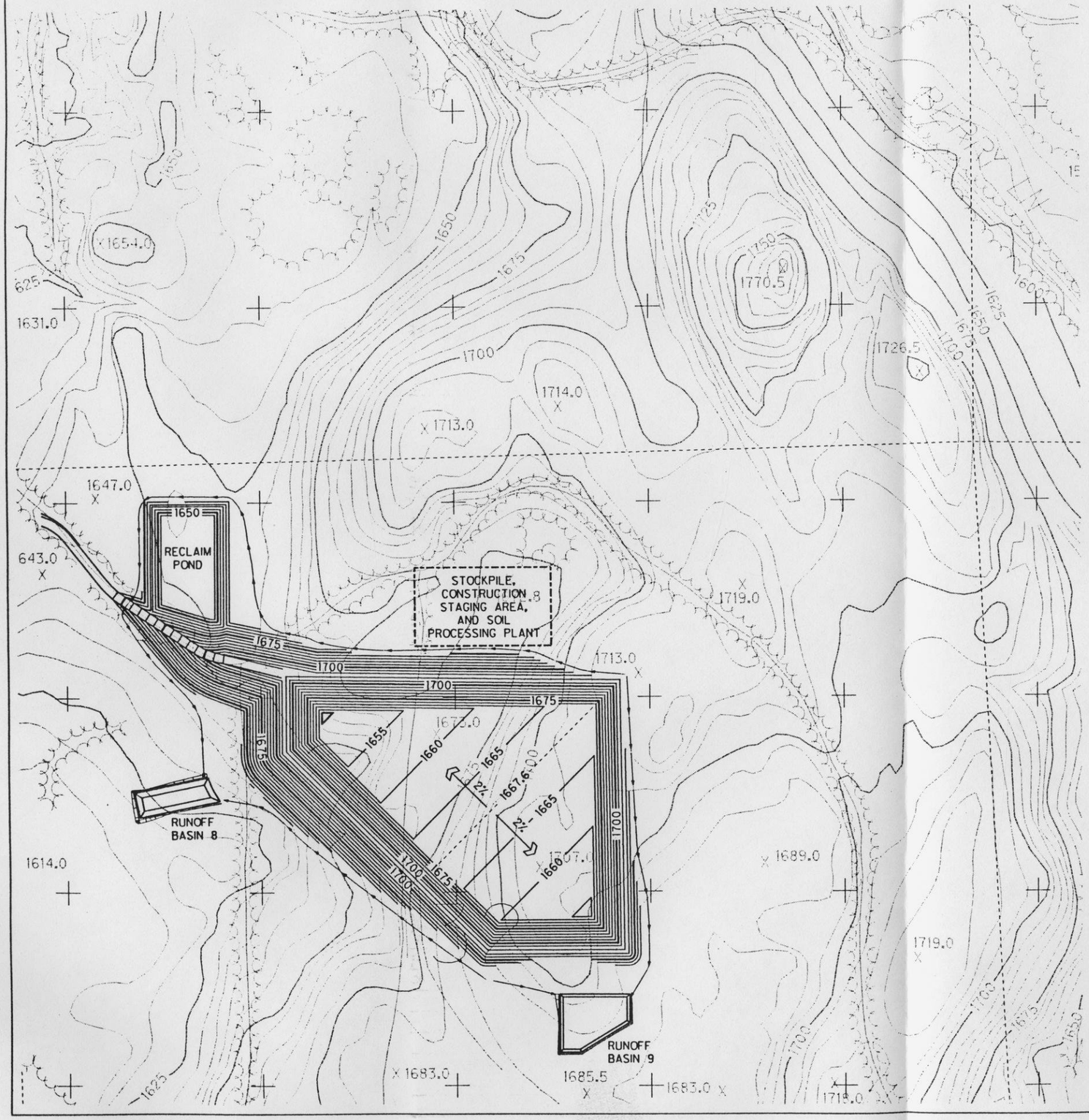
Crandon Mining Company

FIGURE 6.7-2

TMA WATER BALANCE,
DEPTH VS. DRY DENSITY

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

93C049



LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- —1675— EXISTING CONTOUR
- x 1692.0 SPOT ELEVATION
- SECTION LINE
- —1660— PROPOSED CONTOUR
- PROPOSED DRAINAGE FLOW
- 2% PROPOSED SLOPE DIRECTION

N

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1''=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

0 300' 600'
SCALE

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

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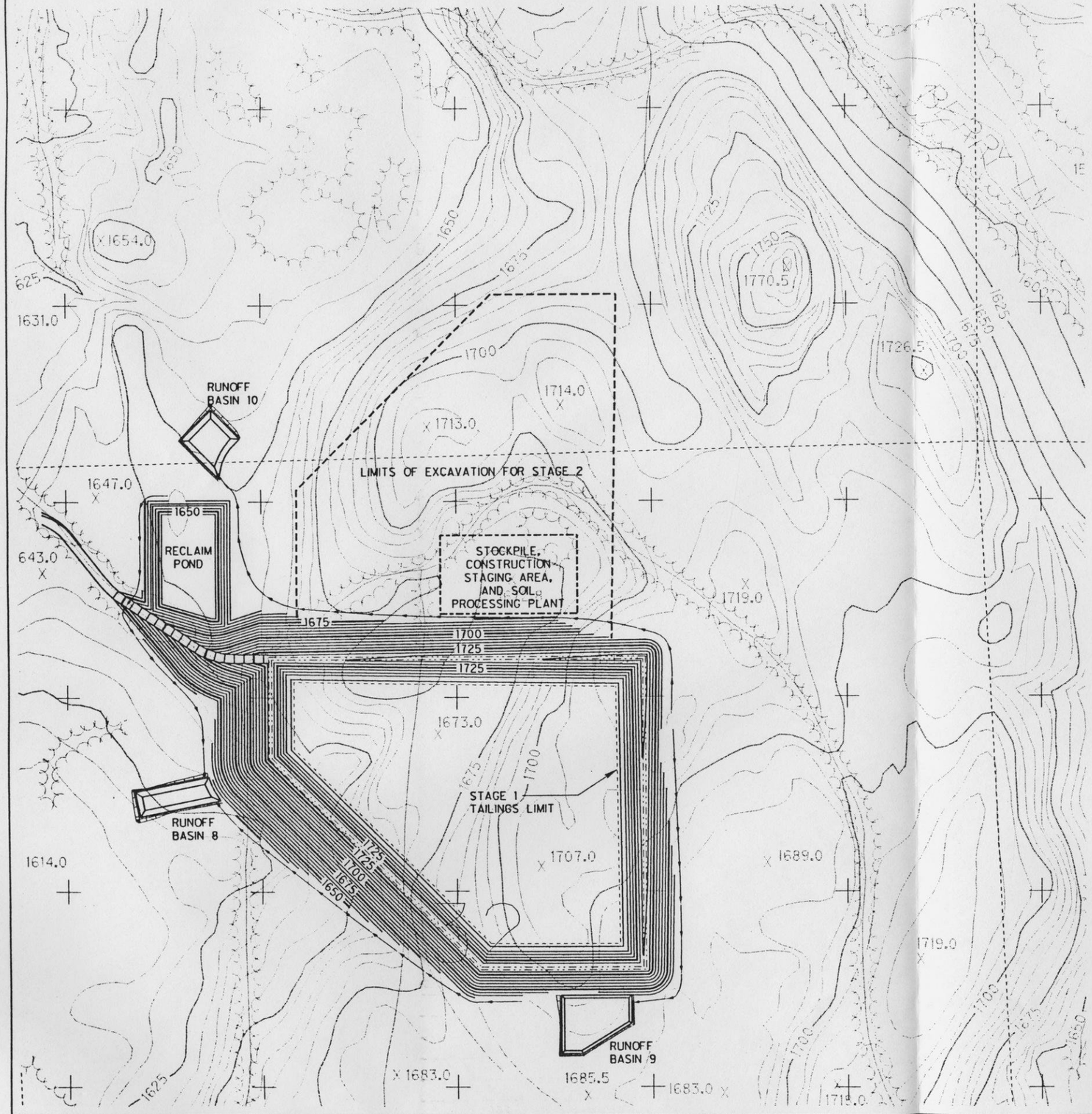
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FIGURE 6.12-1
TMA 1
STAGE 1 CONSTRUCTION

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JRB2

/d3/dl/users/jpr1/4c54fs1.dgn

93C049



N

LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- —1675— EXISTING CONTOUR
- x 1692.0 SPOT ELEVATION
- SECTION LINE
- —1660— PROPOSED CONTOUR
- PROPOSED DRAINAGE FLOW
- 2% PROPOSED SLOPE DIRECTION

NOTES:

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3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

0 300' 600'
SCALE

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

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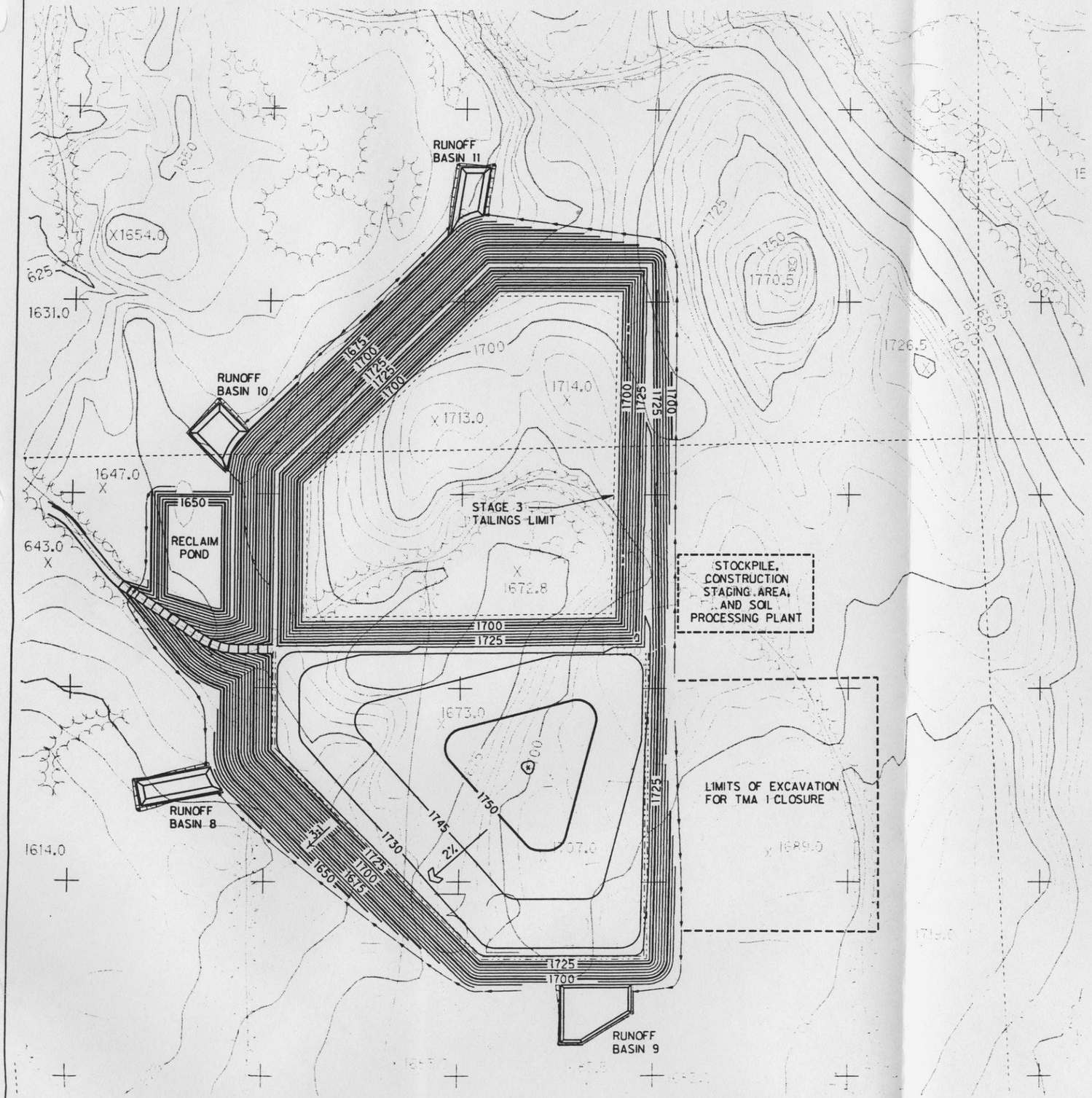
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APPROVED BY:	GWS	DATE: MAY '95

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FIGURE 6.12-2
TMA I STAGE 2 CONSTRUCTION

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JRB2

/d3/d1/users/jpr1/4c54fs2.dgn



LEGEND

- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- x 1692.0 SPOT ELEVATION
- SECTION LINE
- 1660 — PROPOSED CONTOUR
- PROPOSED DRAINAGE FLOW
- 2% PROPOSED SLOPE DIRECTION

N

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
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3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

0 300' 600'

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

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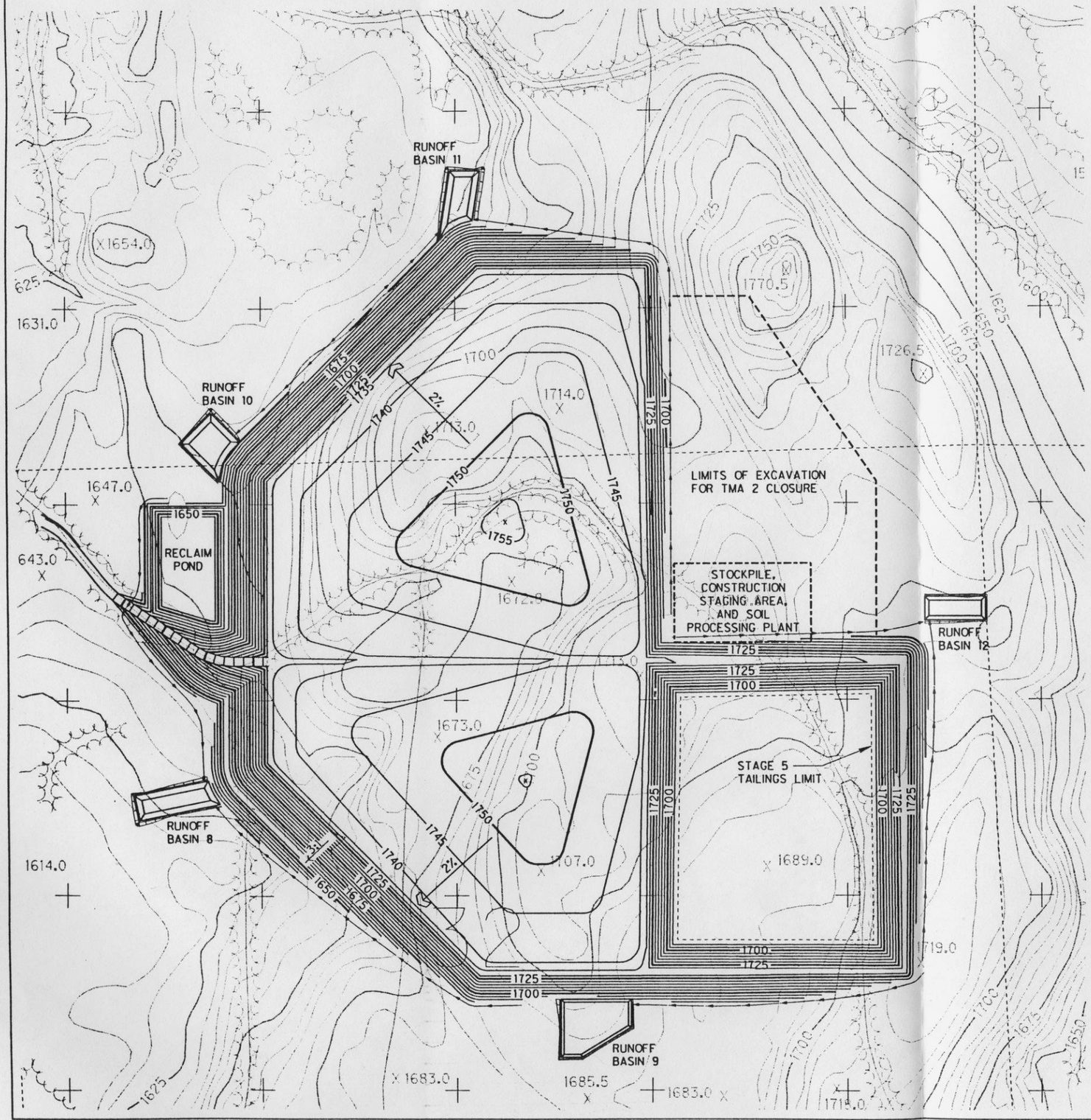
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APPROVED BY:	REM	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

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FIGURE 6.12-3
TMA 1 CLOSURE AND
TMA 2 CONSTRUCTION

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JRB2

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N

LEGEND

- EXISTING ROAD
- —1675— EXISTING CONTOUR
- x 1692.0 SPOT ELEVATION
- SECTION LINE
- — 1660 — PROPOSED CONTOUR
- PROPOSED DRAINAGE FLOW
- 2% PROPOSED SLOPE DIRECTION

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1''=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

0 300' 600'

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

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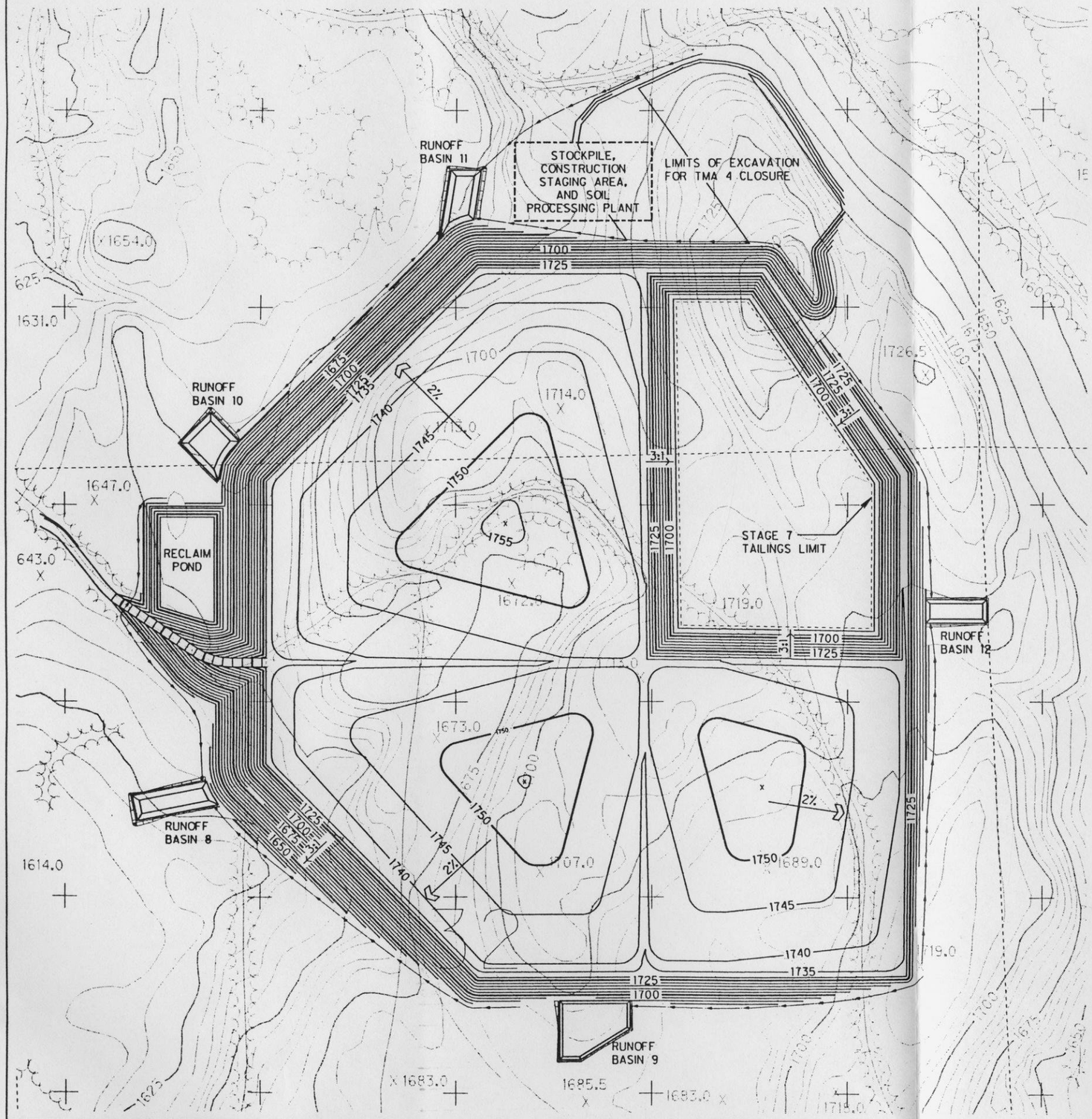
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FIGURE 6.12-4
TMA 2 CLOSURE AND
TMA 3 CONSTRUCTION

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JRB2

/d3/d1/users/jpr1/4c54fs6.dgn

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LEGEND

- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- x 1692.0 SPOT ELEVATION
- SECTION LINE
- 1660 — PROPOSED CONTOUR
- PROPOSED DRAINAGE FLOW
- 2% PROPOSED SLOPE DIRECTION

N

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1''=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
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3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

0 300' 600'

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

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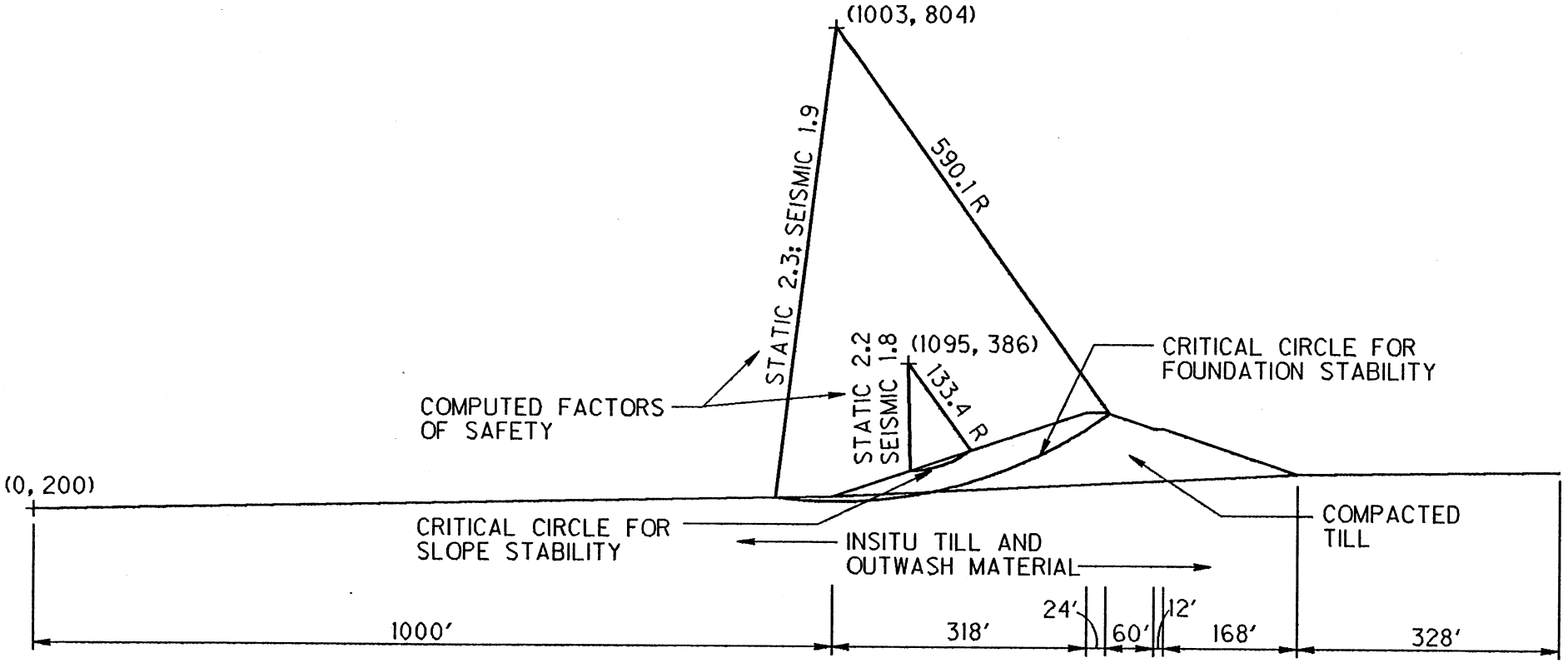
FIGURE 6.12-5
TMA 3 CLOSURE AND
TMA 4 CONSTRUCTION

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JRB2

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SLOPE STABILITY ANALYSIS

SCALE: 1" = 200'

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APPROVED BY:		GWS		DATE: MAY '95

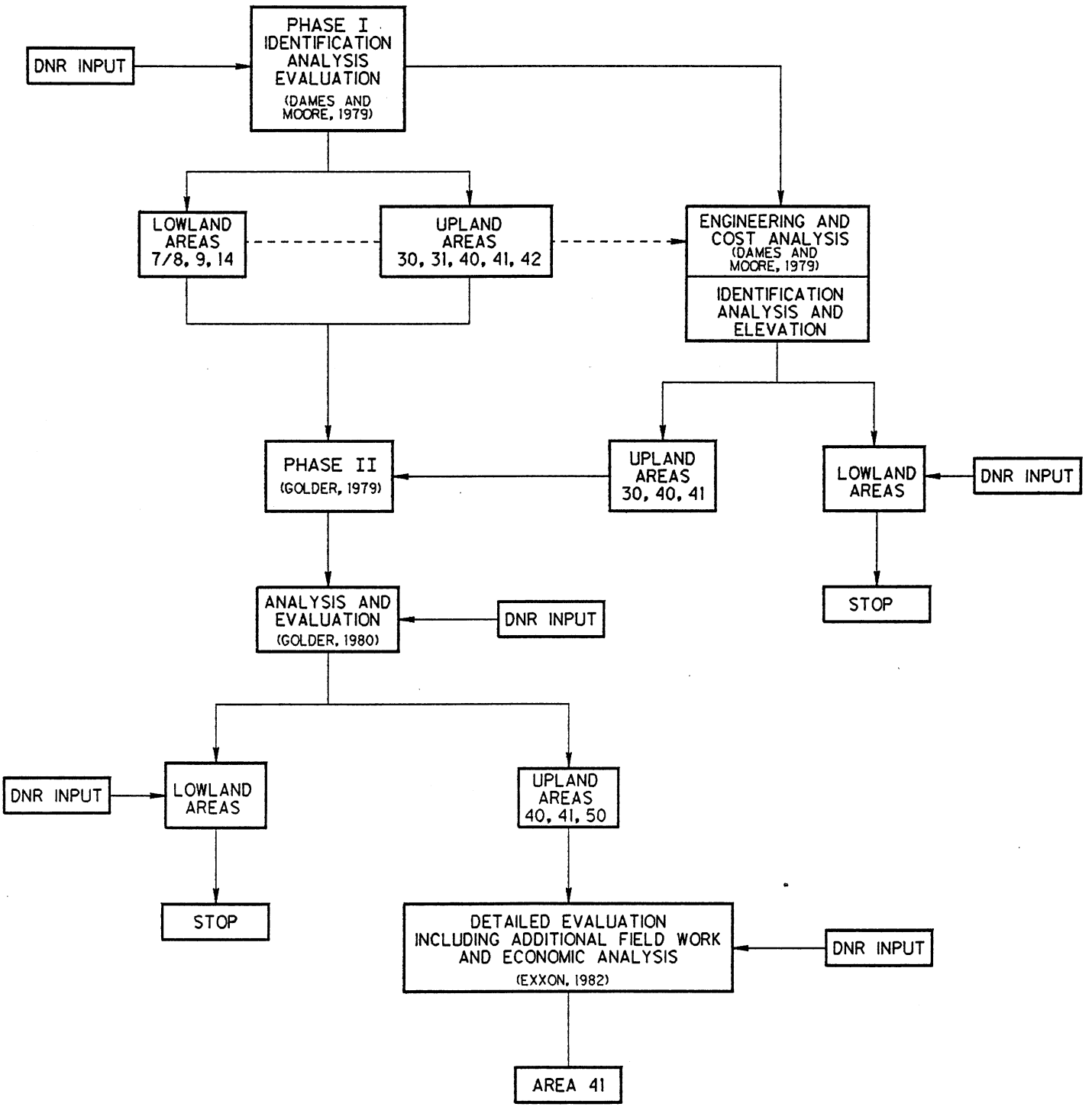
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FIGURE 6.15.1

SLOPE STABILITY ANALYSIS

Scale:	1" = 200'	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JPR

93C049



SOURCE: EXXON, 1985

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APPROVED BY:	GWS	DATE: MAY '95

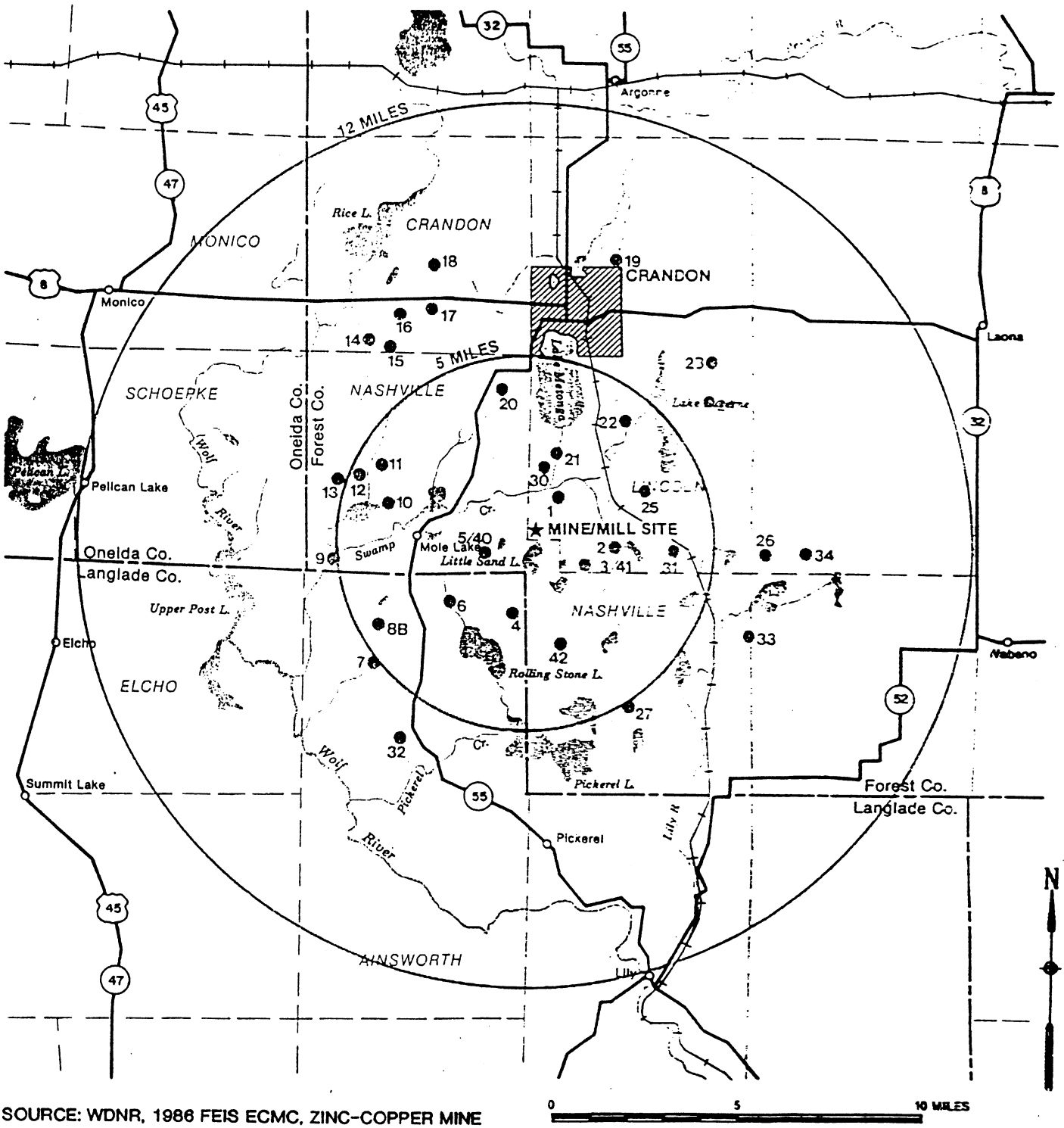
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FIGURE 10.1-1
SITE SCREENING PROCESS
FOR THE TMA

Scale:	NOT TO SCALE	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JOW

/d3/d1/users/jpr1/4c62f1.dgn

93C049



SOURCE: WDNR, 1986 FEIS ECMC, ZINC-COPPER MINE

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APPROVED BY:	GWS	DATE: MAY '95

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FIGURE 10.1-2

ALTERNATIVE TMA SITE LOCATIONS IDENTIFIED IN THE 1980'S

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	JOW

/d3/d1/users/jpr1/4c62f5.dgn

93C049

2282000 E
2284000 E
2286000 E
2288000 E
2290000 E
118000 N
116000 N
114000 N
112000 N

F12
F15
F15 BUR OAK WATERSHED
F33
M4
HEMLOCK CREEK WATERSHED
N1
16
F81
17
14
13
15
F31
1986 PROPOSED TMA AND RECLAIM POND FOOTPRINT
M2
M1
M3
HEMLOCK CREEK
DUCK LAKE WATERSHED
F32
M6
M5
F30
F29
F28
K3
F66
18
F27
19
DEEP HOLE LAKE WATERSHED
F65
L1
DUCK LAKE
20
F64
F63
K2
F25
F26
K1
F22

N

LEGEND

- WETLAND
- F15 — WETLAND ID NUMBER
- STREAMS
- LAKES
- WETLAND DIVIDER
- WATERSHED BOUNDARY LINE
- 1986 PROPOSED TMA AND RECLAIM POND FOOTPRINT

NOTES:

1. BASE MAP DIGITIZED FROM 1'' = 1000' SCALE, MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.

Foth & Van Dyke

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APPROVED BY:	PAE	DATE: FEB. '96
APPROVED BY:	GWS	DATE: FEB. '96

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FIGURE 10.2-1
1986 TMA & RECLAIM POND ALTERNATIVE

Scale: 0 400' 800' | Date: FEBRUARY, 1996

Prepared By: Foth & Van Dyke | By: JRB2

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93C049

2282000 E
2284000 E
2286000 E
2288000 E
2290000 E
118000 N
116000 N
114000 N
112000 N

F12
F15
F15 BUR OAK WATERSHED
F33
M4
HEMLOCK CREEK WATERSHED
N1
HEMLOCK CREEK
M1
M2
M3
M6
M5
K3
16
F81
17
14
13
15
F31
F32
F30
F29
F28
DUCK LAKE WATERSHED
18
F27
19
20
DEEP HOLE LAKE WATERSHED
F66
F65
F64
F63
F25
F26
F22
DUCK LAKE
TMA SOUTH ALTERNATIVE DISTURBANCE BOUNDARY
TMA SOUTH ALTERNATIVE FOOTPRINT
L1
K2
K1

N

LEGEND

- WETLAND
- F15 — WETLAND ID NUMBER
- STREAMS
- LAKES
- WETLAND DIVIDER
- WATERSHED BOUNDARY LINE
- TMA SOUTH ALTERNATIVE FOOTPRINT
- TMA SOUTH ALTERNATIVE DISTURBANCE BOUNDARY

NOTES:

1. BASE MAP DIGITIZED FROM 1" = 1000' SCALE, MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.

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APPROVED BY:	PAE	DATE: FEB. '96
APPROVED BY:	GWS	DATE: FEB. '96

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FIGURE 10.2-2

TMA SOUTH ALTERNATIVE

Scale: 0 400' 800' Date: FEBRUARY, 1996

Prepared By: Foth & Van Dyke By: JRB2

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93C049

2282000 E
2284000 E
2286000 E
2288000 E
2290000 E
118000 N
116000 N
114000 N
112000 N

PROPOSED TMA BORROW AREA
F15
F12
RUNOFF BASIN 11
HEMLOCK CREEK WATERSHED
F15 BUR OAK WATERSHED
F33
M4
TMA NORTH ALTERNATIVE DISTURBANCE BOUNDARY
TMA NORTH ALTERNATIVE FOOTPRINT
RUNOFF BASIN 10
1995 PROPOSED TMA RECLAIM POND
16
F81
17
N1
RUNOFF BASIN 12
14
13
15
F31
M2
M1
M3
RUNOFF BASIN 8
F32
M6
M5
F30
F29
F28
DUCK LAKE WATERSHED
RUNOFF BASIN 9
F66
K3
18
F27
19
DEEP HOLE LAKE WATERSHED
F65
L1
DUCK LAKE
20
F64
F63
K2
HEMLOCK CREEK

N

LEGEND

- WETLAND
- F15 — WETLAND ID NUMBER
- STREAMS
- LAKES
- WETLAND DIVIDER
- WATERSHED BOUNDARY LINE
- 1995 PREFERRED TMA FOOTPRINT
- 1995 PREFERRED TMA DISTURBANCE BOUNDARY

NOTES:

1. BASE MAP DIGITIZED FROM 1" = 1000' SCALE, MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.

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APPROVED BY:	PAE	DATE: FEB. '96
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FIGURE 10.2-3

TMA NORTH ALTERNATIVE

Scale: 0 400' 800' Date: FEBRUARY, 1996

Prepared By: Foth & Van Dyke By: JRB2

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93C049

2282000 E
2284000 E
2286000 E
2288000 E
2290000 E
118000 N
116000 N
114000 N
112000 N

PROPOSED TMA BORROW AREA
F15
F12
F15 BUR OAK WATERSHED
F15 SETBACK ALTERNATIVE DISTURBANCE BOUNDARY
F33
M4
F15 SETBACK ALTERNATIVE TMA FOOTPRINT
N1
16
F81
17
HEMLOCK CREEK WATERSHED
14
13
15
F31
M2
M1
M3
HEMLOCK CREEK
PROPOSED TMA RECLAIM POND
F32
M6
M5
F30
F66
F29
F28
DUCK LAKE WATERSHED
K3
L1
18
DUCK LAKE
19
DEEP HOLE LAKE WATERSHED
F65
20
F27
F64
F63
K2

N

LEGEND

- WETLAND
- F15 — WETLAND ID NUMBER
- STREAMS
- LAKES
- WETLAND DIVIDER
- WATERSHED BOUNDARY LINE
- F15 SETBACK ALTERNATIVE TMA FOOTPRINT
- F15 SETBACK ALTERNATIVE DISTURBANCE BOUNDARY

NOTES:

1. BASE MAP DIGITIZED FROM 1'' = 1000' SCALE, MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
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FIGURE 10.2-4

F15 SETBACK ALTERNATIVE

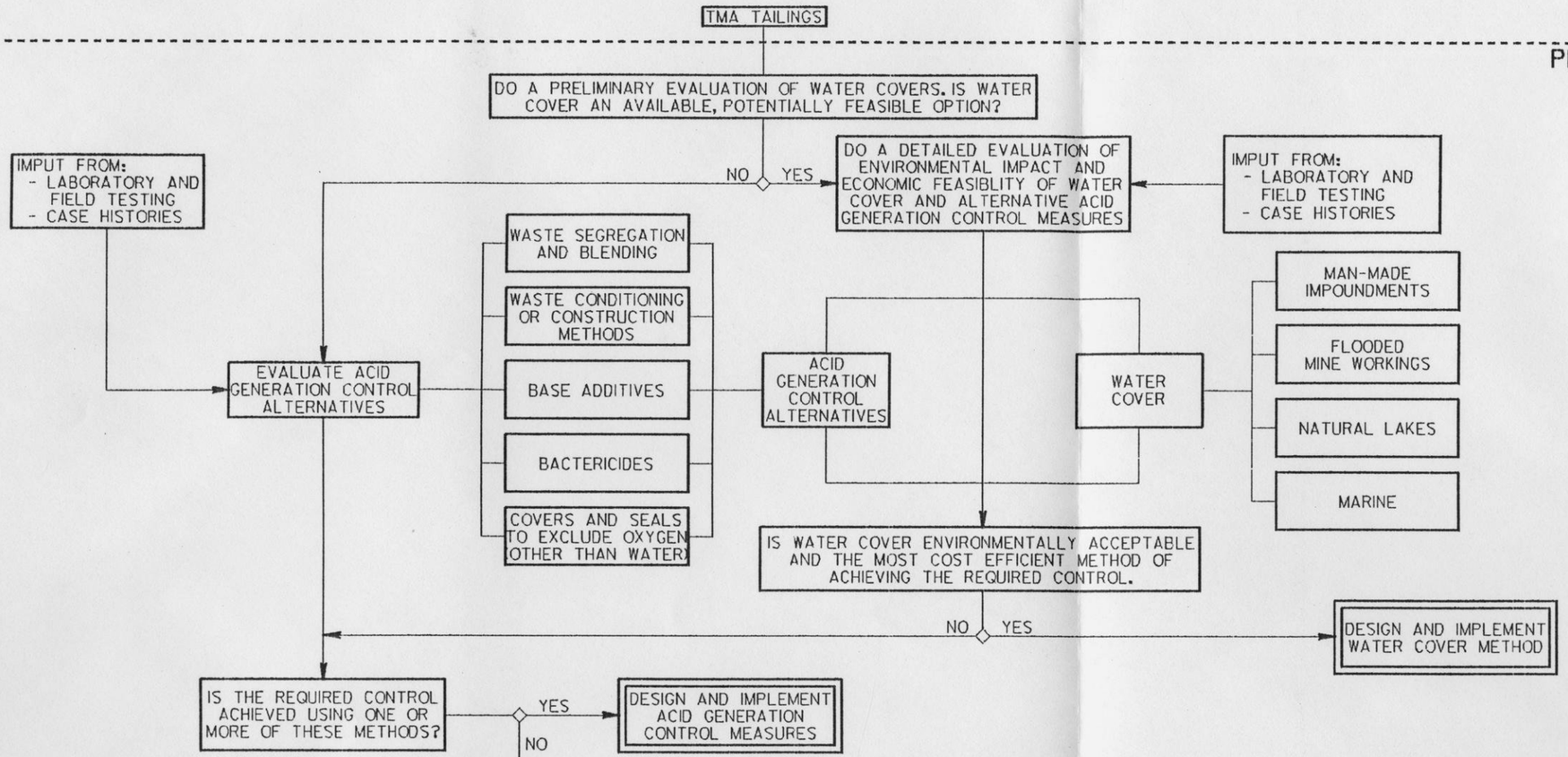
Scale: 0 400' 800' Date: FEBRUARY, 1996

Prepared By: Foth & Van Dyke By: JRB2

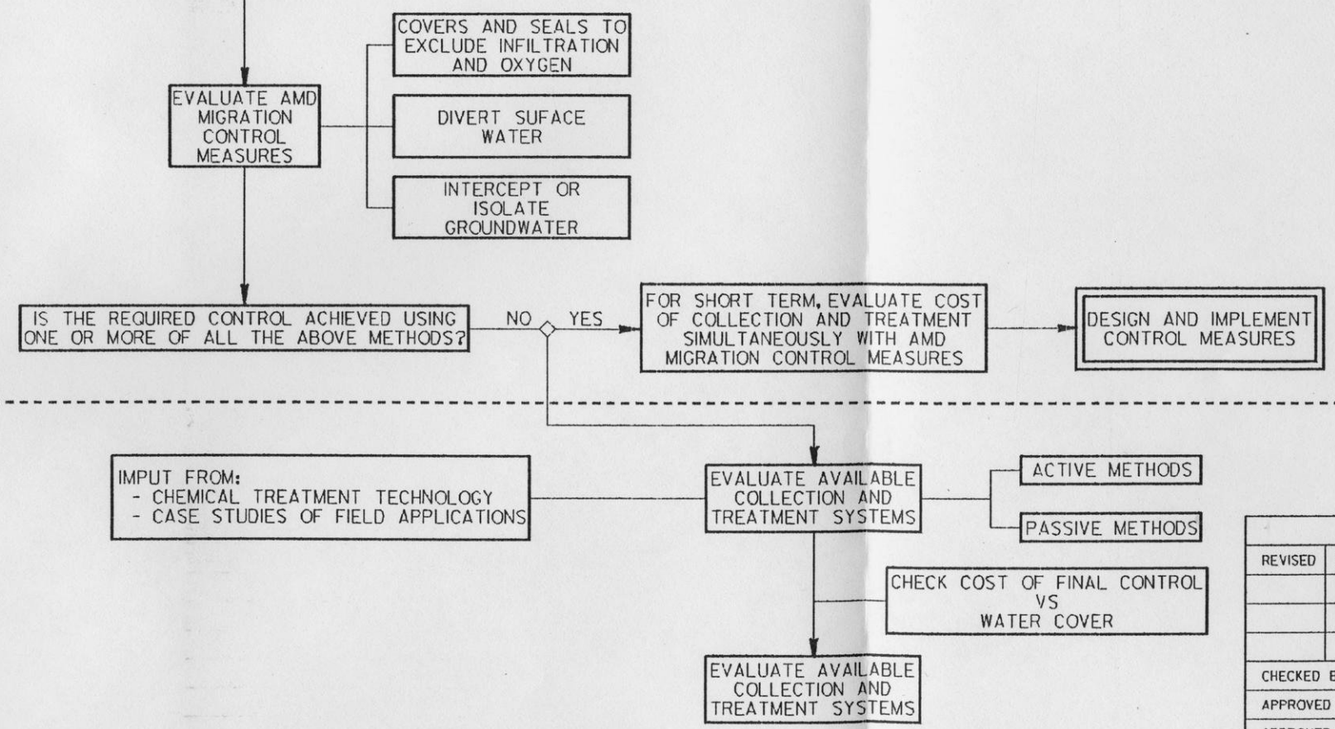
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ACID GENERATING FACILITIES
PRIMARY CONTROL / ACID GENERATION CONTROL



SECONDARY CONTROL / MIGRATION CONTROL



TERTIARY CONTROL / COLLECTION AND TREATMENT

SOURCE:
B.C. TASK FORCE,
DRAFT TECHNICAL GUIDE,
VOLUME I,
AUGUST, 1989

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FIGURE 10.3-1

FLOW CHART SHOWING APPROACH TO ACID ROCK DRAINAGE CONTROL

Scale:	NOT TO SCALE	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

c:\cadwork\3c49f137.dgn

93C049

2,260,000 E 2,265,000 E 2,270,000 E 2,275,000 E 2,280,000 E 2,285,000 E 2,290,000 E

130,000 N

125,000 N

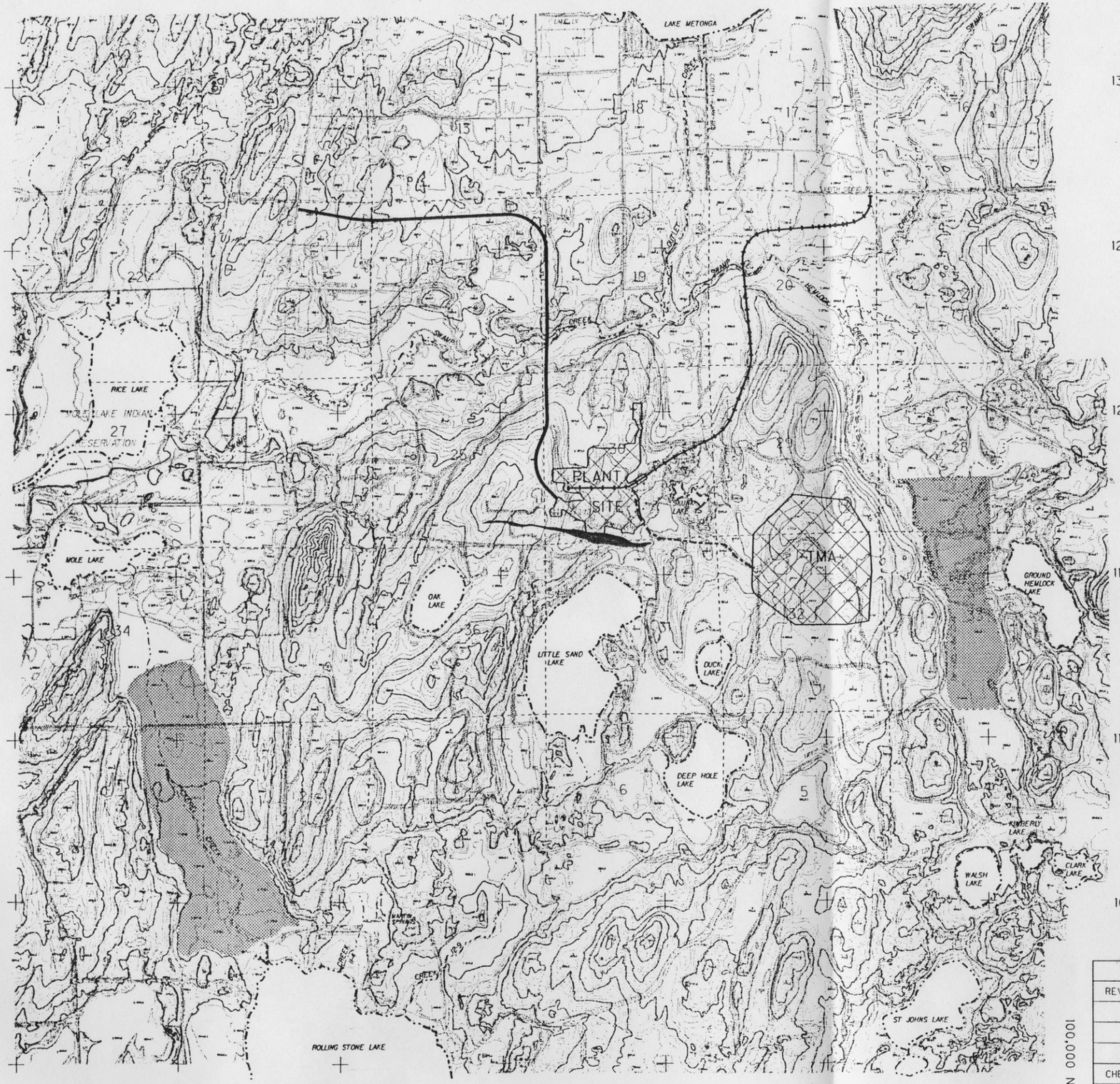
120,000 N

115,000 N

110,000 N

105,000 N

100,000 N



N

LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- 1692.6 × EXISTING SPOT ELEVATION
- SECTION LINE
- ORE BODY
- PROPOSED ACCESS ROAD
- PROPOSED TMA ACCESS ROAD AND TAILINGS PIPELINE ROUTE
- PROPOSED RAILROAD SPUR
- PROPOSED FACILITIES
- AREA EVALUATED FOR SUB-AQUEOUS (UNDERWATER) DISPOSAL

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
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3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. ORE BODY OUTLINE IS REPRESENTATIVE OF THE SUBCROP AT .THE BASE OF THE OVERBURDEN.

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

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APPROVED BY:	GWS	DATE: MAY '95

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FIGURE 10.3-2

ALTERNATE SITES FOR SUB-AQUEOUS (UNDERWATER) DISPOSAL

Scale: 0 1750' 3500' Date: MAY, 1995

Prepared By: Foth & Van Dyke By: GAM

/d3/dl/users/jpr1/4c54ff2.dgn

93C049

Drawings for the *TMA Feasibility Report/Plan of Operation*

Crandon Mining Company

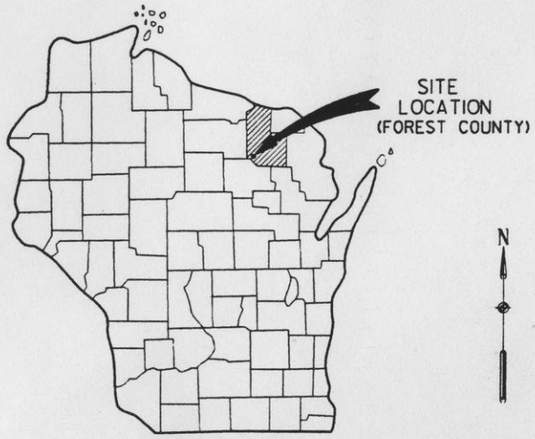
CRANDON PROJECT
FEASIBILITY REPORT / PLAN OF OPERATION
TAILINGS MANAGEMENT AREA

FOREST COUNTY, WISCONSIN

MAY, 1995

Foth & Van Dyke

INDEX	
DRAWING NO.	DESCRIPTION
1	TITLE SHEET
2	EXISTING CONDITIONS - CRANDON PROJECT
3	EXISTING CONDITIONS - TMA
4	GROUNDWATER TABLE CONTOUR MAP OCTOBER, 1994
5	GEOLOGIC CROSS SECTION LOCATION MAP
6	GEOLOGIC CROSS SECTION F'''-F'''
7	GEOLOGIC CROSS SECTION G'''-G'''
8	GEOLOGIC CROSS SECTION H'''-H'''
9	GEOLOGIC CROSS SECTION J'''-J'''
10	GEOLOGIC CROSS SECTION K'''-K'''
11	GEOLOGIC CROSS SECTION L'''-L'''
12	GEOLOGIC CROSS SECTION O'''-O'''
13	SITE SEQUENCING PLAN
14	SUBBASE GRADES
15	BASE GRADES
16	PIPING PLAN
17	TOP OF TILL GRADES AND TAILINGS DISTRIBUTION PIPING
18	FINAL GRADES
19	TMA-1, STAGE I PHASING PLAN
20	TMA-1, STAGE II PHASING PLAN
21	TMA-1, FINAL GRADES AND TMA-2, BASE GRADES
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23	ENGINEERING CROSS SECTIONS
24	ENGINEERING CROSS SECTIONS
25	ENGINEERING CROSS SECTIONS
26	ENGINEERING CROSS SECTIONS
27	ENGINEERING CROSS SECTIONS
28	DETAILS
29	DETAILS
30	DETAILS
31	DETAILS



COUNTY LOCATION MAP

SITE LOCATION MAP

SCALE: 1" = 10,000

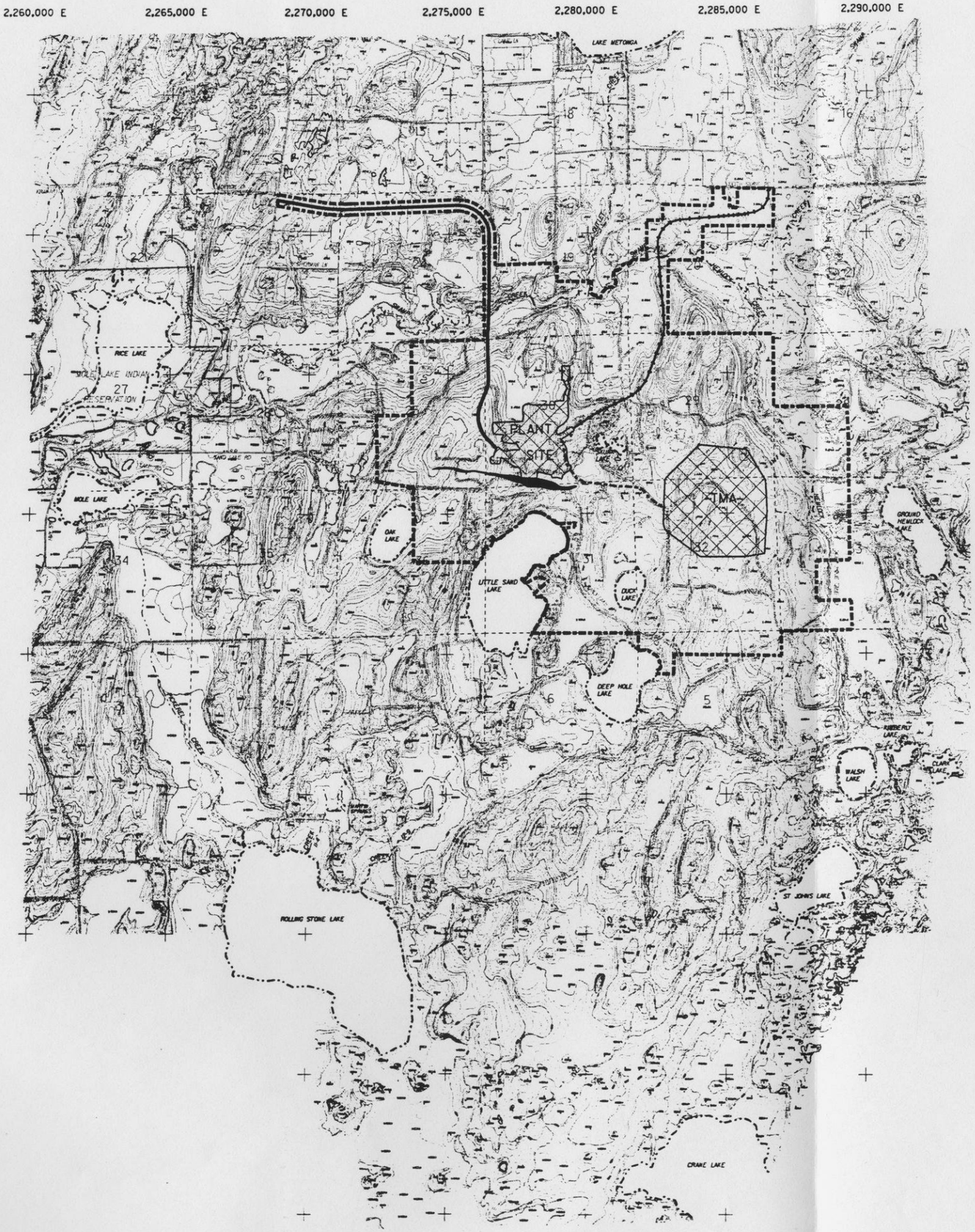
WISCONSIN
Nandakumaran Paruvakat
E-25845
Green Bay, Wis.
PROFESSIONAL ENGINEER

5-30-95

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION

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LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- EXISTING CONTOUR
- SPOT ELEVATION
- SECTION LINE
- ORE BODY
- PROPOSED ACCESS ROAD
- PROPOSED HAUL ROAD
- PROPOSED RAILROAD SPUR
- PROPOSED FACILITIES
- PROJECT AREA

N

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. ORE BODY OUTLINE IS REPRESENTATIVE OF THE SUBCROP AT THE BASE OF THE OVERBURDEN.

SCALE
0 2000' 4000'

Crandon Mining Company

Foth & Van Dyke

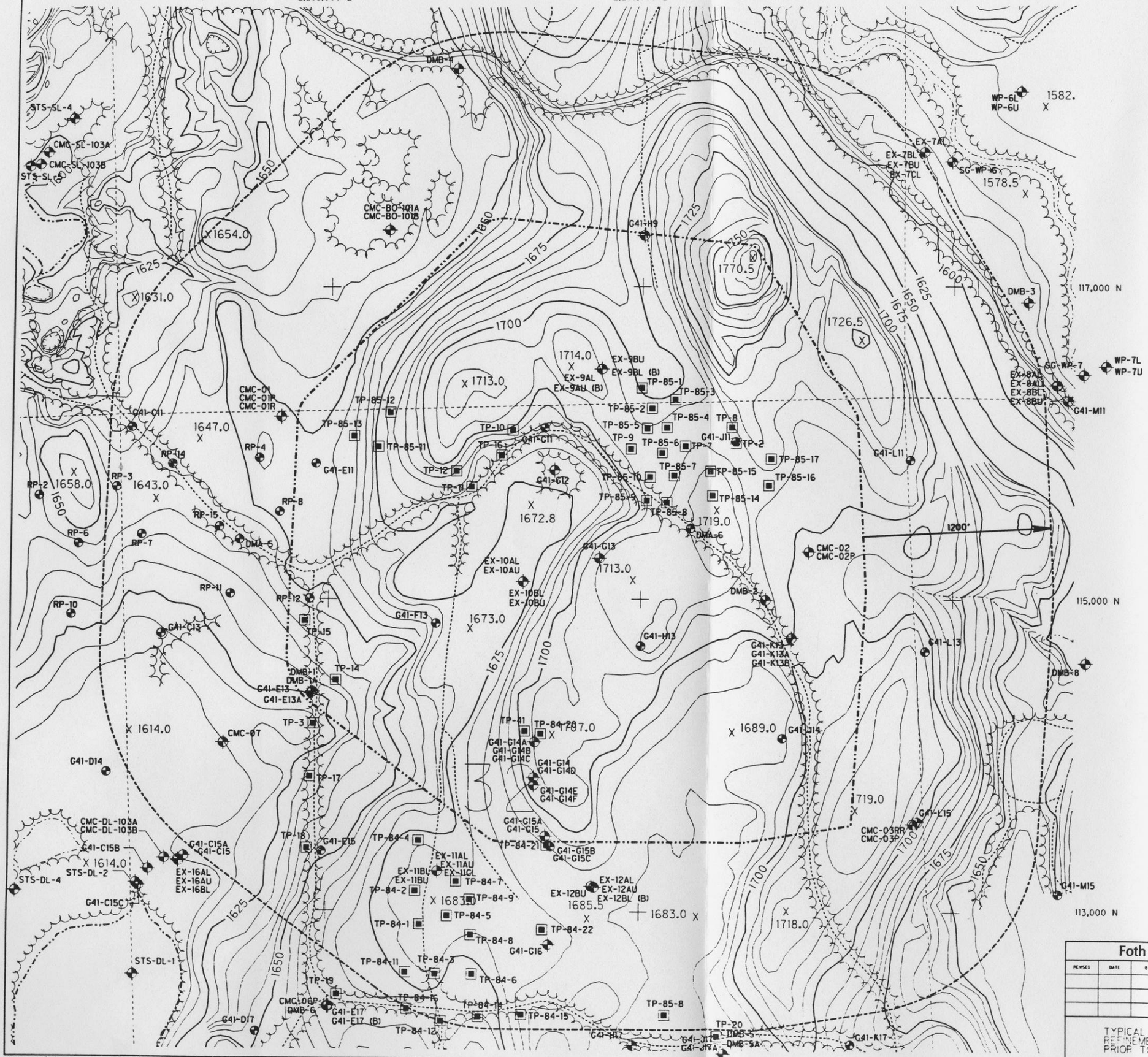
REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

TITLE
EXISTING CONDITIONS - CRANDON PROJECT

SCALE AS SHOWN	STATE WISCONSIN		COUNTY FOREST		
DRAWN BY GAM, JPRI	DATE 5/95	CHECKED BY JKSI	DATE 5/95		
APPROVED BY REM	DATE 5/95	APPROVED BY NXP	DATE 5/95		
APPROVED BY GWS	DATE 5/95	CRANDON MINING COMPANY DEM	DATE 5/95		
DRAWING NO. 2	SHEET 2 OF 31	REVISION NO. 0			

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LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- —1675— EXISTING CONTOUR
- 1692.0 X SPOT ELEVATION
- SECTION LINE
- PERIMETER OF MINE WASTE SITE (AS PER NR182)
- COMPLIANCE BOUNDARY
- SOIL BORING LOCATION (SEE NOTE 5)
- MONITORING WELL LOCATION (SEE NOTE 5)
- TEST PIT LOCATION (SEE NOTE 5)

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. SOIL BORING, TEST PIT OR MONITORING WELL NUMBERS ADJACENT TO SYMBOL.

SCALE
0 300' 600'

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

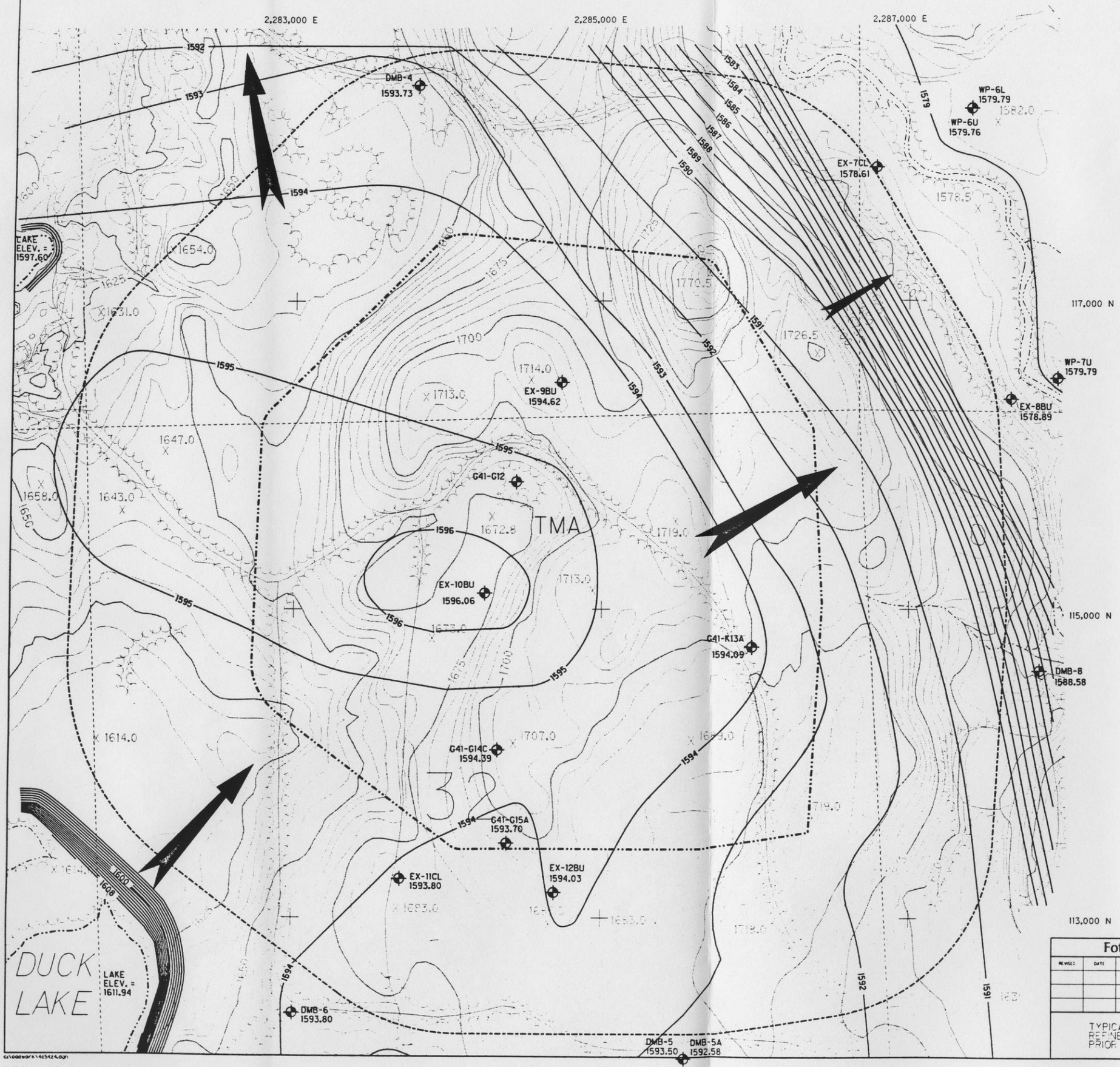
TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

TITLE: EXISTING CONDITIONS - TMA

SCALE: AS SHOWN	STATE: WISCONSIN	COUNTY: FOREST	
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95
DRAWING NO.: 3		SHEET 3 OF 3	REVISION NO.: 0

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LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- —1675— EXISTING CONTOUR
- 1692.0 X SPOT ELEVATION
- SECTION LINE
- PERIMETER OF MINE WASTE SITE (AS PER NR182)
- COMPLIANCE BOUNDARY
- —1580— GROUNDWATER CONTOUR (CONTOUR INTERVAL = 10 FEET)
- GROUNDWATER FLOW DIRECTION
- EX-14BU 1592.41 MONITORING WELL LOCATION, NUMBER AND WATER TABLE ELEVATION

N

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. GROUNDWATER ELEVATIONS MEASURED OCTOBER, 1984.

SCALE
0 300' 600'

Crandon Mining Company

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

TITLE
GROUNDWATER TABLE CONTOUR MAP
OCTOBER, 1984

SCALE AS SHOWN	STATE WISCONSIN		COUNTY FOREST		
DRAWN BY GAM, JPRI	DATE 5/95	CHECKED BY JKSI	DATE 5/95		
APPROVED BY REM	DATE 5/95	APPROVED BY NXP	DATE 5/95		
APPROVED BY GWS	DATE 5/95	CRANDON MINING COMPANY DEM	DATE 5/95		
DRAWING NO. 4	SHEET 4 OF 31	REVISION NO. 0			

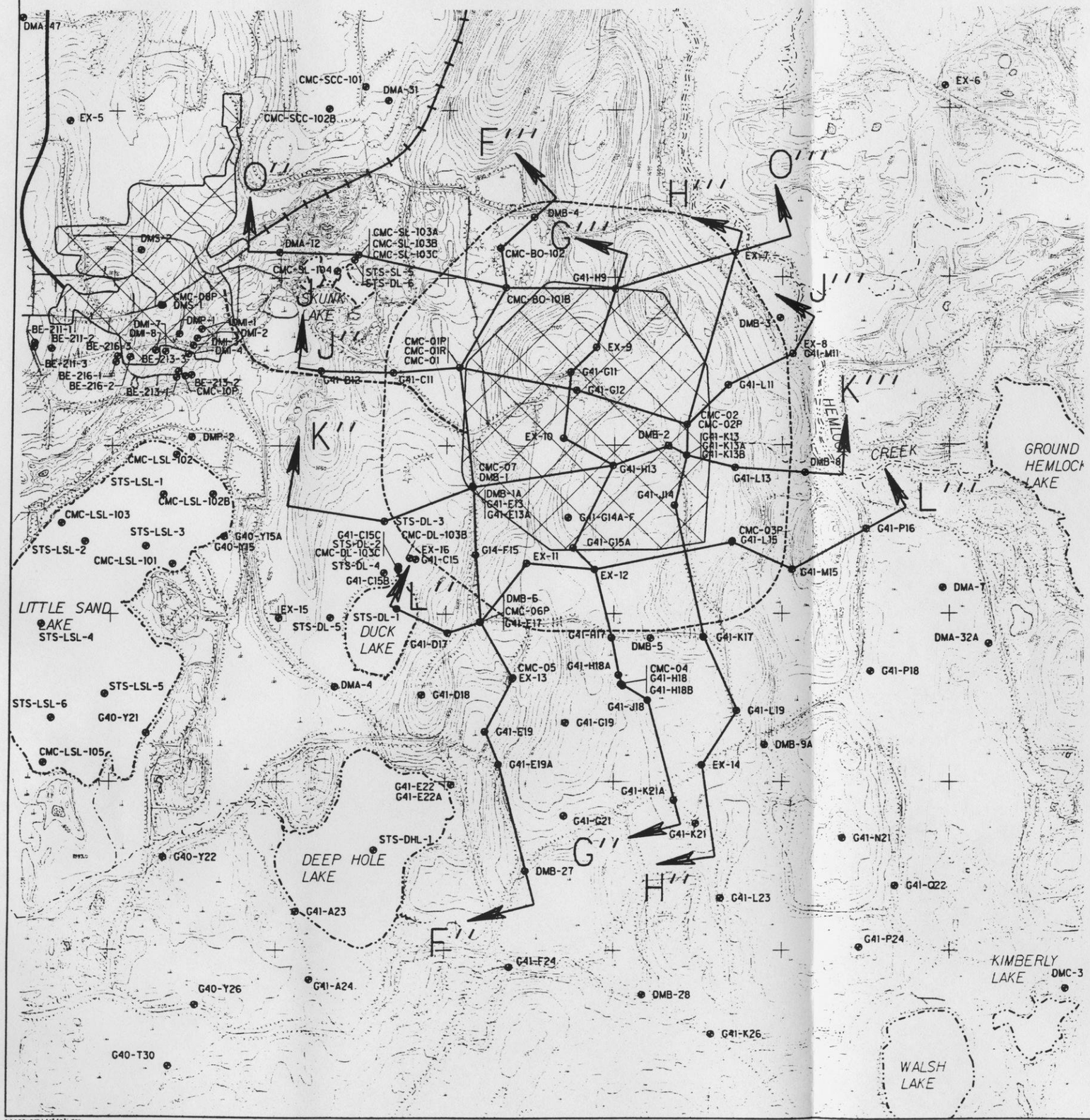
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120,000 N | 117,500 N | 115,000 N | 112,500 N | 110,000 N | 107,500 N

N



LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- EXISTING CONTOUR
- TREE LINE
- SPOT ELEVATION
- SECTION LINE
- PROPOSED ACCESS ROAD
- PROPOSED HAUL ROAD
- PROPOSED RAILROAD SPUR
- PROPOSED FACILITIES

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

SCALE
0 800' 1600'

Crandon Mining Company

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

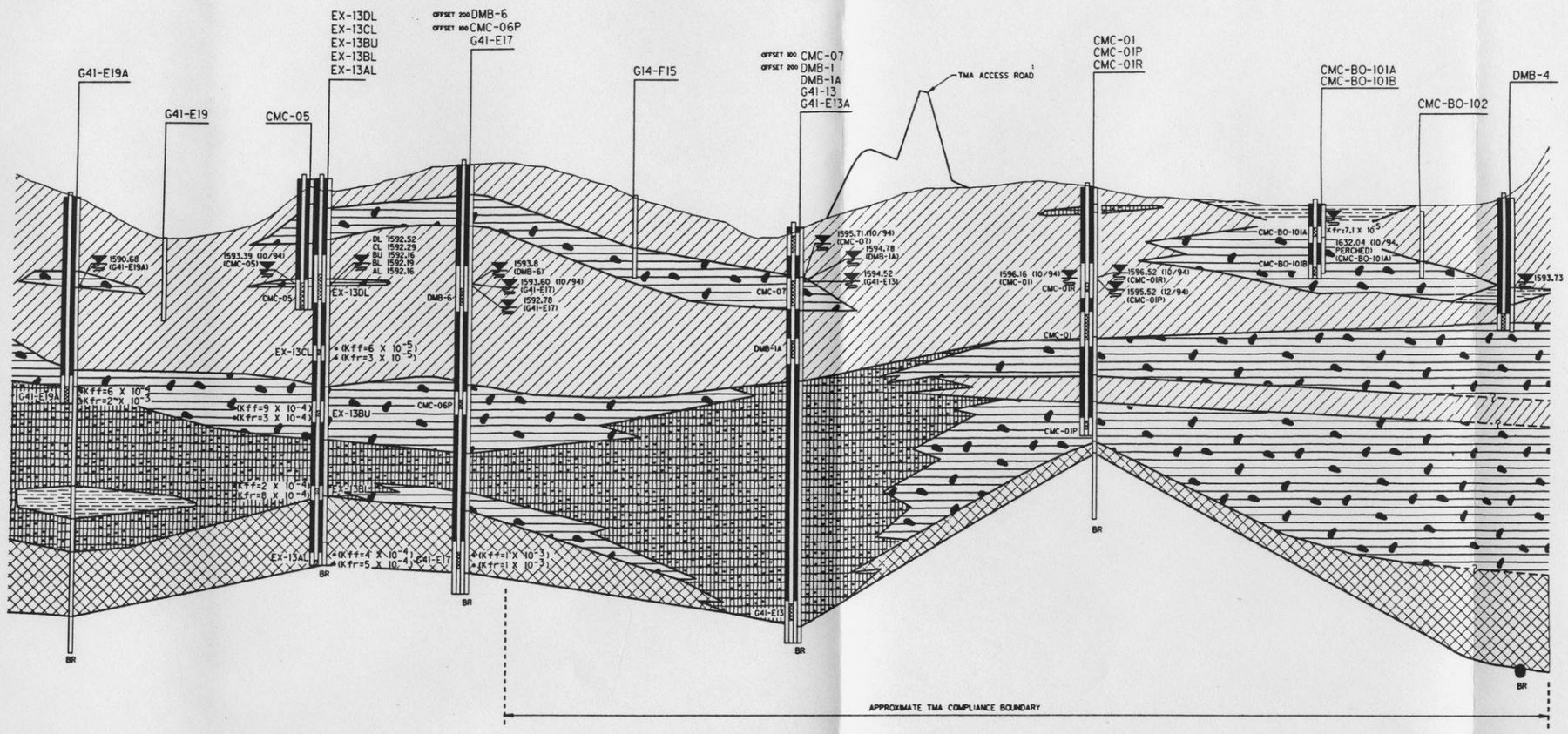
TITLE
GEOLOGIC CROSS SECTION LOCATION MAP

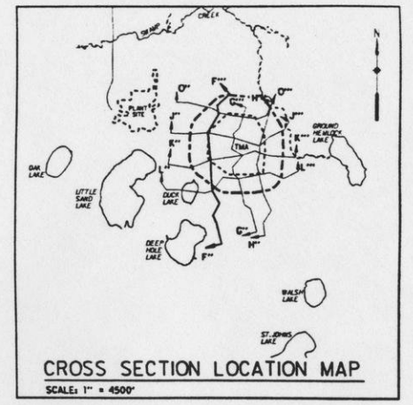
SCALE AS SHOWN	STATE WISCONSIN		COUNTY FOREST		
DRAWN BY GAM, JPRI	DATE 5/95	CHECKED BY JKSI	DATE 5/95		
APPROVED BY REM	DATE 5/95	APPROVED BY NXP	DATE 5/95		
APPROVED BY GWS	DATE 5/95	CRANDON MINING COMPANY DEM	DATE 5/95		
DRAWING NO. 5	SHEET 5 OF 31	REVISION NO. 0			

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05/26/95

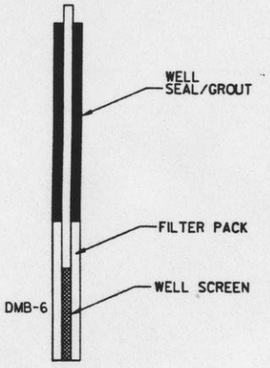
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SECTION F''-F'''



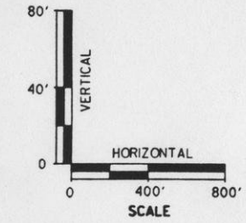


WELL CONSTRUCTION INFORMATION



NOTES:

1. WELLS ARE LOCATED IN INDIVIDUAL BORINGS, THE FILTER PACK IS SILICA SAND, AND THE ANNULAR SPACE AROUND THE WELL CASING WAS BACKFILLED WITH BENTONITE PELLETS AND CEMENT/BENTONITE GROUT AND CONCRETE.
2. THIS CROSS SECTION DERIVED FROM THE REGIONAL CROSS SECTION F-F' OF THE EIR, 1995.



LEGEND

- Qtl UNDIFFERENTIATED LATE WISCONSIN TILL — NASHVILLE MEMBER, COPPER FALLS FORMATION AND THE MAPLEVILLE MEMBER, HORICON FORMATION
- Qte UNDIFFERENTIATED EARLY-WISCONSIN-TILL AND PRE-WISCONSIN TILL — MARATHON AND LINCOLN FORMATIONS, MAY INCLUDE MASSIVE SAPROLITE
- Qowf FINE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND SILT OF WISCONSINAN AGE
- Qowl COARSE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND GRAVEL OF WISCONSINAN AGE
- INFERRED GEOLOGIC CONTACT
- 1594.58 (EX-12BL) WATER LEVEL ELEVATION (10/84)• UNLESS INDICATED
- LL LIQUID LIMIT
- PI PLASTICITY INDEX
- P200 PERCENT FINES
- USCS UNIFIED SOIL CLASSIFICATION SYSTEM
- KL LABORATORY HYDRAULIC CONDUCTIVITY IN CM/SEC.
- Ql LACUSTRINE DEPOSITS — GLACIO-LACUSTRINE, BEDDED, SILT AND CLAY WITH MINOR AMOUNTS OF SAND OF WISCONSINAN, HOLOCENC AND RECENT AGE.
- Qhlc ICE-CONTACT DEPOSITS — ABLATION TILL, FLOW TILL, GLACIO-FLUVIAL SILT, CLAY AND SAND OF WISCONSINAN AGE
- Ms MASSIVE SAPROLITE — UNSTRUCTURED CLAY, SILT AND SAND, HIGHLY WEATHERED BEDROCK
- Ss STRUCTURED SAPROLITE BEDROCK — CLAY, SILT AND SAND WITH RELIC BEDROCK STRUCTURES. PARTLY WEATHERED BEDROCK
- BR PCu UNDIFFERENTIATED BEDROCK — PRECAMBRIAN METAVOLCANIC BEDROCK. MAY INCLUDE STRUCTURED SAPROLITE
- BR INFERRED UNDIFFERENTIATED BEDROCK ELEVATION BASED UPON STS 1984 BEDROCK ELEVATION MAP
- •Kff FIELD PERMEABILITY - FALLING HEAD TEST IN CM/SEC.
- •Kfr FIELD PERMEABILITY - RISING HEAD TEST IN CM/SEC.
- •(Kff) •(Kfr) WELL ABANDONED

Crandon Mining Company - TMA Laboratory Soil Data

Boring ID	Sample #	Depth(ft.)	% Gravel	% Sand	P200	% Silt	% Clay	USCS	Other
G41-E19	S-5	7.5-8.5	7	65	28			SM	
G41-E19A	S-3	20-21.5	11	72	17			SM	
	S-4	30-31.5	21	51	28			SM	
	S-10	90-91.5	18	65	17			SM	
	S-12	110-111	32	60	8			SM-SP	
	S-18	170-171.3	1	96	3			SP	
	S-19	180-182	2	48	50			SM	
	S-24	232-233	31	62	7			SM-SP	
EX-13	S-2	15-16.5	61	35	4			GW	
	S-3	26-27	12	42	46			SM	
	S-4	25-27	28	60	12			SM-SP	
	S-5	30-32	42	46	12			SM-SP	
EX-13B	S-6	35-37	4	78	18			SM	
	S-7	41-42.5	7	75	18	15	3	SM	
	S-8	51-52.5	16	63	21			SM	
	S-10	71-72.5	11	74	15	12	3	SM	
	S-12	92-93.5	12	65	23	19	4	SM	
	S-13	102-103.5	14	73	13			SM	
	S-14	110-111.5	19	65	16			SM	
	S-16	131-132.5	35	58	7			SM-SP	
EX-13C	S-1	152-153.5	67	26	7	6	1	GP-GM	
	S-2	162-163.5	32	82	6			SM-SP	
	S-3	172-173.5	3	90	7			SM-SP	
	S-4	183-184.5	7	86	7			SM-SP	
	S-5	194-195	32	41	27	20	7	SM	
	S-6	205-206	12	63	25	21	4	SM	
	S-7	215.6-216.5	6	88	6			SM-SP	
	S-8	226-226.5	18	10	72	53	19	CL	
CMC-05	S-14	73.4-74.2	5.9	74.4	19.7			SM	
CMC-06A	S-26	145-147	34.3	55.5	10.2			SP-SM	
G41-E17	S-2	12-13.5	31	54	15			SM	
	S-7	40-41.5	7	68	25	18	7	SM	
	S-16	86-87.5	6	71	23			SM	
	S-24	153.5-155	37	54	9	3	6	SP-SM	
	S-28	195-196	6	82	12			SM-SP	
	S-30	214-215.5	0	10	90			ML	
	S-31	224.5-226	0	1	99	51	12	ML	
	S-32c	234-235	3	34	63	39	24	ML	
	S-32b	234-235	7	35	58	46	12	ML	
	S-33	245.5-246.5	31	49	20			SM	
CMC-07	S-8	40-42	9.8	71.5	18.7			SM	
G41-E13	S-4	45-47	1.2	70.2	28.6	24.2	8.4	SM	
	S-5	50-51.5	11	72	16			SM	
	S-6	59-61.5	7	71	22	20	2	SM	
	S-9	90-91.5	35	52	13			SM	
	S-13 & 14	65-65.5	23	59	18			SM	
	S-21	180-180.8	2	81	17			SM	
	S-22	210-211	3	84	13			SM	
	S-25	240-241	1	95	4			SM	
	S-30	290-291	0	95	5			SP-SM	
	S-34	230-231	0	93	7			SP-SM	
CMC-01	S-2	82.5-84.5	8.2	69.1	22.7			SM	
CMC-01R	S-1	49-48.7	12.9	62.7	25.4	17.5	8.4	SC	
	S-2	51-55	15.3	61.6	23.1			SM	
CMC-01P	S-24	138-140	6.2	57	36.8	13.4	6.0	CL	
	S-25	143-145	3.1	65.9	31.0			CL	
CMC-BO-101A	S-1	14-16	0.4	22.3	75.3	53.6	22.7	CL	LL:28 PI: 8
CMC-BO-101B	S-3	4-6	0	53.7	46.3	32.5	13.8	SC-SM	PL:3910-6 LL:19 PI: 5
	S-7	12-14	0	4.4	95.6	73.3	22.3	CL	KL:2.9X10-9 LL:28 PI: 8
	S-18	36-38	1.9	95.4	2.6			SP	
CMC-BO-102	S-9	16-18	17.6	82.4	20			SC	
	S-11	35-37	0	82.9	17.1			SM	

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

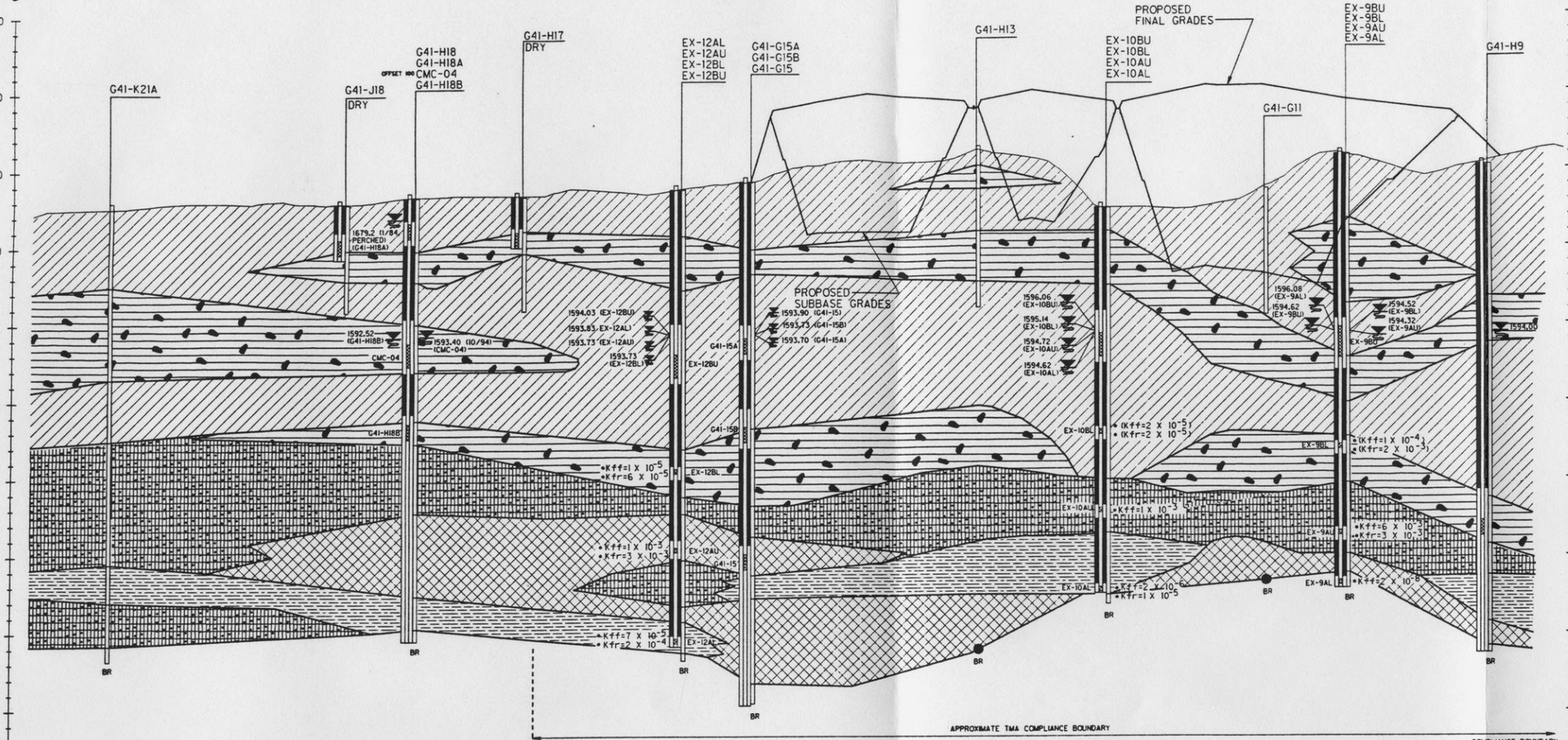
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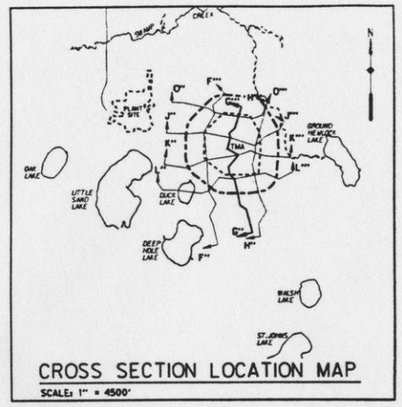
SCALE: VERT. 1" = 40' HOR. 1" = 400'	STATE: WISCONSIN	COUNTY: FOREST	
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95
DRAWING NO.: 6		SHEET: 6 OF 31	REVISION NO.: 0

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SECTION G''-G'''

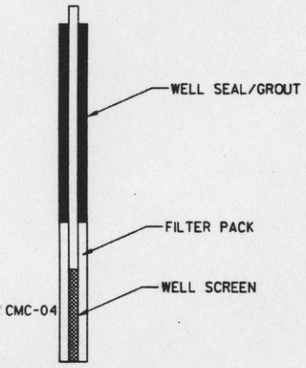




CROSS SECTION LOCATION MAP

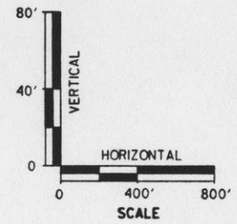
SCALE: 1" = 4500'

WELL CONSTRUCTION INFORMATION



NOTES:

1. WELLS ARE LOCATED IN INDIVIDUAL BORINGS, THE FILTER PACK IS SILICA SAND, AND THE ANNULAR SPACE AROUND THE WELL CASING WAS BACKFILLED WITH BENTONITE PELLETS AND CEMENT/BENTONITE GROUT AND CONCRETE.
2. THIS CROSS SECTION DERIVED FROM THE REGIONAL CROSS SECTION G-G' OF THE EIR, 1995.



LEGEND

- Qtl UNDIFFERENTIATED LATE WISCONSIN TILL — NASHVILLE MEMBER, COPPER FALLS FORMATION AND THE MAPLEVILLE MEMBER, HORICON FORMATION
- Qte UNDIFFERENTIATED EARLY-WISCONSIN-TILL AND PRE-WISCONSIN TILL — MARATHON AND LINCOLN FORMATIONS, MAY INCLUDE MASSIVE SAPROLITE
- Qowf FINE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND SILT OF WISCONSINAN AGE
- Qowl COARSE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND GRAVEL OF WISCONSINAN AGE
- INFERRED GEOLOGIC CONTACT
- 1594.58 (EX-12BL) WATER LEVEL ELEVATION (10/84)• UNLESS INDICATED
- LL LIQUID LIMIT
- PI PLASTICITY INDEX
- P200 PERCENT FINES
- USCS UNIFIED SOIL CLASSIFICATION SYSTEM
- KL LABORATORY HYDRAULIC CONDUCTIVITY IN CM/SEC.
- Qi LACUSTRINE DEPOSITS — GLACIO-LACUSTRINE, BEDDED, SILT AND CLAY WITH MINOR AMOUNTS OF SAND OF WISCONSINAN, HOLOCENC AND RECENT AGE.
- Qhlc ICE-CONTACT DEPOSITS — AB..ATION TILL, FLOW TILL, GLACIO-FLUVIAL SILT, CLAY AND SAND OF WISCONSINAN AGE
- Ms MASSIVE SAPROLITE — UNSTRUCTURED CLAY, SILT AND SAND, HIGHLY WEATHERED BEDROCK
- Ss STRUCTURED SAPROLITE BEDROCK — CLAY, SILT AND SAND WITH RELIC BEDROCK STRUCTURES, PARTLY WEATHERED BEDROCK
- BR PCu UNDIFFERENTIATED BEDROCK — PRECAMBRIAN METAVOLCANIC BEDROCK, MAY INCLUDE STRUCTURED SAPROLITE
- BR INFERRED UNDIFFERENTIATED BEDROCK ELEVATION BASED UPON STS 1984 BEDROCK ELEVATION MAP
- •Kff FIELD PERMEABILITY - FALLING HEAD TEST IN CM/SEC.
- •Kfr FIELD PERMEABILITY - RISING HEAD TEST IN CM/SEC.
- •(Kff) •(Kfr) WELL ABANDONED

Crandon Mining Company - TMA Laboratory Soil Data

Boring ID	Sample #	Depth (ft.)	% Gravel	% Sand	P200	% Silt	% Clay	USCS	Other
G41-H18	S-5	20-21.5	23	65	12			SW	
	S-9	40-41	18	78	4			SP	
G41-H18A	S-2	13-14.5	0	39	61	54	7	SM	
	S-3	16.5-18	11	51	38	30	8	SM	
	S-4	20-21.5	31	66	3			SP	
CMC-04	S-22	108-110	2.5	86.6	10.9			SP-SM	
G41-H17	S-3	9-10.5	30	66	4			SP	
	S-6	24-25.5	43	54	3			GP	
	S-13	54-54.5	50	53	15			SM	
	S-16	69-69.6	7	58	35			SM	
EX-12	S-1	10-12	9	83	8			SM-SP	
	S-2	20-22	18	52	30	23	7	SM	
	S-4	41-43	75	6	19	18	1	GM	
	S-5	48-49	66	32	2			GP	
	S-6	59-60	50	47	3			SM-GM	
	S-7	69-70.5	20	36	44			SM	
	S-8	80-81.5	48	36	16	13	3	SM	
	S-9	90-91.5	7	78	15			SM	
	S-11	110.5-112	6	69	25			SM	
	S-14	141-142.5	26	60	14	12	2	SM	
	S-19	192-192.5	47	46	7			SM-SP	
	S-20	203-204.5	1	93	6			SM-SP	
	S-21	213-214.5	10	67	23	21	2	SM	
	S-22	223.5-225	13	69	18			SM	
	S-24	244.2-245.5	1	81	18			SM	
	S-27	274.5-276.5	9	66	25	21	4	SM	
	S-28	285-287	0	48	62	55	7	ML	
	S-29	295-297	0	4	96	84	12	ML	
G41-G15A	S-3	18-19	24	53	23	12	11	SM	
	S-4	23-24	14	66	20			SM	
	S-7	38-38.2	45	29	26			GM	
	S-9	103.5-105.5	18	55	27			SM	
	S-10	131.5-133	7	70	23			SM	
G41-H13	S-1	0-1.5	10	52	38	31	7	SM	
	S-1	29-30.5	14	63	23			SM	
	S-15	73-72.2	2	84	14			SM	
	S-22	164-164.8	25	59	16			SM	
EX-10	S-1	10.5-12.5	6	58	46	41	5	SM	
	S-2	20.5-22.5	24	72	4			SP	
	S-4	41.5-43.5	24	70	6			SP-SM	
	S-5	51.5-53.5	8	68	24	19	5	SM	
	S-6	62-64	11	65	24	18	6	SM	
	S-9	92.5-94.5	6	70	24	20	4	SM	
	S-10	101-103	7	79	14			SM	
	S-13	123.5-125.5	15	65	20			SM	
	S-17	174.5-176.5	5	88	7			SM-SP	
	S-18	185-186.5	3	93	6			SM-SP	
	S-21	215.5-217	2	61	37	35	2	SM	
	S-22	229-229	0	7	93	91	2	ML	
	S-23	236-237	0	8	92	80	12	CL	
	S-24	246.5-250.5	9	89	8			SM-SP	
G41-G11	S-1	0-1.5	15	53	32	20	12	SM	
	S-4	30-33.5	23	59	18			SM	
	S-6	60-61.5	25	57	18			SM	
	S-14	140-141	19	76	10			SM-SP	
	S-15	145-146	16	63	21	17	4	SM	
	S-17	155-156	30	52	18			SM	
EX-9	S-1	4-4.5	5	92	3			SM	
	S-2	20-22	27	61	12			SM	
	S-3	30-31.5	25	58	17	13	4	SM	
	S-5	50-52	32	57	11			SM-SP	
	S-7	70-71.7	37	57	6			SM-SP	
	S-9	91-93	36	59	5			SP	
	S-10	101.5-103.5	24	56	26	18	2	SM	
	S-12	121-124.5	12	69	19	17	2	SM	
	S-13	133.5-135	27	66	7			SM-SP	
	S-14	144.5-144	8	83	9			SM-SP	
	S-16	161-165	31	47	22	20.5	1.5	SM	
	S-18	181-184.5	29	60	11			SM-SP	
	S-20	201-205	12	75	13			SM	
	S-25	251-256	1	81	18			SM	
G41-H9	S-3	21-22.5	14	68	18			SM	
	S-4	40-41	26	62	12			SM	
	S-8	76-71.8	29	58	14			SM	
	S-10	90-91	49	44	7			SM-SP	
	S-15	140-141.5	17	63	20			SM	
	S-16	150-151.3	14	64	22			SM	
	S-21	200-200.8	1	92	8			SM-SP	
	S-22	210.5-211.4	14	64	22			SM	

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REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

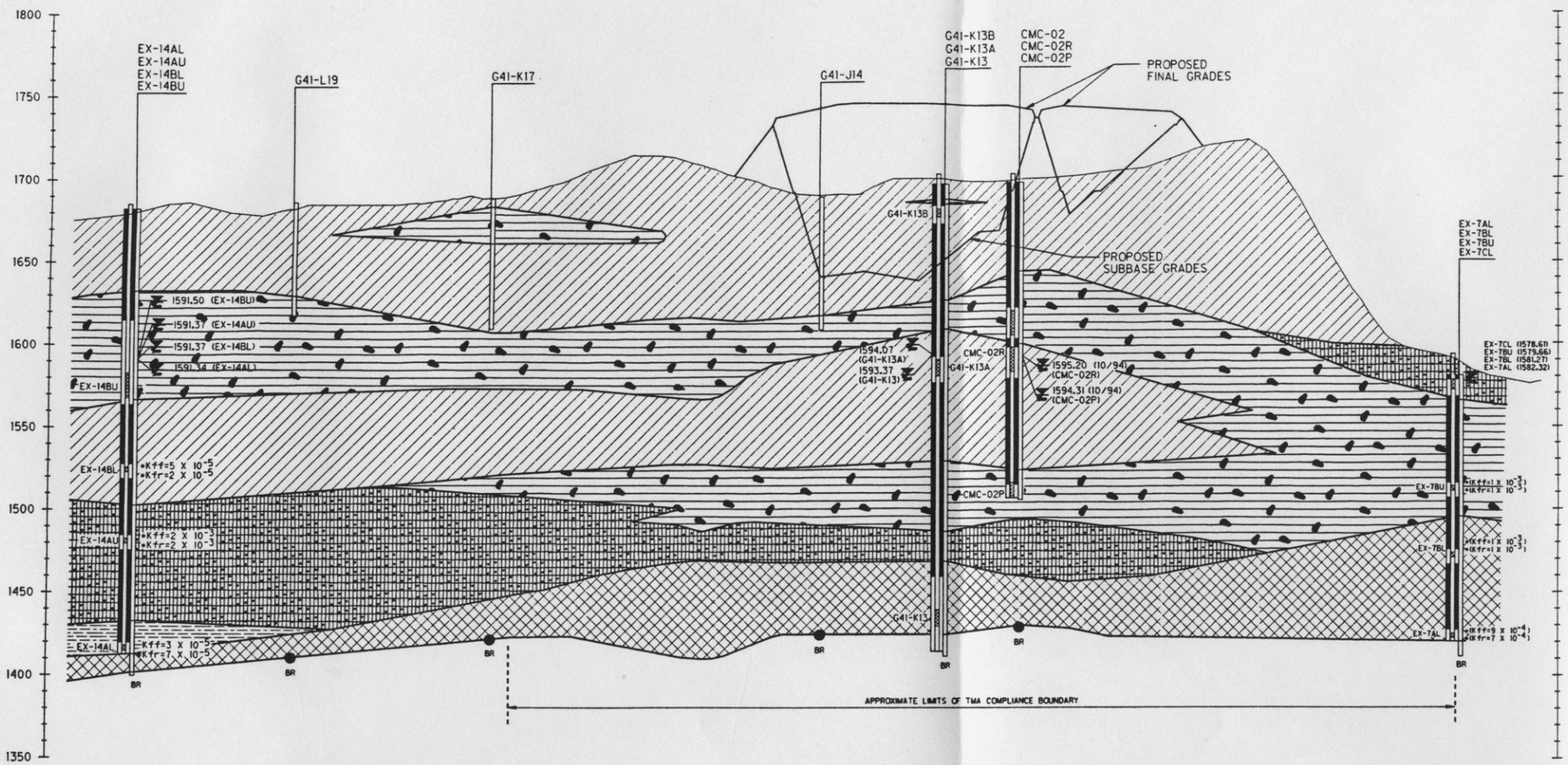
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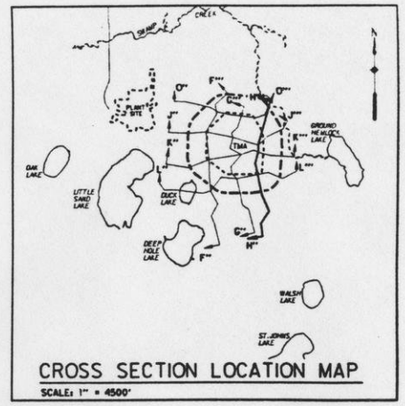
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APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO.: 7	SHEET: 7 OF 31	REVISION NO.: 0			

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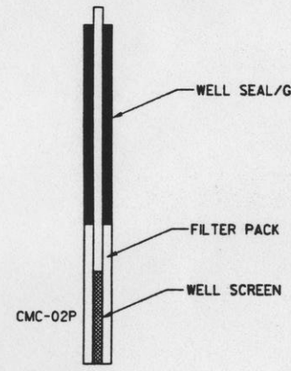
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SECTION H''-H'''



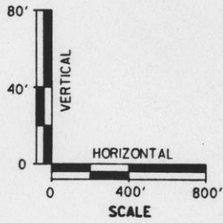


WELL CONSTRUCTION INFORMATION



NOTES:

1. WELLS ARE LOCATED IN INDIVIDUAL BORINGS, THE FILTER PACK IS SILICA SAND, AND THE ANNULAR SPACE AROUND THE WELL CASING WAS BACKFILLED WITH BENTONITE PELLETS AND CEMENT/BENTONITE GROUT AND CONCRETE.
2. THIS CROSS SECTION DERIVED FROM THE REGIONAL CROSS SECTION H-H' OF THE EIR, 1995.



LEGEND

- Qtl UNDIFFERENTIATED LATE WISCONSIN TILL — NASHVILLE MEMBER, COPPER FALLS FORMATION AND THE MAPLEVILLE MEMBER, HORICON FORMATION
- Qte UNDIFFERENTIATED EARLY-WISCONSIN-TILL AND PRE-WISCONSIN TILL — MARATHON AND LINCOLN FORMATIONS, MAY INCLUDE MASSIVE SAPROLITE
- Qowf FINE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND SILT OF WISCONSINAN AGE
- Qowl COARSE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND GRAVEL OF WISCONSINAN AGE
- INFERRED GEOLOGIC CONTACT
- 1594.58 (EX-12BL) — WATER LEVEL ELEVATION (10/84)• UNLESS INDICATED
- LL LIQUID LIMIT
- PI PLASTICITY INDEX
- P200 PERCENT FINES
- USCS UNIFIED SOIL CLASSIFICATION SYSTEM
- KL LABORATORY HYDRAULIC CONDUCTIVITY IN CM/SEC.
- Ql LACUSTRINE DEPOSITS — GLACIO-LACUSTRINE, BEDDED, SILT AND CLAY WITH MINOR AMOUNTS OF SAND OF WISCONSINAN, HOLOCENC AND RECENT AGE.
- Qhlc ICE-CONTACT DEPOSITS — ABLATION TILL, FLOW TILL, GLACIO-FLUVIAL SILT, CLAY AND SAND OF WISCONSINAN AGE
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- Ss STRUCTURED SAPROLITE BEDROCK — CLAY, SILT AND SAND WITH RELIC BEDROCK STRUCTURES, PARTLY WEATHERED BEDROCK
- BR PCu UNDIFFERENTIATED BEDROCK — PRECAMBRIAN METAVOLCANIC BEDROCK, MAY INCLUDE STRUCTURED SAPROLITE
- BR INFERRED UNDIFFERENTIATED BEDROCK ELEVATION BASED UPON STS 1984 BEDROCK ELEVATION MAP
- •Kff FIELD PERMEABILITY - FALLING HEAD TEST IN CM/SEC.
- •Kfr FIELD PERMEABILITY - RISING HEAD TEST IN CM/SEC.
- •(Kff) •(Kfr) WELL ABANDONED

Crandon Mining Company - TMA Laboratory Soil Data

Boring ID	Sample #	Depth (ft.)	% Gravel	% Sand	P200	% Silt	% Clay	USCS	Other
EX-14	S-1	2-4	0	31	69	61	8	ML	--
	S-2	10-12	27	56	17	14	3	SW	--
	S-3	21-22	42	50	8	--	--	SW-SP	--
	S-4	30-31.5	15	67	18	16	2	SW	--
	S-5	40-41.5	24	62	14	--	--	SW	--
	S-6	50.9-51.9	29	63	8	--	--	SW-SP	--
	S-7	60.5-62.0	18	76	6	--	--	SW-SP	--
	S-8	71-73	45	49	6	--	--	SW-SP	--
	S-9	81-82	35	57	8	--	--	SW-SP	--
	S-11	101.5-103.5	15	69	6	--	--	SW-SP	--
	S-13	122-124	12	70	18	--	--	SW	--
	S-15	142.5-144.5	16	70	14	--	--	SW	--
	S-18	173.175	18	69	13	12	1	SW	--
	S-19	183.5-185.5	0	94	6	--	--	SW-SP	--
	S-22	214-215.5	10	79	11	--	--	SW-SP	--
	S-23	224.5-226	1	76	23	--	--	SW	--
	S-25	245-246	0	47	53	52	1	ML	--
	S-27	264.5-266	10	75	15	--	--	SW	--
	S-28	276-277.5	25	57	18	--	--	SW	--
G41-L19	S-2	5-10.5	22	62	16	13	3	SW	--
	S-5	24-24.5	16	70	14	--	--	SW	--
G41-K17	S-6	24-25.5	27	66	7	--	--	SW-SP	--
	S-12	54-54.5	2	88	12	--	--	SW-SP	--
	S-16	74-74.4	8	73	19	--	--	SW	--
G41-K13	S-1	120-120.9	6	75	19	10	9	SW	--
	S-3	142-142.5	8	71	21	--	--	SW	--
	S-6	172-173	26	65	9	--	--	GW-SW	--
	S-14	254-254.7	22	61	17	--	--	SW	--
G41-K13A	S-2	5-6.5	12	65	23	--	--	SW	--
	S-8	35-35.3	20	60	20	--	--	SW	--
	S-13	60-60.6	10	70	20	7	13	SW	--
	S-15	70-70.3	41	45	14	--	--	GW-SW	--
	S-16	75-75.5	33	57	10	2	8	SP-SW	--
	S-20	95-96	10	70	20	--	--	SW	--
	S-23A	116-116.3	8	82	10	--	--	SP-SW	--
G41-K13B	S-1	1-2.5	0	27	63	55	7	ML	--
	S-3	5.5-11	0	13	87	78	9	CL-ML	LL:22 PI: 4
	S-5	19.5-20	20	64	16	--	--	SW	--
CMC-02P	S-31	191-193	30	53.6	16.4	--	--	SW	--
EX-7	S-1	61.5-63.5	11	80	9	--	--	SW-SP	--
	S-2	120-122	7	52	41	39	2	SW	--
	S-3	162-163	61	27	12	10	2	GP-SW	--

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REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

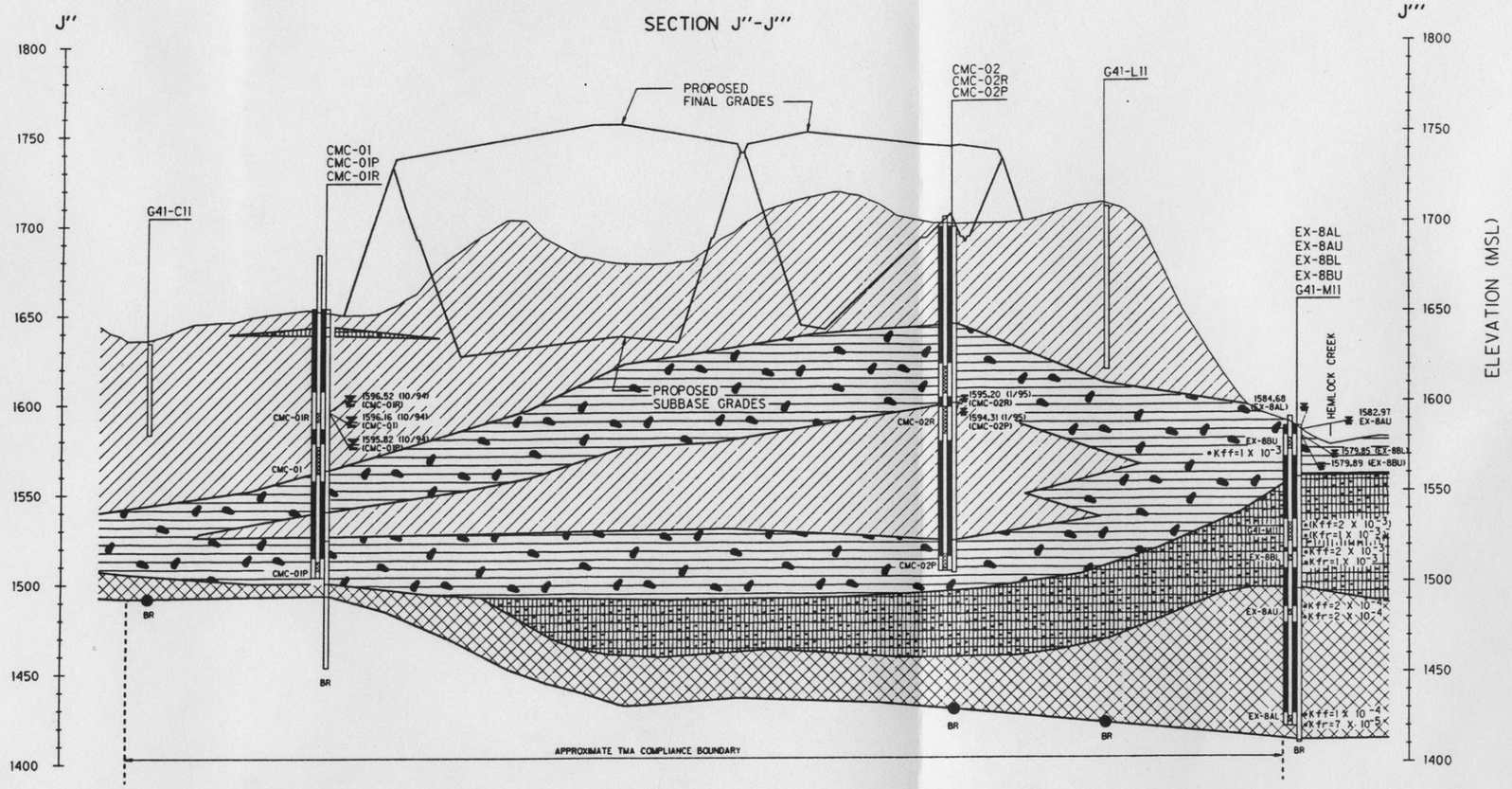
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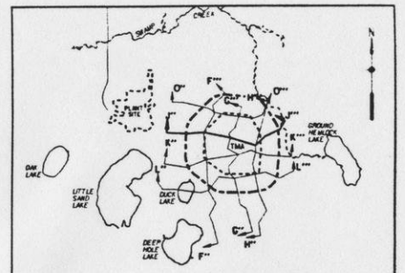
SCALE VERT: 1" = 40' HOR: 1" = 400'	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO.: 8		SHEET: 8 OF 31	REVISION NO.: 0		

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05/26/95 pbd

SECTION J''-J'''

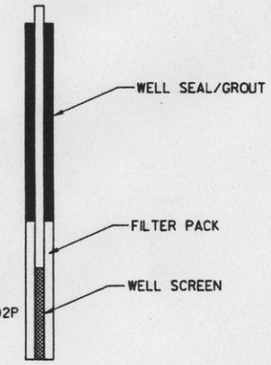




LEGEND

- Qtl UNDIFFERENTIATED LATE WISCONSIN TILL — NASHVILLE MEMBER, COPPER FALLS FORMATION AND THE MAPLEVILLE MEMBER, HORICON FORMATION
- Qte UNDIFFERENTIATED EARLY-WISCONSIN-TILL AND PRE-WISCONSIN TILL — MARATHON AND LINCOLN FORMATIONS, MAY INCLUDE MASSIVE SAPROLITE
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- INFERRED GEOLOGIC CONTACT
- 1594.58 (EX-12BL) — WATER LEVEL ELEVATION (10/84)• UNLESS INDICATED
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- •Kfr FIELD PERMEABILITY - RISING HEAD TEST IN CM/SEC.
- •(Kff) •(Kfr) WELL ABANDONED

WELL CONSTRUCTION INFORMATION

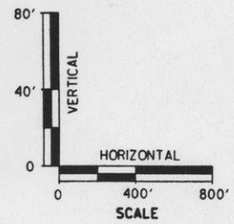


NOTES:

1. WELLS ARE LOCATED IN INDIVIDUAL BORINGS, THE FILTER PACK IS SILICA SAND, AND THE ANNULAR SPACE AROUND THE WELL CASING WAS BACKFILLED WITH BENTONITE PELLETS AND CEMENT/BENTONITE GROUT AND CONCRETE.
2. THIS CROSS SECTION DERIVED FROM THE REGIONAL CROSS SECTION J-J' OF THE EIR, 1995.

Crandon Mining Company - TMA Laboratory Soil Data

Boring ID	Sample #	Depth (ft.)	% Gravel	% Sand	P200	% Silt	% Clay	USCS	Other
G41-C11	S-4	10-11.5	16	60	24			SM	
	S-9	35-36.5	13	67	20			SM	
CMC-01	S-2	82.5-84.5	8.2	69.1	22.7			SM	
CMC-01R	S-1	48-48.7	11.9	62.2	25.9	17.5	8.4	SC	
	S-2	53-55	15.3	61.6	23.1			SM	
CMC-01P	S-24	138-140	6.3	74.3	19.4	13.4	6.0	SC	
	S-25	143-145	3.1	83.9	13.0			SM	
CMC-02P	S-31	191-193	30	53.6	16.4			SM	
G41-L11	S-2	5-6.5	13	63	24			SM	
	S-4	10-11.5	11	65	24	20	4	SM	
	S-6	20-21.5	28	52	20			SM	
	S-10	40-40.5	36	48	16			SM	
	S-12	50-50.5	10	68	22			SM	
	S-14	60-61	8	67	25			SM	
	S-16	70-71	10	70	20			SM	
	S-18	80-80.5	44	44	12			GM-SM	
G41-M11	S-2	10-12	11	87	2			SM	
	S-5	38.5-40	0	98	2			SP	
	S-8	68-70	0	63	37			SM	
Ex-8	S-1	1-1.3	1	96	3			SP	
	S-3	20-22	22	68	10			SM-SP	
	S-5	40-42	0	96	4			SP	
	S-8	71-73	1	87	12	10	2	SM	
	S-10	92-94	22	51	27	21	6	SM	
	S-12	112-114	18	71	11	10	1	SM-SP	
	S-14	133-135	29	60	11			SM-SP	
	S-15	143-145	16	63	21	17	4	SM	
	S-17	164-166	31	50	19			SM	



TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

Crandon Mining Company

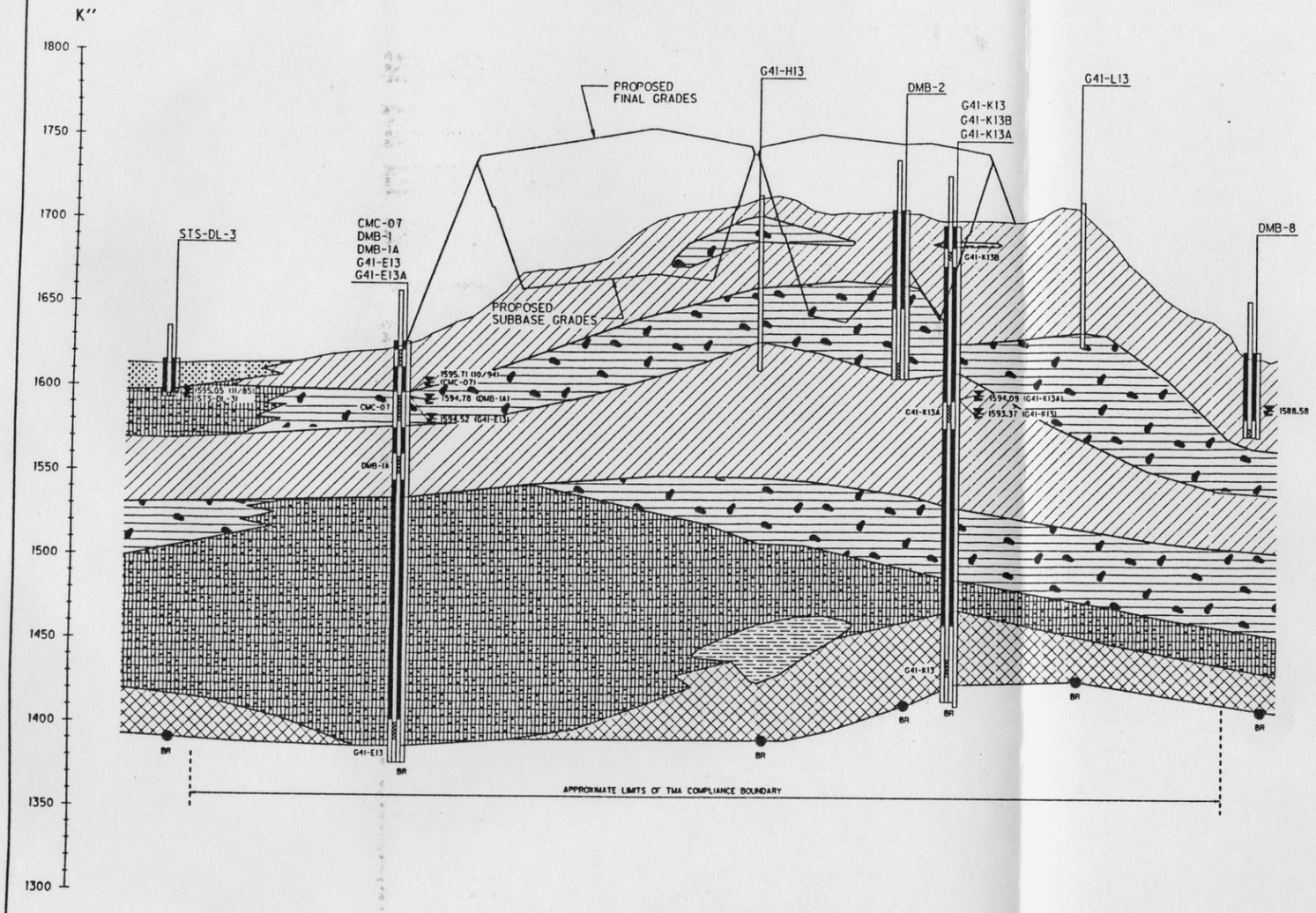
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APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO.: 9	SHEET 9 OF 31	REVISION NO.: 0			

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SECTION K''-K'''



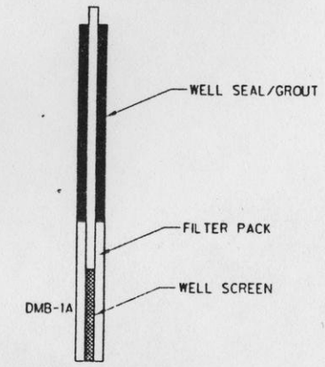
LEGEND

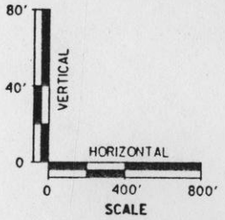
- Qtl UNDIFFERENTIATED LATE WISCONSIN TILL — NASHVILLE MEMBER, COPPER FALLS FORMATION AND THE MAPLEVILLE MEMBER, HORICON FORMATION
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- INFERRED GEOLOGIC CONTACT
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- LL LIQUID LIMIT
- PI PLASTICITY INDEX
- P200 PERCENT FINES
- USCS UNIFIED SOIL CLASSIFICATION SYSTEM
- Ql LACUSTRINE DEPOSITS — GLACIO-LACUSTRINE, BEDDED, SILT AND CLAY WITH MINOR AMOUNTS OF SAND OF WISCONSINAN, HOLOCENC AND RECENT AGE.
- Qhfc ICE-CONTACT DEPOSITS — ABLATION TILL, FLOW TILL, GLACIO-FLUVIAL SILT, CLAY AND SAND OF WISCONSINAN AGE
- Ms MASSIVE SAPROLITE — UNSTRUCTURED CLAY, SILT AND SAND, HIGHLY WEATHERED BEDROCK
- Ss STRUCTURED SAPROLITE BEDROCK — CLAY, SILT AND SAND WITH RELIC BEDROCK STRUCTURES, PARTLY WEATHERED BEDROCK
- BR PCu UNDIFFERENTIATED BEDROCK — PRECAMBRIAN METAVOLCANIC BEDROCK, MAY INCLUDE STRUCTURED SAPROLITE
- BR INFERRED UNDIFFERENTIATED BEDROCK ELEVATION BASED UPON STS 1984 BEDROCK ELEVATION MAP
- *Kff FIELD PERMEABILITY - FALLING HEAD TEST IN CM/SEC.
- *Kfr FIELD PERMEABILITY - RISING HEAD TEST IN CM/SEC.
- *(Kff) *(Kfr) WELL ABANDONED
- KL LABORATORY HYDRAULIC CONDUCTIVITY IN CM/SEC.

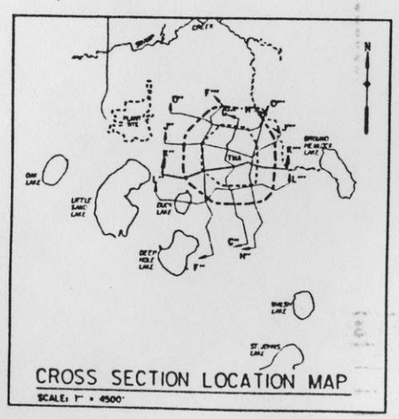
Crandon Mining Company - TMA Laboratory Soil Data

Boring ID	Sample *	Depth(ft.)	% Gravel	% Sand	P200	% Silt	% Clay	USCS	Other
G41-E13	S-3	10-11.5	11	73	16				
	S-5	20-21.5	7	71	22	20	2	SM	
	S-9	40-41.5	35	52	13			SM	
	S-13 & 14	65-65.5	23	59	18			SM	
	S-21	100-100.8	2	81	17			SM	
	S-22	110-111	3	84	13			SM	
	S-25	140-141	1	95	4			SP	
	S-30	190-191	0	95	5			SP-SM	
	S-34	230-231	0	93	7			SP-SM	
CMC-07	S-8	40-42	9.8	71.5	18.7			SM	
G41-H13	S-9	45-47	1.2	70.2	28.6	2.4	26.2	SM	
	S-1	0-1.5	10	52	38	31		SM	
	S-7	29-30.5	14	63	23			SM	
	S-15	71-72.2	2	84	14			SM	
	S-22	104-104.8	25	59	16			SM	
G41-K13	S-1	120-120.9	6	75	19	10	9	SM	
	S-3	142-142.5	8	71	21			SM	
	S-6	172-173	26	65	9			GM-SM	
	S-14	254-254.7	22	63	17			SM	
G41-K13A	S-2	5-6.5	12	65	23			SM	
	S-8	35-35.3	20	60	20			SM	
	S-13	60.60.6	10	70	20	7	13	SM	
	S-15	70-70.3	41	45	14			GM-SM	
	S-16	75-75.5	33	57	10	2	8	SP-SM	
	S-20	95-96	10	70	20			SM	
	S-23A	116-116.3	8	82	10			SP-SM	
G41-K13B	S-1	1-2.5	0	27	63	56	7	ML	
	S-3	9.5-11	0	13	87	78	9	CL-ML	LL:22 PI: 4
	S-5	19.5-20	20	64	16			SM	

WELL CONSTRUCTION INFORMATION







CROSS SECTION LOCATION MAP

SCALE: 1" = 4500'

NOTES:

1. WELLS ARE LOCATED IN INDIVIDUAL BORINGS. THE FILTER PACK IS SILICA SAND, AND THE ANNULAR SPACE AROUND THE WELL CASING WAS BACKFILLED WITH BENTONITE PELLETS AND CEMENT/BENTONITE GROUT AND CONCRETE.
2. THIS CROSS SECTION DERIVED FROM THE REGIONAL CROSS SECTION K-K' OF THE EIR, 1995.

