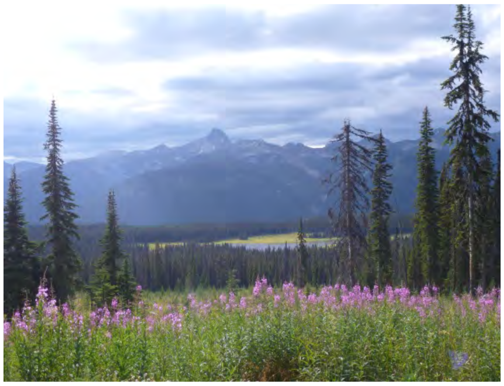
Appendix 5-D

Mine Waste and Water Management Design Report

HARPER CREEK PROJECT

Application for an Environmental Assessment Certificate / Environmental Impact Statement

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT



MINE WASTE AND WATER MANAGEMENT DESIGN REPORT

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VA101-458/11-1
Rev 0
September 26, 2014

Knight Piésold
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ISO 14001 - EMS 550121
OHSAS 18001 - OHS 550122

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT DESIGN REPORT
VA101-458/11-1

Rev	Description	Date	Approved
0	Issued in Final	September 26, 2014	KB

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EXECUTIVE SUMMARY

KP completed engineering studies to update the design of the mine waste and water management facilities to contribute to an updated project FS. The FS update is being completed in conjunction with Merit Consultants International Inc. and Nilsson Mine Services Ltd. The facilities include the Tailings Management Facility (TMF), waste rock and overburden stockpiles, low-grade ore stockpiles, and open pit, and associated water management infrastructure.

The principle design objectives for the waste rock stockpiles and TMF are to ensure protection of the regional groundwater and surface water during both operations and in the long-term (after closure), and to achieve effect reclamation at mine closure. The design and location of the waste rock stockpiles and TMF has taken into account the following requirements:

- situating the TMF and waste rock facilities away from sensitive environmental features including fish bearing drainages
- clustering the facilities to minimize the overall footprint
- permanent, secure, and total confinement of all solid waste materials within engineered disposal facilities
- control, collection, and removal of free-draining liquids from the waste and tailings facilities during operations for recycling as process water to the maximum practical extent
- prevention of acid rock drainage and minimization of metal leaching from reactive tailings and waste rock, and
- staged development of the facility over the life of the project.

The TMF was designed to permanently store tailings and potentially acid generating (PAG) waste rock generated during operation of the mine. Specific overall features of the TMF are listed below:

- cofferdams and sediment control ponds to manage water during construction by either routing water around the TMF or directing water to the TMF for collection
- two zoned water-retaining earth-rockfill dams referred to as the main embankment and north embankment
- designated PAG waste rock stockpile areas within the TMF
- downstream water management ponds for seepage and storm water management
- collection channels that route water to the TMF and collection ponds
- diversion channels that route water away from the TMF and collection ponds to the downstream receiving environment
- tailings distribution system
- tailings beaches
- reclaim water system, and
- supernatant water pond.

Two tailings streams will be generated in the process plant and transported to the TMF. The two types of tailings are designated as rougher scavenger (bulk) tailings and cleaner scavenger (cleaner) tailings. Bulk tailings will be transported by the bulk tailings distribution pipeline primarily to the main embankment during the first 24 years of operation, and later in mine life to the north embankment during between Years 19 and 24. The bulk tailings will be discharged from the embankment crests using multiple spigots to build extensive tailings beaches to hydraulically isolate the supernatant pond from the TMF embankments. Cleaner tailings will be transported to a separate location within the TMF by the cleaner tailings pipeline. Deposition of cleaner tailings will occur in an area that maintains the tailings solids in a subaqueous state perpetually. The tailings pipelines will be extended towards the open pit in Year 23. Tailings deposition will occur in the open pit beginning late in Year 24 with the start-up of low grade ore processing and will continue until the end of operations.

Seepage will be primarily controlled by the low-permeability core zone constructed prior to the development of the tailings beach, the core zone key trench, and the low-permeability subgrade materials. Seepage from the TMF will result from infiltration of ponded water directly through the embankment fill and the natural ground, and from expulsion of pore water as the tailings mass consolidates.

Special design provisions incorporated into the tailings embankment design to minimize seepage losses include the development of extensive tailings beaches (which isolate the supernatant pond from the embankment), embankment drainage collection systems, and toe drains at the downstream toe of the embankments to reduce seepage gradients. Additional seepage collection ditches along the toe of the embankments will collect seepage and surface runoff, and direct the flow to the pumpback systems.

The primary objective of the closure and reclamation initiatives will be to eventually return the TMF to a self-sustaining facility that satisfies the end land use objectives. The TMF is designed to maintain long-term stability, protect the downstream environment, and manage surface water.

The project has undergone a series of design changes since the last feasibility study to both optimize the mine site footprint and general arrangement of the project, and to reduce and mitigate the potential environmental impacts resulting from the development of the project. These design changes include the following significant modifications:

- earlier saturation of the PAG waste rock stockpile within the TMF
- modification of the TMF embankment cross section to improve constructability, reduce seepage potential, and improve long-term access to downstream monitoring features
- relocation and reconfiguration of the non-PAG waste rock stockpile, overburden, and topsoil stockpiles, and
- separation of low-grade ore by geochemical classification, and relocation of stockpiles to reduce the potential for unrecoverable seepage.

The arrangement of the project infrastructure has been optimized to reduce the potential for environmental impacts by collection and storage of surface water and groundwater in contact with mine facilities for the duration of operations, and discharge of water from the TMF pond in closure. Predictions of changes to streamflow were completed using this water management strategy and corresponding predictions of water quality were also produced.

The project area generates a net surplus of water for the entire operating life of the mine. Fresh water diversions were designed in order to divert as much fresh water as practical. Diversion of runoff by gravity from much of the undisturbed catchment area reporting to the TMF pond was impractical as a long-term solution due to the shape of the TMF catchment. The TMF pond was predicted to accumulate nearly 200 Mm^3 by the end of mine life prior to TMF discharge, with an annual water surplus in excess of 5 Mm^3 per year.

The storage of all surplus water in the TMF pond is a primary driver for TMF embankment construction, and requires adequate tailings distribution to maintain the minimum required tailings beaches adjacent to the embankments. This increases the complexity of the TMF design and monitoring requirements. Reducing surplus water would have a beneficial impact on the project.

It is recommended that a conceptual surplus water management plan be developed to investigate options for removal of surplus fresh water and operational discharge of mine water to support permitting of the project. The following concepts should be considered:

- Staged capture of undisturbed runoff from the TMF and pumped diversion to T-creek during operations
- Operational discharge from the Non-PAG waste rock stockpile water management pond, and
- Operational discharge from the TMF supernatant pond and water management ponds.

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ABBREVIATIONS

AP	acid potential
ARD	acid rock drainage
B.C.	British Columbia
BMPs	best management practices
CDA	Canadian Dam Association
EDGM	earthquake design ground motion
EL	elevation
FS	Feasibility Study
HCMC	Harper Creek Mining Corp.
hp	horsepower
IDF	inflow design flood
km	kilometre
KP	Knight Piésold Ltd.
LGO	low-grade ore
m	metres
MAP	mean annual precipitation
MCE	maximum credible earthquake
MDE	maximum design earthquake
Merit	Merit Consultants International Inc.
ML	metal leaching
Mm^3	million cubic metres
m^3/s	cubic metres per second
MSC	Meteorological Services of Canada
Mt	million tonnes
MTOs	material take-offs
Nilsson	Nilsson Mine Services Ltd.
Non-PAG	non potentially-acid generating
NP	neutralization potential
OBE	operations basis earthquake
PAG	potentially-acid generating
PEA	Preliminary Economic Assessment
PET	potential evapotranspiration
PGA	peak ground acceleration
PLT	plate load test
PMF	probable maximum flood
PMP	probable maximum precipitation
QA	quality assurance
RMR	rock mass rating
RQD	rock quality designation
SPMDD	standard proctor maximum dry density
SRK	SRK Consulting Inc.
The project	Harper Creek Project
TMF	tailings management facility

TPD .... tonnes per day
t/m^3 .... tonnes per cubic metre
UCS .... unconfined compressive strength
USCS .... Unified Soil Classification System
WMP .... water management pond
YMI .... Yellowhead Mining Inc.

1 – INTRODUCTION

1.1 PROJECT DESCRIPTION

The Harper Creek Project is a large copper-gold-silver deposit located approximately 150 km northeast of Kamloops in south-central British Columbia (BC), as shown on Figure 1.1. The project involves a conventional truck-shovel open pit mine and 70,000 tonnes per day (TPD) processing plant, which is designed to process the copper sulphide ore and produce marketable copper concentrate over a 28 year mine life. Road access to the project site will be over existing Highway 5 from Kamloops to Vavenby, and then via upgraded existing secondary and forestry roads from Vavenby to the project site. The Canadian National rail line is located 24 km away and will connect the property to the Port of Vancouver, an approximate rail distance of 560 km. Power interconnection to the provincial grid is anticipated at a tie-in location near Vavenby.

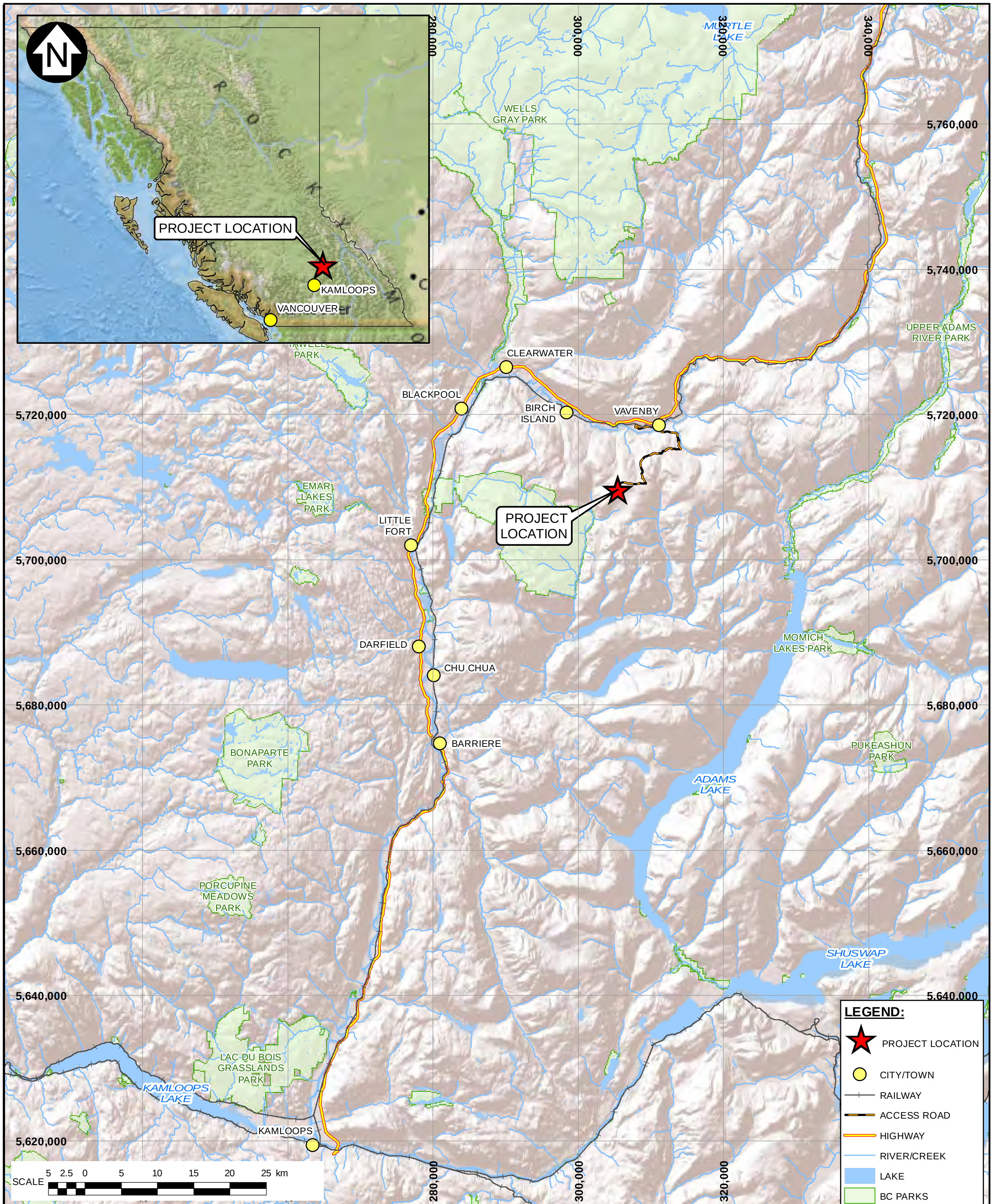
1.2 PROJECT HISTORY

Copper mineralization was first discovered at Harper Creek in 1966 by Noranda Exploration Company and Quebec Cartier Mining Company, a wholly owned subsidiary of US Steel. The two companies carried out surface exploration independently in separate areas, and then cooperatively under a joint venture agreement. Drilling on the main deposit totaled 25,806 m in 161 holes. An economic assessment of the deposit in 1972 concluded that the rate of return for the proposed operation was not sufficient to warrant production, and the joint venture was terminated in 1974.

The properties lay idle until 1986, when Aurun Mines optioned the Quebec Cartier ground. A pre-feasibility study, commissioned by Aurun Mines, was completed by Phillips Baratt Kaiser Engineering Ltd. in 1986. This agreement was terminated in 1991, and the Quebec Cartier claims were transferred to Cygnus Mines Limited, another wholly owned subsidiary of US Steel. In 1996, American Comstock Exploration Ltd. purchased the Noranda ground and acquired an option on the Cygnus claims. American Comstock drilled eight diamond holes and shortly thereafter dropped the option. Over the next few years, some of the Noranda claims were abandoned, and in 2004 American Comstock sold six legacy claims to Argent Resources Ltd. A series of negotiations and agreements then ensued whereby the properties were consolidated and mineral tenures were variously sold or optioned to Yellowhead Mining Inc. (YMI). In 2006, YMI began the company's first phase of field exploration on the Harper Creek claims. YMI completed a series of exploration phases and resource estimate updates between 2006 and 2012.

An independent Technical Report and Preliminary Economic Assessment (PEA) for the Harper Creek Project was completed by Wardrop Engineering Inc., a Tetra Tech Company and filed on SEDAR on April 1, 2011 (Wardrop, 2011). The report concluded that the project offered exceptional potential as one of the largest undeveloped copper projects in Canada and recommended YMI proceed to a Feasibility Study (FS).

YMI subsequently commissioned Merit Consultants International Inc., Knight Piésold Ltd., Nilsson Mine Services Ltd., All North Consultants, and other specialist consultants to undertake a FS for the Project. The Technical Report for the FS was filed on SEDAR on March 29, 2012 (Merit, 2012).



NOTES:

1. BASE MAP: TRIM AND NTS MAPPING, ESRI ARCGIS ONLINE SHADED RELIEF.
2. COORDINATE GRID IS IN METRES. COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N.
3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:750,000 FOR 8.5x11 (LETTER) PAPER.

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

PROJECT LOCATION

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P/A NO. VA101-458/11 | REF NO. 1

FIGURE 1.1 REV 0

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1.3 SITE SELECTION

Knight Piésold Ltd. (KP) was commissioned in December 2010 to review the alternatives assessment of the Tailings Management Facility (TMF) locations identified in earlier internal studies conducted by YMI. Three potential TMF sites were identified in these studies. These sites were located to the west, south, and east of the open pit and were identified as TMF-1, TMF-2, and TMF-3, respectively. The option TMF-1 was located within the Harper Creek valley and would have a direct impact on fish habitat. TMF-3 was located at a prohibitively large distance from the plant site in comparison to the other options, and would have caused additional disturbance in an otherwise un-impacted catchment area. TMF-2 presented itself as the preferred location for a mine waste management facility for a variety of reasons.

TMF-2 (the TMF) is located at the upper reaches of its catchment area, which reduces the complexity of water diversion measures and limits the flow reduction to the downstream watercourse to the maximum extent practical. It is located in a shallow bowl shaped basin that drains towards Harper Creek down a steep unnamed bedrock channel (henceforth referred to as T-creek), which acts as a natural fish barrier. The presence of the fish barrier reduces the direct environmental impacts that would occur compared to other options. The TMF is located near to the mine site, which allows for the clustering of facilities to reduce overall cost of mine waste and water management, and limits the extents of the mine site and impacted areas.

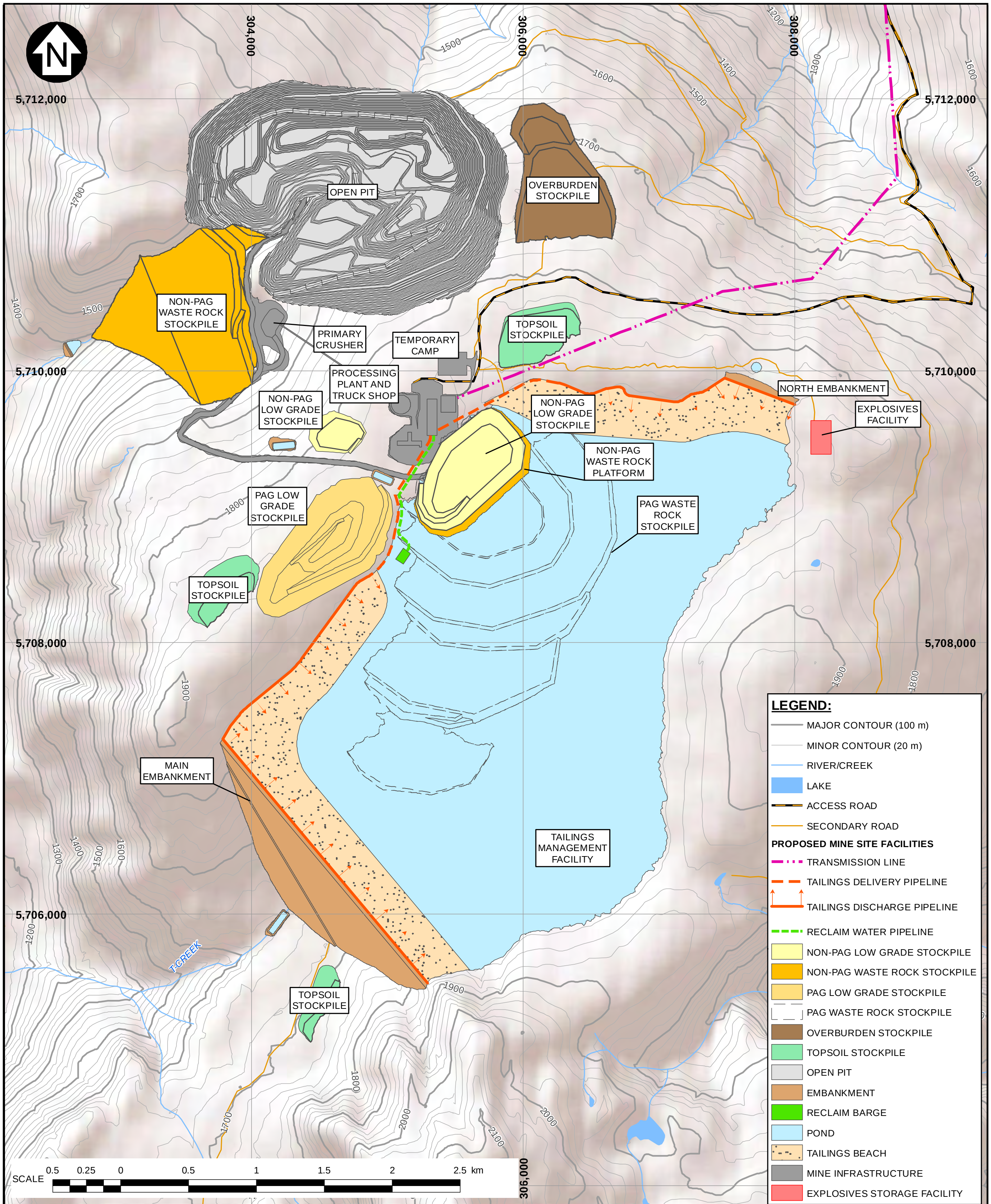
1.4 SCOPE OF WORK

KP completed engineering studies to update the design of the mine waste and water management facilities to contribute to an updated project FS. The FS update is being completed in conjunction with Merit Consultants International Inc. (Merit) and Nilsson Mine Services Ltd. (Nilsson). The facilities include the TMF, waste rock and overburden stockpiles, low-grade ore stockpiles, and open pit. An overview of the project area general arrangement is shown on Figure 1.2. A package of design drawings were developed for the FS to facilitate preparation of the material take-offs to complete an economic evaluation of the project by Merit. The design drawings are included in this report as Appendix A. The design of these facilities is the subject of this report.

1.5 REFERENCE REPORTS

The following KP reports were considered in the preparation of the update to the FS.

- **2011 Site Investigation** – KP report *2011 Geotechnical Site Investigation Factual Report*, Ref. No. VA101-458/3-1 dated February 29, 2012.
- **2012 Site Investigation** – KP report *2012 Geotechnical Site Investigation Factual Report*, Ref. No. VA101-458/7-1 dated July 25, 2013.
- **Terrain Mapping** – KP report *Reconnaissance Terrain Mapping*, Ref. No. VA101-458/4-4 dated November 28, 2012.
- **Watershed Modelling** – KP report *Watershed Modelling*, Ref. No. VA101-458/14-1 Rev 0 dated August 25, 2014.
- **Numerical Groundwater Modelling** – KP report Numerical Groundwater Modelling, Ref. No. VA101-458/14-2 Rev A dated September 10, 2014.
- **Water Quality Modelling –** KP report Water Quality Predictions, Ref. No. VA101-458/14-3 Rev A dated July 30, 2014.



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3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:40,000 FOR 8.5x11 (LETTER) PAPER.
4. FACILITIES REPRESENT MAXIMUM FOOTPRINT FOR MINE LIFE AND ARE YEAR 24, EXCEPT:
   - OPEN PIT TAILINGS (YEAR 24) ARE REMOVED
   - PIPELINES ARE YEAR 20
   - PAG LOW GRADE STOCKPILE IS YEAR 15
   - NON-PAG LOW GRADE STOCKPILE (WEST OF PLANT SITE) IS YEAR 3.

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HARPER CREEK PROJECT

PROJECT LAYOUT - MAXIMUM FOOTPRINT

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FIGURE 1.2

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2 – MINE WASTE CHARACTERIZATION

2.1 GENERAL

Mine waste characterization studies were completed to support the FS design and included assessments of the mine waste rock, low-grade ore, overburden, and tailings. These studies were used to evaluate the metal leaching (ML) and acid rock drainage (ARD) potential of these materials, and to establish mine waste and water management concepts for the design of the project facilities.

2.2 WASTE ROCK

ML/ARD potential was characterized by SRK Consulting Inc. (SRK) using a sample set tested specifically for acid-base accounting and a larger sample set analyzed for trace elements as part of exploration activities. Conclusions from the assessment indicated that correlation of ARD potential with rock types were weak. The study suggested block modelling of ARD potential would be required to estimate volumes of potentially-acid generating (PAG) waste rock and Non-PAG waste rock, and to evaluate the potential to segregate these waste rock materials during mining. Finally, the study suggested that based on elevated concentrations of several elements, leaching under non-acidic conditions may also be a consideration for waste rock management (SRK, 2012).

The assessment by SRK described the methodology for establishing site specific waste rock classification criteria. Site specific neutralization potential (NP*) can be compared against the acid potential (AP) in order to classify the waste rock for waste rock management.

The waste rock classification criteria developed for the project were as follows:

- PAG – NP*/AP < 2
- Non-PAG – NP*/AP > 2

Block modelling of waste rock by the classification criteria listed above was included in development of the mine plan and mine waste production schedule for the project by Nilsson. The waste rock tonnages expected during development of the open pit are as follows:

- PAG – 237 million tonnes (Mt)
- Non-PAG – 265 Mt

The time to acid generation of the PAG waste rock was estimated to be on the order of decades.

2.3 LOW-GRADE ORE

The ML/ARD potential of low-grade ore (LGO) can be classified using the same criteria as the waste rock shown above. The LGO will be segregated during mining, temporarily placed in surface stockpiles, and processed in the last four years of operations. Block modelling of LGO was also completed during development of the mine plan. The expected tonnages to be stockpiled during development of the open pit are as follows:

- PAG LGO – 86 Mt
- Non-PAG LGO – 47 Mt

2.4 TAILINGS

The project milling operation will produce two tailings streams using conventional milling methods to process the ore. The process includes a primary crusher, primary grinding circuit, flotation circuit, and regrinding and secondary flotation circuits. The mill is expected to operate at a nominal throughput of 70,000 TPD. The two types of tailings are designated as rougher scavenger (bulk) tailings and cleaner scavenger (cleaner) tailings. The bulk tailings stream consists of approximately 93% of the total tailings stream with cleaner tailings representing the remaining balance of 7%. The bulk tailings slurry concentration was estimated to be 34.5% by dry weight, with a solids density of 2.66 tonnes per cubic metre (t/m^3). The cleaner tailings slurry concentration was estimated to be 32.7% by dry weight, with a solids density of 3.11 t/m^3.

Lock cycle metallurgical test work produced one sample each of cleaner and bulk tailings. The geochemical characteristics of both tailings types were evaluated. The cleaner tailings contained high levels of sulphur and are PAG, while the bulk tailings contained lower levels of sulphur and are Non-PAG (SRK, 2012).

Two tailings samples from the expected bulk tailings stream were provided by YMI for testing. The test program included index testing to enable geotechnical classification of the materials, and slurry settling, air drying, consolidation and permeability testing to determine the characteristics of the tailings for a range of conditions expected to be representative of field conditions. Test work was completed on tailings samples at solid contents of 35, 45, and 55%. Complete results and details of the tailings testing are provided in Appendix B. A description of the test work and a summary of the results for tailings with a solids content of 35% are provided below.

The specific gravity of the tailings solids was determined to be 2.79 and the material can be described as a non-plastic, fine-grained sandy-silt with traces of clay. The particle size distribution of the tailings sample comprised approximately 46-52% fine sand, 44-50% silt, and 4% clay. The Unified Soil Classification System (USCS) has been used for describing and categorizing soil within groups to allow for the development of distinct soil properties. The tailings can be classified as sand with fines (SM) and a fine-grained soil with very fine sands (ML) depending on the particle size distribution.

Undrained settling, drained settling, and air drying tests were carried out to provide information on the effect of initial slurry solids content on the settling and permeability characteristics of the material and the effect on water recovery and achieved density. Slurry settling (sedimentation) tests provided an estimate of the density to which the tailings slurry will settle in a sub-aqueous environment, under drained and undrained conditions. These tests provided an indication of the tailings dry density achieved in a storage facility after settling and before any significant consolidation occurs. Air drying tests were carried out on the tailings samples to determine the effect of air drying after initial slurry settling and removal of supernatant water.

The tests were performed for a target solids content equal to 35% and the main findings were as follows:

- The settled dry density of the tailings was 1.2 t/m^3 for undrained and drained settling conditions, with a measured supernatant water release of approximately 75%.
- The tailings slurry took up to four days to complete undrained settling and less than two days to complete drained settling.
- A tailings dry density of 1.5 t/m^3 was achieved under air drying conditions.

Laboratory tests carried out to determine the consolidation and permeability characteristics of the tailings included slurry consolidometer, a low stress slurry consolidation test and a falling head permeability test (conducted on settled tailings after completion of drained settling). Relationships between coefficient of consolidation, void ratio and vertical coefficient of permeability versus effective stress have been developed for the tailings. The calculated coefficients of consolidation for the tailings range from 20 m^2/year at very low stresses (representing unconsolidated or fresher tailings near surface) to over 1600 m^2/year at high stresses (representing more consolidated or deeper tailings within the deposit). The permeability of the tailings ranged from $1x10^{-4}$ cm/second at low stresses to $3x10^{-5}$ cm/second at high stresses.

Tailings will be conveyed by pipelines and discharged from the embankment crests into the TMF, or into the open pit during LGO processing. Rheology testing to determine slurry flow characteristics of the tailings slurry was not completed as part of the study.

2.5 OVERBURDEN

The overburden in the deposit area generally ranged from scarce to greater than 10 m in thickness. Overburden was scarce in the southeast area of the deposit with only a thin veneer of topsoil overlying bedrock. Bedrock at surface was typically weathered and rippable.

The thickest regions of overburden were identified in the central and northwest areas of the proposed open pit. The overburden in these thicker areas was characterized through visual classification in machine excavated test pits and existing road cuts, and completion of five (5) particle size analyses. The details of the site investigation and laboratory program were presented in the 2011 Site Investigation Report (Knight Piésold Ltd., 2012a). The overburden typically consisted of silty-sand with some gravel, and is classified by the USCS as a coarse grained soil with gravel and fines (SM-SC and GM-GC).

The USCS classification group allows for comparison of anticipated geotechnical properties of the soil with published typical ranges of these properties. These properties include permeability, shear strength, compaction characteristics, workability and volume change potential of a soil, and how it will be affect by water, frost and other physical conditions.

The overburden tonnage estimated during stripping of the open pit was 39 Mt.

2.6 MINE WASTE MANAGEMENT CONCEPTS

Waste management concepts for the various classifications of mine waste material are outlined below:

- Non-PAG waste rock
  - Used to construct the TMF embankments, mine site roads, and Non-PAG LGO platform.
  - Surplus and unsuitable materials disposed of in one on-land waste stockpile near the pit.
  - On-land stockpile progressively reclaimed during operations as final slopes and grades are reached.
- PAG waste rock
  - Used to construct the upstream zone of the TMF main embankment during first five years.
  - Surplus co-disposed of within the TMF in such a manner that it is typically flooded within 1 year by the supernatant pond.
- Overburden
  - Best available material used to construct low-permeability zone of the TMF embankment raises.

  - Used to construct TMF embankment shell zone when Non-PAG waste rock is unavailable.
  - Surplus and unsuitable materials disposed of in one on-land waste stockpile to the east of the open pit.
  - On-land stockpile progressively reclaimed during operations as final slopes and grades are reached.
- Non-PAG LGO
  - Up to 7.5 Mt temporarily stockpiled near primary crusher and processed within first five years of operations.
  - Balance stockpiled within TMF basin on a Non-PAG waste rock platform at an elevation above the ultimate extents of the TMF.
  - Balance processed during the final four years of operations.
- PAG LGO
  - Stockpiled adjacent to the TMF basin on an engineered sub-grade.
  - Processed during the final four years of operations.

3 – SITE CHARACTERIZATION

3.1 PHYSIOGRAPHY

The Project is located in the Shuswap Highlands, which are characterized by gently sloping plateau areas dissected by deep valleys. The topographic relief in the region is steep to moderate with elevations ranging from 450 m in the North Thompson River valley to highs of 1900 m on the ridges surrounding the mine site area.

The mine site is situated on gently sloping upland ridges flanked by steepened valley slopes. These valleys include the Harper Creek valley to the west and the Barriére River valley to the east, with the moderately sloped Thompson River valley to the north. The ground surface elevation of the deposit area ranges from 1575 m to 1800 m, and the plant site is situated at an elevation of 1840 m. The elevation of the TMF area ranges from 1600 m to 1900 m. The area was historically glaciated and mountain tops are typically rounded. The mine area is covered mostly by thick coniferous forest with heavy underbrush, however, in some places there are open clear cuts. Much of the Harper Creek area has been logged and at higher elevations there are small marshy alpine meadows.

The TMF is located within a broad, shallow valley, which drains southward down a steep bedrock canyon into Harper Creek at an elevation of 1100 m. The side slopes of the TMF basin are gentle to moderately sloped, and the centre of the basin features hummocky terrain with swampy, poorly drained areas.

The Project is situated on the watershed divide between Harper Creek and the North Thompson River. Harper Creek flows south from the Project site and discharges into the western end of North Barriére Lake, just upstream of the lake outlet. Barriére River flows out of the lake, flowing in a southwesterly direction for approximately 25 km before meeting the North Thompson River 58 km north-northeast of Kamloops. Jones and Baker Creek both drain north facing watersheds in the mine site area and flow approximately 5 km from their headwaters to the North Thompson River.

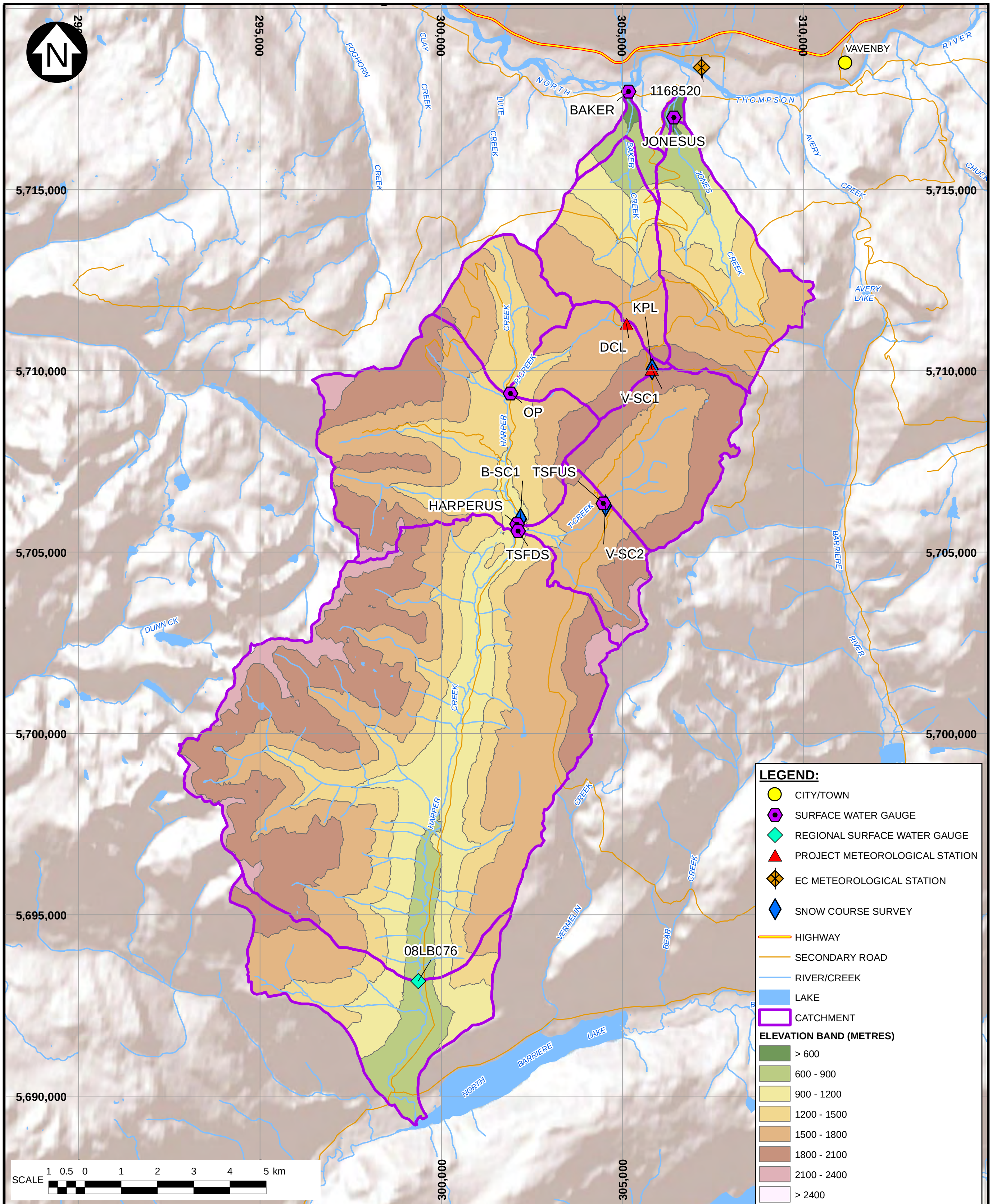
3.2 HYDROMETEOROLOGY

3.2.1 General

Meteorological and hydrological data have been collected at the project site since late 2007 and 2008, respectively. The short-term site-specific data have been correlated with long-term regional data to quantify the meteorological and hydrological characteristics of the project area for the purpose of water balance modelling, engineering design, and environmental assessment. The key findings are summarized in the sections that follow.

3.2.2 Monitoring Locations

Hydrometric data are being collected at six stations in the project area to support hydrometric characterization of the mine site area. The hydrometric stations are identified as OP, HARPERUS, TSFUS, TSFDS, BAKER, and JONESUS. Meteorological data are being collected at two climate stations, which are identified as DCL (elevation 1680 m) and KPL (elevation 1837 m). The monitoring locations for the project site are shown on Figure 3.1.



NOTES:

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3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:150,000 FOR 8.5x11 (LETTER) PAPER.

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BASELINE MONITORING LOCATIONS

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FIGURE 3.1

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3.2.3 Mean Annual Precipitation

The long-term mean annual precipitation (MAP) for the project area was estimated to be 1025 mm, at an elevation of approximately 1680 m. This value was derived from the DCL meteorological station and several regional stations operated by the Meteorological Services of Canada (MSC) branch of Environment Canada.

3.2.4 Monthly Precipitation Distribution

The monthly distribution of precipitation was estimated for the purpose of water management planning. Approximately 61% of the annual precipitation falls as snow between October and April. The remaining 39% of the annual precipitation falls as rain, which may occur any month of the year, but largely falls in the period of April to September. The monthly precipitation statistics that define these distributions are summarized in Table 3.1.

Table 3.1 Monthly Precipitation Distribution

Unit	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Total Precipitation (mm)	132	76	68	37	57	82	69	64	57	122	122	137	1025
%/month	13%	7%	7%	4%	6%	8%	7%	6%	6%	12%	12%	13%	100%
Rain (mm)	2	1	1	25	57	82	69	64	57	41	2	2	404
% Precipitation as Rain/Month	1%	1%	2%	67%	100%	100%	100%	100%	100%	34%	1%	1%	39%
Snow – SWE (mm)	130	75	67	12	0	0	0	0	0	81	120	135	620
% Precipitation as Snow/Month	99%	99%	98%	33%	0%	0%	0%	0%	0%	66%	99%	99%	61%

NOTES:
1. SWE = SNOW WATER EQUIVALENT.
2. ADOPTED FROM WATERSHED MODELLING REPORT (Knight Piésold Ltd., 2014a).
3. PRECIPITATION DISTRIBUTIONS APPLY TO EL. 1680 m, AND CAN BE SCALED TO OTHER ELEVATIONS BY APPLYING OROGRAPHIC FACTORS OF 5% PER 100 m DURING THE NON-FREEZE MONTHS (MAY – SEPTEMBER) AND 10% PER 100 m DURING THE WINTER MONTHS (OCTOBER TO APRIL).

3.2.5 Evapotranspiration / Lake Evaporation

Lake evaporation for the site was estimated according to common empirical equations for potential evapotranspiration (PET). PET values are generally representative of lake evaporation. The empirical Thornthwaite equation was used with the measured site temperature record and long-term synthetic temperature record to estimate a mean lake evaporation value (potential evapotranspiration) of 412 mm.

3.2.6 Return Period Extreme Precipitation

Estimates of extreme precipitation are required for developing water management designs. Estimated 24 hour extreme rainfall values were prepared for a range of return periods and for the probable maximum precipitation (PMP). Extreme precipitation events and corresponding return period for the project site are summarized in Table 3.2.

Table 3.2 Extreme Precipitation Return Period Values

Return Period (years)	24-Hour Extreme Rainfall (mm)
2	35
5	46
10	53
20	60
50	69
100	75
200	82
500	91
1000	97
PMP	300

3.2.7 Mean Annual Runoff

Regional runoff patterns are characterized by low flows during the winter months when precipitation falls almost exclusively as snow, high flows during the spring and early summer snowmelt freshet, low flows during the dry late summer months, and moderate flows during the fall months as precipitation increases. The change in runoff with elevation is quite evident. Lower elevation watersheds generate an earlier spring freshet as above freezing temperatures arrive earlier at the lower elevations.

