Klohn Crippen Berger

January 9, 2009

Pacific Booker Minerals Inc.
#1702 - 1166 Alberni Street
Vancouver, British Columbia
V6E 3Z3

Erik Tornquist
Executive Director

Dear Mr. Tornquist:

Morrison Copper/Gold Project - Geotechnical Feasibility Study – Rev. 1

We are pleased to submit Revision 1 of our report on the Morrison Copper/Gold Project - Geotechnical Feasibility Study. The study presents the feasibility design of the tailings storage facility, mine waste rock dump, low grade ore stockpile, and the associated facilities. The design covers the geotechnical, water management and civil aspects of the works. Revision 1 addresses minor errors in typography and cross-references and corrects inconsistencies in Section 9 and Section 12 of the report.

We appreciate the opportunity to work with you on this very interesting project and wish you success in moving forward towards an operating mine.

Yours truly,

KLOHN CRIPPEN BERGER LTD.

Terence Jibiki, P.Eng.
Project Manager

090109R-Geotechnical Feasibility Study Rev1.doc
File: M09382A01.500

EXECUTIVE SUMMARY

This report presents the geotechnical feasibility study for Pacific Booker Minerals Inc. (PBM) proposed Morrison Copper/Gold Project, located 65 km northeast of Smithers in north-central British Columbia. The Morrison mine will be a 30,000 tpd open pit operation with ore processed in a conventional milling plant, with the copper/gold concentrate transported to the Port of Stewart for shipment to offshore smelters. The mine will produce approximately 224 Mt of tailings and 170 Mt of waste rock.

The tailings storage facility (TSF) is located approximately 3 km northeast of the open pit and approximately 190 m higher in elevation. The TSF will initially be formed with a 50 m high Starter Dam, which will be expanded, with ongoing mining, to an ultimate height of approximately 95 m. The ultimate TSF will also include a 45 m high North Dam and a 35 m high West Dam, which will be constructed during operations. The TSF will be operated as a zero discharge system with seepage recycled and process water returned to the mill for reuse.

The majority of the mine waste rock is potentially acid generating and will be stored adjacent to the open pit, with all drainage recycled to the plant site. A temporary stockpile will store low grade ore, which will be milled later in the mine life. On closure the waste rock dumps will be covered with a low permeability cover and reclaimed.

Site Conditions

The site is in hilly, forested terrain, which receives approximately 550 mm of precipitation annually (40% as snowfall). The area is in a low to moderate seismic zone and the maximum credible earthquake (MCE) is magnitude M_W=6.2 and a peak ground acceleration PGA=0.13 g. The foundation soils for the tailing dams and the waste rock dump consists of medium dense glacial till overlying bedrock. The glacial till has a low permeability and has the potential to generate excess pore pressures during loading. Accordingly, piezometers will be installed in the tailings dams and waste rock dump foundations to confirm the predicted conditions. The low permeability tailings also provides for additional containment of potential seepage from the tailings facility.

Tailings and Waste Rock Characterization

The total tailings is a silty sand with approximately 55% less than the 75 micron particle size and typically contains < 1% sulphides. The dams will be raised with cycloned sand, which is classified as non-potentially acid generating (NAG). The cyclone overflow tailings and the remainder of the total tailings will be spigotted into the impoundment and are classified as NAG to low PAG (low potential for acid generating). The majority (i.e. 90%) of waste rock is classified as PAG and the remainder is NAG. The lag time for acid generation for the waste rock could be a long time and a more detailed assessment of the acid rock drainage characterization is included in the environmental impact assessment for the project.

Tailings Storage Facility

The TSF covers an area of approximately 5 km^2, with an uphill drainage area of approximately 5 km^2. Current drainage from the TSF flows south into a small creek and into Morrison Lake and a small amount of drainage flows northwards into Nakinilerak Lake. Given the potential environmental damage and substantial clean-up costs, the tailings storage facility could be categorized as a "Very High" classification facility (according to the Canadian Dam Association (2007)). However, the selected criteria for flood and seismic design have been upgraded to meet the more conservative "Extreme" classification to reflect the potential for future land use in the area. Accordingly, the tailings dams are designed for the maximum credible earthquake (MCE) and the maximum probable flood (PMF). The Main Starter Dam will be constructed as a homogeneous fill dam using glacial till borrow material from the interior of the TSF and from stripping of the open pit. A sand and gravel blanket drain will be placed under the downstream toe of the dam to control seepage. The dams will be raised by the centerline method with a central glacial till core and cycloned sand on the downstream and upstream sides. The downstream slopes of the dams will be 3H:1V. Seepage collection systems, downstream of each dam, will include a dam and water return system to recycle seepage and cyclone sand drainage water back to the impoundment. A pump barge will return water, via a buried pipeline, to the plantsite. The tailings delivery system includes 2 pump stations and an approximately 760 mm diameter HDPE and HDPE-lined-steel pipeline to the crest of the dams. Cyclowash cyclones, located on the dam crest, will be used to cyclone sand for construction of the dams between March and October of each year.

Mine Area, Low Grade Ore Stockpile and Waste Rock Dumps

The waste rock dump and low grade ore stockpile are located adjacent to the open pit and plantsite area. Development of the open pit will require drainage of Booker Lake and Ore Pond, and removal of soft sediments. Overburden from stripping of the open pit will be stockpiled near the open pit and potentially used for construction of the tailings Starter Dam. Organic bearing material will be stockpiled for use in reclamation. The waste rock dump will cover an area of approximately 220 ha and reach a maximum height of approximately 150 m. Foundation preparation for the waste rock dump and low grade ore stockpile will include removal of marshy soils and any other weak or soft materials. Piezometers will be installed in the glacial till to monitor construction pore pressures.

The majority of the waste rock is potentially acid generating and could begin to leach metals at some time during operations or in the future. NAG waste rock will be preferentially placed in drainage channels and towards the south side of the waste rock dump. Waste rock will be placed to an overall final slope of 2.75H:1V. A soil cover will be placed over the dump surfaces to minimize infiltration of water. Low grade ore will be temporarily stockpiled and milled later in the mine life.

Clean surface water will be diverted around the waste dump and disturbed mine areas. Contact water, from the plant site, waste dump and open pit areas will be collected and recycled to the mill. Runoff water from overburden dumps organic bearing material stockpiles will be directed to sediment ponds prior to release. A fresh water supply will be provided from Morrison Lake.

Environment and Closure

The tailings have low to no potential for acid generation, nonetheless, they will be stored in a saturated impoundment, which will further preclude the risk of acid generation. The tailings process water contains low metal concentrations and meets guidelines for drinking water and wildlife use. Seepage from the tailings impoundment is estimated to be in the order of 5.6 L/s to 7.4 L/s and the majority of this seepage will be collected with the seepage recovery systems. On closure, the tailings impoundment will be closed as a lake and the dam slopes will be revegetated.

After closure the open pit will fill with water, which will assist in reducing the area of pit wall rock that would be exposed to oxidation and, therefore, potentially acid generating. The waste rock dumps will be covered with a low permeability soil cover. A provision for a water treatment plant is included to treat contaminated seepage water from the waste rock dumps and from the remaining exposed pit wall rock above the pit lake level. The plant site would be decommissioned and industrial waste would be stored in the base of the open pit and any hazardous waste would be either disposed in an engineered facility or shipped off site. The disturbed areas would be covered with soil and reclaimed.

Management, operation and closure of the TSF will follow the guidelines developed by the Mining Association of Canada. Accordingly, an Operations, Maintenance and Surveillance (OMS) manual and an Emergency Preparedness Plan (EPP) will be prepared prior to operations to guide responsible management of the facility

TABLE OF CONTENTS

PAGE

TABLE OF CONTENTS
(continued)

TABLE OF CONTENTS
(continued)

TABLE OF CONTENTS
(continued)

TABLE OF CONTENTS
(continued)

TABLE OF CONTENTS
(continued)

TABLES

TABLE OF CONTENTS
(continued)

TABLE OF CONTENTS
(continued)

FIGURES

TABLE OF CONTENTS
(continued)

APPENDICES

DRAWINGS

General

TABLE OF CONTENTS
(continued)

Tailings Storage Facility

Waste Rock Dump and Stockpiles

TSF Access Roads

1. INTRODUCTION

1.1 General

This report presents the geotechnical feasibility study for Pacific Booker Minerals Inc. (PBM) proposed Morrison Copper/Gold Project, located 65 km northeast of Smithers in north-central British Columbia (Drawing D-1001). The project is situated on the east side of Morrison Lake. Access to the site is by road, with barge access across Babine Lake, which is located just south of the mine. Previous mining in the general area included the Bell and Granisle mines, which are currently closed. The Morrison mine will be a 30,000 tpd open pit operation with ore processed in a conventional milling process and the copper/gold concentrate transported to the Port of Stewart, B.C., for shipment to offshore smelters. The general arrangement of the mine facilities are shown on Drawing D-1002.

The mine will produce approximately 224 Mt of tailings and 170 Mt of waste rock. The tailings storage facility (TSF) is located approximately 3 km northeast of the open pit, approximately 190 m higher in elevation than the plant site. The TSF will initially be formed with a 50 m high Starter Dam, which will be expanded with ongoing mining to an ultimate height of approximately 95 m. The ultimate TSF will also include a 45 m high North Dam and a 35 m high West Dam, which will be constructed during operations. The dams will be constructed with a homogeneous glacial till starter dam and raised by the centreline method with a central low permeability core and a downstream shell using compacted cycloned sand. The TSF will be operated as a zero discharge system with seepage recycled and process water returned to the mill for reuse.

The majority of the mine waste rock is potentially acid generating and will be stored adjacent to the open pit, with all drainage recycled to the plant for use in processing. A temporary stockpile will store low grade ore, which will be milled as required during the mine life. On closure the waste rock dumps will be covered with a low permeability soil cover and reclaimed. A provision for a water treatment plant is included due to the potential need for treatment of acidic seepage from the waste dumps and/or open pit at some point in the future.

The scope of work for the geotechnical feasibility study included the following components:

- Assessment of alternatives for storage of waste rock and tailings;
- Site investigations and laboratory testing;
- Water management, including: water balance; diversions; seepage recovery; sediment control; drainage of Booker Lake and Ore Pond; bridge crossings; runoff collection; and fresh water supply system from Morrison Lake;
- Design of tailing dams and the deposition plan for the TSF;
- Design of tailings pumping and water reclaim systems;
- Design of access roads to the TSF;
- Design of waste rock dumps and low grade ore stockpiles;
- Design overburden and organic sediment storage piles; and
- Develop a bill of quantities and capital cost estimate for the above items.

1.2 Previous Studies

The Morrison Copper/Gold Project has been under review and assessment since 2002 and the main studies, prior to the work presented in this report, are summarized as follows:

Knight Piesold – Initial Waste Management Site Alternatives Study (2002):

- This report is the initial study of potential waste management facilities within a 6 km radius of the deposit. A basic overview of the regional surficial sediment types was presented, and a comparative table summarizing preliminary environmental and engineering factors for the potential site alternatives were developed and summarized. The conceptual layouts of four sites were mapped based on 20 m contours.

Knight Piesold – Report on Initial Site Visit and Updated Concepts for Waste Management (2003):

- An onsite assessment (July 7 to 9, 2003) of the Morrison Property area was conducted to identify other possible alternatives for dam placement prior to the commencement of pre-feasibility studies. This report presents the findings from the visit, which included a possible new waste management site (Site E), and new waste management concepts at Sites A and E.

Knight Piesold – Tailings and Waste Rock Management Input to Scoping Study (2004):

- In this report, conceptual level design layouts and costs estimates have been completed for four potential tailings and mine waste rock management options. The four options have been developed at two sites termed sites A and B, as follows:
    - all wastes at Site A;
    - all wastes at Site B using a central till core;
    - all wastes at Site B using a HPDE lined dam; and
    - combined storage at Sites A and B.
- Each option considered the co-disposal of tailings with waste rock. The non-acid generating (NAG) waste rock would be used to construct the embankment dams and potentially acid generating (PAG) waste rock would be contained within the tailings deposit either by placing it in the upstream shell of the dam or within the tailings basin for inundation by the

tailings. Cycloned sand tailings, if non-acid generating, were also considered for dam construction. The conceptual layout of each option is also presented in the report.

Knight Piesold – Geotechnical Site Investigation Report (2006a):

- This report contains geotechnical site investigation data gathered from a geotechnical site investigation and groundwater quality monitoring program. The investigation included: the waste management facility (WMF); the proposed plant site; and groundwater quality monitoring installations for the open pit. The investigation consisted of 17 geotechnical drill holes, 17 groundwater monitoring wells and 35 test pits.

1.3 Site Selection Study

The TSF site selection study included a review of all previous sites identified by Knight Piesold and optimization of the selected site. Seven sites were assessed, including some combinations of sites and these are shown in plan in Figure 1.1 (note that the coloured circles on the figure indicated environmental sampling locations in 2003). Supporting details are included in Appendix I. A comparison summary of the alternatives is included in Table 1.1.

The screening level site selection study identified two main alternatives: Site B, which is the selected site; and Site E, which is located on the southwest side of Morrison Lake. The assessment concluded that Site E, which had a similar cost to Site B, had significant risk associated with the following: potential soft foundations; a large catchment area and water management components; and loss of aquatic habitat.

Accordingly, Site B was selected for the feasibility design presented in this report. Additional optimization of the design included some modifications to the alignment of the dams and consideration of alternatives for minimizing seepage and the use of different construction materials.

Table 1.1 Comparison Summary of Waste Storage Alternatives

SITE	PRIMARY ADVANTAGES	PRIMARY DISADVANTAGES	RISKS/ OPPORTUNITIES
Site A	- Low pumping head - Short haul distance	- Large dam volume - Fish habitat disturbance - Large diverted catchment - Low potential to expand facility	- Proximity to Morrison Lake may result in costly long term seepage control. - If NAG waste rock is not available, fill borrow volumes may be prohibitive.
Site B	- Morrison Lake seepage control - Storage efficiency - Diversion/water management	- High head - Affects second watershed (Nakinilerak Lake) - Large terrestrial footprint	
Site C	- Fewer wetlands; less wildlife issues	- Potential resource in Hearne Hill - Little advantage over Site B	
Site D		- High pumping costs - Aquatic impacts and fish compensation - Similar to Site C	
Site E	- Low pumping cost - Downhill haulage cost - High storage efficiency - Low dam - Potential to expand	- Large catchment area/major diversion - Fish compensation - Creek crossing - Potentially poor foundation conditions	- The location may be prone to soft foundation conditions.
Site F	- High storage efficiency - Potential to expand	- Long over-land haul distance - Creek crossing	- Bridge or conveyor over lake could reduce waste rock haul costs, but increase capital cost
Site G	- Near pit	- Large dam volume - No significant advantages over Site A.	
Site H	- Small dam - Potential to expand	- Lake habitat - Water management - Very little advantage over Site E	

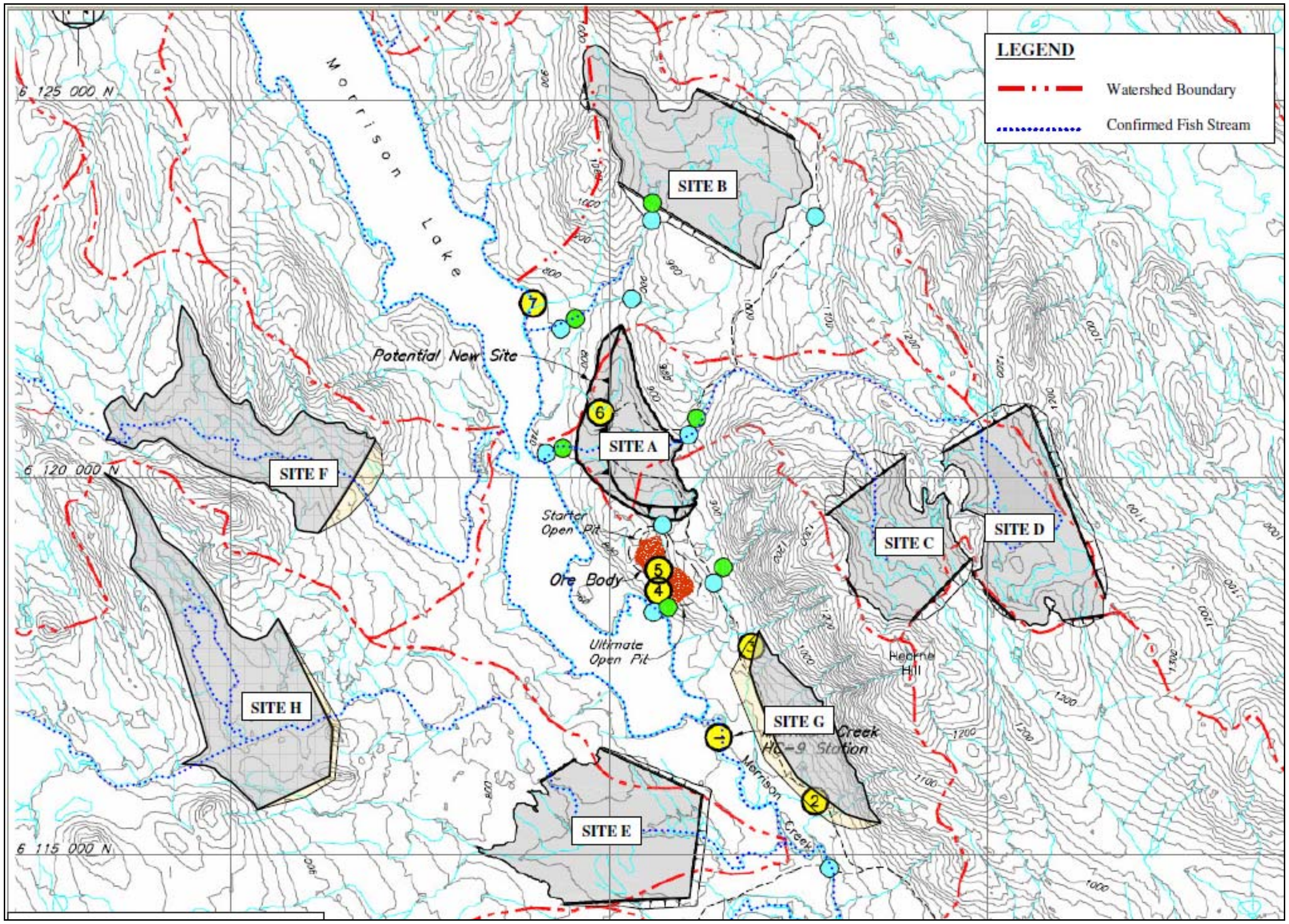


Figure 1.1 Plan Showing Mine Waste Storage Alternatives (original map extracted from KP 2003)

1.4 Report Disclaimer

This report is an instrument of service of Klohn Crippen Berger Ltd. and has been prepared for the exclusive use of Pacific Booker Minerals Inc. The contents of this report may not be relied upon by any other party without the express written permission of Klohn Crippen Berger. The contents of this report reflect Klohn Crippen Berger's best judgement in light of the information available to it at the time of preparation. Any use which a third party makes of this report, or any reliance on or decisions to be made based on it, are the responsibility of such third parties. Klohn Crippen Berger Ltd. accepts no responsibility for damages, if any, suffered by any third party as a result of decisions made or actions based on this report.

The analyses, conclusions and recommendations contained in this report are based on data derived from a limited number of test holes obtained from widely spaced subsurface explorations. The methods used indicate subsurface conditions only at the specific locations where samples were obtained or where *in situ* tests would infer, only at the time they were obtained, and only to the depths penetrated. The samples and tests cannot be relied on to accurately reflect the nature and extent of strata variations that usually exist between sampling or testing locations. Klohn Crippen Berger has endeavoured to comply with generally accepted geotechnical practice common to the local area. Klohn Crippen Berger makes no warranty, express or implied.

The recommendations included in this report have been based in part on assumptions about strata variations between test holes that will not become evident until construction or further investigation. Accordingly, Klohn Crippen Berger should be retained to perform construction observation and thereby provide a complete professional geotechnical engineering service through the observational method. If variations or other latent conditions become evident during construction, Klohn Crippen Berger will re-evaluate this report's recommendations.

The cost estimate is based on information available to date and is only our opinion of probable cost for budgetary purposes. Actual costs depend on many factors which can change with time, including site and market conditions at the time of construction.

2. DESIGN BASIS

2.1 General

Waste produced from open pit mining and tailings from ore processing will be stored in separate locations on the project site. Drawing D-1002 shows the general site arrangement.

The Tailings Storage Facility (TSF) is designed to store all tailings generated from the mill over the mine life. Additional tailings storage is also achieved by the use of tailings for the construction of cycloned sand dams.

Waste rock will be stored in the dump sites located around the north and east perimeter of the open pit, in a hillside dump. NAG will also be selectively placed in drainage inverts and in the portion of the dump which fall outside of the natural surface water catchment for the open pit. Low grade ore (LGO) will be stockpiled against the waste rock dump, directly east of the plant site, and processed towards the end of mine operations.

Stripped overburden and organic material will be stockpiled for remediation on closure, with excess material stored in a storage dump.

Waste rock will be produced throughout the mine life and a summary of estimated tonnage each year is presented in Table 2.1.

Table 2.1 Annual Tailings and Open Pit Waste Production (kilotonnes)

Year	Tailings	Open Pit Waste				
	Mill Feed	Overburden	Unknown	PAG	NAG	Total Open Pit Waste
-1	-	2,551	8	4,936	206	7,701
1	9,855	5,468	-	8,932	433	14,834
2	10,950	343	(0)	7,186	330	7,860
3	10,950	-	0	4,182	439	4,621
4	10,950	973	0	11,029	110	12,111
5	10,950	329	-	7,858	67	8,254
6	10,950	482	-	6,395	674	7,551
7	10,950	36	0	6,858	1,440	8,334
8	10,950	1,699	3	9,254	1,027	11,983
9	10,950	2,120	-	8,839	2,069	13,028
10	10,950	66	0	9,172	1,334	10,572
11	10,950	-	0	7,655	1,195	8,850
12	10,950	720	(0)	12,662	549	13,931
13	10,950	164	(0)	11,564	462	12,191
14	10,950	192	(0)	12,922	723	13,836
15	10,950	-	(0)	10,907	2,343	13,250
16	10,950	-	0	8,751	2,549	11,300
17	10,950	-	43	2,427	537	3,008
18	10,950	-	0	119	497	616
19	10,950	-	0	31	258	289
20	10,950	-	-	-	-	-
21	6,346	-	-	-	-	-
TOTAL	**224,251**	**15,144**	**54**	**151,679**	**17,242**	**184,120**

2.2 Dam Failure Classification

The Canadian Dam Association (CDA, 2007) has developed a classification scheme that can be used to provide guidance on the "standard of care" for design of dams. The standards are determined by the potential consequence classification of the tailings facility, which is then used to select criteria for the flood and seismic design. The assessment was based on consideration of the potential incremental life safety, socioeconomic, financial and environmental consequences of failure.

The tailings facility is located in an unpopulated area approximately 70 km northeast of Smithers between Morrison Lake and Nakinilerak Lake. The project is on Crown Land and current industrial land use in the area consists of forestry. The project area is used by the First Nations for recreation and Country Food use. In the event of an incident at the tailings impoundment, discharge from the tailings facility will first inundate the land in the immediate vicinity, and then enter Nakinilerak Lake or Morrison Lake. The tailings exposed to the atmosphere could potentially become acid generating.

In the event of a failure incident, any tailings and supernatant entering Nakinilerak Lake and Morrison Lake will temporarily elevate the suspended sediment and dissolved metal concentrations in the lakes near the entry area. Morrison Creek provides a spawning ground for local Coho and Sockeye fish, which are considered to be "critical" environmental components of the area. Little is known about Nakinilerak Lake, but it is also expected to be an important habitat for local fish stocks.

In the event of a failure incident, it is expected that the environmental clean-up would require recovery of the tailings, clean-up of the affected water, and construction of a new containment facility. Restoration of the area is considered to have a rating between "highly possible" to "possible, but impractical".

In contrast to a substantial environmental impact, a tailings dam failure would have minimal impact to human life, mine, and public infrastructure due to its remote location and lack of any development downstream.

Given the potential environmental damage and substantial clean-up costs, the tailings storage facility could be classified as a "Very High" consequence facility; although the selected criteria for flood and seismic design is based on an "Extreme" consequence facility to reflect the potential for future changes to land use in the area.

2.3 Design Criteria Summary

The tailings dam is designed to international standards, using International Congress of Large Dams ICOLD Guidelines (1990) and Canadian Dam Safety Guidelines (CDA, 2007), BC MEMPR, and BC Dam Safety Regulations. The main design criteria are summarized in Table 2.2 and discussed in the following sections.

The design criteria for the waste rock dump and LGO piles are lower than the tailings dam due to the significantly reduced consequence of failure. For example, a failure of the waste rock dump may result in localized slumping of the rock dumps but would not lead to any significant environmental damage and the incremental life safety consequence is low.

Table 2.2 Summary of Design Criteria

Item	Criteria
Storage Capacity	
Tailings Storage Facility:	
• Starter Dam	• Year 1 tailings production (9.9 Mt tailings @ 1.4 t/m^3)
• Ultimate Dam	• Total tailings production (224 Mt tailings @ 1.5 t/m^3 less cycloned sand)
Waste Rock Dumps:	
• Waste Rock Dump	• Total NAG, PAG waste (169 Mt @ 2 t/m^3)
• Overburden Dump	• Total Overburden and Unknown, less amounts used in dam construction (15.2 Mt @ 1.7 t/m^3)
• Organic Sediment Storage	• Excavated lake bottom sediments
Temporary Stockpiles:	
• Low Grade Ore Stockpile	• 38 Mt @ 2 t/m^3
• Organic Stockpiles	• Organic bearing soils from foundation areas (volume as required).
Water & Flood Management during Operation	
• Diversion of upland catchment, if required for water balance purposes	• 1: 100 year peak flow
• Flood management – dam safety	• storage of 30 day PMF or discharge of PMF peak flow with an emergency spillway
• Flood discharge	• flows exceeding 1:200 year peak flow can be discharged.
Seismic Return Period	
Tailings Dam – During operation and closure	• MCE PGA= 0.13 and M_w6.2
Waste Dumps – During closure	• 1000 year return period PGA = 0.06 g

Table 2.2 Summary of Design Criteria (cont'd)

Item	Criteria
Geotechnical Factors of Safety	
Tailings Dam	
• Operational & Closure	• Static FoS = 1.5 • Post-Construction FOS = 1.3 • Pseudo-static FoS = 1.0 (seismic coefficient = 50% of MCE PGA = 0.065 g) • FoS = 1.2 to 1.3 for post earthquake condition
PAG Dump, Low Grade Ore Stockpile, and Overburden Dump • Operational & Closure	• Static FoS = 1.4 • Post-Construction FoS = 1.2 • Pseudo-static FoS = 1.1 (seismic coefficient = 50% of 1000 year PGA = 0.03 g)
Organic Stockpiles • Operational Only	• Static FoS = 1.3 • Post-Construction FoS = 1.2
Environment - Operations	
• Total seepage flows out of waste dump	• Seepage to be collected and returned to the process plant
Closure	

• Flood Handling
  • Diversion Ditches
  • Spillway
	• All diversion ditches to be decommissioned.
	• PMF period routed peak flow.
	• Contingency spillway to ensure Main Dam would not be breached.

• Reclamation	• Dam slopes to be reclaimed. • Impoundment area to be combination of reclaimed beaches, if possible, and water pond areas. • PAG waste on surface to be covered and seepage treated.

2.4 Design Earthquakes

The design earthquake selected for the tailings dam satisfies the Canadian Dam Safety Guidelines (CDA, 2007) guideline for a "Very High" consequence dam, which recommends an annual exceedance probability of 1:5000 year. However, the design criterion has been upgraded to the maximum credible earthquake based on a deterministic basis. The design criterion also meets the 1:10,000 return period.

The design earthquake selected for the waste dumps was based on a low consequence of failure as the dump would deform but would not fail catastrophically. A return period of 1,000 years was conservatively selected.

2.5 Design Floods

The selection of the inflow design flood (IDF) for the TSF considered the Canadian Dam Safety Guidelines (CDA, 2007), which recommends an IDF of 2/3 between 1/1000-year and probable maximum flood (PMF). Similarly for the design earthquake criterion, the IDF flood criterion has been upgraded to the PMF. In addition, flood criterion considers combinations of rain on snow events.

The design flood can be managed with storage or an open channel spillway or, in part, with pumping. Two potential scenarios were considered, as follows:

<u>Scenario A – 30 day PMP</u>

The 30 day PMP of 0.54 m over the TSF catchment area would, assuming a runoff coefficient of 1.0, produce approximately 5.3 Mm^3. Storage of flood waters would require approximately 10 m of freeboard on the Starter Dam, reducing to 2 m of freeboard on the Ultimate Dam. Management of the flood waters could also be achieved with storage of the 7 day PMP value of 0.35 m, which is equivalent to 3.4 Mm^3, and pumping or discharge via an emergency spillway for the remaining flood waters.

Scenario B - 2 week 200 year return period (rain on snow)

This event is equivalent to 0.32 m of precipitation over the TSF catchment area, or 3.2 Mm^3. The precipitation from such and event could be stored in the impoundment, with an emergency spillway constructed to pass any additional flows.

Considering scenarios A and B, KCBL recommend a flood management design storage criterion of the 7 day PMP. This will also provide storage for a 2 week, 200 year rain on snow event, with an emergency spillway to discharge any flows in excess of this criterion. Freeboard of 1 m above the flood volume will be provided to permit construction of the spillway.

On closure, a permanent spillway in rock will be constructed to safely pass the peak flow from a PMF.

The design flood criteria selected for various other components of the water management facilities are summarized in Table 2.3. The expected operating life of the mine was taken into account in the selection of the design floods for temporary facilities, such as the surface runoff diversion ditches, the Starter Dam emergency spillway, the seepage collection ditches and the seepage recovery pond. Based on current resources, the mine is expected to be active for about 21 years. During this time all facilities related to the tailings impoundment would be closely and frequently monitored, with personnel, equipment and materials expected to be readily available in the event that remedial measures are required under routine and/or emergency maintenance.

Table 2.3 Flood Design Criteria for Water Management Facilities

Facility	Design Flood Return Period (years)	Flood Storage & Freeboard Allowance	Comments
Tailings Impoundment - Operations	PMF	1 m freeboard and storage of 3.4 Mm^3	Store 7 day PMP of 0.35 m and discharge excess flows in an emergency spillway. Assume upland ditches are not operational and a runoff coefficient of 1.0.
Tailings Impoundment on Closure	PMF	2 m above spillway invert	Routing of peak flow from the PMF over a closure spillway.
Surface Water Diversion Ditches	100		Ditches may be required for temporary water control.
Tailings Dam Closure Spillway	PMF		Assume the upland surface water diversion ditches have been decommissioned.
Seepage Collection Ditches	100		-
Seepage Recovery Pond Spillway	500		Assume the upland surface water diversion ditch is functioning.
Overburden Sediment Ponds	10		
Plant Site and Waste Dump Diversion Ditches	100		

2.6 TSF Design Seepage

The environmental design is based on protecting aquatic habitat in Morrison Lake and Nakinilerak Lake. The receiving surface water quality will meet BC aquatic life water quality guidelines (30-day-average values), which generally are more strict than drinking water guidelines.

The design basis for determining the “allowable” seepage rate from the impoundment is typically based on comparison of the tailings supernatant water quality with receiving water quality criteria for drinking water, irrigation and aquatic habitat. The assessment is based on determining the fate and transport of potential contaminants from the impoundment to the receiving environment. During operations, seepage through the dam will be collected with a seepage collection ditch and pond for return to the impoundment, as required. However, on closure it is preferable to have a passive system, which does not require ongoing controls.

Accordingly the design seepage rate is "as low as reasonably possible" and, given the generally low permeability of the impoundment soils, is targeted as < 5 L/s.

2.7 Waste Dump Design and Construction

The waste dump design is based on waste rock placed in 10 m to 20 m high lifts with intra-slope benches built to an average overall slope of 2.75H:1V. The minimum 2.75H:1V slope facilitates re-contouring of the dump slopes for reclamation at mine closure. No stability issues are expected but monitoring will be carried out to assess the dump stability during construction. If instability is observed, then lower tip head heights may be required to allow consolidation and strength gain in the foundation soils.

The water table within the waste rock dump is depressed due to the high permeability nature of the competent coarse waste rock. Water exiting at the toe of the dump will be collected in ditches or pipelines and routed to the process water tank. Furthermore, the upslope catchment area will be diverted by the clean water diversion ditch to be constructed around the upslope perimeter of the dump. As a result the waste dump will only receive precipitation falling directly in the immediate area of the dump.

Hydrology Methodology:

Total precipitation is based on frequency analyses of total precipitation (rain and snow). The runoff coefficient for flood event is 1.0. Runoff coefficient for annual runoff is 0. 5.

Diversion Channels:

Selected diversions, where possible, will divert average flows. Peak flood flows will be managed as surface runoff assuming a breach of diversion ditches.

2.8 Closure

The closure criteria for the tailings impoundment includes the following:

- PAG tailings will remain permanently saturated. NAG tailings, if they are available, could also be used to develop beaches above the final impoundment lake level;
- A "dam safety" width of 100 m of NAG rock will be placed adjacent to the upstream crest of the Main Dam and the North Dam to maintain the permanent water pond away from the crest of the dam; and
- A permanent spillway, excavated in rock, will be designed to pass the probable maximum flood (PMF) peak flow. The spillway will be located to minimize any potential for blockage due to natural hazards. A secondary spillway should also be located to provide an additional level of safety.

3. SITE INVESTIGATIONS AND LABORATORY PROGRAMS

3.1 Previous Studies

A geotechnical site investigation program was completed by Knight Piesold Ltd. of Vancouver, British Columbia, between November 2005 and April 2006 (Knight Piesold 2006a). The area of investigation includes the tailings storage facility (TSF), an earlier proposed plant site near the open pit, and groundwater quality monitoring installations for the open pit. The investigation consisted of 17 geotechnical drill holes, 17 groundwater monitoring wells and 35 test pits.

Drilling methods consisted of ODEX drilling through the overburden and rotary drilling in bedrock using HQ Triple Tube. Standard Penetration tests and Shelby tube samples were collected in soils, packer permeability tests were completed in competent bedrock, along with the collection of geotechnical data. The results show a range of depth from 4 m to 20 m of moist, stiff till throughout the TSF area. The TSF area showed a mixture of sedimentary and volcanic bedrock beneath the overburden. Investigations at the plant site and surrounding area showed a consistent, moist, stiff till overburden with both volcanic and sedimentary bedrock. Groundwater monitoring wells were installed in geotechnical drill holes on the Morrison property, but groundwater was not sampled by KCBL.

Locations of the 2006 site investigation program are shown in Drawing D-1003. Relevant geotechnical data is included in Appendix II.

3.2 2007 KCBL Site Investigation Program

KCBL completed a two-phased site investigation program in 2007 to support the feasibility design of the TSF and a proposed plant site east of Booker Lake. Later, the plant site would be relocated to the knoll west of Booker Lake. The investigation consisted of two phases:

- Phase I – Geophysics: Electrical resistivity (ERT) survey of proposed dam alignments and plant site; and
- Phase II – Drill holes and test pits in overburden and bedrock, standpipe installation, hydraulic conductivity testing, and laboratory testing.

The site investigation plan is shown in D-1003. The following sections summarize the findings of the site investigation program. Further details can be found in KCBL report “2007 Geotechnical Site Investigation” dated November 19, 2008 (Appendix III).

3.2.1 Phase I - Geophysics

Six electrical resistivity lines totalling approximately 9.5 km were surveyed between May 4 and June 12, 2007 by Frontier Geosciences Inc (Frontier). Lines were located as shown in Drawing D-1003. Data processing and inversion were done by Frontier. The inverted resistivity sections were interpreted by KCBL. The interpreted resistivity sections are presented in KCBL (2008).

In general, the interpreted resistivity sections correlate well with drill hole data and available regional structural data suggesting that data quality is good. Three main units were identified: conductive overburden, resistive bedrock and conductive bedrock. The conductive overburden unit is interpreted to be till. Highly conductive areas likely correspond to regions with higher moisture content, while resistive layers are interpreted to be layers of coarse material within the till. The extent and thickness of the overburden unit varies across the site. The northwest region covered by RL-KC07-4A and RL-KC07-4B shows only small patches of overburden with a maximum thickness of approximately 10 m, while the other areas show large extents of overburden, averaging approximately 15 m thick but in places showing >30 m (greater than the depth of the survey). Given that the resolution of the resistivity data is approximately 2 m vertical and 5 m horizontal, it is not suitable for identifying small or thin features, thus the till appears to be relatively homogeneous.

The bedrock was broadly classified into two types: resistive and conductive. Conductive bedrock was interpreted to be siltstone or other fine grained sedimentary rock, and resistive bedrock was interpreted to be sandstone. The highly resistive bedrock on lines RL-KC07-4A and RL-KC07-4B could be sandstone or volcanics, but given the resistivity values and local geology observations, is more likely to be basalt. Several local faults were interpreted on the inverted resistivity sections. Some, such as the one at the NW end of RL-KC07-1B are interpreted from a change in resistivity, while others are interpreted from linear anomalies.

3.2.2 Phase II - Drilling and Test Pitting Program

The drilling program was designed using the results of the geophysical survey in which specific areas of interest were identified for drilling. The Phase II program was conducted from November 11 to December 17, 2007, and was completed under the technical supervision of KCBL. The drilling program is summarized below:

Tailings Storage Facility

- 10 boreholes at 5 sites consisting of one deep and one shallow hole per site.
    - 5 deep boreholes drilled between 11 m and 39 m into bedrock, to total depths of between 35 m and 58 m.
    - 5 shallow boreholes in overburden, with final depths between 11 m and 25 m.
- 13 piezometers constructed in 10 boreholes at 5 sites.