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

TITLE: GEOLOGIC CROSS SECTION K''-K'''

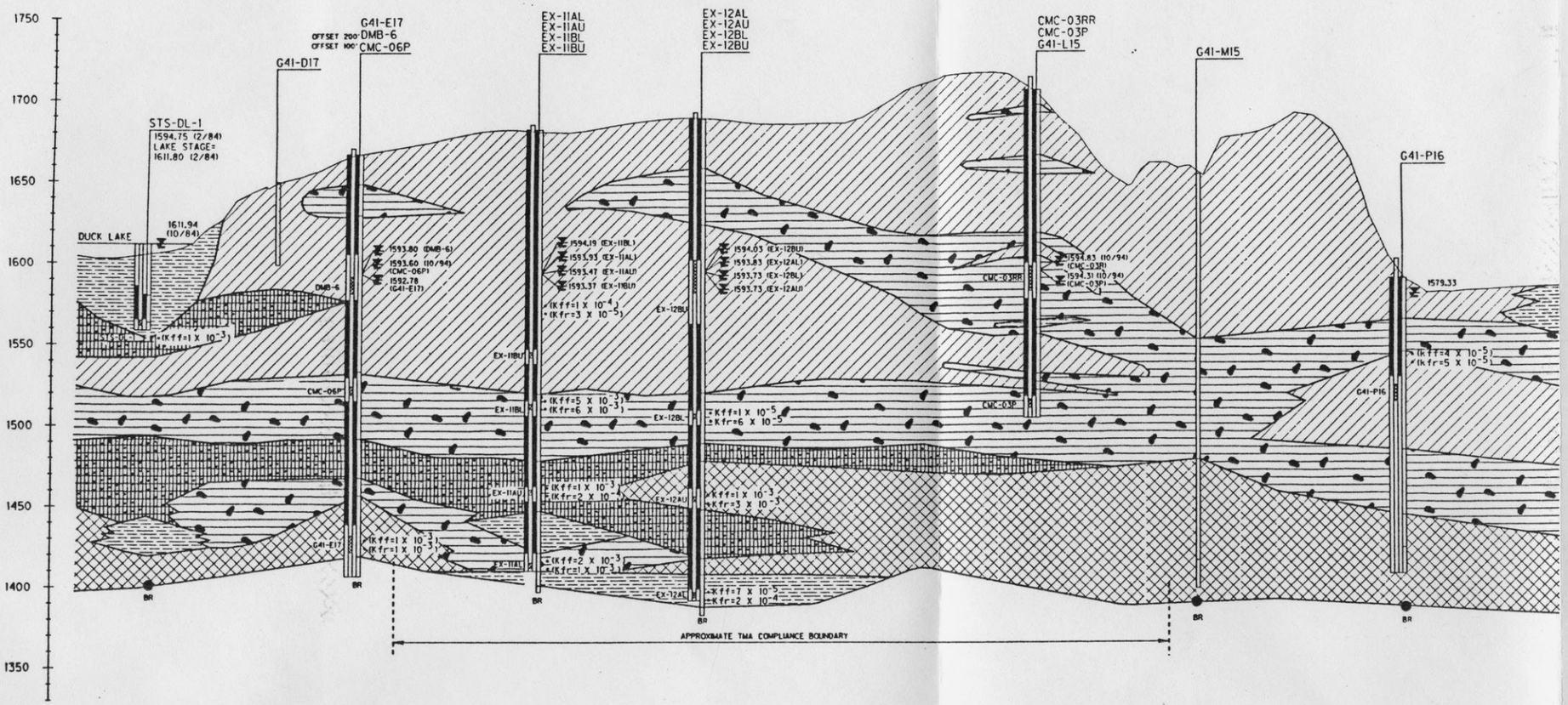
SCALE: VERT. 1" = 40' HOR. 1" = 400'	STATE: WISCONSIN	COUNTY: FOREST	
DRAWN BY: CAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95
DRAWING NO.: 10	SHEET: 10 OF 31	REVISION NO.: 0	

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05/26/95 bmt

SECTION L'' - L'''

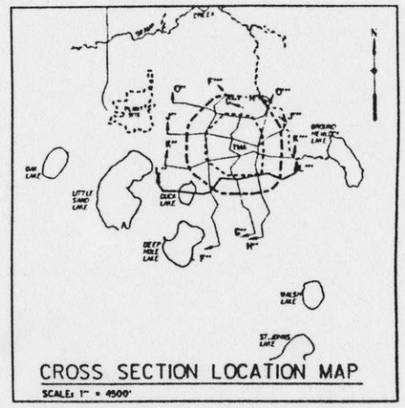


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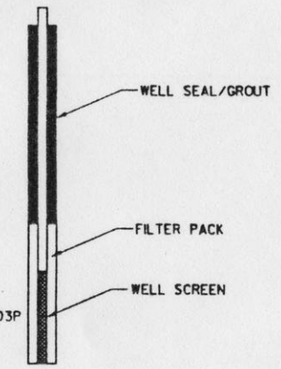
- Qtl UNDIFFERENTIATED LATE WISCONSIN TILL — NASHVILLE MEMBER, COPPER FALLS FORMATION AND THE MAPLEVILLE MEMBER, HORICON FORMATION
- Qte UNDIFFERENTIATED EARLY-WISCONSIN-TILL AND PRE-WISCONSIN TILL — MARATHON AND LINCOLN FORMATIONS, MAY INCLUDE MASSIVE SAPROLITE
- Qowf FINE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND SILT OF WISCONSINAN AGE
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- PCu UNDIFFERENTIATED BEDROCK — PRECAMBRIAN METAVOLCANIC BEDROCK, MAY INCLUDE STRUCTURED SAPROLITE
- ——— INFERRED GEOLOGIC CONTACT
- 1594.58 (EX-12BL) WATER LEVEL ELEVATION (10/84)* UNLESS INDICATED
- BR INFERRED UNDIFFERENTIATED BEDROCK ELEVATION BASED UPON STS 1984 BEDROCK ELEVATION MAP
- LL LIQUID LIMIT
- PI PLASTICITY INDEX
- P200 PERCENT FINES
- USCS UNIFIED SOIL CLASSIFICATION SYSTEM
- KL LABORATORY HYDRAULIC CONDUCTIVITY IN CM/SEC.
- *Kff FIELD PERMEABILITY - FALLING HEAD TEST IN CM/SEC.
- *Kfr FIELD PERMEABILITY - RISING HEAD TEST IN CM/SEC.
- *(Kff) *(Kfr) WELL ABANDONED

Crandon Mining Company - TMA Laboratory Soil Data

Boring ID	Sample #	Depth (ft.)	% Gravel	% Sand	P200	% Silt	% Clay	USCS	Other
STS-DL-1	10	25.5-28	0	22	78	69	9	CL	KL:1.3X10-6 LL:30.4
	15	38-40	0	7	93	83	10	CL	KL:1.0X10-7 PI:9.1
G41-D17									
G41-E17	S-2	12-13.5	31	54	15			SW	
	S-7	40-41.5	7	68	25	18	7	SW	
	S-16	86-87.5	6	71	23			SW	
	S-24	153.5-155	37	54	9	3	6	SW-SM	
	S-28	195-196	6	82	12			SW-SP	
	S-30	214-215.5	0	10	90			ML	
	S-31	224.5-226	0	1	99	51	32	ML	
	S-32a	234-235	3	34	63	39	24	ML	
	S-32b	234-235	3	35	62	46	32	ML	
	S-33	245.5-246.5	31	69	20			SW	
CMC-06P	S-26	145-147	34.5	55.5	10.2			SP-SM	
EX-11	S-1	4-6	9	71	20			SW	
	S-2	10-12	37	60	23			SW	
	S-3	20.5-22.6	21	61	18			SW	
	S-4	30.5-32.5	9	63	26	22	4	SW	
	S-5	41-45	32	66	22			SW	
	S-6	45-49	34	64	22	18	4	SW	
	S-7	50-54	6	70	24			SW	
	S-8	55-57	9	69	22			SW	
	S-9	60-64	9	71	20			SW	
	S-10	65-69	24	60	16	13	3	SW	
	S-12	82.5-84.5	7	67	26	22	4	SW	
	S-16	123.5-125.5	20	67	13	11	2	SW	
	S-19	154-156	31	71	18			SW	
	S-20	164.5-166.5	37	56	7			SW-SP	
	S-22	185-187	8	86	6			SW-SP	
	S-24	205.5-207.5	1	95	4			SP	
	S-27	236-237	0	77	23			SW	
	S-29	256.5-258.5	0	4	96	74	22	CL	
	S-30	267-269	22	77	12			SW-SP	
	S-31	277-278.5	0	2	98	86	12	CL	
EX-12	S-1	10-12	9	82	9			SW-SP	
	S-2	20-22	18	52	30	23	7	SW	
	S-4	41-43	34	6	59	38	1	GW	
	S-5	48-49	44	32	7			GW	
	S-6	59-60	56	41	3			SW-GW	
	S-7	69-70.5	29	36	44			SW	
	S-8	80-81.5	49	36	15	33	3	SW	
	S-9	90-91.5	9	79	15			SW	
	S-11	110.5-112	6	63	25			SW	
	S-14	143-142.5	21	62	14	12	2	SW	
	S-15	152-152.5	44	46	7			SW-SP	
	S-16	203-204.5	1	93	6			SW-SP	
	S-21	213-214.5	30	67	23	21	2	SW	
	S-22	223.5-225	11	69	18			SW	
	S-24	244.2-245.5	1	81	18			SW	
	S-27	274.5-276.5	9	66	25	21	4	SW	
	S-28	285-287	0	48	52	45	7	ML	
	S-29	295-297	0	4	96	84	12	ML	
G41-L15	S-2	4.5-6	14	66	20	15	5	SW	
	S-7	29-30.5	3	48	15			SW	
	S-9	39-40.5	13	77	10			SP-SM	
CMC-03RR	S-3	113-115	21.5	54.8	10.7			SW	
CMC-03P	S-37	390-392	37.4	54.1	12.1			SW	
G41-M15	S-5	46-48	4	96	8			SW-SP	
	S-12	90-91.5	2	61	37	31	6	SW	
	S-22	121-122.5	24	78	8			SW-SP	
	S-26	153-154.5	29	63	8			SW-SP	
	S-28	173-174.5	29	64	10			SW-SP	
G41-P16	S-4	36.5-38	6	54	40			GP	
	S-8	56-57.5	13	72	17			SW	
	S-9	86-87.5	24	63	13			SW-SP	
	S-12	90-91.5	34	61	16			SW	
	S-26	148.5-149.9	29	51	20			SW	

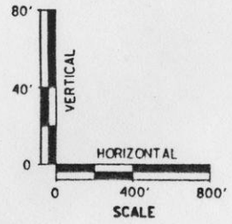


WELL CONSTRUCTION INFORMATION



NOTES:

1. WELLS ARE LOCATED IN INDIVIDUAL BORINGS, THE FILTER PACK IS SILICA SAND, AND THE ANNULAR SPACE AROUND THE WELL CASING WAS BACKFILLED WITH BENTONITE PELLETS AND CEMENT/BENTONITE GROUT AND CONCRETE.
2. THIS CROSS SECTION DERIVED FROM THE REGIONAL CROSS SECTION L-L' OF THE EIR, 1995.



Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

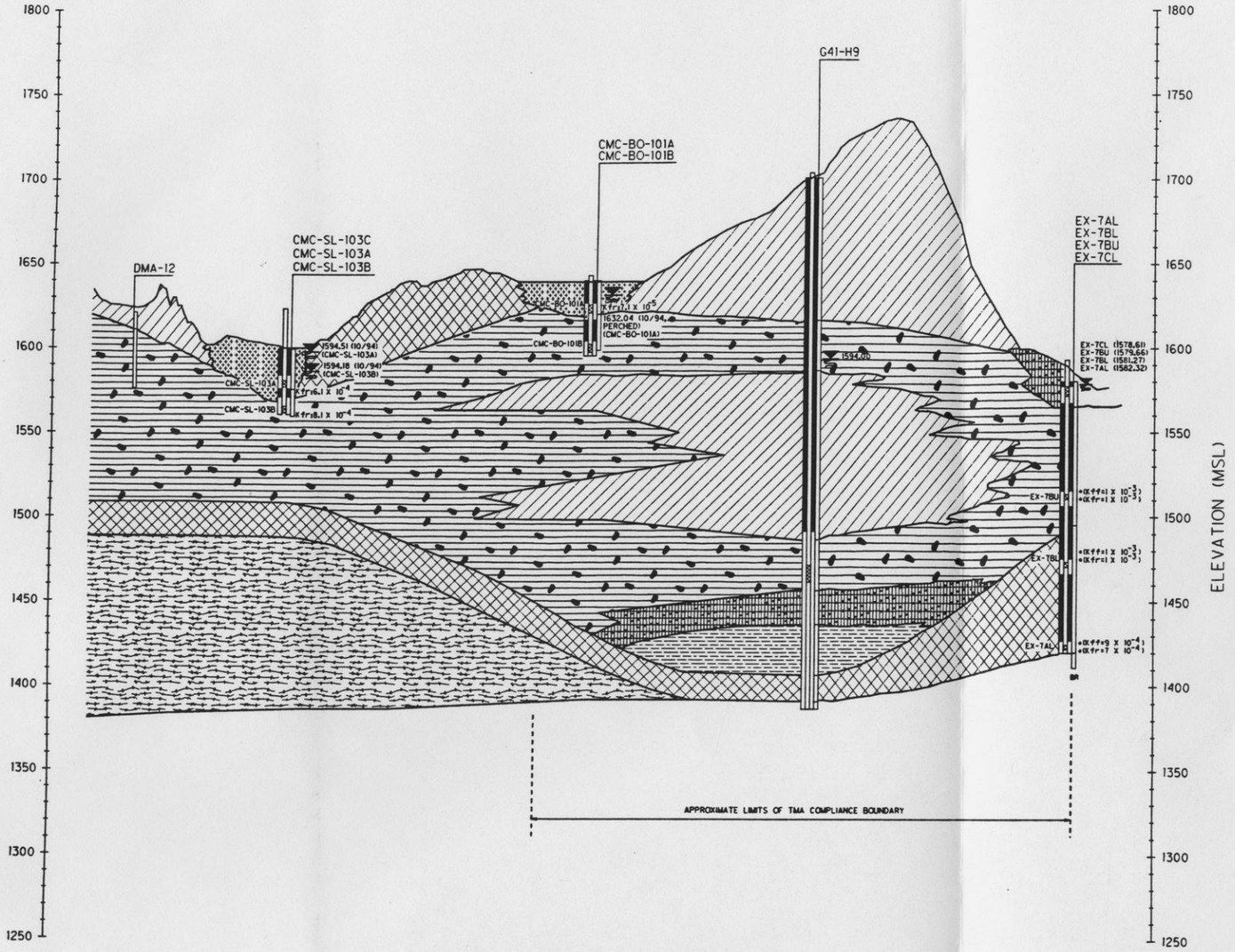
TITLE: GEOLOGIC CROSS SECTION L''-L'''

SCALE VERT: 1" = 40' HORIZ: 1" = 400'	STATE: WISCONSIN	COUNTY: FOREST	
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95
DRAWING NO: 11		SHEET 11 OF 31	REVISION NO: 0

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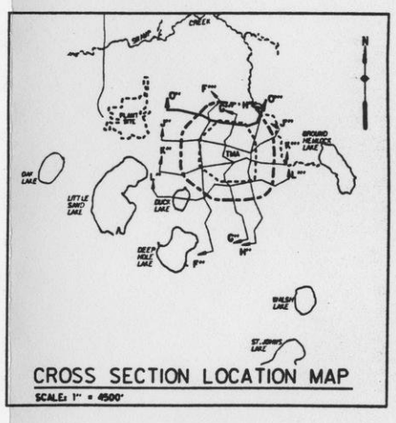
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SECTION O''-O'''



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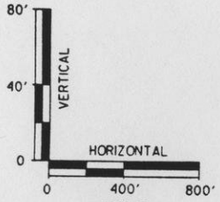
- Qtl UNDIFFERENTIATED LATE WISCONSIN TILL — NASHVILLE MEMBER, COPPER FALLS FORMATION AND THE MAPLEVILLE MEMBER, HORICON FORMATION
- Qte UNDIFFERENTIATED EARLY-WISCONSIN-TILL AND PRE-WISCONSIN TILL — MARATHON AND LINCOLN FORMATIONS, MAY INCLUDE MASSIVE SAPROLITE
- Qowf FINE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND SILT OF WISCONSINAN AGE
- Qowl COARSE-GRAINED OUTWASH DEPOSITS — GLACIAL FLUVIAL STRATIFIED SAND AND GRAVEL OF WISCONSINAN AGE
- INFERRED GEOLOGIC CONTACT
- 1615.99 (STS-DL-3) WATER LEVEL ELEVATION (10/84)* UNLESS INDICATED
- LL LIQUID LIMIT
- PI PLASTICITY INDEX
- P200 PERCENT FINES
- USCS UNIFIED SOIL CLASSIFICATION SYSTEM
- Ql LACUSTRINE DEPOSITS — GLACIO-LACUSTRINE, BEDDED, SILT AND CLAY WITH MINOR AMOUNTS OF SAND OF WISCONSINAN, HOLOCENE AND RECENT AGE.
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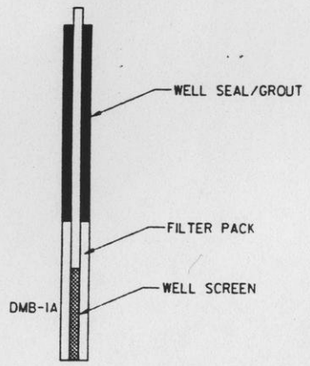


CROSS SECTION LOCATION MAP

SCALE: 1" = 4500'

WELL CONSTRUCTION INFORMATION





Crandon Mining Company - TMA Laboratory Soil Data

Boring ID	Sample #	Depth (ft.)	% Gravel	% Sand	P200	% Silt	% Clay	USCS	Other
CMC-SL-103A	S-4	6-8.5	0.1	37.4	42.5	19.6	22.9	SC-SM	KL: 4.2 x 10^{-7}
CMC-SL-103B	ST-4	26-28	0.0	14.8	85.2	61.1	24.1	CL	KL: 3.5 x 10^{-6}
	S-9	38-40	25.5	70.1	4.4	—	—	SP	
CMC-BC-101A	S-1	14-16	0.4	23.3	73.3	53.6	22.7	CL	LL:28 PI: 8
CMC-BC-101B	S-3	4-6	0	53.7	46.3	32.5	13.8	SC-SM	KL:1X10-6 LL:19 PI: 5
	S-7	12-14	0	4.4	95.6	73.3	22.3	CL	KL:2.9X10-9 LL:28 PI: 8
	S-18	36-38	1.8	95.6	2.6			SP	
G41-H9	S-3	21-22.5	14	68	18			SM	
	S-5	40-41	26	62	12			SM	
	S-8	70-71.8	28	58	14			SM	
	S-10	90-91	49	44	7			SM-SP	
	S-15	140-141.5	17	63	20			SM	
	S-16	150-151.3	14	64	22			SM	
	S-21	200-200.8	1	92	8	101.0		SM-SP	
	S-22	210.5-211.4	14	64	22			SM	
Ex-7	S-1	61.5-63.5	11	80	9	--	--	SM-SP	--
	S-2	120-122	7	52	41	39	2	SM	--
	S-3	162-163	61	27	12	10	2	GP-SM	--

NOTES:

1. WELLS ARE LOCATED IN INDIVIDUAL BORINGS, THE FILTER PACK IS SILICA SAND, AND THE ANNULAR SPACE AROUND THE WELL CASING WAS BACKFILLED WITH BENTONITE PELLETS AND CEMENT/BENTONITE GROUT AND CONCRETE.
2. THIS CROSS SECTION DERIVED FROM THE REGIONAL CROSS SECTION O-O' OF THE EIR, 1995.

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

TITLE: GEOLOGIC CROSS SECTION O''-O'''

SCALE VERT: 1" = 40' HOR: 1" = 400'	STATE: WISCONSIN	COUNTY: FOREST	
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95
DRAWING NO.: 12	SHEET 12 OF 31	REVISION NO.: 0	

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SITE PREPARATION /OPERATION

CELL	STAGE	MINE WASTE VOLUME (CY)	EARTHWORK (CY) CUT ①	EARTHWORK (CY) FILL	12" LPS LAYER (CY)	GEOMEMBRANE (SF)	GEOTEXTILE (CUSHIONING) (SF)	24" DRAINAGE LAYER (CY)	GEOTEXTILE (FILTER) (SF)	18" (BASE) TILL LAYER (CY)	6" TILL FINES LAYER (CY)	18" SIDEWALL TILL LAYER (CY)
1	I	2032900	1339400	911300	73500	1965800	1965800	93000	1270400	69700	23300	36200
	II	2153600	100	1807000	19900	522600	522600	N.A.	N.A.	N.A.	N.A.	30400
2	III	3391200	4038500	145900	89900	2398100	2398100	101800	1389700	76300	25500	53400
	IV	4174600	1200	1616600	33000	885400	885400	N.A.	N.A.	N.A.	N.A.	50700
3	V	1586400	2197900	7100	54000	1430500	1430500	55800	762000	41800	14000	35100
	VI	2649800	2800	614300	26900	725700	725700	N.A.	N.A.	N.A.	N.A.	41100
4	VII	1705600	3435100	700	59900	1588200	1588200	62000	845900	46500	15500	37100
	VIII	2852600	45200	178000	27100	732900	732900	N.A.	N.A.	N.A.	N.A.	43700
TOTALS		20546700	11060200	5280900	384200	10249200	10249200	312600	4268000	234300	78300	327700

① INCLUDES TOPSOIL

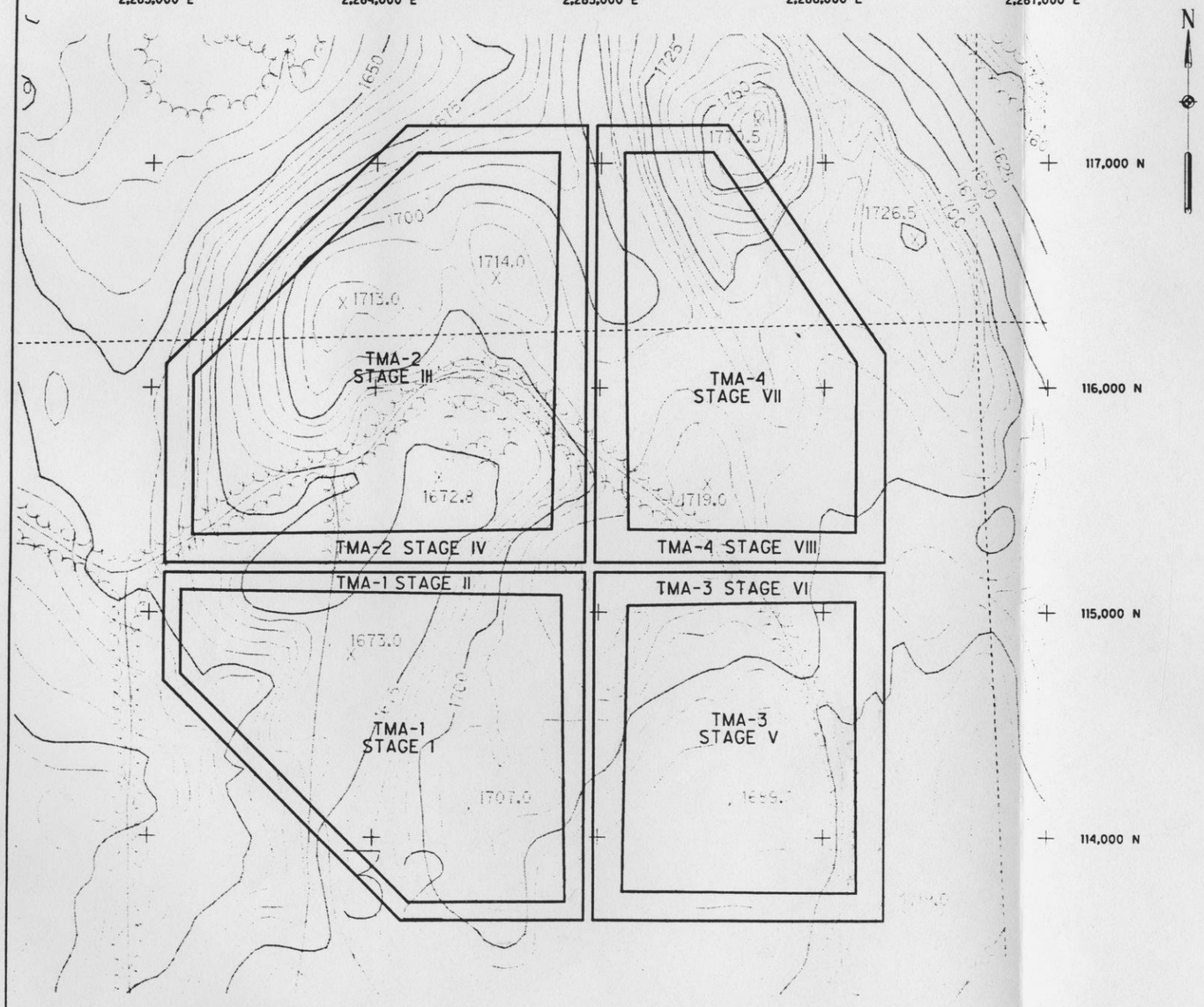
(CY) - CUBIC YARDS
(SF) - SQUARE FEET
LPS - LOW PERMEABILITY SOILS
N.A. - NOT APPLICABLE
ALL VALUES ARE INPLACE QUANTITIES

CLOSURE

CELL	STAGE	FILL MATERIAL (CY) ①	36" SOIL GRADING (CY)	12" LPS LAYER (CY)	GEOMEMBRANE (SF)	12" DRAINAGE LAYER (CY)	36" ROOTING LAYER (CY)	6" TOPSOIL LAYER (CY)
1	I	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.
	II	373600	258900	87500	2420400	88500	258000	44300
2	III	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.
	IV	585600	342100	115400	3179700	116400	341100	58300
3	V	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.
	VI	307900	221800	75100	2076300	75900	220200	38000
4	VII	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.
	VIII	328300	238900	80800	2234200	81600	238100	40900
TOTALS		1595400	1061700	358800	9910600	362400	1057400	181500

① FILL MATERIAL MAY INCLUDE TAILINGS. WASTE ROCK. TILL SOIL OR OTHER MATERIALS.

(CY) - CUBIC YARDS
(SF) - SQUARE FEET
LPS - LOW PERMEABILITY SOILS
N.A. - NOT APPLICABLE



NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.

LEGEND

- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- 1692.0 X EXISTING SPOT ELEVATION

SCALE
0 300' 600'

Crandon Mining Company

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

TITLE: SITE SEQUENCING PLAN

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO. 13	SHEET 13 OF 31	REVISION NO. 0			

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2,283,000 E

2,284,000 E

2,285,000 E

2,286,000 E

117,000 N

116,000 N

115,000 N

114,000 N

N

RECLAIM POND

PROPOSED INTERIOR BERM SEE DETAIL (5/28)

PROPOSED PERIMETER BERM SEE DETAIL (1/28)

PROPOSED BENCH BERM SEE DETAILS (3/28) (4/28)

PROPOSED SUMP SEE DETAIL (1/30)

EX-11CL

EX-12BU

LEGEND

- STREAMS
- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- 1692.0 X — SPOT ELEVATION
- EX-9BU — EXISTING MONITORING WELL LOCATION AND NUMBER
- SECTION LINE
- 1725 — PROPOSED CONTOUR (SUBBASE GRADE)
- 3:1 — PROPOSED SLOPE RATIO (TYP.)
- 2% — PROPOSED SLOPE (TYP.)
- ■ PROPOSED SUMP LOCATION

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING SITE SUBBASE GRADES. (BOTTOM OF 1' LOW PERMEABILITY SOIL LAYER).

SCALE
0 200' 400'

Crandon Mining Company

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

TITLE: SUBBASE GRADES

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO.: 14	SHEET 14 OF 31	REVISION NO.: 0			

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2,283,000 E

2,284,000 E

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2,286,000 E

117,000 N

116,000 N

115,000 N

114,000 N

RECLAIM POND

PROPOSED INTERIOR BERM SEE DETAIL (5/28)

TOE OF BERM W/ DRAINAGE LAYER SEE DETAIL (1/29)

PROPOSED PERIMETER BERM SEE DETAIL (1/28)

PROPOSED BENCH BERM SEE DETAILS (3/28) (4/28)

TOE OF BERM W/O DRAINAGE LAYER SEE DETAIL (6/28)

PROPOSED SUMP SEE DETAIL (1/30)

N

LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- 1692.0 X — SPOT ELEVATION
- SECTION LINE
- 1725 — PROPOSED CONTOUR (SUBBASE GRADE)
- 3:1 — PROPOSED SLOPE RATIO (TYP.)
- 2% — PROPOSED SLOPE (TYP.)
- ■ PROPOSED SUMP LOCATION

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING SITE BASE GRADES. (TOP OF 1' LOW PERMEABILITY SOIL LAYER).

SCALE 0 200' 400'

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

Crandon Mining Company

TITLE: BASE GRADES

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO.: 15	SHEET 15 OF 31	REVISION NO.: 0			

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2,286,000 E

117,000 N

116,000 N

115,000 N

114,000 N

PROPOSED CLEANOUT SEE DETAILS (2/30)

PROPOSED LEACHATE FORCEMAIN SEE DETAIL (5/28)

PROPOSED PERFORATED LEACHATE COLLECTION PIPE SEE DETAIL (1/29)

MANHOLE

DISCHARGE MANHOLE

RECLAIM PIPE CROSSING INTERIOR BERM SEE DETAIL (4/29)

PROPOSED RECLAIM PIPE SEE DETAIL (3/28)

PUMP STATION

RECLAIM POND

PROPOSED SUMP SEE DETAIL (1/30)

PIPELINE TO PLANT SITE FOR WATER REUSE

BYPASS PIPELINE

MANHOLE

RECLAIM PIPE AT PERIMETER BERM SEE DETAIL (6/29)

PROPOSED DECANT STRUCTURE SEE DETAIL (3/30)

PROPOSED SIDE SLOPE RISER SEE DETAILS (1/30)

N

LEGEND

- EXISTING ROAD
- 1675 EXISTING CONTOUR
- 1692.0 X SPOT ELEVATION
- EX-9BU MONITORING WELL LOCATION AND NUMBER
- SECTION LINE
- 1725 PROPOSED CONTOUR (BASE GRADE)
- 3:1 PROPOSED SLOPE RATIO (TYP.)
- 2% PROPOSED SLOPE (TYP.)
- PROPOSED RECLAIM PIPE
- PROPOSED LEACHATE PIPE
- PROPOSED PERFORATED LEACHATE COLLECTION PIPE
- PROPOSED DECANT STRUCTURE
- PROPOSED VALVE
- PROPOSED SIDE SLOPE RISER
- PROPOSED CLEANOUT RISER
- PROPOSED SUMP LOCATION

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING SITE BASE GRADES. (TOP OF 1' LOW PERMEABILITY SOIL LAYER).

SCALE
0 200' 400'

Crandon Mining Company

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

TITLE: PIPING PLAN

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO.: 16	SHEET 16 OF 31	REVISION NO.: 0			

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2,284,000 E

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2,286,000 E

117,000 N

116,000 N

115,000 N

114,000 N

RECLAIM POND

TAILINGS PIPELINE FROM MILL TO TMA

PROPOSED INTERIOR BERM SEE DETAIL (5/28)

PROPOSED PERIMETER BERM SEE DETAILS (1/28) (3/29)

PROPOSED BENCH BERM SEE DETAIL (3/28)

N

LEGEND

- LAKES
- STREAMS
- EXISTING ROAD
- —1675— EXISTING CONTOUR
- 1692.0 X SPOT ELEVATION
- SECTION LINE
- —1725— PROPOSED CONTOUR (TOP OF TILL GRADE)
- 3:1 PROPOSED SLOPE RATIO (TYP.)
- 2% PROPOSED SLOPE (TYP.)
- PROPOSED VALVE
- PROPOSED MASS DISCHARGE LINE
- PROPOSED TAILINGS DISCHARGE PIPELINE AND SPIGOT ARRANGEMENT

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING SITE TOP OF TILL GRADES. (TOP OF 6" TILL FINES LAYER AT THE BASE)

SCALE
0 200' 400'

Crandon Mining Company

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

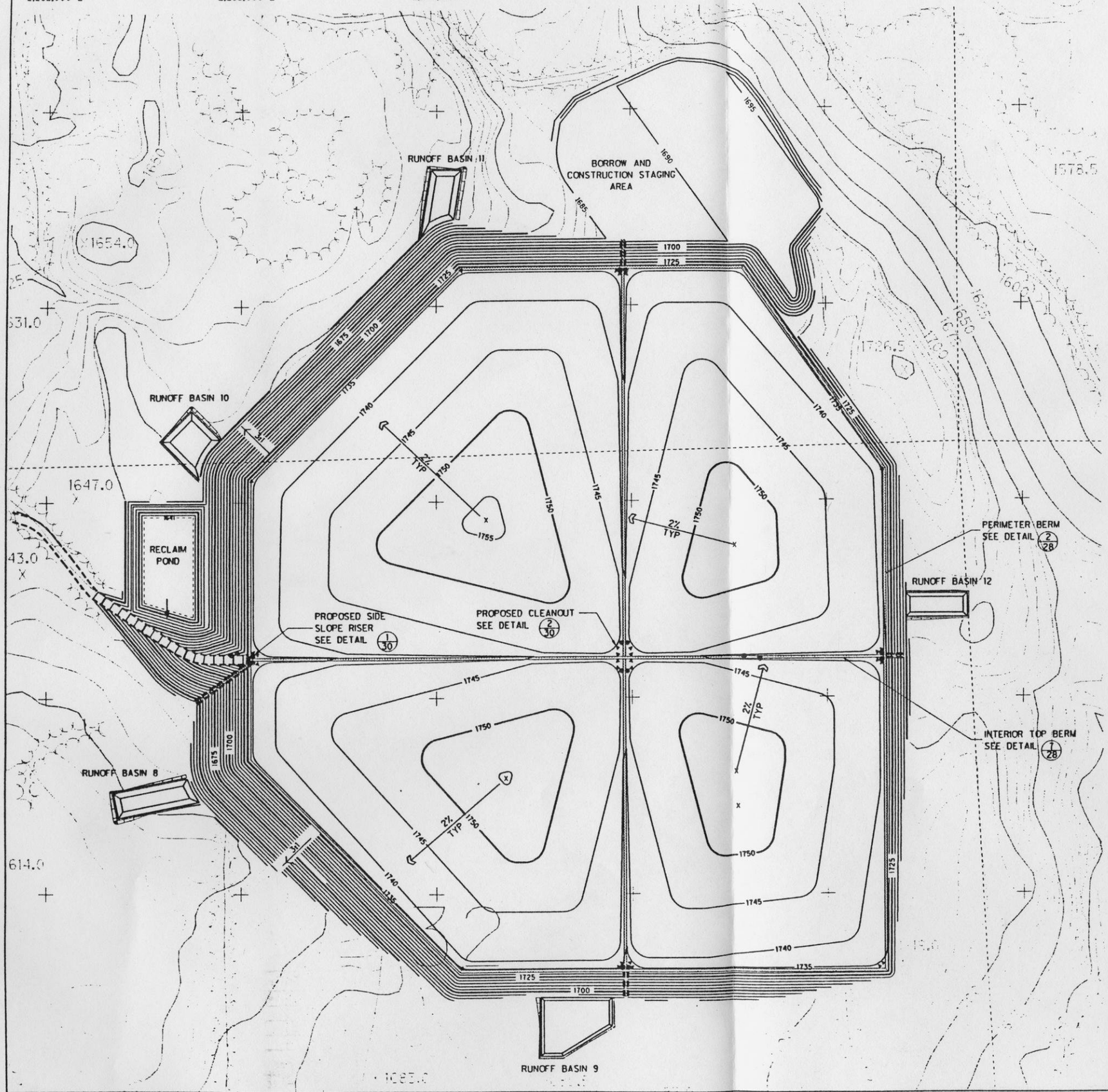
TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

TITLE
TOP OF TILL GRADES AND TAILINGS DISTRIBUTION PIPING

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO. 17	SHEET 17 OF 31	REVISION NO. 0			

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LEGEND

- EXISTING ROAD
- —1675— EXISTING CONTOUR
- 1692.0 X SPOT ELEVATION
- SECTION LINE
- —1725— PROPOSED CONTOUR (FINAL GRADE)
- 3:1 PROPOSED SLOPE RATIO (TYP.)
- 2% PROPOSED SLOPE (TYP.)
- PROPOSED CLEANOUT
- PROPOSED SIDE SLOPE RISER
- PROPOSED VALVE
- PROPOSED SPILLWAY

N

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING SITE FINAL GRADES. (TOP OF FINAL COVER).
6. REFER TO DRAWING NO. 22 FOR DRAINAGE FEATURES.

SCALE
0 250' 500'

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Foth & Van Dyke

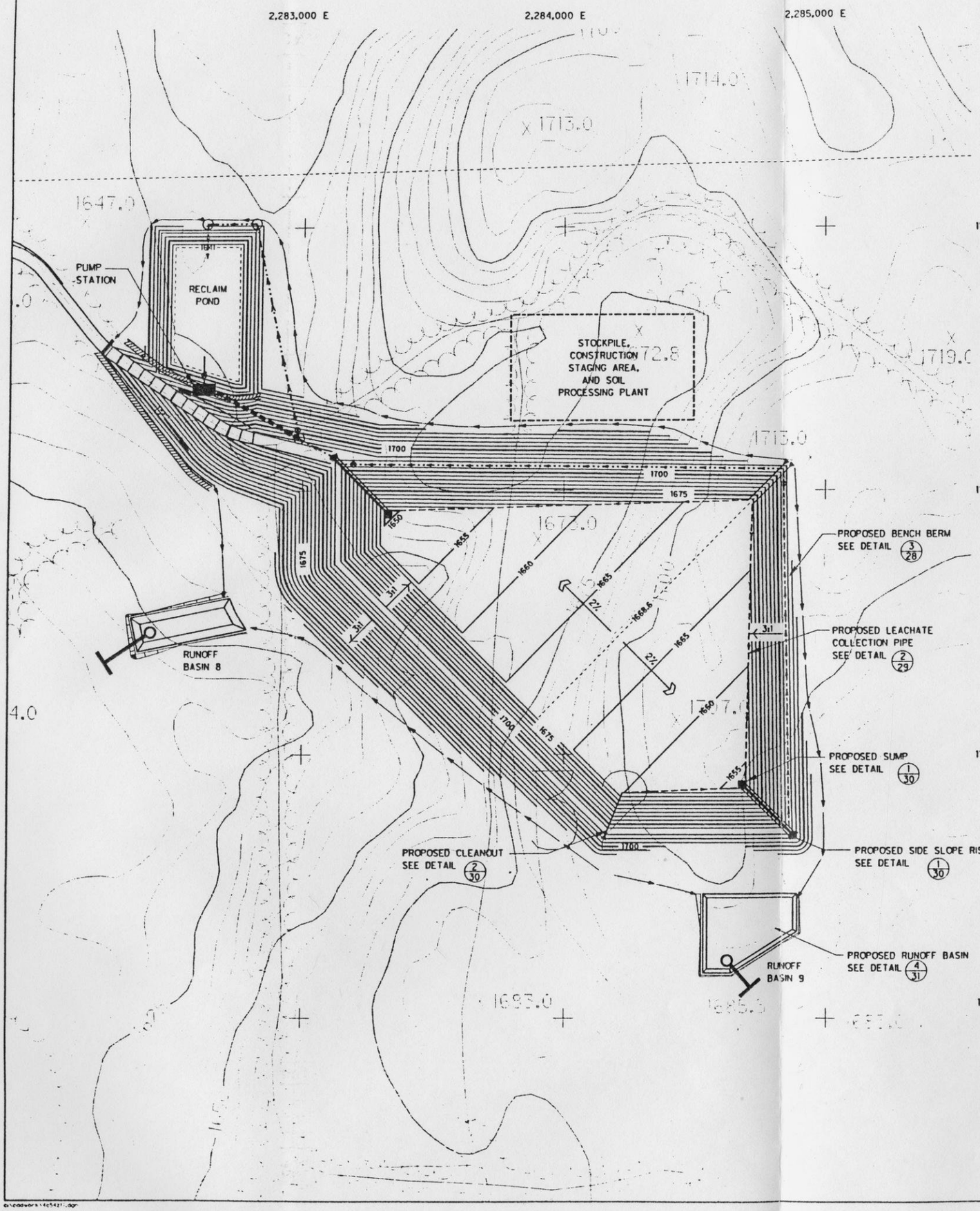
REVISED	DATE	BY	DESCRIPTION

Crandon Mining Company

TITLE
FINAL GRADES

SCALE AS SHOWN	STATE WISCONSIN		COUNTY FOREST		
DRAWN BY GAM, JPRI	DATE 5/95	CHECKED BY JKSI	DATE 5/95		
APPROVED BY REM	DATE 5/95	APPROVED BY NXP	DATE 5/95		
APPROVED BY GWS	DATE 5/95	CRANDON MINING COMPANY DEM	DATE 5/95		
DRAWING NO. 18	SHEET 18 OF 51	REVISION NO. 0			

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CONSTRUCTION QUANTITIES

ITEM / DESCRIPTION	EST. QNTY.
SITE PREPARATION	
CLEARING AND GRUBBING	59.5 AC
TOPSOIL STRIPPING	72,000 CY
SITE GRADING (CUT)	1,267,400 CY
SITE GRADING (FILL)	911,300 CY
ACCESS ROAD	5,725 LF
LINER	
LOW PERMEABLE SOIL LAYER (12")	73,500 CY
GEOMEMBRANE	1,965,800 SF
GEOTEXTILE (CUSHION)	1,965,800 SF
DRAINAGE LAYER (24")	93,000 CY
GEOTEXTILE (FILTER)	1,270,400 SF
TILL LAYER (18") BASE	69,700 CY
TILL FINES LAYER (6")	23,300 CY
TILL LAYER (18") SIDEWALL	36,200 CY
LEACHATE COLLECTION SYSTEM	
STONE	660 CY
6" PERFORATED PIPE	2,960 LF
6" CLEANOUT RISER	1,230 LF
SUMP / SIDE SLOPE RISER	500 LF
LEACHATE FORCEMAIN	1,360 LF
DECANT STRUCTURE	1 LS
DECANT PIPE	1,690 LF
TAILINGS PIPE	5,550 LF
MASS DISCHARGE PIPE	340 LF
DRAINAGE	
DOWNSLOPE SPILLWAY	N.A.
DIVERSION BERM	1,250 LF
SEDIMENTATION BASIN	2 LS
LANDSCAPING	
TOPSOIL	16,500 CY
SEED, FERTILIZE, MULCH	20.5 AC

LEGEND

- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- 1692.0 — SPOT ELEVATION
- SECTION LINE
- 1725 — PROPOSED CONTOUR (BASE GRADE)
- 3:1 — PROPOSED SLOPE RATIO (TYP.)
- 2% — PROPOSED SLOPE (TYP.)
- PROPOSED RECLAIM PIPE
- PROPOSED LEACHATE PIPE
- PROPOSED PERFORATED LEACHATE COLLECTION PIPE
- PROPOSED CLEANOUT RISER
- PROPOSED SIDE SLOPE RISER
- TEMPORARY DRAINAGE DITCH
- PROPOSED DRAINAGE BERM
- − 1% → FLOW DIRECTION
- PROPOSED MANHOLE

N

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING TMA-1, STAGE 1 BASE GRADES (TOP OF 1' LOW PERMEABILITY SOIL LAYER).

SCALE
0 200' 400'

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

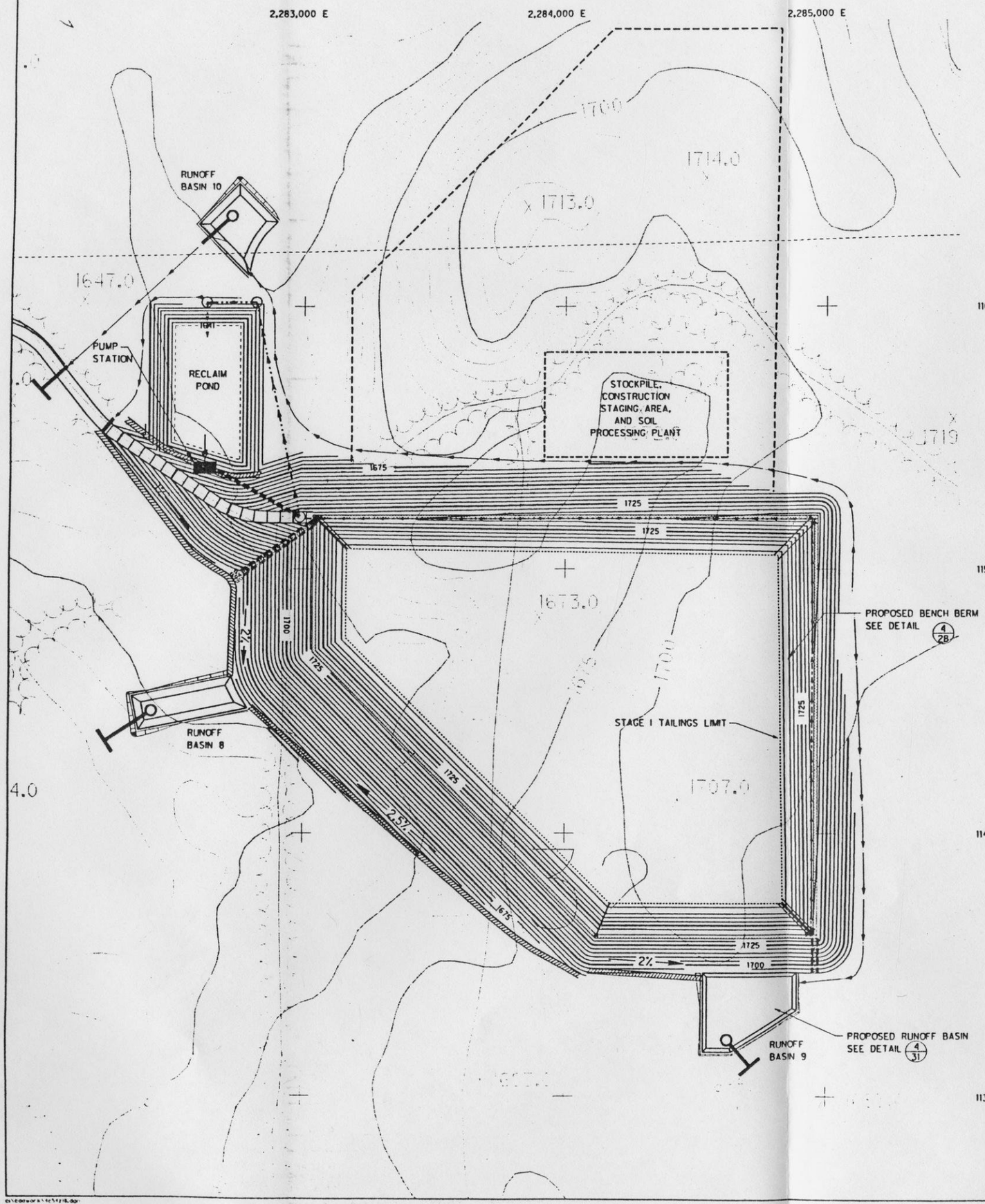
TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

TITLE
TMA-1, STAGE 1 PHASING PLAN

SCALE AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPR	DATE: 5/95	CHECKED BY: JKS	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO. 19		SHEET 19 OF 31	REVISION NO. 0		

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CONSTRUCTION QUANTITIES

ITEM / DESCRIPTION	EST. QNTY.
SITE PREPARATION	
CLEARING AND GRUBBING	27.5 AC
TOPSOIL STRIPPING	50,000 CY
SITE GRADING (CUT)	0 CY
SITE GRADING (FILL)	1,856,100 CY
ACCESS ROAD	6,350 LF
LINER	
LPS LAYER (12")	19,900 CY
GEOMEMBRANE	522,600 SF
GEOTEXTILE (CUSHION)	522,600 SF
DRAINAGE LAYER (24")	N.A.
GEOTEXTILE (FILTER)	N.A.
TILL LAYER (18") BASE	N.A.
TILL FINES LAYER (6")	N.A.
TILL LAYER (18") SIDEWALL	30,400 CY
LEACHATE COLLECTION SYSTEM	
6" CLEANOUT RISER	720 LF
SUMP / SIDE SLOPE RISER	255 LF
LEACHATE FORCEMAIN	200 LF
DECANT PIPE	200 LF
TAILINGS PIPE	6,230 LF
MASS DISCHARGE PIPE	260 LF
DRAINAGE	
DOWNSLOPE SPILLWAY	530 LF
DIVERSION BERM	2,700 LF
SEDIMENTATION BASIN	1 LS
LANDSCAPING	
TOPSOIL	23,150 CY
SEED, FERTILIZE, MULCH	28.7 AC

LEGEND

- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- 1692.0 SPOT ELEVATION
- SECTION LINE
- 1725 — PROPOSED CONTOUR (BASE GRADE)
- 3:1 PROPOSED SLOPE RATIO (TYP.)
- 2% PROPOSED SLOPE (TYP.)
- PROPOSED CLEANOUT RISER
- PROPOSED SIDE SLOPE RISER
- TEMPORARY DRAINAGE DITCH
- PROPOSED DRAINAGE BERM
- PROPOSED CULVERT
- 2% FLOW DIRECTION
- APPROXIMATE BORROW LIMITS
- PROPOSED RECLAIM PIPE
- PROPOSED LEACHATE PIPE
- PROPOSED MANHOLE

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING TMA-1, STAGE II BASE GRADES (TOP OF 1' LOW PERMEABILITY SOIL LAYER).

SCALE
0 200' 400'

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

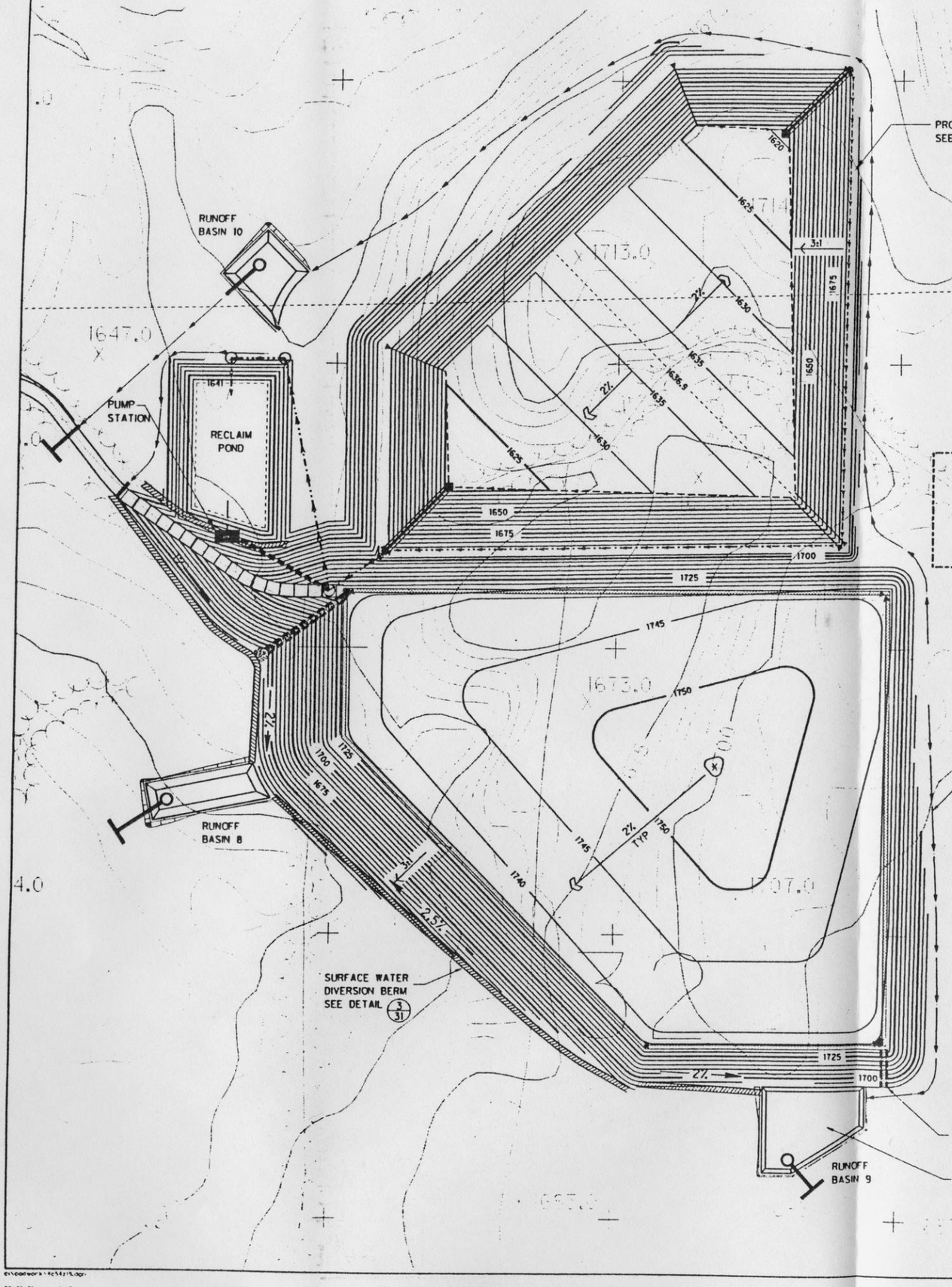
TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

TITLE
TMA-1, STAGE II PHASING PLAN

SCALE AS SHOWN	STATE WISCONSIN	COUNTY FOREST	
DRAWN BY GAM, JPRI	DATE 5/95	CHECKED BY JKSI	DATE 5/95
APPROVED BY REM	DATE 5/95	APPROVED BY NYP	DATE 5/95
APPROVED BY GWS	DATE 5/95	CRANDON MINING COMPANY DEM	DATE 5/95
DRAWING NO. 20		SHEET 20 OF 31	REVISION NO. 0

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CONSTRUCTION QUANTITIES

ITEM / DESCRIPTION	EST. QNTY.
SITE PREPARATION	
CLEARING AND GRUBBING	60.6 AC
TOPSOIL STRIPPING	73,300 CY
SITE GRADING (CUT)	3,965,200 CY
SITE GRADING (FILL)	145,900 CY
ACCESS ROAD	6,040 LF
LINER	
LPS LAYER (12")	89,900 CY
GEOMEMBRANE	2,398,100 SF
GEOTEXTILE (CUSHION)	2,398,100 SF
DRAINAGE LAYER (24")	101,800 CY
GEOTEXTILE (FILTER)	1,389,700 SF
TILL LAYER (18") BASE	76,350 CY
TILL FINES LAYER (6")	25,450 CY
TILL LAYER (18") SIDEWALL	53,400 CY
LEACHATE COLLECTION SYSTEM	
STONE	725 CY
6" PERFORATED PIPE	3,260 LF
6" CLEANOUT RISER	2,200 LF
SUMP / SIDE SLOPE RISER	645 LF
LEACHATE FORCEMAIN	1,670 LF
DECANT STRUCTURE	1 QTY
DECANT PIPE	1,600 LF
TAILINGS PIPE	6,000 LF
MASS DISCHARGE PIPE	200 LF
FINAL COVER	
FILL MATERIAL	373,600 CY
GRADING LAYER (36")	258,900 CY
CLAY LAYER (12")	87,500 CY
GEOMEMBRANE	2,420,400 SF
DRAINAGE LAYER (12")	88,500 CY
ROOTING LAYER (36")	258,000 CY
TOPSOIL (6")	44,300 CY
SEED, FERTILIZE, MULCH	58.3 AC
LANDSCAPING	
TOPSOIL	16,500 CY
SEED, FERTILIZE, MULCH	20.5 AC

N

LEGEND

- EXISTING ROAD
- 1675 — EXISTING CONTOUR
- 1692.0 — SPOT ELEVATION
- SECTION LINE
- 1725 — PROPOSED CONTOUR (BASE GRADE)
- 3:1 — PROPOSED SLOPE RATIO (TYP.)
- 2% — PROPOSED SLOPE (TYP.)
- PROPOSED CLEANOUT RISER
- PROPOSED SIDE SLOPE RISER
- TEMPORARY DRAINAGE DITCH
- PROPOSED SURFACE WATER DIVERSION BERM
- 2% — FLOW DIRECTION
- PROPOSED RECLAIM PIPE
- PROPOSED LEACHATE PIPE
- PROPOSED MANHOLE

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING TMA-1, STAGE II FINAL GRADES (TOP OF FINAL COVER) AND TMA-2, STAGE III BASE GRADES (TOP OF LOW PERMEABILITY SOIL LAYER).

SCALE
0 200' 400'

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION: REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

TITLE: TMA-1, FINAL GRADES AND TMA-2, BASE GRADES

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JPSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NYP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO: 21		SHEET: 3 OF 3	REVISION NO: 0		

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2,282,000 E
2,283,000 E
2,284,000 E
2,285,000 E
2,286,000 E
2,287,000 E

118,000 N
117,000 N
116,000 N
115,000 N
114,000 N

PROPOSED DRAINAGE DITCH SEE DETAIL (2/31)
RUNOFF BASIN 11
BORROW AREA
RUNOFF BASIN 10
RECLAIM POND
PROPOSED RUNOFF BASIN SEE DETAIL (4/31)
RUNOFF BASIN 12
ACCESS ROAD SEE DETAILS (2/28) (7/28)
PROPOSED SURFACE WATER DIVERSION BERM SEE DETAIL (3/31)
RUNOFF BASIN 8
RUNOFF BASIN 9
PROPOSED SPILLWAY SEE DETAIL (1/31)

LEGEND

- EXISTING ROAD
- 1675 EXISTING CONTOUR
- 1692.0 SPOT ELEVATION
- SECTION LINE
- 1725 PROPOSED CONTOUR (FINAL GRADE)
- 3:1 PROPOSED SLOPE RATIO (TYP.)
- 2% PROPOSED SLOPE (TYP.)
- PROPOSED BERM
- PROPOSED SPILLWAY
- PROPOSED DRAINAGE OUTLET
- 0.5% FLOW DIRECTION

NOTES:

1. TOPOGRAPHIC BASE MAP DIGITIZED FROM 1"=1000' SCALE, 5' CONTOUR INTERVAL MAP PREPARED BY AERO-METRIC ENGINEERING, INC., SHEBOYGAN, WISCONSIN. DATE OF PHOTOGRAPHY APRIL 28, 1976.
2. HORIZONTAL DATUM BASED ON WISCONSIN STATE PLANE COORDINATE SYSTEM - NORTH ZONE.
3. VERTICAL DATUM BASED ON MEAN SEA LEVEL DATUM. CONTOUR INTERVAL IS FIVE FEET.
4. COUNTY AND TOWNSHIP LINES DIGITIZED FROM 7.5' SERIES USGS MAPS.
5. THIS DRAWING SHOWS CONTOURS DEPICTING SITE FINAL GRADES. (TOP OF FINAL COVER).
6. FOR EROSION CONTROL REFER TO THE CMC MPA, 1995 SUBMITTAL.

SCALE
0 250' 500'

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

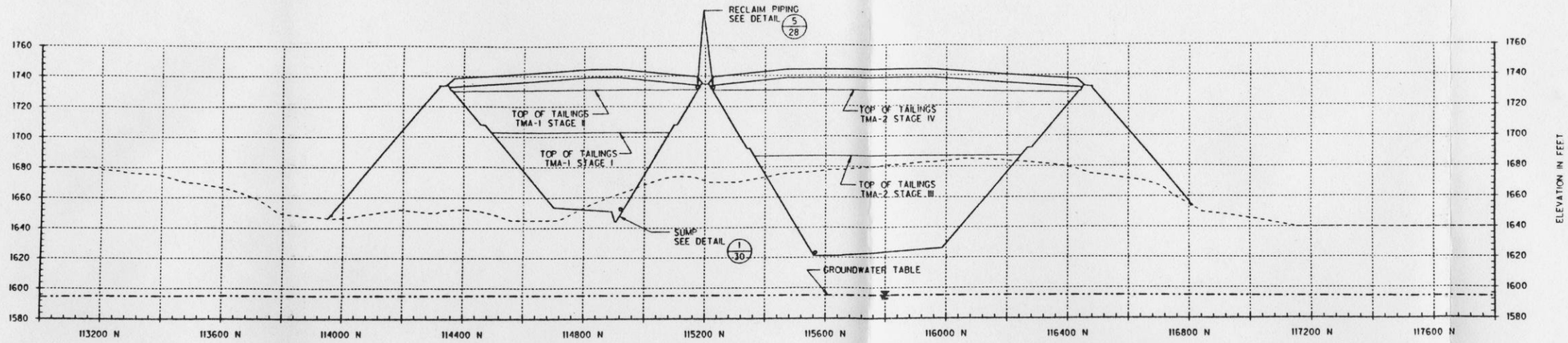
TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

Crandon Mining Company

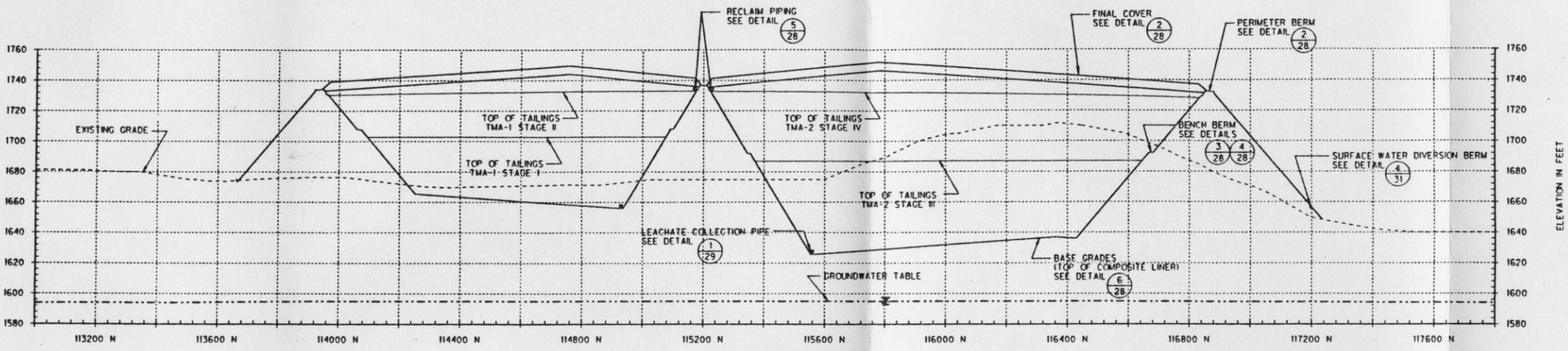
TITLE
DRAINAGE PLAN

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	RANDOM MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO. 22	SHEET 22 OF 31	REVISION NO. 0			

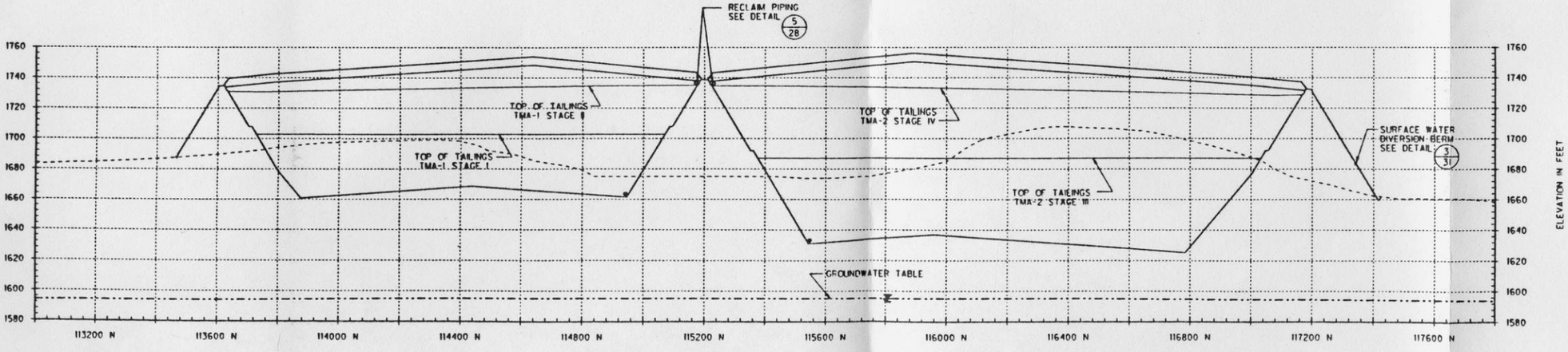
© 1995 Foth & Van Dyke and Associates, Inc.



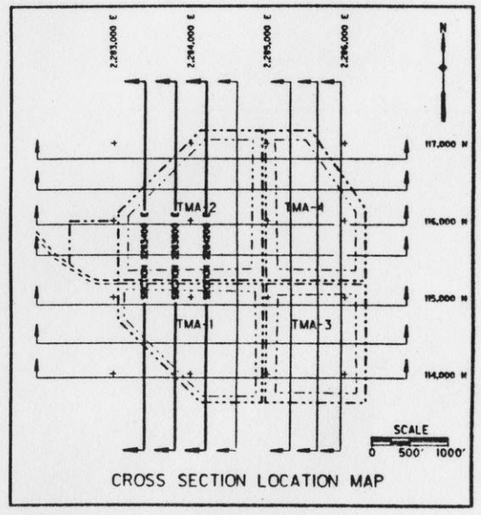
SECTION 2283400 E



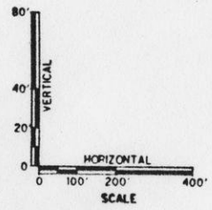
SECTION 2283800 E



SECTION 2284200 E



CROSS SECTION LOCATION MAP



Crandon Mining Company

Foth & Van Dyke

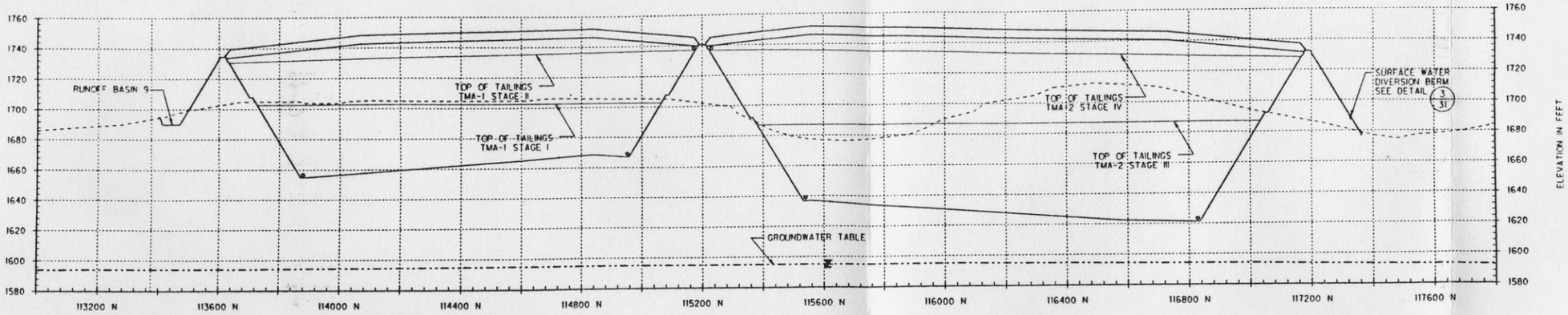
REVISED	DATE	BY	DESCRIPTION

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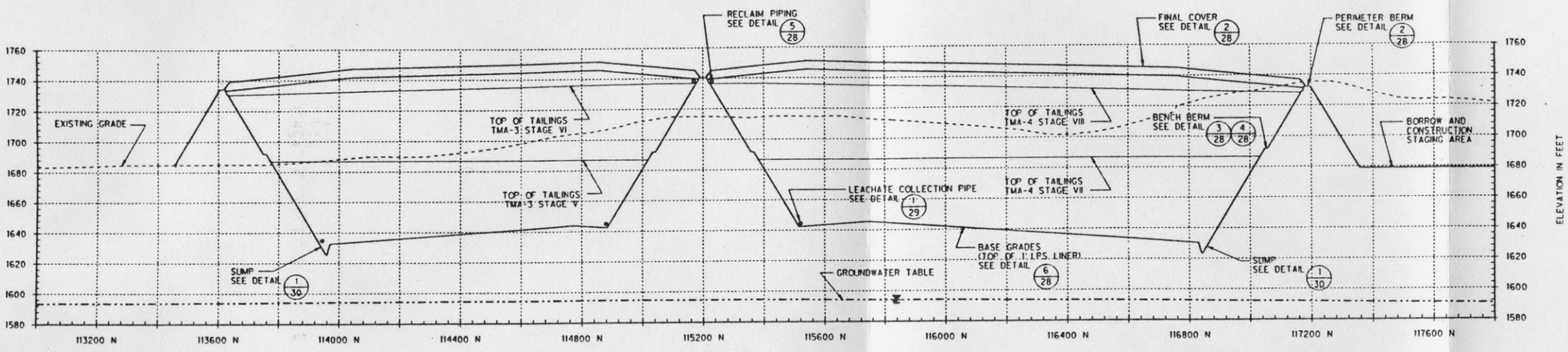
TITLE: ENGINEERING CROSS SECTIONS

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO.: 23		SHEET: 23 OF 31	REVISION NO.: 0		

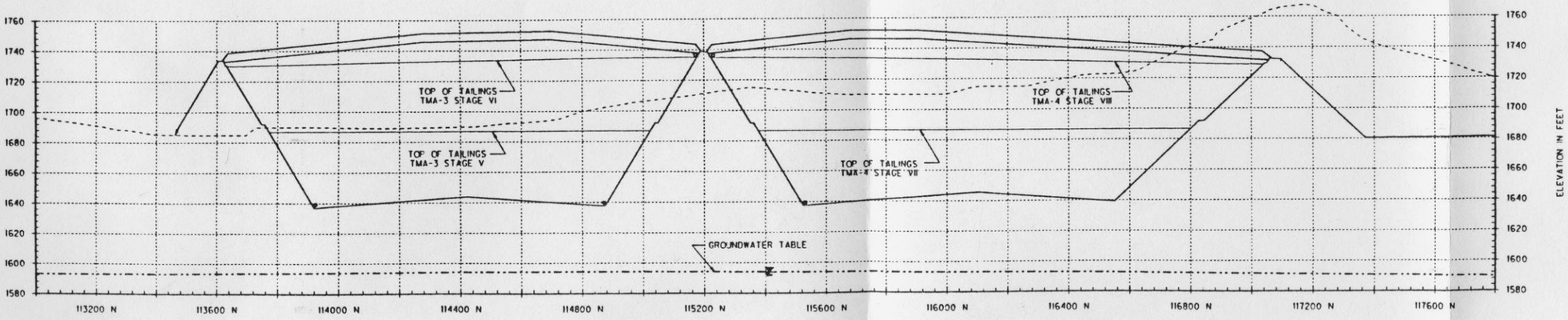
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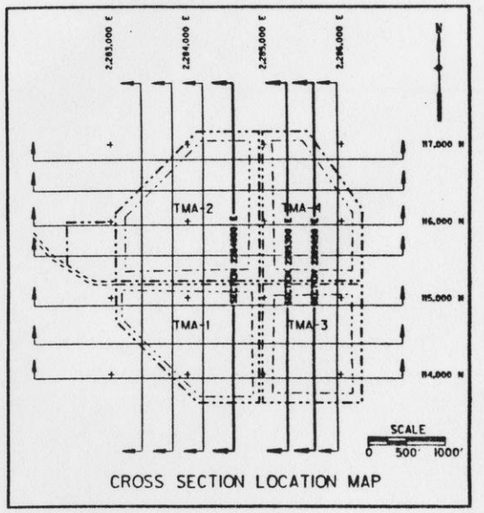
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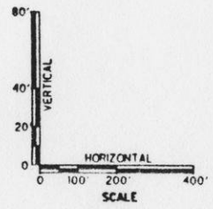
SECTION 2285300 E



SECTION 2285650 E



CROSS SECTION LOCATION MAP



Crandon Mining Company

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

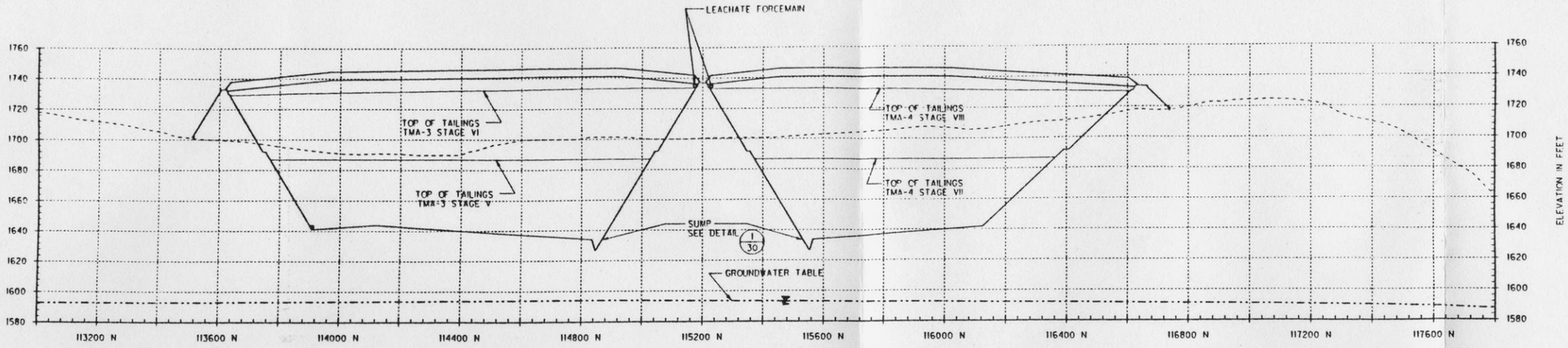
TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

TITLE

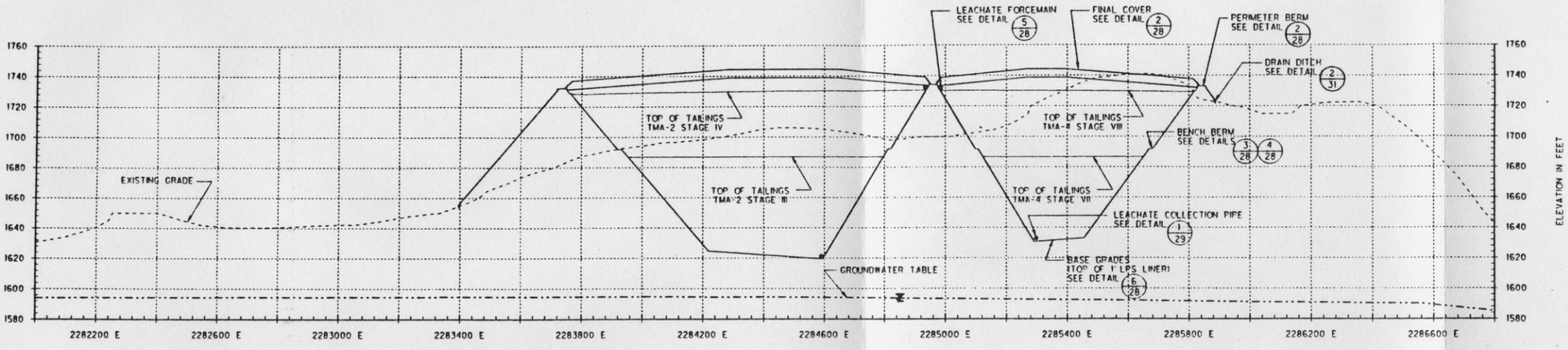
ENGINEERING CROSS SECTIONS

SCALE: AS SHOWN	STATE: WISCONSIN		COUNTY: FOREST		
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKSI	DATE: 5/95		
APPROVED BY: PEM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	FOR CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO. 24		SHEET 24 OF 31	REVISION NO. 0		

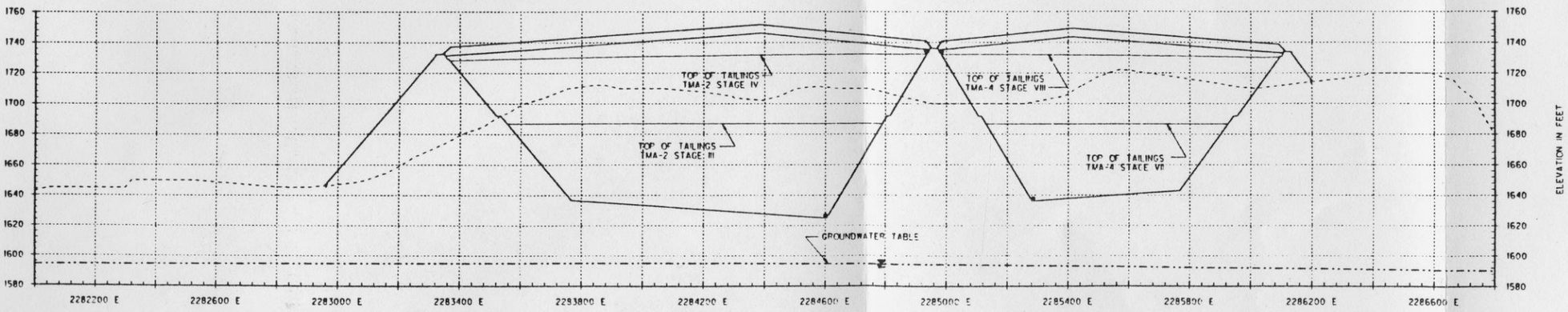
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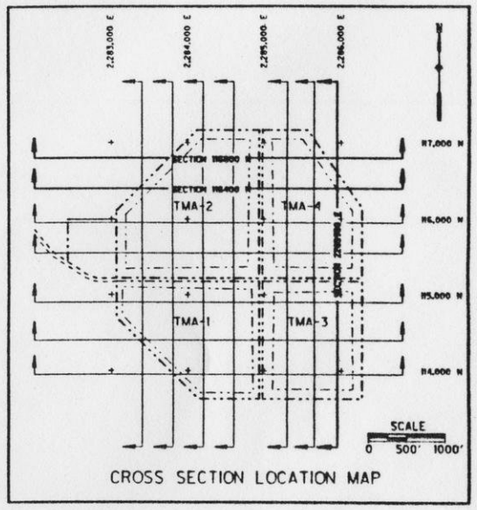
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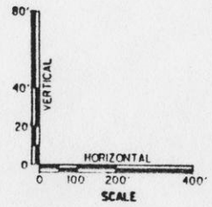
SECTION 116800 N



SECTION 116400 N



CROSS SECTION LOCATION MAP



Crandon Mining Company

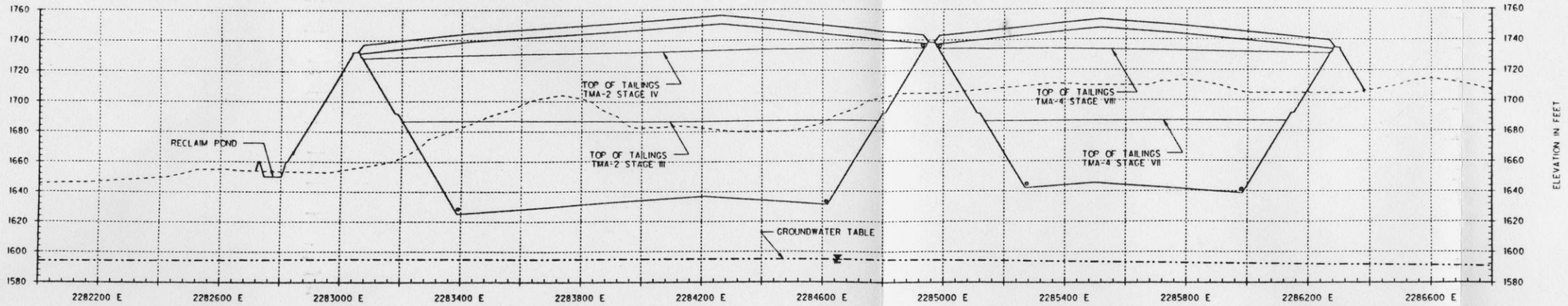
Foth & Van Dyke

NO.	DATE	BY	DESCRIPTION

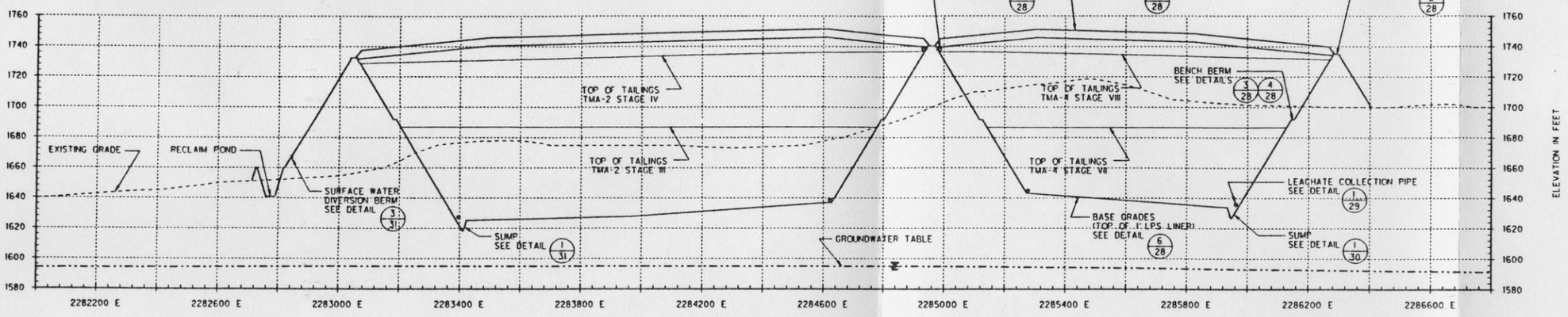
TYPICAL REPRESENTATION. REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

TITLE: ENGINEERING CROSS SECTIONS

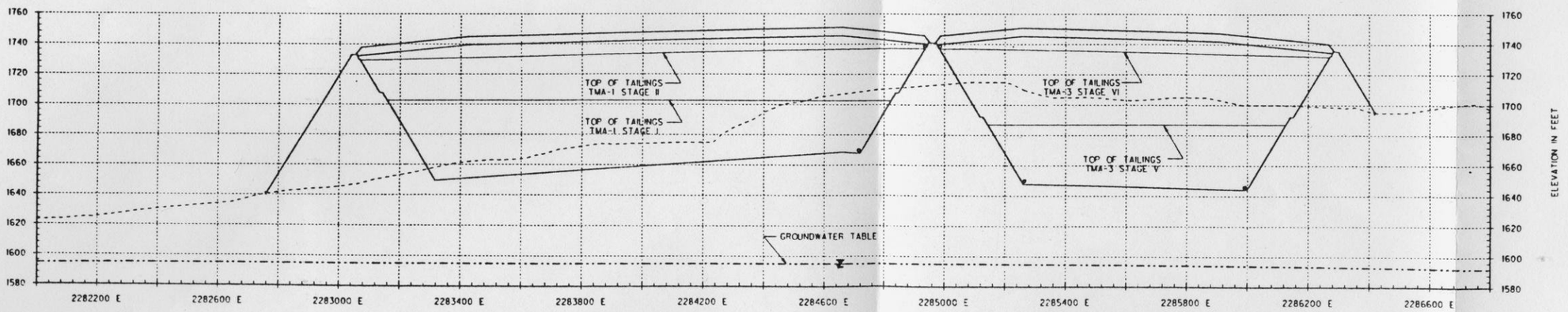
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DRAWN BY: GAM, JPR	DATE: 5/95	CHECKED BY: JFS	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: DWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO. 25		SHEET NO. 25			



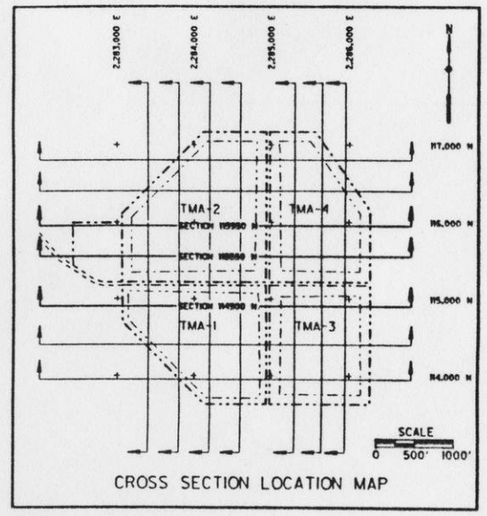
SECTION 115950 N



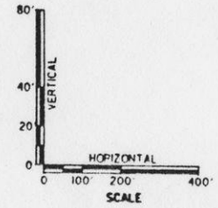
SECTION 115550 N



SECTION 114900 N



CROSS SECTION LOCATION MAP



Foth & Van Dyke

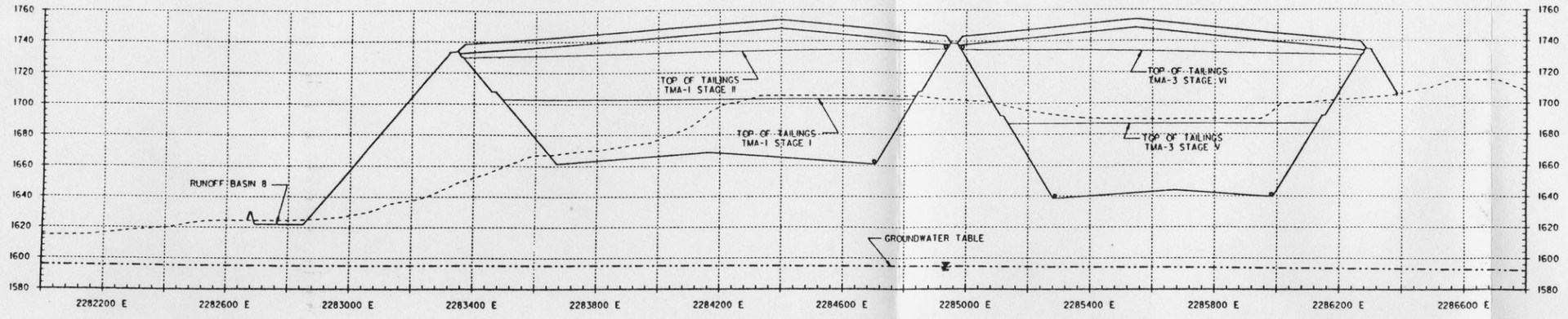
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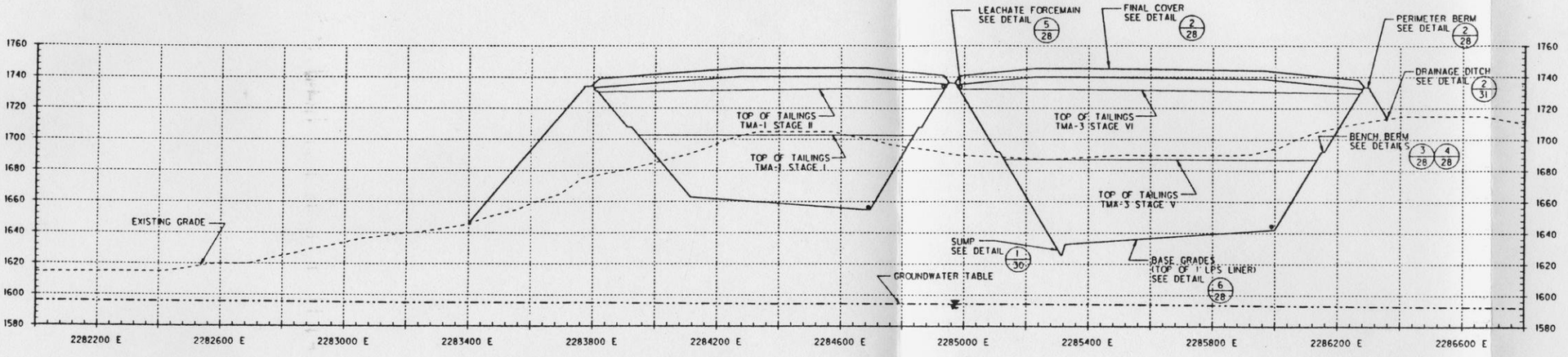
Crandon Mining Company

TITLE: ENGINEERING CROSS SECTIONS

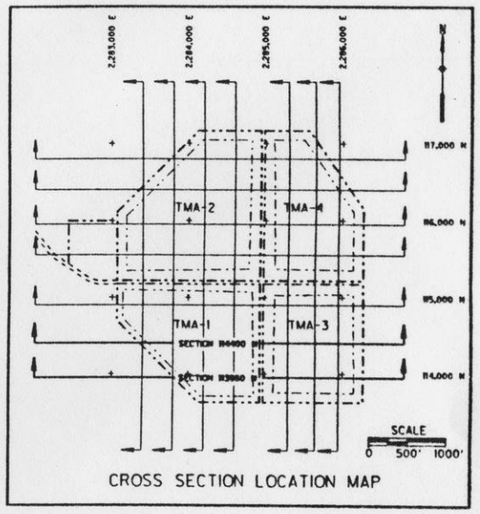
SCALE: AS SHOWN	STATE: WISCONSIN	COUNTY: FOREST	
DRAWN BY: GAM, JPR	DATE: 5/95	CHECKED BY: JKSJ	DATE: 5/95
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95
APPROVED BY: GWS	DATE: 5/95	BRANDON MINING COMPANY: DEM	DATE: 5/95
DRAWING NO.: 26		SHEET 26 OF 31	REVISION NO. C



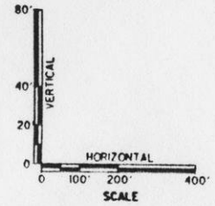
SECTION 114400 N



SECTION 113950 N



CROSS SECTION LOCATION MAP



Foth & Van Dyke

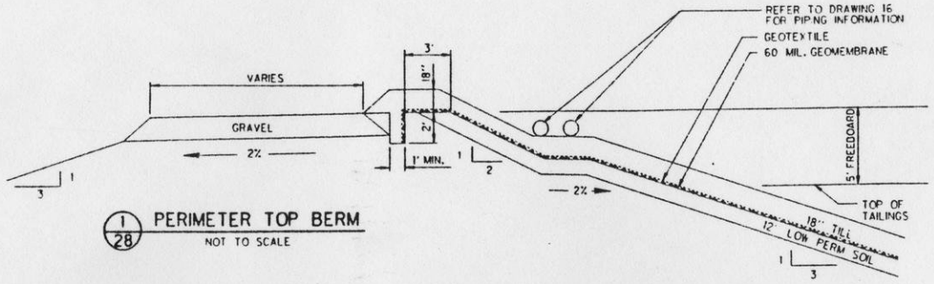
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TYPICAL REPRESENTATION. REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION.

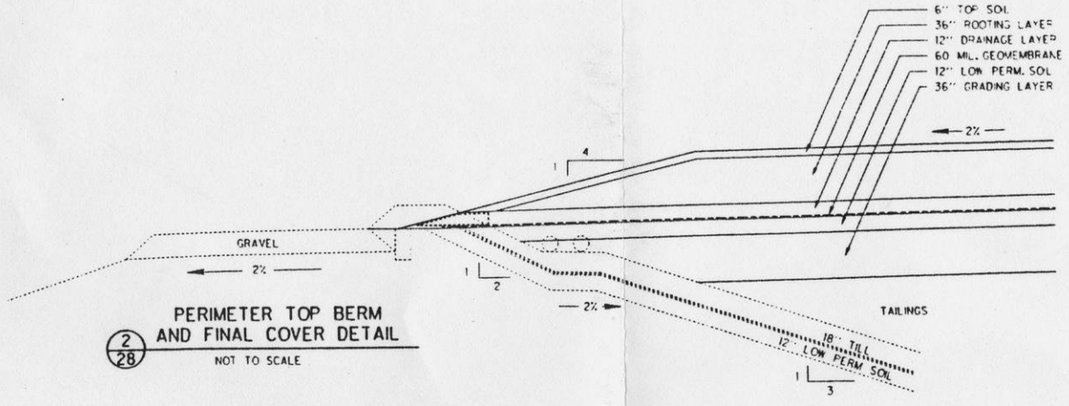
Crandon Mining Company

TITLE
ENGINEERING CROSS SECTIONS

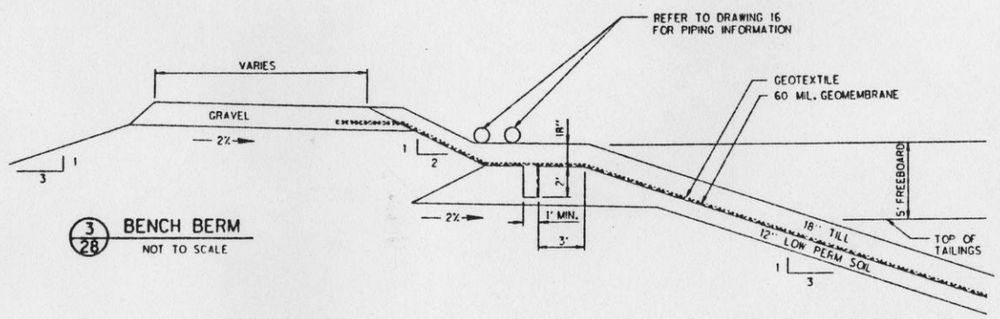
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DRAWING NO: 27		SHEET: 27 OF 31	REVISION NO: 0		



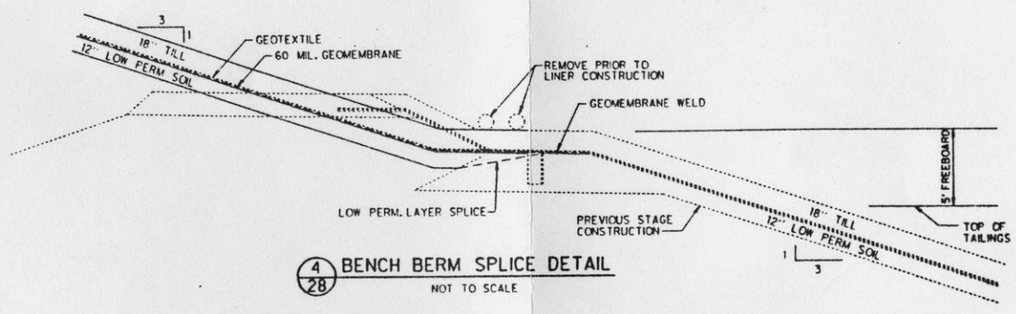
1/28 PERIMETER TOP BERM
NOT TO SCALE



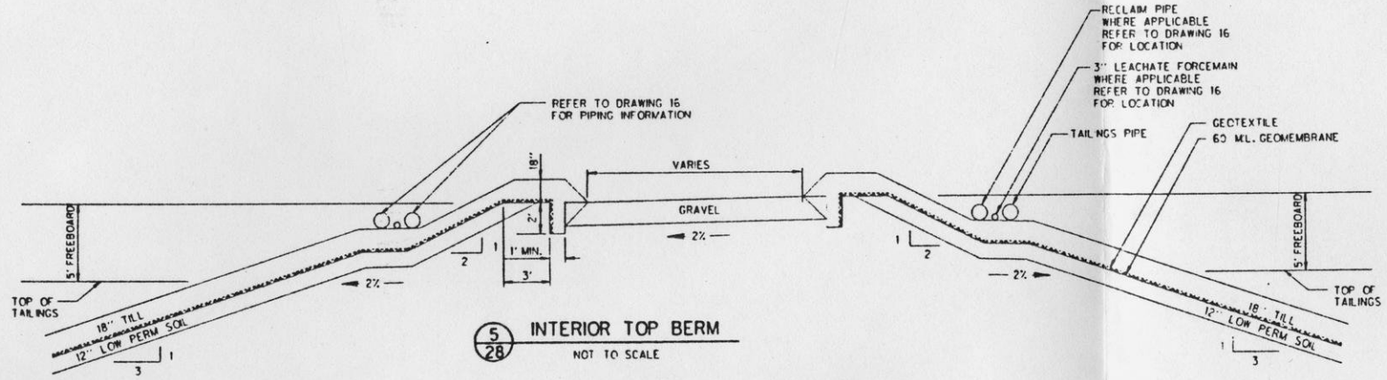
2/28 PERIMETER TOP BERM AND FINAL COVER DETAIL
NOT TO SCALE



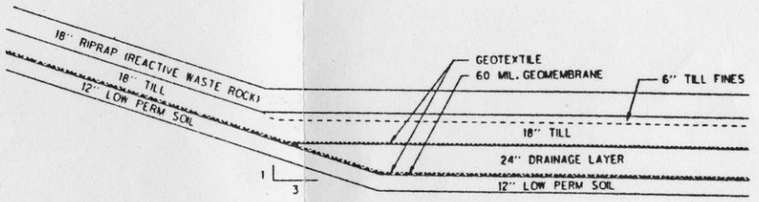
3/28 BENCH BERM
NOT TO SCALE



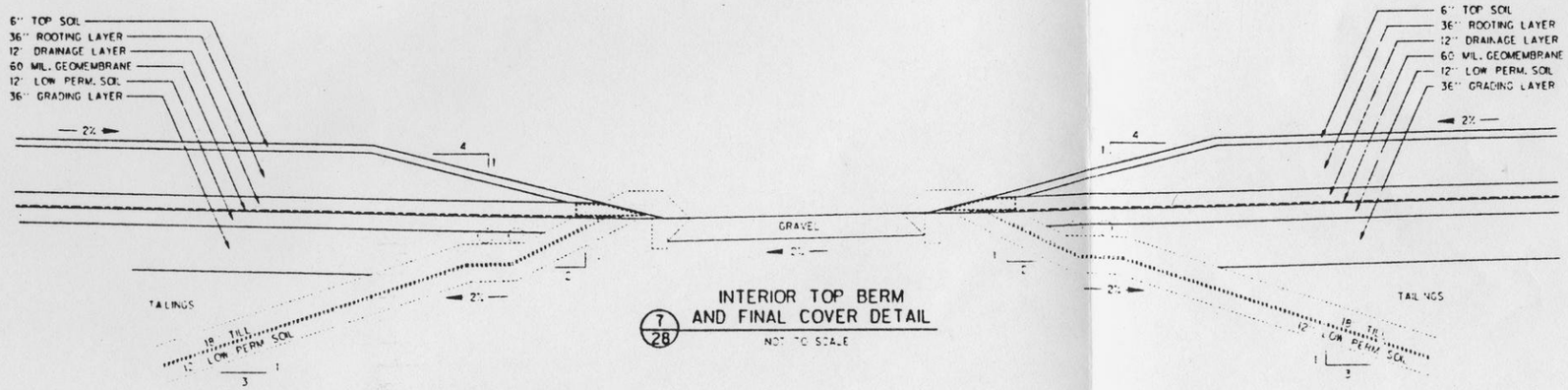
4/28 BENCH BERM SPLICE DETAIL
NOT TO SCALE



5/28 INTERIOR TOP BERM
NOT TO SCALE



6/28 TOE OF BERM W/O DRAINAGE LAYER
NOT TO SCALE



7/28 INTERIOR TOP BERM AND FINAL COVER DETAIL
NOT TO SCALE

Foth & Van Dyke

REVISIONS	DATE	BY	DESCRIPTION

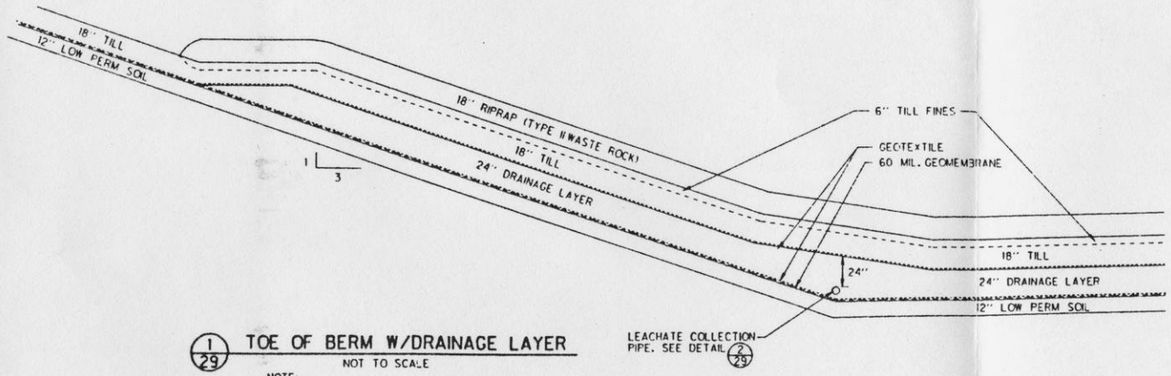
TYPICAL REPRESENTATION; REFINEMENTS MAY BE MADE PRIOR TO CONSTRUCTION

Crandon Mining Company

TITLE: DETAILS

SCALE: AS SHOWN	STATE: WISCONSIN			COUNTY: FOREST	
DRAWN BY: GAM, JPRI	DATE: 5/95	CHECKED BY: JKS!	DATE: 5/95		
APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95		
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO.: 28		SHEET: 28 OF 30	REVISION NO.: 0		

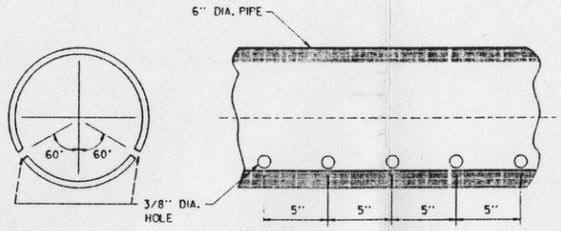
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1/29 TOE OF BERM W/DRAINAGE LAYER

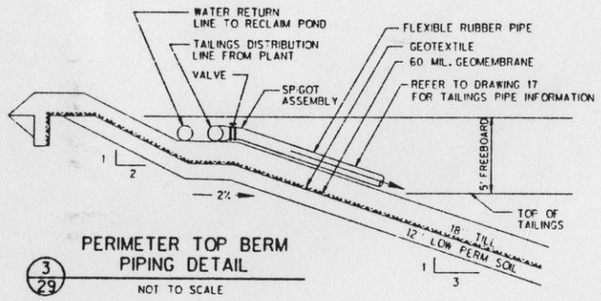
NOT TO SCALE

NOTE:
DRAINAGE LAYER ELEVATION CORRESPONDS TO THE ELEVATION OF THE HIGH POINT IN THE TMA CELL BOTTOM.



2/29 LEACHATE COLLECTION PIPE PERFORATION DETAIL

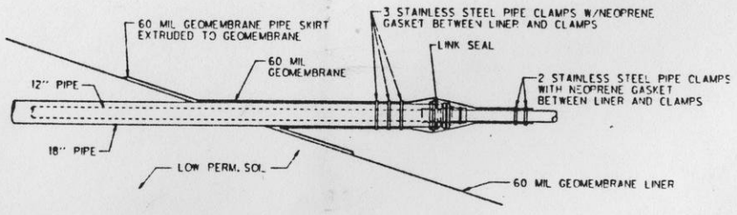
NOT TO SCALE



3/29 PERIMETER TOP BERM PIPING DETAIL

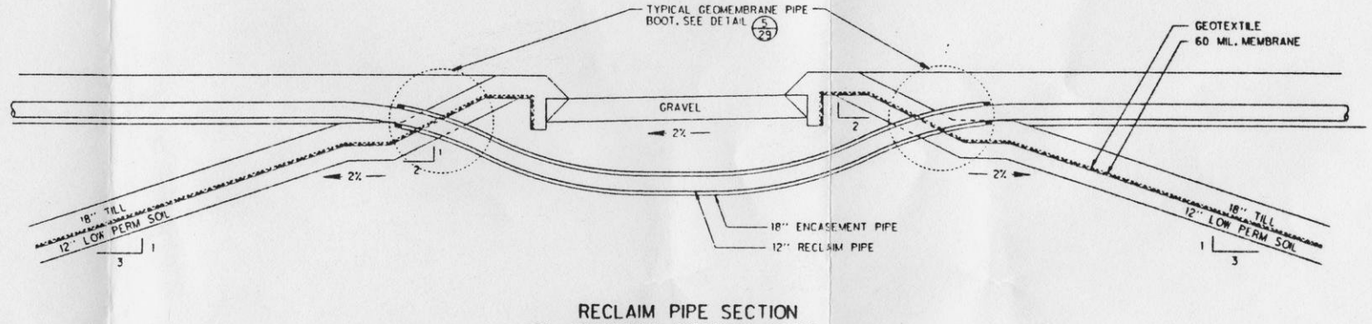
NOT TO SCALE

NOTE:
SPIGOT PIPE EXTENDS TO THE TOP OF THE TAILINGS AND IS CUT OFF OR EXTENDED AS REQUIRED TO DISCHARGE TAILINGS ON TO THE TAILINGS BEACHES.



5/29 GEOMEMBRANE PIPE BOOT DETAIL

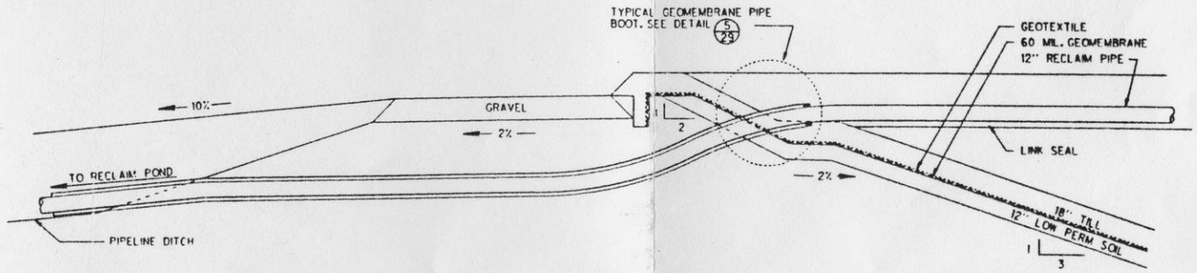
NOT TO SCALE



RECLAIM PIPE SECTION

4/29 RECLAIM PIPE SECTION CROSSING INTERIOR BERM

NOT TO SCALE



6/29 RECLAIM PIPE SECTION AT PERIMETER BERM

NOT TO SCALE

Foth & Van Dyke

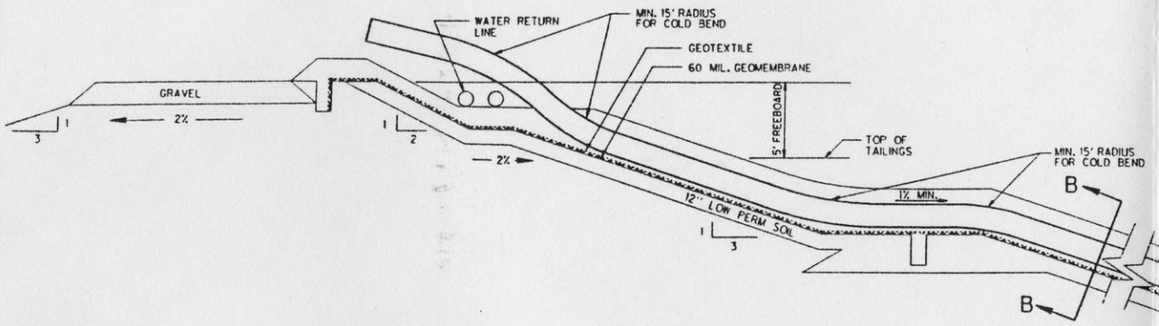
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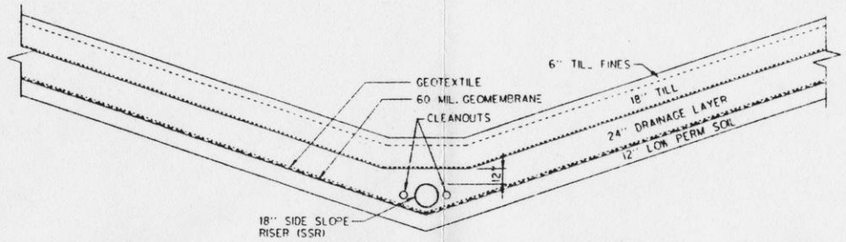
Crandon Mining Company

TITLE: DETAILS

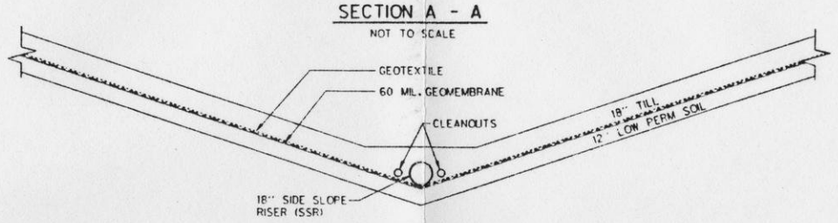
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APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95		
DRAWING NO: 29		SHEET: 29 OF 31	DIVISION OF: C		



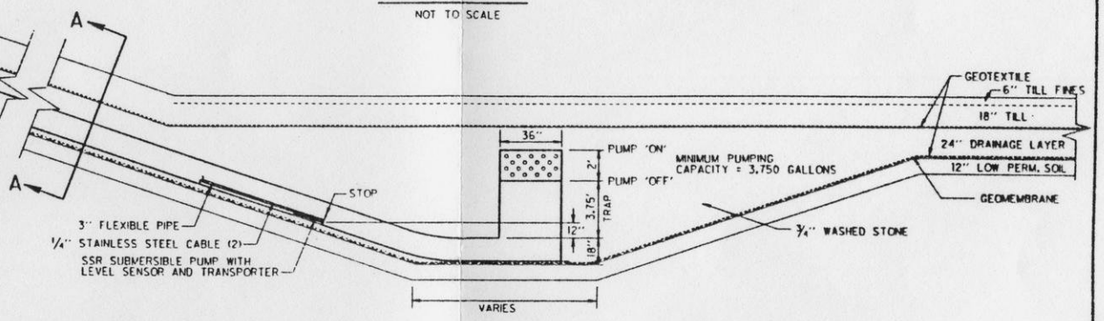
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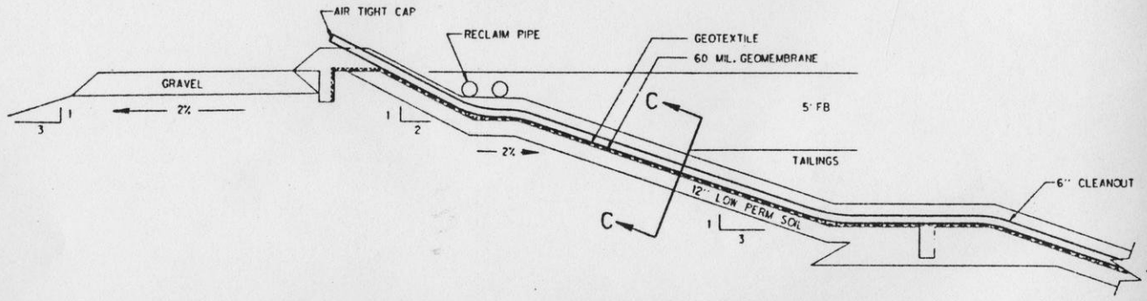


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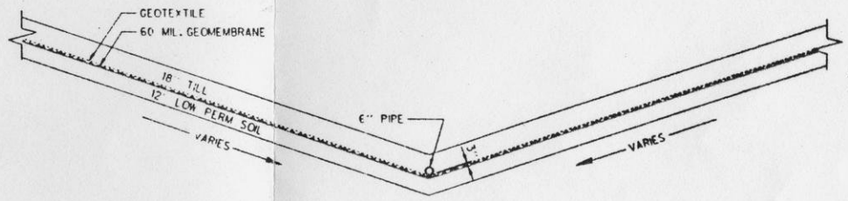


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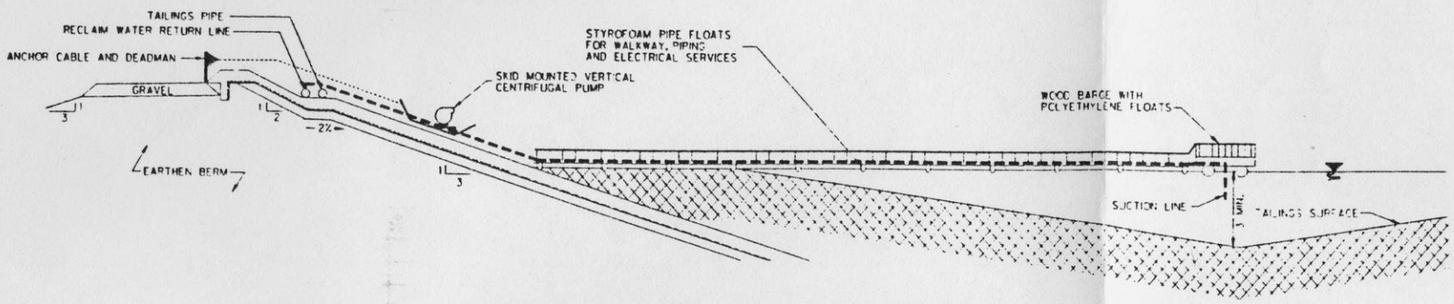




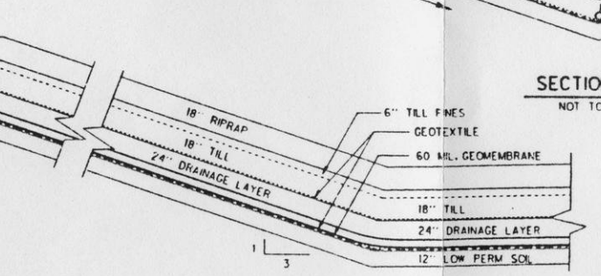
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NOT TO SCALE



SECTION C - C
NOT TO SCALE



3/30 TYPICAL DECANT STRUCTURE DETAIL
NOT TO SCALE



Crandon Mining Company

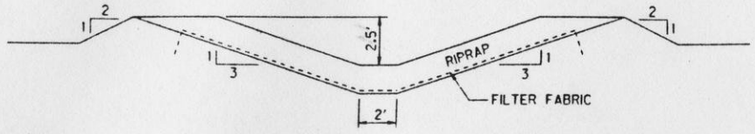
Foth & Van Dyke

REVISIONS	DATE	BY	DESCRIPTION

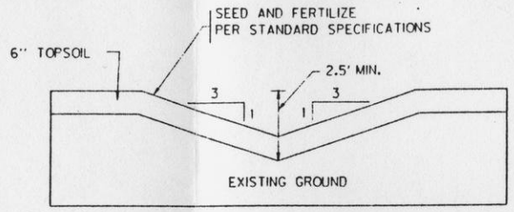
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TITLE: DETAILS

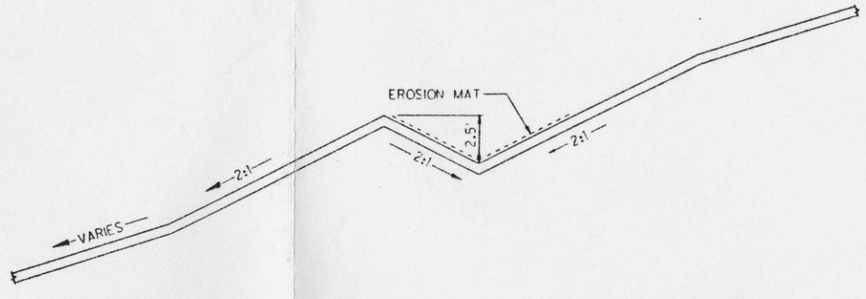
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APPROVED BY: REM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95
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DRAWING NO.: 30	SHEET: 30 OF 31	REVISION NO.: 0	



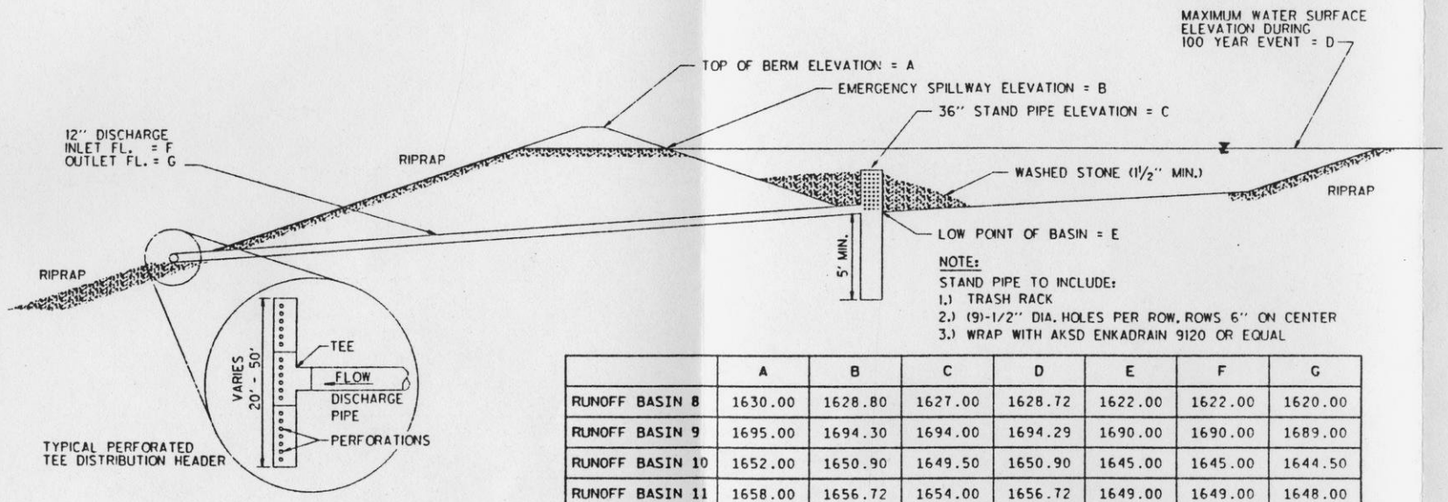
(1/31) TYPICAL RIPRAPPED SPILLWAY
NOT TO SCALE



(2/31) DRAINAGE DITCH DETAIL "V" NOTCH
NOT TO SCALE



(3/31) TYPICAL SURFACE WATER DIVERSION BERM
NOT TO SCALE



NOTE:
STAND PIPE TO INCLUDE:
1.) TRASH RACK
2.) (9)-1/2" DIA. HOLES PER ROW, ROWS 6" ON CENTER
3.) WRAP WITH AKSD ENKADRAIN 9120 OR EQUAL

	A	B	C	D	E	F	G
RUNOFF BASIN 8	1630.00	1628.80	1627.00	1628.72	1622.00	1622.00	1620.00
RUNOFF BASIN 9	1695.00	1694.30	1694.00	1694.29	1690.00	1690.00	1689.00
RUNOFF BASIN 10	1652.00	1650.90	1649.50	1650.90	1645.00	1645.00	1644.50
RUNOFF BASIN 11	1658.00	1656.72	1654.00	1656.72	1649.00	1649.00	1648.00
RUNOFF BASIN 12	1702.00	1701.10	1700.50	1701.04	1696.00	1696.00	1695.50

(4/31) TYPICAL SECTION - RUNOFF BASIN
NOT TO SCALE

Foth & Van Dyke

REVISE:	DATE	BY	DESCRIPTION

TYPICAL REPRESENTATION:
REFINEMENTS MAY BE MADE
PRIOR TO CONSTRUCTION.