The unit runoff and hydrograph shape of the TMF area are expected to most appropriately represent streamflow patterns in the project area. The annual hydrograph in the Project area has a uni-modal shape, with the majority of runoff occurring in May and June during the snowmelt freshet.

Estimates of mean monthly and annual unit runoff were calculated in the Watershed Modelling Report for the project (Knight Piésold Ltd., 2014a). These estimates are summarized in Table 3.3 for the TMF area.

Table 3.3 Project Site Long-term Unit Runoff

Station	Unit	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
TSFUS	Unit Runoff (l/s/km^2)	0.0	0.0	0.0	8.2	67.3	91.2	37.9	9.4	5.0	4.4	1.2	0.0	18.7
	Discharge (m^3/s)	0.00	0.00	0.00	0.12	1.01	1.37	0.57	0.14	0.08	0.07	0.02	0.00	0.28
	Runoff (mm)	0.0	0.0	0.0	21	180	236	102	25	13	12	3	0	592
	% of Total Runoff	0	0	0	4	30	40	17	4	2	2	1	0	100
	10 year wet discharge (m^3/s)	0.00	0.00	0.00	0.36	1.47	2.03	1.25	0.30	0.25	0.17	0.06	0.00	0.40
	10 year dry discharge (m^3/s)	0.00	0.00	0.00	0.00	0.57	0.76	0.08	0.01	0.00	0.00	0.00	0.00	0.18

NOTES:
1. TMF AREA IS REPRESENTATIVE OF THE PROJECT AREA DUE TO ITS MEDIAN ELEVATION AND DRAINAGE AREA BEING SIMILAR TO THAT OF THE PROJECT AREA FACILITIES.
2. THE CALCULATION OF RUNOFF IN mm ASSUMES 28 DAYS IN FEBRUARY.
3. ADOPTED FROM TABLES F-1 AND F-2 OF THE WATERSHED MODELLING REPORT (Knight Piésold Ltd., 2014a).

The mean annual unit runoff for the TMF area was estimated to be 18.7 l/s/km^2, as shown in Table 3.3. The highest monthly runoff tends to occur in June, with 91.2 l/s/km^2, and the lowest occurs in December to March, when surface flows are negligible. The annual unit runoff equates to an annual runoff depth of 592 mm.

3.3 REGIONAL SURFICIAL GEOLOGY

The surficial deposits and landforms within the project area are from the Fraser Glaciation, the last period of ice sheet glaciation in British Columbia. The project is predominantly covered with glacial till with colluvium and bedrock exposures becoming more widespread at higher elevations. Mountain tops are rounded from the large ice sheet thickness during the last glaciation.

The majority of the Project site is located in mountainous upland watershed area that drains into the low-lying valleys of Harper, Baker and Jones Creeks and into the North Thompson or North Barrieré River valleys. In these upland areas, a discontinuous veneer of overburden covers bedrock. Overburden deposits are predominantly glacial till, colluvium with some organic material. At locations where overburden is thin or absent, weathered bedrock including schist, phyllite, and granodiorite intrusions are present at the surface interspersed with pockets of glacial till. Glaciolacustrine deposits have been identified in portions of the T-creek valley. Glaciolacustrine materials were deposited during a period of deglaciation as a result of meltwater detention caused by ice damming of major drainages (Lett et al., 1999).

Surficial materials within the lower-lying river and creek valleys of Harper, Baker, Jones Creeks and North Thompson or North Barrieré River valleys are composed of fluvial and glaciofluvial deposits. Glaciofluvial materials typically comprised of sand, silt and gravel were deposited along valleys as outwash from ablation of glacial ice (Paulen et al., 2000). These materials are present in the lower reaches of the Harper Creek, Jones Creek, and Baker Creek watersheds as well as in the North Thompson and North Barrieré River valleys. Fluvial materials have been mapped in the major drainages, including the North Thompson River valley and the lower reaches of Jones and Baker Creeks. These areas are generally outside the mine footprint and only road, power line and railway alignments are affected by these materials.

3.4 REGIONAL BEDROCK GEOLOGY

The regional geology consists of deformed and metamorphosed Lower Cambrian and Upper Devonian to Mississippian sedimentary and volcanic rocks with sills and dikes consisting of foliated granite to diorite. These rock units comprise what is known as the Eagle Bay Assemblage. This assemblage is intruded by Middle to Upper Jurassic and Cretaceous granitic plutons. Eocene-age Kamloops Group volcanic rocks overlay the Eagle Bay Assemblage rocks.

The regional structural geology consists typically of east-west striking, low to moderately dipping stratigraphy. Thrust faults disrupt the stratigraphic sequence by positioning Cambrian rocks overtop of younger Paleozoic strata. A series of steeply southeast-dipping normal faults are present, hosting Tertiary dikes.

The Harper Creek deposit is an extensive volcanogenic sulphide system, with a mineralized zone spanning 2000 m along strike, 2000 m down dip and lies within a 1000 m thickness of volcano-sedimentary stratigraphy. The deposit is hosted in the Eagle Bay Assemblage, specifically within the Lower Paleozoic and Greenstone Belts. The deposit is interpreted to be a polymetallic volcanogenic sulphide deposit comprised of lenses of disseminated, banded and fracture-filling iron and copper sulphides. The mineralization consists of chalcopyrite with accessory pyrite, magnetite and pyrrhotite. There are significant amounts of Au and Ag present within the mineralized zone. The mineralization is tabular and strikes east-west, dipping at 15° to 25°, with sulphide lenses up to tens of metres thick. This tabular mineralization comprises the central and west zones of the pit. There is a broad lower-grade zone of Cu with Au/Ag that is linked to multi-phased stringer or feeder zones within the eastern zone of the pit area (Knight Piésold Ltd., 2013b).

3.5 SEISMICITY

3.5.1 General

A seismicity assessment was carried out for the project, including a review of the regional seismicity and a probabilistic seismic hazard analysis (Appendix C). A seismic hazard analysis is required to provide seismic design parameters for the design of the TMF and for other facilities at the project site, including mine waste rock stockpiles and water management dams. Design ground motion parameters provided by the seismic hazard analysis include peak ground acceleration, spectral acceleration (defining uniform hazard spectra) and earthquake magnitude.

3.5.2 Regional Tectonics and Seismicity

The project is situated within south-eastern B.C., where the level of recorded historical seismic activity has been low. The maximum earthquake magnitude for this region of B.C. is estimated to be about magnitude 7.0, with an upper bound estimate of magnitude 7.4, based on historical earthquake data and the regional tectonics (Adams and Halchuk, 2003).

The level of seismicity in the interior of B.C. and the Rocky Mountain region drops off rapidly with distance from the west coast to the north. The largest earthquake recorded in the southern Cordillera region was an event of about magnitude 6.0 in 1918, located in the Valemont area of the Rocky Mountain trench. More recently, a magnitude 5.4 earthquake occurred near Prince George in 1986 causing minor damage, and a magnitude 5.3 earthquake occurred in 2001 east of Dawson Creek.

The seismic hazard along the west coast of B.C. is significant due to the subduction zone earthquakes along offshore faults and within the subducting oceanic tectonic plate. There is potential for very large earthquakes of magnitude 8.0 to 9.0 along this Cascadia subduction zone. However, such an event would be located over 450 km southwest of the project site, and therefore the amplitude of ground motions experienced at the site would be very low due to attenuation over such a large distance. Peak ground accelerations on rock at the project site from a great subduction earthquake would likely be less than 0.05g.

There is also potential for intraslab earthquakes, occurring deep within the subducted Juan de Fuca plate that extends eastward beneath the North American plate. These events, which have the potential to be as large as about Magnitude 7.5, would likely occur over 300 km to the southwest, at a depth of over 40 km. Ground motions on rock experienced at the project site for this type of subduction earthquake are likely to be less than 0.1g. The seismic hazard at the project is predominantly from potential shallow earthquakes occurring closer to the site.

3.5.3 Seismic Hazard Analysis

The seismic hazard for the project has been defined using probabilistic methods of analysis. Historical earthquake data and regional tectonics were examined to identify potential seismic sources and maximum earthquake magnitude for each source.

The ground motions experienced at the project site are dependent on the regional ground motion attenuation characteristics and the earthquake source mechanism. The attenuation models for shallow crustal earthquakes were based on a set of four ground motion attenuation models, known as the New Generation Attenuation (NGA) relations (Earthquake Spectra, 2008). Attenuation relationships used for the interface subduction and intraslab subduction earthquake source zones were based on relationships developed specifically for oceanic subduction zone earthquakes (Youngs, 1997 and Atkinson, 2003).

The computer program EZ-FRISK was used to develop a seismic hazard model for B.C. and the surrounding regions (EZ-FRISK, 2008). The model was used to determine the relationships between PGA and annual frequency of occurrence for the project site. Median hazard values of PGA were determined for return periods up to 10,000 years. Predicted values for lower return periods were compared with those provided by the NRC seismic hazard database and were very similar. A summary of the probabilistic hazard analysis is provided in Table 3.4.

Table 3.4 Summary of Probablistic Seismic Hazard Analysis

Return Period (Years)	Probability of Exceedance[1] (%)	Peak Ground Acceleration (PGA)[2]	
		Median PGA (g)	Estimated Mean PGA[3] (g)
100	21	0.03	0.04
500	4	0.07	0.08
1,000	2	0.10	0.11
2,500	1	0.14	0.16
5,000	0.5	0.16	0.19
10,000	0.2	0.23	0.26

NOTES:
1. PROBABILITY OF EXCEEDANCE CALCULATED FOR A DESIGN OPERATING LIFE OF 28 YEARS.
2. PEAK GROUND ACCELERATIONS ARE FOR SOFT ROCK / VERY DENSE SOIL (Vs_{30} = 560 M/SEC).
3. MEAN PGA VALUES ESTIMATED AS 1.15 X MEDIAN VALUES.

Deaggregation of the probabilistic seismic hazard results was carried out to provide the relative contributions of all potential seismic sources, and to more accurately define the characteristics of potential earthquakes contributing to the seismic hazard. The findings indicate that the seismic hazard for the project site is predominantly from shallow crustal earthquakes in this region of south-eastern B.C.

A design earthquake magnitude 7.0 and 7.3 have been selected for earthquake return periods of 5,000 and 10,000 years, respectively, based on the review of regional tectonics and historical seismicity, and the findings of deaggregation of the probabilistic seismic hazard.

4 – GEOLOGICAL, GEOTECHNICAL AND HYDROGEOLOGICAL CONDITIONS

4.1 GENERAL

4.1.1 Project Area Geotechnical Characterization

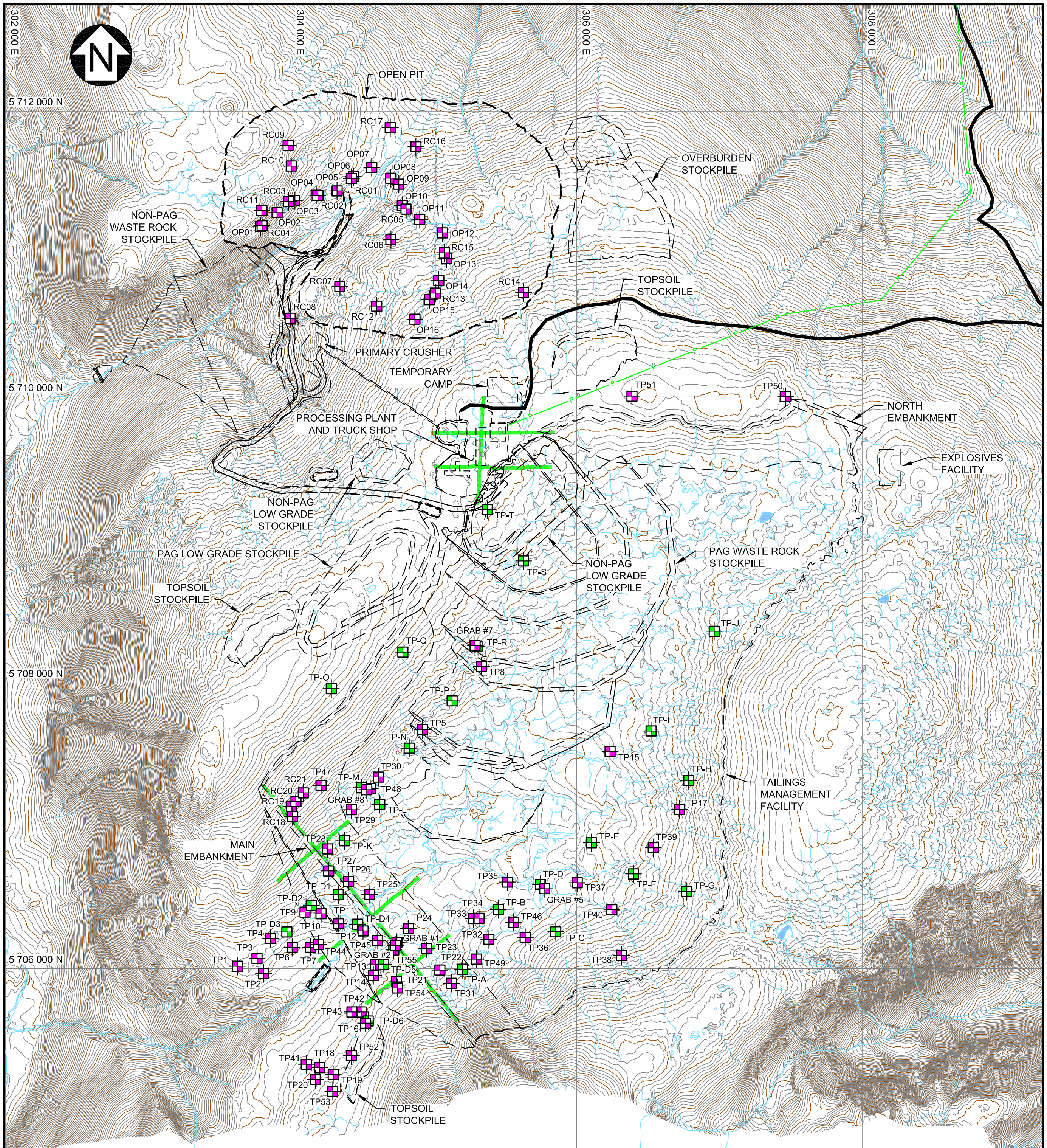
Site investigations were conducted in 2011 and 2012 to evaluate geotechnical and hydrogeological conditions for the proposed TMF and waste dumps. Drillhole, ground geophysics, and test pit locations were adjusted as the program progressed as a greater understanding of site conditions was acquired. The data collected during the site investigation programs was used to characterize the geology, hydrogeology, and geotechnical conditions at the site. Very little pre-existing geotechnical or hydrogeological information was available prior to 2011. The factual data from the 2011 and 2012 site investigation programs were reported on previously in the following documents:

- **2011 Site Investigation** – KP report *2011 Geotechnical Site Investigation Factual Report*, Ref. No. VA101-458/3-1 dated February 29, 2012.
- **2012 Site Investigation** – KP report *2012 Geotechnical Site Investigation Factual Report*, Ref. No. VA101-458/7-1 dated July 25, 2013.
- **Terrain Mapping** – KP report *Reconnaissance Terrain Mapping*, Ref. No. VA101-458/4-4 dated November 28, 2012.

The site investigation programs included the following:

- Excavation of 71 test pits and logging of 21 pre-existing road cuts.
- 32 geotechnical drillholes in and around the TMF, waste dump, and plant site areas.
- 28 overburden drillholes around the TMF and open pit, terminating in shallow bedrock.
- 7 geomechanical (oriented core) drillholes in the open pit.
- Installation of 20 long-term monitoring wells at 11 locations across the project area.
- Installation of 31 standpipe piezometers in geotechnical and geomechanical drillholes.
- In-situ packer testing conducted in bedrock in all geotechnical and geomechanical drillholes.
- Response testing conducted in all standpipe piezometers and monitoring wells.
- Laboratory rock mass strength and direct shear testing of bedrock.
- Laboratory index testing of overburden material.
- Seismic refraction surveys along the TMF main embankment and plant site areas.

The simplified project layout including the test pits and drillholes from all investigations at the site are illustrated on Figure 4.1 and Figure 4.2, respectively. Additional details on field data collection methods and findings can be found in the reference reports listed above (Knight Piésold Ltd., 2012a, 2012b, and 2013).



400 200 0 400 800 1200 1600 2000 m
SCALE A

LEGEND:

- TEST PIT (KP)
- TEST PIT (KLOHN CRIPPEN BERGER, 2008)
- MINE ACCESS ROAD
- SEISMIC REFRACTION LINE

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	WAL		

HARPER CREEK MINING CORP.

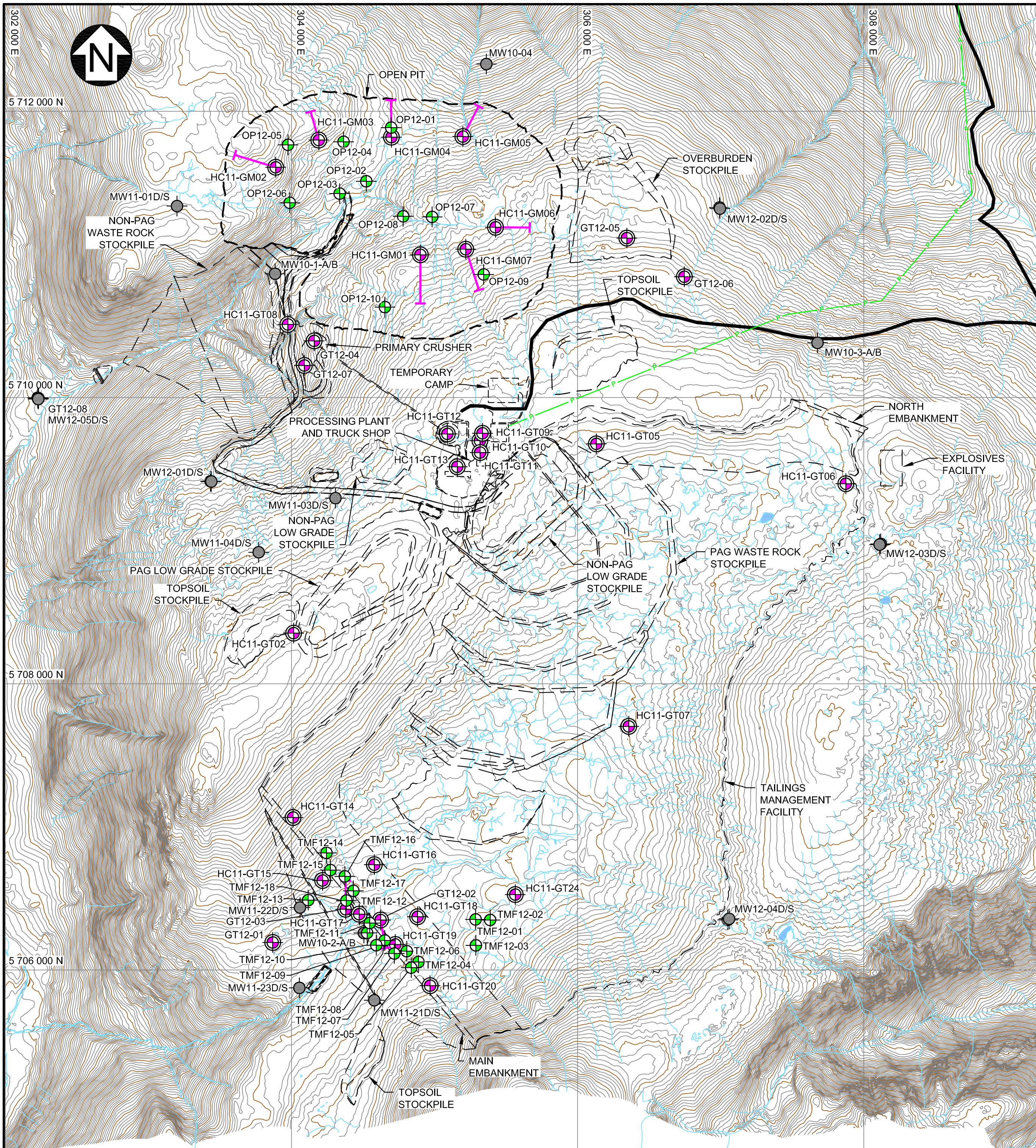
HARPER CREEK PROJECT

TEST PIT AND SEISMIC REFRACTION PLAN

Knight Piésold CONSULTING

P/A NO.	REF NO.
VA101-458/11	1

FIGURE 4.1 REV 0



LEGEND:

- GEOTECHNICAL DRILLHOLE WITH PIEZOMETER
- MONITORING WELL
- GEOTECHNICAL DRILLHOLE
- MINE ACCESS ROAD

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

SCALE A: 400 200 0 400 800 1200 1600 2000 m

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

DRILLHOLE AND MONITORING WELL PLAN

Knight Piésold CONSULTING

P/A NO.	REF NO.
VA101-458/11	1

FIGURE 4.2 REV 0

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	WAL		

4.1.2 Surficial Material Types

The stratigraphic units (corresponding USCS classification) encountered in the project area are as follows:

- **Organic Deposits** – thin veneer of topsoil across much of the project area and accumulated in poorly drained areas as swamps as a result of the decomposition of vegetation (OL, Pt).
- **Colluvium Deposits** – thin layers of colluvium, typically boulder gravel with some silt and sand, are found along the base of some steeper slopes developed on the steeper valley side slopes as a result of soil creep and landslides (GM, GW-GP).
- **Glaciolacustrine Deposits** – classified as fine grained soils silts and clays (ML-CL).
- **Glacial Till** – identified as coarse grained soils with gravels and fines (SM-SC and GM-GC).
- **Weathered Bedrock** – deformed and metamorphosed, sedimentary and volcanic rocks.

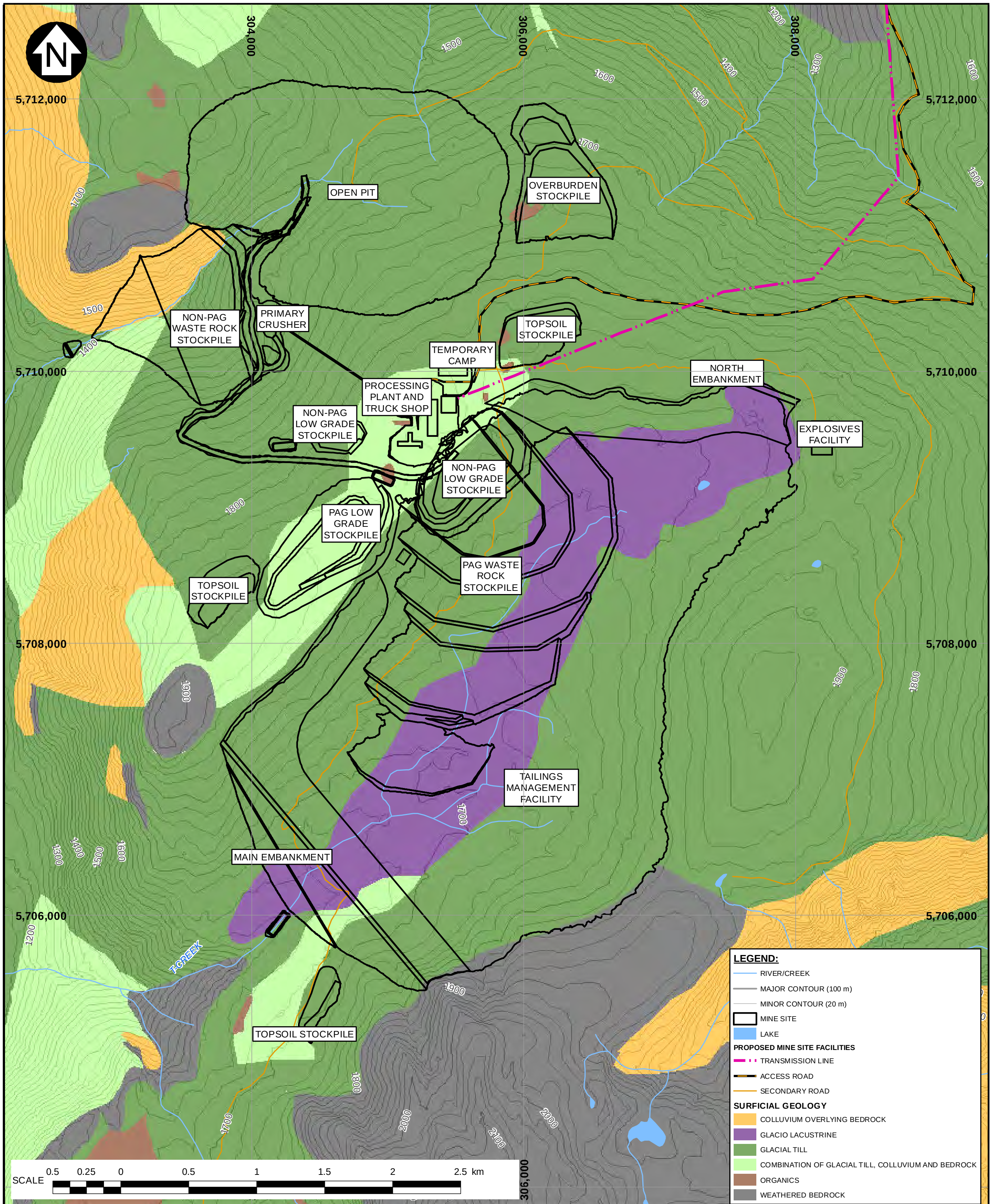
The distribution of the surficial materials at the site is shown on Figure 4.3 and descriptions of each stratigraphic unit are provided in the sections below.

Organic soils accumulated in swamps as a result of the decomposition of vegetation with the retreat of the ice sheet. A thin veneer of topsoil ranging from 0.1 to 0.5 m deep covers much of the project area. Thicker layers of organics are present within the poorly drained areas of the property, particularly in the centre of the TMF basin, and consist of brown and block spongy fibrous peat to organic silt wet fibric to mesic plant material in various stages of decomposition.

Colluvium has developed locally on the stepper valley side slopes as a result of soil creep and landslides. A surface veener of colluvium is expected in the steeper areas of terrain and weathered bedrock colluvium is expected to be more prevalent on the moderately steep, south-facing slopes in the project area. The colluvium is comprised of silty sand, gravel and cobbles and the consistency of this material is expected to vary locally. Colluvium was only encountered in one area of the project footprint – the south facing slope of the P-creek valley. This material is also expected in the lower reaches of Harper, Jones and Baker Creeks.

Glaciolacustrine deposits developed from glacial lakes locally on the flat mountain-top areas as the ice retreated. Fine sediments accumulated in the glacial lakes varying from silt with some fine sand to fine to coarse sand. These deposits when encountered in the TMF basin area were generally up to 2 m thick and underlain by glacial till.

Glacial till deposits are present in the valleys of the project area and in a discontinuous blanket on mountain crests and slopes. Glacial till was deposited at the base of the ice sheet and is found thickest in the valley bottoms and thinner on valley side slopes and discontinuous over the bedrock on topographic highs. Glacial till thickness within the TMF area ranges from 1 m to 12 m, and typically is greater than 4 m thick. Glacial till generally comprised fine to coarse gravel with trace to some sand and silt and trace cobbles. The site investigation programs indicated that the glacial till on the east side of the valley contains a slightly higher proportion of fines than that on the west side.



NOTES:

1. BASE MAP: TRIM AND NTS MAPPING, ESRI ARCGIS ONLINE SHADED RELIEF.
2. COORDINATE GRID IS IN METRES. COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N.
3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:40,000 FOR 8.5x11 (LETTER) PAPER.
4. FACILITIES REPRESENT MAXIMUM FOOTPRINT FOR MINE LIFE.

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	19AUG'14	ISSUED WITH REPORT	JAS	AMD	DDF	KJB

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

SURFICIAL GEOLOGY

Knight Piésold CONSULTING

P/A NO. VA101-458/11

REF NO. 1

FIGURE 4.3

REV 0

SAVED: M:\1\01\00458\11\A\GIS\Figs\Report 1 - Mine Waste and WM\SurficialGeology_Rev0.mxd; Aug 19, 2014 2:35 PM; cczembor

4.1.3 Bedrock Geotechnical Properties

Bedrock outcrop exposure is rare and generally restricted to higher elevations in the area, and it is typically overlain by 1 to 15 m of overburden. Bedrock within and surrounding the immediate project area consisted of intrusives, orthogneiss, fault zones, phyllites, schists, quartz eye schists and silica altered host rocks (Knight Piésold Ltd., 2013).

A cumulative summary of the rock mass properties grouped by lithology is presented in Table 4.1.

Table 4.1 Summary of Rock Mass Quality

Lithology [1]	RQD (%)				RMR^{89}				
	# of Runs	Mean	Median	St. Dev.	# of Discontinuities	Mean	Median	St. Dev.	Description
Intrusives	151	72	79	25	831	69	68	11	GOOD
Orthogneiss	580	74	85	27	3182	67	67	10	GOOD
Fault Zone	42	60	69	36	144	57	57	11	FAIR
Phyllite	394	64	75	33	2117	65	64	10	GOOD
Schist	436	77	88	26	898	63	63	10	GOOD
Schist (w/Quartz Eyes)	859	75	85	27	2236	63	63	9	GOOD
Silica Altered Zone	110	74	85	28	258	66	67	8	GOOD

NOTES:
1. ADOPTED FROM TABLE 4.1 OF THE 2012 GEOTECHNICAL SITE INVESTIGATION FACTUAL REPORT (Knight Piésold Ltd., 2013).

Rock strength properties were grouped by failure types and by testing methods and are shown in Table 4.2. Many samples of phyllite and schist selected for unconfined compressive strength (UCS) testing failed along foliation planes within the rock, providing significantly lower UCS values. The rock strength values for failure through intact rock and failure through foliation are presented separately. Point load test (PLT) samples do not differentiate between intact failure and foliation failure, and are presented in their own category as well.

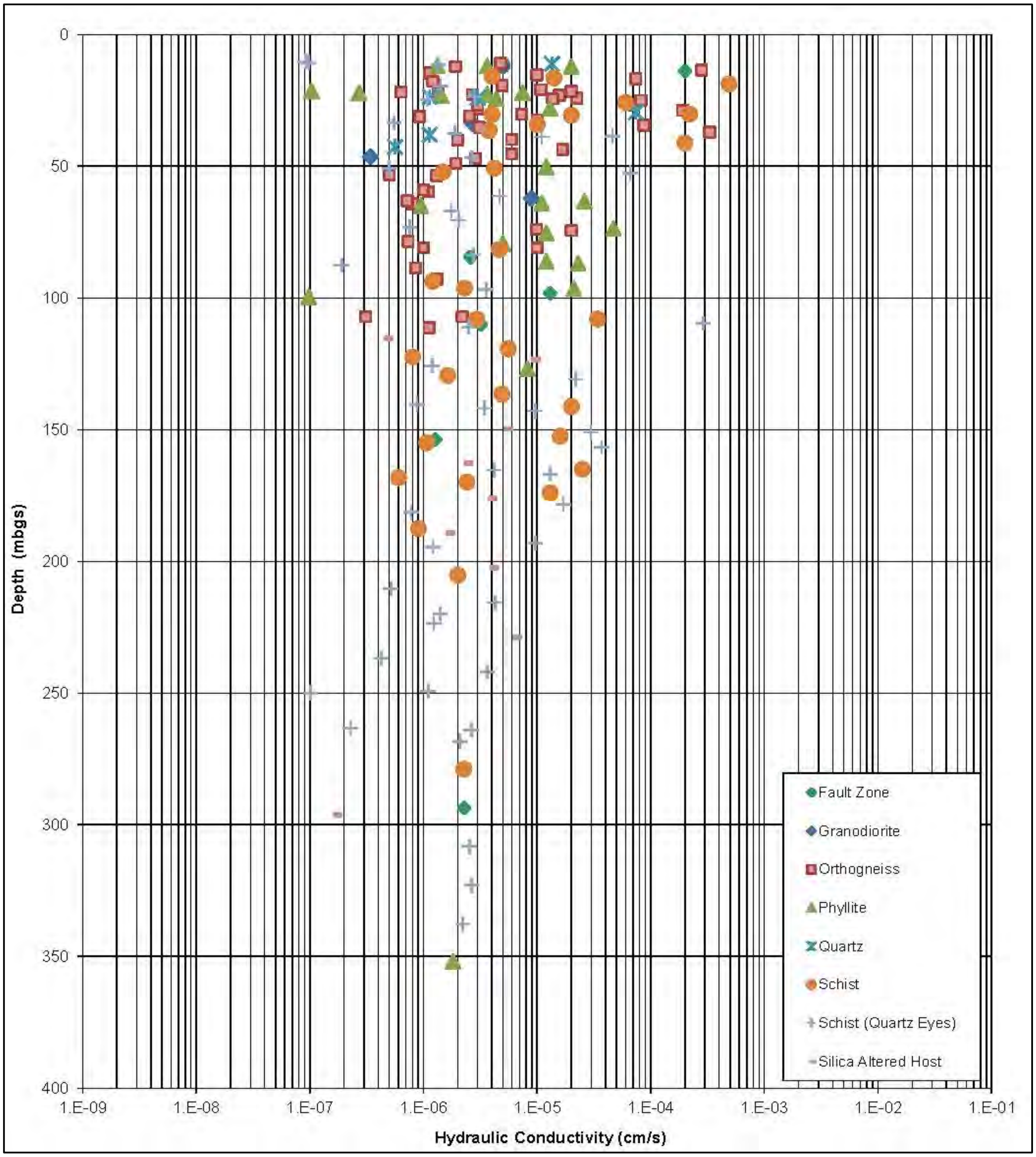
Table 4.2 Summary of Rock Mass Strength Properties

Lithology	Mean Rock Strength (MPa) [1]			Mean Young's Modulus (GPa) [3]	Mean Poisson's Ratio [3]	Direct Shear	
	UCS		PLT [2]			Mean Peak Friction	Mean Residual Friction
	Foliation Break	Intact					
Intrusives	-	120 (2)	-	78	0.234	-	-
Orthogneiss	-	138 (10	119 (49)	67	0.199	-	-
Fault Zone	-	-	-	-		-	-
Phyllite	39 (5)	80 (2)	22 (27)	44	0.150	36	29
Schist	26 (2)	91 (3)	23 (38)	44	0.290	42	37
Schist (w/Quartz Eyes)	53 (11)	93 (4)	25 (83)	48	0.196	37	32
Silica Altered Zone	37 (1)	-	29 (12)	52	0.273	37	32

NOTES:

1. ADOPTED FROM TABLE 4.2 OF THE 2012 GEOTECHNICAL SITE INVESTIGATION FACTUAL REPORT (Knight Piésold Ltd., 2013).

Hydrogeological testing was completed during the site investigation programs to estimate the in situ hydraulic conductivity of the rock mass in the project area and to develop an understanding of the variability of rock mass permeability by rock lithology and depth. Lugeon testing (single packer) was completed in all geotechnical and geomechanical drillholes, and falling head response testing was conducted following standpipe piezometer or monitoring well installation. Testing of the rock mass generally indicated low hydraulic conductivity values ranging from $6x10^{-5}$ m/s to $1x10^{-9}$ m/s. A plot of hydraulic conductivities values measured during the testing compared with test interval depth and separated by rock lithology type is shown on Figure 4.4.



NOTES:

1. ADAPTED FROM TABLE 3.5 OF THE 2012 GEOTECHNICAL SITE INVESTIGATION FACTUAL REPORT (Knight Piésold Ltd., 2013).

Figure 4.4 Hydraulic Conductivity Testing Summary

4.2 FOUNDATION DESIGN CONSIDERATIONS

4.2.1 Tailings Management Facility

The TMF is located in the southeast section of the mine footprint in a broad valley with gentle side slopes in the headwaters of the Harper Creek Catchment. The TMF drains southward down a steep bedrock canyon into Harper Creek. The dominant surficial material type is glacial till, consisting stiff to dense, moist, sands and gravels with some silt and clay ranging from <1 to 16 m thick. Extensive areas are mantled by glacial till. Organic swamps were encountered locally in the centre of the TMF basin, typically overlying glaciolacustrine deposits. The glaciolacustrine deposits are expected to be several metres deep, and overlie the glacial till deposits in the TMF basin. These deposits are expected to be weaker than the underlying glacial till. Glaciolacustrine material will be removed if encountered within the footprint of the TMF embankment, and will require further investigation during detailed design in other areas of the TMF. The overburden is covered by a thin veneer of organics and topsoil featuring hummocky terrain with swampy, poorly drained areas typically ranging in thickness from 0.1 to 0.5 m and is expected to be thicker in inaccessible areas. The slopes of the TMF are moderately steep with the steepest slopes along the west and east sides of the impoundment area. Valley side slopes consist of a combination of glacial till, colluvium and weathered bedrock. Thin layers of colluvium, typically boulder gravel with some silt and sand, are found along the base of some steeper slopes.

Orthogneiss is the dominant bedrock at the TMF with some granodiorite intrusions. Quartz monzonite is the primary lithology downstream of the main embankment. Overall, the rock quality is ‘GOOD’ with an average RMR of 68. The rock strength ranges from 114 to 206 MPa with an average of 150 MPa. The permeability at the TMF is generally low with hydraulic conductivity values typically ranging from 1×10^{-9} to 1×10^{-7} m/sec. The groundwater is shallow at the TMF with water levels generally less than 2 m below ground surface.

4.2.2 Non-PAG Waste Rock Stockpile

The Non-PAG Waste Rock Stockpile is located in the upper portion of the catchment of a tributary watercourse ‘P-Creek’ on moderate to moderately steep south-facing slopes and moderate north-facing slopes. Bedrock is generally overlain by a blanket of glacial till comprised of silt, sand and gravel, trace clay ranges in thickness from 6 to 25 m. The surface veneer of colluvium is interpreted to be generally present in the steeper areas and weathered bedrock colluvium is expected to be more prevalent on the moderately steep, south-facing slopes. The bedrock comprises alternating layers of schists, quartz eye schists and phyllites. The average RMR and rock strength of the bedrock is 49 and 52 MPa, respectively. The static water level ranges from 4 to 6 m below ground surface and the hydraulic conductivity ranges from 7×10^{-8} to 1×10^{-7} m/sec.

4.2.3 Low-grade Ore Stockpiles

The overburden ranges from 1 to 4 m in thickness and it mainly consists of silty sand with gravel, glacial till. Bedrock is mainly orthogneiss with small layers of quartz eye schists the average RMR and UCS are 59 and 115 MPa, respectively. Static water levels range from artesian conditions to 6 m below ground surface. The hydraulic conductivity ranges from 7×10^{-7} to 1×10^{-7} m/sec.

4.2.4 Overburden Stockpile

The overburden at the overburden stockpile site ranges in thickness from 2 to 6 m and mainly consists of silty sand and gravel materials. The bedrock is primarily quartz eye schists with phyllite layers. The average UCS, RMR and RQD of the bedrock are 49 MPa, 51 and 58 %. The hydraulic conductivity ranges from 9×10^{-7} to 1×10^{-6} m/sec. Static water level is less than 1 m below ground surface.

4.2.5 Topsoil Stockpiles

There are three proposed topsoil stockpile sites located to the east and west of the plant site, and south of the TMF. The overburden near the east topsoil stockpile is 6 m thick and is comprised of silty sand and gravel. Bedrock is orthogneiss and minor quartz eye schists with an average RMR value of 59. The overburden near the west topsoil stockpile is approximately 3 m thick and is comprised of silt and gravel. Bedrock is orthogneiss with an average RMR value of 63. The overburden near the south topsoil stockpile is 5 m thick and is comprised of sand and gravel. The bedrock is quartz monzonite with an average RMR value of 77.

5 – MINE WASTE MANAGEMENT

5.1 GENERAL

The principle design objectives for the waste rock stockpiles and TMF are to ensure protection of the regional groundwater and surface water during both operations and in the long-term (after closure), and to achieve effect reclamation at mine closure. The design and location of the waste rock stockpiles and TMF has taken into account the following requirements:

- situating the TMF and waste rock facilities away from sensitive environmental features including fish bearing drainages
- clustering the facilities to minimize the overall footprint
- permanent, secure, and total confinement of all solid waste materials within engineered disposal facilities
- control, collection, and removal of free-draining liquids from the waste and tailings facilities during operations for recycling as process water to the maximum practical extent
- prevention of acid rock drainage and minimization of metal leaching from reactive tailings and waste rock, and
- staged development of the facility over the life of the project.