East of Booker Lake (Previous Plant Site)

- 4 boreholes at 4 sites with a maximum depth of 25 m, and up to 4 m into bedrock when encountered.
- 3 piezometers were installed in 3 boreholes.
- 1 borehole was cored 39 m into bedrock, to a total depth of 47 m, to ensure the plant site area does not contain ore mineralization.

Drilling Methods

The drill holes were completed using some or all of the following drilling methods, depending on the ground conditions:

- ODEX 90 hammer pushing 4.5” casing in overburden with a hole diameter of 123 mm; and
- HQ triple tube mud rotary diamond drilling in rock with a hole diameter of 96 mm.

The drilling program is summarized in Table 3.1. Drill hole logs are presented in Appendix II. Standpipe installation methods are described in Section 3.1 and summarized in Table 3.2.

Standard Penetration Testing

Field soil logging was carried out on Standard Penetration Test (SPT) split spoon samples taken every 1.5 m (5 ft). Select samples were submitted to the KCBL soil lab for visual classification, grain size analysis, moisture content and Atterberg Limit tests. Pocket penetrometer readings were taken on “disturbed” split spoon samples in the field. These readings provide a crude measure of the relative consistency of cohesive soils.

Measured blow counts (N) were corrected for overburden pressure, hammer efficiency, rod length, and sampling method to produce $(N_1)_{60}$ values. Profiles of $(N_1)_{60}$ are plotted in geological sections on Drawings D-1005, D-1006 and D-1007.

Table 3.1 2007 Drilling Program

Drilling Hole ID	Location	Date Started (2007)	Northing (m)#	Easting (m)#	Collar Elevation (m)*	Depth to Bedrock (m)	Hole Depth (m)
DH07-1A	North Dam	Nov. 11	6,125,281	671,989	973	20.44	49.4
DH07-1B	North Dam	Nov. 15	6,125,279	671,996	973	>17.4	17.4
DH07-2A	North Dam	Nov. 16	6,125,496	671,403	990	23.9	35.1
DH07-2B	North Dam	Nov. 18	6,125,493	671,396	990	>11.0	11.0
DH07-3A	Main Dam	Nov. 19	6,123,345	671,446	974	21.9	41.6
DH07-3B	Main Dam	Nov. 22	6,123,335	671,450	974	>15.4	15.4
DH07-4A	Main Dam	Nov. 23	6,123,637	671,060	960	12.8	46.2
DH07-4B	Main Dam	Nov. 26	6,123,634	671,070	960	>11.4	11.4
DH07-5A	Main Dam	Nov. 27	6,123,951	670,477	935	19.2	21.5
DH07-5B	Main Dam	Nov. 29	6,123,965	670,477	935	19.2	58.2
DH07-6	Plant Site	Dec. 3	6,120,025	671,245	863	>23.2	23.2
DH07-7	Plant Site	Dec. 4	6,120,115	671,105	851	>22.9	22.9
DH07-8	Plant Site	Dec. 5	6,120,422	671,193	877	5.2	9.1
DH07-9	Plant Site	Dec. 5	6,120,197	671,101	841	22.3	25.3
DH07-10	Plant Site	Dec. 6	6,120,299	671,036	845	8.2	47.5

Notes:
Coordinates were determined by handheld Global Positioning System (GPS).
* Elevations were estimated from 2 m contours provided by PBM
Packer testing was done in bedrock; falling head tests were performed in overburden. See below for details.

Standpipe Installations

Sixteen standpipe piezometers were installed during the 2007 Site Investigation Program as summarized in Table 3.2 to monitor groundwater piezometric levels. Static water levels were measured, but response times in the overburden were typically much too slow to get an accurate water level. Water levels in bedrock responded much more quickly. Table 3.2 also shows piezometric levels collected in April 2008, by Rescan. These levels should not have residual effects from the drilling. Minor artesian levels were observed in DH07-1A, however this will not affect the tailings dam design.

Table 3.2 2007 Standpipe Installations[1]

Drilling Hole ID	Nested Piezo	Location	Installation Date (2007)	Total Hole Depth (mbg[2])	Piezo Stickup (mags[3])	Screen Depth (mbg[2])	Filter Pack Interval (m)	Geologic Unit at Screen Depth	Static Water Level (mbg[2])[4]	Static Water Level (mbg)[5]
DH07-1A	-	North Dam	Nov. 15	49.4	0.90	43.1 - 49.2	42.8 – 49.4	Sandstone and siltstone	-4.5 (artesian)	(artesian)
DH07-1B	-	North Dam	Nov. 16	17.4	0.83	14.2 - 17.2	13.6 – 17.4	Gravelly clay/silt (TILL)	unknown	Frozen
DH07-2A	-	North Dam	Nov. 18	35.1	0.97	31.7 - 34.7	31.1 – 35.1	Siltstone	27.7	27.74
DH07-2B	-	North Dam	Nov. 19	11	0.93	7.6 - 10.7	7 – 11	Gravelly clay (TILL)	unknown	6.32
DH07-3A	-	Main Dam	Nov. 22	41.6	0.92	38.4 - 41.5	37.8 – 41.6	Siltstone	10.7	8.59
DH07-3B	-	Main Dam	Nov. 23	15.4	0.86	12 - 15.1	11.6 – 15.4	Gravelly clay (TILL)	unknown	10.72
DH07-4A	S1	Main Dam	Nov. 25	46.2	0.82	43 - 46	42.5 – 46.2	Siltstone	9.7	10.28
DH07-4A	S2	Main Dam	Nov. 25	46.2	0.84	33.4 - 36.4	32.9 – 36.7	Sandy siltstone	10.5	10.94
DH07-4B	S1	Main Dam	Nov. 26	11.4	0.90	9.8 - 11.3	9.4 – 11.4	Gravelly clay (TILL)	unknown	10.77
DH07-4B	S2	Main Dam	Nov. 26	11.4	0.92	3 - 4.6	2.7 – 4.7	Gravelly clay (TILL)	unknown	4.04
DH07-5A	S1	Main Dam	Nov. 28	21.5	0.85	19.2 - 21.3	19.1 – 21.5	Volcanoclastic rock	9.3	11.47
DH07-5A	S2	Main Dam	Nov. 28	21.5	0.87	13.7 - 15.2	13.6 – 15.4	Gravel and clay (TILL)	10	10.62
DH07-5B	-	Main Dam	Dec. 2	58.2	0.88	55 - 58.1	54.3 – 58.2	Sandstone	unknown	11.50
DH07-6	-	Plantsite	Dec. 3	23.2	0.86	21 - 22.6	20.7 – 23.2	Silty clay (TILL)	unknown	-
DH07-7	-	Plantsite	Dec. 4	22.9	0.91	21 - 22.6	20.7 – 22.9	Clay, some gravel (TILL)	unknown	-
DH07-9	-	Plantsite	Dec. 6	25.3	0.91	18.3 - 19.8	18 – 19.8	Silty clay (TILL)	unknown	-

Notes:
1. Pipe diameter: 26 mm.
2. mbg – metres below ground.
3. mags – metres above ground surface.
4. Static water levels measured from Nov. 15 to Dec. 6, 2007 during the drilling program.
5. Static water levels measured on April 6 2008 by Rescan.

3.2.3 Test Pitting

The test pit program consisted of 8 test pits and was conducted from December 5 to December 7, 2007. A 345 Cat excavator supplied by Babine Lodge was used to dig the test pits. Test pit depths ranged from 0.8 m to 6.0 m. Grab samples were taken for moisture content, grain size and Atterberg limit tests. The test pit program is summarized in Table 3.3. The test pit logs are presented in Appendix III.

Table 3.3 2007 Test Pit Program

Test Pit	Date (2007)	Northing	Easting	Elevation (m)	Depth (m)	Depth to Bedrock (m)	Surface Material
TP07-1	Dec. 5	6,120,423	670,641	821	5	Unknown	Glacial Till
TP07-2	Dec. 6	6,121,305	670,015	795	6	Unknown	Sand and Gravel
TP07-3	Dec. 6	6,121,117	669,994	795	6	Unknown	Sand and Gravel
TP07-4	Dec. 6	6,120,999	669,939	789	6	Unknown	Sand and Gravel
TP07-5	Dec. 5	6,120,486	670,347	828	6	Unknown	Glacial Till
TP07-6	Dec. 6	6,120,827	669,928	776	6	Unknown	Glacial Till
TP07-7	Dec. 7	6,123,188	672,197	1,040	2	1	Glacial Till
TP07-8	Dec. 7	6,123,524	672,499	1,025	3.4	2.4	Glacial Till

3.3 2007 KCBL Laboratory Test Program

3.3.1 General

Geotechnical testing of selected representative soil samples was performed in KCBL’s Vancouver laboratory. Grab samples were collected from test pit excavations, and SPT split spoon samples were obtained at regular intervals in overburden drill holes.

3.3.2 Geotechnical Tests

A suite of geotechnical laboratory tests was performed on selected soil samples to characterize gradation, plasticity and compaction properties. The following is a summary of the tests performed:

- 104 moisture content tests (ASTM D2216) to determine *in situ* moisture contents;
- 21 washed sieve analyses (ASTM D422) to determine gradation;
- 10 hydrometer analyses (ASTM D422) to determine gradation of the fine portion;
- 10 Atterberg Limit tests (ASTM D4318) to assess the soil classification of the fine portion; and
- 2 Standard Proctor tests (ASTM D698) to determine a moisture-density relationship.

All the test results are presented in the report in Appendix III. The results are discussed further in Section 5.1.

3.4 2008 KCBL Site Investigation Program

Following the 2007 investigation the proposed location of the plant site changed from the east side of Booker Lake to the knoll on the west side of Booker Lake. This made room for a larger waste rock dump and low grade ore stockpile around the north and east perimeter of the open pit. These changes prompted the need for further subsurface information which led to the 2008 site investigation, completed in September 2008. Data from the geotechnical investigations are included in Appendix IV and comprised the following:

- Geological and geomorphology mapping;
- Drilling and standpipe installation program;
- Test pit program; and
- Geotechnical laboratory testing.

3.4.1 Geological and Geomorphology Mapping

To assess the terrain on site for potential geo-hazards such as mass movements, debris flows, soil creep etc that may preclude development, KCBL undertook a terrain hazard assessment as part of the site investigation process. This assessment was carried out as part of both the 2007 and 2008 site investigations and comprised the following techniques:

- Review and interpretation of colour stereo aerial photographs;
- Review of existing geological maps and data;
- Terrain modeling;
- Walkover and flyover surveys; and
- Engineering geological mapping.

The geomorphology mapping is shown on Drawing D-1009.

3.4.2 2008 Drilling Program

The 2008 drilling program was carried out jointly by KCBL and Rescan to provide both geotechnical data as well as environmental data and installations. During drilling of select monitoring well installations supervised by Rescan, KCBL recorded geotechnical data. The 2008 geotechnical drilling program is summarized in Table 3.4.

The geotechnical drilling program ran from September 12 to September 22, 2008 under KCBL's supervision and was undertaken in the waste rock dump and low grade ore stockpile foundations and at the new plant site west of Booker Lake. The program comprised the following:

- 2 boreholes (DH08-1 and DH08-2) drilled into overburden with SPT and LPT split spoon samples taken every 1.5 m to 3 m, and then drilled 3 m into the underlying bedrock. The LPT is a larger diameter penetration sampler used for coarse grained soils and LPT data is corrected to the SPT values for geotechnical assessment purposes;
- 2 pairs of monitoring wells, MW08-1(A,B) and MW08-3(A,B), installed into bedrock under the supervision of Rescan, but with geotechnical information and SPT, LPT and hydraulic conductivity data in overburden recorded by KCBL in one of each of the pairs; and
- 6 thin walled "Shelby" tube samples were collected in DH08-1A, MW08-1, and MW08-3, however recovery was limited because of the presence of gravel and cobbles in the soil.

Table 3.4 KCBL 2008 Drilling Program

<table>
<tr><th>Drilling Hole ID</th><th>Location</th><th>Date Started</th><th>Northing (m)</th><th>Easting (m)</th><th>Collar Elevation (m)*</th><th>Depth to Bedrock (m)</th><th>Hole Depth (m)</th></tr>
<tr><td>DH08-1A</td><td>W.R. Dump</td><td>Sep 12, 2008</td><td rowspan="2">6,120,064</td><td rowspan="2">670,403</td><td rowspan="2">819</td><td>14.17</td><td>20.12</td></tr>
<tr><td>DH08-1B</td><td>W.R. Dump</td><td>Sep 12, 2008</td><td>N/A</td><td>12.8</td></tr>
<tr><td>DH08-2</td><td>New plant site</td><td>Sep 13, 2008</td><td>6,120,472</td><td>670,743</td><td>795</td><td>6.26</td><td>12.3</td></tr>
<tr><td>MW08-1</td><td>W.R. Dump</td><td>Sep 15, 2008</td><td>6,119,626</td><td>671,032</td><td>849</td><td>55.88</td><td>86.2</td></tr>
<tr><td>MW08-3</td><td>LGO Stockpile</td><td>Sep 21, 2008</td><td>6,120,820</td><td>669,975</td><td>781</td><td>14.8</td><td>13.9</td></tr>
</table>

Notes:
Coordinates were determined by handheld Global Positioning System (GPS).
* Elevations were estimated from 2 m contours provided by PBM.

The 2008 drilling program also consisted of the following boreholes completed under the supervision of Rescan:

- An additional borehole (DH08-3) was completed in the process plant foundation. This borehole provided geotechnical data and provided samples for mineralogical testing; and
- Monitoring well MW08-2(A, B) was completed west of the proposed open pit.

3.4.3 Standpipe Piezometer Installations

During the 2008 site investigation, three standpipe piezometers were installed under the supervision of KCBL. These installations are summarized in Table 3.5, along with monitoring well installations completed by Rescan in boreholes in which KCBL collected geotechnical data.

Falling head tests and packer tests were carried out during drilling and in monitoring well installations to determine hydraulic conductivity of the overburden and rock.

Table 3.5 2008 Standpipe Installations[1]

Drilling Hole ID	Location	Installation Date	Total Hole Depth (mbg[2])	Screen Depth (mbg[2])	Filter Pack Interval (m)	Geologic Unit at Screen Depth	Static Water Level (mbg[2])[3]
DH08-1A	west of Booker Lake	Sep 13, 2008	20.1	16.15 – 19.2	15.24 – 20.12	Bedrock - Wacke	Unknown
DH08-1B	west of Booker Lake	Sep 14, 2008	12.8	9.14 – 12.19	8.53 – 12.8	Till	1.52
DH08-2	New Plant site	Sep 15, 2008	12.4	8.81 – 11.89	8.23 – 12.4	Bedrock – Shale	4.7
MW08-1A[4]	NE of open pit	Sep. 19, 2008	86.2	72.09~78.18	70.26~78.64	Bedrock	Unknown
MW08-1B[4]	NE of open pit	Sep 20, 2008	30.5	23.78~29.87	?~30.18	Till	Unknown
MW08-3A[4]	N of low grade ore stockpile	Sep. 21, 2008	36.2	30.48~35.05	28.96~35.51	Bedrock	Unknown
MW08-3B[4]	N of low grade ore stockpile	Sep. 22, 2008	14.8	8.99~13.72	8.23~13.89	Till	Unknown

Notes:
1. Pipe diameter: 50.1 mm.
2. mbg – meters below ground.
3. Static water levels measured on Oct. 10 2008 by Rescan.
4. Data from Rescan.

3.5 2008 Test Pit Program

The 2008 test pit program comprised excavation and logging of 17 test pits using a 325 Cat excavator supplied by Babine Lodge. Test pit depth ranged from 1.9 m to 6.0 m. Grab samples were taken for moisture content, grain size, and Atterberg tests. Undisturbed block and Shelby tube samples were obtained in softer ground for triaxial and consolidation testing. The test pit program is summarized in Table 3.6.

Table 3.6 2008 Test Pit Program

Test Pit	Date	Northing	Easting	Elevation (m)	Depth (m)	Depth to Bedrock (m)	Surface Material
TP08-A	8/9/2008	6118979	671357	833	2.8	>2.8	Glacial till
TP08-B	5/9/2008	6118943	670003	740	4.5	>4.5	Clay & gravels
TP08-C	8/9/2008	6120181	670918	822	1.9	1.9	Gravel
TP08-D	6/9/2008	6119105	669765	742	3.1	>3.1	Sand and gravel
TP08-E	9/9/2008	6120584	670128	806	4.5	>4.5	Glacial till
TP08-F	10/9/2008	6120526	669865	796	3.6	3.6	Glacial till
TP08-H	5/9/2008	6121659	670147	797	3.5	>3.5	Glacial till
TP08-I	5/9/2008	6121770	669960	808	2.9	>2.9	Sand & gravel
TP08-J	9/9/2008	6118497	671645	817	2.8	2.8	Glacial till
TP08-K	9/9/2008	6118700	671485	819	3.5	3.5	Glacial till
TP08-L	6/9/2008	6120054	670234	836	3.2	3.2	Glacial till
TP08-M	6/9/2008	6120172	669766	819	1.9	1.9	Clay
TP08-N	6/9/2008	6119673	669954	825	2.3	>2.3	Sand & gravel
TP08-O	17/9/2008	6121293	671037	906	4.9	>4.9	Glacial till
TP08-P	18/9/2008	6121210	670848	905	1.1	1.1	Glacial till
TP08-Q	18/9/2008	6121029	670810	866	6.0	>6.0	Glacial till
TP08-R	18/9/2008	6121121	670474	824	4.4	>4.4	fill

In addition to the above, KCBL undertook excavation and logging of 7 test pits within the proposed open pit area on behalf of Rescan. Samples collected from these test pits were sent to Rescan for environmental testing purposes.

3.5.1 2008 Geotechnical Laboratory Testing

Geotechnical testing of selected representative soil samples was performed in KCBL's Vancouver laboratory. The laboratory tests performed can be summarized as follows:

- 74 moisture content tests (ASTM D2216) to determine *in situ* moisture contents;
- 20 washed sieve analyses (ASTM D422) to determine gradation;
- 31 soil classification/visual descriptions (ASTM D2488) to provide accurate description of the samples;

- 19 Atterberg Limit tests (ASTM D4318) to assess the soil classification of the fine portion;
- 1 direct shear test (ASTM D3080) on a remoulded sample collected in 2007, to estimate shear strength of the glacial till;
- 1 one-dimensional consolidation test (ASTM D2435) to assess the change in void ratio and settlement rate with increased loading; and
- 1 consolidated-undrained triaxial test (ASTM D4767) to determine the shear strength and pore pressure response of the glacial till sample.

4. SITE CONDITIONS

4.1 Location, Access and Physiography

The Morrison Property is located in north-central British Columbia, centered on Latitude 55°11' north, Longitude 126°19' west on NTS map sheet 93M/01. The property is accessed from Granisle/Topley Landing on the south side of Babine Lake via a barge across the lake (4 km), thence along Nose Bay (2.5 km), Jinx (8 km), and Hagen (32 km) Forest Service Roads.

The Morrison Property lies within the Babine River/Lake drainage basin, which drains into the Skeena River, on the rolling uplands of the Nechako Plateau. This is an area of northwesterly trending ridges and valleys. The largest valleys are filled with long, narrow lakes, including nearby Morrison Lake. Most of the area is an upland surface that stands 733 m to 1370 m (Hearne Hill) above sea level (Ogryzlo et al, 1995). Numerous small lakes and bogs fill depressions resulting from the irregular topography of glacial sediments.

4.2 Geology

4.2.1 General

The Morrison deposit is located in the Intermontane Belt of central British Columbia, in the Stikine volcanic arc terrane near its northern margin with the Cache Creek terrane (Schiarizza and MacIntyre 1999; Gabrielse and Yorath, 1989).

The topography is controlled by northwesterly trending block faulting related to Eocene extension (Figure 4.1). The block faulting has resulted in older Hazelton Group (Lower to Middle Jurassic) volcanic and sedimentary rock being exposed in uplands, and younger Bowser Lake Group (Middle to Upper Jurassic) sedimentary rocks exposed in lowlands, in the areas surrounding Morrison Lake (MacIntyre 2001).

4.2.2 Bedrock Geology

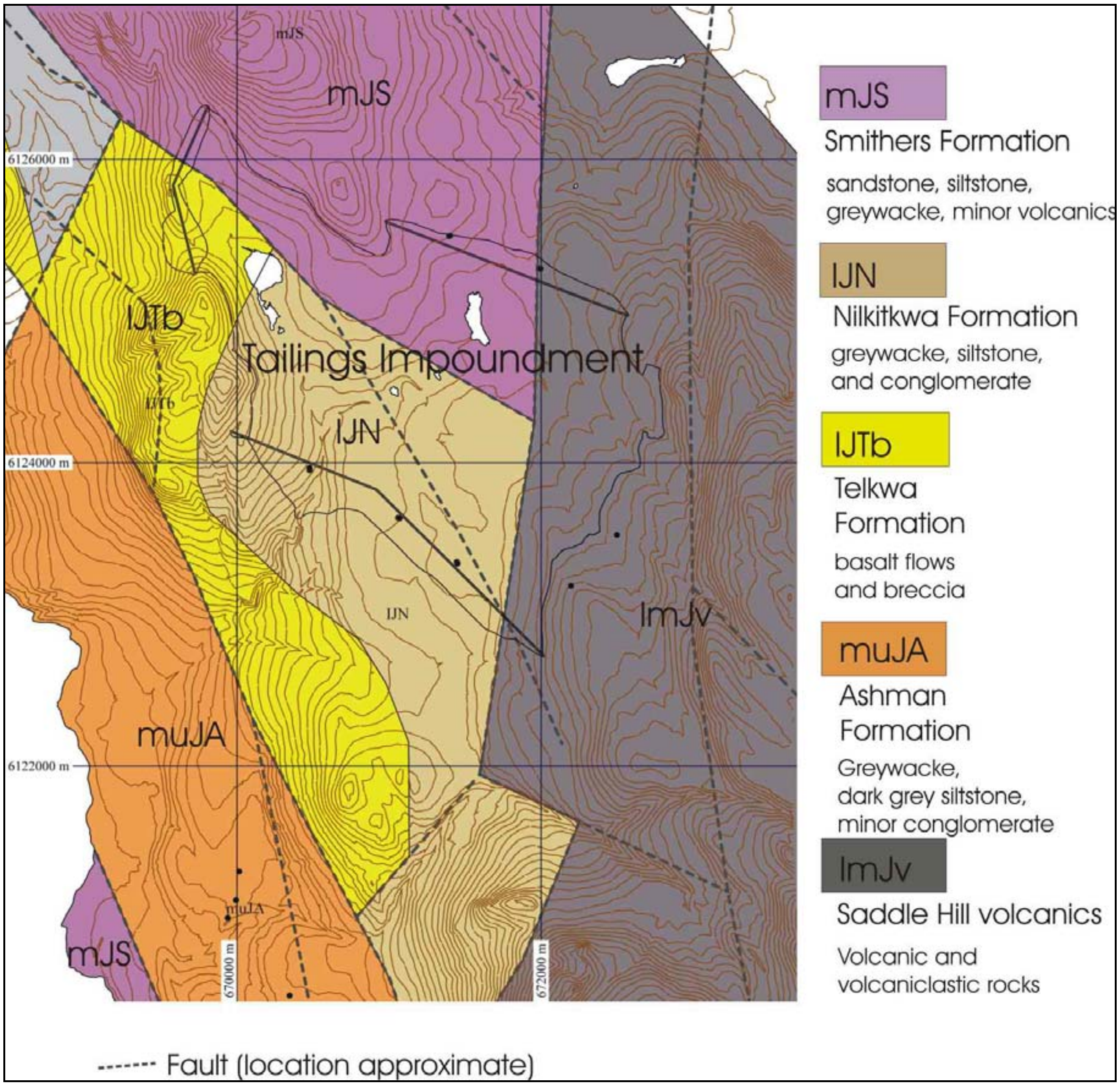
In the vicinity of the Tailings Facility, the Ashman Formation is the only Bowser Lake Group rocks exposed. Hazelton Group rocks exposed around the site include Smithers Formation, Nilkitkwa Formation, Telkwa Formation, and Saddle Hill volcanics (Table 4.1). Eocene intrusive rocks are also present, and are host to the Morrison ore deposit. However, as these intrusive rocks were not observed in the vicinity of the tailings facility, they are not described here.

Table 4.1 Rock Types Within the Tailings Storage Facility Footprint.

Group	Formation	Description
Bowser Lake Group	Ashman Formation	Dark grey siltstone, silty and sandy argillite, minor granule to pebble conglomerate, thin to medium bedded, locally fossiliferous.
Hazelton Group	Smithers Formation	Greenish grey to maroon, well-bedded, fossiliferous sandstone, siltstone, wacke and volcanic pebble conglomerate. Locally glauconitic.
	Saddle Hill Volcanics	Brown weathering, greenish grey to green basalt flows, breccias, and tuffs. Locally amygdaloidal and vesicular, with minor volcaniclastic rocks.
	Nilkitkwa Formation	Dark grey, well bedded sand and siltstones, greywacke, with minor pebble conglomerate. [Overlies Telkwa Formation].
	Telkwa Formation	Maroon to greenish grey amygdaloidal basalt flows and breccias. Calcite and chlorite filled amygdules. [Underlies Nilkitkwa Formation.]

(Based on MacIntyre et al. 1997 and MacIntyre 2001).

Varying degrees of alteration have occurred throughout the rocks of the area, resulting in enrichment in carbonate with lesser chlorite and biotite. Intense clay carbonate alteration is primarily associated with faults and related shears.



(Based on MacIntyre 2001)

Figure 4.1 Bedrock Geology of the Tailings Impoundment

4.2.3 Structural Geology

Four major tectonic events have been documented in the region (MacIntyre et al. 1997). Mid to Late Jurassic folding and uplift was followed by mid Cretaceous contraction that produced northwest trending folds and northeast directed thrusts. Crustal extension in

Late Eocene time produced north trending grabens and horsts. The most recent tectonic event was during the Miocene, resulting in tilted fault blocks.

The most prominent structural feature is the northwest trending Morrison Fault, which dextrally disrupted the porphyry system, with displacement estimated at 300 m (Kimura, 2003). Smaller dextral *en-echelon* oriented faults lie sub-parallel to the Morrison Fault (Simpson, 2007). Intense clay carbonate alteration is associated with fault zones. Mineralized fractures 2 cm to 10 cm apart are visible in trenches and outcrops. The fractures have orientations in all directions but mostly dip steeply and trend northerly, parallel to the strike of the Morrison fault. At the northern end of the deposit, the strikes of the fractures swing to the east and northeast (Carter, 1973; Richards, 1974).

4.2.4 Surficial Geology and Geomorphology

The current landscape is the result of recent geologic processes and the legacy of erosion and deposition by the Cordilleran Ice Sheet, which likely covered the Nechako Plateau several times during the Pleistocene. The most recent glacial event is known as the Fraser Glaciation, which reached its maximum ice extent between 25,000 and 12,000 years ago. The surficial material in the study area appears to have been deposited during this glaciation and in the post-glacial period.

Surficial materials include glacial till, glaciofluvial gravels, colluvium, and organics. Silt- and clay-rich glacial till is by far the most common unit, and overlies fractured fine grained sedimentary and volcanic rocks. Steeper slopes are typically covered by a thin veneer of glacial till or colluvium overlying bedrock, and flat areas between northwesterly trending ridges have a thicker cover of glacial till. The low permeability glacial till creates poor drainage and wet conditions in flat areas where till has filled depressions in the bedrock. Flat areas are poorly drained, and tend to be swampy with accumulations of organic sediments of 1 m to 2 m deep.

Well developed flutings and drumlins oriented parallel to Morrison Lake are a dominant feature of the area, and are the result of ice flowing southeastward from the Coast Mountains (Levson 2002). Below about 950 m elevation, glaciolacustrine sediments are widespread to the south around Babine Lake, but are rare around Morrison Lake. Glaciofluvial deposits are present in isolated fan-deltas at elevations of about 800 m, and may be present at lower elevations as well.

Surficial mapping and terrain stability mapping was carried out in 2008 and is presented in Drawing D-1009. Previous terrain mapping covering the project site was undertaken for British Columbia Ministry of Forests by Klohn Crippen (1998). This mapping was done at Terrain Survey Intensity Level (TSIL) D (BC Ministry of Environment 1995), assessed terrain stability at a reconnaissance scale, and identified unstable and possibly unstable areas.

Main Dam Foundation

The Main Dam straddles a broad flat area for much of its length, with steeper terrain at the right (west) and left (east) abutments. The central flat area is underlain by clayey glacial till varying in thickness from 0 m to 21 m. Glacial till masks the undulating bedrock, in the rolling plateau topography (see RL-KC07-1A). Bedrock was observed to outcrop at the crest of small slopes separating flat areas. Water saturated organic silt and peat deposits between 1 m and 2 m thick occur in the poorly drained areas between the small steps. The right (west) abutment is a moderately sloping gully, transitioning into a bedrock controlled slope rising to the west. In the vicinity of the gully, coarser fluvial and glaciofluvial sediments overlie glacial till. Post-glacial fluvial sediments appear to be thin and restricted to the modern channel. The bedrock-controlled slope has a decreasing thickness of glacial till with increasing elevation, and accumulations of colluvium (reworked glacial till) were seen in the resistivity profile at the toe of steeper slopes.

North Dam Foundation

The North Dam straddles a saddle separating southward drainage towards Morrison Lake and the Skeena River, from northward drainage towards Nakinilerak Lake and the Fraser River. Topography in the vicinity of the saddle is subdued with generally gentle slopes. The right (east) abutment has a thin cover of glacial till between 2 m and 5 m thick, and the left (west) abutment has a variable thickness of glacial till between 1 m and 24 m thick. Glacial till was the only surficial sediment observed besides very thin and localized fluvial and organic sediments. Glacial till is very similar to that observed at the Main Dam foundation. However, a clay layer, overlying the bedrock contact in DH07-1A was observed, which could be a pre-glacial or interglacial lacustrine deposit.

West Dam Foundation

No drilling was performed at the West Dam, however, resistivity data show a very thin cover of surficial material over bedrock, typically less than 2 m thick.

The overburden deposits at each of the dam sites were classified into three generalized units based on physical and depositional characteristics, such as method of deposition, gradation, and permeability. These soil units are classified and described as follows:

- Water saturated surface organics;
- Permeable glaciofluvial sand and gravel; and
- Dense impermeable glacial till.

Waste Rock Dump Foundation

The waste rock dump sits against the hillside to the east and north of the proposed open pit. The foundations are comprised primarily of the predominant clayey glacial till deposit with thickness varying up to 50 m. Glacial till thickness is generally at its maximum at the base of the hillside where it fills in a bedrock valley, and thins with increased elevation. Occasional bog deposits present localized soft surficial sediments and organic matter. In addition, soft sediments are expected to be present in Booker Lake and the Pit Pond, which will be drained and excavated for preparation of the waste dump. A small rotational slide on the upper slopes of the hillside is identifiable in aerial photographs of the site (Drawing D-1009). The movement appears to be localized in nature and is near the upper-most limit of the dump foundation, posing negligible risk to construction of waste rock dump. There was no evidence for significant mass movements on site.

Plant Site

The new plant site is located on the rocky knoll north of the open pit. Three test pits and two drill holes were done in this area. Highly to moderately weathered bedrock is exposed on the very top of the knoll, with overburden cover increasing with a drop in elevation, (i.e. with distance from the knoll top). The overburden comprises mostly dense, low permeability gravelly clay, inferred as a glacial till, together with local areas of sandy, cobbley, gravel material inferred as completely weathered bedrock. The water table in this area is approximately 4 m below the ground surface.

4.2.5 Surficial Material Types

The main soils in the project area are described in the following sections.

- **Water Saturated Surface Organics**: PEAT (PT), dark brown, moist to wet, fine to coarse fibrous, some silt. At > 0.3 m depth is: ORGANIC SILT (OL), low plasticity, soft, dark brown, wet, massive, low dry strength, rapid dilatancy, organics are amorphous to fine fibrous. Pockets of organics are located in flat poorly drained areas.

- **Permeable Glaciofluvial Sand and Gravel**: GRAVEL (GW), fine to medium grained, fine to coarse sandy to some sand, loose, rounded, brownish grey, up to trace cobbles, no fines, moist, glaciofluvial.

Glaciofluvial sediments were not observed within the footprint of the TSF, but are suspected within the gully of the Main Dam.

- **Low Permeability Glacial Till:** Sandy Lean CLAY (CI), trace to some gravel, low to intermediate plasticity, soft to very stiff, brown, no odour, moist, uncemented, high dry strength, slow dilatancy, glacial till. Till is uniform and structureless, with rare lenses of gravel and sand, usually mixed with fines. It is widespread.

4.3 Seismic Hazard Assessment

Probabilistic and deterministic seismic hazard analyses were conducted to derive the Maximum Credible Earthquake (MCE) Peak Ground Acceleration (PGA) for the project site. A brief summary of the seismic hazard assessment is presented below. See Appendix V for the details.

4.3.1 Probabilistic Seismic Hazard Assessment

The probabilistic seismic hazard assessment was carried out using the Cornell-McGuire approach embodied in the computer program Ez-Frisk (Risk Engineering Inc.). The Geological survey of Canada's (GSC) seismic source zone models GSC-H and GSC-R were used in the hazard assessment. The GSC-H and GSC-R models were used by GSC to develop the seismic hazard maps for the 2005 NBCC (National Building Code of Canada) (Adams and Halchuck, 2003).

Uncertainties in the seismic hazard assessment should be considered and addressed quantitatively to obtain reliable estimate of ground motions. Two types of uncertainties are normally considered in probabilistic analyses, namely the aleatory uncertainty and the epistemic uncertainty. The aleatory uncertainty or the random uncertainty is due to the physical variability of the earthquake processes such as the randomness of the location of the earthquakes and the scatter in the earthquake ground motions. This uncertainty is readily incorporated within the Cornell-McGuire analysis frame work by integrating over the statistical distribution in the ground motion relations and by considering the randomness in earthquake location.

The epistemic uncertainty or the model uncertainty is due to incomplete understanding of the physical models governing the earthquake occurrence and ground motion generation (i.e. selection and characterization of sources, recurrence rates, maximum magnitude ground motion attenuation equations). The epistemic uncertainty is handled in the present assessment through the use of sensitivity analyses as recommended by CDA (2007).

The sensitivity analyses in the present assessment were conducted to determine the effect of the following key uncertainties:

- Source zone models GSC-H and GSC-R;
- Source zone model parameters such as the maximum magnitude, recurrence rate and depth; and
- Ground motion attenuation equations.

Uncertainty in the attenuation equations is one of the significant contributors to overall uncertainty in a seismic hazard assessment and it is often assessed by considering alternative sets of attenuation equations. The assessment used the attenuation equation by Boore, Joyner Fumal (1997) and the NGA (New Generation Attenuation Model Project) equations proposed by Boore and Atkinson (2007), Campbell and Bozorgnia (2007), and Chiou and Youngs (2006) for the shallow crustal sources that dominate the hazard at our project site.

Based on the probabilistic seismic hazard assessment, the MCE PGA was estimated as 0.13 g and Figure 4.2 shows the corresponding hazard curve. The 1000 year return period PGA is 0.06g. Based on the de-aggregation analyses, which shows the magnitude-distance contributions to the probabilistic hazard, an earthquake magnitude of M_w6.2 was selected for seismic deformation and liquefaction assessment.

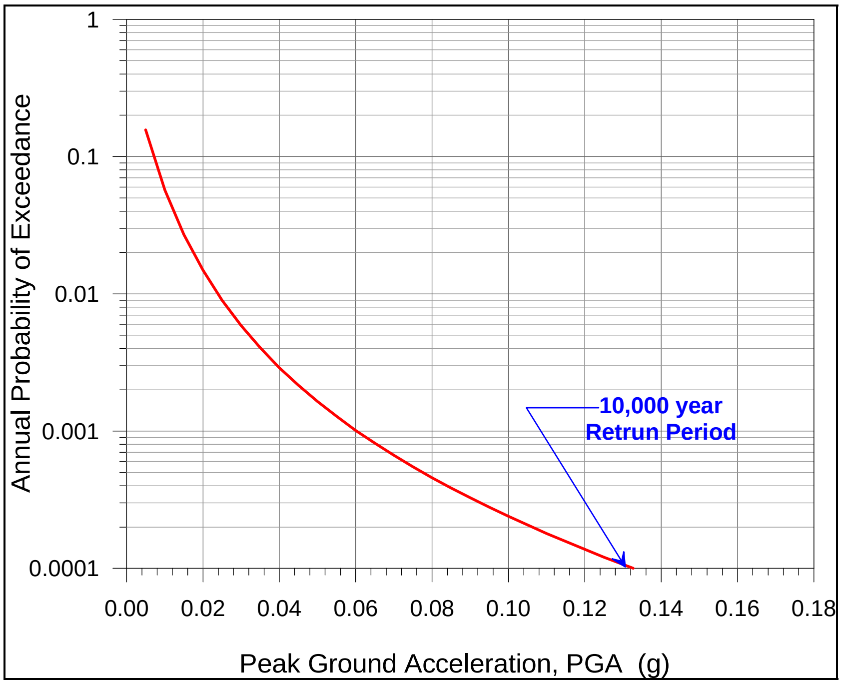


Figure 4.2 Seismic Hazard Curve from Probabilistic Seismic Hazard Assessment

4.3.2 Deterministic Seismic Hazard Assessment

A deterministic seismic hazard assessment was also carried out for the project site considering two potential earthquake scenarios. In Scenario 1, an earthquake of magnitude M_w8.5 occurring at the Queen Charlotte Fault was considered. The estimated PGA from this scenario earthquake is 0.05 g.