Crandon Mining Company

TITLE
DETAILS

SCALE	STATE: WISCONSIN	COUNTY: FOREST	
DRAWN BY: GAM, JPR	DATE: 5/95	CHECKED BY: JWSI	DATE: 5/95
APPROVED BY: PEM	DATE: 5/95	APPROVED BY: NXP	DATE: 5/95
APPROVED BY: GWS	DATE: 5/95	CRANDON MINING COMPANY: DEM	DATE: 5/95
DRAWING NO. 31		SHEET 31 OF 31	REVISION NO. 0

Appendix A

Mine Waste Volume Calculation

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 1/5
Prepared by: REM Date: 12/27/94
Checked by: NxP Date: 12/30/94

CALculate Tailings Quantity in GROSS TERMS

GIVEN | Assumptions

1. TOTAL ORE production ~ 56×10^6 TONS for 28 years
2. 40% of the total ore production is tailings
3. average tailing density ~ 97 PCF for Zn and Cu tailings assuming subaerial tailings disposal.
4. Zn ore ~ 32×10^6 Tons (57%) AND Cu ore ~ 24×10^6 TONS (43%) = 56×10^6 TONS (TOTAL ore production)

CALculate TOTAL Volume and Tons OF ALL TAILINGS

TONS

56×10^6 TONS x .40 = 22.4×10^6 TONS for 28 YRS

CY

22.4×10^6 TONS x $\frac{2000 \text{ lb}}{\text{TON}}$ x $\frac{1 \text{ CF}}{97 \text{ lbs}}$ x $\frac{1 \text{ cy}}{27 \text{ CF}}$ = 1.7106×10^7 CY

SAY 17.11×10^6 CY

CALculate Quantity and Volume OF ZN and Cu Tailings

ZN Tailings for 16 YRS

TONS 22.4×10^6 TONS x .57 = 1.2768×10^7 SAY 12.77×10^6 TONS

CY 17.11×10^6 CY x .57 = 9.7527×10^6 SAY 9.75×10^6 CY

ANNUAL TONS AND CY FOR ZN

TONS

$\frac{12.77 \times 10^6 \text{ TONS}}{16 \text{ YRS}}$ = 7.9813×10^5 SAY $.80 \times 10^6$ TONS/YR

CY

$\frac{9.75 \times 10^6 \text{ CY}}{16 \text{ YRS}}$ = 6.0938×10^5 SAY $.61 \times 10^6$ CY/YR

With 15% Contingencies

9.75×10^6 CY x .15 = 1.4625×10^6 + 9.75×10^6 CY = 1.1213×10^7

SAY 11.21×10^6 CY

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 2/5
Prepared by: REM Date: 12/27/94
Checked by: N×P Date: 12/30/94

Calculate Quantity and Volume of Cu Tailings

Cu Tailings for 12yrs

Tons

22.40×10^6 Tons $\times .43 = 9.632$ say 9.63×10^6 Tons

Cy

17.11×10^6 cy $\times .43 = 7.3573$ say 7.36×10^6 cy

Calculate Annual Cy and tons

Tons

$\frac{9.36 \times 10^6 \text{ Tons}}{12 \text{ yrs}} = 8.025$ say $.80 \times 10^6$ Tons/yr

Cy

$\frac{7.36 \times 10^6 \text{ cy}}{12 \text{ yrs}} = 6.1333$ say $.61 \times 10^6$ cy/yr

Calculate Total Cu w/ 15% Contingency

$7.36 \times 10^6 \times .15 = 1.104$ say 1.1×10^6 cy $+ 7.36 \times 10^6$ cy $=$ 8.46×10^6 cy

Total Zn and Cu Tailings Volume

w/o Contingencies = (Zn) 9.75×10^6 + (Cu) 7.36×10^6 = 17.11×10^6 cy

w/ Contingencies = (Zn) 11.21×10^6 + (Cu) 8.46×10^6 = 19.67×10^6 cy

Foth & Van Dyke

Client: CMC — Scope I.D.: 93C049
Project: TMA — Page: 3/5
Prepared by: REM — Date: 12/21/94
Checked by: N x P — Date: 12/30/94

TMA WASTE CHARACTERISTICS AND Quantities Summary

ORE	WASTE DESCRIPTION	DENSITY (PCF)	(Annual) x 10^6 TONS	(Annual) x 10^6 CY	TOTAL Sto (x 10^6) TONS	TOTAL Sto (x 10^6) CY
ZINC						
	TAILINGS	97	.80	.64	12.77	9.75
	WEST VENT RAISE ROCK	111.3 (loose)	—	—	.05	.03
	High Sulfur Waste Rock	111.3 (loose)	—	—	.10	.07
	WTP Sludge (51%)*	47.2	.02	.03	.26	.40
	TOTALS				13.18	10.25
	15% Contingency (Tailings cy only*)					1.46
	TOTAL ZN w/ Contingency					11.71
COPPER						
	Tailings	93 PCF	.80	.64	7.67	7.36
	**WTP Sludge (43%)	47.2			.2	.30
	OTHER WASTE ***					.11
	TOTAL				7.87	7.77
	15% Contingency (Tailings cy only*)					1.10
	TOTAL Cu w/ Contingency (Tailings only)					8.87
	TOTAL Tons w/o Contingency				30.75	

TOTAL TMA Volume (w/ contingencies and other wastes)

ZN = 11.71 x 10^6 CY

CU = 8.87 x 10^6 CY

TOTAL = 20.58 x 10^6 CY

** WTP Sludge
Assume proportion to TOTAL Tailings ie ZN = 57%, Cu = 43%.

*** Assume "other wastes" are proportional to tailings for Cu

.15/9.75 = x/7.36
x = .11323
avg = .11/4 = .0275 avg .03

* Tailings Contingencies
ZN = 9.75 x .15 = 1.46 x 10^6 CY
Cu = 7.36 x .15 = 1.10 x 10^6 CY

Foth & Van Dyke

Client: CMC — Scope I.D.: 93C049
Project: TMA — Page: 4/5
Prepared by: REM — Date: 12/28/94
Checked by: N.P — Date: 12/30/94

TMA Preliminary Sizing of Cells and Phases

Given / Assumptions & Revised 1/11/94 due to change to Sub aerial deposition

1. 4 cells for Zinc and 4 cells for Copper
2. 2 phases (stages of construction) per cell ∴ 8 phases Total
3. Contingency of 15% added to the final phase of Zn/Cu cells
4. γ for Zn and Cu = 97 pcf
5. Tailings generation rate equal to 40% of mining rate, (5,500 Tons/day) 360 days per year for 56×10^6 Ton ore body
6. TMA Cell and phase life as follows:

Zinc — TMA 1 - Total Life 7 yrs - Phase I - 3 yrs / Phase II - 4 yrs

TMA 2 - Total Life 9 yrs - Phase III - 3.5 yrs / Phase IV - 5.5 yrs + 15% Contingency

16 yrs

Copper — TMA 3 - Total Life - 5 yrs - Phase V - 2.0 yrs / Phase VI - 3.0 yrs

TMA 4 - Total Life - 7 yrs - Phase VII - 3.0 yrs / Phase VIII - 4.0 yrs + 15% Contingency

12 yrs

(Check Annual Tailings production)

Calculate Annual production for tailings for Zn and Cu (check of pg 1/T and 2/T)

Zinc (Tailings)

$$56 \times 10^6 \text{ Tons} \times .40 \times \frac{1}{16 \text{ yrs}} \times .57 (Zn) = 7.98 \times 10^5 \frac{\text{Tons}}{\text{yr}}$$

$$7.98 \times 10^5 \frac{\text{Tons}}{\text{yr}} \times \frac{2000 \text{ lb}}{\text{Ton}} \times \frac{1 \text{ CF}}{97 \text{ lbs}} \times \frac{1 \text{ cy}}{27 \text{ CF}} = 6.093929 \times 10^5 \text{ say } .61 \times 10^6 \text{ cy/yr}$$

Copper

$$56 \times 10^6 \text{ Tons} \times .40 \times \frac{1}{12 \text{ yrs}} \times .43 = 8.0267 \times 10^5 \text{ Tons/yr}$$

$$8.0267 \times 10^5 \frac{\text{Tons}}{\text{yr}} \times \frac{2000 \text{ lb}}{\text{Ton}} \times \frac{1 \text{ CF}}{97 \text{ lbs}} \times \frac{1 \text{ cy}}{27 \text{ CF}} = 6.1296 \times 10^5 \text{ say } .61 \times 10^6 \text{ cy/yr}$$

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 5/5
Prepared by: REM Date: 12/28/94
Checked by: N x P Date: 12/30/94

Calculate Cy's for ~~TMA~~ Zinc 1 and 2, Phases I, II, III and IV

Phase I - 3yrs x .61 x 10^6 cy/yr = 1.83 x 10^6 cy

Phase II - 4yrs x .61 x 10^6 cy/yr = 2.44 x 10^6 cy

Phase III - 3.5yrs x .61 x 10^6 cy/yr = 2.14 x 10^6 cy

Phase IV - 5.5yrs x .61 x 10^6 cy/yr = 3.36 x 10^6 cy

(w/o contingency) SUB TOTAL 9.77 ← S/B 9.75 diff due to rounding

+15% Contingency (pg 1) 1.46

TOTAL Zn Tailings → 11.23

Phase IV then is 3.36 + 1.46 = 4.82 x 10^6 cy

Copper Calculate Cy for TMA 3 and 4, Phases V, VI, VII and VIII

Phase V - 2yrs x .61 x 10^6 cy/yr = 1.22 x 10^6 cy

Phase VI - 3yrs x .61 x 10^6 cy/yr = 1.83 x 10^6 cy

Phase VII - 3yrs x .61 x 10^6 cy/yr = 1.83 x 10^6 cy

Phase VIII - 4yrs x .61 x 10^6 cy/yr = 2.44 x 10^6 cy

w/o contingency subtotal 7.32 ← S/B 7.36 diff due to rounding

(+15% Contingency pg 2) 1.10

TOTAL Cu Tailings → 8.42

Phase VIII then is 2.44 + 1.10 = 3.54 x 10^6 cy

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: TMA	Page: 1/2
	Prepared by: REM	Date: 12/20/94
	Checked by: N>P	Date: 5-25-95

CALCULATIONS FOR TMA - "OTHER WASTE" QUANTITIES

"OTHER WASTES" ASSUME include the following:

1. • Rock from WEST Vent raise CONSTRUCTION ASSUME:
   - 18' ϕ, 2000' length, in-situ rock γ = 2.84 g/cc (p. 13, 1986 Exxon Design Criteria) (2.84 g/cc = 177 pcf)
2. • High Sulfur WASTE rock
   - estimate 100,000 TONS
   - γ = 177 pcf
   - F. waste rock @ ~24" ϕ with loose γ = 111.3 pcf (1985 FR p1-22)
3. • SLudge from WASTEwater Treatment Plant
   - γ = 47.2 lbs/CF (FR, p 1-22)
   - estimate (conservative) - 700,000 T over life of mine
   - mine site life of 28 years

4. • LABORATORY WASTES
   - Assume that this quantity is very small and therefore there is no need to design the TMA for any additional capacity.

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 2 | 2
Prepared by: REM Date: 12/20/94
Checked by: N.P Date: 5-25-95

DETERMINE OTHER WASTE Quantities AND Volumes

Quantity

① Calculate West Vent Raise Wastes

$\frac{\pi}{4} \times (18\text{ ft})^2 \times 2000\text{ ft} \times 177\frac{\text{lbs}}{\text{CF}} \times \frac{1\text{ TON}}{2000\text{ lbs}} \times (1\times10^{-6}) = .045\text{ m Tons}$

$.045 \times 10^6$ Tons

Volume (Loose) 45000

$.045\times10^6\text{ Tons} \times \frac{2000\text{ lbs}}{\text{Ton}} \times \frac{1\text{ CF}}{111.3\text{ lbs}} \times \frac{1\text{ cy}}{27\text{ CF}} = 2.9949\times10^4$ say 30,000 cy or $.03\times10^6$ cy

② Calculate Quantity and Volume of High Sulfur Waste Rock

Assume TOTAL Quantity = 100,000 Tons

Calculate Volume

$100{,}000\text{ Tons} \times \frac{2000\text{ lbs}}{\text{Ton}} \times \frac{1\text{ CF}}{111.3\text{ lbs}} \times \frac{1\text{ cy}}{27\text{ CF}} = 66553.526$ say 67,000 cy use $.07\times10^6$ cy

③ Calculate Quantity and Volume of WTP Sludge

ASSUME TOTAL sludge generation over 28 years = 700,000 cy $.7\times10^6$ cy which is very conservative

Annual quantity = $\frac{700{,}000}{28\text{ yrs}} = 25{,}000$ cy/yr

Determine Quantity

$700{,}000\text{ cy} \times \frac{27\text{ CF}}{1\text{ cy}} \times \frac{47.2\text{ lbs}}{\text{CF}} \times \frac{1\text{ TON}}{2000\text{ lbs}} = 446{,}040$ Tons say $.45\times10^6$ Tons

Annual Tonnage = $\frac{450{,}000\text{ Tons}}{28\text{ yrs}} = 16071$ say 16,100 Tons/yr

27000 TONS x 2000 lbs x / TON

Appendix B

Memoranda From Steffens, Robertson, and Kirsten, USA Regarding Tailings Laboratory Physical Testing

STEFFEN ROBERTSON AND KIRSTEN
Consulting Engineers and Scientists

MEMORANDUM

TO: Ron Meister - Foth & Van Dyke

FROM: Rick Frechette, Rob Dorey - Steffen Robertson and Kirsten (SRK), U.S., Inc.

DATE: December 12, 1994

RE: **TAILINGS TESTING AND DENSITY PROJECTIONS FOR CRANDON PROJECT STEFFEN ROBERTSON AND KIRSTEN (SRK), U.S., INC. PROJECT NO. 71501**

1.0 INTRODUCTION

Testing of slimes and whole tailings samples submitted by Foth and Van Dyke (FVD) has been performed by Steffen Robertson and Kirsten (SRK), U.S., Inc. The purpose of the testing was to develop data for use in the preliminary design of a tailings management system for the proposed Crandon Project. The results are indicative of the possible range of densities that may be achieved under various operating conditions. An estimate of anticipated tailings densities under the assumed operating conditions is presented later in this memo. The samples tested and the data produced have also been evaluated in the context of the results of several previous studies. The results of this evaluation are presented at the end of this memorandum.

2.0 TEST PROGRAM

In addition to index testing, the testing included column settling, pan evaporation and one dimensional consolidation. The tailings samples consist of fine grained non-plastic sand to silt size particles. Grain size and specific gravity data and plasticity testing are summarized in Table 1.

TABLE 1
LABORATORY INDEX TESTING

	Grain Size -75μ	Grain Size -20μ	Specific Gravity (SG)	Plasticity
Slimes	100%	92%	3.32	Non-plastic
Whole	90%	47%	3.52	Non-plastic

Grain size analyses on tailings from several metallurgical tests (including Pb and Cu concentrates) were completed earlier by Lakefield Research. Grain size testing of the tailings samples by SRK appear to show that the samples are similar to those tested by Lakefield Research.

The tests for column and pan evaporation were conducted on a 45% solids slurry. SRK's test summary report is attached.

The column test simulates the settling of solids from a slurry and their initial consolidation under subsequent tailings placement. The results typically indicate the initial in-situ density achieved under sub-aqueous conditions. Two tests, one with drainage at the base of the column and one without drainage were performed.

The results for the column settling tests (Table 2) with regard to drained vs. undrained conditions show that the undrained values are approximately equal to the drained. This reflects behavior characteristic of low permeability material undergoing a relatively slow consolidation process. That is, the ability of the material to drain and consolidate is limited by the material's permeability.

The pan evaporation tests simulate the density which would be achieved at the point when the material reaches a moisture content where air entry to the interstitial space occurs. The results indicate the density which may occur under sub-aerial, beach conditions where drying takes place.

The consolidation tests simulate the consolidation of the tailings at depth within the facility. The test results indicate the potential densities which would exist under conditions where primary consolidation is complete and all excess pore water pressures are dissipated following the end of tailings deposition.

A summary of the laboratory testing results relative to density is given in Table 2.

TABLE 2
LABORATORY DENSITY RESULTS

Condition	Dry Unit Weight					
	Slimes			Whole		
	Dry Density pcf	Wet Density pcf	Ret. Moist.	Dry Density pcf	Wet Density pcf	Ret. Moist.
Drained Column Test	60-65	102-110	70%	80-85	120-129	50%
Undrained Column Test	60-65	102-110	70%	85-95	120-133	40%
Pan Evap. Test	80	116	45%	112	140	25%
Consolidation Test (30' load)	89	130	40%	124	157	22%
Consolidation Test (60' load)	92	127	38%	127	152	20%

3.0 DATA EVALUATION AND ANALYSIS

The above data provide the input for the analysis of conditions which may exist during and following deposition of tailings. The following is a preliminary analysis of these data designed to indicate the probable range of in-situ densities that may exist. Key operational data provided by FVD and used in the analysis include:

- life of project anticipated composition of tailings deposit - 80% slimes, 20% whole tailings;
- maximum rate of rise of tailings deposit - 10 to 12 ft/yr;
- maximum total tailings thickness of 85 ft;
- bottom drainage of the impoundment.

Based on the testing, the coefficient of consolidation, c_v, was estimated at 0.091 and 0.127 cm^2/sec for the slimes and whole tailings, respectively. Using these values as input to the Terzaghi one-dimensional consolidation model, a time to 90 percent consolidation of approximately 6 months is indicated for the mixed tailings, assuming an 85-ft maximum tailings deposit thickness. This indicates that the majority of settlement of the tailings surface would occur within one year of the end of operations. During operations, however, the continuous deposition of tailings results in a progressive increase in loading of the buried tailings which results, in saturated conditions, in the development of pore water pressures which must dissipate to achieve consolidation of the material. Portions of the tailings will be at different degrees of consolidation within a vertical profile of the material.

The following consolidation densities (Table 3) have been calculated using the Terzaghi model for corresponding degree of consolidation shown:

TABLE 3
CONSOLIDATION DENSITIES

	90% Consolidation	80% Consolidation	70% Consolidation
Slimes	87 pcf*	83 pcf	79 pcf
Whole Tails	120 pcf	114 pcf	108 pcf
Mixed Tails (80/20)**	94 pcf	89 pcf	85 pcf

* All values are dry density.
** Based on weighted average of slimes and whole tailings densities.

To evaluate the time dependent degree of consolidation of the increasing thickness of the tailings, an alternative solution to the consolidation equation to that of Terzaghi's may be used. A theoretical solution to the consolidation of a uniformly deposited clay layer (Gibson, 1958) was employed to evaluate possible tailings densities based on an assumed rate of rise. This method has been used successfully in several cases of predicting tailings consolidation. Summarized below (Table 4) is the estimated degree of consolidation for the tailings for a range of assumed rate of rise for the surface of the tailings or the rate of increase in tailings thickness in the impoundment.

TABLE 4
DENSITIES BY GIBSON ANALYSIS

	Slimes		Whole Tails	
Assumed Rate of Rise (ft/yr)	Estimated Degree of Consolidation, U (%)	Dry Density (pcf)	Estimated Degree of Consolidation, U (%)	Dry Density (pcf)
10	89	87	91	120
12	83	86	88	119
20	80	83	85	117
30	75	81	80	114

The operation of the impoundment will influence the in-situ density of the tailings materials.

Key aspects of the operation which may affect the average densities achieved in the facility include:

- Ratio of whole tailings to slimes deposited with time over the life of the facility;
- Sequence of whole tailings deposition relative to slimes deposition;
- The rate of rise of the facility or cells within the facility;
- The method of solution management on the surface of the tailings (sub-aqueous vs sub-aerial); and
- The water balance of the facility in terms of the size and variations in any free water pond on the surface of the tailings. This would involve the inclusion of seasonal climatic conditions (precipitation and evaporation) and would predict the percentage of the surface area under water over time.

The operation of the underground mine is scheduled such that mine backfilling will result in a relatively consistent delivery of 80 percent slimes and 20 percent whole tailings to the TMA. The operation of the TMA will involve the staged development of 4 cells, two for the initial zinc tailings followed by two for copper tailings. In general, apart from the short period when the next cell is commissioned, depositions will occur in one cell only. An average rate of rise of 12 ft is indicated for each cell.

Based on the data summarized in Table 4, a dry density of 93 pcf is indicated.

Limited data is available for the copper tailings. Review of the description of the ore, however, suggest that similar physical properties would occur for these tailings as for the zinc tailings. With the assumption of a similar average specific gravity for both copper and zinc tailings, the same dry density of 93 pcf is suggested.

Based upon the relative values of pan evaporation tests and the consolidation test results, the final average density following post-operational consolidation is unlikely to be significantly affected by depositional method. During operations, however, a small increase in density is indicated by the test results. This may account for a 2 to 5 pcf increase in operating densities.

An estimate of the typical overall hydraulic conductivity for the tailings deposits may be made using Terzaghi's theory and the consolidation test results. An order of magnitude k value of 10^{-6} cm/s is estimated for the slimes, with the value for the slimes approximately 1/2 that of the whole tailings. The mixed tailings will exhibit percolation characteristics similar to that of the slimes.

In regard to post-closure consolidation, the bulk of settlement of the material can be anticipated to occur within one year of deposition in the facility or part thereof. Long-term consolidation may continue for an extended period of time, however, its extent is indicated to be limited following the initial year of consolidation.

For a maximum tailings thickness of 85 ft it is possible to estimate the order of magnitude of post-closure consolidation. If the operational density increases by the 6 to 8 percent suggested by the test results, a consolidation of the post-closure surface of approximately 4 to 6 ft may be estimated. Of this consolidation, 3 to 5 ft can be anticipated to occur within one year of the end of deposition.

4.0 COMPARISONS TO PREVIOUS TESTING AND CASE HISTORIES

To compare the results of the current test program and previous test work and predictions of in-situ tailings density, a compilation of such is given in Table 5. The validity of the application of the results from any test program in predictions of in-situ density is highly dependent upon the representative nature of the samples tested. The specific gravity of the sample is also of significant impact upon the absolute density values. The specific gravity of the tailings products range from 2.93 to 3.91 and average 3.46.

The following compares and contrasts each previous test program to the current program.

The Knight & Piesold (KP) studies (June 1981, December 1982) included one sample (FS6) REM ~~(B56)~~ which may be considered similar to the whole tailings in the current test. Adjusting the data from the column tests and pan evaporation tests for specific gravity, the data is not significantly different considering the difference in test methods and procedures. The recommended design dry density for the whole tailings of 127 pcf is, however, based upon fully drained and sub-aerial deposition. It is also interesting that the KP studies show little variation in "settled" or "pan evaporation" densities for a wide range of initial slurry densities.

TABLE 5
SUMMARY OF CRANDON TAILINGS TESTING
SRK PROJECT #71501 - TASK 7

SOURCE OF DATA	DESCRIPTION OF SAMPLE	GRAIN SIZE % PASSING 20 MICRON	SPECIFIC GRAVITY	MAX DESIGN DRY DENSITY (PCF)	MAX DESIGN DRY DENSITY USING AVG S.G. (PCF)	SLURRY DENSITY (% SOLIDS BY WT)	DRAINED COLUMN SETTLED DRY DENSITY (PCF)	DRAINED COLUMN SETTLED VOID RATIO	UNDRAINED COLUMN SETTLED DRY DENSITY (PCF)	UNDRAINED COLUMN SETTLED VOID RATIO	PAN EVAP DRY DENSITY (PCF)	PAN EVAP VOID RATIO	CONSOL DRY DENSITY (PCF)	CONSOL MAX LOAD (PSF)	CONSOL VOID RATIO	Cv (CM^2/SEC)	k (CM/SEC)	COMMENTS ON RECOMMENDED DESIGN DENSITIES
SRK (SEPT 1994)	MASSIVE FINE TAILINGS	82	3.32	70	73	45%	80-85	2.19 - 2.45	80-85	2.19 - 2.45	80	1.59	82	25,600	1.25	0.091	1.00E-06	DENSITY BASED ON COLUMN
	MASSIVE WHOLE TAILINGS	47	3.52	100	96	45%	80-85	1.56 - 1.75	85-95	1.31 - 1.56	112	0.96	127	25,600	0.73	0.127		DENSITIES
KNIGHT & PIESOLD (JUNE 1981) (DEC 1982)	MASSIVE / STRINGER ZINC TAILS (FS6)	56	3.46	127	127	VARIED	96.61	1.19	96.74	1.23	124.82	0.73					3.00E-05	DESICCATED DENSITY USED
GOLDER (FEB 1981)	ZINC COMPOSITE SLIMES (-20 MICRON)	96	3.16 (3.22)	94 (95)	103													S.G. ASSUMED, ASSUMED e = 1.1
STS CONSULTANTS (JUNE 1983)	TAILINGS (-30 MICRON)		3.37			55%								32,000	0.73		5.00E-06	NO RECOMMENDATION MADE S.G. DATA NOT INCLUDED
DAMES & MOORE (APRIL 1978)	RAW STRINGER (S2)	44	2.93		132								112.1	25,600	0.83	0.15		NO RECOMMENDATION MADE
	RAW MASSIVE (M2)	50	3.91		123								139.4	25,600	0.75	0.15		
FEASIBILITY STUDY CSMRI (1982)	TAILINGS FINES - MASSIVE ORE	67	3.8															BASED ON CONSOLIDATED DENSITY
	TAILINGS FINES - MASSIVE/ STRINGER ORE	75	3.2	106	121								92.4	80,000	1.18		5.00E-06	ASSUMED e = 0.85 FOR DESIGN DENSITY
LAKEFIELD (RECENT DATA)	WHOLE TAILINGS	50	3.57															NO RECOMMENDATION MADE
	SLIMES	83	3.73															

NOTES:

1. GRAIN SIZE DETERMINED AS PER ASTM D-422
2. SPECIFIC GRAVITY DETERMINED AS PER ASTM D-854
3. CONSOLIDATION DETERMINED AS PER ASTM D-2435
4. PARENTHESIZED VALUES IN THE GOLDER (FEB 1981) RESULTS REPRESENT RESULTS CONDUCTED ON +/- 30 MICRON MATERIAL AS OPPOSED TO +/- 20 MICRON MATERIAL

The Golder data contains one sample which may be considered similar to the slimes tested. This sample is referenced as zinc composite slimes on Table 5. The Golder study uses assumed void ratios in the calculation of the densities recommended, however, which are significantly lower than both the KP studies and the current work.

The Dames and Moore study has two tailings samples (S2 and M2) which may be considered similar to the whole tailings tested. Both samples under consolidation testing show values similar to the recent consolidation test when the results are adjusted for specific gravity.

The CSMRI study tested two slimes samples which fall between the sand and slimes tested. The value obtained from the consolidation testing of the finer grained sample indicates a density of 92.4 pcf. This test, however, is at a significantly higher ultimate load.

For comparative purposes, the achieved operational densities (whole-tails) for the Grey Eagle and Cannon Mines, adjusting for specific gravity, were 98 and 107 pcf, respectively. Values for the Crandon Project may be lower than these values. Some cycloning was conducted at Grey Eagle, and limited beaching was achieved. More substantial beaching and reduction of water ponding was achieved at Cannon. In summary, considering the range of materials tested and the test method selected in the studies as most representative of in-situ conditions, the data is consistent.

5.0 DESIGN DENSITIES

The test program described in this memorandum and the potential range of operating conditions allow the prediction of the general range of in-situ dry densities that may exist within the Crandon Project tailings facility. These values are presented in Table 6.

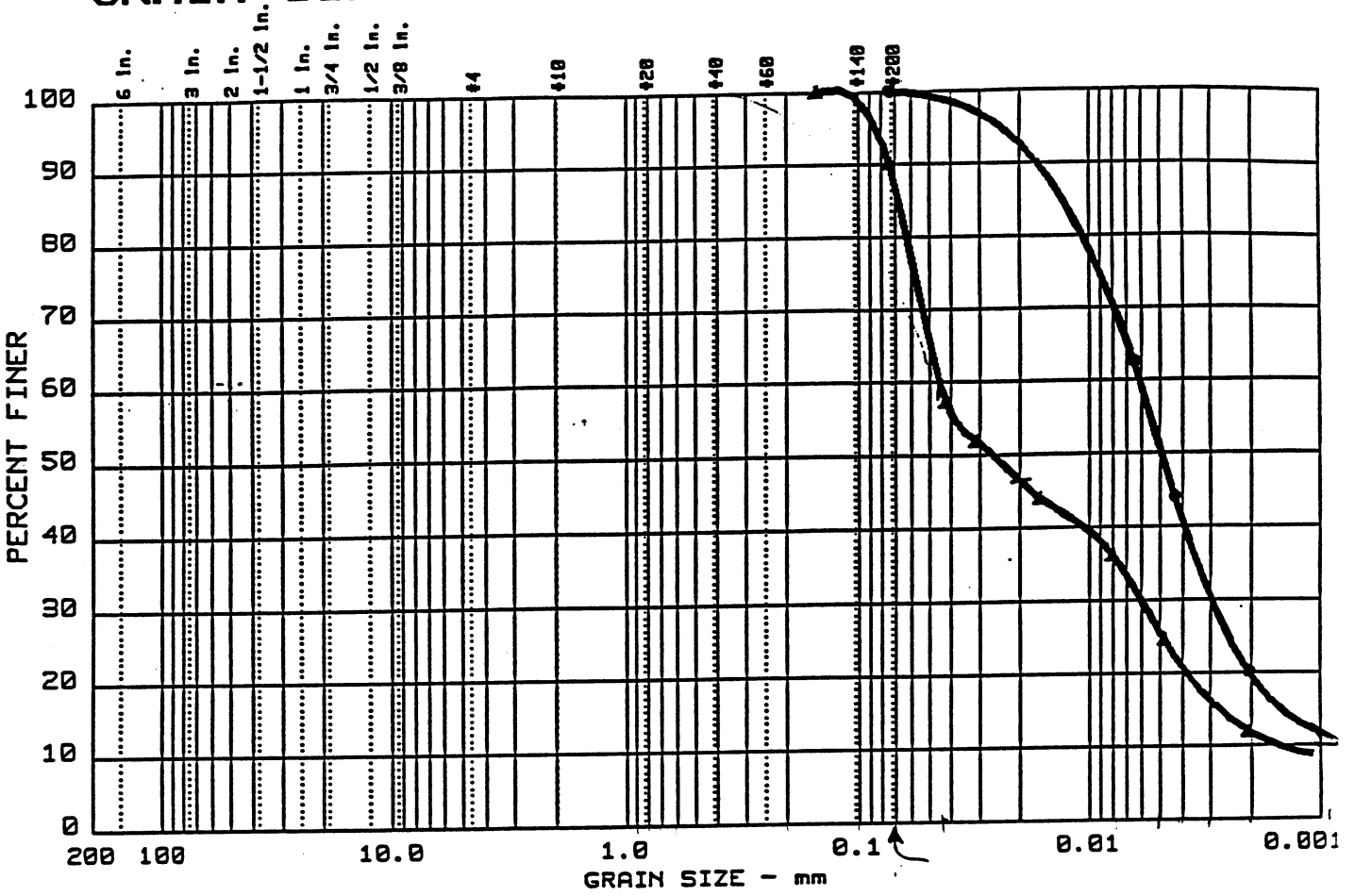
TABLE 6
DESIGN TAILINGS DRY DENSITIES
(all densities in pcf)

Operating Densities		Final Densities
Sub-Aqueous	Sub-Aerial	
93	97	100

RF/RD:lp
Attachments

LABORATORY DATA

GRAIN SIZE DISTRIBUTION TEST REPORT



	Test	% +3"	% GRAVEL	% SAND	% FINES
●	4	0.0	0.0	0.0	100.0
▲	6	0.0	0.0	10.0	90.0

	LL	PI	D_{85}	D_{60}	D_{50}	D_{30}	D_{15}	D_{10}	C_c	C_u
●		-			0.00	0.003	0.0015			
▲					0.03	0.006	0.0027	0.0015	0.52	29.5

MATERIAL DESCRIPTION	USCS	AASHTO
● SILT	ML	
▲ SILT, slightly sandy	ML	

Project No.: F107101
Project: CRANDON
● Location: Fine Tailings
▲ Location: Whole Tailings

Date: 6/7/94

GRAIN SIZE DISTRIBUTION TEST REPORT
STEFFEN ROBERTSON AND KIRSTEN (U.S.) INC.
Consulting Engineers and Scientists

Remarks:

Figure No. _____

GRAIN SIZE DISTRIBUTION TEST DATA

Test No.: 4

:e: 6/7/94
roject No.: F107101
roject: CRANDON

Sample Data

ocation of Sample: Fine Tailings
ample Description: SILT
USCS Class: ML
ASHTO Class:
Liquid limit:
Plasticity index:

Notes

emarks:

Fig. No.:

Mechanical Analysis Data

	Initial
ry sample and tare=	80.70
are =	0.00
ry sample weight =	80.70

re for cumulative weight retained= 0

Sieve	Cumul. Wt. retained	Percent finer
# 200	0.00	100.0

Hydrometer Analysis Data

eparation sieve is number 4
ercent -# 4 based on complete sample= 100.0
Weight of hydrometer sample: 80.7
alculated biased weight= 80.70
utomatic temperature correction
Composite correction at 20 deg C =-3

eniscus correction only= 1
Specific gravity of solids= 3.32
Specific gravity correction factor= 0.891
ydrometer type: 152H Effective depth L= 16.294964 - 0.164 x Rm

Elapsed time, min	Temp, deg C	Actual reading	Corrected reading	K	Rm	Eff. depth	Diameter mm	Percent finer
19.0	24.0	58.5	56.5	0.0110	59.5	6.5	0.0064	62.3
60.0	24.0	42.0	40.0	0.0110	43.0	9.2	0.0043	44.1
365.0	25.0	20.0	18.3	0.0108	21.0	12.9	0.0020	20.2
5668.0	25.0	11.0	9.3	0.0108	12.0	14.3	0.0005	10.2

Fractional Components

Gravel/Sand based on #4 sieve
Sand/Fines based on #200 sieve
% + 3 in. = 0.0 % GRAVEL = 0.0 % SAND = 0.0
% SILT =100.0 % CLAY = 0.0 (% CLAY COLLOIDS = 11.7)

D85= 0.01 D60= 0.006 D50= 0.005
D30= 0.0029 D15= 0.00148

GRAIN SIZE DISTRIBUTION TEST DATA Test No.: 6

ate: 6/7/94
Project No.: F107101
Project: CRANDON

Sample Data

Location of Sample: Whole Tailings
Sample Description: SILT, slightly sandy
USCS Class: ML Liquid limit:
AASHTO Class: Plasticity index:

Notes

Remarks:

Fig. No.:

Mechanical Analysis Data

	Initial
Dry sample and tare=	80.40
Tare =	0.00
Dry sample weight =	80.40

Tare for cumulative weight retained= 0

Sieve	Cumul. Wt. retained	Percent finer
# 100	0.00	100.0
# 200	8.00	90.0

Hydrometer Analysis Data

Separation sieve is number 4
Percent -# 4 based on complete sample= 100.0
Weight of hydrometer sample: 72
Calculated biased weight= 72.00
Automatic temperature correction
Composite correction at 20 deg C =-3

Meniscus correction only= 1
Specific gravity of solids= 3.52
Specific gravity correction factor= 0.870
Hydrometer type: 152H Effective depth L= 16.294964 - 0.164 x Rm

Elapsed time, min	Temp, deg C	Actual reading	Corrected reading	K	Rm	Eff. depth	Diameter mm	Percent finer
0.5	24.0	49.5	47.5	0.0105	50.5	8.0	0.0421	57.3
1.0	24.0	45.0	43.0	0.0105	46.0	8.8	0.0311	51.9
4.0	24.0	38.5	36.5	0.0105	39.5	9.8	0.0165	44.0

Elapsed time, min	Temp, deg C	Actual reading	Corrected reading	K	Rm	Eff. depth	Diameter mm	Percent finer
19.0	24.0	32.0	30.0	0.0105	33.0	10.9	0.0080	36.2
60.0	24.0	22.5	20.5	0.0105	23.5	12.4	0.0048	24.7
370.0	24.0	12.0	10.0	0.0105	13.0	14.2	0.0021	12.0
5674.0	24.0	9.0	7.0	0.0105	10.0	14.7	0.0005	8.4

Fractional Components

Gravel/Sand based on #4 sieve
Sand/Fines based on #200 sieve
% + 3 in. = 0.0 % GRAVEL = 0.0 % SAND = 10.0
% SILT = 90.0 % CLAY = 0.0 (% CLAY COLLOIDS = 8.6)

D85= 0.07 D60= 0.045 D50= 0.026
D30= 0.0060 D15= 0.00270 D10= 0.00152
Cc = 0.5248 Cu = 29.5121

DRAINED SETTLING TEST

Project	Crandon	Tested By	WAS
Project No.		Checked By	TM
Sample No.	Whole Tailings	Date	7/26/94

INITIAL SAMPLE DATA (Lift #1)

Init. Wet Wt. + Pan(g)	460.60	Init. Solids Content	45.6%
Init. Dry Wt. + Pan(g)	323.60	Init. Moisture Content	119.3%
Pan Wt. Only (g)	208.80	Specific Gravity	3.52

INITIAL SAMPLE DATA (Lift #2 and Lift #3)

Init. Wet Wt. + Pan(g)	507.90	Init. Solids Content	36.6%
Init. Dry Wt. + Pan(g)	329.50	Init. Moisture Content	173.5%
Pan Wt. Only (g)	226.70	Specific Gravity	3.52

LIFT #1 (Bottom Lift Covered by 2 Additional Lifts)

Total Settlement Time = 184.0 hours

Sample Dia (cm)	6.30	Init. Wet Wt. (g)	694.9
Area (cm^2)	31.16	Init. Void Ratio	4.607
Init. Ht. (cm)	16.20	Init. Dry Density (pcf)	39.2
Init. Vol (cm^3)	504.87	Final Moisture Content (%)	45.3%
Final Ht. (cm)	7.50	Final Void Ratio	1.596
Final Vol (cm^3)	233.73	Final Dry Density (pcf)	84.6

LIFT #2 (Middle Lift Covered by 1 Additional Lift)

Total Settlement Time = 93.8 hours

Sample Dia (cm)	6.30	Init. Wet Wt. (g)	495.0
Area (cm^2)	31.16	Init. Void Ratio	6.756
Init. Ht. (cm)	12.80	Init. Dry Density (pcf)	28.3
Init. Vol (cm^3)	398.91	Final Moisture Content (%)	49.1%
Final Ht. (cm)	4.50	Final Void Ratio	1.727
Final Vol (cm^3)	140.24	Final Dry Density (pcf)	80.6

LIFT #3 (Top Lift)

Total Settlement Time = 64.5 hours

Sample Dia (cm)	6.30	Init. Wet Wt. (g)	616.5
Area (cm^2)	31.16	Init. Void Ratio	6.735
Init. Ht. (cm)	15.90	Init. Dry Density (pcf)	28.4
Init. Vol (cm^3)	495.52	Final Moisture Content (%)	50.4%
Final Ht. (cm)	5.70	Final Void Ratio	1.773
Final Vol (cm^3)	177.64	Final Dry Density (pcf)	79.2

FINAL SAMPLE DATA

Final Dry Wt. + Pan (g)	1027.20	Final Total Volume (cm^3)	532.91
Pan Wt. Only (g)	347.30	Final Moisture Content (%)	49.9%
Final Dry Wt.	679.90	Final Avg. Void Ratio	1.758
Final Total Ht. (cm)	17.10	Final Avg. Dry Density (pcf)	79.6

UNDRAINED SETTLING TEST

Project	Crandon	Tested By	WAS
Project No.		Checked By	TM
Sample No.	Whole Tailings	Date	7/26/94

INITIAL SAMPLE DATA (Lift #1)

Init. Wet Wt. + Pan(g)	460.60	Init. Solids Content	45.6%
Init. Dry Wt. + Pan(g)	323.60	Init. Moisture Content	119.3%
Pan Wt. Only (g)	208.80	Specific Gravity	3.52

INITIAL SAMPLE DATA (Lift #2 and Lift #3)

Init. Wet Wt. + Pan(g)	507.90	Init. Solids Content	36.6%
Init. Dry Wt. + Pan(g)	329.50	Init. Moisture Content	173.5%
Pan Wt. Only (g)	226.70	Specific Gravity	3.52

LIFT #1 (Bottom Lift Covered by 2 Additional Lifts)

Total Settlement Time = 184.0 hours

Sample Dia (cm)	5.94	Init. Wet Wt. (g)	639.8
Area (cm^2)	27.75	Init. Void Ratio	4.287
Init. Ht. (cm)	15.80	Init. Dry Density (pcf)	41.5
Init. Vol (cm^3)	438.38	Final Moisture Content (%)	36.2%
Final Ht. (cm)	6.80	Final Void Ratio	1.276
Final Vol (cm^3)	188.67	Final Dry Density (pcf)	96.5

LIFT #2 (Middle Lift Covered by 1 Additional Lift)

Total Settlement Time = 93.8 hours

Sample Dia (cm)	5.94	Init. Wet Wt. (g)	398.0
Area (cm^2)	27.75	Init. Void Ratio	6.581
Init. Ht. (cm)	11.30	Init. Dry Density (pcf)	29.0
Init. Vol (cm^3)	313.52	Final Moisture Content (%)	38.3%
Final Ht. (cm)	3.50	Final Void Ratio	1.348
Final Vol (cm^3)	97.11	Final Dry Density (pcf)	93.5

LIFT #3 (Top Lift)

Total Settlement Time = 64.5 hours

Sample Dia (cm)	5.94	Init. Wet Wt. (g)	565.5
Area (cm^2)	27.75	Init. Void Ratio	6.508
Init. Ht. (cm)	15.90	Init. Dry Density (pcf)	29.3
Init. Vol (cm^3)	441.15	Final Moisture Content (%)	42.7%
Final Ht. (cm)	5.30	Final Void Ratio	1.503
Final Vol (cm^3)	147.05	Final Dry Density (pcf)	87.8

FINAL SAMPLE DATA

Final Dry Wt. + Pan (g)	966.40	Final Total Volume (cm^3)	418.95
Pan Wt. Only (g)	346.60	Final Moisture Content (%)	39.2%
Final Dry Wt.	619.80	Final Avg. Void Ratio	1.378
Final Total Ht. (cm)	15.10	Final Avg. Dry Density (pcf)	92.4

PAN EVAPORATION TEST

Project	Crandon	Tested By	WAS
Project No.		Checked By	TM
Sample No.	Whole Tailings	Date	7/26/94

INITIAL SAMPLE DATA

Init. Wet Wt. + Pan(g)	460.60	Init. Solids Content	45.6%
Init. Dry Wt. + Pan(g)	323.60	Init. Moisture Content	119.3%
Pan Wt. Only (g)	208.80	Specific Gravity	3.52

FINAL SAMPLE DATA

Final Wet Wt. + Pan (g)	274.50	Subsample Ht. (cm)	1.07
Final Dry Wt. + Pan (g)	265.40	Supsample Dia. (cm)	4.93
Pan Wt. Only (g)	228.90	Void Ratio	0.969
Final Moisture Content (%)	24.9%	Dry Density (pcf)	111.6

137.00
9.1

91

375

DRAINED SETTLING TEST

Project	Crandon	Tested By	WAS
Project No.		Checked By	TM
Sample No.	Fine Tailings	Date	7/22/94

INITIAL SAMPLE DATA

Init. Wet Wt. + Pan(g)	499.50	Init. Solids Content	46.2%
Init. Dry Wt. + Pan(g)	354.00	Init. Moisture Content	116.3%
Pan Wt. Only (g)	228.90	Specific Gravity	3.32

LIFT #1 (Bottom Lift Covered by 2 Additional Lifts)

Total Settlement Time = 184.0 hours

Sample Dia (cm)	6.30	Init. Wet Wt. (g)	725.9
Area (cm^2)	31.16	Init. Void Ratio	4.085
Init. Ht. (cm)	16.50	Init. Dry Density (pcf)	40.7
Init. Vol (cm^3)	514.21	Final Moisture Content (%)	67.3%
Final Ht. (cm)	10.50	Final Void Ratio	2.236
Final Vol (cm^3)	327.23	Final Dry Density (pcf)	64.0

LIFT #2 (Middle Lift Covered by 1 Additional Lift)

Total Settlement Time = 93.8 hours

Sample Dia (cm)	6.30	Init. Wet Wt. (g)	485.0
Area (cm^2)	31.16	Init. Void Ratio	4.396
Init. Ht. (cm)	11.70	Init. Dry Density (pcf)	38.4
Init. Vol (cm^3)	364.63	Final Moisture Content (%)	68.5%
Final Ht. (cm)	7.10	Final Void Ratio	2.275
Final Vol (cm^3)	221.27	Final Dry Density (pcf)	63.3

LIFT #3 (Top Lift)

Total Settlement Time = 64.5 hours

Sample Dia (cm)	6.30	Init. Wet Wt. (g)	570.0
Area (cm^2)	31.16	Init. Void Ratio	4.180
Init. Ht. (cm)	13.20	Init. Dry Density (pcf)	40.0
Init. Vol (cm^3)	411.37	Final Moisture Content (%)	69.2%
Final Ht. (cm)	8.40	Final Void Ratio	2.297
Final Vol (cm^3)	261.78	Final Dry Density (pcf)	62.8

FINAL SAMPLE DATA

Final Dry Wt. + Pan (g)	1162.20	Final Total Volume (cm^3)	810.28
Pan Wt. Only (g)	353.50	Final Moisture Content (%)	70.0%
Final Dry Wt.	808.70	Final Avg. Void Ratio	2.325
Final Total Ht. (cm)	26.00	Final Avg. Dry Density (pcf)	62.3

UNDRAINED SETTLING TEST

Project	Crandon	Tested By	WAS
Project No.		Checked By	TM
Sample No.	Fine Tailings	Date	7/22/94

INITIAL SAMPLE DATA

Init. Wet Wt. + Pan(g)	499.50	Init. Solids Content	46.2%
Init. Dry Wt. + Pan(g)	354.00	Init. Moisture Content	116.3%
Pan Wt. Only (g)	228.90	Specific Gravity	3.32

LIFT #1 (Bottom Lift Covered by 2 Additional Lifts)

Total Settlement Time = 184.0 hours			
Sample Dia (cm)	5.94	Init. Wet Wt. (g)	705.0
Area (cm^2)	27.75	Init. Void Ratio	3.831
Init. Ht. (cm)	17.10	Init. Dry Density (pcf)	42.9
Init. Vol (cm^3)	474.44	Final Moisture Content (%)	63.5%
Final Ht. (cm)	11.00	Final Void Ratio	2.107
Final Vol (cm^3)	305.20	Final Dry Density (pcf)	66.7

LIFT #2 (Middle Lift Covered by 1 Additional Lift)

Total Settlement Time = 93.8 hours			
Sample Dia (cm)	5.94	Init. Wet Wt. (g)	309.0
Area (cm^2)	27.75	Init. Void Ratio	3.834
Init. Ht. (cm)	7.50	Init. Dry Density (pcf)	42.9
Init. Vol (cm^3)	208.09	Final Moisture Content (%)	66.9%
Final Ht. (cm)	5.00	Final Void Ratio	2.223
Final Vol (cm^3)	138.73	Final Dry Density (pcf)	64.3

LIFT #3 (Top Lift)

Total Settlement Time = 64.5 hours			
Sample Dia (cm)	5.94	Init. Wet Wt. (g)	674.0
Area (cm^2)	27.75	Init. Void Ratio	3.964
Init. Ht. (cm)	16.80	Init. Dry Density (pcf)	41.7
Init. Vol (cm^3)	466.12	Final Moisture Content (%)	75.8%
Final Ht. (cm)	.11.90	Final Void Ratio	2.516
Final Vol (cm^3)	330.17	Final Dry Density (pcf)	58.9

FINAL SAMPLE DATA

Final Dry Wt. + Pan (g)	1118.90	Final Total Volume (cm^3)	774.09
Pan Wt. Only (g)	350.50	Final Moisture Content (%)	70.6%
Final Dry Wt.	768.40	Final Avg. Void Ratio	2.343
Final Total Ht. (cm)	27.90	Final Avg. Dry Density (pcf)	62.0

PAN EVAPORATION TEST

Project	Crandon	Tested By	WAS
Project No.		Checked By	TM
Sample No.	Fine Tailings	Date	7/22/94

INITIAL SAMPLE DATA

Init. Wet Wt. + Pan(g)	499.50	Init. Solids Content	46.2%
Init. Dry Wt. + Pan(g)	354.00	Init. Moisture Content	116.3%
Pan Wt. Only (g)	228.90	Specific Gravity	3.52

FINAL SAMPLE DATA

Final Wet Wt. + Pan (g)	286.40	Subsample Ht. (cm)	1.68
Final Dry Wt. + Pan (g)	267.90	Supsample Dia. (cm)	4.93
Pan Wt. Only (g)	226.80	Void Ratio	1.739
Final Moisture Content (%)	45.0%	Dry Density (pcf)	80.2

CONSOLIDATION TEST

CLIENT SRK

JOB NO. 2030-16

BORING NO. Whole
DEPTH
SAMPLE NO. Crandon
SOIL DESCR.

SAMPLED
TEST STARTED 7-22-94 SR CJW
TEST FINISHED 8-1-94
SETUP NO. ATT-08

SAMPLE DIA.(in) 2.410

INITIAL HT.(in) 1.000

MOISTURE/DENSITY DATA		BEFORE TEST	AFTER TEST
Wt. Soil & Ring(s)	(g)	235.5	166.7
Wt. Ring(s)	(g)	45.5	45.5
Wt. Soil	(g)	190.0	121.2
Wet Density	PCF	158.7	154.4
Wt. Wet Soil & Pan	(g)	193.7	124.9
Wt. Dry Soil & Pan	(g)	106.5	106.5
Wt. Lost Moisture	(g)	87.2	18.4
Wt. of Pan Only	(g)	3.7	3.7
Wt. of Dry Soil	(g)	102.8	102.8
Moisture Content	%	84.9	17.9
Dry Density	PCF	85.8	130.9
Max. Dry Density	PCF		
Percent Compaction			

LOAD (PSF)	CONSOL. (IN.)
30	0.2192
100	0.2385
200	0.2488
400	0.2625
800	0.2747
1600	0.2897
3200	0.3046
6400	0.3234
12800	0.3398
25600	0.3595
6400	0.3562
1600	0.3522
400	0.3484
100	0.3444

LOAD (PSF)	LOG p	CONSOL. (IN.)	DEFL. (IN.)	VOID RATIO e
30	1.477	0.2192	-0.2192	0.9990
100	2.000	0.2385	-0.2385	0.9496
200	2.301	0.2488	-0.2488	0.9232
400	2.602	0.2625	-0.2625	0.8882
800	2.903	0.2747	-0.2747	0.8569
1600	3.204	0.2897	-0.2897	0.8185
3200	3.505	0.3046	-0.3046	0.7804
6400	3.806	0.3234	-0.3234	0.7323
12800	4.107	0.3398	-0.3398	0.6903
25600	4.408	0.3595	-0.3595	0.6399
6400	3.806	0.3562	-0.3562	0.6483
1600	3.204	0.3522	-0.3522	0.6585
400	2.602	0.3484	-0.3484	0.6683
100	2.000	0.3444	-0.3444	0.6785

Sp. Gr. (g/cc)	3.520
Init. Ht. Solids (cm)	0.992
Init. Ht. Voids (cm)	1.548
Init. Void Ratio	1.5602

TIME READING DATA Whole

800 psf Load

Elapsed Time (min)	SQRT Time (min)	Dial Reading (in)	Defl. (in)
0.0	0.00	0.0439	0.0000
0.1	0.32	0.0515	-0.0076
0.3	0.50	0.0523	-0.0084
0.5	0.71	0.0528	-0.0089
1.0	1.00	0.0531	-0.0092
2.0	1.41	0.0534	-0.0095
4.0	2.00	0.0538	-0.0099
9.0	3.00	0.0542	-0.0103
16.0	4.00	0.0545	-0.0106
30.0	5.48	0.0549	-0.0110
60.0	7.75	0.0553	-0.0114
120.0	10.95	0.0557	-0.0118
240.0	15.49	0.0561	-0.0122
480.0	21.91	0.0560	-0.0121

1600 psf Load

Elapsed Time (min)	SQRT Time (min)	Dial Reading (in)	Defl. (in)
0.0	0.00	0.0567	0.0000
0.1	0.32	0.0672	-0.0105
0.3	0.50	0.0679	-0.0112
0.5	0.71	0.0682	-0.0115
1.0	1.00	0.0685	-0.0118
2.0	1.41	0.0689	-0.0122
4.0	2.00	0.0693	-0.0126
9.0	3.00	0.0697	-0.0130
16.0	4.00	0.0700	-0.0133
30.0	5.48	0.0705	-0.0138
60.0	7.75	0.0709	-0.0142
128.0	11.31	0.0714	-0.0147
240.0	15.49	0.0719	-0.0152
485.0	22.02	0.0722	-0.0155

3200 psf Load

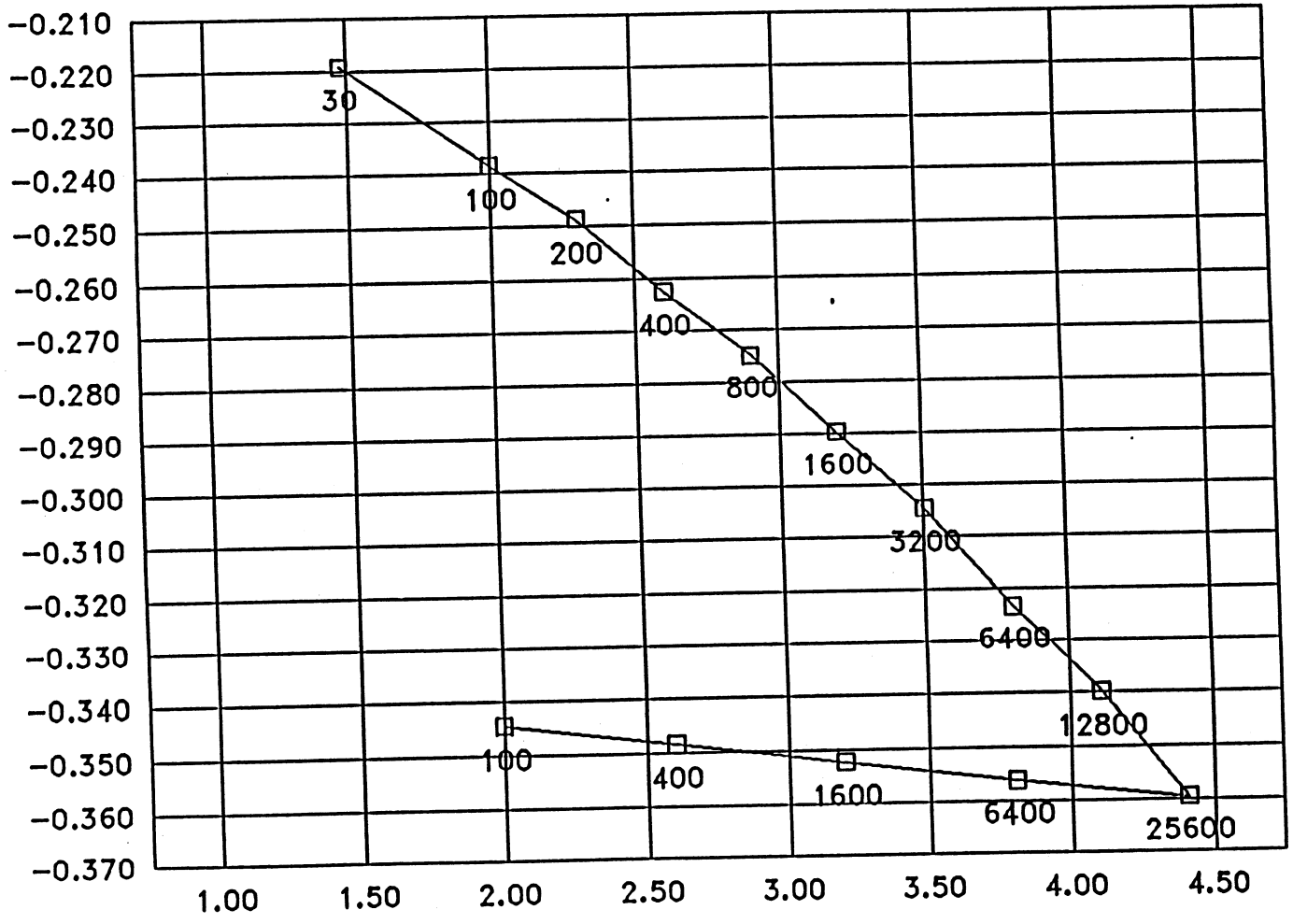
Elapsed Time (min)	SQRT Time (min)	Dial Reading (in)	Defl. (in)
0.0	0.00	0.0726	0.0000
0.1	0.32	0.0835	-0.0109
0.3	0.55	0.0839	-0.0113
0.5	0.71	0.0844	-0.0118
1.0	1.00	0.0847	-0.0121
2.0	1.41	0.0851	-0.0125
4.0	2.00	0.0855	-0.0129
9.0	3.00	0.0861	-0.0135
16.0	4.00	0.0864	-0.0138
30.0	5.48	0.0868	-0.0142
60.0	7.75	0.0873	-0.0147
120.0	10.95	0.0878	-0.0152
260.0	16.12	0.0883	-0.0157
480.0	21.91	0.0886	-0.0160

6400 psf Load

Elapsed Time (min)	SQRT Time (min)	Dial Reading (in)	Defl. (in)
0.0	0.00	0.0889	0.0000
0.1	0.32	0.1032	-0.0143
0.3	0.50	0.1044	-0.0155
0.5	0.71	0.1049	-0.0160
1.0	1.00	0.1054	-0.0165
2.0	1.41	0.1058	-0.0169
4.0	2.00	0.1062	-0.0173
9.0	3.00	0.1067	-0.0178
16.0	4.00	0.1074	-0.0185
30.0	5.48	0.1078	-0.0189
60.0	7.75	0.1082	-0.0193
120.0	10.95	0.1085	-0.0196
285.0	16.88	0.1088	-0.0199
480.0	21.91	0.1091	-0.0202

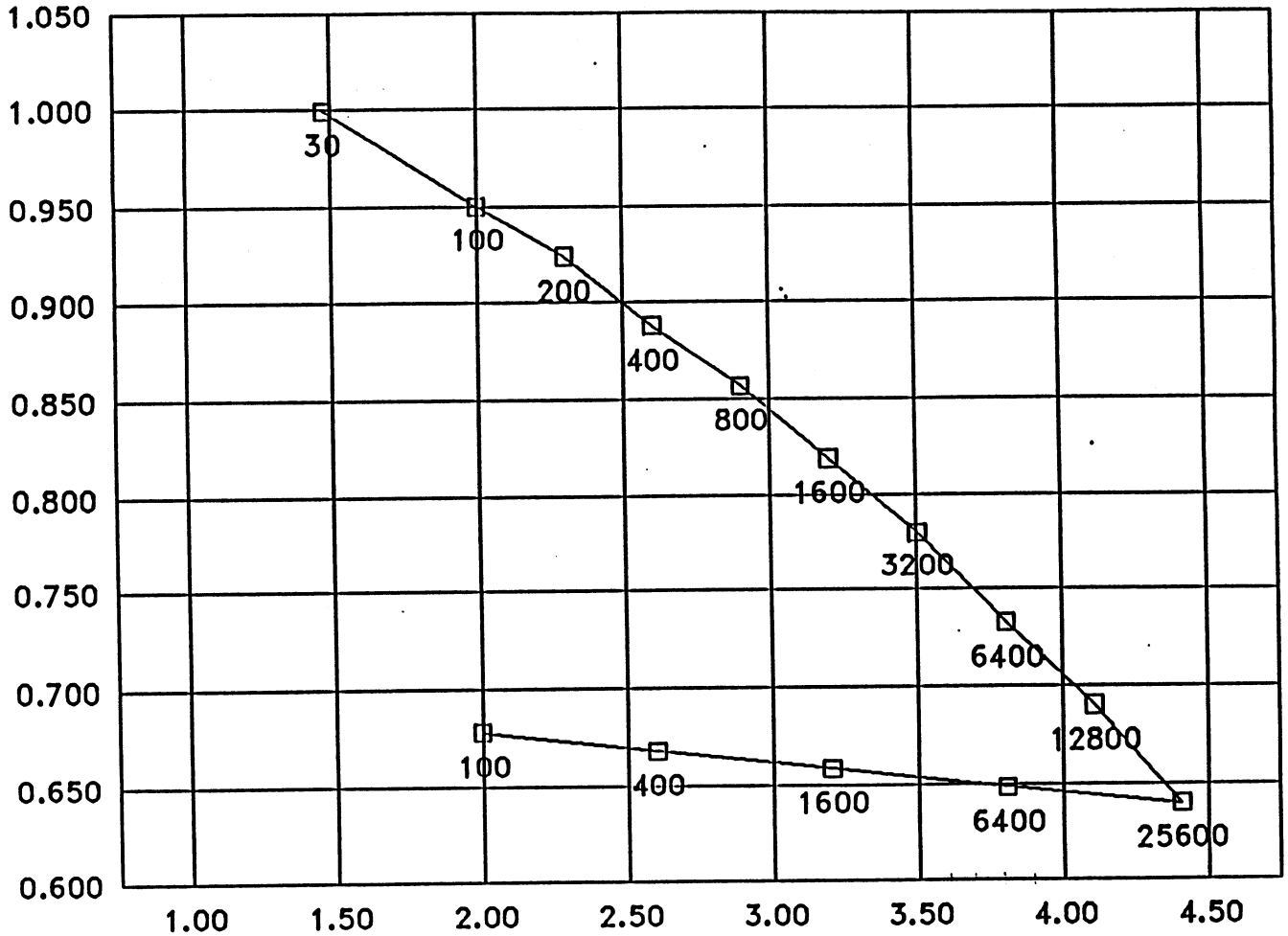
CONSOLIDATION TEST DATA

Whole Crandon



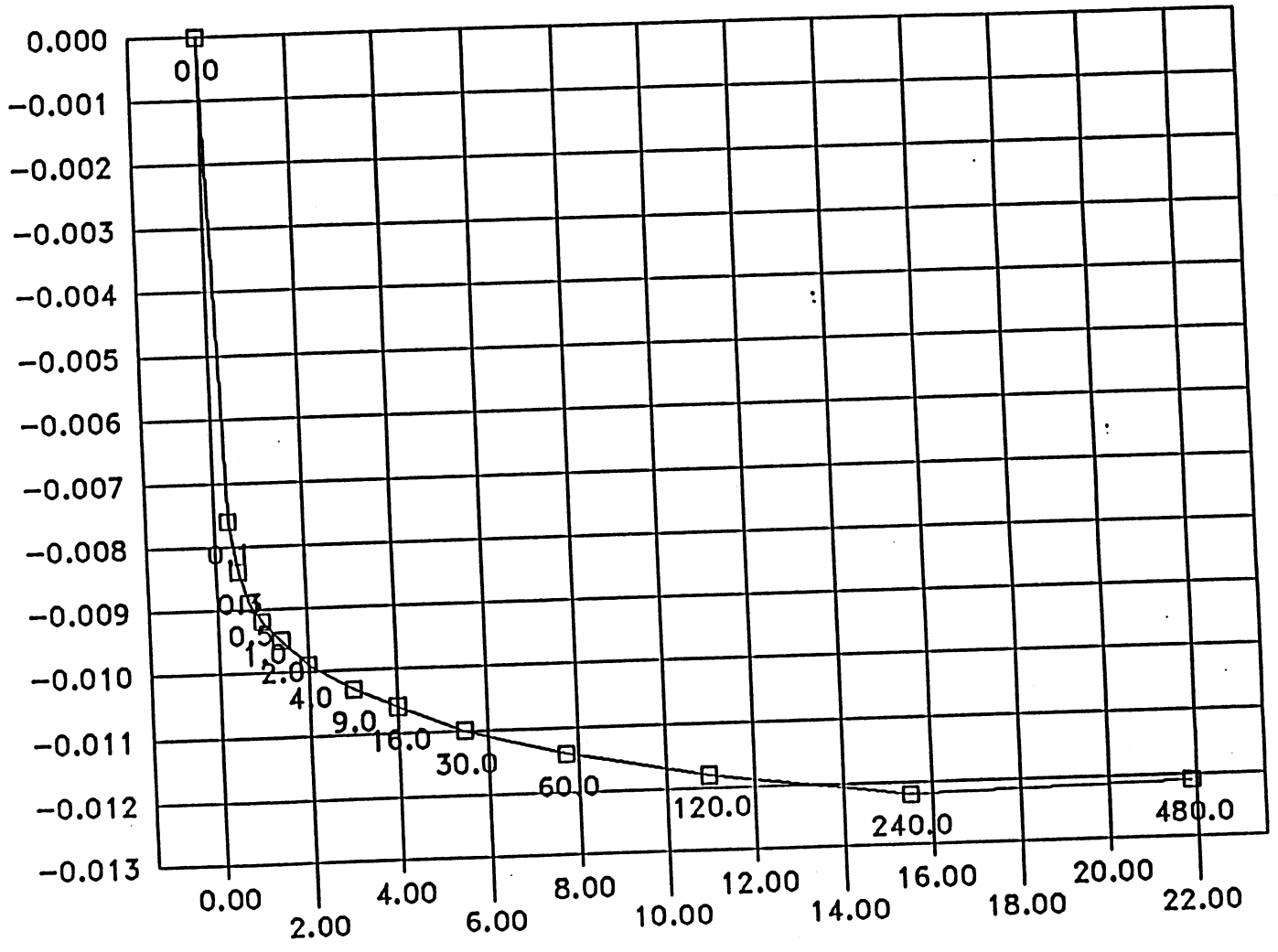
CONSOLIDATION TEST DATA

Whole Crandon



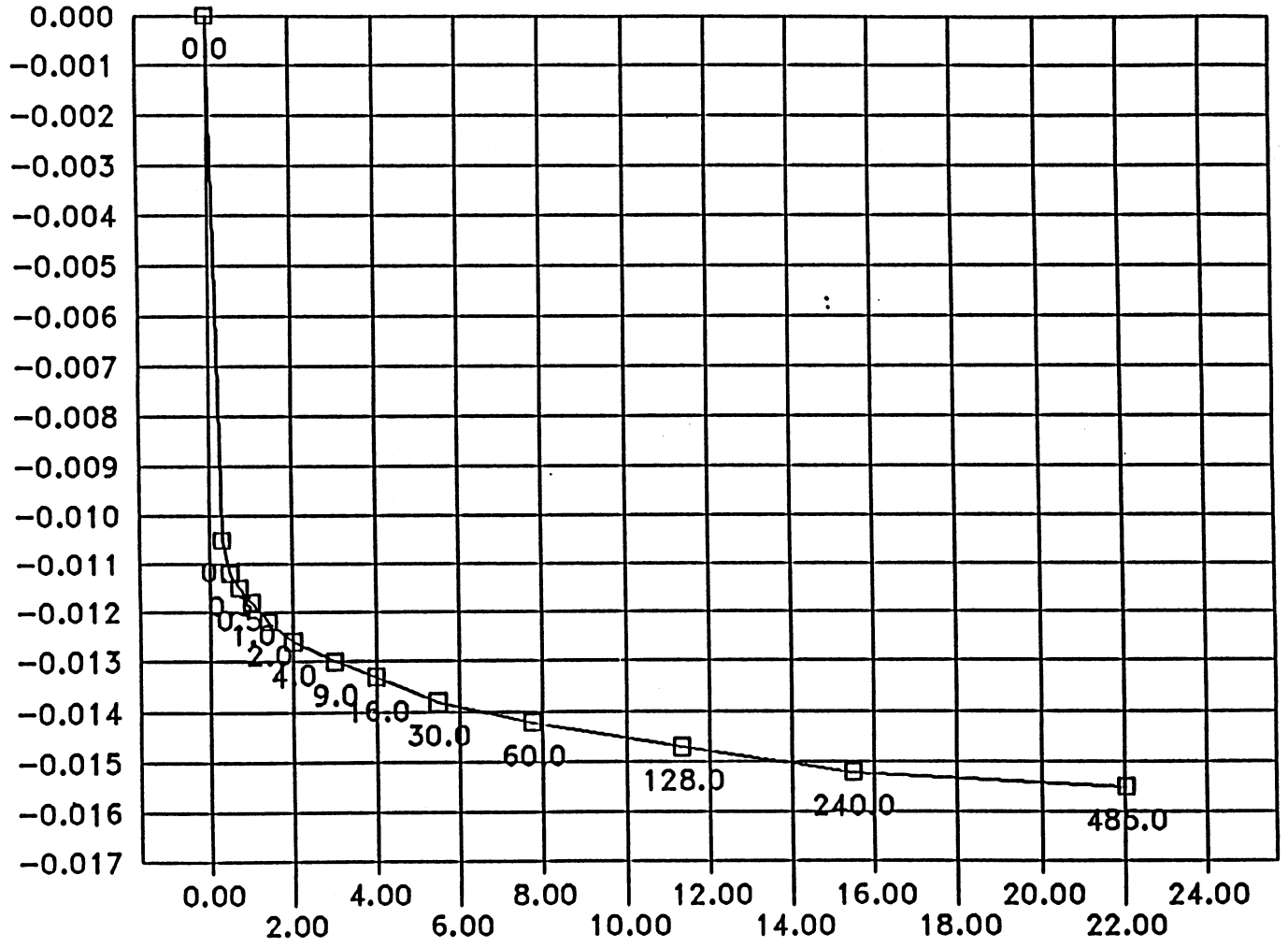
TIME READING DATA

Whole Crandon 800 psf Load



TIME READING DATA

Whole Crandon 1600 psf Load



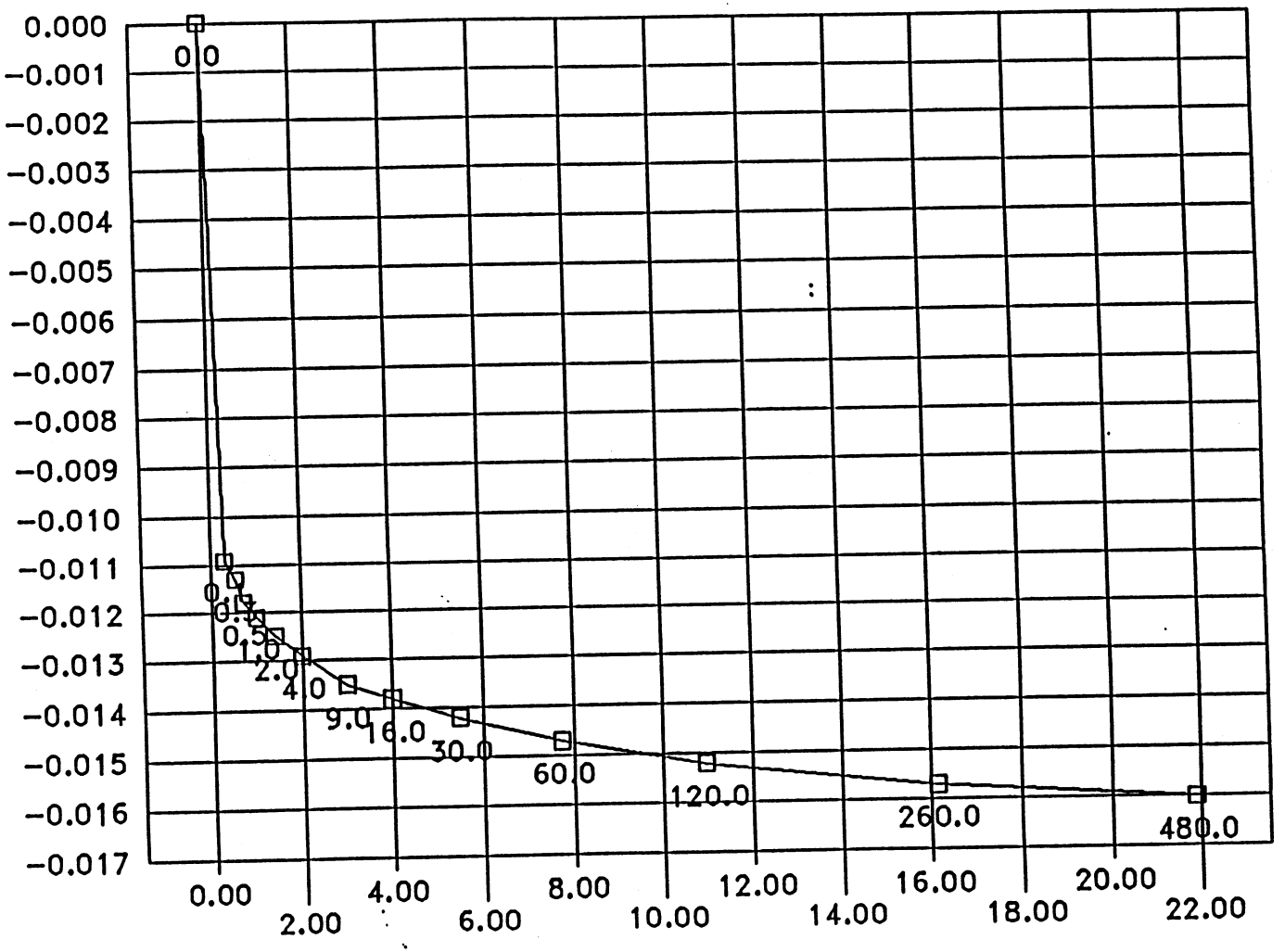
Deflection in inches

SQUARE ROOT of TIME

□ Time In Minutes

TIME READING DATA

Whole Crandon 3200 psf Load



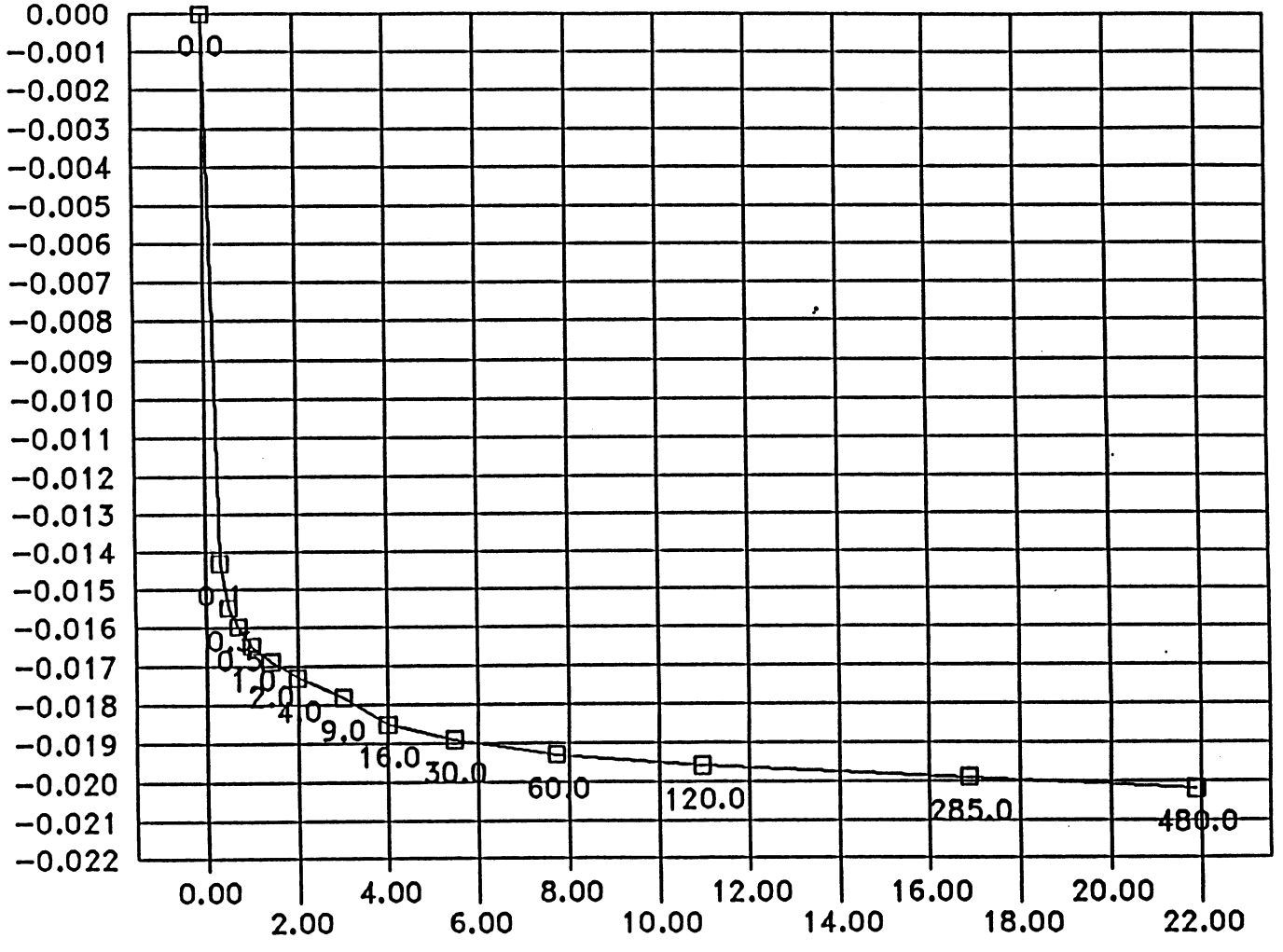
Deflection in inches

SQUARE ROOT of TIME

□ Time in Minutes

TIME READING DATA

Whole Crandon 6400 psf Load



CONSOLIDATION TEST DATA SHEET

SRCCWH
SRCEWH
SCRCCWH 1/2 3/4

SRCQWH

JOB SRK — JOB NO. 2030-16

DATE SAMPLED CRANDON — BORING NO. WHOLE

TEST STARTED 7-22-94 BY SR — TEST FINISHED ____ BY ____ DEPTH ____

SOIL TYPE TAILINGS

CONSOLIDOMETER NO. ATT-8 — INITIAL SAMPLE HEIGHT 1.00" = 2.54 cm A_o = 29.431 cm²
SG = 3.52

MOISTURE & DENSITY DATA	BEFORE TEST	AFTER TEST
WT. OF RING(S)+ SOIL g	235.48	166.48
WT. OF RING(S) ONLY g	45.48	
WT. OF SOIL (NET)		
WET DENSITY IN LBS/CU. FT.*		
WT. OF WET SOIL + DISH NO. KØ		
INT. OF DRY SOIL + DISH		
WT. OF LOST MOISTURE		
WT. OF DISH ONLY g	3.70	3.70
WT. OF DRY SOIL		
MOISTURE AS % OF DRY WT.		
DRY DENSITY IN LBS/CU. FT.		

TEST INSTRUCTIONS

LOAD CYCLE 30.4 100 200 400 800 TR 1600 TR 3200 TR 6400 TR 12800 25600
RE-BOUND X 2

SPECIAL INSTRUCTION AND NOTES

DRY WT. SOIL + PAN + RING = 151.9
WS = 102.78g
HS = 102.79/3.52/29.431 = .9922
HV = 2.54 - .9922 = 2.560 1.54"
C_o = 2.560/.9922 = 2.580
FH = 1.00" - .3444 = .6556"

DATE	TIME	ELAPSED TIME IN MINS.	LOAD (PSF)	DIAL READING (.0001 IN.)	CONSOL UNDER THIS LOAD	TOTAL CONSOLIDATION
7-21-94	1135	2	30.4	.1830		
	1136	3		.2106		
	1137	4		.2168		
	1138			.2192		.2192
	1324		100		New ZERO!	
	1325	1		.0157		
	1326	2		.0162		
	1423			.0179		
	1606			.0184		
7-23-94	0847			.0192		
	1016			.0193		.2385
			200			
	1019	1		.0268		
	1020	2		.0272		
	1025			.0278		
	1110			.0287		
	1325			.0293		
	1519			.0296		
	1714			.0297	.2489	.2488
			400			
	1716	1		.0409		
	1717	2		.0411		
	1735			.0424		
	1843			.0431		
7-24-94	0907			.0439	.2631	.2625
			800			
		.1		.0515		
		.25		.0523		
		.5		.0528		
	0917	1		.0531		

DATE	TIME	ELAPSED TIME IN MINS.	LOAD (PSF)	DIAL READING (.0001 IN.)	CONSOL UNDER THIS LOAD	TOTAL CONSOLIDATION
7-24-94	0918	2	800	.0534		
	0920	4		.0538		
	0925	9		.0542		
	0932	16		.0545		
	0946	30		.0549		
	1016	60		.0553		
	1116	120		.0557		
	1316	240		.0561		
	1716	480		.0564		
7-25-94	0835			.0567	.2759	.2747
			1600			
		.1		.0672		
		.25		.0679		
		.5		.0682		
	0836	1		.0685		
	0837	2		.0689		
	0839	4		.0693		
	0844	9		.0697		
	0851	16		.0700		
	0905	30		.0705		
	0935	60		.0709		
1043	~~1035~~	~~120~~	128	.0714		
	1235	240		.0719		
	~~1655~~	480	485	.0722		
7-26-94	0830			.0726	.2918	.2897
			3200			
		.1		.0835		
		.25		.0839		
		.5		.0844		
	0831	1		.0847		

ADVANCED TERRA TESTING

CONTINUED ON REVERSE SIDE

WHOLE TAILINGS

NSOLIDATION TEST DATA SHEET (CONTINUED)

	TIME	E-LAPSED TIME IN MINS.	LOAD (PSF)	DIAL READING (.0001 IN.)	CONSOL UNDER THIS LOAD	TOTAL CONSOL-IDATION
.7	0832	2	3200	.0951		
	0834	4		.0850		
	0839	9		.0861		
	0846	16		.0864		
	0900	30		.0868		
	0930	60		.0873		
	1030	120		.0878		
	1230	240	210	.0883		
	1630	480		.0886		
94	809			.0889		
	0920			.0889	.3081	.3046
	9:40	0	6400			
		.1		.1032		
		.25		.1044		
		.5		.1049		
	9:41	1		.1054		
	942	2		.1058		
	944	4		.1062		
	949	9		.1067		
	956	16		.1074		
	10:10	30		.1078		
	1040	60		.1082		
	11:40	120		.1085		
	~~12:40~~	~~240~~	285	.1088		
	17:40	480		.1091		
94	0815			.1092	.3284	.3234
	0936		1			
			12800			
	0937	1		.1243		
	0938	2		.1246		
	1339			.1273		
	1504			.1274	.3466	.3398
			25600			
	1505	1		.1453		
	1506	2		.1457		
94	0820			.1491		
	0935			.1491	.3689	.3595
			6400			
	0936	1		.1454		
	0937	2		.1454		
	1041			.1458	.3644	.3562
			1600			
	1042	1		.1401		
	1043	2		.1400		
	1221			.1396		
	1415			.1396	.3588	.3522
			400			
	1416	1		.1356		
	1417	2		.1354		
	1605			.1345		
	1648			.1345	.3537	.3484
			100			
	1649	1		.1314		
	1650	2		.1313		
-94	0955			.1296		
24	0812			.1296	.3488	.3444

DATE	TIME	E-LAPSED TIME IN MINS.	LOAD (PSF)	DIAL READING (.0001 IN.)	CONSOL UNDER THIS LOAD	TOTAL CONSOL-IDATION

CONSOLIDATION TEST

CLIENT SRK

JOB NO. 2030-16

BORING NO. Fine
DEPTH
SAMPLE NO. Crandon
SOIL DESCR.

SAMPLED
TEST STARTED 7-22-94 SR CJW
TEST FINISHED 8-1-94
SETUP NO. ATT-07

SAMPLE DIA.(in) 2.410

INITIAL HT.(in) 1.000

MOISTURE/DENSITY DATA		BEFORE TEST	AFTER TEST
Wt. Soil & Ring(s)	(g)	194.1	158.9
Wt. Ring(s)	(g)	44.4	44.4
Wt. Soil	(g)	149.7	114.5
Wet Density	PCF	125.0	130.3
Wt. Wet Soil & Pan	(g)	153.4	118.1
Wt. Dry Soil & Pan	(g)	89.0	89.0
Wt. Lost Moisture	(g)	64.3	29.1
Wt. of Pan Only	(g)	3.7	3.7
Wt. of Dry Soil	(g)	85.4	85.4
Moisture Content	%	75.4	34.1
Dry Density	PCF	71.3	97.2
Max. Dry Density	PCF		
Percent Compaction			

LOAD (PSF)	CONSOL. (IN.)
30	0.0696
100	0.1018
200	0.1188
400	0.1392
800	0.1607
1600	0.1838
3200	0.2087
6400	0.2302
12800	0.2666
25600	0.2974
6400	0.2912
1600	0.2813
400	0.2736
100	0.2664

LOAD (PSF)	LOG p	CONSOL. (IN.)	DEFL. (IN.)	VOID RATIO e
30	1.477	0.0696	-0.0696	1.7060
100	2.000	0.1018	-0.1018	1.6124
200	2.301	0.1188	-0.1188	1.5630
400	2.602	0.1392	-0.1392	1.5036
800	2.903	0.1607	-0.1607	1.4411
1600	3.204	0.1838	-0.1838	1.3740
3200	3.505	0.2087	-0.2087	1.3015
6400	3.806	0.2302	-0.2302	1.2390
12800	4.107	0.2666	-0.2666	1.1332
25600	4.408	0.2974	-0.2974	1.0436
6400	3.806	0.2912	-0.2912	1.0617
1600	3.204	0.2813	-0.2813	1.0904
400	2.602	0.2736	-0.2736	1.1128
100	2.000	0.2664	-0.2664	1.1338

Sp. Gr. (g/cc)	3.320
Init. Ht. Solids (cm)	0.874
Init. Ht. Voids (cm)	1.667
Init. Void Ratio	1.9084

TIME READING DATA Fine

800 psf Load

Elapsed Time (min)	SQRT Time (min)	Dial Reading (in)	Defl. (in)
0.0	0.00	0.0706	0.0000
0.1	0.32	0.0789	-0.0083
0.3	0.50	0.0828	-0.0122
0.5	0.71	0.0848	-0.0142
1.0	1.00	0.0861	-0.0155
2.0	1.41	0.0869	-0.0163
4.0	2.00	0.0878	-0.0172
9.0	3.00	0.0887	-0.0181
16.0	4.00	0.0893	-0.0187
30.0	5.48	0.0899	-0.0193
60.0	7.75	0.0906	-0.0200
120.0	10.95	0.0913	-0.0207
240.0	15.49	0.0920	-0.0214
480.0	21.91	0.0926	-0.0220

1600 psf Load

Elapsed Time (min)	SQRT Time (min)	Dial Reading (in)	Defl. (in)
0.0	0.00	0.0931	0.0000
0.1	0.32	0.1049	-0.0118
0.3	0.50	0.1080	-0.0149
0.5	0.71	0.1093	-0.0162
1.0	1.00	0.1102	-0.0171
2.0	1.41	0.1110	-0.0179
4.0	2.00	0.1118	-0.0187
9.0	3.00	0.1126	-0.0195
16.0	4.00	0.1133	-0.0202
30.0	5.48	0.1140	-0.0209
60.0	7.75	0.1147	-0.0216
120.0	10.95	0.1151	-0.0220
240.0	15.49	0.1161	-0.0230
490.0	22.14	0.1167	-0.0236

3200 psf Load

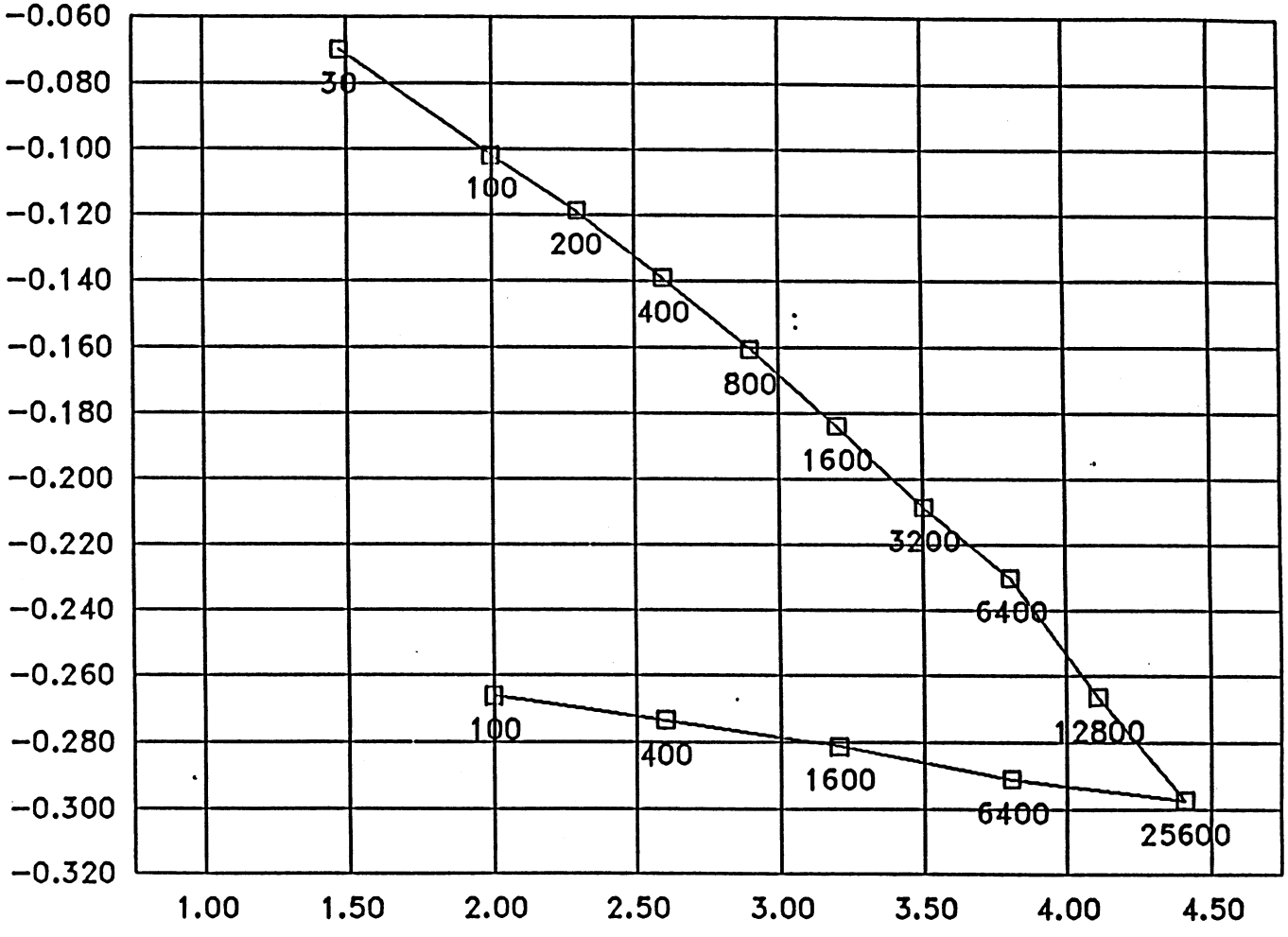
Elapsed Time (min)	SQRT Time (min)	Dial Reading (in)	Defl. (in)
0.0	0.00	0.1174	0.0000
0.1	0.32	0.1329	-0.0155
0.3	0.55	0.1350	-0.0176
0.5	0.71	0.1359	-0.0185
1.0	1.00	0.1368	-0.0194
2.0	1.41	0.1375	-0.0201
4.0	2.00	0.1382	-0.0208
9.0	3.00	0.1391	-0.0217
16.0	4.00	0.1398	-0.0224
30.0	5.48	0.1404	-0.0230
60.0	7.75	0.1412	-0.0238
120.0	10.95	0.1419	-0.0245
265.0	16.28	0.1426	-0.0252
480.0	21.91	0.1431	-0.0257

6400 psf Load

Elapsed Time (min)	SQRT Time (min)	Dial Reading (in)	Defl. (in)
0.0	0.00	0.1438	0.0000
0.1	0.32	0.1482	-0.0044
0.3	0.50	0.1561	-0.0123
0.5	0.71	0.1585	-0.0147
1.0	1.00	0.1596	-0.0158
2.0	1.41	0.1604	-0.0166
4.0	2.00	0.1612	-0.0174
9.0	3.00	0.1622	-0.0184
16.0	4.00	0.1628	-0.0190
30.0	5.48	0.1635	-0.0197
60.0	7.75	0.1643	-0.0205
120.0	10.95	0.1649	-0.0211
275.0	16.58	0.1655	-0.0217

CONSOLIDATION TEST DATA

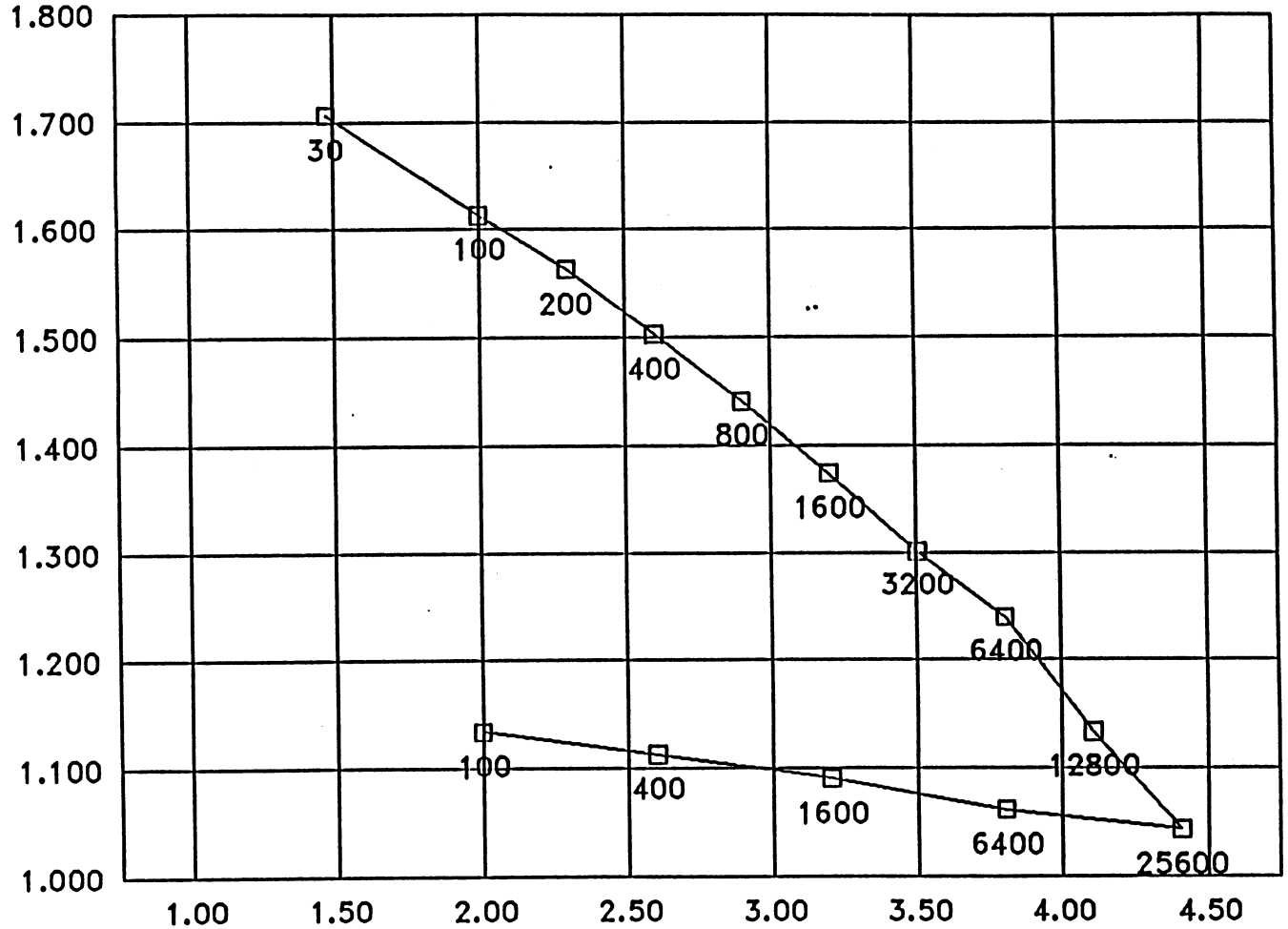
Fine Crandon



CONSOL (IN./IN.)

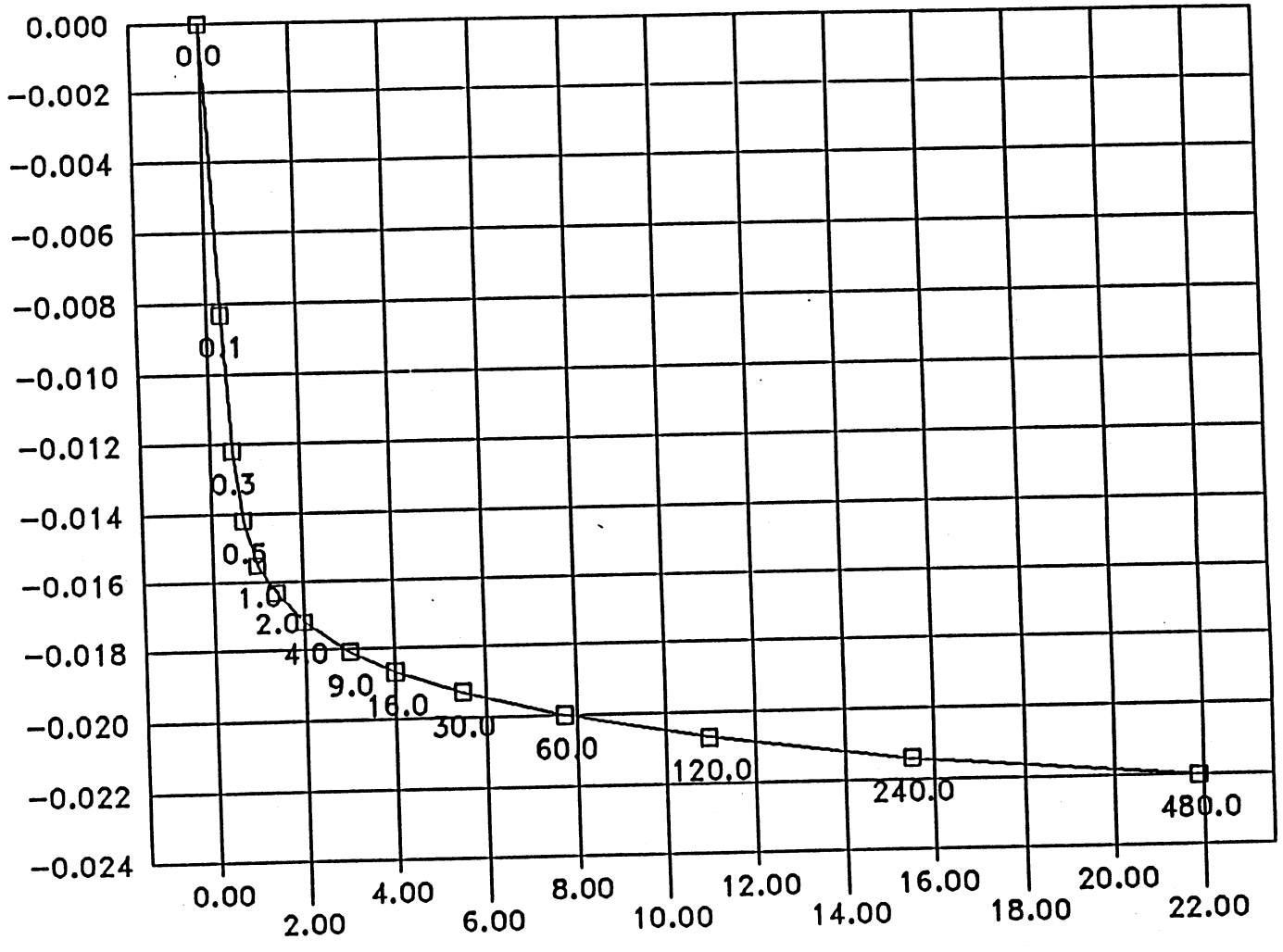
CONSOLIDATION TEST DATA

Fine Crandon



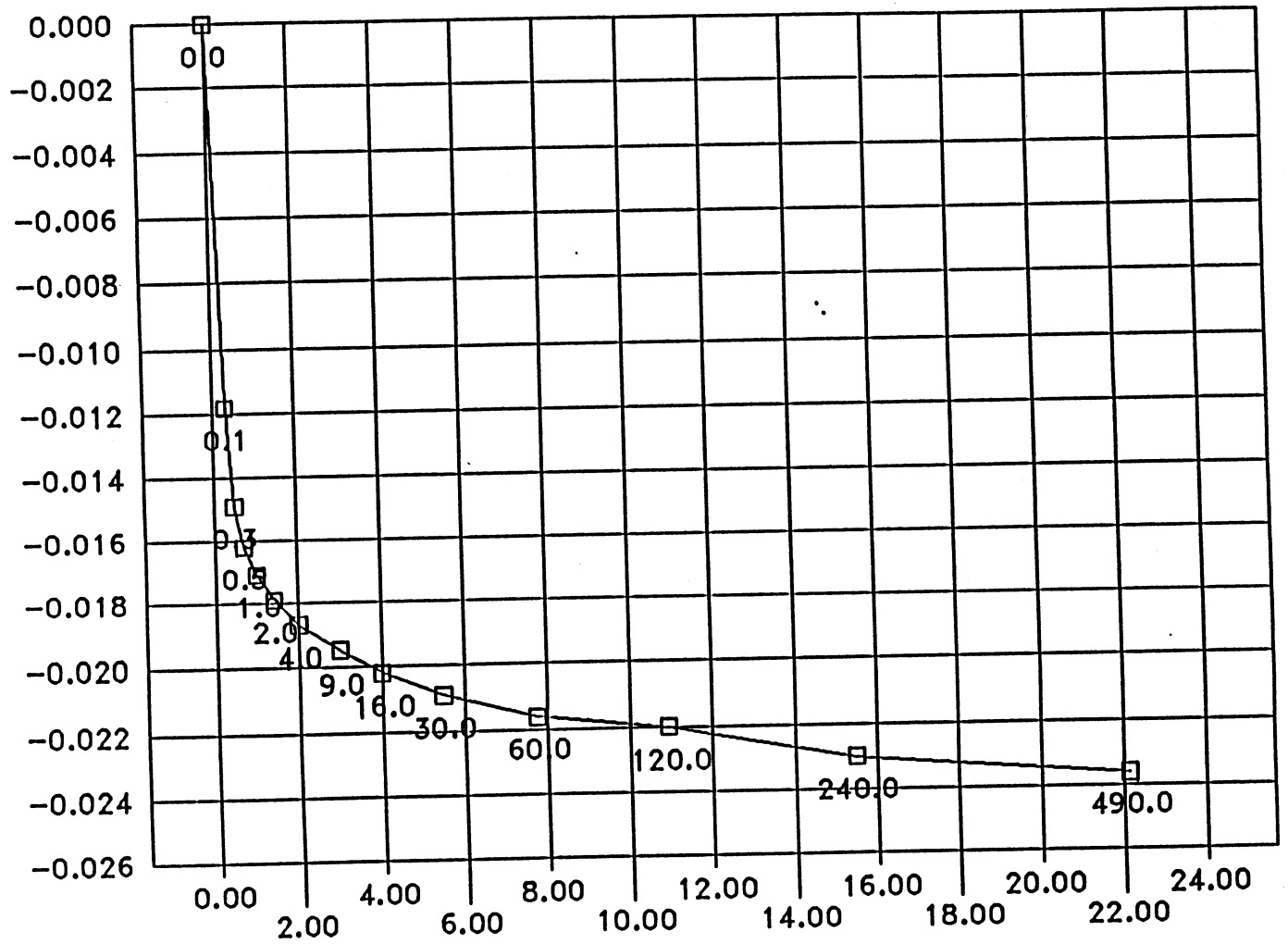
TIME READING DATA

Fine Crandon 800 psf Load



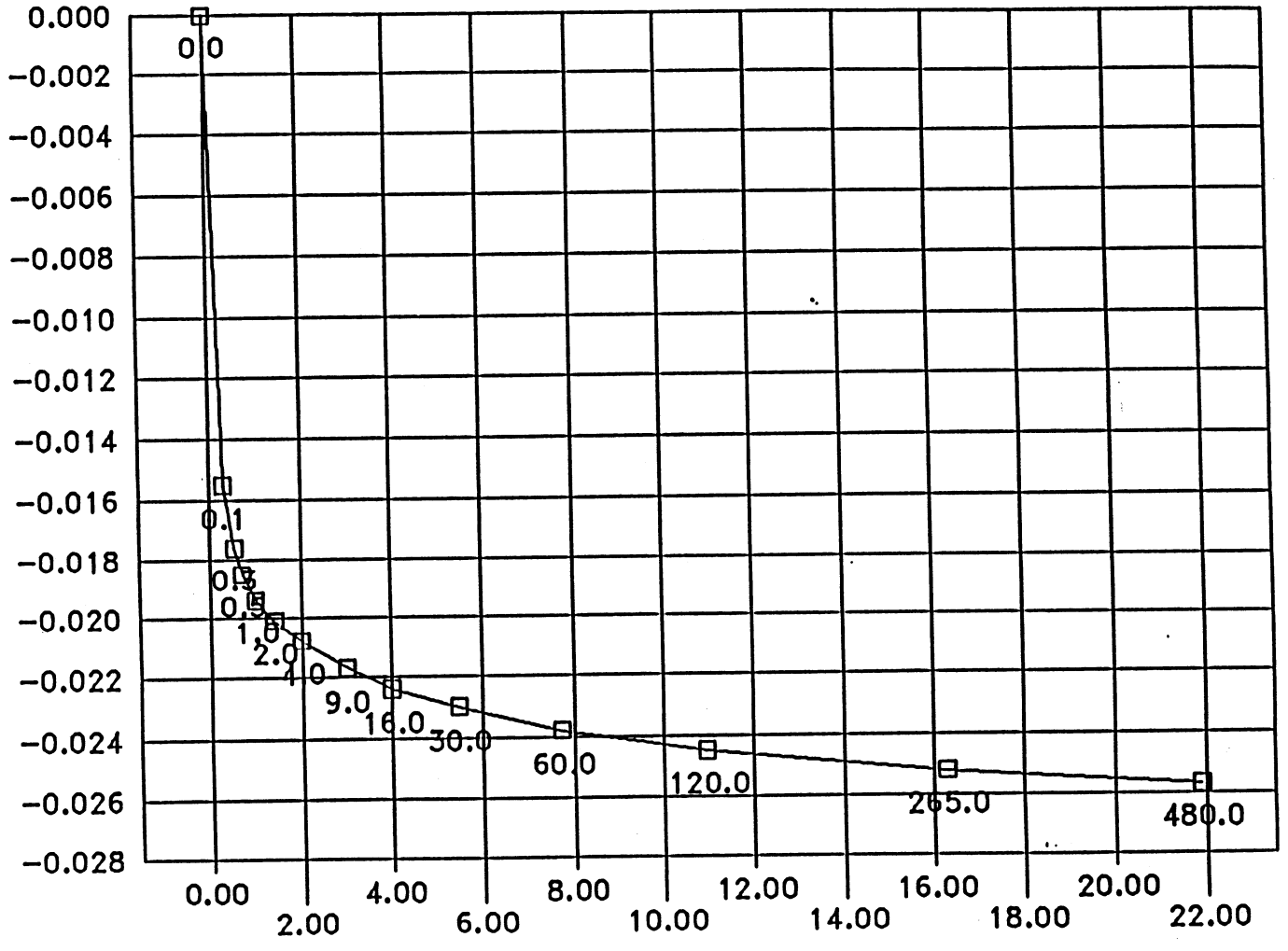
TIME READING DATA

Fine Crandon 1600 psf Load



TIME READING DATA

Fine Crandon 3200 psf Load



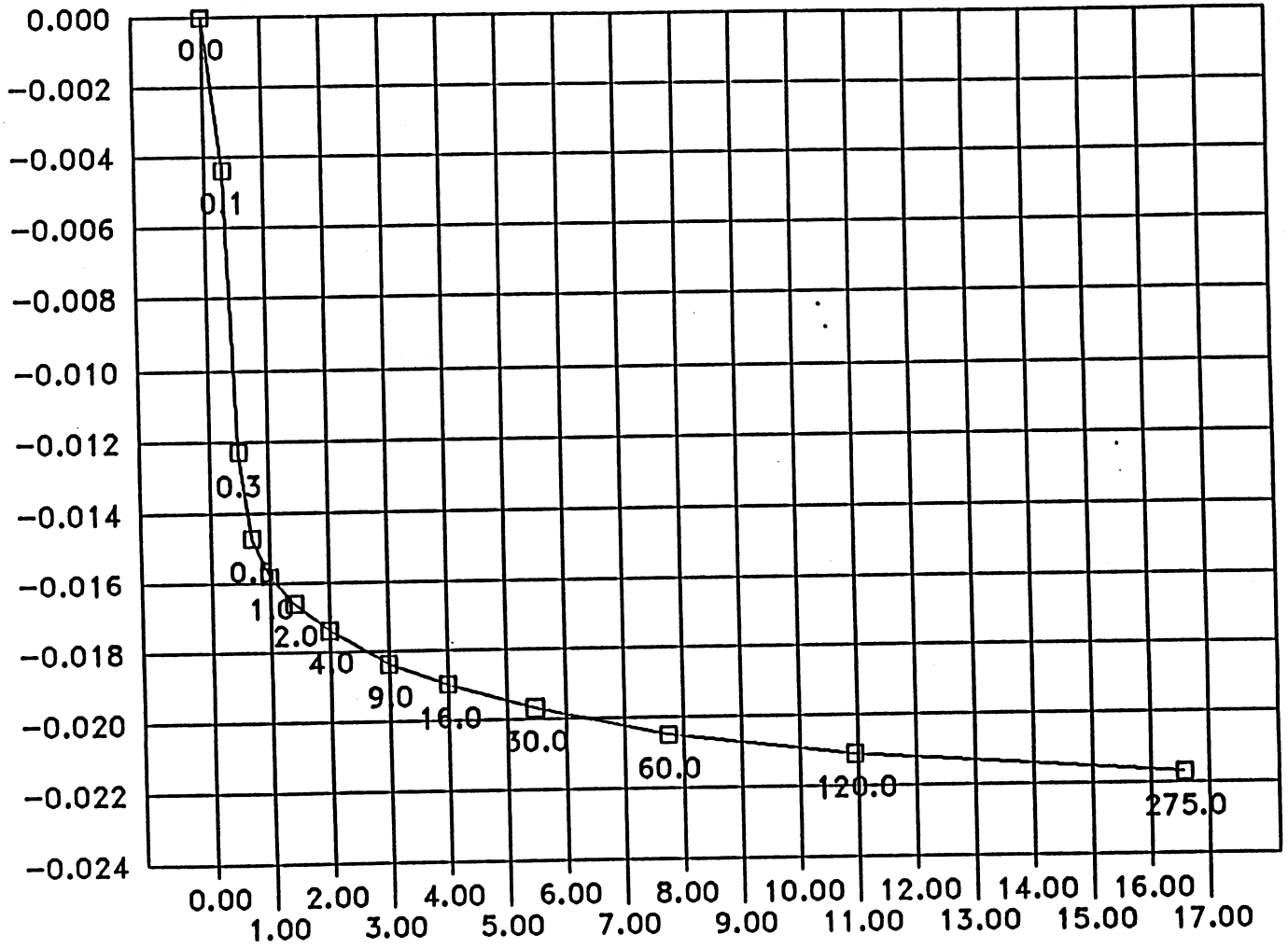
Deflection in inches

SQUARE ROOT of TIME

□ Time in Minutes

TIME READING DATA

Fine Crandon 6400 psf Load



Deflection in inches

SQUARE ROOT of TIME

□ Time in Minutes

CONSOLIDATION TEST DATA SHEET

SRCEFI
SRCEFI 1 2 3 4

SRCEFI

JOB SRK — JOB NO. 2030-16

DATE SAMPLED CRANDON — BORING NO. FINE

TEST STARTED 7-21-94 BY CW/SR TEST FINISHED ____ BY ____ DEPTH ____

SOIL TYPE TAILINGS

CONSOLIDOMETER NO. ATT-97 — INITIAL SAMPLE HEIGHT 1.00" = 2.54cm A_o = 29.431 cm^2 SG = 3.32

MOISTURE & DENSITY DATA	BEFORE TEST	AFTER TEST
WT. OF RING(S)+ SOIL	194.09 / 149.62	152.87
WT. OF RING(S) ONLY	44.41	
WT. OF SOIL (NET)		
WET DENSITY IN LBS/CU. FT.*		
WT. OF WET SOIL + DISH NO. 52		
INT. OF DRY SOIL + DISH		
WT. OF LOST MOISTURE		
WT. OF DISH ONLY	3.68	3.68
WT. OF DRY SOIL		
MOISTURE AS % OF DRY WT.		
DRY DENSITY IN LBS/CU. FT.		

TEST INSTRUCTIONS

LOAD CYCLE 30 100 200 400 800 TR 1600 TR 3200 6400 TR 12800 25600 ____

SPECIAL INSTRUCTION AND NOTES

DRY WT. SOIL + PAN + RING = 133.4
WS = 85.35
Hs = 85.35 / 3.32 / 29.431 = .8735
Hv = 2.54 - .8735 = 1.667
eo = 1.667 / .8735 = 1.908
FH = 1.00" - .2664 = .7336"

DATE	TIME	E-LAPSED TIME IN MINS.	LOAD (PSF)	DIAL READING (.0001 IN.)	CONSOL UNDER THIS LOAD	TOTAL CONSOL-IDATION
7-21-94	1641		30	.0000		
	1642	1		.0077		
	1643	2		.0126		
	1647	6		.0264		
	1656	15		.0430		
	1745			.0655		
7-22-94	0807			.0668		
	0929			.0681		
	1048			.0696		.0696
			100			NEW ZERO!
	1049	1		.0171		
	1050	2		.0237		
	1146			.0314		
	1423			.0320		
	1607			.0322		.1018
			200			
	1608	1		.0417		
	1609	8		.0437		
7-23-94	0847			.0495		
	1016			.0496	.1192	.1188
			400			
	1019	1		.0634		
	1020	2		.0647		
	1025			.0661		
	1110			.0680		
	1325			.0692		
	1519			.0697		
	1713			.0699		
7-24-94	0906			.0706	.1402	.1392

DATE	TIME	E-LAPSED TIME IN MINS.	LOAD (PSF)	DIAL READING (.0001 IN.)	CONSOL UNDER THIS LOAD	TOTAL CONSOLIDATION
7-24-94			800			
		.1		.0789		
		.25		.0828		
		.5		.0848		
	0912	1		.0861		
	0913	2		.0869		
	0915	4		.0878		
	0920	9		.0887		
	0927	16		.0893		
	0941	30		.0899		
	1011	60		.0906		
	1111	120		.0913		
	1311	240		.0920		
	1711	480		.0926		
7-25-94	0830			.0931	.1627	.1607
			1600			
		.1		.1049		
		.25		.1080		
		.5		.1093		
	0831	1		.1102		
	0832	2		.1110		
	0834	4		.1118		
	0839	9		.1126		
	0846	16		.1133		
	0900	30		.1140		
	0930	60		.1147		
	1030	120		.1151		
	1230	240		.1161		
	1630	480	490	.1167		
7-26-94	0825			.1174	.1870	.1837

ADVANCED TERRA TEST

CONTINUED ON REVERSE SIDE

FINE TAILINGS

IDATION TEST DATA SHEET (CONTINUED)

TIME	E-LAPSED TIME IN MINS.	LOAD (PSF)	DIAL READING (.0001 IN.)	CONSOL UNDER THIS LOAD	TOTAL CONSOL-IDATION
	.1	3200	.1329		
	25		.1350		
	.5		.1359		
2826	1		.1368		
2827	2		.1375		
10829	4		.1382		
0834	9		.1391		
841	16		.1398		
0855	30		.1404		
0925	60		.1412		
1025	120		.1419		
1225	240	265	.1426		
1625	480		.1431		
74 809			.1438		
0920			.1438	.2134	.2087
9:50	0	6400			
	.1		1482		
	.25		.1561		
	.5		.1585		
9:51	1		.1596		
9:52	2		.1604		
9:54	4		.1612		
9:59	9		.1622		
10:06	16		.1628		
10:20	30		.1635		
10:50	60		.1643		
11:50	120		.1649		
5 ~~3:50~~	~~240~~	275	.1655		
17:50	480				
74 0814			.1663		
0935			.1664	.2360	.2302
		12800			
0936	1		.1985		
0937	2		.1993		
1339			.2045		
1501			.2047	.2743	.2666
		25600			
1502	1		.2309		
1503	2		.2316		
-94 0821			2377		
0934			.2378	.3074	.2974
		6400			
0935	1		.2314		
0936	2		.2314		
1041			.2310		
1221			.2310	.3006	.2912
		1600			
1222	1		.2231		
1223	2		.2229		
1417			.2220		
1603			.2219	.2915	.2813
		400			
1604	1		.2138		
1605	2		.2135		
1649			.2122		
-30-94 0815			.2111	.2807	.2736

DATE	TIME	E-LAPSED TIME IN MINS.	LOAD (PSF)	DIAL READING (.0001 IN.)	CONSOL UNDER THIS LOAD	TOTAL CONSOL-IDATION
7-30-94			100			
	1207	1		.2066		
	1208	2		.2062		
7-31-94	0955			.2035		
8-1-94	0812			.2033	.2729	.2664

STEFFEN ROBERTSON AND KIRSTEN
Consulting Engineers and Scientists

MEMORANDUM

TO: Rob Dorey

FR: Troy Meyer

RE: Laboratory Test Results for Crandon Tailings

DATE: January 25, 1995

SRK Project No. 71501

Fine tailings samples were tested to determine the capillary-moisture relationship using a ceramic plate extractor. The moisture retention tests were conducted in accordance with ASTM D2325 test method which covers the determination of capillary-moisture relationships for coarse- and medium-textured soils.

Samples of fine tailings (-#200 sieve) were remolded at target dry densities of 70 and 90 pcf. The samples were then saturated and placed in contact with a saturated porous ceramic plate installed inside a pressure chamber. Duplicates were prepared for each sample and were placed diametrically opposed to each other on the ceramic plate. The bottom of the plate is covered with a rubber membrane and is maintained at atmospheric pressure during the test by means of a small drain tube through the side of the pressure chamber. Upon applying a pressure to the chamber, a pressure drop is created across the plate and the saturated soil samples establish equilibrium with the water in the plate. The water, held at a tension less than the pressure drop across the ceramic plate, will then move out of the soil, through the plate, and out through the drain tube. Once the water has stopped flowing, the drain tube is clamped, the pressure is released and the weight of the wet sample is determined. This process is repeated for several pressures (soil-water tensions) and the dry weight of the sample is determined at the conclusion of the test. Moisture contents of the samples were determined for each pressure and a capillary-moisture curve was prepared for each sample. Pressures of 0.3, 0.5, 1, 2, 5, and 10 bars were used and were plotted with the corresponding volumetric moisture contents.

Upon review of the test data, the calculated dry densities of the duplicate samples varied due to swell in the samples during saturation. A specific gravity of 3.32 was used to calculate the volume of each sample in the saturated condition. It was assumed that the volume remained constant throughout the test.

The field capacity is defined as the volumetric water content of a soil or waste layer at a capillary head of 0.33 bars and refers to the water content after a prolonged period of gravity

drainage. This value was found for each sample by interpolation and is reported for each sample. The wilting point is the volumetric water content of a soil or waste layer at a capillary pressure of 15 bars and refers to the water content that can be achieved by plant transpiration. This value was found by extrapolation and is reported for each sample.

The following table summarizes the field capacity and wilting point values found for each sample.

SUMMARY OF MOISTURE RETENTION TEST RESULTS

Sample	Dry Density (pcf)	Field Capacity		Wilting Point	
		% Vol.	% Dry Mass	% Vol.	% Dry Mass
#1	61.0	43.5	44.5	27.16	27.8
#1 (R)	67.5	46.5	43.0	29.1	26.9
#2	69.9	47.4	42.3	28.1	25.1
#2 (R)	75.3	49.2	40.8	30.5	25.3

The test data and results of the capillary moisture retention tests are shown on the included test data sheet. The capillary moisture curves are also included on the following pages.