5.2 DESIGN BASIS

Design and operating criteria have been developed for the FS to facilitate preparation of drawings and material take-offs to support an overall project economic evaluation. The design criteria reflect the FS mine plan and operating strategy.

Key considerations for the development of the TMF design are summarized as follows:

- Final mine material movement schedule for the TMF design was provided to KP by Harper Creek Mining Corp. (HCMC) on March 3, 2014.
- Initial staging of the starter TMF embankment allows for storage of one year of tailings, PAG waste rock, an operational pond volume of 12 Mm^3, and storage of the inflow design flood (IDF) with at least 1 m of freeboard for wave run-up.
- Annual staging of the TMF embankment lifts to allow for storage of the next year of tailings and waste rock disposal, storage of the predicted operational pond volume, and storage of the IDF with at least 1 m of freeboard for wave run-up.
- Conventional slurry tailings disposal with tailings solids approximately 35% by weight.
- Water for the process plant sourced from the TMF supernatant pond at a flow rate of 5,520 m^3/hour.

The design and operating criteria for the design of the project are presented on Table 5.1 below.

Table 5.1 Design and Operating Criteria

ITEM	DESIGN CRITERIA
1.0 GENERAL	
Site Coordinates	Approximately 305 000 E , 5 710 000 N (UTM NAD 83, ZONE 11)
Site Elevation	Approximately 1350 to 1850 masl
Codes and Standards	Health Safety and Reclamation Code for Mines in British Columbia (2008), Mines Act (RSBC 1996), ASTM, CDA Dam Safety Guidelines (2007) and associated revisions (2013) and bulletins (2014), and related codes.
Mine Production	Total ore milled = 718 million tonnes (Mt)
	Throughput = 70,000 tonnes per day (TPD).
	Open pit stripping ratio = approximately 0.8:1
	Mine Life = approximately 28 years
	Active Open Pit Mining = 24 years, Low Grade Ore Processing = 4 years
2.0 MINE WASTE MANAGEMENT	
2.1 Waste Properties	
Tailings	Total tailings production = 718 Mt
	Tailings Disposed in TMF = 585 Mt and Tailings Disposed in Open Pit during Years 24 to 28 = 133 Mt
	Dry density = 1.30 t/m3.
	Bulk Tailings Slurry Solids Content = 34.5%, Cleaner Tailings = 32.7%
	Bulk Tailings Specific Gravity of Solids = 2.66, Cleaner Tailings = 3.11
Potentially Acid Generating (PAG) Waste Rock	PAG co-disposed with tailings = 237 Mt
	Waste Rock Specific Gravity = 2.7
	Dry density = 2.2 t/m^3.
Non-Potentially Acid Generating (NON-PAG) Waste Rock	NON-PAG used to construct TMF embankments or disposed in surface stockpiles = 265 Mt
	Waste Rock Specific Gravity = 2.7
	Dry density = 2.2 t/m^3.
Non-Potentially Acid Generating Overburden	Overburden used to construct TMF embankments or disposed in surface stockpiles = 39 Mt
	Overburden Specific Gravity = 2.7
	Dry density = 2.0 t/m^3.
Low Grade Ore (LGO)	PAG LGO placed in surface stockpile = 86 Mt (Processed in Years 24 to 28)
	NON-PAG LGO placed in surface stockpile = 47 Mt (Processed in Years 24 to 28)
	LGO Specific Gravity = 2.7
	Dry density = 2.2 t/m^3.
2.2 Tailings Management Facility (TMF)	
Function	One impoundment provides for secure and permanent storage of tailings, PAG waste rock, and management of mine water for reuse in the process.

ITEM	DESIGN CRITERIA
Concept	585 Mt of tailings co-disposed with 237 Mt of PAG waste rock within one impoundment formed by two embankments. Main embankments raised in stages and constructed using the centreline method. North embankment built in one stage in Year 18.
Storage Capacity	Cofferdam - 1 in 10 year wet August, September, October and November, plus 1 in 10 year 24-hour storm event, plus construction dewatering allowance, and 1 m freeboard.
	Stage 1 Embankment - 1 year of tailings production and co-disposed waste rock, allowance for minimum 12 Mm^3 process water pond, plus IDF and freeboard.
	Raised Embankment - next year of tailings and co-disposed waste rock, process water pond, plus IDF and freeboard.
	Ultimate Embankment at Closure - 585 Mt tailings and 237 Mt co-disposed waste rock plus, 183 Mm^3 pond storage, and freeboard to attenuate the IDF.
CDA Consequence Classification	Very High
Inflow Design Flood (IDF)	Probable Maximum Flood (PMF) adopted for IDF.
	IDF Volume = 10 Mm^3 (assumes 697 mm IDF runoff depth and 1489 hectare catchment area)
Design Freeboard	Minimum 1 m above design storage capacity for wave runup; assumes tailings beaches developed at 0.5% slope from dam crest.
Operational Criteria	Flood management: Inflows are contained within the impoundment. The storage volume assumes any diversion systems are non-functional during the IDF.
	Supernatant water reclaimed for re-use in mill process.
	Excess water monitored and TSF raises managed appropriately.
Closure Criteria	Surface runoff of non-contact water routed to natural streams.
	TSF Closure Spillways: Pass PMF sized for the 24 hour PMP and 1/100 year snowmelt (498 mm) without consideration of the runoff attenuation provided by storage below the spillway crest.
Tailings Distribution (by others)	Bulk tailings delivery line to TMF with spigot discharge points every 200 m along main embankment crest for Years 1 to 24. Tailings discharge points along north and east of the TMF and from the north embankment crest between Years 19 and 24. Multiple point discharge, except in emergencies.
	Cleaner tailings delivery line to TMF to subaqueous disposal point near to reclaim barge access.
	Low grade ore tailings delivery to open pit between Years 24 and 28.
	Gravity discharge from mill used where sufficient head is available. Pump stations where gravity discharge is not sufficient.

<table>
<tr><th>ITEM</th><th colspan="2">DESIGN CRITERIA</th></tr>
<tr><td rowspan="2">Seepage</td><td colspan="2">Measures to control seepage include:
- lower permeability core zone with filter/transition zones for raised embankments
- lower permeability tailings deposit
- seepage collection ponds and pump-back systems downstream of TMF embankments</td></tr>
<tr><td colspan="2">Collected seepage is monitored and managed appropriately.</td></tr>
<tr><td rowspan="3">Seismic</td><td colspan="2">Earthquake Design Ground Motion (EDGM) = ½ between 1/2475 and MCE, Maximum Credible Earthquake (MCE) = 1/10,000 year event</td></tr>
<tr><td colspan="2">Magnitude, EDGM = 7.3 and MCE = 7.3</td></tr>
<tr><td colspan="2">Peak horizontal ground acceleration (PGA), EDGM = 0.21g (mean hazard value) and MCE = 0.26g</td></tr>
<tr><td rowspan="5">Embankment Stability</td><td colspan="2">Permanent embankment slopes to be no steeper than 2H:1V to facilitate reclamation, and achieving the minimum required Factors of Safety (FS) for the following loading conditions:</td></tr>
<tr><td>End of construction (starter dam and dam raises)</td><td>FS = 1.3</td></tr>
<tr><td>Long term (at closure)</td><td>FS = 1.5</td></tr>
<tr><td>Seismic (Pseudo-static loading condition)</td><td>FS = 1.0</td></tr>
<tr><td>Seismic (Post-earthquake loading condition; full liquefaction of tailings assumed)</td><td>FS = 1.5</td></tr>
<tr><td rowspan="2">Embankment Crest Width</td><td colspan="2">Minimum 30 m on downstream side during mine fleet construction periods to facilitate 2-way haul traffic and reduce turn and dump time.</td></tr>
<tr><td colspan="2">Minimum 30 m working surfaces during downstream step-outs.</td></tr>
<tr><td colspan="3">2.3 NON-PAG Waste Rock Stockpile</td></tr>
<tr><td>Function</td><td colspan="2">One engineered stockpile provides for secure and permanent storage of excess NON-PAG waste rock.</td></tr>
<tr><td>Surface Water</td><td colspan="2">Diverted around dumps to downstream environment using diversions field fit to advancing fill platforms.</td></tr>
<tr><td>Seepage and Runoff</td><td colspan="2">Seepage and contact water collected and routed to water management ponds, and subsequently pumped to the TMF.</td></tr>
<tr><td rowspan="3">Closure Criteria</td><td colspan="2">Surface runoff of non-contact water routed to natural streams.</td></tr>
<tr><td colspan="2">Seepage and contact water collected and routed to TMF until water quality suitable for release to downstream environment.</td></tr>
<tr><td colspan="2">Slopes to be covered with salvaged overburden and revegetated.</td></tr>
<tr><td colspan="3">2.4 Overburden Stockpile</td></tr>
<tr><td>Function</td><td colspan="2">One engineered stockpile provides for secure and permanent storage of excess overburden from pit stripping.</td></tr>
</table>

ITEM	DESIGN CRITERIA
Surface Water	Diverted around dumps to downstream environment using diversions field fit to advancing fill platforms.
Seepage and Runoff	Seepage and contact water collected with open channel ditch near stockpile toe and routed to open pit until Year 10.
	Contact water routed through sediment control pond and released to downstream environment after Year 10.
	Collection ditch remains in place to collect seepage and contact water if unsuitable for release.
Closure Criteria	Surface runoff of non-contact water routed to natural streams.
	Final slopes and grades progressively revegetated.
	Contact water routed through sediment control pond and released until reclamation is complete.
2.5 Low Grade Ore Stockpiles	
Function	Two engineered stockpiles provide for separate temporary storage of Non-PAG low-grade ore and one stockpile for PAG low-grade ore.
Surface Water	Diverted around stockpiles to downstream environment using diversions field fit to advancing fill platforms.
Seepage and Runoff	Seepage and contact water collected and routed to water management ponds, and subsequently pumped to the TMF.
Closure Criteria	Stockpiles removed and low grade ore processed between Years 24 and 28.
	Final slopes and grades progressively reclaimed and revegetated during low grade processing.
	Surface runoff of non-contact water routed to natural streams.
	Contact water routed to water management ponds and TMF until water quality suitable for release to downstream environment.
3.0 HAUL ROADS, SERVICE ROADS, AND DIVERSION TRAILS	
Function	Haul roads for construction equipment from pit rim and borrow areas to embankments for delivery of embankment construction materials and waste rock.
	Service roads for access to pipeworks (tailings, & reclaim systems) and diversion trails for runoff management.
Dimensions	Haul roads were sized according to Health Safety and Reclamation Code for Mines in British Columbia (2008). Allowances for safety barriers and ditches are included as required.
	Service roads will consist of 10 m road platform for pipeline bench and adjacent service vehicle access with allowance for ditches and safety barriers as required.
	Diversion trails will consist of 6 m road platform for service equipment and access suitable for a 4WD light truck for monitoring.
Operational Criteria	Roads to be accessible year round; maintenance and snow removal to be appropriate for intended frequency of use.

ITEM	DESIGN CRITERIA
Closure Criteria	Roads will be partially reclaimed to maintain access suitable for a 4WD light truck.
4.0 WATER MANAGEMENT	
4.1 Water Management Ponds	
Function	Collect seepage and contact water and convey to the TMF for storage.
	Remove sediment from storm water and discharge to downstream environment.
Surface Water	Diverted around ponds to downstream environment using diversions ditches.
Seepage and Runoff	Collect seepage and contact water, and manage pond level within dead storage zone by pumping daily flows to the TMF.
Design Storms Events	Live storage to accommodate passing the 1 in 10 year 24-hour storm event with a retention time of at least 20 hours.
	Outlet spillway to manage the 1 in 200 year 24-hour storm event.
Closure Criteria	Surface runoff of non-contact water routed to natural streams.
	Seepage and contact water collected and routed to TMF before release to downstream environment.
	Ponds removed and reclaimed once water quality is suitable for release to natural streams.
4.2 Water Management Pipelines and Pump-back Systems	
Function	Convey seepage and contact water to the TMF for storage.
Materials	Steel or HDPE
Alignment	Placed on surface along service roads or diversion trails.
Design Criteria	One pipeline to the TMF from each water management pond.
	Capable of delivering peak mean monthly flow from the associated area without storage.
	One standby unit at each pump station.
	Discharge points along pipeline for drain back will be directed towards contact water collection ditches or ponds.
Restraint	Periodic mounding of overburden material from adjacent ditch excavation and small safety berms to prevent excessive movement.
Pipeline Lengths (from the water management pond in each area)	NON-PAG waste rock stockpile: 4,200 m
	NON-PAG LGO stockpile: 1,800 m
	PAG LGO stockpile: 200 m
	TMF Main Embankment: Year 1 = 1,300 m, Year 5 = 1,350 m, Year 10 = 1,400 m, Ultimate = 1,450 m
	TMF North Embankment: 300 m
Design Elevations	NON-PAG waste rock stockpile water management pond: EL. 1375 m
	NON-PAG LGO stockpile water management pond: EL. 1723 m
	PAG LGO stockpile water management pond: EL. 1828 m
	Highest elevation on route to TMF: 1840 m
	TMF main water management pond: EL. 1635 m

ITEM	DESIGN CRITERIA
	TMF north water management pond: EL. 1815 m
	TMF Main Embankment: Year 1 = 1731 m, Year 5 = 1763 m, Year 10 = 1791 m, Ultimate = 1836 m
	TMF North Embankment: Year 18 = 1836 m
Closure Criteria	Pipelines removed once water quality is suitable for release to natural streams.
	Service roads and diversion trails reclaimed following removal of pipelines.
4.3 Open Pit Water Management System	
Function	Transfer water from the pit excavation to the TMF for recycle to the milling process.
Surface Water	Diverted around and away from open pit using diversion ditches and convey to downstream environment
Design Criteria	Base inflow = predicted seepage inflow plus average precipitation.
	Dewatering system capable of removing base inflow plus 1 in 10 year 24-hour storm event in 10 days.
	One pipeline to the TMF from the open pit.
	Pump selection based on 20% surge capacity.
Materials	Steel or HDPE
Alignment	Placed on surface along pit walls and access ramps within open pit, and then running adjacent to the mine haul road to TMF.
Restraint	Periodic mounding of overburden material from adjacent ditch excavation and safety berms to prevent excessive movement.
Pipeline Length	From Open Pit centroid to TMF (along mine haul road): 4,800 m
Design Elevations	Open Pit (bottom elevation): Year 1 = 1588 m, Year 5 = 1480 m, Year 10 = 1432 m, Year 24 = 1324 m
	Highest elevation on route to TMF: 1840 m
Closure Criteria	Reclaim barge and pipeline relocated to open pit at closure.
	Operations pit dewatering system removed following reclaim barge relocation.
	Pump water from open pit with reclaim barge to the TMF (subsequently flows to Harper Creek via the TMF closure spillway and T-creek).
4.4 Process Water Reclaim System (by others)	
Function	Reclaim water from the supernatant pond and transport to the process water head tank at the mill site.
Capacity	Reclaim system provides 100% of annual average water neeed for tailings delivery to the TMF
General Criteria	Water extracted from supernatant pond using a floating barge pump station.
	One reclaim pipeline from the TMF to process water head tank at mill site.
Alignment	Placed on surface along service roads.
Closure Criteria	Reclaim barge and pipeline relocated to open pit in closure.
	Pump water from open pit with reclaim barge to the TMF (subsequently flows to Harper Creek via the TMF closure spillway and T-creek).

5.3 TAILINGS DAM HAZARD CLASSIFICATION

The Canadian Dam Association (CDA) Dam Safety Guidelines (2013 revision) were used to determine the dam classification and suggested minimum inflow design flood (IDF) and earthquake design ground motion (EDGM) for the project tailings dams. The tailings dams were classified by considering the potential incremental consequences of a failure. The dam safety classification for the project tailings dams is VERY HIGH. The following suggested design flood and earthquake levels were adopted from the CDA guidelines for the project:

- IDF – 2/3 between 1 in 1,000 year return period and probable maximum flood (PMF)
- EDGM – 1/2 between the 1 in 2,475 year return period and Maximum Credible Earthquake (MCE)

A draft technical bulletin released by the CDA in 2014, entitled Application of Dam Safety Guidelines to Mining Dams, suggests that in closure of the TMF, a mining dam should be designed for the PMF and MCE regardless of dam classification. The following design event levels were adopted for closure of the TMF:

- IDF – PMF
- EDGM – MCE (1 in 10,000 year return period)

5.4 LAYOUT AND OPERATING STRATEGY

5.4.1 General

The filling schedule for the TMF was based on the detailed mine schedule and is presented in Table 5.2. Specific overall features of the TMF are listed below:

- cofferdams and sediment control ponds to manage water during construction by either routing water around the TMF or directing water to the TMF for collection
- two zoned water-retaining earth-rockfill dams referred to as the main embankment and north embankment
- designated PAG waste rock stockpile areas within the TMF
- downstream water management ponds for seepage and storm water management
- collection channels that route water to the TMF and collection ponds
- diversion channels that route water away from the TMF and collection ponds to the downstream receiving environment
- tailings distribution system
- tailings beaches
- reclaim water system, and
- supernatant water pond.

Table 5.2 TMF Filling Schedule

YEAR	TAILINGS SOLIDS		WASTE ROCK INUNDATED IN TMF			TMF STORAGE REQUIREMENTS		
	Total Tailings Cumulative	Total Tailings Solids	Annual Total	Cumulative Total	Cumulative Total	Total Tailings and Waste Rock	Supernatant Pond Allowance	Total Required Storage Volume
	tonnes	m^3	tonnes	tonnes	m^3	m^3	m^3	m^3
PRE-PRODUCTION	0	0	0	0	0	0	12,000,000	12,000,000
1	22,992,222	17,686,325	6,399,349	6,399,349	2,908,795	20,595,120	12,000,000	32,595,120
2	48,539,058	37,337,737	7,712,368	14,111,716	6,414,417	43,752,153	21,000,000	64,752,153
3	74,086,140	56,989,339	4,890,041	19,001,757	8,637,162	65,626,501	28,000,000	93,626,501
4	99,633,733	76,641,333	10,013,086	29,014,843	13,188,565	89,829,898	34,000,000	123,829,898
5	125,181,326	96,293,327	8,384,309	37,399,152	16,999,614	113,292,942	39,000,000	152,292,942
6	150,728,602	115,945,079	15,778,129	53,177,281	24,171,491	140,116,570	45,000,000	185,116,570
7	176,275,881	135,596,832	19,100,137	72,277,417	32,853,372	168,450,203	51,000,000	219,450,203
8	201,823,590	155,248,916	18,040,388	90,317,806	41,053,548	196,302,464	57,000,000	253,302,464
9	227,371,319	174,901,015	15,613,545	105,931,350	48,150,614	223,051,629	64,000,000	287,051,629
10	252,918,982	194,553,063	11,088,602	117,019,953	53,190,888	247,743,951	71,000,000	318,743,951
11	278,466,689	214,205,145	9,805,409	126,825,361	57,647,892	271,853,037	78,000,000	349,853,037
12	304,014,226	233,857,097	11,509,526	138,334,887	62,879,494	296,736,591	85,000,000	381,736,591
13	329,561,581	253,508,908	10,879,182	149,214,070	67,824,577	321,333,486	93,000,000	414,333,486
14	355,109,288	273,160,991	10,600,687	159,814,757	72,643,071	345,804,062	101,000,000	446,804,062
15	380,656,982	292,813,063	11,685,721	171,500,478	77,954,763	370,767,826	109,000,000	479,767,826
16	406,204,907	312,465,313	9,964,802	181,465,280	82,484,218	394,949,531	117,000,000	511,949,531
17	431,752,846	332,117,574	12,643,647	194,108,927	88,231,330	420,348,904	125,000,000	545,348,904
18	457,300,713	351,769,779	8,284,431	202,393,358	91,996,981	443,766,760	134,000,000	577,766,760
19	482,848,512	371,421,932	8,438,820	210,832,178	95,832,808	467,254,740	142,000,000	609,254,740
20	508,396,261	391,074,047	7,261,133	218,093,311	99,133,323	490,207,370	150,000,000	640,207,370
21	533,943,874	410,726,057	3,289,972	221,383,284	100,628,765	511,354,822	159,000,000	670,354,822
22	559,491,457	430,378,044	3,257,650	224,640,933	102,109,515	532,487,559	168,000,000	700,487,559
23	585,038,826	450,029,866	6,298,357	230,939,291	104,972,405	555,002,271	177,000,000	732,002,271
24	585,038,826	450,029,866	3,773,351	234,712,642	106,687,565	556,717,431	183,000,000	739,717,431
25	585,038,826	450,029,866	2,646,358	237,359,000	107,890,455	557,920,321	163,000,000	720,920,321
26	585,038,826	450,029,866	0	237,359,000	107,890,455	557,920,321	165,000,000	722,920,321
27	585,038,826	450,029,866	0	237,359,000	107,890,455	557,920,321	170,000,000	727,920,321
28	585,038,826	450,029,866	0	237,359,000	107,890,455	557,920,321	170,000,000	727,920,321

NOTES:

1. ASSUME TAILINGS DENSITY = 1.3 t/m^3.
2. ASSUME WASTE ROCK DENSITY = 2.2 t/m^3.
3. INFLOW DESIGN FLOOD (IDF) VOLUME = 10,000,000 m^3.
4. WASTE PRODUCTION SCHEDULE PROVIDED BY ALASTAIR TIVER OF YELLOWHEAD MINING INC. ON MARCH 3, 2014.
5. TAILINGS TO BE DEPOSITED INTO THE OPEN PIT STARTING AT THE BEGINNING OF YEAR 24 CONFIRMED BY ALASTAIR TIVER IN MEETING ON FEBRUARY 18, 2014.
6. SUPERNATANT POND ALLOWANCE BASED ON PEAK ANNUAL TMF POND VOLUME FROM THE LIFE OF MINE WATERSHED MODEL FOR AVERAGE CONDITIONS.

A filling curve was developed for the facility and includes the approximate rate of rise of the tailings and waste rock horizon, supernatant pond allowance, and IDF freeboard. The filling curve for the TMF is shown on Figure 5.1.

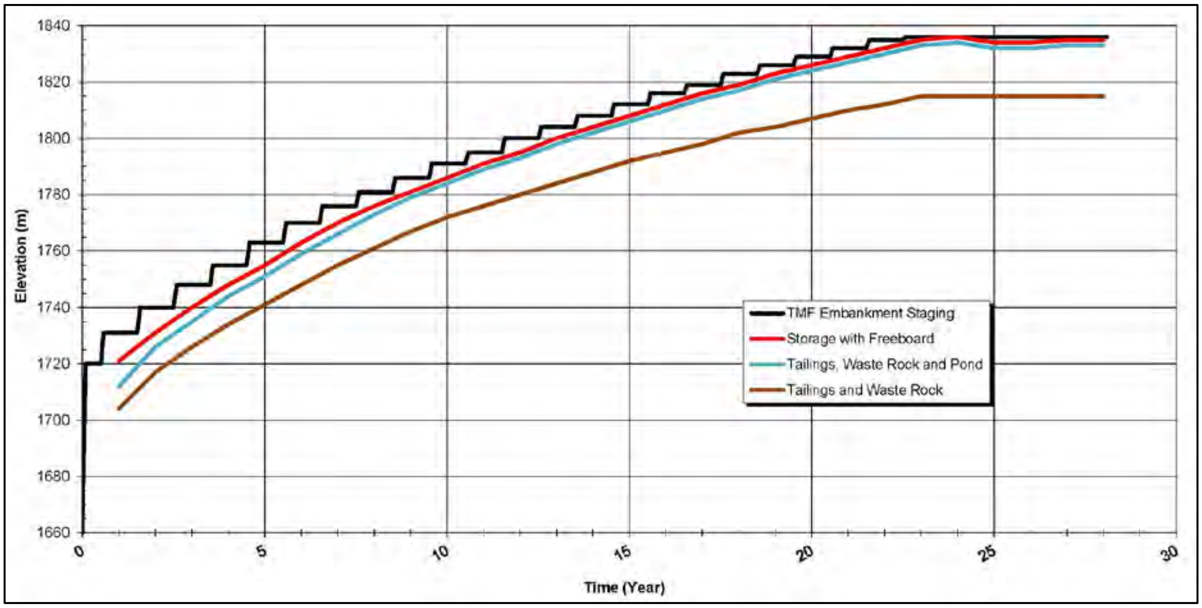


Figure 5.1 TMF Filling Curve

5.4.2 PAG Waste Rock Stockpile Area

The PAG waste rock stockpile area (henceforth referred to as the PAG disposal area) within the TMF footprint will be developed as part of preproduction construction to provide a location for PAG disposal from the pit stripping to expose the orebody. The PAG disposal area will be developed at the same or similar rate of rise as the TMF filling level but will be several metres higher than the tailings pond to provide a dry, stable placement surface for truck traffic. The design objective for the PAG area is to flood the waste rock within one year of placement.

The maximum elevation of the disposal area will remain at an elevation where it can be flooded by the supernatant pond in the case of premature closure. The disposal area will expand as fill platforms with overall slopes at angle of repose. The tailings beaches will provide a low-permeability barrier between the coarse permeable waste rock and the tailings embankments. The fill platform will rise slightly above and with the TMF filling level from the start of mining until Year 24 when mining ceases. The fill platforms of the PAG disposal area will be progressively covered by tailings and the supernatant pond during operations and will be flooded during closure of the TMF.

The general arrangement of the TMF during year 10 and approximate extents of the advancing PAG waste rock stockpile are shown on Figure 5.2.

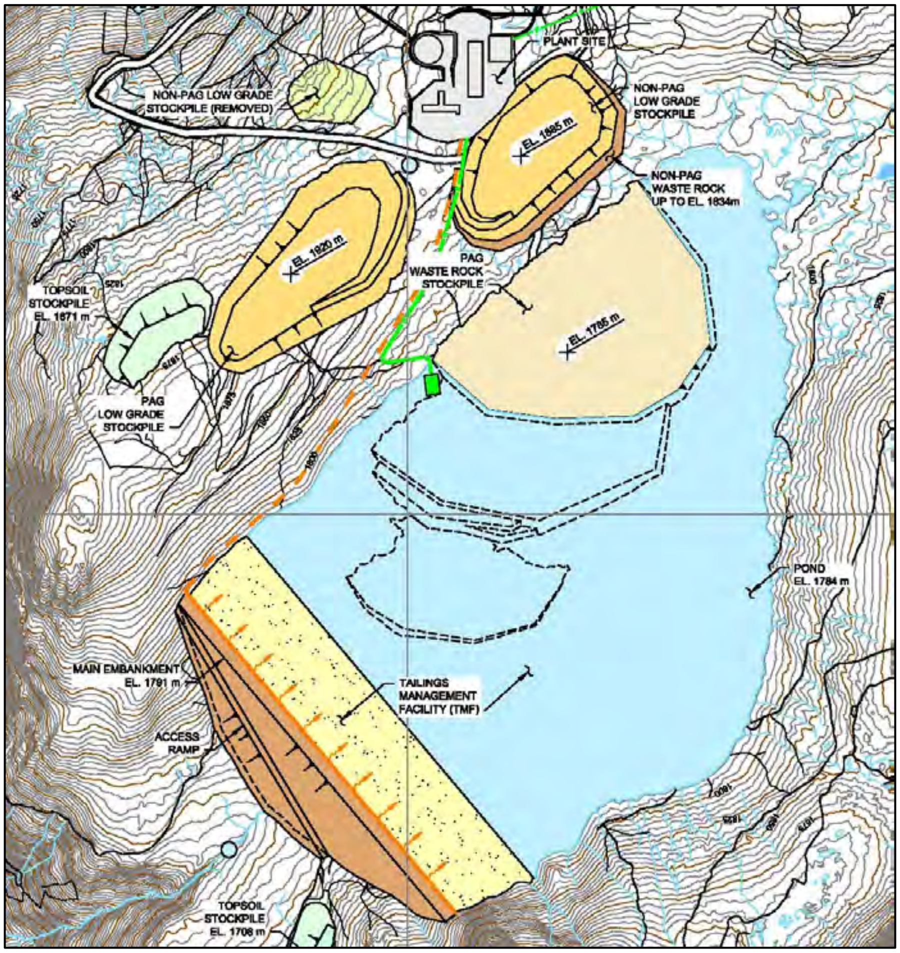


Figure 5.2 TMF Filling Year 10

5.4.3 Tailings Consolidation Considerations

The TMF filling schedule was based on an average settled density of 1.3 tonnes/m^3. Tailings consolidation testing was completed for the project and indicated the following settled density values for the various stages of tailings deposit consolidation:

- Undrained settled dry density was 1.2 tonnes/m^3 and took approximately four days to complete.
- Drained settled dry density was 1.2 tonnes/m^3 and took approximately two days to complete.
- Air dried final dry density was 1.5 tonnes/m^3.

A comparison between the average settled density of the tailings deposit used in the layout of the facility and the drained settled density determine by laboratory testing was completed. There is the potential to underestimate storage capacity in the TMF of 1.5 Mm^3 per year of tailings solids based on the variability of tailings density, which typically corresponds to less than 1 m of embankment height. This incremental difference can be managed within the annual excess storage capacity provided by the operational freeboard of the TMF above and beyond the needs for IDF storage.

5.5 SITE PREPARATION

Site investigations were carried out within the TMF embankment footprint area to characterize the depth to low permeability sub-grade material suitable for the foundation of the embankment. The sub-grade material will be defined by the following criteria:

- USCS material classification of SM-SC and GM-GC
- dense and/or compact material
- greater than 15% fines (defined by % passing the #200 sieve size), and
- underlying material acts as an aquitard.

The embankment foundations will be cleared and stripped in preparation for fill placement for each stage of the embankment. A cut-off trench will be excavated below the embankment core zone to intersect the low permeability sub-grade material based on the criteria above. The cut-off trench is estimated to be nominally 2 m deep, although the depth may vary locally.

5.6 EMBANKMENT CONSTRUCTION

5.6.1 Construction Materials

The total construction fill requirement for the main embankment is 58.4 Mm^3 of material, which will be provided from pit stripping (55.7 Mm^3) and external borrow sources (2.7 Mm^3). The earth-rockfill dams will comprise the following zones:

- The core zone (Zone S) will be constructed from low-permeability glacial till from nearby external borrows and from pit stripping. The material will consist of well-graded silty sand with some gravel with a fines content of 20% to 60% passing the #200 sieve. This material will generally require no processing except for the removal of oversized particles. The material will be placed in maximum 300 mm lifts loose and compacted by combination of smooth drum vibratory rollers and pad foot compactors to 95% standard proctor maximum dry density (SPMDD).
- The filter zone (Zone F) will be constructed with clean, fine to coarse sand. It will be placed adjacent to and downstream of the core zone to prevent piping of the core zone material and to reduce pore pressures within the embankment. This material will be a processed non-reactive sand material produced in a quarry downstream of the main embankment. Zone F will be placed and spread in maximum 600 mm lifts loose and compacted by four to six passes with smooth drum vibratory rollers.
- The transition zone (Zone T) will be constructed adjacent to and downstream of the filter Zone F. It will be constructed with processed non-reactive sand and gravel material produced in a quarry downstream of the main embankment. The transition zone will prevent the migration of fines from the core zone and Zone F into the pervious downstream shell zone (Zone C). Zone T will be placed and spread in maximum 600 mm lifts loose and compacted by four to six passes with smooth-drum vibratory rollers.
- Shell zones (Zone C) will be constructed on both the upstream and downstream sides of the dam with random fill consisting of overburden and specific waste rock material types from the open pit. Compaction will be done with trucks across the main fill by routing haul truck patterns to produce a uniformly compacted lift. A vibratory smooth drum roller will be used on the edges of lifts with a minimum four to six passes. The lift thickness and specified maximum particle sizes will be based on the truck placement fleet as follows:

- o Contractor fleet: placed and spread in maximum 1,000 mm lifts with a maximum particle size of 1,000 mm.
- o Mine fleet: placed and spread in maximum 2,000 mm lifts with a maximum particle size of 2,000 mm.

5.6.2 Cofferdam

The initial stage of the TMF main embankment is the cofferdam, which will eventually be incorporated into the upstream shell zone of the Stage 1 embankment. It was designed to an elevation of 1683 m with an embankment crest 10 m-wide and 1.5H:1V slopes, upstream and downstream. The cofferdam will be constructed entirely of locally borrowed Zone S material from the southeast side of the TMF impoundment, located within 2 km of the dam. The total construction volume requirement of the cofferdam was estimated to be 400,000 m^3. A site plan and cross section showing the cofferdam arrangement, site preparation specifications, and construction water management ditch layout are shown on Figure 5.3.

All contact runoff water during construction of the cofferdam will be collected in a downstream sediment control pond to remove sediment, prior to release, thereby preventing sediment laden water from entering the downstream watercourse. After the cofferdam has achieved an elevation of 1683 m, contact water will be managed within the TMF impoundment created by the cofferdam. The cofferdam will be constructed entirely of Zone S material from one borrow area to limit the need for sediment and erosion control in multiple areas for this initial phase of construction.

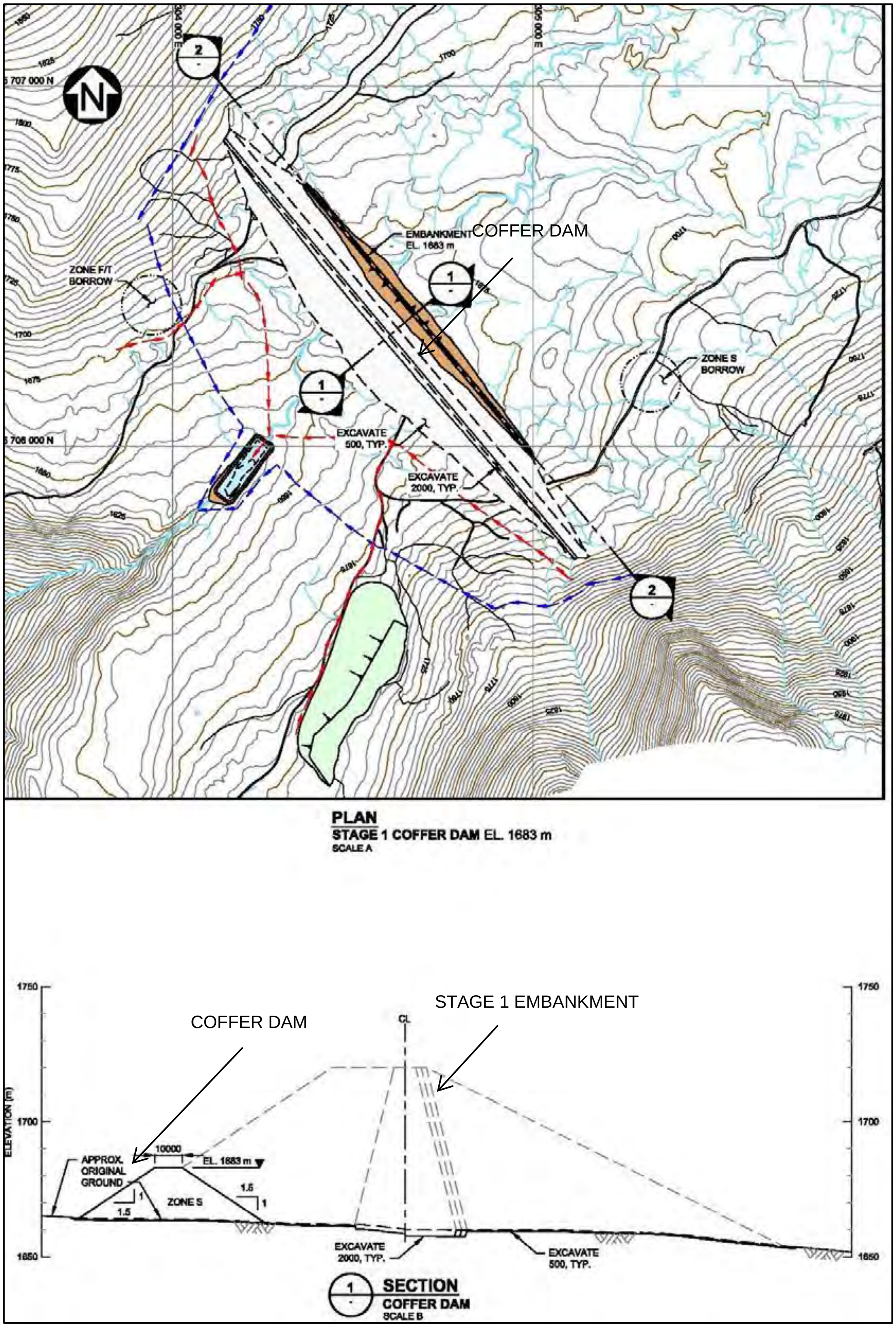


Figure 5.3 Cofferdam Plan and Section

Initial impoundment of water behind the cofferdam will be planned to occur in August following the annual freshet, which generally provides the vast majority of the run-off at the project site. The cofferdam has been sized to provide storage capacity for four months (August through November) of statistically wet conditions for the project site, in addition to a 10 year return period design flood, with an allowance for construction dewatering and freeboard. It is intended to provide secure isolation for construction of Stage 1 of the main embankment, including the foundation seepage collection drains, the foundation key-in for the core zone, and to allow the construction of Stage 1 to advance above the cofferdam elevation.

5.6.3 Stage 1 Main Embankment

Construction of Stage 1 will commence immediately following completion of the cofferdam to reach an elevation of 1700 m by May, which will provide storage capacity for a maximum pond volume of 12 Mm^3 in time to collect and store the annual freshet. The annual freshet will generate the vast majority of the start-up water for the process plant. Stage 1 construction will continue throughout the year to reach elevation 1720 m (approximately 70 m in height at the maximum dam section) prior to the start of operation of the process plant. Stage 1 will provide an impoundment capable of securely storing process start-up water, one year of process tailings and PAG waste rock, site contact water, and the Inflow Design Flood (IDF) with at least 1 m of freeboard for wave run-up.

The Stage 1 design incorporates upstream and downstream shell zones comprised of general fill (Zone C). The embankment has a core zone of low-permeability (Zone S) material and two downstream filter/transition layers (Zones F and T), which will maintain the integrity of the core zone and control seepage flow that passes through the core. The seepage will be collected in a longitudinal drain running the length of the embankment and directed to an outlet drain near the center of the embankment. Seepage flow will be directed in the outlet drain to a downstream water management pond for collection and recycle of contact water to the TMF.

Construction of Stage 1 will require approximately 7.35 Mm^3 of material, which will be provided from pit stripping (5.55 Mm^3) and external borrow sources (1.8 Mm^3). The volumes of each material zone required for the cofferdam and the Stage 1 embankment are presented below in Table 5.3.

Table 5.3 Stage 1 Main Embankment Volumes

ZONE - MATERIAL TYPE	STAGE 1 EMBANKMENT VOLUMES		
	PHASE		TOTALS
(units)	COFFERDAM	STAGE 1	(m^3)
ZONE C - General Fill (m^3)	-	5,547,000	**5,547,000**
ZONE F - Filter (m^3)	-	118,000	**118,000**
ZONE T - Transition (m^3)	-	118,000	**118,000**
ZONE S - Core Zone (m^3)	402,000	1,169,000	**1,571,000**
TOTALS	**402,000**	**6,952,000**	**7,354,000**

The site plan and typical cross section for the Stage 1 embankment are shown on Figure 5.4.