In the Scenario 2, random floating earthquake with magnitudes, M_w4, M_w4.5 and M_w5 in the vicinity of the project site was considered. The attenuation equations by Boore, Joyner Fumal (1997), Boore and Atkinson (2007), Campbell and Bozorgnia (2007), and

Chiou and Youngs (2006) were used in the estimation of PGA. The estimated PGAs ranged between 0.03 g and 0.10 g.

4.3.3 Recommendations for MCE Ground Motions

Based on the seismic hazard assessment, an MCE PGA of 0.13 g is recommended for the Morrison project site and it should be associated with an earthquake magnitude of M_w6.2 in seismic deformation and liquefaction assessments. The recommended ground motion is applicable to Site Class C (i.e. very dense soil or soft rock with shear wave velocity of 360 m/s to 760 m/s in the top 30 m) conditions (NEHRP, 2003).

4.4 Climate

4.4.1 Weather Stations

Regional Weather Stations

Environment Canada has established several meteorological stations in North Central British Columbia. Data from five stations in the region were used for the site characterization. The meteorological stations and their periods of record are summarized in Table 4.2.

Table 4.2 Regional Weather Stations

Station Name	Station No.	Lat.	Long.	Elevation (m)	Period of Record	Complete Years of Record
Babine Lake at Pinkut Creek	1070573	54.45	-125.46	713	1969-2005	29
Fort Babine	1072902	55.32	-126.62	716	1944-1975	19
Smithers Airport	1077500	54.82	-127.18	522	1942-2007	62
Topley Landing	1078209	54.81	-126.16	722	1962-2005	39
Takla Landing	1097970	55.47	-125.97	854	1962-1991	20

Morrison Project Site Data

An on-site meteorological station was installed in June 2006. The station logs hourly measurements. Site data are included in Appendix II.

4.4.2 Precipitation (Rainfall and Snowfall)

Mean Annual Precipitation

KCBL estimated the mean annual precipitation for the project area based on the available regional precipitation data. Average annual precipitation between the five surrounding stations has shown to be relatively consistent. Average annual precipitation for each station is presented in Table 4.3 and the mean annual precipitation from this data is 535 mm.

The tailings area average elevation is approximately 950 m. The above regional stations are located at lower elevations. An elevation-precipitation relationship was established (see Appendix II). The estimated mean annual precipitation at the tailings area is 550 mm.

The Smithers Airport regional data were available up to December 2007. Overlap of data between the site and Smithers Airport station was available from June 2006 through December 2007. The correlation between the total precipitation measured at the site weather station and Smithers Airport was used to estimate the site mean annual precipitation. The data used for determining the correlation are included in Appendix II. The estimated mean annual precipitation at the mine site, based on this data would be 528 mm.

Sufficient correlation, however, between the estimated mean annual precipitation at the mine site and the regional average annual precipitation was observed. Precipitation at the site is therefore assumed to be the average between the five regional stations, with elevation modification, for an annual precipitation of 550 mm.

Table 4.3 Monthly Precipitation Distribution

Station Name	Oct (%)	Nov (%)	Dec (%)	Jan (%)	Feb (%)	Mar (%)	Apr (%)	May (%)	Jun (%)	Jul (%)	Aug (%)	Sep (%)	Mean Annual Precipitation (mm)
Regional Stations													
Babine Lake at Pinkut Creek	8.7	9.3	10.7	11.1	7.4	5.6	4.9	6.9	10.6	8.5	8.0	8.4	493
Fort Babine	10.8	10.3	12.2	10.5	7.5	4.8	5.3	5.9	7.5	8.7	8.1	8.6	596
Smithers Airport	12.0	10.8	10.1	10.1	5.4	4.4	4.2	6.6	9.2	9.1	8.3	9.9	517
Topley Landing	9.2	10.3	10.6	10.2	6.4	5.5	4.4	7.3	10.1	9.0	8.3	8.6	537
Takla Landing	10.0	10.5	9.9	10.3	6.7	5.4	3.4	6.3	9.8	8.9	8.9	9.8	534
Average	**10.1**	**10.2**	**10.7**	**10.4**	**6.7**	**5.1**	**4.4**	**6.6**	**9.4**	**8.8**	**8.3**	**9.1**	**535**

Note: Based on hydrological year, October to September for periods with complete data.

Rescan (2008) estimated the mean annual precipitation only from the regional stations. PRISM (Parameter-elevation Regression on Independent Slopes Model) was used to estimate the mine site mean annual precipitation. Using the data from Topley Landing, PRISM estimated a mean annual precipitation of 750 mm at the mine site. The PRISM mode, however, was not validated with site data and KCBL believes that this method overestimates the precipitation at the mine and, consequently, the higher value was not used for design.

Monthly Precipitation Distribution

Distribution of precipitation throughout an average hydrological year is presented in Table 4.3 for each of the five regional stations. It can be observed that the distribution of precipitation is similar at all of the five regional stations.

Comparison between site precipitation distribution and that measured at the Smithers Airport from October 2006 through Sept 2007 illustrated that it is reasonable to assume that the distribution of precipitation at the site can be characterized by the surrounding regional stations. The annual distribution of precipitation is calculated as the average of each of the five regional stations.

Average annual rainfall and average annual snowfall, along with monthly distribution of each are presented in Table 4.4 and Table 4.5 respectively.

Table 4.4 Monthly Rainfall Distribution

Station Name	Oct (%)	Nov (%)	Dec (%)	Jan (%)	Feb (%)	Mar (%)	Apr (%)	May (%)	Jun (%)	Jul (%)	Aug (%)	Sep (%)	Mean Annual Rainfall (mm)
Regional Stations													
Babine Lake at Pinkut Creek	13.2	5.7	1.1	1.2	0.7	1.4	5.7	11.4	17.8	14.3	13.5	14.0	293
Fort Babine	16.0	5.8	0.8	0.4	0.3	0.8	5.8	10.6	13.6	15.8	14.6	15.5	329
Smithers Airport	15.8	7.2	2.7	2.7	1.5	1.9	4.6	9.5	13.6	13.5	12.3	14.6	348
Topley Landing	12.8	4.2	0.8	1.1	0.9	1.3	4.9	12.1	17.4	15.5	14.2	14.8	313
Takla Landing	15.6	2.8	0.2	0.2	0.5	0.4	4.1	10.8	17.2	15.6	15.7	17.1	304
Average	**14.7**	**5.1**	**1.1**	**1.1**	**0.8**	**1.2**	**5.0**	**10.9**	**15.9**	**14.9**	**14.1**	**15.2**	**317**

Note: Based on hydrological year, October to September for periods with complete data.

Table 4.5 Monthly Snowfall Distribution

Station Name	Oct (%)	Nov (%)	Dec (%)	Jan (%)	Feb (%)	Mar (%)	Apr (%)	May (%)	Jun (%)	Jul (%)	Aug (%)	Sep (%)	Mean Annual Snowfall (mm)
Regional Stations													
Babine Lake at Pinkut Creek	2.1	14.6	24.8	25.6	17.1	11.7	3.6	0.4	0.0	0.0	0.0	0.1	200
Fort Babine	4.4	15.7	26.1	22.8	16.3	9.7	4.7	0.2	0.0	0.0	0.0	0.1	268
Smithers Airport	3.8	17.8	25.5	26.4	13.5	9.4	3.1	0.4	0.0	0.0	0.0	0.1	200
Topley Landing	4.0	18.8	24.3	22.9	14.1	11.3	3.7	0.7	0.0	0.0	0.0	0.1	225
Takla Landing	2.8	17.4	23.7	24.7	15.6	12.4	2.7	0.3	0.0	0.0	0.0	0.3	221
Average	**3.5**	**17.6**	**25.1**	**24.0**	**14.9**	**10.8**	**3.5**	**0.4**	**0.0**	**0.0**	**0.0**	**0.1**	**223**

Note: Based on hydrological year, October to September for periods with complete data.

4.4.3 Evaporation

The only regional station reporting lake evaporation measurements is Topley Landing, whose data are available from the Canada Climate Normals. Lake evaporation at the site is assumed to have the same total annual lake evaporation, as well as annual distribution as that reported in the Topley Landing Climate Normals. Annual lake evaporation is presented in Table 4.6.

4.4.4 Temperature

Average monthly temperatures for the regional meteorological stations are presented in Table 4.7. Note that temperature is relatively consistent throughout all regional stations. Site temperature data were compared with the Smithers Airport temperature data for the period of data overlap (June 2006 to Dec 2007), and the temperature distribution was similar, with temperatures consistently 2^{o} C to 3^{o} C lower at site. The difference in temperature is attributed to the difference in elevation between the Smithers Airport regional station (522 m) and the site station (~850 m). Site temperature data is included in Appendix II.

4.4.5 Wind

Regional wind information was available from the Canada Climate Normals for Babine Lake at Pinkut Creek, and for the Smithers Airport. Table 4.8 outlines the wind information available through the Canada Climate Normals. Wind speed and direction were measured at the Morrison site weather station since June 2006. Site wind data are included in Appendix II.

Table 4.6 Monthly Lake Evaporation Distribution

Station Name	Oct (%)	Nov (%)	Dec (%)	Jan (%)	Feb (%)	Mar (%)	Apr (%)	May (%)	Jun (%)	Jul (%)	Aug (%)	Sep (%)	Mean Annual Evaporation (mm)
Topley Landing	0.0	0.0	0.0	0.0	0.0	0.0	0.0	21.5	23.1	24.7	19.9	10.8	389

Table 4.7 Monthly Temperatures

Station Name	Oct (%)	Nov (%)	Dec (%)	Jan (%)	Feb (%)	Mar (%)	Apr (%)	May (%)	Jun (%)	Jul (%)	Aug (%)	Sep (%)	Mean Annual Temperature (°C)
Regional Stations													
Babine Lake at Pinkut Creek	4.5	-2.0	-6.5	-9.3	-6.6	-2.0	3.2	7.9	12.0	14.7	14.3	10.0	3.3
Fort Babine	3.0	-3.5	-9.7	-14.0	-7.8	-4.0	1.4	6.6	10.9	13.0	12.3	7.9	1.3
Smithers Airport	4.5	-2.3	-7.2	-9.3	-5.0	-0.7	4.5	9.2	12.8	14.8	14.2	9.9	3.8
Topley Landing	3.9	-2.6	-7.8	-10.6	-6.3	-1.8	3.3	8.1	12.2	14.3	13.9	9.4	3.0
Takla Landing	4.5	-3.6	-6.1	-7.3	-9.3	-2.3	2.8	7.1	11.3	14.0	13.6	9.2	2.8
Average	**4.1**	**-2.8**	**-7.4**	**-10.1**	**-7.0**	**-2.2**	**3.0**	**7.8**	**11.8**	**14.2**	**13.7**	**9.3**	**2.9**

Table 4.8 Regional Wind Data

	Oct	Nov	Dec	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep
Smithers Airport												
Average												
Speed (km/h)	6.2	6.7	6.9	7.3	7.4	6.9	7.6	7.4	6.8	5.8	5.3	5
Most Frequent Direction	SE	SE	SE	SE	SE	SE	SE	NW	NW	NW	SE	SE
Max Hourly												
Speed (km/h)	58	64	51	66	56	55	61	51	57	50	46	56
Direction	SE	SW	SW	SW	SW	W	SW	NW	N	SE	SW	SW
Maximum Gust												
Speed (km/h)	89	106	111	120	120	107	93	100	74	78	74	81
Direction	SW	E	S	SW	W	SW	SW	SW	S	NE	SE	S
Babine Lake at Pinkut Creek												
Average												
Speed (km/h)	8	9.3			7.9					9	8.5	8
Most Frequent Direction	S				W						W	S
Max Hourly												
Speed (km/h)	40	50	53	48	50	47	51	40	35	42	40	45
Direction	W	NE	E	NE	NE	E	NE	NE	W	W	W	NE

4.5 Hydrology

4.5.1 Regional Hydrometric Stations

Environment Canada has established several hydrometric stations in North Central British Columbia. Data from seven hydrometric stations in the region were used for the site characterization. The hydrometric stations and periods of record are summarized in Table 4.9.

Table 4.9 Regional Hydrometric Stations

Station Name	Station No.	Lat.	Long.	Mean Basin Elevation (m)	Drainage Basin Area (km^2)	Flow Regime	Period of Record	Complete Years of Record
Babine River at Babine	08EC001	55.321	-126.628	1,120	6,350	Natural	1929-1985	39
Fulton River at the Mouth	08EC002	54.814	-126.171	N/A	1,400	Regulated	1963-1970	5
Pinkut Creek Near Tintagel	08EC004	54.415	-125.428	1,211	809	Regulated	1961-2006	40
Morrison River at Outlet of Morrison Lake	08EC008	55.172	-126.303	N/A	414	Natural	1965-1970	1
Babine River at Outlet of Nilkitwa Lake	08EC013	55.43	-126.70	1,125	6,760	Natural	1972-2006	34
Twain Creek Tributary Near Babine Lake	08EC014	54.56	-125.91	1,253	6	Natural	1998-2006	7
Kazchek Cr Near Mouth	08EJ005	54.90	-125.14	N/A	881	Natural	1980-1989	5

4.5.2 Annual Runoff Coefficient

KCBL calculated the average annual runoff coefficient for the drainage basins of hydrometric stations included in Table 4.9. The average annual discharge was used to calculate the average annual runoff volume. The average annual rainfall was calculated assuming an annual rainfall depth of 550 mm. Calculated runoff coefficients are presented in Table 4.10 and ranged between 0.35 and 0.61. An average annual runoff coefficient of 0.5 was estimated from the regional data. The mean annual runoff is estimated 275 mm.

Table 4.10 Annual Runoff Coefficients

Station Name	Basin Area (km^2)	Annual Rainfall (mm)	Annual Runoff ($10^6 m^3$)	Annual Runoff Coefficient
Babine River at Babine	6,350	550	1,455	0.42
Fulton River at the Mouth	1,400	550	471	0.61
Pinkut Creek Near Tintagel	809	550	158	0.35
Morrison River at Outlet of Morrison Lake	414	550	121	0.53
Babine River at Outlet of Nilkitwa Lake	6,760	550	1,547	0.42
Twain Creek Tributary Near Babine Lake	5.5	550	1.9	0.61
Kazchek Cr Near Mouth	881	550	208	0.43
Average				**0.48**

Rescan (2007) estimated an annual runoff coefficient of 0.6. Data from four selected regional hydrometric stations were used by Rescan, in conjunction with the assumption of a mean annual precipitation of 750 mm. The site flow data were not included in determining the runoff coefficient. KCBL have used 0.5 as the best estimate of the runoff coefficient based on the data presented above.

4.5.3 Runoff Distribution

Mean monthly flows were calculated for each of the regional hydrometric stations for years from which there are complete years of record. The monthly flows are shown in Table 4.11. The distribution of runoff for each station is shown in Table 4.12. The runoff distribution for the site was calculated as the average between Morrison River (08EC008), Twain Creek Tributary (08EC014), and Kazchek Creek (08EJ005). The other stations were considered unsuitable since they had either controlled flow regimes, or the drainage area was too large to be representative of the site. Estimated site runoff distribution is presented in Table 4.12.

Table 4.11 Regional Mean Monthly Stream Flow

Station Name	Oct (m^3/s)	Nov (m^3/s)	Dec (m^3/s)	Jan (m^3/s)	Feb (m^3/s)	Mar (m^3/s)	Apr (m^3/s)	May (m^3/s)	Jun (m^3/s)	Jul (m^3/s)	Aug (m^3/s)	Sep (m^3/s)	Average Annual (m^3/s)
Regional Stations													
Babine River at Babine	35.8	31.2	26.7	22.2	19.6	17.6	17.3	58.4	115.7	95.9	66.1	45.2	46.0
Fulton River at the Mouth	11.2	8.8	6.2	4.6	4.7	3.7	5.5	63.6	43.0	14.3	8.1	4.7	14.9
Pinkut Creek Near Tintagel	2.3	2.7	2.4	2.2	2.1	2.2	3.3	16.2	15.2	5.8	3.0	2.7	5.0
Morrison River at Outlet of Morrison Lake	0.9	2.1	1.4	1.3	1.2	0.8	2.7	17.0	10.6	3.1	1.6	3.2	3.8
Babine River at Outlet of Nilkitwa Lake	34.6	30.6	26.1	23.4	22.4	21.5	28.6	84.8	120.5	93.2	59.7	41.3	48.9
Twain Creek Tributary Near Babine Lake	0.01	0.0	0.0	0.0	0.0	0.00	0.09	0.38	0.16	0.03	0.01	0.03	0.1
Kazchek Cr Near Mouth	1.9	2.3	2.2	2.5	2.2	2.1	5.4	25.9	19.8	9.4	3.3	1.6	6.6

Table 4.12 Monthly Flow Distribution

Station Name	Oct (%)	Nov (%)	Dec (%)	Jan (%)	Feb (%)	Mar (%)	Apr (%)	May (%)	Jun (%)	Jul (%)	Aug (%)	Sep (%)
Regional Stations												
Babine River at Babine	6.5	5.7	4.8	4.0	3.6	3.2	3.1	10.6	21.0	17.4	12.0	8.2
Fulton River at the Mouth	6.3	5.0	3.5	2.6	2.6	2.1	3.1	35.7	24.1	8.0	4.6	2.6
Pinkut Creek Near Tintagel	3.8	4.5	4.0	3.6	3.5	3.7	5.5	27.1	25.4	9.6	4.9	4.5
Morrison River at Outlet of Morrison Lake	2.0	4.6	3.0	2.8	2.5	1.8	5.9	37.1	23.1	6.7	3.5	7.0
Babine River at Outlet of Nilkitwa Lake	5.9	5.2	4.4	4.0	3.8	3.7	4.9	14.5	20.5	15.9	10.2	7.0
Twain Creek Tributary Near Babine Lake	0.8	0.0	0.0	0.0	0.0	0.0	12.5	53.6	23.0	4.6	1.1	4.4
Kazchek Cr Near Mouth	2.4	2.9	2.8	3.2	2.7	2.7	6.8	32.9	25.2	12.0	4.3	2.1
Morrison Site Estimate	**1.7**	**2.5**	**1.9**	**2.0**	**1.8**	**1.5**	**8.4**	**41.2**	**23.8**	**7.8**	**3.0**	**4.5**

4.5.4 Intensity-Duration -Frequency Relationships

The Intensity-Duration-Frequency (IDF) data were available from Environment Canada stations and are summarized in Table 4.13.

Table 4.13 Regional Stations with IDF

Station Name	Station No.	Lat.	Long.	Elevation (m)	Period of Record	Complete Years of Record
Short Duration IDF Stations						
Germansen Landing	1183090	55.47	-124.42	746	1964-1990	26
MacKenzie	1184790	55.18	-123.08	697	1971-1990	20
Ft St James Airport	1092972	54.25	124.15	716	1980-1990	11
Burns Lake	1091169	54.14	-125.46	704	1969-1990	21
Quick	1076638	54.37	-126.54	533	1963-1990	25
Smithers Airport	1077500	54.82	-127.18	522	1971-1990	20
1 to 30 day Duration IDF Stations						
Ft Babine	1072902	55.32	-126.62	716	1944-1975	26 (20)*
Topley	1078209	54.81	-126.16	722	1962-2005	39 (39)*
Takla	1097970	55.47	-125.97	854	1962-1991	25 (-)*

* Complete years available for rainfall only analysis (Complete years available for rainfall + snowmelt analysis)

Only rainfall IDF data were available for the Probable Maximum Precipitation (PMP). Table 4.14 and Table 4.15 summarize the IDF relationships.

Table 4.14 Rainfall Only IDF

Duration		Total Depth (mm)							
(hours)	(days)	2 yr	5 yr	10 yr	25 yr	50 yr	100 yr	200 yr	PMP
0.08		3.4	5.2	6.3	7.8	8.8	9.9	11.2	33.4
0.17		4.4	6.5	7.9	9.7	11.0	12.3	13.9	41.4
0.25		5.1	7.5	9.1	11.0	12.5	13.9	15.8	46.9
0.5		6.5	9.5	11.4	13.7	15.5	17.3	19.5	58.2
1		8.4	11.9	14.2	17.1	19.3	21.4	24.2	72.1
2		10.8	15.1	17.8	21.4	23.9	26.5	30.0	89.4
6	0.25	16.0	21.8	25.5	30.3	33.8	37.3	42.2	125.6
12	0.5	20.5	27.5	32.0	37.7	42.0	46.2	52.3	155.7
24	1	26.3	34.6	40.1	47.0	52.2	57.3	64.8	193.0
48	2	32.4	41.5	47.7	55.6	61.6	67.5	80.8	238.0
72	3	37.5	47.6	54.6	63.5	70.2	76.8	92.0	269.0
96	4	41.5	52.5	60.0	69.7	77.0	84.2	100.9	293.4
120	5	45.0	56.6	64.7	75.0	82.7	90.5	108.3	313.9
144	6	48.0	60.3	68.7	79.6	87.7	95.9	114.8	331.7
168	7	50.7	63.5	72.3	83.7	92.2	100.7	120.6	347.5
192	8	53.2	66.5	75.6	87.4	96.2	105.1	125.8	361.8
216	9	55.4	69.2	78.6	90.8	100.0	109.1	130.7	374.9
240	10	57.6	71.7	81.5	94.0	103.4	112.9	135.1	387.0
360	15	66.5	82.3	93.2	107.3	117.9	128.5	153.8	437.4
480	20	73.7	90.8	102.6	117.8	129.3	140.8	168.6	477.1
600	25	79.8	97.9	110.5	126.7	139.0	151.2	181.1	510.4
720	30	85.2	104.2	117.4	134.5	147.4	160.3	191.9	539.3

Table 4.15 Rainfall + Snowmelt IDF

Duration		Total Depth of Rainfall + Snowmelt						
(hours)	(days)	2 yr	5 yr	10 yr	25 yr	50 yr	100 yr	200 yr
24	1	24.3	32.0	37.1	43.6	48.4	53.2	57.6
48	2	36.7	48.8	57.0	67.2	74.8	82.4	89.3
72	3	46.7	62.6	73.2	86.6	96.6	106.5	115.3
96	4	55.5	74.6	87.5	103.7	115.7	127.7	138.3
120	5	63.3	85.6	100.4	119.2	133.2	147.1	159.2
144	6	70.6	95.6	112.4	133.6	149.4	165.0	178.7
168	7	77.4	105.1	123.6	147.1	164.6	181.9	197.0
192	8	83.8	114.1	134.3	159.9	179.0	198.0	214.3
216	9	89.9	122.6	144.4	172.1	192.7	213.2	230.9
240	10	95.7	130.7	154.2	183.9	205.9	227.9	246.8
360	15	121.8	167.5	198.1	236.9	265.8	294.5	318.8
480	20	144.6	199.8	236.8	283.6	318.5	353.2	382.4
600	25	165.1	229.0	271.8	326.1	366.4	406.7	440.3
720	30	184.1	256.0	304.3	365.4	411.0	456.3	494.0

4.6 Hydrogeology

4.6.1 General Regime

Groundwater recharge in the TSF area is limited to precipitation infiltration at higher elevations. Generally, groundwater flow is restricted to fractured siltstone, sandstone and volcanic bedrock to depths of 20 m to 30 m and is generally confined by the overlying glacial tills which also limit groundwater recharge from precipitation.

As described in Section 4.2.4 above, glacial tills are present across the majority of the TSF area, thinning along the steep slopes. Where the glacial till is thin or absent, groundwater seepage is visible, with discharged groundwater flowing over the glacial till to lower lying areas, eventually exiting the site via creeks flowing to the south-southwest. Groundwater that is not discharged along the steeper slopes remains within the fractured zone of the bedrock and flows under the glacial till. Groundwater generally follows the topography, ultimately exiting to the south-southwest of the TSF area until discharging to surface in creeks at lower elevations that are not underlain by glacial tills. Groundwater that passes under creeks will discharge into Morrison Lake or continue south as part of the regional groundwater regime.

4.6.2 Hydraulic Conductivity

Hydraulic conductivity testing was conducted during drilling investigations by KCBL in 2007 and Knight Piesold Consulting (KPC) in 2006. Packer tests were completed in the glacial tills, the shallow fractured bedrock and the deeper, more competent bedrock. The results of the testing are described in detail in the KPC report entitled "Geotechnical Site Investigation Report (Ref. No. VA101-102/7-1), issued in July, 2006, and the KCBL report "2007 Geotechnical Site Investigation", issued in May, 2008. Additionally, KCBL reviewed and incorporated falling head test results conducted by Rescan in monitoring wells MW07-1 through MW07-8, installed in October and November, 2007. The results of all hydraulic conductivity testing are summarized by depth in Table 4.16.

For the fractured bedrock zone encountered at depths generally between 10 m and 40 m below ground surface, the test results indicate hydraulic conductivities ranging from $1.5x10^{-7}$ m/s to $5.1x10^{-6}$ m/s with a geometric mean of $8.7x10^{-7}$ m/s. The hydraulic conductivity decreased with depth: bedrock tested between approximately 40 m and 50 m has a range of $7.0x10^{-8}$ m/s to $1.8x10^{-6}$ m/s and a geometric mean of $3.1x10^{-7}$ m/s. Tests conducted at depths greater than 50 m below ground surface indicate conductivities ranging from less than $6.6x10^{-8}$ m/s to $2.4x10^{-7}$ m/s.

Hydraulic conductivity testing performed in the glacial tills indicated very low hydraulic conductivity. The tests were generally incomplete due to long recovery times. The hydraulic conductivity of the glacial tills is estimated to be on the order of $1x10^{-10}$ m/s.

Table 4.16 Summary of Hydraulic Conductivity Testing

Drill Hole	Test Interval (mbg) Top	Bottom	Hydraulic Conductivity K (m/s)	Geologic Unit
DH06-2	9.1	39.5	5.1E-07	Volcanic
DH06-3	6.7	36.9	3.3E-07	Volcanic
DH06-4	11.0	41.5	7.4E-07	Volcanic
DH06-6	9.6	36.7	1.4E-06	Volcanic
DH06-7	12.8	43.3	5.1E-06	Volcanic/Siltstone/Sandstone
DH06-11	8.8	36.9	7.2E-06	Siltstone
DH07-1A	23.35	35.66	6.0E-07	Sandstone/mudstone
DH07-2A	26.20	35.10	2.4E-07	Siltstone
DH07-3A	24.23	35.51	1.6E-06	Sandstone
DH07-4A	27.89	36.12	1.5E-07	Sandstone/siltstone
DH07-5B	26.37	42.98	9.2E-07	Meta-sedimentary
MW07-01A	16.76	28.04	1.2E-06	Volcanic/Siltstone
MW07-02A	4.57	21.94	6.8E-07	Limestone
	Geometric Mean		8.7E-07	
DH07-1A	35.51	49.38	7.0E-08	Sandstone/Mudstone
DH07-3A	35.51	41.61	1.8E-06	Siltstone
DH07-4A	36.12	46.18	2.3E-07	Siltstone/Sandstone
	Geometric Mean		3.1E-07	
DH06-1	59.4	89.9	2.4E-07	Volcanic
DH07-5B	45.26	58.22	6.6E-08	Siltstone/Sandstone Cong.
	Geometric Mean		1.3E-07	

4.6.3 Groundwater Flow Direction

Groundwater elevations are only available in the immediate vicinities of the north and main dams, with one additional groundwater elevation measurement from monitoring well MW07-2 installed to the east of the TSF area. However, the available data supports the assumption that groundwater elevation gradients generally follow the surficial topography.

Based on the surficial topography and the available groundwater elevation measurements, groundwater flow in the TSF area flows from the higher surrounding elevations toward the centre of the TSF and then in a south to south-westerly direction under the proposed Main Dam alignment. A groundwater divide is evident in the vicinity of the North Dam where groundwater flows to the north-north-east. Groundwater elevation data is not available in the vicinity of the West Dam; however, it is likely that an east-west groundwater flow divide exists in this area.

The total groundwater flux through the fractured bedrock underlying the Main Dam alignment can be estimated assuming an average fracture zone thickness of 20 m, and average hydraulic conductivity of $8x10^{-7}$ m/s and a dam length of 1200 m. The assumptions provide an estimate of less than 1.0 L/s flowing through the fractured bedrock below the Main Dam.

4.6.4 Groundwater Quality

Water quality results from the 2007 groundwater monitoring well installations were provided by Rescan and included in Appendix VI.

5. GEOTECHNICAL CHARACTERIZATION

5.1 Foundation Soils

5.1.1 Tailings Storage Facility

The geotechnical characteristics of the glacial till in the TSF area are derived from data collected during the 2006 Knight Piesold studies (Appendix II) and the KCBL 2007 and 2008 site investigations (Appendices III and IV, respectively). Table 5.1 summarizes the available in-situ data collected from geotechnical boreholes. In general, the glacial till is clayey sand to sandy clay, with some fine gravel, with low to intermediate plasticity.

Table 5.1 Glacial Till Properties from Borehole Logs

Location	Borehole No.	Consistency[1]	Pocket Penetrometer – Su (kPa)	SPT $(N_1)_{60}$ or (*N*) (blows/0.3m)
North Dam	DH07-1A	Stiff to Very Stiff	60 – 100 (avg. 70)	13 – 57 (med. 27)
	DH07-2A	Stiff to Very Stiff	60 – 270 (avg. 80)	12 – 62 (med. 20)
	DH06-12(KP)	Stiff to Very Stiff	n/a	19 – 43 (med. 26)
	DH06-11(KP)	Very Stiff to Hard	n/a	38
	DH06-10(KP)	Stiff to Very Stiff	n/a	14 – 37 (med. 19)
Main Dam	DH07-3A	Very Stiff to Hard	100 – 270 (avg. 150)	22 – 58 (med. 32)
	DH07-4A	Very Stiff to Hard	60 – 170 (avg. 100)	22 – 62 (med. 28)
	DH07-5A	Very Stiff to Hard	~150 kPa	22 – 82 (med. 53)
	DH06-7(KP)	Stiff to Very Stiff	n/a	13 – 24 (med. 19)
	DH06-3(KP)	Firm to hard	n/a	6 – 50 (med. 27)
	DH06-2(KP)	Soft to very stiff	n/a	2 – 21 (med. 5)
	DH06-1(KP)	n/a	n/a	n/a

Note: 1: The consistency presented in Table 4-1 is based upon Pocket Penetrometer – Su (kPa) and SPT N values. Field observation of the disturbed samples indicated that the foundation till is generally firm to very stiff.

Table 5.2 Summary of Glacial Till Index Properties

Location	Moisture Content (w) (%)	Fines Content (%<74 µm)	Liquid Limit (LL) (%)	Plastic Limit (PL) (%)	Plasticity Index (PI) (%)
Main Dam	7 to 24 (mean =13)	33 to 80 (mean=46)	22 to 36 (mean=32)	13 to 19 (mean=16)	3 to 23 (mean=15)
North Dam	10 to 26 (mean=13)	38 to 66 (mean=47)	27 to 45 (mean=34)	13 to 17 (mean=15)	12 to 28 (mean=18)

Note: Numbers in brackets are the average for the set of samples.

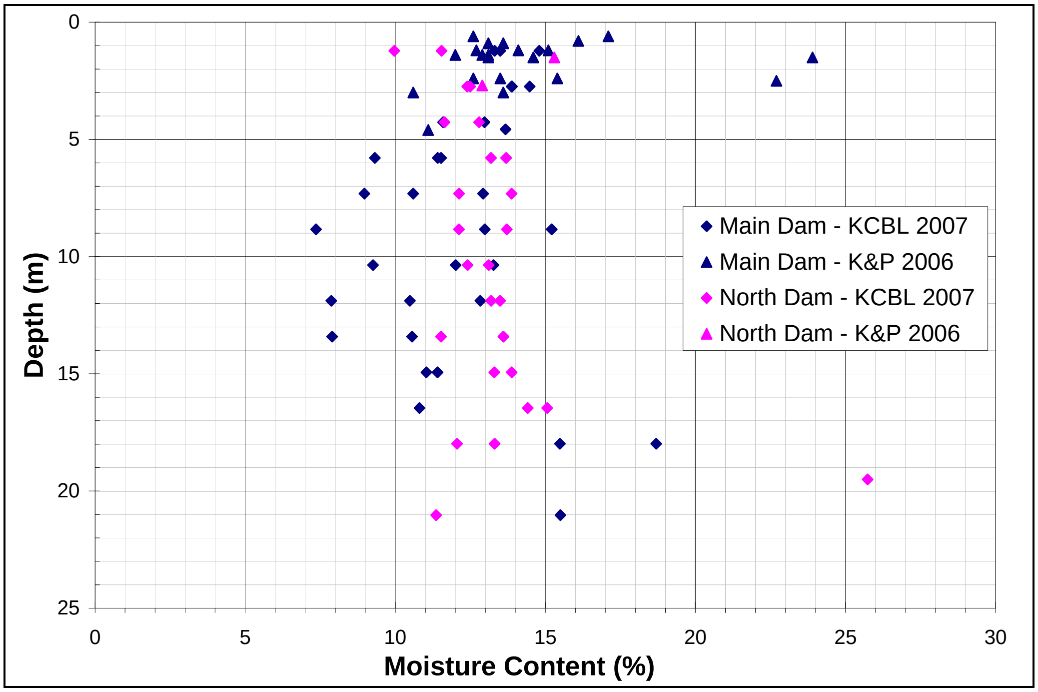


Figure 5.1 Moisture Content Distribution of Glacial Till

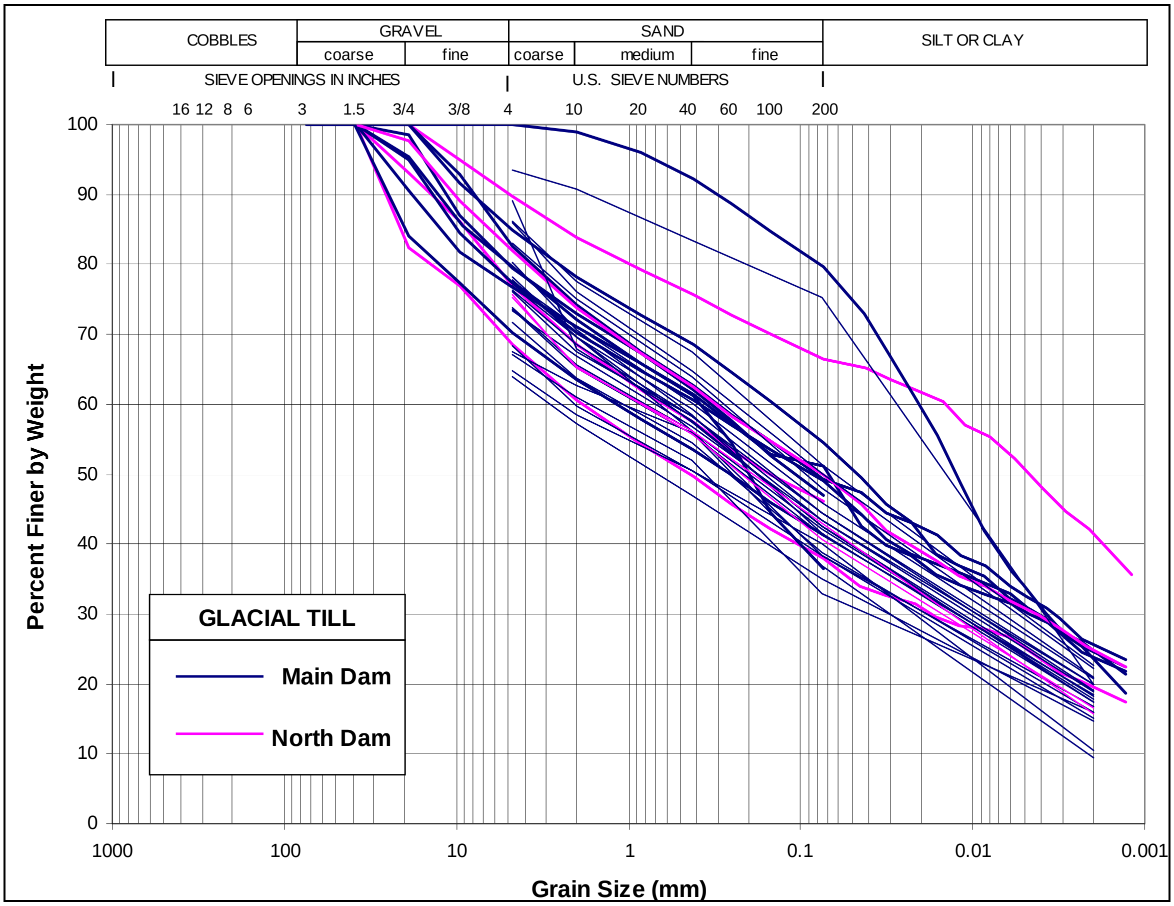


Figure 5.2 Grain Size Distribution of Glacial Till

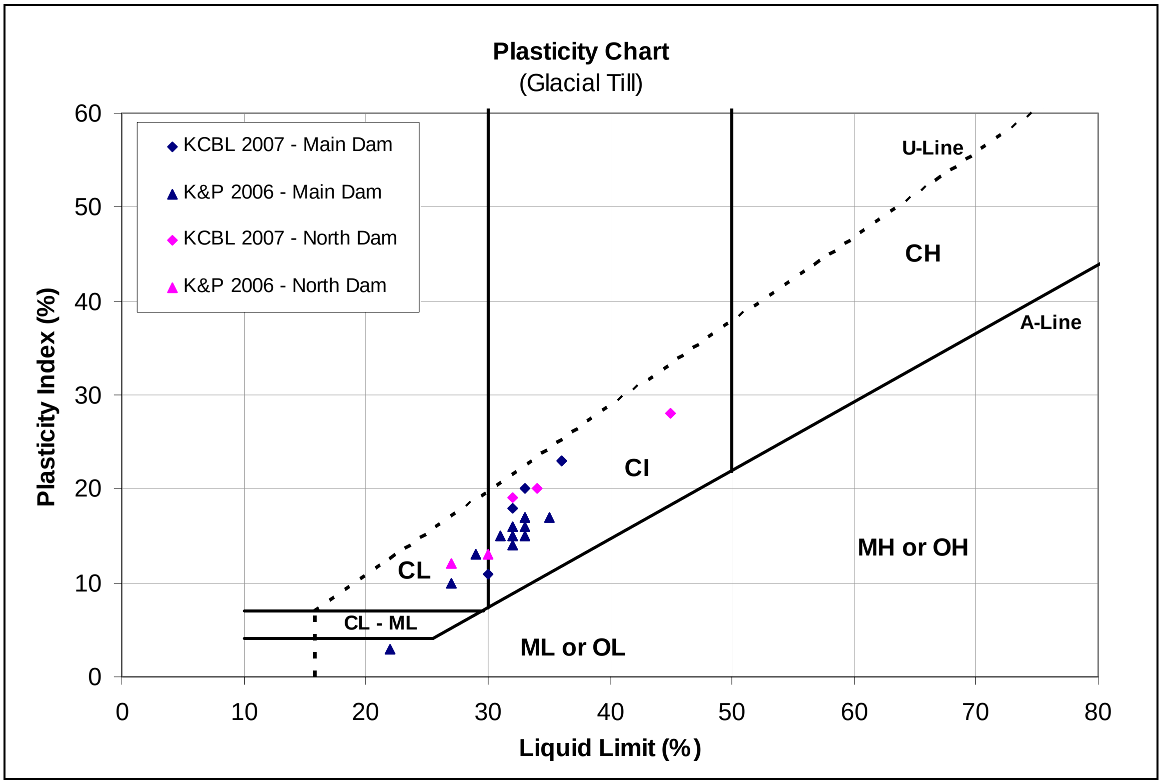


Figure 5.3 Atterberg Limits of Glacial Till

5.1.2 Waste Rock Dump

The geotechnical characteristics of the glacial till in the waste rock dump area are derived from data collected during the 2008 geotechnical site investigation program. Table 5.3 summarizes the available *in-situ* data collected from geotechnical boreholes. Table 5.4 and Figure 5.4 summarize the index properties of the glacial till. In general, the glacial till is clayey sand to sandy clay, with some fine gravel, with low to intermediate plasticity.