CAPILLARY MOISTURE RETENTION TEST
ASTM D 2325

Project: Crandon
Project No: F107101

Started: 11/2/94
Finished: 11/23/94

Tested By: MC
Checked By: TM

Sample ID: FINE TAILINGS

Sample	Ring Mass g	Ring Volume cc	Initial Mass g	Sat. Mass g	0.3 Bar Mass g	0.5 Bar Mass g	1 Bar Mass g	2 Bar Mass g	5 Bar Mass g	10 Bar Mass g	Dry Mass Filter + Ring + Dish g	Dish Wt. g
Filter Only			0.368	0.820	0.684	0.614	0.570	0.522	0.513	0.499	4.568	4.200
#1	3.799	23.4352	27.127	44.060	37.621	37.296	36.866	35.652	34.361	33.959	31.305	4.232
#1(R)	3.485	21.3163	26.960	41.740	37.177	36.948	36.485	35.560	33.881	33.560	31.105	4.185
#2	3.680	25.8204	32.910	50.535	45.573	45.229	44.638	43.385	42.419	41.387	37.152	4.177
#2(R)	3.789	24.4022	33.202	49.579	45.959	45.516	44.790	43.481	42.571	41.861	37.805	4.216

Sample	Dry Mass g	Unit Wt. g/cc	Sat. Mass g	Total H20 g	Sat. MC		0.3 Bar			0.5 Bar		
					% DM	% Vol.	Ret H20	% DM	% Vol.	Ret H20	% DM	% Vol.
#1	22.906	0.977	39.441	16.535	72.19%	70.56%	10.232	44.67%	43.66%	9.977	43.56%	42.57%
#1(R)	23.067	1.082	37.435	14.368	62.29%	67.40%	9.941	43.10%	46.64%	9.782	42.41%	45.89%
#2	28.927	1.120	46.035	17.108	59.14%	66.26%	12.282	42.46%	47.57%	12.008	41.51%	46.51%
#2(R)	29.432	1.206	44.970	15.538	52.79%	63.67%	12.054	40.96%	49.40%	11.681	39.69%	47.87%

Sample	1 Bar			2 Bar			5 Bar			10 Bar		
	Ret H20	% DM	% Vol.	Ret H20	% DM	% Vol.	Ret H20	% DM	% Vol.	Ret H20	% DM	% Vol.
#1	9.591	41.87%	40.93%	8.425	36.78%	35.95%	7.143	31.18%	30.48%	6.755	29.49%	28.82%
#1(R)	9.363	40.59%	43.92%	8.486	36.79%	39.81%	6.816	29.55%	31.98%	6.509	28.22%	30.54%
#2	11.461	39.62%	44.39%	10.256	35.45%	39.72%	9.299	32.15%	36.01%	8.281	28.63%	32.07%
#2(R)	10.999	37.37%	45.07%	9.738	33.09%	39.91%	8.837	30.03%	36.21%	8.141	27.66%	33.36%

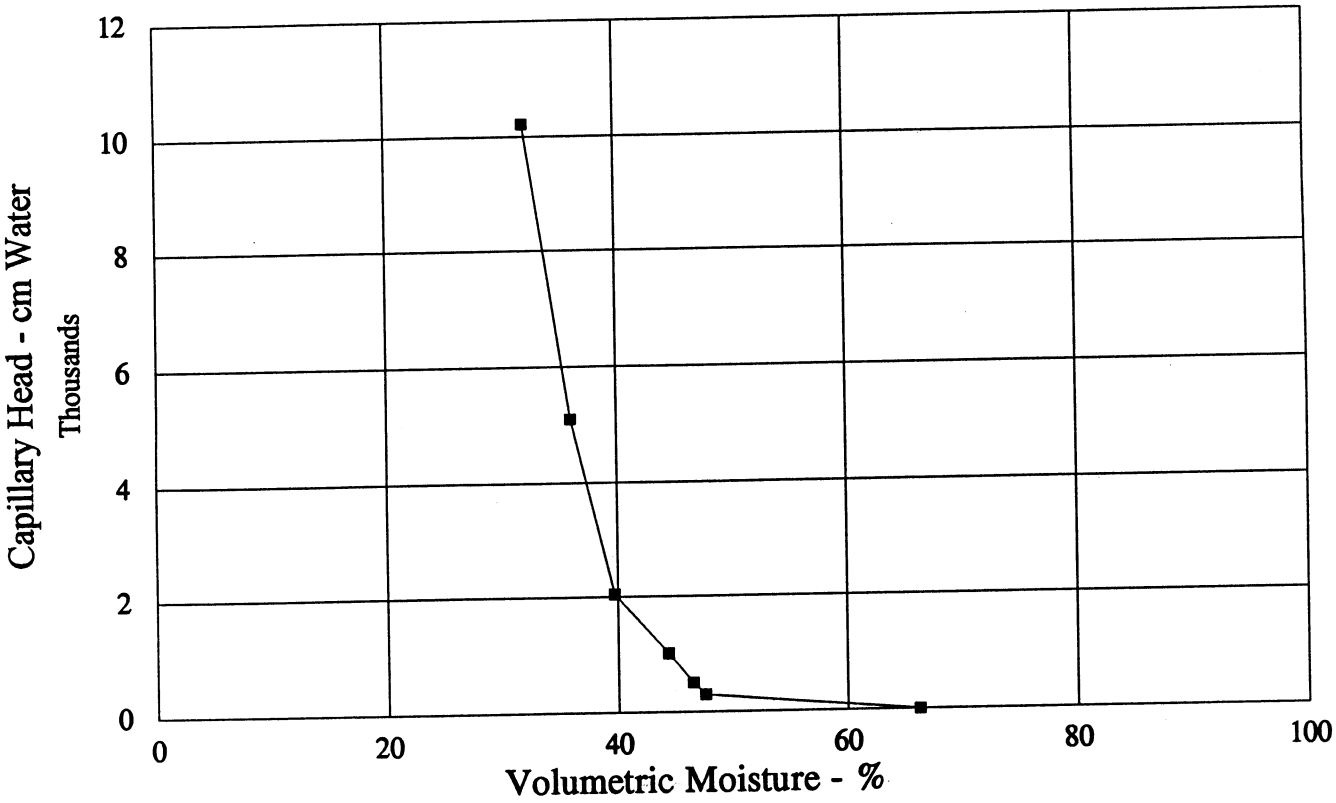
Sample	Vol. MC (%) Sat.	Vol. MC (%) 0.3 Bar	Vol. MC (%) 0.5 Bar	Vol. MC (%) 1 Bar	Vol. MC (%) 2 Bar	Vol. MC (%) 5 Bar	Vol. MC (%) 10 Bar
#1	70.56	43.66	42.57	40.93	35.95	30.48	28.82
#1(R)	67.40	46.64	45.89	43.92	39.81	31.98	30.54
#2	66.26	47.57	46.51	44.39	39.72	36.01	32.07
#2(R)	63.67	49.40	47.87	45.07	39.91	36.21	33.36

Heads Used							
Bar	0.0	0.3	0.5	1.0	2.0	5.0	10.0
cm H20	0.0	305.9	509.9	1019.7	2039.4	5098.6	10197.2

%DM = Moisture Content By Dry Mass
%Vol. = Volumetric Moisture Content

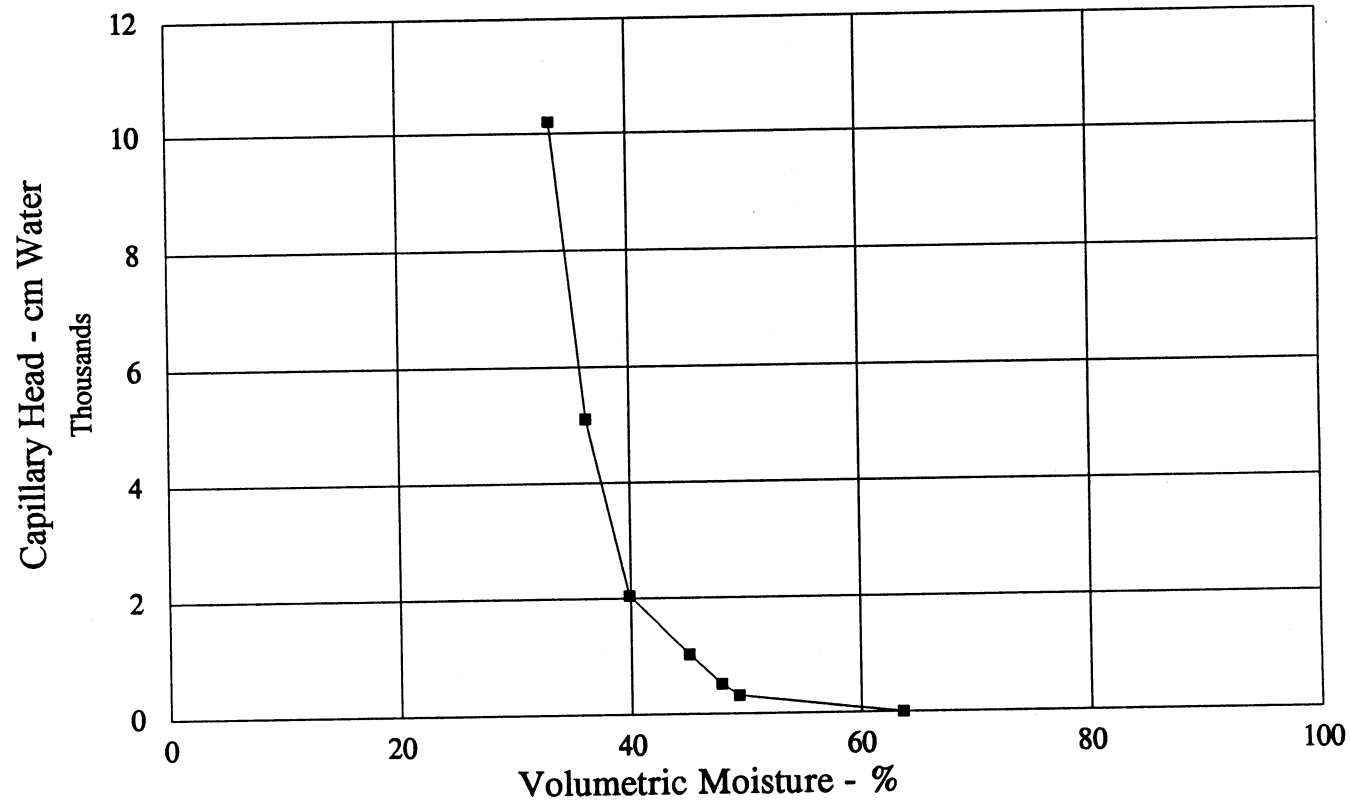
CAPILLARY MOISTURE CHARACTERISTIC CURVE

Fine Tailings #2



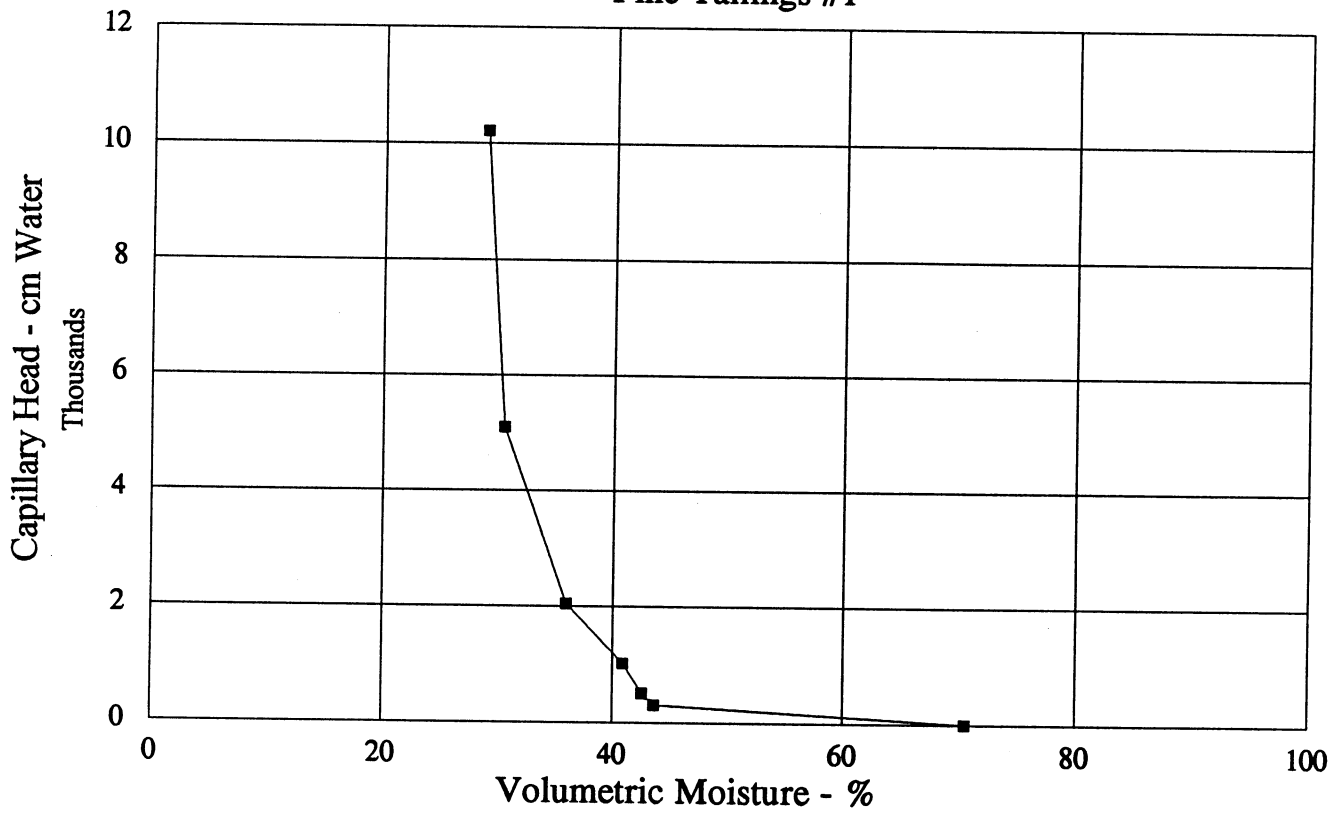
CAPILLARY MOISTURE CHARACTERISTIC CURVE

Fine Tailings #2 (R)



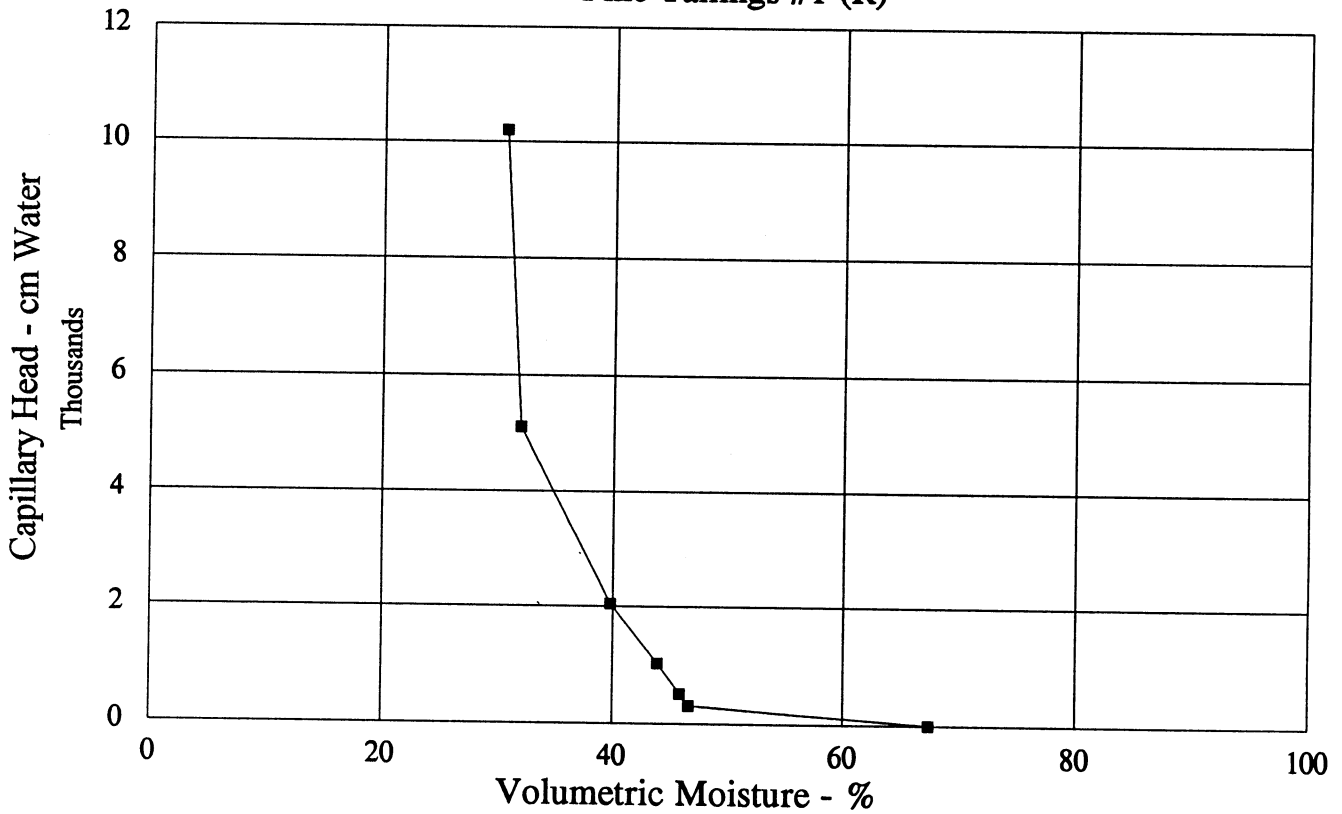
CAPILLARY MOISTURE CHARACTERISTIC CURVE

Fine Tailings #1



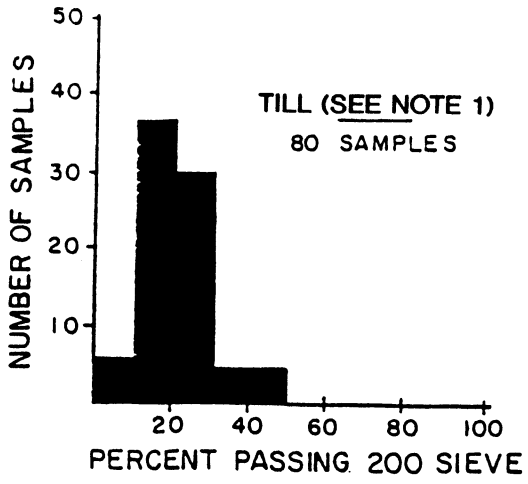
CAPILLARY MOISTURE CHARACTERISTIC CURVE

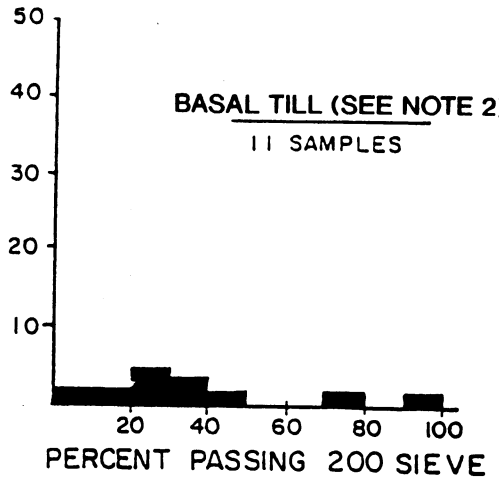
Fine Tailings #1 (R)

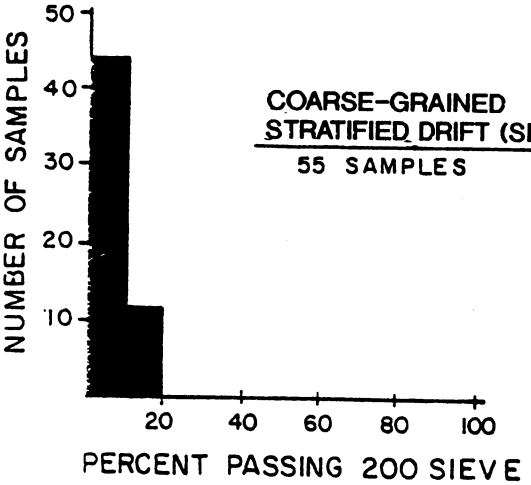


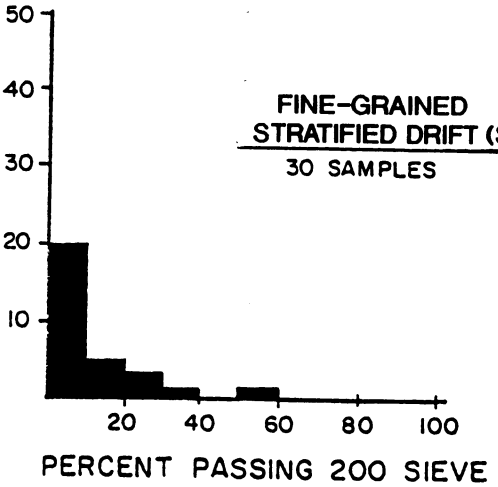
Appendix C

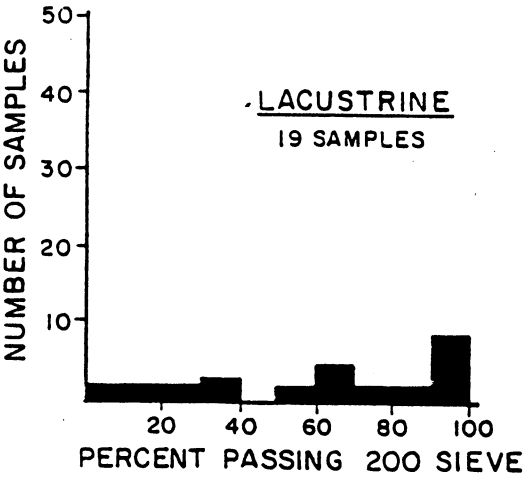
Summary of Soil Laboratory Test Data From Exxon 1985 Feasibility Report











NOTES:

1. CURRENTLY TERMED LATE WISCONSINIAN TILL
2. CURRENTLY TERMED PRE- OR EARLY WISCONSINIAN TILL
3. CURRENTLY TERMED COARSE GRAINED OUTWASH
4. CURRENTLY TERMED FINE GRAINED OUTWASH

SOURCE:
EXXON 1985 DATA (STS-84)

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	NXP	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

Crandon Mining Company

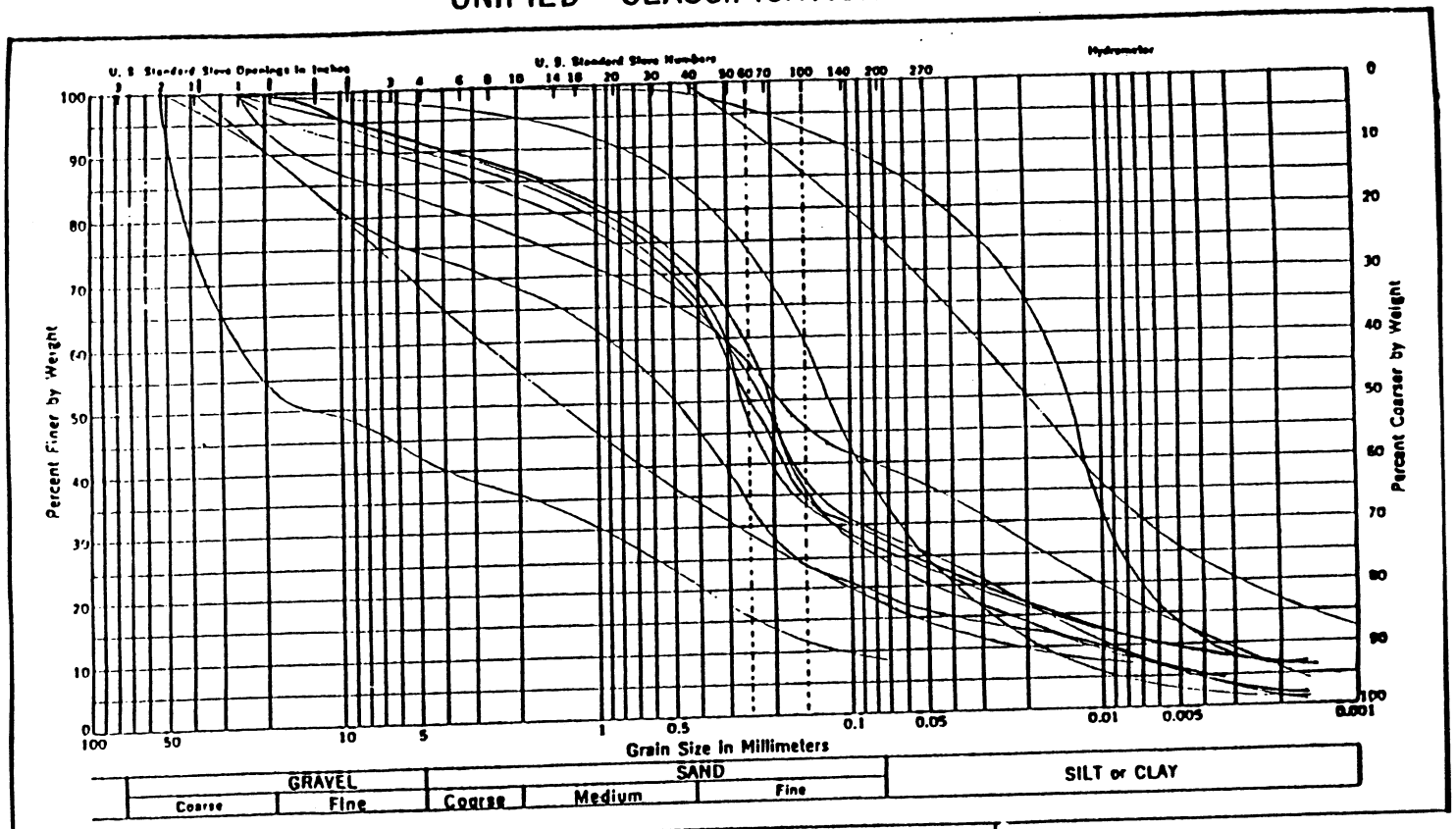
P200 DISTRIBUTION FOR VARIOUS GLACIAL DEPOSITS

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

/d3/dl/users/jprl/3c49f134.dgn

93C049

UNIFIED CLASSIFICATION SYSTEM



SAMPLE NO.	W.C.	LL	PL	PI	CLASSIFICATION
					PRE- TO EARLY WISCONSINAN TILL

REFERENCE: STS-1984

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	SVDI	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

Crandon Mining Company

REPRESENTATIVE GRAIN SIZE CURVES
PRE-TO EARLY WISCONSINAN TILL

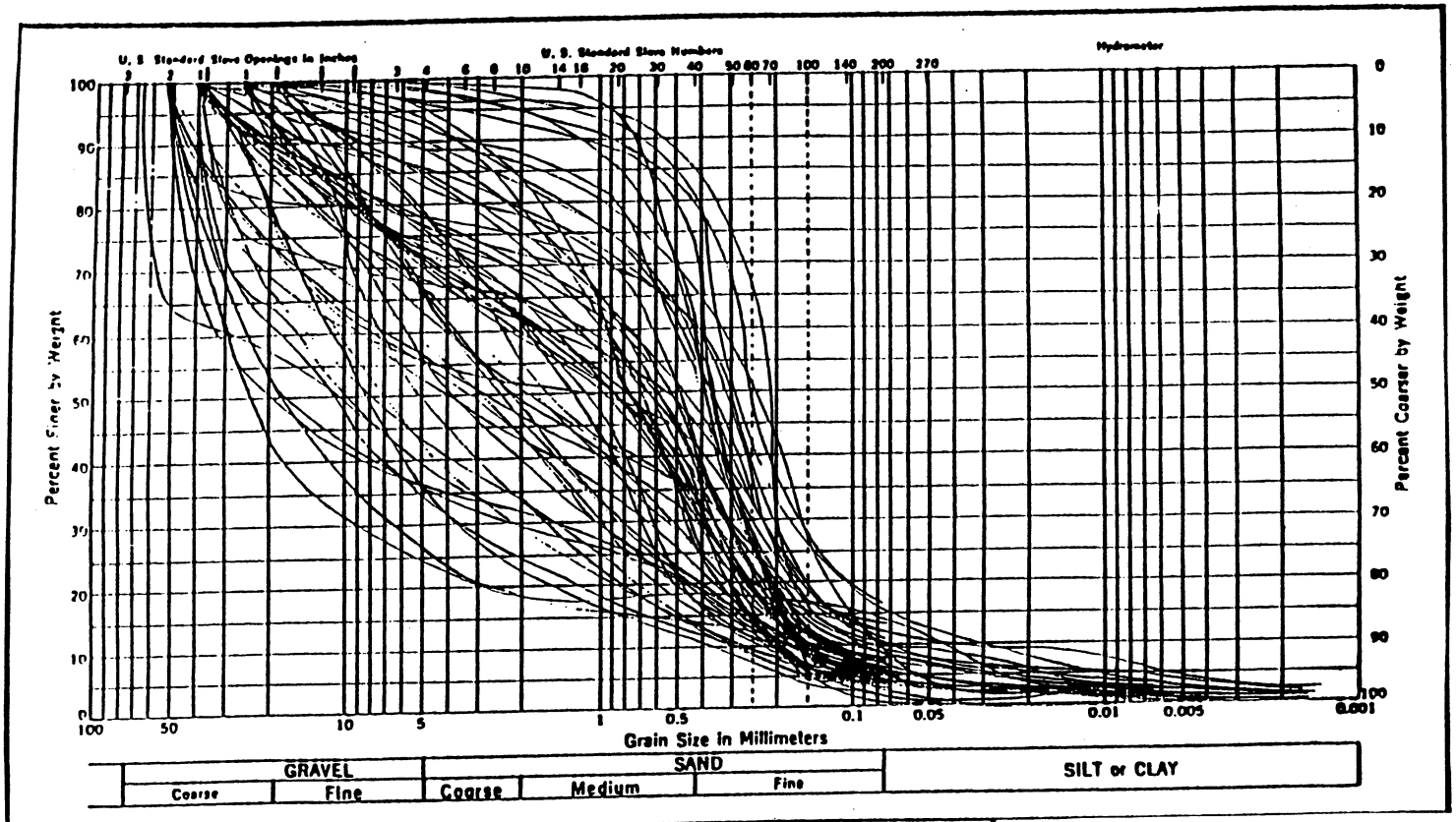
Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

c:\cadwork\4c33fig.dgn

C-2

93C049

UNIFIED CLASSIFICATION SYSTEM



SAMPLE NO.	W.C.	LL	PL	PI	CLASSIFICATION
					COARSE GRAINED OUTWASH
					SP, SW, SM-SP, SM, GP-GM, GP, GW, GM

REFERENCE: STS-1984

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	SVDI	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

Crandon Mining Company

REPRESENTATIVE GRAIN SIZE CURVES
OF COARSE GRAINED OUTWASH DEPOSITS

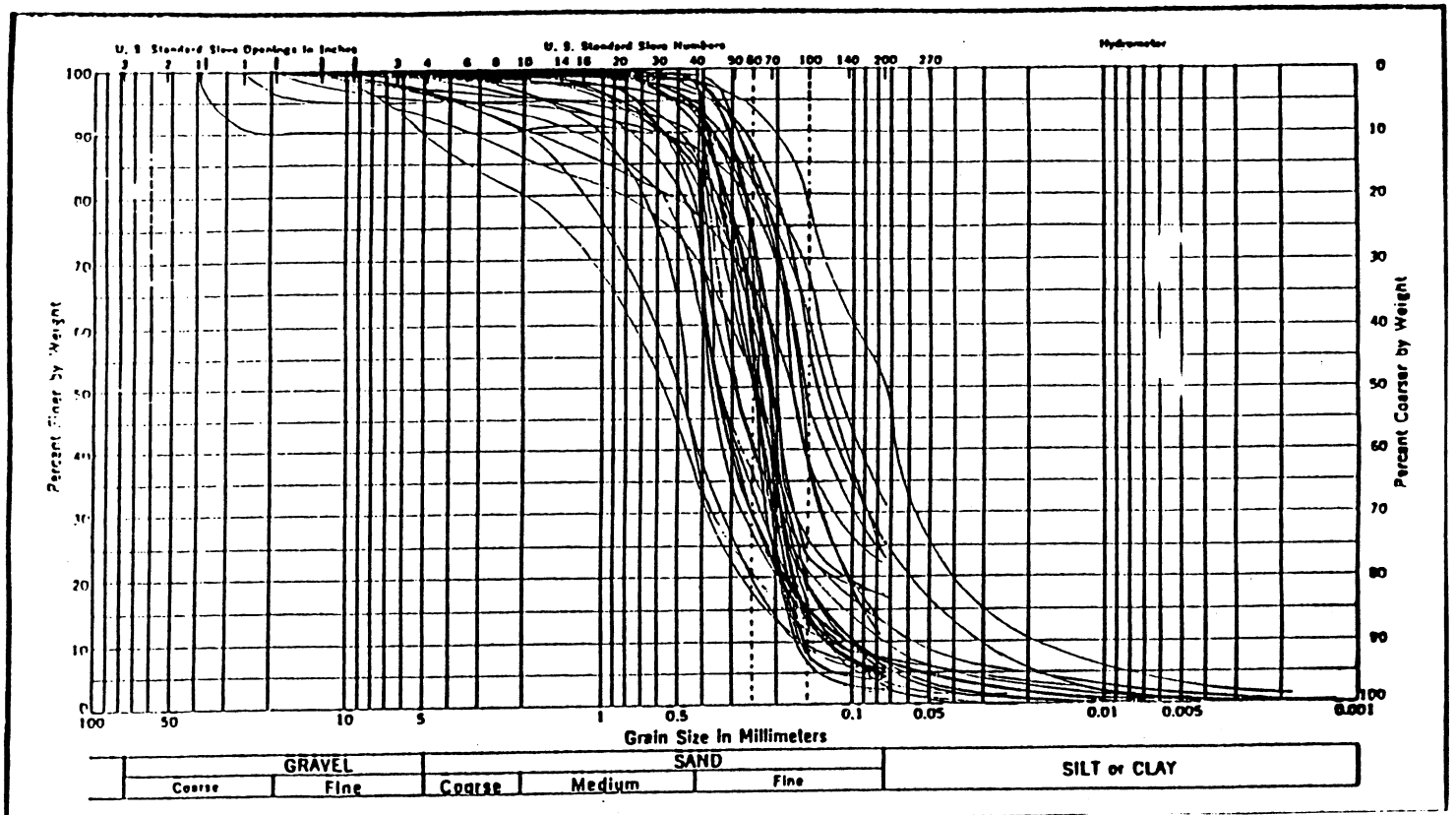
Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

c:\cadwork\4c33fig.dgn

C-3

93C049

UNIFIED CLASSIFICATION SYSTEM



SAMPLE NO.	W.C.	LL	PL	PI	CLASSIFICATION
					FINE GRAINED OUTWASH
					SP, SM-SP, SM, ML

REFERENCE: STS-1984

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	SVDI	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

Crandon Mining Company

REPRESENTATIVE GRAIN SIZE CURVES
OF FINE GRAINED OUTWASH DEPOSITS

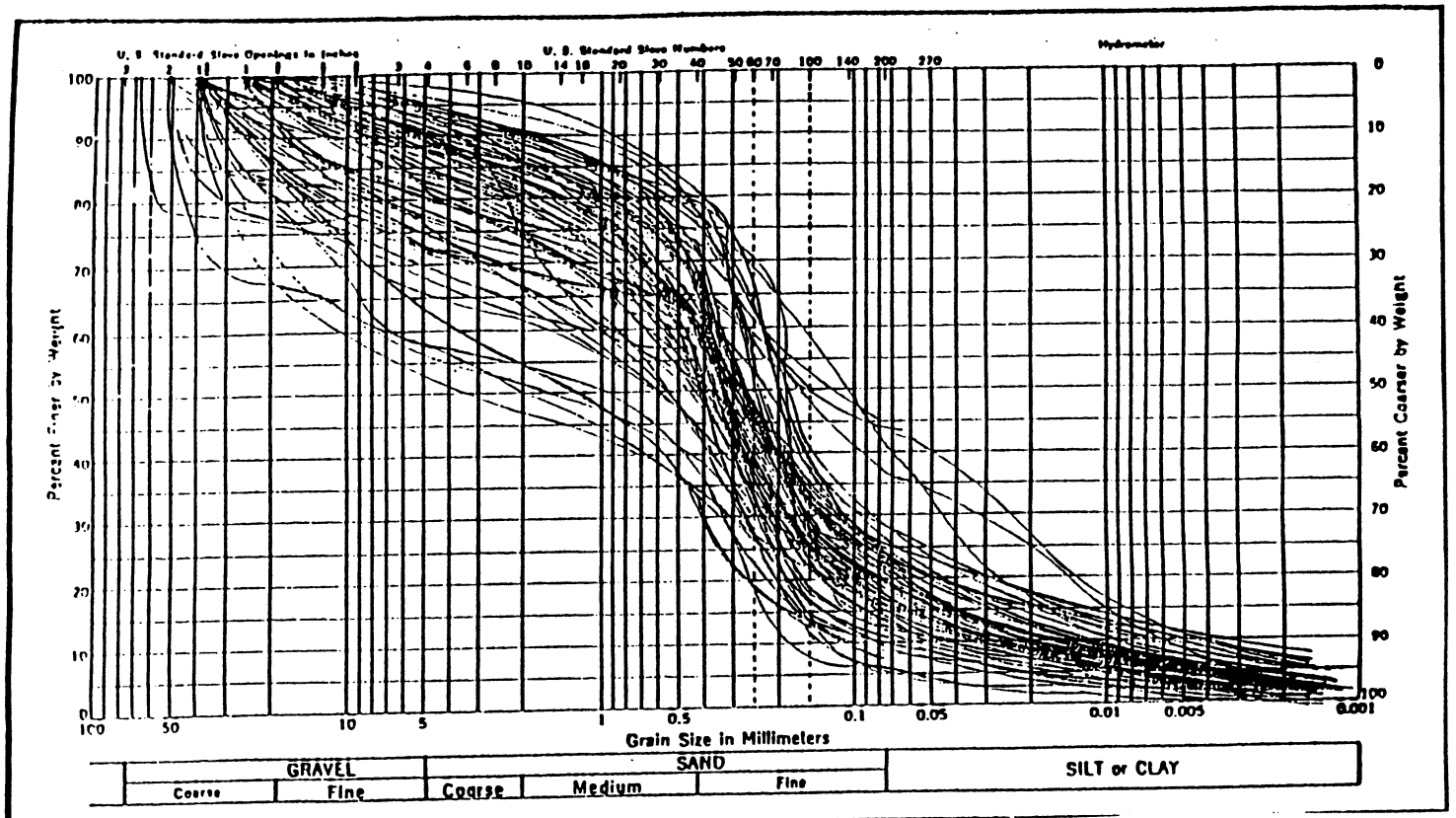
Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

c:\cadwork\4c33fig.dgn

C-4

93C049

UNIFIED CLASSIFICATION SYSTEM



SAMPLE NO.	W.C.	LL	PL	PI	CLASSIFICATION
					LATE WISCONSINAN TILL

REFERENCE: STS-1984

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	SVDI	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

Crandon Mining Company

REPRESENTATIVE GRAIN SIZE CURVES OF LATE WISCONSINAN TILL

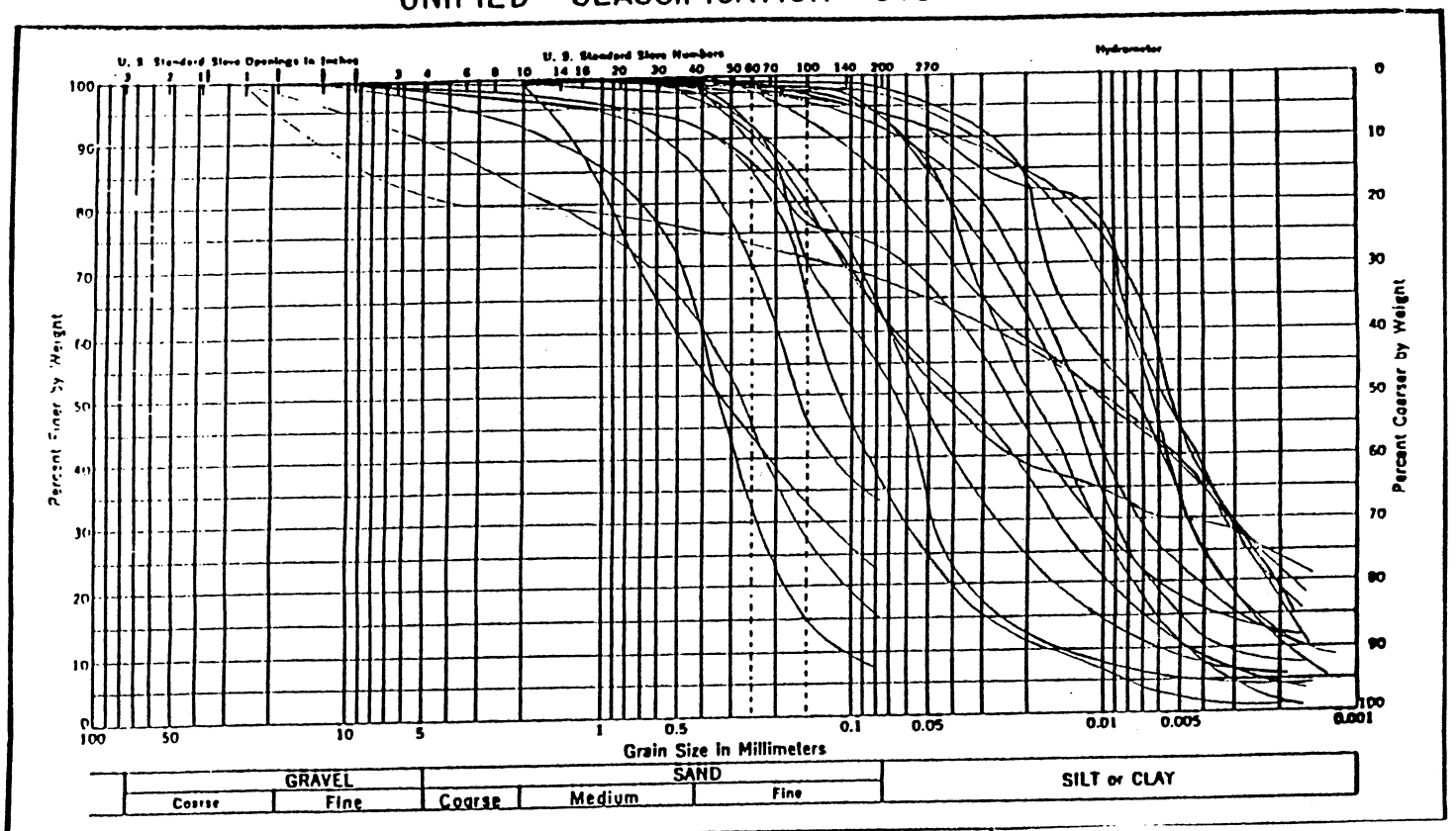
Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

c:\cadwork\4c33fig.dgn

C-5

93C049

UNIFIED CLASSIFICATION SYSTEM



SAMPLE NO.	W.C.	LL	PL	PI	CLASSIFICATION
					Lacustrine SM-SP, SM, ML, CL
					Note: Gradation Curves Represent
					Composite Samples and Not Individual
					Varves

Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: MAY '95
APPROVED BY:	SVDI	DATE: MAY '95
APPROVED BY:	GWS	DATE: MAY '95

Crandon Mining Company

REPRESENTATIVE GRAIN SIZE CURVES
OF LACUSTRINE DEPOSITS

Scale:	AS SHOWN	Date:	MAY, 1995
Prepared By:	Foth & Van Dyke	By:	BSH

REFERENCE: STS-1984

c:\cadwork\4c33fig.dgn

C-6

93C049

Appendix D

Geomembrane Design Calculations

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 1/4
Prepared by: NXP Date: 4/18/95
Checked by: J.B. Smith Date: 5/3/95

REQUIRED INTERFACE FRICTION PROPERTIES

TMA GEOMEMBRANE LINER.

Purpose: To determine the minimum interface friction properties between the geomembrane and the clay below and the geomembrane and the Geotextile above.

Scope: Determine the mobilized friction angles in the till layer and along the membrane - till interface to obtain a factor of safety of 1.2 against sliding of the Till layer placed on the geomembrane

CONCLUSION: The Minimum interface friction angle required is 21.2°.

In order to achieve this friction angle with the clay below and with the geotextile above, a textured membrane will be necessary

Foth & Van Dyke	Client: CMC	Scope I.D.: 93 C049
	Project: TMA	Page: 2/4
	Prepared by: NXP	Date: 4-18-95
	Checked by: [signature]	Date: 5/3/95

Method of Analysis and Assumptions

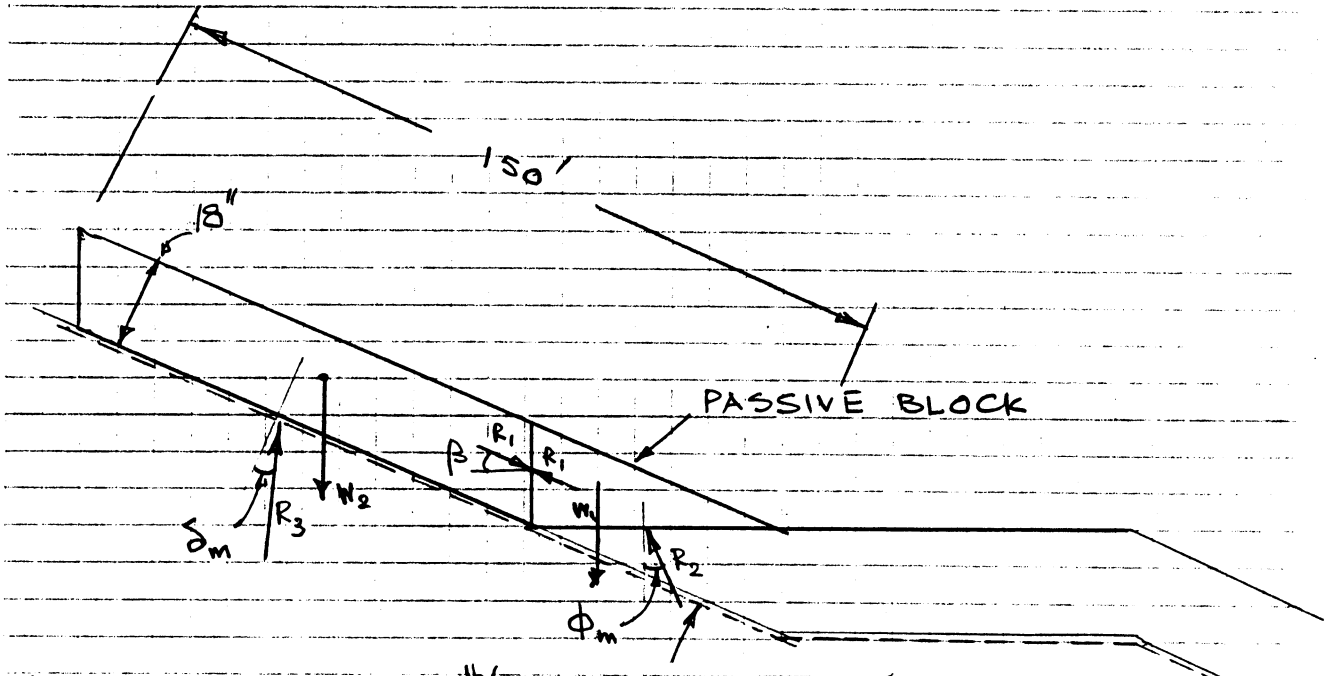
Assumptions – Slope Angle 18.43° (3H to 1V) ✓

Angle of internal friction of till = 32° ✓

Thickness of till cover = 1.5' ✓

Density of till layer = 120 pcf. ✓

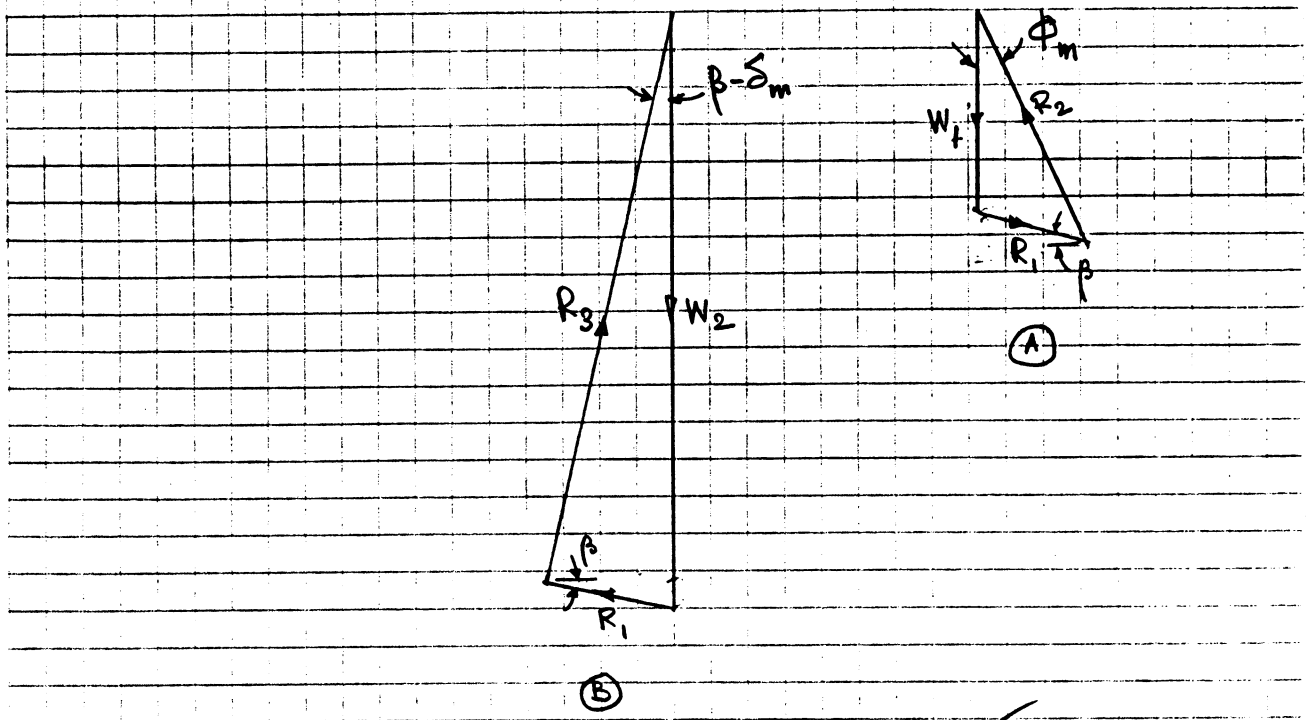
Maximum Length of slope = 150' ✓



$W_2 = 145 \times 1.5 \times 120 = 26100$ lb/ft Geomembrane ✓

$W_1 = \frac{4.5 \times 1.5 \times 120}{2} = 405$ lb/ft ✓

Force Polygons



From Force Polygon (A) $(W_1 + R_1 \sin\beta)\tan\phi_m = R_1 \cos\beta$ ✓

$R_1(\cos\beta - \sin\beta\tan\phi_m) = W_1\tan\phi_m$ ✓

$$R_1 = \frac{W_1\tan\phi_m}{\cos\beta - \sin\beta\tan\phi_m}$$ ✓

From Force Polygon (B) $$\tan(\beta-\delta_m) = \frac{R_1\cos\beta}{W_2 - R_1\sin\beta}$$ ✓ ✓

$$\tan\phi_m = \frac{\tan\phi}{1.2} = \frac{\tan 32^\circ}{1.2}, \quad \phi_m = 27.51^\circ$$ ✓

$$R_1 = \frac{405(.521)}{\cos 18.43^\circ - \sin 18.43^\circ \tan 27.51^\circ} = \frac{210.9}{.95 - .165} = \frac{265.3}{R_1}$$ ✓

$$\tan(\beta - \delta_m) = \frac{265.3\,(.95)}{26100 - 265.3(-.316)} = \frac{252.04}{26016}$$ ✓

$$= .00097$$ ✓

$$\therefore \beta - \delta_m = .56^\circ$$ ✓

$$\therefore \delta_m = \beta - .56 = 18.43 - .56$$
$$= 17.87^\circ$$ ✓

$$\delta = \tan^{-1}(1.2 \tan 17.87^\circ)$$

$$= \underline{\underline{21.2^\circ}}$$ ✓ 21.16°

Appendix E

Wave Action and Riprap Calculations

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA DESIGN Page: 1/11
Prepared by: REM Date: 2/20/95
Checked by: N.P Date: 4/25/95

DESIGN RIPRAP FOR EMBANKMENT SLOPES DURING FINAL TAILINGS DEPOSITION

GIVEN:

1. TMA 2, the largest cell has maximum fetch = 2600' (F) (see p. 11)
2. WATER DEPTH – min = 3', max = 5', AV = 4'
3. TMA 2, largest cell has maximum width = 600' at 1/3 of 1800' assuming partially filled w/ tailings (see pg. 11.)
4. maximum wind velocity = 90 mph (Thom, H.C.S., ASCE Tech. publication, July, 1968) for 100 yr. return period
5. assumed wind direction is parallel to maximum fetch SW – NE
6. Fetch Effectiveness for Rectangular Fetches

B = Average reservoir width / F = Straight line fetch = $\frac{600'}{2600'} = .23$

for $\frac{B}{F} = .23$, $\frac{F_E}{F} = .43$

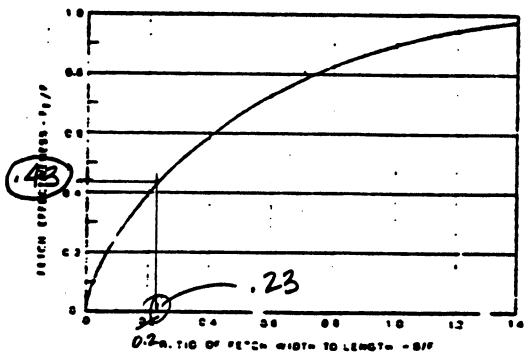


FIG. 14. Fetch width-length ratio vs. fetch effectiveness for rectangular fetches (after Saville 1954). Notes: for regular shaped reservoirs only; B = average reservoir width; F = straight line fetch; FF = effective fetch = fetch effectiveness × F.

(FROM Peters + Towle "Upstream Slope protection for Earthdams in Prairie Provinces", 1978)

FF (effective fetch) = fetch effectiveness × F
= .43 × 2600'
FF = 1118
SAY 1120 ft or .21 miles

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 2/11
Prepared by: REM Date: 2/20/95
Checked by: N.P Date: 4/25/95

Calculate WIND SETup (as per Water Resources Engineering, Linsley and Franzini 1964)

$$Z_s = \frac{V_w^2 FF}{1400 d} \quad \text{where}$$

Z_s = rise in feet above still water level
V_w = WIND SPEED (90 mph)
FF = EFFECTIVE FETCH (.21 miles)
D = Average depth (4 feet)

$$Z_s = \frac{(90 \text{ mph})^2 (.21)}{1400 (4)} = \frac{1701}{5600} = .30 \text{ feet}$$

Calculate Significant Wave Height (Z_w) where

$$Z_w = 0.034 V_w^{1.06} FF^{0.47}$$

$$Z_w = (.034) \underset{(117.9)}{(90)}^{1.06} \underset{(.568)}{(.30)}^{0.47}$$

$Z_w = 2.28$ say 2.3 Ft = the average height of the highest one-third of the waves

Maximum Wave Height (H_x)

$1.87(Z_w) = 1.87(2.3) = 4.3 \text{ ft} = H_x$

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 3/11
Prepared by: RBM Date: 2/28/95
Checked by: N.P Date: 4/25/95

Calculate Wave lenght λ

$\lambda = 5.12\ T_w^2$ where λ = wave length
T_w = wave period given by

$T_w = 0.46\ V_w^{0.44}\ F^{0.28}$

$T_w = 0.46\ (90\text{mph})^{0.44}\ (.21)^{0.28}$
$\quad\quad\quad

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: TMA	Page: 4/11
	Prepared by: REM	Date: 2/22/95
	Checked by: N.P	Date: 4/25/95

Calculate Design run-up

Design run-up (ft) = $H_x \frac{Z_n}{Z_w}$ where $H_x = 1.87 Z_w$

$\frac{Z_n}{Z_w} = .42$

$H_x = 2.3 ft (1.87)$

$H_x = 4.301$ say 4.3 ft.

(4.3 Ft) (.42) = Design run-up (ft) = 1.8 Ft.

This is very conservative since the maximum wave height (H_x) was used rather than the Significant wave height (S_w)

Determine if Freeboard exist with maximum the conservative maximum wave height $H_x = 4.3$ ft, Required freeboards = Wave run-up + Design Precipitation

Design Precipitation for 100yr 24hr rainfall event = 5.0" or .42'
Assume 30% of the pond area is water and the remainder tailings which will have 100% run-off to the 30% pond area then
Design Precipitation = $\frac{.42}{.3}$ = 1.4 ft + 1.8ft = 3.2 ft okay

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: TMA	Page: 5/11
	Prepared by: RBM	Date: 2/22/95
	Checked by: N x P	Date: 4/25/95

b. **Wave Protection**

The gradation of the riprap shall conform to the same criteria as established for revetment riprap. The theoretical stone size shall be selected from Figure 2.

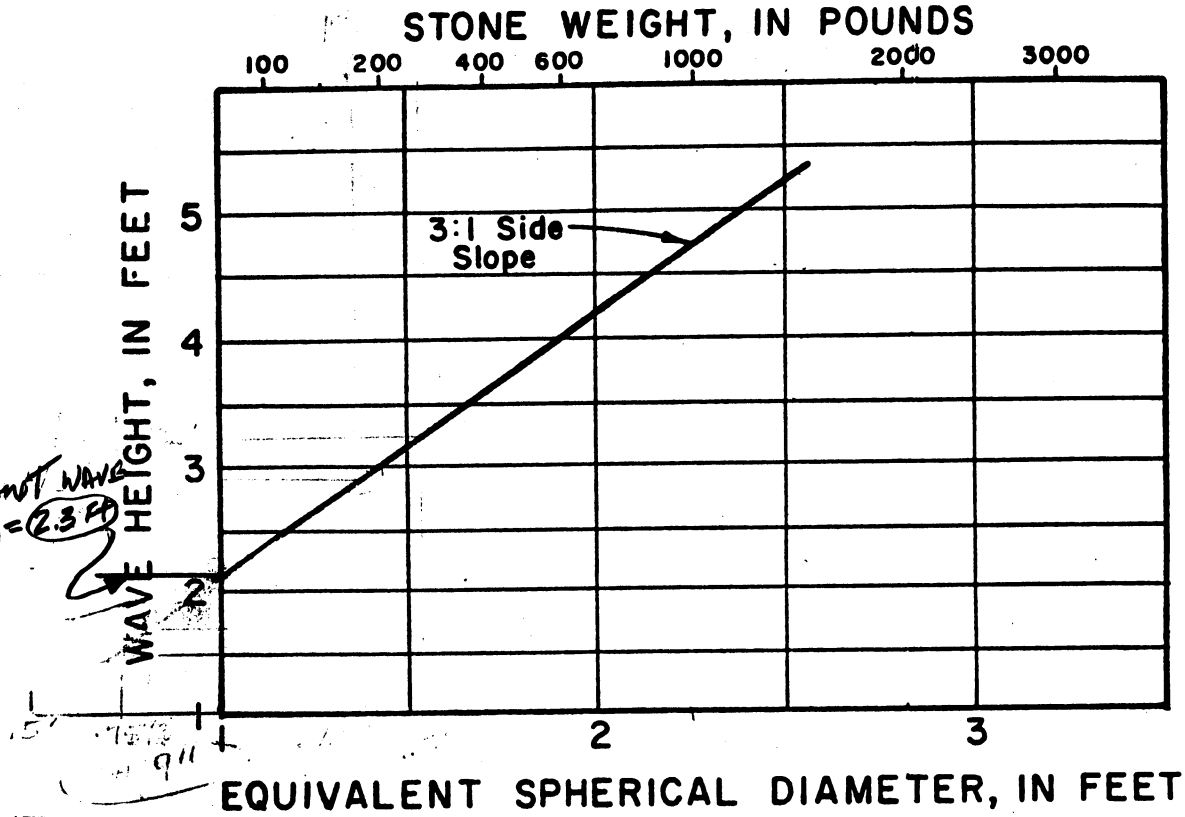


FIGURE 2

SIGNIFICANT WAVE HEIGHT 2_W = (2.3 ft)

GIVEN:

Significant wave height of 2.3' against 3:1 side slope using Figure 2 from USDA - SCS Technical Guide the theoretical size = 12" meaning that the D_{50} s/B = 12"

Determine riprap layer thickness as 1½ times the maximum stone diameter = 18" — assume gradation consistent w/ Soil Class A-2 with % passing as follows per Section 02290 (Specs) 100% - 150# (~15") 80% - ~90# (~12"), 50% - 40# (9"), 20% - 7" (7#) and 10% - 2# (~3") (Same Wisconsin DOT Section 606 on pg 8) GRADATION is slightly small however calculations are conservative!

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: TMA	Page: 6/11
	Prepared by: REM	Date: 2/22/95
	Checked by: N.P	Date: 4/25/95

SECTION 606. RIPRAP

606.1 Description. This work shall consist of furnishing and placing riprap, (with or without grout), medium random riprap or heavy riprap, as the case may be, in accordance with the requirements of the plans and these specifications.

606.2 Materials. Materials used in this work shall meet the requirements for the class of material named and as provided in the specifications.

Stone for riprap shall be durable field or quarry stone of approved quality. It shall be sound, hard, dense, resistant to the action of air and water, and free from seams, cracks or other structural defects.

Stone pieces for Riprap shall be of a size and shape approved by the engineer and, except for those used for chinking, shall range in weight from approximately 25 to 150 pounds, with not less than approximately 50 percent of the pieces weighing more than 60 pounds.

Stone pieces for Medium Random Riprap shall be well graded ranging in weight up to 200 pounds or more. Not less than approximately 50 percent of the total volume shall consist of pieces weighing 80 pounds or more with not less than 80 percent weighing 15 pounds or more.

Stone pieces for Heavy Riprap shall be of a size and shape approved by the engineer and shall be well graded, ranging in weight up to 400 pounds or more. Not less than approximately 50 percent of the total volume shall consist of pieces weighing 150 pounds or more, and not less than approximately 80 percent of the total volume shall consist of pieces weighing 40 pounds or more.

When so provided, waste concrete slabs may be substituted for the above-designated stone. In such case, the concrete shall be sound, and free of reinforcement and the slabs shall meet the size requirements as specified for stone.

Mortar for grouted riprap shall comply with the requirements for mortar for Mortar Rubble Masonry as set forth in Section 518.

Determine if filter layer is required based on the filter criteria

$$\frac{D_{15}\text{ (riprap)}}{D_{85}\text{ (TILL)}} < 5 \qquad \frac{D_{15}\text{ riprap}}{D_{15}\text{ (TILL)}} < 40$$

D_{15} RIPRAP = 120mm
D_{85} TILL = 5mm
D_{15} TILL = 0.05mm

$$\frac{120\text{mm}}{5\text{mm}} < 5 \qquad \frac{120}{0.05} < 40$$

$24 \not< 5$ $2400 \not< 40$ ∴ FILTER is required Between Riprap and Till

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 7/11
Prepared by: REM Date: 3/16/95
Checked by: N-P Date: 4/25/95

- Assume that GEOTEXTILE FILTER WILL BE USED AS A FILTER
- Determine the Apparent Size opening (AOS) or O_{95} of the geotextile
  - Koerner, in Designing with Geosynthetics, 1989 recommends using recommendation of GIROUD, J.P, 1983 Geotextile Filter Criteria as follows:

d_{50} TILL = .3 mm

$C_U > 3$ (TILL is 7.5 to 30 use 20)

FOR DENSE TILL $D_R > 80\%$ $O_{95} < 18\, d_{50}/C_U$

$$O_{95} < \frac{18(.3)}{20} = \frac{5.4}{20} = 0.27 \text{ mm}$$

$O_{95} < 0.27$ mm which is between #50(.30) and #60(.25) sieve

∴ select geotextile w/ AOS of < #60(.25mm) such as:

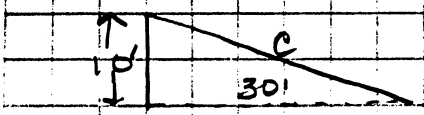
Product	AOS
Amoco 1199	0.212(70)
TREVIRA 011/200	0.210(70)
NICOLON S400	0.212(70)
MIRAFI 140NL	0.210(70)

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 8/11
Prepared by: RGM Date: 3/16/95
Checked by: N.P Date: 4/25/95

Determine RipRap Required for Final Deposition

Assume Riprap will extend 10 vertical feet at the top of the 3:1 slope



$10^2 + 30^2 = C^2$

$C = 31.62$ ft say 32'

32 □'/FT

Determine Maximum area for riprap on all four cells from pg 2 Scaling off drawing

TMA 2 = 3000'
TMA 1 = 2800'
TMA 3 = 2800'
TMA 4 = 2900'
11,600 lin ft

Since riprap will be place on the tailings assume a 30% loss in volume estimates

11,600' x 32 □' x 1.5' (Thick) x 1/27 x 1.3 = 26,809 say 26,800 cy

RipRap Required to cover Base and LCS (See pg 10) 234,500 cy

TOTAL @ 1B Thick. → 261,300 cy

Determine IF Sufficient riprap is available from Mine WASTE Rock and Till Processing

Mine WASTE Rock

Assume the following quantities

- High Sulfur waste Rock = 100,000 Tons
- Other WASTE Rock = 600,000 Tons
  700,000 Tons
- Loose density = 111.3 pcf (Exxon, 1985)

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 9/11
Prepared by: RBM Date: 3/16/95
Checked by: N x P Date: 4/25/95

Calculate Loose Quantity of Waste Rock (High + Low Sulfur)

$700{,}100 \text{ tons} \times \frac{2{,}000 \text{ lbs}}{\text{ton}} \times \frac{1 \text{ CF}}{111.3 \text{ lbs}} \times \frac{1 \text{ cy}}{27 \text{ CF}} = 465874.68$

Say 465,900 cy

Rock for Riprap from Other Sources

1% > 6" from total Cut

$11{,}060{,}200 \times .01 = 110{,}600$ cy

>2" and >3" from Processing soils as per soil balance = 65,300 cy

Total Rock Available for Rip Rap (Loose)

	cy
Mine Waste Rock -	465,900
>6" from Cut	110,600
>2" and >3" from Processing	65,300
	641,800 cy

As per Hansbro, 1983 Memo to WDNR re: refuse + demo waste the following rock quantities are required for construction

ITEM	cy (in-place)	cy Loose (+20%)
Base Course (Roads)	31,900	38,300
Sub Base (Roads)	37,700	45,200
Railroad Ballast + Sub Ballast	26,700	32,000
Haul Road to TMA and Berm Road	NA	32,500
		148,000 cy (Loose)

Foth & Van Dyke

Client: CMC Scope I.D.: 93049
Project: TMA Page: 10/11
Prepared by: RBM Date: 3/16/94
Checked by: N.P Date: ~~4~~ 12/3/95

TOTAL AVAILABLE for RIPRAP - 641,800 cy

REQUIRED for on-site CONSTRUCTION (148,000 cy)

REQUIRED for INTERNAL RIPRAP on interior side slopes (261,300 cy)

NET → 232,500 cy

Also proposed to Riprap Cell Bottoms prior to filling each cell with water. At a minimum we need to cover the LCS drainage areas which results in riprap required as follows.

TMA 1 Stage I 1,255,500 □' x 1.5' x 1/27 = 69,750 cy

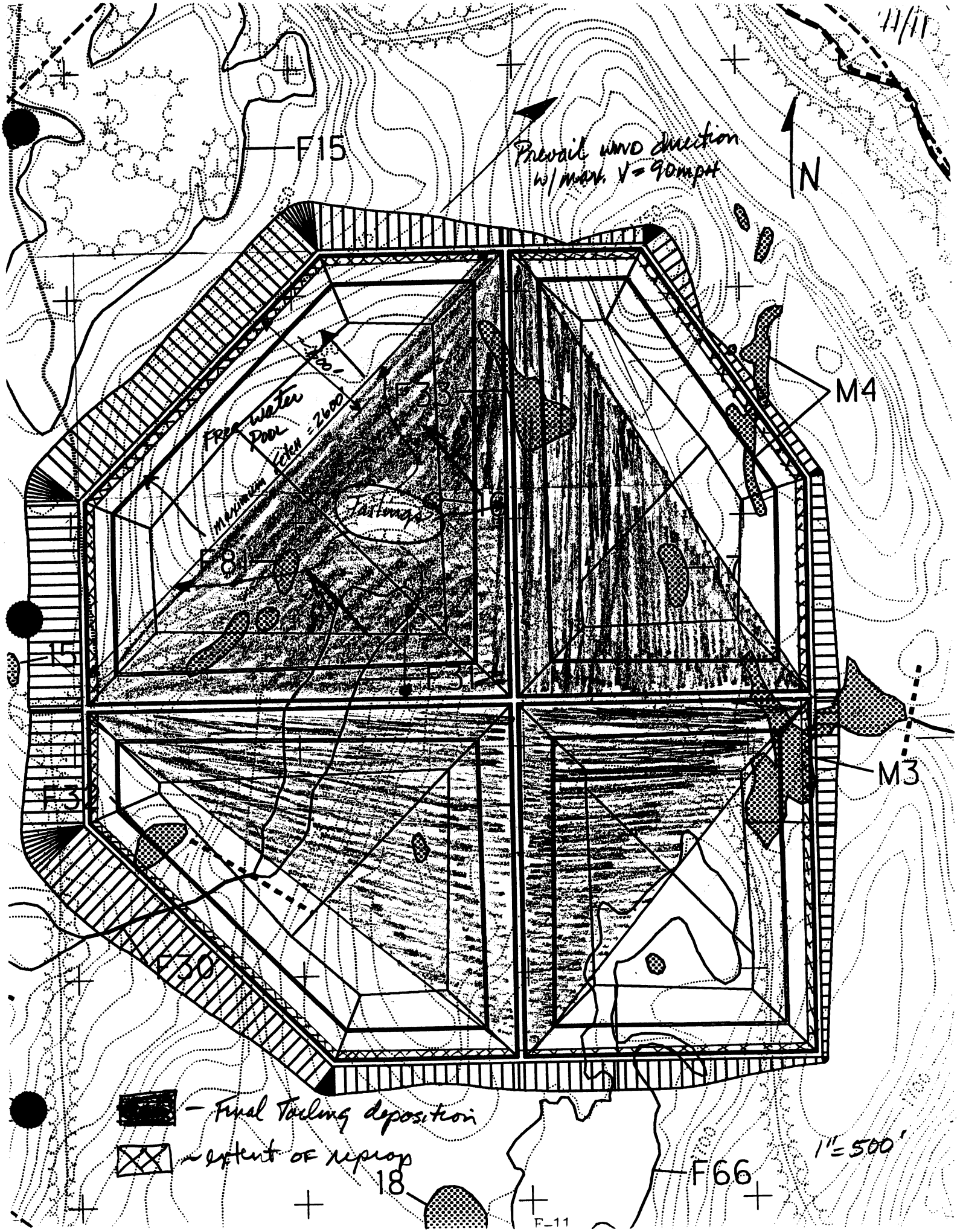
TMA 2 Stage III 1,374,300 □' x 1.5' x 1/27 = 76,350 cy

TMA 3 Stage V 753,300 □' x 1.5' x 1/27 = 41,850 cy

TMA 4 Stage VII 837,000 □' x 1.5 x 1/27 = 46,500 cy

234,450 say 234,500

▲ 232,500 cy is available for riprap for storm water management system and other uses.


F15
Prevail. wind direction
w/ max. V = 90mph
N
M4
F33
Free water
Pool
Maximum Elev. = 2600
Fastenal
F8
F5
F32
M3
F3
F30
F66
18
F-11
- Final Tailing deposition
- extent of riprap
1" = 500'

Appendix F

Filter Design Calculations For Leachate Collection System Drainage Layers

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 1 | 9
Prepared by: REM Date: 3/9/95
Checked by: N.P Date: 4/25/95

FILTER DESIGN FOR TMA leachate Drainage LAYER

Purpose: Design a filter system to inhibit the migration of tailings fines into the leachate drainage layer

GIVEN:

1. DESIGN CRITERIA for PROTECTIVE filters AS follows: (from NAVFAC DM-7.1)

TO AVOID movement of Particles from the Base:

$$\frac{D_{15}\ \text{Filter (F)}}{D_{85}\ \text{Base (B)}} \leq 5$$

$$\frac{D_{50}\ F}{D_{50}\ B} < 25$$

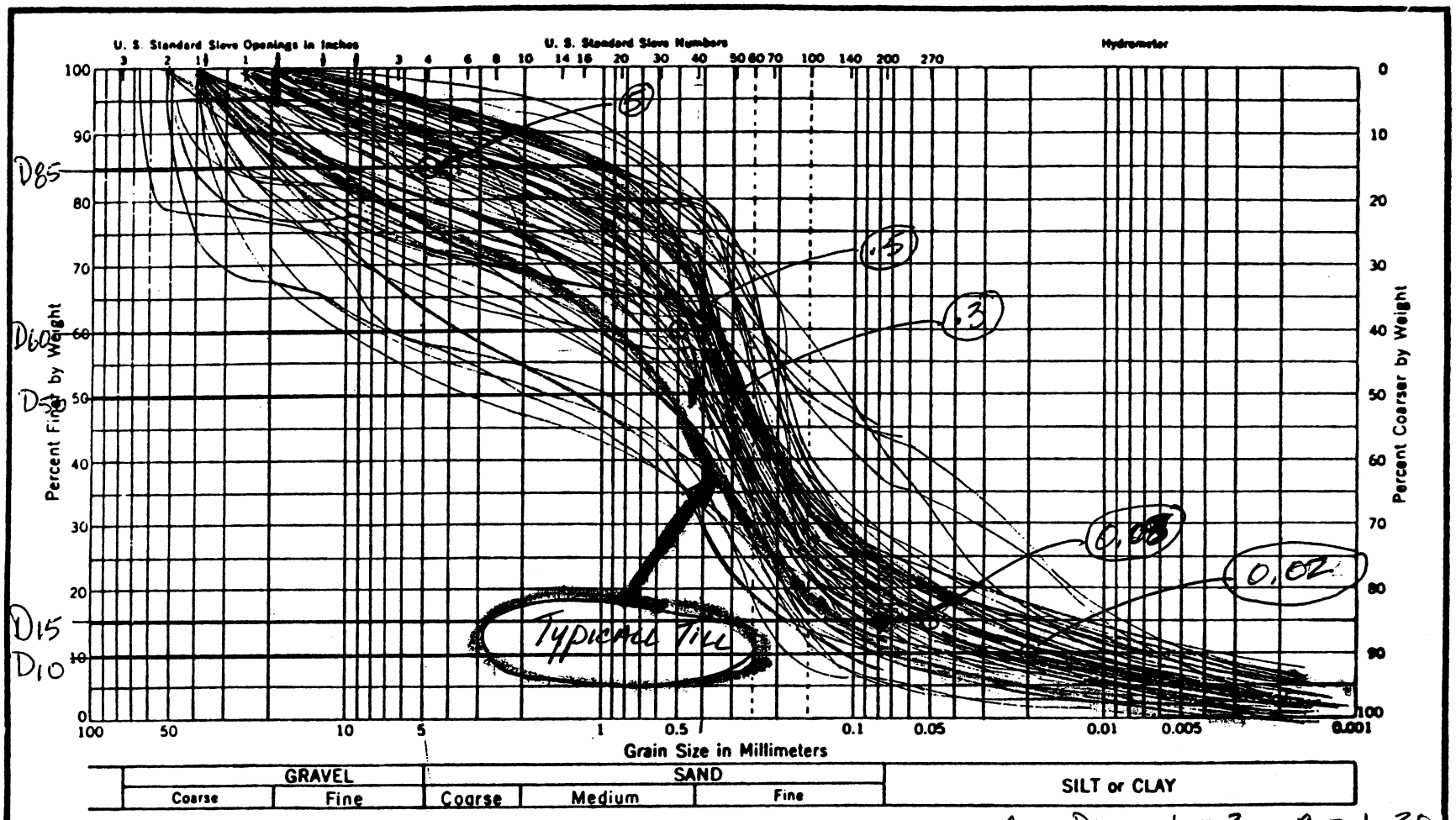
$$\frac{D_{15}\ F}{D_{15}\ B} < 20$$

- TO AVOID Head loss $\frac{D_{15F}}{D_{15B}} > 4$
- TO AVOID segregation, Filter should contain no sizes > 3" - max is 2"
- TO AVOID internal movements of fines Filter should have no more than 5% passing NO. 200 Sieve (For this application this condition is not critical)

2. GRAIN SIZE CURVES FOR Fine Tailings and TILL AS PER pg 2 and 5 with properties AS FOLLOWS.

	FILTER (TILL)	BASE (fine Tailings)	BASE (whole tailings)
D_{15}, mm	0.08	.0015	0.0027
D_{50}, mm	0.3	.005	0.03
D_{85}, mm	.4	0.01	0.07
C_u	$\frac{.15}{0.02} = 25$	slimes $\frac{.006}{.001} = 6$	$\frac{.045}{.0015} = 30$

UNIFIED CLAS、ICATION SYSTEM



$C_u = \frac{D_{60}}{D_{10}} = \frac{.6 - .3}{0.08 - 0.01} = 7.5 \text{ to } 30$

SAMPLE NO.	W.C.	LL	PL	PI	CLASSIFICATION
					Glacial Till SM-SP, SM

TITLE

COMPOSITE GRADATION CURVES

Ref: Exxon (1985.4)

F-2

930049
2/9
REM

Foth & Van Dyke	Client: CMC	Scope I.D.: 930049
	Project: TMA	Page: 3/9
	Prepared by: REM	Date: 3/9/95
	Checked by: N.P	Date: 4/25/95

Fine Tailings

$\frac{D_{15}F}{D_{15}B} \leq 20$ $\quad \frac{0.05}{.0015} = 33.3$ Criteria Not met

$\frac{D_{50}F}{D_{50}B} < 25$ $\quad \frac{0.3}{.005} = 60$ CRITERIA NOT MET

$\frac{D_{15}F}{D_{85}B} \leq 5$ $\quad \frac{0.08}{0.01} = 8$ CRITERIA NOT MET

Whole Tailings

$\frac{D_{15}F}{D_{15}B} \leq 20$ $\quad \frac{0.05}{.0027} = 18.5$ OKAY

$\frac{D_{50}F}{D_{50}B} < 25$ $\quad \frac{0.3}{0.03} = 10$ OKAY

$\frac{D_{15}F}{D_{85}B} \leq 5$ $\quad \frac{0.08}{0.07} = 1.11$ OKAY

TO AVOID HEAD loss in Filter $\frac{D_{15}F}{D_{15}B} > 4$

Fine Tailings = 33 okay
Whole Tailings = 18.5 okay

Initial deposit of tailings will be primarily whole tailings since there are no stopes to backfill during the initial mining. However fine tailing will separate and settle out in the center of the cell so to be conservative a filter must be provided.

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA Page: 4 | 9
Prepared by: REM Date: 3/9/95
Checked by: N.P Date: 4/25/95

Since drainage layer will be processed from on-site till soils, a waste material consisting of the P40 till material is available for use as a filter material below the tailings and above the unprocessed till layer.

Determine gradation of this Fine filter layer assuming till is processed to 100% P40 and the waste product consisting of P40 till is used as a filter layer for the tailings

Average Properties
Unprocessed Till

50% -.5mm ∴ divide values of Unprocessed Till by .57 and plot gradation on Pg 5

$\frac{50\%(.5)}{.5} = 1\ (100\%)$ P40

$\frac{.47}{.57} = .824\ (82\%)$ P60

$\frac{.42}{.57} = .737\ (74\%)$ P100

$\frac{.35}{.57} = .614\ (61\%)$ P200

$\frac{.10}{.57} = .175\ (18\%)$ P.002

Fine FILTER

D_{15} - .0016
D_{50} .038
D_{85} .22
D_{60} .07
D_{10} .001

$C_u = \frac{.07}{.001} = 70$

Compare Fine FILTER to Tailings

$\frac{D_{15}F}{D_{15}B} \leq 5$ $\frac{.0016}{.0015} = 1.07$ OKAY

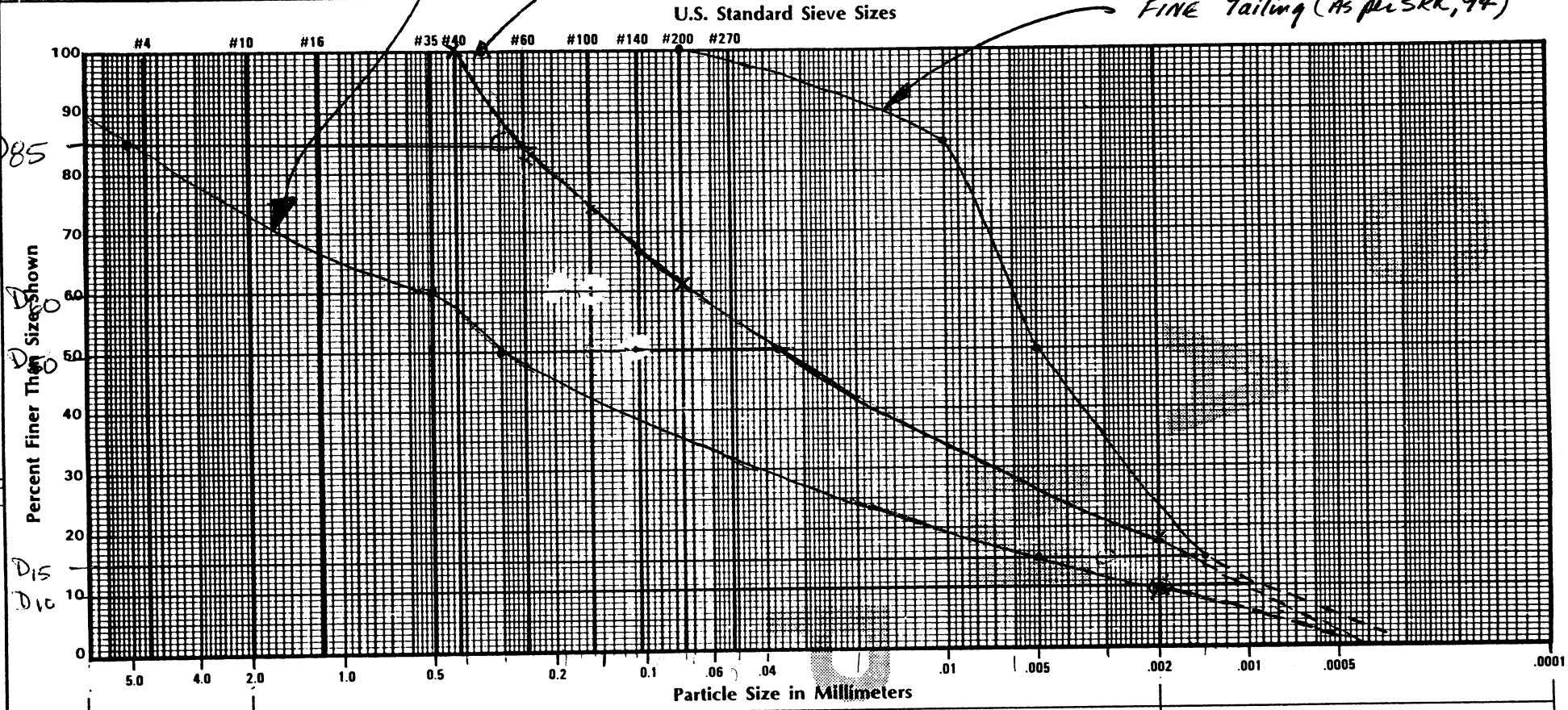
$\frac{D_{50}F}{D_{50}B} < 25$ $\frac{.038}{.005} = 7.6$ OKAY

X-Till Fines (i.e. P40)

Till (as per Exxon, '85)

GRAIN SIZE DISTRIBUTION CURVE - USDA

Fine Tailing (as per SRK, '94)



	Sand			Silt	Clay
	Coarse	Medium	Fine		
	% =	% =	% =	% =	% =

Location Sampled: _____ Elev. or Depth: _____ Date Sampled: _____

Sample Source: _____ Sampled Moisture Content (%): _____ Report No.: _____

USDA Soil Textural Classification: _____

Soil Classification (ASTM: D2487): _____

Munsell Color Code: _____

Date Received: _____

Atterberg Limits: LL _____ PL _____ PI _____

Foth & Van Dyke

Client: _____ Scope I.D.: _____

Project: _____ Page: _____

Prepared by: _____ Date: _____

Checked by: _____ Date: _____

FORM CES020 (1/91)

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: TMA	Page: 6/9
	Prepared by: REM	Date: 3/9/95
	Checked by: N.P	Date: 4/25/95

$\frac{D_{15}\ F}{D_{85}\ B} < 20$ $\quad \frac{.0016}{0.01} = 0.16$ okay

Therefore P40 material from Processing Till is okay as a filter for Tailings

Check Fine FILTER (P40 material against TILL) where Fine Filter is BASE and TILL is the FILTER

$\frac{D_{15}\ F}{D_{15}\ B} \leq 5$ $\quad \frac{0.05}{.0016} = 31.3$ OKAY

$C_u B = 70$ ∴ is broadly graded and $\frac{D_{15}\ F}{D_{15}\ B}$ may be increased to 40

$\frac{D_{50}\ F}{D_{50}\ B} \leq 25$ $\quad \frac{0.3}{0.038} = 7.9$ OKAY

$\frac{D_{15}\ F}{D_{85}\ B} \leq 20$ $\quad \frac{0.08}{.22} = .36$ OKAY

Check TILL (Unprocessed) vs. Drainage layer
- Assume Drainage layer with gradation 100% P 2" and 100% P40 - No. 40 - 42% ∴ 58% → = 100%

Left curve

1mm – 61%	organic; sieved	$100 - \frac{39}{.58} = 32.8\%$
2mm – 69%		$100 - \frac{31}{.58} = 46.6\%$
5mm – 75%		$100 - \frac{25}{.58} = 56.9\%$
10mm 82%		$100 - \frac{18}{.58} = 68.9\%$
20mm 90%		$100 - \frac{10}{.58} = 82.8\%$
2"		100%

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: TMA	Page: 7/9
	Prepared by: REM	Date: 3/9/94
	Checked by: N.P	Date: 4/25/95

Right Curve

No. 40 - 67% finer - 33% remaining

1mm - 85% $\frac{100-15}{.38} = 55\%$

2mm - 90% $100 - \frac{10}{.33} = 70\%$

5mm 95% $100 - \frac{5}{.33} = 85\%$

15mm 100%

Plot this range of Drainage layer gradation as drainage layer grain size curve on pg. 8 Determine for if Filter Criteria is met with Drainage layer as filter and Till as base

FILTER RANGE OR Values

D_{85} - 25-5
D_{60} - 6-1.3
D_{50} - 3-.9
D_{15} - .6-.48
D_{10} - .52-.44

$C_u = \frac{D_{60}}{D_{10}} = \frac{6 \text{ to } 1.3}{.52 \text{ to } .45} = 11.5 \text{ to } 2.9$

TO AVOID HEAD LOST $\frac{D_{15F}}{D_{15B}} \; S/B > 4$

OKAY SINCE range is 7.5 to 6

$\frac{D_{15F}}{D_{15B}} \leq 5 \quad \frac{.6 \text{ to } .48}{.08} = 7.5 \text{ to } 6$

$\frac{D_{50F}}{D_{50B}} \leq 25 \quad \frac{3 \text{ to } .9}{.3} = 10 \text{ to } 3$ okay

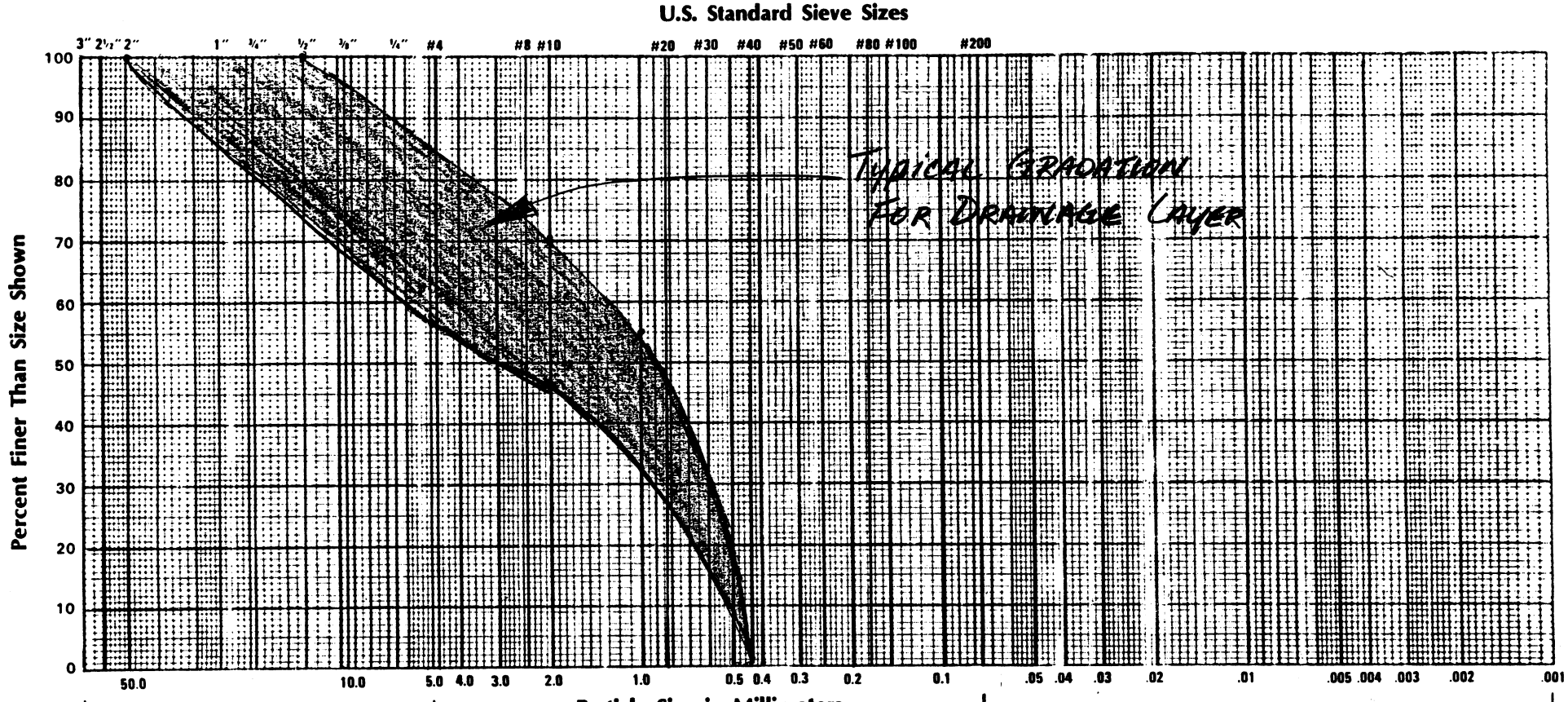
$\frac{D_{15F}}{D_{85B}} < 20 \quad \frac{.6 \text{ to } .48}{4} = .15 \text{ to } .12$ okay

Since $\frac{D_{15F}}{D_{15B}} \leq 5$ criteria is <u>not</u> met, a geotextile filter is required

FOR $C_u > 3$, Koerner (Designing w/ Geosynthetics, 1989) recommends using relationships formulated by GIROUD as the most conservative method for filtration design

(Geotextile Apparent Opening Size) $O_{95} < 13.5 \, D_{50}/C_u$

GRAIN SIZE DISTRIBUTION CURVE



Gravel		Sand			Silt	Clay
Coarse	Fine	Coarse	Medium	Fine		
% =	% =	% =	% =	% =	% =	% =

Location Sampled: _____ Elev. or Depth: _____ Date Sampled: _____

Sample Source: _____ Sampled Moisture Content (%): _____ Report No.: _____

Soil Classification (ASTM: D2487): _____

Atterberg Limits: LL_____ PL_____ PI _____

Munsell Color Code: _____

Date Received: _____

Coefficients: Cc = _____ Cu= _____

Foth & Van Dyke

Client: _____ Scope I.D.: _____

Project: _____ Page: _____

Prepared by: _____ Date: _____

Checked by: _____ Date: _____

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Foth & Van Dyke

Client: CMC Scope I.D.: 93C045
Project: TMA Page: 9/9
Prepared by: REM Date: 3/9/95
Checked by: N.P Date: 4/25/95

FOR UNPROCESSED TILL Average Properties

$\frac{D_{60}}{D_{10}} = \frac{.5}{.02} = 25 = C_u$ $D_{50} = .3$

$O_{95} < \frac{13.5(.3)}{25}$

$O_{95} < 0.162$ which is between an #80 and #100 sieve

∴ AOS OF FABRIC S/B < 0.160 Examples of Suitable geotextiles and their AOS's are:

Typar 3601 - 0.10 (140)
Nicolon S1000 - 0.15 (100)
MIRAFI 180N 0.177 (80)
MIRAFI 1100N 0.149 (100)

Determine the hole diameter in the leachate collection pipe given Drainage layer as per pg 7

TO AVOID movement OF THE FILTER into the DRAIN pipe perforations

$\frac{D_{85}\ F}{\text{Hole Diameter}} > 1.0 \text{ to } 1.2$

$\frac{9.0}{X} > 1.0 \text{ to } 1.2$

X = Hole Diameter S/B 9.0 to 7.5mm or ~ 3/8" diameter

Check range OF permeability using Permeability of SANDS and SAND-Gravel Mixtures from NAVFAC 7.1 (see pg 10)

Appendix G

Perforated Leachate Pipe Strength Analysis

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA-PIPE CRUSH Page: 1 17B
Prepared by: QBKirsch Date: 4/25/95
Checked by: N.P Date: 5-8-95

LEACHATE PIPE CRUSHING ANALYSIS

PURPOSE: TO SELECT THE PROPER 6" PE PIPE SDR FOR THE LEACHATE COLLECTION PIPE DESIGN FOR MAXIMUM LOADING CONDITIONS IN TMA 1-4.

GIVEN: - PIPE DIAMETER of 6.625 in

- GREATEST DEPTH OF FILL OVER THE PIPE IS 120 FT CONSISTING OF
  - 2 FT of STONE
  - 118 FT of TAILINGS.

NOTE: ASSUME TOTAL VERTICAL COLUMN @ TAILINGS. BECAUSE GRADING LAYER MATERIAL IS UNKNOWN AT THIS TIME.

- TAILINGS DRY DENSITY $\gamma_d = 97$ #/CF

- UNIT WEIGH of STONE BACKFILL IS @ 130 #/FT^3

CONTINUED ON NEXT PAGE:

CONCLUSIONS: FOR DESIGN CONSIDERATIONS of 120 FT of BURIAL A 6.625 in OD PE SDR 17 PIPE will DEFLECT 2.9 %. THUS DEFLECTION IS BELOW MAXIMUM ALLOWABLE of 5% DEFLECTION RECOMMENDED BY PIPE MANUFACTURES

GIVEN : CONTINUED :
- MANUFACTURES MAXIMUM ALLOWABLE PIPE DEFLECTION FOR 6 in PE PIPE is 5% (POLYPIPE INDUSTRIES)

SOLUTION : CALCULATE PIPE DEFLECTION (1) AND IMPOSED DESIGN / CONSTRUCTION METHODS TO LIMIT DEFLECTIONS LESS THAN MAXIMUM ALLOWABLE MANUFACTURES RECOMMENDATIONS

EQUATION #1 $\Delta X = \dfrac{D_L K W_c}{\dfrac{2E}{3(SDR)^3} + 0.061E'}$

WHERE Δx = DEFLECTION (INCHES)
D_L = LAG FACTOR, ASSUMED (1.25 - 1.5) USE = 1.5 (CONSERVATIVE)
K = BEDDING CLASS FACTOR CONSERVATIVE $K = 0.1$ USE
W_c = MARSTONS LOAD PER UNIT LENGTH LBS / LINEAL INCH

E = PIPE MODULUS OF ELASTICITY (30,000 psi, CONSERVATIVE)

EQUATION #2 $SDR = OD/T$ where OD = OUTSIDE PIPE DIAMETER
T = PIPE WALL THICKNESS

E' = MODULUS OF PASSIVE SOIL RESISTANCE (CRUSHED ROCK)

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA - PIPE CRUSH Page: 3/178
Prepared by: J.B. KIRSCH Date: 4/25/95
Checked by: N.P Date: 5-8-95

Solution - Continued:

(2) DETERMINE Column Load on Collection Pipe:

(Maximum Prism Load) $w = 34\%$ - SEE (A) NEXT PAGE $(130\ \frac{lb}{ft^3})(118\ ft)$

Equation #3 • TAILINGS $\gamma_d = 97\ lb/cf$; $\gamma_{moist} = (\gamma_d)(1+w) = (97)(1.34) = 15340$

STONE BACKFILL $130\ lb/cf \cdot 2\ ft = +\ 260$

PIPE $+\ 0$

$15600\ lb/ft^2$

Maximum Column Load on Collection Pipe = $15,600\ lbs/sf$ =

(3) Equation #4 DETERMINE UNIT WEIGHT ON PIPE

$15600\ lb/ft^2 \cdot \frac{1}{120\ ft} = 130\ lb/ft^3$ UNIT WEIGHT

(4) DETERMINE MARSTON LOAD W_c, PER UNIT LENGTH

Equation #5 $W_c = C_c w B_c^2$

WHERE C_c = LOAD COEFFICIENT WHICH IS A FUNCTION OF THE PROJECTION RATIO (p) SETTLEMENT RATIO (r_{sd}) AND RATIO OF FILL HEIGHT (H) to PIPE WIDTH (B_c) SEE FIGURE #1

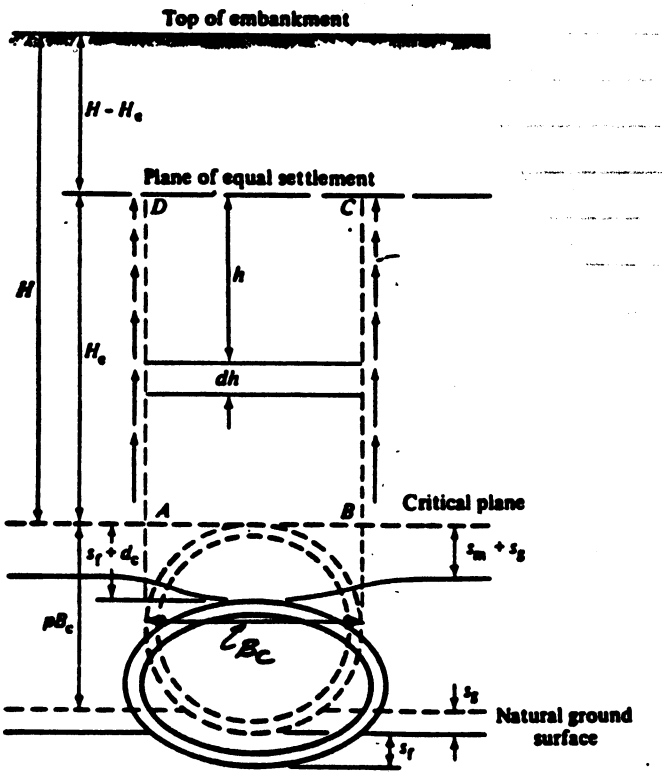
w = UNIT WEIGHT = 130 lb/ft^3

B_c = OUTSIDE DIAMETER OF PIPE, FEET (BREADTH OF CONDUIT)

$6.625\ in / 12\ in/ft = 0.552\ FT$

Foth & Van Dyke

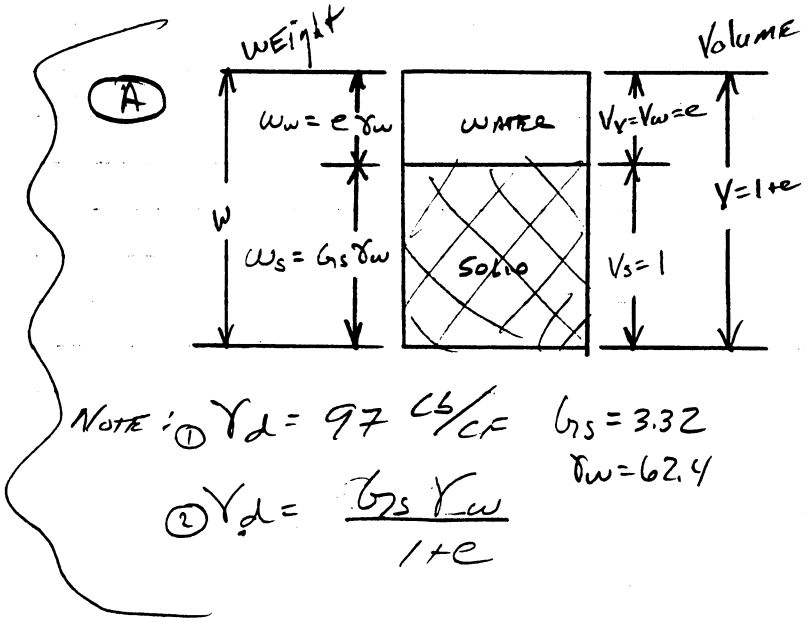
Client: CMC Scope I.D.: 93C049
Project: TMA - PIPE CRUSH Page: 4/178
Prepared by: BKIRSCH Date: 4/26/95
Checked by: N.P Date: 5-8-95



SOURCE:

SPANGLER, M.G. & HANDY, R.L. (1982). SOIL ENGINEERING, HARPER & ROW, NEW YORK, NY

Figure [illegible] Settlements that influence loads on positive projecting conduits (incomplete ditch condition). Key same as in Figure 26.12.



NOTE: ① $\gamma_d = 97$ LB/CF $G_s = 3.32$ $\gamma_w = 62.4$

② $\gamma_d = \dfrac{G_s \gamma_w}{1+e}$

③ $1+e = \dfrac{62.4 \cdot 3.32}{97}$

$1+e = 2.14$

$e = 1.14$

$w = \dfrac{e}{G_s}$

$w\% = \dfrac{e}{G_s} \times 100$

$w\% = \dfrac{1.14}{3.32} \times 100 =$

$w\% = 34\%$

Foth & Van Dyke	Client: CMC	Scope I.D.: 930049
	Project: TMA- PIPE CRUSH	Page: 5/17B
	Prepared by: JB Kiesel	Date: 4/26/95
	Checked by: N.P	Date: 5-8-95

SOLUTION:
(CONTINUED)

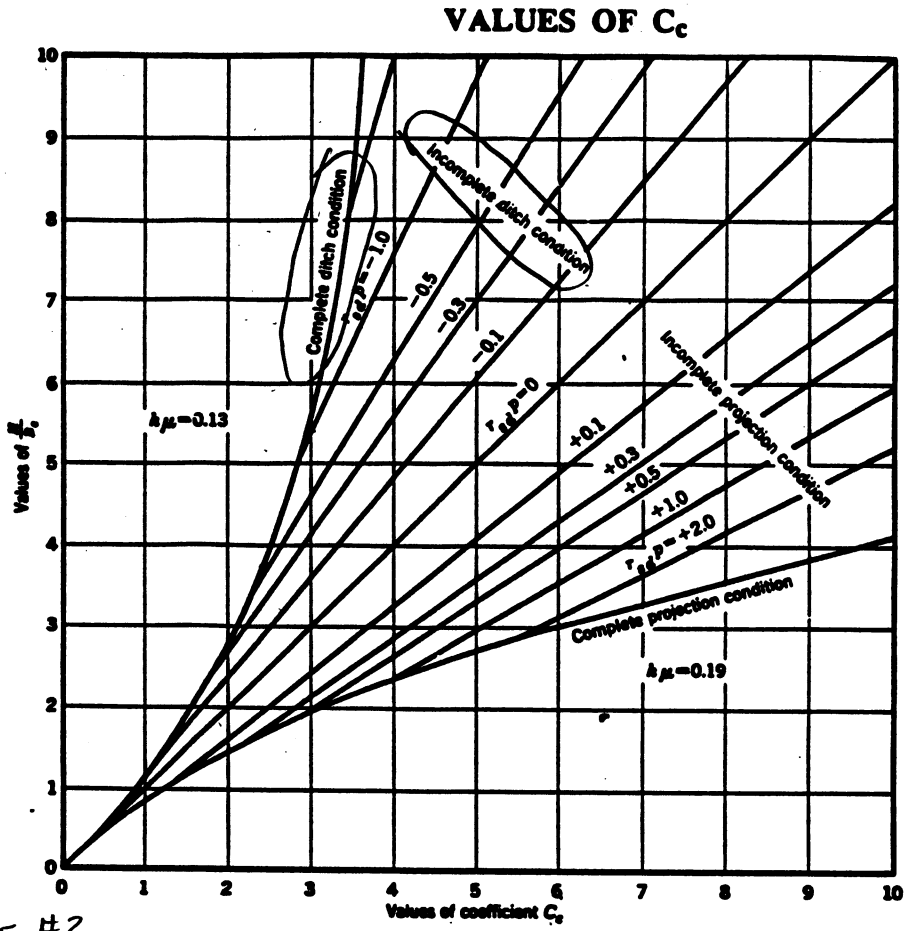


FIGURE #2

Diagram for coefficient C_c for positive projecting conduits.

SOURCE: GRAVITY SANITARY SEWER DESIGN AND CONSTRUCTION. "MANUALS & REPORTS ON ENGINEERING PRACTICE NO. 60," AMERICAN SOCIETY OF CIVIL ENGINEERS AND "MANUAL OF PRACTICE FD-5," WATER POLLUTION CONTROL FEDERATION. 1982.

4-a) NOTE: FOR FLEXIBLE PIPE UNDER MOST CONDITIONS THE PRODUCT $r_{sd} \cdot P$, IS LESS THAN ZERO (NEGATIVE VALUE) FOR . , AN - EMBANKMENT SITUATION SUCH AS A LANDFILL CONSTRUCTION, AN INCOMPLETE DITCH CONDITION EXISTS. BECAUSE $r_{sd} \cdot P$ IS NEGATIVE THE TOTAL PRISM LOAD IS NOT APPLIED TO THE PIPE.

4-b FOR THE PIPE VERTICAL PRESSURE ON TOP OF THE PIPE.

USE $K_o = 0.5$

K_o = RATIO of LATERAL EARTH PRESSURES to VERTICAL EARTH PRESSURES

4-c DETERMINE $K_{o\mu}$:

Equation #6 $\mu = \tan\phi = \tan 10^\circ = 0.17$

ϕ = INTERNAL FRICTION ANGLE of TAILINGS

Equation #7 $K_{o\mu} = (0.5)(0.17) = 0.088 = \underline{0.09}$

4-d DETERMINE $r_{sd} = \dfrac{(S_m + S_g) - (S_f + d_c)}{S_m}$

Equation #8 SEE Figure #1

WHERE:

r_{sd} = SETTLEMENT RATIO

S_m = COMPRESSION STRAIN of the SIDE COLUMN of height $p \cdot B_c$ where: p = PROJECTION RATION, B_c = WIDTH of PIPE

S_g = SETTLEMENT OF THE NATURAL SURFACE ADJACENT to CONDUIT

S_f = SETTLEMENT of the conduit into its Foundation

d_c = SHORTENING OF THE VERTICAL HEIGHT of the conduit

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA - PIPE CRUSHING Page: 7 / 17B
Prepared by: JBKirsch Date: 4/26/95
Checked by: N.P. Date: 5-8-95

4.d- continued FOR DETERMINING f_{sd}

FOR f_{sd} analysis ASSUME: $S_g = 0$

$S_f = 0$

Equation #9 S_m = Load STRAIN = $\left(\frac{\text{Load of Prism}}{E_B \text{ of Stone}}\right)(B_c)$

Load of Prism = (UNIT WEIGHT)(Height)(width) = L_p

Equation #10 $L_p = \left(130.0 \frac{\text{LB}}{\text{FT}^3}\right)(120 \text{ FT})(0.55 \text{ FT}) = 8,580 \frac{\text{LB}}{\text{FT}}$

or $L_p < 9,000$ (USE 9,000 LB/FT

E_B of Stone of 5×10^6 PSF (CONSERVATIVE VALUE) SEE ATTACHMENT #1.

Equation #9 (AGAIN)

$$S_m = \frac{9.000 \text{ LB/FT}}{5 \times 10^6 \text{ LB/FT}^2} \cdot B_c = 0.0018 B_c$$

USE $0.002 B_c$

d_c = (% of DEFLECTION)(B_c)

% of DEFLECTION = MAXIMUM RECOMMENDED BY MANUFACTURER

5% FOR 6" PE PIPE

Foth & Van Dyke

Client: CMC Scope I.D.: 930049
Project: TMA-Pipe Crushing Page: 8/178
Prepared by: GB Kirsch Date: 4/26/95
Checked by: N.P Date: 5-8-95

Equation 8 Again FOR DETERMINING r_{sd}

4.d continued $r_{sd} = \dfrac{(s_m + s_g) - (s_f + d_c)}{s_m}$

where $s_m = 0.002 B_c$

$s_g = 0$

$s_f = 0$

$d_c = 0.05 B_c$

$\therefore r_{sd} = \dfrac{[(0.002 B_c) + 0] - [0 + 0.05 B_c]}{0.005 B_c}$

$r_{sd} = \dfrac{0.002 B_c - 0.05 B_c}{0.005 B_c} = -9.6$

$r_{sd} = -9.6$

4-e DETERMINE RATIO of C_c FOR $K_{\mu} = 0.09$
NOTE: K_{μ} WAS DETERMINED in #4-c.

TABLE 1 Values[a] of C_c in Terms of H/B_c

Incomplete Projection Condition $K\mu = 0.19$		Incomplete Ditch Condition $K\mu = 0.13$	
$r_{sd}p$	Equation	$r_{sd}p$	Equation
+0.1	$C_c = 1.23H/B_c - 0.02$	−0.1	$C_c = 0.82H/B_c + 0.05$
+0.3	$C_c = 1.39H/B_c - 0.05$	−0.3	$C_c = 0.69H/B_c + 0.11$
+0.5	$C_c = 1.50H/B_c - 0.07$	−0.5	$C_c = 0.61H/B_c + 0.20$
+0.7	$C_c = 1.59H/B_c - 0.09$	−0.7	$C_c = 0.55H/B_c + 0.25$
+1.0	$C_c = 1.69H/B_c - 0.12$	−1.0	$C_c = 0.47H/B_c + 0.40$
+2.0	$C_c = 1.93H/B_c - 0.17$		

FROM FIGURE 2 THE INCOMPLETE DITCH CONDITION CURVES ARE BOUNDED BY THE COMPLETE DITCH CONDITION CURVE.

Foth & Van Dyke

Client: CMC Scope I.D.: 930049
Project: TMA - PIPE CRUSHING Page: 9/17B
Prepared by: G.B KIRSCH Date: 4/27/95
Checked by: N.P Date: 5-8-95

THE COMPLETE DITCH CONDITION C_c VALUE IS DEFINED BY: $C_c = \frac{e^{\pm K_u (H/B_c)} - 1}{\pm 2K_u}$

Equation #11

THE RATIO OF $K_u = 0.09$, $K_u = 0.13$ IS 1.44 :

THUS C_c (1.44) WILL BE APPLIED FOR $K_u = 0.09$

TABLE #2 r_{sdp}, C_c K_u/K_u

Equation #12

FOR $K_u = 0.09$, $H/B_c = 120/0.55$

1. r_{sdp}	C_c (1)	= C_c K_u/K_u (2)	MODIFIED C_c VALUES
-0.1	$C_c = 0.82\ H/B_c + 0.05$	= 178.3 (1.44)	256.8
-0.3	$C_c = 0.69\ H/B_c + 0.11$	= 150.1 (1.44)	216.1
-0.5	$C_c = 0.61\ H/B_c + 0.20$	= 132.8 (1.44)	191.2
-0.7	$C_c = 0.55\ H/B_c + 0.25$	= 119.8 (1.44)	172.5
-1.0	$C_c = 0.47\ H/B_c + 0.40$	= 102.6 (1.44)	147.7

(1) C_c EQUATIONS ARE FROM TABLE #1

(2) C_c VALUES OF $K_u = 0.13$ EQUATIONS ARE MULTIPLIED BY K_u/K_u RATIO WHICH IS $\frac{0.13}{0.09} = 1.44$

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: PIPE CRUSHING Page: 10/17B
Prepared by: E. Kuesel Date: 4/28/95
Checked by: N. P Date: 5-8-95

FIGURE 3 - GRAPHICAL REPRESENTATION OF TABLE 2

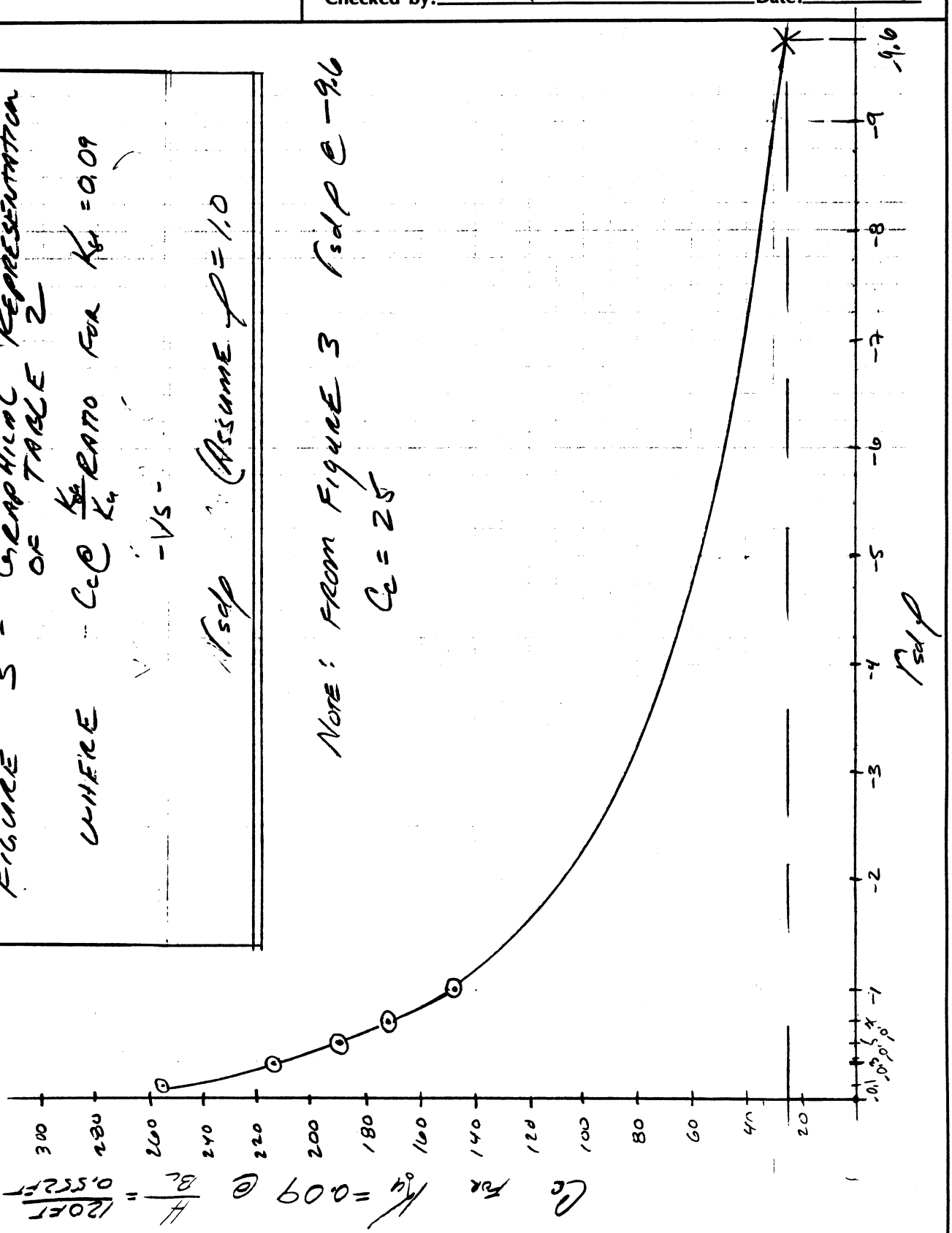
WHERE C_c @ $\frac{K_\mu}{K_4}$ RATIO FOR $K_\mu = 0.09$

$-V_s$ -

V_{sfp} (ASSUME $C_c = 1.0$

NOTE: FROM FIGURE 3 V_{sfp} @ -9.6

$C_c = 25$



Foth & Van Dyke	Client: CMC Scope I.D.: 93C049 Project: TMA - PIPE CRUSHING Page: 11/17 Prepared by: JB KIRSCH Date: 4/27/95 Checked by: N - P Date: 5-8-95

4-f. From #4

MARSON LOAD W_c

Equation from #5 Again $W_c = C_c w B_c^2$

$C_c = 25$ From Figure 3

$w = 130$ LB/FT³ From #3

$B_c = 6.625 \text{ in} / 12 \text{ in/ft} = 0.552$ FT

$W_c = (25)(130 \text{ LB/FT}^3)(0.552 \text{ FT})^2$

$W_c = 990.$ LB/LINEAL FT

$W_c = 990 \text{ LB/LIN.FT} \cdot \frac{1 \text{ FT}}{12 \text{ in}} = 82.5 \text{ LB/LINEAL INCH}$

(5) Modulus of Passive Soil Resistance, E'

E' = 1,000 psi for dumpen crushed Rock.

SEE TABLE #3.

Foth & Van Dyke	Client: CMC · Scope I.D.: 93C049 Project: PIPE CRUSHING Page: 12 / 17D Prepared by: QBKIRSCH Date: 4/27/95 Checked by: N.P Date: 5-8-95

TABLE #3

AVERAGE VALUES OF MODULUS OF SOIL REACTION, E'
(For Initial Flexible Pipe Deflection)

Soil type-pipe bedding material (Unified Classification System[a]) (1)	E' for Degree of Compaction of Bedding, in pounds per square inch			
	Dumped (2)	Slight, <85% Proctor, <40% relative density (3)	Moderate, 85%-95% Proctor, 40%-70% relative density (4)	High, >95% Proctor, >70% relative density (5)
Fine-grained Soils (LL > 50)[b] Soils with medium to high plasticity CH, MH, CH-MH	No data available; consult a competent soils engineer; Otherwise use E' = 0			
Fine-grained Soils (LL < 50) Soils with medium to no plasticity, CL, ML, ML-CL, with less than 25% coarse-grained particles	50	200	400	1,000
Fine-grained Soils (LL < 50) Soils with medium to no plasticity, CL, ML, ML-CL, with more than 25% coarse-grained particles Coarse-grained Soils with Fines GM, GC, SM, SC[c] contains more than 12% fines	100	400	1,000	2,000
Coarse-grained Soils with Little or no Fines GW, GP, SW, SP[c] contains less than 12% fines	E' FOR STONE BACKFILL			
→ Crushed Rock	200	1,000	2,000	3,000
	1,000	3,000	3,000	3,000
Accuracy in Terms of Percentage Deflection[d]	±2	±2	±1	±0.5

[a]ASTM Designation D 2487, USBR Designation E-3.
[b]LL = Liquid limit.
[c]Or any borderline soil beginning with one of these symbols (i.e. GM-GC, GC-SC).
[d]For ±1% accuracy and predicted deflection of 3%, actual deflection would be between 2% and 4%.

Note: Values applicable only for fills less than 50 ft (15 m). Table does not include any safety factor. For use in predicting initial deflections only, appropriate Deflection Lag Factor must be applied for long-term deflections. If bedding falls on the borderline between two compaction categories, select lower E' value or average the two values. Percentage Proctor based on laboratory maximum dry density from test standards using about 12,500 ft-lb/cu ft (598,000 J/m^3) (ASTM D 698, AASHTO T-99, USBR Designation E-11). 1 psi = 6.9 kPa.

SOURCE: "Soil Reaction for Buried Flexible Pipe" by Amster K. Howard, U.S. Bureau of Reclamation, Denver, Colorado. Reprinted with permission from American Society of Civil Engineers.

SOURCE: UNI-BELL PVC PIPE ASSOCIATION (1991)
HANDBOOK OF PVC PIPE: DESIGN AND CONSTRUCTION
DALLAS, TX p. 207

(6) SOLVE FOR DEFLECTION FROM EQUATION #1.

EQUATION #1 AGAIN

$$\Delta x = \frac{D_L \cdot K \, W_c}{\frac{2E}{3(SDR)^3} + 0.061 E'}$$

WHERE $D_L = 1.5$

$K = 0.1$

$W_c = 82.5$ LB/LINEAL INCH

$E = 30,000$ psi

SDR = ASSUME SDR 17

$E' = 1000$ psi

$$\therefore \Delta x_{@SDR17} = \frac{(1.5)(0.1)(82.5 \text{ lb/in})}{\frac{2 \cdot (30,000 \text{ psi})}{3 \cdot (17)^3} + 0.061(1,000 \text{ psi})} = \frac{12.37}{4.07 + 61}$$

$$\Delta x_{@SDR17} = 0.191 \text{ inches}$$ ✓

(7) EQUATION #13 DETERMINE % DEFLECTION $= \frac{\Delta x}{B_c} \times 100$

where $\Delta x_{@SDR17} = 0.191$ inches, $B_c = 6.625$ inches

$$\frac{0.191 \text{ in}}{6.625 \text{ in}} \times 100 = 2.9\%$$ ✓

CONTINUED

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: PIPE CRUSHING Page: 14/17b
Prepared by: JBKIRSCH Date: 4/28/95
Checked by: N.P Date: 5-8-95

Conclusion: FOR DESIGN CONSIDERATION OF 120 FT OF COVER MATERIALS, A 6.625 in OD. PE SDR 17 PIPE will DEFLECT 2.9%. THUS DEFLECTION IS BELOW MAXIMUM ALLOWABLE PIPE DEFLECTION of 5% RECOMMENDED BY PIPE & MANUFACTURES.

REFERENCE:

SPANGLER, M.G. and HANDY, R.L., (1982) Soil Engineering, HARPER & ROW, PUBLISHERS, NY. NY.

Foth & Van Dyke	Client: CMC Scope I.D.: 93C044 Project: PIPE CRUSHING Page: 15/17B Prepared by: JBKiesel Date: 4/28/95 Checked by: NVP Date: 5-8-95

ATTATCHMENT #1. Modulus OF ELASTICITY FOR STONE BACKFILL

TASK: DETERMINE THE MODULUS OF ELASTICITY (E_B) FOR THE STONE BACKFILL AROUND THE Collection PIPE.

Solution:

$$G_1 = \frac{3,230}{(1+e)} (2.97-e)^2 \bar{\sigma}_o^{1/2}$$

WHERE $E_B = G_1 \cdot 2(1+\mu)$

Thus:

$$E_B = \frac{6,460 (2.97-e)^2 \bar{\sigma}_o^{1/2} (1+\mu)}{1+e}$$

WHERE: G_1 = SHEARE MODULUS
E_B = MODULUS of ELASTICITY in KPa
e = Void Ratio = 0.65

ASSUME: No Compaction
$\gamma_{d\ stone}$ = 100 LB/CF
$G_{s\ stone}$ = 2.65 (specific Gravity)
μ = Poisson's Ratio

$$\frac{\mu}{1-\mu} = 0.5 \quad \therefore \mu = 0.33$$

$\bar{\sigma}_o$ = EFFECTIVE CONFINING STRESS IN KPa

ATTACHMENT #1 (CONTINUED)

Determine $\bar{\sigma}_o$:

$$\bar{\sigma}_o = 1/3 (\sigma_z + 2\sigma_h)$$

where: σ_z = VERTICAL STRESS AT WORST CASE

$$\sigma_h = K_o \sigma_z$$

$$K_o = 1 - \sin\phi = 0.5 \quad \text{where } \phi = 30°$$

σ_z = 15,600 lb/ft² = 15.6 Kips/ft²

σ_z = 15.6 Kips/ft² (FROM MAX. PRISM LOAD ON LEACHATE COLLECTION PIPE. WORST CASE)

= 17.4 Kip/ft²

N/m² = Pa

σ_z = 748 KPa ✓ (100 KPa; Kips = 1000)

$\bar{\sigma}_o$ = 1/3 (748 + (2)(0.5)(748))

$\bar{\sigma}_o$ = 499 KPa ✓ = EFFECTIVE CONFINING STRESS

Foth & Van Dyke	Client: CMC	Scope I.D.: 93.044
	Project: PIPE CRUSHING	Page: 17 of 17 Pgs.
	Prepared by: JB KIRSCH	Date: 4/28/95
	Checked by: N x P	Date: 5-8-95

ATTACHMENT #1 (continued)

SOLVE FOR Modulus of ELASTICITY (E_B):

FROM ABOVE $E_B = \dfrac{6{,}460\,(2.97 - e)^2\,\bar{\sigma}_o^{1/2}\,(1+\mu)}{1+e}$

WHERE: $\bar{\sigma}_o$ = 499 KPa ✓ - EFFECTIVE CONFINING STRESS

μ = 0.33 - POISSON'S RATIO

e = 0.65 - Void Ratio

$$E_B = \frac{6{,}460\,(2.97 - 0.65)^2\,(499\ KPa)^{1/2}\,(1 + 0.33)}{1 + 0.65}$$

E_B = 625,454 KPa ✓

$$E_B = 625{,}454\ KPa \left(\frac{Kip/FT^2}{48}\right)\left(\frac{1000\,Lb}{1\,Kip}\right) = 13{,}030{,}000\ PSF$$ ✓

$= 13 \times 10^6$ psf

FOR A CONSERVATIVE VALUE WE WILL USE 5×10^6 PSF $= E_B$ ✓.

SOURCE: BOWLES, JOSEPH E. (1982). FOUNDATION ANALYSIS AND DESIGN, McGRAW-HILL, INC., NEW YORK, N.Y.

Fig. 2

ATTACHMENT #2

PIPE INFORMATION: MINIMUM WALL, in.
WEIGHT PER FOOT, #
NOMINAL I.D., in.

O.D., NOMINAL	in. ACTUAL	SDR 7	7.3	9	9.3	11	13.5	15.5	17	21	26	32.5
1	1.315	0.188 0.28 0.924	0.180 0.28 0.940	0.146 0.23 1.011	0.141 0.22 1.021	0.120 0.19 1.066						
1-1/4	1.660	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
1-1/2	1.900	0.271 0.59 1.335	0.260 0.57 1.359	0.211 0.48 1.461	0.204 0.47 1.475	0.173 0.40 1.541	0.141 0.33 1.607	0.123 0.29 1.645				
2	2.375	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
3	3.500	0.500 2.02 2.460	0.479 1.95 2.503	0.389 1.63 2.691	0.376 1.58 2.717	0.318 1.36 2.838	0.259 1.13 2.961	0.226 0.99 3.030	0.206 0.91 3.072	0.167 0.75 3.153	0.135 0.61 3.220	
4	4.500	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
5	5.563	0.795 5.10 3.910	0.762 4.92 3.978	0.618 4.11 4.277	0.598 4.00 4.319	0.506 3.44 4.511	0.412 2.86 4.706	0.359 2.51 4.816	0.327 2.31 4.882	0.265 1.89 5.012	0.214 1.54 5.118	0.171 1.24 5.207
6	6.625	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
8	8.625	1.232 12.26 6.062	1.182 11.83 6.167	0.958 9.89 6.632	0.927 9.61 6.696	0.784 8.27 6.994	0.639 6.87 7.296	0.556 6.04 7.468	0.507 5.54 7.570	0.411 4.54 7.771	0.332 3.70 7.935	0.265 2.99 8.073
10	10.750	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
12	12.750	1.821 26.79 8.961	1.747 25.86 9.117	1.417 21.60 9.803	1.371 20.99 9.898	1.159 18.08 10.339	0.944 15.00 10.786	0.823 13.20 11.039	0.750 12.11 11.190	0.607 9.92 11.487	0.490 8.09 11.730	0.392 6.52 11.934
14	14.000	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
16	16.000	2.286 42.18 11.246	2.192 40.72 11.441	1.778 34.02 12.302	1.720 33.06 12.422	1.455 28.47 12.975	1.185 23.63 13.535	1.032 20.79 13.853	0.941 19.07 14.042	0.762 15.62 14.415	0.615 12.74 14.720	0.492 10.27 14.976
18	18.000	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
20	20.000		2.740 63.63 14.301	2.222 53.16 15.378	2.151 51.65 15.527	1.818 44.48 [illegible]	1.481 36.92 16.919	1.290 32.49 17.316	1.176 29.80 17.553	0.952 24.41 18.019	0.769 19.91 18.400	0.615 16.05 [illegible]

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Appendix H

Leachate Sump Size Determination and Pump Sizing

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA - Sump Size Page: 1/7 + A, B Attach
Prepared by: JBKiasch Date: 4/12/95
Checked by: MOF Date: 5/2/95

LEACHATE SUMP SIZE DETERMINATION

PURPOSE: DETERMINE THE LEACHATE SUMP STORAGE CAPACITY FOR THE LARGEST TMA CELL

SCOPE: DETERMINE THE MAXIMUM SUMP STORAGE CAPACITY FOR CMC - TMA 2 (LARGEST TMA CELL)
Use HELP model runs @ 2x PRECIPITATION; Open Case
Use LEACHATE GENERATION FOR MAXIMUM GENERATION MONTH
Use 6 cycles PER DAY of Sump Storage CAPACITY EVACUATION.

Conclusions:

SUMP SIZE REQUIRED FOR TMA-2 (LARGEST TMA CELL) DURING OPERATIONS AND @ 6 cycles PER DAY is 1,634 CF

Foth & Van Dyke	Client: CMC Scope I.D.: 93C049 Project: TMA - Sump Size Page: 2/7 Prepared by: QBKiesel Date: 4/12/95 Checked by: MDF Date: 5/2/95

Assumptions:

1) TMA 2 has the maximum surface area of the 4 - TMA cells.

2) There are 2 sumps per TMA cell

3) Porosity of stone in the sump is 0.30

4) The sump evacuation pump is designed to run at 6 cycles/day.

Provided

2x

1) The maximum monthly leachate collected for sideslope HELP model @ 2x precipitation.

Foth & Van Dyke	Client: CMC Scope I.D.: 93C049 Project: TMA-Sump Size Page: 3/7 Prepared by: JBKiesel Date: 4/12/95 Checked by: MDF Date: 5/2/95

PROVIDED: CONTINUED

2) THE MAXIMUM MONTHLY LEACHATE COLLECTED FOR BASE HELP MODEL @ 2X PRECIPITATION.

3) SURFACE AREA FOR TMA 2 SIDESLOPE + BASE

Solution:

(1) DETERMINE VOLUME OF LEACHATE COLLECTED (gal/day) FOR TMA 2 @ 2X PRECIPITATION VALUE FOR SIDE SLOPE (SS)

Eq 1 $LD_{ss} \times SA_{ss} = Q_{ss}$

WHERE LD_{ss} = LEACHATE LATERAL DRAINAGE FROM PEAK MONTH = 0.4472 in/month FROM ATTACHMENT A

SA_{ss} = 1,863,670 FT^2 FROM CADD DESIGN FILES

Q_{ss} = QUANTITY GENERATED PER DAY FOR SIDESLOPE. (gal/day) For Side Slope (SS)

Eq 1 $\left(0.4476 \frac{in}{month}\right)\left(\frac{1 FT}{12 in}\right)\left(1,863,670 FT^2\right)\left(\frac{7.48 gal}{FT^3}\right)\left(\frac{1 month}{30 day}\right) = Q_{ss}$

Q_{ss} = 17,332. gal/day SS ✓

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: TMA - Sump Size	Page: 4/7
	Prepared by: QB Kiesch	Date: 4/12/95
	Checked by: MDF	Date: 5/2/95

Solution Continued:

(2) DETERMINE VOL OF LEACHATE COLLECTED (gal/day) FOR TMA 2 @ 2X PRECIPITATION VALUE FOR BASE (B).

Eq #2 $LD_B \times SA_B = Q_B$

WHERE: LD_B = LEACHATE LATERAL DRAINAGE FROM PEAK month = 1.0544 in/month FROM ATTACHMENT B.

SA_B = 1,214,970 FT^2 FROM CAD DESIGN FILES

Q_B = QUANTITY GENERATED PER DAY FOR BASE (GAL/DAY)

Eq 2: $\left(1.0544 \frac{in}{month}\right)\left(\frac{1 FT}{12 in}\right)\left(1,214,970 FT^2\right)\left(7.48 \frac{gal}{FT^3}\right)\left(\frac{1 month}{30 day}\right) = Q_B$

Q_B = 26,618 gal/day ✓

(3) Summation of $Q_{SS} + Q_{BASE} = Q_T$ Equation #3

Q_T = 17,332 + 26,618 gal/day

Q_T = 43,950 gal/day ✓

(4) Q_T / # of Sumps for TMA CELL 2 Equation #4

$Q_{T/2}$ = 43,950 / 2 = 21,975 GAL/DAY PER Sump ✓

Foth & Van Dyke

Client: CMC Scope I.D.: 93049
Project: TMA - Sump SIZE Page: 5 / 7
Prepared by: J.B KIRSCH (RBM) Date: 4/12/95
Checked by: MDF Date: 5/8/95

Solution Continued:

(5) DETERMINE VOLUME OF STORAGE CAPACITY of Sump, STONE & Liquid CAPACITY) AT POROSITY of 0.30. FOR STONE FILL

Equation #5: Vol x Porosity = $Q_{T/2}$

Volume = FT^3 of Storage Capacity of Sump PER DAY.