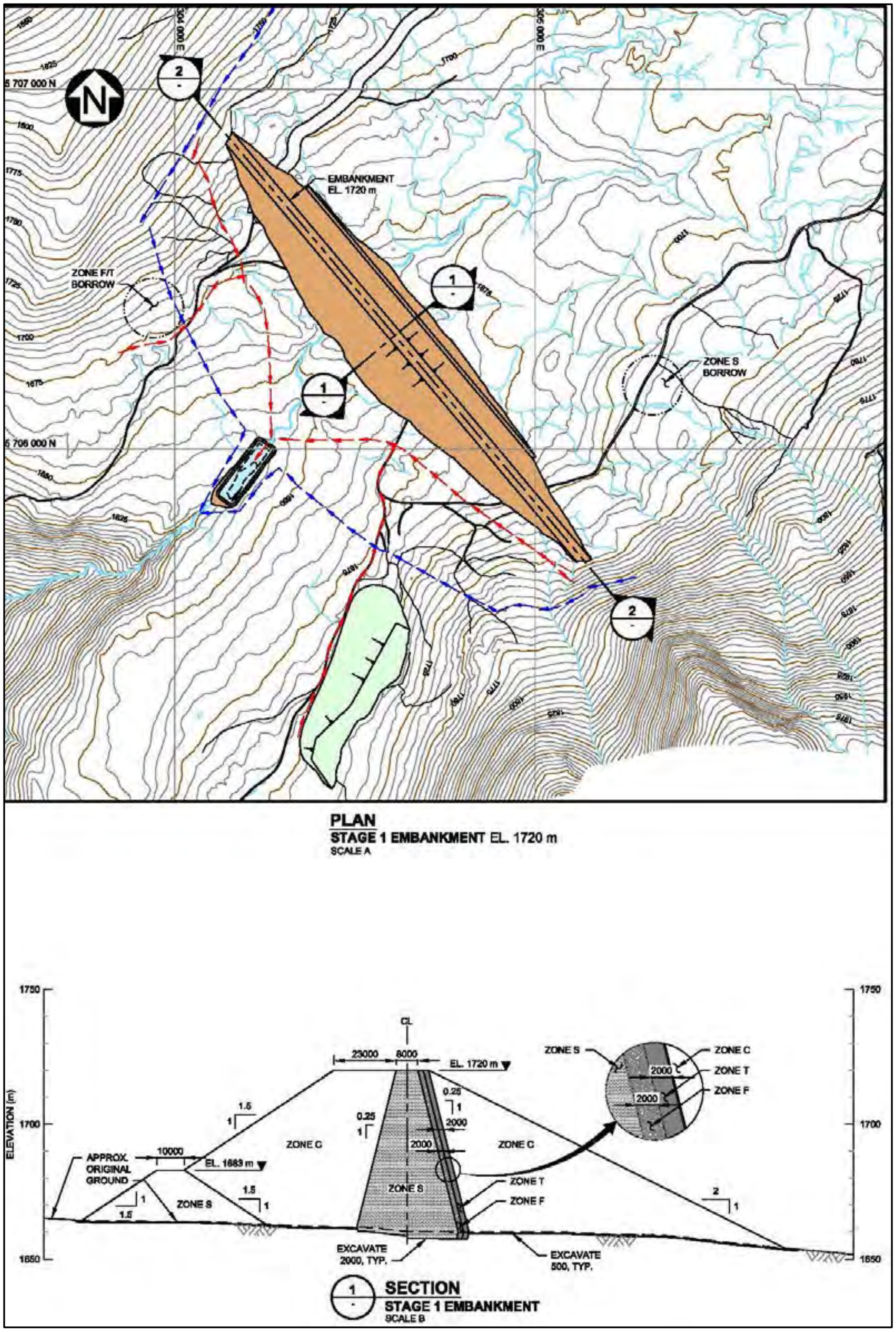


Figure 5.4 Stage 1 Main Embankment Plan and Section

5.6.4 Main Embankment Expansions

Construction of subsequent stages of the main embankment will commence following the start of process plant operation and will be completed using the centreline method of construction. The expansion of the embankment will consist of two major work areas – downstream step-outs and crest raises.

Downstream step-outs of the main embankment shell zone (Zone C) will be constructed in sections at least 30 m-wide using non-PAG waste rock from the open pit. An access ramp will be built into each step-out to allow on-going access to the embankment toe for downstream construction. Each step-out will support one or more vertical embankment crest raises.

Crest raises, constructed on an annual basis, provide storage for the upcoming year of tailings, PAG waste rock, and site contact water. The height of the annual raise varies from 11 m to 3 m depending on storage characteristics of the TMF and the volume of waste to be managed in the upcoming year.

The total fill requirement for the main embankment is 58.4 Mm^3 of construction material, which will be provided from pit stripping (55.7 Mm^3) and external borrow sources (2.7 Mm^3). The volume of each material zone that is required for the main embankment is presented below in Table 5.4.

Table 5.4 Sustaining Embankment Volumes

ZONE - MATERIAL TYPE	MAIN EMBANKMENT VOLUMES		
	PHASE		TOTALS
(units)	STAGE 1 TOTAL	SUSTAINING	(m^3)
ZONE C - General Fill (m^3)	5,547,000	48,738,000	**54,285,000**
ZONE F - Filter (m^3)	118,000	452,000	**570,000**
ZONE T - Transition (m^3)	118,000	457,000	**575,000**
ZONE S - Core Zone (m^3)	1,571,000	1,393,000	**2,964,000**
TOTALS	**7,354,000**	**51,040,000**	**58,394,000**

The final stage of the main embankment was designed to reach an elevation of 1836m, which is approximately 185m in height at the maximum dam section. It will be capable of securely storing over 585 Mt of process tailings, 237 Mt of PAG waste rock, site contact water, and the IDF with at least 1 m of freeboard for wave run-up. The staged expansion of the main embankment is shown on Figure 5.5.

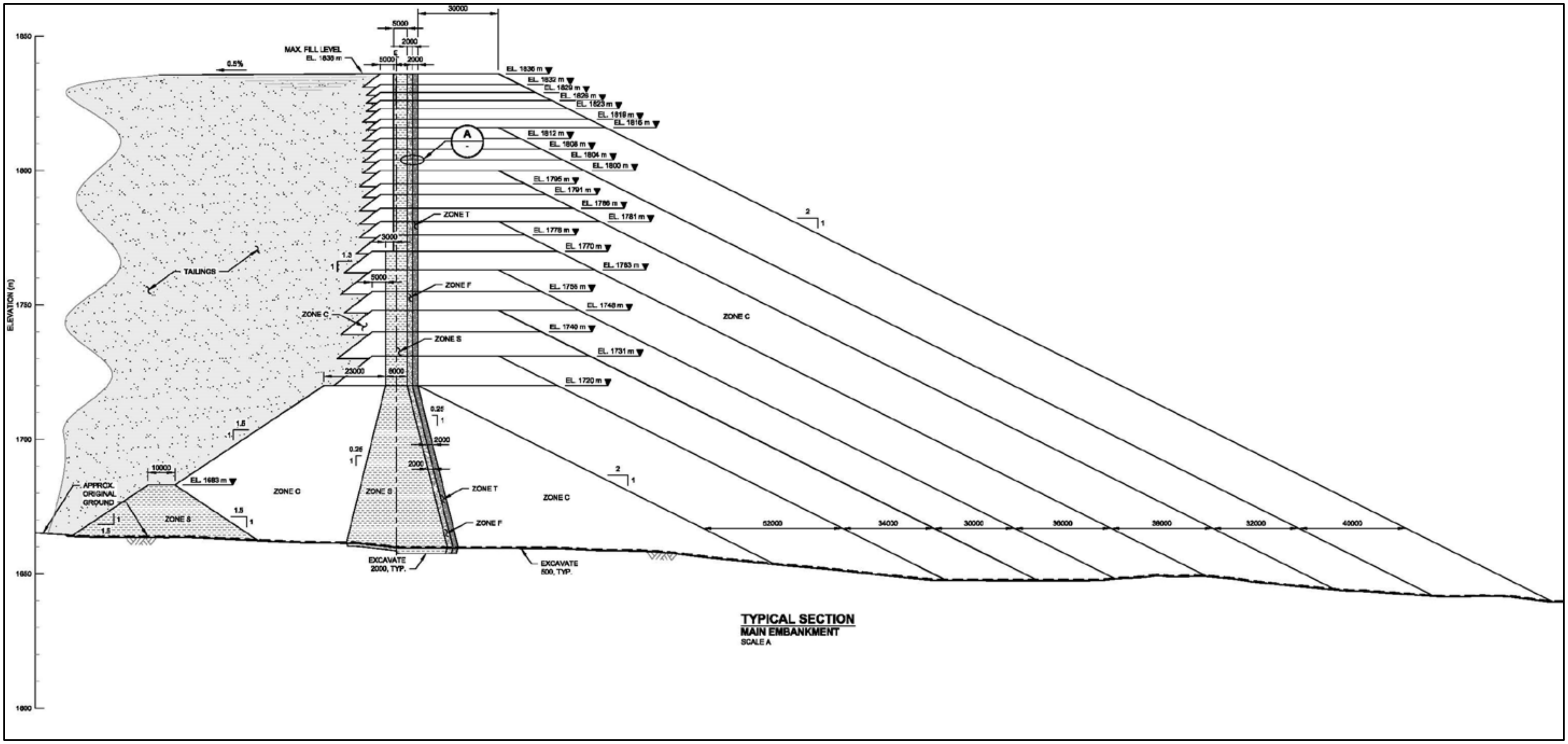


Figure 5.5 Main Embankment Staged Expansions

5.6.5 North Embankment

An embankment on the north side of the TMF will be constructed during Year 19 to provide containment at the drainage divide between the TMF basin and the Jones Creek catchment. The north embankment will be constructed in one stage, which will be approximately 11 m high. The embankment was designed with a 12 m crest width and 2H:1V slopes on the upstream and downstream sides. The embankment will be comprised of Zone C material, with an upstream low-permeability (Zone S) layer 4 m-thick and two downstream filter/transition layers (Zones F and T) to maintain the integrity of the Zone S layer. The total construction volume requirement of the cofferdam was estimated to be 107,000 m^3, including 15,000 m^3 of Zone S, 77,000 m^3 of Zone C and 15,000 m^3 of Zones F and T combined.

5.6.6 Winter Construction Considerations

Open pit mining will be a continuous process, and the construction of the tailings embankments will reflect this. Winter construction will be necessary to reduce material rehandle and to maintain the required crest elevation from year to year. Zone C material will be placed in the downstream shell zones during the winter. The core zone and filter/transition zone materials will be placed in the summer months as much as possible, but from time to time will be done in the winter if delays are encountered during summer construction.

Precedent exists for the construction of earthfill structures in freezing conditions, with special considerations. These include construction of the Mt. Milligan Tailings Storage Facility, which is located in central B.C.

To meet density requirements, embankment fill, particularly in the core, filter, and transition zones, must be compacted before it freezes. Haul time and compaction methods will address this priority. For example, sheep's foot packers leave depressions in the fill that increase the rate of heat loss and collect snow, and it may be more efficient to have loaded haul trucks make several passes over a lift before dumping.

It is possible for a lift to freeze after it has been compacted without significantly reducing its density. Material may be placed on top of a frozen lift provided that ice, snow, and loose frozen material are removed first and the density has not been significantly altered. An acceptable fill surface is 90% free of ice and snow, with the remaining 10% consisting of small discontinuous patches. Small areas will be prepared immediately in front of the advancing lift and covered as quickly as possible.

Borrow areas require careful management in freezing conditions, as frozen material must be separated and spoiled. Snow and frozen material will be removed only from the immediate work area to minimizing refreezing of exposed surfaces.

Haul roads will require extra attention in winter to maintain safety and prevent haul time from increasing dramatically. Extra equipment such as sand trucks and graders will be used, along with a supply of road sand.

Spoil factors and equipment downtime could increase as temperatures decrease. The loss of efficiency during winter months can be reduced by clearly outlining a set of construction procedures in advance. All QA staff, operators, and supervisors will be aware of the procedures. It may not be possible to place core zone material properly in temperatures below approximately -15°C, even with quality procedures in place. It may be more efficient to place coarse rockfill in the embankment shell zones during these times and to schedule overburden material removal from the pit in the summer months so it can be utilized efficiently in embankment construction.

5.7 SEEPAGE CONTROL MEASURES AND SEEPAGE ANALYSES

5.7.1 Seepage Control Measures

Seepage will be primarily controlled by the low-permeability core zone constructed prior to the development of the tailings beach, the core zone key trench, and the low-permeability subgrade materials. Seepage from the TMF will result from infiltration of ponded water directly through the embankment fill and the natural ground, and from expulsion of pore water as the tailings mass consolidates.

Special design provisions incorporated into the tailings embankment design to minimize seepage losses include the development of extensive tailings beaches (which isolate the supernatant pond from the embankment), embankment drainage collection systems, and toe drains at the downstream toe of the embankments to reduce seepage gradients. Additional seepage collection ditches along the toe of the embankments will collect seepage and surface runoff, and direct the flow to the pumpback systems.

Groundwater monitoring wells have been installed in the downstream area below the TMF embankments, and additional monitoring wells will be installed prior to commissioning of the TMF. The monitoring wells will be used to monitoring groundwater quality downstream of the TMF, and the information collected at these locations can be used to locate groundwater recovery wells, if required.

5.7.2 Seepage Analyses

Seepage analyses were carried out for the TMF to determine the potential for seepage through the embankments and foundation materials at the final embankment (closure) configuration and along the northwestern and southeastern flanks of the TMF impoundment. The modelling approach and complete results are presented in Appendix D and a summary is provided in the following section. The following sections were analyzed for the TMF and are identified on Figure 5.6:

- Main Embankment: Sections 1, 2 and 3
- North Embankment: Section 6
- East Saddle: Section 4
- West Saddle: Section 5

The seepage rate through the foundation materials and embankment fill zones will be influenced by the following factors:

- Permeability of the natural glacial till materials that blanket the basin
- Permeability of the bedrock foundation
- The thickness and permeability of the tailings stored within the TMF
- Permeability of the embankment core zones
- Seepage gradients in the embankment and foundation zones, and
- The seepage area (increases during operations).

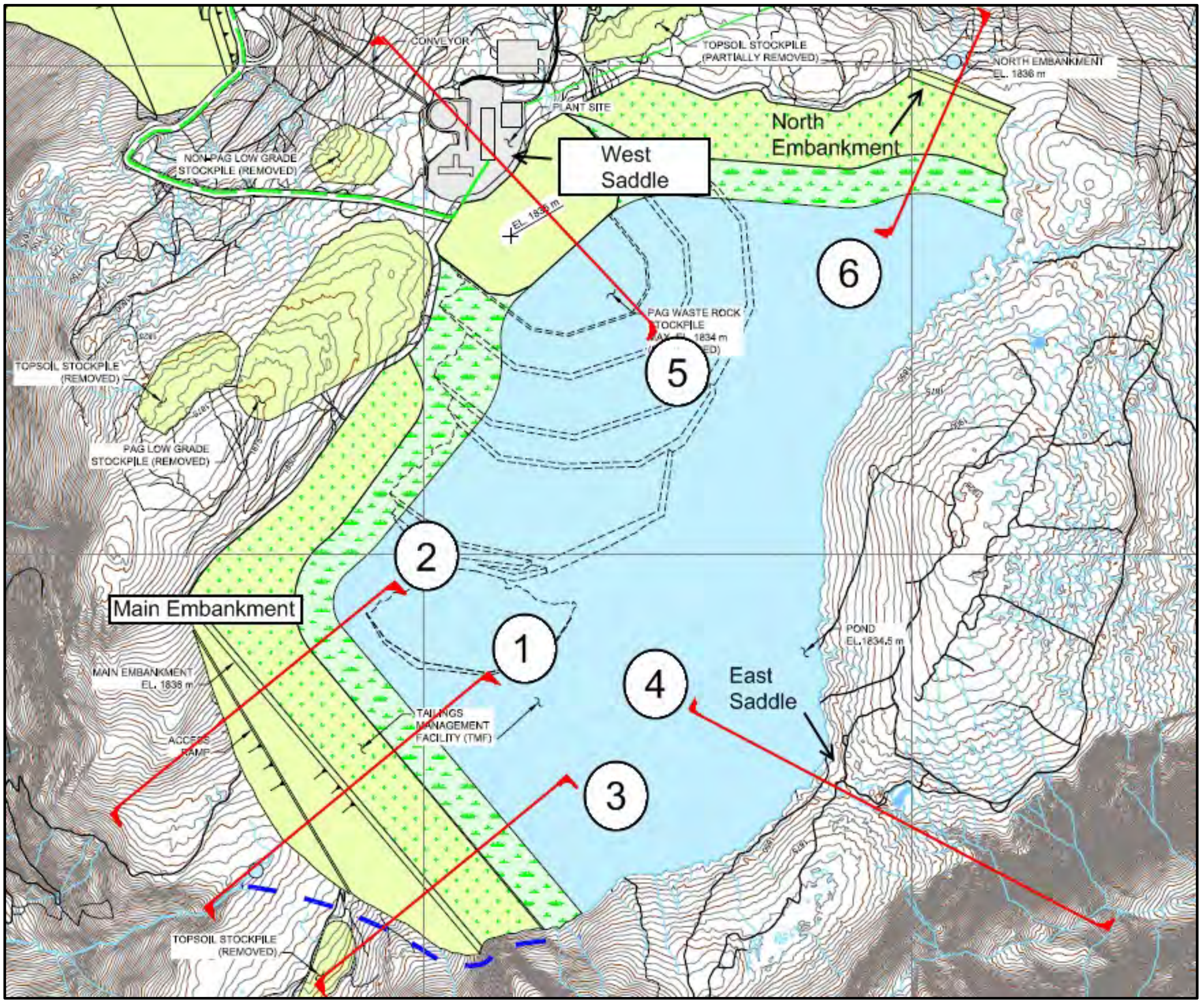


Figure 5.6 Location of Seepage Analysis Sections (2D)

The seepage flow rate is expected to vary over the life of the project as the TMF gradually fills with tailings, PAG waste rock and supernatant water. The tailings deposit will increase in thickness during operations and the tailings mass will decrease in permeability due to on-going self-weight consolidation. The seepage estimates are reported by means of the following metrics:

- Total seepage (L/s) indicates the total seepage that is expected to permeate through the TMF embankments and foundation for each section.
- Unrecoverable seepage (L/s) indicates the total seepage that is expected to be unrecoverable with the planned seepage control measures in place, and could reach the downstream receiving environment.

The results of the seepage analyses for the TMF are shown on Table 5.5.

Table 5.5 Expected Case Seepage Estimates

TMF Location	Total Seepage	Unrecoverable Seepage
	(L/s)	(L/s)
Main Embankment	14	1
North Embankment	0.1	0.1
East Saddle	0.1	0.1
West Saddle	< 0.1	< 0.1

The tailings beach is expected to extend approximately 300 m from the main embankment crest during normal operating conditions and following closure of the TMF. The length of the tailings beach will substantially influence the total seepage rates. An upper bound scenario was modelled to estimate the effect on seepage rates if the supernatant pond was allowed to extend to the main embankment crest. The result was an order of magnitude increase in total seepage to a maximum of approximately 160 L/s. The unrecoverable seepage did not increase in this scenario, indicating the proposed seepage control measures will still be effective. Seepage would be captured at the downstream water management pond and could be recycled for long-term storage within the TMF. The seepage collection measures are considered to be a robust solution for a wide range of embankment seepage flows.

5.8 EMBANKMENT STABILITY ANALYSES

Stability analyses were carried out to confirm the stability of the embankment under both static and seismic loading conditions (Appendix D). These analyses comprised checking the stability of the embankment arrangement for each of the following cases:

- Static conditions during operations and post-closure.
- Earthquake loading from the operating basis earthquake (OBE) and the maximum design earthquake (MDE).
- Post-earthquake conditions using residual (post-liquefaction) tailings strengths.

The stability analyses were carried out using the limit equilibrium computer program SLOPE/W. Factors of safety have been computed using the Morgenstern-Price method.

In accordance with international recommendations (ICOLD, 1995) and standard industry practice, the minimum acceptable factor of safety for the tailings embankment under static conditions is 1.3 for short-term operating conditions and 1.5 for long-term (steady-state and post-closure) of the TSF. A factor of safety of less than 1.0 is acceptable under earthquake loading conditions provided that calculated embankment deformations resulting from the seismic loading are not significant and that the post-earthquake stability of the embankment maintains a factor of safety greater than 1.2, implying there is no flow slide potential.

The seismic stability assessment of the TMF has included estimation of seismically induced deformations of the dam from the OBE and MDE events. The OBE has been defined as the 1 in 500 year earthquake with a mean Peak Ground Acceleration (PGA) of 0.08g and design earthquake magnitude of 7.0. The MDE corresponds to the halfway between the 1 in 2,475 year earthquake and the 1 in 10,000 year earthquake with a mean PGA of 0.21g and design earthquake magnitude of 7.3. The 1 in 10,000 year earthquake was also considered to demonstrate the robustness of the embankment design to seismic loading. The PGA for the 1 in 10,000 year event was 0.26g.

The stability analyses results satisfy the factor of safety design criteria and shows the proposed design is adequate to maintain both short term (operational) and long term (post-closure) stability. The seismic analyses indicate any embankment deformations during earthquake loading from the OBE, MDE, and 1 in 10,000 year event would be minor and would not have a significant impact of the available embankment freeboard or result in any loss of embankment integrity. The results indicate the embankments are not dependent on tailings strength to maintain overall stability and integrity.

5.9 TAILINGS DISTRIBUTION

The KP scope of work for the FS design specifically excluded the design of the tailings distribution system. An operational tailings deposition strategy is considered essential to the operational criteria of the TMF, and is described in the following section.

Two tailings streams will be generated in the process plant and transported to the TMF. The two types of tailings are designated as rougher scavenger (bulk) tailings and cleaner scavenger (cleaner) tailings. Bulk tailings will be transported by the bulk tailings distribution pipeline primarily to the main embankment during the first 24 years of operation, and later in mine life to the north embankment during between Years 19 and 24. The bulk tailings will be discharged from the embankment crests using multiple spigots to build extensive tailings beaches to hydraulically isolate the supernatant pond from the TMF embankments. Single point discharge will only be permitted during tailings distribution pipeline relocation. The tailings beaches are expected to develop at an average slope of approximately 0.5%, with coarser tailings settling out first in the vicinity of the embankment and finer slime tailings settling out later. A tailings beach width of approximately 300 m will be maintained, corresponding to a minimum freeboard of 1.5 m. Bulk tailings will also be deposited along the northern and western perimeters of the TMF to manage the size and location of the supernatant pond, and to progressively cover portions of the PAG waste rock stockpile.

Cleaner tailings will be transported to a separate location within the TMF by the cleaner tailings pipeline. Deposition of cleaner tailings will occur in an area that maintains the tailings solids in a subaqueous state perpetually.

The tailings pipelines will be extended towards the open pit in Year 23. Tailings deposition will occur in the open pit beginning late in Year 24 with the start-up of low grade ore processing and will continue until the end of operations. The distribution pipelines in the TMF area will be removed at this time.

5.10 GEOTECHNICAL MONITORING INSTRUMENTATION

Geotechnical instrumentation will be installed along three planes through the main embankment and one plane through the north embankment. The instrumentation will be installed during construction and over the life of the project. The geotechnical instrumentation will be comprised of vibrating wire piezometers, slope inclinometers and surface movement monuments, and will be installed in the foundations, embankment fill, tailings beach and on the embankment crests. Surface movement monuments will be spaced approximately 150 m apart on the embankment crests. Slope inclinometers will be placed downstream of each instrumentation plane to monitor any displacement due to embankment loading. Vibrating wire piezometers will be placed in key areas in the foundation and Stage 1 embankment fill to monitor performance of the design. Vibrating wires will also be placed upstream in the tailings beach, in the earthfill core zone, and downstream of the embankment drain every 10 m of height during the staged embankment raises. The location, types, and number of instrumentation will be further evaluated during detailed design of the project. A summary of the instrumentation in each plane is provided in Table 5.6.

Table 5.6 TMF Instrumentation Summary

Instrumentation Plane (and Location)	**Vibrating Wire Piezometers**			**Slope Inclinometer**
	Foundation	**Fill**	**Tailings**	
1 (Main Embankment)	4	24	12	2
2 (Main Embankment)	4	32	12	2
3 (Main Embankment)	4	24	12	2
4 (North Embankment)	4	8	2	2
Total	**16**	**88**	**38**	**8**

Instrumentation monitoring will be routinely completed during construction and operations. Measurements during construction will be taken and analysed on a routine basis to monitor the response of the embankment fill and the foundation from the loading of the embankment fill. The frequency of monitoring for the piezometers and inclinometers may be decreased once the effects of initial construction have dissipated. Surface monuments will be surveyed at least twice per year during operations. An Operations Maintenance and Surveillance (OMS) Manual will be prepared following initial construction and prior to commissioning of the TMF, and will provide comprehensive operating instructions and monitoring frequencies for the TMF and related facilities.

5.11 WASTE ROCK, OVERBURDEN AND LOW-GRADE ORE STOCKPILES

5.11.1 General

Mine waste and low-grade ore stockpile areas have been identified and subsequently refined and optimized to minimize surface and seepage water control requirements. Waste rock and overburden from pit stripping and low-grade ore will be separated by geochemical classification and stockpiled in different areas around the site. The stockpiles are shown on Figure 5.7 and discussed in the following sections.

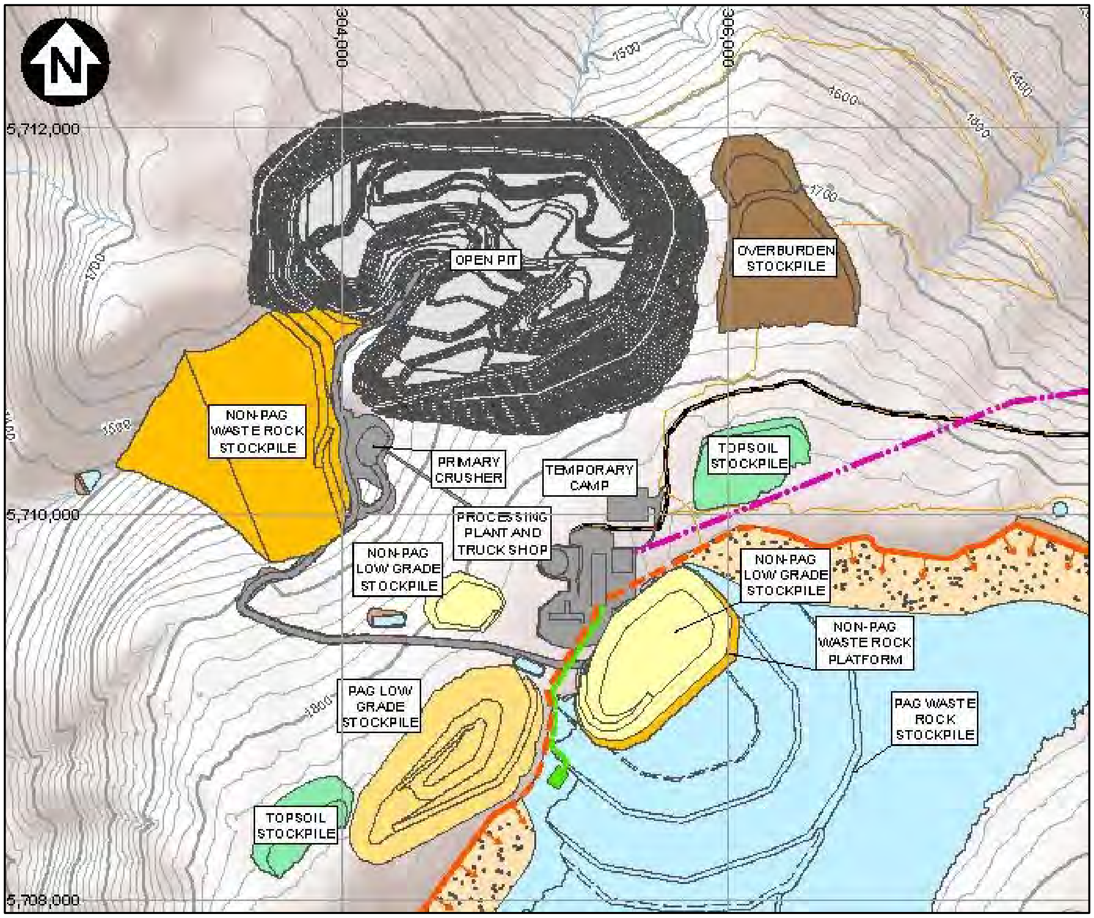


Figure 5.7 Stockpile Layout

5.11.2 Non-PAG Waste Rock Stockpile

The non-PAG waste rock stockpile is situated in the P-creek drainage to the southwest of the open pit. The stockpile will be constructed as a valley fill. The stockpile will be constructed using a descending method of construction during the first ten years in order to create access to the dump toe for the mine haul trucks. Construction of the stockpile will proceed as ascending construction thereafter, once the ultimate toe of the stockpile is established. The final elevation of the stockpile reaches an elevation of approximately 1660 m with a dump volume of approximately 265 Mt. An assessment of the stability rating for the stockpile has been carried out using the waste dump and stockpile stability rating system as described in the interim guidelines provided by the B.C. Mine Waste Rock Pile Research Committee (BCMWRPRC, 1991). The non-PAG waste rock stockpile was classified as Class III, Moderate Hazard. The specific key factors affecting stability, rated condition, and point ratings for classification of the stockpile are shown in Table 5.7.

Table 5.7 Non-PAG Waste Rock Stockpile Stability Classification

Key Factors Affecting Stability	Condition	Point Rating
Dump Height	100 - 200 m	100
Dump Volume	Large	100
Dump Slope	Moderate	50
Foundation Slope	Moderate	50
Degree of Confinement	Confined	0
Foundation Type	Intermediate	100
Dump Material Quality	Moderate	100
Method of Construction	Mixed	100
Piezometric & Climatic Conditions	Intermediate	100
Dumping Rate	Moderate	100
Seismicity	Moderate	50
DUMP STABILITY RATING		**850**
	Class	**Failure Hazard**
Dump Stability Class	III	Moderate

A preliminary assessment of stability of the non-PAG waste rock stockpile was carried out for the final design height of the stockpile. Details of assessment are included in Appendix D and described in the following section. The stability analyses were carried out using the limit equilibrium computer program SLOPE/W (Geostudio, 2007). A systematic search was performed using this program to obtain the minimum factor of safety from a number of potential slip surfaces. Factors of safety (FS) were computed using the Morgenstern-Price Method.

In accordance with provincial guidelines (BCMWRPRC, 1991) and standard industry practice, the minimum acceptable FS for waste stockpiles under static conditions was 1.3 for short-term operating conditions, 1.5 for long-term conditions after reclamation and abandonment and 1.0 for a pseudo-static analysis.

The results of the stability analyses satisfy the requirements for FS as indicated in the guidelines. The static FS for long-term conditions for the critical slip surface was 1.52. The pseudo-static analysis was completed using a mean peak ground acceleration of 0.8g corresponding to the 1 in 475 year event (corresponding to an event with a 10% probability of exceedance in 50 years). The pseudo-static FS was 1.38. The estimated yield acceleration was 0.19g, which is equivalent to an event with a return period of 1 in 5,000 years.

5.11.3 Overburden Stockpile

Overburden from pit stripping not used in the construction will be placed in one stockpile situated to the east of the open pit in the headwaters of the Jones Creek catchment. It will be a sidehill fill with an ascending method of construction. The overburden stockpile was designed to contain 39 Mt of material with a final elevation of 1760 m. The foundation is competent and gently sloped to the north. An assessment of the stability rating for the stockpile has been carried out using the waste dump and stockpile stability rating system as described in the interim guidelines (BCMWRPRC, 1991) and presented in Table 5.8. The specific key factors affecting stability, rated condition, and point ratings for classification of the stockpile were assessed. The overburden stockpile was classified as Class II, Low Hazard.

Table 5.8 Overburden Stockpile Stability Classification

Key Factors Affecting Stability	Condition	Point Rating
Dump Height	50 - 100 m	50
Dump Volume	Medium	50
Dump Slope	Moderate	50
Foundation Slope	Flat	0
Degree of Confinement	Mod. Confined	50
Foundation Type	Competent	0
Dump Material Quality	Poor	200
Method of Construction	Favorable	0
Piezometric & Climatic Conditions	Intermediate	100
Dumping Rate	Slow	0
Seismicity	Moderate	50
DUMP STABILITY RATING		**550**
	Class	**Failure Hazard**
Dump Stability Class	II	Low

5.11.4 Non-PAG Low-Grade Ore Stockpiles

Non-PAG LGO will be placed in two surface stockpiles. The first stockpile will be situated to the west of the plant site in the P-Creek catchment. It will be a temporary sidehill fill that will be processed entirely during the first five years of operations. It was designed to contain 4 Mt of LGO with a final elevation of 1830 m.

The second stockpile will be situated in the TMF basin to the southeast of the plant site area. A platform of non-PAG waste rock (approximately 15 Mt) will be constructed to an elevation of 1834 m, prior to placement of the LGO. The platform will be a sidehill fill constructed at a slope no steeper than 26 degrees with a competent, gently sloped foundation. The Non-PAG LGO stockpile will be a heaped fill constructed on the platform and eventually processed by end of Year 28. It was designed to contain up to 43 Mt of material with a final elevation of 1900 m. An assessment of the stability rating for the Non-PAG LGO stockpile has been carried out using the waste dump and stockpile stability rating system as described in the interim guidelines (BCMWRPRC, 1991) and presented in Table 5.9. The combined stockpile, including both the platform and heaped fill, was used in the stability rating assessment.

The specific key factors affecting stability, rated condition, and point ratings for classification of the stockpile were assessed. The Non-PAG LGO stockpile was classified as Class II, Low Hazard. The stockpile classification does not consider the removal of the LGO fill, or the inevitable rise of the PAG waste rock stockpile to encompass the LGO stockpile platform by the end of mining. Both of these factors will increase the long-term stability of this stockpile.

Table 5.9 Non-PAG LGO Stockpile Stability Classification

Key Factors Affecting Stability	Condition	Point Rating
Dump Height	100 - 200 m	100
Dump Volume	Medium	50
Dump Slope	Moderate	50
Foundation Slope	Flat	0
Degree of Confinement	Mod. Confined	50
Foundation Type	Competent	0
Dump Material Quality	Moderate	100
Method of Construction	Mixed	100
Piezometric & Climatic Conditions	Intermediate	100
Dumping Rate	Slow	0
Seismicity	Moderate	50
DUMP STABILITY RATING		**600**
	Class	**Failure Hazard**
Dump Stability Class	II	Low

5.11.5 PAG Low-Grade Ore Stockpile

The PAG LGO will be placed in a single stockpile to the southwest of the plant site and north of the TMF. The stockpile was designed as a ridge crest fill with ascending construction, and will also include a portion of heaped fill up to the final design elevation. It was designed to contain up to 86 Mt of material with a final elevation of 2000 m. The overall stockpile slope angles will be approximately 26 degrees and will be constructed over a competent, gently sloped foundation. The footprint will be cleared and stripped of topsoil in preparation for stockpile placement. A 500 mm layer of compacted Zone S material from overburden stripping in the pit will be placed on the PAG LGO stockpile foundation. The sub-grade treatment will extend approximately 10 m beyond the stockpile edge to allow for runoff collection from the stockpile.

An assessment of the stability rating for the Non-PAG LGO stockpile has been carried out using the waste dump and stockpile stability rating system as described in the interim guidelines (BCMWRPRC, 1991) and presented in Table 5.10. The specific key factors affecting stability, rated condition, and point ratings for classification of the stockpile were assessed. The PAG LGO stockpile was classified as Class III, Moderate Hazard. It should be noted that the duration of the hazard is short-term, since the stockpile will be processed and the area reclaimed prior to the end of operations.

Table 5.10 PAG LGO Stockpile Stability Classification

Key Factors Affecting Stability	Condition	Point Rating
Dump Height	100 - 200 m	100
Dump Volume	Medium	50
Dump Slope	Moderate	50
Foundation Slope	Flat	0
Degree of Confinement	Unconfined	100
Foundation Type	Competent	0
Dump Material Quality	Moderate	100
Method of Construction	Favourable	0
Piezometric & Climatic Conditions	Intermediate	100
Dumping Rate	Moderate	100
Seismicity	Moderate	50
DUMP STABILITY RATING		**650**
	Class	**Failure Hazard**
Dump Stability Class	III	Moderate

5.11.6 Topsoil Stockpiles

Topsoil will be removed from key areas and placed in stockpiles for reclamation purposes. Three topsoil stockpile locations were identified as part of the study. The first two (East and West) are shown on a preceding page, on Figure 5.7, and the third (South) is situated downstream of the TMF main embankment. The details of each topsoil stockpile, including volume, height, crest elevation, slope angle and foundation conditions are presented in Table 5.11.

Table 5.11 Topsoil Stockpile Details

Stockpile Location	Volume	Height	Crest Elevation	Dump Slope Angle	Foundation Conditions
	(Mm^3)	(m)	(masl)	(ratio)	
East Topsoil	1.2	14	1840	5H:1V	Gentle slope (6%), competent
West Topsoil (LGO)	0.7	9	1871	5H:1V	Gentle slope (11%), competent
South Topsoil (TMF)	0.4	12	1708	5H:1V	Gentle, concave slope (13%), competent

5.11.7 Recommendations

The guidelines (BCMWRPRC, 1991) provide suggestions on the recommended levels of effort for investigation and design of mine rock and overburden piles. Consideration should be given to the failure hazards noted above and the recommended levels of effort noted below.

Low failure hazard rating:

- Thorough site investigation, including test pits
- Samples may be required, including limited lab index testing
- Basic stability analyses required, and
- Routine visual and instrument monitoring.

Moderate failure hazard rating:

- Detailed, phased site investigation including test pits and drilling or other subsurface investigations
- Detailed lab testing including index properties, shear strength and durability likely required
- Detailed stability analyses, possibly including parametric studies, required
- Stage II detailed design report may be required for approval/permitting
- Moderate restrictions on construction (eg. limiting loading rate, lift thickness, and/or material quality), and
- Detailed instrument monitoring to confirm design, document behavior and establish loading limits.

Additional site investigation during the permitting phase is recommended to support waste stockpile detailed design. Site investigation effort should focus on the Non-PAG waste rock stockpile area and LGO stockpiles, and include foundation area wide test pit coverage and lab index testing.

Basic stability analyses for all facilities are recommended for the next phase of project development, including a more detailed analysis, incorporating details from the site investigation, for the Non-PAG waste rock stockpile.

6 – WATER MANAGEMENT SYSTEM DESIGN

6.1 WATER MANAGEMENT OBJECTIVES

The overall water management objective is to provide sufficient water to support the mill water requirements and maintain potentially acid generating (PAG) materials in the TMF in a subaqueous state, while mitigating environmental impacts to downstream receiving waters. Water will be controlled to minimize erosion in areas disturbed by construction activities and to prevent release of sediment-laden water to the receiving environment. This includes collection and diversion of surface water runoff and constructing and operating sediment control ponds, seepage collection systems, and pumpback systems. The key facilities requiring water management planning are the:

- open pit
- process plant site, truck shop, and laydown areas
- roads
- TMF
- Non-PAG waste rock stockpile
- low-grade ore stockpiles, and
- overburden stockpile.

The key elements for water management are:

- water management ponds
- water pumpback systems
- collection and diversion channels
- seepage prevention and collection measures
- TMF water reclaim system
- surface and groundwater monitoring systems, and
- sediment and erosion control measures including sediment control ponds for the facilities listed above.

Water within the Project area will be recycled and used to the maximum practical extent by collecting runoff from the mine site area. Site runoff water will be collected and stored within the TMF and used to inundate the PAG waste rock and tailings solids to prevent the onset of acid rock drainage and minimize metal leaching. Excess water will be stored in the supernatant pond within the TMF and recycled to the mill for use in the process. The water supply sources for the Project are as follows:

- runoff from the catchment above the Project site
- direct precipitation onto the TMF and runoff from the mine site facilities
- water recycle from the TMF supernatant pond
- groundwater and surface water from open pit dewatering

The following sections describe the water management strategies, design elements, and facilities through the construction (preproduction), operations and closure phases of the Project.