Table 5.3 SPT Values for Glacial Till – Waste Rock Dump

Bore Hole	Number of Tests	N_{60}[1]				$(N_1)_{60}$			
	SPT or LPT	min	max	mean	median	min	max	mean	median
DH08-1A	9	31	96	32	50	30	119	65	56
DH08-2	5	43	157	82	68	48	172	98	89
MW08-1	14	36	158	57	46	17	60	34	30
MW08-3	4	7	48	30	30	14	45	31	32

NOTES:
1. N_{60}: N_{SPT} normalized to 60% hammer efficiency

Table 5.4 2008 Laboratory Index Test Results – Waste Rock Dump

Test Hole Location	Sample no.	Depth (m)	W (%)	W_L (%)	W_P (%)	I_P (%)	% Gravel	% Sand	% Fines	Material Type
DH08-1	SPT 4	5.7	11	32	14	18	13.1	33.3	53.6	Glacial till
DH08-1	SPT 9	13.4	14	31	14	17	10.9	31.1	58	Glacial till
DH08-2	SPT- 4	4	11	29	13	16	29.8	30.9	39.3	Glacial till
MW08-1	LPT-11	31.7	13	26	13	13	18.4	39.4	42.2	Glacial till
MW08-1	SHLBY-1	24.1		28	13	15	12.3	42.3	45.4	Glacial till
MW08-3	LPT-2	5.8	22	25	14	11	4.6	28.5	66.9	Glacial till
TP08-A		2	10	24	16	8	24	39.5	36.5	Glacial till
TP08-E		2.6	12	29	15	14	19.3	34.9	45.8	Glacial till
TP08-F		2	16	29	14	15	10	36.9	53.1	Glacial till
TP08-J		1	12	27	20	7	15.6	39.1	45.3	Glacial till
TP08-J		2	14	40	21	19	14.5	37.5	48	Glacial till
TP08-K		2	13	22	15	7	26.2	49.7	24.1	Gravelly Sand
TP08-L		1	9	30	16	14	24.6	30.8	44.6	Glacial till

W – Natural water content
W_L – Liquid Limit
W_P – Plastic Limit
I_P – Plasticity Index

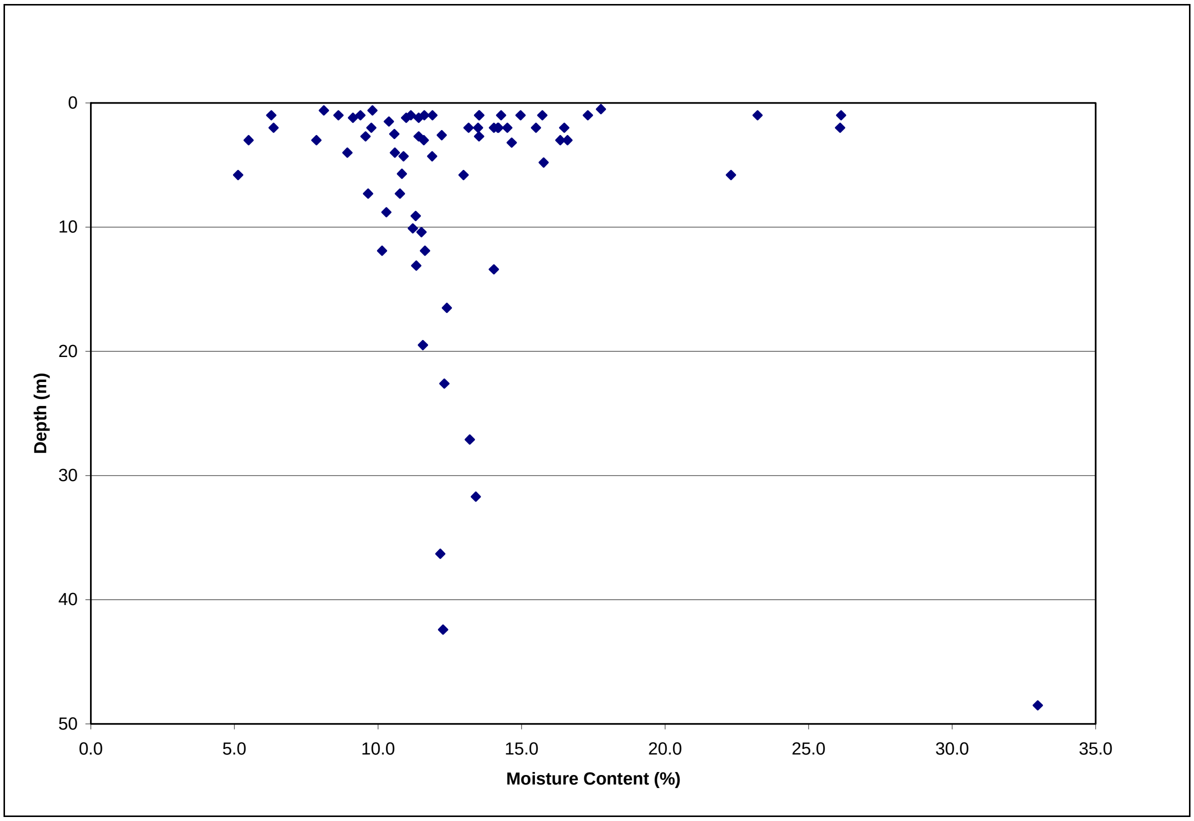


Figure 5.4 Moisture Content Against Depth for the Waste Rock Dump

5.2 Granular Borrow

Sand and Gravel Borrow Area #1

A potential sand and gravel borrow area is located approximately 650 m southeast of the Open Pit (see Drawing D-1002). The aerial extent of granular deposit is estimated to be 85,000 m^2 based on interpretation of the local topography.

The depth of sand and gravel was determined from site investigation data (Knight Piésold, 2006). Test pits TP06-41, TP06-42, TP06-43, and TP06-44 show that the sand and gravel deposit extends to a minimum depth of 3 m. Test pit stratigraphy is shown in the attached logs and test pit locations are shown on Drawing D-1003.

The borrow area is estimated to contain 250,000 m^3 of sand and gravel, based on an estimated average depth of 3 m.

Knight Piésold lab tests indicate that the borrow area material has the following grain size distribution.

Table 5.5 Sand & Gravel Borrow Area #1 Sample Grain Size Distributions

Hole	Sample id*	Depth (m)	% Gravel	% Sand	% Fines
TP06-41	TP06-41	0.8	6.4	69.8	23.8
TP06-41	TP06-41	2.4	73.6	24.6	1.8
TP06-42	TP06-42	0.9	67.3	28.4	4.3
TP06-42	TP06-42	2.7	58.6	35.7	5.7
TP06-43	TP06-43	1.2	40.2	41.7	18.2
TP06-43	TP06-43	2.4	61.8	21.4	16.8
TP06-44	TP06-40	0.9	33.3	63.4	3.3
TP06-44	TP06-40	2.7	28.5	61.7	9.7

*Note: Test Pit TP06-44 does correspond to Sample ID TP06-40.

Sand and Gravel Borrow Area #2

A potential sand and gravel borrow area is located roughly 1500 m northwest of the Open Pit as shown in Drawing D-1003. Surface area of the borrow site is estimated to be 180,000 m^2 based on interpretation of the local topography.

The depth of sand and gravel was determined from site investigation data (KCBL, 2007) Test pits TP07-02, TP07-03, and TP07-04 show that the sand and gravel deposit extends to a minimum depth of 6 m. Test pit stratigraphy is shown in the attached logs and test pit locations are shown on Drawing D-1003. Table 5.6 and Figure 5.5 summarize the grain size distribution.

Table 5.6 Sand & Gravel Borrow Area #2 Sample Grain Size Distributions

Hole	Depth (m)	%Gravel	%Sand	%Fines
TP07-2	3	46.8	46.0	7.2
TP07-2	5.8	22.2	72.7	5.1
TP07-3	3	60.9	36.4	2.7
TP07-4	3	54.0	43.3	2.7

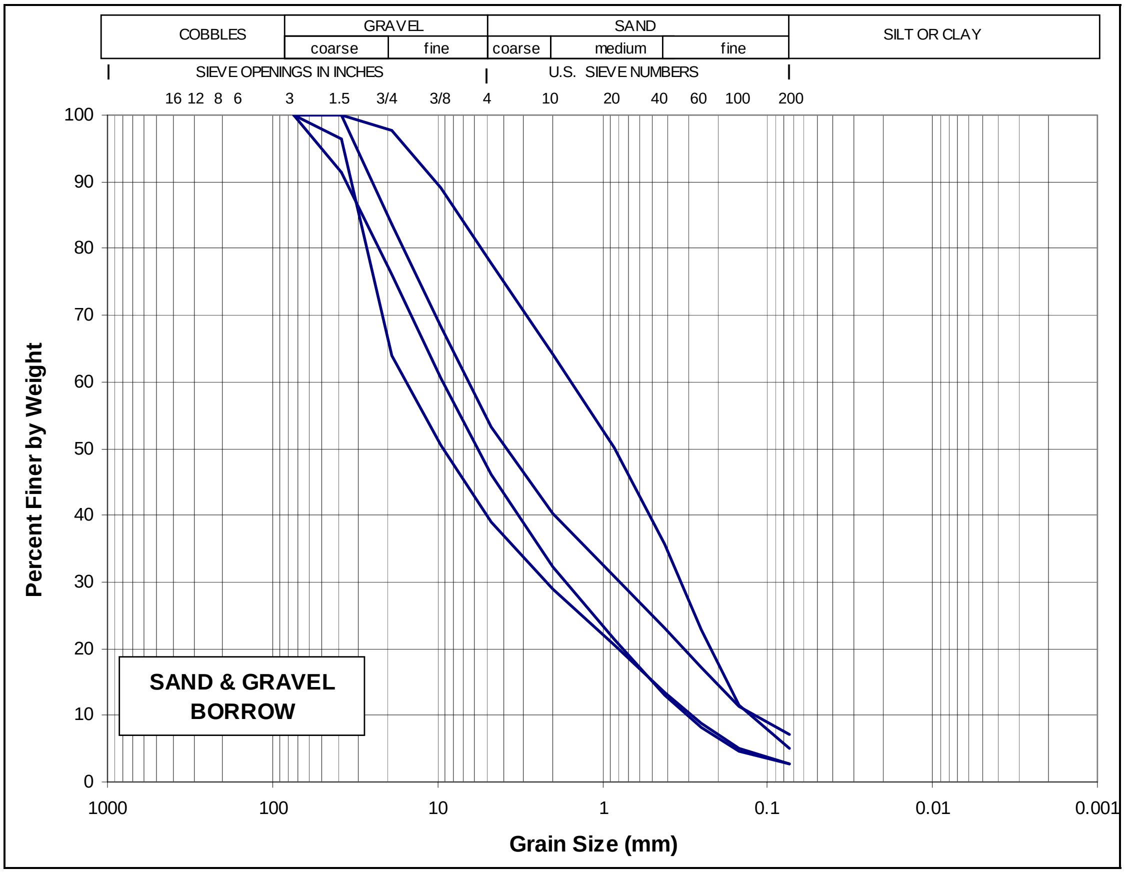


Figure 5.5 Sand & Gravel Borrow Area #2 Sample Grain Size Distribution Curves

The borrow area is estimated to contain 1 M m^3 of sand and gravel based on an estimated average depth of 6 m.

5.3 Tailings Characterization

5.3.1 Geotechnical

The Morrison tailings is a relatively coarse grind in comparison with other copper tailings in British Columbia. KCBL received two portions of the total tailings from SGS Lakefield Research Ltd. (SGS): a coarse fraction (retained on a 270 micron mesh) and a fine fraction (passing a 270 micron mesh). The total tailings (combined at a ratio of 65% coarse fraction and 35% fine fraction) have approximately 53% fines. Gradation analyses of the samples are shown on Figure 5.6. Laboratory tests were carried out on the materials and the results are summarized in Table 5.7.

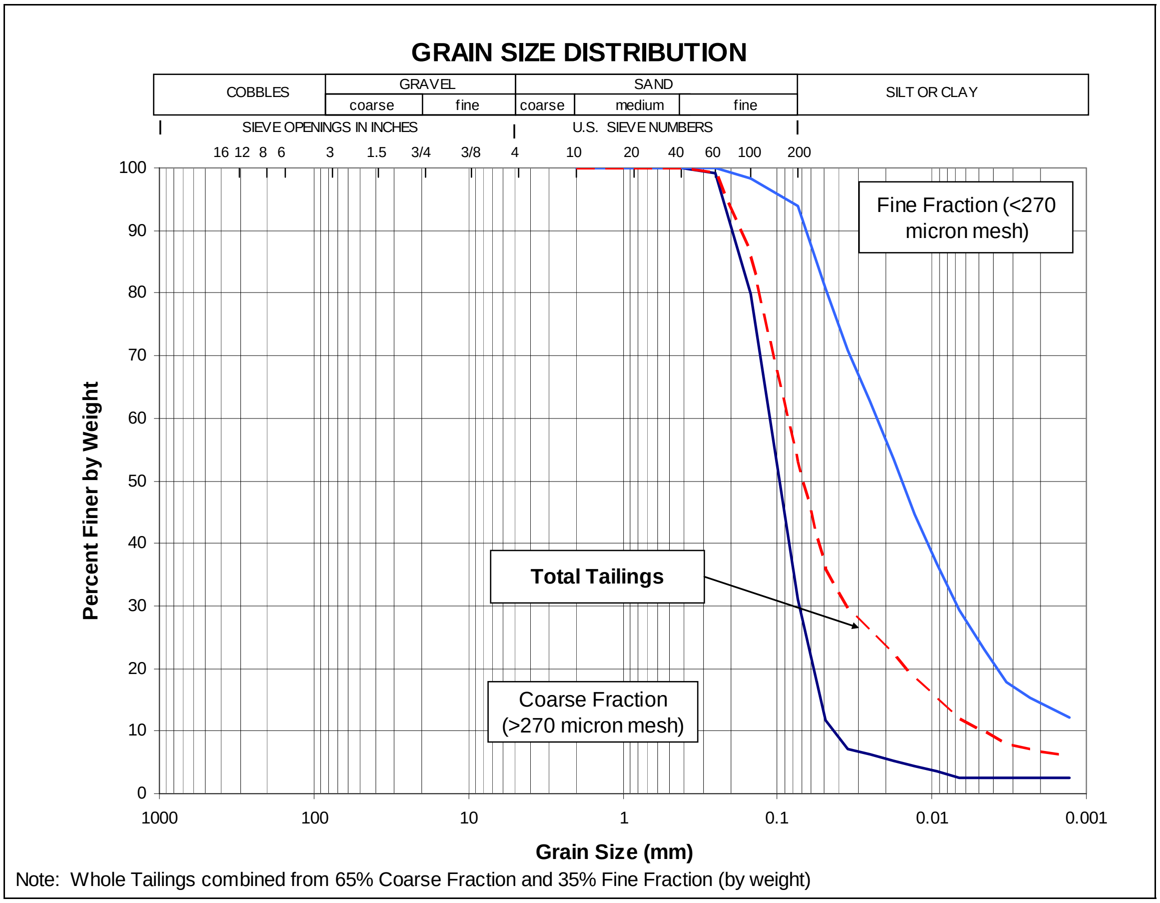


Figure 5.6 Grain Size Distribution Curve of Morrison Tailings

Table 5.7 Summary of Geotechnical Testing of Tailings

Parameter	Coarse Fraction	Fine Fraction	Total Tailings	Cycloned Sand (18% fines)
Specific Gravity	2.73	2.81	2.76	
Maximum Standard Proctor Dry Density and Optimum Moisture Content				1530 kg/m^3 18.0 %
% Fines (< 75 micron)	31.1	94.1	53	18.0
Jar settling tests – 5 days starting @ 33% solids by weight. Measured %solids by weight.	-	61%	66%	-
Consolidation test at low stress <10 kPa -final density	-	-	1.5 t/m^3	-
Consolidation test (@700 kpa)				
Final void ratio	-	-	0.644	-
Final density	-	-	1.67 t/m^3	-
Permeability (m/s)	-	-	2.8×10^{-6}	-
C_v (m^2/year)	-	-	36	-
M_v (cm^2/N)	-	-	8.0×10^{-4}	-
C_c	-	-	0.085	-

The tailings are coarse, thus settled density in the impoundment is expected to be higher than normal copper tailings. Tailings densities of: 1.4 t/m^3 for the starter dam; and 1.5 t/m^3 for the final dam, have been selected. Actual values will be measured during operations and the impoundment versus volume curve will be adjusted accordingly.

At the downstream shell, compacted cycloned sand with coarse angular particles, is assumed to have a friction angle of 35 degrees. The field specification for the sand will be a maximum of 20% fines (< 75 micron size).

6. GEOCHEMICAL CHARACTERIZATION AND WATER QUALITY

6.1 Tailings

6.1.1 General

Locked-cycle flotation tests (LCT) were carried out during 2005 to 2007 and the test program is summarized in Table 6.1.

Table 6.1 Summary of Tailings Locked Cycle Samples

Year	Number of Samples	Description
2005	10	Series of rougher-scavenger and cleaner-scavenger tailings during metallurgical test work for Cycles 3+4+5 in Tests F46, F47, F48, F51 and F52.
2006	10	Series of rougher-scavenger and cleaner-scavenger tailings during metallurgical test work for Tests F53, C1 to C5. Static ABA tests were carried out on the ten samples.
2006	2	Composites of 90% rougher-scavenger and 10% cleaner-scavenger tailings from Tests F46 and F47 for humidity cell testing.
2007	2	Samples were prepared from locked-cycle testing of samples generated during the metallurgical test program at SGS representing the finer fraction (-270 mesh) and coarser fraction (+270 mesh).

The results of the geochemical testing of the tailings produced with the 2005 and 2006 locked cycle programs are included Appendix VI (“Morrison Project – Prediction of Metal Leaching and Acid Rock Drainage, Phase 1” by Minesite Drainage Assessment Group, March 22, 2007).

Geochemical characterization of the tailings produced with the 2007 locked cycle program is presented in the Interim Report “Environmental Testing of Morrison Tailings Solids and Supernatant” by SGS Lakefield Research Ltd., dated October 4, 2007. This interim report is included in Appendix VI. The results of ongoing humidity cell tests of the coarse and finer tailings are also included in Appendix VI.

The 2007 metallurgical testing program was designed to increase the efficiency of the sulphide removal in the mill to reduce the ARD potential of the tailings. As such, the 2007 test results are considered to be the representative samples for geochemical and physical characterization of the tailings.

6.1.2 Summary of 2006 Testing

The 2006 program of geochemical testing was carried out on five sets of rougher tailings and five sets of cleaner tailings and two composite samples. The approximate weighted average is 90% rougher tailings to 10% cleaner tailings. Static testing was carried out on all samples. Table 6.2 summarizes the static testing performed on 2006 tailings samples.

Table 6.2 Summary of 2006 Tailings Static Geochemistry

Sample	Paste pH	% S	Available NP (t $CaCO_3$/1000t)	Adjusted SNPR
F46 – Rougher/Scavenger Composite	8.2	0.19	35	8.0
F47 – Rougher/Scavenger Composite	7.7	1.07	73	2.33
F46 Bulk Rougher Tails Cycles 3+4+5	7.6	0.06	27.1	14.4
F47 Bulk Rougher Tails Cycles 3+4+5	8.0	0.13	47.4	11.7
F48 Bulk Rougher Tails Cycles 3+4+5	8.0	0.28	12.2	1.4
F51 Bulk Rougher Tails Cycles 3+4+5	8.2	0.10	31.2	10
F52 Bulk Rougher Tails Cycles 3+4+5	7.5	0.05	30.4	19
F46 Bulk Cleaner Tails Cycles 3+4+5	8.2	0.84	43.3	1.65
F47 Bulk Cleaner Tails Cycles 3+4+5	7.6	8.29	41.0	0.16
F48 Bulk Cleaner Tails Cycles 3+4+5	7.9	1.46	25.2	0.55
F51 Bulk Cleaner Tails Cycles 3+4+5	7.7	5.07	43.1	0.27
F52 Bulk Cleaner Tails Cycles 3+4+5	7.7	2.87	43.2	0.48

NOTE: NP - Neutralization Potential; SNPR – Sulphide Net Potential Ratio

ABA testing was carried out on a total tailings sample and a low sulphide tailings produced with an additional sulphide flotation circuit and the results are presented in Table 6.2 and Table 6.3, respectively.

Table 6.3 Modified Acid Base Accounting of Low Sulphide and Total Tailings

Sample ID		**Test F21 Low 'S' Tails -**			**Test F17 Combined Total Tailings**	
Paste pH	Units	9.05	Pyrite tails	combined	8.75	Rougher
Fizz Rate	---	2			2	
Sample	weight(g)	2.03			1.96	
HCl added	mL	32.8			30.5	
HCl	Normality	0.1			0.1	
NaOH	Normality	0.1			0.1	
NaOH to	pH=8.3 mL	13.45			10.9	
Final pH	units	1.74			1.96	
NP	t CaCO3/1000t	47.7			50	
AP	t CaCO3/1000 t	1.25			7.5	
Net NP	t CaCO3/1000 t	46.4			42.5	
NP/AP	**ratio**	38.1			6.7	
S	%	0.132	0.62	0.32	0.26	0.16
Sulphide	%	0.04			0.24	
SO4	%	< 0.4			< 0.4	
C	%	0.832			0.907	
Carbonate	%	2.25			1.54	

Summary

Testing indicates the majority of the rougher tailings will be NAG – only one test indicates a SNPR of less than 2.0 (range of 1.4 to 19). All of the cleaner tailings are PAG. The two composite samples and the weighted averages of rougher/cleaner indicate that the total tailings are predicted to be NAG.

An assessment of the potential sulphide content in the cycloned sand was made with the following two methodologies:

- A total %S of the combined tailings was calculated on the basis of the ratio of 10% cleaner tailings % sulphide and 90% rougher tailings %S, which equates to a weighted average of 0.94%S for the combined sample. In the worst case, this %S would be assumed to be weighted equally throughout the particle size range and the cycloned sand, therefore, could have 0.94%S.

- If the sulphide distribution is assumed to be more associated with the fines, which the testing indicates, then a reduction in fines content through the cycloning process could reduce the %S from 0.94%S to 0.3%S. This %S would be the lowest possible and the actual is estimated to be in the order of 0.5%, which is still considered low.

The conclusion of the assessment is that the cycloned tailings sand is predicted to have a low potential for acid rock drainage. In addition, a sulphide flotation circuit could also be added to further reduce the sulphide content, if required.

6.1.3 2007 Tailings Geochemical Testing

One of the objectives of the 2007 tailings test program was to confirm the geochemical properties of the cycloned sand for use in construction of the dam. Cyclone sand will be produced by cycloning the total tailings stream to reduce the fines content from approximately 55% to 18%.

The 2007 tailings test program assessed the potential benefit of a sulphide floatation circuit to reduce the sulphide content of the coarse tailings, which will be used for construction of the cycloned sand dam. The combined rougher and cleaner tailings, samples F25 and F26, was screened on a 270 mesh sieve and a pyrite float was carried out on the coarse fraction. The pyrite was then combined with the fine tailings. Geochemical testing of the coarse and fine tailings (sample F28) is included in Appendix VI. Results of the acid base accounting (ABA) tests are presented in Table 6.4.

The ABA tests indicate the coarse tailings are potentially acid consuming, and the higher sulphide fine tailings have a very low potential for acid rock drainage.

Table 6.4 Modified Acid Base Accounting of Coarse and Fine Tailings (F28)

Parameter	Units	Coarse Fraction	Fine Fraction
% < 75 micron sieve size	%	25	95
Paste pH	Units	9.31	8.76
Fizz Rate	---	2	1
Sample	weight(g)	2.0	1.0
HCl added	mL	40.50	47.60
HCl	Normality	0.1	0.1
NaOH	Normality	0.1	0.1
NaOH to	pH=8.3 mL	20.10	23.45
Final pH	Units	1.57	1.72
NP	t CaCO3/1000t	50.5	61.0
AP	t CaCO3/1000 t	3.4	14.4
Net NP	t CaCO3/1000 t	47.1	46.6
NP/AP	**Ratio**	14.9	4.2
S	%	0.094	0.542
Sulphide	%	0.11	0.46
SO4	%	< 0.4	< 0.4
C	%	0.744	0.955
Carbonate	%	2.19	2.40

Tailing Humidity Cells Testing

Humidity cell testing of the coarse and fine tailings have been in progress for at least 48 weeks and the results are included in Appendix VI. The coarse and fine tailings still remain neutral with a pH of 7.65 and 6.77, respectively. The leachate water quality of the humidity cell tests for the coarse and fine tailings are summarized in Table 6.5 and Table 6.6 respectively.

Table 6.5 Humidity Cell Leachate Water Quality - Coarse Fraction

Parameter	Units	BCWQ 30 day aquatic guideline	Week 0	Week 48
LIMS			10434-JUN	10345-MAY08
Hum Cell Leachate Vol	mLs		707	969
pH	unitless	9	7.68	7.65
Conductivity	uS/cm		308	110
Alkalinity	mg/L as $CaCO_3$		51	42
Acidity	mg/L as $CaCO_3$		< 2	< 2
SO_4	mg/L	100 (Dissolved)	58	5.2
Hg	mg/L		< 0.0001	< 0.0001
Ag	mg/L		< 0.00003	0.00003
Al	mg/L	0.05	0.0347	0.0207
As	mg/L	0.005	0.0020	0.0003
Ba	mg/L		0.0767	0.287
Be	mg/L		< 0.00004	< 0.00002
B	mg/L		< 0.002	< 0.002
Bi	mg/L		< 0.00002	0.00002
Ca	mg/L		24.3	11.8
Co	mg/L	0.004	0.00242	0.000062
Cd	mg/L	0.00006 $_{(3)}$	< 0.00006	0.000006
Cr	mg/L	0.009 {Cr(III)}	< 0.0003	< 0.0005
Cu	mg/L	0.008 $_{(3)}$	0.0022	0.001
Fe	mg/L	0.3	< 0.01	< 0.01
Li	mg/L		< 0.002	< 0.002
K	mg/L		11.1	2.6
Mg	mg/L		6.63	3.95
Mn	mg/L	1.5 $_{(3)}$	0.0712	0.0195
Mo	mg/L	1	0.0209	0.00267
Na	mg/L		14.7	0.14
Ni	mg/L	0.15 $_{(3)}$	0.0047	0.0003
Pb	mg/L	0.20 $_{(3)}$	0.00013	< 0.00002
Sb	mg/L	0.02	0.0038	0.00043
Se	mg/L	0.002	0.001	< 0.001
Sn	mg/L		0.0010	0.00003
Si	mg/L		1.35	0.37
Ti	mg/L		< 0.0002	< 0.0001
V	mg/L		0.00022	0.00021
Zn	mg/L	0.09 $_{(3)}$	0.0026	< 0.001

Table 6.6 Humidity Cell Leachate Water Quality - Fine Fraction

Parameter	Units	BCWQ 30 day Aquatic Guideline	Week 0	Week 48
LIMS			10434-JUN	10345-MAY08
Hum Cell Leachate Vol	mLs		845	972
pH	unitless	9	6.31	6.77
Conductivity	uS/cm		69	87
Alkalinity	mg/L as $CaCO_3$		4	4
Acidity	mg/L as $CaCO_3$		8	< 2
SO_4	mg/L	100 (Dissolved)	4.3	21
Hg	mg/L		< 0.0001	< 0.0001
Ag	mg/L		< 0.00003	0.00005
Al	mg/L	0.05	0.0027	0.0023
As	mg/L	0.005	0.0005	0.0002
Ba	mg/L		0.116	0.00744
Be	mg/L		< 0.00004	< 0.00002
B	mg/L		< 0.002	< 0.002
Bi	mg/L		< 0.00002	< 0.00001
Ca	mg/L		3.98	5.73
Co	mg/L	0.004	0.00279	0.000086
Cd	mg/L	0.00006 $_{(3)}$	< 0.00006	0.000007
Cr	mg/L	0.009 {Cr(III)}	0.0005	< 0.0005
Cu	mg/L	0.008 $_{(3)}$	0.0005	< 0.0005
Fe	mg/L	0.3	0.29	< 0.01
Li	mg/L		< 0.002	< 0.002
K	mg/L		1.69	1.61
Mg	mg/L		0.926	3.81
Mn	mg/L	1.5 $_{(3)}$	0.0268	0.0162
Mo	mg/L	1	0.00412	0.00257
Na	mg/L		1.56	0.54
Ni	mg/L	0.15 $_{(3)}$	0.0025	0.0004
Pb	mg/L	0.20 $_{(3)}$	0.00004	0.00002
Sb	mg/L	0.02	0.0007	0.00014
Se	mg/L	0.002	< 0.001	< 0.001
Sn	mg/L		0.0018	< 0.00001
Si	mg/L		0.13	0.14
Ti	mg/L		< 0.0002	0.0001
V	mg/L		0.00035	0.00008
Zn	mg/L	0.09 $_{(3)}$	0.0091	0.001

6.1.4 Summary of Tailings Geochemistry

The main conclusions of the tailings geochemical testing are:

- The total tailings contain relatively low sulphide concentrations up to 1.0%S. The sulphides are primarily associated with the finer cleaner scavenger tailings, which comprise 10% of the total tailings and can contain up to 8.3%S. The cleaner tailings have a typical gradation of approximately 95% silt sizes and 5% sand sizes. The rougher tailings can contain 0.05%S to 0.28%S and have typical gradation of 25% silt size and 75% sand size. The rougher and cleaner tailings will be mixed together and pumped to the TSF.

- The cyclone sand will contain a small fraction of the cleaner tailings and are predicted to be NAG and to have geochemical properties similar to the rougher scavenger tailings.

- Testing of a floatation circuit that would remove pyrite from the coarse fraction (+ #270 sieve size) produced coarse tailings with a high NP/AP ratio. In addition it appears that even the higher sulphide fine tailings have a high NP/AP ratio, thus neither product are predicted to be PAG. The testing does suggest, however, that a sulphide floatation circuit may have some benefit in the future if actual %S concentrations were to increase beyond those predicted in the mining assessment.

6.2 Tailings Supernatant Water

Tailings will be pumped from the mill to the TSF at 33% solids by weight and excess reclaim water will be recycled back to the process plant. The water quality of the tailings impoundment will be partially diluted by precipitation and runoff of fresh water derived from the upland catchment areas.

Aging tests of the tailings supernatant have been completed by SGS and the results for the coarse and fine fractions are presented in Table 6.7 and Table 6.8 respectively. The whole tailings consist of a combination of 65% coarse fraction and 35% fine fraction.

Table 6.7 Fresh and Aged Tailings Supernatant Analyses - Coarse Fraction

Parameter	Units	BCWQ 30 day Aquatic Guideline	Day 0	Aged Day 61
LIMS			10151-Jun07	10157-AUG07
pH	units	9	8.39	8.33
Acidity	mg/L as $CaCO_3$		< 2	< 2
Alkalinity	mg/L as $CaCO_3$		89	63
Conductivity	µS/cm		464	528
TDS	mg/L		317	585
TSS	mg/L		197	< 2
EMF	mV		168	9
F	mg/L		0.35	0.37
Cl	mg/L		38	38
SO_4	mg/L	100 (Dissolved)	75	98
NO_2	as N mg/L		< 0.06	< 0.06
NO_3	as N mg/L		1.11	< 0.05
NO_2+NO_3	as N mg/L		1.11	< 0.06
NH_3+NH_4	as N mg/L		0.1	< 0.1
Thiosalts	as S_2O_3 mg/L		< 10	< 10
Metals			**Dissolved**	**Dissolved**
Hg	mg/L		< 0.0001	< 0.0001
Ag	mg/L		< 0.00003	< 0.00003
Al	mg/L	0.05	0.04	0.05
As	mg/L	0.005	0.0019	0.0027
Ba	mg/L		0.165	0.101
Be	mg/L		< 0.00004	< 0.00004
B	mg/L		0.03	0.035
Bi	mg/L		< 0.00002	0.00009
Ca	mg/L		40.1	36.3
Cd	mg/L	0.00006	0.00044	< 0.00006
Co	mg/L	0.004	0.000469	0.000178
Cr	mg/L	0.009 {Cr(III)}	0.0005	0.0009
Cu	mg/L	0.008 $_{(3)}$	0.0023	0.0058
Fe	mg/L	0.3	0.01	< 0.01
K	mg/L		12.8	17.8
Li	mg/L		< 0.002	< 0.002
Mg	mg/L		9.56	12.7
Mn	mg/L	1.5 $_{(3)}$	0.0435	0.004
Mo	mg/L	1	0.0362	0.063
Na	mg/L		35.4	33.8
Ni	mg/L	0.15 $_{(3)}$	0.006	0.0023
Pb	mg/L	0.20 $_{(3)}$	0.00006	0.00014
Sb	mg/L	0.02	0.0049	0.0051
Se	mg/L	0.002	< 0.001	0.001

Table 6.7 Fresh and Aged Tailings Supernatant Analyses - Coarse Fraction (cont'd)

Parameter	Units	BCWQ 30 day Aquatic Guideline	Day 0	Aged Day 61
Si	mg/L		2.02	2.68
Sn	mg/L		0.0046	< 0.0003
Ti	mg/L		0.0009	0.0011
V	mg/L		0.00028	0.00099
Zn	mg/L	0.09 $_{(3)}$	0.0034	0.0032

Table 6.8 Fresh and Aged Tailings Supernatant Analyses - Fine Fraction

Parameters	Units	BCWQ 30 day Aquatic Guideline	Day 0	Aged Day 61
LIMS			10151-Jun07	10157-AUG07
pH	units	9	8.34	8.3
Acidity	mg/L as $CaCO_3$		<2	< 2
Alkalinity	mg/L as $CaCO_3$		78	66
Conductivity	µS/cm		353	433
TDS	mg/L		234	254
TSS	mg/L		74	< 2
EMF	mV		170	-1
F	mg/L		0.37	0.51
Cl	mg/L		28	26
SO_4	mg/L	100 (Dissolved)	53	64
NO_2	as N mg/L		< 0.06	< 0.06
NO_3	as N mg/L		< 0.05	< 0.05
NO_2+NO_3	as N mg/L		< 0.06	< 0.06
NH_3+NH_4	as N mg/L		< 0.1	< 0.1
Thiosalts	as S_2O_3 mg/L		< 10	< 10
Metals			**Dissolved**	**Dissolved**
Hg	mg/L		< 0.0001	< 0.0001
Ag	mg/L		< 0.00003	< 0.00003
Al	mg/L	0.05	0.03	0.02
As	mg/L	0.005	0.0021	0.0025
Ba	mg/L		0.124	0.0865
Be	mg/L		< 0.00004	< 0.00004
B	mg/L		0.02	0.023
Bi	mg/L		< 0.00002	< 0.00002
Ca	mg/L		30.4	30.9
Cd	mg/L	0.00006	0.00011	< 0.00006
Co	mg/L	0.004	0.000275	0.000126
Cr	mg/L	0.009 {Cr(III)}	0.0005	0.0007

Table 6.8 Fresh and Aged Tailings Supernatant Analyses - Fine Fraction (cont'd)

Parameters	Units	BCWQ 30 day Aquatic Guideline	Day 0	Aged Day 61
Cu	mg/L	0.008 (3)	0.0008	0.0077
Fe	mg/L	0.3	0.01	< 0.01
K	mg/L		12.6	15.6
Li	mg/L		< 0.002	< 0.002
Mg	mg/L		7.87	10.3
Mn	mg/L	1.5 (3)	0.0256	0.0197
Mo	mg/L	1	0.0772	0.087
Na	mg/L		23.2	26.2
Ni	mg/L	0.15 (3)	0.003	0.0013
Pb	mg/L	0.20 (3)	< 0.00002	0.00014
Sb	mg/L	0.02	0.0044	0.0023
Se	mg/L	0.002	< 0.001	< 0.001
Si	mg/L		1.28	2.2
Sn	mg/L		0.0015	< 0.0003
Ti	mg/L		0.0006	0.0006
V	mg/L		0.00026	0.00116
Zn	mg/L	0.09 (3)	0.002	0.0013

6.3 Waste Rock

6.3.1 General

Waste materials from the open pit will comprise overburden soils (glacial tills and interglacial silts and clays) and waste rock. The waste rock contains both non-potentially acid generating (NAG) and potentially acid generating rocks (PAG), however we understand that it may be difficult to spatially separate the rocks, and for practical purposes a high percentage of the waste rock is classified as PAG. During operations, a geochemistry characterization program will be carried out and NAG rock will be preferentially placed in drainage channels and towards the south side of the waste rock dump.