Porosity = 0.3

$Q_{T/2}$ = 21,975 gpd ~ 22,000 gpd/day ✓

Solve Equation #5 for Volume:

$$Vol = \frac{Q_{T/2}}{Porosity} = \frac{(22{,}000\ gpd/day)\left(\frac{1\ CF}{7.48\ gal}\right)}{0.30}$$

$$Vol = 9{,}804\ CF/day$$ ✓

(6) DETERMINE Sump Volume of Capacity FOR/cycle

Equation "6": Volume / # of Cycles = Sump Storage Volume PER cycle

WHERE Volume = 9,804 CF/day

# of Cycles = 6/day

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA - Sump Size Page: 6/7
Prepared by: JB Kosch Date: 4/12/95
Checked by: MDF Date: 5/3/95

Equation #6

9,804 CF/day / 6 cycle/day =

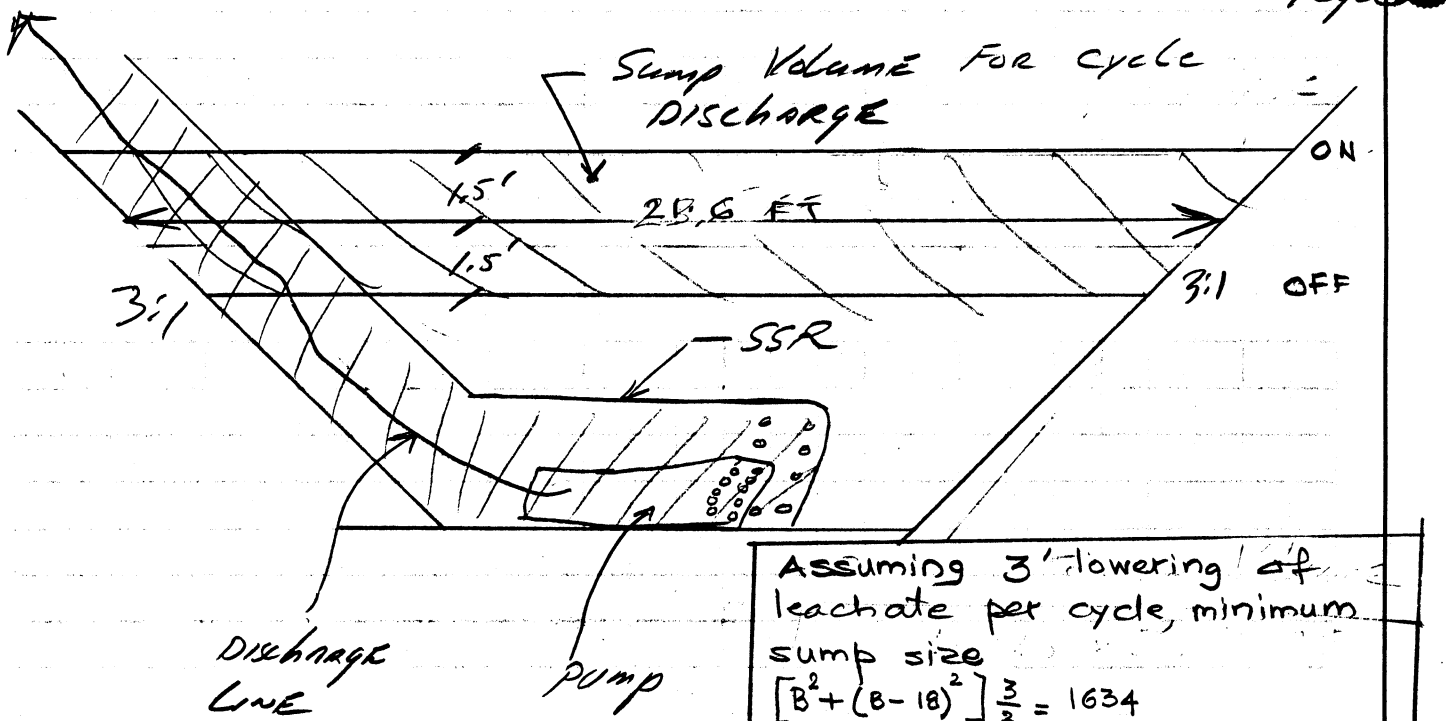
1,634 CF/cycle - SUMP volume with stone @ 0.30 porosity ✓

Also -

22,000 gal/day / 6 cycle/day =

3,667 gal/cycle of LEACHATE DISCHARGED ✓

which is equivalent to 490 CF/cycle ✓



Assuming 3' lowering of leachate per cycle, minimum sump size

$[B^2 + (B-18)^2]\frac{3}{2} = 1634$

$B^2 + B^2 - 36B + 324 = 1089$

$2B^2 - 36B - 765 = 0$

$B = 30.6$ ft

Foth & Van Dyke

Client: CMC Scope I.D.: 93049
Project: TMA - Sump Size Page: 7/7 1A+1B
Prepared by: QB/CA Date: 4/12/95
Checked by: MD. Fem Date: 5/8/95

ATTACHMENT A

HELP MODEL FOR LEACHATE

Sump STORAGE CAPACITY - DETERMINATION

— SIDESLOPE —

SS-45 FT TAILINGS
NO COVER
300 FT DRAIN
2 X PRECIP

A1

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**********************************************************************************
**********************************************************************************
**                                                                              **
**                                                                              **
**              HYDROLOGIC EVALUATION OF LANDFILL PERFORMANCE                   **
**                HELP MODEL VERSION 3.01   (14 OCTOBER 1994)                   **
**                   DEVELOPED BY ENVIRONMENTAL LABORATORY                      **
**                     USAE WATERWAYS EXPERIMENT STATION                        **
**                FOR USEPA RISK REDUCTION ENGINEERING LABORATORY               **
**                                                                              **
**                                                                              **
**********************************************************************************
**********************************************************************************


PRECIPITATION DATA FILE:     C:\HELP3\round6\ss182x\op.D4
TEMPERATURE DATA FILE:       C:\HELP3\round6\ss182x\OP.D7
SOLAR RADIATION DATA FILE:   C:\HELP3\round6\ss182x\OP.D13
EVAPOTRANSPIRATION DATA:     C:\HELP3\round6\ss182x\op.D11
SOIL AND DESIGN DATA FILE:   C:\HELP3\round6\ss182x\SS300452.D10
OUTPUT DATA FILE:            C:\HELP3\round6\ss182x\ss300452.OUT


TIME:   8:57     DATE:   4/ 9/1995


**********************************************************************************

     TITLE:  CMC TMA: SS FULL NO COV 2xPRCIP 300 FT DR LENGTH:SS300452xR6

**********************************************************************************


     NOTE:  INITIAL MOISTURE CONTENT OF THE LAYERS AND SNOW WATER
              WERE SPECIFIED BY THE USER.
```

A-2

LAYER 1

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	60.00	INCHES
POROSITY	=	0.5803	VOL/VOL
FIELD CAPACITY	=	0.5100	VOL/VOL
WILTING POINT	=	0.3400	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5803	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.399999999000E-05	CM/SEC

LAYER 2

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5529	VOL/VOL
FIELD CAPACITY	=	0.5066	VOL/VOL
WILTING POINT	=	0.3366	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5529	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.365999995000E-05	CM/SEC

LAYER 3

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5445	VOL/VOL
FIELD CAPACITY	=	0.5033	VOL/VOL
WILTING POINT	=	0.3333	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5445	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.333000003000E-05	CM/SEC

A-3

LAYER 4

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5385	VOL/VOL
FIELD CAPACITY	=	0.5000	VOL/VOL
WILTING POINT	=	0.3300	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5385	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.300000011000E-05	CM/SEC

LAYER 5

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5338	VOL/VOL
FIELD CAPACITY	=	0.4965	VOL/VOL
WILTING POINT	=	0.3265	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5338	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.266000006000E-05	CM/SEC

LAYER 6

TYPE 2 - LATERAL DRAINAGE LAYER
MATERIAL TEXTURE NUMBER 10

THICKNESS	=	18.00	INCHES
POROSITY	=	0.3980	VOL/VOL
FIELD CAPACITY	=	0.2440	VOL/VOL
WILTING POINT	=	0.1360	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.2440	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.119999997000E-03	CM/SEC
SLOPE	=	33.33	PERCENT
DRAINAGE LENGTH	=	300.0	FEET

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LAYER 7

TYPE 4 - FLEXIBLE MEMBRANE LINER
MATERIAL TEXTURE NUMBER 35

THICKNESS	=	0.06	INCHES
POROSITY	=	0.0000	VOL/VOL
FIELD CAPACITY	=	0.0000	VOL/VOL
WILTING POINT	=	0.0000	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.0000	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.199999996000E-12	CM/SEC
FML PINHOLE DENSITY	=	1.00	HOLES/ACRE
FML INSTALLATION DEFECTS	=	4.00	HOLES/ACRE
FML PLACEMENT QUALITY	=	3 - GOOD	

LAYER 8

TYPE 3 - BARRIER SOIL LINER
MATERIAL TEXTURE NUMBER 16

THICKNESS	=	12.00	INCHES
POROSITY	=	0.4270	VOL/VOL
FIELD CAPACITY	=	0.4180	VOL/VOL
WILTING POINT	=	0.3670	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.4270	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.100000001000E-06	CM/SEC

GENERAL DESIGN AND EVAPORATIVE ZONE DATA

NOTE: SCS RUNOFF CURVE NUMBER WAS USER-SPECIFIED.

SCS RUNOFF CURVE NUMBER	=	0.00	
FRACTION OF AREA ALLOWING RUNOFF	=	0.0	PERCENT
AREA PROJECTED ON HORIZONTAL PLANE	=	200.000	ACRES
EVAPORATIVE ZONE DEPTH	=	8.0	INCHES
INITIAL WATER IN EVAPORATIVE ZONE	=	4.642	INCHES
UPPER LIMIT OF EVAPORATIVE STORAGE	=	4.642	INCHES
LOWER LIMIT OF EVAPORATIVE STORAGE	=	2.720	INCHES
INITIAL SNOW WATER	=	0.000	INCHES
INITIAL WATER IN LAYER MATERIALS	=	304.698	INCHES
TOTAL INITIAL WATER	=	304.698	INCHES
TOTAL SUBSURFACE INFLOW	=	0.00	INCHES/YEAR

EVAPOTRANSPIRATION AND WEATHER DATA

NOTE: EVAPOTRANSPIRATION DATA WAS OBTAINED FROM
GREEN BAY WISCONSIN

MAXIMUM LEAF AREA INDEX	=	0.00
START OF GROWING SEASON (JULIAN DATE)	=	130
END OF GROWING SEASON (JULIAN DATE)	=	275
AVERAGE ANNUAL WIND SPEED	=	10.10 MPH

AVERAGE 1ST QUARTER RELATIVE HUMIDITY = 73.00 %
AVERAGE 2ND QUARTER RELATIVE HUMIDITY = 68.00 %
AVERAGE 3RD QUARTER RELATIVE HUMIDITY = 74.00 %
AVERAGE 4TH QUARTER RELATIVE HUMIDITY = 76.00 %

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NOTE: PRECIPITATION DATA WAS SYNTHETICALLY GENERATED USING
COEFFICIENTS FOR GREEN BAY WISCONSIN

NORMAL MEAN MONTHLY PRECIPITATION (INCHES)

JAN/JUL	FEB/AUG	MAR/SEP	APR/OCT	MAY/NOV	JUN/DEC
2.16	1.64	3.24	4.70	6.36	7.16
7.46	8.28	7.98	4.92	3.92	2.90

NOTE: ***Precipitation values are 2 times the normal mean monthly values to represent ponding ontop of the TMA, per phone conversation with Paul Schroeder on 3/28/95.***

NOTE: TEMPERATURE DATA WAS SYNTHETICALLY GENERATED USING
COEFFICIENTS FOR GREEN BAY WISCONSIN

NORMAL MEAN MONTHLY TEMPERATURE (DEGREES FAHRENHEIT)

JAN/JUL	FEB/AUG	MAR/SEP	APR/OCT	MAY/NOV	JUN/DEC
10.20	14.20	26.00	40.40	53.20	61.50
66.10	63.30	54.90	44.70	30.50	15.50

NOTE: SOLAR RADIATION DATA WAS SYNTHETICALLY GENERATED USING
COEFFICIENTS FOR GREEN BAY WISCONSIN

STATION LATITUDE = 45.29 DEGREES

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ANNUAL TOTALS FOR YEAR 1

	INCHES	CU. FEET	PERCENT
PRECIPITATION	53.54	38870036.000	100.00
RUNOFF	0.000	0.000	0.00
EVAPOTRANSPIRATION	33.148	24065186.000	61.91
DRAINAGE COLLECTED FROM LAYER 6	4.6617	3384362.500	8.71
PERC./LEAKAGE THROUGH LAYER 8	0.968434	703082.750	1.81
AVG. HEAD ON TOP OF LAYER 8	473.0399		
CHANGE IN WATER STORAGE	14.762	10717411.000	27.57
SOIL WATER AT START OF YEAR	304.698	221210656.000	
SOIL WATER AT END OF YEAR	307.106	222958768.000	
SNOW WATER AT START OF YEAR	0.000	0.000	0.00
SNOW WATER AT END OF YEAR	0.879	638377.312	1.64
ANNUAL WATER BUDGET BALANCE	0.0000	-4.976	0.00

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ANNUAL TOTALS FOR YEAR 10

	INCHES	CU. FEET	PERCENT
PRECIPITATION	56.70	41164192.000	100.00
RUNOFF	0.000	0.000	0.00
EVAPOTRANSPIRATION	37.088	26926108.000	65.41
DRAINAGE COLLECTED FROM LAYER 6	5.1352	3728190.750	9.06
PERC./LEAKAGE THROUGH LAYER 8	1.145948	831958.312	2.02
AVG. HEAD ON TOP OF LAYER 8	543.8355		
CHANGE IN WATER STORAGE	199.769	145032096.000	352.33
SOIL WATER AT START OF YEAR	306.968	222858800.000	
SOIL WATER AT END OF YEAR	306.660	222635296.000	
SNOW WATER AT START OF YEAR	1.712	1242647.370	3.02
SNOW WATER AT END OF YEAR	2.037	1478597.000	3.59
ANNUAL WATER BUDGET BALANCE	-186.4382	**********	-328.82

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AVERAGE MONTHLY VALUES IN INCHES FOR YEARS 1 THROUGH 10

	JAN/JUL	FEB/AUG	MAR/SEP	APR/OCT	MAY/NOV	JUN/DEC
PRECIPITATION						
TOTALS	1.75	1.62	3.26	4.92	6.91	8.20
	9.82	8.33	7.32	4.44	3.65	2.70
STD. DEVIATIONS	1.20	1.02	1.74	1.81	3.74	3.52
	4.10	3.17	2.81	2.31	1.29	1.56
RUNOFF						
TOTALS	0.000	0.000	0.000	0.000	0.000	0.000
	0.000	0.000	0.000	0.000	0.000	0.000
STD. DEVIATIONS	0.000	0.000	0.000	0.000	0.000	0.000
	0.000	0.000	0.000	0.000	0.000	0.000
EVAPOTRANSPIRATION						
TOTALS	0.423	0.455	1.216	3.272	5.457	6.478
	6.603	5.487	3.486	2.022	1.006	0.424
STD. DEVIATIONS	0.074	0.118	0.441	0.573	0.682	0.344
	0.492	0.368	0.176	0.224	0.194	0.058
LATERAL DRAINAGE COLLECTED FROM LAYER 6						
TOTALS	0.4097	0.3826	0.4179	0.3979	0.4235	0.4275
	0.4434	0.4444	0.4330	0.4476	0.4316	0.4382
STD. DEVIATIONS	0.0801	0.0079	0.0071	0.0066	0.0167	0.0175
	0.0134	0.0101	0.0003	0.0000	0.0026	0.0029
PERCOLATION/LEAKAGE THROUGH LAYER 8						
TOTALS	0.0868	0.0833	0.0910	0.0860	0.0930	0.0961
	0.0999	0.1002	0.0980	0.1013	0.0975	0.0979
STD. DEVIATIONS	0.0303	0.0041	0.0024	0.0024	0.0059	0.0061
	0.0046	0.0034	0.0001	0.0001	0.0009	0.0010

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AVERAGES OF MONTHLY AVERAGED DAILY HEADS (INCHES)

DAILY AVERAGE HEAD ACROSS LAYER 8

AVERAGES	485.3058	524.7463	521.5541	513.7458	528.2382	550.6833
	552.8095	554.1472	557.8800	557.9999	555.9827	546.2274
STD. DEVIATIONS	167.5049	20.4282	8.6694	9.2347	21.2458	22.9316
	16.4125	12.1832	0.3762	0.0706	3.3166	3.6182

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AVERAGE ANNUAL TOTALS & (STD. DEVIATIONS) FOR YEARS 1 THROUGH 10

	INCHES	CU. FEET	PERCENT
PRECIPITATION	62.93 (5.750)	45688636.0	100.00
RUNOFF	0.000 (0.0000)	0.00	0.000
EVAPOTRANSPIRATION	36.329 (1.2520)	26374646.00	57.727
LATERAL DRAINAGE COLLECTED FROM LAYER 6	5.09729 (0.15381)	3700634.250	8.09968
PERCOLATION/LEAKAGE THROUGH FROM LAYER 8	1.13119 (0.05742)	821243.312	1.79748
AVERAGE HEAD ACROSS TOP OF LAYER 8	537.443 (22.687)		
CHANGE IN WATER STORAGE	109.524 (62.2791)	79514160.00	174.035

****

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: Sump Size	Page: B
	Prepared by: JBK	Date: 4/12/95
	Checked by: MD Fun	Date: 5/2/95

ATTACHMENT B

HELP MODEL FOR LEACHATE Sump Storage Capacity DETERMINATION

- BASE -

BASE: 90 FT. TAILINGS, NO COVER
2X PRECIP.

```
**********************************************************************
**********************************************************************
**                                                                  **
**                                                                  **
**              HYDROLOGIC EVALUATION OF LANDFILL PERFORMANCE       **
**                HELP MODEL VERSION 3.01   (14 OCTOBER 1994)       **
**                   DEVELOPED BY ENVIRONMENTAL LABORATORY          **
**                     USAE WATERWAYS EXPERIMENT STATION            **
**              FOR USEPA RISK REDUCTION ENGINEERING LABORATORY     **
**                                                                  **
**                                                                  **
**********************************************************************
**********************************************************************
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B1

for use AL

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PRECIPITATION DATA FILE:     C:\HELP3\ROUND6\SS183X\OP.D4
TEMPERATURE DATA FILE:       C:\HELP3\ROUND6\SS183X\OP.D7
SOLAR RADIATION DATA FILE:   C:\HELP3\ROUND6\SS183X\OP.D13
EVAPOTRANSPIRATION DATA:     C:\HELP3\ROUND6\SS183X\OP.D11
SOIL AND DESIGN DATA FILE:   C:\HELP3\ROUND6\BASE3X\FULL2X.D10
OUTPUT DATA FILE:            C:\HELP3\ROUND6\BASE3X\FULL2X.OUT


TIME:    6: 4     DATE:    4/ 7/1995


**********************************************************************
     TITLE:  CMC TMA: FULL WIHTH 90 FT OF TAILING UNCOVERED: FILE (FULL)-FULL2X R6
**********************************************************************


     NOTE:  INITIAL MOISTURE CONTENT OF THE LAYERS AND SNOW WATER
              WERE SPECIFIED BY THE USER.
```

B-2

LAYER 1

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5803	VOL/VOL
FIELD CAPACITY	=	0.5100	VOL/VOL
WILTING POINT	=	0.3400	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5803	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.399999999000E-05	CM/SEC

LAYER 2

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5529	VOL/VOL
FIELD CAPACITY	=	0.5066	VOL/VOL
WILTING POINT	=	0.3366	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5529	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.365999995000E-05	CM/SEC

LAYER 3

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5445	VOL/VOL
FIELD CAPACITY	=	0.5033	VOL/VOL
WILTING POINT	=	0.3333	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5445	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.333000003000E-05	CM/SEC

LAYER 4

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5385	VOL/VOL
FIELD CAPACITY	=	0.5000	VOL/VOL
WILTING POINT	=	0.3300	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5385	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.300000011000E-05	CM/SEC

B-3

LAYER 5

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5338	VOL/VOL
FIELD CAPACITY	=	0.4965	VOL/VOL
WILTING POINT	=	0.3265	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5338	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.266000006000E-05	CM/SEC

LAYER 6

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5297	VOL/VOL
FIELD CAPACITY	=	0.4930	VOL/VOL
WILTING POINT	=	0.3230	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5297	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.232999992000E-05	CM/SEC

LAYER 7

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5259	VOL/VOL
FIELD CAPACITY	=	0.4915	VOL/VOL
WILTING POINT	=	0.3215	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5259	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.199999999000E-05	CM/SEC

LAYER 8

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5226	VOL/VOL
FIELD CAPACITY	=	0.4905	VOL/VOL
WILTING POINT	=	0.3205	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5226	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.150000005000E-05	CM/SEC

B-4

LAYER 9

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 0

THICKNESS	=	120.00	INCHES
POROSITY	=	0.5195	VOL/VOL
FIELD CAPACITY	=	0.4900	VOL/VOL
WILTING POINT	=	0.3200	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.5195	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.999999997000E-06	CM/SEC

LAYER 10

TYPE 1 - VERTICAL PERCOLATION LAYER
MATERIAL TEXTURE NUMBER 10

THICKNESS	=	24.00	INCHES
POROSITY	=	0.3980	VOL/VOL
FIELD CAPACITY	=	0.2440	VOL/VOL
WILTING POINT	=	0.1360	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.2440	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.119999997000E-03	CM/SEC

LAYER 11

TYPE 2 - LATERAL DRAINAGE LAYER
MATERIAL TEXTURE NUMBER 21

THICKNESS	=	24.00	INCHES
POROSITY	=	0.3970	VOL/VOL
FIELD CAPACITY	=	0.0320	VOL/VOL
WILTING POINT	=	0.0130	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.0320	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.300000012000	CM/SEC
SLOPE	=	2.00	PERCENT
DRAINAGE LENGTH	=	940.0	FEET

B-5

LAYER 12

TYPE 4 - FLEXIBLE MEMBRANE LINER
MATERIAL TEXTURE NUMBER 35

THICKNESS	=	0.06	INCHES
POROSITY	=	0.0000	VOL/VOL
FIELD CAPACITY	=	0.0000	VOL/VOL
WILTING POINT	=	0.0000	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.0000	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.199999996000E-12	CM/SEC
FML PINHOLE DENSITY	=	1.00	HOLES/ACRE
FML INSTALLATION DEFECTS	=	4.00	HOLES/ACRE
FML PLACEMENT QUALITY	=	3 - GOOD	

LAYER 13

TYPE 3 - BARRIER SOIL LINER
MATERIAL TEXTURE NUMBER 16

THICKNESS	=	12.00	INCHES
POROSITY	=	0.4270	VOL/VOL
FIELD CAPACITY	=	0.4180	VOL/VOL
WILTING POINT	=	0.3670	VOL/VOL
INITIAL SOIL WATER CONTENT	=	0.4270	VOL/VOL
EFFECTIVE SAT. HYD. COND.	=	0.100000001000E-06	CM/SEC

GENERAL DESIGN AND EVAPORATIVE ZONE DATA

NOTE: SCS RUNOFF CURVE NUMBER WAS USER-SPECIFIED.

SCS RUNOFF CURVE NUMBER	=	0.00	
FRACTION OF AREA ALLOWING RUNOFF	=	0.0	PERCENT
AREA PROJECTED ON HORIZONTAL PLANE	=	200.000	ACRES
EVAPORATIVE ZONE DEPTH	=	8.0	INCHES
INITIAL WATER IN EVAPORATIVE ZONE	=	4.642	INCHES
UPPER LIMIT OF EVAPORATIVE STORAGE	=	4.642	INCHES
LOWER LIMIT OF EVAPORATIVE STORAGE	=	2.720	INCHES
INITIAL SNOW WATER	=	0.000	INCHES
INITIAL WATER IN LAYER MATERIALS	=	593.472	INCHES
TOTAL INITIAL WATER	=	593.472	INCHES
TOTAL SUBSURFACE INFLOW	=	0.00	INCHES/YEAR

EVAPOTRANSPIRATION AND WEATHER DATA

B-6

NOTE: EVAPOTRANSPIRATION DATA WAS OBTAINED FROM
GREEN BAY WISCONSIN

MAXIMUM LEAF AREA INDEX	=	0.00
START OF GROWING SEASON (JULIAN DATE)	=	130
END OF GROWING SEASON (JULIAN DATE)	=	275
AVERAGE ANNUAL WIND SPEED	=	10.10 MPH
AVERAGE 1ST QUARTER RELATIVE HUMIDITY	=	73.00 %
AVERAGE 2ND QUARTER RELATIVE HUMIDITY	=	68.00 %
AVERAGE 3RD QUARTER RELATIVE HUMIDITY	=	74.00 %
AVERAGE 4TH QUARTER RELATIVE HUMIDITY	=	76.00 %

NOTE: PRECIPITATION DATA WAS SYNTHETICALLY GENERATED USING
COEFFICIENTS FOR GREEN BAY WISCONSIN

NORMAL MEAN MONTHLY PRECIPITATION (INCHES)

JAN/JUL	FEB/AUG	MAR/SEP	APR/OCT	MAY/NOV	JUN/DEC
2.16	1.64	3.24	4.70	6.36	7.16
7.46	8.28	7.98	4.92	3.92	2.90

NOTE: ***Precipitation values are 2 times the normal mean monthly values to represent ponding ontop of the TMA per Paul Schroeder phone conversation of 3/28/95.***

NOTE: TEMPERATURE DATA WAS SYNTHETICALLY GENERATED USING
COEFFICIENTS FOR GREEN BAY WISCONSIN

NORMAL MEAN MONTHLY TEMPERATURE (DEGREES FAHRENHEIT)

JAN/JUL	FEB/AUG	MAR/SEP	APR/OCT	MAY/NOV	JUN/DEC
10.20	14.20	26.00	40.40	53.20	61.50
66.10	63.30	54.90	44.70	30.50	15.50

NOTE: SOLAR RADIATION DATA WAS SYNTHETICALLY GENERATED USING
COEFFICIENTS FOR GREEN BAY WISCONSIN

STATION LATITUDE = 45.29 DEGREES

B-7

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ANNUAL TOTALS FOR YEAR 1

	INCHES	CU. FEET	PERCENT
PRECIPITATION	53.54	38870036.000	100.00
RUNOFF	0.000	0.000	0.00
EVAPOTRANSPIRATION	33.148	24065186.000	61.91
DRAINAGE COLLECTED FROM LAYER 11	10.5703	7674029.000	19.74
PERC./LEAKAGE THROUGH LAYER 13	0.000816	592.597	0.00
AVG. HEAD ON TOP OF LAYER 13	0.7976		
CHANGE IN WATER STORAGE	9.821	7130318.000	18.34
SOIL WATER AT START OF YEAR	593.472	430860544.000	
SOIL WATER AT END OF YEAR	594.602	431681088.000	
SNOW WATER AT START OF YEAR	0.000	0.000	0.00
SNOW WATER AT END OF YEAR	0.879	638377.312	1.64
ANNUAL WATER BUDGET BALANCE	-0.0001	-88.554	0.00

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ANNUAL TOTALS FOR YEAR 10

	INCHES	CU. FEET	PERCENT
PRECIPITATION	56.70	41164192.000	100.00
RUNOFF	0.000	0.000	0.00
EVAPOTRANSPIRATION	37.089	26926302.000	65.41
DRAINAGE COLLECTED FROM LAYER 11	12.4146	9013030.000	21.90
PERC./LEAKAGE THROUGH LAYER 13	0.000956	694.359	0.00
AVG. HEAD ON TOP OF LAYER 13	0.9403		
CHANGE IN WATER STORAGE	141.084	102426928.000	248.83
SOIL WATER AT START OF YEAR	594.330	431483488.000	
SOIL WATER AT END OF YEAR	593.718	431039008.000	
SNOW WATER AT START OF YEAR	1.712	1242647.370	3.02
SNOW WATER AT END OF YEAR	2.037	1478597.000	3.59
ANNUAL WATER BUDGET BALANCE	-133.8881	-97202768.000	-236.13

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AVERAGE MONTHLY VALUES IN INCHES FOR YEARS 1 THROUGH 10

B-9

	JAN/JUL	FEB/AUG	MAR/SEP	APR/OCT	MAY/NOV	JUN/DEC
PRECIPITATION						
TOTALS	1.75	1.62	3.26	4.92	6.91	8.20
	9.82	8.33	7.32	4.44	3.65	2.70
STD. DEVIATIONS	1.20	1.02	1.74	1.81	3.74	3.52
	4.10	3.17	2.81	2.31	1.29	1.56
RUNOFF						
TOTALS	0.000	0.000	0.000	0.000	0.000	0.000
	0.000	0.000	0.000	0.000	0.000	0.000
STD. DEVIATIONS	0.000	0.000	0.000	0.000	0.000	0.000
	0.000	0.000	0.000	0.000	0.000	0.000
EVAPOTRANSPIRATION						
TOTALS	0.423	0.455	1.216	3.272	5.457	6.478
	6.603	5.487	3.486	2.022	1.006	0.424
STD. DEVIATIONS	0.074	0.118	0.441	0.573	0.682	0.344
	0.492	0.368	0.176	0.224	0.194	0.058
LATERAL DRAINAGE COLLECTED FROM LAYER 11						
TOTALS	0.9558	0.8812	1.0468	1.0200	1.0544	1.0204
	1.0544	1.0544	1.0204	1.0544	1.0204	1.0544
STD. DEVIATIONS	0.3360	0.2250	0.0238	0.0010	0.0000	0.0002
	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
PERCOLATION/LEAKAGE THROUGH LAYER 13						
TOTALS	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001
	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001
STD. DEVIATIONS	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000

B-10

AVERAGES OF MONTHLY AVERAGED DAILY HEADS (INCHES)

DAILY AVERAGE HEAD ACROSS LAYER 13

AVERAGES	0.8463	0.8701	0.9336	0.9400	0.9403	0.9403
	0.9403	0.9403	0.9403	0.9403	0.9403	0.9403
STD. DEVIATIONS	0.2972	0.2221	0.0213	0.0009	0.0000	0.0000
	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000

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B-11

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AVERAGE ANNUAL TOTALS & (STD. DEVIATIONS) FOR YEARS 1 THROUGH 10

	INCHES	CU. FEET	PERCENT
PRECIPITATION	62.93 (5.750)	45688636.0	100.00
RUNOFF	0.000 (0.0000)	0.00	0.000
EVAPOTRANSPIRATION	36.329 (1.2519)	26374804.00	57.727
LATERAL DRAINAGE COLLECTED FROM LAYER 11	12.23701 (0.58578)	8884069.000	19.44481
PERCOLATION/LEAKAGE THROUGH FROM LAYER 13	0.00094 (0.00004)	684.563	0.00150
AVERAGE HEAD ACROSS TOP OF LAYER 13	0.926 (0.045)		
CHANGE IN WATER STORAGE	78.353 (44.0573)	56884272.00	124.504

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CMC TMA PUMP CURVE DETERMINATION
TMA CELL 1 NORTH SUMP LEACHATE PUMP
STAGE II CONDITION

SCOPE: 93C049 FILE: CMCPUMP.WQ1 (JBK DISK TO REM)

PREPARED BY: [signature] DATE: 4/22/95

CHECKED BY: [signature] DATE: 5/10/95

LINE	ITEM	Ke	Le
3"	ENTRANCE	0.50	-
"	PUMP	-	-
"	CHECK VALVE	2.00	-
"	3:1 BEND (18.42)	0.10	-
"	142	-	142
"	3:1 BEND (18.42)	0.10	-
"	20	-	20
"	3:1 BEND (18.42)	0.10	-
"	142	-	142
"	3:1 BEND (18.42)	0.10	-
"	AIR REL. VALVE	2.00	-
"	GATE VALVE	0.30	-
"	REDUCER	0.25	-
TOTAL		5.45	304

NOTE:
TERMINATION AT CENTER OF TMA 1-4 INTO GRAVITY PIPE TO RECLAIM POND

PIPE ROUGHNESS COEFFICIENT = 120.00

FLOW	PIPE DIA	VELOCITY	Ke	HKe	Le	SLOPE	HLe	Ht	STATIC	TOTAL LOSS
0	3.072	0.00	5.45	0.00000	304	0.00000	0.00	0.00	85	85
25	3.072	1.08	5.45	0.09905	304	0.00245	0.75	0.84	85	86
50	3.072	2.16	5.45	0.39619	304	0.00884	2.69	3.08	85	88
75	3.072	3.25	5.45	0.89142	304	0.01873	5.69	6.58	85	92
100	3.072	4.33	5.45	1.58475	304	0.03189	9.69	11.28	85	96
125	3.072	5.41	5.45	2.47617	304	0.04818	14.65	17.12	85	102
150	3.072	6.49	5.45	3.56569	304	0.06751	20.52	24.09	85	109
175	3.072	7.57	5.45	4.85330	304	0.08979	27.30	32.15	85	117
200	3.072	8.65	5.45	6.33901	304	0.11495	34.94	41.28	85	126
225	3.072	9.74	5.45	8.02281	304	0.14293	43.45	51.47	85	136
250	3.072	10.82	5.45	9.90470	304	0.17369	52.80	62.71	85	148

CMC TMA PUMP CURVE DETERMINATION
TMA CELL 1 SOUTH SUMP LEACHATE PUMP
STAGE II CONDITION

SCOPE: 93C049 FILE: CMCPUMP.WQ1 (JBK DISK TO REM)

PREPARED BY: DATE: 4/22/95

CHECKED BY: DATE: 5/10/95

LINE	ITEM	Ke	Le
3"	ENTRANCE	0.50	-
"	PUMP	-	-
"	CHECK VALVE	2.00	-
"	3:1 BEND (18.42)	0.10	-
"	133	-	133
"	3:1 BEND (18.42)	0.10	-
"	20	-	20
"	3:1 BEND (18.42)	0.10	-
"	133	-	133
"	3:1 BEND (18.42)	0.10	-
"	AIR REL. VALVE	2.00	-
"	GATE VALVE	0.30	-
"	775	-	775
"	AIR REL. VALVE	2.00	-
"	775	-	775
"	GATE VALVE	0.30	-
"	REDUCER	0.25	-
TOTAL		7.75	1,836

NOTE:
TERMINATION AT CENTER OF TMA 1-4 INTO GRAVITY PIPE TO RECLAIM POND

PIPE ROUGHNESS COEFFICIENT = 120.00

FLOW	PIPE DIA	VELOCITY	Ke	HKe	Le	SLOPE	HLe	Ht	STATIC	TOTAL LOSS
0	3.072	0.00	7.75	0.00000	1,836	0.00000	0.00	0.00	90	90
25	3.072	1.08	7.75	0.14085	1,836	0.00245	4.50	4.65	90	95
50	3.072	2.16	7.75	0.56339	1,836	0.00884	16.24	16.80	90	107
75	3.072	3.25	7.75	1.26762	1,836	0.01873	34.38	35.65	90	126
100	3.072	4.33	7.75	2.25355	1,836	0.03189	58.54	60.80	90	151
125	3.072	5.41	7.75	3.52117	1,836	0.04818	88.46	91.98	90	182
150	3.072	6.49	7.75	5.07048	1,836	0.06751	123.95	129.02	90	219
175	3.072	7.57	7.75	6.90149	1,836	0.08979	164.85	171.75	90	262
200	3.072	8.65	7.75	9.01419	1,836	0.11495	211.04	220.06	90	310
225	3.072	9.74	7.75	11.40858	1,836	0.14293	262.42	273.83	90	364
250	3.072	10.82	7.75	14.08466	1,836	0.17369	318.90	332.98	90	423

CMC TMA PUMP CURVE DETERMINATION
TMA CELL 2 NORTH SUMP LEACHATE PUMP
STAGE II CONDITION

SCOPE: 93C049 FILE: CMCPUMP.WQ1 (JBK DISK TO REM)

PREPARED BY: [signature] DATE: 4/22/95

CHECKED BY: [signature] DATE: 5/10/95

LINE	ITEM	Ke	Le
3"	ENTRANCE	0.50	-
"	PUMP	-	-
"	CHECK VALVE	2.00	-
"	3:1 BEND (18.42)	0.10	-
"	191	-	191
"	3:1 BEND (18.42)	0.10	-
"	20	-	20
"	3:1 BEND (18.42)	0.10	-
"	191	-	191
"	3:1 BEND (18.42)	0.10	-
"	AIR REL. VALVE	2.00	-
"	GATE VALVE	0.30	-
"	1000	-	1000
"	AIR REL. VALVE	2.00	-
"	1000	-	1000
"	GATE VALVE	0.30	-
"	REDUCER	0.25	-
TOTAL		7.75	2,402

NOTE:
TERMINATION AT CENTER OF TMA 1-4 INTO GRAVITY PIPE TO RECLAIM POND

PIPE ROUGHNESS COEFFICIENT = 120.00

FLOW	PIPE DIA	VELOCITY	Ke	HKe	Le	SLOPE	HLe	Ht	STATIC	TOTAL LOSS
0	3.072	0.00	7.75	0.00000	2,402	0.00000	0.00	0.00	125	125
25	3.072	1.08	7.75	0.14085	2,402	0.00245	5.89	6.03	125	131
50	3.072	2.16	7.75	0.56339	2,402	0.00884	21.25	21.81	125	147
75	3.072	3.25	7.75	1.26762	2,402	0.01873	44.98	46.25	125	171
100	3.072	4.33	7.75	2.25355	2,402	0.03189	76.59	78.84	125	204
125	3.072	5.41	7.75	3.52117	2,402	0.04818	115.73	119.25	125	244
150	3.072	6.49	7.75	5.07048	2,402	0.06751	162.16	167.23	125	292
175	3.072	7.57	7.75	6.90149	2,402	0.08979	215.67	222.57	125	348
200	3.072	8.65	7.75	9.01419	2,402	0.11495	276.10	285.12	125	410
225	3.072	9.74	7.75	11.40858	2,402	0.14293	343.32	354.73	125	480
250	3.072	10.82	7.75	14.08466	2,402	0.17369	417.21	431.29	125	556

CMC TMA PUMP CURVE DETERMINATION
TMA CELL 2 SOUTH SUMP LEACHATE PUMP
STAGE II CONDITION

SCOPE: 93C049 FILE: CMCPUMP.WQ1 (JBK DISK TO REM)

PREPARED BY: JBK DATE: 4/22/95

CHECKED BY: [signature] DATE: 5/10/95

LINE	ITEM	Ke	Le
3"	ENTRANCE	0.50	-
"	PUMP	-	-
"	CHECK VALVE	2.00	-
"	3:1 BEND (18.42)	0.10	-
"	191	-	191
"	3:1 BEND (18.42)	0.10	-
"	20	-	20
"	3:1 BEND (18.42)	0.10	-
"	191	-	191
"	3:1 BEND (18.42)	0.10	-
"	AIR REL. VALVE	2.00	-
"	GATE VALVE	0.30	-
"	REDUCER	0.25	-
TOTAL		5.45	402

NOTE:
TERMINATION AT CENTER OF TMA 1-4 INTO GRAVITY PIPE TO RECLAIM POND

PIPE ROUGHNESS COEFFICIENT = 120.00

FLOW	PIPE DIA	VELOCITY	Ke	HKe	Le	SLOPE	HLe	Ht	STATIC	TOTAL LOSS
0	3.072	0.00	5.45	0.00000	402	0.00000	0.00	0.00	115	115
25	3.072	1.08	5.45	0.09905	402	0.00245	0.99	1.09	115	116
50	3.072	2.16	5.45	0.39619	402	0.00884	3.56	3.95	115	119
75	3.072	3.25	5.45	0.89142	402	0.01873	7.53	8.42	115	123
100	3.072	4.33	5.45	1.58475	402	0.03189	12.82	14.40	115	129
125	3.072	5.41	5.45	2.47617	402	0.04818	19.37	21.84	115	137
150	3.072	6.49	5.45	3.56569	402	0.06751	27.14	30.70	115	146
175	3.072	7.57	5.45	4.85330	402	0.08979	36.09	40.95	115	156
200	3.072	8.65	5.45	6.33901	402	0.11495	46.21	52.55	115	168
225	3.072	9.74	5.45	8.02281	402	0.14293	57.46	65.48	115	180
250	3.072	10.82	5.45	9.90470	402	0.17369	69.82	79.73	115	195

CMC TMA PUMP CURVE DETERMINATION
TMA CELL 3 NORTH AND CELL 4 SOUTH SUMP LEACHATE PUMP
STAGE II CONDITION

SCOPE: 93C049 FILE: CMCPUMP.WQ1 (JBK DISK TO REM)

PREPARED BY: [signature] DATE: 4/22/95

CHECKED BY: [signature] DATE: 5/10/95

LINE	ITEM	Ke	Le
3"	ENTRANCE	0.50	-
"	PUMP	-	-
"	CHECK VALVE	2.00	-
"	3:1 BEND (18.42)	0.10	-
"	167	-	167
"	3:1 BEND (18.42)	0.10	-
"	20	-	20
"	3:1 BEND (18.42)	0.10	-
"	167	-	167
"	3:1 BEND (18.42)	0.10	-
"	AIR REL. VALVE	2.00	-
"	GATE VALVE	0.30	-
"	675	-	675
"	AIR REL. VALVE	2.00	-
"	675	-	675
"	GATE VALVE	0.30	-
"	REDUCER	0.25	-
TOTAL		7.75	1,704

NOTE:
TERMINATION AT CENTER OF TMA 1-4 INTO GRAVITY PIPE TO RECLAIM POND

PIPE ROUGHNESS COEFFICIENT = 120.00

FLOW	PIPE DIA	VELOCITY	Ke	HKe	Le	SLOPE	HLe	Ht	STATIC	TOTAL LOSS
0	3.072	0.00	7.75	0.00000	1,704	0.00000	0.00	0.00	110	110
25	3.072	1.08	7.75	0.14085	1,704	0.00245	4.18	4.32	110	114
50	3.072	2.16	7.75	0.56339	1,704	0.00884	15.07	15.63	110	126
75	3.072	3.25	7.75	1.26762	1,704	0.01873	31.91	33.18	110	143
100	3.072	4.33	7.75	2.25355	1,704	0.03189	54.33	56.59	110	167
125	3.072	5.41	7.75	3.52117	1,704	0.04818	82.10	85.62	110	196
150	3.072	6.49	7.75	5.07048	1,704	0.06751	115.03	120.11	110	230
175	3.072	7.57	7.75	6.90149	1,704	0.08979	153.00	159.90	110	270
200	3.072	8.65	7.75	9.01419	1,704	0.11495	195.87	204.88	110	315
225	3.072	9.74	7.75	11.40858	1,704	0.14293	243.56	254.96	110	365
250	3.072	10.82	7.75	14.08466	1,704	0.17369	295.97	310.06	110	420

CMC TMA PUMP CURVE DETERMINATION
TMA CELL 3 SOUTH SUMP LEACHATE PUMP
STAGE II CONDITION

SCOPE: 93C049 FILE: CMCPUMP.WQ1 (JBK DISK TO REM)

PREPARED BY: [signature] DATE: 4/22/95

CHECKED BY: [signature] DATE: 5/10/95

LINE	ITEM	Ke	Le
3"	ENTRANCE	0.50	-
"	PUMP	-	-
"	CHECK VALVE	2.00	-
"	3:1 BEND (18.42)	0.10	-
"	167	-	167
"	3:1 BEND (18.42)	0.10	-
"	20	-	20
"	3:1 BEND (18.42)	0.10	-
"	167	-	167
"	3:1 BEND (18.42)	0.10	-
"	AIR REL. VALVE	2.00	-
"	GATE VALVE	0.30	-
"	775	-	775
"	AIR REL. VALVE	2.00	-
"	775	-	775
"	GATE VALVE	0.30	-
"	REDUCER	0.25	-
TOTAL		7.75	1,904

NOTE:
TERMINATION AT CENTER OF TMA 1-4 INTO GRAVITY PIPE TO RECLAIM POND

PIPE ROUGHNESS COEFFICIENT = 120.00

FLOW	PIPE DIA	VELOCITY	Ke	HKe	Le	SLOPE	HLe	Ht	STATIC	TOTAL LOSS
0	3.072	0.00	7.75	0.00000	1,904	0.00000	0.00	0.00	110	110
25	3.072	1.08	7.75	0.14085	1,904	0.00245	4.67	4.81	110	115
50	3.072	2.16	7.75	0.56339	1,904	0.00884	16.84	17.40	110	127
75	3.072	3.25	7.75	1.26762	1,904	0.01873	35.66	36.92	110	147
100	3.072	4.33	7.75	2.25355	1,904	0.03189	60.71	62.96	110	173
125	3.072	5.41	7.75	3.52117	1,904	0.04818	91.74	95.26	110	205
150	3.072	6.49	7.75	5.07048	1,904	0.06751	128.54	133.61	110	244
175	3.072	7.57	7.75	6.90149	1,904	0.08979	170.95	177.86	110	288
200	3.072	8.65	7.75	9.01419	1,904	0.11495	218.86	227.87	110	338
225	3.072	9.74	7.75	11.40858	1,904	0.14293	272.14	283.55	110	394
250	3.072	10.82	7.75	14.08466	1,904	0.17369	330.71	344.79	110	455

CMC TMA PUMP CURVE DETERMINATION
TMA CELL 4 NORTH SUMP LEACHATE PUMP
STAGE II CONDITION

SCOPE: 93C049 FILE: CMCPUMP.WQ1 (JBK DISK TO REM)

PREPARED BY: DATE: 4/22/95

CHECKED BY: DATE: 5/10/95

LINE	ITEM	Ke	Le
3"	ENTRANCE	0.50	-
"	PUMP	-	-
"	CHECK VALVE	2.00	-
"	3:1 BEND (18.42)	0.10	-
"	175	-	175
"	3:1 BEND (18.42)	0.10	-
"	20	-	20
"	3:1 BEND (18.42)	0.10	-
"	175	-	175
"	3:1 BEND (18.42)	0.10	-
"	AIR REL. VALVE	2.00	-
"	GATE VALVE	0.30	-
"	1000	-	1000
"	AIR REL. VALVE	2.00	-
"	1000	-	1000
"	GATE VALVE	0.30	-
"	REDUCER	0.25	-
TOTAL		7.75	2,370

NOTE:
TERMINATION AT CENTER OF TMA 1-4 INTO GRAVITY PIPE TO RECLAIM POND

PIPE ROUGHNESS COEFFICIENT = 120.00

FLOW	PIPE DIA	VELOCITY	Ke	HKe	Le	SLOPE	HLe	Ht	STATIC	TOTAL LOSS
0	3.072	0.00	7.75	0.00000	2,370	0.00000	0.00	0.00	115	115
25	3.072	1.08	7.75	0.14085	2,370	0.00245	5.81	5.96	115	121
50	3.072	2.16	7.75	0.56339	2,370	0.00884	20.96	21.53	115	137
75	3.072	3.25	7.75	1.26762	2,370	0.01873	44.38	45.65	115	161
100	3.072	4.33	7.75	2.25355	2,370	0.03189	75.57	77.82	115	193
125	3.072	5.41	7.75	3.52117	2,370	0.04818	114.19	117.71	115	233
150	3.072	6.49	7.75	5.07048	2,370	0.06751	160.00	165.07	115	280
175	3.072	7.57	7.75	6.90149	2,370	0.08979	212.79	219.70	115	335
200	3.072	8.65	7.75	9.01419	2,370	0.11495	272.42	281.44	115	396
225	3.072	9.74	7.75	11.40858	2,370	0.14293	338.75	350.16	115	465
250	3.072	10.82	7.75	14.08466	2,370	0.17369	411.65	425.74	115	541

Appendix I

Side Slope Riser (SSR) Pipe Strength Analysis

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA SSR PIPE Page: 1/10
Prepared by: QB/KIASEK Date: 4/28/95
Checked by: N.P Date: 5/8/95

SSR PIPE CRUSHING ANALYSIS

PURPOSE : TO SELECT PROPER 18" PE PIPE SDR FOR SIDE SLOPE RISER WITH MAXIMUM LOADING CONDITIONS EXISTING IN TMA 1-4.

GIVEN - PIPE DIAMETER = 18"

- GREATES DEPTH OF FILL OVER THE SSR PIPE IS 110 FT
  - 5 FT STONE BACKFILL
  - 105 FT of TAILINGS

NOTE : ASSUME TOTAL VERTICAL COLUMN IS TAILINGS, BECAUSE GRADING LAYER MATERIAL IS UNKNOWN AT THIS TIME.

- TAILINGS DRY DENSITY = $\gamma_d = 97$ LB/CF

Conclusion = 18" PE PIPE SDR 11 WILL NOT EXCEED MAXIMUM MANUFACTURES RECOMMENDATION of 3% DEFLECTION FOR DESIGN SENARIO.

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA - SSR PIPE Page: 2/18
Prepared by: JBKinsick Date: 4/29/95
Checked by: N.P Date: 5-8-95

GIVEN Continued:

- UNIT WEIGHT of STONE BACKFILL is @ 130 LB/CF

- MANUFACTURES MAXIMUM ALLOWABLE PIPE DEFLECTION FOR 18" PE PIPE is 3%. (POLY PIPE INDUSTRIES)

Solution - CALCULATE PIPE DEFLECTION AND IMPOSED DESIGN / CONSTRUCTION METHODS TO LIMIT DEFLECTION LESS THAN MAXIMUM ALLOWABLE MANUFACTURES RECOMMENDATIONS

Equation #1

$$\Delta x = \frac{D_L \cdot K \cdot W_c}{\frac{2E}{3(SDR)^3} + 0.061E'}$$

WHERE: Δx = DEFLECTION (INCHES)

D_L = LAG FACTOR (1.25 - 1.5) ASSUME = 1.5 (CONSERVATIVE)

K = BEDDING CONSTANT CLASS FACTOR

W_c = MARSTON LOAD PER UNIT LENGTH LBS/LINEAL INCH

E' = Modulus of Passive Resistance of Crushed Rock

E = PIPE Modulus of ELASTICITY

Foth & Van Dyke

Client: CMC Scope I.D.: 930049
Project: TMA - SSR PIPE Page: 3/18
Prepared by: QBKiesz Date: 4/30/95
Checked by: N.P Date: 5-8-95

EQUATION #2 SDR = OD/T WHERE OD = OUTSIDE PIPE DIAMETER
T = WALL THICKNESS

(2) DETERMINE COLUMN LOAD ON SSR PIPE
(MAXIMUM DRUM LOAD)

EQUATION #3 • TAILINGS @ γ_d = 97 LB/CF ω = 34%

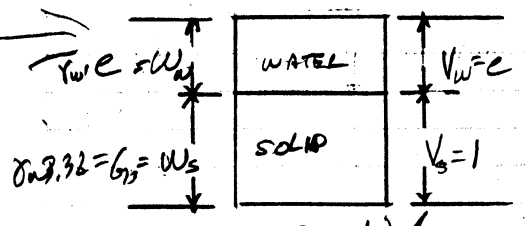
$\therefore \gamma_{moist} = (\gamma_d)(1+\omega) = (97\ LB/CF)(1+0.34) = 130.0\ LB/CF$

TAILINGS @ 105 FT · 130 LB/CF = 13,650 LB/FT²
SAME @ 5 FT 130 LB/CF = 650 LB/FT²
14,300 LB/FT²

MAXIMUM COLUMN LOAD ON SSR.
= 14,300 LB/FT²

(3) EQUATION #4 DETERMINE UNIT WEIGHT ON SSR

$(14,300\ LB/FT^2)\left(\frac{1}{110\ FT}\right) = 130.0\ LB/CF$



$\gamma_d = 97\ LB/CF$

$\gamma_d = \frac{G_s \cdot \gamma_w}{1+e}$

$1+e = \frac{G_s \cdot \gamma_w}{\gamma_d}$

$1+e = \frac{3.32 \cdot \gamma_w}{97}$

$1+e = 2.14$

$e = 1.14$

$w = \frac{e}{G_s}$

$w\% = \frac{1.14}{3.32} * 100 = 34\%$

Foth & Van Dyke	Client: CMC Scope I.D.: 93C049 Project: TMA-SSR PIPE Page: 4/18 Prepared by: JB Kursik Date: 4/30/95 Checked by: N.P Date: 5-8-95

(4) DETERMINE MARSTON LOAD W_c, PER UNIT LENGTH

EQUATION #5. $W_c = C_c W B_c^2$

WHERE: C_c = LOAD COEFFICIENT WHICH IS A FUNCTION OF THE PROJECTION RATIO (p) SETTLEMENT RATIO (r_{sd}) AND RATIO OF FILL HEIGHT (H) TO PIPE WIDTH (B_c) - SEE FIGURE #1

W = UNIT WIDTH = 1.44 S^{-1} LB/CF

B_c = OUTSIDE DIAMETER OF PIPE, FEET = (BREATH of CONDUIT)

18 in/12 in/ft = 1.5 ft

(SEE ATTACHMENT #2)

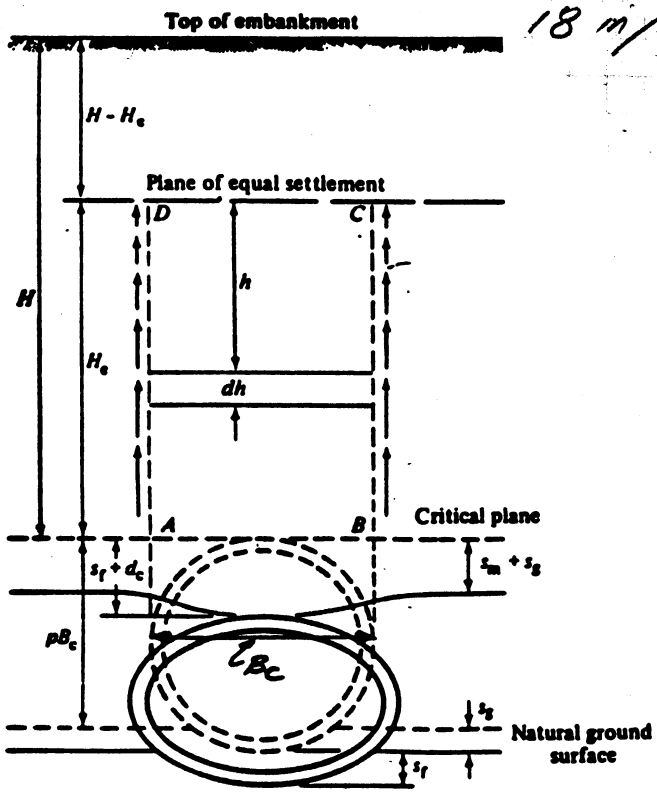


Figure 1 Settlements that influence loads on positive projecting conduits (incomplete ditch condition). Key same as in Figure 26.12.

SOURCE:

SPANGLER, M. G. & HANDY, R. L. (1982). SOIL ENGINEERING, HARPER & ROW, NEW YORK, NY

Foth & Van Dyke	Client: CMC	Scope I.D.: 93049
	Project: TMA - SSR - PIPE	Page: 5/18
	Prepared by: RKinsil	Date: 4/30/95
	Checked by: N.P	Date: 5-8-95

SOLUTION:
(CONTINUED)

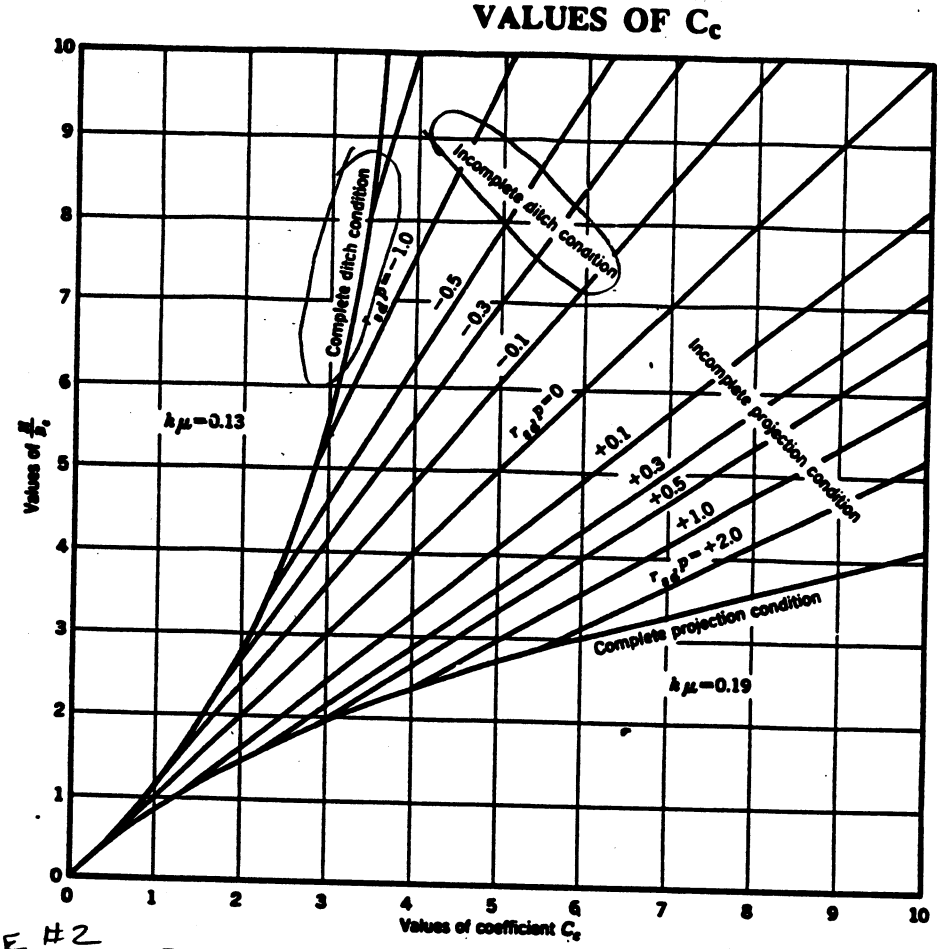


FIGURE #2

Diagram for coefficient C_c for positive projecting conduits.

SOURCE: GRAVITY SANITARY SEWER DESIGN AND CONSTRUCTION. "MANUALS & REPORTS ON ENGINEERING PRACTICE NO. 60," AMERICAN SOCIETY OF CIVIL ENGINEERS AND "MANUAL OF PRACTICE FD-5," WATER POLLUTION CONTROL FEDERATION. 1982.

4-a) NOTE: FOR FLEXIBLE PIPE UNDER MOST CONDITIONS THE PRODUCT $r_{sd} \cdot P$, IS LESS THAN ZERO (NEGATIVE VALUE) FOR A EMBANKMENT SITUATION SUCH AS A LANDFILL CONSTRUCTION, AN INCOMPLETE DITCH CONDITION EXISTS. BECAUSE $r_{sd} \cdot P$ IS NEGATIVE THE TOTAL PRISM LOAD IS NOT APPLIED TO THE PIPE.

4-b FOR THE PIPE VERTICAL PRESSURE (STRESS)

USE $K_o = 0.5$

K_o = Ratio of LATERAL EARTH PRESSURES to VERTICAL EARTH PREASSURES

4-c. DETERMINE $K_o u =$

Equation #6 WHERE $u = \tan \phi = \tan 10^\circ = 0.17$

ϕ = INTERNAL FRICTION ANGLE of TAKINGS

Equation #7 $K_o u = (0.5)(0.17) = 0.088 \approx 0.09$

4-d DETERMINE $r_{sd} = \dfrac{(S_m + S_g) - (S_f + d_c)}{S_m}$

WHERE:

r_{sd} = SETTELMENT RATIO (SEE Figure #1)

S_m = Compression STRAIN of the SIDE COLUMN of HEIGHT $p \cdot B_c$

where p = PROJECTION RATIO
B_c = WIDTH of PIPE.

S_g = SETTELMENT of the NATURAL SURFACE ADJACENT to the CONDUIT.

S_f = SETTELMENT of the CONDUIT INTO its FOUNDATION.

d_c = SHORTENING of the VERTICAL HEIGHT of the CONDUIT

4-d (ε_{sd}) - DETERMINATION CONTINUED

FOR ANALYSIS ASSUME $S_g = 0$, $S_f = 0$

Equation #9 S_M = LOAD STRAIN = $\left(\frac{\text{LOAD of PRISM}}{E_B \text{ of STONE}}\right)(B_c)$

Equation #10 L_p = LOAD of PRISM (unit weight)(HEIGHT)(WIDTH)

$L_p = \left(130 \frac{LB}{FT^3}\right)(1.10\,FT)(1.5\,FT) = 21{,}450 \frac{LB}{FT}$

$L_p = 21{,}450\ LB/FT$

E_B of STONE @ $5 \times 10^6\ LB/FT^2$ (CONSERVATIVE VALUE)

SEE ATTACHMENT #1.

Equation #9 (AGAIN)

$S_M = \frac{21{,}450\ LB/FT}{5 \times 10^6\ LB/FT^2} \cdot B_c = 0.004\ B_c$

d_c = (% DEFLECTION)(B_c)

% DEFLECTION = MAXIMUM RECOMMENDED PIPE DEFLEXATION BY MANUFACTURE @ 3% FOR 18" HDPE PIPE.

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA-SSR PIPE Page: 8/18
Prepared by: QBKinsel Date: 4/30/95
Checked by: N.P Date: 5-8-95

4-d r_{sd} – DETERMINATION Continued

Equation #8 Again: $r_{sd} = \frac{(S_m + S_g) - (S_f + d_c)}{S_m}$

where: $S_m = 0.0125\ B_c$

$S_g = 0$

$S_f = 0$

$d_c = 0.03\ B_c$

$r_{sd} = \frac{(0.004\ B_c + 0) - (0 + 0.03\ B_c)}{0.004\ B_c}$

$r_{sd} = \frac{0.004\ B_c - 0.03\ B_c}{0.004\ B_c} = -6.5$

$r_{sd} = -6.5$

4-e DETERMINE RATIO of C_c FOR $K_{o\mu} = 0.09$
NOTE: $K_{o\mu}$ WAS DETERMINED IN #4-c

TABLE 1 Values* of C_c in Terms of H/B_c

Incomplete Projection Condition $K\mu = 0.19$		Incomplete Ditch Condition $K\mu = 0.13$	
$r_{sd}p$	Equation	$r_{sd}p$	Equation
+0.1	$C_c = 1.23H/B_c - 0.02$	−0.1	$C_c = 0.82H/B_c + 0.05$
+0.3	$C_c = 1.39H/B_c - 0.05$	−0.3	$C_c = 0.69H/B_c + 0.11$
+0.5	$C_c = 1.50H/B_c - 0.07$	−0.5	$C_c = 0.61H/B_c + 0.20$
+0.7	$C_c = 1.59H/B_c - 0.09$	−0.7	$C_c = 0.55H/B_c + 0.25$
+1.0	$C_c = 1.69H/B_c - 0.12$	−1.0	$C_c = 0.47H/B_c + 0.40$
+2.0	$C_c = 1.93H/B_c - 0.17$		

FROM FIGURE #2 THE INCOMPLETE DITCH CONDITION CURVES ARE BOUNDED BY THE COMPLETE DITCH CONDITION CURVE.

Foth & Van Dyke

Client: CMC- Scope I.D.: 93C049
Project: TMA-SSR PIPE Page: 9/18
Prepared by: JB Kirsch Date: 4/30/95
Checked by: N.P Date: 5-8-95

THE COMPLETE DITCH CONDITION C_c VALUE IS DEFINED BY $C_c = \frac{e^{\pm K_u (H/B_c)} - 1}{\pm 2K_u}$

EQUATION #11

THE RATIO OF $K_o = 0.09$, $K_u = 0.13$

C_c 1.44.

THUS. C_c (1.44) WILL BE APPLIED FOR $K_u = 0.09$

TABLE #2

f_{sdp}, $C_c^{K_u/K_a}$ FOR $K_u = 0.09$,

$H/B_c = 110\,FT/1.5\,FT$

f_{sdp}	C_c (1)	$C_c^{K_u/K_a}$ (2)	MODIFIED C_c VALUES
-0.1	$C_c = 0.82^{H/B_c} + 0.05$	60.18 (1.44)	86.7
-0.3	$C_c = 0.69^{H/B_c} + 0.11$	50.71 (1.44)	73.0
-0.5	$C_c = 0.61^{H/B_c} + 0.20$	44.9 (1.44)	64.7
-0.7	$C_c = 0.55^{H/B_c} + 0.25$	40.58 (1.44)	58.4
-1.0	$C_c = 0.47^{H/B_c} + 0.40$	34.9 (1.44)	50.20

(1) C_c EQUATIONS ARE FROM TABLE #1

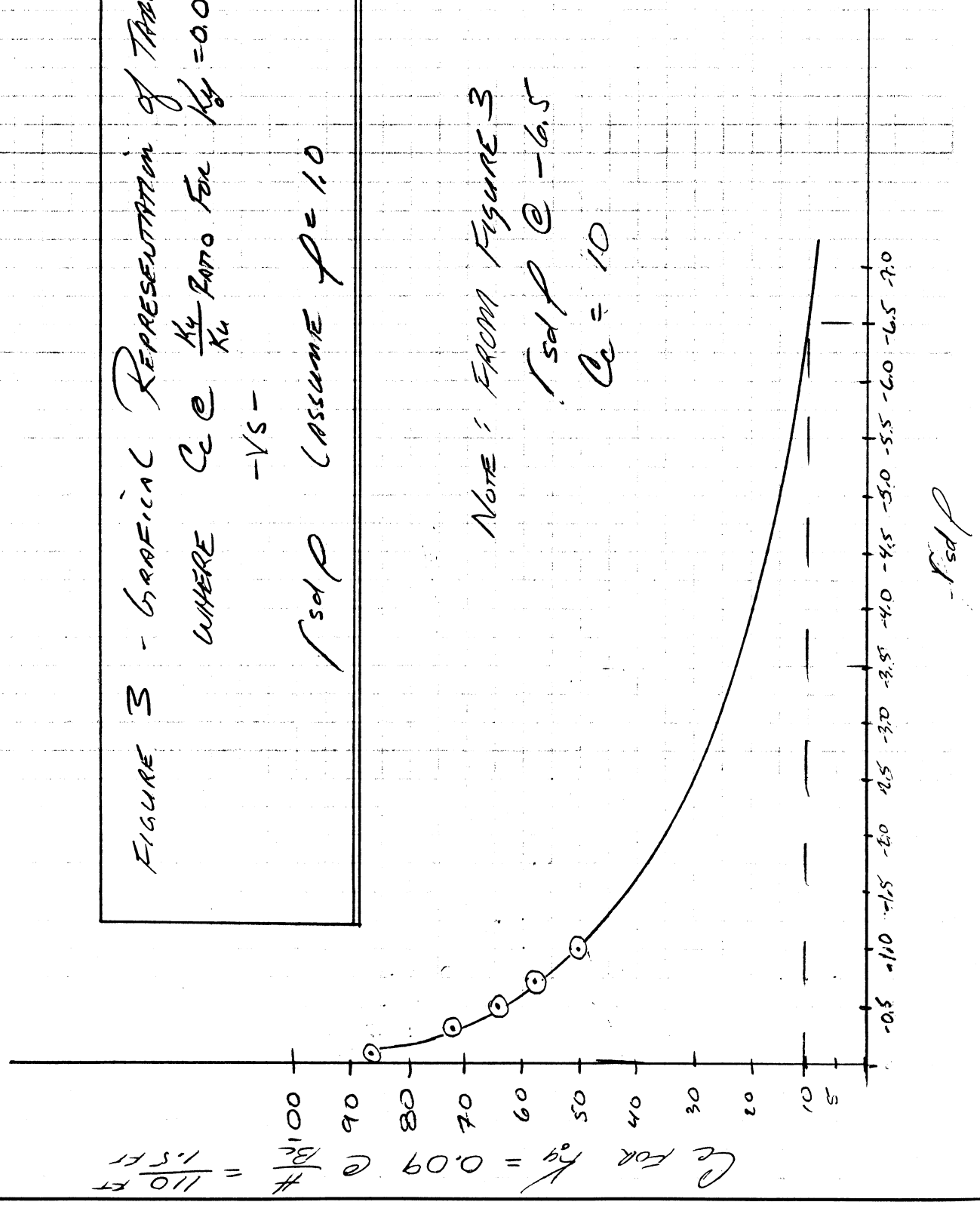
(2) C_c VALUES OF $K_u = 0.13$ EQUATIONS ARE MULTIPLIED BY $\frac{K_u}{K_o}$ RATIO WHICH IS $\frac{0.13}{0.09} = 1.44$

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA-SSR PIPE Page: 10/18
Prepared by: JBKirsch Date: 4/30/95
Checked by: N. P Date: 5-8-95

FIGURE 3 - GRAPHICAL REPRESENTATION OF TABLE #2
WHERE C_e @ $\frac{K_y}{K_u}$ RATIO FOR K_y = 0.09
-VS-
$I_{sd}P$ (ASSUME P_e = 1.0

NOTE: FROM FIGURE 3
$I_{sd}P$ @ -6.5
C_e = 1.0



C_e FOR $\frac{K_y}{K_u}$ = 0.09 @ $\frac{\#}{\text{SEC}}$ = $\frac{110 \text{ FT}}{1.5 \text{ FT}}$

Foth & Van Dyke

Client: CMC Scope I.D.: 930049
Project: TMA - SSR PIPE Page: 11/18
Prepared by: JB KIRSCH Date: 4/30/95
Checked by: N.P Date: 5-8-95

4-5 FROM #4

EQUATION #5 (AGAIN) $W_c = C_c W B_c^2$

$C_c = .10$ FROM FIGURE 3

$W = 130$ LB/CF

$B_c = 18"/12in = 1.5$ FT

$W_c = (.10)(130)(1.5)^2 =$

$W_c = 2925$ LB/LINEAL FT

$W_c = 2925 \frac{LB}{LIN FT} \cdot \frac{1 FT}{12in} = 244$ LB/IN LINEAL ✓

(5) Modulus of Passive Soil Resistance, E'

E' = 1,000 psi FOR DUMPED CRUSHED ROCK

SEE TABLE #3.

Foth & Van Dyke	Client: CMC Scope I.D.: 93C049 Project: TMA - SSR Page: 12/18 Prepared by: CB Kirsch Date: 4/30/95 Checked by: N.P Date: 5-8-95

TABLE #2

AVERAGE VALUES OF MODULUS OF SOIL REACTION, E'
(For Initial Flexible Pipe Deflection)

Soil type-pipe bedding material (Unified Classification System[a]) (1)	E' for Degree of Compaction of Bedding, in pounds per square inch			
	Dumped (2)	Slight, <85% Proctor, <40% relative density (3)	Moderate, 85%-95% Proctor, 40%-70% relative density (4)	High, >95% Proctor, >70% relative density (5)
Fine-grained Soils (LL > 50)[b] Soils with medium to high plasticity CH, MH, CH-MH	No data available; consult a competent soils engineer; Otherwise use E' = 0			
Fine-grained Soils (LL < 50) Soils with medium to no plasticity, CL, ML, ML-CL, with less than 25% coarse-grained particles	50	200	400	1,000
Fine-grained Soils (LL < 50) Soils with medium to no plasticity, CL, ML, ML-CL, with more than 25% coarse-grained particles Coarse-grained Soils with Fines GM, GC, SM, SC[c] contains more than 12% fines	100	400	1,000	2,000
Coarse-grained Soils with Little or no Fines GW, GP, SW, SP[c] contains less than 12% fines	E' FOR STONE BACKFILL			
→ Crushed Rock	200	1,000	2,000	3,000
	1,000	3,000	3,000	3,000
Accuracy in Terms of Percentage Deflection[d]	±2	±2	±1	±0.5

[a]ASTM Designation D 2487, USBR Designation E-3.
[b]LL = Liquid limit.
[c]Or any borderline soil beginning with one of these symbols (i.e. GM-GC, GC-SC).
[d]For ±1% accuracy and predicted deflection of 3%, actual deflection would be between 2% and 4%.

Note: Values applicable only for fills less than 50 ft (15 m). Table does not include any safety factor. For use in predicting initial deflections only, appropriate Deflection Lag Factor must be applied for long-term deflections. If bedding falls on the borderline between two compaction categories, select lower E' value or average the two values. Percentage Proctor based on laboratory maximum dry density from test standards using about 12,500 ft-lb/cu ft (598,000 J/m^3) (ASTM D 698, AASHTO T-99, USBR Designation E-11). 1 psi = 6.9 kPa.

SOURCE: "Soil Reaction for Buried Flexible Pipe" by Amster K. Howard, U.S. Bureau of Reclamation, Denver, Colorado. Reprinted with permission from American Society of Civil Engineers.

SOURCE: UNI-BELL PVC PIPE ASSOCIATION (1991)
HANDBOOK OF PVC PIPE: DESIGN AND CONSTRUCTION
DALLAS, TX p. 207

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: TMA SSR PIPE Page: 13/18
Prepared by: JB Kasel Date: 4/30/95
Checked by: N.P Date: 5-8-95

(6) SOLVE FOR DEFLECTION FROM EQUATION #1

EQUATION #1 AGAIN

$$\Delta_x = \frac{D_L \cdot K \cdot W_c}{\frac{2E}{3(SDR)^3} + 0.061E'}$$

WHERE $D_L = 1.5$

$K = 0.1$

$W_c = 244.$ lb/in. LINEAR

$E = 30{,}000$ psi

SDR = ASSUME 11.

$E' = 1{,}000$ psi

$$\Delta_{x\ @\ SDR\,11} = \frac{(1.5)(0.1)(244.\ lb/in)}{\frac{2(30{,}000\ psi)}{3(11)^3} + 0.061(1{,}000\ psi)}$$

$$\Delta_{x\ @\ SDR\,11} = \frac{36.6}{15.064\ 61} = 0.481\ inches = \Delta_{x\ @\ SDR\,11}$$

(7) EQUATION #13 : DETERMINE % DEFLECTION

$$\frac{\Delta x}{Bc} \times 100 = \frac{0.481\ in}{18\ in} \times 100 = 2.67\%$$ ✓

Foth & Van Dyke

Client: CMC Scope I.D.: 93049
Project: TMA - SSR - Pipe Page: 14/18
Prepared by: BKirsch Date: 4/30/95
Checked by: D.P Date: 5-8-95

Conclusion: FOR DESIGN CONSIDERATION OF 110 Ft OF COVER MATERIAL A 18" OD PE PIPE SDR 11 will DEFLECT 2.67%. This DETERMINES DEFLECTION UNDER WORST CASE DESIGN SENARIO IS BELOW MAXIMUM Allowable PIPE DEFLECTION OF 3% RECOMMENDED BY PE PIPE MANUFACTURES

REFERENCES

1) ~~Boodes~~
Spangler, M.G., and Handy, R.L., (198), Soil Engineering, Harper & Row, Publishers, N.Y., NY.

Foth & Van Dyke	Client: CMC	Scope I.D.: 93044
	Project: PIPE CRUSHING	Page: 15/18
	Prepared by: JBKinsel	Date: 4/28/95
	Checked by: N.P	Date: 5-8-95

ATTACHMENT #1 - Modulus OF ELASTICITY FOR STONE BACKFILL

TASK : DETERMINE THE MODULUS OF ELASTICITY (E_B) FOR THE STONE BACKFILL AROUND THE Collection PIPE.

Solution:

$$G_1 = \frac{3,230}{(1+e)} (2.97-e)^2 \bar{\sigma}_o^{1/2}$$

WHERE $E_B = G_1 2(1+\mu)$

Thus:

$$E_B = \frac{6,460 (2.97-e)^2 \bar{\sigma}_o^{1/2} (1+\mu)}{1+e}$$

WHERE: G_1 = SHEARE MODULUS
E_B = MODULUS of ELASTICity in KPa
e = Void Ratio = 0.65

ASSUME: No Compaction
γd_{stone} = 100 LB/CF
G_s stone = 2.65 (SPECIFIC GRAVITY)
μ = POISSON's Ratio

$$\frac{\mu}{1-\mu} = 0.5 \quad \therefore \mu = 0.33$$

$\bar{\sigma}_o$ = EFFECTIVE CONFINING STRESS IN KPa

ATTACHMENT #1 Continued:

DETERMINE $\bar{\sigma}_o$

$\bar{\sigma}_o = 1/3\,(\sigma_z + 2\sigma_h)$

WHERE: σ_z = VERTICAL STRESS AT WORST CASE

$\sigma_h = K_o \sigma_z$ = HORIZONTAL CONFINING STRESS

$K_o = 1 - \sin\phi = 0.5$ WHERE $\phi = 30°$

$\sigma_z = 14{,}300\ lb/ft^2 = 14.3\ kips/ft$

(SEE Equation # 3 OF CALCULATION)

$\sigma_z = 14.3\ kips/ft \cdot 47.880\ kN/m^2$ $\qquad N/m^2 = Pa$

$\sigma_z = 685\ KPa$

$\bar{\sigma}_o = 1/3\left[(685\ KPa) + (2)(0.5)(685\ KPa)\right]$

$\bar{\sigma}_o = 456.5\ KPa$ = EFFECTIVE CONFINING STRESS.

Foth & Van Dyke	Client: CMC	Scope I.D.: 92049
	Project: SSR-PAPE	Page: 17/18
	Prepared by: JBKissil	Date:
	Checked by: N.P	Date: 5-8-95

ATTACHMENT #1 (CONTINUED)

SOLVE FOR MODULUS OF ELASTICITY (E_b) OF BEDDING STONE

$$E_b = \frac{6{,}460(2.97-e)^2 \bar{\sigma}_o^{1/2}(1+\mu)}{1+e}$$

where: $\bar{\sigma}_o = 456.5$ KPa = EFFECTIVE CONFINING STRESS
$\mu = 0.33$ = POISSON'S RATIO
$e = 0.65$ = VOID RATIO

$$E_b = \frac{6{,}460(2.97-0.65)^2 \cdot (456.5\ KPa)^{1/2}(1+.33)}{1+0.65}$$

$E_b = 598{,}791$ KPa

$$E_b = (598\,791\ KPa)\left(\frac{1\ kip}{47.880\ KPa}\right)\left(\frac{1000\ lbs/ft^2}{1\ kip}\right)$$

$E_b = 12{,}506{,}000$ lbs/ft²

FOR CONSERVATIVE VALUE USE

$E_b = 5 \times 10^6$ PSF

SOURCE: BOWLES, J.E. (1982), FOUNDATION ANALYSIS AND DESIGN, McGRAW HILL, INT, NY, NY.

ATTACHMENT #2

Fig. 2

PIPE INFORMATION: MINIMUM WALL, in.
WEIGHT PER FOOT, #
NOMINAL I.D., in.

O.D., NOMINAL	in. ACTUAL	SDR 7	7.3	9	9.3	11	13.5	15.5	17	21	26	32.5
		0.188	0.180	0.146	0.141	0.120						
1	1.315	0.28	0.28	0.23	0.22	0.19						
		0.924	0.940	1.011	1.021	1.066						
		0.278	0.267	0.216	0.209	0.177	0.144	0.126				
1-1/4	1.660	0.42	0.41	0.37	0.36	0.31	0.25	0.22				
		1.167	1.187	1.276	1.289	1.346	1.404	1.434				
		0.271	0.260	0.211	0.204	0.173	0.141	0.123				
1-1/2	1.900	0.59	0.57	0.48	0.47	0.40	0.33	0.29				
		1.335	1.359	1.461	1.475	1.541	1.607	1.645				
		0.348	0.329	0.265	0.255	0.210	0.170	0.148	0.130			
2	2.375	0.93	0.90	0.73	0.70	0.58	0.47	0.41	0.36			
		1.669	1.698	1.826	1.844	1.926	2.009	2.056	2.084			
		0.500	0.479	0.389	0.376	0.318	0.259	0.226	0.206	0.167	0.135	
3	3.500	2.02	1.95	1.63	1.58	1.36	1.13	0.99	0.91	0.75	0.61	
		2.460	2.503	2.691	2.717	2.838	2.961	3.030	3.072	3.153	3.220	
		0.643	0.616	0.500	0.484	0.409	0.333	0.296	0.265	0.214	0.173	0.138
4	4.500	3.34	3.22	2.69	2.61	2.25	1.87	1.64	1.51	1.24	1.01	0.81
		3.163	3.218	3.460	3.494	3.649	3.807	3.896	3.949	4.054	4.140	4.212
		0.795	0.762	0.618	0.598	0.506	0.412	0.359	0.327	0.265	0.214	0.171
5	5.563	5.10	4.92	4.11	4.00	3.44	2.86	2.51	2.31	1.89	1.54	1.24
		3.910	3.978	4.277	4.319	4.511	4.706	4.816	4.882	5.012	5.118	5.207
		0.946	0.908	0.736	0.712	0.602	0.491	0.427	0.390	0.315	0.255	0.204
6	6.625	7.23	6.98	5.83	5.67	4.88	4.05	3.56	3.27	2.68	2.18	1.76
		4.656	4.737	5.094	5.143	5.372	5.604	5.736	5.814	5.969	6.095	6.201
		1.232	1.182	0.958	0.927	0.784	0.639	0.556	0.507	0.411	0.332	0.265
8	8.625	12.26	11.83	9.89	9.61	8.27	6.87	6.04	5.54	4.54	3.70	2.99
		6.062	6.167	6.632	6.696	6.994	7.296	7.468	7.570	7.771	7.935	8.073
		1.536	1.475	1.194	1.156	0.977	0.796	0.694	0.632	0.512	0.413	0.331
10	10.750	19.04	18.38	15.36	14.92	12.85	10.67	9.39	8.61	7.05	5.75	4.64
		7.556	7.687	8.266	8.346	8.717	9.044	9.307	9.435	9.685	9.890	10.062
		1.821	1.747	1.417	1.371	1.159	0.944	0.823	0.750	0.607	0.490	0.392
12	12.750	26.79	25.86	21.60	20.99	18.08	15.00	13.20	12.11	9.92	8.09	6.52
		8.961	9.117	9.803	9.898	10.339	10.786	11.039	11.190	11.487	11.730	11.934
		2.000	1.918	1.556	1.505	1.273	1.037	0.903	0.824	0.667	0.538	0.431
14	14.000	32.29	31.18	26.05	25.31	21.80	18.09	15.92	14.60	11.96	9.75	7.87
		9.840	10.011	10.764	10.869	11.353	11.843	12.121	12.287	12.613	12.880	13.104
		2.286	2.192	1.778	1.720	1.455	1.185	1.032	0.941	0.762	0.615	0.492
16	16.000	42.18	40.72	34.02	33.06	28.47	23.63	20.79	19.07	15.62	12.74	10.27
		11.246	11.441	12.302	12.422	12.975	13.535	13.853	14.042	14.415	14.720	14.976
		2.571	2.466	2.000	1.935	1.636	1.333	1.161	1.059	0.857	0.692	0.554
18	18.000	53.39	51.54	43.06	41.84	36.03	29.90	26.31	24.14	19.77	16.12	13.00
		12.651	12.871	13.840	13.974	14.596	15.227	15.585	15.798	16.217	16.560	16.848
			2.740	2.222	2.151	1.818	1.481	1.290	1.176	0.952	0.769	0.615
20	20.000		63.63	53.16	51.65	44.48	36.92	32.49	29.80	24.41	19.91	16.05
			14.301	15.378	15.527	16.218	16.919	17.316	17.553	18.019	18.400	18.720
				2.444	2.366	2.000	1.630	1.416	1.294	1.048	0.846	0.677

9/18

Appendix J

Leachate Volume Calculations for Post-Closure Conditions

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: Leachate Volumes Page: 1 of 2
Prepared by: N X P Date: 5-26-95
Checked by: REM Date: 5/26/95

Purpose: Estimate Post-Closure Leachate Volumes at the TMA site

Scope: Using i) output from HELP model runs for the case of a closed cell ii) proportional areas of sideslopes and base and iii) the schedule of closure for each cell, calculate the leachate generation rate post closure

Method of Calculation:

Attachment A shows leachate collected in each TMA cell after closure. The lateral drainage from the drainage layer above the baseliner through the years are listed beginning year 9 for TMA 1 (Page AA 1)

This is based on the assumption that TMA 1 is closed at the end of year 8.

Similarly the same data are provided for TMA-2, 3 and 4 starting at years 20, 25 and 32 respectively, based on assumed closure times. (Pages AA 4, AA 7 and AA 10)

Attachment A also shows the sideslope areas and base areas for TMAs 1, 2, 3 and 4 (see page A.A 1)

These areas have been used to multiply the leachate collection rate (as given by HELP model) to obtain quantities and are given in Gallons per day in Attachment A

Attachment B provides the summation of leachate quantities starting from year 1 (i.e start up of TMA 1)

Post closure leachate quantities are the values starting from year 32. (Please note that these computations are based on some assumptions regarding schedule of closure and slight variations in this schedule will not impact leachate quantities)

From Attachment B, it may be noted that the maximum leachate collected post closure is 32929 gal/day or 22.9 gpm.

For Design purposes use 25 gpm.

Foth & Van Dyke

Client: CMC Scope I.D.: 93 C049
Project: Leachate Volumes Page: AA. 0
Prepared by: MDF Date: 4/7/95
Checked by: J.B.K Date: 5/25/95

ATTACHMENT A

LEACHATE COLLECTED IN INDIVIDUAL TMAs (1 thru 4)

MDF 5/8/95 60 mil HDPE liner and cover
leakage collected

Leachate From closed TMA-1

A.A 1

SURFACE AREA (PLAN VIEW) FT^2

	SS	BASE
TMA 1	1229380	1100380
TMA 2	1863670	1214970
TMA 3	324770	671090
TMA 4	1425960	724510

SS = SIDESLOPE AT 33.33% SLOPE
BASE = BASE AT 2% SLOPE

TMA 1

	leachate collected from layer12		leachate collected from layer 17	
	ss		base	
YEAR	in/year	gall/DAY	in/yr	gall/DAY
1	0.00	0.00	0.00	0.00
2	0.00	0.00	0.00	0.00
3	0.00	0.00	0.00	0.00
4	0.00	0.00	0.00	0.00
5	0.00	0.00	0.00	0.00
6	0.00	0.00	0.00	0.00
7	0.00	0.00	0.00	0.00
8	0.00	0.00	0.00	0.00
9	4.2651	8955	10.5703	19864
10	3.7063	7781	12.4146	23329
11	3.1115	6533	12.4146	23329
12	2.7014	5672	9.1470	17189
13	4.4630	9370	2.7483	5165
14	2.6732	5612	2.3434	4404
15	0.7623	1600	2.0960	3939
16	0.6655	1397	1.9077	3585
17	0.6224	1307	1.7548	3298
18	0.5850	1228	1.6291	3061
19	0.5525	1160	1.5192	2855
20	0.5285	1110	1.4265	2681
21	0.5030	1056	1.3381	2515
22	0.4801	1008	1.2625	2372
23	0.4583	962	1.1942	2244
24	0.4386	921	1.1366	2136
25	0.4176	877	1.0677	2006
26	0.3986	837	1.0108	1899
27	0.3806	799	0.9655	1814
28	0.3644	765	0.9265	1741
29	0.3469	728	0.8857	1664
30	0.3313	696	0.8501	1597
31	0.3163	664	0.8180	1537
32	0.3028	636	0.7905	1485

33	0.2884	605	0.7631	1434
34	0.2754	578	0.7397	1390
35	0.2630	552	0.7175	1348
36	0.2518	529	0.6984	1312
37	0.2399	504	0.6770	1272
38	0.2291	481	0.4570	859
39	0.1382	290	0.0063	12
40	0.1319	277	0.0000	0
41	0.1260	265	0.0000	0
42	0.1207	253	0.0000	0
43	0.1150	241	0.0000	0
44	0.1099	231	0.0000	0
45	0.1050	220	0.0000	0
46	0.1006	211	0.0000	0
47	0.0959	201	0.0000	0
48	0.0920	193	0.0000	0
49	0.0884	186	0.0000	0
50	0.0851	179	0.0000	0
51	0.0815	171	0.0000	0
52	0.0783	164	0.0000	0
53	0.0752	158	0.0000	0
54	0.0724	152	0.0000	0
55	0.0693	145	0.0000	0
56	0.0666	140	0.0000	0
57	0.0640	134	0.0000	0
58	0.0616	129	0.0000	0
59	0.0590	124	0.0000	0
60	0.0567	119	0.0000	0
61	0.0544	114	0.0000	0
62	0.0524	110	0.0000	0
63	0.0502	105	0.0000	0
64	0.0482	101	0.0003	1
65	0.0463	97	0.0003	1
66	0.0446	94	0.0005	1
67	0.0428	90	0.0003	1
68	0.0411	86	0.0004	1
69	0.0394	83	0.0003	1
70	0.0380	80	0.0003	1
71	0.0364	76	0.0002	0
72	0.0350	73	0.0004	1
73	0.0336	71	0.0006	1
74	0.0323	68	0.0002	0
75	0.0310	65	0.0003	1
76	0.0298	63	0.0008	2
77	0.0286	60	0.0004	1
78	0.0275	58	0.0002	0
79	0.0264	55	0.0004	1

80	0.0254	53	0.0003	1
81	0.0244	51	0.0004	1
82	0.0235	49	0.0001	0
83	0.0225	47	0.0001	0
84	0.0216	45	0.0001	0
85	0.0207	43	0.0005	1
86	0.0200	42	0.0002	0
87	0.0191	40	0.0002	0
88	0.0184	39	0.0001	0
89	0.0177	37	0.0004	1
90	0.0170	36	0.0005	1
91	0.0163	34	0.0009	2
92	0.0157	33	0.0003	1
93	0.0151	32	0.0006	1
94	0.0145	30	0.0003	1
95	0.0139	29	0.0007	1
96	0.0134	28	0.0001	0
97	0.0128	27	0.0003	1
98	0.0124	26	0.0001	0

Leachate From Closed TMA-2

AA 4

TMA 2

YEAR	leachate collected from layer12 ss in/year	gal/DAY	leachate collected from layer 17 base in/yr	gall/DAY
1	0.00	0.00	0.00	0.00
2	0.00	0.00	0.00	0.00
3	0.00	0.00	0.00	0.00
4	0.00	0.00	0.00	0.00
5	0.00	0.00	0.00	0.00
6	0.00	0.00	0.00	0.00
7	0.00	0.00	0.00	0.00
8	0.00	0.00	0.00	0.00
9	0.00	0.00	0.00	0.00
10	0.00	0.00	0.00	0.00
11	0.00	0.00	0.00	0.00
12	0.00	0.00	0.00	0.00
13	0.00	0.00	0.00	0.00
14	0.00	0.00	0.00	0.00
15	0.00	0.00	0.00	0.00
16	0.00	0.00	0.00	0.00
17	0.00	0.00	0.00	0.00
18	0.00	0.00	0.00	0.00
19	0.00	0.00	0.00	0.00
20	4.2651	13575	10.5703	21932
21	3.7063	11796	12.4146	25759
22	3.1115	9903	12.4146	25759
23	2.7014	8598	9.1470	18979
24	4.4630	14204	2.7483	5702
25	2.6732	8508	2.3434	4862
26	0.7623	2426	2.0960	4349
27	0.6655	2118	1.9077	3958
28	0.6224	1981	1.7548	3641
29	0.5850	1862	1.6291	3380
30	0.5525	1758	1.5192	3152
31	0.5285	1682	1.4265	2960
32	0.5030	1601	1.3381	2776
33	0.4801	1528	1.2625	2620
34	0.4583	1459	1.1942	2478
35	0.4386	1396	1.1366	2358
36	0.4176	1329	1.0677	2215
37	0.3986	1269	1.0108	2097
38	0.3806	1211	0.9655	2003
39	0.3644	1160	0.9265	1922
40	0.3469	1104	0.8857	1838
41	0.3313	1054	0.8501	1764
42	0.3163	1007	0.8180	1697

43	0.3028	964	0.7905	1640
44	0.2884	918	0.7631	1583
45	0.2754	877	0.7397	1535
46	0.2630	837	0.7175	1489
47	0.2518	801	0.6984	1449
48	0.2399	764	0.6770	1405
49	0.2291	729	0.4570	948
50	0.1382	440	0.0063	13
51	0.1319	420	0.0000	0
52	0.1260	401	0.0000	0
53	0.1207	384	0.0000	0
54	0.1150	366	0.0000	0
55	0.1099	350	0.0000	0
56	0.1050	334	0.0000	0
57	0.1006	320	0.0000	0
58	0.0959	305	0.0000	0
59	0.0920	293	0.0000	0
60	0.0884	281	0.0000	0
61	0.0851	271	0.0000	0
62	0.0815	259	0.0000	0
63	0.0783	249	0.0000	0
64	0.0752	239	0.0000	0
65	0.0724	230	0.0000	0
66	0.0693	221	0.0000	0
67	0.0666	212	0.0000	0
68	0.0640	204	0.0000	0
69	0.0616	196	0.0000	0
70	0.0590	188	0.0000	0
71	0.0567	180	0.0000	0
72	0.0544	173	0.0000	0
73	0.0524	167	0.0000	0
74	0.0502	160	0.0000	0
75	0.0482	153	0.0003	1
76	0.0463	147	0.0003	1
77	0.0446	142	0.0005	1
78	0.0428	136	0.0003	1
79	0.0411	131	0.0004	1
80	0.0394	125	0.0003	1
81	0.0380	121	0.0003	1
82	0.0364	116	0.0002	0
83	0.0350	111	0.0004	1
84	0.0336	107	0.0006	1
85	0.0323	103	0.0002	0
86	0.0310	99	0.0003	1
87	0.0298	95	0.0008	2
88	0.0286	91	0.0004	1
89	0.0275	88	0.0002	0

90	0.0264	84	0.0004	1
91	0.0254	81	0.0003	1
92	0.0244	78	0.0004	1
93	0.0235	75	0.0001	0
94	0.0225	72	0.0001	0
95	0.0216	69	0.0001	0
96	0.0207	66	0.0005	1
97	0.0200	64	0.0002	0
98	0.0191	61	0.0002	0
99	0.0184	59	0.0001	0
100	0.0177	56	0.0004	1
101	0.0170	54	0.0005	1
102	0.0163	52	0.0009	2
103	0.0157	50	0.0003	1
104	0.0151	48	0.0006	1
105	0.0145	46	0.0003	1
106	0.0139	44	0.0007	1
107	0.0134	43	0.0001	0
108	0.0128	41	0.0003	1
109	0.0124	39	0.0001	0

Leachate From Closed TMA-3

AA 7

TMA 3

YEAR	leachate collected from layer12 ss in/year	gall/DAY	leachate collected from layer 17 base in/yr	gall/DAY
1	0.00	0.00	0.00	0.00
2	0.00	0.00	0.00	0.00
3	0.00	0.00	0.00	0.00
4	0.00	0.00	0.00	0.00
5	0.00	0.00	0.00	0.00
6	0.00	0.00	0.00	0.00
7	0.00	0.00	0.00	0.00
8	0.00	0.00	0.00	0.00
9	0.00	0.00	0.00	0.00
10	0.00	0.00	0.00	0.00
11	0.00	0.00	0.00	0.00
12	0.00	0.00	0.00	0.00
13	0.00	0.00	0.00	0.00
14	0.00	0.00	0.00	0.00
15	0.00	0.00	0.00	0.00
16	0.00	0.00	0.00	0.00
17	0.00	0.00	0.00	0.00
18	0.00	0.00	0.00	0.00
19	0.00	0.00	0.00	0.00
20	0.00	0.00	0.00	0.00
21	0.00	0.00	0.00	0.00
22	0.00	0.00	0.00	0.00
23	0.00	0.00	0.00	0.00
24	0.00	0.00	0.00	0.00
25	4.2651	2366	10.5703	12114
26	3.7063	2056	12.4146	14228
27	3.1115	1726	12.4146	14228
28	2.7014	1498	9.1470	10483
29	4.4630	2475	2.7483	3150
30	2.6732	1483	2.3434	2686
31	0.7623	423	2.0960	2402
32	0.6655	369	1.9077	2186
33	0.6224	345	1.7548	2011
34	0.5850	324	1.6291	1867
35	0.5525	306	1.5192	1741
36	0.5285	293	1.4265	1635
37	0.5030	279	1.3381	1534
38	0.4801	266	1.2625	1447
39	0.4583	254	1.1942	1369
40	0.4386	243	1.1366	1303
41	0.4176	232	1.0677	1224
42	0.3986	221	1.0108	1158

43	0.3806	211	0.9655	1107
44	0.3644	202	0.9265	1062
45	0.3469	192	0.8857	1015
46	0.3313	184	0.8501	974
47	0.3163	175	0.8180	937
48	0.3028	168	0.7905	906
49	0.2884	160	0.7631	875
50	0.2754	153	0.7397	848
51	0.2630	146	0.7175	822
52	0.2518	140	0.6984	800
53	0.2399	133	0.6770	776
54	0.2291	127	0.4570	524
55	0.1382	77	0.0063	7
56	0.1319	73	0.0000	0
57	0.1260	70	0.0000	0
58	0.1207	67	0.0000	0
59	0.1150	64	0.0000	0
60	0.1099	61	0.0000	0
61	0.1050	58	0.0000	0
62	0.1006	56	0.0000	0
63	0.0959	53	0.0000	0
64	0.0920	51	0.0000	0
65	0.0884	49	0.0000	0
66	0.0851	47	0.0000	0
67	0.0815	45	0.0000	0
68	0.0783	43	0.0000	0
69	0.0752	42	0.0000	0
70	0.0724	40	0.0000	0
71	0.0693	38	0.0000	0
72	0.0666	37	0.0000	0
73	0.0640	35	0.0000	0
74	0.0616	34	0.0000	0
75	0.0590	33	0.0000	0
76	0.0567	31	0.0000	0
77	0.0544	30	0.0000	0
78	0.0524	29	0.0000	0
79	0.0502	28	0.0000	0
80	0.0482	27	0.0003	0
81	0.0463	26	0.0003	0
82	0.0446	25	0.0005	1
83	0.0428	24	0.0003	0
84	0.0411	23	0.0004	0
85	0.0394	22	0.0003	0
86	0.0380	21	0.0003	0
87	0.0364	20	0.0002	0
88	0.0350	19	0.0004	0
89	0.0336	19	0.0006	1

90	0.0323	18	0.0002	0
91	0.0310	17	0.0003	0
92	0.0298	17	0.0008	1
93	0.0286	16	0.0004	0
94	0.0275	15	0.0002	0
95	0.0264	15	0.0004	0
96	0.0254	14	0.0003	0
97	0.0244	14	0.0004	0
98	0.0235	13	0.0001	0
99	0.0225	12	0.0001	0
100	0.0216	12	0.0001	0
101	0.0207	11	0.0005	1
102	0.0200	11	0.0002	0
103	0.0191	11	0.0002	0
104	0.0184	10	0.0001	0
105	0.0177	10	0.0004	0
106	0.0170	9	0.0005	1
107	0.0163	9	0.0009	1
108	0.0157	9	0.0003	0
109	0.0151	8	0.0006	1
110	0.0145	8	0.0003	0
111	0.0139	8	0.0007	1
112	0.0134	7	0.0001	0
113	0.0128	7	0.0003	0
114	0.0124	7	0.0001	0

Leachate From Closed TMA-4

AA10

TMA 4

YEAR	leachate collected from layer12 ss in/year	gall/DAY	leachate collected from layer 17 base in/yr	gall/DAY
1	0.00	0.00	0.00	0.00
2	0.00	0.00	0.00	0.00
3	0.00	0.00	0.00	0.00
4	0.00	0.00	0.00	0.00
5	0.00	0.00	0.00	0.00
6	0.00	0.00	0.00	0.00
7	0.00	0.00	0.00	0.00
8	0.00	0.00	0.00	0.00
9	0.00	0.00	0.00	0.00
10	0.00	0.00	0.00	0.00
11	0.00	0.00	0.00	0.00
12	0.00	0.00	0.00	0.00
13	0.00	0.00	0.00	0.00
14	0.00	0.00	0.00	0.00
15	0.00	0.00	0.00	0.00
16	0.00	0.00	0.00	0.00
17	0.00	0.00	0.00	0.00
18	0.00	0.00	0.00	0.00
19	0.00	0.00	0.00	0.00
20	0.00	0.00	0.00	0.00
21	0.00	0.00	0.00	0.00
22	0.00	0.00	0.00	0.00
23	0.00	0.00	0.00	0.00
24	0.00	0.00	0.00	0.00
25	0.00	0.00	0.00	0.00
26	0.00	0.00	0.00	0.00
27	0.00	0.00	0.00	0.00
28	0.00	0.00	0.00	0.00
29	0.00	0.00	0.00	0.00
30	0.00	0.00	0.00	0.00
31	0.00	0.00	0.00	0.00
32	4.2651	10386	10.5703	13079
33	3.7063	9026	12.4146	15360
34	3.1115	7577	12.4146	15360
35	2.7014	6578	9.1470	11318
36	4.4630	10868	2.7483	3400
37	2.6732	6510	2.3434	2899
38	0.7623	1856	2.0960	2593
39	0.6655	1621	1.9077	2360
40	0.6224	1516	1.7548	2171
41	0.5850	1425	1.6291	2016
42	0.5525	1345	1.5192	1880

43	0.5285	1287	1.4265	1765
44	0.5030	1225	1.3381	1656
45	0.4801	1169	1.2625	1562
46	0.4583	1116	1.1942	1478
47	0.4386	1068	1.1366	1406
48	0.4176	1017	1.0677	1321
49	0.3986	971	1.0108	1251
50	0.3806	927	0.9655	1195
51	0.3644	887	0.9265	1146
52	0.3469	845	0.8857	1096
53	0.3313	807	0.8501	1052
54	0.3163	770	0.8180	1012
55	0.3028	737	0.7905	978
56	0.2884	702	0.7631	944
57	0.2754	671	0.7397	915
58	0.2630	640	0.7175	888
59	0.2518	613	0.6984	864
60	0.2399	584	0.6770	838
61	0.2291	558	0.4570	565
62	0.1382	337	0.0063	8
63	0.1319	321	0.0000	0
64	0.1260	307	0.0000	0
65	0.1207	294	0.0000	0
66	0.1150	280	0.0000	0
67	0.1099	268	0.0000	0
68	0.1050	256	0.0000	0
69	0.1006	245	0.0000	0
70	0.0959	234	0.0000	0
71	0.0920	224	0.0000	0
72	0.0884	215	0.0000	0
73	0.0851	207	0.0000	0
74	0.0815	198	0.0000	0
75	0.0783	191	0.0000	0
76	0.0752	183	0.0000	0
77	0.0724	176	0.0000	0
78	0.0693	169	0.0000	0
79	0.0666	162	0.0000	0
80	0.0640	156	0.0000	0
81	0.0616	150	0.0000	0
82	0.0590	144	0.0000	0
83	0.0567	138	0.0000	0
84	0.0544	132	0.0000	0
85	0.0524	128	0.0000	0
86	0.0502	122	0.0000	0
87	0.0482	117	0.0003	0
88	0.0463	113	0.0003	0
89	0.0446	109	0.0005	1

90	0.0428	104	0.0003	0
91	0.0411	100	0.0004	0
92	0.0394	96	0.0003	0
93	0.0380	93	0.0003	0
94	0.0364	89	0.0002	0
95	0.0350	85	0.0004	0
96	0.0336	82	0.0006	1
97	0.0323	79	0.0002	0
98	0.0310	75	0.0003	0
99	0.0298	73	0.0008	1
100	0.0286	70	0.0004	0
101	0.0275	67	0.0002	0
102	0.0264	64	0.0004	0
103	0.0254	62	0.0003	0
104	0.0244	59	0.0004	0
105	0.0235	57	0.0001	0
106	0.0225	55	0.0001	0
107	0.0216	53	0.0001	0
108	0.0207	50	0.0005	1
109	0.0200	49	0.0002	0
110	0.0191	47	0.0002	0
111	0.0184	45	0.0001	0
112	0.0177	43	0.0004	0
113	0.0170	41	0.0005	1
114	0.0163	40	0.0009	1
115	0.0157	38	0.0003	0
116	0.0151	37	0.0006	1
117	0.0145	35	0.0003	0
118	0.0139	34	0.0007	1
119	0.0134	33	0.0001	0
120	0.0128	31	0.0003	0
121	0.0124	30	0.0001	0

ATTACHMENT B

CUMULATIVE LEACHATE FROM ALL TMAs

A.B.1

LEACHATE FROM CLOSED TMA 1-4

60 mil CAP
60 mil BASE

TMA 1 -4

YEAR	ss gall/DAY	base gall/DAY	combine gall/DAY
1	0.00	0.00	1.00
2	0.00	0.00	1.00
3	0.00	0.00	1.00
4	0.00	0.00	1.00
5	0.00	0.00	1.00
6	0.00	0.00	1.00
7	0.00	0.00	1.00
8	0.00	0.00	1.00
9	8955	19864	28818
10	7781	23329	31111
11	6533	23329	29862
12	5672	17189	22860
13	9370	5165	14535
14	5612	4404	10016
15	1600	3939	5539
16	1397	3585	4982
17	1307	3298	4604
18	1228	3061	4290
19	1160	2855	4015
20	14684	24613	39297
21	12852	28273	41125
22	10911	28131	39042
23	9560	21223	30783
24	15125	7838	22964
25	11750	18983	30733
26	5319	20476	25795
27	4643	20001	24643
28	4244	15865	20109
29	5066	8194	13260
30	3937	7435	11372
31	2769	6899	9668
32	12992	19527	32519
33	11504	21425	32929
34	9938	21095	31034
35	8833	16765	25598
36	13019	8563	21582
37	8561	7803	16364
38	3815	6902	10717
39	3325	5663	8988
40	3140	5312	8451

41	2975	5003	7978
42	2827	4735	7562
43	2703	4512	7215
44	2576	4301	6876
45	2459	4112	6570
46	2348	3941	6289
47	2246	3793	6039
48	2142	3632	5773
49	2045	3073	5119
50	1698	2055	3754
51	1624	1969	3593
52	1550	1896	3446
53	1482	1828	3310
54	1415	1536	2951
55	1309	985	2295
56	1249	944	2194
57	1195	915	2110
58	1142	888	2030
59	1094	864	1958
60	1046	838	1883
61	1001	565	1567
62	762	8	770
63	729	0	729
64	698	1	699
65	671	1	671
66	641	1	642
67	615	1	615
68	589	1	590
69	565	1	566
70	541	1	542
71	519	0	520
72	499	1	500
73	480	1	481
74	460	0	461
75	442	1	443
76	424	2	427
77	408	2	410
78	392	1	393
79	376	2	378
80	361	2	363
81	348	2	350
82	334	1	335
83	320	1	322
84	308	2	309
85	296	2	297
86	284	1	285
87	273	3	275

A.B.3

88	262	2	264
89	252	2	254
90	242	2	244
91	232	3	235
92	223	3	226
93	215	2	217
94	206	1	207
95	198	2	200
96	190	2	192
97	183	2	184
98	175	1	176
99	144	1	145
100	138	1	139
101	133	2	134
102	127	3	130
103	122	1	124
104	118	2	120
105	113	1	114
106	108	2	111
107	104	1	106
108	100	2	101
109	97	1	98
110	55	1	55
111	53	1	53
112	51	1	51
113	48	1	49
114	47	1	48
115	38	0	39
116	37	1	38
117	35	0	36
118	34	1	35
119	33	0	33
120	31	0	32
121	30	0	30

Appendix K

Surface Water Management Design Calculations Including TR55 Hydrology and POND 2 Models

Prepared By: SRB
Checked By: MDL

CRANDON MINING COMPANY

TMA RUNOFF BASIN 8

QTR-55 HYDROLOGY
POND-2 HYDROLOGY

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:51:22
Watershed file: --> TMA1 .WSD
Hydrograph file: --> TMA1 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN B 3/95
100 YEAR STORM EVENT

>>>> Input Parameters Used to Compute Hydrograph <<<<

Subarea Description	AREA (acres)	CN	Tc (hrs)	* Tt (hrs)	Precip. (in)	Runoff (in)	Ia/p input/used	
1-1	28.80	60.0	0.50	0.20	5.00	1.30	.27	.30
1-2	15.14	60.0	0.50	0.30	5.00	1.30	.27	.30
2-1	17.86	60.0	0.50	0.30	5.00	1.30	.27	.30
1-5	6.15	60.0	0.50	0.00	5.00	1.30	.27	.30
6-1	0.18	60.0	0.10	0.30	5.00	1.30	.27	.30
6-2	0.46	60.0	0.10	0.20	5.00	1.30	.27	.30
5-1	1.97	60.0	0.30	0.20	5.00	1.30	.27	.30
5-2	2.58	60.0	0.30	0.20	5.00	1.30	.27	.30

* Travel time from subarea outfall to composite watershed outfall point.
Total area = 73.14 acres or 0.11428 sq.mi
Peak discharge = 52 cfs

WARNING: Drainage areas of two or more subareas differ by a factor of 5 or greater.

>>>> Computer Modifications of Input Parameters <<<<<

Subarea Description	Input Values Tc (hr)	Input Values * Tt (hr)	Rounded Values Tc (hr)	Rounded Values * Tt (hr)	Ia/p Interpolated (Yes/No)	Ia/p Messages
1-1	0.49	0.17	0.50	0.20	No	--
1-2	0.49	0.29	0.50	0.30	No	--
2-1	0.51	0.27	0.50	0.30	No	--
1-5	0.50	0.00	**	**	No	--
6-1	0.10	0.25	0.10	0.30	No	--
6-2	0.10	0.21	0.10	0.20	No	--
5-1	0.30	0.18	0.30	0.20	No	--
5-2	0.30	0.15	0.30	0.20	No	--

* Travel time from subarea outfall to composite watershed outfall point.
** Tc & Tt are available in the hydrograph tables.

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:51:22
Watershed file: --> TMA1 .WSD
Hydrograph file: --> TMA1 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN : 8 3/95
100 YEAR STORM EVENT

>>>> Summary of Subarea Times to Peak <<<<

Subarea	Peak Discharge at Composite Outfall (cfs)	Time to Peak at Composite Outfall (hrs)
1-1	23	12.7
1-2	11	12.8
2-1	13	12.8
1-5	5	12.4
6-1	0	0.0
6-2	1	12.3
5-1	2	12.4
5-2	2	12.3
Composite Watershed	52	12.7

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:51:22
Watershed file: --> TMA1 .WSD
Hydrograph file: --> TMA1 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 3/95
100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	11.0 hr	11.3 hr	11.6 hr	11.9 hr	12.0 hr	12.1 hr	12.2 hr	12.3 hr	12.4 hr
1-1	0	0	0	0	0	0	2	5	11
1-2	0	0	0	0	0	0	0	1	2
2-1	0	0	0	0	0	0	0	1	2
1-5	0	0	0	0	0	1	2	4	5
6-1	0	0	0	0	0	0	0	0	0
6-2	0	0	0	0	0	0	0	1	1
5-1	0	0	0	0	0	0	1	1	2
5-2	0	0	0	0	0	0	1	2	2
Total (cfs)	0	0	0	0	0	1	6	15	25

Subarea Description	12.5 hr	12.6 hr	12.7 hr	12.8 hr	13.0 hr	13.2 hr	13.4 hr	13.6 hr	13.8 hr
1-1	18	22	23	20	14	10	7	6	5
1-2	5	8	10	11	10	7	5	4	3
2-1	5	9	12	13	11	8	6	4	3
1-5	5	5	4	3	2	1	1	1	1
6-1	0	0	0	0	0	0	0	0	0
6-2	0	0	0	0	0	0	0	0	0
5-1	2	2	1	1	1	0	0	0	0
5-2	2	2	2	1	1	1	0	0	0
Total (cfs)	37	48	52	49	39	27	19	15	12

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:51:22
Watershed file: --> TMA1 .WSD
Hydrograph file: --> TMA1 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN B 3/95

100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	14.0 hr	14.3 hr	14.6 hr	15.0 hr	15.5 hr	16.0 hr	16.5 hr	17.0 hr	17.5 hr
1-1	4	4	3	3	3	2	2	2	2
1-2	2	2	2	2	1	1	1	1	1
2-1	3	2	2	2	2	1	1	1	1
1-5	1	1	1	1	1	0	0	0	0
6-1	0	0	0	0	0	0	0	0	0
6-2	0	0	0	0	0	0	0	0	0
5-1	0	0	0	0	0	0	0	0	0
5-2	0	0	0	0	0	0	0	0	0
Total (cfs)	10	9	8	8	7	4	4	4	4

Subarea Description	18.0 hr	19.0 hr	20.0 hr	22.0 hr	26.0 hr
1-1	2	2	1	1	0
1-2	1	1	1	1	0
2-1	1	1	1	1	0
1-5	0	0	0	0	0
6-1	0	0	0	0	0
6-2	0	0	0	0	0
5-1	0	0	0	0	0
5-2	0	0	0	0	0
Total (cfs)	4	4	3	3	0

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:51:22
Watershed file: --> TMA1 .WSD
Hydrograph file: --> TMA1 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 1 3/95
100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
11.0	0	14.8	8
11.1	0	14.9	8
11.2	0	15.0	8
11.3	0	15.1	8
11.4	0	15.2	8
11.5	0	15.3	7
11.6	0	15.4	7
11.7	0	15.5	7
11.8	0	15.6	6
11.9	0	15.7	6
12.0	0	15.8	5
12.1	1	15.9	5
12.2	6	16.0	4
12.3	15	16.1	4
12.4	25	16.2	4
12.5	37	16.3	4
12.6	48	16.4	4
12.7	52	16.5	4
12.8	49	16.6	4
12.9	44	16.7	4
13.0	39	16.8	4
13.1	33	16.9	4
13.2	27	17.0	4
13.3	23	17.1	4
13.4	19	17.2	4
13.5	17	17.3	4
13.6	15	17.4	4
13.7	14	17.5	4
13.8	12	17.6	4
13.9	11	17.7	4
14.0	10	17.8	4
14.1	10	17.9	4
14.2	9	18.0	4
14.3	9	18.1	4
14.4	9	18.2	4
14.5	8	18.3	4
14.6	8	18.4	4

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:51:22
Watershed file: --> TMA1 .WSD
Hydrograph file: --> TMA1 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN B 3/95
100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
18.6	4	22.4	3
18.7	4	22.5	3
18.8	4	22.6	3
18.9	4	22.7	2
19.0	4	22.8	2
19.1	4	22.9	2
19.2	4	23.0	2
19.3	4	23.1	2
19.4	4	23.2	2
19.5	4	23.3	2
19.6	3	23.4	2
19.7	3	23.5	2
19.8	3	23.6	2
19.9	3	23.7	2
20.0	3	23.8	2
20.1	3	23.9	2
20.2	3	24.0	2
20.3	3	24.1	1
20.4	3	24.2	1
20.5	3	24.3	1
20.6	3	24.4	1
20.7	3	24.5	1
20.8	3	24.6	1
20.9	3	24.7	1
21.0	3	24.8	1
21.1	3	24.9	1
21.2	3	25.0	1
21.3	3	25.1	1
21.4	3	25.2	1
21.5	3	25.3	1
21.6	3	25.4	0
21.7	3	25.5	0
21.8	3	25.6	0
21.9	3	25.7	0
22.0	3	25.8	0
22.1	3	25.9	0
22.2	3		
22.3	3		

```
**********************************************************************
*                                                                    *
* CRANDON MINING COMPANY          93C049.27      FOTH & VAN DYKE     *
*       TMA Runoff Basin 8                          3/95             *
*                                                                    *
*                                                                    *
*                                                                    *
**********************************************************************
```

Inflow Hydrograph: TMA1 .HYD
Rating Table file: TMA1 .PND

----INITIAL CONDITIONS----
Elevation = 1621.99 ft
Outflow = 0.00 cfs
Storage = 0.00 ac-ft

GIVEN POND DATA			INTERMEDIATE ROUTING COMPUTATIONS	
ELEVATION (ft)	OUTFLOW (cfs)	STORAGE (ac-ft)	2S/t (cfs)	2S/t + O (cfs)
1621.99	0.0	0.000	0.0	0.0
1622.49	0.0	0.006	1.4	1.4
1622.99	0.0	0.013	3.3	3.3
1623.49	0.0	0.024	5.8	5.8
1623.99	0.0	0.038	9.1	9.1
1624.49	0.0	0.055	13.2	13.2
1624.99	0.0	0.076	18.3	18.3
1625.49	0.0	0.424	102.7	102.7
1625.99	0.0	0.794	192.1	192.1
1626.49	0.0	1.181	285.8	285.8
1626.99	0.0	1.586	383.9	383.9
1627.49	8.9	2.010	486.4	495.3
1627.99	9.3	2.453	593.5	602.8
1628.49	9.6	2.914	705.2	714.8
1628.99	10.0	3.396	821.7	831.7
1629.49	10.4	3.897	943.1	953.5
1629.99	10.7	4.419	1069.4	1080.1
1630.49	11.0	4.962	1200.7	1211.7
1630.99	11.3	5.526	1337.2	1348.5
1631.49	23.0	6.111	1478.9	1501.9
1631.99	47.7	6.719	1626.0	1673.7

Time increment (t) = 0.100 hrs.

POND-2 Version: 5.16 S/N: 1295130172
EXECUTED: 04-24-1995 06:38:11

Pond File: TMA1 .PND
Inflow Hydrograph: TMA1 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
11.000	0.00	-----	0.0	0.0	0.00	1621.99
11.100	0.00	0.0	0.0	0.0	0.00	1621.99
11.200	0.00	0.0	0.0	0.0	0.00	1621.99
11.300	0.00	0.0	0.0	0.0	0.00	1621.99
11.400	0.00	0.0	0.0	0.0	0.00	1621.99
11.500	0.00	0.0	0.0	0.0	0.00	1621.99
11.600	0.00	0.0	0.0	0.0	0.00	1621.99
11.700	0.00	0.0	0.0	0.0	0.00	1621.99
11.800	0.00	0.0	0.0	0.0	0.00	1621.99
11.900	0.00	0.0	0.0	0.0	0.00	1621.99
12.000	0.00	0.0	0.0	0.0	0.00	1621.99
12.100	1.00	1.0	1.0	1.0	0.00	1622.36
12.200	6.00	7.0	8.0	8.0	0.00	1623.82
12.300	15.00	21.0	29.0	29.0	0.00	1625.05
12.400	25.00	40.0	69.0	69.0	0.00	1625.29
12.500	37.00	62.0	131.0	131.0	0.00	1625.65
12.600	48.00	85.0	216.0	216.0	0.00	1626.12
12.700	52.00	100.0	316.0	316.0	0.00	1626.64
12.800	49.00	101.0	411.7	417.0	2.64	1627.14
12.900	44.00	93.0	486.8	504.7	8.93	1627.53
13.000	39.00	83.0	551.5	569.8	9.18	1627.84
13.100	33.00	72.0	604.8	623.5	9.36	1628.08
13.200	27.00	60.0	645.8	664.8	9.47	1628.27
13.300	23.00	50.0	676.7	695.8	9.55	1628.41
13.400	19.00	42.0	699.5	718.7	9.61	1628.51
13.500	17.00	36.0	716.2	735.5	9.67	1628.58
13.600	15.00	32.0	728.8	748.2	9.71	1628.63
13.700	14.00	29.0	738.3	757.8	9.75	1628.67
13.800	12.00	26.0	744.7	764.3	9.77	1628.70
13.900	11.00	23.0	748.2	767.7	9.78	1628.72
14.000	10.00	21.0	749.6	769.2	9.79	1628.72
14.100	10.00	20.0	750.0	769.6	9.79	1628.72
14.200	9.00	19.0	749.4	769.0	9.79	1628.72
14.300	9.00	18.0	747.9	767.4	9.78	1628.71
14.400	9.00	18.0	746.3	765.9	9.77	1628.71
14.500	8.00	17.0	743.8	763.3	9.77	1628.70
14.600	8.00	16.0	740.3	759.8	9.75	1628.68
14.700	8.00	16.0	736.8	756.3	9.74	1628.67
14.800	8.00	16.0	733.3	752.8	9.73	1628.65
14.900	8.00	16.0	729.9	749.3	9.72	1628.64
15.000	8.00	16.0	726.5	745.9	9.71	1628.62
15.100	8.00	16.0	723.1	742.5	9.69	1628.61
15.200	8.00	16.0	719.7	739.1	9.68	1628.59
15.300	7.00	15.0	715.4	734.7	9.67	1628.58
15.400	7.00	14.0	710.1	729.4	9.65	1628.55

Pond File: TMA1 .PND
Inflow Hydrograph: TMA1 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
15.500	7.00	14.0	704.8	724.1	9.63	1628.53
15.600	6.00	13.0	698.6	717.8	9.61	1628.50
15.700	6.00	12.0	691.4	710.6	9.59	1628.47
15.800	5.00	11.0	683.3	702.4	9.57	1628.43
15.900	5.00	10.0	674.2	693.3	9.54	1628.39
16.000	4.00	9.0	664.2	683.2	9.52	1628.35
16.100	4.00	8.0	653.2	672.2	9.49	1628.30
16.200	4.00	8.0	642.3	661.2	9.46	1628.25
16.300	4.00	8.0	631.5	650.3	9.43	1628.20
16.400	4.00	8.0	620.7	639.5	9.40	1628.15
16.500	4.00	8.0	609.9	628.7	9.37	1628.11
16.600	4.00	8.0	599.2	617.9	9.34	1628.06
16.700	4.00	8.0	588.6	607.2	9.31	1628.01
16.800	4.00	8.0	578.1	596.6	9.28	1627.96
16.900	4.00	8.0	567.6	586.1	9.24	1627.91
17.000	4.00	8.0	557.2	575.6	9.20	1627.86
17.100	4.00	8.0	546.9	565.2	9.16	1627.81
17.200	4.00	8.0	536.6	554.9	9.12	1627.77
17.300	4.00	8.0	526.5	544.6	9.08	1627.72
17.400	4.00	8.0	516.4	534.5	9.05	1627.67
17.500	4.00	8.0	506.4	524.4	9.01	1627.63
17.600	4.00	8.0	496.4	514.4	8.97	1627.58
17.700	4.00	8.0	486.5	504.4	8.93	1627.53
17.800	4.00	8.0	476.9	494.5	8.84	1627.49
17.900	4.00	8.0	468.7	484.9	8.06	1627.44
18.000	4.00	8.0	461.9	476.7	7.42	1627.41
18.100	4.00	8.0	456.2	469.9	6.87	1627.38
18.200	4.00	8.0	451.4	464.2	6.41	1627.35
18.300	4.00	8.0	447.3	459.4	6.03	1627.33
18.400	4.00	8.0	443.9	455.3	5.70	1627.31
18.500	4.00	8.0	441.0	451.9	5.43	1627.30
18.600	4.00	8.0	438.6	449.0	5.20	1627.28
18.700	4.00	8.0	436.6	446.6	5.01	1627.27
18.800	4.00	8.0	434.9	444.6	4.85	1627.26
18.900	4.00	8.0	433.5	442.9	4.71	1627.25
19.000	4.00	8.0	432.3	441.5	4.60	1627.25
19.100	4.00	8.0	431.3	440.3	4.50	1627.24
19.200	4.00	8.0	430.4	439.3	4.42	1627.24
19.300	4.00	8.0	429.7	438.4	4.36	1627.23
19.400	4.00	8.0	429.1	437.7	4.30	1627.23
19.500	4.00	8.0	428.6	437.1	4.25	1627.23
19.600	3.00	7.0	427.4	435.6	4.13	1627.22
19.700	3.00	6.0	425.5	433.4	3.95	1627.21
19.800	3.00	6.0	423.9	431.5	3.80	1627.20
19.900	3.00	6.0	422.5	429.9	3.67	1627.20
20.000	3.00	6.0	421.4	428.5	3.56	1627.19

POND-2 Version: 5.16 S/N: 1295130172
EXECUTED: 04-24-1995 06:38:11

Pond File: TMA1 .PND
Inflow Hydrograph: TMA1 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
20.100	3.00	6.0	420.5	427.4	3.47	1627.19
20.200	3.00	6.0	419.7	426.5	3.40	1627.18
20.300	3.00	6.0	419.0	425.7	3.33	1627.18
20.400	3.00	6.0	418.4	425.0	3.28	1627.17
20.500	3.00	6.0	418.0	424.4	3.24	1627.17
20.600	3.00	6.0	417.6	424.0	3.20	1627.17
20.700	3.00	6.0	417.2	423.6	3.17	1627.17
20.800	3.00	6.0	416.9	423.2	3.14	1627.17
20.900	3.00	6.0	416.7	422.9	3.12	1627.17
21.000	3.00	6.0	416.5	422.7	3.10	1627.16
21.100	3.00	6.0	416.3	422.5	3.08	1627.16
21.200	3.00	6.0	416.2	422.3	3.07	1627.16
21.300	3.00	6.0	416.1	422.2	3.06	1627.16
21.400	3.00	6.0	416.0	422.1	3.05	1627.16
21.500	3.00	6.0	415.9	422.0	3.04	1627.16
21.600	3.00	6.0	415.8	421.9	3.03	1627.16
21.700	3.00	6.0	415.8	421.8	3.03	1627.16
21.800	3.00	6.0	415.7	421.8	3.02	1627.16
21.900	3.00	6.0	415.7	421.7	3.02	1627.16
22.000	3.00	6.0	415.7	421.7	3.02	1627.16
22.100	3.00	6.0	415.6	421.7	3.01	1627.16
22.200	3.00	6.0	415.6	421.6	3.01	1627.16
22.300	3.00	6.0	415.6	421.6	3.01	1627.16
22.400	3.00	6.0	415.6	421.6	3.01	1627.16
22.500	3.00	6.0	415.5	421.6	3.01	1627.16
22.600	3.00	6.0	415.5	421.5	3.01	1627.16
22.700	2.00	5.0	414.7	420.5	2.93	1627.15
22.800	2.00	4.0	413.1	418.7	2.78	1627.15
22.900	2.00	4.0	411.8	417.1	2.65	1627.14
23.000	2.00	4.0	410.7	415.8	2.55	1627.13
23.100	2.00	4.0	409.8	414.7	2.46	1627.13
23.200	2.00	4.0	409.0	413.8	2.39	1627.12
23.300	2.00	4.0	408.4	413.0	2.33	1627.12
23.400	2.00	4.0	407.8	412.4	2.27	1627.12
23.500	2.00	4.0	407.4	411.8	2.23	1627.12
23.600	2.00	4.0	407.0	411.4	2.19	1627.11
23.700	2.00	4.0	406.7	411.0	2.16	1627.11
23.800	2.00	4.0	406.4	410.7	2.14	1627.11
23.900	2.00	4.0	406.2	410.4	2.11	1627.11
24.000	2.00	4.0	406.0	410.2	2.10	1627.11
24.100	1.00	3.0	405.0	409.0	2.00	1627.10
24.200	1.00	2.0	403.3	407.0	1.84	1627.09
24.300	1.00	2.0	401.9	405.3	1.71	1627.09
24.400	1.00	2.0	400.7	403.9	1.59	1627.08
24.500	1.00	2.0	399.7	402.7	1.50	1627.07
24.600	1.00	2.0	398.8	401.7	1.42	1627.07

Pond File: TMA1 .PND
Inflow Hydrograph: TMA1 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
24.700	1.00	2.0	398.1	400.8	1.35	1627.07
24.800	1.00	2.0	397.5	400.1	1.30	1627.06
24.900	1.00	2.0	397.0	399.5	1.25	1627.06
25.000	1.00	2.0	396.6	399.0	1.21	1627.06
25.100	1.00	2.0	396.3	398.6	1.18	1627.06
25.200	1.00	2.0	396.0	398.3	1.15	1627.05
25.300	1.00	2.0	395.7	398.0	1.12	1627.05
25.400	0.00	1.0	394.7	396.7	1.02	1627.05
25.500	0.00	0.0	393.0	394.7	0.86	1627.04
25.600	0.00	0.0	391.5	393.0	0.72	1627.03
25.700	0.00	0.0	390.3	391.5	0.61	1627.02
25.800	0.00	0.0	389.3	390.3	0.51	1627.02
25.900	0.00	0.0	388.4	389.3	0.43	1627.01

********** SUMMARY OF ROUTING COMPUTATIONS **********

Pond File: TMA1 .PND
Inflow Hydrograph: TMA1 .HYD
Outflow Hydrograph: OUT .HYD

Starting Pond W.S. Elevation = 1621.99 ft

***** Summary of Peak Outflow and Peak Elevation *****

Peak Inflow = 52.00 cfs
Peak Outflow = 9.79 cfs
Peak Elevation = 1628.72 ft

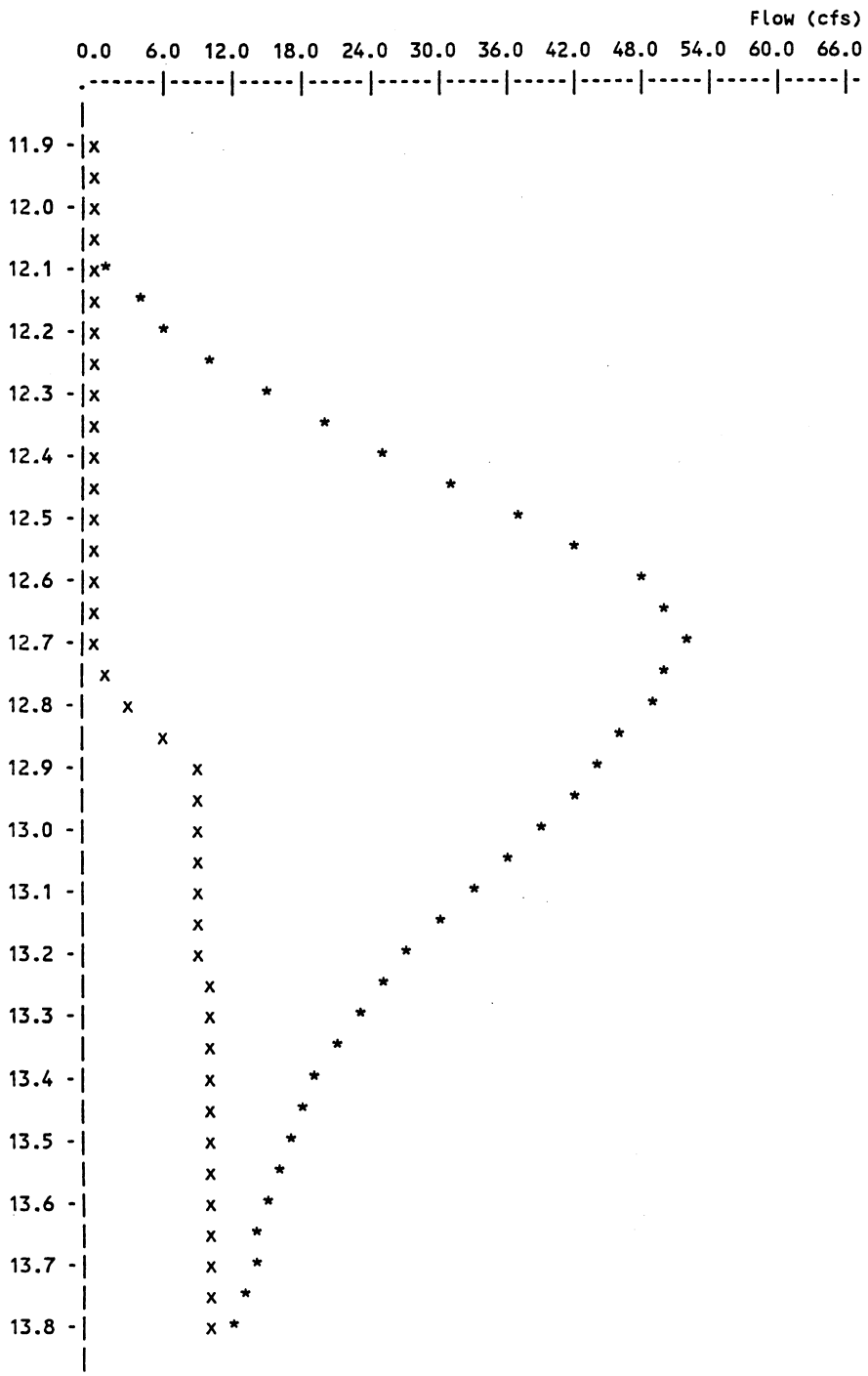
***** Summary of Approximate Peak Storage *****

Initial Storage	=	0.00 ac-ft
Peak Storage From Storm	=	3.14 ac-ft
Total Storage in Pond	=	3.14 ac-ft

Pond File: TMA1 .PND
Inflow Hydrograph: TMA1 .HYD
Outflow Hydrograph: OUT .HYD

EXECUTED: 04-24-1995
06:38:11

Peak Inflow = 52.00 cfs
Peak Outflow = 9.79 cfs
Peak Elevation = 1628.72 ft



* File: TMA1 .HYD Qmax = 52.0 cfs
x File: OUT .HYD Qmax = 9.8 cfs

CRANDON MINING COMPANY

TMA RUNOFF BASIN 9

QTR-55 HYDROLOGY
POND-2 HYDROLOGY

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:54:44
Watershed file: --> TMA2 .WSD
Hydrograph file: --> TMA2 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 9 3/95
100 YEAR STORM EVENT

>>>> Input Parameters Used to Compute Hydrograph <<<<

Subarea Description	AREA (acres)	CN	Tc (hrs)	* Tt (hrs)	Precip. (in)	Runoff (in)	Ia/p input/used	
3-1	15.46	60.0	0.50	0.10	5.00	1.30	.27	.30
3-2	12.21	60.0	0.50	0.20	5.00	1.30	.27	.30
1-3	12.37	60.0	0.50	0.20	5.00	1.30	.27	.30
1-4	9.14	60.0	0.50	0.00	5.00	1.30	.27	.30
3-4A	6.20	60.0	0.30	0.30	5.00	1.30	.27	.30

* Travel time from subarea outfall to composite watershed outfall point.
Total area = 55.38 acres or 0.08653 sq.mi
Peak discharge = 45 cfs

>>>> Computer Modifications of Input Parameters <<<<<

Subarea Description	Input Values Tc (hr)	Input Values * Tt (hr)	Rounded Values Tc (hr)	Rounded Values * Tt (hr)	Ia/p Interpolated (Yes/No)	Ia/p Messages
3-1	0.50	0.14	0.50	0.10	No	--
3-2	0.50	0.15	0.50	0.20	No	--
1-3	0.50	0.16	0.50	0.20	No	--
1-4	0.50	0.00	**	**	No	--
3-4A	0.30	0.29	0.30	0.30	No	--

* Travel time from subarea outfall to composite watershed outfall point.
** Tc & Tt are available in the hydrograph tables.

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:54:44
Watershed file: --> TMA2 .WSD
Hydrograph file: --> TMA2 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN - 9 3/95
100 YEAR STORM EVENT

>>>> Summary of Subarea Times to Peak <<<<

Subarea	Peak Discharge at Composite Outfall (cfs)	Time to Peak at Composite Outfall (hrs)
3-1	13	12.6
3-2	10	12.7
1-3	10	12.6
1-4	8	12.4
3-4A	6	12.6
Composite Watershed	45	12.6

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:54:44
Watershed file: --> TMA2 .WSD
Hydrograph file: --> TMA2 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN ' 9 3/95

100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	11.0 hr	11.3 hr	11.6 hr	11.9 hr	12.0 hr	12.1 hr	12.2 hr	12.3 hr	12.4 hr
3-1	0	0	0	0	0	0	1	4	8
3-2	0	0	0	0	0	0	1	2	5
1-3	0	0	0	0	0	0	1	2	5
1-4	0	0	0	0	0	1	3	6	8
3-4A	0	0	0	0	0	0	0	1	3
Total (cfs)	0	0	0	0	0	1	6	15	29

Subarea Description	12.5 hr	12.6 hr	12.7 hr	12.8 hr	13.0 hr	13.2 hr	13.4 hr	13.6 hr	13.8 hr
3-1	12	13	12	10	7	5	4	3	2
3-2	8	9	10	9	6	4	3	2	2
1-3	8	10	10	9	6	4	3	2	2
1-4	8	7	6	4	3	2	2	2	1
3-4A	5	6	5	4	3	2	1	1	1
Total (cfs)	41	45	43	36	25	17	13	10	8

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:54:44
Watershed file: --> TMA2 .WSD
Hydrograph file: --> TMA2 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 9 3/95
100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	14.0 hr	14.3 hr	14.6 hr	15.0 hr	15.5 hr	16.0 hr	16.5 hr	17.0 hr	17.5 hr
3-1	2	2	2	1	1	1	1	1	1
3-2	2	2	1	1	1	1	1	1	1
1-3	2	2	1	1	1	1	1	1	1
1-4	1	1	1	1	1	1	1	1	1
3-4A	1	1	1	1	1	0	0	0	0
Total (cfs)	8	8	6	5	5	4	4	4	4

Subarea Description	18.0 hr	19.0 hr	20.0 hr	22.0 hr	26.0 hr
3-1	1	1	1	1	0
3-2	1	1	1	0	0
1-3	1	1	1	0	0
1-4	1	0	0	0	0
3-4A	0	0	0	0	0
Total (cfs)	4	3	3	1	0

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:54:44
Watershed file: --> TMA2 .WSD
Hydrograph file: --> TMA2 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN 9 3/95

100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
11.0	0	14.8	6
11.1	0	14.9	5
11.2	0	15.0	5
11.3	0	15.1	5
11.4	0	15.2	5
11.5	0	15.3	5
11.6	0	15.4	5
11.7	0	15.5	5
11.8	0	15.6	5
11.9	0	15.7	5
12.0	0	15.8	4
12.1	1	15.9	4
12.2	6	16.0	4
12.3	15	16.1	4
12.4	29	16.2	4
12.5	41	16.3	4
12.6	45	16.4	4
12.7	43	16.5	4
12.8	36	16.6	4
12.9	30	16.7	4
13.0	25	16.8	4
13.1	21	16.9	4
13.2	17	17.0	4
13.3	15	17.1	4
13.4	13	17.2	4
13.5	12	17.3	4
13.6	10	17.4	4
13.7	9	17.5	4
13.8	8	17.6	4
13.9	8	17.7	4
14.0	8	17.8	4
14.1	8	17.9	4
14.2	8	18.0	4
14.3	8	18.1	4
14.4	7	18.2	4
14.5	7	18.3	4
14.6	6	18.4	4

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:54:44
Watershed file: --> TMA2 .WSD
Hydrograph file: --> TMA2 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN 9 3/95

100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
18.6	3	22.4	1
18.7	3	22.5	1
18.8	3	22.6	1
18.9	3	22.7	1
19.0	3	22.8	1
19.1	3	22.9	1
19.2	3	23.0	1
19.3	3	23.1	1
19.4	3	23.2	1
19.5	3	23.3	1
19.6	3	23.4	1
19.7	3	23.5	1
19.8	3	23.6	1
19.9	3	23.7	1
20.0	3	23.8	1
20.1	3	23.9	1
20.2	3	24.0	0
20.3	3	24.1	0
20.4	3	24.2	0
20.5	2	24.3	0
20.6	2	24.4	0
20.7	2	24.5	0
20.8	2	24.6	0
20.9	2	24.7	0
21.0	2	24.8	0
21.1	2	24.9	0
21.2	2	25.0	0
21.3	2	25.1	0
21.4	2	25.2	0
21.5	2	25.3	0
21.6	1	25.4	0
21.7	1	25.5	0
21.8	1	25.6	0
21.9	1	25.7	0
22.0	1	25.8	0
22.1	1	25.9	0
22.2	1		
22.3	1		

```
**************************************************************************
*                                                                        *
* CRANDON MINING COMPANY          93C049.27       FOTH & VAN DYKE        *
*       TMA Runoff Basin 9                              3/95             *
*                                                                        *
*                                                                        *
*                                                                        *
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Inflow Hydrograph: TMA2 .HYD
Rating Table file: TMA2 .PND

----INITIAL CONDITIONS----
Elevation = 1690.00 ft
Outflow = 0.00 cfs
Storage = 0.00 ac-ft

GIVEN POND DATA			INTERMEDIATE ROUTING COMPUTATIONS	
ELEVATION (ft)	OUTFLOW (cfs)	STORAGE (ac-ft)	2S/t (cfs)	2S/t + O (cfs)
1690.00	0.0	0.000	0.0	0.0
1690.20	0.0	0.002	0.5	0.5
1690.40	0.0	0.004	1.1	1.1
1690.60	0.0	0.007	1.7	1.7
1690.80	0.0	0.010	2.5	2.5
1691.00	0.0	0.014	3.3	3.3
1691.20	0.0	0.018	4.3	4.3
1691.40	0.0	0.022	5.3	5.3
1691.60	0.0	0.027	6.5	6.5
1691.80	0.0	0.032	7.8	7.8
1692.00	0.0	0.038	9.2	9.2
1692.20	0.0	0.044	10.7	10.7
1692.40	0.0	0.051	12.4	12.4
1692.60	0.0	0.059	14.3	14.3
1692.80	0.0	0.067	16.3	16.3
1693.00	0.0	0.083	20.0	20.0
1693.20	0.0	0.446	107.9	107.9
1693.40	0.0	0.812	196.5	196.5
1693.60	0.0	1.180	285.6	285.6
1693.80	0.0	1.552	375.5	375.5
1694.00	7.6	1.926	466.0	473.6
1694.20	7.8	2.303	557.2	565.0
1694.40	7.9	2.682	649.1	657.0
1694.60	9.1	3.065	741.6	750.7
1694.80	13.6	3.450	834.9	848.5

Time increment (t) = 0.100 hrs.