6.2 MINE DEVELOPMENT AND MINE WATER MANAGEMENT

6.2.1 General

The sequence of mine development, mine waste production and management, and associated mine water management is essential to the design of water management systems and the modelling of predicted water quantity and quality effects. There are five stages of mine development that were considered:

- Construction (two preproduction years, referred to as Year -2 and Year -1)
- Operations I (during active mining in the open pit, Year 1 through a portion of Year 24)
- Operations II (during low-grade ore processing, from end of active mining through Year 28)
- Closure (during active closure and reclamation phase while open pit and TMF are filling), and
- Post Closure (steady-state long-term closure condition following active closure).

A water management plan indicating water movement strategies for each of the five stages of mine development are shown on Figures 6.1 to 6.5. A summary of the mine development sequence and mine water management plans are described in the following sections.

6.2.2 Construction

The project begins with two years of construction prior to mine operation. The construction phase includes the following development activities:

- Collection channels and water management ponds are constructed downstream of key development areas.
- Topsoil is stripped from the open pit and key areas of the TMF and stored in topsoil stockpiles.
- Overburden and non-PAG waste rock is stripped from the open pit and used in construction of mine facilities (roads, crusher pad, plant site grading, TMF embankments, or stored in the overburden or non-PAG waste rock stockpiles.
- PAG waste rock is stripped from the open pit and placed in the TMF for long-term storage or used in construction of the upstream zone of the main embankment (upstream of core zone).
- LGO encountered during pit pre-stripping is stored in one of three surface stockpiles, depending on geochemical classification and grade of the material.
- The cofferdam for the TMF main embankment is constructed to EL. 1683 m, followed immediately by construction of the Stage 1 embankment to EL. 1720 m.
- Tailings distribution system to the main embankment is constructed, but inactive.
- Water reclaim system from the TMF to the process plant is constructed, but inactive.
- Diversion channels are constructed to route non-contact water to the downstream receiving environment.
- Water management ponds function as sediment control ponds with surface water discharge to receiving environment with water quality meeting federal and provincial discharge standards.
- Water management pump systems and pipelines are installed at all water management ponds.

6.2.3 Operations I

The first operational phase includes the active mining and ore processing period and is approximately 24 years long. This period includes the following activities:

- Overburden and non-PAG waste rock is stripped from the open pit and used in on-going construction of the TMF embankments or stored in the overburden or non-PAG waste rock stockpiles.
- PAG waste rock is stripped from the open pit and placed in the TMF for long-term storage, or used in construction of the upstream zone of the main embankment (first five years only) and subsequently buried with NAG waste and flooded.
- LGO encountered during mining is stored in one of three surface stockpiles, depending on geochemical classification and grade of the material. Stockpiled LGO is used periodically to supplement volume and grade of ore processing.
- Crushing and milling of ore underway at the crusher pad and process plant, respectively.
- Tailings from ore processing conveyed as slurry to TMF for long-term storage.
- Supernatant water from TMF is reclaimed and reused in ore processing.
- Portion of supernatant water is separated, treated, and used as fresh water in the mill for reagent makeup, gland seals, and most purposes other than potable water.
- Diversion channels continue to route non-contact water to the downstream receiving environment and are adjusted as required due to the expanding mine site.
- Seepage and surface runoff from the non-PAG waste rock stockpile, PAG LGO stockpile, and the non-PAG LGO stockpile outside the TMF are collected in water management ponds and pumped to the TMF for long-term storage and reuse.
- Surface runoff from the overburden stockpile is conveyed by collection ditch to the open pit during until approximately Year 10, and then routed through a sediment control pond with surface water discharge to receiving environment if water quality meets federal and provincial discharge standards.
- Pit wall seepage and surface runoff from the open pit is conveyed to the TMF for long-term storage and reuse.

6.2.4 Operations II

The second operational phase commences when active mining is complete in the open pit. This phase consists of low-grade ore processing and includes the following activities:

- Crushing and milling of LGO underway at crusher pad and process plant, respectively.
- Concurrent reclamation activities begin in key areas around the project including non-PAG waste rock stockpile, TMF tailings beaches, and TMF embankments.
- Tailings from LGO processing conveyed as slurry to the open pit for long-term storage.
- Water from TMF is reclaimed and reused in ore processing for first 18 months as the open pit begins to fill, and then water is reclaimed from the open pit for the remainder of operations.
- Open pit water management system is partially decommissioned and removed. Pit inflows from seepage and surface runoff are allowed to accumulate in the pit.
- Seepage and surface runoff from the non-PAG waste rock stockpile and PAG LGO stockpile are collected in water management ponds and pumped to the TMF for long-term storage.
- TMF is allowed to fill with surface runoff and inflows from site water management systems, diluting concentrations of key parameters in the TMF pond.
- The closure spillway is constructed at southeastern end of the TMF, but remains inactive until TMF pond has filled.

6.2.5 Closure

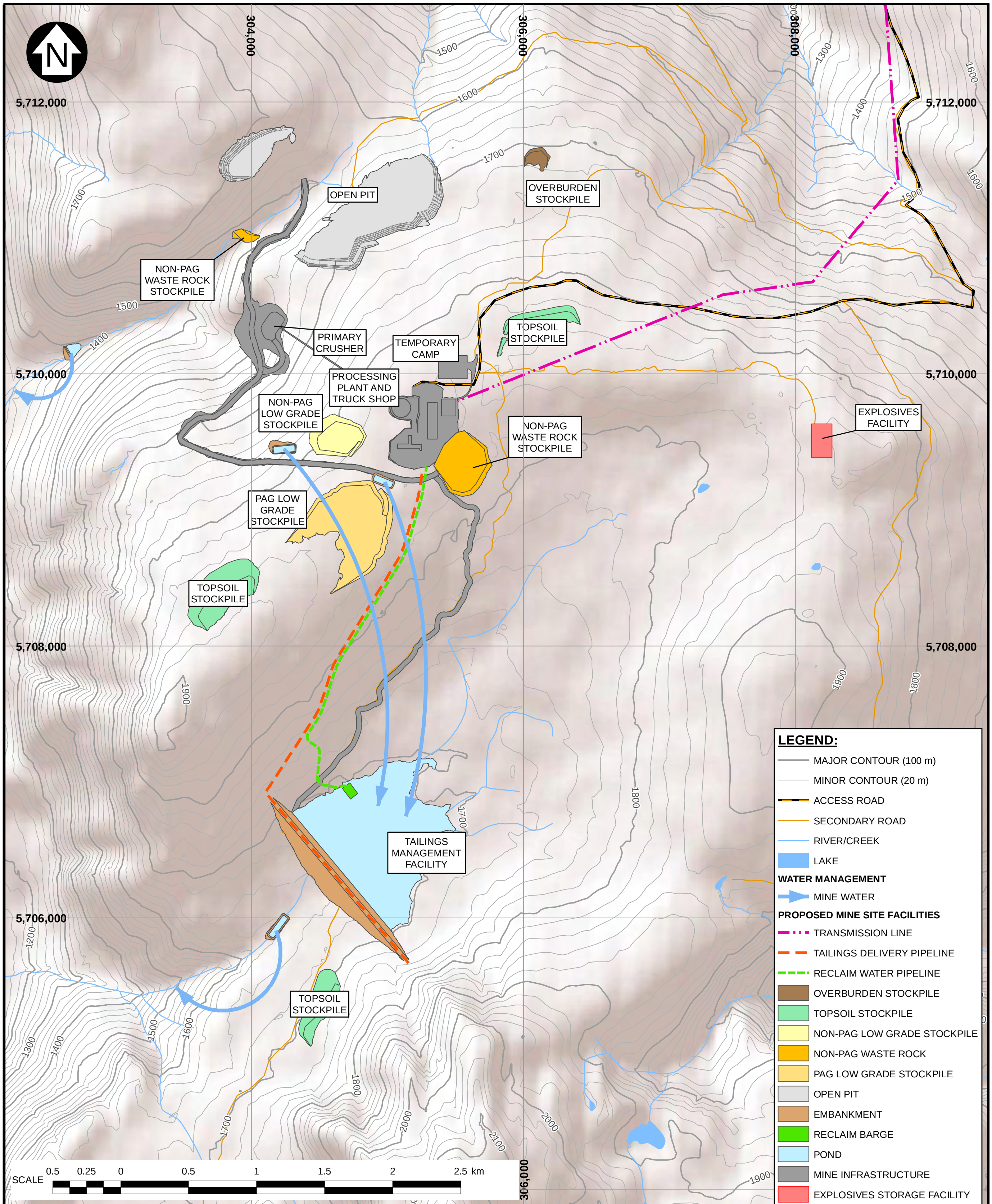
There is a period of time following LGO processing where the mine site is being actively reclaimed and has not yet reached end land use objectives. Activities during this period include:

- Final reclamation activities occur for plant site, tailings distribution, LGO stockpiles and associated water management ponds and pipelines.
- Seepage and surface runoff from the non-PAG waste rock stockpile is collected in the water management pond and pumped to the TMF for long-term storage.
- Pit inflows from seepage and surface runoff are allowed to accumulate in the pit diluting concentrations of key parameters in the pond.
- Open pit water management system for closure is commission, but remains inactive.
- The TMF closure spillway becomes active and excess site water is discharged via the spillway to the downstream receiving environment.

6.2.6 Post Closure

The active closure period continues until the open pit pond reaches the final design elevation (EL. 1530 m), and the following activities will occur in perpetuity during the post closure period:

- Excess pit inflows from seepage and surface runoff are pumped to the TMF for long-term storage and discharge management.
- Seepage and surface runoff from the non-PAG waste rock stockpile is collected in the water management pond and pumped to the TMF for long-term storage.
- The TMF closure spillway remains active and excess site water is discharged via the spillway to the downstream receiving environment.



NOTES:

1. BASE MAP: TRIM AND NTS MAPPING, ESRI ARCGIS ONLINE SHADED RELIEF.
2. COORDINATE GRID IS IN METRES. COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N.
3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:40,000 FOR 8.5x11 (LETTER) PAPER.

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	CAC	BB	KJB

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

MINE WATER MANAGEMENT PLAN CONSTRUCTION

Knight Piésold CONSULTING

P/A NO. VA101-458/11

REF NO. 1

FIGURE 6.1

REV 0

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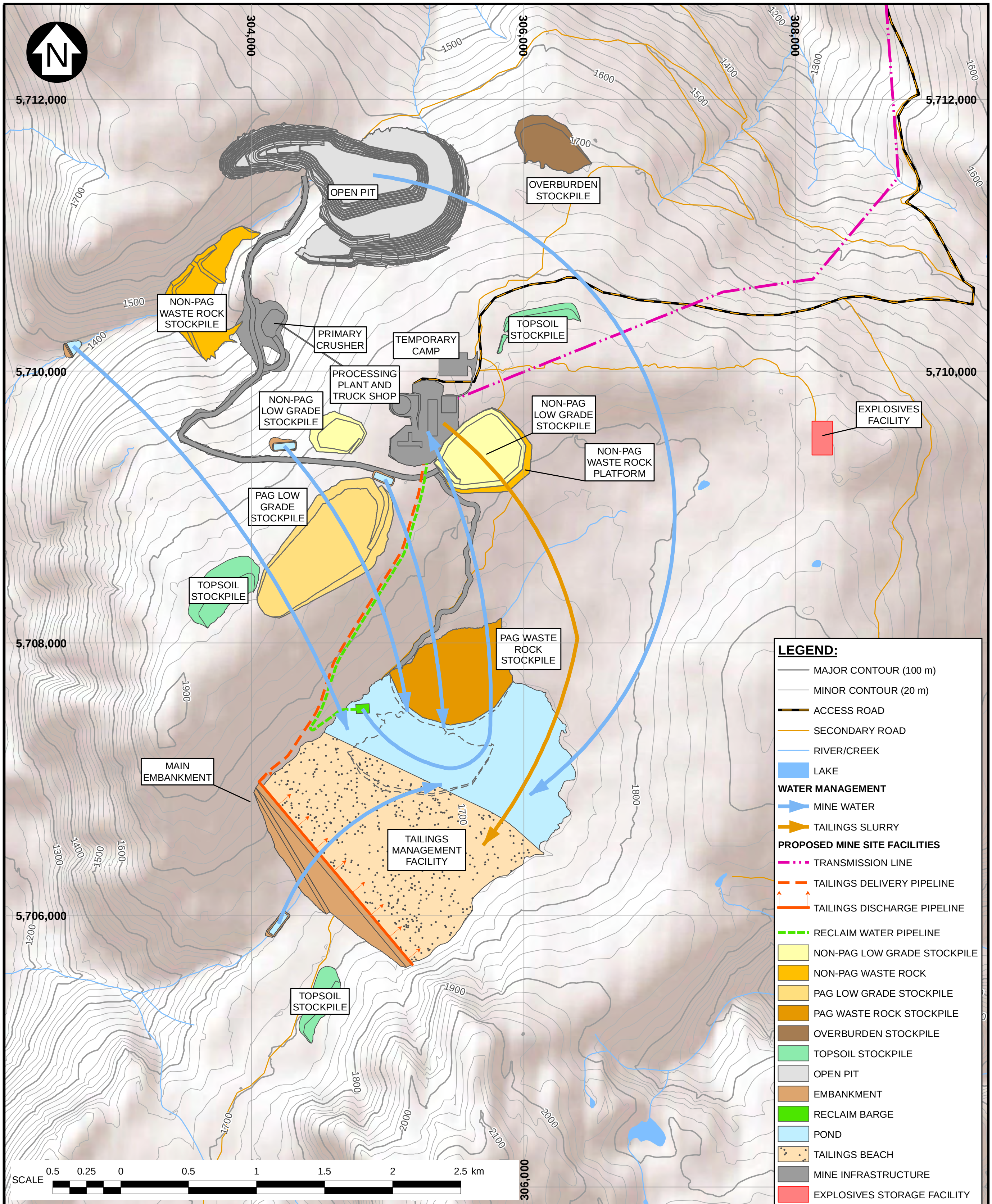

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NOTES:

1. BASE MAP: TRIM AND NTS MAPPING, ESRI ARCGIS ONLINE SHADED RELIEF.
2. COORDINATE GRID IS IN METRES. COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N.
3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:40,000 FOR 8.5x11 (LETTER) PAPER.

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HARPER CREEK MINING CORP.

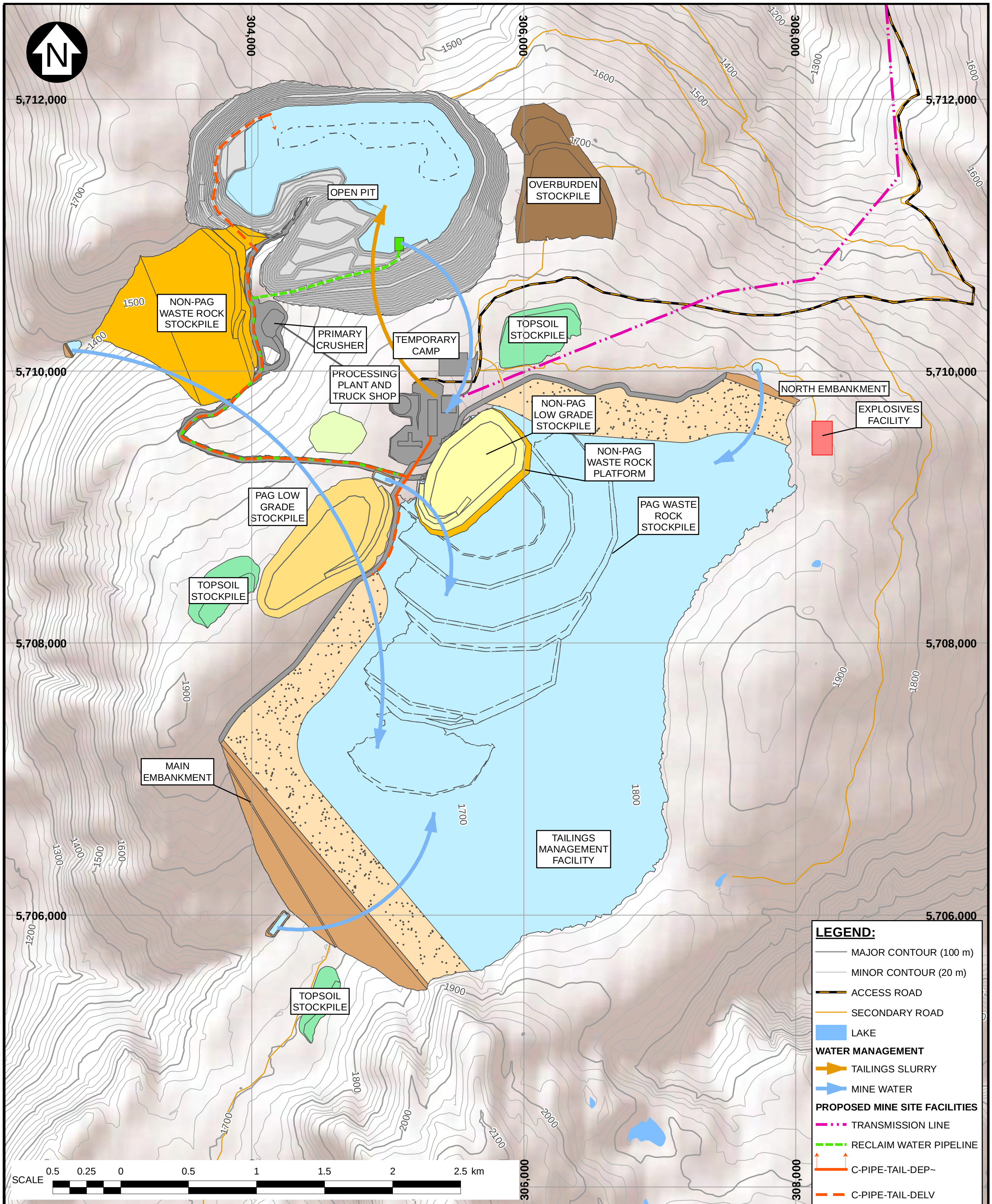
HARPER CREEK PROJECT

MINE WATER MANAGEMENT PLAN
OPERATIONS I

Knight Piésold CONSULTING

P/A NO. VA101-458/11 | REF NO. 1

FIGURE 6.2 | REV 0



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NOTES:

1. BASE MAP: TRIM AND NTS MAPPING, ESRI ARCGIS ONLINE SHADED RELIEF.
2. COORDINATE GRID IS IN METRES. COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N.
3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:40,000 FOR 8.5x11 (LETTER) PAPER.

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	CAC	BB	KJB

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

MINE WATER MANAGEMENT PLAN
OPERATIONS II

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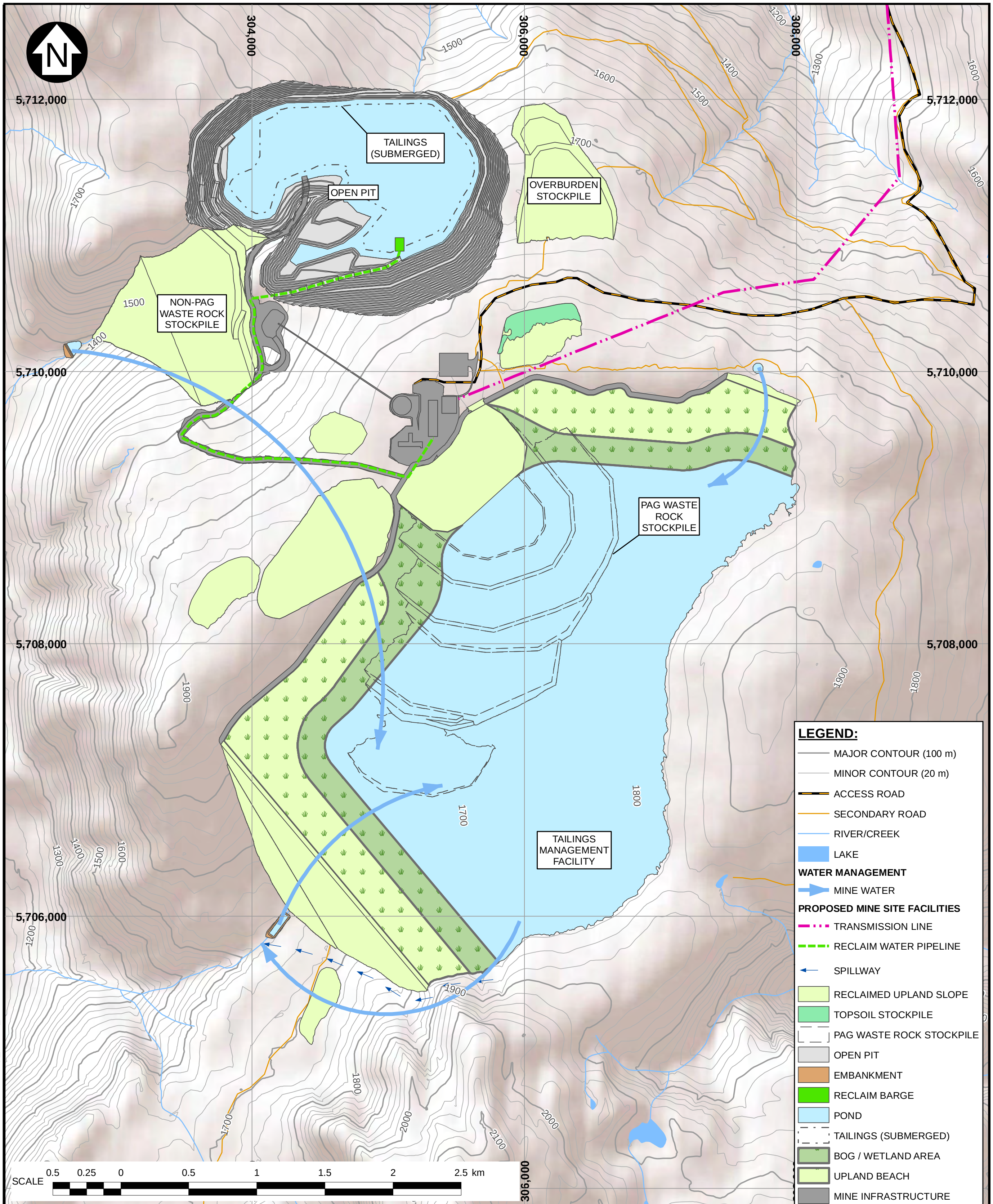
P/A NO. VA101-458/11

REF NO. 1

FIGURE 6.3

REV 0



NOTES:

1. BASE MAP: TRIM AND NTS MAPPING, ESRI ARCGIS ONLINE SHADED RELIEF.
2. COORDINATE GRID IS IN METRES. COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N.
3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:40,000 FOR 8.5x11 (LETTER) PAPER.

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

MINE WATER MANAGEMENT PLAN
CLOSURE

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P/A NO. VA101-458/11 | REF NO. 1

FIGURE 6.4 | REV 0

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	CAC	BB	KJB

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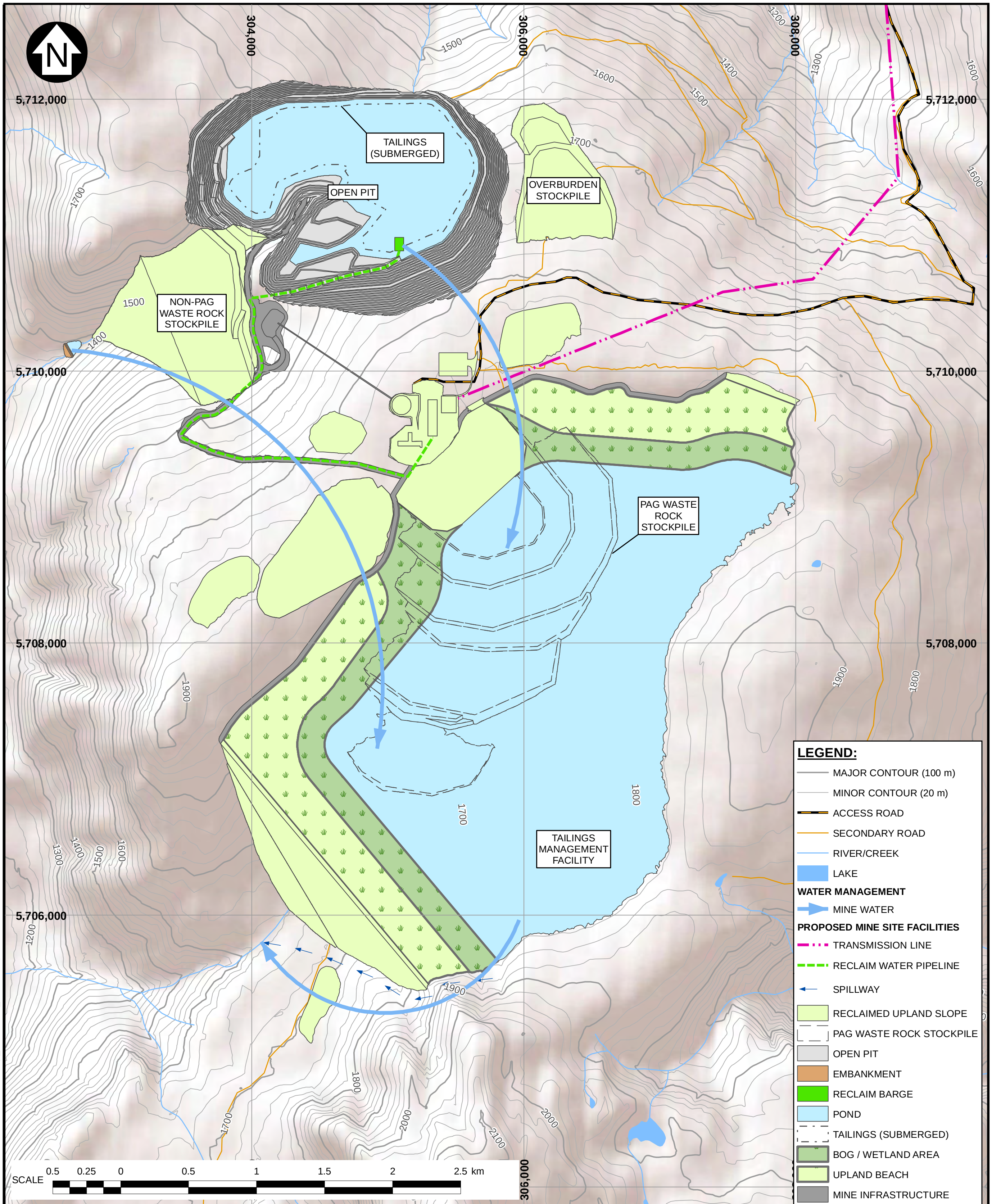

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NOTES:

1. BASE MAP: TRIM AND NTS MAPPING, ESRI ARCGIS ONLINE SHADED RELIEF.
2. COORDINATE GRID IS IN METRES. COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N.
3. THIS FIGURE IS PRODUCED AT A NOMINAL SCALE OF 1:40,000 FOR 8.5x11 (LETTER) PAPER.

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	CAC	BB	KJB

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

MINE WATER MANAGEMENT PLAN
POST CLOSURE

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P/A NO. VA101-458/11

REF NO. 1

FIGURE 6.5

REV 0

6.3 CONSTRUCTION WATER MANAGEMENT

Eight discrete areas of development have been identified within the Project boundary that will require an Erosion and Sediment Control Plan (ESCP) to be prepared during permitting of the project:

- construction camp
- process plant site and truck shop
- primary crusher
- open pit
- non-PAG waste rock stockpile
- overburden stockpile
- low-grade ore stockpiles, and
- tailings management facility (TMF).

The ESCP will describe specific surface water control elements and measures will be implemented in these areas to minimize erosion and prevent sediment discharge into surrounding areas. The plan will follow guidelines and recommendations provided by the BC Ministry of Environment in the guidance document Developing a Mining Erosion and Sediment Control Plan (BC MOE, 2014).

Surface water sediment mobilization and erosion will be managed throughout the site by:

- installing sediment controls prior to construction activities
- limiting the disturbance to the minimum practical extent
- reducing water velocity across the ground, particularly on exposed surfaces and in areas where water concentrates
- progressively rehabilitating disturbed land and constructing drainage controls to improve the stability of rehabilitated land
- scarifying the surface in rehabilitation areas to promote infiltration
- protecting natural drainages and watercourses by constructing appropriate sediment control devices such as collection and diversion ditches, sediment traps, and sediment ponds
- restricting access to rehabilitated areas, and
- constructing surface drainage controls to intercept surface runoff.

Subsurface water will be controlled by the use of sump pits, wells, or removable pump stations to draw down the natural water table and provide dry, stable construction areas. Excavations will be kept stable and workable by pumping water collected in the excavation sump pits to sediment control devices such as temporary holding ponds, sediment basins, or sediment filter bags where required.

An adaptive management approach will be implemented that allows sediment and erosion control works to be field-fit to suit conditions encountered during construction. Best management practices (BMPs) will be implemented before and during construction. Regular monitoring and maintenance of implemented BMPs will ensure success of the plan. The temporary sediment and erosion control features will be reclaimed after the soils and sediments have stabilized. The following is a summary list of BMPs that may be used at the Project site depending upon conditions encountered:

- vegetation management and revegetation
- mulching
- rolled erosion control products
- surface roughening
- recontouring

- silt fencing
- temporary sediment traps and sediment basins
- filter bags
- flocculants
- collection or diversion ditches
- culverts, and
- exfiltration areas.

In addition to the BMPs described above, a water management pond has been designed for each major area of disturbance. The ponds were designed in accordance with the Guidelines for Assessing the Design, Size and Operation of Sedimentation Ponds Used in Mining (BC MOELP, 1996). The ponds were designed to accommodate a live storage equal to the 1 in 10-year, 24-hour storm event with 0.5 m of freeboard and to settle out sediment particles sized 0.01 mm (and larger), while providing a retention time of at least 20 hours. Each pond and pond outlet spillway was designed to withstand a 1 in 200-year, 24-hour storm event, per the guidelines above. The collection and diversion ditches will be designed for the 1 in 10-year, 24-hour storm event. The collection and diversion ditches will be constructed adjacent to access trails consisting of a 6 m road platform for service equipment and access suitable for a four wheel drive light truck for monitoring. Ditches will typically be at least 1 m-deep and trapezoidal shaped with a 1 m-base width and 2H:1V slopes.

6.4 TAILINGS MANAGEMENT FACILITY

6.4.1 TMF Supernatant Pond

Surplus water will be stored on site within the TMF supernatant pond. Water from the TMF pond will be reclaimed using a dedicated barge-mounted pump station and re-used in ore processing. The water will be discharged from the mill as tailings slurry to the TMF pond. The pond volume will increase steadily during operations as the pond water is recycled and additional site water is pumped to the TMF for storage. The predicted TMF pond volume is summarized in Section 6.7 and additional details are provided in Appendix E.

The following sections provide the specifications of infrastructure and design flows for each water management system planned for operation of the project, including the water management ponds, pump systems, and pipelines.

6.4.2 Main Embankment

The water management pond for the main embankment will be situated at a topographic low point downstream of the embankment as shown on Figure 6.6. Collection ditches (red arrows on Figure 6.6) will be situated immediately downstream of the advancing embankment to collect storm water runoff and route the flows to the water management pond. Diversion ditches (blue arrows) will be constructed outside of the maximum extents of the embankment to direct runoff from undisturbed areas away from the construction area. The embankment seepage collection drains will be directed to the pond, which will allow for temporary storage and pumpback of seepage flows to the TMF. The seepage collection drains will be routed through a sump to allow for water quality monitoring prior to discharging to the water management pond.

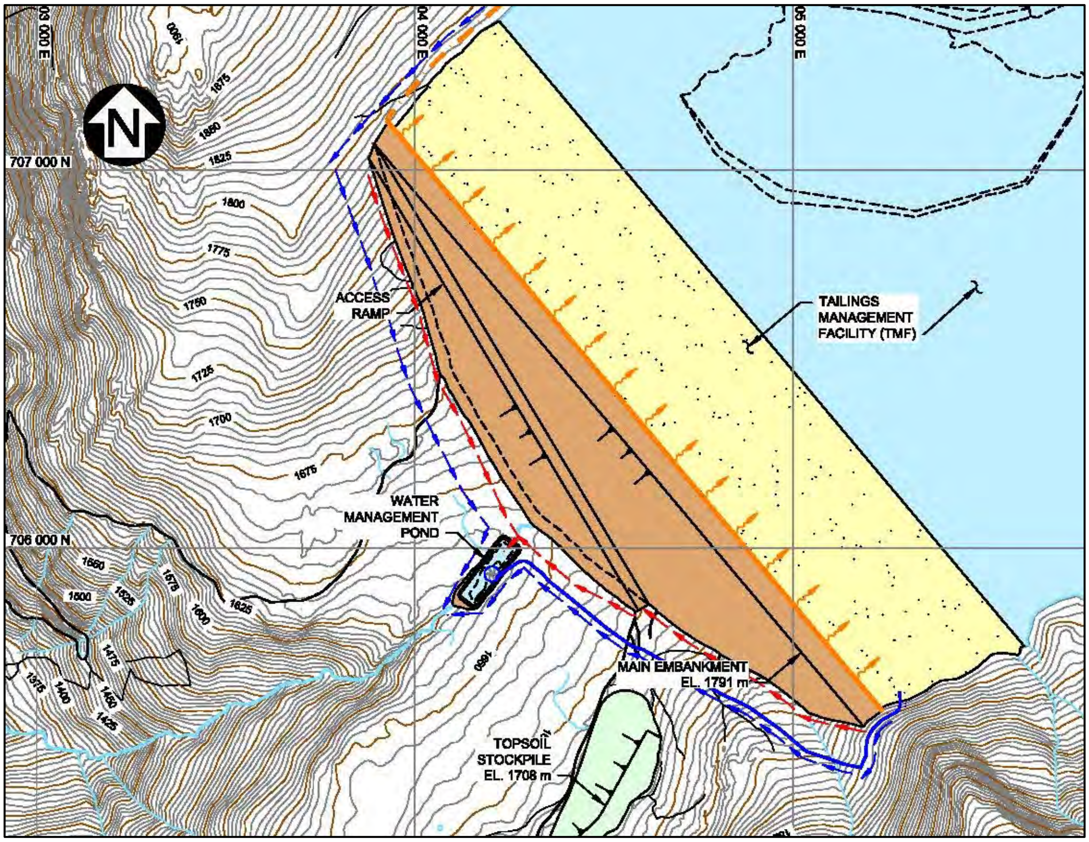

Figure 6.6 Main Embankment WMP

The pond will be rectangular with a 5:1 length to width ratio. It was designed to accommodate a live storage equal to the 1 in 10-year, 24-hour storm event and to settle out sediment particles sized 0.01 mm (and larger), while providing a retention time of at least 20 hours. The primary outlet consists of a 240 mm riser pipe situated at EL. 1637 m. The secondary outlet consists of a 5 m wide spillway and was designed to withstand a 1 in 200-year, 24-hour storm event with freeboard of at least 0.5 m. Additional design details for the pond, including minimum pond dimensions and peak event flows, are provided in Table 6.1.

Table 6.1 Main Embankment WMP Details

Main Embankment WMP	
Base Dimensions (W x L)	35 m x 175 m
Minimum Depth (D)	4 m
Peak Inflow (Q_{10})	1.99 m^3/s
Peak Outflow (Q_{10})	0.104 m^3/s
Peak Inflow (Q_{200})	5.05 m^3/s
Peak Outflow (Q_{200})	2.21 m^3/s

The pond dimensions provided above are the minimum dimensions required to provide adequate capacity to manage the design inflows. The pond design exceeds the minimum required rectangular dimensions and will include additional storage capacity below the base dimensions indicated above. Additional layout details for the pond are provided in the design drawings in Appendix A.

The water management pond has been designed with a pumpback system and pipeline to convey seepage and storm water inflows to the TMF for storage. The pumpback system will consist of four 200 horsepower (hp) in-line centrifugal pumps (three operating and one installed spare). The system will require one booster station to reach the TMF, consisting of the same pump arrangement as the intake system. The system was designed to pump a maximum design flow of 460 m^3/hour (0.128 m^3/s) through a 14 inch HDPE DR11 pipeline.

6.4.3 North Embankment

The water management pond for the north embankment will be situated to the north of the TMF in the headwaters of the Jones Creek catchment. This water management pond was predicted to be required around Year 18 when the TMF impoundment has expanded towards north and an embankment is needed to provide containment at the drainage divide between the TMF basin and the Jones Creek catchment. Collection ditches will be situated immediately downstream of the embankment to collect storm water runoff and the limited TMF seepage predicted to flow in this direction. The pond was designed to accommodate a live storage equal to the 1 in 10-year, 24-hour storm event and to settle out sediment particles sized 0.01 mm (and larger), while providing a retention time of at least 20 hours. The primary outlet consists of a 100 mm riser pipe situated at EL. 1822 m. The secondary outlet consists of a 5 m wide spillway and was designed to withstand a 1 in 200-year, 24-hour storm event with freeboard of at least 0.5 m. The pond will be rectangular with a 5:1 length to width ratio. Additional design details for the pond are provided in Table 6.2.

Table 6.2 North Embankment WMP Details

North Embankment WMP	
Base Dimensions (W x L)	10 m x 50 m
Minimum Depth (D)	3.5 m
Peak Inflow (Q_{10})	0.23 m^3/s
Peak Outflow (Q_{10})	0.01 m^3/s
Peak Inflow (Q_{200})	0.60 m^3/s
Peak Outflow (Q_{200})	0.38 m^3/s

The water management pond has been designed with a pumpback system and pipeline to convey seepage and storm water inflows to the TMF for storage. The pumpback system will consist of two 10 hp in-line centrifugal pumps (one operating and one installed spare). The system was designed to pump a maximum design flow of 35 m^3/hour (0.010 m^3/s) through a 4 inch HDPE DR11 pipeline.

6.5 WASTE ROCK, OVERBURDEN AND LOW-GRADE ORE STOCKPILES

6.5.1 Non-PAG Waste Rock Stockpile

The water management pond for the non-PAG waste rock stockpile was designed as an instream pond within P-Creek downstream of the ultimate extents of the waste rock stockpile as shown on Figure 6.7. Collection ditches (red arrows on Figure 6.7) will be situated around the advancing stockpile to collect storm water runoff and route the flows to the water management pond. Diversion ditches (blue arrows) will be constructed outside of the maximum extents of the embankment to direct runoff from undisturbed areas away from stockpile to maintain flow to the downstream environment to the maximum practical extent.

The pond will be created by constructing a 10.5 m high water retaining dam across P-Creek. The dam will be constructed in one stage. The embankment was designed with a 10 m crest width at EL. 1387 m and 2H:1V slopes on the downstream side. The upstream slope was flattened to 3H:1V to maintain stability during rapid dewatering following a storm event. The embankment will be comprised of Zone C material, with an upstream low-permeability (Zone S) layer 4 m-thick and two downstream filter/transition layers (Zones F and T) to maintain the integrity of the Zone S layer. The upstream Zone S layer will be keyed into low permeability subgrade materials, as described in Section 5.5.

The pond level will be maintained at the minimum operating level by a level-actuated pumpback system to convey stockpile infiltration and storm water inflows to the TMF for storage. The pumpback system will consist of four 300 hp in-line centrifugal pumps (three operating and one installed spare). The system will require four booster stations to reach the TMF, consisting of the same pump arrangement as the intake system. The intake system will be a small floating station, which requires 1.5 m of submergence to operate, and adjacent booster station. The system was designed to pump a maximum design flow of 910 m^3/hour (0.253 m^3/s) through a 20 inch HDPE DR11 pipeline.