PAG waste rock will be placed adjacent to the open pit and mill site, as shown on Drawing D-1002. A low grade ore stockpile, overburden storage dumps and organic bearing sediment stockpiles will also be located near the plant site.

The waste rock, over a period of time, is anticipated to potentially become acidic and leach metals. A proportion of the precipitation falling on the waste rock dump (WRD) will infiltrate and emerge, near the rim of the open pit, as effluent. The effluent will, at some point in the future, require management.

Geochemical characterization of the waste rock is being carried out by the Minesite Drainage Assessment Group (MDAG) and will be reported in the Environmental Impact Assessment. The following section provides a preliminary estimate of potential leachate water quality based on observations from nearby Bell and Granisle Mines, and from humidity cell leachate data developed by MDAG.

The effluent water quality was estimated from humidity cell leachate analyses and field test pad data from the project as well as from accumulated data from Bell and Granisle Mines, which are located a few kilometres along strike (Morin and Hutt, 2001).

6.3.2 Comparison of Morrison, Bell and Granisle Leachate Geochemistry

Information on leachate water quality pertinent to estimation of the expected effluent water quality from the Morrison Copper/Gold project is available from four different sources. Morin and Hutt (2007) summarize humidity cell and field test pad data available for the waste rock. These authors also provide a summary of humidity cell leachate chemistry for tailings from the project. In addition, Morin and Hutt (2007) provide summary data from Bell and Granisle Mines, which lie along strike from the project and are located in similar geological and mineralogical situations. Figure 6.1 provides a comparison of the waste rock leachate water quality data from all of these sources.

Gradation information was not available for the humidity cell samples. Consequently a laboratory to field coefficient of 0.1 was utilized.

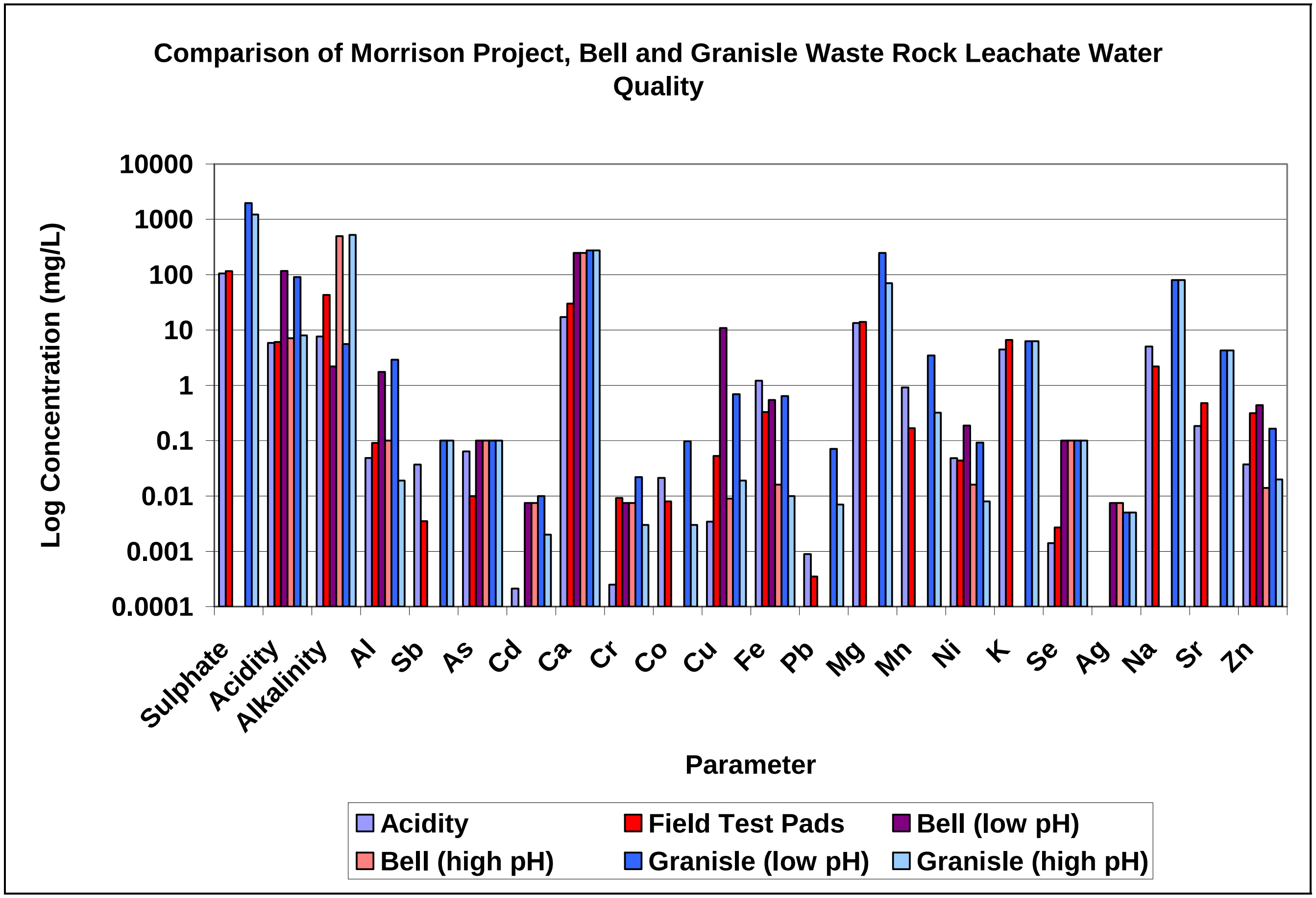


Figure 6.1 Comparison of Available Waste Rock Leachate Water Quality

Examination of the available leachate water quality data for the sources of bedrock described above indicates that the concentrations of various parameters are broadly similar amongst the data from the various sources. Consequently, base case and upper bound limits of expected leachate water quality at the project were estimated using the median (base case) and the ninety-fifth percentile (P_{95}) for the upper bound. Statistically selected data for the predicted waste rock leachate are provided in Table 6.9.

Table 6.9 Un-equilibrated Base Case and Upper Bound Estimates of Waste Rock Leachate Water Quality, Morrison Project

Parameters	BCWQG Aquatic Life (mg/L)	Base Case Waste Rock (mg/L)	Upper Bound Case (mg/L)
Sulphate	100	**672**	**1861.25**
Alkalinity		25.34	519.275
Al	0.05	**0.0955**	**2.615**
Sb	0.02	**0.06855**	**0.1**
As	0.005	**0.1**	**0.1**
Cd	0.005	0.00475	**0.009375**
Ca		248	275
Cr	0.009	0.0075	**0.0188**
Co	0.11	0.01465	0.085645
Cu	0.002	**0.036**	**8.348**
Fe	0.3	**0.435**	**1.068**
Pb	0.003	**0.003945**	**0.0614**
Mg		42	221.3
Mn	0.05	**0.621**	**3.0963**
Ni	0.025	**0.0461**	**0.164**
K		6.25	6.5475
Se	0.01	**0.1**	**0.1**
Ag	0.0001	**0.005**	**0.0075**
Na		42.525	80
Zn	0.0075(30 day) 0.033 (max)	**0.1012**	**0.40725**

Note: values in bold exceed BCWQG for aquatic life

Base case and upper bound estimates of several parameters for waste rock leachate water quality exceed BCWQG aquatic life standards.

6.4 Construction Materials for the Tailings Dams

The tailings dams will be constructed using locally available glacial till and sand and gravels. These materials were derived from a wide variety of soil and rock types located some distance from the Morrison Copper/Gold project site and were deposited by glaciers or glacially derived meltwaters. Consequently, it is unlikely that the borrow materials will be potentially acid generating. Nonetheless, a program of ABA testing for all borrow materials will be carried out as part of the QA/QC program for dam fills. Material that is potentially acid generating will not be used for construction.

The geochemical characterization of the cycloned sand used for dam construction is presented in Section 5.1 of this report.

7. TAILINGS FACILITY DESIGN

7.1 General

The tailings storage facility (TSF) is located approximately 4 km northeast of the mine as shown in Drawing D-1001, near elevation 1000 m (approximately 180 m in elevation above the millsite elevation). The TSF will cover an area of approximately 5 km^2 and will initially be formed with a Main Dam (95 m ultimate height). Construction of the North Dam (45 m ultimate height) will start in approximately Year 1 and the West Dam (35 m ultimate height) would be required beginning in Year 3. The Main and North dams will be built with a homogeneous fill starter dam and raised using the centerline cycloned sand method. Cyclone overflow and total tailings will be spigotted from the dams and the east side of the impoundment and a reclaim pond will be formed along the west side of the impoundment. A floating pump barge will be used to return excess process water back to the plantsite for reuse. The TSF catchment area is relatively small (total 4.6 km^2 catchment area above the final TSF) and only a small portion of surface runoff water need be diverted during average year conditions.

The facility will be operated as a “zero” discharge system and seepage collection dams will be constructed downstream of the dams to collect and return any seepage water to the TSF. Seepage losses from the impoundment are predicted to be relatively low due to the low permeability glacial till, which blankets most of the impoundment area, and the use of a low permeability core zone in the dams.

On closure, the dam slopes will be revegetated and the TSF will become a natural water pond. All potentially acid generating tailings, if present, will be permanently saturated with a minimum water cover. A closure spillway, excavated in bedrock in the left (east) abutment of the Main Dam, will be constructed for long term management of flood waters.

The facility has been designed to store 224 Mt of mine tailings, or approximately 150 Mm^3 of tailings. The storage-elevation curve for the TSF is shown in Figure 7.1.

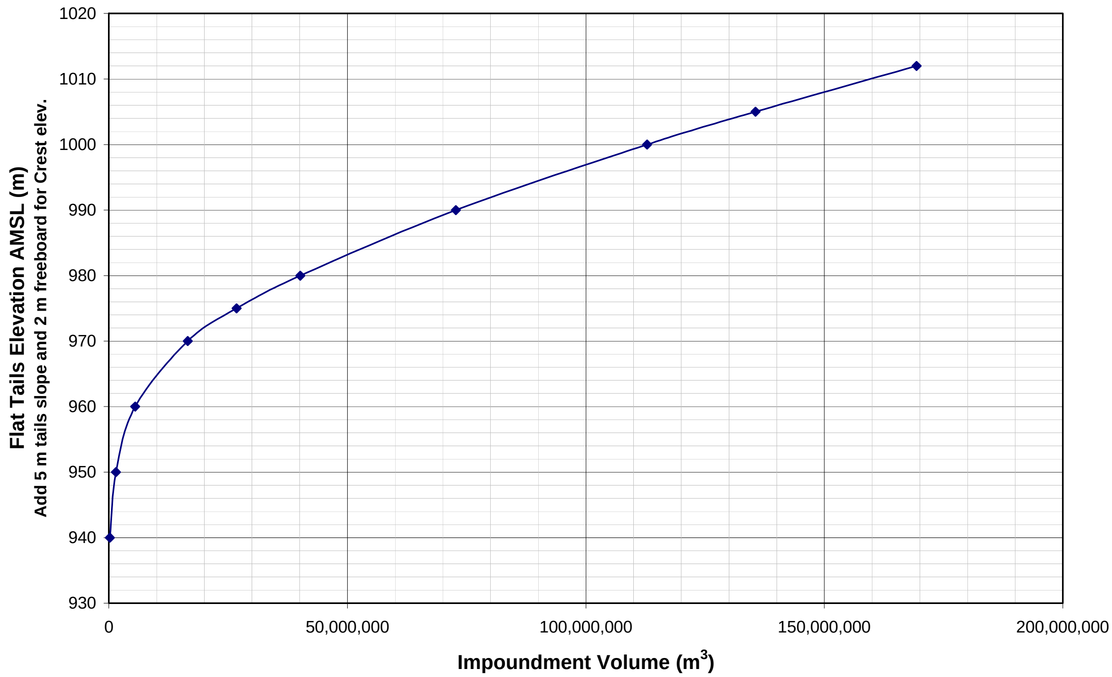


Figure 7.1 Volume - Elevation Curve - Tailings Impoundment

7.2 Dam Design

The Starter Dam for the Main Dam will be a 50 m high homogeneous compacted earthfill dam designed to store 1 year of tailings. The upstream slope will be 2H:1V and the downstream slope will be 3H:1V. Seepage and piping control will be with a 1 m thick sand and gravel blanket filter drain placed under the downstream shell of the dam. The starter dams for the North Dam and West Dam will be constructed later in the mine life and will consist of an approximately 5 m to 10 m high homogeneous fill dam, with a downstream blanket drain.

The Main Dam and the North Dam will be raised by the centerline cyclone sand method, with compacted cycloned sand placed in the downstream shell of the dam. A vertical 10 m wide low permeability core zone would be extended from the starter dam to the ultimate dam crest. The cyclone sand will act as a natural filter zone for the glacial till core to control the risk of piping.

Skid mounted cyclones will be located along the crest of the dams and the cyclone underflow would be used for dam fill and the cyclone overflow would be discharged into the impoundment, forming a beach and water reclaim pond. When the cyclones are not operating, total tailings would be spigotted from the east side of the impoundment. Later in the mine life, when the North Dam is required, a tailings line will be directed to the North Dam for cycloning.

The foundation area will be stripped of all loose and deleterious material. Topsoil will be stripped and stored for use in reclamation. A seepage cutoff trench will be excavated to approximately 3 m deep along the upstream toe of the dam to ensure a positive connection between the dam core and the foundation. The trench will be 10 m wide and will extend a minimum of 2 m into glacial till.

The main zones in the dams are described as follows:

<u>Glacial Till Dam Fill and Core Zone</u>

Glacial till will be borrowed from areas within the impoundment as shown in plan on Drawing D-1101, and from stripping the open pit. The borrow areas will be stripped and topsoil stockpiled for use in reclamation. The glacial till is a well graded silt-sand-gravel mixture, which will be placed in 300 mm thick loose lifts and compacted with a minimum of 6 passes with a 10 tonne vibratory roller. The material has a low permeability and is anticipated to be an excellent construction material. The moisture content will be controlled in the field within a range suitable for meeting the compaction specification of 95% of the Modified Proctor Density. Scarifying between lifts may be required if smooth surfaces are observed on the surfaces. Coarser graded borrow material will be preferentially placed towards the downstream side of the dam.

Any borrow pits developed within the impoundment will leave a minimum of 2 m of glacial till over the underlying bedrock to minimize the potential for seepage into the bedrock.

Sand and Gravel Blanket Drain

Sand and gravel will be obtained from Borrow Area #2, located approximately 1.5 km northwest of the plant site as shown in plan on Drawing D-1002. The sands and gravels typically contain 3% to 5% fines (< 75 micron sieve size) and the material may need to be washed as part of the processing to produce a material with < 3% fines. Additional testing will be carried out to confirm the requirements for wash processing. The sand and gravel borrow will be placed in 450 mm thick lifts and compacted with a minimum of 6 passes with a 10 tonne vibratory roller.

Cycloned Sand

Tailings will be delivered to the crest of the dam in a pipeline and sand will be produced for dam construction with skid-mounted hydrocyclone units located along the crest of the dam. The cyclone underflow, at a solids density of approximately 70% by weight, will be discharged with a flexible pipe into construction cells located parallel to the dam centerline. Each cell will be infilled with approximately 0.5 m thickness of tailings, which will be spread with bulldozers. Movement of the dozers, considered with the downward hydraulic gradients, provide sufficient compaction of the sand. An *in situ* density of 95% of the Standard Proctor Density will be specified. Excess water will be decanted from the cell and directed towards the seepage recovery pond for reclaim to the impoundment.

Cycloned sand will also be used to support the upstream toe of the central glacial till core. Cyclone sand, for the upstream zone, would be discharged directly to the upstream slope and would not be compacted.

The gradation criterion of the cyclone sand is < 17% fines (75 micron) to allow drainage and compaction of the sand. In addition, the quantities of sand required for the dams, and the available construction/cycloning months, requires that approximately 24% of the total tailings be recovered for 8 months of dam construction. A preliminary selection of the type of cyclone required for production of adequate quantities and quality of cycloned sand for dam construction has been carried out by KREBS Engineers. Their analysis recommended the use of six gMAX26-SRC hydrocyclones in parallel, which resulted in 24% recovery rate and a sand product with 11% fines. Each set of six hydrocyclones will be mounted on mobile skids for sand distribution along the dam crest.

7.3 Dam Stability Analysis

7.3.1 Geotechnical Design Properties

The geotechnical properties used for design are summarized in Table 7.1, and the selection of properties is summarized in the following sections.

Table 7.1 Summary of Geotechnical Design Properties

Soil Unit	Bulk Unit Weight (kN/m^3)	Static Drained Shear Strength	Pore Pressure Response $\bar{B}$	Static Undrained Shear Strength – Cohesion[1]
Cycloned Sand at Downstream	18	φ'=35˚ C'=0 kpa	0	-
Cycloned Sand at Upstream	19	φ'=29˚ C'=0 kpa	0	-
Tailings	19	φ'=29˚ C'=0 kpa		-
Till Core	21	φ'=32˚ C'=0 kpa	0	
Start Dam-Till	21	φ'=32˚ C'=0 kpa		
Glacial Till Foundation	21	φ'=37˚ C'=0 kpa	0.5 Starter Dam 0.3 Ultimate Dam	S_u=200 kPa
Blanket Drainage Sand	18	φ'=35˚ C'=0 kpa	0	-
Bedrock		impenetrable	0	drainage layer

Note1: For Starter Dam only

Undrained Shear Strength

The stability of the Main Starter Dam was checked for the undrained strength condition in the foundation glacial till. The undrained shear strength was estimated using correlations with the blow counts from Standard Penetration tests (SPT) of DH07-5B, located in the Starter Dam area. The data indicated a SPT-N value range of 25 to 80, with an average of over 30 blows per foot. Laboratory index tests indicate that the foundation till was medium plastic and using the correlation proposed by Terzaghi and Peck (1971), the unconfined compressive strength, U_c was 400 kPa and the undrained shear strength-cohesion, $C=U_c/2$, was 200 kPa.

Drained Shear Strength

- Glacial till foundation: A triaxial test carried out on an undisturbed glacial till sample indicated an internal drained angle of friction of 37°, which is consistent with dense glacial till strengths observed from other projects.

- Compacted glacial till: Three direct shear tests were carried out on remolded samples in the laboratory, which indicated that the drained internal friction angle of till was in the range of 31˚ to 33˚ and 32˚ was selected for design.

- Downstream cyclone sand and blanket sand: Cyclone sand placed in the downstream shell and the sand drainage blanket will be compacted to at least 90% of the relative density and a lower bound strength estimate indicates an internal angle of friction of 35˚. The cyclone sand and blanket sand is free draining material and the drained shear strength applies.

- Upstream cyclone sand: The cyclone sand placed in the upstream zone will be deposited hydraulically without mechanical compaction. Conservatively, we assumed that the internal friction for the cyclone sand was 29˚. The cyclone sand would be free draining and the drained strength would apply for the stability analysis.

- Tailings: The drained strength of the tailings would be similar to the cycloned sand, in the order of 29°. The tailings, however, would be loose and saturated and, therefore, the strength during seismic loading would reduce to the residual shear strength that has been estimated as $S_u/\sigma'=0.07$.

<u>Piezometric Levels and Pore Pressure Response</u>

The piezometric surface was assumed to be 4 m below the dam crest and horizontal from tailing pond to the central till core; the piezometric surface then drops concavely toward the blanket drainage layer.

Initially, an excess pore pressure, Δu_0, will be generated in the saturated glacial till foundation soils due to the construction of the Starter Dam. The pore pressure response $\overline{B}$ parameter is defined as the ratio of excess pore pressure, Δu_1, to the total applied stress σ:

- $\overline{B} = \Delta u_1/\sigma$

After loading, the pore pressures will dissipate with time, although additional pore pressures will also be generated when additional load is added each year. The initial pore pressure response parameter for the saturated glacial till foundation was assessed as follows:

- The glacial till is expected to be overconsolidated due to the ice loading during deposition. Empirical values for lightly overconsolidated, or dense fine sand silt, are in the range of 0.3 to 0.7. For a low plastic dense till the equivalent $\overline{B}$ value is estimated to be in the order of 0.5.
- A triaxial test carried out on an undisturbed sample of glacial till indicated a $\overline{B}$ parameter measured in the triaxial test varied from 0.7 for an initial load of 200 kPa, reducing to 0.3 at the higher stress loads.
- A $\overline{B}$ parameter of 0.5 was selected for design. The actual pore pressure response will be measured in the field with piezometers, and the design would then be adjusted, if required, to meet the required factors of safety for stability.

As the dam is raised over the life of the mine the total stress (σ) in the foundation will increase gradually. However, during this period, the void water will also drain out gradually and the excess pore pressures will dissipate. The reduction in the effective $\overline{B}$ value was estimated with a 1-D consolidation model. We assumed that foundation till consolidation coefficient, C_v, was 3 x 10^{-3} cm^2/s. The thickness of foundation till was 30 m with double drainage layers: the top drainage layer was the blanket drainage to be placed on the existing grade; and the bottom drainage layer was the bedrock underlying the foundation till.

The consolidation analysis indicated that the $\overline{B}$ value was not uniform along the depth of the till layer; the maximum $\overline{B}$ value was at the middle of the layer where the drainage path was the longest, and the minimum $\overline{B}$ value occurred at the top and the bottom of the layer, where the drainage path were zero. Assuming an initial $\overline{B}$ value of 1.0, the effective $\overline{B}$ value dropped to 0.26 during 20 years of dam construction and to 0.07 ten years after the dam is completed. However, starting with an initial $\overline{B}$ value of 0.5 would result in even lower values for closure and post closure.

7.3.2 Stability

Stability analysis was carried out using GeoSlope International SLOPE/W (2004). The Main Dam, North Dam and West Dam sites are as shown in the Drawing D-1102, the sections analysed for stability are shown in Appendix VII (refer to Figures VII-2.1 to VII-3.16). Stability analyses were carried out for the Starter Dam and Ultimate Dam cases and used the following conditions:

- Both effective strength and total strength (using undrained strength in the foundation) analyses were carried out for the Main Starter Dam.
- Strength and pore pressure parameters used are summarized in Table 7.1.
- The pseudostatic seismic analysis used a 0.065 g effective horizontal acceleration.
- Upper bound displacements were assessed using relationships proposed by Hynes-Griffin and Franklin (1984).

Starter Dam analyses using an undrained shear strength material model for the till foundation are summarized in Table 7.2. The effective strength analysis for the Starter Dam used a $\overline{B}$ parameter = 0.5. The critical slip surfaces are presented in Appendix VII (Refer to Figure VII-2.1, Figure VII-2.4 and Figure VII-2.5).

Table 7.2 Summary of Stability for the Main Starter Dam

Dam Location-Section	Factor of Safety	
	Undrained Strength Analyses	Effective Strength Analyses
Section D-D' 2.0H:1V upstream Without Tailings[1]	1.56 (Figure VII-2.1)	
Section D-D' 3H:1V downstream	1.31 (Figure VII-2.4)	1.54 (Figure VII-2.5)

Note 1: With tailing present at the upstream, the upstream will have higher FOS

The analyses for ultimate dams using drained material strength for till were summarized in Table 7.3. The effective strength analysis for the Starter Dam used a $\overline{B}$ parameter = 0.5. The critical slip faces were presented in Appendix VII (Refer to Figure VII 3.1 to Figure VII 3.15). The analyses indicate that the slopes are stable with FoS > 1.3 at operation stage and FoS > 1.5 at closure stage under static load**.** Under seismic load, pseudostatic analyses indicated that the slopes were stable with FoS > 1.1, the upper bound of displacement at the design earthquake events was less than 1.0 m**.**

The main factors controlling stability of the Ultimate Dams were the elevation of piezometric surface and the effective pore pressure response parameter $\overline{B}$ of the glacial till foundation. A conservative piezometric surface was used for the analyses and the actual surface is expected to be lower. The pore pressure response parameter $\overline{B}$ value also conservatively assumed a higher initial value, which then reduces during the life of the mine. The estimate was also based on a till thickness of 30 m, which is the maximum thickness. Dam areas with thinner till layers would consolidate faster.

Table 7.3 Summary of Stability Analyses Results

Dam	Section	Slope note 1	Stage	FoS note 2 – Operations ($\overline{B}$ =0.3)		FoS note 2 – Closure ($\overline{B}$ =0)	
				Static (Refer Figure in Appendix)	Pseudostatic seismic load = 0.065 (Refer Figure in Appendix)	Static (Refer Figure in Appendix)	Pseudostatic seismic load = 0.065 (Refer Figure in Appendix)
Main Dam	C-C'	3H:1V Downstream	Ultimate	FOS=1.42 (Figure VII-3.1)	FOS=1.12 (Figure VII-3.2)	FOS=1.80 (Figure VII-3.3)	FOS=1.45 (Figure VII-3.4)
	D-D'	3H:1V Downstream	Ultimate	FOS=1.34 (Figure VII-3.5)	FOS=1.09 (Figure VII-3.6)	FOS=1.80 (Figure VII-3.7)	FOS=1.42 (Figure VII-3.8)
North Dam	E-E'	3H:1V Downstream	Ultimate	FOS=1.51 (Figure VII-3.9)	FOS=1.19 (Figure VII-3.10)	FOS=1.85 (Figure VII-3.11)	FOS=1.48 (Figure VII-3.12)
West Dam	G-G'	2.5H:1V Downstream	Ultimate	FOS=1.42 (Figure VII-3.13)	FOS=1.17 (Figure VII-3.14)	FOS=1.59 (Figure VII-3.15)	FOS=1.28 (Figure VII-3.16)

Note 1: Section D-D’ Starter Dam and Section G-G’ upstream 2H: 1V slope, other sections have a vertical upstream with ~1% tailing beach slope.

Note 2: The upstream slopes have higher FoS than the corresponding downstream slopes as specified in Table 8-3

7.4 Seepage Analysis

The tailings facility is designed to minimize seepage, as far as practical, with low permeability dams and a seepage cutoff key into the low permeability glacial till foundations and bedrock. Seepage analyses have been carried out to model potential seepage flows through the dam and through the TSF impoundment area and foundations.

Seepage analysis was carried out using GeoSlope International SEEP/W (2004), which is a two-dimensional, finite element numerical model that simulates the movement and pore-water pressure distribution within porous materials such as soil and rock. Two models were constructed and were used to quantify steady-state tailings water seepage rates to the foundation soils through a representative section of the TSF and the Main Dam.

The models were developed for the Main Dam and then used to estimate seepage rates for the North Dam and West Dam.

7.4.1 Groundwater Model

General

The baseline groundwater flow and hydrogeologic characterization of the TSF impoundment area and dam foundations are presented in Section 4.6 of this report. As described in that section, groundwater in the tailings impoundment area flows predominately through fractured bedrock south-westerly towards "Main Dam" Creek and Morrison Lake. The majority of the fractured bedrock within the TSF is overlain by glacial tills of relatively low hydraulic conductivity. Based on the results of test pitting and drilling programs, a geophysical study and visual surface observations, the tills appear to taper to a thin veneer, or are completely absent, on steep slopes within the TSF and upslope to the east of the TSF. The hydrogeologic model assumes that the source of groundwater recharge is primarily infiltrating precipitation in areas of exposed bedrock, generally at higher elevations to the east of the TSF. Seepage below the dam is assumed to occur along a 1280 m length of the dam, between the slopes underlying the eastern and western sections.

Groundwater discharge has been observed on the slopes within the TSF and the slopes downslope to the south of the Main Dam. Precipitation infiltrating to the east of the TSF migrates into the TSF, partially discharging from the bedrock slopes and into creeks as runoff, with the remaining groundwater flowing under the glacial tills and discharging from exposed bedrock slopes further downslope and to the south of the TSF.

Model Description

The TSF seepage model is a two dimensional section originating to the east of the TSF above the area of drill hole MW07-2A/B, and generally following the surface topography to the north-south running creek in the vicinity of drill hole DH06-2. The location of the hydrogeologic section is shown in Drawing D-1003. Two versions of the model were used: one where a substantial thickness of glacial till was present below the dam (refer Figure VIII-2); and another representing sections of the Main Dam underlain by fractured bedrock (refer Figure VIII-3). Baseline versions of each, representing existing conditions, were developed and calibrated against baseline precipitation and base flow in the downstream creek.

The primary groundwater flow path through the fractured bedrock is represented by two zones, each approximately 20 m thick. The hydraulic conductivities used were based on the results of packer testing and rising head tests. The deep zone of fractured bedrock, generally encountered between 35 m and 50 m below ground surface, was assigned a hydraulic conductivity of 3×10^{-7} m/s. The shallow fractured bedrock, encountered generally between ground surface and a depth of 35 m, was divided into two units. The easterly unit was assigned a hydraulic conductivity of 6.5×10^{-7} m/s while the south-westerly section was assigned a hydraulic conductivity of 9.5 x 10^{-7} m/s. The hydraulic conductivity assigned to the glacial till foundation was 1 x 10^{-8} m/s while the underlying deep bedrock was considered a no-flow boundary.

The tailings were assigned a hydraulic conductivity of 1 x 10^{-6} m/s which is considered conservatively high. A constant head of 1015 m was applied across the tailings and represents the TSF at closure. The dam core was assigned a hydraulic conductivity of 1 x 10^{-8} m/s, the same as used for the glacial till foundation, of which, the dam core is composed.

Precipitation is represented as a unit flux of 1.25 x 10^{-9} m/s/m across the surface of the section, with the exception of the tailings impoundment where a constant head of 1015 m is applied. The precipitation flux, as well as the hydraulic conductivities assigned, were confirmed during calibration of the model.

The closure scenarios represented by the two models include saturated tailings to an elevation of 1015 m. Of the 2480 m total length of the Main Dam, the models are assumed to be representative of the approximately 1280 m length of dam between the two slopes underlying the eastern and western sections of the Main Dam. Based on the geophysical study, the model with till underlying the dam represents approximately 1175 m of the total length and the model with fractured bedrock underlying the dam represents approximately 105 m of the total length.

7.4.2 Model Calibration

Before the dam and tailings were added to the models, the models were calibrated to existing conditions. Hydraulic conductivities and precipitation rate were varied until a reasonable fit to existing depth to groundwater and creek base flow was established. Base flow was estimated from data collected at flow measuring station MCS-7. The lowest flows measured were between 1 L/s and 2 L/s.

Assuming that the model represents a 1280 m wide discharge area, and that all of the seepage discharges to the creek, a calibrated discharge to the north-south creek of approximately 2.4 L/s results for both baseline models (Figure VIII-4 till present and Figure VIII-5 till not present). Because the flows measured at station MCS-7 include flows from several creeks, of which the north-south creek, used for the south-westerly terminus of the model, is only one, the model estimate of discharge is likely higher than actual. However, for the purposes of seepage modelling, and recognizing the potential variability of site conditions, the calibration is considered adequate.

7.4.3 Model Results

The model simulations indicate that an increase in groundwater discharge to the north-south creek will be approximately 0.6 L/s, assuming all groundwater seeping under the dam reports to the creek. This groundwater, however, could potentially flow subsurface and emerge, eventually, in Morrison Lake. In consideration of the accuracy of the model and groundwater predictions, expected groundwater flows could range up to a half order of magnitude higher than those calculated.

Total seepage discharge at the downstream toe of the dam is estimated to be approximately 2.3 L/s. This seepage will be collected, with a seepage recovery pond and pumpback system, and returned to the tailings facility.

The modeling results indicate a greater increase in groundwater discharge to the north-south creek when glacial till is present under the dam. Without the glacial till confining groundwater in the underlying fractured bedrock, some of the additional head in the aquifer due to the tailings is released with groundwater that would otherwise discharge to the creek when the glacial till is present, discharging at the toe of the dam. Though a lower flux increase is evident, there is potentially more mixing of tailings water with groundwater and hence the groundwater seepage water quality could be similar to the impoundment water quality, (see Table 6.7 and Table 6.8). The use of the process water quality would assume that no allowance has been considered for attenuation and adsorption of metals along potential groundwater flow paths.

The hydrogeologic conditions at the North and West Dam sites are similar to the Main Dam, however potential seepage flows are estimated to be lower because of the lower dam heights.

Table 7.4 Summary of Predicted Seepage Flows from the Tailings Facility

Dam	Seepage Estimates (L/s)		
	Dam Structure reporting to Seepage Pond	Dam Foundations	
		Base Case	Upper Bound
Main Dam	2.4	0.6	3
North Dam	1.5	0.4	2
West Dam	0.5	0.25	1

7.5 Tailings Deposition Plan

Tailings will be deposited from the cyclone overflow from the dam crests and from the east side of the impoundment. The average tailings beach slope is estimated to be approximately 1%; the slope will vary from > 1% near the spigot discharge and flatten until it meets the water pond. At the pond, the beach slope typically steepens to approximately 4% and then flattens out towards the reclaim barge.

For startup of mine operations a free water pond of approximately 100,000 m^3 will be required to provide water for both startup of the mine and for settling of the initial tailings discharge. The size of the water pond will vary seasonally and a minimum volume will be maintained for settling capacity and pumping requirements. The water management components are described in Section 9 of this report.

7.6 Instrumentation and Monitoring

The geotechnical instrumentation for the dam will include the following:

- Electric piezometers will be installed in the glacial till foundation to monitor the pore pressure response due to loading. “Trigger” levels will be determined from the stability analyses to ensure that the pore pressures are within the predicted range and that stability is maintained.
- Two inclinometers will be installed in the each of the Main Dam and North Dam. The inclinometers will monitor potential deformation of the dam and/or the foundations.
- Surface survey monuments will be installed along the dam crest.

Engineering and Environmental Monitoring for TSF will include the following:

- Construction QA/QC will be carried out for the Starter Dam construction and for ongoing raising of the tailings dam.
- Groundwater monitoring wells, installed downstream of the TSF, will be used to monitor potential groundwater flows and contaminant transport.
- Impoundment water quality tests will be carried out monthly to document the actual water quality of the impoundment throughout operations.
- Seepage flows and seepage water quality downstream of the dam will be monitored at least monthly.
- The climate stations will continue to be used.

General monitoring of the operations will include:

- Daily inspection of the cyclone operations, seepage recovery pond and tailings pipelines.
- All "as-built" conditions will be documented.
- Tailing volumes and water volumes will be measured and reconciled on an annual basis. This will include the TSF water balance model, which will be continually updated to incorporate actual hydrometric conditions, tailings and water volumes and storage characteristics.

7.7 Civil Works

7.7.1 Tailings and Water - Pumping and Piping

The tailings delivery and reclaim systems are presented in Appendix X and the main components are summarized as follows:

Tailings Pumping

Two stage pumping is required: Pump Station No. 1 is located within the mill buildings and others (Wardrop) are designing all building auxiliaries; Pump Station No. 2 is located midway between the plant site and the TSF. The building for Pump Station No. 2 will be a 13 m wide by 23 m long steel clad, pre-constructed building, complete with heating, electrical works, overhead crane, etc. Power supply will be with an underground cable designed by Wardrop. The pumps include two Peerless 8175, 4 x 6 x 14 end suction centrifugal ANSI pumps.

Tailings Pipeline

The tailings delivery pipeline will be HDPE and HDPE-lined-steel (for the higher pressure sections of the line), approximately 760 mm diameter. The pipeline will be laid on surface in a ditch and emergency storage ponds will be provided to contain any spills. The pipeline will deliver total tailings to the crest of the Main Dam and, starting in year 2, to the crest of the North Dam.

Cyclone System

The cyclone system will consist of 6 skid mounted hydrocyclones, which will be used to produce cycloned sand for dam construction. A supply line, which is "tapped" into the water reclaim line, will be used to supply the hydrocyclones. The water is pumped around the top of the tailings dams via two sets of booster pumps and a separate HDPE header pipe around the perimeter of the tailings pond.