Pond File: TMA2 .PND
Inflow Hydrograph: TMA2 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
11.000	0.00	-----	0.0	0.0	0.00	1690.00
11.100	0.00	0.0	0.0	0.0	0.00	1690.00
11.200	0.00	0.0	0.0	0.0	0.00	1690.00
11.300	0.00	0.0	0.0	0.0	0.00	1690.00
11.400	0.00	0.0	0.0	0.0	0.00	1690.00
11.500	0.00	0.0	0.0	0.0	0.00	1690.00
11.600	0.00	0.0	0.0	0.0	0.00	1690.00
11.700	0.00	0.0	0.0	0.0	0.00	1690.00
11.800	0.00	0.0	0.0	0.0	0.00	1690.00
11.900	0.00	0.0	0.0	0.0	0.00	1690.00
12.000	0.00	0.0	0.0	0.0	0.00	1690.00
12.100	1.00	1.0	1.0	1.0	0.00	1690.38
12.200	6.00	7.0	8.0	8.0	0.00	1691.84
12.300	15.00	21.0	29.0	29.0	0.00	1693.02
12.400	29.00	44.0	73.0	73.0	0.00	1693.12
12.500	41.00	70.0	143.0	143.0	0.00	1693.28
12.600	45.00	86.0	229.0	229.0	0.00	1693.47
12.700	43.00	88.0	317.0	317.0	0.00	1693.67
12.800	36.00	79.0	392.8	396.0	1.59	1693.84
12.900	30.00	66.0	445.9	458.8	6.45	1693.97
13.000	25.00	55.0	485.6	500.9	7.66	1694.06
13.100	21.00	46.0	516.2	531.6	7.73	1694.13
13.200	17.00	38.0	538.6	554.2	7.78	1694.18
13.300	15.00	32.0	555.0	570.6	7.81	1694.21
13.400	13.00	28.0	567.4	583.0	7.82	1694.24
13.500	12.00	25.0	576.7	592.4	7.83	1694.26
13.600	10.00	22.0	583.0	598.7	7.84	1694.27
13.700	9.00	19.0	586.3	602.0	7.84	1694.28
13.800	8.00	17.0	587.7	603.3	7.84	1694.28
13.900	8.00	16.0	588.0	603.7	7.84	1694.28
14.000	8.00	16.0	588.3	604.0	7.84	1694.28
14.100	8.00	16.0	588.6	604.3	7.84	1694.29
14.200	8.00	16.0	588.9	604.6	7.84	1694.29
14.300	8.00	16.0	589.2	604.9	7.84	1694.29
14.400	7.00	15.0	588.6	604.2	7.84	1694.29
14.500	7.00	14.0	586.9	602.6	7.84	1694.28
14.600	6.00	13.0	584.2	599.9	7.84	1694.28
14.700	6.00	12.0	580.5	596.2	7.83	1694.27
14.800	6.00	12.0	576.9	592.5	7.83	1694.26
14.900	5.00	11.0	572.2	587.9	7.82	1694.25
15.000	5.00	10.0	566.6	582.2	7.82	1694.24
15.100	5.00	10.0	561.0	576.6	7.81	1694.23
15.200	5.00	10.0	555.3	571.0	7.81	1694.21
15.300	5.00	10.0	549.7	565.3	7.80	1694.20
15.400	5.00	10.0	544.2	559.7	7.79	1694.19

Pond File: TMA2 .PND
Inflow Hydrograph: TMA2 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
15.500	5.00	10.0	538.6	554.2	7.78	1694.18
15.600	5.00	10.0	533.1	548.6	7.76	1694.16
15.700	5.00	10.0	527.6	543.1	7.75	1694.15
15.800	4.00	9.0	521.1	536.6	7.74	1694.14
15.900	4.00	8.0	513.7	529.1	7.72	1694.12
16.000	4.00	8.0	506.3	521.7	7.71	1694.11
16.100	4.00	8.0	498.9	514.3	7.69	1694.09
16.200	4.00	8.0	491.5	506.9	7.67	1694.07
16.300	4.00	8.0	484.2	499.5	7.66	1694.06
16.400	4.00	8.0	476.9	492.2	7.64	1694.04
16.500	4.00	8.0	469.7	484.9	7.62	1694.02
16.600	4.00	8.0	462.5	477.7	7.61	1694.01
16.700	4.00	8.0	455.8	470.5	7.35	1693.99
16.800	4.00	8.0	450.1	463.8	6.83	1693.98
16.900	4.00	8.0	445.3	458.1	6.40	1693.97
17.000	4.00	8.0	441.3	453.3	6.02	1693.96
17.100	4.00	8.0	437.8	449.3	5.71	1693.95
17.200	4.00	8.0	434.9	445.8	5.45	1693.94
17.300	4.00	8.0	432.5	442.9	5.22	1693.94
17.400	4.00	8.0	430.4	440.5	5.03	1693.93
17.500	4.00	8.0	428.7	438.4	4.87	1693.93
17.600	4.00	8.0	427.2	436.7	4.74	1693.92
17.700	4.00	8.0	426.0	435.2	4.62	1693.92
17.800	4.00	8.0	424.9	434.0	4.53	1693.92
17.900	4.00	8.0	424.0	432.9	4.44	1693.92
18.000	4.00	8.0	423.3	432.0	4.38	1693.92
18.100	4.00	8.0	422.6	431.3	4.32	1693.91
18.200	4.00	8.0	422.1	430.6	4.27	1693.91
18.300	4.00	8.0	421.6	430.1	4.23	1693.91
18.400	4.00	8.0	421.3	429.6	4.19	1693.91
18.500	4.00	8.0	420.9	429.3	4.16	1693.91
18.600	3.00	7.0	419.8	427.9	4.06	1693.91
18.700	3.00	6.0	418.0	425.8	3.90	1693.90
18.800	3.00	6.0	416.5	424.0	3.76	1693.90
18.900	3.00	6.0	415.2	422.5	3.64	1693.90
19.000	3.00	6.0	414.2	421.2	3.54	1693.89
19.100	3.00	6.0	413.2	420.2	3.46	1693.89
19.200	3.00	6.0	412.5	419.2	3.39	1693.89
19.300	3.00	6.0	411.8	418.5	3.33	1693.89
19.400	3.00	6.0	411.3	417.8	3.28	1693.89
19.500	3.00	6.0	410.8	417.3	3.23	1693.89
19.600	3.00	6.0	410.4	416.8	3.20	1693.88
19.700	3.00	6.0	410.1	416.4	3.17	1693.88
19.800	3.00	6.0	409.8	416.1	3.14	1693.88
19.900	3.00	6.0	409.6	415.8	3.12	1693.88
20.000	3.00	6.0	409.4	415.6	3.10	1693.88

Pond File: TMA2 .PND
Inflow Hydrograph: TMA2 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
20.100	3.00	6.0	409.2	415.4	3.08	1693.88
20.200	3.00	6.0	409.0	415.2	3.07	1693.88
20.300	3.00	6.0	408.9	415.0	3.06	1693.88
20.400	3.00	6.0	408.8	414.9	3.05	1693.88
20.500	2.00	5.0	407.9	413.8	2.97	1693.88
20.600	2.00	4.0	406.3	411.9	2.82	1693.87
20.700	2.00	4.0	404.9	410.3	2.69	1693.87
20.800	2.00	4.0	403.7	408.9	2.58	1693.87
20.900	2.00	4.0	402.7	407.7	2.49	1693.87
21.000	2.00	4.0	401.9	406.7	2.42	1693.86
21.100	2.00	4.0	401.2	405.9	2.35	1693.86
21.200	2.00	4.0	400.6	405.2	2.30	1693.86
21.300	2.00	4.0	400.1	404.6	2.25	1693.86
21.400	2.00	4.0	399.7	404.1	2.21	1693.86
21.500	2.00	4.0	399.3	403.7	2.18	1693.86
21.600	1.00	3.0	398.2	402.3	2.07	1693.85
21.700	1.00	2.0	396.3	400.2	1.91	1693.85
21.800	1.00	2.0	394.8	398.3	1.77	1693.85
21.900	1.00	2.0	393.5	396.8	1.65	1693.84
22.000	1.00	2.0	392.4	395.5	1.55	1693.84
22.100	1.00	2.0	391.5	394.4	1.46	1693.84
22.200	1.00	2.0	390.7	393.5	1.39	1693.84
22.300	1.00	2.0	390.1	392.7	1.33	1693.84
22.400	1.00	2.0	389.5	392.1	1.28	1693.83
22.500	1.00	2.0	389.0	391.5	1.24	1693.83
22.600	1.00	2.0	388.6	391.0	1.20	1693.83
22.700	1.00	2.0	388.3	390.6	1.17	1693.83
22.800	1.00	2.0	388.0	390.3	1.14	1693.83
22.900	1.00	2.0	387.8	390.0	1.12	1693.83
23.000	1.00	2.0	387.6	389.8	1.10	1693.83
23.100	1.00	2.0	387.4	389.6	1.09	1693.83
23.200	1.00	2.0	387.2	389.4	1.07	1693.83
23.300	1.00	2.0	387.1	389.2	1.06	1693.83
23.400	1.00	2.0	387.0	389.1	1.05	1693.83
23.500	1.00	2.0	386.9	389.0	1.04	1693.83
23.600	1.00	2.0	386.9	388.9	1.04	1693.83
23.700	1.00	2.0	386.8	388.9	1.03	1693.83
23.800	1.00	2.0	386.7	388.8	1.03	1693.83
23.900	1.00	2.0	386.7	388.7	1.02	1693.83
24.000	0.00	1.0	385.8	387.7	0.94	1693.82
24.100	0.00	0.0	384.2	385.8	0.80	1693.82
24.200	0.00	0.0	382.9	384.2	0.67	1693.82
24.300	0.00	0.0	381.7	382.9	0.57	1693.81
24.400	0.00	0.0	380.8	381.7	0.48	1693.81
24.500	0.00	0.0	380.0	380.8	0.41	1693.81
24.600	0.00	0.0	379.3	380.0	0.34	1693.81

POND-2 Version: 5.16 S/N: 1295130172
EXECUTED: 04-24-1995 06:39:40

Pond File: TMA2 .PND
Inflow Hydrograph: TMA2 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH

ROUTING COMPUTATIONS

TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
24.700	0.00	0.0	378.7	379.3	0.29	1693.81
24.800	0.00	0.0	378.2	378.7	0.24	1693.81
24.900	0.00	0.0	377.8	378.2	0.21	1693.81
25.000	0.00	0.0	377.4	377.8	0.17	1693.80
25.100	0.00	0.0	377.1	377.4	0.15	1693.80
25.200	0.00	0.0	376.9	377.1	0.12	1693.80
25.300	0.00	0.0	376.7	376.9	0.11	1693.80
25.400	0.00	0.0	376.5	376.7	0.09	1693.80
25.500	0.00	0.0	376.4	376.5	0.08	1693.80
25.600	0.00	0.0	376.2	376.4	0.06	1693.80
25.700	0.00	0.0	376.1	376.2	0.05	1693.80
25.800	0.00	0.0	376.0	376.1	0.05	1693.80
25.900	0.00	0.0	376.0	376.0	0.04	1693.80

********** SUMMARY OF ROUTING COMPUTATIONS **********

Pond File: TMA2 .PND
Inflow Hydrograph: TMA2 .HYD
Outflow Hydrograph: OUT .HYD

Starting Pond W.S. Elevation = 1690.00 ft

***** Summary of Peak Outflow and Peak Elevation *****

Peak Inflow = 45.00 cfs
Peak Outflow = 7.84 cfs
Peak Elevation = 1694.29 ft

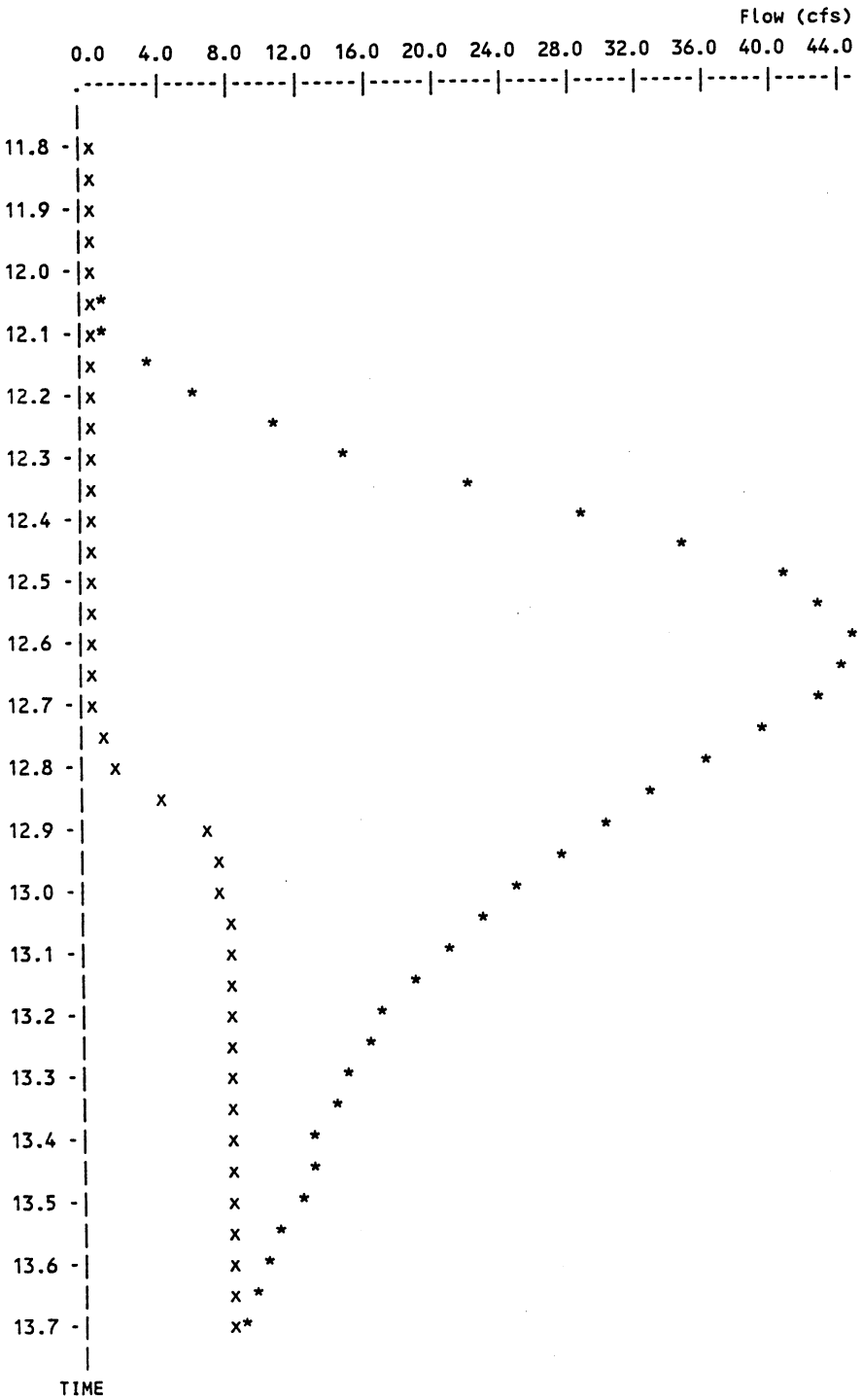
***** Summary of Approximate Peak Storage *****

Initial Storage	=	0.00 ac-ft
Peak Storage From Storm	=	2.47 ac-ft
Total Storage in Pond	=	2.47 ac-ft

Pond File: TMA2 .PND
Inflow Hydrograph: TMA2 .HYD
Outflow Hydrograph: OUT .HYD

EXECUTED: 04-24-1995
06:39:40

Peak Inflow = 45.00 cfs
Peak Outflow = 7.84 cfs
Peak Elevation = 1694.29 ft



* File: TMA2 .HYD Qmax = 45.0 cfs
x File: OUT .HYD Qmax = 7.8 cfs

CRANDON MINING COMPANY

TMA RUNOFF BASIN 10

QTR-55 HYDROLOGY
POND-2 HYDROLOGY

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:55:10
Watershed file: --> TMA3 .WSD
Hydrograph file: --> TMA3 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 10 3/95
100 YEAR STORM EVENT

>>>> Input Parameters Used to Compute Hydrograph <<<<

Subarea Description	AREA (acres)	CN	Tc (hrs)	* Tt (hrs)	Precip. (in)	Runoff (in)	Ia/p input/used	
2-4	21.98	60.0	0.50	0.10	5.00	1.30	.27	.30
2-2	17.72	60.0	0.50	0.20	5.00	1.30	.27	.30
2-5	6.12	60.0	0.50	0.10	5.00	1.30	.27	.30
2-6B	8.52	60.0	0.50	0.30	5.00	1.30	.27	.30

* Travel time from subarea outfall to composite watershed outfall point.
Total area = 54.34 acres or 0.08491 sq.mi
Peak discharge = 42 cfs

>>>> Computer Modifications of Input Parameters <<<<<

Subarea Description	Input Values Tc (hr)	Input Values * Tt (hr)	Rounded Values Tc (hr)	Rounded Values * Tt (hr)	Ia/p Interpolated (Yes/No)	Ia/p Messages
2-4	0.50	0.10	**	**	No	--
2-2	0.50	0.20	**	**	No	--
2-5	0.50	0.10	**	**	No	--
2-6B	0.50	0.33	0.50	0.30	No	--

* Travel time from subarea outfall to composite watershed outfall point.
** Tc & Tt are available in the hydrograph tables.

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:55:10
Watershed file: --> TMA3 .WSD
Hydrograph file: --> TMA3 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 10 3/95
100 YEAR STORM EVENT

>>>> Summary of Subarea Times to Peak <<<<

Subarea	Peak Discharge at Composite Outfall (cfs)	Time to Peak at Composite Outfall (hrs)
2-4	19	12.6
2-2	14	12.6
2-5	5	12.5
2-6B	6	12.7
Composite Watershed	42	12.6

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:55:10
Watershed file: --> TMA3 .WSD
Hydrograph file: --> TMA3 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN 10 3/95

100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	11.0 hr	11.3 hr	11.6 hr	11.9 hr	12.0 hr	12.1 hr	12.2 hr	12.3 hr	12.4 hr
2-4	0	0	0	0	0	0	2	5	11
2-2	0	0	0	0	0	0	1	3	7
2-5	0	0	0	0	0	0	0	1	3
2-6B	0	0	0	0	0	0	0	0	1
Total (cfs)	0	0	0	0	0	0	3	9	22

Subarea Description	12.5 hr	12.6 hr	12.7 hr	12.8 hr	13.0 hr	13.2 hr	13.4 hr	13.6 hr	13.8 hr
2-4	17	19	17	15	10	7	5	4	4
2-2	11	14	14	13	9	6	4	3	3
2-5	5	5	5	4	3	2	1	1	1
2-6B	3	4	6	6	5	4	3	2	2
Total (cfs)	36	42	42	38	27	19	13	10	10

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:55:10
Watershed file: --> TMA3 .WSD
Hydrograph file: --> TMA3 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN 10 3/95

100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	14.0 hr	14.3 hr	14.6 hr	15.0 hr	15.5 hr	16.0 hr	16.5 hr	17.0 hr	17.5 hr
2-4	3	3	2	2	2	2	2	1	1
2-2	3	2	2	2	2	1	1	1	1
2-5	1	1	1	1	1	0	0	0	0
2-6B	1	1	1	1	1	1	1	1	1
Total (cfs)	8	7	6	6	6	4	4	3	3

Subarea Description	18.0 hr	19.0 hr	20.0 hr	22.0 hr	26.0 hr
2-4	1	1	1	1	0
2-2	1	1	1	1	0
2-5	0	0	0	0	0
2-6B	1	0	0	0	0
Total (cfs)	3	2	2	2	0

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:55:10
Watershed file: --> TMA3 .WSD
Hydrograph file: --> TMA3 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN 10 3/95

100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
11.0	0	14.8	6
11.1	0	14.9	6
11.2	0	15.0	6
11.3	0	15.1	6
11.4	0	15.2	6
11.5	0	15.3	6
11.6	0	15.4	6
11.7	0	15.5	6
11.8	0	15.6	6
11.9	0	15.7	5
12.0	0	15.8	5
12.1	0	15.9	4
12.2	3	16.0	4
12.3	9	16.1	4
12.4	22	16.2	4
12.5	36	16.3	4
12.6	42	16.4	4
12.7	42	16.5	4
12.8	38	16.6	4
12.9	32	16.7	4
13.0	27	16.8	3
13.1	23	16.9	3
13.2	19	17.0	3
13.3	16	17.1	3
13.4	13	17.2	3
13.5	12	17.3	3
13.6	10	17.4	3
13.7	10	17.5	3
13.8	10	17.6	3
13.9	9	17.7	3
14.0	8	17.8	3
14.1	8	17.9	3
14.2	7	18.0	3
14.3	7	18.1	3
14.4	7	18.2	3
14.5	6	18.3	3
14.6	6	18.4	3

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:55:10
Watershed file: --> TMA3 .WSD
Hydrograph file: --> TMA3 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN 10 3/95

100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
18.6	2	22.4	2
18.7	2	22.5	2
18.8	2	22.6	2
18.9	2	22.7	2
19.0	2	22.8	2
19.1	2	22.9	2
19.2	2	23.0	2
19.3	2	23.1	1
19.4	2	23.2	1
19.5	2	23.3	1
19.6	2	23.4	1
19.7	2	23.5	1
19.8	2	23.6	1
19.9	2	23.7	1
20.0	2	23.8	1
20.1	2	23.9	1
20.2	2	24.0	1
20.3	2	24.1	1
20.4	2	24.2	1
20.5	2	24.3	1
20.6	2	24.4	1
20.7	2	24.5	1
20.8	2	24.6	1
20.9	2	24.7	1
21.0	2	24.8	1
21.1	2	24.9	1
21.2	2	25.0	0
21.3	2	25.1	0
21.4	2	25.2	0
21.5	2	25.3	0
21.6	2	25.4	0
21.7	2	25.5	0
21.8	2	25.6	0
21.9	2	25.7	0
22.0	2	25.8	0
22.1	2	25.9	0
22.2	2		
22.3	2		

```
**********************************************************************
*                                                                    *
* CRANDON MINING COMPANY          93C049.27      FOTH & VAN DYKE     *
*       TMA Runoff Basin 10                          3/95            *
*                                                                    *
*                                                                    *
*                                                                    *
**********************************************************************
```

Inflow Hydrograph: TMA3 .HYD
Rating Table file: TMA3 .PND

----INITIAL CONDITIONS----
Elevation = 1645.00 ft
Outflow = 0.00 cfs
Storage = 0.00 ac-ft

GIVEN POND DATA			INTERMEDIATE ROUTING COMPUTATIONS	
ELEVATION (ft)	OUTFLOW (cfs)	STORAGE (ac-ft)	2S/t (cfs)	2S/t + O (cfs)
1645.00	0.0	0.000	0.0	0.0
1645.50	0.0	0.006	1.4	1.4
1646.00	0.0	0.014	3.3	3.3
1646.50	0.0	0.024	5.9	5.9
1647.00	0.0	0.038	9.2	9.2
1647.50	0.0	0.055	13.3	13.3
1648.00	0.0	0.078	19.0	19.0
1648.50	0.0	0.393	95.2	95.2
1649.00	0.0	0.723	175.0	175.0
1649.50	8.0	1.069	258.6	266.6
1650.00	8.5	1.429	345.9	354.4
1650.50	8.9	1.806	437.1	446.0
1651.00	9.3	2.199	532.3	541.6
1651.50	9.6	2.609	631.4	641.0
1652.00	10.0	3.036	734.7	744.7
1652.50	10.4	3.480	842.3	852.7
1653.00	10.7	3.942	954.1	964.8
1653.50	11.0	4.422	1070.2	1081.2
1654.00	11.3	4.921	1190.9	1202.2
1654.50	11.7	5.438	1316.0	1327.7

Time increment (t) = 0.100 hrs.

POND-2 Version: 5.16 S/N: 1295130172
EXECUTED: 04-24-1995 06:41:15

Pond File: TMA3 .PND
Inflow Hydrograph: TMA3 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
11.000	0.00	-----	0.0	0.0	0.00	1645.00
11.100	0.00	0.0	0.0	0.0	0.00	1645.00
11.200	0.00	0.0	0.0	0.0	0.00	1645.00
11.300	0.00	0.0	0.0	0.0	0.00	1645.00
11.400	0.00	0.0	0.0	0.0	0.00	1645.00
11.500	0.00	0.0	0.0	0.0	0.00	1645.00
11.600	0.00	0.0	0.0	0.0	0.00	1645.00
11.700	0.00	0.0	0.0	0.0	0.00	1645.00
11.800	0.00	0.0	0.0	0.0	0.00	1645.00
11.900	0.00	0.0	0.0	0.0	0.00	1645.00
12.000	0.00	0.0	0.0	0.0	0.00	1645.00
12.100	0.00	0.0	0.0	0.0	0.00	1645.00
12.200	3.00	3.0	3.0	3.0	0.00	1645.93
12.300	9.00	12.0	15.0	15.0	0.00	1647.65
12.400	22.00	31.0	46.0	46.0	0.00	1648.18
12.500	36.00	58.0	104.0	104.0	0.00	1648.56
12.600	42.00	78.0	180.8	182.0	0.61	1649.04
12.700	42.00	84.0	249.1	264.8	7.84	1649.49
12.800	38.00	80.0	312.4	329.1	8.36	1649.86
12.900	32.00	70.0	365.2	382.4	8.62	1650.15
13.000	27.00	59.0	406.5	424.2	8.80	1650.38
13.100	23.00	50.0	438.7	456.5	8.94	1650.56
13.200	19.00	42.0	462.6	480.7	9.05	1650.68
13.300	16.00	35.0	479.3	497.6	9.12	1650.77
13.400	13.00	29.0	490.0	508.3	9.16	1650.83
13.500	12.00	25.0	496.6	515.0	9.19	1650.86
13.600	10.00	22.0	500.2	518.6	9.20	1650.88
13.700	10.00	20.0	501.8	520.2	9.21	1650.89
13.800	10.00	20.0	503.4	521.8	9.22	1650.90
13.900	9.00	19.0	503.9	522.4	9.22	1650.90
14.000	8.00	17.0	502.5	520.9	9.21	1650.89
14.100	8.00	16.0	500.1	518.5	9.20	1650.88
14.200	7.00	15.0	496.7	515.1	9.19	1650.86
14.300	7.00	14.0	492.4	510.7	9.17	1650.84
14.400	7.00	14.0	488.1	506.4	9.15	1650.82
14.500	6.00	13.0	482.8	501.1	9.13	1650.79
14.600	6.00	12.0	476.6	494.8	9.10	1650.76
14.700	6.00	12.0	470.4	488.6	9.08	1650.72
14.800	6.00	12.0	464.3	482.4	9.05	1650.69
14.900	6.00	12.0	458.3	476.3	9.03	1650.66
15.000	6.00	12.0	452.3	470.3	9.00	1650.63
15.100	6.00	12.0	446.3	464.3	8.98	1650.60
15.200	6.00	12.0	440.4	458.3	8.95	1650.56
15.300	6.00	12.0	434.6	452.4	8.93	1650.53
15.400	6.00	12.0	428.8	446.6	8.90	1650.50

Pond File: TMA3 .PND
Inflow Hydrograph: TMA3 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
15.500	6.00	12.0	423.0	440.8	8.88	1650.47
15.600	6.00	12.0	417.3	435.0	8.85	1650.44
15.700	5.00	11.0	410.7	428.3	8.82	1650.40
15.800	5.00	10.0	403.1	420.7	8.79	1650.36
15.900	4.00	9.0	394.6	412.1	8.75	1650.31
16.000	4.00	8.0	385.2	402.6	8.71	1650.26
16.100	4.00	8.0	375.8	393.2	8.67	1650.21
16.200	4.00	8.0	366.6	383.8	8.63	1650.16
16.300	4.00	8.0	357.4	374.6	8.59	1650.11
16.400	4.00	8.0	348.3	365.4	8.55	1650.06
16.500	4.00	8.0	339.3	356.3	8.51	1650.01
16.600	4.00	8.0	330.4	347.3	8.46	1649.96
16.700	4.00	8.0	321.5	338.4	8.41	1649.91
16.800	3.00	7.0	311.8	328.5	8.35	1649.85
16.900	3.00	6.0	301.3	317.8	8.29	1649.79
17.000	3.00	6.0	290.8	307.3	8.23	1649.73
17.100	3.00	6.0	280.4	296.8	8.17	1649.67
17.200	3.00	6.0	270.2	286.4	8.11	1649.61
17.300	3.00	6.0	260.1	276.2	8.05	1649.55
17.400	3.00	6.0	250.2	266.1	7.96	1649.50
17.500	3.00	6.0	242.0	256.2	7.09	1649.44
17.600	3.00	6.0	235.3	248.0	6.38	1649.40
17.700	3.00	6.0	229.7	241.3	5.79	1649.36
17.800	3.00	6.0	225.1	235.7	5.30	1649.33
17.900	3.00	6.0	221.3	231.1	4.90	1649.31
18.000	3.00	6.0	218.2	227.3	4.57	1649.29
18.100	3.00	6.0	215.6	224.2	4.29	1649.27
18.200	3.00	6.0	213.4	221.6	4.07	1649.25
18.300	3.00	6.0	211.7	219.4	3.88	1649.24
18.400	3.00	6.0	210.2	217.7	3.73	1649.23
18.500	2.00	5.0	208.2	215.2	3.51	1649.22
18.600	2.00	4.0	205.7	212.2	3.25	1649.20
18.700	2.00	4.0	203.6	209.7	3.03	1649.19
18.800	2.00	4.0	201.9	207.6	2.85	1649.18
18.900	2.00	4.0	200.5	205.9	2.70	1649.17
19.000	2.00	4.0	199.4	204.5	2.58	1649.16
19.100	2.00	4.0	198.4	203.4	2.48	1649.15
19.200	2.00	4.0	197.6	202.4	2.39	1649.15
19.300	2.00	4.0	197.0	201.6	2.33	1649.15
19.400	2.00	4.0	196.5	201.0	2.27	1649.14
19.500	2.00	4.0	196.0	200.5	2.22	1649.14
19.600	2.00	4.0	195.6	200.0	2.18	1649.14
19.700	2.00	4.0	195.3	199.6	2.15	1649.13
19.800	2.00	4.0	195.1	199.3	2.12	1649.13
19.900	2.00	4.0	194.9	199.1	2.10	1649.13
20.000	2.00	4.0	194.7	198.9	2.08	1649.13

POND-2 Version: 5.16 S/N: 1295130172
EXECUTED: 04-24-1995 06:41:15

Pond File: TMA3 .PND
Inflow Hydrograph: TMA3 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
20.100	2.00	4.0	194.6	198.7	2.07	1649.13
20.200	2.00	4.0	194.5	198.6	2.06	1649.13
20.300	2.00	4.0	194.4	198.5	2.05	1649.13
20.400	2.00	4.0	194.3	198.4	2.04	1649.13
20.500	2.00	4.0	194.2	198.3	2.03	1649.13
20.600	2.00	4.0	194.2	198.2	2.03	1649.13
20.700	2.00	4.0	194.1	198.2	2.02	1649.13
20.800	2.00	4.0	194.1	198.1	2.02	1649.13
20.900	2.00	4.0	194.1	198.1	2.02	1649.13
21.000	2.00	4.0	194.0	198.1	2.01	1649.13
21.100	2.00	4.0	194.0	198.0	2.01	1649.13
21.200	2.00	4.0	194.0	198.0	2.01	1649.13
21.300	2.00	4.0	194.0	198.0	2.01	1649.13
21.400	2.00	4.0	194.0	198.0	2.01	1649.13
21.500	2.00	4.0	194.0	198.0	2.00	1649.13
21.600	2.00	4.0	194.0	198.0	2.00	1649.13
21.700	2.00	4.0	194.0	198.0	2.00	1649.13
21.800	2.00	4.0	193.9	198.0	2.00	1649.13
21.900	2.00	4.0	193.9	197.9	2.00	1649.13
22.000	2.00	4.0	193.9	197.9	2.00	1649.13
22.100	2.00	4.0	193.9	197.9	2.00	1649.13
22.200	2.00	4.0	193.9	197.9	2.00	1649.13
22.300	2.00	4.0	193.9	197.9	2.00	1649.13
22.400	2.00	4.0	193.9	197.9	2.00	1649.13
22.500	2.00	4.0	193.9	197.9	2.00	1649.13
22.600	2.00	4.0	193.9	197.9	2.00	1649.13
22.700	2.00	4.0	193.9	197.9	2.00	1649.13
22.800	2.00	4.0	193.9	197.9	2.00	1649.13
22.900	2.00	4.0	193.9	197.9	2.00	1649.13
23.000	2.00	4.0	193.9	197.9	2.00	1649.13
23.100	1.00	3.0	193.1	196.9	1.91	1649.12
23.200	1.00	2.0	191.6	195.1	1.75	1649.11
23.300	1.00	2.0	190.4	193.6	1.62	1649.10
23.400	1.00	2.0	189.3	192.4	1.51	1649.09
23.500	1.00	2.0	188.5	191.3	1.42	1649.09
23.600	1.00	2.0	187.8	190.5	1.35	1649.08
23.700	1.00	2.0	187.2	189.8	1.29	1649.08
23.800	1.00	2.0	186.7	189.2	1.24	1649.08
23.900	1.00	2.0	186.3	188.7	1.20	1649.07
24.000	1.00	2.0	186.0	188.3	1.16	1649.07
24.100	1.00	2.0	185.7	188.0	1.13	1649.07
24.200	1.00	2.0	185.5	187.7	1.11	1649.07
24.300	1.00	2.0	185.3	187.5	1.09	1649.07
24.400	1.00	2.0	185.2	187.3	1.08	1649.07
24.500	1.00	2.0	185.1	187.2	1.06	1649.07
24.600	1.00	2.0	185.0	187.1	1.05	1649.07

Pond File: TMA3 .PND
Inflow Hydrograph: TMA3 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
24.700	1.00	2.0	184.9	187.0	1.04	1649.07
24.800	1.00	2.0	184.8	186.9	1.03	1649.06
24.900	1.00	2.0	184.8	186.8	1.03	1649.06
25.000	0.00	1.0	183.9	185.8	0.94	1649.06
25.100	0.00	0.0	182.3	183.9	0.77	1649.05
25.200	0.00	0.0	181.1	182.3	0.64	1649.04
25.300	0.00	0.0	180.0	181.1	0.53	1649.03
25.400	0.00	0.0	179.1	180.0	0.43	1649.03
25.500	0.00	0.0	178.4	179.1	0.36	1649.02
25.600	0.00	0.0	177.8	178.4	0.30	1649.02
25.700	0.00	0.0	177.3	177.8	0.24	1649.02
25.800	0.00	0.0	176.9	177.3	0.20	1649.01
25.900	0.00	0.0	176.6	176.9	0.17	1649.01

********** SUMMARY OF ROUTING COMPUTATIONS **********

Pond File: TMA3 .PND
Inflow Hydrograph: TMA3 .HYD
Outflow Hydrograph: OUT .HYD

Starting Pond W.S. Elevation = 1645.00 ft

***** Summary of Peak Outflow and Peak Elevation *****

Peak Inflow = 42.00 cfs
Peak Outflow = 9.22 cfs
Peak Elevation = 1650.90 ft

***** Summary of Approximate Peak Storage *****

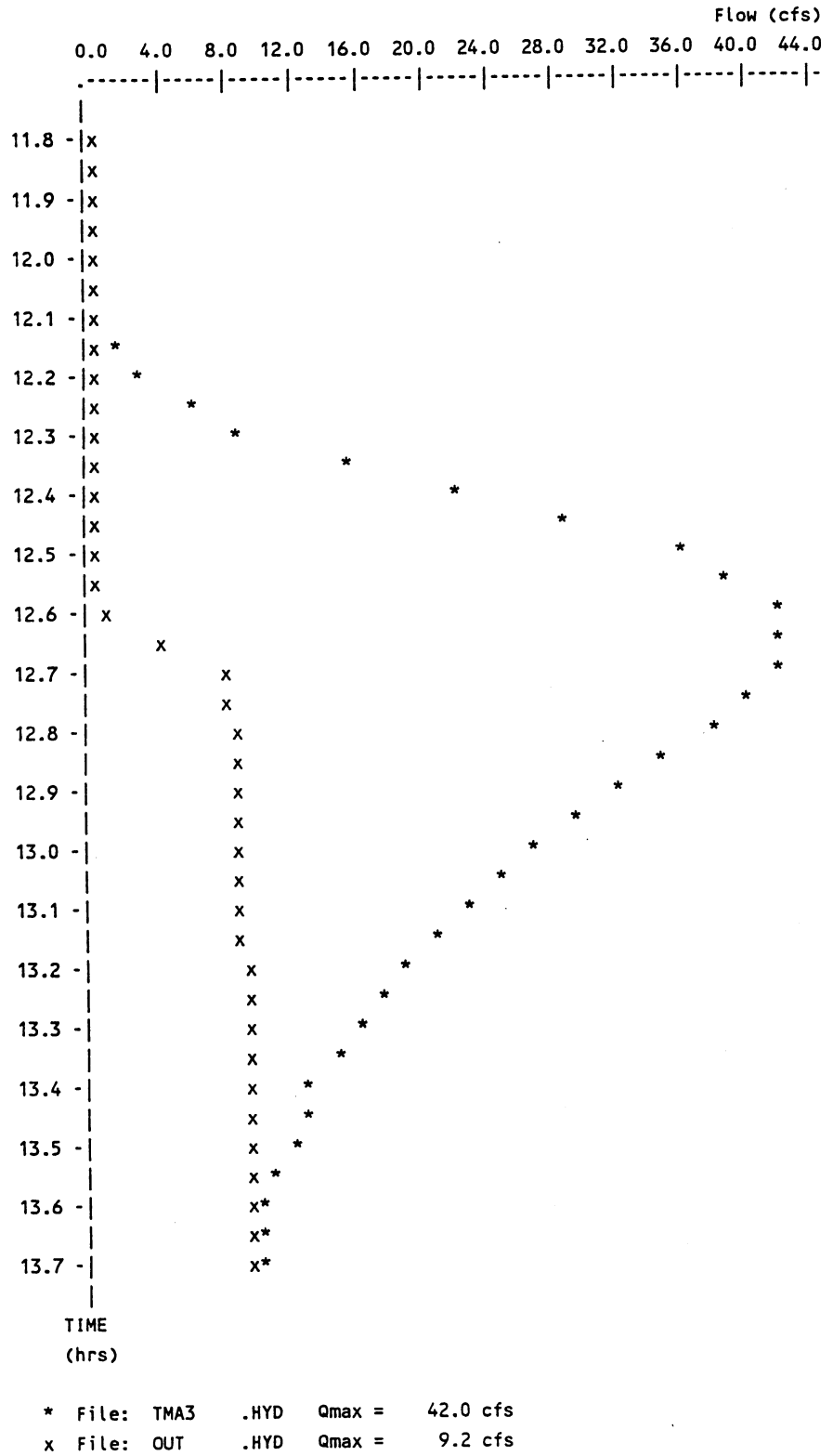
Initial Storage = 0.00 ac-ft
Peak Storage From Storm = 2.12 ac-ft

Total Storage in Pond = 2.12 ac-ft

Pond File: TMA3 .PND
Inflow Hydrograph: TMA3 .HYD
Outflow Hydrograph: OUT .HYD

EXECUTED: 04-24-1995
06:41:15

Peak Inflow = 42.00 cfs
Peak Outflow = 9.22 cfs
Peak Elevation = 1650.90 ft



* File: TMA3 .HYD Qmax = 42.0 cfs
x File: OUT .HYD Qmax = 9.2 cfs

CRANDON MINING COMPANY

TMA RUNOFF BASIN 11

QTR-55 HYDROLOGY
POND-2 HYDROLOGY

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:56:56
Watershed file: --> TMA4 .WSD
Hydrograph file: --> TMA4 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN II 3/95
100 YEAR STORM EVENT

>>>> Input Parameters Used to Compute Hydrograph <<<<

Subarea Description	AREA (acres)	CN	Tc (hrs)	* Tt (hrs)	Precip. (in)	Runoff (in)	Ia/p input/used	
2-6	1.88	60.0	0.40	0.10	5.00	1.30	.27	.30
2-3	18.55	60.0	0.50	0.30	5.00	1.30	.27	.30
4-2	16.46	60.0	0.50	0.30	5.00	1.30	.27	.30
4-4	2.10	60.0	0.50	0.10	5.00	1.30	.27	.30
STOCKPILE	25.90	60.0	0.75	0.00	5.00	1.30	.27	.30
4-1A	1.60	60.0	0.10	0.50	5.00	1.30	.27	.30
4-3A	17.10	60.0	0.20	0.50	5.00	1.30	.27	.30

* Travel time from subarea outfall to composite watershed outfall point.
Total area = 83.59 acres or 0.13061 sq.mi
Peak discharge = 60 cfs

WARNING: Drainage areas of two or more subareas
differ by a factor of 5 or greater.

>>>> Computer Modifications of Input Parameters <<<<<

Subarea Description	Input Values Tc (hr)	Input Values * Tt (hr)	Rounded Values Tc (hr)	Rounded Values * Tt (hr)	Ia/p Interpolated (Yes/No)	Ia/p Messages
2-6	0.40	0.10	**	**	No	--
2-3	0.50	0.28	0.50	0.30	No	--
4-2	0.49	0.26	0.50	0.30	No	--
4-4	0.47	0.14	0.50	0.10	No	--
STOCKPILE	0.69	0.00	0.75	0.00	No	--
4-1A	0.10	0.52	0.10	0.50	No	--
4-3A	0.25	0.48	0.20	0.50	No	--

* Travel time from subarea outfall to composite watershed outfall point.
** Tc & Tt are available in the hydrograph tables.

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:56:56
Watershed file: --> TMA4 .WSD
Hydrograph file: --> TMA4 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN II 3/95
100 YEAR STORM EVENT

>>>> Summary of Subarea Times to Peak <<<<

Subarea	Peak Discharge at Composite Outfall (cfs)	Time to Peak at Composite Outfall (hrs)
2-6	2	12.4
2-3	14	12.8
4-2	12	12.8
4-4	2	12.5
STOCKPILE	18	12.7
4-1A	2	12.6
4-3A	14	12.7
Composite Watershed	60	12.7

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:56:56
Watershed file: --> TMA4 .WSD
Hydrograph file: --> TMA4 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN II 3/95
100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	11.0 hr	11.3 hr	11.6 hr	11.9 hr	12.0 hr	12.1 hr	12.2 hr	12.3 hr	12.4 hr
2-6	0	0	0	0	0	0	0	1	2
2-3	0	0	0	0	0	0	0	1	2
4-2	0	0	0	0	0	0	0	1	2
4-4	0	0	0	0	0	0	0	0	1
STOCKPILE	0	0	0	0	0	0	2	5	9
4-1A	0	0	0	0	0	0	0	0	1
4-3A	0	0	0	0	0	0	0	1	4
Total (cfs)	0	0	0	0	0	0	2	9	21

Subarea Description	12.5 hr	12.6 hr	12.7 hr	12.8 hr	13.0 hr	13.2 hr	13.4 hr	13.6 hr	13.8 hr
2-6	2	2	1	1	1	0	0	0	0
2-3	6	10	13	14	12	8	6	4	4
4-2	5	9	11	12	11	7	5	4	3
4-4	2	2	2	1	1	1	0	0	0
STOCKPILE	14	17	18	17	13	10	7	6	5
4-1A	1	2	1	1	1	0	0	0	0
4-3A	9	13	14	14	9	6	4	3	3
Total (cfs)	39	55	60	60	48	32	22	17	15

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:56:56
Watershed file: --> TMA4 .WSD
Hydrograph file: --> TMA4 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN II 3/95

100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	14.0 hr	14.3 hr	14.6 hr	15.0 hr	15.5 hr	16.0 hr	16.5 hr	17.0 hr	17.5 hr
2-6	0	0	0	0	0	0	0	0	0
2-3	3	3	2	2	2	2	1	1	1
4-2	3	2	2	2	2	1	1	1	1
4-4	0	0	0	0	0	0	0	0	0
STOCKPILE	4	3	3	3	2	2	2	2	2
4-1A	0	0	0	0	0	0	0	0	0
4-3A	3	2	2	2	2	1	1	1	1
Total (cfs)	13	10	9	9	8	6	5	5	5

Subarea Description	18.0 hr	19.0 hr	20.0 hr	22.0 hr	26.0 hr
2-6	0	0	0	0	0
2-3	1	1	1	1	0
4-2	1	1	1	1	0
4-4	0	0	0	0	0
STOCKPILE	2	1	1	1	0
4-1A	0	0	0	0	0
4-3A	1	1	1	1	0
Total (cfs)	5	4	4	4	0

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:56:56
Watershed file: --> TMA4 .WSD
Hydrograph file: --> TMA4 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN II 3/95
100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
11.0	0	14.8	9
11.1	0	14.9	9
11.2	0	15.0	9
11.3	0	15.1	9
11.4	0	15.2	9
11.5	0	15.3	8
11.6	0	15.4	8
11.7	0	15.5	8
11.8	0	15.6	8
11.9	0	15.7	7
12.0	0	15.8	7
12.1	0	15.9	6
12.2	2	16.0	6
12.3	9	16.1	6
12.4	21	16.2	6
12.5	39	16.3	5
12.6	55	16.4	5
12.7	60	16.5	5
12.8	60	16.6	5
12.9	54	16.7	5
13.0	48	16.8	5
13.1	40	16.9	5
13.2	32	17.0	5
13.3	27	17.1	5
13.4	22	17.2	5
13.5	20	17.3	5
13.6	17	17.4	5
13.7	16	17.5	5
13.8	15	17.6	5
13.9	14	17.7	5
14.0	13	17.8	5
14.1	12	17.9	5
14.2	11	18.0	5
14.3	10	18.1	5
14.4	10	18.2	5
14.5	9	18.3	5
14.6	9	18.4	5

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:56:56
Watershed file: --> TMA4 .WSD
Hydrograph file: --> TMA4 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN II 3/95
100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
18.6	4	22.4	4
18.7	4	22.5	4
18.8	4	22.6	3
18.9	4	22.7	3
19.0	4	22.8	3
19.1	4	22.9	3
19.2	4	23.0	3
19.3	4	23.1	3
19.4	4	23.2	3
19.5	4	23.3	3
19.6	4	23.4	3
19.7	4	23.5	2
19.8	4	23.6	2
19.9	4	23.7	2
20.0	4	23.8	2
20.1	4	23.9	2
20.2	4	24.0	2
20.3	4	24.1	2
20.4	4	24.2	2
20.5	4	24.3	2
20.6	4	24.4	2
20.7	4	24.5	2
20.8	4	24.6	1
20.9	4	24.7	1
21.0	4	24.8	1
21.1	4	24.9	1
21.2	4	25.0	1
21.3	4	25.1	1
21.4	4	25.2	1
21.5	4	25.3	1
21.6	4	25.4	1
21.7	4	25.5	0
21.8	4	25.6	0
21.9	4	25.7	0
22.0	4	25.8	0
22.1	4	25.9	0
22.2	4		
22.3	4		

```
**********************************************************************
*                                                                    *
* CRANDON MINING COMPANY          93C049.27       FOTH & VAN DYKE    *
*       TMA Runoff Basin 11                            3/95          *
*                                                                    *
*                                                                    *
*                                                                    *
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Inflow Hydrograph: TMA4 .HYD
Rating Table file: TMA4 .PND

----INITIAL CONDITIONS----
Elevation = 1648.99 ft
Outflow = 0.00 cfs
Storage = 0.00 ac-ft

GIVEN POND DATA			INTERMEDIATE ROUTING COMPUTATIONS	
ELEVATION (ft)	OUTFLOW (cfs)	STORAGE (ac-ft)	2S/t (cfs)	2S/t + O (cfs)
1648.99	0.0	0.000	0.0	0.0
1649.49	0.0	0.006	1.4	1.4
1649.99	0.0	0.013	3.3	3.3
1650.49	0.0	0.024	5.8	5.8
1650.99	0.0	0.038	9.1	9.1
1651.49	0.0	0.055	13.2	13.2
1651.99	0.0	0.076	18.3	18.3
1652.49	0.0	0.386	93.4	93.4
1652.99	0.0	0.713	172.6	172.6
1653.49	0.0	1.053	254.9	254.9
1653.99	0.0	1.407	340.6	340.6
1654.49	8.9	1.775	429.6	438.5
1654.99	9.3	2.157	522.1	531.4
1655.49	9.6	2.554	618.1	627.7
1655.99	10.0	2.965	717.6	727.6
1656.49	10.4	3.392	820.8	831.2
1656.99	10.7	3.833	927.6	938.3
1657.49	11.0	4.290	1038.2	1049.2
1657.99	11.3	4.763	1152.7	1164.0
1658.49	23.0	5.252	1271.0	1294.0
1658.99	47.7	5.757	1393.3	1441.0

Time increment (t) = 0.100 hrs.

Pond File: TMA4 .PND
Inflow Hydrograph: TMA4 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
11.000	0.00	-----	0.0	0.0	0.00	1648.99
11.100	0.00	0.0	0.0	0.0	0.00	1648.99
11.200	0.00	0.0	0.0	0.0	0.00	1648.99
11.300	0.00	0.0	0.0	0.0	0.00	1648.99
11.400	0.00	0.0	0.0	0.0	0.00	1648.99
11.500	0.00	0.0	0.0	0.0	0.00	1648.99
11.600	0.00	0.0	0.0	0.0	0.00	1648.99
11.700	0.00	0.0	0.0	0.0	0.00	1648.99
11.800	0.00	0.0	0.0	0.0	0.00	1648.99
11.900	0.00	0.0	0.0	0.0	0.00	1648.99
12.000	0.00	0.0	0.0	0.0	0.00	1648.99
12.100	0.00	0.0	0.0	0.0	0.00	1648.99
12.200	2.00	2.0	2.0	2.0	0.00	1649.66
12.300	9.00	11.0	13.0	13.0	0.00	1651.46
12.400	21.00	30.0	43.0	43.0	0.00	1652.15
12.500	39.00	60.0	103.0	103.0	0.00	1652.55
12.600	55.00	94.0	197.0	197.0	0.00	1653.14
12.700	60.00	115.0	312.0	312.0	0.00	1653.82
12.800	60.00	120.0	415.4	432.0	8.31	1654.46
12.900	54.00	114.0	510.8	529.4	9.29	1654.98
13.000	48.00	102.0	593.7	612.8	9.55	1655.41
13.100	40.00	88.0	662.1	681.7	9.82	1655.76
13.200	32.00	72.0	714.0	734.1	10.03	1656.02
13.300	27.00	59.0	752.7	773.0	10.18	1656.21
13.400	22.00	49.0	781.1	801.7	10.29	1656.35
13.500	20.00	42.0	802.4	823.1	10.37	1656.45
13.600	17.00	37.0	818.5	839.4	10.42	1656.53
13.700	16.00	33.0	830.6	851.5	10.46	1656.58
13.800	15.00	31.0	840.6	861.6	10.49	1656.63
13.900	14.00	29.0	848.6	869.6	10.51	1656.67
14.000	13.00	27.0	854.6	875.6	10.52	1656.70
14.100	12.00	25.0	858.5	879.6	10.54	1656.72
14.200	11.00	23.0	860.4	881.5	10.54	1656.72
14.300	10.00	21.0	860.3	881.4	10.54	1656.72
14.400	10.00	20.0	859.2	880.3	10.54	1656.72
14.500	9.00	19.0	857.2	878.2	10.53	1656.71
14.600	9.00	18.0	854.1	875.2	10.52	1656.70
14.700	9.00	18.0	851.1	872.1	10.51	1656.68
14.800	9.00	18.0	848.1	869.1	10.51	1656.67
14.900	9.00	18.0	845.1	866.1	10.50	1656.65
15.000	9.00	18.0	842.1	863.1	10.49	1656.64
15.100	9.00	18.0	839.2	860.1	10.48	1656.63
15.200	9.00	18.0	836.2	857.2	10.47	1656.61
15.300	8.00	17.0	832.3	853.2	10.46	1656.59
15.400	8.00	16.0	827.4	848.3	10.45	1656.57

Pond File: TMA4 .PND
Inflow Hydrograph: TMA4 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
15.500	8.00	16.0	822.5	843.4	10.43	1656.55
15.600	8.00	16.0	817.7	838.5	10.42	1656.52
15.700	7.00	15.0	811.9	832.7	10.40	1656.50
15.800	7.00	14.0	805.1	825.9	10.38	1656.46
15.900	6.00	13.0	797.4	818.1	10.35	1656.43
16.000	6.00	12.0	788.8	809.4	10.32	1656.39
16.100	6.00	12.0	780.2	800.8	10.28	1656.34
16.200	6.00	12.0	771.7	792.2	10.25	1656.30
16.300	5.00	11.0	762.3	782.7	10.21	1656.26
16.400	5.00	10.0	751.9	772.3	10.17	1656.21
16.500	5.00	10.0	741.7	761.9	10.13	1656.16
16.600	5.00	10.0	731.5	751.7	10.09	1656.11
16.700	5.00	10.0	721.4	741.5	10.05	1656.06
16.800	5.00	10.0	711.4	731.4	10.01	1656.01
16.900	5.00	10.0	701.4	721.4	9.98	1655.96
17.000	5.00	10.0	691.5	711.4	9.94	1655.91
17.100	5.00	10.0	681.7	701.5	9.90	1655.86
17.200	5.00	10.0	672.0	691.7	9.86	1655.81
17.300	5.00	10.0	662.4	682.0	9.82	1655.76
17.400	5.00	10.0	652.8	672.4	9.78	1655.71
17.500	5.00	10.0	643.4	662.8	9.74	1655.67
17.600	5.00	10.0	634.0	653.4	9.70	1655.62
17.700	5.00	10.0	624.6	644.0	9.67	1655.57
17.800	5.00	10.0	615.4	634.6	9.63	1655.52
17.900	5.00	10.0	606.2	625.4	9.59	1655.48
18.000	5.00	10.0	597.1	616.2	9.56	1655.43
18.100	5.00	10.0	588.0	607.1	9.54	1655.38
18.200	5.00	10.0	579.0	598.0	9.51	1655.34
18.300	5.00	10.0	570.0	589.0	9.48	1655.29
18.400	5.00	10.0	561.1	580.0	9.45	1655.24
18.500	4.00	9.0	551.3	570.1	9.42	1655.19
18.600	4.00	8.0	540.5	559.3	9.39	1655.13
18.700	4.00	8.0	529.8	548.5	9.35	1655.08
18.800	4.00	8.0	519.1	537.8	9.32	1655.02
18.900	4.00	8.0	508.6	527.1	9.28	1654.97
19.000	4.00	8.0	498.1	516.6	9.24	1654.91
19.100	4.00	8.0	487.7	506.1	9.19	1654.85
19.200	4.00	8.0	477.4	495.7	9.15	1654.80
19.300	4.00	8.0	467.2	485.4	9.10	1654.74
19.400	4.00	8.0	457.1	475.2	9.06	1654.69
19.500	4.00	8.0	447.1	465.1	9.01	1654.63
19.600	4.00	8.0	437.1	455.1	8.97	1654.58
19.700	4.00	8.0	427.3	445.1	8.93	1654.53
19.800	4.00	8.0	418.1	435.3	8.61	1654.47
19.900	4.00	8.0	410.5	426.1	7.77	1654.43
20.000	4.00	8.0	404.4	418.5	7.08	1654.39

Pond File: TMA4 .PND
Inflow Hydrograph: TMA4 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
20.100	4.00	8.0	399.3	412.4	6.52	1654.36
20.200	4.00	8.0	395.2	407.3	6.06	1654.33
20.300	4.00	8.0	391.8	403.2	5.69	1654.31
20.400	4.00	8.0	389.0	399.8	5.38	1654.29
20.500	4.00	8.0	386.8	397.0	5.13	1654.28
20.600	4.00	8.0	384.9	394.8	4.93	1654.27
20.700	4.00	8.0	383.4	392.9	4.76	1654.26
20.800	4.00	8.0	382.2	391.4	4.62	1654.25
20.900	4.00	8.0	381.2	390.2	4.51	1654.24
21.000	4.00	8.0	380.3	389.2	4.41	1654.24
21.100	4.00	8.0	379.7	388.3	4.34	1654.23
21.200	4.00	8.0	379.1	387.7	4.28	1654.23
21.300	4.00	8.0	378.6	387.1	4.23	1654.23
21.400	4.00	8.0	378.3	386.6	4.19	1654.23
21.500	4.00	8.0	378.0	386.3	4.15	1654.22
21.600	4.00	8.0	377.7	386.0	4.12	1654.22
21.700	4.00	8.0	377.5	385.7	4.10	1654.22
21.800	4.00	8.0	377.3	385.5	4.08	1654.22
21.900	4.00	8.0	377.2	385.3	4.07	1654.22
22.000	4.00	8.0	377.1	385.2	4.06	1654.22
22.100	4.00	8.0	377.0	385.1	4.05	1654.22
22.200	4.00	8.0	376.9	385.0	4.04	1654.22
22.300	4.00	8.0	376.9	384.9	4.03	1654.22
22.400	4.00	8.0	376.8	384.9	4.03	1654.22
22.500	4.00	8.0	376.8	384.8	4.02	1654.22
22.600	3.00	7.0	375.9	383.8	3.93	1654.21
22.700	3.00	6.0	374.4	381.9	3.76	1654.20
22.800	3.00	6.0	373.2	380.4	3.62	1654.19
22.900	3.00	6.0	372.2	379.2	3.51	1654.19
23.000	3.00	6.0	371.3	378.2	3.42	1654.18
23.100	3.00	6.0	370.6	377.3	3.34	1654.18
23.200	3.00	6.0	370.1	376.6	3.28	1654.17
23.300	3.00	6.0	369.6	376.1	3.23	1654.17
23.400	3.00	6.0	369.3	375.6	3.19	1654.17
23.500	2.00	5.0	368.1	374.3	3.06	1654.16
23.600	2.00	4.0	366.4	372.1	2.87	1654.15
23.700	2.00	4.0	365.0	370.4	2.71	1654.14
23.800	2.00	4.0	363.8	369.0	2.58	1654.14
23.900	2.00	4.0	362.9	367.8	2.48	1654.13
24.000	2.00	4.0	362.1	366.9	2.39	1654.12
24.100	2.00	4.0	361.5	366.1	2.32	1654.12
24.200	2.00	4.0	360.9	365.5	2.26	1654.12
24.300	2.00	4.0	360.5	364.9	2.21	1654.11
24.400	2.00	4.0	360.2	364.5	2.17	1654.11
24.500	2.00	4.0	359.9	364.2	2.14	1654.11
24.600	1.00	3.0	358.8	362.9	2.03	1654.10

POND-2 Version: 5.16 S/N: 1295130172
EXECUTED: 04-24-1995 06:43:52

Pond File: TMA4 .PND
Inflow Hydrograph: TMA4 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
24.700	1.00	2.0	357.1	360.8	1.84	1654.09
24.800	1.00	2.0	355.8	359.1	1.69	1654.08
24.900	1.00	2.0	354.6	357.8	1.56	1654.08
25.000	1.00	2.0	353.7	356.6	1.46	1654.07
25.100	1.00	2.0	353.0	355.7	1.38	1654.07
25.200	1.00	2.0	352.4	355.0	1.31	1654.06
25.300	1.00	2.0	351.9	354.4	1.25	1654.06
25.400	1.00	2.0	351.4	353.9	1.21	1654.06
25.500	0.00	1.0	350.3	352.4	1.08	1654.05
25.600	0.00	0.0	348.5	350.3	0.88	1654.04
25.700	0.00	0.0	347.1	348.5	0.72	1654.03
25.800	0.00	0.0	345.9	347.1	0.59	1654.02
25.900	0.00	0.0	344.9	345.9	0.48	1654.02

********** SUMMARY OF ROUTING COMPUTATIONS **********

Pond File: TMA4 .PND
Inflow Hydrograph: TMA4 .HYD
Outflow Hydrograph: OUT .HYD

Starting Pond W.S. Elevation = 1648.99 ft

***** Summary of Peak Outflow and Peak Elevation *****

Peak Inflow = 60.00 cfs
Peak Outflow = 10.54 cfs
Peak Elevation = 1656.72 ft

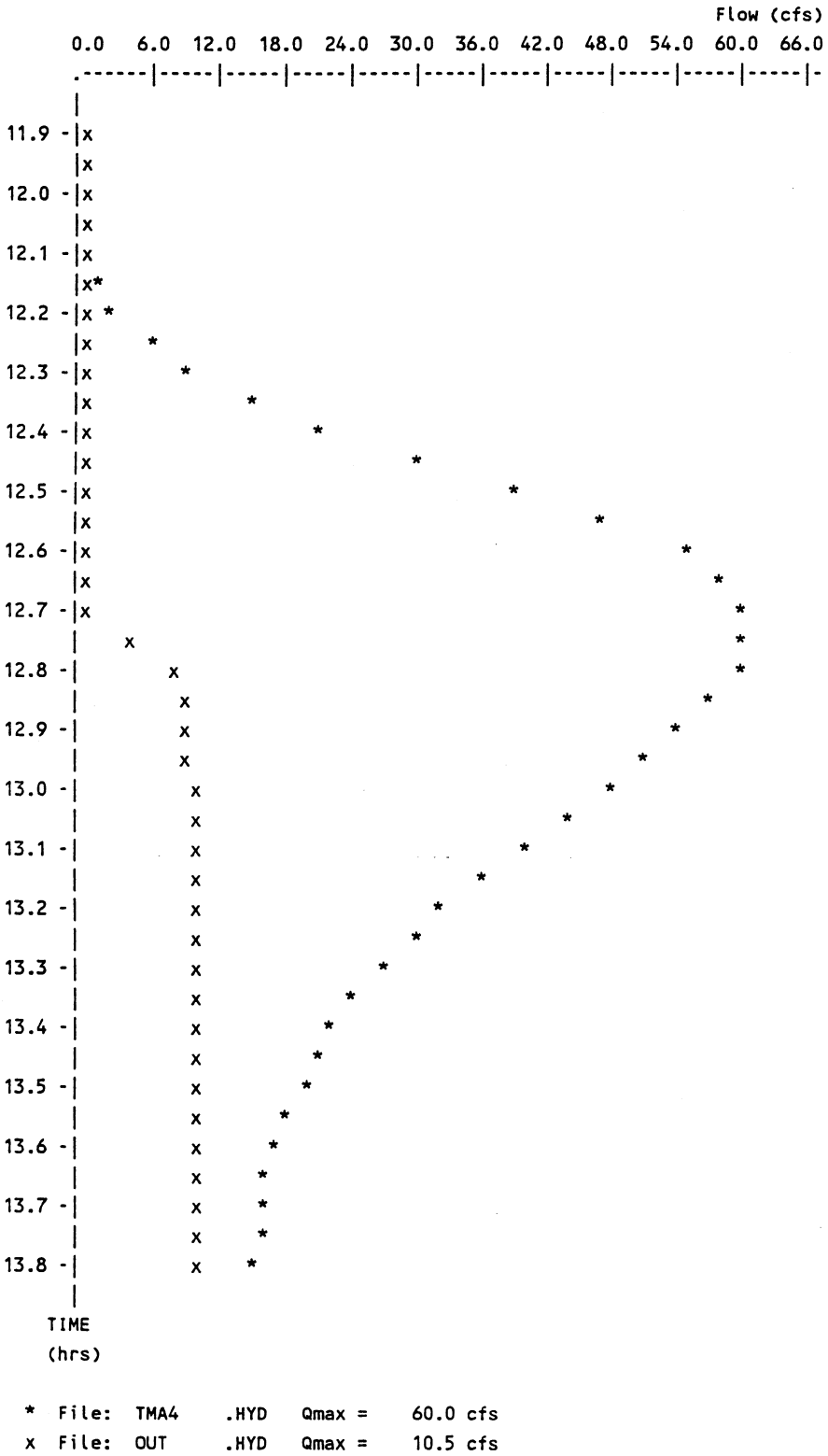
***** Summary of Approximate Peak Storage *****

Initial Storage	=	0.00 ac-ft
Peak Storage From Storm	=	3.60 ac-ft
Total Storage in Pond	=	3.60 ac-ft

Pond File: TMA4 .PND
Inflow Hydrograph: TMA4 .HYD
Outflow Hydrograph: OUT .HYD

EXECUTED: 04-24-1995
06:43:52

Peak Inflow = 60.00 cfs
Peak Outflow = 10.54 cfs
Peak Elevation = 1656.72 ft



* File: TMA4 .HYD Qmax = 60.0 cfs
x File: OUT .HYD Qmax = 10.5 cfs

CRANDON MINING COMPANY

TMA RUNOFF BASIN 12

QTR-55 HYDROLOGY
POND-2 HYDROLOGY

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:57:33
Watershed file: --> TMA5 .WSD
Hydrograph file: --> TMA5 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN 12 3/95

100 YEAR STORM EVENT

>>>> Input Parameters Used to Compute Hydrograph <<<<

Subarea Description	AREA (acres)	CN	Tc (hrs)	* Tt (hrs)	Precip. (in)	Runoff (in)	Ia/p input	Ia/p used
3-3	8.60	60.0	0.50	0.10	5.00	1.30	.27	.30
4-1B	7.46	60.0	0.50	0.10	5.00	1.30	.27	.30
4-3B	9.58	60.0	0.50	0.10	5.00	1.30	.27	.30
3-4B	12.17	60.0	0.50	0.20	5.00	1.30	.27	.30

* Travel time from subarea outfall to composite watershed outfall point.
Total area = 37.81 acres or 0.05908 sq.mi
Peak discharge = 31 cfs

>>>> Computer Modifications of Input Parameters <<<<<

Subarea Description	Input Values Tc (hr)	Input Values * Tt (hr)	Rounded Values Tc (hr)	Rounded Values * Tt (hr)	Ia/p Interpolated (Yes/No)	Ia/p Messages
3-3	0.50	0.11	0.50	0.10	No	--
4-1B	0.50	0.10	**	**	No	--
4-3B	0.50	0.13	0.50	0.10	No	--
3-4B	0.50	0.24	0.50	0.20	No	--

* Travel time from subarea outfall to composite watershed outfall point.
** Tc & Tt are available in the hydrograph tables.

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:57:33
Watershed file: --> TMA5 .WSD
Hydrograph file: --> TMA5 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 12 3/95
100 YEAR STORM EVENT

>>>> Summary of Subarea Times to Peak <<<<

Subarea	Peak Discharge at Composite Outfall (cfs)	Time to Peak at Composite Outfall (hrs)
3-3	7	12.6
4-1B	6	12.5
4-3B	8	12.6
3-4B	10	12.7
Composite Watershed	31	12.7

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:57:33
Watershed file: --> TMA5 .WSD
Hydrograph file: --> TMA5 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 12 3/95
100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	11.0 hr	11.3 hr	11.6 hr	11.9 hr	12.0 hr	12.1 hr	12.2 hr	12.3 hr	12.4 hr
3-3	0	0	0	0	0	0	1	2	4
4-1B	0	0	0	0	0	0	1	2	4
4-3B	0	0	0	0	0	0	1	2	5
3-4B	0	0	0	0	0	0	1	2	5
Total (cfs)	0	0	0	0	0	0	4	8	18

Subarea Description	12.5 hr	12.6 hr	12.7 hr	12.8 hr	13.0 hr	13.2 hr	13.4 hr	13.6 hr	13.8 hr
3-3	6	7	7	6	4	3	2	2	1
4-1B	6	6	6	5	3	2	2	1	1
4-3B	7	8	8	6	4	3	2	2	2
3-4B	8	9	10	9	6	4	3	2	2
Total (cfs)	27	30	31	26	17	12	9	7	6

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:57:33
Watershed file: --> TMA5 .WSD
Hydrograph file: --> TMA5 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 12 3/95
100 YEAR STORM EVENT

Composite Hydrograph Summary (cfs)

Subarea Description	14.0 hr	14.3 hr	14.6 hr	15.0 hr	15.5 hr	16.0 hr	16.5 hr	17.0 hr	17.5 hr
3-3	1	1	1	1	1	1	1	1	1
4-1B	1	1	1	1	1	1	1	0	0
4-3B	1	1	1	1	1	1	1	1	1
3-4B	2	2	1	1	1	1	1	1	1
Total (cfs)	5	5	4	4	4	4	4	3	3

Subarea Description	18.0 hr	19.0 hr	20.0 hr	22.0 hr	26.0 hr
3-3	1	0	0	0	0
4-1B	0	0	0	0	0
4-3B	1	1	0	0	0
3-4B	1	1	1	0	0
Total (cfs)	3	2	1	0	0

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:57:33
Watershed file: --> TMA5 .WSD
Hydrograph file: --> TMA5 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27

BASIN 12 3/95

100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
11.0	0	14.8	4
11.1	0	14.9	4
11.2	0	15.0	4
11.3	0	15.1	4
11.4	0	15.2	4
11.5	0	15.3	4
11.6	0	15.4	4
11.7	0	15.5	4
11.8	0	15.6	4
11.9	0	15.7	4
12.0	0	15.8	4
12.1	0	15.9	4
12.2	4	16.0	4
12.3	8	16.1	4
12.4	18	16.2	4
12.5	27	16.3	4
12.6	30	16.4	4
12.7	31	16.5	4
12.8	26	16.6	4
12.9	22	16.7	4
13.0	17	16.8	3
13.1	14	16.9	3
13.2	12	17.0	3
13.3	10	17.1	3
13.4	9	17.2	3
13.5	8	17.3	3
13.6	7	17.4	3
13.7	6	17.5	3
13.8	6	17.6	3
13.9	6	17.7	3
14.0	5	17.8	3
14.1	5	17.9	3
14.2	5	18.0	3
14.3	5	18.1	3
14.4	5	18.2	3
14.5	4	18.3	3
14.6	4	18.4	3

TR-55 TABULAR HYDROGRAPH METHOD
Type II Distribution
(24 hr. Duration Storm)

Executed: 04-13-1995 14:57:33
Watershed file: --> TMA5 .WSD
Hydrograph file: --> TMA5 .HYD

CRANDON MINING COMPANY
FOTH AND VAN DYKE 93C049.27
BASIN 12 3/95
100 YEAR STORM EVENT

Time (hrs)	Flow (cfs)	Time (hrs)	Flow (cfs)
18.6	2	22.4	0
18.7	2	22.5	0
18.8	2	22.6	0
18.9	2	22.7	0
19.0	2	22.8	0
19.1	2	22.9	0
19.2	2	23.0	0
19.3	2	23.1	0
19.4	2	23.2	0
19.5	2	23.3	0
19.6	1	23.4	0
19.7	1	23.5	0
19.8	1	23.6	0
19.9	1	23.7	0
20.0	1	23.8	0
20.1	1	23.9	0
20.2	1	24.0	0
20.3	1	24.1	0
20.4	1	24.2	0
20.5	1	24.3	0
20.6	1	24.4	0
20.7	1	24.5	0
20.8	1	24.6	0
20.9	1	24.7	0
21.0	0	24.8	0
21.1	0	24.9	0
21.2	0	25.0	0
21.3	0	25.1	0
21.4	0	25.2	0
21.5	0	25.3	0
21.6	0	25.4	0
21.7	0	25.5	0
21.8	0	25.6	0
21.9	0	25.7	0
22.0	0	25.8	0
22.1	0	25.9	0
22.2	0		
22.3	0		

```
**********************************************************************
*                                                                    *
* CRANDON MINING COMPANY          93C049.27      FOTH & VAN DYKE     *
*      TMA Runoff Basin 12                            3/95           *
*                                                                    *
*                                                                    *
*                                                                    *
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Inflow Hydrograph: TMA5 .HYD
Rating Table file: TMA5 .PND

----INITIAL CONDITIONS----
Elevation = 1696.00 ft
Outflow = 0.00 cfs
Storage = 0.00 ac-ft

GIVEN POND DATA			INTERMEDIATE ROUTING COMPUTATIONS	
ELEVATION (ft)	OUTFLOW (cfs)	STORAGE (ac-ft)	2S/t (cfs)	2S/t + O (cfs)
1696.00	0.0	0.000	0.0	0.0
1696.20	0.0	0.002	0.5	0.5
1696.40	0.0	0.004	1.1	1.1
1696.60	0.0	0.007	1.7	1.7
1696.80	0.0	0.010	2.5	2.5
1697.00	0.0	0.014	3.3	3.3
1697.20	0.0	0.018	4.3	4.3
1697.40	0.0	0.022	5.3	5.3
1697.60	0.0	0.027	6.5	6.5
1697.80	0.0	0.032	7.8	7.8
1698.00	0.0	0.038	9.2	9.2
1698.20	0.0	0.044	10.7	10.7
1698.40	0.0	0.051	12.4	12.4
1698.60	0.0	0.059	14.3	14.3
1698.80	0.0	0.067	16.3	16.3
1699.00	0.0	0.078	18.9	18.9
1699.20	0.0	0.197	47.7	47.7
1699.40	0.0	0.319	77.1	77.1
1699.60	0.0	0.443	107.2	107.2
1699.80	0.0	0.570	137.9	137.9
1700.00	0.0	0.700	169.3	169.3
1700.20	0.0	0.832	201.4	201.4
1700.40	0.0	0.968	234.1	234.1
1700.60	8.1	1.106	267.6	275.7
1700.80	8.3	1.247	301.7	310.0
1701.00	8.5	1.391	336.6	345.1
1701.20	8.6	1.538	372.1	380.7
1701.40	8.8	1.688	408.4	417.2
1701.60	8.9	1.840	445.4	454.3
1701.80	9.1	1.996	483.1	492.2
1702.00	9.3	2.155	521.6	530.9

GIVEN POND DATA			INTERMEDIATE ROUTING COMPUTATIONS	
ELEVATION (ft)	OUTFLOW (cfs)	STORAGE (ac-ft)	2S/t (cfs)	2S/t + O (cfs)
1702.20	9.4	2.317	560.8	570.2
1702.40	9.6	2.483	600.8	610.4
1702.60	10.7	2.651	641.5	652.2
1702.80	15.2	2.822	683.0	698.2

Time increment (t) = 0.100 hrs.

Pond File: TMA5 .PND
Inflow Hydrograph: TMA5 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
11.000	0.00	-----	0.0	0.0	0.00	1696.00
11.100	0.00	0.0	0.0	0.0	0.00	1696.00
11.200	0.00	0.0	0.0	0.0	0.00	1696.00
11.300	0.00	0.0	0.0	0.0	0.00	1696.00
11.400	0.00	0.0	0.0	0.0	0.00	1696.00
11.500	0.00	0.0	0.0	0.0	0.00	1696.00
11.600	0.00	0.0	0.0	0.0	0.00	1696.00
11.700	0.00	0.0	0.0	0.0	0.00	1696.00
11.800	0.00	0.0	0.0	0.0	0.00	1696.00
11.900	0.00	0.0	0.0	0.0	0.00	1696.00
12.000	0.00	0.0	0.0	0.0	0.00	1696.00
12.100	0.00	0.0	0.0	0.0	0.00	1696.00
12.200	4.00	4.0	4.0	4.0	0.00	1697.15
12.300	8.00	12.0	16.0	16.0	0.00	1698.78
12.400	18.00	26.0	42.0	42.0	0.00	1699.16
12.500	27.00	45.0	87.0	87.0	0.00	1699.47
12.600	30.00	57.0	144.0	144.0	0.00	1699.84
12.700	31.00	61.0	205.0	205.0	0.00	1700.22
12.800	26.00	57.0	251.1	262.0	5.44	1700.53
12.900	22.00	48.0	282.7	299.1	8.24	1700.74
13.000	17.00	39.0	304.9	321.7	8.37	1700.87
13.100	14.00	31.0	319.0	335.9	8.45	1700.95
13.200	12.00	26.0	328.0	345.0	8.50	1701.00
13.300	10.00	22.0	333.0	350.0	8.51	1701.03
13.400	9.00	19.0	335.0	352.0	8.52	1701.04
13.500	8.00	17.0	334.9	352.0	8.52	1701.04
13.600	7.00	15.0	332.9	349.9	8.51	1701.03
13.700	6.00	13.0	328.9	345.9	8.50	1701.00
13.800	6.00	12.0	324.0	340.9	8.48	1700.98
13.900	6.00	12.0	319.1	336.0	8.45	1700.95
14.000	5.00	11.0	313.2	330.1	8.41	1700.91
14.100	5.00	10.0	306.5	323.2	8.38	1700.88
14.200	5.00	10.0	299.8	316.5	8.34	1700.84
14.300	5.00	10.0	293.2	309.8	8.30	1700.80
14.400	5.00	10.0	286.7	303.2	8.26	1700.76
14.500	4.00	9.0	279.3	295.7	8.22	1700.72
14.600	4.00	8.0	270.9	287.3	8.17	1700.67
14.700	4.00	8.0	262.7	278.9	8.12	1700.62
14.800	4.00	8.0	256.4	270.7	7.13	1700.58
14.900	4.00	8.0	252.6	264.4	5.91	1700.55
15.000	4.00	8.0	250.3	260.6	5.16	1700.53
15.100	4.00	8.0	248.9	258.3	4.71	1700.52
15.200	4.00	8.0	248.0	256.9	4.43	1700.51
15.300	4.00	8.0	247.5	256.0	4.26	1700.51
15.400	4.00	8.0	247.1	255.5	4.16	1700.50

Pond File: TMA5 .PND
Inflow Hydrograph: TMA5 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
15.500	4.00	8.0	247.0	255.1	4.10	1700.50
15.600	4.00	8.0	246.8	255.0	4.06	1700.50
15.700	4.00	8.0	246.8	254.8	4.04	1700.50
15.800	4.00	8.0	246.7	254.8	4.02	1700.50
15.900	4.00	8.0	246.7	254.7	4.01	1700.50
16.000	4.00	8.0	246.7	254.7	4.01	1700.50
16.100	4.00	8.0	246.7	254.7	4.01	1700.50
16.200	4.00	8.0	246.7	254.7	4.00	1700.50
16.300	4.00	8.0	246.7	254.7	4.00	1700.50
16.400	4.00	8.0	246.6	254.7	4.00	1700.50
16.500	4.00	8.0	246.6	254.6	4.00	1700.50
16.600	4.00	8.0	246.6	254.6	4.00	1700.50
16.700	4.00	8.0	246.6	254.6	4.00	1700.50
16.800	3.00	7.0	246.0	253.6	3.81	1700.49
16.900	3.00	6.0	245.1	252.0	3.49	1700.49
17.000	3.00	6.0	244.5	251.1	3.30	1700.48
17.100	3.00	6.0	244.1	250.5	3.18	1700.48
17.200	3.00	6.0	243.9	250.1	3.11	1700.48
17.300	3.00	6.0	243.7	249.9	3.07	1700.48
17.400	3.00	6.0	243.6	249.7	3.04	1700.48
17.500	3.00	6.0	243.6	249.6	3.03	1700.47
17.600	3.00	6.0	243.6	249.6	3.02	1700.47
17.700	3.00	6.0	243.5	249.6	3.01	1700.47
17.800	3.00	6.0	243.5	249.5	3.01	1700.47
17.900	3.00	6.0	243.5	249.5	3.00	1700.47
18.000	3.00	6.0	243.5	249.5	3.00	1700.47
18.100	3.00	6.0	243.5	249.5	3.00	1700.47
18.200	3.00	6.0	243.5	249.5	3.00	1700.47
18.300	3.00	6.0	243.5	249.5	3.00	1700.47
18.400	3.00	6.0	243.5	249.5	3.00	1700.47
18.500	2.00	5.0	242.9	248.5	2.81	1700.47
18.600	2.00	4.0	241.9	246.9	2.49	1700.46
18.700	2.00	4.0	241.3	245.9	2.30	1700.46
18.800	2.00	4.0	241.0	245.3	2.18	1700.45
18.900	2.00	4.0	240.7	245.0	2.11	1700.45
19.000	2.00	4.0	240.6	244.7	2.07	1700.45
19.100	2.00	4.0	240.5	244.6	2.04	1700.45
19.200	2.00	4.0	240.5	244.5	2.03	1700.45
19.300	2.00	4.0	240.4	244.5	2.02	1700.45
19.400	2.00	4.0	240.4	244.4	2.01	1700.45
19.500	2.00	4.0	240.4	244.4	2.01	1700.45
19.600	1.00	3.0	239.8	243.4	1.81	1700.44
19.700	1.00	2.0	238.8	241.8	1.49	1700.44
19.800	1.00	2.0	238.2	240.8	1.30	1700.43
19.900	1.00	2.0	237.8	240.2	1.18	1700.43
20.000	1.00	2.0	237.6	239.8	1.11	1700.43

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Pond File: TMA5 .PND
Inflow Hydrograph: TMA5 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH		ROUTING COMPUTATIONS				
TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
20.100	1.00	2.0	237.5	239.6	1.07	1700.43
20.200	1.00	2.0	237.4	239.5	1.04	1700.43
20.300	1.00	2.0	237.3	239.4	1.03	1700.43
20.400	1.00	2.0	237.3	239.3	1.02	1700.43
20.500	1.00	2.0	237.3	239.3	1.01	1700.42
20.600	1.00	2.0	237.3	239.3	1.01	1700.42
20.700	1.00	2.0	237.3	239.3	1.00	1700.42
20.800	1.00	2.0	237.3	239.3	1.00	1700.42
20.900	1.00	2.0	237.3	239.3	1.00	1700.42
21.000	0.00	1.0	236.7	238.3	0.81	1700.42
21.100	0.00	0.0	235.7	236.7	0.49	1700.41
21.200	0.00	0.0	235.1	235.7	0.30	1700.41
21.300	0.00	0.0	234.7	235.1	0.18	1700.40
21.400	0.00	0.0	234.5	234.7	0.11	1700.40
21.500	0.00	0.0	234.3	234.5	0.07	1700.40
21.600	0.00	0.0	234.3	234.3	0.04	1700.40
21.700	0.00	0.0	234.2	234.3	0.03	1700.40
21.800	0.00	0.0	234.2	234.2	0.02	1700.40
21.900	0.00	0.0	234.2	234.2	0.01	1700.40
22.000	0.00	0.0	234.2	234.2	0.01	1700.40
22.100	0.00	0.0	234.1	234.2	0.00	1700.40
22.200	0.00	0.0	234.1	234.1	0.00	1700.40
22.300	0.00	0.0	234.1	234.1	0.00	1700.40
22.400	0.00	0.0	234.1	234.1	0.00	1700.40
22.500	0.00	0.0	234.1	234.1	0.00	1700.40
22.600	0.00	0.0	234.1	234.1	0.00	1700.40
22.700	0.00	0.0	234.1	234.1	0.00	1700.40
22.800	0.00	0.0	234.1	234.1	0.00	1700.40
22.900	0.00	0.0	234.1	234.1	0.00	1700.40
23.000	0.00	0.0	234.1	234.1	0.00	1700.40
23.100	0.00	0.0	234.1	234.1	0.00	1700.40
23.200	0.00	0.0	234.1	234.1	0.00	1700.40
23.300	0.00	0.0	234.1	234.1	0.00	1700.40
23.400	0.00	0.0	234.1	234.1	0.00	1700.40
23.500	0.00	0.0	234.1	234.1	0.00	1700.40
23.600	0.00	0.0	234.1	234.1	0.00	1700.40
23.700	0.00	0.0	234.1	234.1	0.00	1700.40
23.800	0.00	0.0	234.1	234.1	0.00	1700.40
23.900	0.00	0.0	234.1	234.1	0.00	1700.40
24.000	0.00	0.0	234.1	234.1	0.00	1700.40
24.100	0.00	0.0	234.1	234.1	0.00	1700.40
24.200	0.00	0.0	234.1	234.1	0.00	1700.40
24.300	0.00	0.0	234.1	234.1	0.00	1700.40
24.400	0.00	0.0	234.1	234.1	0.00	1700.40
24.500	0.00	0.0	234.1	234.1	0.00	1700.40
24.600	0.00	0.0	234.1	234.1	0.00	1700.40

Pond File: TMA5 .PND
Inflow Hydrograph: TMA5 .HYD
Outflow Hydrograph: OUT .HYD

INFLOW HYDROGRAPH | ROUTING COMPUTATIONS

TIME (hrs)	INFLOW (cfs)	I1+I2 (cfs)	2S/t - O (cfs)	2S/t + O (cfs)	OUTFLOW (cfs)	ELEVATION (ft)
24.700	0.00	0.0	234.1	234.1	0.00	1700.40
24.800	0.00	0.0	234.1	234.1	0.00	1700.40
24.900	0.00	0.0	234.1	234.1	0.00	1700.40
25.000	0.00	0.0	234.1	234.1	0.00	1700.40
25.100	0.00	0.0	234.1	234.1	0.00	1700.40
25.200	0.00	0.0	234.1	234.1	0.00	1700.40
25.300	0.00	0.0	234.1	234.1	0.00	1700.40
25.400	0.00	0.0	234.1	234.1	0.00	1700.40
25.500	0.00	0.0	234.1	234.1	0.00	1700.40
25.600	0.00	0.0	234.1	234.1	0.00	1700.40
25.700	0.00	0.0	234.1	234.1	0.00	1700.40
25.800	0.00	0.0	234.1	234.1	0.00	1700.40
25.900	0.00	0.0	234.1	234.1	0.00	1700.40

********** SUMMARY OF ROUTING COMPUTATIONS **********

Pond File: TMA5 .PND
Inflow Hydrograph: TMA5 .HYD
Outflow Hydrograph: OUT .HYD

Starting Pond W.S. Elevation = 1696.00 ft

***** Summary of Peak Outflow and Peak Elevation *****

Peak Inflow = 31.00 cfs
Peak Outflow = 8.52 cfs
Peak Elevation = 1701.04 ft

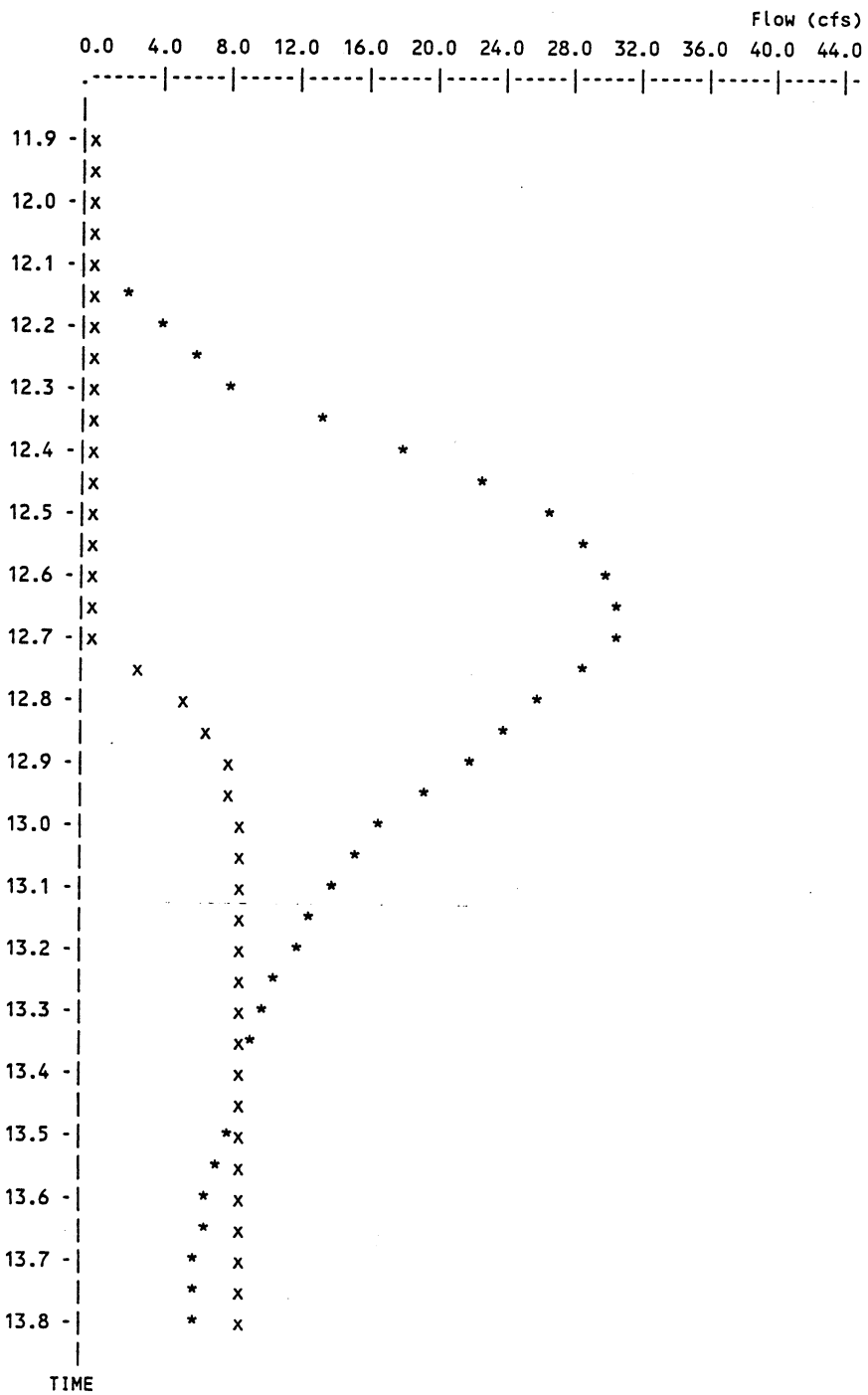
***** Summary of Approximate Peak Storage *****

Initial Storage	=	0.00 ac-ft
Peak Storage From Storm	=	1.42 ac-ft
Total Storage in Pond	=	1.42 ac-ft

Pond File: TMA5 .PND
Inflow Hydrograph: TMA5 .HYD
Outflow Hydrograph: OUT .HYD

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Peak Inflow = 31.00 cfs
Peak Outflow = 8.52 cfs
Peak Elevation = 1701.04 ft



* File: TMA5 .HYD Qmax = 31.0 cfs
x File: OUT .HYD Qmax = 8.5 cfs

Appendix L

Revised Universal Soil Loss Equation (RUSLE) for Final Cover and Exterior Berm Slopes

Foth & Van Dyke

Client: CMC Scope I.D.: 93C019
Project: Soil Loss Page: 1/12
Prepared by: MDF Date: 4-21-95
Checked by: JBK Date: 5-1-95

Purpose: To estimate soil loss from the exterior slopes of TMA

Scope: Use Revised Universal Soil Loss Equation for estimating annual soil loss

CALCULATIONS:

Annual Soil Loss (Tons/Acre) = R K LS C P

R = 95 (See p. 2)
LS = 6.495; 7.885, 4.911 for different scenarios (see p3 thru 7)

K = 0.24 (See p 8)

C = 0.02 (See p 11)

P = 1 (See p 12)

SOIL LOSS

A = (95)(.24)(6.495)(.02)(1.0) = .196 ton/Acre/yr for LS = 6.495

A = (95)(.24)(7.885)(.02)(1.0) = 3.60 ton/Acre/yr for LS = 7.885

A = (95)(.24)(4.911)(.02)(1.0) = 2.24 ton/Acre/yr for LS = 4.911

CONCLUSION: The above soil loss values are acceptable. Note that Mulching @ 4t/acre is required

p. 2/9

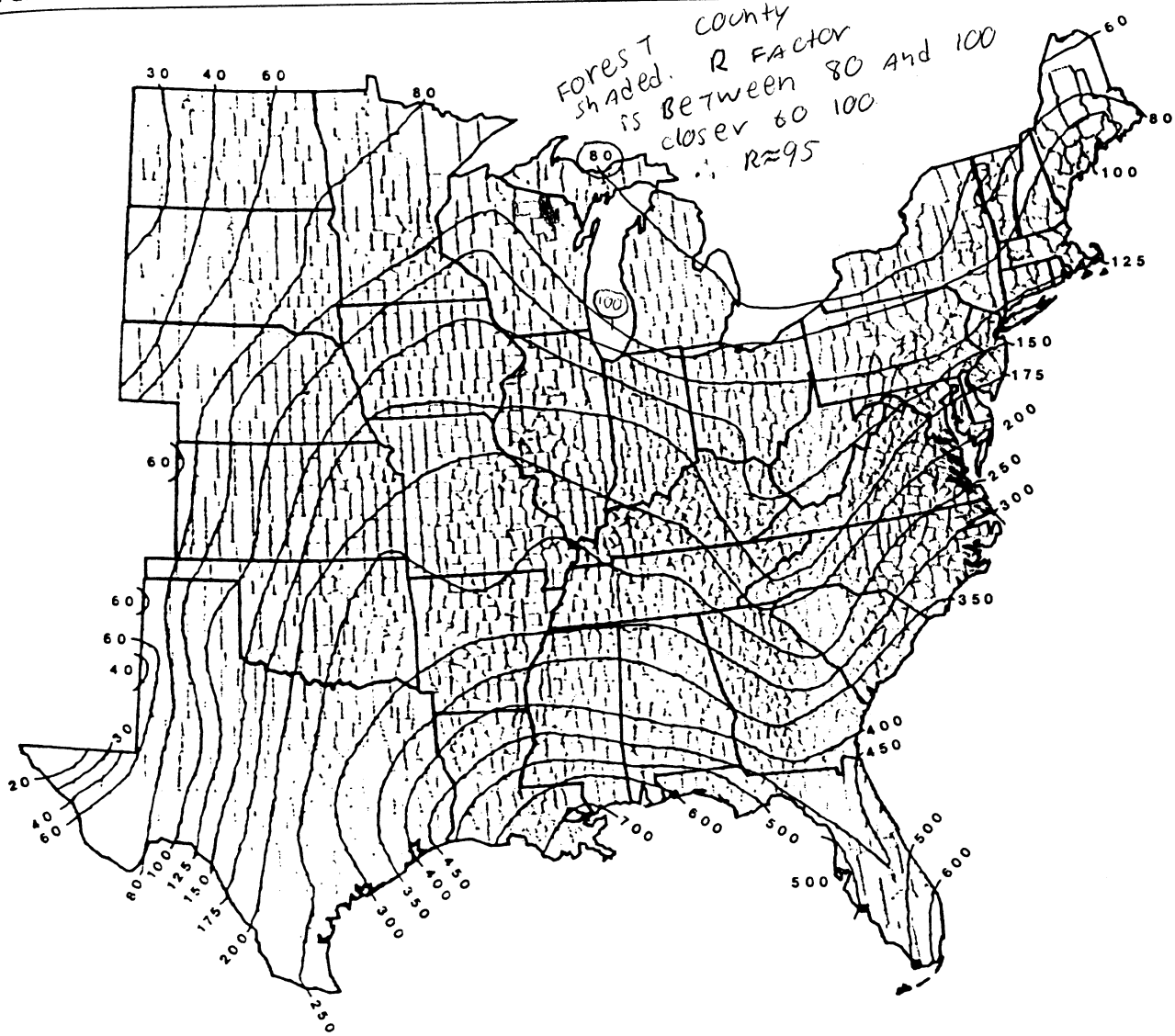


Figure 8.6 Isolines of annual R factor for the Eastern United States (after Renard *et al.*, 1993b). Maps for areas in and west of the Rockies are in Appendix 8A. R factor is in ft · tonsf · in./acre · hr · year. To convert to SI units, MJ · mm ha · h · y. multiply by 17.02.

Units on the annual R factor are hundreds of ft · tonsf · in./acre · hr · year. The hundreds term comes from the 100 divisor in Eq. (8.31). In further reference, the units on R will be given as ft · tonsf · in./acre · hr · year for simplicity and the hundreds will be understood. The units on single storm erosivity would be ft · tonsf · in./acre · hr · storm.

To convert the annual R to SI units, MJ · mm/ha · hr · year, multiply the English Units by 17.02. A summary of other conversion factors is given in the Appendix. As a note ft · tonsf · in./acre · hr · year means

$$\frac{\text{ft tonsf in.}}{\text{acre hr year}}.$$

Average Annual R Values

Using Eq. (8.31), Wischmeier and Smith (1965) computed values for the R factor for stations east of the Rockies and extended the values to the Western States utilizing a correlation between 2-year, 6-hr rainfall and the R factor (Wischmeier and Smith, 1978). In the development of RUSLE, additional analyses were made on longer data bases and on hourly rainfall for stations in and west of the Rockies. The data are summarized in Fig. 8.6 for the eastern United States.

Corrections for Ponding

Studies have shown that ponded water on flat slopes tends to absorb rainfall energy and retard erosion

Foth & Van Dyke

Client: CMC — Scope I.D.: 93C049
Project: estimate LS factor — Page: 3/10
Prepared by: MDF — Date: 4/21/95
Checked by: [signature] — Date: 5/1/95

DETERMINE LS

purpose: solve for the LS factor as shown in Design Hydrology and sedimentology for small catchments pg 263. using REVISED UNIVERSAL Soil Loss Equation (RUSLE)

assumptions: convex slope in two sections. section 1: 1400 ft at 2% slope section 2: 320 ft at 33⅓% slope

Section slope segment	slope %	M slope expon*	θ°	uniform slope LS factor**	SAF***	segment LS FACTOR (col 5 * col 6)
1	2.0	0.14	1.15	0.373	.906	.338
2	33.3	0.50	18.42	9.808	1.29	12.652
					Σ	12.99

* TABLE 8.6 - These values are not applicable to Thawing soils. (to account for thawing use a slope length exponent of 0.5 for All slopes) (see page 4);
** see page
*** see page 6

M for slope = 33%

$$\frac{0.52 - 0.49}{40 - 30} = \frac{0.52 - x}{40 - 33.3}$$

Interpolate 33.3% value slope exponent

$$\frac{.03}{10} = \frac{0.52 - x}{6.7}$$

$$0.201 = 5.2 - 10x$$

$$-4.999 = -10x$$

$$x = 0.4999 \approx 0.50$$ ✓

Solve for θ°

$\theta = \tan^{-1}(.02) = 1.1457^\circ \approx 1.15^\circ$

$\theta = \tan^{-1}(.333) = 18.4178^\circ \approx 18.42^\circ$ ✓

Table 8.6 Slope Length Exponent m in Eq. (8.43) (after McCool *et al.*, 1993)[a]

Percentage slope	Rill/interrill ratio		
	Low[b]	Moderate[c]	High[d]
0.2	0.02	0.04	0.07
0.5	0.04	0.08	0.16
1.0	0.08	0.15	0.26
2.0	0.14	0.24	0.39
3.0	0.18	0.31	0.47
4.0	0.22	0.36	0.53
5.0	0.25	0.40	0.57
6.0	0.28	0.43	0.60
8.0	0.32	0.48	0.65
10.0	0.35	0.52	0.68
12.0	0.37	0.55	0.71
14.0	0.40	0.57	0.72
16.0	0.41	0.59	0.74
20.0	0.44	0.61	0.76
25.0	0.47	0.64	0.78
30.0	0.49	0.66	0.79
40.0	0.52	0.68	0.81
50.0	0.54	0.70	0.82
60.0	0.55	0.71	0.83

[a] Values in table are not applicable to thawing soils. See text for explanation.
[b] $\beta = 1/2$ value from Eq. (8.45) in Eq. (8.44).
[c] $\beta = 1 \times$ value from Eq. (8.45) in Eq. (8.44).
[d] $\beta = 2 \times$ value from Eq. (8.45) in Eq. (8.44).

would be $A_i(\lambda_i - \lambda_{i-1})$, and the average erosion per unit area over the entire slope length would be

$$A = R \sum_{i=1}^{n} K_i C_i P_i S_i \frac{\left[\lambda_i^{m+1} - \lambda_{i-1}^{m+1}\right]}{\lambda_e 72.6^m}, \qquad (8.47)$$

where λ_e is the total slope length. Equation (8.47) can also be used to evaluate the effects of variation in K, C, and P over the slope length.

An alternate method for evaluating irregular slopes is the use of a slope length adjustment factor (SAF). If the slope is divided into n increments of equal length ΔX, then

$$A = R \sum_{i=1}^{n} K_i C_i P_i S_i \frac{\left[(i\,\Delta X)^{m+1} - ([i-1]\,\Delta X)^{m+1}\right]}{n\,\Delta X 72.6^m}. \qquad (8.48)$$

Dividing by n times the soil loss from a uniform slope of equal length and assuming constant values of K_i C_i P_i along the slope, a slope adjustment factor can be developed for each segment, or

$$\mathrm{SAF}_i = \frac{A_i}{A} = \frac{i^{m+1} - (i-1)^{m+1}}{n^m}, \qquad (8.49)$$

where n is the number of segments and SAF is the slope adjustment factor. The sum of the SAF_i for a given slope is equal to the number of segments n; thus the average erosion over the slope is

$$A = \frac{R}{n} \sum_{i=1}^{n} K_i C_i P_i S_i L_i (\mathrm{SAF})_i. \qquad (8.50a)$$

where L_i is the slope length factor calculated from Eq. (8.43) using the m value corresponding to the segment steepness. In the development of a SAF relationship, R, K, C, and P remain constant over all segments; thus Eq. (8.50a) can be solved for an equivalent LS factor

$$LS = \frac{1}{n} \sum_{i=1}^{n} S_i L_i (\mathrm{SAF})_i. \qquad (8.50b)$$

Factors calculated from Eq. (8.50b) are given in Table 8.7. An example of its use is given in Example Problem 8.5.

Example Problem 8.5. Estimating *LS* factors

A soil that is very susceptible to rilling has a slope length of 210 ft and an average slope of 15%. Estimate the LS factor if:

(1) the slope is uniform
(2) the slope is convex with slopes of 10, 15, and 20% on segments 1, 2, and 3
(3) the slope is concave with slopes of 20, 15, and 10% on segments 1, 2, and 3.

Assume that the soil is not freezing and thawing.

Solution:

1. Uniform slope. The slope angle is

$$\theta = \tan^{-1} 0.15 = 8.53°.$$

From Eq. (8.45) for soils moderately susceptible to rilling,

$$\beta = \frac{11.16 \sin 8.53}{3.0(\sin 8.53)^{0.8} + 0.56} = 1.37.$$

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: LS CALCULATION	Page: 5/10
	Prepared by: MDF	Date: 4/21/95
	Checked by: [illegible] Kissel	Date: 5/1/95

** solve for uniform slope LS factor

EQ(8.40) * EQ(8.43) (From p. 6) EQ 8.40 (see Attachment A)

$\sin\theta = \sin 1.15^\circ = 0.02 < 0.09$ (use EQ 8.39)

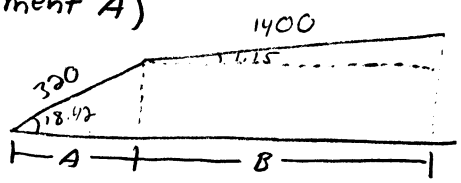
$\sin 18.42^\circ = 0.32 \geq 0.09$ (use EQ 8.40)

$S_1 = 10.8 \sin(1.15) + 0.03 = 0.247$ ✓

$S_2 = 16.8 \sin(18.42) - 0.50 = 4.808$ ✓

use EQ 8.43 (see attachment A)

$L = \left(\frac{\lambda}{72.6}\right)^m$ $\lambda =$



$\cos 18.42 = \frac{A}{320}$ $A = 303.61$ ft ✓

$\cos 1.15 = \frac{B}{1400}$ $B = 1399.72$ ft ✓

λ = horizontal projection of length = A + B

m = slope length exponent (already chosen using low rill/interrill ratio)

$L_2 = \left(\frac{303.61}{72.6}\right)^{0.50} = 2.04$ ✓

$L_1 = \left(\frac{1399.72}{72.6}\right)^{0.14} = 1.51$ ✓

$L_1 * S_1 = (1.51)(0.247) = 0.373 = (LS)_1$ ✓

$L_2 * S_2 = (2.04)(4.805) = 9.808 = (LS)_2$ ✓

Table 8.7 Adjustment Factors to LS to Estimate Soil Loss on Irregular Slopes (after McCool *et al.*, 1993)[a]

Number of segments	Sequential number of segments	Slope length exponent								
		0.05	0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8
$n=$ 2	$i=$ 1	0.97	0.93	0.87	0.81	0.76	0.71	0.66	0.62	0.57
	2	1.03	1.07	1.13	1.19	1.24	1.29	1.34	1.38	1.43
3	1	0.95	0.90	0.80	0.72	0.64	0.58	0.52	0.46	0.42
	2	1.01	1.02	1.04	1.05	1.06	1.05	1.05	1.04	1.03
	3	1.04	1.08	1.16	1.23	1.30	1.37	1.43	1.50	1.55
4	1	0.93	0.87	0.76	0.66	0.57	0.50	0.44	0.38	0.33
	2	1.00	1.00	0.98	0.96	0.94	0.92	0.88	0.85	0.82
	3	1.03	1.05	1.09	1.13	1.16	1.18	1.20	1.22	1.23
	4	1.04	1.08	1.17	1.25	1.33	1.40	1.48	1.55	1.62
5	1	0.92	0.85	0.73	0.62	0.53	0.45	0.38	0.32	0.28
	2	0.99	0.97	0.94	0.90	0.86	0.82	0.77	0.73	0.69
	3	1.01	1.03	1.04	1.05	1.06	1.06	1.06	1.05	1.03
	4	1.03	1.06	1.12	1.17	1.21	1.25	1.29	1.32	1.35
	5	1.05	1.09	1.17	1.26	1.34	1.42	1.50	1.58	1.65

[a]Soil loss factors = $[i^{1+m} - (i-1)^{1+m}]/n^m$, where i is the sequential number of segment, m is the slope length exponent, and n is the number of segments. Values are forced to give a factor total equal to n.

2. *Convex slope.* Values are tabulated below.

Slope segment	Slope percentage	Slope expon. Table 8.6	θ (°)	Uniform slope *LS* factor[a]	*SAF*, Table 8.7	Segment *LS* factor (Column 5 × Column 6)
1	10	0.68	5.71	2.41	0.47	1.13
2	15	0.73	8.53	4.33	1.04	4.50
3	20	0.76	11.31	6.26	1.53	9.58
						Σ=15.21

[a]Equation (8.40) x Eq. (8.43) with λ=210 ft.

From Eq. (8.50b)

$$LS = \Sigma/n = 15.21/3 = 5.07.$$

3. *Concave slope.* Values are tabulated below.

Slope segment	Slope percentage	Slope expon. Table 8.6	θ (°)	Uniform slope *LS* factor[a]	*SAF*, Table 8.7	Segment *LS* factor (Column 5 × Column 6)
1	20	0.76	11.31	6.26	0.44	2.75
2	15	0.73	8.53	4.33	1.04	4.50
3	10	0.68	5.71	2.41	1.49	3.59
						Σ=10.84

[a]Equation (8.40) x Eq. (8.43) with λ=210 ft.

Foth & Van Dyke

Client: CMC — Scope I.D.: 93C049
Project: LS CALCULATION — Page: 7/10
Prepared by: MDF — Date: 4/21/95
Checked by: C.B. Klasek — Date: 5/1/95

Interpolating SAF values from Table 8.7, (2 segments). (Pg C 6)

Slope exponent = 0.14

0.10	0.93
0.14	x
0.20	0.87

$$\frac{0.93-0.87}{.10-.20} = \frac{0.93-x}{.10-.14}$$

$$\frac{0.06}{-0.1} = \frac{0.93-x}{-0.04}$$

$$-.0024 = -.093 + .1x$$

$$.0906 = .1x$$

SAF For $m=0.14 = x = .906$ ✓ $i=1$

Slope exponent = 0.50

SAF From table = 1.29 ✓ $i=2$

Segment LS Factor

= Uniform slope LS Factor * SAF

= (.373)(.906) = .338 ✓ $i=1$

= (9.808)(1.29) = 12.652 ✓ $i=2$

LS calculation

LS = Σ segments / n n = 2

= (.338 + 12.652) / 2

= 12.99 / 2

= 6.495 ✓

Conclusion
The LS Factor is 6.495

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: LS CALCS Page: 8/10
Prepared by: MDF Date: 4/21/95
Checked by: CHECKED Date: 5/1/95

CALCULATION ACCOUNTING FOR THAWING SOILS

segment	slope	expon	θ	uniform slope LS	SAF	segment LS
1	2.0	0.5	1.15	4.39 ✓	0.71	3.117 ✓
2	33.3	0.5	18.42	9.808 ✓	1.29	12.652 ✓

Σ 15.77

Uniform slope LS (1)

$S = 10.8 \sin(1.15) + 0.03 = 0.247$ ✓

$L = \left(\frac{\lambda}{72.6}\right)^n = \left(\frac{1399.72}{72.6}\right)^{0.5} = 4.39$ ✓

SAF
From Table 8.7 is 0.71 (See p 6

$(4.39)(0.71) = 3.117$

$LS = (3.117 + 12.652) / 2$

$= 7.8845$ ✓

LS ACCOUNTING FOR THAWING SOILS is 7.885 ✓

Foth & Van Dyke	Client: CMC	Scope I.D.: 93C049
	Project: LS CALCS	Page: 9/19
	Prepared by: MDF	Date: 4/21/95
	Checked by: [illegible]	Date: 5/1/95

Average values of slope length

measured = horizontal projection

Avg 33.3% side $\approx 0.75'' \times \frac{200'}{1''} = 150'$

Avg 2% top $\approx 9.5'' \times \frac{200'}{1''} = 1900'$

$L_2 = \left(\frac{150}{72.6}\right)^{0.5} = 1.44$ ✓

$L_1 = \left(\frac{1900}{72.6}\right)^{0.5} = 5.12$ ✓

$S_2 = 16.8 \sin \theta_2 - 0.50$
$= 16.8 \sin 18.42 - 0.50$
$= 4.808$

$\theta =$ 33.33% slope $= 18.42°$

$S_1 = 10.8 \sin \theta + 0.03$
$10.8 \sin 1.15° + 0.03$
$S_1 = 0.247$

$\theta =$ 2% slope $= 1.15°$

$L_1 S_1 = (5.12)(0.247) = 1.26$ ✓ SAF = 0.71

$L_2 S_2 = (1.44)(4.808) = 6.92$ ✓ SAF = 1.29

segment 1 $(1.26)(0.71) = 0.895$ ✓ SEE P 6

segment 2 $(6.92)(1.29) = 8.927$ ✓

$\Sigma / n = LS$

$LS = (8.927 + 0.895)/2 = \underline{4.911}$ ✓

greater than 15 ft, the S factor from the USLE was modified significantly by McCool *et al.* (1987, 1993) after extensive evaluation of the original USLE data base. The modified version is

$$S = 10.8 \sin\theta + 0.03; \quad \sin\theta < 0.09 \qquad (8.39)$$

$$S = 16.8 \sin\theta - 0.50; \quad \sin\theta \geq 0.09, \qquad (8.40)$$

where θ is the slope angle. Based on an evaluation of data from disturbed lands with slopes up to 84%, McIssac *et al.* (1987) developed an equation similar to (8.39) and (8.40) with exponents in the same range; thus McCool *et al.* (1993) recommend that Eqs. (8.39) and (8.40) also be used for disturbed lands.