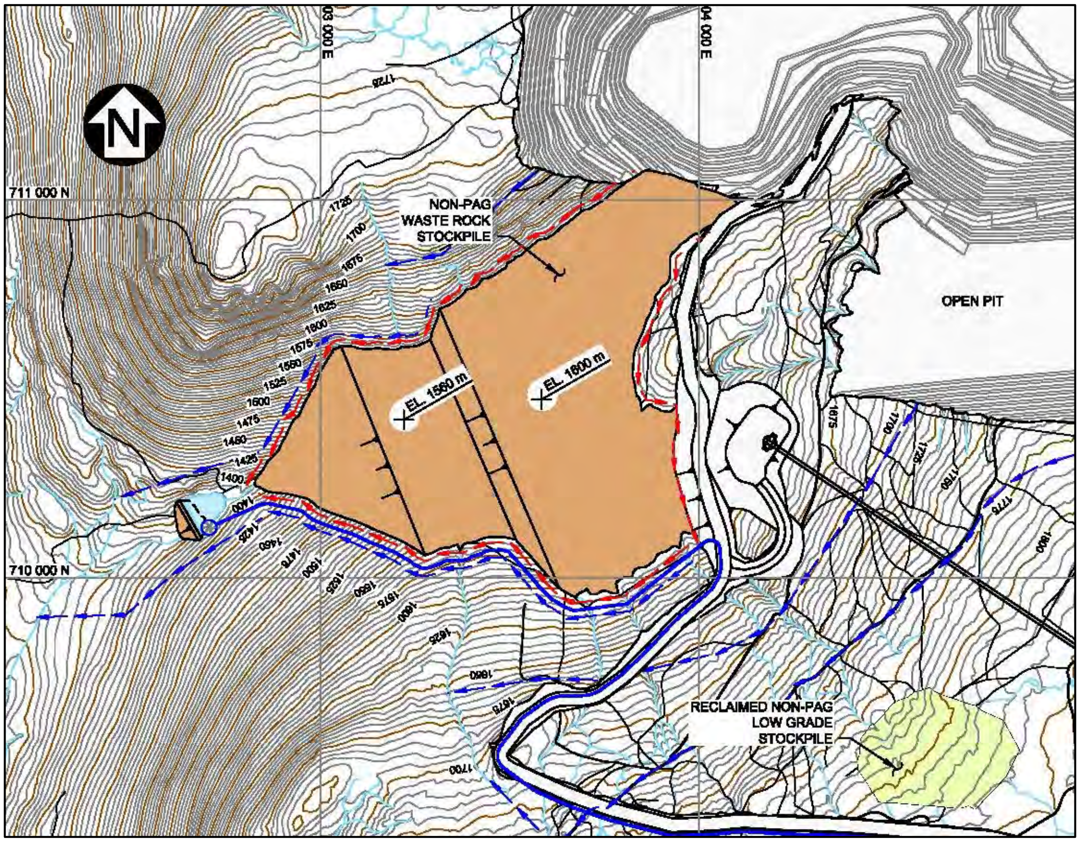

Figure 6.7 Non-PAG Waste Rock Stockpile WMP

The pond was designed with 1.5 m of dead storage below EL. 1377.5. It was designed to capture an inflow volume of over 24,000 m^3 in excess of the dead storage capacity, which is equivalent to the 1 in 10-year, 24-hour storm event. A 5 m wide spillway with an invert at EL. 1384.5 m was designed to withstand a 1 in 200-year, 24-hour storm event with dam freeboard of at least 0.5 m. Additional layout details are included in the design drawing package in Appendix A. Design details for the pond, including peak event flows, are provided in Table 6.3.

Table 6.3 Non-PAG Waste Rock Stockpile WMP Details

Non-PAG Waste Rock WMP	
Dam Height (H)	10.5 m
Pond Operating Depth (D)	7 m
Peak Inflow (Q_{10})	4.34 m^3/s
Peak Outflow (Q_{10})	0.16 m^3/s
Peak Inflow (Q_{200})	10.9 m^3/s
Peak Outflow (Q_{200})	2.55 m^3/s

6.5.2 Overburden Stockpile

Water from the overburden stockpile area will be managed by selective routing of runoff to prevent sediment laden water from entering the downstream receiving environment. Runoff during the first ten years of operations will be captured in a collection ditch downslope of the overburden stockpile. The collection ditch will route runoff to the open pit water management system for collection. The stockpile was designed as an ascending sidehill fill, and will reach an elevation of 1730 m during Year 10. The dump surface will then step back creating a large bench at elevation 1730 m, and then be lifted another 30 m over the remainder of the mine life. A sediment and erosion control pond will be constructed on the bench at elevation 1730 to manage sediment removal from surface runoff during the remainder of construction of the overburden stockpile. The collection ditch downslope will remain in place, but inactive, to allow for operational flexibility. The water management plan for the overburden stockpile for Year 10 is shown on Figure 6.8.

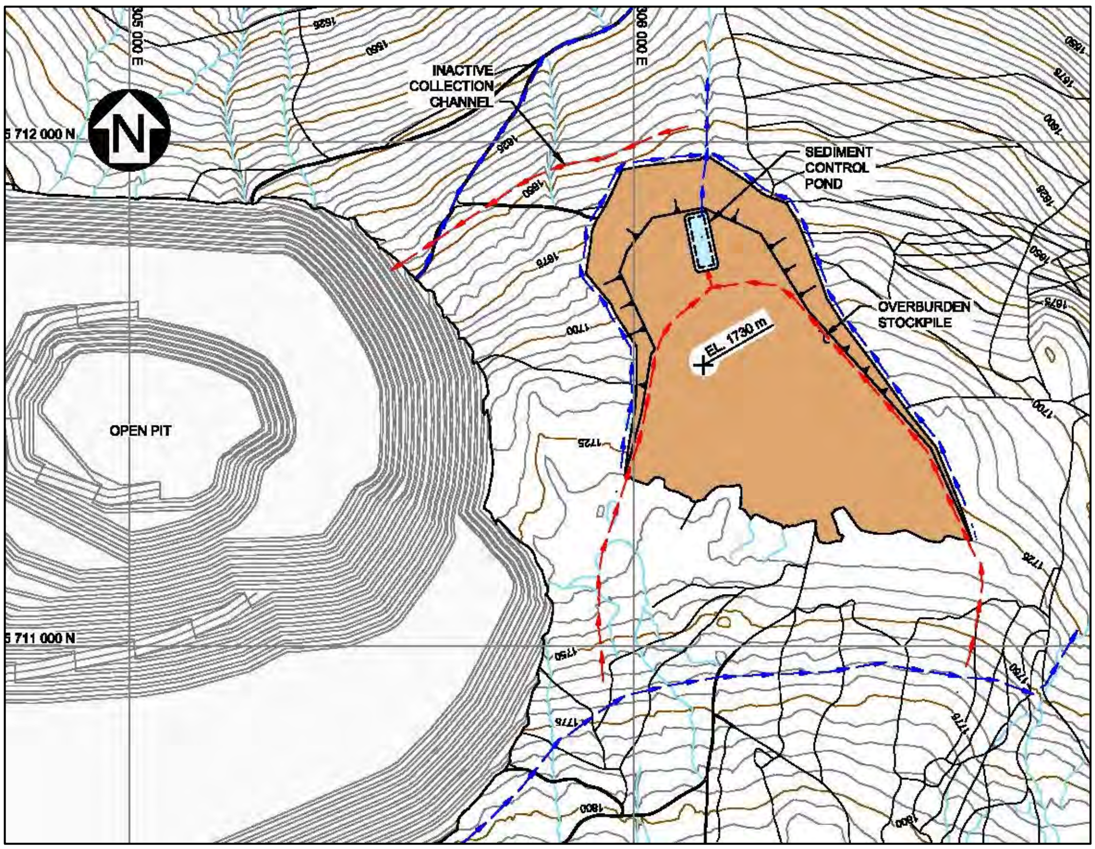

Figure 6.8 Overburden Stockpile Water Management in Year 10

The pond will be rectangular with a 5:1 length to width ratio. The pond will be approximately 25 m-width by 125 m-length by 5 m-depth, but final dimensions will depend on the total disturbed area during early operation of the mine. The pond will be designed to accommodate a live storage equal to the 1 in 10-year, 24-hour storm event with 0.5 m of freeboard and to settle out sediment particles sized 0.01 mm (and larger), while providing a retention time of at least 20 hours. The pond and pond outlet spillway will be designed to withstand a 1 in 200-year, 24-hour storm event. The collection and diversion ditches will be designed for the 1 in 10-year, 24-hour storm event.

6.5.3 PAG Low-Grade Ore Stockpile

The water management pond for the PAG low-grade ore stockpile will be situated at a topographic low point adjacent to the primary mine haul road from the pit to the TMF as shown on Figure 6.9. Collection ditches (red arrows on Figure 6.9) will be situated immediately downstream of the advancing stockpile to collect storm water runoff and stockpile infiltration, and route the flows to the water management pond. Diversion ditches (blue arrows) will be constructed outside of the maximum extents of the stockpile to direct runoff from undisturbed areas away from the stockpile construction area.

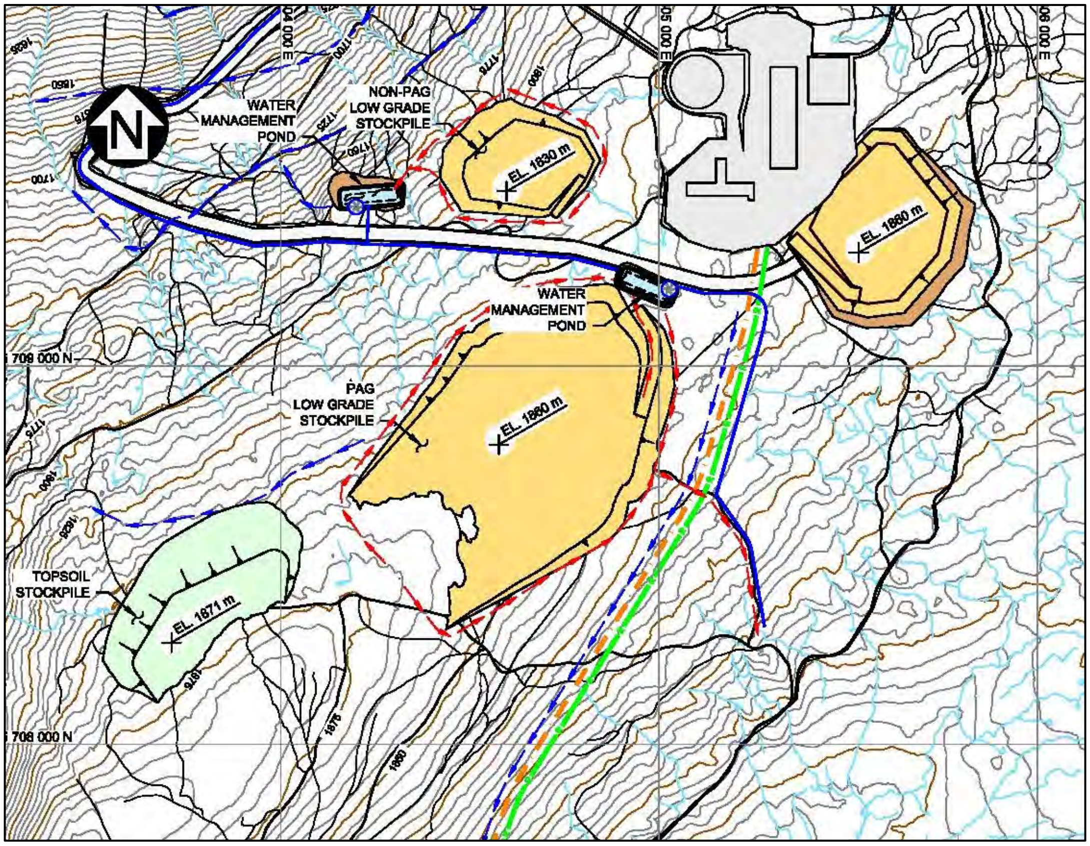

Figure 6.9 PAG Low-Grade Stockpile WMP

The pond will be rectangular with a 5:1 length to width ratio. It was designed to accommodate a live storage equal to the 1 in 10-year, 24-hour storm event and to settle out sediment particles sized 0.01 mm (and larger), while providing a retention time of at least 20 hours. The primary outlet consists of a 200 mm riser pipe situated at EL. 1833 m. The secondary outlet consists of a 5 m wide spillway and was designed to withstand a 1 in 200-year, 24-hour storm event with freeboard of at least 0.5 m. Additional design details for the pond, including minimum pond dimensions and peak event flows, are provided in Table 6.4.

The water management pond has been designed with a pumpback system and pipeline to convey stockpile infiltration and storm water runoff to the TMF for storage. The pumpback system will consist of two 40 hp in-line centrifugal pumps (one operating and one installed spare). The system was designed to pump a maximum design flow of 395 m^3/hour (0.109 m^3/s) through a 12 inch HDPE DR11 pipeline.

Table 6.4 PAG Low-Grade Stockpile WMP Details

PAG Low-Grade Stockpile WMP	
Base Dimensions (W x L)	28 m x 125 m
Minimum Depth (D)	5 m
Peak Inflow (Q_{10})	1.34 m^3/s
Peak Outflow (Q_{10})	0.085 m^3/s
Peak Inflow (Q_{200})	3.09 m^3/s
Peak Outflow (Q_{200})	2.10 m^3/s

6.5.4 Non-PAG Low-Grade Ore Stockpiles

Non-PAG LGO will be placed in two surface stockpiles. The first stockpile will be situated to the west of the plant site in the P-Creek catchment (first five years only). The second stockpile will be situated in the TMF basin to the southeast of the plant site area. Both stockpiles are shown on Figure 6.9.

Runoff and infiltration from the stockpile situated in the TMF will be controlled with drainage ditches and will report directly to the TMF supernatant pond, which is described in Section 6.4.1.

The water management pond for the temporary stockpile situated in the P-Creek catchment pond will be rectangular with a 5:1 length to width ratio. It was designed to accommodate a live storage equal to the 1 in 10-year, 24-hour storm event and to settle out sediment particles sized 0.01 mm (and larger), while providing a retention time of at least 20 hours. The primary outlet consists of a 200 mm riser pipe situated at approximately EL. 1780 m. The secondary outlet consists of a 5 m wide spillway and was designed to withstand a 1 in 200-year, 24-hour storm event with freeboard of at least 0.5 m. Additional design details for the pond, including minimum pond dimensions and peak event flows, are provided in Table 6.5.

Table 6.5 Non-PAG Low-Grade Stockpile WMP Details

Non-PAG Low-Grade Stockpile WMP	
Base Dimensions (W x L)	28 m x 140 m
Minimum Depth (D)	5 m
Peak Inflow (Q_{10})	2.13 m^3/s
Peak Outflow (Q_{10})	0.085 m^3/s
Peak Inflow (Q_{200})	5.62 m^3/s
Peak Outflow (Q_{200})	2.27 m^3/s

The water management pond has been designed with a pumpback system and pipeline to convey stockpile infiltration and storm water runoff to the TMF for storage. The pumpback system will consist of three 250 hp in-line centrifugal pumps (two operating and one installed spare). The system was designed to pump a maximum design flow of 375 m^3/hour (0.104 m^3/s) through a 12 inch HDPE DR11 pipeline.

6.6 OPEN PIT WATER MANAGEMENT

6.6.1 General

Open pit development will have a significant impact on the local hydrogeologic regime. The open pit will become a groundwater discharge area during excavation, and will progressively increase in size over the active mining period of the project. The existing groundwater table is at or near surface, and progressive development of the pit will result in a gradual lowering of the groundwater table in the vicinity of the excavation.

Pit water management systems are typically comprised of a combination of surface water diversion ditches, groundwater depressurization and pit dewatering pumping systems. These measures will be implemented in stages using an observational approach during pit development. This will enable an assessment of the pit slope drainage capability and the requirements for depressurization installations to remain quite flexible.

A conceptual water management plan has been developed for controlled removal of both groundwater inflows and precipitation runoff from within the pit, which include allowances for:

- Diversion ditches to collect surface runoff, snowmelt and seepage along the pit crest.
- A series of pumps and collection systems which transfer water from the pit excavation to the TMF for recycle to the milling process.

6.6.2 Surface Diversion Ditches

Diversion ditches along the pit crest are required to divert the surface runoff away from the pit during operations. It is recommended a staged sequence of diversion ditches be utilized to minimize surface water inflows during all phases of pit development. These surface runoff ditches will capture and divert the majority of all runoff and snowmelt before the water reaches the pit and will reduce power requirements for pumping water from the deeper levels of the open pit. It may be necessary to include a low permeability liner along select sections of these ditches in order to reduce seepage losses depending on conditions encountered during construction.

6.6.3 Design Basis

The open pit development sequencing throughout mine life was divided into four general stages to evaluate the requirements for surface water management. The design of each stage of the dewatering system considered the extents of the open pit excavation and the following design inflows:

- Groundwater inflows were determined by the Dupuit approximation equation, as described in Section 5 of the 2012 Open Pit Geotechnical Design Report (Knight Piésold Ltd., 2012c).
- Storm water inflows were estimated from the 1 in 10 year 24 hour return period storm event (53 mm).

- Average annual inflows from the Mean Annual Precipitation (MAP) of 1025 mm at EL.1680 m.

The pit dewatering system was designed to remove these inflows over a 10 day period at each stage of open pit development. The estimates of pump system energy were based on removal of groundwater inflows and average precipitation inflows only. The design flows for each stage of pit development are shown on Table 6.6.

Table 6.6 Open Pit Dewatering System Design Flows

PIPELINE SYSTEM		Catchment Area (m^2)	Groundwater Inflow (m^3/h)	Avg. Precip. Inflow (m^3/h)	Storm Event Inflow (m^3/h)	Design Flow (m^3/h)
PIT SYSTEM	Year 1	900,100	32	108	199	340
	Year 5	1,683,150	54	202	372	630
	Year 10	2,492,670	65	299	550	910
	Year 24	3,150,740	140	378	696	1,210

6.6.4 Pit Dewatering System Design

The pit dewatering system will progressively expand over the active mining period of the project. The system will consist of in-pit intake and booster pump stations, out-of-pit booster pump stations, and a pipeline to convey the flow from the Open Pit to the TMF.

The number of pumps and number of booster stations will increase throughout operations. The in-pit pump stations were designed with skid-mounted diesel drive pumps so they could be easily relocated within the pit. The system was designed so the out-of-pit pump stations would remain in a fixed location for the duration of the project. These booster stations will include electric drive pumps in permanent pump houses.

The in-pit pump stations will consist of one to three 300 hp pumps each capable of conveying a design flow of 400 m^3/hour, and will operate without an installed spare pump. The out-of-pit pump stations will include two to four 250 hp pumps each capable of conveying the same design flow of 400 m^3/hour. An installed spare was included at each permanent pump house.

The system was designed to pump a maximum design flow of 1200 m^3/hour (0.333 m^3/s) through a 16 inch HDPE DR13.5 pipeline from pit base to pit crest, and then 20 inch HDPE DR11 pipeline from pit crest to the TMF.

Material takeoffs for the development of the pit dewatering system showing each year of system expansion are included in Appendix F. A summary of the pump system configuration and energy requirements are included in Table 6.7.

Table 6.7 Open Pit Dewatering Pump System Configuration

Dewatering System		Pump Station Configuration	Number of Pump Stations	Total System Pump Energy (Diesel US Gal/yr)	Total System Pump Energy (MWhr/yr)
Base to Pit Crest	Year 1	Skid-mounted pumps, diesel drive, 300 hp Year 1: One pump + one standby Year 5: Two pumps Year 10: Three pumps Year 24: Three pumps Pump Model - Godwin HL225M	1	54,000	-
	Year 5		2	215,000	-
	Year 10		2	317,000	-
	Year 24		3	510,000	-
Pit Crest to TMF	Year 1	Permanent pump station installations, electric drive, 250 hp Year 1: One pump + one standby Year 5: Two pumps + one standby Year 10: Three pumps + one standby Year 24: Three pumps + one standby Pump Model - Pioneer SC86C21	2	-	1,019
	Year 5		2	-	1,927
	Year 10		2	-	2,869
	Year 24		2	-	4,340

6.7 MINE SITE WATER BALANCE

6.7.1 Introduction

A monthly operational and closure water balance was developed for the project using the GoldSim© software package. The model estimated the magnitude and extent of any water surplus and/or deficit conditions in the TMF based on a range of possible climatic conditions. The modelling timeline included:

- One year of pre-production (Year -1)
- 28 years of operations (Years 1 to 28) at a nominal milling rate of 70,000 dry metric tonnes per day, and
- 17 years of closure.

The model incorporated the following major project components:

- Open Pit
- Mill
- TMF

- PAG Waste Rock Stockpile stored within the TMF
- Non-PAG Waste Rock and Overburden Stockpiles, and
- Non-PAG and PAG Low-Grade Stockpiles.

The model input parameters and results of the water balance were described previously (Knight Piésold Ltd., 2014b). The referenced document is included as Appendix E and the results are summarized in the following sections.

6.7.2 Results

The TMF pond was predicted to be in a net surplus condition for the entire operating life of the mine, indicating the system, including the TMF and contributing mine site catchment areas, is able to supply more than enough water to meet the mill process water requirements. The potential variability of climate conditions over the project life was addressed by systematically varying climatic inputs to the water balance based on the 96 year historical precipitation and temperature record developed for the project (Knight Piésold Ltd., 2014a). The water balance model was run with 96 iterations for each year of simulated mine life, enabling a large number of combinations of wet, dry, and median months and years of precipitation and corresponding temperature values to be considered. Model outputs, in particular flow volumes, were then compiled as distributions for each month in each year from which probabilities of occurrence could be determined. The probabilities of occurrence presented in the water balance results represent the following conditions:

- Median scenario – 50% chance of the value being equaled or exceeded in any given month or year
- 95th percentile scenario – 5% chance that the water volume of flow rate will be equaled or exceeded in any given month or year (also referred to as the 95th percentile wet), and
- 5th percentile scenario – 95% chance that a water volume or flow will be equaled or exceeded in any given month or year (also referred to as the 95th percentile dry).

The predicted TMF pond volumes for the three scenarios described above are shown on Figure 6.10. The results of the monthly water balance model indicate that:

- The TMF pond was predicted to be in a surplus condition throughout operations and is able to supply all the process water required to support mill processing from Years 1 to 25.5. As of Year 25.5, when LGO is processed through the mill, the open pit was able to supply all the process water required for the mill to the end of operations in Year 28.
- The TMF pond ranges from a minimum of 12 Mm^3 at start-up to a maximum of 196 Mm^3 at the end of operations, under median conditions.
- The mine site provides sufficient water for reuse in the process such that additional make-up water will not be required, as envisaged in the early stages of project development. Fresh non-potable water required for the mine process should be sourced from treated TMF reclaim water.

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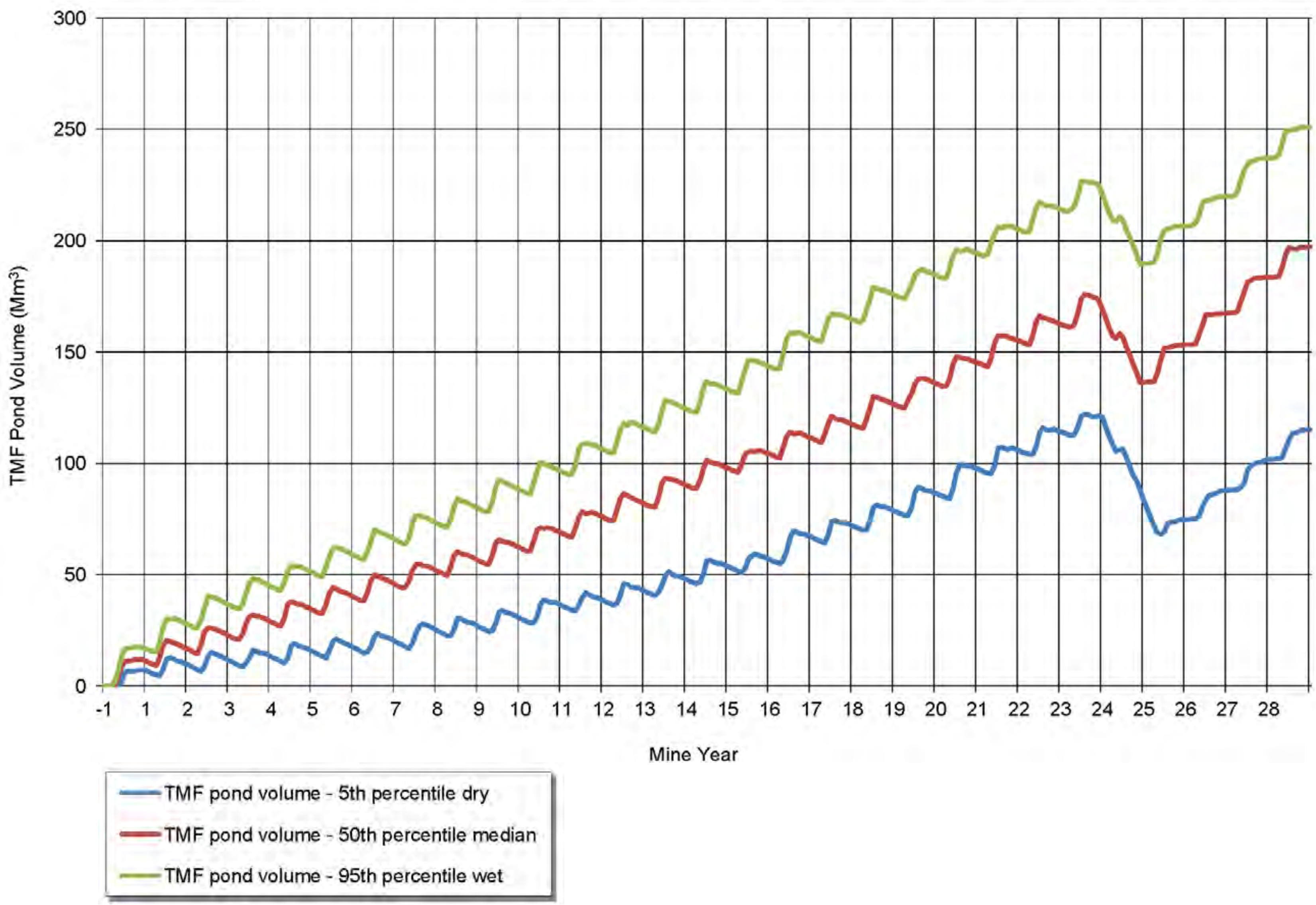


Figure 6.10 Predicted TMF Pond Volumes

7 – CLOSURE AND RECLAMATION

7.1 GENERAL

The primary objective of the closure and reclamation initiatives will be to eventually return the TMF to a self-sustaining facility that satisfies the end land use objectives. The TMF is designed to maintain long-term stability, protect the downstream environment, and manage surface water. Activities that will be carried out during operations and at closure to achieve these objectives are discussed in the following sections.

7.2 PROGRESSIVE RECLAMATION

The project design allows for substantial reclamation activities to occur during the final five years of operations (reclamation of embankment and stockpiles, as an example), leaving only the LGO footprints and infrastructure to be reclaimed in the years following closure.

Closure and reclamation activities will commence about five years into mining operations. The activities have been split into concurrent reclamation (Years 5 to 28) and final reclamation (Years 29 to 33). A general description of reclamation activities that will occur in each phase are as follows:

Concurrent reclamation activities:

- Non-PAG LGO stockpile (small stockpile) – apply soil cover and revegetation
- Overburden Stockpile footprints – apply soil cover and revegetation
- Non-PAG Waste Rock Stockpile – apply overburden cap, soil cover and revegetation
- TMF Embankments – apply overburden cap, soil cover and revegetation
- Tailings Beaches – apply soil cover and revegetation
- Tailings Beaches – construct wetlands at TMF pond margins
- TMF – construct spillway on eastern abutment of main embankment

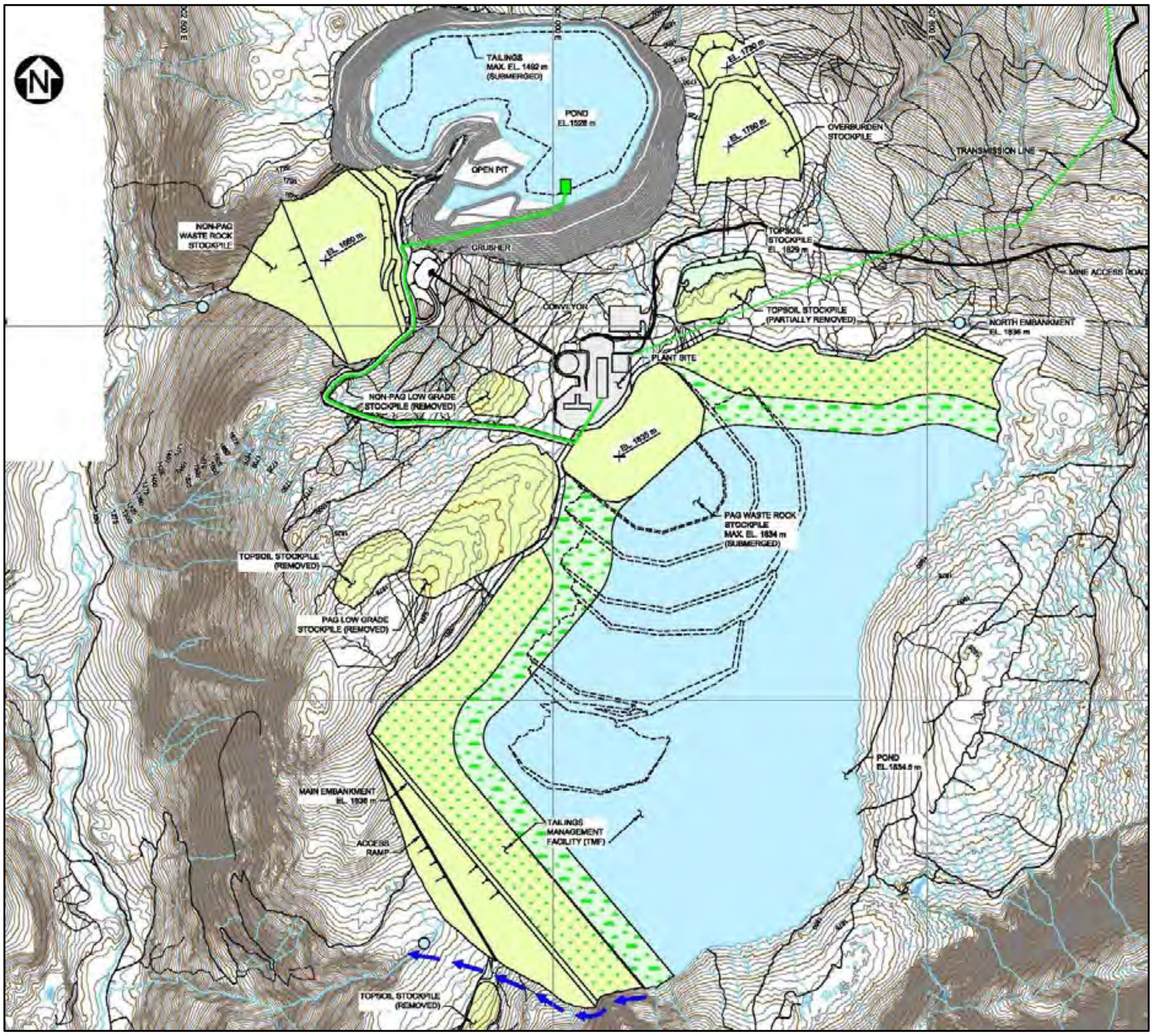
7.3 DECOMMISSIONING AND CLOSURE

Upon mine closure, surface facilities will be removed in stages and full reclamation of the TSF will be initiated. The closure plan is compatible with a premature closure event. General aspects of the final reclamation activities include:

- Topsoil stockpiles – remove and use to apply soil cover to project facilities
- PAG LGO stockpile footprint – remove subgrade and place in TMF, apply soil cover and revegetation
- Non-PAG LGO stockpile footprint – apply soil cover from PAG LGO subgrade material and revegetation
- LGO Water Management Ponds – decommission, remove, and revegetation
- Crusher, Conveyor and Plant Site – remove structures, apply soil cover and revegetation
- Crusher Pad – apply overburden cap, soil cover and revegetation
- Pipelines and Pump Stations – remove mechanical equipment, apply soil cover and revegetation
- Open Pit – construct emergency spillway on northern edge (lowest point of pit rim)
- TMF Water Management Ponds – decommission, remove, and revegetation
- Roads – decommission major haul roads and maintain sufficient road for light vehicle access

The waste rock stockpiles and embankments will have a cap applied using material from the overburden stockpiles, to facilitate water storage and release, and limit infiltration through the underlying materials. A soil cover of approximately 300 mm will be applied and revegetated with native species. Some areas will be reforested with the same species as existed prior to mine development. The plant site, crusher and conveyor will have a soil cover applied and then revegetated, once all structures have been dismantled and removed from site. Access roads will be reclaimed, unless they are required for long-term access to the site. An illustration of the general arrangement of the project in Year 28 is shown on Figure 7.1.

Figure 7.1 General Arrangement Year 28



Excess water from the TMF will be released through the spillway on the east abutment once all tailings deposition is complete (after Year 28) and the TMF pond has reached the spillway invert. At this time, water from TMF water management pond will also be released if water quality is suitable for release to the downstream receiving environment. The TMF spillway will release water to T-Creek, a tributary of Harper Creek. Once the pit has reached an elevation between 1530 m, excess water will be pumped and released to the TMF and subsequently flow through the TMF spillway to the downstream receiving environment. The lowest elevation of the pit wall is expected to be elevation 1555 m, which allows for over 25 m of freeboard to manage storm inflows.

8 – MATERIAL TAKEOFFS

8.1 GENERAL

The quantities (material takeoffs) for developing the initial and sustaining capital cost estimate and operating costs were derived from the drawings of this feasibility design report for the mine waste facilities and water management systems for the Project. Material takeoffs (MTOs) generated for the project include site preparation, heavy civil construction, piping, mechanical equipment, and operating power. The MTOs were transmitted to Merit to derive unit costs, and to compile the initial capital, sustaining capital and operational costs. The final MTOs are provided in Appendix F.

The costs for the development of these facilities have been compiled and combined in a separate Technical Report for the Feasibility Study. The final report included the detailed cost estimate was completed by Merit in July 2014 (Merit et al., 2014).

8.2 DRAWING PACKAGES

MTOs were derived from the feasibility design drawings where sufficient detail was available. Quantities provided were "neat-line" measured or calculated quantities with no allowance for design growth included. Allowances were provided based on recent experience where the level of design detail was not sufficient for measurement or calculation of quantities. The feasibility design drawing package is included as Appendix A. The drawing package includes the following:

- General Arrangement Drawings (9 Drawings)
- Tailings Embankment Construction (5 Drawings)
- Mine Water Management (6 Drawings)

8.3 DEVELOPMENT OF MATERIAL TAKEOFFS

8.3.1 Site Preparation and Heavy Civil Works

Site preparation and civil earthworks MTOs were provided as neat, in place quantities, with no allowance for swell, compaction, and waste. Mass earthworks quantities were generated from a 3D model of the works using LiDAR topography as the original ground. The backfill materials and levels of compaction were specified in the designs drawings. Civil works and structural quantities for building foundations and steel superstructures were based on lump sum allowances for specified plan area building sizes.

8.3.2 Piping

Overland piping for the water management systems were measured from drawings and provided as neat-line quantities without allowance for design growth, waste, and snaking. The MTOs were provided as lengths by type of service with material, grade and diameter specified.

8.3.3 Mechanical

Major mechanical equipment (pumps and motors) were specified in the MTOs. Budgetary quotations were solicited from reliable vendors for all pumps and motors, and provided to Merit for consideration during preparation of the cost estimates.

9 – SUMMARY AND RECOMMENDATIONS

The project has undergone a series of design changes since the last feasibility study (Knight Piésold Ltd., 2012d) to both optimize the mine site footprint and general arrangement of the project, and to reduce and mitigate the potential environmental impacts resulting from the development of the project. These design changes include the following significant modifications:

- earlier saturation of the PAG waste rock stockpile within the TMF
- modification of the TMF embankment cross section to improve constructability, reduce seepage potential, and improve long-term access to downstream monitoring features
- relocation and reconfiguration of the non-PAG waste rock stockpile, overburden, and topsoil stockpiles, and
- separation of low-grade ore by geochemical classification, and relocation of stockpiles to reduce the potential for unrecoverable seepage.

Additional geotechnical site investigations should be completed during the permitting phase to support detailed design of the project. Site investigation effort should focus on confirming the foundation conditions for the waste rock stockpiles and low-grade ore stockpiles, and the extent of glaciolacustrine deposits within the TMF basin and the footprint of the PAG waste rock stockpile.

The arrangement of the project infrastructure has been optimized to reduce the potential for environmental impacts by collection and storage of surface water and groundwater in contact with mine facilities for the duration of operations, and discharge of water from the TMF pond in closure. Predictions of changes to streamflow were completed (Knight Piésold Ltd., 2014a) using this water management strategy and corresponding predictions of water quality were also produced (Knight Piésold Ltd., 2014c).

The project area was predicted to generate a net surplus of water for the entire operating life of the mine. Fresh water diversions were designed in order to divert as much fresh water as practical. Diversion of runoff by gravity from much of the undisturbed catchment area reporting to the TMF pond was impractical as a long-term solution due to the shape of the TMF catchment. The TMF pond was predicted to accumulate nearly 200 Mm^3 by the end of mine life prior to TMF discharge, with an annual water surplus in excess of 5 Mm^3 per year.

The storage of all surplus water in the TMF pond is a primary driver for TMF embankment construction, and requires adequate tailings distribution to maintain the minimum required tailings beaches adjacent to the embankments. This increases the complexity of the TMF design and monitoring requirements. Reducing surplus water would have a beneficial impact on the project.

It is recommended that a conceptual surplus water management plan be developed to investigate options for removal of surplus fresh water and operational discharge of mine water to support permitting of the project. The following concepts should be considered:

- Staged capture of undisturbed runoff from the TMF and pumped diversion to T-creek during operations
- Operational discharge from the Non-PAG waste rock stockpile water management pond, and
- Operational discharge from the TMF supernatant pond and water management ponds.

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11 – CERTIFICATION

This report was prepared, reviewed and approved by the undersigned.