Water Reclaim System

The reclaim system consists of a reclaim barge with pumps, a reclaim line from the tailings pond to the reclaim/fire water tank, then continuing to the plant site. A preliminary design for the barge was provided by Chamco and consists of a 12 m long by 8.5 m wide and 2.4 m deep fabricated steel hull, complete with electrical, winch stands, guard rails and access ramp. The pump system will include three Peerless 1 stage 20HH vertical pumps, with motors, power and controls.

The reclaim pipeline will be a buried HDPE pipeline.

Seepage Pond Reclaim System

The seepage pump stations will have a precast concrete intake with two Peerless 4 or 5 stage M14LD vertical turbine pumps. The pipeline will be a 7.15 inch diameter HDPE SDR 9 pipe from the pump station to the TSF.

Fresh/Process Water System

The fresh/process water make-up system consists of a pump house located on the shore of Morrison Lake, and a discharge pipeline to the plant site. Two Peerless 5 stage M12LD vertical turbine pumps will draw water from a well, connected to the lake by HDPE or steel pipe. The pumps will be housed in a pre-engineered steel frame building. The pipeline will be a 8.625 inch OD HDPE, SDR 11 pipe from the pump house to the plant site.

7.7.2 Access Roads for the Tailings Facility

Access roads for the TSF are shown on Drawing D-1002, and details are provided on Drawings D-1401 and D-1402. The main portion of the road will run from the Plant Site to the east abutment of the Main Dam. A slightly smaller branch road will provide access to the west abutment of the Main Dam, the Reclaim Pipeline and Pond/Barge and the West Dam. The main design assumptions for the road are summarized in Table 7.5.

Table 7.5 Dam Access Road Design Assumptions

	Main Dam Access Road	Reclaim Pipeline Access Road
Design Vehicle	40 ton Articulated Truck Length = 11.0 m Width = 3.5 m Tire O.D. = 1.86 m	30 ton Articulated Truck Length = 9.9 m Width = 2.9 m Tire O.D. = 1.61 m
Maximum Grade	15% for Short Pitch, 10% Sustained Grade	15% for Short Pitch, 10% Sustained Grade
Travel Width	10.5 m (2 Lanes)	6.0 m (1 Lane)
Design Speed (For Grades of 8% or less)	50 km/hr	50 km/hr
Min. Curve Radius (for 50 km/hr)	100 m	100 m
Cross Slope (Sloping towards the hillside)	4%	4%

The travel widths listed were determined based on 3 times the design vehicle width for the 2-lane road and 2 times the design vehicle width for the single lane road based on British Columbia Mines Act Health and Safety standards. The single lane road should be constructed with 10.0 m wide pullouts every 500 m along the length to allow passing.

In addition to the travel width specified, the Main Dam and West Dam roads also require a shoulder barrier on the edge of the haulage road opposite the natural slope. This barrier is required for the full length of the road for safety and must be at least 75% of the height of the largest tire on any vehicle hauling on the road. This corresponds to a 1.4 m high barrier for the Main Dam access road and a 1.2 m high barrier for the West Dam branch road.

The cross slope on the roads are specified as 4% sloping into the natural slope. A drainage ditch is required on the inside of the road for the full road length. Drainage culverts are also required to allow passage of runoff water across the road and to pass existing streams. These culverts will likely consist of up to 850 mm diameter CSP pipes typically every 70 m along the road length.

Maximum cut and fill slope angles will depend on the quality of overburden found on site. In general, for coarse grained soils, cut and fill slopes shall be no steeper than 1.5H:1V. For cuts in rock, or consolidated soils and fills using well graded rock, steeper slopes are acceptable but should be evaluated on site.

8. WASTE ROCK DUMP AND STOCKPILES

8.1 General

Development of the open pit will require storage of approximately 170 Mt of mine waste rock, of which approximately 17 Mt is classified as non-potentially acid generating (NAG) and the remainder is potentially acid generating (PAG). In addition, approximately 19 Mt of low grade ore (LGO) will be stockpiled near the plant site for processing later in the mine life. Pre-stripping of the open pit will require drainage of Booker Lake and Ore Pond, and removal of soft sediments in the lakes, as well as removal of localized boggy soils. Organic material will be stripped and stored in a separate facility for use in reclamation of the mine upon closure. Overburden soils will be used to construct the Starter Dam for the TSF and the organic sediment storage containment; excess overburden will be stored in a stockpile west of the open pit for reclamation of the waste rock dump slopes. Drawing D - 1201 shows the layouts of the storage facilities and surface water management works.

Diversion ditches will direct clean surface runoff water around the waste rock dump and plant site and into the adjacent drainages. All contact drainage water from the waste rock, low grade ore, open pit and plant site, will be collected and recycled to the plant site for process water. Drainage from the overburden and organic bearing material stockpiles will be treated in sediment ponds and discharged.

The waste dumps will be constructed to accommodate reclamation with 2.5H:1V inter-bench slopes, with 5 m wide benches every 20 m in height, for an overall slope of 2.75H:1V. The dump slopes will be covered with a low permeability glacial till layer to reduce infiltration and a veneer of organic material to promote revegetation. Seepage from the waste rock dumps may, over a period of time, become acidic and require treatment with a high density sludge water treatment plant. The water treatment plant will treat potentially contaminated runoff from the open pit.

8.2 Design Basis

Table 8.1 summarizes the design criteria for dump stability, which have been adopted from the BCMWRPRC (1991) and are generally higher than that required for a low hazard rating, which is the classification for this project. The Factor of Safety (FoS) criteria selected for static stability are 1.2 for operating and 1.4 for closure, which are conventionally used for engineered waste dumps. Stability analyses were conducted using effective frictional strength parameters and estimated operating pore water pressures in the dump materials and underlying foundation soils. Stability evaluations were performed on all dumps with the exception of the small organic stockpile dumps.

Table 8.1 Stability Criteria for Dump Design

Loading Condition	Design Standard	Design Criteria
1. Static Loading Conditions	Two-Dimensional Limit-Equilibrium Factor of Safety with operating pore pressures. Deep-seated failure surfaces through dump foundation.	- FoS = 1.2 For operating conditions - FoS = 1.4 For long-term closure conditions
2. Pore Water Pressure Conditions[1]	Using $\overline{B}$ pore water pressure determination.	- $\overline{B}$ = 0.4 for operating conditions and - $\overline{B}$ = 0 for long-term closure
3. Seismic Loading Conditions[1,2]	Pseudo-Static Deformation Analyses	- <1m horizontal displacement for long-term closure conditions - FoS =1.1 for long-term closure conditions - 1,000 – year return period event for operation and closure: 50% of PGA=0.03g

Note: 1. Pore water pressure conditions and seismic loading conditions were not applied to organic stockpiles.
2. Only the closure waste rock dump was subjected to a seismic horizontal displacement analysis.

The seismic deformation criterion of < 1 m is conservative and reflects the desire to maintain the integrity of the dump closure cover, as opposed to geotechnical stability. Allowable deformations could be in the order of 10 m horizontal without initiating any significant stability concerns.

8.3 Storage Requirements and Construction Methodology

Storage Requirements

The approximate volumes of waste rock, low grade ore, organic bearing material and overburden are summarized in Table 8.2.

Table 8.2 Summary of Waste Rock, Low Grade Ore and Overburden Quantities

<table>
<tr><th colspan="2">Dump Site</th><th>Footprint (ha)</th><th>Toe Elevation (m)</th><th>Crest Elevation (m)</th><th>Storage Volume (m^3)</th><th>Tonnage (t)</th></tr>
<tr><td colspan="2">Waste Rock Dump</td><td>175</td><td>810</td><td>991</td><td>85,000,000</td><td>170,000,000</td></tr>
<tr><td colspan="2">Low Grade Ore Stockpile</td><td>20</td><td>776</td><td>910</td><td>19,000,000</td><td>38,000,000</td></tr>
<tr><td colspan="2">Overburden Stockpile</td><td>30</td><td>740</td><td>800</td><td>7,410,000</td><td>12,600,000</td></tr>
<tr><td rowspan="2">Overburden and Organic Sediment Dump</td><td>Organics & Weak Sediments</td><td rowspan="2">18</td><td rowspan="2">796</td><td rowspan="2">840</td><td>1,150,000</td><td>1,730,000</td></tr>
<tr><td>Overburden</td><td>555,000</td><td>1,050,000</td></tr>
<tr><td colspan="2">Organic Bearing Material Stockpile No. 1</td><td>2.6</td><td>820</td><td>860</td><td>325,000</td><td>550,000</td></tr>
<tr><td colspan="2">Organic Bearing Material Stockpile No. 2</td><td>5.3</td><td>743</td><td>766</td><td>113,000</td><td>190,000</td></tr>
</table>

Organic Bearing Material (OBM) Stockpile No. 1 is sized to store a stripping depth of 0.3 m over the ultimate open pit area of approximately 108 ha. OBM Stockpile No. 2 is sized to store a stripping depth of 0.3 m over the plant site area of approximately 38 ha.

Table 8.2 summarizes the annual production of waste rock during open pit development. Figure 8.1 shows the rate at which the waste rock dump surface will rise, assuming even distribution of the PAG and NAG waste rock over the dump area.

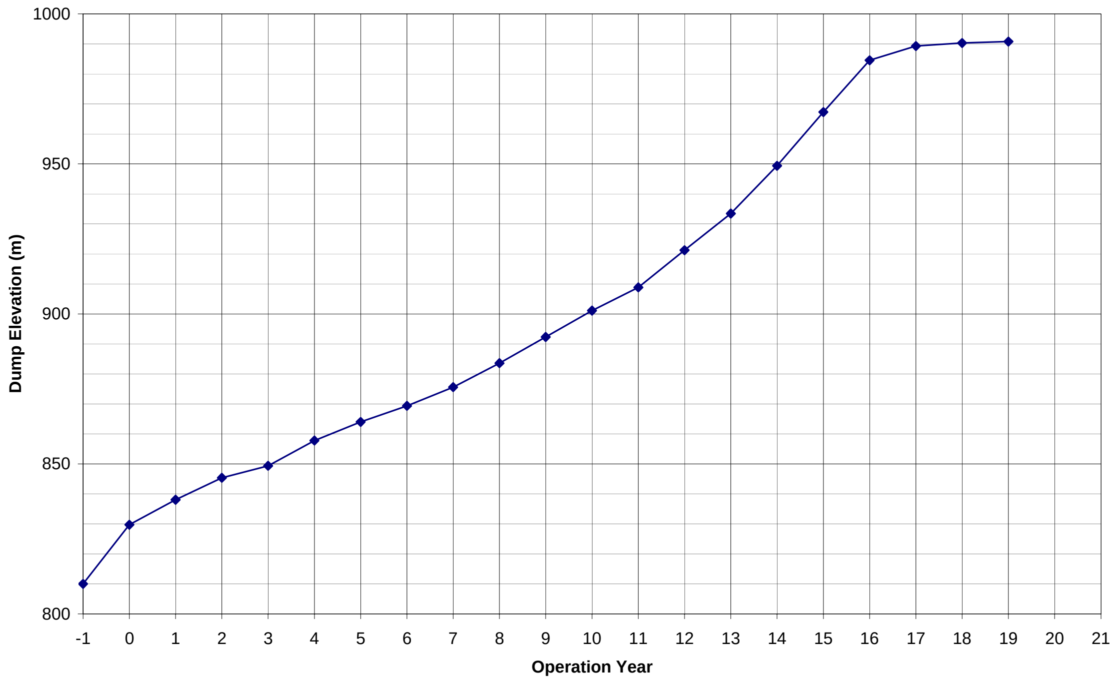


Figure 8.1 Waste Rock Dump Raising Schedule

Foundation Preparation and Drainage of Booker Lake and Ore Pond

Removal of saturated, loose and soft soils will, as far as practical, be carried out in the winter months when the ground is frozen. Pre-drainage of areas will also, as far as is practical, be carried out prior to any excavation and will include the construction of and use of drainage ditches.

Estimated water and sediment volumes in Booker Lake and Ore Pond are summarized in Table 8.3. The volumes of sediment could vary from the assumed values, although reasonably conservative values were selected for planning purposes.

Table 8.3 Water and Sediment Volumes in Booker Lake & Ore Pond

Lake	Area (ha)	Water Depth (m)		Assumed Sediment Thickness (m)	Volumes (m^3)	
		Maximum	Average		Water	Sediment
Booker Lake	15	23	10	4	1,500,00	600,000
Ore Pond	2.5	6	4	2	100,000	50,000

Drainage of Ore Pond and Booker Lake, and the removal of the saturated sediments will be carried out primarily over the winter months when frozen ground will facilitate excavation and storage of soft materials. Drainage will be carried out with a barge mounted pump, which will provide control over the pumping rate and pumping location to minimize the risk of pumping water with elevated suspended sediment. Ore Pond will be drained first and soft sediment removed. Ore Pond will then be used as a sediment retention pond for Booker Lake, particularly when the drainage of Booker Lake is near the final stage and there is a higher risk of suspended sediments in the discharge water. During lake and pond drainage, inflows to Booker Lake will be diverted by temporary ditches towards the adjacent drainage basin to the southeast. The natural boggy area of the south drainage basin will attenuate flows and allow sediments to settle before water flows into Morrison Lake.

The general program for drainage and removal of soft sediments is summarized as follows:

- Ore Pond: The pond level will be drawn down in the fall months by direct pumping of water from a barge mounted pump. Pumping will be carried out until the remaining water volume is too small to permit pumping without inclusion of elevated suspended sediment. The required pumping capacity, to allow drawdown of 80% of the water in one month is 40 L/s (140 m^3/hr). At this stage, pumping will stop and the remaining water and exposed sediments will be allowed to freeze over the winter months. Excavation of the remaining water (ice) and frozen sediments will be carried out during the winter.

- Booker Lake – Pre-Drainage: Booker Lake will be substantially dewatered in the summer/fall months over a 6 month period. Approximately 90% of the water volume will be discharged from a barge mounted pump to preclude elevated suspended sediment in the discharge water. Dewatering will require a pumping capacity of approximately 90 L/s (320 m^3/hr) over the six month period.

- Booker Lake – Final Drainage and Sediment Removal: Final drainage of the lake will occur over the winter months. The remaining water will either freeze and be excavated, or will be pumped to the cleaned out Ore Pond, which will act as a settling pond to control elevated suspended sediment. Frozen and unfrozen sediments will be excavated and placed in the storage area.

Overburden and Organic Sediment Dump

The storage dump for the organic/sediment materials, which are expected to include soft, weak materials, will be constructed concurrently with placement of more competent overburden (glacial till soils) in the downstream shell of the pile to maintain a stable outer slope. This will allow placement of the weaker unconsolidated soils inside the perimeter of the pile.

Stripped overburden and organic material will be placed in 2 m high lifts. The temporary stockpiles will have an average overall outer slope of 3H:1V.

General Rock Dump and Low Grade Ore Stockpile Construction

NAG and PAG waste rock and low grade ore will be placed in 10 m to 20 m high lifts, with 5 m wide intra-slope benches to achieve an average overall slope of 2.75H:1V.

The waste rock dump and low grade ore stockpile should have a minimum setback of 100 m from the toe to the edge of the open pit at ground surface.

8.4 Geotechnical Conditions

Zones of weak soils occur in localized boggy areas and in Booker Lake. In addition, a soft to firm clay layer occurs up to 6 m deep beneath the west side of the low grade ore stockpile. All soft, weak or deleterious materials will be removed from the critical foundation areas beneath the slopes of the waste rock dump and low grade ore stockpile.

After removal of soft materials, the foundations of the waste rock dump and low grade ore stockpile and overburden and organic sediment storage areas will consist primarily of a medium dense glacial till overlying bedrock. The glacial till is a saturated fine soil that can generate pore pressures due to loading from construction of the piles. The geotechnical properties of the soil were determined from laboratory testing of an undisturbed sample, as described in Section 7.3.1 of this report. The shear strength of the glacial till is relatively high, $\varphi = 36°$ and the main geotechnical consideration is the potential pore pressure generation during loading. The $\overline{B}$ pore pressure response parameter measured in a consolidated undrained triaxial test indicated an initial value of 0.7, reducing to 0.3 at higher stress levels (dump heights). The $\overline{B}$ value was also assessed on the basis of a consolidation test on the glacial till. The existing test holes indicate that the glacial till is typically up to 30 m thick, except for one area adjacent to the open pit where the overburden can be up to 55 m thick. The determination of the $\overline{B}$ value was based on the following:

- The thickness of foundation till was assumed to be 55 m with double drainage layers; the top drainage layer is waste rock placed on the existing grade, and the bottom drainage layer is bedrock underlying the foundation till.
- The maximum dump height was assumed to be 100 m, inducing approximately 2000 kPa of increased stress. The laboratory consolidation test indicates that the consolidation coefficient Cv of the glacial till ranges from 3.4×10^{-3} cm^2/s to 4.7×10^{-3} cm^2/s over the stress levels of 450 kPa

to 1800 kPa. A Cv value of 4.0 x 10^{-3} cm^2/s was assumed for consolidation analysis.

- An initial $\overline{B}$ value of 0.5 was applied to annual load increases, based on triaxial test results.
- To simplify the loading sequence, we assumed that an annual average of 10 m of waste rock would be placed on the dump uniformly over the 21 years of operation.

The maximum $\overline{B}$ value is at the middle of the layer where the drainage path is the longest. The $\overline{B}$ value decreases more slowly than at the TSF, due to the greater till thickness. At year 20 of operation, the calculated $\overline{B}$ value drops to 0.41. The $\overline{B}$ value decreases to less than 0.1 within 35 years after mine closure and eventually will approach zero. A $\overline{B}$ value of 0.4 was assumed for post-construction stability analysis of the ultimate waste rock dump and LGO stockpile, and a $\overline{B}$ value of 0 assumed for long term closure.

Waste rock is a free draining material with relatively high shear strength. The phreatic surface was assumed to 1 m above the base of the dump for the purpose of analysis. Based on a poorly graded, loose dump rock and/or gravel, we conservatively assumed a waste rock drained strength $\phi' = 38°$. The overburden and organics placed in the stockpiles are assumed to have lower strengths. Very weak lake-bottom sediments will require containment with competent overburden soils. The geotechnical properties used for design are summarized in Table 8.4.

Table 8.4 Summary of Geotechnical Parameters – Waste Dump Facilities

Material	Total Unit Weight (kN/m^3)	Static Drained Strength (ϕ')	Pore Pressure Response Parameter
Waste Rock	22	38°	
Low Grade Ore	22	38°	
Colluvium/Till Foundation	21	36°	$\overline{B}$ = 0.4 during operation stages $\overline{B}$ = 0.0 at closure stage
Loose Overburden	18	30°	
Organic and saturated sediments	16	27°	Ru = 0.3

Note: Ru is the pore pressure ratio defined by $Ru = u/\sigma_v$

8.5 Waste Rock Dump and Stockpile Stability

Two-dimensional limit equilibrium stability analyses were performed using the computer software SLOPE/W and circular slip surfaces were analyzed by the Morgenstern-Price method.

The design sections used for the stability analyses are shown on Drawing D-1201 and representative cross sections are shown on Drawing D-1202 and D-1203. Soil profiles and groundwater elevation are shown on Figure VII-4.1 for section A, Figure VII-4.9 for Section C and Figure VII-4.13 for Section D.

The geotechnical parameters used for the analyses are summarized in Table 8.4. The following general conditions were also used for the analyses:

- The foundation glacial till was assumed to have a pore pressure response of $\overline{B}$ =0.4 for operations and $\overline{B}$ =0 at closure. The rate of loading is low because the dumps will be incrementally raised over the life of the mine.

In addition, piezometers will be installed in the foundation soils to measure the actual pore pressures;

- Pseudo-static seismic analysis assumed 0.03 g horizontal acceleration, which is equivalent to 50% of the 1,000 year return period PGA; and
- Upper bound seismic displacements were assessed using relationships proposed by Hynes-Griffin and Franklin (1984).

The results of the stability analysis are summarized in Table 8.5 and the figures for critical slip faces are attached in Appendix VII (refer to Figure VII-4.1 to Figure VII-4.20).

The analysis indicates that 2.75H:1V rock dump slopes have a FoS higher than the Stability Criteria for Dump Design listed in Table 8.1. The analysis also indicates that waste rock dump displacement at the design earthquake event is less than 1.0 m.

Table 8.5 Summary of Stability Analyses - Waste Dumps and Overburden Piles

Facility	Crest El (m)	Operations ($\overline{B}$ =0.4)		Closure ($\overline{B}$ =0)	
		Static	Pseudostatic	Static	Pseudostatic
Waste Dump (A-A) Right Slope	940	1.70 (VII-4.1)	1.53 (VII-4.2)	1.96 (VII-4.3)	1.76 (VII-4.4)
Waste Dump (A-A) Left Slope	940	2.00 (VII-4.5)	1.78 (VII-4.6)	2.33 (VII-4.7)	2.11 (VII-4.8)
Waste Dump (C-C) Right Slope	990	1.55 (VII-4.9)	1.41 (VII-4.10)	1.83 (VII-4.11)	1.75 (VII-4.12)
Waste Dump (D-D) Right Slope	910	1.58 (VII-4.13)	1.52 (VII-4.14)	1.58 (VII-4.15)	1.73 (VII-4.16)
Waste Dump (D-D) Left Slope	910	1.83 (VII-4.17)	1.67 (VII-4.18)	2.58 (VII-4.19)	2.06 (VII-4.20)

Note: Representative Slope/W output figures are included in Appendix VII (figure numbers in brackets).

8.6 Monitoring and Environmental Guidelines

The waste storage facilities will be constructed and operated to meet safety and environmental requirements. The main monitoring and control aspects are summarized for the construction and operation phases as follows:

Construction

The main activity during the pre-mining construction phase is the dewatering of Booker Lake and Ore Pond and the stockpiling of sediments and overburden. The monitoring/control requirements for this stage include:

- Monitoring of total suspended solids in the discharge water from drainage of the lake and pond.
- Placement of silt fences around the toes of the overburden and organic sediment piles, along with construction of a collection ditch and settling pond to control the suspended sediment levels.
- Progressive reseeding and reclamation of the piles to control erosion.
- Inspection and assessment of stripped foundation areas of the outer shells of the waste rock dumps to confirm adequate soil strength for construction of the dumps and stockpiles.

Operations

- Electric piezometers will be installed beneath the slopes of the waste rock dump and low grade ore stockpiles. The purpose of the piezometers is to measure the actual pore pressure response of the glacial till due to loading by the dumps. This will be used to confirm the predicted values and the stability of the dumps as they are raised.
- Where possible, progressive reclamation of the waste rock dump will be carried out.

- Inspections of the waste rock dump and overburden stockpiles will be carried out by mine personnel and any unusual or unexpected conditions will be reported to the mine geotechnical engineer for action.

- The mine waste rock is potentially acid generating, and although the lag time for depletion of the residual alkalinity is uncertain. It is possible that some contaminated seepage/runoff water from the waste rock dump will occur during operations. The water quality of seepage from the waste rock dump will be monitored, possibly monthly or on a quarterly basis, to assist in the ongoing assessment of the waste rock geochemistry. All contaminated seepage water will be recycled back to the plant site during operations.

9. WATER MANAGEMENT

9.1 General

The mine facilities will be operated as a closed system with all process water and "contact" surface water collected and recycled to the plant site for use in the milling process. Clean surface water will be diverted around the facilities to reduce the quantity of contact water. Nonetheless, in approximately Year 8, there is a net surplus water balance, which will require storage in the TSF until closure. A water reclaim pump barge and pipeline will return water from the TSF to the plant. A water balance model is presented, which links the mine area drainage and the TSF. The water balance accounts for all inflows, both into the mine area and into the TSF, as well as all losses to tailing voids, groundwater, evaporation and processing.

The main water management components for the mine are summarized as follows and details of the components are provided in the following subsections.

Tailings Storage Facility (TSF)

- Seepage recovery ponds to collect seepage water and construction water from the cyclone sand process, which is returned to the TSF. The seepage pond has an emergency spillway for dam safety;
- Flood management for the TSF and for the Seepage Recovery dams for dam safety;
- Operation of the TSF will reduce, during operations, the stream flows in the creek immediately downstream of the Main dam, and, to a much lesser extent, the stream flows downstream of the North dam;
- Surface water diversion ditches to reduce water inflows during operations; and
- Upon mine closure, a permanent spillway will be constructed to provide for long term dam safety.

Mine and Plant site Facilities

- A "clean" water intake on Morrison Lake to provide clean water for the plant site;
- "Clean" water diversion ditches to reduce water inflows into the waste rock dump, open pit and plant site areas;
- Collection ditches and sumps to collect runoff from the waste rock dumps, open pit and plant site areas for return to the process plant; and
- Sediment control ponds downstream of overburden stockpiles to provide settling of suspended solids prior for discharge.

9.2 Water Balance

A monthly water balance model was developed for four stages of the mine: Starter Dam (year 2), Operations (year 10), Ultimate dam (year 21) and Closure. Two subcomponents of the water balance were assessed:1) the TSF; and 2) the open pit, plant site & waste rock dump areas, and are discussed below. The two sub-components are linked into an overall water balance, which is used to determine requirements for diversion, and storage. The monthly water balance is tracked on an annual basis, with seasonal storage provided within the TSF. The precipitation and evaporation parameters are based on the average monthly values.

Tailings Facility Water Balance

The inflows and outflows from the TSF model are discussed as follows:

Inflows:

- Precipitation and runoff:
    - Precipitation/runoff over the total tailings beach and reclaim pond area assumed a runoff coefficient = 1.0.

- Precipitation/runoff from the uphill catchment areas, above the active tailings/decant pond area, and from the dam slopes and seepage pond catchment areas, assumed a runoff coefficient = 0.5.

- The runoff flows were adjusted for monthly runoff distribution influenced by snow and snowmelt.

- Tailings Transport Water: The "whole" tailings transport water delivered to the TSF is based on 30,000 tpd of tailings solids at a density of 34% solids by weight. Tailings will then be discharged into the impoundment as either total tailings directly from the plant site, or as part of the operation of the cyclones during the 8 month dam construction period. During dam construction we have assumed that whole tailings will be discharged to the impoundment for 15% of the time due to periodic cyclone plant maintenance.

  - During cyclone operations it is necessary to "locally" add approximately 164 m^3/hr of water to the cyclone feed water to operate the cyclowash cyclones. This water, which will be pumped from the reclaim pond and added into the cyclone underflow and overflow, is cancelled with a "loss" in the outflows, resulting in no net change in water use due to the cyclowash system. The cyclone operation assumes a sand underflow recovery of 24% of the total tailings solids of 30,000 tpd.

  - The cyclone overflow is estimated to have a density of 27.5% solids by weight.

  - The cyclone underflow process water assumes that the underflow density of discharge to the downstream shell of the dam is 72% solids by weight and that the cycloned sand in-situ will have an effective water volume of 85% solids by weight. This solids density is equivalent to a residual moisture content in the compacted tailings of 18%. The remaining water is assumed to report to the seepage recovery pond and returned to the impoundment.

- Seepage: Seepage through the impoundment dams will be primarily collected in the seepage recovery pond and returned to the impoundment. The estimated seepage reclaim is 4 m^3/hr.

Outflows:

- Voids: The main water losses are to the voids of the tailings. Based on interpretation of settling tests the tailings in the impoundment are estimated to settle to an effective dry density of 1.3 t/m^3 (72% solids by weight) for the Starter Impoundment to 1.4 t/m^3 (75% solids by weight) for year 10 and 1.5 t/m^3 (77% solids by weight) for the Ultimate Impoundment.

- Seepages: Most seepage is expected to report to the Seepage Recovery Pond; however some seepage can be expected to report to deeper groundwater flows as discussed in Section 7.4 of this report. Of the 10 m^3/hr of total seepage loss, approximately 4 m^3/hr will be reclaimed and 6 m^3/hr will report to deeper groundwater flows as the net loss to seepage.

- Evaporation: Evaporation losses were assumed to apply to the total wet tailings beach and decant pond areas.

Plant Site, Open Pit, Low Grade Ore Stockpile and Waste Dump Water Balance

Inflows:

- Precipitation: Precipitation/runoff from the waste dumps and plant site areas assumed an annual runoff coefficient = 0.7, while a runoff coefficient = 1.0 was used for the open pit. A runoff coefficient = 0.4 was used for short events.

- Groundwater: Open pit groundwater flow will be collected in the base of the open pit and returned to the plant site. The estimated groundwater inflow was assumed to vary from 50 m^3/hr for the Operation Case to 100 m^3/hr for the Ultimate Pit case.

- Ore void transport water from the open pit to the mill is estimate as 3% moisture content or 39 m^3/h (Wardrop supplied estimate).

- Morrison Lake fresh water intake requirements were determined by Wardrop to be 88 m^3/hr, which includes: pump gland water (43 m^3/hr); mill cooling water (45 m^3/hr); reagent mix water (3 m^3/hr); and potable water (3 m^3/hr).

- TSF water reclaim, which will be controlled, as required, to balance the water requirements for the plant site.

Outflows:

- Tailings transport water was based on 30,000 tpd solids at 34% solids by weight.
- Ore concentrate (Wardrop supplied estimate) water loss is estimated to be 1 m^3/hr.
- Potable water use (Wardrop supplied estimate) of 3 m^3/hr.

The detailed worksheets for the monthly water balance models for the various stages of mine operations are included in Appendix IX and the results are summarized in Table 9.1. The water balance indicates that there will be a net surplus of water in approximately Year 8 of operations, which will need to be stored in the tailings impoundment until closure. The runoff collection from the mine area will be monitored and, if the quality meets discharge requirements, it may be released to reduce the storage requirements in the TSF. The estimated accumulated volume of water at the end of mine life is approximately 14 Mm^3, which is equivalent to approximately 4 m of additional dam height for the Ultimate Dam.

Table 9.1 Water Balance Summary

Parameters	Average Annual Flows (m^3/hr)			
	Starter	Operations	Pre-closure	Closure
	Year 2	Year 10	Year 21	
TSF Water Inputs (m3/hr)				
Whole Tailings Water (direct discharge)	1,062	1,062	1,062	-
Cyclone Overflow Water (85% Operational)	1,416	1,416	1,416	-
Cyclone Underflow (sand) Water (85% Operational)	67	67	67	-
Precipitation on pond	76	261	318	286
Runoff from undiverted catchment	102	19	19	167
Excess Plantsite Catchment runoff	-	-	-	-
Seepage reclaim	4	4	4	-
Subtotal	**2,727**	**2,829**	**2,886**	**454**
TSF Water Losses (m3/hr)				
Pond evapor.	54	184	225	202
Tailing voids	211	190	171	-
Cyclone Overflow Voids	230	189	170	-
Cycloned Sand Voids	30	30	30	-
Seepage	10	10	10	10
Pump to Cyclowash	109	109	109	-
Water reclaim to process plant	2,083	2,033	1,983	-
Subtotal	**2,727**	**2,746**	**2,698**	**212**
Plant Area Water Inputs (m3/hr)				
Water reclaim to process plant (see above)	2,083	2,033	1,983	n/a
Open Pit Dewatering	-	50	100	n/a
Runoff from undiverted catchment	89	134	132	n/a
Seepage from diversion ditch	-	-	-	n/a
Ore Void Water (3% MC)	39	39	39	n/a
Make-up to Freshwater Tank	87	87	87	n/a
Subtotal	**2,366**	**2,410**	**2,409**	**n/a**
Plant Area Water Losses (m3/hr)				
Tailings Transport Water	2,405	2,405	2,405	n/a
Concentrate Load Out	1	1	1	n/a
Freshwater to Potable Water	3	3	3	n/a
Subtotal	**2,409**	**2,409**	**2,409**	**n/a**
Net Balance (m3/hr)	**-**	**83**	**188**	**242**

Note: The net balance (surplus) will be stored in the TSF.

9.3 TSF - Water Management Components

9.3.1 Seepage Recovery Ponds and Dam

Seepage recovery ponds and collection dams will be built downstream of both the Main Dam and the North Dam of the TSF. The Seepage Dams are located approximately 300 m downstream of the toe of the Main and North Dams (refer to Drawing D-1002). The dams will intercept seepage water, local runoff and residual water from the cyclone sand construction of the dams. Water captured in the pond will be recycled to the tailings impoundment.

Seepage Ponds Storage and Water Reclaim Requirements

The Seepage Ponds inflows include the following:

- Surface water: Diversion ditches will be constructed to minimize the catchment area of the seepage recovery ponds, which will reduce their total catchment area from approximately 80 ha to 40 ha. The basis for the snowmelt inflow is based on the 2 year return period flow presented in Table 4.15 for combined snowmelt and precipitation, a catchment of 40 ha and a runoff coefficient of 0.5.

- The cyclone sand water is calculated as the difference between the cyclone underflow water (72% solids by weight) and the residual moisture in the cycloned sand (85% solids by weight).

A summary of the inflow sources and potential flows into the Seepage Ponds is presented in Table 9.2.

Table 9.2 Summary of Inflow Components to Each Seepage Pond

Source	Inflows							
	Daily		Weekly		Monthly		Annual	
	Flow (m^3/hr)	Volume (m^3)	Flow (m^3/hr)	Volume (m^3)	Flow (m^3/hr)	Volume (m^3)	Flow (m^3/hr)	Volume (m^3)
Seepage	4	96	4	670	4	2,800	4	35,000
Cyclone sand water	64	1,540	64	10,800	64	45,000	40	360,000
Snowmelt and/or precipitation	200	4,800	92	15,400	51	37,000	13	110,000
Total	268	6,436	160	26,870	119	84,800	57	505,000

The maximum precipitation event is the snowmelt/precipitation combination, which is generally restricted to the spring freshet. Accordingly, the pumping rate of 120 m^3/hr has been selected to manage the monthly "wet" month inflow, while the overall average reclaim rate, over the year, is 57 m^3/h. In addition to providing this capacity we recommend a storage volume of 50,000 m^3 to accommodate wetter months and/or the breakdown of pumps

The flood storage criterion for the Seepage Recovery Ponds is to store the 200 year, 7 day inflow from a rain/snowmelt event, which is equivalent to 77 mm of precipitation. The catchment area for both seepage ponds, assuming the local diversions fail, is approximately 80 ha, which assuming a runoff coefficient of 0.8 for the event, equates to a flood storage volume requirement of 46,000 m^3. The flood storage volume will be additional to seasonal storage requirements for the seepage pond, for a total storage requirement of approximately 100,000 m^3. Larger flows will be discharged over the emergency spillway.

Seepage Dam Requirements

A dam elevation of 890 m is required for the South Seepage Dam to provide 100,000 m^3 of storage, with a freeboard of 1.5 m over the spillway invert; a dam elevation of 965 m is required for the North Seepage Dam.

The dams will be constructed as homogenous glacial till fill dams, with a downstream blanket drain, with upstream and downstream slopes of 2.5H:1V.

Seepage Pond Intake

The seepage pond intake will consist of a concrete intake "box" with a trash rack, leading to the pump house.

9.3.2 Tailing Facility Flood Management

The flood management criteria for the TSF are presented in Section 2.5 (Design Floods) of this report and summarized as follows:

- Storage of the 7 day PMF which is approximately 3.4 Mm^3. This storage criteria would also be equivalent to storing a 2 week-200 year rain on snow event; and
- Freeboard of 1 m above the stored flood level.

Larger events could be handled with an emergency response plan, which will include a plan for construction of a bypass spillway.

9.3.3 Changes in Water Flows

The TSF will modify flows in the creek downstream of the Main Dam and to a lesser extent, the drainage in the North Dam area. The estimated change in flows was calculated on the basis of the modified catchment areas and a runoff coefficient of 0.5. The catchment area reduces from 13.4 km^2 to 7.5 km^2. The change in flow applies to the lower reach of the creek, in the area of the alluvial fan near the confluence with Morrison Lake. The projected changes are shown in Table 9.3, and generally result in an operational reduction in flow to approximately 55% of the baseline flow. Upon closure of the mine, the flows should be re-established to near baseline flow conditions.

Table 9.3 Estimated Flow Reductions in Creek Downstream of the Main Dam due to the TSF

Flow Condition		Oct (%)	Nov (%)	Dec (%)	Jan (%)	Feb (%)	Mar (%)	Apr (%)	May (%)	Jun (%)	Jul (%)	Aug (%)	Sep (%)
	(%)	1.7	2.5	1.9	2.0	1.8	1.5	8.4	41.2	23.8	7.8	3.0	4.5
Baseline	(m^3/s)	0.023	0.034	0.026	0.027	0.024	0.021	0.115	0.564	0.260	0.106	0.040	0.062
With TSF	(m^3/s)	0.013	0.019	0.014	0.015	0.013	0.012	0.063	0.310	0.143	0.058	0.022	0.034

The change in water flows to the drainage northward to Nakinilerak Lake is small; the immediate drainage to the pond adjacent to Nakinilerak Lake will see a catchment area reduction from approximately 3.5 km^2 to 3.1 km^2, or about a 10% reduction in flow. However, Nakinilerak Lake, the main receiving water body, has a very large catchment area (i.e., >1,000 ha) and the impact from the TSF would not be measureable.

Surface runoff above the Waste Rock Dump will be diverted to the north and south to avoid contact with active mine areas. The diversion towards the north will increase the flow in the primary creek north of the plantsite by less than 5% (due to its large natural catchment). The surface flows towards the southeast of the open pit will be reduced by approximately 20% due to collection of Waste Rock Dump runoff and seepage in this catchment.

9.3.4 Morrison Lake – Fresh Water Intake

The fresh water supply will be from Morrison Lake as shown on Drawing D-1002. The fresh water flow requirement is approximately 88 m^3/hr (24 L/s). The intake will consist of a 1 m diameter perforated corrugated steel pipe (CSP) placed horizontally in a trench near the shoreline, approximately 2 m below average lake level. The intake will be screened to prevent fish ingress. The CSP pipe and trench will be backfilled with drain gravel and lead to a 1.5 m diameter CSP riser pipe. The riser pipe will be protected with a small shed and will house the pump and control panel.