For slope lengths less than 15 ft, the S factor is not as strongly related to slope (slope exponent less than 1.0) since rilling would not have been initiated. The recommended factor is

$$S = 3.0(\sin\theta)^{0.8} + 0.56. \qquad (8.41)$$

Under conditions where thawing of recently tilled soils is occurring and surface runoff is the primary factor causing erosion (typical of the Pacific Northwest in the spring), the S factor should be (McCool *et al.*, 1987, 1993)

$$S = 4.25(\sin\theta)^{0.6}, \quad \sin\theta \geq 0.09. \qquad (8.42)$$

For thawing soils with slopes less than 9%, Eq. (8.39) should be used.

The S factor in the RUSLE is significantly modified from the original USLE as a result of an extensive reevaluation of the original data base, addition of the factors for short slope lengths, and new values for thawing soils (McCool *et al.*, 1987). The original data base did not include values beyond 20%. When using the quadratic form of the equation for S developed for the original USLE, projections beyond 20% yielded unreasonably high values for erosion. The RUSLE equation with the linear function corrects this problem.

Table 8.4 K Value Estimates based on Textural Information (English Units) (Soil Conservation Service, 1978)

Texture	Estimated K value[a]
Topsoil	
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Fine sandy loam, loamy very fine sand, sandy loam	0.24
Loamy fine sand, loamy sand	0.17
Sand	0.15
Silt loam, silty clay loam, very fine sandy loam	0.37
Subsoil and Residual Material	
Outwash Soils	
Sand	0.17
Loamy sand	0.24
Sandy loam	0.43
Gravel, fine to moderate fine	0.24
Gravel, medium to moderate coarse	0.49
Lacrustrine Soils	
Silt loam and very fine sandy loam	0.37
Silty clay loam	0.28
Clay and silty clay	0.28
Glacial Till	
Loam, fine to moderate fine subsoil	0.32
Loam, medium subsoil	0.37
Clay loam	0.32
Clay and silty clay	0.28
Loess	0.37
Residual	
Sandstone	0.49
Siltstone, nonchannery	0.43
Siltstone, channery	0.32
Acid clay shale	0.28
Calcareous clay shale or limestone residuum	0.24

[a]These values are typical based only on textural information. Values for an actual soil can be considerably different due to different structure and infiltration.

[b]Units on K in this table are English units (tons•acre•hr/hundreds•acre•ft•tonsf•in.). To convert to metric units (t•ha•h/ha•MJ•mm), multiply K values by 0.1317.

Slope Length Factor

The slope length factor was developed by McCool *et al.* (1989, 1993) from the original USLE data base augmented with theoretical considerations. The L factor retains its original form

$$L = \left[\frac{\lambda}{72.6}\right]^m, \qquad (8.43)$$

where λ is the slope length in feet, 72.6 ft is the length of a standard erosion plot, and m is a variable slope length exponent. Slope length, λ, is the horizontal projection of plot length, not the length measured along the slope. The difference in horizontal projections and slope lengths becomes important on steeper slopes.

The slope length exponent is related to the ratio of rill to interrill erosion, β (Foster *et al.*, 1977b; McCool *et al.*, 1989, 1993), by

$$m = \frac{\beta}{1+\beta}. \qquad (8.44)$$

From Eq. (8.50b)

$$LS = \Sigma/n = 10.84/3 = 3.61.$$

4. Summary.

Concave $LS = 3.61$
Uniform $LS = 4.33$
Convex $LS = 5.07$.

Thus the convex shape has the highest LS factor.

Estimating Slope Lengths for Watersheds

Slope length estimates for fields or watersheds with nonplanar surfaces require considerable professional judgment. A large number of slope lengths occur in any given real watershed. Erosion can be estimated for each of these and area weighted to determine average erosion on a watershed.

Slope length is defined as the slope distance from the point of origin of overland flow to the point of

Table 8.8 Selected USLE C Values for Construction, Mining, and Forest Lands.

Condition	C factor	References
1. Bare soil conditions		
Undisturbed except scraped	0.66–1.30	a
Compacted		
Smooth	1.00–1.40	a,b
Root raked	0.90–1.20	a
Disk tillage		
Fresh	1.00	a
After one rain	0.89	a
2. Mulch		
Straw		
0.5 tons/ac	0.30	a,d
1.0 tons/ac	0.18	a,d
2.0 tons/ac	0.09	a,d
4.0 tons/ac	0.02	a,d
Wood chips		
0.5 tons/ac	0.90	a,d
2.0 tons/ac	0.70	a,d
4.0 tons/ac	0.42	a,d
6.0 tons/ac	0.22	a,d
3. Chemical binders		
Asphalt emulsion, 605 gal/ac	0.14-0.52	a
Aquatan, Terra-tack	0.67	a
4. Seedings		
No prepared seedbed		
New planting	0.64	a
After 60 days	0.54	a
Prepared seedbed		
New planting	0.40	a
After 60 days	0.05	a
5. Undisturbed forest		
100–75% canopy, 100–90% litter	0.0001–0.001	c
35–20% canopy, 70–40% litter	0.003–0.009	c
6. Permanent pasture and brush cover		
0% canopy, 80% ground cover		
Grass	0.013	c
Weeds	0.043	c
50% Brush, 80% ground cover		
Grass	0.012	c
7. Mechanically prepared woodland sites		
Burned, 10% cover at ground		
Good soil	0.240	c
Poor soil	0.360	c
Burned, 0% cover at ground		
Good soil	0.260	c
Poor soil	0.450	c
Disked, 0% cover at ground		
Good soil	0.720	c
Poor soil	0.940	c

For the C value and the entire soil loss equation to be valid 4.0 tons/acre of straw mulch must be applied

Note. Additional values are given in Appendix 8B.
[a]Transportation Research Board (1980).
[b]Barfield *et al.* (1988).
[c]Wischmeier and Smith (1978).
[d]Meyer *et al.* (1972). C factors for mulch vary depending on slope length and steepness. Slope length limits apply (see Appendix 8B, Table 8B.5).

VALUES OF FACTOR P

Practice	Land slope (percent)				
	1.1-2	2.1-7	7.1-12	12.1-18	18.1-24
	(Factor P)				
Contouring (P_c)	0.60	0.50	0.60	0.80	0.90
Contour strip cropping (P_{sc})					
R-R-M-M[1]	0.30	0.25	0.30	0.40	0.45
R-W-M-M	0.30	0.25	0.30	0.40	0.45
R-R-W-M	0.45	0.38	0.45	0.60	0.68
R-W	0.52	0.44	0.52	0.70	0.90
R-O	0.60	0.50	0.60	0.80	0.90
Contour listing or ridge planting (P_{cl})	0.30	0.25	0.30	0.40	0.45
Contour terracing (P_t)[2]	[3] $0.6/\sqrt{n}$	$0.5/\sqrt{n}$	$0.6/\sqrt{n}$	$0.8/\sqrt{n}$	$0.9/\sqrt{n}$
No support practice	1.0	1.0	1.0	1.0	1.0

Appendix M

Revised Universal Soil Loss Equations (RUSLE) for Interior Berms

Foth & Van Dyke

Client: CMC Scope I.D.: 93C049
Project: Soil loss calc for TMA Berm Page: 1/11
Prepared by: MJZ Date: 4/4/95
Checked by: JBKiesel Date: 5/1/95

Soil loss calculation: Interior Berms

Purpose: determine soil loss for TMA Berm with and without vegetative cover

Reference: Haan, CT, et al. Design Hydrology and Sedimentology for Small Catchments. (1994).

Conclusion: @ 50' length 33.3% Bare Ground Soil Loss is 214.2 ton/ac/yr
@ 50' length 33.3% Prepared Seeding Soil Loss is 4.08 ton/ac/yr

Equation: $A = RKLSCP$

A = average soil loss per unit area (ton/ac/yr)

	Bare Ground 95	Prepared Mulch 95
R = rainfall/runoff factor		
K = soil erodibility factor	0.32	0.32
SL = slope length factor	3.87	3.87
C = cover and management factor	1.05	0.02
P = supporting conservation factor	1.0	1.0

Calculation

CMC is located in Forrest county which has been highlighted on the attached map. The annual R isoline is approximately

$R = 95 \left(\frac{100 ft \cdot tonsf \cdot in}{Ac\ hr\ yr}\right)$ SEE ATTACHED FIG 8.6

The soil in Question is a glacial till. The average value of the K values for glacial till is $\frac{.32+.37+.22+.23}{4} = 0.3225$

$K = 0.32$

see the attached sheet for K values - TABLE 8.4

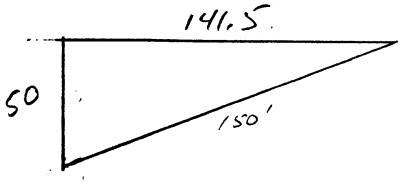
Foth & Van Dyke

Client: Case Scope I.D.: 93C044
Project: Soil Loss Page: 2/11
Prepared by: MSZ Date: 4/4/95
Checked by: RPK Date: 5/1/95

DETERMINE L, S. FACTOR

$L = \left[\frac{\lambda}{72.6}\right]^m$ - SEE Equation 8.43 of ATTACHMENTS

where λ = SLOPE length (horizontal projection) of a [illegible] FT LENGTH of SLOPE



$A^2 = 150^2 - 50^2$

$A = 141.5 = \lambda$

m = VARIABLE SLOPE LENGTH EXPONENT

Due to compaction Rill/interrill susceptibility RATIO.

FOR COMPACTED SOILS ASSUME Low Ratio

$M = \frac{B}{1+B}$ - (SEE EQUATION 8.44 of ATTACHMENT)

$B = \frac{11.16 \sin\theta}{3.0 (\sin\theta)^{0.8} + 0.56}$ $\theta = 18.42°$ FOR 3:1 SLOPE (SEE Equation 8.45 of ATTACHMENTS)

$B = \frac{11.16 \sin 18.42°}{3.0 (\sin 18.42°)^{0.8} + 0.56} = 2.0109$

$B_{Low} = 2.0109/2 = 1.0055$ (SEE NOTE TABLE 8.6 of ATTACHMENT)

$m = \frac{1.0055}{1+1.0055} = 0.5014$

$L = \left[\frac{141.5}{72.6}\right]^{0.5014} = 1.3973$

$S = 16.8 \sin\theta - 0.50$ For $\sin\theta \geq 0.09$

$\theta = 18.42° = 3{:}1$ slope (SEE Equation 8.40 of ATTACHMENTS)

$S = 16.8 \sin 18.42° - 0.50 = 4.8085$

$LS = (4.8029)(1.3973) =$

$LS = 6.711$

The value for C will be chosen for two conditions

1) no cover

Bare soil, compacted, Root Raked =

- USE C = 1.05 (SEE ATTACHED TABLE 8.8)

2) planted vegetation

Mulch planting @ 4 tons/ac

USE C = 0.02 tons/ac

P = 1.0 for All cases of no support practice

see Attached Table

Foth & Van Dyke

Client: CMC — Scope I.D.: 930049
Project: SOIL LOSS CALCS — Page: 4/11
Prepared by: MJZ — Date: 4/4/95
Checked by: [signature] — Date: 5/1/95

soil loss with no vegetation - BARE GROUND - 50 LENGTH @ 33.33%

$$A = (95)(0.32)(6.711)(1.05)(1.0) = \boxed{214.2 \text{ ton/AC/yr}}$$

soil loss with vegetation - PREPARED SEEDING - 50 LENGTH @ 33.33%

$$A = (95)(0.32)(6.711)(0.02)(1.0) = \boxed{4.08 \text{ ton/AC/yr}}$$

Soil Loss of 214.2 ton/acre/yr translates into an erosion of $\left(\frac{214.2 \text{ tons}}{\text{ACRE} \cdot \text{YR}}\right)\left(\frac{2000 \text{ lb}}{\text{ton}}\right)\left(\frac{1 \text{ ACRE}}{43560 \text{ FT}^2}\right)\left(\frac{12 \text{ in}}{1 \text{ FT}}\right)\left(\frac{1 \text{ CF}}{100 \text{ lb}}\right) = 1.18 \text{ in}/\text{FT}^3/\text{YR}$

assuming a unit weight of 100 pcf for the fill layers. Since the average "life" of bare slopes is 4 years or less, this will translate into about 4.7 inches of erosion or less near the top reaches. This is acceptable especially in view of the possibility of tailings covering the slopes during deposition.

If rill formation is found to be excessive, establishment of vegetation or other erosion controls can be resorted to.

ATTACHMENTS

SOURCE #1-

HAAN, C.T., BARFIELD, B.J. and HAYES, J.C. (1994) DESIGN HYDROLOGY AND SEDIMENTOLOGY FOR SMALL CATCHMENTS, ACADEMIC PRESS, HARCOURT BRACE & COMPANY, SAN DIEGO, CA. 588pp

SOURCE #2

DESIGN AND CONSTRUCTION OF COVERS FOR SOLID WASTE LANDFILLS, EPA 600/2-79-165, AUGUST, 1979

Soil Loss 93C049 6/11

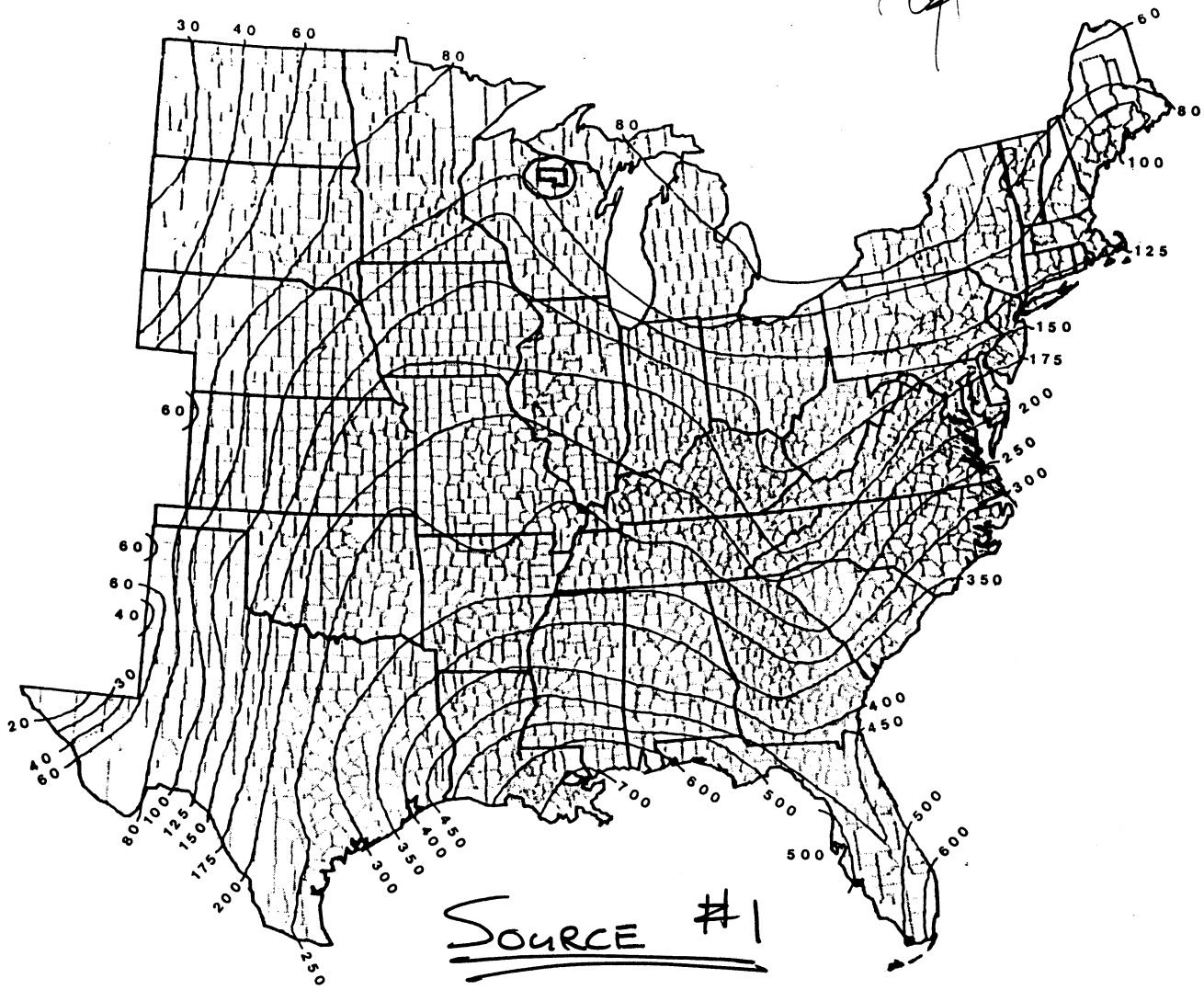


Figure 8.6 Isolines of annual R factor for the Eastern United States (after Renard *et al.*, 1993b). Maps for areas in and west of the Rockies are in Appendix 8A. R factor is in ft · tonsf · in./acre · hr · year. To convert to SI units, MJ · mm/ha · h · y, multiply by 17.02.

Units on the annual R factor are hundreds of ft · tonsf · in./acre · hr · year. The hundreds term comes from the 100 divisor in Eq. (8.31). In further reference, the units on R will be given as ft · tonsf · in./acre · hr · year for simplicity and the hundreds will be understood. The units on single storm erosivity would be ft · tonsf · in./acre · hr · storm.

To convert the annual R to SI units, MJ · mm/ha · hr · year, multiply the English Units by 17.02. A summary of other conversion factors is given in the Appendix. As a note ft · tonsf · in./acre · hr · year means

$$\frac{\text{ft tonsf in.}}{\text{acre hr year}}.$$

Average Annual R Values

Using Eq. (8.31), Wischmeier and Smith (1965) computed values for the R factor for stations east of the Rockies and extended the values to the Western States utilizing a correlation between 2-year, 6-hr rainfall and the R factor (Wischmeier and Smith, 1978). In the development of RUSLE, additional analyses were made on longer data bases and on hourly rainfall for stations in and west of the Rockies. The data are summarized in Fig. 8.6 for the eastern United States.

Corrections for Ponding

Studies have shown that ponded water on flat slopes tends to absorb rainfall energy and retard erosion

greater than 15 ft, the S factor from the USLE was modified significantly by McCool *et al.* (1987, 1993) after extensive evaluation of the original USLE data base. The modified version is

$$S = 10.8 \sin\theta + 0.03; \quad \sin\theta < 0.09 \qquad (8.39)$$

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where θ is the slope angle. Based on an evaluation of data from disturbed lands with slopes up to 84%, McIssac *et al.* (1987) developed an equation similar to (8.39) and (8.40) with exponents in the same range; thus McCool *et al.* (1993) recommend that Eqs. (8.39) and (8.40) also be used for disturbed lands.

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For thawing soils with slopes less than 9%, Eq. (8.39) should be used.

The S factor in the RUSLE is significantly modified from the original USLE as a result of an extensive reevaluation of the original data base, addition of the factors for short slope lengths, and new values for thawing soils (McCool *et al.*, 1987). The original data base did not include values beyond 20%. When using the quadratic form of the equation for S developed for the original USLE, projections beyond 20% yielded unreasonably high values for erosion. The RUSLE equation with the linear function corrects this problem.

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Clay and silty clay	0.28
Glacial Till	
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Loess	0.37
Residual	
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[a]These values are typical based only on textural information. Values for an actual soil can be considerably different due to different structure and infiltration.

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Slope Length Factor

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The slope length exponent is related to the ratio of rill to interrill erosion, β (Foster *et al.*, 1977b; McCool *et al.*, 1989, 1993), by

$$m = \frac{\beta}{1+\beta}. \qquad (8.44)$$

Table 8.5 Experimental K Value Estimates for Disturbed Lands (English Units)

Reclaimed soil or residual material	Location of experimental site	K Exp[a]/Nomo[b]	Reference
Hosmer silt loam	Indiana	0.387/0.485[c]	Stein *et al.* (1983)
Alfred silt loam	Southern Indiana	0.812/0.485	
Ava silt loam	Southern Indiana	0.842/0.478	
Graded overburden	Southern Indiana	0.197–0.835/ 0.250–0.478	
Clinton silt loam[d]	Western Illinois	0.370/0.360	Mitchell *et al.* (1983)
Tama silty clay loam[d]	Western Illinois	0.210/0.310	
Hosmer silt loam[d]	Southern Indiana	0.450–0.650/ 0.470	
Sadler silt loam (A horizon)	Western Kentucky	0.415/0.385	Barfield *et al.* (1988)
Sadler silt loam (B horizon)	Western Kentucky	0.380/0.640	
Shale spoil material	Western Kentucky	0.140/0.180	

[a]Values measured experimentally with rainfall simulators.
[b]Values calculated from Wischmeier *et al.* (1971) nomograph shown in Fig. 8.9.
[c]Values in English units of tons•acre•hr/hundreds•acre•ft•tonsf•in. To convert to metric units of t•a•h/ha•MJ•mm, multiply by 0.1317.
[d]The dominant soil series. Some mixing occurred with other series.

For soils that are classed as being moderately susceptible to erosion, McCool *et al.* (1989) proposed that

$$\beta_{\text{mod}} = \frac{11.16 \sin\theta}{3.0(\sin\theta)^{0.8} + 0.56}, \qquad (8.45)$$

where θ is the slope angle. Thus, the slope exponent is a function of the slope angle θ.

Soils in the RUSLE are classed as having low, moderate, or high susceptibility to rill erosion. Equation (8.45) is for soils that are moderately susceptible to erosion. Conversions for soils that have low or high susceptibility to erosion are given in Table 8.6. Values in Table 8.6 are based on the assumption that moderately erodible soils have a β defined by Eq. (8.45), soils highly susceptible to rilling have a β that is twice that given by Eq. (8.45), and soils with low susceptibility to rilling have a β that is defined by half that given by Eq. (8.45).

For soils in the Pacific Northwest, or other soils that are exposed to runoff during thawing without sufficient rainfall energy to cause interrill erosion, the values in Table 8.6 should not be used. Instead, McCool *et al.* (1989) recommend that a slope length exponent of 0.5 be used for all slopes. When runoff on thawing soils is exposed to rainfall sufficient to cause significant interrill erosion, the slope length exponent for the low rill to interrill erosion ratio should be used (column 1 in Table 8.6). For rangeland soils, the use of a low rill to interrill erosion ratio is proposed. Selection of the appropriate column to use in Table 8.6 requires professional judgement. The assistance of a soil scientist may be helpful.

Combined Length and Slope Factors

Combined slope length and slope steepness factors were calculated using the factors from Eqs. (8.39) to (8.43). These combination factors are given in Fig. 8.13 for all susceptibilities and for thawing soils.

Irregular and Segmented Slopes

Soil loss is strongly impacted by slope shape (Foster and Huggins, 1979). A convex shape will have greater erosion than a uniform slope by as much as 30%. A concave slope will have less erosion than a uniform slope. Foster and Wischmeier (1974) developed a procedure for evaluating the impact of irregular slopes by dividing the slope into segments. The soil loss per unit area from the ith segment is

$$A_i = RK_iC_iP_iS_i\left[\frac{\lambda_i^{m+1} - \lambda_{i-1}^{m+1}}{(\lambda_i - \lambda_{i-1})72.6^m}\right], \qquad (8.46)$$

where λ_i and λ_{i-1} are the slope lengths at the start and end of segment i, and K_i, C_i, P_i, and S_i are USLE factors for segment i. Equation (8.46) can be used for each segment i. The total erosion from each segment

Table 8.6 Slope Length Exponent m in Eq. (8.43) (after McCool *et al.*, 1993)[a]

Percentage slope	Rill/interrill ratio		
	Low[b]	Moderate[c]	High[d]
0.2	0.02	0.04	0.07
0.5	0.04	0.08	0.16
1.0	0.08	0.15	0.26
2.0	0.14	0.24	0.39
3.0	0.18	0.31	0.47
4.0	0.22	0.36	0.53
5.0	0.25	0.40	0.57
6.0	0.28	0.43	0.60
8.0	0.32	0.48	0.65
10.0	0.35	0.52	0.68
12.0	0.37	0.55	0.71
14.0	0.40	0.57	0.72
16.0	0.41	0.59	0.74
20.0	0.44	0.61	0.76
25.0	0.47	0.64	0.78
30.0	0.49	0.66	0.79
40.0	0.52	0.68	0.81
50.0	0.54	0.70	0.82
60.0	0.55	0.71	0.83

[a] Values in table are not applicable to thawing soils. See text for explanation.
[b] β = 1/2 value from Eq. (8.45) in Eq. (8.44).
[c] β = 1 × value from Eq. (8.45) in Eq. (8.44).
[d] β = 2 × value from Eq. (8.45) in Eq. (8.44).

would be $A_i(\lambda_i - \lambda_{i-1})$, and the average erosion per unit area over the entire slope length would be

$$A = R \sum_{i=1}^{n} K_i C_i P_i S_i \frac{\left[\lambda_i^{m+1} - \lambda_{i-1}^{m+1}\right]}{\lambda_e 72.6^m}, \qquad (8.47)$$

where λ_e is the total slope length. Equation (8.47) can also be used to evaluate the effects of variation in K, C, and P over the slope length.

An alternate method for evaluating irregular slopes is the use of a slope length adjustment factor (SAF). If the slope is divided into n increments of equal length ΔX, then

$$A = R \sum_{i=1}^{n} K_i C_i P_i S_i \frac{\left[(i\,\Delta X)^{m+1} - ([i-1]\,\Delta X)^{m+1}\right]}{n\,\Delta X 72.6^m}. \qquad (8.48)$$

Dividing by n times the soil loss from a uniform slope of equal length and assuming constant values of K_i C_i P_i along the slope, a slope adjustment factor can be developed for each segment, or

$$\mathrm{SAF}_i = \frac{A_i}{A} = \frac{i^{m+1} - (i-1)^{m+1}}{n^m}, \qquad (8.49)$$

where n is the number of segments and SAF is the slope adjustment factor. The sum of the SAF_i for a given slope is equal to the number of segments n; thus the average erosion over the slope is

$$A = \frac{R}{n} \sum_{i=1}^{n} K_i C_i P_i S_i L_i (\mathrm{SAF})_i. \qquad (8.50a)$$

where L_i is the slope length factor calculated from Eq. (8.43) using the m value corresponding to the segment steepness. In the development of a SAF relationship, R, K, C, and P remain constant over all segments; thus Eq. (8.50a) can be solved for an equivalent LS factor

$$LS = \frac{1}{n} \sum_{i=1}^{n} S_i L_i (\mathrm{SAF})_i. \qquad (8.50b)$$

Factors calculated from Eq. (8.50b) are given in Table 8.7. An example of its use is given in Example Problem 8.5.

Example Problem 8.5. Estimating *LS* factors

A soil that is very susceptible to rilling has a slope length of 210 ft and an average slope of 15%. Estimate the LS factor if:

(1) the slope is uniform
(2) the slope is convex with slopes of 10, 15, and 20% on segments 1, 2, and 3
(3) the slope is concave with slopes of 20, 15, and 10% on segments 1, 2, and 3.

Assume that the soil is not freezing and thawing.

Solution:

1. Uniform slope. The slope angle is

$$\theta = \tan^{-1} 0.15 = 8.53°.$$

From Eq. (8.45) for soils moderately susceptible to rilling,

$$\beta = \frac{11.16 \sin 8.53}{3.0(\sin 8.53)^{0.8} + 0.56} = 1.37.$$

From Eq. (8.50b)

$$LS = \Sigma/n = 10.84/3 = 3.61.$$

4. Summary.

Concave $LS = 3.61$
Uniform $LS = 4.33$
Convex $LS = 5.07$.

Thus the convex shape has the highest LS factor.

Estimating Slope Lengths for Watersheds

Slope length estimates for fields or watersheds with nonplanar surfaces require considerable professional judgment. A large number of slope lengths occur in any given real watershed. Erosion can be estimated for each of these and area weighted to determine average erosion on a watershed.

Slope length is defined as the slope distance from the point of origin of overland flow to the point of

Table 8.8 Selected USLE C Values for Construction, Mining, and Forest Lands.

Condition	C factor	References
1. Bare soil conditions		
Undisturbed except scraped	0.66–1.30	*a*
Compacted		
Smooth	1.00–1.40	*a,b*
Root raked	0.90–1.20	*a*
Disk tillage		
Fresh	1.00	*a*
After one rain	0.89	*a*
2. Mulch		
Straw		
0.5 tons/ac	0.30	*a,d*
1.0 tons/ac	0.18	*a,d*
2.0 tons/ac	0.09	*a,d*
4.0 tons/ac	0.02	*a,d*
Wood chips		
0.5 tons/ac	0.90	*a,d*
2.0 tons/ac	0.70	*a,d*
4.0 tons/ac	0.42	*a,d*
6.0 tons/ac	0.22	*a,d*
3. Chemical binders		
Asphalt emulsion, 605 gal/ac	0.14-0.52	*a*
Aquatan, Terra-tack	0.67	*a*
4. Seedings		
No prepared seedbed		
New planting	0.64	*a*
After 60 days	0.54	*a*
Prepared seedbed		
New planting	0.40	*a*
After 60 days	0.05	*a*
5. Undisturbed forest		
100–75% canopy, 100–90% litter	0.0001–0.001	*c*
35–20% canopy, 70–40% litter	0.003–0.009	*c*
6. Permanent pasture and brush cover		
0% canopy, 80% ground cover		
Grass	0.013	*c*
Weeds	0.043	*c*
50% Brush, 80% ground cover		
Grass	0.012	*c*
7. Mechanically prepared woodland sites		
Burned, 10% cover at ground		
Good soil	0.240	*c*
Poor soil	0.360	*c*
Burned, 0% cover at ground		
Good soil	0.260	*c*
Poor soil	0.450	*c*
Disked, 0% cover at ground		
Good soil	0.720	*c*
Poor soil	0.940	

Note. Additional values are given in Appendix 8B.
[a]Transportation Research Board (1980).
[b]Barfield *et al.* (1988).
[c]Wischmeier and Smith (1978).
[d]Meyer *et al.* (1972). C factors for mulch vary depending on slope length and steepness. Slope length limits apply (see Appendix 8B, Table 8B.5).

VALUES OF FACTOR P

Practice	Land slope (percent)				
	1.1-2	2.1-7	7.1-12	12.1-18	18.1-24
	(Factor P)				
Contouring (P_c)	0.60	0.50	0.60	0.80	0.90
Contour strip cropping (P_{sc})					
R-R-M-M[1]	0.30	0.25	0.30	0.40	0.45
R-W-M-M	0.30	0.25	0.30	0.40	0.45
R-R-W-M	0.45	0.38	0.45	0.60	0.68
R-W	0.52	0.44	0.52	0.70	0.90
R-O	0.60	0.50	0.60	0.80	0.90
Contour listing or ridge planting (P_{cl})	0.30	0.25	0.30	0.40	0.45
Contour terracing (P_t)[2]	[3] $0.6/\sqrt{n}$	$0.5/\sqrt{n}$	$0.6/\sqrt{n}$	$0.8/\sqrt{n}$	$0.9/\sqrt{n}$
No support practice	1.0	1.0	1.0	1.0	1.0

SOURCE #2

Soil Loss

Appendix N

Tailings Management Area Groundwater Quality Performance Evaluation

(Note: This appendix was replaced by the May 1997 Foth & Van Dyke report titled *Crandon Project Tailings Management Area Groundwater Quality Performance Evaluation*. The report was issued as part of a May 15, 1997 update to the Crandon Project *Environmental Impact Report* (EIR). The report is discussed in Section 4.2.5.10 of the EIR and provided in its entirety in Appendix 4.2-12 of the EIR.)

Foth & Van Dyke
Memorandum

October 18, 1995

TO: Jerry W. Sevick, Foth & Van Dyke

CC: Don Moe, Crandon Mining Company
Master File

FR: Steve Lehrke, Foth & Van Dyke SL
Marek Zaluski, Foth & Van Dyke MMAZ

RE: Crandon Project - Groundwater Quality Performance Assessment for the Tailings Management Area

1 Introduction

Crandon Mining Company (CMC) is in the process of applying for necessary permits to mine an ore body to extract metals. The project site is located in Forest County, Wisconsin, five miles south of the City of Crandon, and two miles east of State Trunk Highway 55.

Numerous documents have been developed to satisfy requirements for the necessary project permits. Together, these documents provide engineering details pertaining to the construction, operation and reclamation of the mine, plant site and tailings facilities, as well as a detailed description of the project's environmental setting and a detailed evaluation of project impacts, including plans for the mitigation of impacts.

The objective of this memorandum is to compare the performance of the TMA as designed to potential groundwater quality standards. The information from the comparison will be used while preparing the remaining sections of the project's EIR. The memorandum has been prepared based on:

- The data and descriptive materials that are available in project documents issued to date. The pertinent information in these documents is incorporated into this memorandum by reference, and therefore will not be repeated here.
- The results of the evaluation establishing the source term for the TMA. The source term is defined as the quality of the small amount of leachate that will potentially percolate from the TMA and reach the groundwater system.
- The results of the project's solute transport modeling analysis, which takes into consideration the volume and timing of leachate percolation from the TMA.
- A review of current groundwater quality regulations that will likely be applied to the TMA.
- Appropriate statistical calculations.

A similar performance analysis for the reflooded mine has been determined to be unnecessary since the project's regional groundwater modeling work has shown that particle travel times to the compliance boundary would be very long (about 2,000 years for the best engineering judgement case).

2 Regulatory Background

Groundwater quality associated with the TMA is regulated under Chapter NR 182, Wis. Admin. Code. According to this code, groundwater quality standards at the compliance boundary are to be based on three key criteria:

- Primary and secondary Maximum Contaminant Levels (MCLs) as promulgated by state or national drinking water standards;
- The baseline concentration where that concentration exceeds the MCL; and
- The need to protect public health, safety and welfare.

For the State of Wisconsin, the MCLs are addressed in Chapter NR 809 Wis. Admin. Code entitled "Safe Drinking Water." Standards relating to the protection of the public health, safety and welfare are addressed in Chapter NR 140, Wis. Admin. Code entitled "Groundwater Quality" where Enforcement Standards (ESs) and Preventive Action Limits (PALs) are specified. In most cases the numerical values for the MCLs of NR 809 and for the ESs of NR 140 are the same or very similar. Any exemptions granted by the WDNR under NR 182 must not exceed the levels required to protect public health, safety and welfare.

The term "baseline concentration" is defined in NR 182 as "the concentration of a substance in groundwater . . . as determined by monitoring before mining operations." The same regulation defines the term "background concentration" as "the concentration of a substance in groundwater . . . as determined by monitoring at locations which are not to be affected by the mining site."

In addition, NR 182 also defines "statistically significant change" as "an amount of change determined by the use of statistical tests for measuring significance at the 95 percent confidence level." This term is used in conjunction with the code's requirement that a mining waste site "shall not cause concentrations of substances in groundwater at or beyond the compliance boundary in excess by a statistically significant amount of the groundwater quality standard" established per NR 182.

Given the above discussion, the groundwater compliance assessment for the TMA was accomplished in three steps. The first was to calculate estimated incremental groundwater changes at the compliance boundary due to the TMA without considering baseline groundwater concentrations. The compliance boundary is expected to extend radially from the TMA for a distance of 1,200 feet.

The second step was to evaluate existing groundwater quality data within the proposed TMA compliance boundary. This analysis was performed to determine if any baseline concentrations exceed MCLs, which would then require the establishment of an alternative concentration limit (ACL) for those parameters. NR 182.075(1)(a)2 b states that "Where the baseline concentration of a substance subject to a state or national drinking water standard exceeds the MCL set by

state or national drinking water standards, the groundwater quality standard shall be the baseline concentration of that substance . . . ." Since groundwater quality data typically varies over time, a statistical analysis of the data is used to define the baseline concentration. For the purposes of this memorandum, baseline concentrations statistically derived per NR 182.075(1)(a)2 b are referred to as ACLs.

The third step of the analysis involved the comparison of estimated parameter concentrations at the TMA compliance boundary to the MCLs, and ACLs where appropriate, to determine if the TMA as designed will meet potential groundwater standards.

In accordance with NR 182.075, the WDNR is to establish the groundwater quality standards for the TMA as part of the master hearing process. To assist in the completion of the TMA performance assessment, it was assumed that these standards will be the NR 809 primary and secondary MCLs, with ACLs established where appropriate. It is possible that the WDNR will establish standards for parameters not addressed in NR 809. Using the NR 809 list for this analysis is valid since the metals of importance in the TMA source term are included in NR 809.

Using the mean plus two standard deviations as a methodology for calculating ACLs for baseline groundwater quality meets the 95 percent confidence level required by NR 182 only if the data are normally distributed, and the true population mean and standard deviation are known. In completing the statistical analysis of existing site groundwater quality data, it was determined that the data was not always normally distributed. Therefore, in order to comply with the NR 182 requirement to measure statistical significance at the 95 percent confidence level, an alternative statistical method was used. This method, which is applicable to both normally and non-normally distributed data, is described in Section 3 of this memorandum.

It should also be noted that baseline water quality for the wells to be used for compliance monitoring of the TMA is planned to be completed prior to the initiation of construction. Upon the completion of this work it will be necessary to statistically evaluate the data to determine if individual ACLs will need to be established for any of the individual wells to be used in the future monitoring system.

3 Completion of Analysis

3.1 Calculation of Incremental Change at the Compliance Boundary

The long-term quality of the TMA leachate has been estimated by Dr. Richard Lawrence and John Thresher as described in an October 18, 1995 technical memorandum. The concentrations of those leachate parameters for which a NR 809 standard exists is shown in Table 1. These respective concentrations are referred to in this memorandum as the source term. It has been conservatively assumed that upon release of the source term to the groundwater system it will be transported as a nonreactive solute whose concentration is controlled by advection and dispersion processes. TMA transport modeling has been simulated by GeoTrans, Inc. as part of the project's regional groundwater modeling work and is discussed in Section 4 of the project's EIR (Foth & Van Dyke, 1995a). It should be noted that the GeoTrans analysis was conservative in nature since adsorption in both the saturated and unsaturated zones and dilution in the unsaturated zone were not considered. If these factors were incorporated into the analysis they would be expected to result in lower concentrations at the compliance boundary.

The GeoTrans solute transport modeling analysis estimated a relative concentration, shown below, at the TMA compliance boundary for the best engineering judgement and practical worst case scenarios based on unity at the source. This method is standard modeling practice and avoids the need to make simulations for each source term parameter. By knowing the modeled relative concentration values and the source term concentrations, the incremental concentrations at the compliance boundary due to the source term can be determined by simple multiplication. The estimated source term incremental concentrations for the TMA parameters of interest are shown in Table 1.

TMA Relative Concentration at Compliance Boundary	
Best Engineering Judgement	0.011
Practical Worst Case Scenario	0.026

3.2 Evaluation of Existing Baseline Water Quality

The 16 groundwater monitoring wells used for TMA groundwater quality validation in 1994 and 1995 as described in Section 3.6 of the EIR (Foth & Van Dyke, 1995a) were used to define current groundwater quality in the area. It should be noted that a different set of wells will be used to establish baseline groundwater quality conditions at the TMA prior to construction and for monitoring of groundwater quality once mining begins. An analysis similar to that described below or one for each parameter for each well will need to be completed once baseline data is collected to determine if ACLs are needed for these new wells.

The process used to determine how baseline water quality in the vicinity of the TMA compares to the MCLs was to statistically determine the tolerance limit for the entire data set for each parameter. The tolerance limit is an estimate of the upper 95th percentile of the population with 95 percent confidence. Once this was completed the tolerance limit for each parameter was compared to the NR 809 MCL. Where the tolerance limit was greater than the MCL, the tolerance limit was assumed to be an ACL for the respective parameter for NR 182 compliance demonstration purposes.

The data used for the analysis is shown in Table 2. The statistical analyses were completed using the statistical methods provided in the USEPA guidance documents "Statistical Analysis of Ground-Water Monitoring Data at RCRA Facilities - Interim Final Guidance" (USEPA, 1989), "Statistical Analysis of Ground-Water Monitoring Data at RCRA Facilities - Addendum to Interim Final Guidance" (USEPA, 1992) and "Solid Waste Disposal Facility Criteria - Technical Manual" (USEPA, 1993).

Several types of statistical tolerance limits may be calculated. These include parametric, non-parametric and Poisson. The type of tolerance limit calculated and the treatment of nondetects are dependent on both the distributional shape and the percentage of nondetects in the baseline data. USEPA (1992) discusses the criteria used to choose the appropriate type of tolerance limit. These criteria are outlined in the flowchart illustrated in Figure 1 and are briefly described as follows.

- If at least 85 percent of the baseline data are above the detection limit, and the data are either normally distributed or normally distributed following a logarithmic transformation, an upper tolerance limit based on the normal distribution is calculated.

- If between 50 percent and 85 percent of the baseline data are above the detection limit, and the data are either normally distributed or normally distributed following a logarithmic transformation, the mean and standard deviation are adjusted using Cohen's adjustment and an upper tolerance limit based on the normal distribution is calculated.

- If less than 50 percent but greater than 10 percent of the baseline data are above detection, or the data is not normally distributed or cannot be transformed to fit a normal distribution, an upper prediction limit based on the normal distribution cannot be justified and a non-parametric upper tolerance limit is used.

- If greater than 0 percent but less than or equal to 10 percent of the baseline data are above detection, a Poisson tolerance limit is calculated.

The algorithm for calculating a parametric tolerance limit is given in USEPA, 1989. The algorithms for calculating non-parametric and Poisson tolerance limits are given in USEPA, 1992. All tolerance limits were calculated as one-sided upper 95 percent limits with 95 percent coverage. Normality of the baseline data was determined with normal probability plots and the Shapiro-Wilk W test. The procedure of the Shapiro-Wilk W test is given in USEPA (1992) and also in Gilbert (1987). References used for performing statistical calculations are provided at the end of the memorandum. Example calculations for tolerance limits are provided in Attachment 1.

Comparing the results of the calculated tolerance limits (ACLs) to applicable NR 809 MCLs showed that baseline conditions are greater than the MCLs for antimony and manganese. Therefore, for the purposes of the TMA compliance demonstration, it will be assumed that the standard for antimony and manganese will be an ACL which is equivalent to the calculated tolerance limit.

3.3 Compliance Demonstration

In order to assess if the TMA meets water quality standards at the compliance boundary it was necessary to add the incremental source term concentration at the compliance boundary to the baseline groundwater quality and compare the result to the NR 809 MCLs and, where appropriate, to the calculated ACLs. The incremental source term concentration at the compliance boundary shown in Table 1 was calculated by multiplying the source term by the modeled relative concentrations. The average baseline groundwater quality was conservatively calculated as the upper 95 percent confidence limit of the mean for each parameter. As with the calculation of the ACLs, the guidance given in USEPA (1989) was used to determine the appropriate type of confidence limit to use. This criteria is outlined in the flowchart illustrated in Figure 2 and briefly described below:

- If at least 85 percent 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 percent but less than 85 percent 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 percent of the baseline data are above the detection limit, a non-parametric upper confidence limit on the median is used.

Since all parameters have at least thirty results, the data sets were considered to be large samples and a test for normality was unnecessary. This is due to the fact that the mean of a large sample is a normally distributed variable.

Table 1 compares the estimated parameter concentrations at the compliance boundary to the estimated groundwater standards or ACLs, where appropriate. In all cases, the estimated concentration for all parameters for both the best engineering judgement and the practical worst case scenarios fall below the standard or ACL. Therefore it can be concluded that the TMA will not cause an increase in groundwater concentrations harmful to public health, safety or welfare.

4 References

Foth & Van Dyke, May, 1995a. *Environmental Impact Report for the Crandon Project.*

Foth & Van Dyke, May, 1995b. *Mine Permit Application for the Crandon Project.*

Foth & Van Dyke, May, 1995c. *Tailings Management Area Feasibility Report/Plan of Operation for the Crandon Project.*

Gilbert, Richard O., 1987. Statistical Methods for Environmental Pollution Monitoring. Van Nostrand Reinhold, New York.

USEPA, 1989. Statistical Analysis of Ground-Water Monitoring Data at RCRA Facilities - Interim Final Guidance. Office of Solid Waste, Washington, D.C.

USEPA, 1992. Statistical Analysis of Ground-Water Monitoring Data at RCRA Facilities - Addendum to Interim Final Guidance. Office of Solid Waste, Washington, D.C.

USEPA, 1993. Solid Waste Disposal Facility Criteria - Technical Manual. Office of Solid Waste.

Table 1

Tailings Management Area Compliance Performance Assessment

Parameter	Units	Source Term[1]	Relative Concentration[2]		Incremental Source Term At Compliance Boundary[3]		Average Baseline Groundwater Quality[4]	Estimated Parameter Concentration[5]		MCL[6]	ACL[7]
			Best Engineering Judgement	Practical Worst Case	Best Engineering Judgement	Practical Worst Case		Best Engineering Judgement	Practical Worst Case		
Aluminum	mg/L	0.08	0.011	0.026	0.001	0.002	<0.200	<0.200	<0.200	0.2[8]	
Antimony	mg/L	0.002	0.011	0.026	0.00002	0.0001	<0.050	<0.050	<0.050	0.006	0.070
Arsenic	mg/L	0.01	0.011	0.026	0.0001	0.0003	<0.005	<0.005	<0.005	0.05	
Barium	mg/L	0.03	0.011	0.026	0.0003	0.001	0.025	0.025	0.026	2	
Beryllium	mg/L	<0.0001	0.011	0.026	<0.000001	<0.000003	-[10]	-	-	0.004	
Cadmium	mg/L	0.01	0.011	0.026	0.0001	0.0003	<0.001	<0.001	<0.001	0.005	
Chromium	mg/L	0.02	0.011	0.026	0.0002	0.0005	<0.005	<0.005	<0.005	0.1	
Copper	mg/L	0.07	0.011	0.026	0.001	0.002	<0.030	<0.030	<0.030	1.3	
Cyanide[9]	ug/L	<80	0.011	0.026	<0.88	<2.08	<20	<20	<20	200	
Iron	mg/L	0.8	0.011	0.026	0.01	0.02	<0.100	<0.100	<0.100	0.3[8]	
Lead	mg/L	0.1	0.011	0.026	0.001	0.0026	<0.002	0.002	0.004	0.015	
Manganese	mg/L	1.4	0.011	0.026	0.015	0.036	0.011	0.026	0.047	0.05[8]	1.5
Mercury	ug/L	0.1	0.011	0.026	0.001	0.003	<0.2	<0.2	<0.2	2	
Nickel	mg/L	0.02	0.011	0.026	0.0002	0.001	<0.030	<0.030	<0.030	0.1	
Selenium	mg/L	0.002	0.011	0.026	0.00002	0.00005	<0.005	<0.005	<0.005	0.05	
Silver	mg/L	0.002	0.011	0.026	0.00002	0.00005	<0.010	<0.010	<0.010	0.1[8]	

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Table 1 *(Continued)*

Parameter	Units	Source Term[1]	Relative Concentration[2] Best Engineering Judgement	Relative Concentration[2] Practical Worst Case	Incremental Source Term At Compliance Boundary[3] Best Engineering Judgement	Incremental Source Term At Compliance Boundary[3] Practical Worst Case	Average Baseline Groundwater Quality[4]	Estimated Parameter Concentration[5] Best Engineering Judgement	Estimated Parameter Concentration[5] Practical Worst Case	MCL[6]	ACL[7]
Sulfate	mg/L	2500	0.011	0.026	27.5	65.0	13.0	41	78	250[8]	
Thallium	mg/L	<0.0001	0.011	0.026	<0.000001	<0.000003	-[10]	-	-	0.002	
TDS	mg/L	3100	0.011	0.026	34.1	80.6	168.3	203	249	500[8]	
Zinc	mg/L	1.1	0.011	0.026	0.012	0.029	<0.020	0.022	0.039	5[8]	

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[1]See the October 18, 1995 TMA Source Term Technical Memorandum by Lawrence and Thresher.
[2]From GeoTrans, Inc. Regional Groundwater Modeling Study (See Section 4.2.5.10 and Appendix 4.2-3 of EIR).
[3]Product of source term value multiplied by the relative concentration.
[4]Value is upper 95 percent confidence limit of the mean.
[5]Equals the incremental source term concentration at the compliance boundary plus the upper 95 percent confidence limit of the mean. For confidence limits below detection the incremental source term was added to one-half the detection limit. For those cases where the estimated parameter concentration was still less than the detection limit, the estimated parameter concentration was set at the detection limit.
[6]NR 809 MCL.
[7]Alternative Concentration Level calculated statistically using baseline water quality data.
[8]Value is secondary MCL.
[9]The source term value is for total cyanide. The cyanide MCL is for free cyanide. Since the source term value is appreciably lower than the MCL, a determination of the source term's free cyanide is not necessary.
[10]No baseline data available.

Table 2
Tailings Management Area
Groundwater Quality Statistical Analysis

Monitoring Well	Date	Aluminum mg/L	Antimony mg/L	Arsenic mg/L	Barium mg/L	Beryllium mg/L	Cadmium mg/L	Chromium mg/L	Copper mg/L	Cyanide ug/L	Iron mg/L	Lead mg/L	Manganese mg/L	Mercury ug/L	Nickel mg/L	Selenium mg/L	Silver mg/L	Sulfate mg/L	Thallium mg/L	TDS mg/L	Zinc mg/L
CMC-01P	May-94	< 0.200	< 0.050	< 0.005	0.012		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.031	< 0.2	< 0.030	< 0.005	< 0.010	12.0		139	< 0.020
CMC-01P	Jun-94	< 0.200	< 0.050	< 0.005	0.015		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	0.029	0.2	< 0.030	< 0.005	< 0.010	10.0		139	< 0.020
CMC-01P	Jul-94	< 0.200	< 0.050	< 0.005	0.015		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.031	0.3	< 0.030	< 0.005	< 0.010	10.0		128	< 0.020
CMC-01P	Aug-94	0.227	< 0.050	< 0.005	0.014		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.023	< 0.2	< 0.030	< 0.005	< 0.010	13.0		140	< 0.020
CMC-01P	Sep-94	< 0.050	< 0.20	< 0.0050	0.013		< 0.0001	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	0.025	< 0.2	< 0.030	< 0.0050	< 0.010	13		150	< 0.020
CMC-01P	Oct-94	< 0.050	< 0.20	< 0.0050	0.011		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.012	< 0.2	< 0.030	< 0.0050	< 0.010	15		170	< 0.020
CMC-01P	Nov-94	< 0.050	< 0.0050	< 0.0050	0.013		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	0.016	< 0.20	< 0.030	< 0.0050	< 0.010	13		180	< 0.020
CMC-01P	Dec-94	< 0.050	< 0.0050	< 0.0050	0.012		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	12		170	< 0.020
CMC-01R	May-94	< 0.200	< 0.050	< 0.005	< 0.010		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.022	< 0.2	< 0.030	< 0.005	< 0.010	70.0		339	< 0.020
CMC-01R	Jun-94	< 0.200	< 0.050	< 0.005	0.030		0.001	< 0.005	< 0.030	< 20	< 0.100	0.017	0.035	0.3	< 0.030	< 0.005	< 0.010	35.0		258	< 0.020
CMC-01R	Jul-94	< 0.200	< 0.050	< 0.005	0.026		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.001	0.023	0.4	< 0.030	< 0.005	< 0.010	23.0		202	< 0.020
CMC-01R	Aug-94	0.274	< 0.050	< 0.005	0.026		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	0.2	< 0.030	< 0.005	< 0.010	21.0		170	< 0.020
CMC-01R	Sep-94	< 0.050	< 0.20	< 0.0050	0.028		0.0005	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	19		180	0.021
CMC-01R	Oct-94	< 0.050	< 0.20	< 0.0050	0.027		0.00032	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	21		180	< 0.020
CMC-01R	Nov-94	< 0.050	< 0.0050	< 0.0050	0.027		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	19		190	< 0.020
CMC-01R	Dec-94	< 0.050	< 0.0050	< 0.0050	0.028		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	16		150	< 0.020
CMC-02P	May-94	< 0.200	< 0.050	< 0.005	0.011		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	13.0		150	< 0.020
CMC-02P	Jun-94	< 0.200	< 0.050	< 0.005	0.015		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	< 0.010	0.2	< 0.030	< 0.005	< 0.010	11.0		139	< 0.020
CMC-02P	Jul-94	< 0.200	< 0.050	< 0.005	0.010		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.003	< 0.010	0.7	< 0.030	< 0.005	< 0.010	13.0		130	< 0.020
CMC-02P	Aug-94	< 0.200	< 0.050	< 0.005	0.011		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	0.8	< 0.030	< 0.005	< 0.010	14.0		141	< 0.020
CMC-02P	Sep-94	< 0.050	< 0.20	< 0.0050	0.011		< 0.0001	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	14		150	< 0.020
CMC-02P	Oct-94	< 0.050	< 0.20	< 0.0050	0.0093		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	20		180	< 0.020
CMC-02P	Nov-94	< 0.050	< 0.0050	0.0055	0.012		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	0.016	< 0.20	< 0.030	< 0.0050	< 0.010	14		150	< 0.020
CMC-02P	Dec-94	< 0.050	< 0.0050	0.0064	0.012		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	13		160	< 0.020
CMC-02R	Oct-94	< 0.050	< 0.20	< 0.0050	0.032		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.14	< 0.2	< 0.030	< 0.0050	< 0.010	17		160	< 0.020
CMC-02R	Nov-94	< 0.050	< 0.0050	< 0.0050	0.040		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	0.050	< 0.20	< 0.030	< 0.0050	< 0.010	17		160	< 0.020
CMC-02R	Dec-94	< 0.050	< 0.0050	< 0.0050	0.041		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.33	< 0.20	< 0.030	< 0.0050	< 0.010	12		160	< 0.020
CMC-02R	Jan-95	< 0.050	< 0.0050	0.085	0.040		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.042	< 0.20	< 0.030	< 0.0050	< 0.010	12		140	< 0.020
CMC-02R	Feb-95	< 0.050	< 0.005	< 0.0050	0.034		0.00031	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.065	< 0.20	< 0.030	< 0.0050	< 0.010	10		150	< 0.020
CMC-02R	Mar-95	< 0.050	< 0.0050	< 0.0050	0.037		0.0003	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.042	< 0.20	< 0.030	< 0.0050	< 0.010	11		180	< 0.020
CMC-02R	Apr-95	< 0.050	< 0.0050	< 0.0050	0.036		0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.022	< 0.20	< 0.030	< 0.0050	< 0.010	10		160	< 0.020
CMC-02R	May-95	< 0.050	< 0.0050	< 0.0050	0.038		0.00013	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.020	< 0.20	< 0.030	< 0.0050	< 0.010	13		180	< 0.020
CMC-03P	May-94	< 0.200	< 0.050	< 0.005	0.022		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.047	< 0.2	< 0.030	< 0.005	< 0.010	3.00		125	< 0.020
CMC-03P	Jun-94	< 0.200	< 0.050	< 0.005	0.023		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.030	0.2	< 0.030	< 0.005	< 0.010	< 2.00		122	< 0.020
CMC-03P	Jul-94	< 0.200	< 0.050	< 0.005	0.002		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	0.020	< 0.2	< 0.030	< 0.005	< 0.010	2.00		123	< 0.020
CMC-03P	Aug-94	< 0.200	< 0.050	< 0.005	0.021		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.011	< 0.2	< 0.030	< 0.005	< 0.010	3.00		116	< 0.020
CMC-03P	Sep-94	< 0.050	< 0.20	< 0.0050	0.020		< 0.0001	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	0.011	< 0.2	< 0.030	< 0.0050	< 0.010	6.0		130	< 0.020
CMC-03P	Oct-94	< 0.050	< 0.20	< 0.0050	0.018		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	5.4		130	< 0.020
CMC-03P	Nov-94	< 0.050	< 0.0050	< 0.0050	0.021		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	5.7		160	< 0.020
CMC-03P	Dec-94	< 0.050	< 0.0050	< 0.0050	0.023		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.034	< 0.20	< 0.030	< 0.0050	< 0.010	2.3		130	< 0.020
CMC-03RR	May-94	< 0.200	< 0.050	< 0.005	0.027		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.016	< 0.2	< 0.030	< 0.005	< 0.010	15.0		129	< 0.020
CMC-03RR	Jun-94	< 0.200	< 0.050	< 0.005	0.024		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.011	< 0.2	< 0.030	< 0.005	< 0.010	11.0		193	< 0.020
CMC-03RR	Jul-94	< 0.200	< 0.050	< 0.005	0.023		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	12.0		125	< 0.020
CMC-03RR	Aug-94	0.206	< 0.050	< 0.005	0.021		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.001	< 0.010	0.3	< 0.030	< 0.005	< 0.010	11.0		120	< 0.020
CMC-03RR	Sep-94	< 0.050	< 0.20	< 0.0050	0.022		< 0.0001	< 0.005	< 0.030	< 20	0.058	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	14		130	< 0.020
CMC-03RR	Oct-94	< 0.050	< 0.20	< 0.0050	0.020		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	17		130	< 0.020
CMC-03RR	Nov-94	< 0.050	< 0.0050	< 0.0050	0.020		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	14		110	< 0.020
CMC-03RR	Dec-94	< 0.050	< 0.0050	< 0.0050	0.021		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	13		110	< 0.020
CMC-04	Jun-94	< 0.200	< 0.050	< 0.005	0.023		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.004	0.227	0.2	< 0.030	< 0.005	< 0.010	10.0		205	< 0.020
CMC-04	Jul-94	< 0.200	< 0.050	< 0.005	0.045		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.003	0.116	< 0.2	< 0.030	< 0.005	< 0.010	8.80		208	< 0.020
CMC-04	Aug-94	< 0.200	< 0.050	< 0.005	0.046		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.024	< 0.2	< 0.030	< 0.005	< 0.010	10.0		184	< 0.020
CMC-04	Sep-94	< 0.050	< 0.20	< 0.0050	0.048		< 0.0001	< 0.005	< 0.030	< 200	< 0.050	< 0.0020	0.031	< 0.2	< 0.030	< 0.0050	< 0.010	9.7		200	< 0.020
CMC-04	Oct-94	< 0.050	< 0.20	< 0.0050	0.019		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	1.1	0.3	< 0.030	< 0.0050	< 0.010	9.8		240	< 0.020
CMC-04	Nov-94	0.13	0.0068	0.0058	0.11		0.00015	0.13	0.11	< 20	0.15	< 0.0030	0.14	< 0.20	0.13	< 0.0050	< 0.010	11		280	0.15
CMC-04	Dec-94	< 0.050	< 0.0050	< 0.0050	0.048		0.0002	< 0.0050	< 0.030	< 20	0.084	< 0.0020	0.032	0.7	< 0.030	< 0.0050	< 0.010	8.1		190	< 0.020
CMC-04	Jan-95	< 0.050	0.0064	0.0055	0.044		0.00015	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.040	< 0.20	< 0.030	< 0.0050	< 0.010	6.3		190	< 0.020
CMC-04	Feb-95													< 0.20							
CMC-05	May-94	< 0.200	< 0.050	< 0.005	0.025		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.171	< 0.2	< 0.030	< 0.005	< 0.010	18.0		221	< 0.020
CMC-05	Jun-94	0.347	< 0.050	< 0.005	0.042		< 0.001	< 0.005	< 0.030	< 20	0.292	0.004	0.117	0.3	< 0.030	< 0.005	< 0.010	11.0		203	< 0.020

Table 2
Tailings Management Area
Groundwater Quality Statistical Analysis

Monitoring Well	Date	Aluminum mg/L	Antimony mg/L	Arsenic mg/L	Barium mg/L	Beryllium mg/L	Cadmium mg/L	Chromium mg/L	Copper mg/L	Cyanide ug/L	Iron mg/L	Lead mg/L	Manganese mg/L	Mercury ug/L	Nickel mg/L	Selenium mg/L	Silver mg/L	Sulfate mg/L	Thallium mg/L	TDS mg/L	Zinc mg/L
CMC-05	Jul-94	< 0.200	< 0.050	< 0.005	0.044		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.003	0.020	< 0.2	< 0.030	< 0.005	< 0.010	8.40		183	< 0.020
CMC-05	Aug-94	< 0.200	< 0.050	< 0.005	0.040		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.012	0.2	< 0.030	< 0.005	< 0.010	10.0		175	< 0.020
CMC-05	Sep-94	< 0.050	< 0.20	< 0.0050	0.040		0.0001	< 0.005	< 0.030	< 20	0.059	< 0.0020	< 0.010	< 0.2	0.037	< 0.0050	< 0.010	11		200	< 0.020
CMC-05	Oct-94	< 0.050	< 0.20	< 0.0050	0.037		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	11		240	< 0.020
CMC-05	Nov-94	< 0.050	0.0050	< 0.0050	0.039		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	10		230	< 0.020
CMC-05	Dec-94	< 0.050	< 0.0050	< 0.0050	0.038		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	7.7		150	< 0.020
CMC-05	Feb-95													< 0.20							
CMC-06P	May-94	< 0.200	< 0.050	< 0.005	0.014		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.038	< 0.2	< 0.030	< 0.005	< 0.010	9.00		100	< 0.020
CMC-06P	Jun-94	< 0.200	< 0.050	< 0.005	0.015		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.024	< 0.2	< 0.030	< 0.005	< 0.010	8.00		83.0	< 0.020
CMC-06P	Jul-94	< 0.200	< 0.050	< 0.005	0.015		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	0.015	0.3	< 0.030	< 0.005	< 0.010	9.40		85.0	< 0.020
CMC-06P	Aug-94	0.231	< 0.050	< 0.005	0.012		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.014	< 0.2	< 0.030	< 0.005	< 0.010	8.00		83.0	< 0.020
CMC-06P	Sep-94	< 0.050	< 0.20	< 0.0050	0.013		< 0.0001	< 0.005	< 0.030	< 20	0.11	< 0.0020	0.011	< 0.2	< 0.030	< 0.0050	< 0.010	9.6		100	< 0.020
CMC-06P	Oct-94	< 0.050	< 0.20	< 0.0050	0.012		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	10		99	< 0.020
CMC-06P	Nov-94	< 0.050	< 0.0050	0.006	0.014		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	0.016	< 0.20	< 0.030	< 0.0050	< 0.010	10		130	< 0.020
CMC-06P	Dec-94	< 0.050	< 0.0050	< 0.0050	0.015		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.010	< 0.20	< 0.030	< 0.0050	< 0.010	8.8		81	< 0.020
CMC-07	May-94	< 0.200	< 0.050	< 0.005	0.012		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	0.012	< 0.2	< 0.030	< 0.005	< 0.010	20.0		219	< 0.020
CMC-07	Jun-94	< 0.200	< 0.050	< 0.005	0.041		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	13.0		185	< 0.020
CMC-07	Jul-94	< 0.200	< 0.050	< 0.005	0.033		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	< 0.010	0.4	< 0.030	< 0.005	< 0.010	11.0		180	< 0.020
CMC-07	Aug-94	< 0.200	< 0.050	< 0.005	0.036		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	13.0		166	< 0.020
CMC-07	Sep-94	< 0.050	< 0.20	< 0.0050	0.030		< 0.0001	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	13		180	< 0.020
CMC-07	Oct-94	< 0.050	< 0.20	< 0.0050	0.032		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	14		220	< 0.020
CMC-07	Nov-94	< 0.050	< 0.0050	< 0.0050	0.027		0.00014	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	12		180	0.021
CMC-07	Dec-94	< 0.050	< 0.0050	< 0.0050	0.026		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	11		180	< 0.020
CMC-07	Feb-95													< 0.20							
DMB-1A	Jun-94	< 0.200	< 0.050	< 0.005	0.025		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.003	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	9.00		171	< 0.020
DMB-1A	Jul-94	< 0.200	< 0.050	< 0.005	0.021		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	9.70		186	< 0.020
DMB-1A	Aug-94	0.203	< 0.050	< 0.005	0.020		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	12.0		167	< 0.020
DMB-1A	Sep-94	< 0.050	< 0.20	< 0.0050	0.019		< 0.0001	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	12		180	< 0.020
DMB-1A	Oct-94	< 0.050	< 0.20	< 0.0050	0.019		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	13		200	< 0.020
DMB-1A	Nov-94	< 0.050	< 0.0050	< 0.0050	0.019		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	12		230	< 0.020
DMB-1A	Dec-94	< 0.050	< 0.0050	< 0.0050	0.020		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	11		240	< 0.020
DMB-1A	Jan-95	< 0.050	< 0.0050	0.086	0.019		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	10		200	< 0.020
DMB-1A	Feb-95													< 0.20							
DMB-6	May-94	< 0.200	< 0.050	< 0.005	0.018		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	16.0		200	< 0.020
DMB-6	Jun-94	< 0.200	< 0.050	< 0.005	0.026		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.003	0.072	0.3	< 0.030	< 0.005	< 0.010	14.0		175	< 0.020
DMB-6	Jul-94	< 0.200	< 0.050	< 0.005	0.025		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	16.0		178	< 0.020
DMB-6	Aug-94	< 0.200	< 0.050	< 0.005	0.023		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	0.3	< 0.030	< 0.005	< 0.010	19.0		167	< 0.020
DMB-6	Sep-94	< 0.050	< 0.20	< 0.0050	0.023		0.0010	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	18		190	< 0.020
DMB-6	Oct-94	< 0.050	< 0.20	< 0.0050	0.022		0.00033	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	22		200	< 0.020
DMB-6	Nov-94	< 0.050	< 0.0050	< 0.0050	0.024		0.00021	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	21		250	< 0.020
DMB-6	Dec-94	< 0.050	< 0.0050	0.0051	0.023		0.00013	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	15		160	< 0.020
EX-13BU	May-94	< 0.200	< 0.050	< 0.005	0.012		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	0.2	< 0.030	< 0.005	< 0.010	4.00		141	< 0.020
EX-13BU	Jun-94	< 0.200	< 0.050	< 0.005	0.032		< 0.001	< 0.005	< 0.030	< 20	0.143	0.002	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	< 2.00		138	< 0.020
EX-13BU	Jul-94	< 0.200	< 0.050	< 0.005	0.014		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.013	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	3.80		143	< 0.020
EX-13BU	Aug-94	0.226	< 0.050	< 0.005	0.011		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	6.00		127	< 0.020
EX-13BU	Sep-94	< 0.050	< 0.20	< 0.0050	0.012		< 0.0001	< 0.005	< 0.030	< 20	0.21	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	7.3		140	< 0.020
EX-13BU	Oct-94	< 0.050	< 0.20	< 0.0050	0.012		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	7.1		160	< 0.020
EX-13BU	Nov-94	< 0.050	< 0.0050	< 0.0050	0.012		0.00018	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	7.7		160	< 0.020
EX-13BU	Dec-94	< 0.050	< 0.0050	< 0.0050	0.014		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	6.3		160	< 0.020
EX-9BL	May-94	< 0.200	0.052	< 0.005	0.019		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	10.0		139	< 0.020
EX-9BL	Jun-94	< 0.200	< 0.050	< 0.005	0.021		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	8.00		128	< 0.020
EX-9BL	Jul-94	< 0.200	< 0.050	< 0.005	0.023		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.001	< 0.010	1	< 0.030	< 0.005	< 0.010	9.20		141	< 0.020
EX-9BL	Aug-94	< 0.200	< 0.050	< 0.005	0.018		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	10.0		142	< 0.020
EX-9BL	Sep-94	< 0.050	< 0.20	< 0.0050	0.019		< 0.0001	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	14		130	< 0.020
EX-9BL	Oct-94	< 0.050	< 0.20	< 0.0050	0.019		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	13		170	< 0.020
EX-9BL	Nov-94	< 0.050	< 0.0050	< 0.0050	0.020		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	12		170	< 0.020
EX-9BL	Dec-94	< 0.050	< 0.0050	< 0.0050	0.020		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	11		150	< 0.020
EX-9BU	May-94	< 0.200	< 0.050	< 0.005	0.011		< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	0.2	< 0.030	< 0.005	< 0.010	12.0		129	< 0.020
EX-9BU	Jun-94	< 0.200	< 0.050	< 0.005	0.014		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	10.0		115	< 0.020

Table 2
Tailings Management Area
Groundwater Quality Statistical Analysis

Monitoring Well	Date	Aluminum mg/L	Antimony mg/L	Arsenic mg/L	Barium mg/L	Beryllium mg/L	Cadmium mg/L	Chromium mg/L	Copper mg/L	Cyanide ug/L	Iron mg/L	Lead mg/L	Manganese mg/L	Mercury ug/L	Nickel mg/L	Selenium mg/L	Silver mg/L	Sulfate mg/L	Thallium mg/L	TDS mg/L	Zinc mg/L
EX-9BU	Jul-94	< 0.200	< 0.050	< 0.005	0.011		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.003	< 0.010	0.3	< 0.030	< 0.005	< 0.010	11.0		128	< 0.020
EX-9BU	Aug-94	< 0.200	< 0.050	< 0.005	0.012			< 0.005	< 0.030	< 20	< 0.100	< 0.001	< 0.010	< 0.2	< 0.030	< 0.005	< 0.010	12.0		134	< 0.020
EX-9BU	Sep-94	< 0.050	< 0.20	< 0.0050	0.011		< 0.0001	< 0.005	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	17		140	< 0.020
EX-9BU	Oct-94	< 0.050	< 0.20	< 0.0050	0.010		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.2	< 0.030	< 0.0050	< 0.010	15		160	< 0.020
EX-9BU	Nov-94	< 0.050	< 0.0050	< 0.0050	0.012		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	13		150	< 0.020
EX-9BU	Dec-94	< 0.050	< 0.0050	< 0.0050	0.012		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	< 0.010	< 0.20	< 0.030	< 0.0050	< 0.010	12		130	< 0.020
G41-H18B	Jun-94	< 0.200	< 0.050	< 0.005	0.023		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.003	1.08	< 0.2	< 0.030	< 0.005	< 0.010	4.00		150	< 0.020
G41-H18B	Jul-94	< 0.200	< 0.050	< 0.005	0.021		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.003	0.918	1	< 0.030	< 0.005	< 0.010	5.10		161	< 0.020
G41-H18B	Aug-94	< 0.200	< 0.050	< 0.005	0.019		< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.001	0.890	< 0.2	< 0.030	< 0.005	< 0.010	10.0		131	< 0.020
G41-H18B	Sep-94	< 0.050	< 0.20	< 0.0050	0.020		< 0.0001	< 0.005	< 0.030	< 200	0.63	< 0.0020	1.1	< 0.2	< 0.030	< 0.0050	< 0.010	7.7		160	< 0.020
G41-H18B	Oct-94	< 0.050	< 0.20	< 0.0050	0.049		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	0.031	< 0.2	< 0.030	< 0.0050	< 0.010	9.3		170	< 0.020
G41-H18B	Nov-94	< 0.050	< 0.0050	< 0.0050	0.019		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0030	1.5	< 0.20	< 0.030	< 0.0050	< 0.010	8.8		110	0.024
G41-H18B	Dec-94	< 0.050	< 0.0050	< 0.0050	0.020		< 0.0001	< 0.0050	< 0.030	< 20	0.092	< 0.0020	1.9	< 0.20	< 0.030	< 0.0050	< 0.010	6.0		180	< 0.020
G41-H18B	Jan-95	< 0.050	< 0.0050	< 0.0050	0.021		< 0.0001	< 0.0050	< 0.030	< 20	< 0.050	< 0.0020	1.5	< 0.20	< 0.030	< 0.0050	< 0.010	4.4		130	< 0.020

Table 2
Tailings Management Area
Groundwater Quality Statistical Analysis

Monitoring Well / Date	Aluminum mg/L	Antimony mg/L	Arsenic mg/L	Barium mg/L	Beryllium mg/L	Cadmium mg/L	Chromium mg/L	Copper mg/L	Cyanide ug/L	Iron mg/L	Lead mg/L	Manganese mg/L	Mercury ug/L	Nickel mg/L	Selenium mg/L	Silver mg/L	Sulfate mg/L	Thallium mg/L	TDS mg/L	Zinc mg/L
# Samples	126	126	126	126	-	127	126	126	126	126	126	126	132	126	126	126	126	-	126	126
% Detections	6.3%	3.1%	6.3%	99.2%	-	13.4%	0.8%	0.8%	0.0%	7.8%	23.4%	43.8%	18.2%	1.6%	0.0%	0.0%	98.4%	-	100.0%	3.1%
Minimum	< 0.05	< 0.005	< 0.005	< 0.01	-	< 0.0001	< 0.005	< 0.03	< 20	< 0.05	< 0.001	< 0.01	< 0.2	< 0.03	< 0.005	< 0.010	< 2	-	81	< 0.02
Maximum	0.347	0.052	0.086	0.11	-	0.001	0.13	0.11	< 200	0.63	0.017	1.9	1	0.13	< 0.005	< 0.010	70	-	339	0.15
Average (1)	< 0.050	< 0.050	< 0.005	0.023	-	< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.002	< 0.010	< 0.20	< 0.030	< 0.005	< 0.010	11.9	-	162.3	< 0.020
Source Term (2)	0.08	0.002	0.01	0.03	< 0.0001	0.01	0.02	0.07	< 80	0.8	0.1	1.4	0.1	0.02	0.002	0.002	2500	< 0.0001	3100	1.1
Relative Concentration (3)																				
Best Eng. Judge.	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011
Practical Worst Case	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026	0.026
Incremental Source Term																				
At Compliance Boundary (4)																				
Best Eng. Judge.	0.001	2.2E-05	0.0001	0.0003	< 1.1E-06	0.0001	0.0002	0.001	< 0.68	0.009	0.0011	0.015	0.001	0.0002	2.2E-05	2.2E-05	27.5	< 1.1E-06	34.1	0.012
Practical Worst Case	0.002	0.0001	0.0003	0.001	< 2.6E-06	0.0003	0.0005	0.002	< 2.08	0.021	0.0026	0.036	0.003	0.001	5.2E-05	5.2E-05	65.0	< 2.6E-06	80.6	0.029
Upper 95% Confidence Limit	< 0.200	< 0.050	< 0.005	0.025	-	< 0.001	< 0.005	< 0.030	< 20	< 0.100	< 0.002	0.011	< 0.2	< 0.030	< 0.005	< 0.010	13.0	-	168.3	< 0.020
Method	Non-Parametric	Non-Parametric	Non-Parametric	Normal		Non-Parametric	Non-Parametric	Non-Parametric	Non-Parametric	Non-Parametric	Non-Parametric	Non-Parametric	Non-Parametric	Non-Parametric	Non-Parametric	Non-Parametric	Normal		Normal	Non-Parametr
Estimated Parameter Concentration (5)																				
Best Eng. Judge.	< 0.200	< 0.050	< 0.005	0.025	-	< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.002	0.026	< 0.2	< 0.030	< 0.005	< 0.010	40	-	202	0.022
Practical Worst Case	< 0.200	< 0.050	< 0.005	0.026	-	< 0.001	< 0.005	< 0.030	< 20	< 0.100	0.004	0.047	< 0.2	< 0.030	< 0.005	< 0.010	78	-	249	0.039
MCL (6)	0.2(8)	0.006	0.05	2	0.004	0.005	0.1	1.3	200	0.3(8)	0.015	0.05(8)	2	0.1	0.05	0.1(8)	250(8)	0.002	500(8)	5(8)
ACL (7)		0.070										1.5								
Method		Poisson										Non-Parametric								

Note: No baseline data exists for Beryllium or Thallium.

Prepared by: SGL
Checked by: MHZ

(1) If at least 85% values were above detection limit, average was calculated replacing non-detects with one-half the detection limit. If between 50% and 85% of the values were detected, the average was calculated using Cohen's adjustment. If less than 50% of the values were detected, the median is given rather than the average.

(2) See the October 18, 1995 TMA Source Term Technical Memorandum by Lawrence and Thresher.

(3) From GeoTrans, Inc. Regional Groundwater Modeling Study (See Section 4 and Appendix 4.2-3 of EIR).

(4) Product of source term value multiplied by the relative cooncentration.

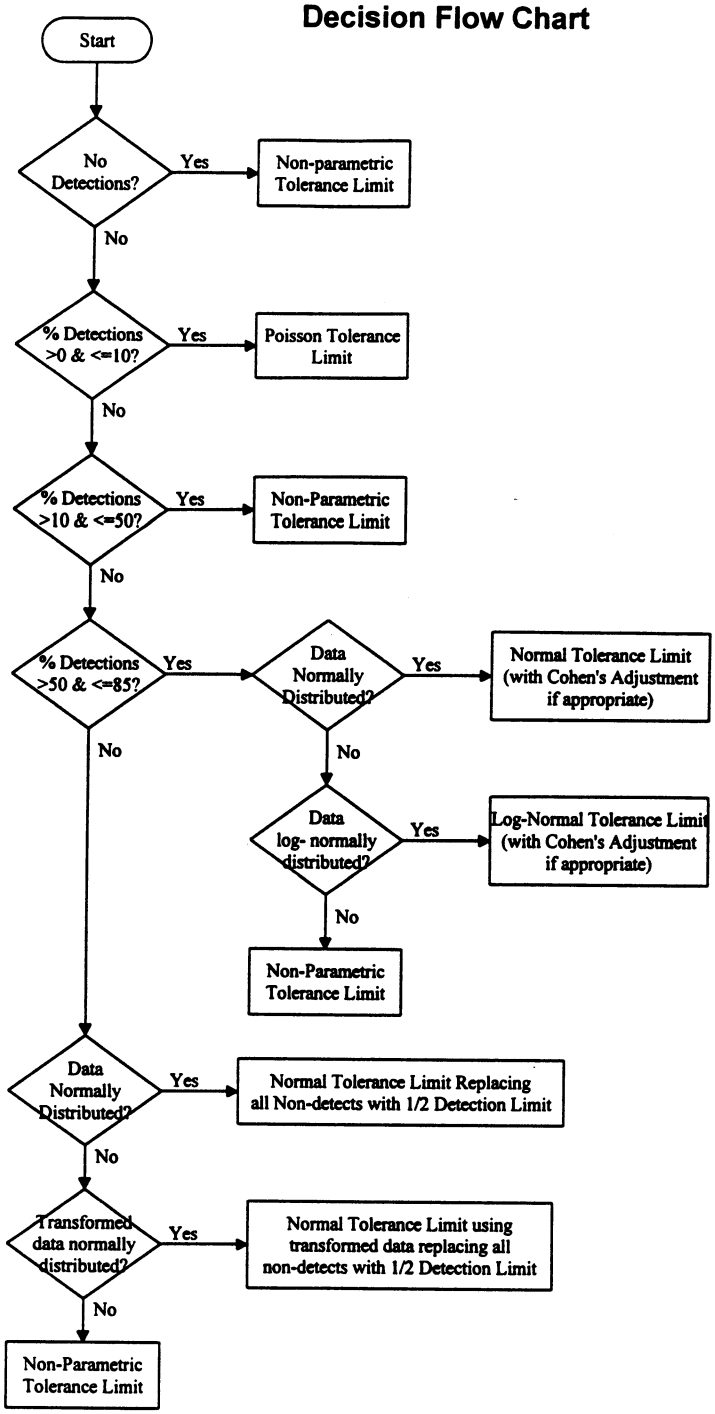
(5) Equals the incremental source term concentration at the compliance boundary plus the upper 95% confidence limit of the mean. For confidence limit below detection the incremental source term was added to one-half the detection limit. For those cases where the estimated parameter concentration was still less than the detection limit, the estimated parameter concentration was set at the detection limit.

(6) NR 809 Maximum Contaminant Limit.

(7) Alternative Concentration Level calculated statistically using baseline water quality data.

(8) Value is secondary MCL.

Statistical Tolerance Limit Decision Flow Chart



Foth & Van Dyke

REVISED	DATE	BY	DESCRIPTION

CHECKED BY:	JKSI	DATE: SEPT.'95
APPROVED BY:	SGL	DATE: SEPT.'95
APPROVED BY:	GWS	DATE: SEPT.'95

Crandon Mining Company

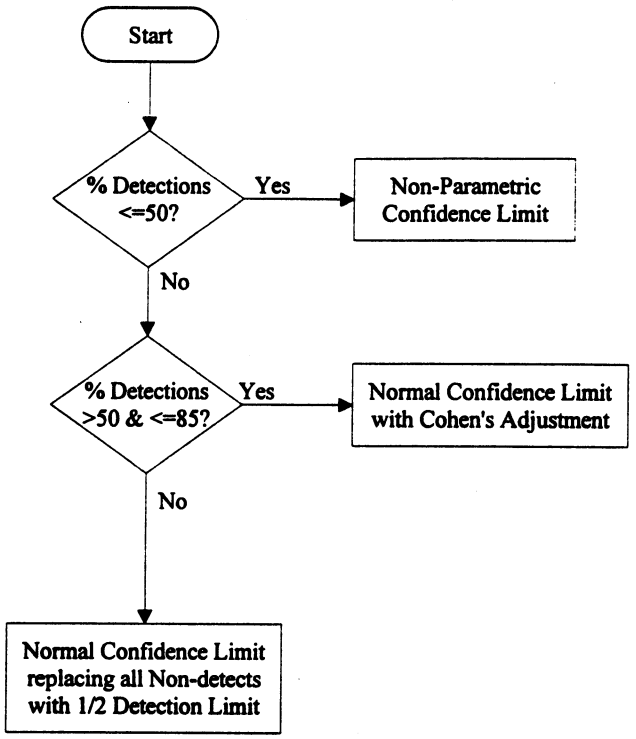
FIGURE 1
STATISTICAL TOLERANCE LIMIT
DECISION FLOW CHART

Scale:	NO SCALE	Date:	SEPTEMBER, 1995
Prepared By:	Foth & Van Dyke	By:	JPRI

/d3/dl/users/jpr1/3c49f160.dgn

93C049

Statistical Confidence Limit Decision Flow Chart



Foth & Van Dyke				
REVISED	DATE	BY	DESCRIPTION	
CHECKED BY:	JKSI			DATE: SEPT.'95
APPROVED BY:	SGL			DATE: SEPT.'95
APPROVED BY:	GWS			DATE: SEPT.'95

Crandon Mining Company

FIGURE 2
STATISTICAL CONFIDENCE LIMIT
DECISION FLOW CHART

Scale:	NO SCALE	Date:	SEPTEMBER, 1995
Prepared By:	Foth & Van Dyke	By:	JPRI

/d3/dl/users/jpr1/3c49f160.dgn

93C049

Attachment 1

Example Calculations for Tolerance Limits

Foth & Van Dyke

Client: Crandon Mining Co. Scope I.D.: 936049
Project: Performance Prediction - TMA Refloated Mine Page: 1
Prepared by: SGL Date: 9/12/95
Checked by: MHZ Date: 9/25/95

Example Calculation

Poisson Tolerance Limit - Aluminum (TMA)

Poisson Count = T_n = 85.9 (see footnote)

(Original data multiplied by a factor of 10 to improve accuracy of Poisson count)

$n = 128$

$\lambda_{T_n} = \frac{1}{2(128)} \cdot \chi^2_{.95}\left[2(85.9)+2\right]$

$= \frac{1}{256} \cdot 205.7 = 0.804$

$\therefore \quad \chi^2_{.05}\left[2k+2\right] \geq 1.608$

$\Rightarrow 2k+2 = 6$ (based on χ^2 table)

$\therefore k = 2$

Since original data was multiplied by a factor of 10, k is divided by 10

$\therefore$ Upper tolerance limit = $2/10$ = **0.2**

Note: $\chi^2_p[n]$ = upper p^{th} percentile of Chi-Square distribution with n degrees of freedom. (To be taken from chi-square table)

T_n and λ_{T_n} are defined in detail on pp. 38-40 of USEPA (1992).

Foth & Van Dyke

Client: Cyprus Mining Co. Scope I.D.: 93C049
Project: Performance Prediction - TMA Refloaded Mine Page: 2
Prepared by: SGL Date: 9/12/95
Checked by: MWR2 Date: 9/25/95

Example Calculation

Non-parametric Tolerance Limit - Manganese (TMA)

$n = 128$

From EPA (1992) Table A-6 it is seen that with 128 values, the second largest provides 95% confidence for 95% coverage of a non-parametric tolerance limit.

Second largest value = 1.5 mg/L
(see G41-H18B, Nov. '94 and June '95)

∴ Upper tolerance limit = **1.5 mg/L**

Reference

USEPA, 1992. Statistical Analysis of Ground-Water Monitoring Data - Addendum to Interim Final Guidance. Office of Solid Waste, Washington D.C.

Foth & Van Dyke

Client: Cravden Mining Co Scope I.D.: 93C049
Project: Performance Prediction - TMA Refloated mine Page: 3
Prepared by: SGC Date: 4/12/95
Checked by: MER Date: 9/25/95

Example Calculation

Upper Confidence Limit (Normal Distribution)

Barium (TMA)

$\bar{X}^* = 0.0232$

$S^* = 0.0130$

$n = 128$ $\quad z_{.95} = 1.645$

*Non-detects replaced with ½ detection limit

Upper confidence limit

$= 0.0232 + 1.645\left(\frac{.0130}{\sqrt{128}}\right)$ (See foot note)

$= \boxed{0.025 \text{ mg/L}}$

Note $\bar{X}$ = mean of data

S = standard deviation of data

n = sample size

$z_{.95}$ = upper 95th percentile of standard normal distribution

Note: See formula on pp 6-3 through 6-5 of USEPA (1989)

Foth & Van Dyke

Client: Cranberry Mining Co. Scope I.D.: 93C049
Project: Performance Prediction - TMA Refloaded mine Page: 4
Prepared by: SGL Date: 9/12/95
Checked by: MJR2 Date: 9/25/95

Example Calculation

Upper Confidence Limit (Normal Distribution - Cohen's Adjustment)

Manganese (Refloaded Mine)

$\bar{X}_{detected} = 0.1649$ Detection Limit = 0.01

$s^2_{detected} = 0.0156$

$n = 48$ $n_{detected} = 25$

$\therefore h = \frac{48-25}{48} = 0.479$ (see foot note)

$\gamma = \frac{0.0156}{(0.1649-0.01)^2} = 0.650$

$\Rightarrow \lambda = 0.958$

$\bar{X}_{adjusted} = 0.1649 - 0.958(0.1649 - .01)$

$= 0.0165$

$s_{adjusted} = \sqrt{0.0156 + 0.958(0.1649 - .01)^2}$

$= 0.196$

$\therefore$ Upper Confidence Limit

$= 0.0165 + \left(\frac{0.196}{\sqrt{48}}\right) 1.645$

$= \boxed{0.063 \text{ mg/L}}$

Note: $\bar{x}_{detected}$ = mean of all data above the detection limit

$s^2_{detected}$ = variance of all data above the detection limit

$n_{detected}$ = sample size of all data above the detection limit.

k, γ, λ, $\bar{x}$ adjusted and s adjusted are defined in detail on pp. 8-6 through 8-10 of USEPA (1989).

Foth & Van Dyke

Client: Cranden Mining Co. Scope I.D.: 93C049
Project: Performance Prediction - TMA Page: 6
Prepared by: SGL Reflooded mine Date: 9/12/95
Checked by: MHR Date: 9/25/95

Example Calculation

Non-Parametric Confidence Limit - Aluminum (TMA)

$n = 128$

upper limit is found as the u^{th} position in the ordered data set where

$$u = \frac{n+1}{2} + \frac{z_{.95}\sqrt{n}}{2}$$ (See Gilbert, 1987 pp 173-174)

$$= \frac{128+1}{2} + \frac{1.645\sqrt{128}}{2}$$

$$= 74^{th} \text{ position}$$

In the ordered data set this corresponds to a value of $\boxed{< 0.2 \text{ mg/L}}$

Memorandum

October 18, 1995

TO: Jerry Sevick, Foth & Van Dyke

CC: Don Moe, Crandon Mining Company
Peter Andersen, GeoTrans, Inc.
Master File

FR: Richard W. Lawrence, Lawrence Consulting Limited
John E. Thresher, Jr., Thresher & Son, Inc.

RE: Crandon Project - TMA Source Term

Following is a discussion of the derivation of the source term for the TMA. We understand that the source term information for the TMA is to be used in conjunction with the project's solute transport modeling work to assess compliance with groundwater standards.

Background

The tailings management area (TMA) at the proposed Crandon Project site will consist of four basins that will be successively filled with fine grained mineral waste referred to as tailings. Tailings will be produced during the primary beneficiation of the ore, a process that will produce ore mineral concentrates. The ore will be crushed within the underground mine and transported to the surface. In the concentrator, the ore will be ground to a fine sized material and pumped as a slurry to flotation cells which selectively recover the ore minerals by separating them from tailings.

The fine grained waste generated during the flotation process will be separated into two size fractions by cycloning. The coarse-sized tailings fraction will be used, when required, to backfill depleted ore stopes within the mine. The fine tailings fraction will be slurried with process water and pumped to one of the four TMA cells. During times when coarse tailings are not required for backfilling depleted stopes, whole tailings will be pumped to the TMA. It is expected that 10 to 20 percent of the tailings deposited in the TMA will consist of whole tailings.

Each of the TMA cells will be lined with a composite liner consisting of a low permeability soil upon which a flexible geomembrane liner will be placed. The geomembrane liner will be covered by a layer of till removed from the site during cell construction. The base of each cell will also contain a layer of gravel sandwiched between the geomembrane liner and the till to facilitate drainage and collection of leachate percolating down through the accumulated tailings mass. The base of the cells will be sloped to enhance leachate collection.

Slurried tailings will be added to the TMA cells using a ring spigot system. The initial filling of the cells will be confined to the low areas at two of the cell corners, after which deposition from the ring spigot system will produce accumulated tailings that are higher at the margins of the cells and lower in the center of the cells. During this mode of deposition, process water will be

ponded centrally and will cover upwards of 60 percent of the surface of each cell. The ponded process water will have a minimum depth of three to five feet.

Physical testing of a Crandon tailings composite produced during locked-cycle metallurgical bench testing was conducted by the Colorado School of Mines Research Institute (CSMRI) in 1982 (CSMRI, 1982). A portion of these tests determined that the hydraulic conductivity of the settled tailings would be expected to be about 5×10^{-6} cm/sec. Testing of a second tailings composite by Steffen Robertson Kirsten (SRK, Lakewood, Colorado) in 1994 confirmed the earlier studies and concluded that the expected hydraulic conductivity of the tailings would be about 1×10^{-6} cm/sec. The descriptions of the tests and resulting data are included in the project's Environmental Impact Report (EIR)(FVD, 1995a) and its Tailings Management Area - Feasibility Report/Plan of Operations (FVD, 1995b).

During the operational life of each cell, excess process water will be pumped from the surface of the active TMA cell to a reclaim pond before being recycled to the milling/flotation process. For this reason, the central portion of each cell will remain saturated during the operational life of each cell. The tailings in the marginal areas of each cell will have moisture contents less than saturation but will be covered by a successive layer of tailings for each rotation of the ring spigot discharge. The design period of rotation is approximately two months but will vary according to climatic conditions.

Each TMA cell will be covered after filling. Prior to the placement of the cover, a grading layer of material will be added to the top of the tailings to produce a domed surface to promote surface drainage. The cover system will be a composite barrier system comprised of a low permeability soil placed on top of the grading layer, which in turn will be covered by a flexible geomembrane liner. Individual layers of gravel, rooting zone soil and topsoil will be placed on top of the geomembrane. As discussed in the TMA Feasibility Report/Plan of Operation (FVD, 1995b), this cover system will greatly restrict the transport of water and air (oxygen) into the deposited tailings.

Under post-closure conditions, the quantity of leachate that could potentially percolate from the base of the cell would be approximately equal to the quantity entering through the top cover. Due to the low permeability of the cover system, this quantity of leachate, as reported in the TMA Feasibility Report/Plan of Operation (FVD, 1995b) is estimated to be very small.

Geochemical Testing of the TMA Tailings Materials

In 1994 tailings materials were generated by Lakefield Research, Lakefield, Ontario (Lakefield) during locked cycle metallurgical bench testing of ore composites from the Crandon deposit (FVD, 1995a). Lakefield separated the whole tailings generated during the locked cycle testing into fine- and coarse-sized fractions. The tailings fractions were shipped in 1994 to Foth & Van Dyke at Green Bay, Wisconsin for testing. Composites of fine, coarse and recombined whole tailings are being tested. Details and results of the test program obtained through May 1995 are presented in the EIR (FVD, 1995a). Results to date will be presented in a waste characterization study status report that will be issued shortly. Final results will be presented in the project's final waste characterization report.

Fine and whole tailings from Lakefield are being evaluated for their leaching characteristics in tests designed to provide data for the performance of the TMA under the design conditions and data to evaluate sulfide oxidation rates and neutralization depletion rates.

In the current CMC test program, fine tailings and whole tailings have been leached under conditions which provide data to allow a prediction of the leachate quality (the source term) under the proposed TMA operating conditions. Both fine and whole tailings were initially leached for four months under saturated conditions using process water from metallurgical testing. This has been followed by leaching under alternating wet/dry conditions using deionized water. This second leaching step has progressed for over four months to date. These tests are continuing. In addition, whole tailings, with and without lime addition, have been subjected to alternating wet/dry cycles for ten months to date to observe the effect of lime amendment on leachate quality. Future test data from these ongoing tests can be used to refine the prediction of the source term, if required.

An earlier series of tests were conducted by the CSMRI (1982). In the CSMRI study, the leachability of several tailings composites was tested in agitated and static (column) tests conducted under neutral, alkaline and acidic conditions. For the column tests, a comparison of wet/dry cycles with saturated (submerged) conditions was carried out. The agitated tests (three cycles of 48 hours) were conducted to provide a practical worst case leachate that could develop as the tailings are in the process of being slurried and deposited within the TMA. The column tests were conducted for 31 weeks to provide a range of potential long term conditions that could occur within the deposited tailings. For the purposes of the TMA source term analysis, the data from the tests on fine zinc tailings (referred to as massive ore) were used. Although these tailings were obtained from an ore composite different from that used in the 1994 Lakefield locked cycle tests, the values obtained from the CSMRI testing under various conditions are valid, based on bulk chemical analyses, to compare with and complement the data from CMC's current waste characterization program.

Derivation of the TMA Source Term

Under the proposed operating conditions of the TMA, significant oxidation of the tailings is unlikely due to: (a) extremely slow oxidation rates in the alkaline process water, (b) saturation and ponding of a significant proportion of the tailings during operations, (c) limited exposure time of unponded tailings before being covered by subsequent deposition events, (d) sufficient carbonate content of the tailings to maintain alkaline conditions while tailings are exposed, and (e) with each successive layer of tailings, oxygen transport into the bulk of the tailings mass becomes less, and is essentially insignificant below the very shallow air-entry zone. In these conditions, leachate quality can be expected to be the result of the non-oxidative reaction between the tailings and process water. During operations, therefore, leachate within the TMA can be expected to remain neutral to alkaline and contain quantities of metals representing near-equilibrium conditions between the tailings and the pore water. During the post-closure period similar leachate characteristics are anticipated since the TMA has been designed to limit oxygen and water entry. These topics will be discussed in greater detail in the project's final waste characterization report.

Derivation of the major ion equilibrium and kinetic non-equilibrium chemistry in a variably saturated porous medium such as a tailings mass could provide the basis for an extremely

complex modeling exercise due to the number of equilibrium chemical reactions such as complexation, cation exchange, and precipitation-dissolution which can take place. Rigorous modeling to predict water quality requires consideration of water flow and heat transport, gas phase concentrations and solute movement as well as the chemical processes for the solids and solutes of interest. Instead of attempting such a detailed analysis a simple derivation of the source term has been completed based on the experimental data obtained in laboratory testing. This simpler approach, presented below, is intended to provide a conservative estimate of water quality and largely ignores complex thermodynamic and kinetic considerations which, if taken into account, would likely provide a lower concentration of most of the ions in the source term.

In developing a solute transport source term for the TMA, results from the current CMC waste characterization test program, results from the test program conducted by the CSMRI (1982) and the analysis of process water from locked-cycle metallurgical testing, reported in Appendix 3.5-32 of the EIR (FVD, 1995a), have been considered.

In the CMC and CSMRI leach tests, the protocols used are typical of standard leaching kinetic tests in which solids are leached with flushes of low ionic strength leachate, typically deionized water, weakly acid or weakly alkaline solutions, in periodic cycles. In such tests, the volume of water added is such that solubilities of ions are rarely exceeded and the products of leaching are removed with each flush. If the solids are free draining and drain completely between each flush cycle, then conditions conducive to the oxidation of contained sulfides are favored, which can lead to the onset of acidic conditions with increased metal loadings in the leachate at each flushing event. The regular removal of leachate does not allow the development of higher-ionic strength or saturated solutions, with respect to specific ions within the pore spaces, and therefore promotes higher leaching kinetics. In practice, the chemistry of pore water in a tailings mass will, in time, reach near-equilibrium so that leaching rates are retarded to a significant degree compared to those obtained in a free draining mass leached with low-ionic strength water. Under such conditions, the complex thermodynamic and kinetic considerations, as previously discussed, would apply. The simplified approach used here essentially assumes that everything that dissolves under the conditions of laboratory testing will be contained in the leachate generated by the TMA and is therefore a worst case scenario.

For the current CMC test programs, the cumulative mass flux of individual cations and anions have been calculated by adding the amounts removed by each flushing cycle. This provides an indication of the total amounts that could be leached from the tailings over long time periods and allows a conservative assessment of sulfide oxidation rates, neutralization rates and metals leaching rates. It should be recognized, however, that the total amounts of individual ions removed over the duration of the tests cannot be used to calculate a water quality for the source term, since the concentrations so derived will exceed solubility limits for many species. In calculating the source term, therefore, the cumulative mass flux of ions obtained from the initial period of testing have been used. Even so, a rigorous analysis of the derived source term could reveal that solubilities for some species are exceeded. It is considered, however, that the values obtained using the simplified approach do represent a practical worst case, and, as previously discussed, a more rigorous analysis is not justified.

Tables 1 and 2 provide test data and calculated values leading to the derivation of a solute transport source term for the TMA. Table 1 provides an analyses of tailings expressed in mg/kg. Analyses for fine and coarse tailings (CMC 94) are from Lakefield Research data as reported in Appendix 3.5-32 of the project's EIR (FVD, 1995a). The analysis for whole tailings is calculated from the fine and coarse tailings analyses, assuming equal weight fractions of the two

components. The CSMRI massive tailings analysis is from Appendix E of the CSMRI (1982) Report.

In Table 2, leaching test results, expressed in mg/kg, from fine and whole tailings used in the current CMC waste characterization program, and from the neutral water leach test conducted on the zinc tailings in the CSMRI study (Table I-2, CSMRI, 1982) are provided. For the tests on TMA fine tailings and TMA whole tailings, the cumulative mass fluxes after 132 days of testing are used. After this period of time, the tailings had been subjected to two flushes under saturated conditions (91 days) and two flushes under free draining, unsaturated conditions (41 days). The mass flux data provided for whole tailings leached under free-draining, wet/dry cycles are the cumulative values from ten flush cycles over the initial 63 days of testing. For some parameters negative values are shown in Table 2. This indicates that there is a net loss of the parameter from the solution into the tailings.

The CSMRI data shown in Table 2 are included for comparison purposes and consist of the total fluxes calculated from the leachate quality data presented in Table J-6 of their 1982 report, which were obtained in a 31-week test under submerged conditions. It can be noted in the CSMRI report that leachate quality and mass flux values for tests conducted under different neutral, acidic and alkaline saturated conditions, and under free draining conditions, are very similar.

Table 2 also includes water quality data for process water generated as part of the Lakefield locked-cycle (30 cycles) metallurgical testing and the corresponding calculated mass flux values based on 13.4 percent solids in the tailings slurry entering the TMA. This percent solids was calculated based on the proportion of 80/20 of fine tailings/whole tailings expected to be disposed of in the TMA, assuming 12 percent and 25 percent solids in the fine tailings and whole tailings slurries, respectively. This sample of process water can be considered to be a test leachate arising from multiple contacts of recycle tailings water with ore and tailings during the extensive locked-cycle test. This water quality can be regarded as being typical of the expected analysis of the water to be discharged with tailings solids from the concentrator to the TMA during operations.

An analysis of the mass flux values for parameters obtained in the various tests shown in Table 2 allows an estimation to be made for the TMA source term. This estimation is shown in Table 3. In the table the estimated analysis of tailings in the TMA has been calculated from the values shown in Table 1 based on the assumption that the tailings will comprise 80 percent fine tailings (12 percent solids) and 20 percent whole tailings (25 percent solids). The estimated extraction of parameters in mg/kg is based on the values shown in Table 2 for the various tests. In the general case, the estimated extraction has been calculated by adding 80 percent of the value for fine tailings and 20 percent of the highest value for whole tailings. In cases when one or more values were available, the highest value was taken. Based on these values, the leachate analysis for the TMA has been calculated assuming a percent solids content for the incoming tailings slurry of 13.4 percent, which is the net value calculated from the 80/20 ratio of fine tailings and whole tailings at the above-stated solids contents. In the case of the Group I and II metals, sodium, potassium, calcium, and magnesium, for which relatively high solubility can be expected in a sulfate system, higher concentrations have been estimated together with sulfate. Estimates are based on the concentrations observed in the tests on TMA fine and whole tailings and with consideration to the solubility of calcium and sulfate which are likely to be the dominant ions in this group. For cyanide, a value of <0.08 mg/L total cyanide was obtained in testing of tailings for backfill. This parameter was not detected in tests on TMA tailings. Total dissolved solids (TDS) analysis is an estimated value. Chloride and fluoride were not analyzed since minerals

containing these parameters are not present in the Crandon ores. In all cases, concentrations are considered to represent a practical worst case for leachate quality under non-acidic conditions.

Thus, tailings slurry will enter the TMA with liquid phase parameter concentrations similar to those derived from the locked-cycle metallurgical testing (as shown in Table 2). Changes in the concentration of parameters will then take place in the TMA due to liquid phase reactions occurring within the surface waters and to solid-liquid interactions as further leaching takes place. In the former case, a reduction of concentrations for some parameters can occur depending on the metal speciation in the process water. For example, if cyanide is used in the mill process, as in the proposed Crandon flowsheet, metals such as copper and zinc can be complexed as cyanide species and be discharged into the TMA in the process water. Degradation of the cyanide compounds within the TMA can then lead to a reduction in the concentration of these metals due to precipitation. However, for the purpose of developing a source term, it is assumed that leaching of parameters within the tailings mass in the TMA will occur over time and therefore increasing the original concentrations to the values shown in Table 3. In developing the source term for the TMA, no allowance has been made for reduction of concentrations in many parameters which will likely occur as leachate percolates down through the tailings mass. Attenuation along the flowpath can occur due to a number of physical, physicochemical, chemical and biological mechanisms including oxidation, precipitation, hydrolysis, reduction, adsorption, complexation and metal uptake by biological organisms and organic matter. In the depths of the TMA, anoxic conditions will prevail and it can be expected that a reduction in the concentration of many parameters will actually be experienced.

References

Foth & Van Dyke, May, 1995a. *Environmental Impact Report for the Crandon Project.*

Foth & Van Dyke, May, 1995b. *Tailings Management Area Feasibility Report/Plan of Operation.*

Colorado School of Mines Research Institute, 1982. *Characterization of Crandon Mill Tailings,* Golden, Colorado. CMC-NOI-149.

Table 1: TAILINGS ANALYSES

		Fine Tailings CMC 94	Coarse Tailings CMC 94	Whole Tailings*	CSMRI Massive Tailings
Al	mg/kg	48,800	18,400	33,600	29,000
Ag	mg/kg	27	16	21	58
As	mg/kg	2,300	2,600	2,450	1,600
Ba	mg/kg	60	50	55	<40
Be	mg/kg	<50	<50	<50	
Bi	mg/kg	20	20	20	
Ca	mg/kg	8,600	5,600	7,100	4,400
Cd	mg/kg	40	20	30	17
Co	mg/kg	55	79	67	
Cr	mg/kg	210	30	120	26
Cu	mg/kg	1,400	500	950	1,260
Fe	mg/kg	170,700	268,000	219,350	291,000
Hg	mg/kg	1.3	1.9	1.6	
K	mg/kg	13,100	3,900	8,500	3,860
Mg	mg/kg	20,500	11,200	15,850	11,100
Mn	mg/kg	670	520	595	450
Mo	mg/kg	<20	<20	<20	
Na	mg/kg	2,700	800	1,750	320
Ni	mg/kg	40	<20	<25	11
Pb	mg/kg	6,300	2,900	4,600	1,540
Sb	mg/kg	64	70	67	<100
Se	mg/kg	30	38	34	<40
Sn	mg/kg	28	22	25	
Ti	mg/kg	1,700	1,200	1,450	360
Tl	mg/kg	20	10	15	<40
Zn	mg/kg	11,200	6,000	8,600	6,210
Au	mg/kg	0.8	1.0	0.9	0.9
C	mg/kg	5,800	4,400	5,100	
Ga	mg/kg	21	13	17	
Ge	mg/kg	<1	<1	<1	
S	mg/kg	170,600	297,000	233,800	299,000
Si	mg/kg	180,000	150,000	165,000	279,000
Te	mg/kg	<3	<3	<3	
U	mg/kg	<20	<20	<20	<10

* Composition of whole tailings based on equal proportions of fine and coarse tailings.

Table 2: EXTRACTION OF PARAMETERS (MASS FLUX) FROM TMA TAILINGS

	Water (cycle 30)		CMC Program 94 (TMA fine tailings at 132 days)		CMC Program 94 (TMA whole tailings at 132 days)		CMC Program 94 Whole Tailings (wet/dry, 63 days)		CSMRI Program 82 (column leach)	
	(mg/L)	(mg/kg)	(mg/kg)	(%)	(mg/kg)	(%)	(mg/kg)	(%)	(mg/kg)	(%)
Al	1.4	0.22	0.25	0.001	-0.12	-0.0004	1.2800	0.004		
Ag	<0.1	0.008					0.01	0.047		
As	<0.1	0.008	0.035	0.002	0.078	0.003	0.179	0.007		
Ba	0.0196	0.003	0.21	0.35	0.03	0.05	0.04	0.073		
Be	0.0006	0.0001								
Bi	<0.1	0.015								
Ca	299	46.3	173.00	2.01	269	3.79	446.0	6.282		
Cd	<0.0002	0.00002	0.010	0.03	0.24	0.80	0.0	0.133		
Co	0.041	0.006	0.009	0.02	0.40	0.60	0.0	0.060		
Cr	0.056	0.009	0.113	0.054	-0.01		0.02			
Cu	0.063	0.010	0.180	0.013	1.5	0.162	0.23	0.024	0.04	0.003
Fe	0.41	0.063	2.57	0.002	14	0.007	11.7	0.005	0.04	<0.001
Hg							0.0005	0.031		
K	10.3	1.59	3.50	0.03	21	0.247	8.1	0.095		
Mg	0.15	0.023	24.0	0.12	250	1.58	102.0	0.644		
Mn	0.0046	0.001	1.20	0.18	36	6.05	9	1.513		
Mo	<0.04	0.003								
Na	30.0	4.64	38.1	1.41			8.8	0.503		
Ni	0.006	0.001					0.1	0.400	0.03	
Pb	0.27	0.042	0.16	0.003	2.34	0.051	1.0	0.022	0.03	0.002
Sb	<0.1	0.008								
Se	0.29	0.044	-0.105	-0.35	0.38	1.12	0.1	0.251		
Sn	<0.1	0.008								
Ti	<0.003	0.0002								
Tl	<0.003	0.0002								
Zn	0.445	0.069	0.80	0.007	21.9	0.25	32	0.370	23.5	0.38
SO4			2553	0.50	859	0.12	1413	0.20		

Note: Negative values in the table indicate there is a net loss of the parameter from the solution into the tailings.

Table 3: TMA SOURCE TERM

	Estimated Analysis TMA Tailings mg/kg *	Estimated Extraction mg/kg	Leachate Analysis mg/L **
Al	45,760	0.50	0.08
Ag	26	0.01	0.002
As	2,330	0.06	0.01
Ba	59	0.18	0.03
Be	<50	0.000	<0.0001
Bi	<20	0.015	0.002
Ca	8,300	227	350
Cd	38	0.06	0.01
Co	57	0.09	0.015
Cr	192	0.15	0.02
Cu	1,310	0.44	0.07
Fe	180,430	4.9	0.8
Hg	1.4	0.0005	0.0001
K	12,180	7.0	15
Mg	19,570	70	95
Mn	655	8.8	1.4
Mo	<20	0.003	0.0005
Na	2,510	32	80
Ni	37	0.10	0.02
Pb	5,960	0.60	0.10
Sb	65	0.01	0.002
Se	31	0.01	0.002
Sn	27	0.01	0.002
Ti	1,650	0.0002	<0.0001
Tl	19	0.0002	<0.0001
Zn	10,680	7.0	1.1
SO4		2,325	2,500
CN (Total)			<0.08
TDS			3100

* Composition of TMA tailings based on 80% fine tailings + 20% whole tailing

** Leachate analysis based on 13.4% solids in tailings slurry entering TMA.

Appendix O

Tailings Management Area (TMA)
Construction Quality Assurance Plan (CQA Plan)

Crandon Mining Company

Tailings Management Area (TMA)
Construction Quality Assurance Plan

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Attachment 1 Example of Forms
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1 Introduction

1.1 Summary

The following sections define the Construction Quality Assurance (CQA) program for construction as required by § NR 182, Wis. Admin Code. This plan is followed during construction to monitor and confirm that the construction features are constructed in accordance with the design and regulatory requirements.

1.2 Purpose and Scope

The purpose of the CQA plan is to provide minimum requirements for construction observation, testing, and documentation activities performed during construction. The purpose is to document that the constructed facility will meet or exceed all design requirements, specifications, regulatory and local approvals. The plan outlines the various organizations and their responsibilities involving the implementation and review of the various CQA activities. The plan also outlines sampling and testing programs to be carried out during the construction. The primary goal of the CQA plan is to provide a means of evaluating and controlling the quality of the constructed facility so that the intent of the design is met.

1.3 Applicable Units

This CQA plan applies to liner and final cover construction of Cells 1, 2, 3 and 4 of the Tailings Management Area (TMA) of Crandon Mining Company's (CMC) Crandon Project.

1.4 Design Summary

The liner design consists of two composite liner configurations, one for the base which includes a leachate drainage layer and one for the interior slopes without leachate drainage layer. The two composite liner systems have the following components listed from top to bottom:

- Composite liner <u>with</u> leachate drainage layer
  - 6" Filter Layer (Till fines)
  - 18" Filter Layer (Till fines)
  - Geotextile
  - 24" Drainage Layer
  - Geotextile
  - Thermoplastic Geomembrane
  - 12" Low Permeable Soil Layer
- Composite liner <u>without</u> leachate drainage layer
  - 18" Till Layer
  - Geotextile
  - Themoplastic Geomembrane
  - 12" Low Permeability Soil Layer

The final cover design consists of a composite cover with the following components (listed from the top to the bottom):

- 6" Topsoil
- 36" Rooting Layer
- 12" Drainage Layer
- Geomembrane
- 12" Low Permeability Soil Layer
- Grading Layer (thickness varies)

Leachate collection system piping, reclaim water piping and tailings delivery and distribution piping will also be installed generally including the following:

- Leachate collection system piping
  - perforated leachate collection system piping
  - leachate cleanout risers
  - sideslope risers
  - leachate transfer line piping
- Reclaim water piping
  - reclaim suction line
  - reclaim force main to process water pond
- Tailings delivery piping
- Tailings distribution piping
  - mass discharge piping and fittings
  - spigotting piping and fittings

2 Responsibility and Authority

2.1 Permitting Agencies

The Wisconsin Department of Natural Resources (WDNR) has the regulatory authority for approval or denial of the development and operational permits required for the Crandon Project, TMA.

2.2 Facility Owner/Operator

CMC is responsible for the design, construction and operation of the facility in compliance with the regulatory requirements. CMC has the control of organizations charged with design, CQA, and construction activities. CMC will provide a construction manager who will be representing CMC in all construction related issues.

2.3 Design Engineer

Foth & Van Dyke and Associates Inc. (F&VD) has the primary responsibility for designing the facility to meet the design and operational requirements of WDNR and CMC. CMC is responsible for providing a design engineer to make necessary design changes if construction problems and/or material changes are identified in the field.

2.4 Construction Contractors

The various construction contractors are responsible for constructing the facility in strict compliance with the design criteria, plans and specifications, local and WDNR approvals. Construction contractors may implement their own quality control program for purposes of monitoring their related construction. The CQA program presented in this document provides the minimum standards for the acceptance of work.

2.5 Construction Quality Assurance Consultant

The CQA Consultant selected by CMC is independent from CMC's project manager, manufacturer(s) of materials used, and contractors. The CQA Consultant is responsible for observing and documenting all construction activities including all required quality control testing and for quality assurance inspections and tests. The CQA Consultant may engage the services of independent consultants to perform the following:

- Survey documentation.
- Soil laboratory testing.
- Geosynthetic laboratory testing.
- Testing and/or inspection of other construction materials.

Independent consultants will report directly to the CQA Consultant.

2.5.1 Construction Quality Assurance Officer

Prior to the start of construction, the CQA Consultant shall designate a person as the CQA Officer. The CQA Officer will be a professional engineer registered in the State of Wisconsin. The CQA Officer is responsible for supervising all the inspection and testing quality assurance/quality control (QA/QC) requirements of this section. The CQA Officer is also responsible for the preparation of a construction certification report following each construction phase.

The specific responsibilities for administering the QA/QC program are the responsibility of the CQA officer and include the following:

- Reviewing plans and specifications for clarity and completeness.
- Educating and training QA/QC personnel on requirements and procedures outlined in the CQA Plan.
- Scheduling and coordinating QA/QC activities.
- Supervising CQA field personnel.
- Confirming that QC data are accurately recorded and maintained.
- Confirming that the correct Quality Control (QC) procedures are used.
- Verifying that raw QA data are properly recorded, reduced, summarized and interpreted.
- Providing associated organizations with reports on QA/QC activities and results.
- Identifying non-conforming construction and verifying corrective measures.
- Preparing the Construction Document Report which confirms that the facility has been constructed in substantial compliance with design criteria, plans and specifications, local and WDNR approvals.

2.5.2 Construction Quality Assurance Monitor(s)

The Construction Quality Assurance Monitor(s) (CQAM), under the direct supervision of the CQA Officer, shall be present to perform inspections and testing during the following construction activities:

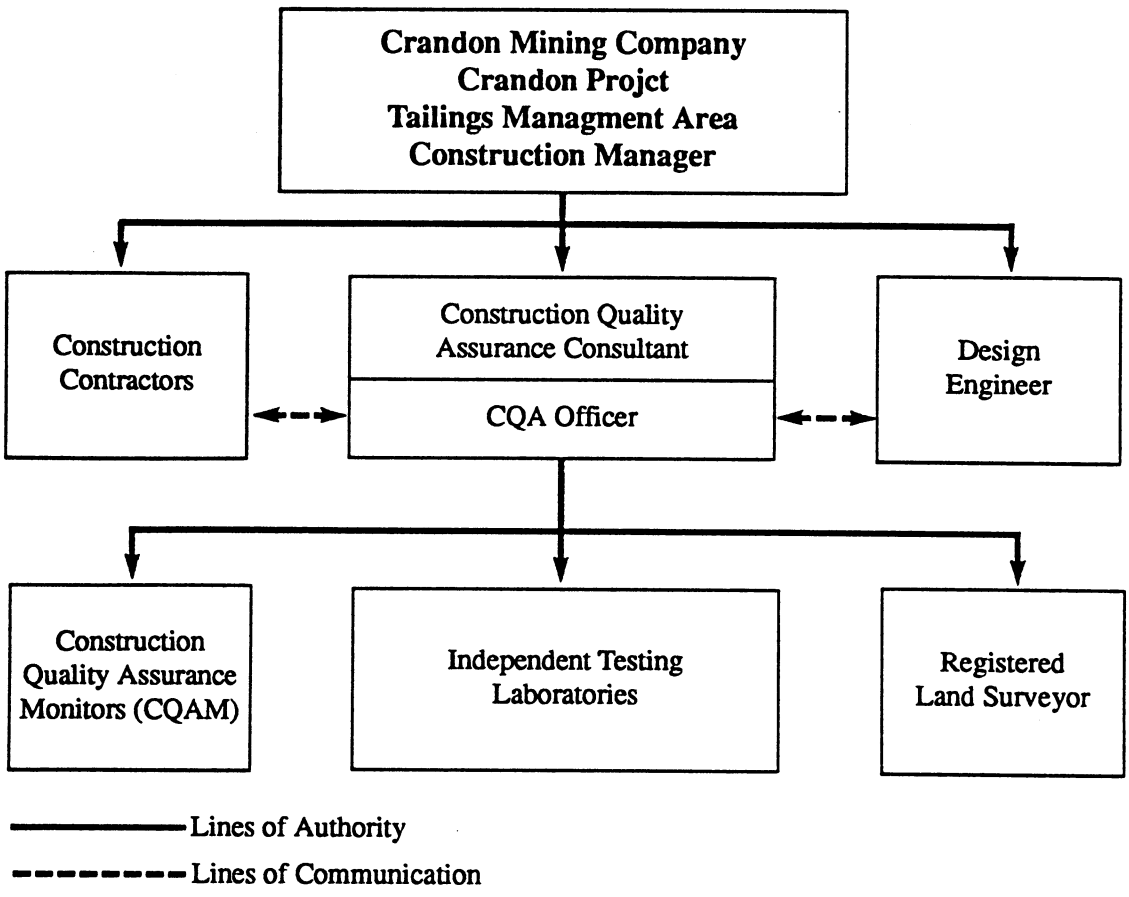
- Construction of the TMA berms and liner support soils
  - Placement of the low permeable soils.
  - Installation of geomembranes and geotextiles.
  - Placement of drainage layer soils.
  - Installation of leachate collection and transfer piping.
- Construction of final cover.
- Construction of surface water control features.
- Reclamation activities.

3 Lines of Communication/Project Meetings

3.1 Lines of Communication

The typical lines of communication necessary during the construction activities are illustrated in the flow chart below.

Lines of Communication for Crandon Mining Company



The CQA Consultant may engage independent consultants for completing the required survey documentation and quality control testing. Quality Control testing includes but is not limited to the following: laboratory soil testing, in-field geosynthetic and soil testing, and laboratory geosynthetic testing.

3.2 Project Meetings

Project meetings are held at regularly scheduled intervals during the course of the project to enhance communication between the organizations involved and to strengthen the responsibilities and authorities of each organization. The CQA Officer is responsible for coordinating and conducting these meetings.

3.2.1 Daily Progress Meetings

A progress meeting is held daily at the work area just prior to commencement or following completion of work. At a minimum, the meeting will be attended by the construction contractor and the QA/QC personnel. This meeting is documented by a member of the CQA Consultant's team. The purposes of the meeting are to:

- Review the previous day's activities and accomplishments.
- Review the work location and activities for the day.
- Identify the contractor's personnel and equipment assignments for the day.
- Discuss any potential construction problems.
- Identify non-conforming construction and determine appropriate corrective measures.

3.2.2 Meetings Regarding Non-Conforming Construction and/or Construction Problems

A special meeting will be held when a construction problem or non-conformance is present or is likely to occur. At a minimum, the meeting will be attended by the construction contractor and CQA Officer. This meeting is documented by the CQA Officer. The purpose of the meeting is to define and resolve problems or recurring work deficiencies.

3.2.3 Preconstruction Meeting

A preconstruction meeting is held prior to construction and prior to each substantial phase of construction, and shall be attended by all parties [contractors, CMC, CQA Officer (or designated CQAM), Design Engineer and a representative of the WDNR]. The meeting is documented by a designated secretary, and minutes will be transmitted to all parties. The purposes of this meeting are to:

- Provide each party with all relevant QA/QC documents and supporting information.
- Familiarize each party with the site-specific QA/QC plan and its role relative to the design criteria, plans, and specifications.
- Determine any changes to the QA/QC plan that are needed to ensure that the facility will be constructed to meet or exceed the specified design.
- Review the responsibilities of each party.

- Review lines of authority and communication for each party.
- Discuss the established procedures or protocol for observations and tests including sampling strategies.
- Discuss the established procedures or protocol for handling construction non-conformance, repairs, and retesting.
- Review methods for documenting and reporting test data.
- Review methods for distributing and storing documents and reports.
- Review compliance of CQA plan with the WDNR and other regulatory approvals.

4 Construction Observation - Record Keeping

4.1 Daily Inspection Report

The CQAM(s) collects all of the samples and performs all of the Quality Control (QC) testing required by the CQA Plan. A daily inspection report is prepared by each inspector for each day of activity. The report contains, at a minimum, the following information:

- Date.
- Type of inspection.
- Summary of weather conditions.
- Summary of any meetings held and attendees.
- Equipment and personnel on the project.
- Summary of construction activities and locations.
- Description of off-site materials received.
- Calibration and recalibration of test equipment.
- Description of procedures used.
- Test locations, procedures, results and test data sheets.
- Summary of samples collected.
- Personnel involved in inspection and sampling activities.
- Signature of the inspector.
- Description of delays in construction activities.
- Detailed description of any problems or non-conforming construction.

4.2 Daily Summary Report

The CQA Officer or the CQAM, under the direct supervision of the CQA Officer, shall prepare a daily summary report which at a minimum contains the following:

- Date.
- Summary of weather conditions.
- Summary of location where construction is occurring.
- Contractors, equipment and personnel on the project.
- Summary of any meetings held and attendees.
- Description of all materials used and references or results of testing and documentation.
- Calibration and recalibration of test equipment.
- Daily inspection reports from each CQAM.
- Description of any construction not meeting the project requirements and how it was corrected.

4.3 Photographs

Photographs shall be obtained for all items of construction. A sufficient number of photographs shall be obtained to document the construction of each construction item (i.e., liners, covers, piping, diversion berms, geosynthetics, etc.). Each photograph shall be a dated 35 mm photograph and shall be recorded in a Photo Log (see Attachment No. 1).

Construction problems and non-conforming work shall be documented with photographs taken before and after the problem or non-conforming work is corrected.

4.4 Test Data Sheets

CQAM will record all test data results on the test data sheets provided in Attachment No. 1. Independent consultants engaged by the CQA Consultant shall submit their test results or data on forms acceptable to and approved by the CQA Consultant.

4.5 Document Control and Record Storage

4.5.1 Daily Records

The daily records maintained during construction activities include, but are not limited to the following:

- Daily inspection reports.
- Daily summary reports.
- Test data sheets from CQAM.
- Test data or documentation data sheets from independent consultants (if any).
- Field book maintained by each CQAM.

Daily records will be copied and forwarded to the CQA Officer on a daily basis.

4.5.2 Storage of Records

All document originals listed in 4.5.1 above will be stored in fireproof cabinets at the construction site. Copies of all documents will be on file at the CMC field office in Crandon, Wisconsin and the CQA Officer's office.

4.6 Construction Problem Identification and Design Changes

4.6.1 Construction Problem Identification

A Non-Conformance Notice is filled out when deviation from the design plans and specifications, or non-conforming work, is observed by CQA personnel. An example of the form is in Attachment 1. The CQA Officer indicates the extent of work required to correct the deficiency and evaluates test results to determine if there were testing errors. These reports must be cross-referenced to laboratory test results, specific test data sheets, and daily summary reports and include all necessary information listed in Section 4.6.2 - Design Changes.

4.6.2 Design Changes

No changes in design will be made in the field without first completing and submitting a Field Modification form for approval by the Design Engineer and CMC. The CQA Officer and Design Engineer shall determine if WDNR approval is required for the proposed design change by contacting the WDNR representative. If WDNR approval is required the CQA Officer shall prepare the information required for a plan modification to the proposed design. If a plan

modification submittal is not required, the CQA Officer shall complete the Field Modification form; examples of which are provided in Attachment 1. The Field Modification form shall include the following:

- Original design given by project plans and specifications.
- Reason(s) for required change in design.
- Proposed design changes including changes in materials, quantities, construction methods, and constructed performance.
- Supporting calculations.

4.6.3 Design Change Construction Documentation

Construction due to design changes shall be documented by the CQA Officer in the Daily Summary Report. All materials, quantities, and construction methods required for the design change shall be recorded. Documentation shall, in general, meet the requirements of Sections 5, 6 and 7.

5 Construction Observation - Testing and Verification

This section outlines minimum requirements for the testing and verification of the components of the liner and final cover system.

5.1 Survey Verification

Record surveys shall be performed by a professional land surveyor registered in the State of Wisconsin. At a minimum, the surveys shall document the following:

Liner and Leachate Collection System

- Subbase grades (bottom of a low permeable soils layer) on a 100-foot grid.
- Top of final surface on berm and embankment grades on a 200-foot grid.
- Top of low permeable soils on a 100-foot grid.
- Top of drainage layer on a 100-foot grid.
- Top of till soils on a 100-foot grid.
- Leachate pipe elevations at 25 foot interval.
- Leachate sump subbase grade and final grade elevation.
- Sideslope riser and cleanout location and elevations.