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
Sept 26, 2014
D. D. FONTAINE
36208

Prepared: ______________________
Daniel Fontaine, P.Eng.
Senior Engineer

Reviewed: ______________________
Bruno Borntraeger, P.Eng.
Specialist Engineer

Approved: ______________________
Ken Brouwer, P.Eng.
President

APPENDIX A

DESIGN DRAWINGS

(Pages A-1 to A-21)

TABLE A.1

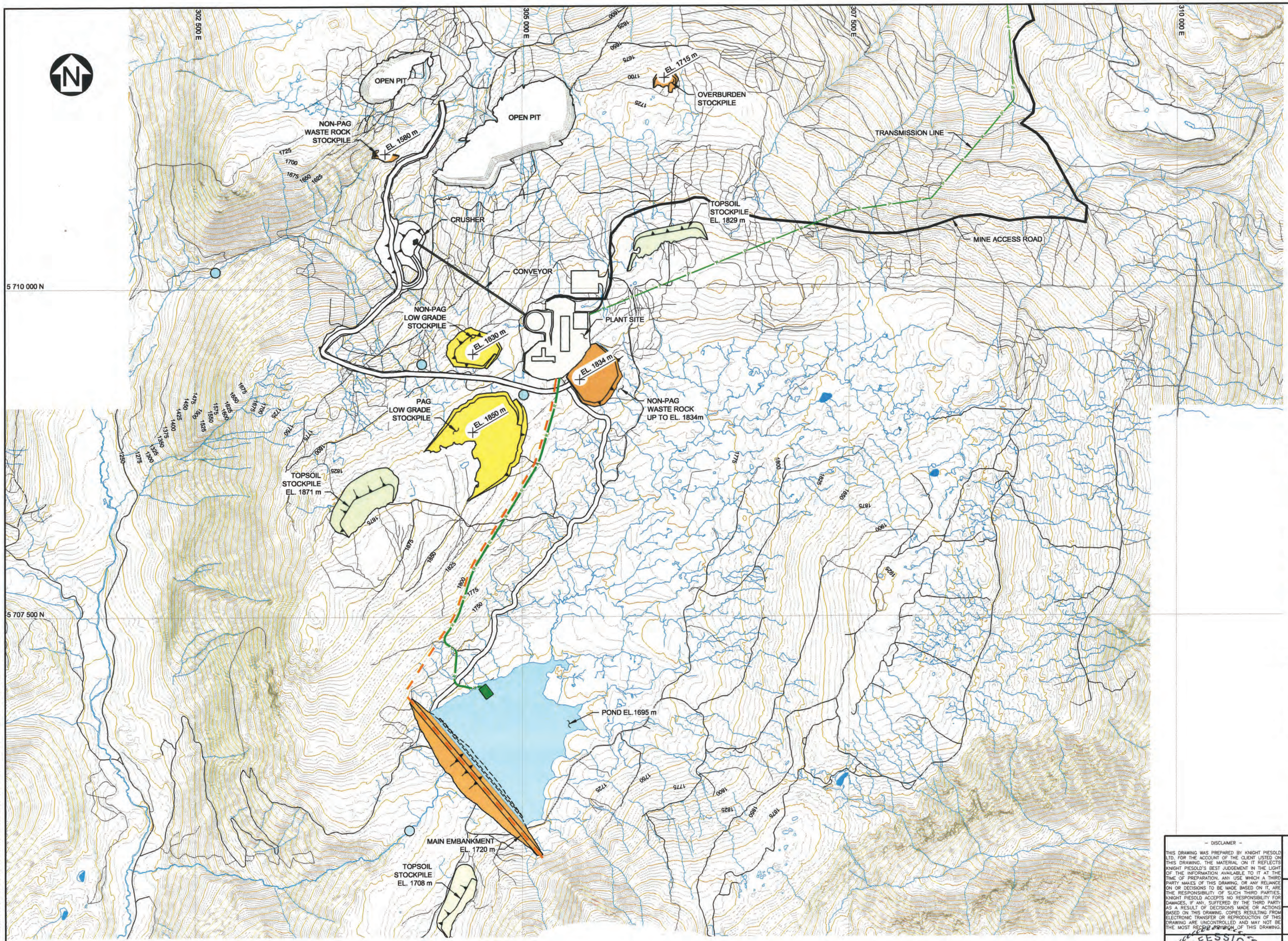
HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT DESIGN REPORT

DRAWING LIST

KP DWG No.	Rev.	Revision Date	Package	Drawing Title Title
C0010	0	26SEP'14	General Arrangements	General Arrangement - Year -1
C0011	0	26SEP'14	General Arrangements	General Arrangement - Year 1
C0013	0	26SEP'14	General Arrangements	General Arrangement - Year 3
C0015	0	26SEP'14	General Arrangements	General Arrangement - Year 5
C0017	0	26SEP'14	General Arrangements	General Arrangement - Year 10
C0019	0	26SEP'14	General Arrangements	General Arrangement - Year 15
C0021	0	26SEP'14	General Arrangements	General Arrangement - Year 20
C0023	0	26SEP'14	General Arrangements	General Arrangement - Year 24
C0025	0	26SEP'14	General Arrangements	General Arrangement - Year 28 - End Of Mine
C0030	0	26SEP'14	TMF	Tailings Management Facility - Main Embankment - Typical Cross Section and Detail
C0031	0	26SEP'14	TMF	Tailings Management Facility - North Embankment - Typical Cross Section
C0035	0	26SEP'14	TMF	Construction Material Specifications
C0040	0	26SEP'14	TMF	Tailings Management Facility - Stage 1 Site Preparation - Plan and Sections
C0045	0	26SEP'14	TMF	Tailings Management Facility - Stage 1 Embankment Construction - Plan and Sections
C0050	0	26SEP'14	Water Management	Water Management - TMF Embankment - Phased Development
C0051	0	26SEP'14	Water Management	Water Management - TMF Embankment - Water Management Pond - Plan and Section
C0055	0	26SEP'14	Water Management	Water Management - Non-PAG Waste Rock Stockpile - Phased Development
C0056	0	26SEP'14	Water Management	Water Management - Non-PAG Waste Rock Stockpile - Water Management Pond - Plan and Section
C0060	0	26SEP'14	Water Management	Water Management - Low-Grade Ore Stockpile - Phased Development
C0065	0	26SEP'14	Water Management	Water Management - Overburden Stockpile - Phased Development



LEGEND:

- MINE WATER
- FRESH WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK / OVERBURDEN
- PAG WASTE ROCK
- LOW-GRADE ORE
- TOPSOIL
- RECLAIM SYSTEM
- WATER MANAGEMENT POND AND PUMPSTATION
- MINE ACCESS ROAD
- EXISTING ACCESS TRAILS
- TRANSMISSION LINE
- RECLAIM WATER PIPELINE
- TAILINGS DELIVERY PIPELINE

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

NOT FOR CONSTRUCTION

SCALE A 300 150 0 500 1000 1500 m

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PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA
D. D. FONTAINE
36208
Sept 25, 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT YEAR -1

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0010	0

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
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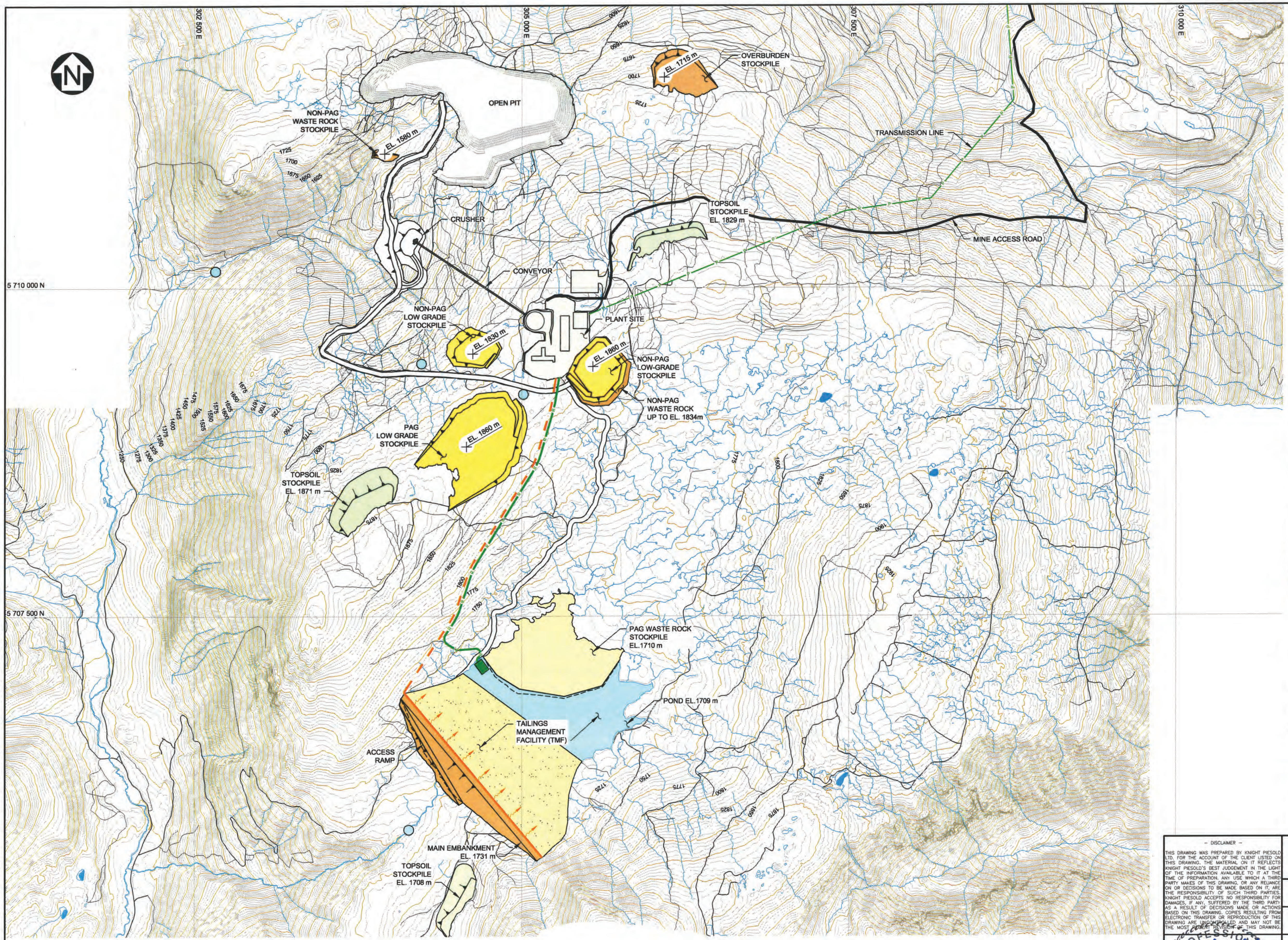

LEGEND:

- TAILINGS BEACH
- MINE WATER
- FRESH WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK / OVERBURDEN
- PAG WASTE ROCK
- LOW-GRADE ORE
- TOPSOIL
- RECLAIM SYSTEM
- WATER MANAGEMENT POND AND PUMPSTATION
- MINE ACCESS ROAD
- EXISTING ACCESS TRAILS
- TRANSMISSION LINE
- RECLAIM WATER PIPELINE
- TAILINGS DELIVERY PIPELINE
- TAILINGS DISCHARGE PIPELINE

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NOT FOR CONSTRUCTION

SCALE A 300 150 0 500 1000 1500 m

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PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA
D. D. FONTAINE
36208
Sept 25, 2014

Knight Piésold
CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT YEAR 1

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0011	0

REFERENCE DRAWINGS

DRG. NO.	DESCRIPTION
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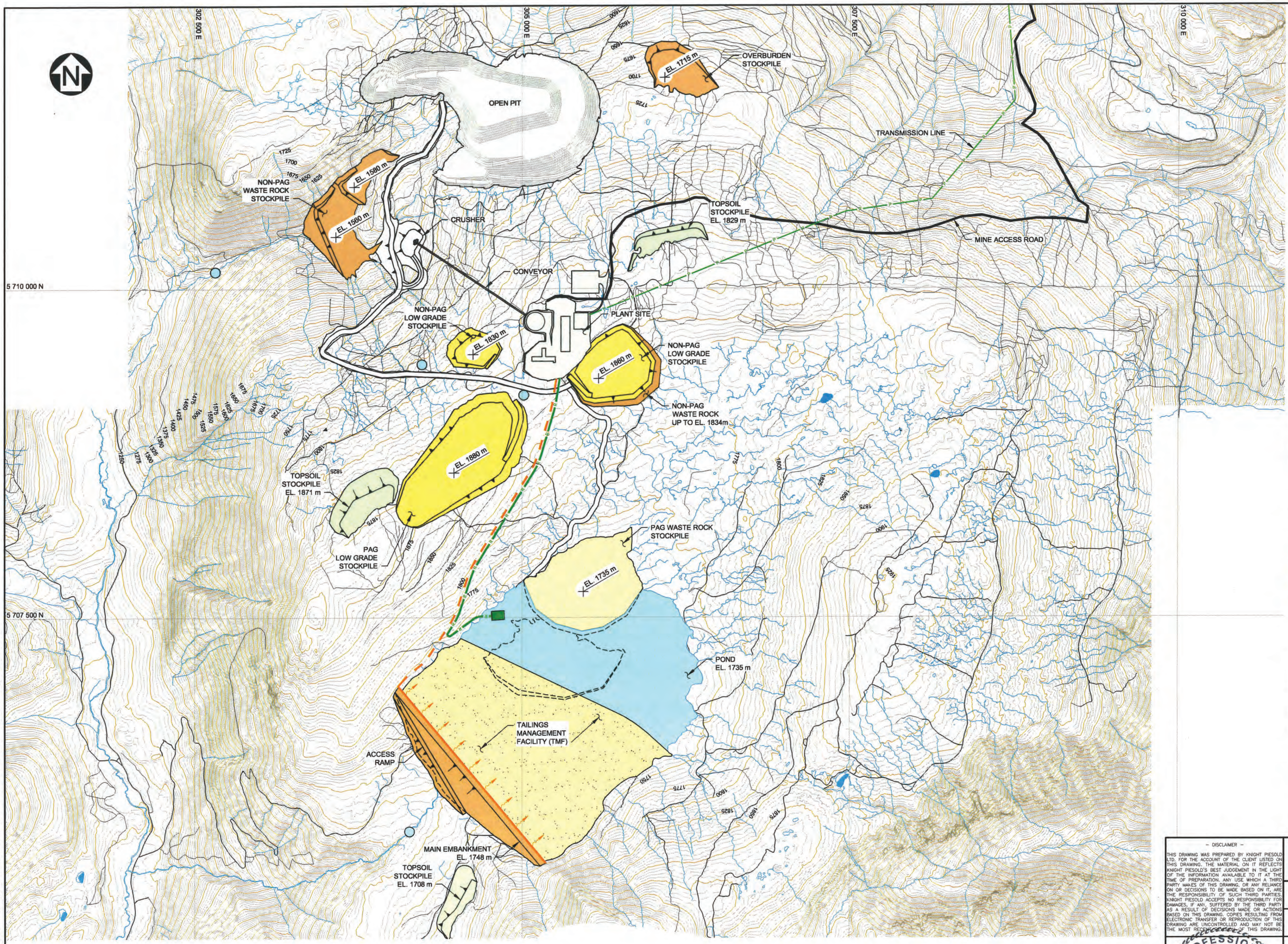

LEGEND:

- TAILINGS BEACH
- MINE WATER
- FRESH WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK / OVERBURDEN
- PAG WASTE ROCK
- LOW-GRADE ORE
- TOPSOIL
- RECLAIM SYSTEM
- WATER MANAGEMENT POND AND PUMPSTATION
- MINE ACCESS ROAD
- EXISTING ACCESS TRAILS
- TRANSMISSION LINE
- RECLAIM WATER PIPELINE
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- TAILINGS DISCHARGE PIPELINE

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SCALE A 300 150 0 500 1000 1500 m

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PROFESSIONAL ENGINEER, PROVINCE OF BRITISH COLUMBIA, D. D. FONTAINE # 36208

Sept 25 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT YEAR 3

P/A NO.	DRAWING NO.	REVISION
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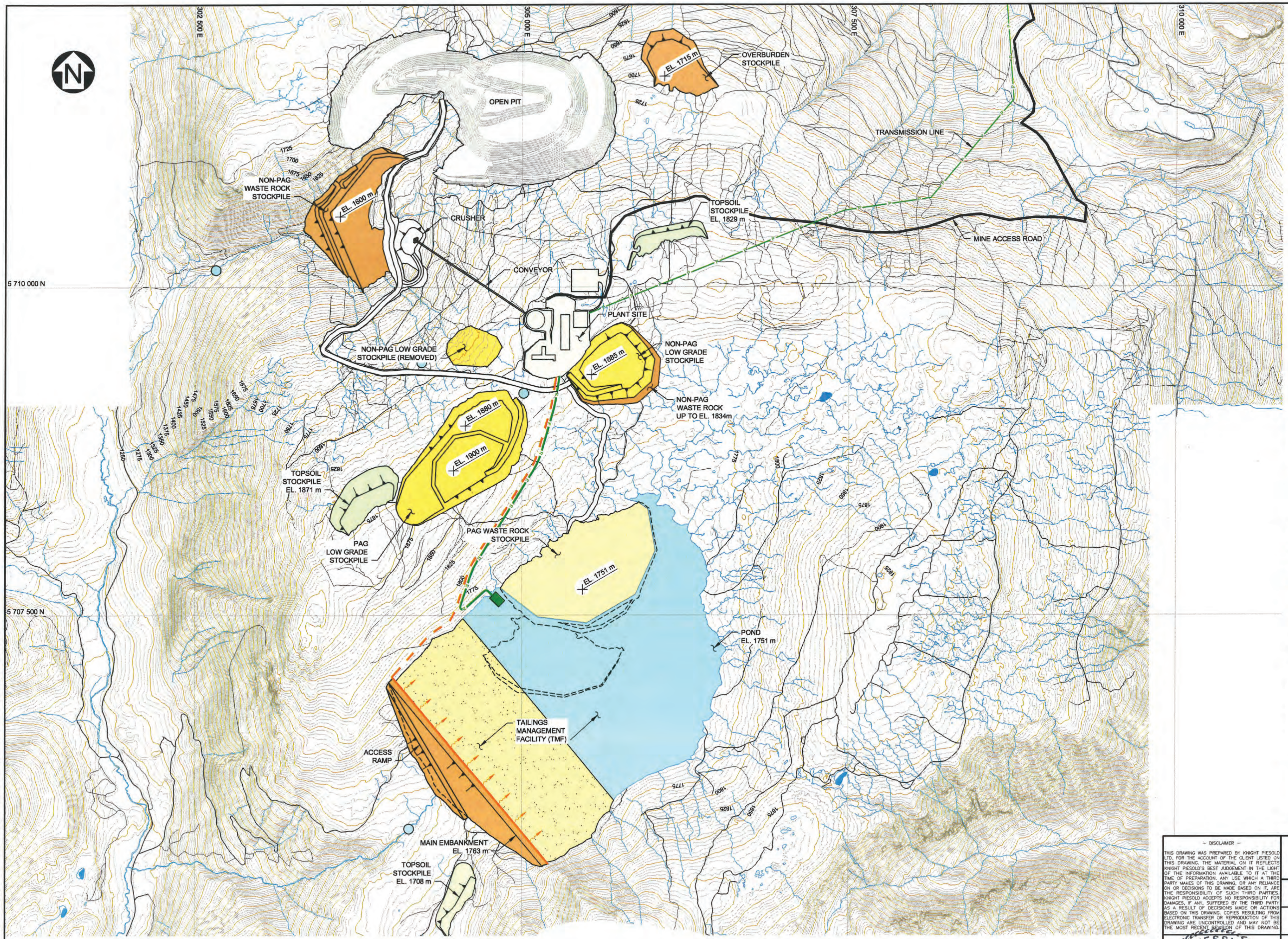

LEGEND:

- TAILINGS BEACH
- MINE WATER
- FRESH WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK / OVERBURDEN
- PAG WASTE ROCK
- LOW-GRADE ORE
- TOPSOIL
- UPLAND SLOPE
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SCALE A 300 150 0 500 1000 1500 m

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PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
D. D. FONTAINE
36208
Sept 25 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT YEAR 5

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0015	0

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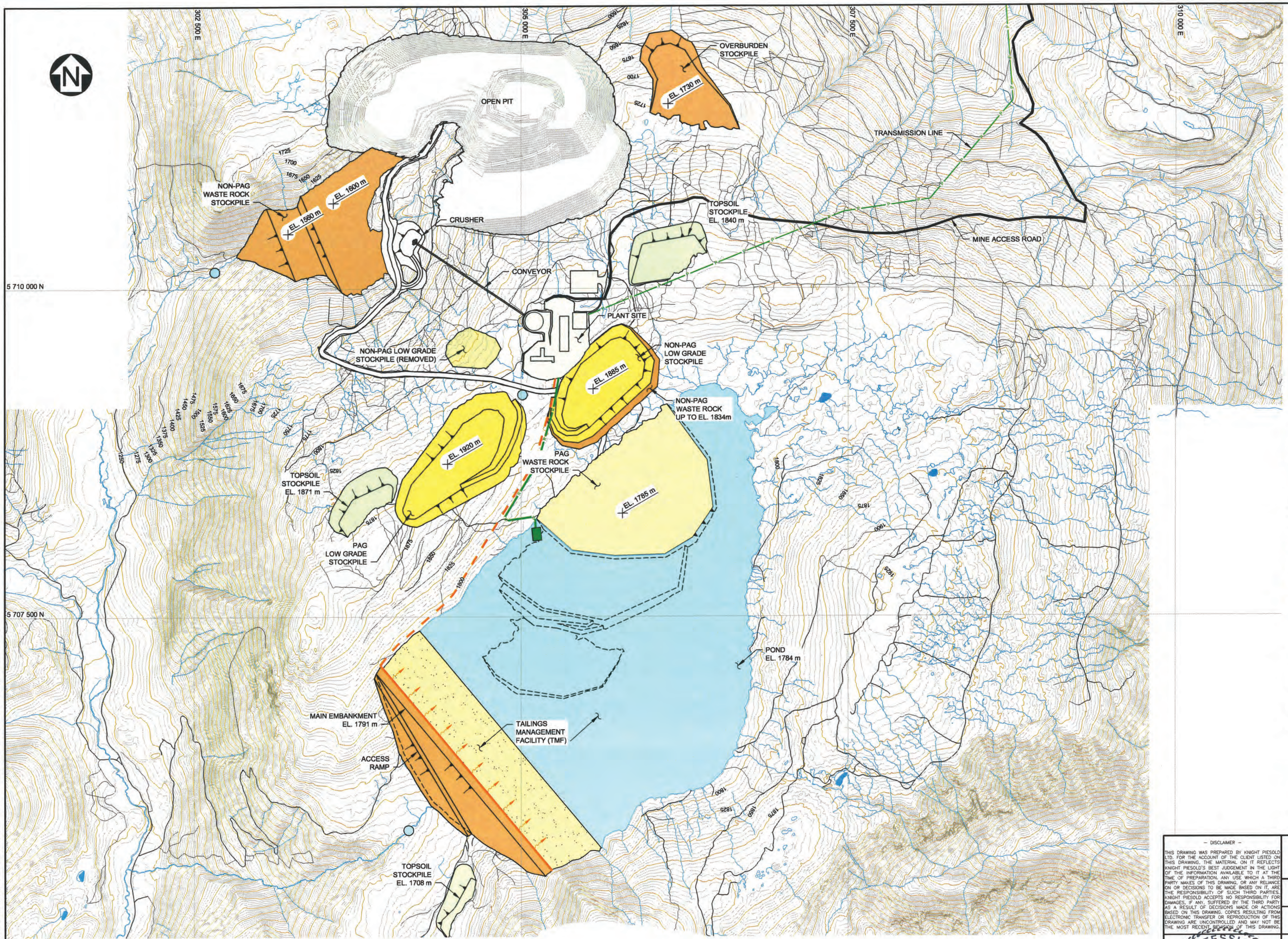

LEGEND:

- TAILINGS BEACH
- MINE WATER
- FRESH WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK / OVERBURDEN
- PAG WASTE ROCK
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300 150 0 500 1000 1500 m
SCALE A

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PROFESSIONAL ENGINEER
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36208
Sept 25 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT YEAR 10

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0017	0

DRG. NO.	DESCRIPTION
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REVISIONS

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REVISIONS

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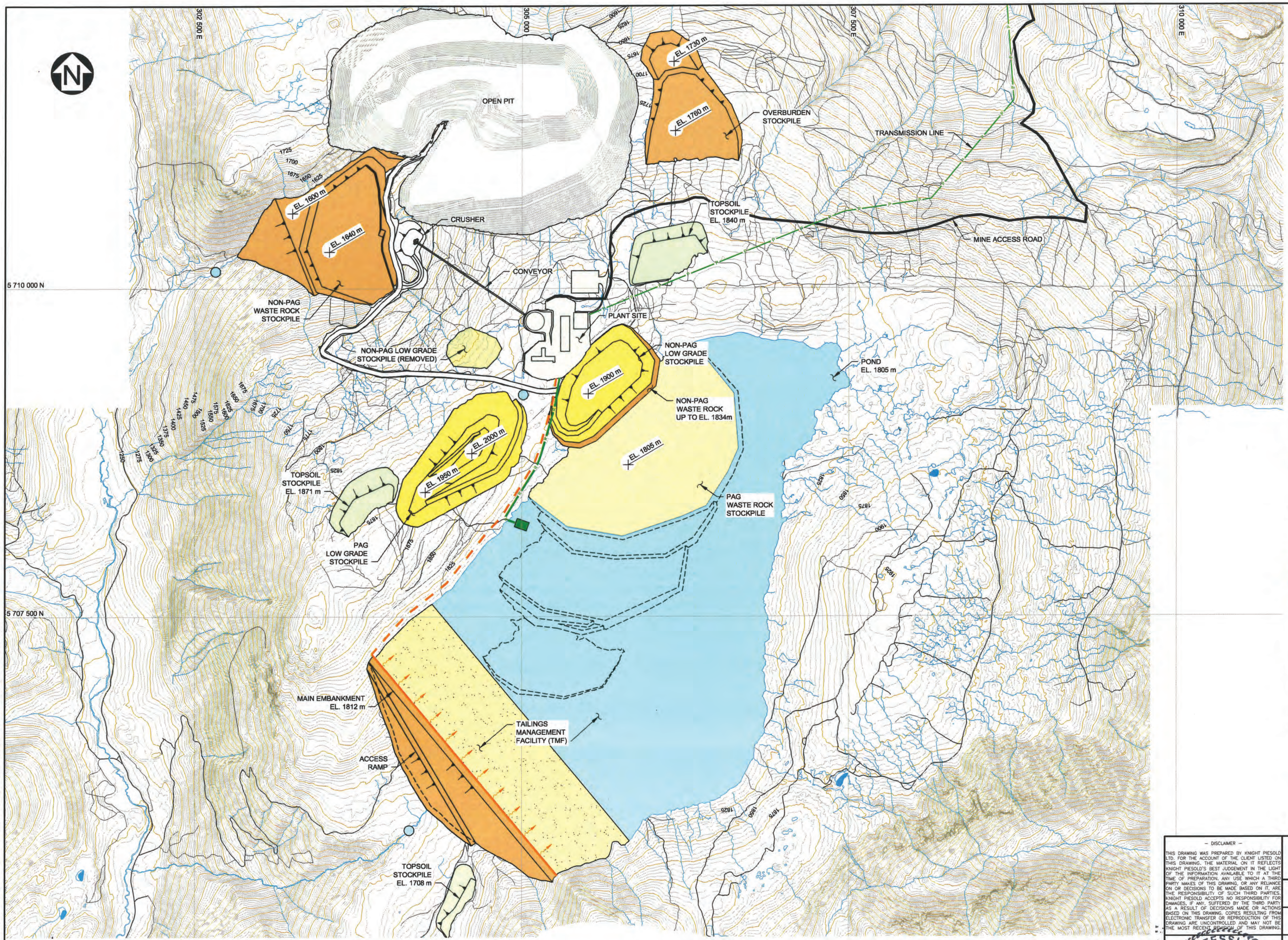

LEGEND:

- TAILINGS BEACH
- MINE WATER
- FRESH WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK / OVERBURDEN
- PAG WASTE ROCK
- LOW-GRADE ORE
- TOPSOIL
- UPLAND SLOPE
- RECLAIM SYSTEM
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NOT FOR CONSTRUCTION

300 150 0 500 1000 1500 m
SCALE A

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PROFESSIONAL ENGINEER — PROVINCE OF BRITISH COLUMBIA — D. D. FONTAINE # 36208 — Sept 25, 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT
YEAR 15

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0019	0

DRG. NO.	DESCRIPTION
-	-

REFERENCE DRAWINGS

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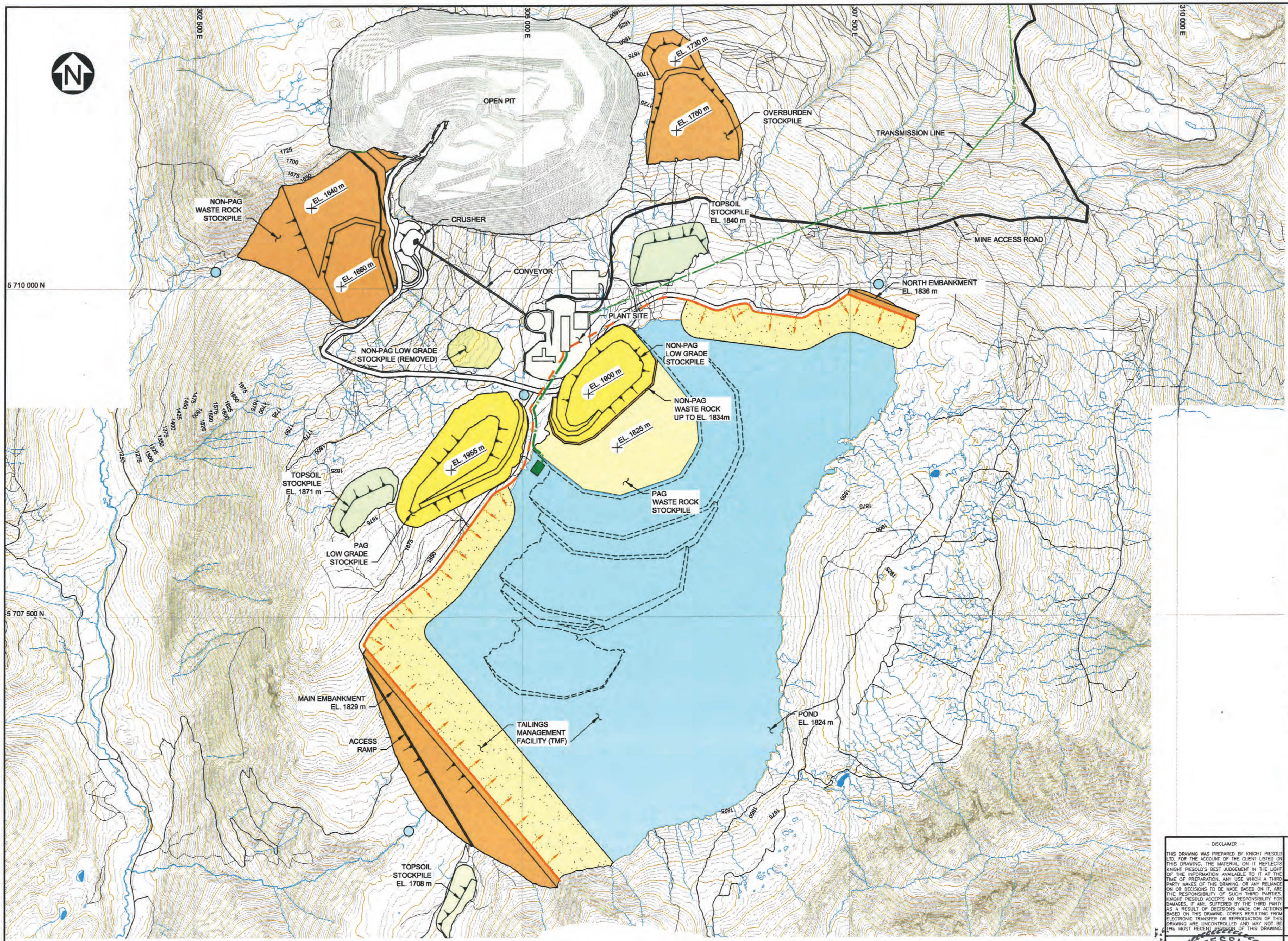

LEGEND:

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- MINE WATER
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PROFESSIONAL ENGINEER D. D. FONTAINE # 36208 BRITISH COLUMBIA Sept 25 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT YEAR 20

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0021	0

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REVISIONS

DRG. NO.	DESCRIPTION
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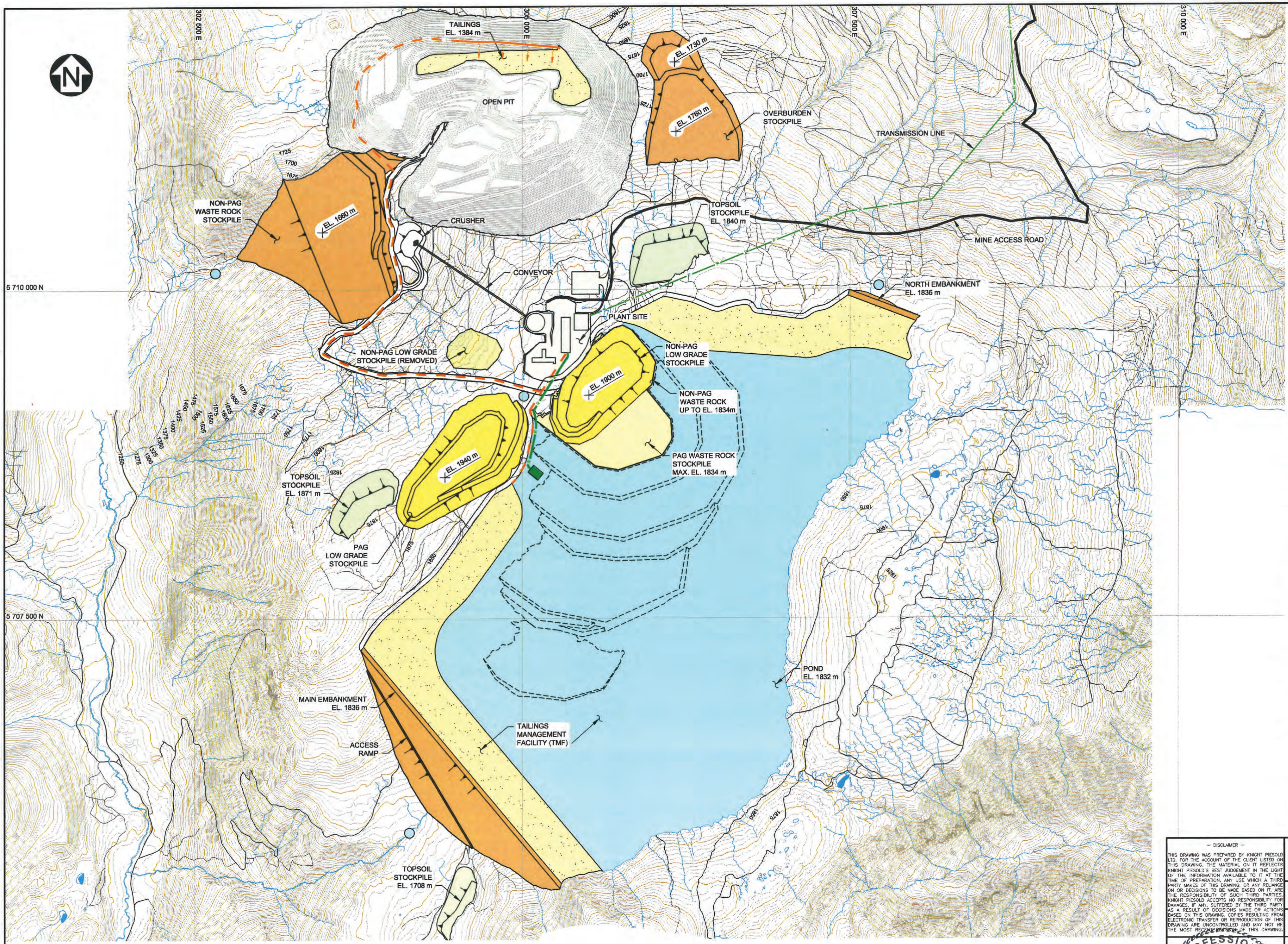

LEGEND:

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- MINE WATER
- FRESH WATER
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- PAG WASTE ROCK
- LOW-GRADE ORE
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SCALE A — 300 150 0 500 1000 1500 m

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PROFESSIONAL ENGINEER — D. D. FONTAINE # 36208 — PROVINCE OF BRITISH COLUMBIA — Sept 24 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT YEAR 24

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0023	0

DRG. NO.	DESCRIPTION
-	-

REFERENCE DRAWINGS

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D

REVISIONS

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	ABN	PB	KJB

REVISIONS

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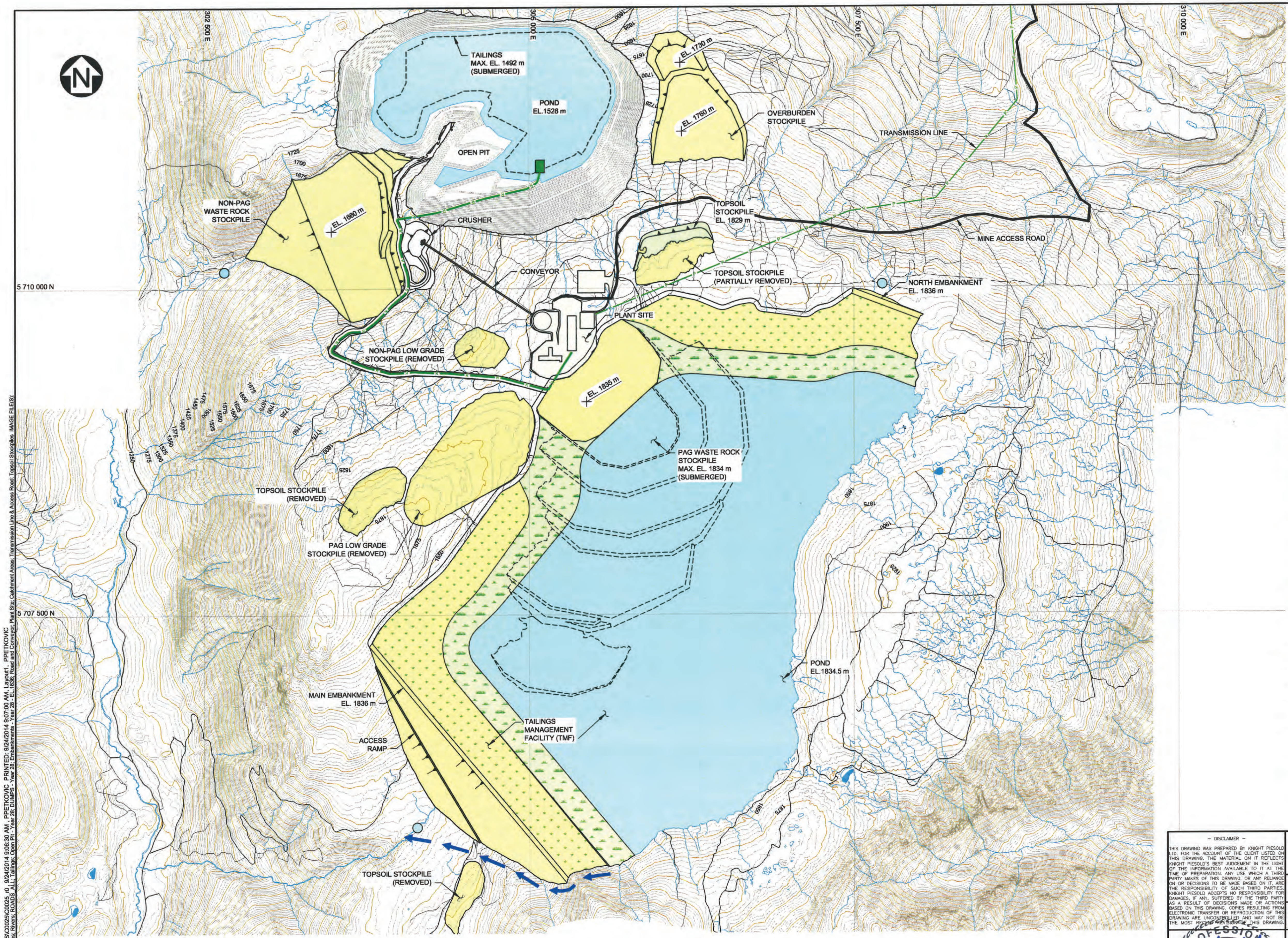

LEGEND:

- UPLAND BEACH
- UPLAND SLOPE
- BOG / WETLAND AREA
- EMERGENT VEGETATION WETLAND
- MINE WATER
- FRESH WATER
- TOPSOIL
- RECLAIM SYSTEM
- WATER MANAGEMENT POND
- MINE ACCESS ROAD
- EXISTING ACCESS TRAILS
- SPILLWAY
- TRANSMISSION LINE
- RECLAIM WATER PIPELINE

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

NOT FOR CONSTRUCTION

SCALE A 300 150 0 500 1000 1500 m

– DISCLAIMER –

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PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
D. D. FONTAINE
36208
Sept 26, 2014