9.3.5 TSF Diversion Ditches and Spillways

Diversion Ditches

To minimize the requirements for water storage in the TSF it is necessary to maximize the amount of surface water diversion. Accordingly, diversion ditches will be constructed for the Year 4, Year 8 and Year 20 TSF impoundments to divert approximately 9 km^2, 5 km^2 and 4.4 km^2, respectively. The ditches are designed to pass the 100 year average snowmelt/runoff flows of 4.4 m^3/s (based on British Columbia Flood Maps). The main diversion ditches, on the east side of the TSF, will typically be 2 m wide and 1.1 m deep. Riprap will be required on slopes steeper than 1%.

Seepage Recovery Pond Spillways

Emergency spillways are required for the Seepage Recovery Dams for dam safety protection. The spillway is sized for the 1000 year return period peak flow of 1.5 m^3/s. The spillway will consist of a 2.0 m wide open channel with 2H:1V side slopes and a 0.5 m water depth plus 0.3 m freeboard. The spillway would be at a 0.5% slope.

TSF Closure Spillway

The closure spillway for the TSF will be constructed after completion of mining operations. The spillway will be located in the left (east) abutment of the Main Dam and will consist of a channel excavated in bedrock and exiting into an existing drainage channel. The spillway is designed for the maximum probable flood peak flow, assuming attenuation of the flood event within the impoundment. The estimated peak flood flow is 4.5 m^3/s, which will require a spillway 2 m wide and 0.7 m deep.

9.4 Plant Site, Open Pit and Waste Rock Dumps

9.4.1 Sediment Control

Sediment ponds will be constructed downstream of the overburden and low grade ore stockpiles as shown on Drawing D-1201. The design of the ponds is based on the following:

- 1:10 year return period storm (based on British Columbia Flood Maps);
- 1:200 year return period for emergency spillway for the pond;
- Settling particle size = 0.03 mm;
- Minimum retention time = 40 minutes;
- Drawdown time = 48 hours; and
- Length of pond = 5 times the width.

The typical sediment pond size is 60 m in length and 12 m wide, which will accommodate a runoff area of 0.24 km^2, which has a 1:10 year peak flow of 0.31 m^3/s and a spillway peak flow of 0.48 m^3/s. The spillway overflow outlet would be 3 m wide and 0.2 m deep. The pond dyke would be 1.6 m high and the 1:200 year flood can be routed with 0.2 m freeboard.

9.4.2 "Clean" Water Diversion Ditches

Diversion ditches will be constructed to divert, as much clean surface water as possible. The two main diversion ditches include:

- Diversion Ditch A, located along the uphill side of the waste rock dump, will divert approximately 2.4 km^2. The design peak 100 year flow, based on British Columbia Flood Maps, is 1.51 m^3/s, requiring a ditch 2 m wide and 0.6 m deep. Riprap will be required on slopes steeper than 1%.

- Diversion Ditch B, located along the uphill side of the Overburden and Organic Sediment Storage Pile, will divert approximately 1 km^2. The design peak 100 year flow, based on British Columbia Flood Maps, is 0.82 m^3/s, requiring a ditch 1.5 m wide and 0.5 m deep. Riprap will be required on slopes steeper than 1%.

9.4.3 Collection of Contact Water

All contact water, from the waste rock dumps, open pit, low grade ore stockpile and plant site areas, will be collected and recycled to the plant site. Runoff collection sumps will be located along the toe of the waste dumps as shown on Drawing D-1201. Sumps, which are located uphill of the open pit area have been sized to rout the average spring freshet runoff of 610 m^3/hr and will require a minimum sump capacity of 100 m^3. Although the pumping capacity could be reduced, to a minimum of 150 m^3/hr, it would result in more water reporting to the open pit during wet periods.

Sumps, in which an overflow could lead to a release to the environment have been sized for the 1:10 year flow of 680 m^3/hr and will require a minimum sump size of 670 m^3.

9.4.4 Access Road Bridge Crossing

The access road from the plant site to the TSF crosses an existing creek, which will be spanned with a corrugated steel pipe (CSP) culvert. The creek, at this location, has a catchment area of 12.3 km^3 and the 1:200 year peak flow (based on the British Columbia Flood Maps) is 10.6 m^3/s. The culvert could consist of an arched CSP pipe with a 3 m wide base and 2 m height. Alternatively, a bridge crossing could be considered in the detail design phase.

10. ENVIRONMENTAL MANAGEMENT PLANS

10.1 General

The environmental management plans (EMP) will evolve from construction, through operations and to closure. The main emphasis will be on water quality, dam safety and habitat restoration. The main components of the environmental management plans are presented in the following sections.

10.2 Construction Environmental Management Plans

Clearing:

The tailings impoundment, open pit, waste rock dump, low grade ore stockpile, plant site and dam footprint areas will be logged of all merchantable timber. All remaining trees in the tailings impoundment area will be dozed to "near" level, for subsequent burial by tailings. Levelling of the trees will reduce the visual aspect of dead trees sticking out of the impoundment. By leaving the levelled trees *in situ*, as opposed to the traditional grubbing and burning, carbon emissions will be reduced.

The dam footprint and borrow areas will be cleared and grubbed of all vegetation and the "refuse" will be stockpiled in the impoundment area, where it will be covered with tailings.

Organic Material Salvage:

As part of the construction, organic bearing material from the dam footprint and borrow areas will be salvaged and stored in designated stockpiles. The stockpiles will be designed with a limited height to maintain the organic content of the stockpiled soils. Areas identified within the impoundment, which contain high quality organic soil, will be selectively "mined" and stored for final reclamation. The estimated quantity of organic bearing material available, and required for final reclamation of the TSF is summarized in

Table 10.1 and Table 10.2, respectively. Organic material salvage will be carried out progressively as the impoundment and dams are raised over the mine life.

Table 10.1 Estimate of Organic Bearing Material* Sources and Volumes

Source	Area (ha)	Average Thickness (m)	Volume (m^3)
Main dam footprint and seepage recovery pond.	44	0.2	88,000
Borrow areas within impoundment(1,2,3)	17	0.2	34,000
Borrow areas outside impoundment	31	0.2	62,000
North dam footprint and seepage recovery pond	18	0.2	36,000
West dam footprint	8	0.2	16,000
Open pit	108	0.2	216,000
Waste rock and low grade ore piles	210	0.2	420,000
Plant site	40	0.2	80,000
Select sources within impoundment (estimated allowance)	15	0.75	112,500
Total	491		1,064,500

* Note: Organic Bearing Material typically consists of 5 cm of organics and 15 cm of soil.

Table 10.2 Estimate of Topsoil Reclamation Requirements

Source	Area (ha)	Average Thickness (m)	Volume (m^3)
Dam crest and slopes	68	0.2	136,000
Seepage pond areas	20	0.2	40,000
Roads and other disturbed areas	20	0.2	40,000
Progressive reclamation of borrow areas outside of the impoundment	31	0.2	62,000
Progressive reclamation of stockpile areas outside of the impoundment (Overburden and North Organic Overburden)	50	0.2	100,000
Allowance for partial reclamation of areas within the perimeter of the impoundment	44	0.2	88,000
Waste rock piles and low grade or stockpiles	223	0.2	446,000
Plant site	40	0.2	80,000
Total	496		992,000

Sediment Control:

Sediment control during construction will be required to protect the downstream water quality (refer to Section 9.4 of this report). Runoff from disturbed areas will be collected in temporary settling ponds prior to discharge. Construction areas will have perimeter ditches to minimize the quantity of runoff water from disturbed areas. Silt fences will be installed around the perimeter of disturbed areas. In addition, other sediment control measures, such as hay bales or additional silt fences will be used, as required.

Mine development will require draining of Booker Lake and Ore Pond, as discussed in Section 8.3, of this report. Water would be ultimately discharged into Morrison Lake. The discharge water quality will be monitored to ensure it meets discharge water requirements.

Surface water diversion ditches will be constructed around the plant site, waste rock dump, low grade ore stockpile and other disturbed areas.

Dust control during construction, particularly with the mine haul roads, will be controlled by watering with conventional water trucks with a spray bar.

10.3 Operations Environmental Management Plans

10.3.1 Tailings Storage Facility

Surface Water Quality

The TSF will be operated as a closed system with no direct discharge of water. The seepage control ponds, located downstream of the dams, will collect drainage from the dam slopes, excess cyclone sand water and dam seepage. The water will be pumped back into the impoundment. The quality of the seepage and collected water is anticipated to be similar to the tailings process water summarized in Table 10.3, where it is compared to guidelines for drinking water and wildlife habitat.

Table 10.3 Operational Water Quality of the Impoundment

Parameter[1]	British Columbia Water Quality Guideline (Wildlife) (mg/L)	Canadian Drinking Water Guideline (mg/L)	Average Aged Process Water (mg/L)
pH			7.21
Conductivity			98.5
Alkalinity			23
Acidity			< 2
SO_4	None Proposed	≤ 500	13.1
Hg			< 0.0001
Ag	None Proposed	None Proposed	0.00004
Al[1]	5	0.1	0.0115
As	0.025	0.025	0.00025
Co	1	1	0.000074
Cd	0.08	0.005	0.0000065
Cr	0.05	0.05	< 0.0005
Cu	0.3	0.5	0.001
Fe	None Proposed	≤ 0.3	< 0.01
Mg			3.88
Mn	None Proposed	≤ 0.05	0.01785
Mo[2]	0.052	0.25	0.00262
Na	None Proposed	≤ 200	0.34
Ni	1	None Proposed	0.00035
Pb	0.1	0.01	0.00002
Sb			0.000285
Se[3]	0.0303	0.01	< 0.001
Sn			0.00003
Si			0.255
Ti			0.0001
V			0.000145
Zn	2	≤ 5	0.001

Notes:
1. All parameters are dissolved concentrations except for Al, which is total.
2. None proposed, livestock watering limit shown.
3. Selenium is based on recommended concentrations for waterfowl, not fish.

Water quality of the impoundment will be monitored monthly during operations. The process water quality could change over the life of the impoundment either due to changes in ore mineralogy, mill process, or due to metalloid build-up resulting from the continual recycling of process water.

If water quality is found to significantly exceed wildlife or drinking water guidelines, it may be necessary to implement management plans to monitor wildlife use, and possibly limit access to wildlife and to post warning signs.

Groundwater

The majority of the tailings impoundment is blanketed with a low permeability glacial till. In addition, the tailings will have a low permeability and the dams are designed to limit seepage. A seepage collection system will return seepage from the toe of the dam back to the impoundment. Nonetheless, some portion of the seepage water could report downstream of the Seepage Recovery Pond.

As presented in Table 7.4, the estimated impoundment foundation seepage downstream of the Main Dam is 0.6 L/s, with an upper bound of 3.0 L/s. This seepage could report to the deeper groundwater regime and potentially discharge at some point downstream or into Morrison Lake. Water that may eventually report to Morrison Lake will have low metal concentrations in the seepage and any potential effect will be substantially mitigated by the huge dilution effect of the lake.

There is a low potential risk that seepage from the TSF could eventually report to the creek downstream of the Main Dam, and modify the water quality. The potential effect could be expected to take a very long time due to the adsorption capacity of the glacial tills.

Similarly, the estimated impoundment foundation seepage downstream of the North Dam is 0.4 L/s, with an upper bound of 2.0 L/s. This seepage is expected to report to the deeper groundwater and may eventually report to Nakinilerak Lake. Water that may eventually report to Nakinilierak Lake will also have low metal concentrations in the seepage and any potential effect will be substantially mitigated by the huge dilution effect of the lake.

The groundwater quality will be monitored with existing shallow and deep groundwater monitoring wells located downstream of the TSF, in additional to the surface water quality monitoring in the downstream receiving creeks.

Geochemistry – Cycloned Sand

The cycloned sand is predicted to be NAG and ongoing testing will be carried out during operations to confirm that the cycloned sand is NAG and that there is a low risk of acid rock drainage (ARD). Static ABA tests will be carried out weekly or bi-weekly to confirm the consistency of the cycloned sand. In addition, a larger scale field leach pad will be constructed with several tailings samples to confirm the longer term leaching properties.

If the cycloned sand does not meet the requirement for a NP:AP ratio > 3.0 for geochemistry it will not be placed in the dam and, therefore, dam materials would then be sourced from borrow areas within the impoundment.

Geochemistry – fine tailings and total tailings

Both the fine tailings (cyclone overflow) and the total tailings are currently predicted to have a low risk of ARD. Nonetheless, the base case strategy is to maintain the tailings in a saturated state to mitigate potential ARD. The geochemistry of the tailings will be monitored bi-weekly or monthly with ABA tests.

Several large scale field leach pads will be constructed to monitor the longer term ML-ARD properties of the tailings to predict closure conditions. Longer term column tests will also be carried out to confirm the expected negligible effect of metal leaching into a surface water cover.

Dam Safety

The dam foundations contain soft to stiff glacial till and the behaviour of the glacial till in response to dam construction will be monitored to confirm design parameters for the stability assessment of the dams. The monitoring will include:

- Electric piezometers will be installed to monitor pore pressure generation and dissipation due to loading by the dam fill and the results will be used to confirm the design assumptions and the Factors of Safety for stability. If required, the dam slopes would be flattened or steepened, depending on the actual pore pressure response; and
- Several inclinometers will be installed in the foundation and extended vertically through the dam fill. The inclinometers will measure horizontal displacement with depth.

10.3.2 Mine Area

Geotechnical Stability

The rate of construction of the waste rock and low grade piles is slow, which should assist the dissipation of pore pressures in the foundation glacial tills. Nonetheless, the geotechnical stability of the waste rock dumps will be confirmed with electric piezometers and surface survey monuments.

Water Quality

Runoff and seepage waters from the waste rock, low grade ore stockpile and open pit areas will be collected and recycled to the milling process. However, leaching of the waste rock, over a period of time, may deplete the available alkalinity and become acid generating. The water quality of the runoff and seeps from the waste rock piles will be monitored during operations, on at least a quarterly basis. If the water quality increases in both pH and metal concentrations it will be necessary to increase the lime consumption in the milling process. This could also influence the tailings water quality.

Potentially non-acid generating waste rock will be identified and preferentially placed in drainage channels and towards the south end of the waste rock dump. NAG rock will be identified by testing blast-hole cuttings and by geologic interpretation of non-sulphide rock types. Testing will consist of standard acid base accounting tests and sulphide determination.

Surface water quality and groundwater quality monitoring stations will be installed to monitor potential effects in the mine area.

Progressive Reclamation

As far as practical, progressive reclamation of all disturbed areas will be carried out during operations. Disturbed areas will be graded to control slopes and runoff; the areas would be covered with a 20 cm thick organic bearing material and reclaimed. Test plots will be carried out to confirm vegetation types and success.

10.4 Specifications and Manuals

As part of good tailings management practice, the following manuals and reviews will be carried out.

- An operations, maintenance and surveillance manual (OMS) will be prepared prior to operations. The manual will follow the Mining Association of Canada Guidelines.
- An emergency preparedness plan (EPP) and emergency response plan will be prepared prior to operations.

- Construction specifications will be prepared for all construction works. A QA/QC program will be carried out for all construction and "As-Constructed" reports will be prepared all construction works.

- An annual Dam Safety Review will be carried out.

11. CLOSURE PLAN

11.1 Tailings Facility

11.1.1 Closure Plan Components

The objective of the closure plan is to ensure long term physical, geochemical and ecological stability of the tailings facility and mine waste rock piles, and disturbed mine areas. The tailings facility will be closed as a large shallow lake with the dam slopes and disturbed areas reclaimed. The general philosophy of closure design is to minimize, as far as practical, the requirements for ongoing maintenance and the risks associated with geochemical and physical stability. A commonly accepted period for closure design is 1,000 years, however longer term potential concerns should be considered in the design of closure facilities.

The main components of the closure plan are summarized as follows:

Dam Safety - Geotechnical

The dams are designed for geotechnical stability for the maximum credible earthquake (MCE). In addition, the long term factors of safety will be higher than the operational factors of safety due to ongoing consolidation of the glacial till foundations. The dam design section is designed as a “simple” robust structure that does not rely on long term performance of filters and drains and can accommodate significant deformations without leading to “failure” of the structure. The dam instrumentation will continue to be read into closure until foundation pore pressures have stabilized and any surface or sub-surface ground movement changes are negligible.

A closure concern is the potential for long term erosion of the dam through natural processes. Reclamation of dam slopes with a vegetative cover will mitigate the erosion risk. In addition, the final dam slopes will have benches for runoff management and will be designed for extreme precipitation events, without ongoing erosion.

The dam safety risk due to water will also be mitigated with placement of a saturated fill zone on the upstream side of the final dam crest. This zone will maintain the permanent lake away from the dams, thereby providing an additional factor of safety in case of any dam incidents. For planning purposes, the zone has provisionally been set at 100 m wide for the Main Dam, 75 m wide for the North Dam and 50 m wide for the West Dam. The width of the zones has been selected to be proportional to the dam height.

The Seepage Recovery Dams will be decommissioned after the seepage water quality has "stabilized" and meets discharge requirements.

Dam Safety - Water Management

The management of flood waters is a critical factor in long term safety of the structure. Overtopping of the dam due to failure of the spillway could lead to dam failure and a catastrophic release of tailings and water. While there are no national or international guidelines for closure spillways, KCBL's philosophy is to construct, as far as practical, a "failsafe" spillway and, if possible, a backup spillway to provide a high level of redundancy. The spillway will be designed to pass the peak flow from a probable maximum flood (PMF) and will be excavated in bedrock. The Morrison TSF provides several spillway locations located in rock and away from the Main Dam. The recommendation is to construct the main closure spillway in the left abutment of the Main Dam to convey water flows for the downstream aquatic environment. In addition, an emergency secondary closure spillway will be constructed in the right abutment of the North Dam, at a higher elevation, to provide a secondary safety release of water in the event of failure of the main spillway.

Tailings Lake Water Quality

On closure, the impoundment will contain up to 14 Mm^3 of residual process water. Dilution of this water, with natural surface water inflow, will eventually change the water quality to baseline concentrations, however, this could take up to 5 to 10 years. Although residual process water contains very low levels of metal concentrations, pond water will be monitored regularly to confirm the pond water meets water quality requirements before allowing discharge through the emergency spillway. If discharge is not permitted, the annual surplus water accumulating in the impoundment, which would be approximately 1 Mm^3/yr, would be pumped to the open pit, using the existing pump barge and reclaim water pipeline, until natural dilution of the tailings lake water meets discharge criteria.

11.1.2 TSF Closure Environmental Management Plans

Dam Safety

The dam instrumentation will continue to be read at an appropriate frequency until steady state conditions have been reached. Monitoring on an annual basis with ground inspections or helicopter surveys would identify if any significant issues, such as spillway blockage or dam erosion, were being initiated.

The requirements of the Canadian Dam Association (2007) are that the dam would require a Dam Safety review every 5 years by a qualified professional engineer.

The quality of the reclamation would be monitored annually until stable vegetation has been established on the dam slopes.

Water Quality

The impoundment water quality and downstream compliance stations will continue to be monitored monthly until steady state conditions have been achieved and the water quality meets discharge water quality requirements. The potential for resuspension of fine tailings during wind events will be monitored. Erosion and sediment control of runoff from the dam slopes would be monitored. Annual monitoring would continue for 5 to 10 years, after which the frequency would be reduced to 5 years and completed as part of the Dam Safety Review.

Ecosystem Stability

The lake will, over a long period of time, accumulate benthic matter and begin to establish aquatic habitat. The monitoring of the ecosystem stability should be carried out as part of the Dam Safety Review every 5 years.

11.2 Mine Area – Closure Plan

11.2.1 Closure Plan Components

Covers and Reclamation

Closure of the PAG waste rock dump will be carried out after decommissioning of the plant site and associated structures. The PAG waste rock dump will be re-sloped to a minimum inter-bench slope of 2.5H:1V, with 5 m wide benches every 20 m in height for an overall slope of 2.75H:1V. A soil cover, consisting of a 1 m thick low permeability barrier, using glacial till, and a 0.2 m organic material layer will be placed over the re-graded surfaces. The surface will be reclaimed with suitable vegetation.

The plant site buildings will be decommissioned and, as far as practical, salvaged for reuse. Inert industrial materials will be placed in the open pit. Any contaminated soils or contaminated industrial materials will be disposed in an engineered landfill or shipped to a suitable waste management facility. Concrete structures will be demolished to ground level; floor slabs will be broken up and left *in situ.* The disturbed areas will be covered with organic bearing material and reclaimed.

Water Quality

The PAG waste rock may eventually become acidic and leach metals. The volume of leachate water will be partially controlled by the cover; however, approximately 10% of precipitation falling on the waste rock dump may infiltrate the cover and emerge as contaminated seepage water. Estimates of potential water quality of the seepage are presented in Table 6.9 of this report. Based on a waste dump surface area of 150 ha, the average seepage rate could be in the order of 3 L/s. The open pit may also become acidic. Consequently, a water treatment plant may be required to treat mine water.

12. CONSTRUCTION: SCHEDULE AND QUANTITIES

12.1 Construction Overview

Construction of the waste management facilities will be conducted in three phases: pre-production, production and closure. Pre-production will include drainage of Booker Lake and Ore Pond, stripping of the open pit and construction of the plant site and associated facilities. The Starter Dam, South Seepage Dam and access roads and the pumping and piping systems for the tailings facility will also be constructed during the pre-production phase.

During production, the tailings dams will be raised on an ongoing basis and low grade ore will be temporarily stockpiled for milling near the end of the mine life.

On closure, the tailings dams will be reclaimed and the tailings pond will become a large lake, with permanent spillways to convey surface water and manage extreme floods. The plant site facilities will be decommissioned and the areas reclaimed. The potentially acid generating waste rock dump will be covered with low permeability soil and revegetated. The open pit will, over a period of time, fill with runoff water and groundwater. A provision for a water treatment plant is included in the design to be able to treat potentially contaminated water from the open pit or seepage from the waste rock dumps.

12.1.1 Pre-Production Phase

During the pre-production phase of construction, all mine infrastructure required to begin mine production will be constructed. The construction of waste management facilities will include:

<u>Tailings Storage Facility</u>

- Clearing and grubbing of the TSF impoundment, dam foundations, access roads and all other construction areas listed below;

- Construction of tailings and reclaim water pipeline access roads, and bridge upgrade;
- Stripping of foundations for the Starter Dam and South Seepage Dam;
- Development of glacial till borrow area within the TSF impoundment to provide embankment fill for the dams;
- Construction of the Main Starter Dam embankment using overburden waste from open pit development, glacial till fill from borrow areas within the TSF impoundment and granular fill from the north sand and gravel borrow area;
- Construction of the South Seepage Dam and installation of the seepage reclaim pump and pipeline returning over the crest of the Starter Dam;
- Installation of the tailings pipeline and booster pump station, and excavation of emergency backflow ponds;
- Installation of the reclaim water pumping barge and return pipeline to the process plant;
- Installation of the tailings distribution pipeline from the Main Dam left abutment to the far side of the Starter Dam;
- Installation of cyclone Cyclowash Pump Station No. 1 and water pipeline along the crest of the Starter Dam;
- Assembly and placement of the skid-mounted cyclone (x2) on the Starter Dam crest; and
- Construction of the TSF clean water diversion ditch.

<u>Open Pit Area</u>

- Clearing and grubbing of low grade ore stockpile and waste rock dump foundation areas up to approximately 845 m elevation, and overburden dump, organic sediment storage and organic bearing material stockpiles;
- Draining and dredging of Booker Lake and Ore Pond;

- Excavation of unsuitable foundation soils (waste rock dump, overburden dump, overburden and organic sediment storage);
- Stockpiling of organic bearing material from open pit and plant site development;
- Construction of the waste rock dump clean water diversion ditch;
- Construction of the fresh water pump station and pipeline from Morrison Lake to the plant site;
- Construction of the dump and stockpile seepage collection ditches, and excavation of the pumping sumps; and
- Development of granular borrow areas.

12.1.2 Production Phase

Waste management facilities will be expanded during the production phase of construction to meet mine production requirements and construction will include:

Tailings Storage Facility

- Year 1
  - Construction of the tailings pipeline and access roads to the North Dam;
  - Construction of an approximately 5 m-high starter dyke at the North Dam using glacial till fill from local borrow areas;
  - Construction of the north seepage collection dam embankment, and installation of a seepage reclaim pump and pipeline returning over the crest of the North Dam;
  - Installation of the cyclone Cyclowash Pump Station No. 2 and water pipeline to the North Dam; and

    - On-going hydraulic placement of cycloned tailings sand in the downstream shell of the Main Dam and, starting Year 2 in the North Dam downstream shell.

- (Year 3) Extension of access roads and initiation of incremental foundation preparation and raising of the West Dam;
- (Year 4) Extension of tailings distribution pipeline to the West Dam;
- On-going clearing and grubbing of the TSF impoundment, dam foundations and access roads;
- On-going stripping and preparation of tailings dam foundations;
- On-going glacial till borrow area development within and outside the TSF impoundment to provide embankment fill for the tailings dams; and
- On-going extension of seepage collection ditches.

<u>Open Pit Area</u>

- On-going clearing and grubbing of the waste rock dump foundation;
- On-going stockpiling of organic bearing material from open pit development;
- On-going granular borrow area development, as needed; and
- Progressive reclamation of waste rock dump slopes with a low-permeability soil cover.

12.1.3 Closure Phase

The closure phase of construction will occur at the end of mining and processing operations. However, some closure construction activities will be initiated during operations to take advantage of the availability of materials or to reduce environmental impact. The closure construction activities will include:

Tailings Storage Facility

- Decommissioning and reclamation of tailings and water pipelines and pump stations;
- Placement of a re-vegetated soil cover on the downstream face of the tailings dams;
- Decommissioning and reclamation of the seepage collection dams when water quality results allow for release to the environment;
- Reclamation of borrow areas and stockpiles outside the TSF impoundment;
- Decommissioning of seepage collection and clean water diversion ditches;
- Construction of a saturated NAG rockfill cover over the tailings beaches adjacent to tailings dams; and
- Construction of a closure spillway on the left abutment of the Main Dam and a backup spillway in the right abutment of the North Dam.

Open Pit Area

- Reclamation of temporary stockpile and plant site foundations;
- Decommissioning and reclamation of the freshwater pipeline and pump station;
- Reclamation of borrow areas;
- Completion of waste rock dump slope reclamation;
- Construction of the closure seepage collection ditches to provide drainage of PAG seepage to the open pit;
- Reclamation of overburden dump slopes; and
- Construction of a water treatment plant to treat pit lake discharge (if required to meet water quality guidelines).

12.2 Annual Fill Requirements

Tailing dam construction will be carried out over the life of the mine, on an annual basis, to provide ongoing storage of the mine tailings. The main construction materials for the dam include: glacial till (for the Starter Dams and central core), random fill (for dam shells), and cycloned sand for the downstream shells and upstream support zones of the dam. The annual fill requirements on shown on Figure 12.1 and the cumulative total volumes are shown in Table 12.1. In addition, sand and gravel fills are required for the blanket filter/drain zone beneath the downstream cycloned sand zone.

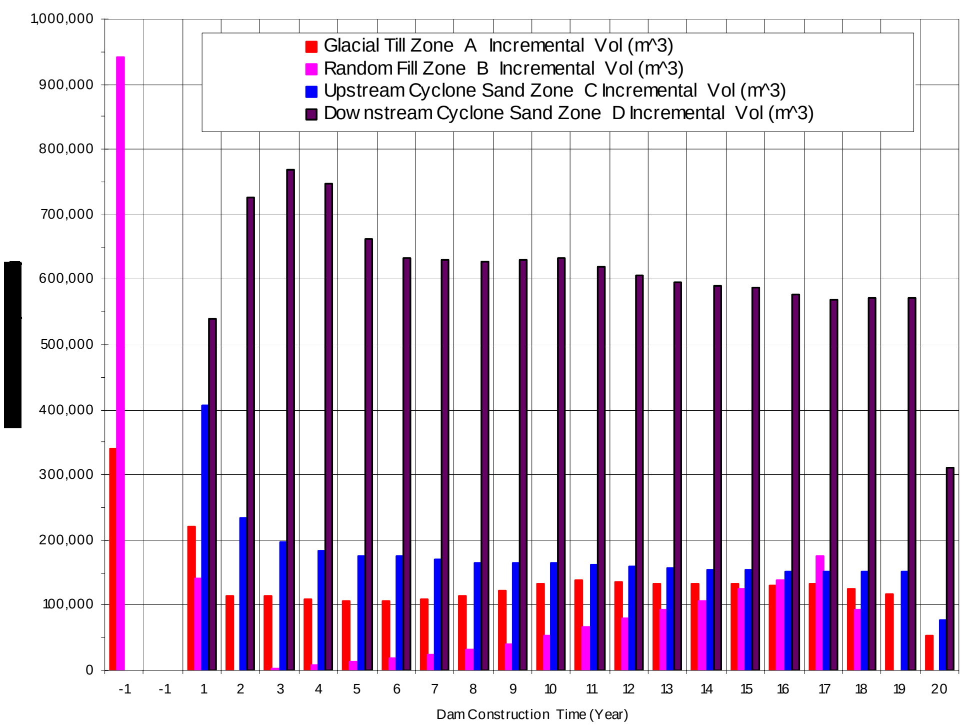


Figure 12.1 Annual Tailings Storage Facility Fill Requirements

Table 12.1 Total Tailing Dams Fill Volumes

Year	Cumulative Volumes (Mm^3)			
	Zone A Glacial Till	Zone B Random Fill	Zone C Upstream Cyclone Sand	Zone D Downstream Cyclone Sand
-2	0.34	0.94	0	0
-1	0.34	0.94	0	0
1	0.56	1.08	0.41	0.54
2	0.68	1.08	0.64	1.27
3	0.79	1.09	0.84	2.03
4	0.90	1.10	1,02	2.78
5	1.01	1.11	1.20	3.44
6	1.11	1.13	1.37	4.08
7	1.22	1.15	1.54	4.71
8	1.34	1.18	1.71	5.34
9	1.46	1.23	1.84	5.97
10	1.60	1.28	2.04	6.60
11	1.73	1.34	2.20	7.22
12	1.87	1.42	2.36	7.83
13	2.00	1.51	2.52	8.42
14	2.14	1.62	2.67	9.01
15	2.27	1.75	2.83	9.60
16	2.40	1.89	2.98	10.18
17	2.51	2.06	3.13	10.75
18	2.66	2.15	3.28	11.32
19	2.77	2.14	3.43	11.89
20	2.83	2.14	3.51	12.20

12.3 Bill of Quantities and Cost Estimate

A Bill of Quantities for pre-production, operations and closure has been developed as the basis for the cost estimate for the project and these are shown in the following tables.

Expressed in 2008 CAD$

Dated - January 9, 2009

Table 12-2 Waste Management Facilities Capital Cost Estimate

KLOHN CRIPPEN BERGER

Capacity - 30,000tpd

WBS Code	Area, Item Description & Equipment Number		Measure Qtty	M Unit	Unit Mhrs	Total Mhrs	Labour Rate	Labour Total Labour	Unit Matls	Materials Total Materials	Unit C Equipmt	Total Constrctn Equipment	Unit Proc Equipmt	Total Process Equipment	Unit Cost	Total Capital Costs	Initial Capital Costs	Sustaining Capital Costs	Sustaining Capital Year
1	**Tailings Storage Facility**																		
1.1	**Site Preparation**																		
1.1.1	**Clear**	Impoundment area	540.9	ha	17.28	9,349	$ 81	$ 757,260	$ -	$ -	$ 4,200.00	$ 2,271,780	$ -	$ -	$ 5,600	$ 3,029,040	$ 840,000	$ 2,189,040	Year 1 to 20
1.1.2	**Clear and Grub**	Dam footprint	72.6573	ha	17.28	1,256	$ 81	$ 101,720	$ -	$ -	$ 4,200.00	$ 305,161	$ -	$ -	$ 5,600	$ 406,881	$ 64,222	$ 342,659	Year 1 to 20
1.1.3	**Strip and Stockpile Organic Bearing Material**	Dam footprint (~0.2m average organic depth)	145314.6	m3	0.02	2,243	$ 81	$ 181,643	$ -	$ -	$ 3.75	$ 544,930	$ -	$ -	$ 5	$ 726,573	$ 114,682	$ 611,891	Year 1 to 20
1.1.4	**Excavate Unsuitable Soils**	Organic zones in D-1009 (assume 3 m depth, 2 km haul)	75000	m3	0.02	1,157	$ 81	$ 93,750	$ -	$ -	$ 3.75	$ 281,250	$ -	$ -	$ 5	$ 375,000	$ -	$ 375,000	Year 1 to 20
1.1.5	**Proof Roll Embankment Footprint**		726573	m2	0.00	561	$ 81	$ 45,411	$ -	$ -	$ 0.19	$ 136,232	$ -	$ -	$ 0	$ 181,643	$ 28,671	$ 152,973	Year 1 to 20
1.1.6	**Borrow Areas - Clear and Grub**	Till borrow (avg.0.2m organic+ 1.7m till)	41.3438	ha	17.28	715	$ 81	$ 57,881	$ -	$ -	$ 4,200.00	$ 173,644	$ -	$ -	$ 5,600	$ 231,525	$ 84,525	$ 147,000	Year 1 to 20
1.1.7	**Borrow Areas - Stockpile Organic Bearing Material**	~0.2m average organic depth	82687.6	m3	0.01	766	$ 81	$ 62,016	$ -	$ -	$ 2.25	$ 186,047	$ -	$ -	$ 3	$ 248,063	$ 90,563	$ 157,500	Year 1 to 20
1.1.8	**Borrow Area - Misc. Ditching**	Assume 100 m of ditch per hectare	41.3438	ha	2.47	102	$ 81	$ 8,269	$ -	$ -	$ 600.00	$ 24,806	$ -	$ -	$ 800	$ 33,075	$ 12,075	$ 21,000	Year 1 to 20
1.1.9	**Temporary Haul Roads to Borrow Areas**	Allowance (assume 4 km of haul road)	1	LS	1,543.21	1,543	$ 81	$ 125,000	$ -	$ -	$ 375,000.00	$ 375,000	$ -	$ -	$ 500,000	$ 500,000	$ 250,000	$ 250,000	Year 1 to 20
1.1.10	**Site Dewatering, Sediment Control, and Drainage**	Allowance	1	LS	308.64	309	$ 81	$ 25,000	$ 5,000.00	$ 5,000	$ 70,000.00	$ 70,000	$ -	$ -	$ 100,000	$ 100,000	$ 50,000	$ 50,000	Year 1 to 20
															Subtotal	$ 5,831,800	$ 1,534,738	$ 4,297,063	
1.2	**Dams (Main, North and West)**																		
1.2.1	**Excavate Cutoff Trench - All**	Assume 3 m x 5 m	82875	m3	0.02	1,279	$ 81	$ 103,594	$ -	$ -	$ 3.75	$ 310,781	$ -	$ -	$ 5	$ 414,375	$ 97,500	$ 316,875	Year 1 to 20
1.2.2	**Zone A - Starter Dam (from Borrow)**	Load, haul, place, compact, includes filling the cutoff trench	359433	m3	0.02	8,875	$ 81	$ 718,866	$ -	$ -	$ 6.00	$ 2,156,598	$ -	$ -	$ 8	$ 2,875,464	$ 2,875,464	$ -	
1.2.3	**Zone A - Annual Raises (from Borrow)**	Load, haul, place, compact	2487803	m3	0.02	61,427	$ 81	$ 4,975,606	$ -	$ -	$ 6.00	$ 14,926,818	$ -	$ -	$ 8	$ 19,902,424	$ -	$ 19,902,424	Year 1 to 20
1.2.4	**Zone B (General Fill) - Starter Dam (from Open Pit Mining)**	Incremental haul, place, compact	600000	m3	0.02	11,111	$ 81	$ 900,000	$ -	$ -	$ 4.50	$ 2,700,000	$ -	$ -	$ 6	$ 3,600,000	$ 3,600,000	$ -	
1.2.5	**Zone B (General Fill) - Starter Dam (from Borrow)**	Load, haul, place, compact	342680	m3	0.02	8,461	$ 81	$ 685,360	$ -	$ -	$ 6.00	$ 2,056,080	$ -	$ -	$ 8	$ 2,741,440	$ 2,741,440	$ -	
1.2.6	**Zone B (General Fill) - Annual Raises (from Borrow)**	Load, haul, place, compact	1210783	m3	0.02	29,896	$ 81	$ 2,421,566	$ -	$ -	$ 6.00	$ 7,264,698	$ -	$ -	$ 8	$ 9,686,264	$ -	$ 9,686,264	Year 4 to 20
1.2.7	**Zone E (Blanket Drain) - All (from Granular Borrow)**	Load, haul, place, compact, + washing (<3% fines)	726573	m3	0.04	26,910	$ 81	$ 2,179,719	$ -	$ -	$ 9.00	$ 6,539,157	$ -	$ -	$ 12	$ 8,718,876	$ 1,376,184	$ 7,342,692	Year 1 to 20
1.2.8	**Zone C - Upstream Cycloned Sand**	Hydraulic placement	3510000	m3	0.003	10,833	$ 81	$ 877,500	$ -	$ -	$ 0.75	$ 2,632,500	$ -	$ -	$ 1	$ 3,510,000	$ -	$ 3,510,000	Year 1 to 20
1.2.9	**Zone D - Downstream Cycloned Sand**	Hydraulic placement with compaction	12204000	m3	0.01	94,167	$ 81	$ 7,627,500	$ -	$ -	$ 1.88	$ 22,882,500	$ -	$ -	$ 2.5	$ 30,510,000	$ -	$ 30,510,000	Year 1 to 20
1.2.10	**Seepage Mitigation - Provisional Sum**	Upstream Till Blanket	1	LS	2,469.14	2,469	$ 81	$ 200,000	$ -	$ -	$ 600,000.00	$ 600,000	$ -	$ -	$ 800,000	$ 800,000	$ -	$ 800,000	Year 1 to 20
															Subtotal	$ 82,758,843	$ 10,690,588	$ 72,068,255	
1.3	**Water Management**																		
1.3.1	**Seepage Collection Dam Fill (South)**	Till from local borrow	250000	m3	0.02	6,173	$ 81	$ 500,000	$ -	$ -	$ 6.00	$ 1,500,000	$ -	$ -	$ 8	$ 2,000,000	$ 2,000,000	$ -	
1.3.2	**Seepage Collection Dam Fill (North)**	Till from local borrow	250000	m3	0.02	6,173	$ 81	$ 500,000	$ -	$ -	$ 6.00	$ 1,500,000	$ -	$ -	$ 8	$ 2,000,000	$ -	$ 2,000,000	Year 1
1.3.3	**Seepage Collection Ditches**	Excavate and side-cast (2 m3/m)	7875	m	0.04	292	$ 81	$ 23,625	$ -	$ -	$ 9.00	$ 70,875	$ -	$ -	$ 12	$ 94,500	$ 15,000	$ 79,500	Year 1 to 20
1.3.4	**Diversion Ditch - Excavation**	Excavate and side-cast (12 m3/m)	3800	m	0.15	563	$ 81	$ 45,600	$ -	$ -	$ 36.00	$ 136,800	$ -	$ -	$ 48	$ 182,400	$ 182,400	$ -	
1.3.5	**Diversion Ditch - Erosion Protection**	Riprap D50 = 175mm; 2 m3/m (includes screening)	7600	m3	0.05	352	$ 81	$ 28,500	$ -	$ -	$ 11.25	$ 85,500	$ -	$ -	$ 15	$ 114,000	$ 114,000	$ -	
1.3.6	**Temporary Ditching - Excavation**	Excavate and side-cast (2 m3/m)	49000	m3	0.04	1,815	$ 81	$ 147,000	$ -	$ -	$ 9.00	$ 441,000	$ -	$ -	$ 12	$ 588,000	$ 84,000	$ 504,000	Year 1 to 20
1.3.7	**Sedimentation Pond - Excavation**	Excavate and side-cast	750	m3	0.01	9	$ 81	$ 750	$ -	$ -	$ 3.00	$ 2,250	$ -	$ -	$ 4	$ 3,000	$ -	$ 3,000	Year 1
															Subtotal	$ 4,981,900	$ 2,395,400	$ 2,586,500	
1.4	**Tailings Delivery and Water Reclaim**																		
	See Separate Estimate																		
1.5	**Closure**																		
1.5.1	**Non-PAG Cover Over Tailing Beach**	Load, haul, place.	472000	m3	0.02	11,654	$ 81	$ 944,000	$ -	$ -	$ 6.00	$ 2,832,000	$ -	$ -	$ 8	$ 3,776,000	$ -	$ 3,776,000	Closure
1.5.2	**Soil Cover on Dam Slope Face Using Till**	Cover Area on 3H:1V slope (1.0 m thickness)	392527	m3	0.02	9,692	$ 81	$ 785,053	$ -	$ -	$ 6.00	$ 2,355,160	$ -	$ -	$ 8	$ 3,140,213	$ -	$ 3,140,213	Closure
1.5.3	**Topsoil Cover on Dam Slope Face Using Organic Bearing Material**	Cover Area on 3H:1V slope (0.2 m thickness)	78505	m3	0.04	2,908	$ 81	$ 235,516	$ -	$ -	$ 6.00	$ 471,032	$ -	$ -	$ 8	$ 706,548	$ -	$ 942,064	Closure
1.5.4	**Erosion Protection**	Per unit length, create a Ditch (1m x 0.5m)	6400	m	0.01	79	$ 81	$ 6,400	$ -	$ -	$ 3.00	$ 19,200	$ -	$ -	$ 4	$ 25,600	$ -	$ 25,600	Closure
1.5.5	**Erosion Protection**	Place D50 = 75mm Gravel (0.5m3/m) (includes screening)	3200	m3	0.05	148	$ 81	$ 12,000	$ -	$ -	$ 11.25	$ 36,000	$ -	$ -	$ 15	$ 48,000	$ -	$ 48,000	Closure
1.5.6	**Reclamation**	Vegetation on dam slopes (hydroseeding)	392527	m2	0.002	727	$ 81	$ 58,879	$ 0.08	$ 29,440	$ 0.53	$ 206,077	$ -	$ -	$ 0.75	$ 294,395	$ -	$ 294,395	Closure
1.5.7	**Closure Spillway - Excavation in Bedrock**	500 m x15 m3/m	7500	m3	0.05	347	$ 81	$ 28,125	$ -	$ -	$ 11.25	$ 84,375	$ -	$ -	$ 15	$ 112,500	$ -	$ 112,500	Closure
1.5.8	**Decommission Diversion Ditches**	Back-fill	3800	m	0.07	281	$ 81	$ 22,800	$ -	$ -	$ 18.00	$ 68,400	$ -	$ -	$ 24	$ 91,200	$ -	$ 91,200	Closure
															Subtotal	$ 8,194,457	$ -	$ 8,429,973	