Cover System

- Top of final tailings or mine waste surface on 100-foot grid.
- Bottom of low permeable soils on 100-foot grid.
- Top of low permeable soils on 100-foot grid.
- Top of drainage layer on 100-foot grid.
- Top of rooting layer on 100-foot grid.
- Top of topsoil layer on 100-foot grid.

5.1.1 Tolerances

The vertical grading tolerance for each soil layer of liner and cover shall be at design grades to 0.1 foot above.

5.2 Thickness Verification

The CQAM(s) shall verify the thickness of the soil components on a 100-foot grid interval. The method of verification may include survey, use of settlement plates, hand augers or hand shoveling. Verification must be done in a manner which does not harm the geosynthetic components of the liner or cover.

5.2.1 Tolerances

Thickness of individual soil components must meet the following minimum tolerances:

Liner System	Minimum Thickness
1. Low Permeable Soil Layer	12"
2. Drainage Layer	24"
3. Till Soil Layer	18" (on slope) 24" (on base)
4. Leachate Pipe Bedding	6"
Cover System	
1. Grading Layer (thickness varies as required to attain a minimum 2% slope as per the drawings)	
2. Low Permeable Soil Layer	12"
3. Drainage Layer	12"
4. Rooting Layer	36"
5. Topsoil Layer	6"

5.3 Soil Testing

The CQAM(s) shall collect samples of each soil component in accordance with Table 5-1. The samples will be sent to the selected soils laboratory for testing.

The CQAM(s) will also be responsible for all in-place testing in accordance with Table 5-1.

The following structures shall be constructed and sampled according to the CQA program in this section:

- Compacted subgrade and embankments
- Clay liner.
- Geomembrane.
- Drainage layer.
- Geotextile.
- Topsoil and rooting layer.

The material acceptability criteria for placed materials tested for quality assurance purposes shall be a maximum of ten (10) percent outliers. No outliers shall be accepted for workmanship acceptance (compaction, geomembrane installation, etc.).

Table 5-1

Construction Quality Assurance Testing Program for Soil Components
Crandon Project

Soil Component	Test/observation	Test Method	Min. Freq.	Acceptance Criteria
Liner System				
Liner Support Soils	In-place density	ASTM D2922	1/acre/ft	Percent Compaction Min. 95% (Standard Proctor, ASTM D698)
Low Permeable Soil Layer	Atterburg Limits	ASTM D4318	1/acre/ft	LL >30
	Grain Size	ASTM D422	1/acre/ft	PI >15
	USCS Soil Class	ASTM D2487	1/acre/ft	P200 >50%
	Hydraulic Conductivity	ASTM D5084[1]	⅓/acre/ft	CL or CH
	In-place Density	ASTM D2922	100-foot grid/ft	Max. 1 x 10^{-7} cm/sec Min. 95% of ASTM D698
	In-place Moist. Cont.	ASTM D2922	100-foot grid/ft	4.
	Stand. Proctor	ASTM D698	1/5000 c.y.	NA
	Thickness	NA[2]	100-foot grid	12" Min.
Drainage Layer	Grain Size	ASTM D422	1/2500 cy	P200 <5%, C_u <4%
	USCS Soil Class	ASTM D2487	1/2500 cy	GP
	Hydraulic Conductivity	ASTM D2434[3]	1/6000 cy	Min. 1 x 10^{-2} cm/sec
	Thickness	NA	100-foot grid	24" Min. LCS
Pipe Backfill	Grain Size	ASTM D422	1/1,000 LF	P200 <5%
Cover System				
Grading Layer	In-Place density	ASTM D2922	1/acre/ft	Percent Compaction Min. 95% (Standard Proctor, ASTM D698)
Low Permeable Soil Layer	In-place density	ASTM D2922	1/acre/ft	Percent Compaction Min. 95% (Standard Proctor, ASTM D698)
	Atterberg Limits	ASTM D4318	1/acre/ft	LL >30 PI >15
Drainage Layer	Grain size	ASTM D422	1/2500 cy	P200 <5%, C_u <4%
	USCS Soil Class	ASTM D2487	1/2500 cy	GP
	Hydraulic Conductivity	ASTM D2434[3]	1/6000 cy	Min. 1 x 10^{-2} cm/sec
	Thickness	NA'	100-foot grid	12" Min. FCDL
Topsoil	pH, nitrogen, phosphorus and potassium, USCS classification		1/5 acres	NA[1]

1. Undisturbed Sample
2. Not Applicable
3. Recompacted Sample
4. -1% to +5% about the optimum moisture content as determined by ASTM D698.

LCS - Leachate Collection System
cm/sec - centimeters per second
cy - cubic yard
lf - lineal feet
C_u - Coefficient of Uniformity

FCDL - Final Cover Drainage Layer
P200 - percent passing the Number 200 Sieve
LL - liquid limit
PL - plastic limit

Prepared by: REM
Checked by: NXP

5.4 CQA Officer Inspection of Subgrade and Foundation

The CQA officer shall perform the following functions:

- Verify that the site conditions are, in general, as interpreted in the plans.
- Ensure that there are no moisture seeps. If *in situ* material provides insufficient strength (i.e., it is soft and yielding and ruts during proof rolling) the soft material shall be removed and replaced with compacted till materials. This material shall be compacted to achieve the density properties necessary listed in Table 5-1.
- Ensure that all trees, stumps, roots, boulders and debris are removed.
- Prohibit the placement of frozen soil. Prohibit the placement of soil onto frozen ground.
- Ensure that the foundation is constructed and graded to provide a smooth, workable surface on which to construct the liner.
- Ensure that the liner base and embankments are graded to the correct elevation and slope.

6 Construction Observation - Geomembrane

The following section summarizes the quality assurance plan proposed for testing and monitoring of the geomembrane liner installation.

6.1 On-Site Quality Assurance

6.1.1 Geomembrane Rolls and Panels

Construction quality assurance monitoring for the rolls and panels include:

1. Monitoring and documenting the unloading of trucks delivering geomembrane rolls to the site.

2. Monitoring the handling and on-site storage of geomembrane rolls.

3. Recording the roll and batch numbers of geomembrane rolls delivered to the site.

4. Review of manufacturer's QA testing for conformance with specifications.

5. Selecting samples from geomembrane rolls delivered to the site for off-site conformance testing. Conformance testing will be performed as outlined in Table 6-1. Samples shall be sent to a geosynthetics testing laboratory for material properties analysis.

6. Fixing a code number to samples and recording the manufacturer's roll numbers of the rolls from which samples are taken.

7. Labeling, packaging and shipping samples to an independent testing laboratory for conformance testing.

8. Interpreting laboratory test results in accordance with the specifications and accepting or rejecting delivered rolls based on results of laboartory testing.

9. Performance of visual review of synthetic liner fabricated at the factory as it is unrolled and deployed at the job site for uniformity, damage, and imperfections, including holes, cracks, thin spots, tears, punctures, blisters, and foreign matter.

Table 6-1

60 Mil HDPE* Material Properties
Crandon Project

Test	Procedure	Manufacturer Test Frequency	Third-Party Conformance Test Frequency
Density	ASTM D792 or ASTM D1505	(See note 1)	1each/100,000 ft^2
Thickness	ASTM D751		1 each/100,000 ft^2
Yield Strength	ASTM D638 NSF modified		1 each/100,000 ft^2
Yield Elongation	ASTM D638 NSF Modified		1 each/100,000 ft^2
Tensile Strength	ASTM D638 NSF Modified		1 each/100,000 ft^2
Tensile Elongation	ASTM D638 NSF Modified		1 each/100,000 ft^2
Modulus of Elasticity	ASTM D638 NSF Modified		NR
Carbon Black Content	ASTM D1603		NR
Carbon Black Dispersion	ASTM D3105 NSF Modified		NR
Environmental Stress Crack	ASTM D746		NR
Low Temperature Brittleness	ASTM D746		NR
Tear Resistance	ASTM D1004		NR
Dimensional Stability	ASTM D1204		NR
Puncture Resistance	FTMS 10/C or Method 2065		NR

NR - Third-Party Conformance Test not required.

Notes: 1. Manufacturers to provide manufacturer quality control data for all the tests listed for each roll.
2. In addition, the following test shall be performed for each resin that has been used in the manufacture of membranes: Melt Index, Carbon Balck Content, and Carbon Black Dispersion.

*High Density Polyethelene (HDPE) is included as an expample of material properties which will be provided when the geomembrane is selected in the final design.

6.1.2 Panel Placement

Quality assurance monitoring for panel placement includes:

1. Obtaining a written acceptance of the subgrade by the geomembrane installer.

2. Evaluating and documenting weather conditions (*e.g.*, temperature, wind) for geomembrane placement and informing the CMC construction manager if requirements for weather conditions are not met, so the CMC construction manager can decide to stop geomembrane placement.

3. Monitoring and documenting geomembrane placement as well as conditions of panels as placed.

    a. Noting panel defects, tears or other deformities.

    b. Measuring panel thicknesses at a minimum rate of five areas measured per roll.

    c. Measuring as-delivered panel lengths.

4. Recording the locations of installed panels and checking that the panels have been installed in accordance with the design plan.

    a. Assigning each panel a unique panel number and identifying that panel with the manufacturer's roll number.

    b. Recording panel numbers and locations on a panel layout diagram.

6.1.3 Geomembrane Field Seam Construction

Quality assurance monitoring and testing to be conducted for seam construction includes:

6.1.3.1 Fusion Welds

1. Monitoring trial fusion seams constructed prior to each seaming sequence to evaluate the seaming crew and equipment.

    a. Record machine temperature, ambient temperature, machine speed, seamer identification, machine number, date and time for all trial seams.

    b. Trial seams will be made at the beginning of each seaming period, at the discretion of the CQAM, at least once every four to five hours (minimum two (2) test seams each day) for each seaming apparatus used that day. Each seamer will make at least one (1) test each day. Six (6) specimens will be cut, and four (4) peel tests will be performed on the inside and outside tracks of the weld and two (2) shear tests will be performed. Field samples will be die cut to one-inch widths.

2. Evaluating and documenting trial seam test results in accordance with the specifications and accepting or rejecting seaming crews and/or equipment.

   a. Observing the performance of peel and shear tests on trial seam samples, shear tests must meet a minimum seam strength specified to pass and peel tests must have a film tearing bond failure and the minimum specified peal adhesion strength.

   b. Retain a portion of each trial seam.

3. Evaluating and documenting the suitability of weather conditions (*e.g.,* temperature, wind, humidity) for seaming and informing the CMC construction manager when weather conditions do not meet the specifications so the CMC construction manager can decide to stop geomembrane seaming.

4. Monitoring seam construction.

5. Assigning a seam number to each seam and recording seam construction data, including seam crew identification, machine number, date and time of seam construction, ambient temperature.

   a. Record the location of all seams on a seam layout diagram.

6. Confirming that the installer's field tensiometer has current calibration documentation. At a minimum, the field tensiometer shall have been calibrated within one year prior to start of project.

6.1.4 Seam Testing and Repair

Items included in the quality assurance for monitoring seam testing and repair include:

1. Monitoring and documenting non-destructive testing done to evaluate continuity of all seams.

   a. Observe seam pressure tests in accordance with project specifications.

   b. Observe air lancing tests in accordance with project specifications.

   c. Marking failed seams for repair.

   d. Document repair and retest of seam.

2. Selecting locations where geomembrane samples will be taken to conduct destructive testing.

   a. A minimum of one destructive test sample will be collected for every 500 lineal feet of field seam.

b. Locations of destructive test samples will be noted on a repair and sample location diagram.

3. Monitoring the cutting of samples by the geomembrane installer.

4. Assigning a unique number to each sample and recording sample locations and other pertinent observations made during sampling.

5. Monitoring the cutting of the sample in three parts: one for the geomembrane installer, one for archiving, and one for testing by the independent laboratory.

6. Monitoring and documenting the field seam destructive tests performed by the geomembrane installer.

7. Labeling, packaging and shipping samples to the independent laboratory for destructive testing.

8. Interpreting laboratory test results and accepting or rejecting seams based on independent laboratory test results.

9. Monitoring and documenting patching of holes caused by sampling.

10. Monitoring and documenting the non-destructive testing of the seams associated with seam repair.

11. Monitoring and documenting the repair of the rejected seams and the non-destructive testing of the seam repairs.

    a. Document passing seam tests between all destructive test locations.

    b. Record all seam repair locations.

12. Monitoring and documenting destructive testing related to seam repair.

    a. Monitoring and documenting one destructive seam sample for every 500 lineal feet of repaired seam as described above.

6.1.5 Defect Repairs

The following quality assurance monitoring and testing will be implemented to monitor defect repairs:

1. Performing systematic visual observation of the entire surface of the geomembrane to locate and document defects and indicate for each defect the type of repair that is required.

2. Monitoring and recording the repair of defects and the non-destructive testing of all repairs.

3. Recording the location and the nature of all defect repairs.

6.1.6 Anchor Trenches

Quality assurance associated with monitoring and testing of anchor trenches shall include the following:

1. Anchor trench excavation shall be monitored for proper depth and location.

2. Geomembrane panels extending into the anchor trench shall be monitored for complete seaming within the anchor trench.

3. Anchor trench backfill operations will be observed and documented.

   a. The length of the open trench shall not exceed the amount of liner to be placed in one day.

   b. The depth of a typical anchor trench shall be documented to conform to approved project drawings.

   c. Backfill shall be placed in thin lifts not to exceed one foot in loose thickness.

   d. Compaction of backfill using hand operated compaction equipment to a minimum of 95% of the maximum dry density as determined by the standard Proctor test (ASTM D 698) within -1% to +5% of optimum moisture content.

   e. Density tests will be performed at a minimum interval of 500 lineal feet of anchor trench.

6.2 Documentation and Reporting

Documenting and reporting methods will be implemented to systematically record results of on-site monitoring and on-site testing. Reporting forms will be used for roll and panel placement, trial weld construction, panel seaming, non-destructive seam testing and destructive seam testing. Unique identifying numbers will be assigned to each panel and seam and used to reference the panel and seam location and test results. Copies of quality assurance forms are included in Attachment 2.

A geomembrane installer's certificate of acceptance of the subgrade will be obtained prior to placement of geomembrane panels. A format for the certificate of acceptance is given in Attachment 2.

Panel location and seam location diagrams will be kept showing the location of all panels and seams, repairs and destructive sample test locations. These location diagrams will be updated on a daily basis and will be available for review by the construction manager.

A photo log will be created containing photos of all phases of the geomembrane liner installation, including deployment, seaming, testing, and anchor trench construction.

Copies of test results for all off-site laboratory testing will be forwarded to the on-site supervisor and will be made available to the construction manager. The laboratory test result documents will be maintained in a job file and submitted with the final certification report (see Section 7).

7 Construction Certification Report

7.1 Summary

Upon completion of the construction of each major phase and prior to placing in service, the CQA Officer shall submit a construction certification report to the agency. This report shall be prepared in accordance with Wisconsin Admin. Code §NR 182. The report shall contain, at a minimum, the following information:

- Based on the Owners/Contractors records and data collected through following the QA/QC measures outlined in Sections 5 and 6. The CQA Officer shall certify that the construction has been prepared and constructed in substantial conformance with the engineering plans and specifications.
- Daily summary reports.
- Detailed narrative describing the construction activities in chronological fashion.
- Analysis and discussion of all QA/QC testing performed with summaries of all test results.
- All raw data and test result reports performed during construction.
- Detailed description and documentation of all material and equipment types and specifications.
- Warranties, shop drawings and operating instructions for equipment and materials.
- Discussion of any construction material or equipment which deviated from the engineering plan, and reasons for deviation.
- Photographs documenting all aspects of facility construction.
- Record drawings containing:
    - Record subbase grade (bottom of clay liner) and/or bottom of clay cap elevations.
    - Record base grade (top of clay liner) elevations. This sheet shall also include a table summarizing the thickness of the clay liner at each survey location.
    - Locations/identification numbers for geomembrane panel layout (with size of panels provided in an accompanying table), seam type, repair location, destructive test location, and each panel and anchor trench location.
    - Thicknesses of granular drainage layers.
    - Locations of all soil tests and samples.
    - Cross-sections.

- As constructed details of TMA piping systems including:

    - leachate collection system piping.
    - reclaim return water piping.
    - tailings delivery piping.
    - tailings distribution system piping.

- Details of stormwater management and erosion control structures.

Attachment 1

Example of Forms

PHOTO LOG

Roll Number	Photo Number	Date Taken	Description

[32-15]94L011

Foth & Van Dyke

Client: _____ Scope I.D.: _____
Project: _____ Page: _____
Prepared by: _____ Date: _____

NON-CONFORMANCE NOTICE

DATE: _____ CONTRACT: _____
CONTRACTOR: _____ FOREMAN: _____
STREET: _____

- ☐ PIPELINE INSTALLATION FROM STATION _____ TO STATION _____
- ☐ COMPACTION FROM STATION _____ TO STATION _____ TYPE _____
- ☐ MANHOLE INSTALLATION AT STATION _____
- ☐ HYDRANT INSTALLATION AT STATION _____
- ☐ VALVE INSTALLATION AT STATION _____
- ☐ EXISTING WATERMAIN REPAIR _____
- ☐ EXISTING SEWER REPAIR _____
- ☐ SEWER LATERAL FOR _____
- ☐ WATER SERVICE FOR _____
- ☐ CATCH BASIN INSTALLATION AT STATION _____
- ☐ CURB AND GUTTER STATION _____ TO STATION _____
- ☐ SIDEWALK STATION _____ TO STATION _____
- ☐ CRUSHED AGGREGATE BASE STATION _____ TO STATION _____
- ☐ BACKFILL STATION _____ TO STATION _____
- ☐ OTHER _____

The above checked item(s) is not in conformance with the plans and specifications and will be deleted from future payment requests until the work is corrected.

COMMENT: _____

RESIDENT INSPECTOR: _____

DISTRIBUTION: ORIGINAL - CONTRACTOR CANARY - OWNER PINK - RESIDENT ENGINEER GOLD - INSPECTOR

Foth & Van Dyke

Client: _____ Scope I.D.: _____
Project: _____ Page: _____
Prepared by: _____ Date: _____
Checked by: _____ Date: _____

FIELD MODIFICATION

Addendum No.: _____

Description of Modification: _____

Basis for Modification: _____

ATTACHMENTS

Revised Drawing: _____

New Drawing: _____

Drawn by: _____ Date: _____
Checked by: _____ Date: _____
Approved by: _____ Date: _____

Copies to: _____ _____

FORM EES012 (1/91)

Foth & Van Dyke

Client:_____ Scope I.D. #:_____

Project:_____

CONFIRMATION OF CLIENTS REQUEST TO PERFORM SERVICES

TYPE OF SERVICES REQUESTED:

- ☐ General Engineering Services
- ☐ Additional Engineering Services Beyond Original Work Scope
- ☐ Other:_____

Addendum No.:_____ Notice No.:_____

Specific Work Requested:_____

Basis for Request:_____

Estimated Cost:_____

Estimated Completion Date:_____

Schedule Impacts:_____

Comments:_____

Authorized by:_____ Date:_____

Authorized Signature:_____ Date:_____

Foth & Van Dyke Project Manager:_____ Notice Date:_____

Copies to:_____

FORM EE5006 (1/91)

Foth & Van Dyke

Client: _____ Scope I.D.: _____

Project: _____ Page: _____

Prepared by: _____ Date: _____

RESIDENT INSPECTION REPORT - SITE

Location: _____

Weather	Temp.		Sky	Rain	Snow	Project Site	
	Lo	Hi				Dry	Muddy

Contractors on Site: _____ Personnel (#)

_____

_____

_____

_____

_____

Other Personnel on Site: _____ Purpose

_____

Report of Observation of Work and Comments: _____

Additional Space on Back

FORM 171 (Rev. 9/98)

DENSITY TESTS OF COMPACTED FILL

Contractor: _____

Compaction Equipment: _____

_____

Weather: _____

Method of Test:

☐ Nuclear Meter *(ASTM: D2922)*

☐ Sand Cone *(ASTM: D1556)*

Area/Location Tested: _____

_____

_____

Report Number: _____

Test No.	Test Location	Depth of Probe	Elevation	Proctor Used	Wet Density *(pcf)*	Moisture Content *(%)*	Dry Density *(pcf)*	Percent Compaction	Remarks

Proctor No.	Soil Classification	Max. Dry Density *(pcf)*	Moisture Content *(%)*	Compaction Spec. *(%)*

General Note:

Density test results are valid only at the locations and elevations tested.

Nuclear Meter Used: _____

Model: _____ Serial No.: _____

Standard Counts: _____

Density: _____ Moisture: _____

Foth & Van Dyke

Client: _____ Scope I.D.: _____

Project: _____ Page: _____

Prepared by: _____ Date: _____

Checked by: _____ Date: _____

FORM 146 (Rev 2-90)

Attachment 2

Geosynthetic CQA Forms

Foth & Van Dyke

PANEL PLACEMENT INFORMATION

PROJECT NO. _____

DATE _____

PAGE _____ OF _____

PANEL NUMBER	ROLL NUMBER	TIME	PANEL SIZE	PANEL LOCATION	SUBGRADE CONDITIONS	AMBIENT TEMP.	WEATHER/ WINDS	PANEL DAMAGE	QA MONITOR	COMMENTS

Foth & Van Dyke

NON-DESTRUCTIVE SEAM TEST INFORMATION

PROJECT NO. _____

DATE _____

PAGE _____ OF _____

TIME	SEAM NUMBER	TESTER INITIALS	AIR TESTING					PASS/FAIL	VACUUM BOX PASS/FAIL	QA MONITOR	COMMENTS
			PRESSURE (PSI)			TIME					
			START	END	DROP	START	END				

Foth & Van Dyke

PANEL SEAMING CHECKLIST

PROJECT NO. _____

DATE _____

PAGE _____ OF _____

START TIME	WEATHER/ WINDS	TEMPERATURE		SEAMER INITIALS	MACHINE NUMBER	TEMP. SETTING	SPEED SETTING	SEAM TYPE	SEAM NUMBER	PANEL NUMBERS	SEAM LENGTH	END TIME	QA MONITOR	COMMENTS
		AMBIENT	SHEET											

Foth & Van Dyke

TRIAL WELD INFORMATION

PROJECT NO. _____

DATE _____

PAGE _____ OF _____

TIME	WEATHER/ WINDS	ABBIENT TEMP.	SEAMER INITIALS	MACHINE NUMBER	EXTR. WELDS		FUSION WELDS			SEAM TEST		PASS/ FAIL	QA MONITOR	COMMENTS
					BARREL TEMP.	PREHEAT TEMP.	WEDGE TEMP.	SPEED SETTING	WHEEL TENSION SETTING	PEEL (LBS/INCH)	SHEAR (LBS/INCH)			

Foth & Van Dyke

DESTRUCTIVE SEAM TEST INFORMATION

PROJECT NO. _____

DATE _____

PAGE _____ OF _____

TIME	SAMPLE IDENTIFICATION	SEAM NUMBER	TESTER INITIALS	FIELD SEAM STRENGTH		PASS/ FAIL	DATE TO LAB PKG. SLIP NO.	LAB PASS/FAIL	QA MONITOR	COMMENTS
				PEEL (LBS/INCH)	SHEAR (LBS/INCH)					

Appendix P

Tailings Management Area and Appurtenances - Typical Assignment Operations Log

Tailings Management Area and Appurtenances
Operating Assignment Log

Crandon Project

Equipment or Area	Description	Activity Description	Frequency	Hours			Comments
Tailings and Reclaim Water Pipeline	Plant Site to TMA	1. record quantity of slurry pumps 2. check pipeline pressure sensing system 3. observe pipeline for leaks (especially at joints) 4. observe containment ditch for damage 5. check sumps for water and/or tailings 6. pump/clean sump	each shift each shift each shift each shift each shift as required				
	Pump Station (Mill)	1. check pumps, tailings pump box	each shift				
	Pump Station (Pond)	1. check pumps,					
Tailings and Reclaim Water Pipelines - on TMA Embankments	TMA Embankment Pipelines	1. observe pipelines for leaks (especially at mechanical joints	each shift				
Tailings Distribution System/ Tailing Cell	Spigots	1. open and close scheduled spigots 2. check spigot discharge pipe and adjust	daily/as required daily/as required				
	End Dump Valves	1. operate as required by conditions	NA				typically used only in winter
	Tailings Surface Conditions	1. observe color/appearance of tailings in non-deposition area	daily				
	Reclaim Water Pump	1. check pump	each shift				
	Reclaim Water Barge	1. check pipeline to/from barge 2. check pond water depth 3. winter-check ice around barge/pipes	each shift each shift each shift				
Leachate Removal and Transfer Systems	Side Slope Risers	1. check water level in sumps 2. check pump operations 3. check that pipe caps covers are in-place	daily				refer to equipment O+M manual
	Leachate Transfer Lines	1. check for leaks	daily				
	Leachate Line Cleanouts	1. check for pipe caps/covers in-place 2. clean collection lines	daily annually				
Process Water Ponds	Ponds	1. check sediment below influent pipe	daily				
	Pumps	1. sample return water at outfall	?				

Surface Water Management Structure	Sedimentation Ponds	1. check for erosion; washouts 2. check water and sediment depth 3. check outfall water clarity	after each storm after each storm after each storm				event of 0.5 inches or more event of 0.5 inches or more event of 0.5 inches or more
	Ditches, Culverts, Spillways, etc.	1. check for erosion, washouts 2. check for siltation 3. check vegetated areas for washouts, rills, etc.	after each storm after each storm after each storm				event of 0.5 inches or more event of 0.5 inches or more event of 0.5 inches or more
	Others	1. check roads for mud tracking 2. check roads for washouts	after each storm after each storm				event of 0.5 inches or more event of 0.5 inches or more

Signatures Date

Shift 1 Operator _____

Shift 2 Operator _____

Shift 3 Operator _____

Manager _____

Appendix P-1

Update of the Crandon Project Biological Function Model

Foth & Van Dyke
Memorandum

January 31, 1996

TO: Don Moe, Crandon Mining Company

CC: Jerry Sevick, Foth & Van Dyke
Master File

FR: Ron Steg, Foth & Van Dyke

RE: Update of the Crandon Project Biological Function Model

As described in Section 3.10 and Appendix 3.10-2 of the CMC Environmental Impact Report (EIR) (Foth & Van Dyke, 1995), the wetland functional values assessment completed for the project in the 1980s considered the following functional values: biological functions, hydrological support, groundwater, storm functions and floodwater storage, shoreline protection, water quality maintenance, cultural and economic, recreation, and education. A separate model was developed for each of these functional values. The purpose of this memorandum is to present updated data regarding the biological function using the biological data base developed through a comprehensive inventory of the flora and fauna existing within a 30 square mile area surrounding the Crandon Site (see Section 3.9 of Foth & Van Dyke, 1995). As part of this memorandum a narrative description of the biological elements that were considered when making decisions regarding the importance of individual wetlands within the study area is presented, rather than completing an update of the wetland functional value scores.

The 1980s Biological Function Model

The biological function model originally developed by Normandeau Associates, Inc. and Interdisciplinary Environmental Planning, Inc. (1982) is described in detail in Appendix 3.10-2 of the Environmental Impact Report (Foth & Van Dyke, 1995). When applied to a specific wetland, the biological function model produced a numerical ranking or score based upon the summation of individual scores for a total of 15 elements important to a wetland's potential for providing biological functional values. Elements considered included: dominant wetland class, number of wetland classes, vegetative interspersion, surrounding habitat, water/cover ratio, number of plant species, proportion of wildlife food plants, vegetative density, wetland juxtaposition, hydrological position, water level fluctuation, surface water connection, percent wetland bordering on open water and size.

This model was developed based on region-specific wetland characteristics and to be responsive to the requirements of NR 132.06(4) of the Wisconsin Administrative code. The biological function model has a potential score range of 29 to 158 with a model mean of 93. When applied to the wetlands in the study area the scores ranged from 35-141, with a mean of 76. The model parameters provide a basis for assessing wetlands in a regional context, whereas the range and mean of the actual scores is representative of those wetlands existing within the study area.

The presence of rare and/or state or federally protected species was not specifically considered in the scoring. Wetlands which provided habitat for, or were frequented by, threatened or endangered species were immediately identified for more thorough analysis. However, these analyses were conducted separately (e.g., the spotted salamander study conducted by Dames & Moore in 1979) and never entered into the biological function scores.

Overlay of New Biological Data

Although not considered quantitatively in the wetland functional value scores presented in the EIR (Foth & Van Dyke, 1995), the presence of rare and/or state or federally protected species was considered during the process of locating project facilities and assessing impacts for the 1995 EIR and permit applications. All wetlands, in the approximate 30 square mile biological study area in which habitat potentially existed for the list of target species presented in Section 3.9 of the EIR, were included in the biological inventories conducted between 1993 and 1995. Obviously, this biological information was not available at the time the wetland functional value scores were originally developed in the 1980s. However, it is important to note that both the quantitative wetland functional value scores as well as the qualitative biological information presented in the following paragraphs has already been considered in the CMC impact assessments and evaluation of alternatives. The following paragraphs present an updated discussion of the wetlands in the study area for which new relevant biological data was collected. Each of these wetlands is listed in Table 1 along with a list of observed endangered, threatened or special concern species.

As shown in Table 1, endangered, threatened or special concern plant or animal species were observed in 13 of the wetlands in the approximate 18 square mile wetland study area. Wetlands Z23, O1 and W1 were the most notable regarding the occurrence of rare species of plants and animals. Within the wetland study area, wetland Z23 consists of approximately 227 acres dominated by bog and conifer swamp. The biological functional value score relates only to this 227 acres. However, wetland Z23 is actually part of a very large wetland system, dominated by conifer swamp, within the headwaters of Pickerel Creek and extends between Mole Lake and Rolling Stone Lake. Wetland O1 is an approximate 114 acre conifer swamp connected to an extensive wetland system along Swamp Creek. Wetland W1 is an approximate 437-acre wetland located within the floodplain of Swamp Creek. Conifer swamp is the dominant wetland type in wetland W1, however, shrub swamp and emergent marsh are interspersed throughout.

Wetlands Z23, O1 and W1 share at least two characteristics important to their biological functional value; all three are relatively large, with at least some areas that have not been recently disturbed, and are generally dominated by conifer swamp. These are important characteristics to many of the listed bird and plant species as well as mammals, specifically the bobcat. Interspersion of a variety of habitat types is also an important characteristic of these wetlands. This is most important in wetlands Z23 and W1 where wetlands border open water habitat and pockets of different wetland habitat types exist adjacent to one another. The presence of host plant species in wetlands Z23 and W1 are the most important feature to the observed butterflies. Scores of 89 and 93 (i.e., below and at the model mean) for wetlands Z23 and O1, respectively, do not quantitatively reflect the biological importance of these wetlands. The score of 101 for wetland W1 is above the model mean, but is likely still not reflective of the biological importance of this wetland. This is particularly true, in that the only known population of *Vaccinium vitis-idaea* in the state exists in this wetland.

Wetlands F28, F16, F64 and Z2 all include a significant bog component. Of these, wetlands F28 and F16 are the best examples of classic bog habitat. Based on the presence of appropriate

habitat and host plants, each of these wetlands provides habitat for at least one Special Concern species of butterfly. Wetland F28 is also relatively large and borders Duck Lake. Duck Lake contains a population of the state threatened algal-leaved pondweed. Wetlands F16, Z2 and F64 were all scored below the model mean for the biological functional value. Greater importance should be placed on these wetlands based on the presence of habitat suitable for the observed rare butterflies. Wetland F28 was scored above the model mean for the biological functional value. This does not quantitatively represent the value of this bog for rare butterfly habitat.

Wetlands F66, F65 and F64 are all part of a contiguous series of wetlands which are hydrologically connected to Deep Hole Lake. Wetland F66 is dominated by conifer swamp but wetlands F65 and F64 contain a variety of habitat types. All three are adjacent to mesic hardwoods and posses a large area of edge habitat relative to their size. A population of the State Special Concern little grape-fern was observed in wetland F66. The State Special Concern pygmy shrew and Proposed State Special Concern four-toed salamander were observed in wetland F65. The State Special Concern bog copper was observed in wetland F64. Biological functional value scores of 94 and 116 for wetlands F66 and F65, respectively, are probably representative of the biological importance of these two wetlands. A score well below the model mean at 78 for wetland F64 undervalues the biological function. The presence of the rare bog copper butterfly warrants greater consideration for this wetland.

Wetlands F7 and F2 are both very high scoring for the biological function. This is a result of their juxtaposition next to Little Sand Lake. As expected, the biological surveys resulted in observations of rare species in both wetlands. However, these wetlands do not warrant any greater concern as a result of these observations.

Wetland F11 consists of an approximate 18 acre conifer swamp. Two species of State Special Concern shrews were observed in this wetland. However, this wetland is not particularly interesting from a biologic standpoint and both species of shrews were found in a variety of habitat types throughout the study area. In other words, this wetland is not necessarily essential for the existence of these two species. As with wetlands F7 and F2, this wetland does not warrant any greater concern as a result of these observations.

Wetland T4 totals approximately 199 acres and includes shrub swamp, conifer swamp and emergent marsh components. This wetland also borders Swamp Creek. As a result of its large size, position adjacent to Swamp Creek and the presence of a variety of adjacent habitat types, this wetland was scored well above the model mean at 118. Two species of State Special Concern plants and the State Special Concern black-backed woodpecker were observed in this wetland. However, the high biological function score already represents the biological value of this wetland.

Conclusions

The CMC biological and wetland assessment studies were conducted independent of one another. Further, the data resulting from these studies was reported separately (e.g., biological resources in Section 3.8 and 3.9 of the EIR and wetlands in Section 3.10). Although the previously reported wetland functional value scores considered the biological function, these scores did not reflect the habitat value associated with the presence of rare or protected plant or animal species cataloged during the 1993, 1994 and 1995 field studies. This memorandum combines the results of CMC's biological studies, that focused on rare, endangered and threatened species, with the results of the previously conducted wetland assessment studies.

The wetland functional value scores have not been updated, however, the importance of wetland habitat for the existence of rare or state and federally protected species has been identified. Each of the wetlands listed in Table 1 provides habitat for at least one rare or protected plant or animal species. Based on this analysis, wetlands O1, W1 and Z23 support the greatest diversity of rare biota in the wetland study area. Wetland F28 also deserves mention as it is the largest contiguous area of bog habitat in the study area and provides a substantial area of habitat for rare butterflies.

References

Foth & Van Dyke, May 1995. *Environmental Impact Report for the Crandon Project.*

Normandeau Associates, Inc. and Interdisciplinary Environmental Planning, Inc., 1982. *Wetlands Assessment Report.*

Table 1

Occurrence of Endangered, Threatened or Special Concern Species in Wetlands in the Vicinity of the Crandon Mine

Wetland	Acres	Biological Function Score	Endangered Species	Threatened Species	Special Concern Species
F2	20.3	141			yellow rail
F7	20.46	129			arctic shrew
F11	18.37	78			arctic shrew pygmy shrew
F16	8.81	74			bog copper bog fritillary inornate ringlet
F28	59.69	104			bog copper dorcus copper jutta arctic algal-leaved pondweed
F64	4.62	78			bog copper
F65	2.16	116			pygmy shrew four-toed salamander
F66	16.27	94			little grape-fern
O1	113.85	89		calypso orchid osprey	American yew northern black current sheathed sedge northern bog sedge marsh horsetail Connecticut warbler evening grossbeak

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Table 1 *(Continued)*

Wetland	Acres	Biological Function Score	Endangered Species	Threatened Species	Special Concern Species
T4	199.02	118			American yew sheathed sedge black-backed woodpecker
W1	436.64	101	mountain cranberry		northern black current American yew sheathed sedge sparse-flowered sedge boreal chickadee blackburnian warbler gray jay bobcat
Z2	15.6	90			bog fritillary
Z23	226.96	93		calypso orchid	bog fritillary jutta arctic black tern sparse-flowered sedge great gray owl[1] American black duck[1] American bittern[1] yellow-bellied flycatcher[1] blackburnian warbler[1] gray jay bobcat[1]

[1]Only the northern portion of wetland Z23 was included in the wetland study area . As a result, the wetland functional value score relates only to that portion of the wetland. These species were observed in the southern portion of this wetland outside of the wetland study area.

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Appendix Q

Crandon Project Pyrite Marketing Update, April 1995, Market Access International

CRANDON PROJECT

PYRITE MARKETING UPDATE

Prepared for

Foth & Van Dyke

Prepared by:

Market Access International, Inc.
200 Union Boulevard, Suite 430
Lakewood, Colorado 80228

April 1995

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1.0 EXECUTIVE SUMMARY

A review of prior studies on the processing and marketing of Crandon Project pyrites and potential derivative products has been conducted with regard to marketing and transportation issues. The earlier studies, which were performed in the 1981-1986 time period, all concluded on the basis of overall process economics that no viable market existed for these products. The process routes which showed the most favorable economics involved production of sulfuric acid as an intermediate in the production of diammonium phosphate (DAP) fertilizer, with an iron pellet by-product. These earlier studies assumed hypothetical processing centers at Green Bay, Wisconsin and Evansville, Indiana in order to determine the sensitivity of process economics to transportation costs.

This re-examination of current market conditions has focused on the most viable products identified in earlier studies, sulfuric acid, iron pellets and DAP, but also reviewed earlier marginal product alternatives in view of the time lapse since the initial work. This evaluation considers both processing and marketing of the 488,323 tons per year of pyrite fines and the 1,221,360 tons per year of total pyrite expected to be generated in the first 15 years of the Crandon Project. The lesser amounts of pyrite and potential products generated in later years would be relatively easier to place in markets, but may have an adverse effect on process economics. Methodology includes both a review of literature and telephone interviews with commercial sources.

It was found that the conclusions of earlier studies remains valid with regard to the lack of viable markets for pyrite concentrates, elemental sulfur, liquid sulfur dioxide, phosphoric acid and by-product gypsum.

The pertinent microeconomic market conditions for sulfuric acid from by-product pyrites have improved from the time of the original work, and the viability of marketing this product improves if it could be processed at Crandon rather than a remote processing center. This option would require installation of a roaster, wet gas cleaner, and an acid plant.

Market conditions for by-product iron pellets have weakened overall from the earlier studies, but if Crandon wished to pursue production of this product rather than wasting the iron cinders, a mixing unit to produce green pellets and a sintering plant for the final product would be required.

Finally, market conditions for DAP are less favorable than in the earlier time frame. Transportation and feed material costs have risen, and DAP prices have declined, resulting in a margin squeeze. If this avenue were to be pursued, it should focus on monoammonium phosphate (MAP) which is replacing DAP in the North American market.

In conclusion, the products which show the most promise of viability, based solely upon market conditions, are sulfuric acid and by-product iron material.

2.0 PURPOSE AND SCOPE

The purpose of this report is to establish current market conditions for pyrite and potential derivative products which would be produced by the Crandon Project. This work includes a review of the following prior studies with regard to marketing and market-related transportation issues. For the sake of consistency, metric ton units in these prior studies have been converted to short tons in this report.

Davy-McKee Engineers and Constructors, June 1981. *Crandon Project Pyrite Processing Study Summary*, 2489/03.

Grefe, Robert, Wisconsin Department of Natural Resources, December 14, 1984. Memorandum with attached "Review of the Crandon Project Reports Submitted by Exxon Minerals Company Assessing Possible Uses for Pyrite Tailings", Final Report, December 21, 1984, Andres A. Trevino.

Coefield, Thomas L. Associates, June 1986. *Crandon Project Pyrite Processing Update*.

The primary effort has been placed upon the four cases last updated by Coefield, which include sulfuric acid, iron pellet and diammonium phosphate products. However, in view of the time elapsed since the earlier studies, the demand and product requirements were also revisited for other potential products which were considered earlier and rejected. Methodology for this current market and transportation assessment includes both a review of literature and telephone interviews with commercial sources.

3.0 BACKGROUND INFORMATION

3.1 McKee Study Summary

This study was undertaken to evaluate the feasibility of directly marketing pyrite tailings from the Crandon Project, and, if not, to determine whether other marketable products could be made from the pyrites using proven technology. The marketing and market-related transportation assessments relied heavily upon *The Marketing Report* by Commodities Research Unit Ltd. (CRU) and *The Transportation Report* by Jones, Bardelmeier & Co. Ltd. respectively, which were incorporated in the full McKee study.

The Crandon Project was expected to produce about 294,000 tons per year pyrite fines and 1,203,400 tons per year total pyrites. In developing process alternatives, the options of utilizing fines only and of utilizing total pyrites were both considered. Hypothetical processing centers were assumed to be located at Green Bay, Wisconsin and Evansville, Indiana, primarily in order to assess the sensitivity of pyrite processing and marketing economics to transportation costs.

The process review identified a number of potential derivative products in addition to pyrite concentrates as follows:

Primary Products

Pyrite concentrates (fines)
Sulfur
Sulfuric acid
Iron pellets
Liquid sulfur dioxide

Secondary Products

Phosphoric acid
Diammonium phosphate (DAP)
Gypsum

Feed Product Supply Requirements

Phosphate rock
Anhydrous ammonia

Products Considered But Rejected

Of these potential products, liquid sulfur dioxide was eliminated from detailed consideration due to the fact that the expected production of 294,500 tons per year was greater than the entire regional market consumption. Similarly, the anticipated 2,640,000 tons per year by-product gypsum was eliminated, since 33,000,000 tons per year are produced in the United States, and only 550-650,000 tons per year are sold, with the balance being landfilled. In addition, the market for phosphoric acid was deemed to be dominated by captive production, and the analysis of this product was incorporated into the DAP analysis. Similarly, the evaluation of elemental sulfur showed good near term demand, but this was based upon the underlying demand for sulfuric acid and it was considered to be more cost-effective to produce acid directly in light of anticipated high production costs for sulfur.

Pyrite Concentrates

Only two potentially viable outlets were identified for direct sale of the pyrite fines, the Copper Range Company at White Pine, Michigan and the Cities Service Company at Copperhill, Tennessee. Copper Range could only use about 10% of the Crandon output and Copperhill would have had to shut down its own production in order to take the Crandon product. At the time it did not appear to present attractive economics and Copperhill did not want to pursue this source.

Sulfuric Acid

The attractiveness of sulfuric acid production was adversely affected by the transportation costs of pyrite to the processing centers. Also, at the time of the study, production of involuntary smelter acid was increasing and its use for oxide leaching had not yet been well established. Consequently, steep discounts were assumed for Crandon product which was assumed to be entirely involuntary production. Thus the economics of this product appeared unattractive at selling prices of $13.61/ton vs. $31.75/ton for acid from burned sulfur in 1978 dollars. This discount structure was projected to continue in the medium term until at least 1990. Therefore, this product was only seen as a potentially viable intermediate in the production of DAP.

Iron Pellets

McKee noted a good potential regional market for iron pellets which could readily absorb Crandon output in terms of both quantity and quality. However, overall market conditions were weak at the time, despite increasing consumption, due to excess supply capacity and these conditions were expected to persist at least until 1990. Consequently, prices were depressed and although Crandon product could probably be placed in the market volumetrically, its optimum potential was seen as a higher quality niche product, perhaps even suitable for direct reduction. A price of $38.10 per ton in constant 1979 dollars was assumed for Crandon product in the analysis extending until 1990.

A process was identified which could recover several nonferrous metals in the production of iron pellets, thus resulting in a credit as an intermediate for smelter feed. All of these metals are sold on terminal markets and hypothetical assumptions were made on their pricing as a smelter credit. It was assumed that realizations could be obtained of 60% of market for base metals and 80% of market for precious metals, an assumption which appears to be overly optimistic. Although the inclusion of iron pellets improved the overall economics of several process routes, McKee noted that the high capital costs for the relatively small pellet plant would not warrant such an investment.

DAP

The DAP market was forecast to see a two percent per year increase in demand for the next 20 years, driven mainly by the export market. Crandon product was assumed to address the Midwest corn belt market, being sold to co-operatives or bulk blenders. A projected 1990 price of $250-260 per ton was used for Crandon product, based upon an underlying F.O.B. Tampa price of $200 per ton, both in 1980 dollars. Although the domestic market was reasonably well saturated, sufficient growth was anticipated to accommodate this new production at competitive pricing.

Availability of the feed materials, phosphate rock and anhydrous ammonia, was expected to present no constraint, both being in ample supply through the year 2000. However, it was noted that the transportation costs for bringing phosphate rock to the processing centers accounted for 13% of overall DAP production cost.

Transportation Considerations

This analysis assumed the pyrite concentrates would move by rail to the processing centers. Green Bay would receive and ship all other commodities except anhydrous ammonia by Lake Michigan for eight months of the year, and revert to rail during freeze-up. Evansville would utilize barge via the Ohio River for all other commodities, and revert to rail for an assumed two month winter freeze-up. Pertinent rates were developed in dollars per metric ton F.O.B. destination in mid-1979 and projected to mid-1985, utilizing an intermodal rate forecasting model tied to the Chase Econometrics July 1979 long term Macro Economic Forecast.

3.2 Trevino Report

This study was undertaken to review the adequacy of the McKee feasibility study, in regard to methodology, data and conclusions, and includes an assessment as to whether the original conclusions were still valid at the time of this later study. Trevino found some minor errors and omissions which did not affect the final conclusions, and found that the McKee study, including its marketing and transportation source studies, was conducted in a professional manner. It was also found that an updated study would not change McKee's conclusions. Comments on the key findings relevant to marketing and transportation matters are provided below.

Transportation

Trevino considered the McKee study's more important flaw was the failure to adequately justify the elimination of the Crandon site as a processing center. Transportation of pyrite to Green Bay added $3,000,000 per year to the processing costs of fines and $13,000,000 per year to processing costs of total pyrite.

Sulfuric Acid

It was also found that McKee failed to fully establish the assumption that Crandon acid would have to bear a heavy discount at a sales realization of $13.61 per ton. However, Trevino's sensitivity analysis at $27.22 per ton still showed this product to be uneconomic so McKee's conclusion was considered valid.

Iron Pellets

Since the production of iron pellets with nonferrous metal credits improved the economics of all processing alternatives, Trevino found the McKee conclusion, that investment in pellet production was unwarranted, to be unsupported by the results of their study. Nevertheless, since the inclusion of pellets still did not sufficiently improve the economics of processing alternatives to acceptable levels, the final conclusion of McKee, that none of the alternatives was viable, was deemed to be valid.

Market Conditions

Trevino conducted an extensive literature review, and concluded that no new market opportunities existed and therefore market conditions did not merit an update. It was also found that the McKee approach of using a constant dollars basis for costs and prices was still valid, since there had been no significant relative shifts in costs and prices.

3.3 Coefield Update

The purpose of this report was to determine whether changes in the project plan, credit for reduced costs of a tailing disposal facility or changed market conditions would individually or in combination change McKee's conclusion that the sale of pyrites or their processed derivative products was unfeasible.

The project plan now contemplated mining the zinc zone first, rather than a blend of ores, and this would result in significantly increased pyrite production in the initial stage. For the first 15 years pyrite production would be 488,323 tons per year as fines and 1,221,360 tons per year in total. During the remaining 14 years of the project life, 282,191 tons per year of fines would be generated and 705,478 tons per year of total pyrite. At the time of the McKee study, the plan called for 294,000 tons per year of fines and 1,203,400 tons per year total pyrite production throughout the project life.

For updating of market conditions, Coefield relied upon a literature review. This update was performed with a focus upon four of the more promising cases identified by McKee as follows:

Base Case - Pyrite fines processed to sulfuric acid, cinders wasted or processed to iron pellets with nonferrous metal credit.

Alternative Case "A" - All pyrite processed to sulfuric acid, cinders wasted or processed to iron pellets with nonferrous metal credit.

Alternative Case "B" - Pyrite fines processed to sulfuric acid and then used in the production of DAP fertilizer, cinders wasted or processed to iron pellets with nonferrous metal credit.

Alternative Case "B-1" - All pyrite processed to sulfuric acid and then used in the production of DAP fertilizer, cinders wasted or processed to iron pellets with nonferrous metal credit.

Table 3-1 was modified from the Coefield report by converting from metric units, and shows the product tonnages and unit prices which were used to develop revenues. The unit prices were the same as established by McKee in the earlier study.

The update of market conditions focused entirely upon sulfur and derivative products, drawing from two articles included with the report, Boyd & Phillips[1] and E&MJ[2]. Coefield concludes that despite a favorable demand outlook for sulfur, pyrite processing will not share in this growth but will decline in relative terms due to the comparatively high cost of this production method. However, no distinction is made between primary pyrite and by-product pyrite processing. Boyd & Phillips projected an increase in the latter as sulfur prices improve (emphasis added). They also noted that the lower material cost offsets to a degree the higher capital cost for processing by-product pyrites, resulting in a required sulfur equivalent price per metric ton of $110 vs. $116 ($99.79 vs. $105.23/ton) for primary pyrites and $105 ($95.25/ton) for smelter offgas, all for production of sulfuric acid at a 15% discounted cash flow rate of return for a new operation. For by-product pyrites this equates to a $53 per tonne ($48.08/ton) realization for sulfuric acid.

The E&MJ article quoted an acid broker to the effect that $10 per short ton would be a good price for smelter acid. As a result, Coefield concluded that the McKee assumed price of $13.61 per ton realization from sulfuric acid was "very optimistic" and may cause total revenues in some of the cases to be overstated.

Coefield concluded that the McKee study was favorable to the economics of pyrite processing and marketing, but despite this, the economic results remained poor. The review of marketing literature did not reveal an outlet for pyrite concentrates and confirmed the market assessment for derivative products made by McKee. Since market conditions were found not to have changed, there was no reason for changing McKee's conclusion that the pyrite or pyrite products had no viable markets.

[1] R. Boyd and T.D. Phillips, April 1985. *Economics of World Sulphur Supply to the End of the Century*. The Fertilizer Society of London.

[2] Engineering and Mining Journal (E&MJ), May 1985. *Marketing Cogenerated Smelter Acid*.

Table 3-1
Annual Revenues - 1979 $

Product	Annual Production Amount	Units	Values per Unit	Yearly Revenues	Total Revenues: Acid Only	Acid and Pellets	DAP Only	DAP and Pellets
Process Only Fines								
Sulfuric Acid	729,068	tons	$13.61	$ 9,922,616	$ 9,922,616	$19,969,396	$114,320,600	$124,367,380
Iron Pellets	263,695	tons	$38.10	10,046,780				
Diammonium Phosphate	571,603	tons	$ 200	114,320,600				
Nonferrous Metals								
Zinc	2,921	tons	$ 417	1,218,057				
Copper	529	tons	$1034	546,986				
Lead	485	tons	$ 522	253,170				
Precious Metals								
Silver	234,379	ozs	$ 6.22	1,457,837				
Gold	10,609	ozs	$248.83	2,639,838				
Subtotal Nonferrous & Precious				$ 6,115,888				
Process All Pyrite								
Sulfuric Acid	1,889,800	tons	$ 13.61	$ 25,720,178	$25,720,178	$50,848,423	$295,231,600	$320,359,845
Iron Pellets	659,534	tons	$ 38.10	25,128,245				
Diammonium Phosphate	1,476,158	tons	$200	295,231,600				
Nonferrous Metals								
Zinc	4,178	tons	$ 417	1,742,226				
Copper	2,083	tons	$1034	2,153,822				
Lead	1,168	tons	$ 522	609,696				
Precious Metals								
Silver	592,859	ozs	$ 6.22	3,687,583				
Gold	27,328	ozs	$248.83	6,800,026				
Subtotal Nonferrous & Precious				$ 14,993,353				

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4.1 Pyrite's Role in the Sulfur Market - Overview

At the time of the Davy McKee study, *Crandon Project Pyrite Processing Study Summary*, 1981, and the Coefield & Associates study *Crandon Project Pyrite Processing Update*, 1986, the world consumption of pyrite concentrates for acid production was a significant factor in total western production of sulfuric acid for industrial consumption. Changes in the world sulfur market have altered the role of pyrite concentrates significantly.

Significant trends in world sulfur production during this period saw total world demand for sulfur build at a slow rate from 1980 to 1987, peak in the years 1987 to 1990, and decline at a moderate rate in 1990 to 1993. Peak world sulfur demand was 60.4 million metric tons (66.6 million tons) in 1988, and declined 13% to 52.5 million metric tons (57.9 million tons) in 1993.[1]

Importantly, the production of sulfur as the result of the de-sulfurization of oil doubled in the period 1980-1993, from less than 6 million metric tons (6.6 million tons) to 12.2 million metric tons (13.4 million tons), and sulfur recovered from natural gas processing increased as well, for the total of sulfur recovered from gas and oil production to account for 55% of world sulfur production.[2]

The increase in world sulfur production recovered from oil has been a large factor in the approximately 60% decline in mined sulfur production in major countries in the period 1980-1993. Much of this decline is attributable to reduced mined production from Poland and the C.I.S. However, U.S. mined sulfur production has dropped significantly during this period due to the environmentally-mandated recovered sulfur production from oil and gas processing.

The effect of this increase in recovered sulfur production during the period 1980-1993, which saw overall world sulfur demand increase only marginally, has been a sustained decrease in world sulfur prices since 1985, and an almost continuous decrease in the inflation-adjusted dollar price of sulfur since 1980.

A benchmark for U.S. sulfur prices has been the ex-Tampa, Florida price for liquid Frasch-process sulfur due to the significant consumption of sulfur in the Tampa area for phosphate fertilizer production. The price for Tampa Frasch-process sulfur held in the $120 to $150 per metric ton ($109-136/ton) range for the period 1981 to 1990, and has declined significantly since then to a low of approximately $50 per metric ton ($45/ton) in 1993[3] and has a current price of approximately $64 per ton.

The trend of recovered sulfur playing a market-determining role in world sulfur prices is irreversible, and recovered sulfur has by the nature of its forced production replaced a portion of mined sulfur production and pyrite production.

World production of pyrite concentrate has declined due to steadily lower sulfur prices as the result of recovered sulfur production and a large reduction in European pyrite demand

due to the rationalization and reduction in capacity of the European fertilizer industry. Overall world pyrite production statistics are strongly distorted by increasing production in China, which is still constructing greenfield pyrite roasters, including two constructed by Boliden Contech,[4] in order to meet its burgeoning industrial demand for sulfuric acid.

Elsewhere in the world, 1990 was a pivotal year in both pyrite production and the sulfur market, with pyrite production outside China declining 50% in the last five years.[5] The principal pyrite production area outside of China has been Europe. Table 4-2 illustrates the precipitous decline in European pyrite production from 1987 to 1993:

Table 4-2
European Pyrite Production
Thousand Metric Tons S (Thousand tons S)

	1987[6]	1990[7]	1993[8]
Cyprus	43 (47)	0 (0)	0 (0)
Finland	340 (375)	357 (394)	350 (386)
Germany	255 (281)	150 (165)	0 (0)
Greece	71 (78)	60 (66)	25 (28)
Italy	291 (321)	290 (320)	175 (193)
Norway	154 (170)	125 (138)	125 (138)
Portugal	132 (146)	95 (105)	95 (105)
Spain	971 (1070)	748 (825)	327 (360)
Sweden	248 (273)	121 (133)	40 (44)
Turkey	50 (55)	65 (72)	33 (36)
	2555 (2816)	2011 (2217)	1170 (1290)

Spain has traditionally been the engine of European pyrite production, with Minas de Tharsis, Rio Tinto Minera SA, and Andaluza de Piritas (APIRSA) having been the major producers. Spanish pyrites were in part consumed in Spain, principally in Huelva, and in part exported to Northern Europe. With the decreasing economic nature of phosphate fertilizer production in Europe, some major pyrite roasters such as Tessenderlo Chemie in Belgium and Kemira Kemi in Sweden have either closed or converted to elemental sulfur burning.[9] The drastic decline in Spanish pyrite production is mirrored elsewhere in Europe, with only Finnish pyrite production maintaining its level of previous years.

In the United States, the changes in the economics of sulfur supply resulted in the virtual elimination of the use of pyrites for roasting.

One of the scenarios examined by Davy McKee and Coefield was the roasting of Crandon Project pyrite concentrate at Copperhill, Tennessee. The Copperhill roasting plant was an outgrowth of a small scale copper mining and smelting operation. This operation evolved into a copper chemical and sulfuric acid production site utilizing an underground

copper/pyrite mine for raw material, four pyrite roasters and two sulfuric acid plants. Each roaster had a capacity of 2,000 tons per day.

During the 1980's, Park Corp. of Ohio had purchased Copperhill, and continued to operate it. The collapse of the main haulage shaft of the mine ultimately forced Copperhill into bankruptcy. B.I.T., a U.S. subsidiary of Trelleborg Boliden, purchased the facility about 1990.

In approximately 1989, the pyrite roasters and one of the two acid plants were taken off line and were replaced with an elemental sulfur burner. The roasters and the unused acid plant are still in existence, but are of negligible scrap value according to B.I.T. site personnel. It is possible that B.I.T. could consider restarting part of the plant if pyrite feed was economically available; however, the estimated cost of transportation from Crandon to Copperhill is $39.33 per dry ton, and together with the condition of the Copperhill plant, renders this option unlikely.

The other possible existing destination for the pyrite concentrate in the U.S. was the Copper Range Co. smelter at White Pine, Michigan. Copper Range during the 1980's sourced pyrite concentrate from Noranda as part of its smelter feed mix. In recent years, most of the sulfur make-up component of the feed mix has been supplied by sulfide copper concentrates supplied from outside the company. Copper Range site personnel have advised that once the current evaluation of the possible re-start and expansion of the smelter is settled, future sulfur feed requirements of the smelter will probably continue to be met through outside sulfide copper concentrate supply. Thus, the possibility of the Crandon Project shipping even small tonnages of pyrite concentrate to Copper range is unlikely.

According to Ms. Joyce Ober, the U.S. Bureau of Mines specialist for sulfur, the sole use of pyrites for sulfuric acid production in the U.S. appears to be a small operation in Arizona that produces the equivalent of 4000 tons per year of sulfur contained.

4.2 Sulfuric Acid

In 1993, the consumption of sulfuric acid in the United States was 13,886,000 metric tons (13,102,057 tons) of sulfur equivalent, broken down as follows:[10]

	Thousand metric tons S content	(Thousand tons S content)
Phosphatic fertilizers	7,906	(8,715)
Copper Leaching	696	(767)
Inorganic pigments & chemicals	549	(605)
Petroleum refining	388	(428)
Pulp mills and paper	304	(335)
Synthetic rubber and plastics	259	(285)
Other uses	1,784	(1,967)

The production of sulfuric acid is accomplished by several alternate routes, with virgin sulfuric acid being produced from elemental sulfur, either mined or recovered. In the past 25 years a substantial volume of byproduct sulfuric acid has been produced from the mandatory processing of nonferrous smelter exhaust gases.

Salient characteristics of the production of smelter acid over the past 25 years are:

A. The early period of production of smelter acid was affected by quality control problems such as black acid production and also by industrial consumer resistance to the product.

B. Smelter acid capacity was added by mandate at a faster rate than the industrial consumer market for the product.

C. Nonferrous smelters are involuntary producers of acid in that they cannot cease production in periods of excess acid supply/ decreased demand, and by the sheer volume of production do not have extensive storage capacity and must sell product as it is produced, sometimes with poor results on an F.O.B. pricing basis.

D. The number of nonferrous smelters in the U.S. has declined at a large rate in the last 25 years. With the failure of Mitsubishi Corp. to build the Texas Copper smelter due to permitting difficulties, it is unlikely that any substantial increase in smelting capacity will occur in the U.S. in the foreseeable future.

E. Large use segments for the leaching of copper ores in the U.S. Southwest and the leaching of uranium in Canada have developed and now consume large amounts of smelter acid, relieving both shipping and pricing pressure on smelter acid producers running integrated operations.

F. The installation of sulfuric acid plants at smelters in the U.S. occurred early on, and the installation of sulfuric acid plants at Canadian smelters is almost complete. The U.S. smelter acid plants were put in place in the 1970s, causing a bulge in smelter acid supply in a market that was developing. The decline in the number of U.S. smelters since the 1970s and the large increase in market demand for smelter acid has more than offset the production arising from the installation of sulfuric acid plants at the Canadian smelters of Inco, Falconbridge, Hudson Bay, and Noranda in recent years.

Thus, the U.S. market for smelter acid has changed dramatically from the time of the McKee and Coefield studies. Smelter acid is now in good demand and meets acceptance by a large number of acid consumers.

In terms of quality, acid produced from the Crandon Project pyrite concentrate should be at least marginally more acceptable to consumers than smelter acid. Consumer acceptance

would then step up to acid produced from burning elemental recovered sulfur and then to acid produced from burning Frasch-process or mined elemental sulfur.

The geography of sulfuric acid production in North America is heavily concentrated in the U.S. Gulf, Eastern, and Southwest regions. The transportation economics of the sulfuric acid market are oriented towards production in the U.S, Gulf and Southwest. Consequently, the U.S. Midwest has traditionally been the premium market for sulfuric acid in the U.S.

Current virgin 100% sulfuric acid pricing is on the order of $68 to $74 per ton in the U.S. Gulf, and $78 to $84 per ton in the Midwest. Smelter acid is currently a $27 to $29 per ton discount from virgin acid. It is thought that this discount may decrease further from current or past levels due to increasing demand for smelter acid and the decreasing real price for sulfur due to the increases in recovered sulfur production.

In the course of this study, GEO Chemicals of Cleveland Ohio was contacted to discuss the sulfuric acid needs of their aluminum sulfate plants in Louisiana, Alabama, and Tennessee. GEO Chemicals purchases 33,000 to 55,000 tons per year of sulfuric acid, and now purchases smelter acid for $40 to $50 per ton. GEO Chemicals expressed interest in diversifying their sourcing of acid and in examining commercial possibilities with a new producer.

The Midwest is the U.S. premium market for sulfuric acid due to the fairly long transportation distances from the major acid production centers, and the Midwest's large industrial usage of acid. Canadian smelter acid traditionally goes to the Eastern U.S. and the eastern portion of the Midwest such as Ohio, northern Indiana, and Michigan. The western portion of the Midwest represents a niche market for sulfuric acid that can absorb the production from a new moderate-sized producer.

If the pyrite concentrate from the Crandon Project is converted into sulfuric acid, Case A would represent a 1.8% increase in U.S. sulfuric acid supply and Case B would represent a 4.6% increase in sulfuric acid supply. Given that the western and upper Midwest is a niche, premium market for acid, it is thought that the sulfuric acid production from roasting the Crandon Project pyrite, at least in Case A, can be absorbed by the market with minimal impact on price and with short transportation distances to consumers. In 1993, pulp mills and paper producers met 36% of their sulfuric acid needs from smelter acid. Wisconsin is a center of paper production, with producers such as International Paper, Appleton, Riverside, Mosinee, Nicolet, Gilbert, Menasha, Flambeau, Badger and Wausau paper mills. These producers are well served by rail transport, and a sulfuric acid production site on the main line of the Wisconsin Central would provide the necessary flexibility for both rail and truck transport of acid. Crandon is on the Wisconsin Central main line; Green Bay is on a branch line.

Any potential production of sulfuric acid at Evansville would impose lower sales prices on product as strong competition from nearby acid producers, Canadian smelter acid, and Gulf Coast production will exist.

Therefore, any production of sulfuric acid from the Crandon Project pyrite concentrates could be most effectively marketed from a Wisconsin production site to paper mills and other nearby sulfuric acid consumers, and most effectively from Crandon itself. Process equipment would include a roasting unit, such as a fluidized-bed reactor, an efficient wet gas cleaning plant and a sulfuric acid plant.

In the current market, it is estimated that an F.O.B. production site value of $36 per ton of sulfuric acid can be obtained if the acid is properly marketed. In addition, previous studies have characterized the Crandon Project pyrite concentrate sulfuric acid potential production as being involuntary. The U.S. Bureau of Mines correctly characterizes pyrite-based sulfuric acid production as voluntary[11], but this would only be true for by-product pyrite such as Crandon product if provisions were made in times of poor acid markets to stockpile pyrite. However, if storage facilities had to be constructed for this contingency, the viability of a pyrite marketing program to avoid pyrite containment cells would be undermined.

4.3 Liquid SO2

The Mckee study has correctly identified that the market for liquid SO2 in the potential process area is too small in relative size to absorb the production from a Crandon Project processing plant exclusively devoted to liquid SO2 production. It appears that a more typical liquid SO2 plant size is 30,000 metric tons (27,216 short tons)[12], an order of magnitude below Crandon product volume.

4.4 Iron Pellets

At the time of the McKee and Coefield studies, the large-scale market for iron ore pellets was a fairly recent development. In 1995, the market for iron ore pellets in the U.S. is both substantial in size and well developed. If the entirety of the iron cinders produced in Case B pyrite roasting were successfully pelletized, the steel production market could absorb the entire tonnage. Pellet processing would require a mixer to produce green pellets and a calciner or belt sintering machine to heat treat this feed material to hard pellets for shipment. Transportation of iron pellets through Wisconsin is now a routine matter; the Wisconsin Central Railroad ships pellets from Minntac in Minnesota on its main line to Geneva Steel in Geneva, Utah as part of a special routing situation[13].

Current pricing for standard iron ore pellets is $30 per ton F.O.B. vessel in the upper Great Lakes.

An alternative to pellet production is the supply of sintered or semi-sintered iron product to either Algoma Steel Company in Sault St. Marie, Ontario or steel mills in the

Chicago area. The Algoma Steel plant rails in considerable quantities of iron ore to its 1.2 million metric ton (1.09 million short ton) per year sinter plant in Wawa, Ontario to ultimately supply its two blast furnaces and one BOF.[14] Algoma Steel is serviced by the Algoma Central Railway, which has been recently purchased by the Wisconsin Central, and Algoma Steel has requested possible product specifications from potential production of iron materials by the Crandon Project.

As part of the production of iron materials, previous studies have examined the recovery of nonferrous and precious metal values from the iron cinders. A chloridization technology was selected to accomplish removal of the majority of these values from the iron cinders.

For purposes of this study, the nonferrous and precious metals recoveries calculated by Coefield will be assumed to be valid, as well as the 60% realization of the base metal contained value in the recovered material, and the 80% realization of the precious metal value of the recovered material. Underlying prices have been updated to zinc $0.65 per pound, lead $0.30 per pound, copper $1.20 per pound, silver $4.95 per ounce, and gold $399.72 per ounce by Pincock, Allen and Holt.

It should be noted that recovery of metal values in similar situations is very problematic. One of the reasons given for the cessation of pyrite roasting by Kemira Kemi of Sweden was the difficulty of disposal of heavy metal containing wastes.[16] This occurred in an area with excellent nonferrous smelting facilities.

If the chloridization technology does not work, it appears that traditional baghouse recovery would be an alternative, with all the attendant problems of disposal of metal containing baghouse dust. Matters may be simplified if the nonferrous residues can be blended with the Crandon Project virgin zinc or copper concentrates. However, it is possible that instead of 60% and 80% realization of the contained metal values, realizations for the nonferrous material may only be on the order of 10% to 50% of the contained metal value.

4.5 Phosphoric Acid

Very little appears to have changed from the time of the McKee and Coefield studies; almost the entirety of phosphoric acid production is utilized in the captive production of phosphate fertilizer. Only 386,000 tons of phosphoric acid were sold as product in 1993. The only new phosphoric acid production capacity was a plant built by Chevron in Rock Springs, Wyoming.[17]

4.6 Gypsum

The possibility of successfully marketing byproduct gypsum from phosphate fertilizer production remains almost as distant as it was in 1981 and 1986. In the course of this study, the gypsum specialist at the U.S. Bureau of Mines, Mr. Gordon Austin, was contacted. Mr. Austin estimates that of the 20 million tons of byproduct gypsum produced in the U.S. in

1993, only 800,000 tons were successfully sold at an average price of $4.00 per ton, and that almost a necessary precondition for selling byproduct gypsum is to have an adjacent, symbiotic wallboard plant. Austin estimates disposal costs of unsold byproduct gypsum at $3.00 per ton, and strongly recommended against Evansville, Indiana as a production site as this site would compete directly against Public Service Co. of Indiana which produces 1,000,000 tons per year of gypsum from its stack scrubbers and can only sell 100,000 tons per year.

4.7 Diammonium Phosphate (DAP) Fertilizer

The market for DAP has changed considerably since 1981. The period prior to 1981 saw extraordinarily strong fertilizer demand. The phosphate fertilizer industry has been in a decline since 1988, and world phosphate fertilizer demand has fallen 20% since 1988.[18] Prices for DAP reached $108 per metric ton ($98 per ton) F.O.B. Tampa early in 1993, a 20 year low, and recovered only to $158 per metric ton ($143 per short ton) in 1994.[19] Current prices for DAP remain at the $140 per ton level.

The depressed state of both the U.S. and world phosphate markets has resulted in the merger of the phosphate assets of IMC Fertilizer and Freeport McMoran's phosphate operations into the entity IMC-Agrico. This merger placed control of a substantial portion of the U.S. industry into the hands of a single party.

A significant market change has occurred in that the export market accounts for the majority of U.S. phosphate fertilizer production, a reversal of the situation in 1981. Of the smaller portion produced for the domestic markets, the use of DAP by farmers has been declining for the last ten years as monoammonium phosphate (MAP) has gained acceptance and even preference in parts of the country. Monoammonium phosphate is priced $4.50 to $5.50 per ton higher than DAP. Therefore, it is recommended that any potential further examination of production of phosphate fertilizer by the Crandon Project include monoammonium phosphate.

FirstMiss Fertilizer Corp. was contacted regarding the potential supply of anhydrous ammonia to Evansville and Green Bay. The current market for anhydrous ammonia is $260 per ton in the U.S. Gulf, and a price indication delivered Evansville is $280 per ton. A price indication for delivery to Green Bay by rail is $285 per ton.

Regarding phosphate rock supply, a viable market for the import of Moroccan phosphate rock into U.S. Gulf ports[20] will tend to keep U.S. suppliers somewhat competitive despite the IMC-Agrico merger. The current published price for 66-68% phosphate rock F.O.B. Tampa is $30 per ton. Transportation and unloading costs for delivery via barge to Evansville or comparable upriver port such as Ft. Madison add $20 per ton to the cost, thereby phosphate rock is estimated to have a delivered Evansville price of $50 per ton.

It is assumed that Moroccan material, which can be competitively imported into the Gulf, can be delivered to Green Bay via ocean vessel for the same cost or less.

It is very interesting to note that while the McKee and Coefield studies were being prepared a modern DAP production facility was available in Ft. Madison, Iowa. The plant was built in 1967 on 1603 acres by Sinclair Oil Co., passed into the ownership of Atlantic Richfield following a merger, and was divested. FirstMiss Fertilizer leased the facility for a number of years and shut down the facility in 1981, with a final shutdown in 1982 following processing of phosphate rock stockpiles. The reason given by a former plant administrator for the shutdown was the increase in natural gas prices from $0.45 per mcf in 1967 to $5.85 per mcf in 1981-82, which made the manufacture of ammonia at the site uneconomic. According to the former administrator, the plant was very carefully mothballed in 1982, and Freeport McMoran, the owner (now IMC-Agrico) kept workers at the site for maintenance until the Mississippi River flood of 1993 broke the river dike and inundated the plant. Interestingly, the plant produced its own sulfuric acid from a sulfur burner at site and acid plant, rather than purchasing smelter acid from the AMAX molybdenum roaster in Ft. Madison. The plant's equipment was left relatively intact with the exception of the ammonia converters which were said to have been sold to China by Freeport. The plant is now in uncertain condition, but the Ft. Madison Economic Development Group appears to be very desirous of the site being utilized. IPSCO, the Canadian steel producer, examined the site before settling on a plant site elsewhere in Iowa. Therefore it is believed that economic incentives may be offered by Ft. Madison to a potential user of the plant.

Table 4-3
1995 Reported Prices for Principal Materials

Phosphoric acid, 75% in tanks, delivered	$61 per ton
Phosphate rock, 66-68% b.p.l., FOB Tampa	30
Sulfur Dioxide, liquid, in tanks, works	230
Sulfuric acid, virgin 100%, Midwest	78-84
virgin 100%, Gulf	68-74
smelter 100%, Midwest	50-55
smelter 100%, Gulf	40-45
Sulfur, Frasch, Tampa	64
DAP, U.S. Gulf, Florida	141
MAP, U.S. Gulf, Florida	145
Iron pellets, FOB vessel, upper Great Lakes	30
Direct reduced iron, FOB vessel, upper Great Lakes	100

1 Foster, Robert B., March 1994. *Sulphur: Difficult Market Conditions Continue*. Engineering and Mining Journal.
[2] Foster, Robert B., op. cit. page 49.
[3] Foster, Robert B., op. cit., page 49.
[4] Kennedy, Alan, Nov. 1992. *Trelleborg Boliden: Global Expansion Continues*. Mining Magazine. Page 11.
[5] Foster, Robert B., op. cit., page 49.
[6] Sulphur Magazine No. 211, Nov.-Dec. 1990. *Fool's Gold Loses Its Lustre*. Page 18.
[7] Ober, Joyce A. *Sulfur 1993*. U.S. Bureau of Mines. PP. 19-25.
[8] Ober, Joyce A., ibid.
[9] Sulphur Magazine No. 211, op. cit., page 18.
[10]Ober, op. cit., page 14.
[11]Ober, op. cit., page 9.
[12]Sulphur Magazine No. 210, Sep.-Oct. 1990. *Record Levels*, page 20.
[13]Trains Magazine, March 1995. *WC: More Ore than Paper?*, page 42.
[14]*1994 Directory of Iron and Steel Plants*, page 96. Association of Iron and Steel Engineers.
[15]Steel Times Magazine, Dec. 1994. Pages 491-492.
[16]Sulphur Magazine No. 211, op. cit, page 18.
[17]Mew, Michael, March 1994. *Phosphate Rock: First Green Shoots of Recovery*. Engineering and Mining Journal, page 52.
[18]Mew, op. cit., page 50.
[19]Mew, op. cit., page 51.
[20]Mew, op. cit., page 53.

5.0 TRANSPORTATION

The transportation considerations and prices for transportation of the Crandon Project pyrite concentrate are also applicable to the movement of Crandon Project nonferrous concentrates. The U.S. and world water shipping system is straining at capacity in 1995, and prices for water-borne transport have increased markedly in the last two years. It is expected that the water-borne freight market will moderate by the time of the inception of shipments, as additional capacity is added or demand is slackened in the water-borne segment. Therefore, it is believed the prices contained in this study represent a near-worst case scenario.

5.1 Barge Shipping Economics

For the barge shipping rates for this study, American Commercial Barge Lines of Jeffersonville, Indiana was contacted. ACBL operates a barge shipyard and operates a large barge fleet on the Mississippi River system.

ACBL advised on an indication basis that there should be little difference in rates for originating movements from St. Paul, MN, Red Wing, MN or La Crosse, WI, hereinafter collectively referred to as "St. Paul." ACBL gave the following indications for movements of sulfide concentrates:

St. Paul	- Evansville, IN	$17.60 per dry ton
	- Chattanooga, TN	22.00
	- East St. Louis/Cahokia	8.80
	- Burnside, LA	16.50
	- Catoosa, OK	22.00

All basis 10% moisture in the cargo and a minimum loading of 1400 wet tons per 1600 ton capacity barge.

ACBL also indicated the following rates:

For iron pellets	
Evansville to Granite City, IL	$7.00 dry ton
For DAP	
Evansville to Ft. Madison, IA	8.50
For phosphate rock (20% H2O assumed)	
Burnside to Evansville	7.78

Burnside, LA was chosen for the southern terminus of movements because it has been the traditional low cost, high volume downriver transfer point. Direct transfer of bulk cargo at Burnside only costs $2.27 per wet ton. For upriver handling points and at Green Bay, this study will assume a material transfer cost of $3.18 per wet ton.

An indication was sought for Chattanooga, TN as a discharge site for pyrite movements to Copperhill, TN. Indications were sought for East St. Louis and Catoosa, OK for informational purposes for the Crandon Project in the potential movements of zinc concentrates to zinc smelters served by these ports.

5.2 Ocean Shipping Considerations

Should the situation regarding pyrite roasting in Europe somehow change from the current and foreseeable situation of there being a lack of viable sales outlets for the Crandon Project pyrite concentrate, the following economics would apply to shipments of concentrates with 10% moisture content, which is of importance due to the International Maritime Organization's regulations governing moisture containing cargos:

Truck transport to Green Bay	$11.18 per dry ton
Loading onto vessel - Green Bay	3.49
Ocean freight - Green Bay to Antwerp or Rotterdam	32.93
Total to C&F Europe	$47.60 per dry ton
Truck transport to La Crosse	$24.19 per dry ton
Loading at river port	3.49
Barge freight to Burnside	16.50
Loading onto vessel Burnside	2.49
Ocean freight - Burnside to Antwerp or Rotterdam (basis 25,000+ tons)	17.96
Total to C&F Europe	$64.63

The ocean freight indications were obtained from M.I.D.-Ship Marine, Inc. of Long island, New York, a large broker of ocean freight. M.I.D.-Ship Marine advises that ocean shipping is in very high demand this year, and that ocean freight prices have exploded. Therefore, these prices for vessel space represent a near-worst case scenario, at perhaps 20% higher than average over the past few years. By the time the Crandon Project enters production it is anticipated that ocean freight rates will have subsided somewhat, in keeping with normal fluctuations of this market.

For the upper Great Lakes to Europe movement, M.I.D.-Ship Marine contacted Federal Navigation of Toronto, which operates 14 "Max Laker" size vessels of 24,000 metric tons capacity, and a variety of smaller bulk vessels on the Lakes. Federal Navigation advised that long-term business from Green Bay was of interest, and that a better price could be achieved than the current $30 per wet ton indicated for May, 1995 loading, inclusive of Welland Canal and St. Lawrence Seaway tolls.

The Green Bay indication was basis a loading rate of 5,500 tons per day, Sundays and holidays included (SHINC) and a vessel discharge rate of 8,800 tons per day (SHINC). The

Burnside indication was basis 8,800 tons per day (SHINC) load, and 8,800 tons per day (SHINC) discharge.

M.I.D.-Ship Marine's domestic arm was contacted regarding the movement of phosphate rock from Tampa to Burnside, LA. M.I.D.-Ship Marine advised that they handle a great deal of ocean barging in the Gulf, and that this movement should cost a maximum of $5.00 per wet ton in the current market. Assuming a 20% moisture content in phosphate rock, the freight to Evansville or Ft. Madison would be:

Ocean barge to Burnside	$6.00 per dry ton
Cargo transfer at Burnside	2.72
Barge to Evansville	7.78
Unloading at Evansville	3.81
Total	$20.31 per dry ton

5.3 Truck Transport Considerations

For the purposes of this study, a number of trucking firms in the Green Bay area were contacted in order to obtain competitive rates for semi truck end-dump hauling of substantial volume. The initial search for an established trucking firm in the area was unsuccessful. The contacting of an area freight forwarder and a truck freight broker in Milwaukee did not yield results.

Given what initially appears to be a commercially under-developed bulk trucking situation in the Crandon - Green Bay area, the commercial ore-hauling firm of MacFarland & Hullinger of Utah and Nevada was contacted to establish comparable rates for this movement. Using the assumption that Wisconsin only allows 40 ton (80,000 pound) gross vehicle weights, which allows cartage of 24 tons of cargo, and that Wisconsin prohibits extra axles or the use of trailers with "pups," MacFarland & Hullinger calculated the following indicative rates:

Crandon to Green Bay	107 miles	$11.18 per dry ton
Crandon to La Crosse	224 miles	$24.19 per dry ton
Crandon to St. Paul	243 miles	$26.40 per dry ton

basis 10% moisture in the concentrates. MacFarland & Hullinger advised that for winter transport the dump trailers would have to be modified to pipe the truck exhaust underneath a stainless steel sheet so as to keep the concentrates from freezing solid.

5.4 Rail Transportation Considerations

Crandon is on the main line of the Wisconsin Central Railroad, which is an aggressive, innovative short line railroad that is rapidly growing.

Mr. John Carey, the iron ore freight specialist of the Wisconsin Central, was contacted with a request for indicative rates for open gondola car shipment of pyrite concentrate and iron pellets. The Wisconsin Central has advised the following indicative rates for pyrite concentrates:

Crandon to Green Bay	$ 7.97 per dry ton
Crandon to White Pine, MI	$18.63 per dry ton
Crandon to Evansville, IN	$37.13 per dry ton
Crandon to Copperhill, TN	$39.33 per dry ton
Crandon to Ft. Madison, IA	$30.53 per dry ton

These are based upon 10% moisture content, and rates for iron pellets and DRI would be approximately reduced by the moisture allowance. Green Bay and White Pine are serviced directly by the Wisconsin Central; for the other destinations another carrier will take part of the freight. In the case of both Evansville and Copperhill, CSX was assumed to be the carrier. Regarding the rail transport of sulfuric acid, the Crandon Project would most likely have to have its own fleet of tank cars, unless selling to a broker with its own equipment. Usually when a shipper provides its own equipment, the rail freight rate is reduced; however, in the case of sulfuric acid, this is a hazardous cargo and consequently would have a higher freight rate than materials such as pyrites.

Table 5-1
Truck and Rail Rates

		$ per dry ton
Pyrite Truck	Crandon - Green Bay	11.18
Truck	Crandon - La Crosse	24.19
Truck	Crandon - St. Paul	26.40
Truck	Crandon - White Pine (est)	12.47
Rail	Crandon - Green Bay	7.97
Rail	Crandon - Evansville	37.13
Rail	Crandon - Copperhill	39.33
Rail	Crandon - White Pine	18.63
Rail	Crandon - Ft. Madison	30.53

Table 5-2
Total Freight and Handling Costs

		$ Per Ton
Pyrite	Truck/barge/truck to Copperhill	60.33
	Truck/barge/ship to Europe(Burnside)	64.65
	Truck/ship to Europe (Green Bay)	47.60
	Truck/barge to Evansville	48.78
Iron Pellets/DRI		
	Truck to FOB vessel, Green Bay	14.35
	Truck/barge Crandon - Granite City	39.98
	Rail Crandon to Chicago (est)	25.00
	Rail Green Bay to Chicago (est)	25.00
	Barge Evansville - Granite City	10.18
Phosphate Rock		
	Barge/barge Tampa - Evansville	20.30
	Barge/barge Tampa to Ft. Madison	20.30
DAP		
	Barge Evansville to Ft. Madison	8.50
	Rail Green Bay to Ft. Madison	27.75

6.0 CONCLUSIONS

In the 14 years since the earliest prior study, it is not surprising that many of the market conditions have changed. No infallible forecasting method has been discovered, so it is prudent in market assessments and forecasts to take a conservative viewpoint. For general reference, current updates on sulfur and phosphate markets from E&MJ, March 1995, are included in Appendix 6.0. In this current study it appears, from marketing considerations alone, that process routes which generate sulfuric acid and iron pellets would have the best potential. These marketing factors would of course have to be incorporated into overall process economics in order to determine the ultimate viability of processing and marketing Crandon Project pyrites.

Conclusions are given as follows for individual products of interest:

Pyrite's Position in Overall Sulfur Market

Primary pyrite production is in relative decline, under pressure by increased supply from recovered sulfur and involuntary smelter acid. By-product pyrite supply has some cost advantage over primary pyrite supply, but still has a higher hurdle cost for new acid production investment than smelter acid. There is still no identifiable U.S. market for Crandon pyrite concentrates, and therefore no change in the market conditions of the earlier studies for this product.

Although theoretically possible, there is no practical market for elemental sulfur produced from pyrite, since it would be more efficient to product sulfuric acid directly from pyrites, rather than through an additional step. Thus, there is no change in market conditions for this product option.

Marginal Products

No new viable market has been identified for liquid sulfur dioxide, phosphoric acid or by-product gypsum. The earlier studies' conclusions for these potential products remains valid.

Sulfuric Acid

Here market conditions have improved, as involuntary smelter acid is being more efficiently utilized in the market, especially in oxide ore leaching near its point of generation. The upper Midwest is a premium market due to its distance from major generation points. Therefore, under current market conditions, Crandon could expect a realization of at least $36 per ton F.O.B. Crandon works. If it had to be produced in Green Bay, net back would decrease by approximately $5.30 per ton due to transportation costs.

Iron Pellets

This product has also seen changed market conditions since the earlier studies. Although well-established as steelmaking feed, prices have suffered due to increases in supply and a shift to steelmaking processes which don't require pellets. Current upper Lake prices are $30 per ton F.O.B. vessel for standard pellets. Nonferrous credits in earlier studies may have been optimistically overstated by 30-40%.

DAP

Here there have been a number of changes from earlier market assessments. There has been a margin squeeze, with raw material and freight costs rising and sales realizations declining. Also, there has been a shift to MAP from DAP in North America, with the majority of DAP marketed for export on a seasonal basis. The fertilizer market does not appear to be as attractive as in earlier studies, and any further consideration of this process route for the Crandon Project should include MAP as a potential product.

An update of revenues on the product volumes of the Coefield report is provided in Table 6-1.

Table 6-1
Annual Revenues - 1995 $

Product	Annual Production		Values per Unit	Yearly Revenues	Total Revenues			
	Amount	Units			Acid Only	Acid and Pellets	DAP Only	DAP and Pellets
Process Only Fines								
Sulfuric Acid	729,068	tons	$ 36	$26,246,448	$26,246,448	$34,157,298	$80,024,420	$87,935,270
Iron Pellets	263,695	tons	$ 30	7,910,850				
Diammonium Phosphate	571,603	tons	$ 140	80,024,420				
Nonferrous Metals								
Zinc	2,921	tons	$ 780	2,278,380				
Copper	529	tons	$1440	761,760				
Lead	485	tons	$ 360	174,600				
Precious Metals								
Silver	234,379	ozs	$ 3.96	928,141				
Gold	10,609	ozs	$319.78	3,392,546				
Subtotal Nonferrous & Precious				$ 7,535,427				
Process All Pyrite								
Sulfuric Acid	1,889,800	tons	$ 36	$ 68,032,800	$68,032,800	$87,818,820	$206,662,120	$226,448,140
Iron Pellets	659,534	tons	$ 30	19,786,020				
Diammonium Phosphate	1,476,158	tons	$ 140	206,662,120				
Nonferrous Metals								
Zinc	4,178	tons	$ 780	3,258,840				
Copper	2,083	tons	$1440	2,999,520				
Lead	1,168	tons	$ 360	420,480				
Precious Metals								
Silver	592,859	ozs	$ 3.96	2,347,722				
Gold	27,328	ozs	$319.78	8,738,948				
Subtotal Nonferrous & Precious				$ 17,765,510				

APPENDIX 6.0

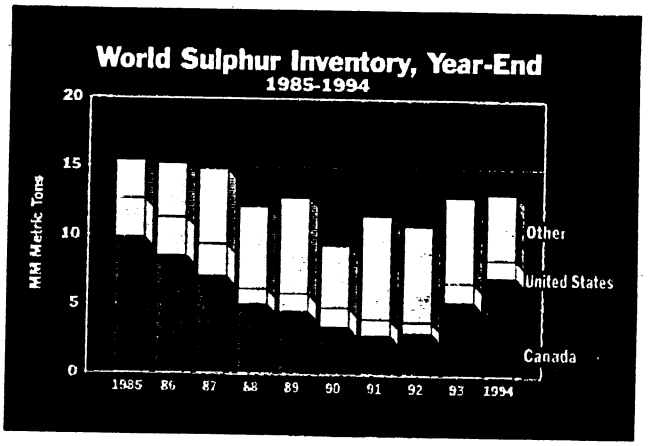


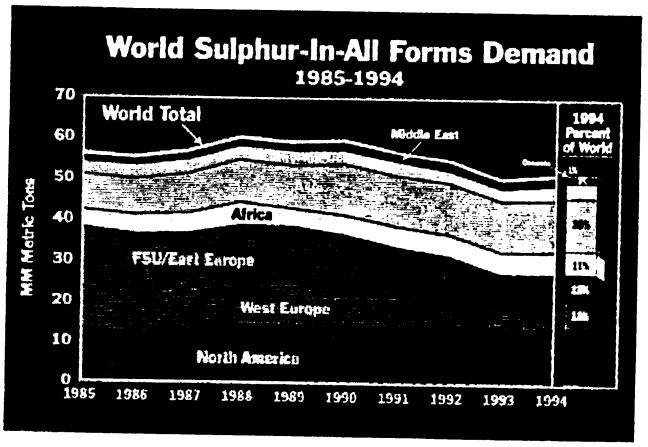
Owens Lake, Calif. No definite startup date has been announced.

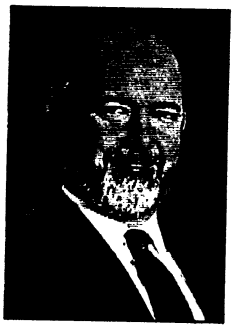
Another potential new supplier is the Wold Trona Co., Casper, Wyo. It has proposed opening a new trona mine south of Green River, that would produce 1M st/yr partially beneficiated, calcined-trona-ore. Wold is in search of investment partners.

Outlook. The year 1995 promises to be better for the U.S. soda-ash industry than 1994. With less competition from caustic soda, and a healthy U.S. economy, the domestic market is expected to continue to show a modest, but not spectacular improvement. Most growth is anticipated to still come from exports; export growth in 1995 is anticipated to match the growth of the past year. Potential increased sales to Asia and South America show the greatest promise.

Tempering the positive outlook for demand growth is uncertainty around exports to the European market. This because of the dumping investigation and currency exchange rates. In addition, the U.S. Bureau of Land Management is evaluating an increase in the royalty for Wyoming trona leases which would place added pressure on industry profitability.







SULPHUR

Suppliers Welcome Rebound in Demand

Robert B. Foster, president, Freeport Sulphur Co.

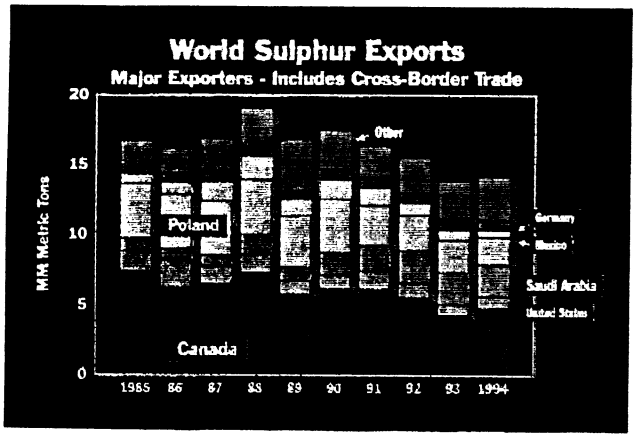
As 1994 ran its course, the 'Chicken Littles' (pessimists) of the industry had to retreat, because 'the sky quit falling,' at least temporarily. Continued withholding of substantial production tonnage by vatting, allowed the first increase in demand since 1988. It had a significant positive impact on sulphur prices in all markets. This demonstrates the increased market volatility that is at least partially unavoidable as voluntary production shrunk to an all-time low as a percentage of the market. It again demonstrated that the surplus or shortage of production to consumption has little to do with market directions, while inventory management, and more recently, trade-pattern changes, dominate market reactions.

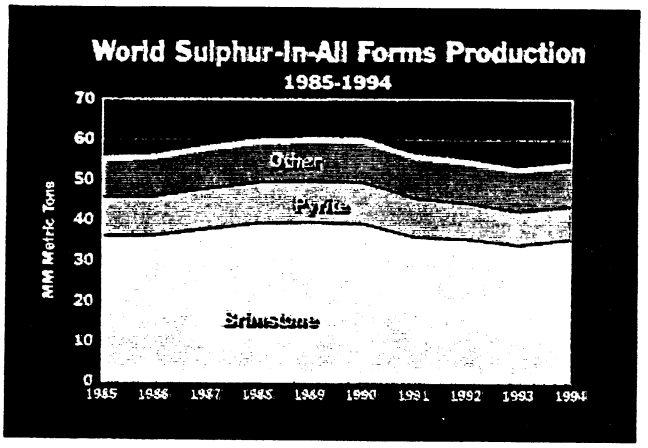
Virtually all changes in market factors were at least as favorable to suppliers as had been expected, and many were more so. Most notable was the increased demand resulting from sustained improvements in the phosphate fertilizer industry, the principal sulphur consumer. While merchant sulphuric acid (H_2SO_4) was expected to be tighter, nobody expected the degree to which it impacted the market. Refinery production was dampened by the refineries' reactions to the shrunken sweet-crude differential, and the Chinese market for elemental sulphur continued to develop. Continuing lack of improvement in the CIS was the biggest negative.

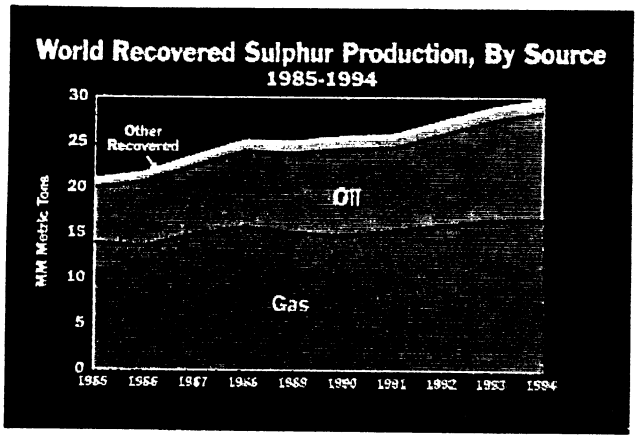
Demand. After reaching a peak of 60.4M mt in 1988, world sulphur-in-all-forms (SAF) demand declined to approximately 50.5M mt in 1993. However, record Chinese phosphate-imports in 1994 helped U.S. sulphur demand rebound by nearly 1.1M mt and helped push world SAF consumption to 51.5M mt. Sulphur demand increased in 1994 in Latin America and Western Europe as well, but was offset by a further decline in the CIS consumption. CIS sulphur demand has declined from a high of 17M mt in 1988, to an estimated 5.1M mt in 1994—or by 70%. It has been the leading problem for sulphur marketers over the past several years.

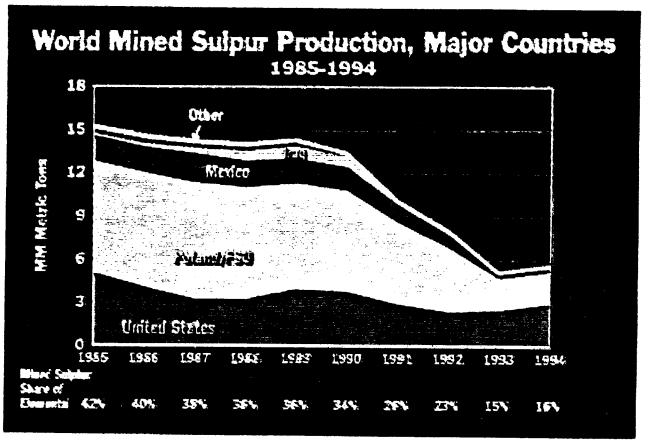
Worldwide, elemental sulphur demand declined to an estimated 32.4M mt in 1993—compared to the high mark of 1988 of 41.7M mt—but rebounded to 33.9M mt last year. North America was the largest SAF demand-region in 1994, and accounted for 35% of overall brimstone demand. SAF consumption in North America improved by nearly 1.1M mt in 1994 over 1993 levels. Elemental consumption accounted for over 75% of the gain. In spite of higher U.S. sulphur-production, reduced imports of elemental sulphur from Canada and Mexico required a U.S. stock draw-down of over 300K mt. A significant rebound in sulphur exports from western Canada aided in the North American sulphur-balance. However, Canadian stocks increased nearly 2M mt after rising nearly 2.4M mt the year before. They were the key factor in the market's overall performance. With a large demand for concentrated phosphate-fertilizer exports to Asia, the prospect for continued high U.S. sulphur consumption appears promising. If anything, U.S. phosphate producers' forward export commitments to mid-1995 could produce the best outlook for U.S. sulphur demand in many years.

SAF demand in Western Europe appears to have bottomed out in 1994, at approximately 6.6M mt. That level represents a decline of around 35% over the past 10 years. Elemental sulphur consumption in the region has been adversely affected by the continued closure of fertilizer plants, reduced fertilizer usage, and increased environmental constraints on non-fertilizer chemical processes. Beginning in 1993, the region reverted to being a net exporter of sulphur. With the forecast calling for static demand and increasing output from refineries, poten-









tial net sulphur exports from western Europe could rise to over 1M mt in the next few years compared to about 200K mt in 1994.

As shown in Chart 1, combined North American and West European sulphur-demand has accounted for between 42-47% of global elemental sulphur-demand since 1985. Of greater significance, however, is the share accounted for by the Commonwealth of Independent States (CIS) which has declined from 19% only 10 years ago to an estimated 11% in 1994. Unquestionably, the chaotic market conditions witnessed these past several years have been precipitated by the decline in the CIS' sulphur demand. It appears demand continued to drift lower in the CIS during 1994. It is doubtful if the magnitude of the inevitable rebound of demand in the CIS will be in the same range; there is no way of telling when it will occur.

Growth must occur if the correlation of phosphate fertilizer consumption to world population continues to hold. Immediate market targets for growth are in Africa—primarily the long-awaited expansion at Jorf Lasfar in Morocco, and the emerging markets in Asia and the Middle East. One forecast calls for an annual increase in sulphur demand of around 850-900K mt/yr over the next three years. Although pessimists would view this as optimistic, that rate of growth is still below the long-term trend recorded in the 20 years up to 1988. Needless to say, global sulphur consumption trends have been wild, to say the least, over the past decade. In the first few years, 1985-1988, world elemental sulphur demand increased an average of 1.5M mt/yr. In the five years that followed, demand decreased almost 1.9M mt/yr only to rise an estimated 1.6M mt in 1994. The erratic swings in brimstone demand and the lack of reaction by the involuntary producers has devastated the voluntary producers of sulphur.

Supply. There are five principal sources of sulphur in the world today, they include:

Estimated 1994 Production (M mt)

	Tons	Percent*
Elemental		
1. Recovered from natural gas	16.7	33
2. Recovered from oil refining/other	13.0	25
3. Mined sulphur	5.4	11
Sub-total	35.1	68
Non-elemental		
4. Smelters/other	10.0	19
5. Pyrite roasting	6.2	12
Sub-total	16.2	32
Total SAF Production	51.3	100

* Totals may not add due to rounding.

Recovered sulphur production continues to rise steadily, primarily due to oil processing, and has grown by nearly 9.0M mt, or over 40% since 1985. Gas sulphur production improved slightly in 1994 due primarily to the first full-year of operation at Shell Canada's Caroline sour-gas processing plant in Alberta and three new plants in Abu Dhabi. The much touted Russian sour-gas plants at Astrakhan continue to produce substantially below their design capacity. The real gains in recovered sulphur production came from refineries in 1994 producing close to 11.7M mt of sulphur or nearly double the level recorded in 1985.

While recovered sulphur production continued to capture a larger share of world elemental output, mined sulphur supply continued to lose

ground. In the past 15 years, mined production of sulphur has dropped from around 50% of total elemental-sulphur production to only 15% last year. A total of 10 mines have been shut down since 1992, two within the United States. Overall, there are only seven remaining sulphur mines of any size and at least three of those are "at risk" depending on selling prices.

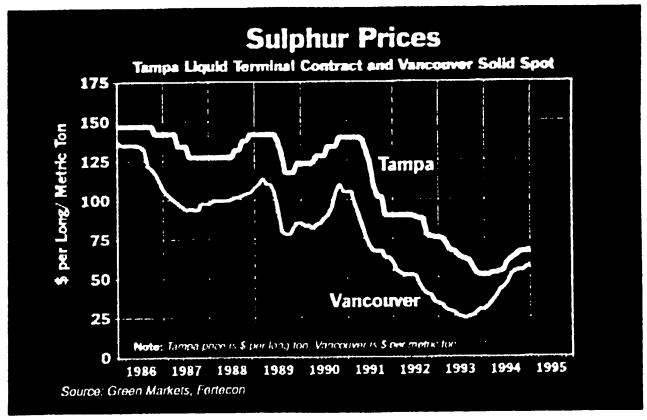
High base-metal prices contributed to strong H_2SO_4 output at metallurgical smelters, and provided nearly 19% of the total production of SAF in 1994. Fortunately for elemental suppliers, a larger percentage of smelter acid remained in that sector as increased demand for leaching of ores bid up merchant H_2SO_4 prices.

Outside of China, pyrite mining and roasting became even less important, as H_2SO_4 production from pyrite in Western Europe has declined 70% over the last 10 years. Of some note, Chinese pyrite output has reversed course slightly over the past two years and could decline further if the Chinese government's plans to close or privatize uneconomic operations takes place.

Gas-recovered sulphur. The processing of sour natural gas to remove entrained H_2S remains the largest current source of elemental sulphur. Estimated 1994 gas-sulphur production totaled 16.7M mt compared to 16.6M mt in 1993. Small decreases in output were recorded in the CIS and the United States, the two largest gas-sulphur producers behind Canada. Saudi Arabia's supply of vatted sulphur was exhausted in August of 1994.

Canada was the only sour-gas producing region recording an increase in year-end stocks in 1994 with an increase of 1.8M mt after adding nearly 2.4M mt in 1993. Preliminary estimates place year-end Canadian stocks at 7.4M mt, the highest level since 1987 and around 4.5M mt higher than on hand at the end of 1991. The outlook is for even higher offshore exports through Vancouver in 1995, coupled with a modest improvement in rail deliveries into the United States. However overall shipments will be insufficient to keep Western Canadian producers from pouring to block again in 1995. Overall sour-gas sulphur output in Western Canada is expected to rise by nearly 400K mt and raise stocks by 1-1.5M mt. Outside of Canada, the mid-year commissioning of three new sour-gas processing plants in the United Arab Emirates more than doubled gas-sulphur production in 1994 to just under 250K mt. In 1995, those new plants are expected to capture 500K mt of sulphur for the full 12 months. After nearly eight years on-stream, the production of sulphur from the Astrakhan gas processing plants near the Caspian Sea in Russia remain a fraction of their combined design rate of 4.3M mt/yr. The achievability of the design capacity is indeterminate. The latest information indicated the number one plant is operating at an intermittent rate of between 50-60% and repairs on the second unit are nearly complete. Deliveries from the plants had more market impact in 1994. Although most went to domestic factories, relatively small amounts exported through the Black Sea were used effectively by North African consumers to buffer pricing increases, despite unreliable loading and shipping arrangements, and uncertain product specifications. The uncertainty concerning the multi-stage Tenghiz project spearheaded by Chevron did not diminish during the year.

Refinery sulphur production. Increased environmental pressures worldwide over the past few years has led to the addition of desulphurization units at several refineries. Pessimists tend to assume these units will operate continuously at capacity, and project increasing surpluses as a result. History does not bear that assumption out. The latest contrary proof is several of those units stood idle or operated at low rates for some months in 1994. Key among underlying factors was the positioning of sweet crudes by oil-producing countries keen on capturing market share and improving cash flow. This led to a heavy discounting of sweeter crudes and a shrunken crude differential. European refinery-



sulphur production dropped precipitously during early 1994, until heavy- and light-crude pricing returned to more historic differentials. However this resulted in a 5% drop in sulphur output. U.S. refinery-sulphur output did not increase as much as some expected, although dependence on high-sulphur Mayan crudes into Gulf Coast refineries dampened the change in crude slates seen elsewhere. U.S. refinery-sulphur production increased to 5.1M mt as compared to 4.8M mt in 1993. Although the crude-slate into U.S. refineries in 1994 averaged 1.15% compared to 1.14% in 1993, a marginal increase in the recovery rate from 59% to 64% associated with new desulphurization units explained the rise in U.S. refinery production. Refinery-sulphur output in the United States, Western Europe and Japan accounted for nearly three-fourths of global oil-recovered sulphur production. The re-commissioning of the war-damaged refineries in Kuwait boosted sulphur output by an estimated 300K mt over the prior year.

World crude slates will continue to be dominated by the decisions of oil-producing countries that can choose to switch production between sweet and sour crudes. The large supply of ultra-light crude recently discovered in Saudi Arabia will no doubt effect the world crude slate over the coming years. Growth rates forecast for oil-recovered sulphur worldwide—projected to average between 300-500K mt/yr—could prove to be optimistic if 1994's experience turns out to be a longer-term trend. North American output from this source is expected to plateau in the near future with most of the incremental supply coming from Western Europe and Eastern Asia (Japan and South Korea).

Sulphur-in-other-forms production (SOF). Sulphur in the form of H_2SO_4 recovered during non-ferrous metal smelting accounts for approximately 10M mt sulphur-equivalent. While there are a number of projects on the drawing boards to boost throughput at base-metal smelters around the globe, it is not expected all of these projects will come to fruition. SOF output improved marginally to 10.5M mt, about 1% higher than registered in 1993. Base-metal prices took off in second-half 1994, boosting producers' profit margins and providing the extra incentive to see new projects through. However, a squeeze on copper smelters' margins, brought on by a concentrate shortage and a reduction in treatment charges to miners, has set back plans for several new projects. Additionally, higher environmental standards forced the closure of several smelters in 1994, leaving some doubt about the future growth in SOF output. H_2SO_4 prices in 1994 reflected the supply/demand pressure and nearly tripled during the year. Besides a somewhat occluded supply outlook for smelter acid, new leaching projects are struggling to source long-term low-cost sources of H_2SO_4 required in the process, which ultimately competes with conventional smelters. The bulk of the new leaching projects are in the United States and Chile.

Sulphur from pyrite. The roasting of pyrite and pyrrhotite is similar to metal smelting operations in that H_2SO_4 is obtained.



However, unlike smelters, the residue or pyrite cinders are the byproduct and the primary product is the H_2SO_4. Pyrite is abundant in several countries, especially in China, which is the largest user and represents the only growing pyrite producer in the world.

As indicated earlier, West European pyrite-production has fallen to an estimated 2.4M mt (H_2SO_4) last year, compared to 7.6M mt as recently as 1988. Major reductions in pyrite use in Western Europe took place in Italy and Spain in 1994, while several East European pyrite producers were also forced to abandon uneconomic operations. It is unlikely any of these operations will come back on-line.

China has vast reserves of pyrite, comprising close to 60% of world sulphur-from-pyrite production. Outside of a few niche pyrite suppliers, China is the only country with an apparent secure source of pyrite. However, even though H_2SO_4 produced from pyrite in China posted double digit growth in 1990-1991, output has leveled off in the past two years at around 11.3M mt (H_2SO_4). China continues to struggle with an over-crowded rail system, which currently permits the movement of H_2SO_4 at little or no cost. If and when the central government begins to rationalize true transportation costs, pyrite production could be threatened. Meanwhile, China's output of pyrite is consumed internally. Recently China has become a market for sulphur exported from Japan, and this could well have an impact on pyrite production levels as well.

Mined sulphur production. Sulphur is currently being mined by either the Frasch (hot-water injection) or open-pit method in four major countries, including the United States, Poland, the CIS, and Iraq. Economic and trade sanctions imposed by the United Nations continue to keep Iraqi sulphur out of the international marketplace. Consequently, Iraqi sulphur output has been curtailed in line with domestic needs and it is estimated production at Mishraq totaled under 300K mt last year.

The big news is the impact of mine closures in the falling market of the last few years. The sharp fall in global sulphur prices beginning in 1991 forced the closure of 12 mines for economic reasons. Since 1990, worldwide mined sulphur-output declined from 13.4M mt to under 5.5M mt in 1994, or over 50%. Within the United States, four mines have been shut down since 1990 and only two are in operation: Main Pass and Culberson. Together, these two mines produced about 2.9M mt in 1994, compared to 2.4M mt the year before, including the Caminada mine. Main Pass surpassed its designed production rate of 2.0M lt/yr in early 1994 and averaged over 6,100 lt/d for the full year.

Polish production also rebounded slightly in 1994, to about 2.0M mt—only 40% of the level of the late 1980's. Under current mining conditions and market prices, the two current operations could continue for many more years. CIS mines are estimated to have produced less than 600K mt last year, or around 33% of 1992's level. With timing dependent on the success of recovered-sulphur projects, and the direction in CIS sulphur demand, it is likely uneconomic sulphur mining in the CIS will eventually cease altogether.

Trade. Maritime trade in sulphur rebounded approximately 1M mt in 1994; trade patterns continued to change. On the downside, it appears exports from Saudi Arabia, Poland, Germany, Mexico and Iran declined last year, while international shipments from Canada, the United States, Kuwait, France, and Japan rose. As previously noted, rail deliveries of sulphur from Canada into the United States dropped approximately 400K mt. For most of the exporters who suffered a decline in sales, much of the cause was due to reduced production and/or lower stocks from which to draw. International sulphur demand was robust in 1994 compared to a year earlier, and permitted a reshaping of market supplies that produced higher pricing. Although exports from the CIS appeared the marketplace in questionable quantities, primarily in the Mediterranean, the onset of winter has forced an end to those sources.

Beginning 1995, trade appears to have returned to a 'more normal' level, albeit sharply below the levels set in the late 1980s. Although trade patterns will continue to change, if the current volume of sales opportunities persists and international demand holds, there is no real reason to expect prices to be impacted as much in 1995 as was recorded in late 1993 and into early 1994.

Sulphur prices. International sulphur prices fell to the lowest mark in nearly 20 years during 1993. In real terms they were probably the lowest ever. However, as market volumes improved in the first quarter of 1994, so too did sulphur prices. Measured by the Vancouver (solid) spot price, prices dropped under $25/mt during the second-half 1993 and improved to the $50-60/mt range by year-end 1994. Liquid prices in Tampa, Fla., the world's largest single market, improved from a low of $45-50/lt to near $70/lt by year's end. Middle East prices were the strongest, rising from the low $30s to nearly $70/mt over the year.

Although the sulphur market has improved, new uncertainties emerge. The resignations of Amoco, Husky and Shell Canada from Prism as of Jan. 1, 1995, will have an impact that is not predictable. Many remember the pricing actions by key competitors during 1991, where competition for spot sales as small as one cargo into Brazil dropped the world price $15/mt in one week. Unquestionably, it was the unified marketing brought on by an enlarged Prism that bore the fruits of better times during 1994. Pessimists would say the fracturing of Canadian marketing must wreak havoc in the market, but lessons can be learned from the past. Continued vatting at the levels required to balance the market will sooner or later change the incremental economic comparisons

Conclusion. Will 1994 prove to be a turning point in the longer term, or just another example of the increased volatility of this new age of sulphur supply dominated by involuntary producers? Whatever the outcome, one fact is certain...the sulphur market has undergone a structural change. It requires replacement of the industry's internal shock absorber, the voluntary sulphur producer. At today's reduced production levels and with only 15% market share, sulphur miners are incapable of serving their traditional role and must remain focused on survival, even with 1994's price improvements. The role of primary swing-supplier has been ceded to the Canadian sour-gas industry, as demonstrated in 1994.

There are clouds on the horizon, as always:

- Will the disproportional impact of unreliable CIS exports seen in 1994 grow?
- What impact will another year without signs of recovery in CIS demand have?

In more rational markets elsewhere, more participants must adjust to living with a long-term excess of production over consumption. This can be done; until 1988 the industry lived with a long-term shortage of production to consumption, and prices fluctuated with other market factors. The industry must continue to improve its ability to adjust to changes in demand, or volatility will continue. More involuntary producers must become voluntary suppliers, and develop the wherewithal to make that transition. Otherwise, pessimists may be right, and 'the sky may start falling' again.



PHOSPHATE

Recovery Blossoms

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The recovery in the phosphate industry which began following the IMC-Agrico merger in mid-1993 continued throughout 1994. By year-end, prices of end-use fertilizers were back at a 20-year average and still heading up. However, the phosphate-rock industry has yet to share in this improvement. Several problems remain, including overcapacity, but exporters have been encouraged enough by the rally in the downstream industry to request a rise in rock prices for 1995.

Phosphate rock is mined in more than 20 countries worldwide and around 90% is used in fertilizers. Other uses include animal-feed supplements and various 'industrial' uses, the most important of which is the production of sodium tripolyphosphate (STPP) as a detergent additive.

There has been a worldwide depression in the use of phosphate fertilizer since 1988 and this has been prolonged by the political and economic changes in Eastern Europe and the former Soviet Union (FSU). In 1994, global phosphate-fertilizer demand is thought to have shown the first increase in six years. Consumption in the FSU is thought to have fallen further, but for the first time, this has been outweighed by increases in other markets, particularly China. This went a long way to bringing the fertilizer market back into balance.

Demand. Following the improving trend in fertilizer consumption, demand for phosphate rock in 1994 is thought to have shown the first increase since 1988. Preliminary estimates put 1994 rock consumption worldwide at just under 130M mt, a rise of 7% over 1993.

The United States has remained the largest consumer with consumption rising an estimated 10.5% to 42.5M mt. The proportion of this total processed domestically continued to increase and reached an estimated 96% in 1994.

As can be seen in Table 2, the increase in U.S. consumption was accounted for almost entirely by higher levels of activity in the production of phosphoric acid, 90% of which is used for fertilizer production. Around two-thirds of phosphate fertilizers produced in the United States are exported and whilst domestic fertilizer sales remained virtually static, exports showed a considerable improvement. This was led by China, which accounts for around 40% of all U.S. fertilizer exports.

Since phosphate-rock exports have become progressively less important to U.S. companies, fertilizer prices have much more impact on balance sheets. The stagnation of rock prices will not worry most U.S. companies as long as fertilizer prices remain firm.

China is the second largest rock consumer in the world, supplying practically all its requirements from a large number of small producers and a few larger mines. Consumption has stagnated in the last few years but new mining projects linked to plans for new fertilizer plants should ensure that rock production increases steadily over the next five years from the current level of 18.3M mt.

Consumption of phosphate rock in the FSU has plummeted in recent years following the political and economic reforms. Rock consumption by domestic plants is thought to have fallen further in 1994 to 12.6M mt, almost one quarter of the pre-1990 level.

Table 1

U.S. Phosphate Rock Statistics (K mt)

	1994e	1993	1992	1991
Marketable Production	40,700	35,138	46,965	48,096
Average value ($/mt)	21.35	21.38	22.53	23.06
Sold or used by Producers	44,120	40,589	45,113	44,707
Exports	3,350	3,575	3,723	5,082
Export value ($/mt)	28.00	28.51	32.29	32.00
Imports	1,800	1,461	1,530	552
Import value ($/mt)	34.00	34.00	36.71	50.73
Consumption (1)	42,570	38,475	42,920	40,177
Producer stocks Dec.	6,000	9,137	12,612	10,168

Sources: USBM and Fertecon Research Centre
(e) Estimated
(1) Sold or Used plus imports minus exports.

Table 2

Phosphate Rock Distribution Pattern (K mt)

	1994e	1993	1992	1991
Wet Process Phosphoric Acid	37,460	33,664	37,790	35,895
Single superphosphate	20	50	23	47
Triple superphosphate	490	450	501	395
Direct Application	0	0	0	0
Elemental Phosphorus	2,800	2,850	3,076	3,288
Ferrophosphorus	0	0	0	0
Total	40,770	37,014	41,390	39,625
Exports	3,350	3,575	3,723	5,082
Grand Total	44,120	40,589	45,113	44,707

Sources: USBM and Fertecon Research Centre Estimates
(e) Estimated

There has been a similar decline in other eastern European countries in the last five years but in some there are signs of a recovery, particularly in Poland and Romania which are in a position to export fertilizers.

Despite the fact that the U.S. had a record corn harvest in 1994 and that India has had good monsoons for six years running, the global level of cereal stocks declined for the second successive year in 1994. Global grain stocks as a percentage of consumption now stand at a level not seen since the early 1970's when there was perceived to be a worldwide "food crisis". The need to increase cereal production from a fixed land area is one factor which will continue to push fertilizer demand.

Production. Higher phosphate-fertilizer consumption in 1994 resulted in the first increase in rock production since 1988. The 1994 global phosphate rock production level is estimated at 124M mt, substantially below consumption mainly due to destocking in the United States. The closure of the US Agri mine in Florida in mid-1994, coupled with a rundown in stock levels at Mobil's Big Four mine, which is due to close early in 1995, were mainly responsible for a 3.4M mt reduction in industry inventories. This follows a similar reduction in 1993 and has



Table 3

U.S. Phosphoric Acid Capacity (K mt P2O5/yr)

	1985	1994
Florida	6,854	7,865
Louisiana	1,788	1,533
Mississippi	220	315
North Carolina	1,146	1,146
Texas	240	220
Idaho	466	599
Wyoming	0	200
Illinois	113	0
Utah	90	0
California	113	0
Iowa	180	0
Total	11,210	11,878

Source: Fertecon Research Centre

Table 4

U.S. Phosphate Rock Imports and Exports (K mt)

Year	Imports	Exports
1980	486	14,276
1981	13	10,395
1982	31	9,842
1983	9	12,010
1984	9	11,528
1985	34	10,878
1986	528	9,010
1987	464	9,008
1988	728	9,018
1989	683	8,979
1990	408	7,152
1991	663	5,622
1992	1,644	3,919
1993	1,461	3,557
1994e	1,800	3,350

Sources: USBM and Fertecon Research Centre (e) Estimated

Table 5

U.S. Phosphate Commodity Exports (K mt)

		1994e	1993	1992	1991
		Average P_2O_5 Content			
Phosphate Rock	32	3,350	3,575	3,919	5,622
Single superphosphate	18	0	0	0	35
Triple superphosphate	46	790	762	1,114	912
Diammonium Phosphate	46	8,300	7,428	8,277	8,789
Monoammonium Phosphate	51	1,400	1,097	885	758
Phosphoric Acid	100	460	324	491	624
Elemental Phosphorus	229	18	17	18	17

Sources: USDC and Fertecon Research Centre
(e) Estimated

Table 6

World Phosphate Rock Capacity (M mt/yr)

	1993	1990
North America	53.15	58.50
Latin America	6.67	6.50
Europe	14.53	35.46
Africa	49.55	47.90
Asia	38.10	34.26
Oceania	2.25	2.25
Total	164.25	184.87

Source: Fertecon Research Centre

left U.S. inventories at an estimated 6M mt, the lowest level for more than a decade.

Phosphate-rock production worldwide peaked in 1988 at 156M mt. Global output fell to 116M mt in 1993 and is not expected to regain the 1988 peak until the end of the century.

A number of mines in the United States remained closed in 1994. Occidental Chemical continued to operate only from its Swift Creek mine in northern Florida; the company's other mine at Suwannee River in the same area remained closed throughout the year. The Wingate Creek mine in central Florida passed into the control of Mulberry Phosphates but also remained closed.

Mobil's Nichols mine remained closed in 1994, although there has been speculation as to whether Mobil will reactivate it for a short while to bridge the gap between the closure of Big Four, which is due to be depleted early in 1994, and the start-up of its new mine at South Fort Meade.

IMC-Agrico resumed maximum output at the Four Corners mine early in 1994 and reactivated the Kingsford mine shortly after but its Lonesome and Payne Creek mines remained closed.

In 1995, two new mines are expected to start production in Florida, the first since Four Corners was commissioned in the mid-1980s. Mobil is due to commence operations at South Fort Meade during the first quarter, whilst later in the year, CF Industries is expected to bring its new South Pasture mine on-stream. The CF mine will replace tonnage currently purchased from IMC under long-term contract.

Elsewhere in the world, the major change in rock production again came in the FSU. The reduced requirement for rock from local production units has resulted in a significant decline in demand. The largest mining area at Apatity on the Kola Peninsula has seen production decline from 20M mt/yr in the late 1980s to around 9M mt in 1993. A further decline is expected in 1994, perhaps to 7.5M mt.

In our analysis we have always equated capacity levels in the FSU with production. We have continued to do this for the sake of consistency, so the capacity of Russian mines has been reduced as production has fallen. Although it is certain that true capacity is now well above production levels in the Kola mines, it is also unlikely that the mines will be able to produce anywhere near 20M mt/yr in future due to deteriorating orebody characteristics and the fact that mining is very costly if free-market conditions are applied.

Global phosphate-rock production levels in 1993 were equivalent to under 70% of installed capacity. Phosphate-rock capacity, excluding what were the centrally planned economies (Soviet Union, China, and Vietnam), has stagnated in the current decade, falling overall by 2M mt/yr in the last four years to 132M mt. Capacity is expected to continue to stagnate, with new additions



Table 7

World Phosphate Rock Reserves (M mt)

	Reserves (1)	Share (%)	Reserve Base (2)	Share (%)
Morocco	5,900	52	21,440	63
United States	1,230	11	4,440	13
South Africa	2,530	22	2,530	7
Brazil	330	3	370	1
Algeria	240	2	240	1
China	210	2	210	1
Syria	190	2	190	1
Jordan	90	1	480	1
Tunisia	0	0	270	1
other	535	5	3,875	11
Total	11,255		34,045	

(1) Phosphate rock reserves at a production cost of less than $40 /mt fob mine. Costs include operating costs, capital servicing, taxes, royalties, and 15% ROI.
(2) Reserve base includes product with production costs (as above) of less than $100 /mt.

Source: USBM Jan 1992

Table 8

Phosphate Rock Production (K mt)

	1994e	1993	1992	1991
Algeria	770	730	1,136	1,090
Brazil	3,700	3,419	2,825	3,210
China	18,585	19,000	18,302	18,027
Egypt	700	537	948	1,367
Finland	650	628	555	473
India	650	967	488	610
Iraq	550	350	250	76
Israel	3,930	3,680	3,595	3,370
Jordan	3,900	4,129	4,296	4,433
Mexico	450	237	515	596
Morocco	19,700	18,193	19,185	17,814
Nauru	680	642	747	530
North Korea	500	500	500	500
Senegal	1,730	1,710	2,301	1,741
South Africa	2,540	2,496	3,080	3,180
Syria	1,200	931	1,266	1,415
Togo	2,050	1,794	2,083	2,965
Tunisia	5,600	5,500	6,400	6,401
United States	40,700	35,138	46,171	48,096
CIS	13,731	14,441	19,971	28,931
others	1,310	1,048	1,044	878
Total	123,626	116,070	135,658	145,703

Source: IFA and Fertecon Research Centre

being offset by depleting reserves. As rock production is expected to rise over the next few years, capacity use should improve and we expect it to reach 87% by the year 2000, a level more indicative of a balanced market.

Trade. Phosphate-rock trade showed the first increase in six years in 1994 rising an estimated 9% to 29.6M mt, of which the United States supplied 3.35M mt.

Two factors reduced international phosphate-rock trade by more than 42% in the 1988 to 1993 period. The primary factor was the fall in global fertilizer use. The second was wholesale closure of phosphoric-acid capacity in Western Europe which had been operated on imported rock. Increased environmental restrictions coupled with the better availability of cheap imported fertilizers from North Africa and elsewhere caused the closure of more than 1M mt P_2O_5/yr of acid capacity in the region in 1992/93 alone.

The turnaround in phosphate-fertilizer demand has resulted in slightly higher rock trade. This has come principally from marginal phosphate-chemicals exporters whose high costs are due to the fact that they use imported rock. Higher phosphate-fertilizer prices have allowed producers such as Fertinal in Mexico, Philphos in Philippines, and Police in Poland, to increase their exports and hence their rock imports have risen also. In addition, higher fertilizer prices have meant that producers in India and elsewhere, who produce fertilizers from imported rock in competition with imported fertilizers, have also been able to produce at a higher rate.

The trend towards processing at sites of rock production rather than at sites of fertilizer consumption is expected to increase. Further plant closures are expected over the next few years in Europe, and possibly elsewhere, which will reduce the requirement for imported rock. These two factors are likely to result in a stagnation in rock trade over the next few years.

Resources. Table 7 lists the major phosphate-rock deposits currently known and the level of reserves and resources estimated by the U.S. Bureau of Mines in its 1992 survey. Morocco has by far the largest resource base of any producer which explains the long-term planning adopted by the state-owned producer, Office Cherifien des Phosphates.

The United States has the second largest resource and a large reserve base, but compared to its large production level (Table 8) it has a limited life expectancy. Some U.S. producers of diammonium phosphate (DAP) have reserves equivalent to less than five years of full production. These depletions are likely to place an increasing strain on U.S. supplies and will undoubtedly result in higher costs of production over the next decade. By the year 2000, five mines are expected to close in Florida and a further five in the five years to 2005. Some of the lost capacity will be replaced by new mines but as depletions accelerate the cost of new mines becomes increasingly onerous.

Outlook. In 1994, phosphate producers' balance sheets in the United States at last showed some signs of recovery and the industry once more began to generate a relatively healthy cash flow, thanks largely to the downstream sector of the business. It must be remembered however, that we are still only one year away from the worst recession the industry has seen in two decades and, not surprisingly, those in financial circles have remained wary of the forecast of better times ahead. This has been reflected in U.S. companies' stock prices which in 1994 by no means always moved in the same direction as fertilizer prices.

The wariness extends to the international financial community as well in that it is still very difficult to secure funding for new fertilizer projects. Producers would argue that with phosphate price cycles averaging around seven years and with it taking three years or so to build a new phosphate complex, the best time to invest is at the bottom of a cycle. However, there is still considerable debt overhang in the industry as a result of the last recession and it may take several years of higher prices before the industry feels good enough to embark on a full-scale reinvestment program.