KLOHN CRIPPEN BERGER

Capacity - 30,000tpd

WBS Code	Area, Item Description & Equipment Number		Measure Qty	M Unit	Unit Mhrs	Total Mhrs	Labour Rate	Labour Total Labour	Unit Matls	Materials Total Materials	Unit C Equipmt	Total Constrctn Equipment	Unit Proc Equipmt	Total Process Equipment	Unit Cost	Total Capital Costs	Initial Capital Costs	Sustaining Capital Costs	Sustaining Capital Year
2	Low Grade Ore Stockpile and Waste Rock Dump																		
2.1	Site Preparation																		
2.1.1	Clear and Grub - Waste Rock Dump		174	ha	17.28	3,007	$ 81	$ 243,600	$ -	$ -	$ 4,200.00	$ 730,800	$ -	$ -	$ 5,600	$ 974,400	$ 974,400	$ -	
2.1.2	Clear and Grub - LGO Stockpile		29.5	ha	17.28	510	$ 81	$ 41,300	$ -	$ -	$ 4,200.00	$ 123,900	$ -	$ -	$ 5,600	$ 165,200	$ 165,200	$ -	
2.1.3	Excavate and Dispose Unsuitable Soils - Waste Rock Dump	Bog area, assuming 4 m average thickness	50000	m3	0.02	772	$ 81	$ 62,500	$ -	$ -	$ 3.75	$ 187,500	$ -	$ -	$ 5	$ 250,000	$ 250,000	$ -	
2.1.3	Excavate and Dispose Unsuitable Soils - LGO Stockpile	LGO (Soft Clay, MW08-3)	646113	m3	0.02	9,971	$ 81	$ 807,641	$ -	$ -	$ 3.75	$ 2,422,923	$ -	$ -	$ 5	$ 3,230,564	$ 3,230,564	$ -	
															Subtotal	$ 4,620,164	$ 4,620,164	$ -	
2.2	Water Management																		
2.2.1	Seepage Collection Ditches	Excavate and side-cast (2 m3/m)	1125	m	0.04	42	$ 81	$ 3,375	$ -	$ -	$ 9.00	$ 10,125	$ -	$ -	$ 12	$ 13,500	$ 13,500	$ -	
2.2.2	Diversion Ditch - Excavation	Excavate and side-cast (16 m3/m)	3600	m	0.15	533	$ 81	$ 43,200		$ -	$ 36.00	$ 129,600	$ -	$ -	$ 48	$ 172,800	$ 172,800	$ -	
2.2.3	Diversion Ditch - Erosion Protection	Riprap D50 = 175mm; 2 m3/m (includes screening)	7200	m3	0.05	333	$ 81	$ 27,000		$ -	$ 11.25	$ 81,000	$ -	$ -	$ 15	$ 108,000	$ 108,000	$ -	
2.2.4	Seepage Collection Pumping Sumps	Cut and Fill	14400	m3	0.02	222	$ 81	$ 18,000	$ -	$ -	$ 3.75	$ 54,000	$ -	$ -	$ 5	$ 72,000	$ 72,000	$ -	
2.2.5	Surface Water Return Pumps		4	ea	740.74	2,963	$ 81	$ 240,000	$ -	$ -	$ 140,000.00	$ 560,000	$ 200,000.00	$ 800,000	$ 400,000	$ 1,600,000	$ 1,600,000	$ -	
2.2.6	Surface Water Return Pipelines		3000	m	0.28	833	$ 81	$ 67,500	$ 75.00	$ 225,000	$ 52.50	$ 157,500	$ -	$ -	$ 150	$ 450,000	$ 450,000	$ -	
2.2.7	Diversion Outlet Erosion Control		1	LS	154.32	154	$ 81	$ 12,500	$ -	$ -	$ 37,500.00	$ 37,500	$ -	$ -	$ 50,000	$ 50,000	$ 50,000	$ -	
															Subtotal	$ 2,466,300	$ 2,466,300	$ -	
2.3	Closure																		
2.3.1	Organic Cover - From Stockpiles	0.2 m thickness	610500	m3	0.02	11,306	$ 81	$ 915,750	$ -	$ -	$ 4.50	$ 2,747,250	$ -	$ -	$ 6	$ 3,663,000	$ -	$ 3,663,000	Year 5 to 19
2.3.2	Low Permeability Cover - From Stockpile	1 m Thick Glacial Till from stockpile	1851360	m3	0.02	45,713	$ 81	$ 3,702,720	$ -	$ -	$ 6.00	$ 11,108,160	$ -	$ -	$ 8	$ 14,810,880	$ -	$ 14,810,880	Year 5 to 19
2.3.3	Erosion Protection	Per unit length, create a Ditch (1m x 0.5m)	22500	m	0.01	278	$ 81	$ 22,500	$ -	$ -	$ 3.00	$ 67,500	$ -	$ -	$ 4	$ 90,000	$ -	$ 90,000	Year 5 to 19
2.3.4	Erosion Protection	Place D50 = 75mm Gravel (0.5m3/m) (includes screening)	11250	m3	0.05	521	$ 81	$ 42,188	$ -	$ -	$ 11.25	$ 126,563	$ -	$ -	$ 15	$ 168,750	$ -	$ 168,750	Year 5 to 19
2.3.4	Reclamation	Vegetation on Slopes	2035000	m2	0.00	3,769	$ 81	$ 305,250	$ 0.08	$ 152,625	$ 0.53	$ 1,068,375	$ -	$ -	$ 0.75	$ 1,526,250	$ -	$ 1,526,250	Year 5 to 19
2.3.5	Decommission Diversion Ditches	Back-fill	3600	m	0.07	267	$ 81	$ 21,600	$ -	$ -	$ 18.00	$ 64,800	$ -	$ -	$ 24	$ 86,400	$ -	$ 86,400	Year 5 to 19
2.3.6	Recontouring and grading (estimate 1.6 m3/m2)	Assume final grading done during operations	2962176	m3	0.01	27,428	$ 81	$ 2,221,632	$ -	$ -	$ 2.25	$ 6,664,896	$ -	$ -	$ 3	$ 8,886,528	$ -	$ 8,886,528	Year 5 to 19
2.3.7	Seepage Water Treatment Plant (Capital)	by Others																	
2.3.8	Treatment Plant Infrastructure Upkeep ($/year)	by Others																	
2.3.9	Seepage Water Treatment Plant (Operating $/year)	by Others																	
2.3.10	Sludge Disposal ($/year)	by Others																	
															Subtotal	$ 29,231,808	$ -	$ 29,231,808	
3	Stockpiles and Borrow Areas																		
3.1	Site Preparation																		
3.1.1	Clear and Grub - Stockpiles		62.8	ha	17.28	1,085	$ 81	$ 87,920	$ -	$ -	$ 4,200.00	$ 263,760	$ -	$ -	$ 5,600	$ 351,680	$ 351,680	$ -	
3.1.2	Strip - Stockpiles	(assume 0.2 m stripping depth)	125600	m3	0.02	1,938	$ 81	$ 157,000	$ -	$ -	$ 3.75	$ 471,000	$ -	$ -	$ 5	$ 628,000	$ 628,000	$ -	
3.1.3	Clear and Grub - Granular Borrow Areas		21.7	ha	17.28	375	$ 81	$ 30,380	$ -	$ -	$ 4,200.00	$ 91,140	$ -	$ -	$ 5,600	$ 121,520	$ 121,520	$ -	
3.1.4	Strip - Borrow Areas	(assume 0.2 m stripping depth)	43400	m3	0.02	670	$ 81	$ 54,250	$ -	$ -	$ 3.75	$ 162,750	$ -	$ -	$ 5	$ 217,000	$ 217,000	$ -	
3.1.5	Excavate Unsuitable Soils	TP08-B - Soft Clay From Overburden Stockpile	231938	m3	0.02	3,579	$ 81	$ 289,922	$ -	$ -	$ 3.75	$ 869,767	$ -	$ -	$ 5	$ 1,159,689	$ 1,159,689	$ -	
															Subtotal	$ 2,477,889	$ 2,477,889	$ -	
3.2	Water Management																		
3.2.1	Seepage Collection Ditches	Excavate and side-cast (2 m3/m)	4876	m	0.04	181	$ 81	$ 14,628	$ -	$ -	$ 9.00	$ 43,884	$ -	$ -	$ 12	$ 58,512	$ 58,512	$ -	
3.2.2	Sedimentation Ponds - Excavation	Excavate and side-cast; 4 x 750 m3	3000	m3	0.01	37	$ 81	$ 3,000	$ -	$ -	$ 3.00	$ 9,000	$ -	$ -	$ 4	$ 12,000	$ 12,000	$ -	
															Subtotal	$ 70,512	$ 70,512	$ -	
3.3	Closure																		
3.3.1	Topsoil Cover	0.2 m thickness	16.9	m3	0.02	0	$ 81	$ 25	$ -	$ -	$ 4.50	$ 76	$ -	$ -	$ 6	$ 101	$ -	$ 101	Closure
3.3.3	Erosion Protection		1	LS	771.60	772	$ 81	$ 62,500	$ -	$ -	$ 187,500.00	$ 187,500	$ -	$ -	$ 250,000	$ 250,000	$ -	$ 250,000	Closure
3.3.5	Reclamation		0.00845	ha	12.35	0	$ 81	$ 8	$ 500.00	$ 4	$ 3,500.00	$ 30	$ -	$ -	$ 5,000	$ 42	$ -	$ 42	Closure
															Subtotal	$ 250,144	$ -	$ 250,144	
4	Booker Lake and Ore Lake																		
4.1	Site Preparation																		
4.1.1	Excavate Lake-Bottom Sediment - Booker Lake	Load, haul, place in contained storage area	600000	m3	0.02	9,259	$ 81	$ 750,000	$ -	$ -	$ 3.75	$ 2,250,000	$ -	$ -	$ 5	$ 3,000,000	$ 3,000,000	$ -	
4.1.2	Lake Dewatering - Booker Lake	Pump downstream	1500000	m3	0.00	2,315	$ 81	$ 187,500	$ -	$ -	$ 0.38	$ 562,500	$ -	$ -	$ 0.5	$ 750,000	$ 750,000	$ -	
4.1.3	Excavate Lake-Bottom Sediment - Ore Lake	Load, haul, place in contained storage area	50000	m3	0.02	772	$ 81	$ 62,500	$ -	$ -	$ 3.75	$ 187,500	$ -	$ -	$ 5	$ 250,000	$ 250,000	$ -	
4.1.4	Lake Dewatering - Ore Lake	Pump downstream	127235	m3	0.00	196	$ 81	$ 15,904	$ -	$ -	$ 0.38	$ 47,713	$ -	$ -	$ 0.5	$ 63,617	$ 63,617	$ -	
4.1.5	Excavate Organic Sediment - Bog	Load, haul, place in contained storage area	13806	m3	0.02	213	$ 81	$ 17,257	$ -	$ -	$ 3.75	$ 51,772	$ -	$ -	$ 5	$ 69,029	$ 69,029	$ -	
4.1.6	Overburden Containment Fill - From Stockpile	Load, haul, place and compact	500000	m3	0.02	9,259	$ 81	$ 750,000	$ -	$ -	$ 4.50	$ 2,250,000	$ -	$ -	$ 6	$ 3,000,000	$ 3,000,000	$ -	
															Subtotal	$ 7,132,646	$ 7,132,646	$ -	

KLOHN CRIPPEN BERGER

Capacity - 30,000tpd

WBS Code	Area, Item Description & Equipment Number		Measure Qtty	M Unit	Unit Mhrs	Total Mhrs	Labour Rate	Labour Total Labour	Unit Matls	Materials Total Materials	Unit C Equipmt	Total Constrctn Equipment	Unit Proc Equipmt	Total Process Equipment	Unit Cost	Total Capital Costs	Initial Capital Costs	Sustaining Capital Costs	Sustaining Capital Year
5	**Infrastructure**																		
5.1	**Roads**																		
5.1.1	**Primary Roads (~ 10 m wide, 30 m clear and grub)**	Includes runaway lanes , 1x200m/km	12.6	km	586.42	7,385	$ 81	$ 598,168	$ -	$ -	$ 142,500.00	$ 1,794,503	$ -	$ -	$ 190,000	$ 2,392,670	$ 825,170	$ 1,567,500	Year 1
5.1.2	**Secondary Roads (~ 6 m wide)**	Includes pull-outs, 1x50m/ 500m	6.9	km	339.51	2,347	$ 81	$ 190,094	$ -	$ -	$ 82,500.00	$ 570,281	$ -	$ -	$ 110,000	$ 760,375	$ 719,125	$ 41,250	Year 1 to 4
															Subtotal	$ 3,153,045	$ 1,544,295	$ 1,608,750	
5.2	**Creek Crossings**																		
5.2.1	**Primary Creek Crossing**	3mx2m Multi-Plate Arch Culvert, 28 m long, 7000 m^3 fill	1	LS	1,234.57	1,235	$ 81	$ 100,000	$ 25,000.00	$ 25,000	$ 125,000.00	$ 125,000	$ -	$ -	$ 250,000	$ 250,000	$ 250,000	$ -	
5.2.2	**Secondary Road/Pipeline Crossings**	0.3m Diam. Culvert Installed	16	each	30.86	494	$ 81	$ 40,000	$ 1,000.00	$ 16,000	$ 6,500.00	$ 104,000	$ -	$ -	$ 10,000	$ 160,000	$ 60,000	$ 100,000	Year 1
												$ -							
															Subtotal	$ 410,000	$ 310,000	$ 100,000	
5.3	**Water and Tailings Pipeline**																		
5.3.1	**Clear and Grub**	Pipeline without Access Road	41000	m2	0.02	633	$ 81	$ 51,250	$ -	$ -	$ 3.75	$ 153,750	$ -	$ -	$ 5	$ 205,000	$ 205,000	$ -	
5.3.2	**Emergency Backflow Ponds - Excavation**	Excavate and Sidecast	20000	m3	0.02	309	$ 81	$ 25,000	$ -	$ -	$ 3.75	$ 75,000	$ -	$ -	$ 5	$ 100,000	$ 100,000	$ -	
															Subtotal	$ 305,000	$ 305,000	$ -	
6	**Monitoring**																		
6.1	**Instrumentation**		1	LS	2,469.14	2,469	$ 81	$ 200,000	$ 200,000.00	$ 200,000	$ -	$ -	$ -	$ -	$ 400,000	$ 400,000	$ 120,000	$ 280,000	Year 1 to 21
															Subtotal	$ 400,000	$ 120,000	$ 280,000	
7	**Other**																		
7.1	**Mobilization/Demobilization - Initial**														8%	$ 2,683,803	$ 2,683,803		
7.2	**Mobilization/Demobilization - On-going**														2%	$ 2,371,450		$ 2,371,450	Year 1 to 20
7.5	**Contractor Crew Housing (Allowance)**	In Granisle or Morrison Camp	1	LS											$ 1,500,000	$ 1,500,000	$ 1,500,000		
7.3	**Engineering - Initial**														6%	$ 2,012,852	$ 2,012,852		
7.4	**Engineering - On-going**														4%	$ 4,742,900		$ 4,742,900	Year 1 to 21
															Subtotal	$ 13,311,004	$ 6,196,654	$ 7,114,350	
Total						468,784		$ 37,971,472		$ 653,069		$ 112,859,968		$ 800,000		$ 165,595,512	$ 39,864,186	$ 125,966,841	
8	**Contingency**																		
8.1	**Recommended Contingency**	for possible variations in Scope of Work, Unit Costs and Unforeseen Circumstances													20%	$ 33,119,102	$ 7,972,837	$ 25,193,368	
Total + Contingency																$ 198,714,614	$ 47,837,024	$ 151,160,210	

13. CONCLUSIONS AND RECOMMENDATIONS

13.1 Conclusions

This report presents the geotechnical feasibility study for Pacific Booker Minerals Inc. (PBM) proposed Morrison Copper/Gold Project, located 65 km northeast of Smithers in north-central British Columbia. The Morrison mine will be a 30,000 tpd open pit operation with ore processed in a conventional milling process and the copper/gold concentrate transported to the Port of Stewart, for shipment to offshore smelters. The mine will produce approximately 224 Mt of tailings and 170 Mt of waste rock.

Site Conditions

- The site is in hilly, forested terrain, which receives approximately 550 mm of precipitation annually (40% as snowfall). The area is in a low to moderate seismic zone and the maximum credible earthquake (MCE) is magnitude M_W=6.2 and a peak ground acceleration PGA=0.13 g. The foundation soils for the tailing dams and the waste rock dump consists of medium dense glacial till overlying bedrock. The glacial till has a low permeability and has the potential to generate excess pore pressures during loading. Accordingly, piezometers will be installed in the tailings dams and waste rock dump foundations to confirm the predicted conditions. The low permeability tailings also provides for additional containment of potential seepage from the tailings facility.

Tailings and Waste Rock Characterization

- The total tailings is a silty sand with approximately 55% less than the 75 micron particle size and typically contains < 1% sulphides. The dams will be raised with cycloned sand that is classified as non-potentially acid generating (NAG). The cyclone overflow tailings and the remainder of the total tailings will be spigotted into the impoundment and are classified as NAG to a low PAG (low potential for acid generating). The majority, (i.e. 90%) of waste rock is classified as PAG and the remainder is NAG. The lag time for acid generation for the waste rock could be a long time and a more detailed assessment of the acid rock drainage characterization is included in the environmental impact assessment for the project.

Tailings Storage Facility

- The TSF covers an area of approximately 5 km^2, with an uphill drainage area of approximately 5 km^2. Current drainage from the TSF flows south into a small creek and into Morrison Lake and a small amount of drainage flows northwards into Nakinilierak Lake. Given the potential environmental damage and substantial clean-up costs, the tailings storage facility could be categorized as a "Very High" classification facility (according to the Canadian Dam Association (CDA 2007)). However, the selected criteria for flood and seismic design have been conservatively upgraded to meet the "Extreme" classification to reflect the potential for future land use in the area. Accordingly, the tailings dams are designed for maximum credible earthquake (MCE) and the maximum probable flood (PMF). The Main Starter Dam will be constructed as a homogeneous fill dam using glacial till borrow material from the interior of the TSF and from stripping of the open pit. A sand and gravel blanket drain will be placed under the downstream toe of the dam to control seepage. The dams will be raised by the centerline method with a central glacial till core and cycloned sand on the downstream and upstream sides. The downstream slopes of the dams will be 3H:1V. Seepage collection systems, downstream of each dam, will include a dam and water return system to recycle seepage and cyclone sand drainage water back to the impoundment. A pump barge will return water, via a buried pipeline, to the plantsite. The tailings delivery system includes 3 pump stations and an approximate 760 mm diameter HDPE pipeline to the crest of the dams. Cyclowash cyclones, located on the dam crest, will be used to cyclone sand for construction of the dams between March and October each year.

Mine Area, Low Grade Ore Stockpile and Waste Rock Dumps

- The waste rock dump and low grade ore stockpile are located adjacent to the open pit and plantsite area. Development of the open pit will require drainage of Booker Lake and Ore Pond, and removal of soft sediments. Overburden from stripping of the open pit will be stockpiled near the open pit and potentially used for construction of the tailings Starter Dam. Organic bearing material will be stockpiled for use in reclamation. The waste rock dump will cover an area of approximately 220 ha and reach a maximum height of approximately 150 m. Foundation preparation for the waste rock dump and low grade ore stockpile will include removal of marshy soils and any other weak or soft materials. Piezometers will be installed in the glacial till to monitor construction pore pressures.

- The majority of the waste rock is potentially acid generating and could begin to leach metals at some time during operations or in the future. NAG waste rock will be preferentially placed in drainage channels and towards the south side of the waste rock dump. Waste rock will be placed to an overall final slope of 2.75H:1V. A soil cover will be placed over the dump surfaces to minimize infiltration of water. Ore will be temporarily stockpiled in a low grade ore dump and milled later in the mine life.

- Clean surface water will be diverted around the waste dump and disturbed mine areas. Contact water from the plant site, waste dump, low grade ore and open pit areas will be collected and recycled to the mill. Runoff water from overburden dumps and organic bearing material stockpiles will be directed to sediment ponds prior to release. A clean water supply will be provided from Morrison Lake.

Environment and Closure

- The tailings have low to no potential for acid generation; nonetheless, they will be stored in saturated impoundment, which will further preclude the risk of acid generation. The tailings process water contains low metal concentrations and meets guidelines for drinking water and wildlife use. Seepage from the tailings impoundment is estimated to be in the order of 5.6 L/s to 7.4 L/s and the majority of seepage will be collected with the seepage recovery systems. On closure, the tailings impoundment will be closed as a lake and the dam slopes will be revegetated.

- After closure the open pit will fill with water, which will assist in reducing the area of pit wall rock that would be exposed to oxidation and, therefore, potentially acid generating. The waste rock dumps will be covered with a low permeability soil cover. A provision for a water treatment plant is provided to treat contaminated seepage water from the waste rock dumps and from the remaining exposed pit wall rock above the pit lake level. The mill site would be decommissioned and industrial waste would be stored in the base of the open pit and any hazardous waste would be either disposed in an engineered facility or shipped off site. The disturbed areas would be covered with soil and reclaimed.

13.2 Recommendations

The main recommendations, as the project moves forward into permitting, detailed design and construction, include the following:

- The foundation glacial till for the tailings dams, waste rock dump and low grade ore stockpiles have the potential to generate significant excess pore pressure when loaded. Consequently, electric piezometers need to be installed in the foundation prior to construction. The piezometers will be monitored during construction and operations to confirm the actual response.
- An operations maintenance and surveillance manual (OMS) will need to be prepared for the tailings impoundment prior to operations. The manual will document operating and maintenance procedures, and monitoring and response plans. The documentation will also include an Emergency Preparedness Plan (EPP), which will document the response plan in the event of such items as a pipeline break or dam safety concern.
- Procedures for identification of NAG waste rock will need to be developed. This could include both geologic identification, as well as blast hole cutting analysis for sulphide and acid base accounting tests.

KLOHN CRIPPEN BERGER LTD.

Terence Jibiki, P.Eng.
Project Manager

Harvey McLeod, P.Eng., P.Geo.
Project Reviewer

REFERENCES

Adams, J., and S. Halchuk, (2003). "Fourth Generation Seismic Hazard Maps of Canada: Values for Over 650 Canadian Localities intended for the 2005 National Building Code of Canada", Geological Survey of Canada, Open File 4459, 155p May 15, 2003.

Appelo, C.A.J. and Postma, D. 2005. *Geochemistry, groundwater and pollution.* (2nd edition) (A.A. Balkema, New York)

Appelo, C.A.J., Van der Weiden, M.J.J., Tournassat, C. and Charlet, L., 2002. *Surface Complexation of ferrous iron and carbonate on ferrihydrite and the mobilization of arsenic: Env. Sci. Technol. 36, 3096-3103.*

Boore, D., Joyner, W. and Fumal, T. (1997). "Equations for Estimating Horizontal Response Spectra and Peak Acceleration from Western North American Earthquakes: A Summary of Recent Work." Seismological Research Letters, 68(1): 128-153.

Boore, D.M., and Atkinson, G.M., (2007). "Boore-Atkinson NGA Ground Motion Relations for the Geometric Mean Horizontal Component and Peak and Spectral Ground Motion Parameters", PEER Report No. 2007/01, Pacific Earthquake Engineering Research Center, May 2007.

British Columbia Mine Waste Rock Pile Research Committee (BCMWRPRC) (1991). Investigation and Design Manual. May 1991.

BC Ministry of Environment (2007). Terrain Mapping. Accessed online at: < http://www.env.gov.bc.ca/terrain/terrain_files/types.html> on December 5, 2008.

BC Ministry of Environment (1999). Mapping and assessing terrain stability guidebook. Second Edition. August 1999.

Canadian Dam Association (2007). Canadian Dam Safety Guidelines.

Campbell, K.W. and Bozorgnia, Y., (2007). "Campbell-Bozorgnia NGA Ground Motion Relations for the Geometric Mean Horizontal Component and Peak and Spectral Ground Motion Parameters". PEER Report No. 2007/02, Pacific Earthquake Engineering Research Center, May 2007.

Carter, N.C. (1973). Geology of the northern Babine Lake area. British Columbia Department of Mines, Preliminary Map 12, 1:63,000.

Chiou, B., and Youngs, R., (2006). "PEER-NGA Empirical Ground motion Model for the Average Horizontal Component of Peak Acceleration and Psuedo-spectral Acceleration for Spectral Periods of 0.01 to 10 Seconds". Pacific Earthquake Engineering interim report for USGS review, June 14, 2006.

Dzombak, D.A. and Morel, F.M.M., 1990. *Surface Complexation modeling: hydrous ferric oxide.* Wiley and Sons, New York, 393 pp.

Environment Canada (2007). Canadian Climate Normals. Accessed online at: <http://climate.weatheroffice.ec.gc.ca/climate_normals/index_e.html> on December 5, 2008.

Environment Canada (2007). Canadian Climate Data. Accessed online at: <http://climate.weatheroffice.ec.gc.ca/climateData/canada_e.html> on December 5, 2008.

Frontier Geosciences Inc. (2007). "Resistivity Imaging Survey", prepared for Pacific booker minerals inc., submitted June, 2007.

Hynes-Griffin and Franklin (1984). Rationalizing the Seismic Coefficient Method. U.S. Army Corps of Engineers, Vicksburg.

Kimura, E.T. (2003). "Geological and Manual Resource Estimation Report on Morrison Copper-Gold Porphyry Deposit at Morrison Lake", for Pacific Booker Ltd., submitted 13 December 2003, Amended 4 February 2003.

Klohn Crippen Consultants Ltd. (1998). TSIL D terrain stability and surface erosion potential mapping and assessment, Morice Forest District TSA, British Columbia. Submitted to British Columbia Ministry of Forests, Prince Rupert Region, December 17, 1998.

Klohn Crippen Berger Ltd. (2008). "Morrison Copper/Goldproject - 2007 Geotechnical Site Investigation" Prepared for Pacific Booker Minerals Inc., submitted November 19, 2008.

Klohn Crippen Berger Ltd. (2008b). "Morrison Copper/Gold Project – Plantsite Foundation Report" Prepared for Pacific Booker Minerals Inc., submitted December 7, 2008.

Knight Piesold Ltd. (2002). "Initial Waste Management Site Alternatives Study", prepared for Pacific Booker Minerals Inc., submitted October 7, 2002.

Knight Piesold Ltd. (2003). "Report on Initial Site Visit and Updated Concepts for Waste Management", prepared for Pacific Booker Minerals Inc., submitted August 20, 2003.

Knight Piesold Ltd. (2004). "Tailings and Waste Rock Management Input to Scoping Study", prepared for Pacific Booker Minerals Inc., submitted June 17, 2004.

Knight Piesold Ltd. (2006a). "Geotechnical Site Investigation Report", prepared for Pacific Booker Minerals inc., submitted July 7, 2006.

Knight Piesold Ltd. (2006b). "2006 Open Pit Geotechnical Investigations", prepared for Pacific Booker Minerals inc., submitted Sept 14, 2006.

Levson, V.M. (2002). Quaternary geology and till geochemistry of the Babine porphyry copper belt, British Columbia (NTS 93 L/9, 16; M/1, 2, 7, 8). British Columbia Ministry of Energy and Mines, Bulletin 110.

MacIntyre, D., Webster, I., and Desjardins, P. (1997). Bedrock Geology of the Old Fort Mountain Map Area, North-Central BC (NTS 93M/01). BCMEMPR Open File 1997-10.

MacIntyre, D. (2001). Geological compilation map, Babine porphyry copper district, central British Columbia (NTS 93L/9,16; 93M/1,2E,7E,8). Geological Survey of Canada, Open File 2001-3, map scale 1:100 000.

Minesite Drainage Assessment Group (2007). "Morrison Project – Prediction of Metal Leaching and Acid Rock Drainage, Phase 1", prepared for Rescan Environmental Services Ltd., submitted March 22, 2007.

Ministry of Environment (2006). British Columbia Water Quality Guidelines. Accessed online at: http://www.env.gov.bc.ca/wat/wq/BCguidelines/approv_wq_guide/approved.html>, on December 5, 2008.

Morin, K.A. and N.M. Hutt (2001). A Comparison of Past Prediction to Current Conditions at Bell Mine. Securing the Future. June 25 – July 1, 2001.

Morin, K.A. and Hutt, N.M. (2007). "Morrison Project –Prediction of Metal Leaching and Acid Rock Drainage, Phase I" . Skelleftea, June 25 – July 1, 2001

National Earthquake Hazard Reduction Program (NEHRP), (2003). "2003 Recommended Provisions for New buildings and Other Structures". Part 1, provisions. FEMA 450, Federal Emergency Management Agency, Washington, D.C.

Ogryzlo, P.L., Dirom, G.E., and Stothart, P.G. (1995.) Morrison Hearne Hill copper gold deposits, Babine region, west-central British Columbia. *In* Porphyry deposits of the northwestern Cordillera of North America. *Edited by* T.G. Schroeter. Canadian Institute of Mining, Metallurgy and Petroleum, Special Vol. 46, pp. 290-303.

Rescan Environmental Services (2008). "Morrison Project 2007 Hydrology Baseline", prepared for Pacific Booker Minerals Inc., submitted March 8, 2008.

Richards, T.A. (1974). Hazelton East Half. Geological Survey of Canada, Open File Map 215.

S.G.S. Lakefield Research Ltd. (2007). "Environment Testing of Morrison Tailings Solids and Supernatant", prepared for Wardrop Mining Engineering Inc. on behalf of Pacific Booker Minerals Inc., submitted October 4, 2007.

Schiarizza, P., and MacIntyre, D. (1999). Geology of the Babine Lake – Takla Lake area, central British Columbia (93 K/11, 12, 13, 14; 93 N/3, 4, 5, 6). *In* Geological Fieldwork 1998. British Columbia Ministry of Energy and Mines, Paper 1999-1, pp. 33-68.

Simpson, R.G. (2007). "Mineral Resource Update – Morrison Project", prepared for Pacific Booker Minerals Inc., submitted May 4, 2007.

Terzaghi, K. and R. Peck (1996), Soil Mechanics in Engineering Practice. Third Edition. New York.

APPENDIX I

Site Selection Study

- Annotative Bibliography
- Minutes of Project Brainstorming Meeting
- December 7, 2006 Correspondence – Site Comparison
- Site B and Site E Comparative Costs

APPENDIX II

Knight Piesold Geotechnical Data (2006)

- Grains Size Distributions
- Compaction Tests
- Drill hole Logs
- Bedrock Drilling Graphs
- Field Tests
- Well Completion Details
- Test Pit Logs

APPENDIX III

KCBL 2007 Geotechnical Site Investigation

2007 Geotechnical Site Investigation

APPENDIX IV

KCBL 2008 Geotechnical Site Investigation and Tailings Testing Data

2008 Geotechnical Site Investigation

- 2008 Drill hole and Test Pit Logs
- Index Test Results

Tailings Testing

- 1-D Consolidation Test
- Grain Size Distributions (Hydrometer)
- Compaction Test ("Cycloned" Sand)
- Jar Settling Tests
- Specific Gravity of Solids

APPENDIX V

Seismic Hazard Assessment

APPENDIX VI

Groundwater and Tailings Geochemical Data

- Groundwater Geochemical Analysis Results (Rescan)
- "Environmental Testing of Morrison Tailings Solids and Supernatant" Interim Report dated October 4, 2007 (SGS)
- Tailings Humidity Cell Testing Data, May 2008 (SGS)
- Tailings Analysis Data, May 2006 (PRA)
- Whole Tailings/Sulphide-Reduced Tailings ABA Results, April 2007 (SGS)
- Compiled Acid Base Accounting and Total-Element Analyses for Mettalurgical-Test Tailings, excerpt from "Prediction of Metal Leaching and Acid Rock Drainage, Phase 2", March 22, 2007 (MDAG)

APPENDIX VII

Stability Analyses

APPENDIX VIII

Seepage Analyses

APPENDIX IX

Water Balance

- Summary Data
- Starter Dam – Average Conditions
- Mid-Operations – Average Conditions
- Pre-Closure – Average Conditions
- Post-Closure – Average Conditions
- Starter Dam – Dry Year Conditions

APPENDIX X

Tailings, Reclaim and Associated Pumping Systems – Feasibility Design Report

DRAWINGS