Knight Piésold
CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

GENERAL ARRANGEMENT
YEAR 28 - END OF MINE

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0025	0

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	ABN	BB	KJB

REVISIONS

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D

REVISIONS

DRG. NO.	DESCRIPTION
-	-

REFERENCE DRAWINGS

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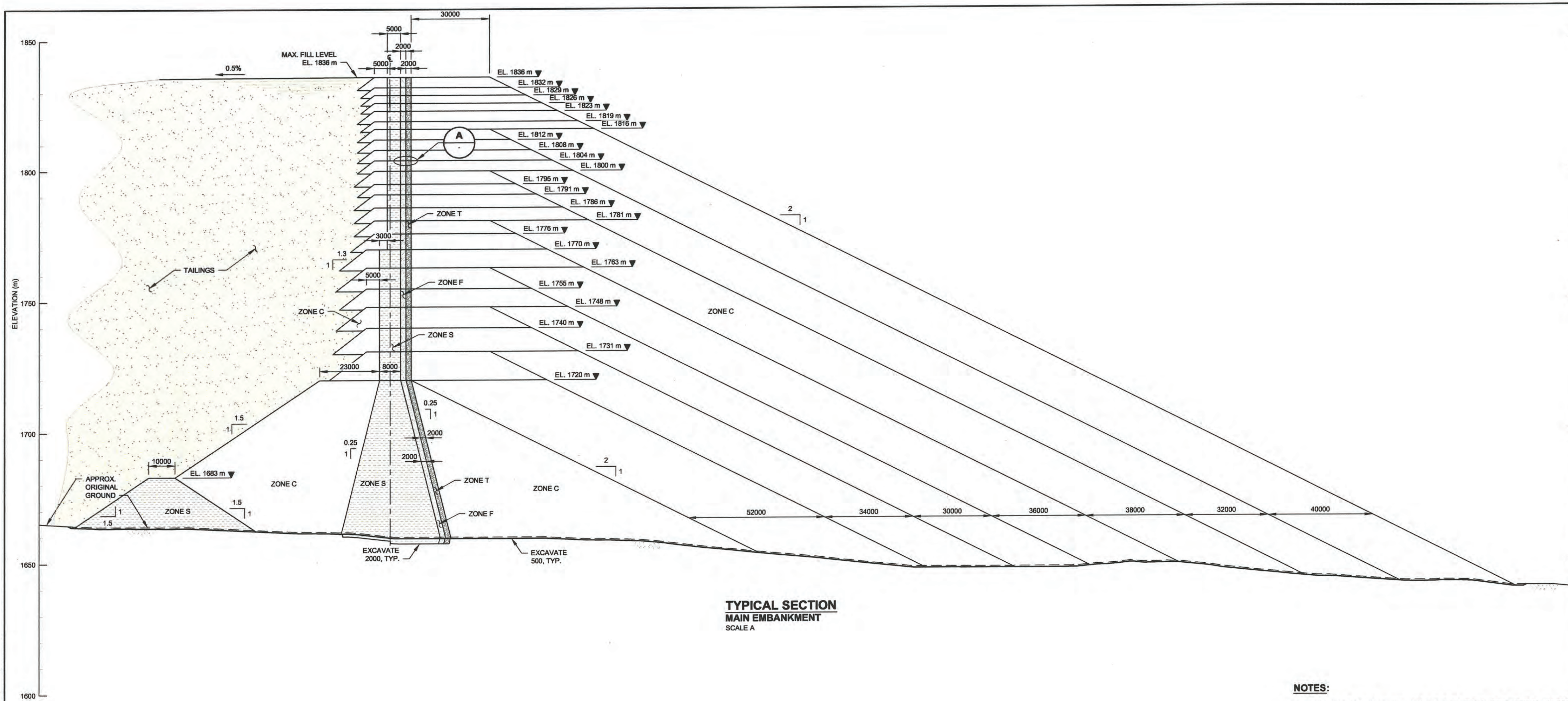
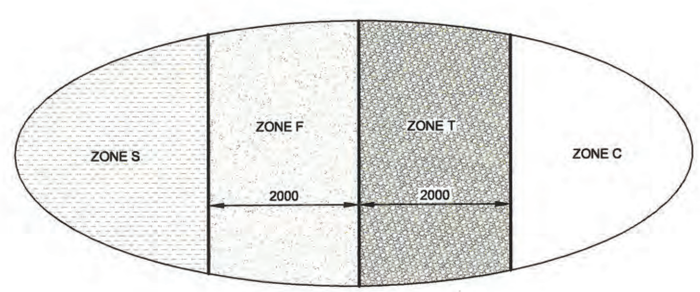


TYPICAL SECTION
MAIN EMBANKMENT
SCALE A



A DETAIL
SCALE B

LEGEND:
- ZONE S (CORE ZONE)
- ZONE F (FILTER)
- ZONE T (TRANSITION)
- ZONE C
- TAILINGS

NOTES:
1. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

NOT FOR CONSTRUCTION

SCALE B 1 0.5 0 1 2 3 4 5 m
SCALE A 15 7.5 0 25 50 75 m

PROFESSIONAL ENGINEER
D. D. FONTAINE
36208
BRITISH COLUMBIA
Sept 25 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
MAIN EMBANKMENT
TYPICAL CROSS SECTION AND DETAIL

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0030	0

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	ABN	BB	KJB

REVISIONS

DRG. NO.	DESCRIPTION
-	-

REFERENCE DRAWINGS

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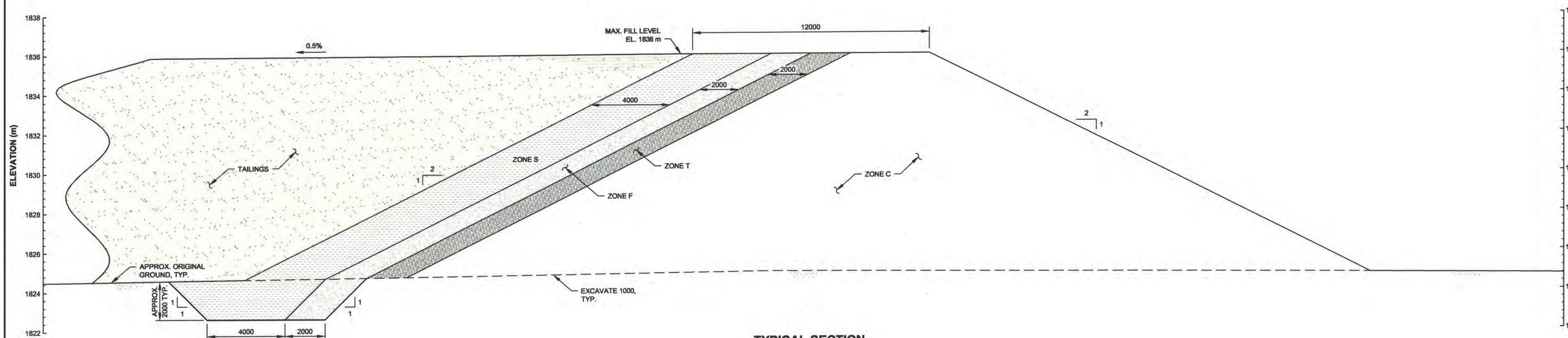

TYPICAL SECTION
NORTH EMBANKMENT
SCALE A

LEGEND:

- ZONE S (CORE ZONE)
- ZONE F (FILTER)
- ZONE T (TRANSITION)
- ZONE C
- TAILINGS

NOTES:

1. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

NOT FOR CONSTRUCTION

SCALE A 2 1 0 2 4 6 8 10 m

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PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
D. D. FONTAIN
36208
Sept 25, 2014

Knight Piésold
CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
NORTH EMBANKMENT
TYPICAL CROSS SECTION

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0031	0

REFERENCE DRAWINGS

DRG. NO.	DESCRIPTION
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REVISIONS

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D

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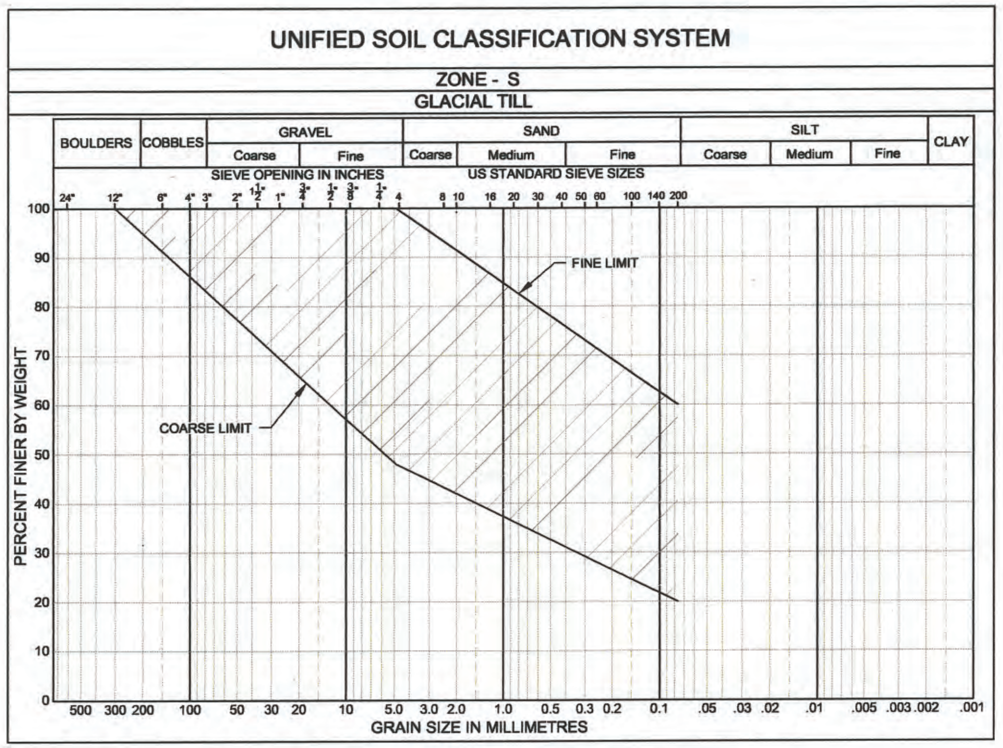

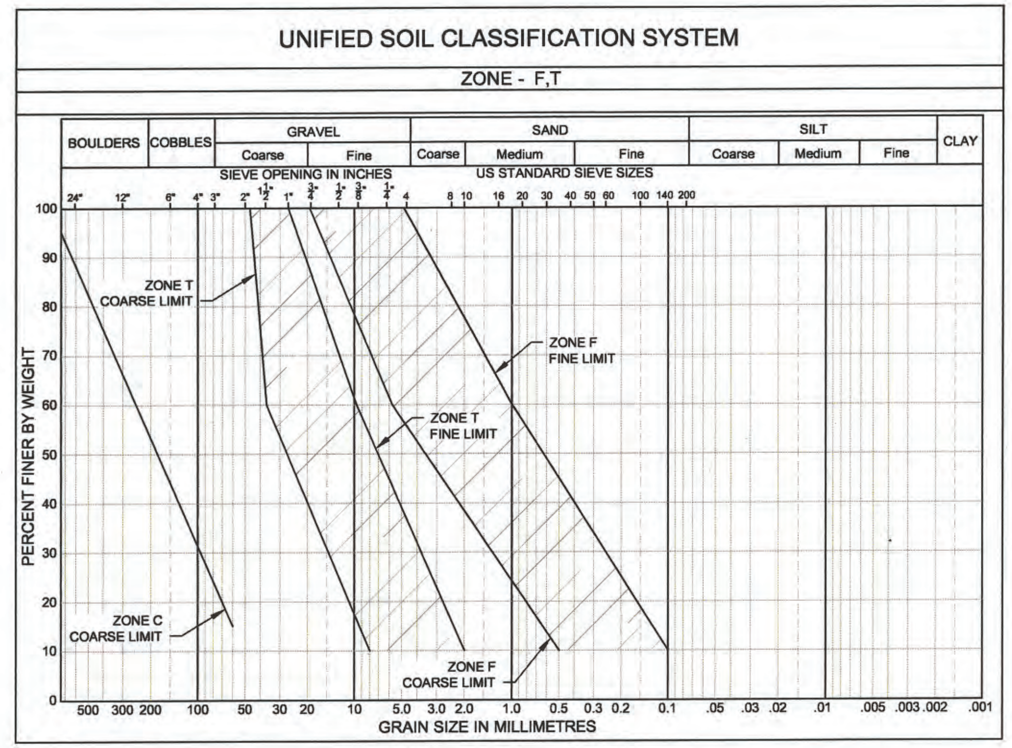

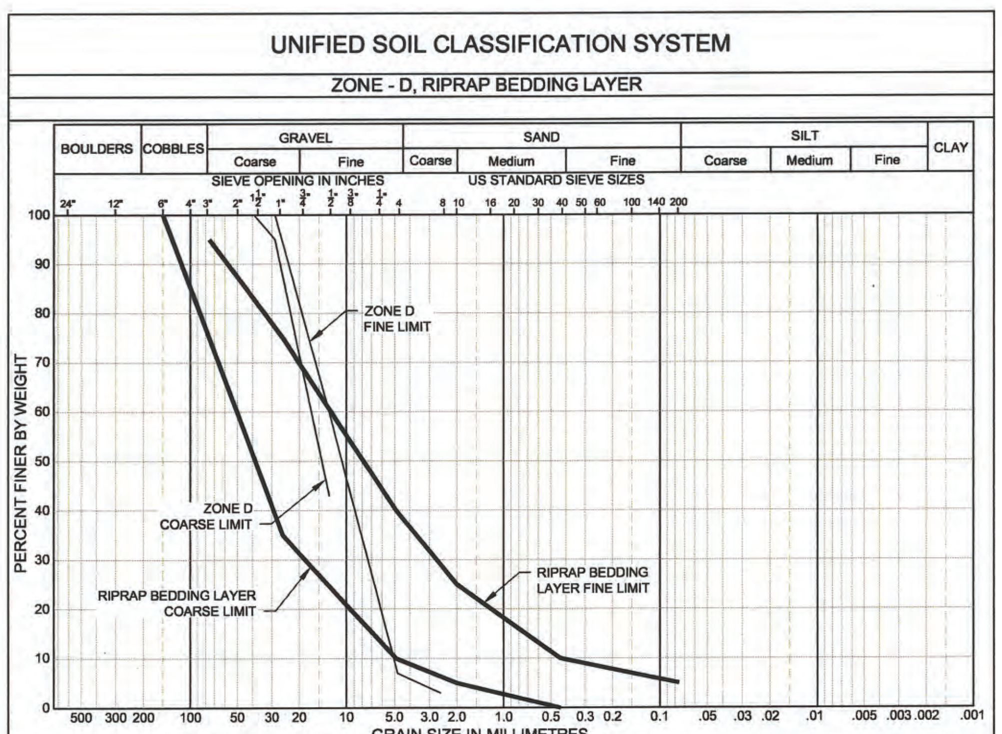

MATERIAL PLACEMENT AND COMPACTION REQUIREMENTS

ZONE	MATERIAL TYPE	LOCATIONS	PLACING AND COMPACTION REQUIREMENTS
S	GLACIAL TILL	CORE ZONE	PLACED, MOISTURE CONDITIONED AND SPREAD IN MAXIMUM 300 MM THICK LAYERS (AFTER COMPACTION). VIBRATORY COMPACTION TO 95% OF STANDARD PROCTOR MAXIMUM DRY DENSITY OR AS APPROVED BY THE ENGINEER
F	FILTER SAND	CHIMNEY DRAIN	PLACED AND SPREAD IN MAXIMUM 600 MM THICK LAYERS AND COMPACTED WITH MINIMUM 4 TO 6 PASSES OF 10 TON SMOOTH DRUM VIBRATORY ROLLER, OR AS APPROVED BY THE ENGINEER
T	GRAVEL	TRANSITION ZONE	PLACED AND SPREAD IN MAXIMUM 600 MM THICK LAYERS AND COMPACTED WITH MINIMUM 4 TO 6 PASSES OF 10 TON SMOOTH DRUM VIBRATORY ROLLER, OR AS APPROVED BY THE ENGINEER.
C	WASTE ROCK OVER BURDEN	SHELL ZONE	CONTRACTOR FLEET TO PLACE AND SPREAD IN MAXIMUM 1000 mm THICK LAYERS. MINING FLEET TO PLACE AND SPREAD IN MAXIMUM 2000 mm THICK LAYERS. UNIFORMLY COMPACTED BY SELECTIVE ROUTING OF HAUL TRUCK TRAFFIC ON MAIN FILL AND BY VIBRATORY ROLLER ON THE FILL EDGES.
D	DRAINAGE GRAVEL	DRAINS	PLACED AROUND DRAINAGE PIPES AND WRAPPED WITH GEOTEXTILE.

NOTES:

1. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.
2. RIPRAP TO BE HARD, DENSE AND DURABLE TO WITHSTAND LONG EXPOSURE TO WEATHERING.
3. RIPRAP SIZE TO MEET OR EXCEED THE SIZE DIMENSIONS SPECIFIED ON THE ROCK INTERMEDIATE DIMENSION (SECONDARY AXIS).
4. RIPRAP STONES SHALL BE ANGULAR IN SHAPE. NO STONE SHALL EXCEED A LENGTH TO BREADTH OR THICKNESS OF 3.
5. WEARING COURSE SHALL BE FREE OF ALL ORGANIC MATTER, AND SOFT FRIABLE PARTICLES, EACH LAYER SHALL BE COMPACTED BY A MINIMUM OF FOUR PASSES OVER THE ENTIRE SURFACE WITH THE SPECIFIED STEEL DRUM VIBRATING ROLLER. WEARING COURSE MATERIAL SHALL CONFORM TO THE FOLLOWING GRADATION:

SIEVE SIZE	PERCENT PASSING
75 MM	100
19 MM	60 - 100
9.5 MM	40 - 83
4.75 MM	20 - 50
2 MM	12 - 30
0.075 MM	5 - 15

RIPRAP REQUIREMENTS

RIPRAP DATA *		APPROXIMATE % OF TOTAL MASS SMALLER THAN GIVEN			
		RIPRAP TYPE			
MASS	SIZE	1	2	3	4
(KG)	(MM)				
2600	1000				100
900	750				50
450	600			100	30
180	450		100	50	
55	300		50	30	10
22	225		30		
7	150			10	
2	100	100	10		
1	75	50			
	50	30			
	25	10			

* MASS TO APPROXIMATE SIZE CONVERSION BASED ON A SPECIFIC GRAVITY OF 2.6 AND A VOLUME AVERAGE BETWEEN A SPHERE AND CUBE.

NOT FOR CONSTRUCTION

PROFESSIONAL ENGINEER
D. D. FONTAINE
36208
Sept 24 2014

Knight Piésold CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

CONSTRUCTION MATERIAL SPECIFICATIONS

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0035	0

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	ABN	BB	KJB

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DRG. NO.	DESCRIPTION
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REFERENCE DRAWINGS

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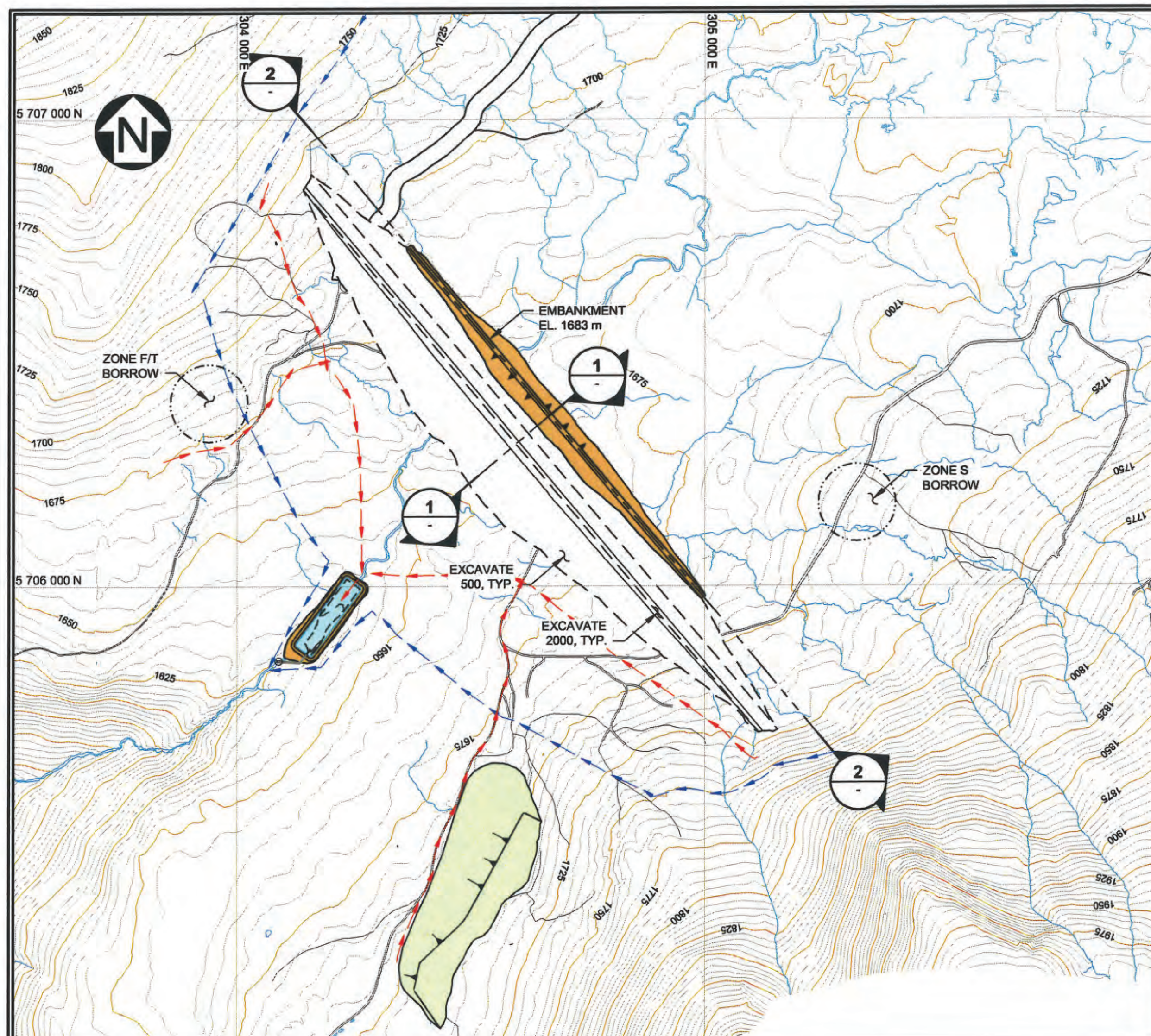
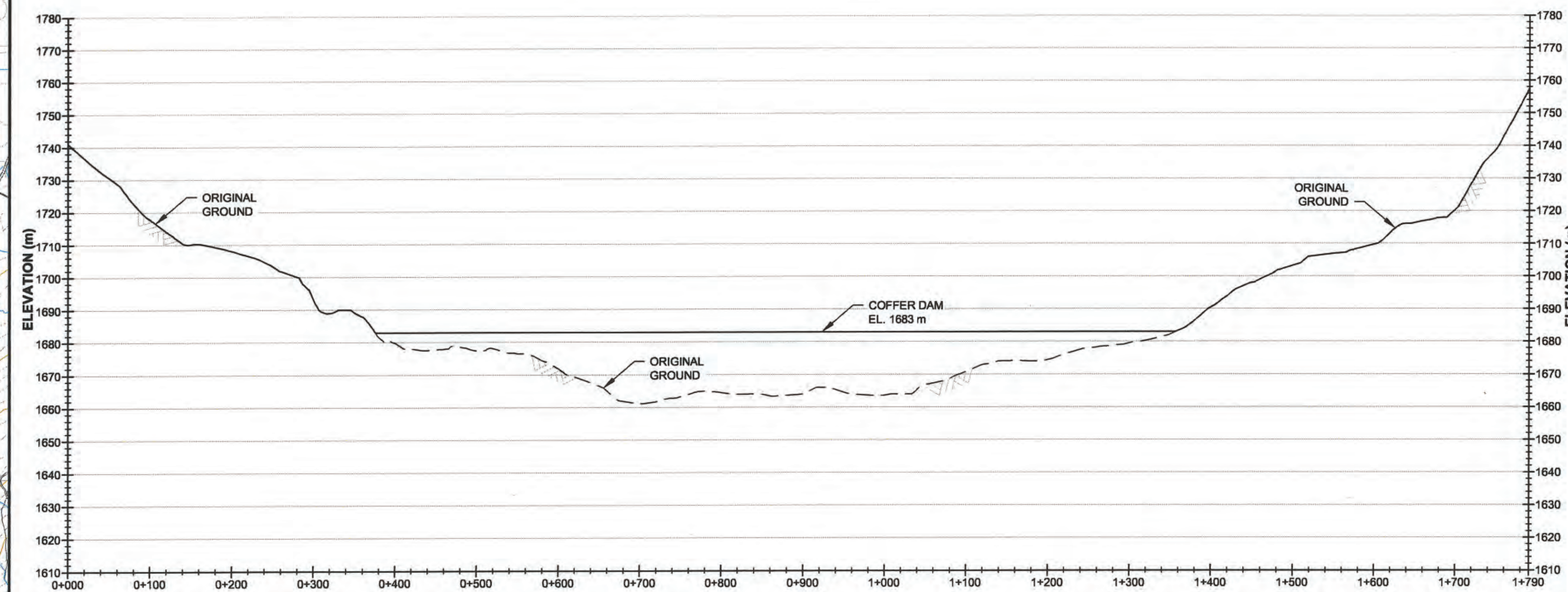


PLAN
STAGE 1 COFFER DAM EL. 1683 m
SCALE A



2 SECTION
STAGE 1 COFFER DAM PROFILE
HORIZONTAL: SCALE C
VERTICAL: SCALE B

LEGEND:

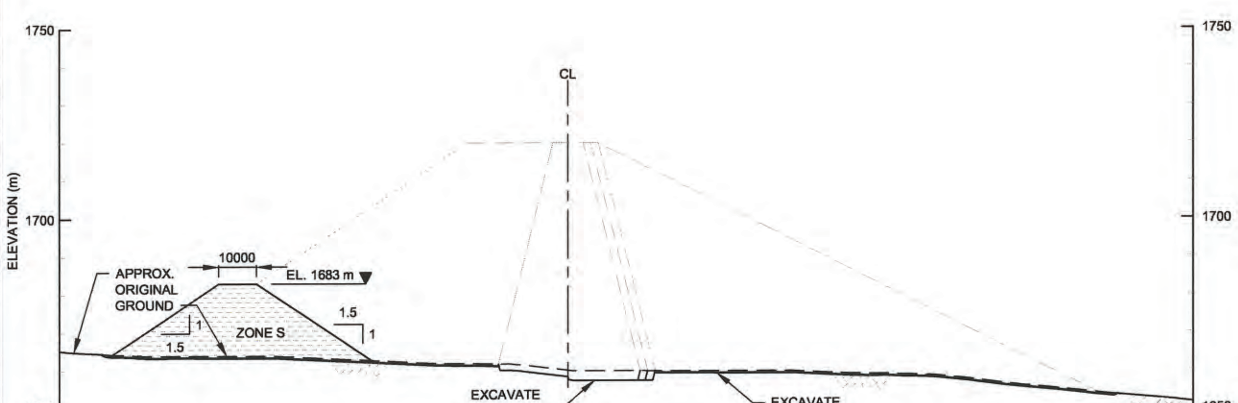
- EMBANKMENT FILL
- TOPSOIL
- ZONE S (CORE ZONE)
- DIVERSION CHANNEL
- COLLECTION CHANNEL

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

NOT FOR CONSTRUCTION

SCALE C 80 40 0 100 200 300 400 m
SCALE B 20 10 0 20 40 60 80 100 m
SCALE A 150 75 0 250 500 750 m



1 SECTION
COFFER DAM
SCALE B

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PROFESSIONAL ENGINEER — PROVINCE OF BRITISH COLUMBIA
D. D. FONTAINE
36208
Sept 25, 2014

Knight Piésold
CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
STAGE 1 SITE PREPARATION
PLAN AND SECTIONS

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0040	0

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	ABN	BB	KJB

REVISIONS

DRG. NO.	DESCRIPTION
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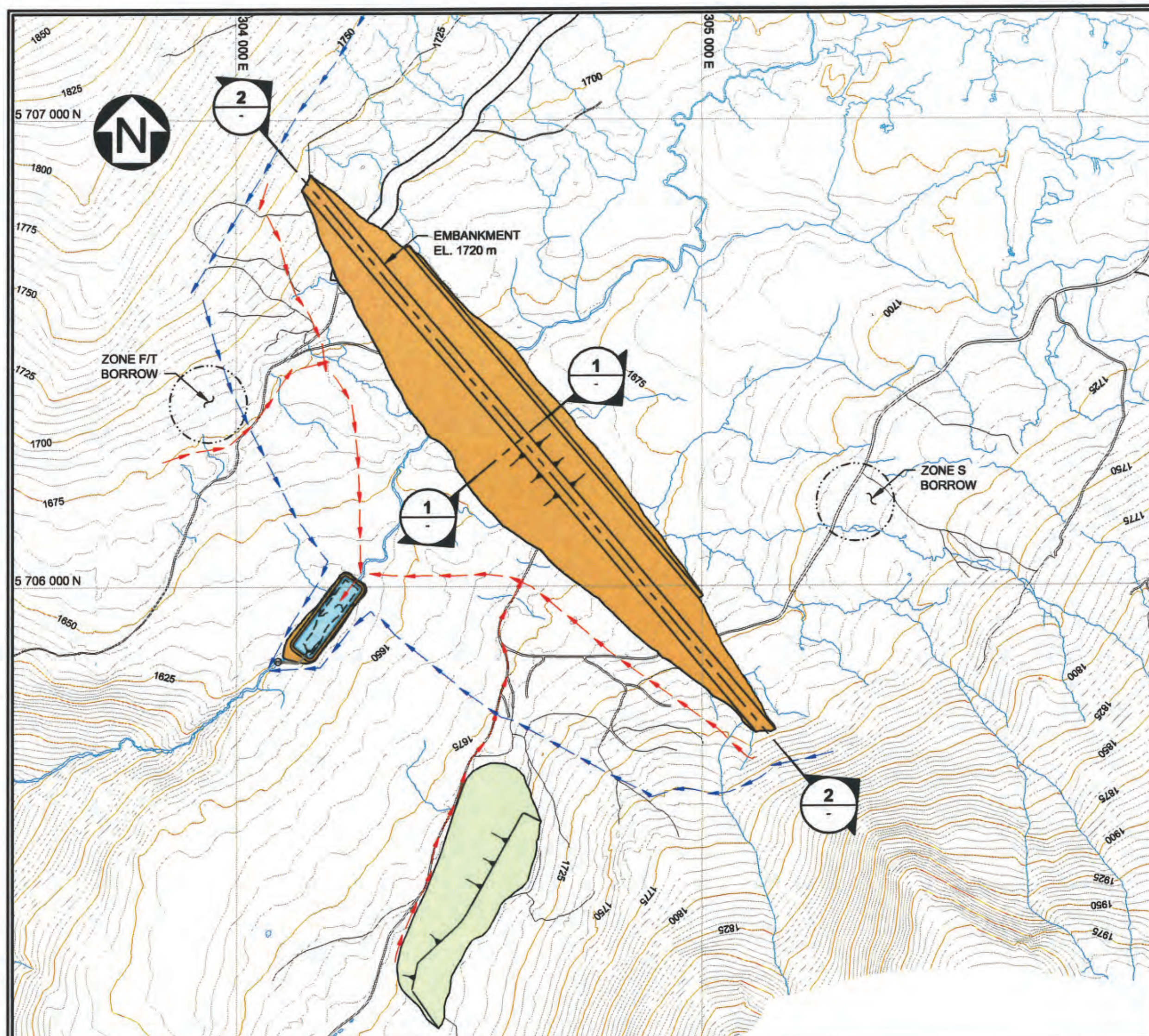
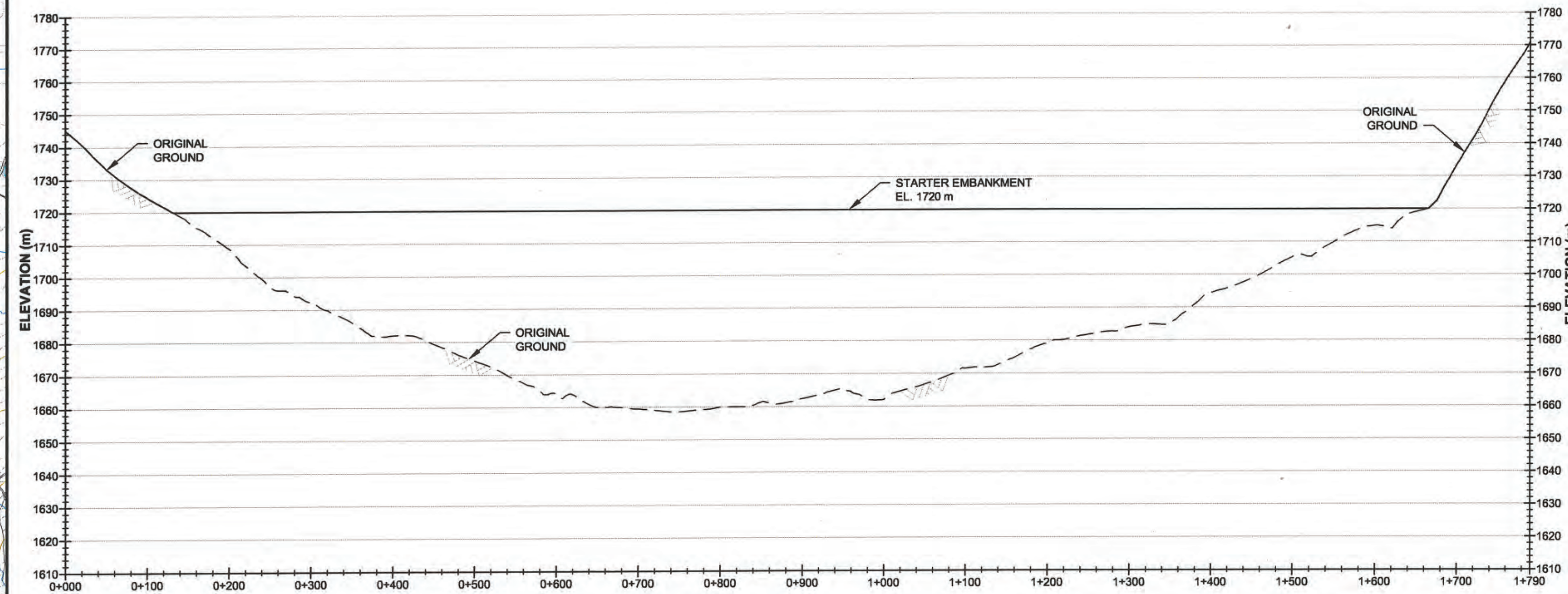


PLAN
STAGE 1 EMBANKMENT EL. 1720 m
SCALE A



2 **SECTION**
STAGE 1 EMBANKMENT PROFILE
HORIZONTAL: SCALE C
VERTICAL: SCALE B

LEGEND:

- EMBANKMENT FILL
- TOPSOIL
- ZONE S (CORE ZONE)
- ZONE F (FILTER)
- ZONE T (TRANSITION)
- DIVERSION CHANNEL
- COLLECTION CHANNEL

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
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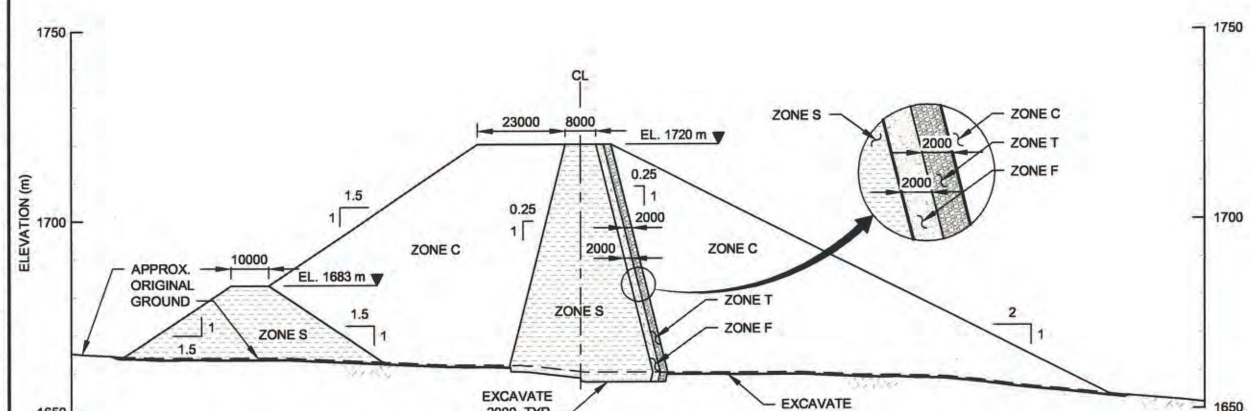
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

NOT FOR CONSTRUCTION

SCALE C 80 40 0 100 200 300 400 m
SCALE B 20 10 0 20 40 60 80 100 m
SCALE A 150 75 0 250 500 750 m



1 **SECTION**
STAGE 1 EMBANKMENT
SCALE B

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DRG. NO.	DESCRIPTION
-	-

REFERENCE DRAWINGS

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D

REVISIONS

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	ABN	AB	KJB

REVISIONS

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PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
D. D. FONTAINE
36208
Sept 25, 2014

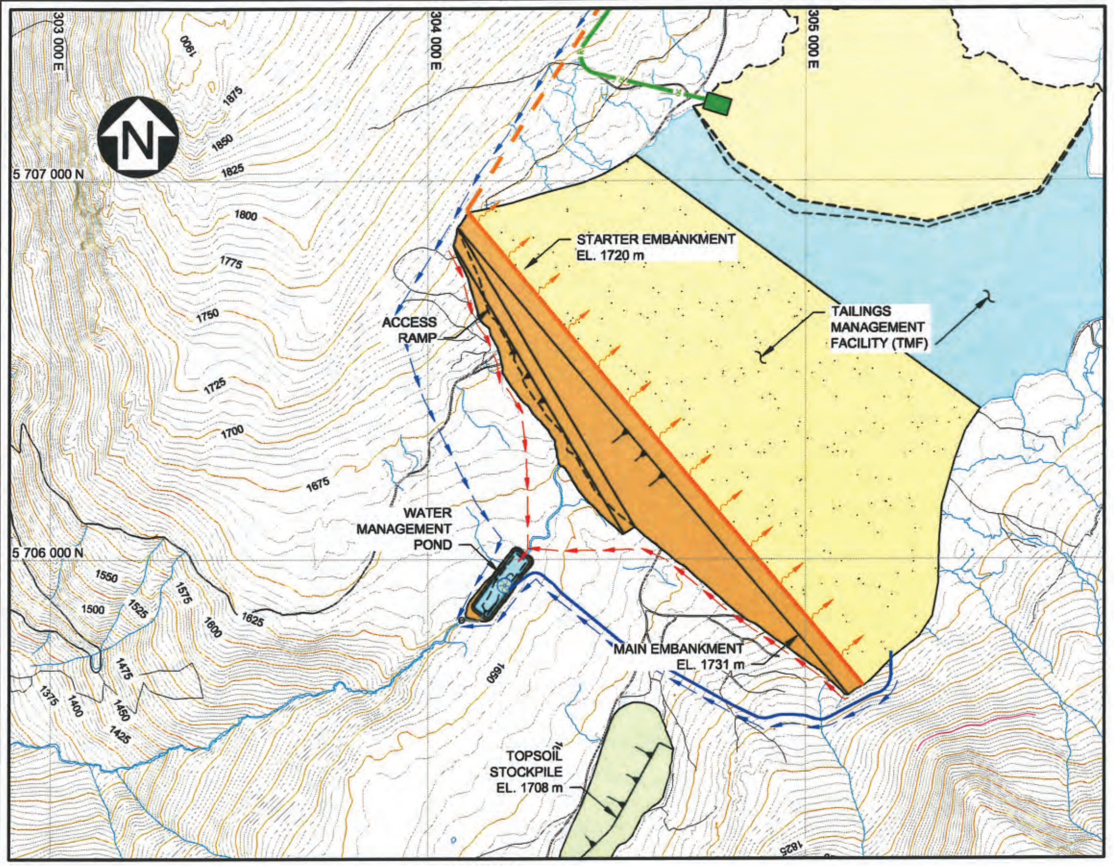
Knight Piésold
CONSULTING

HARPER CREEK MINING CORP.

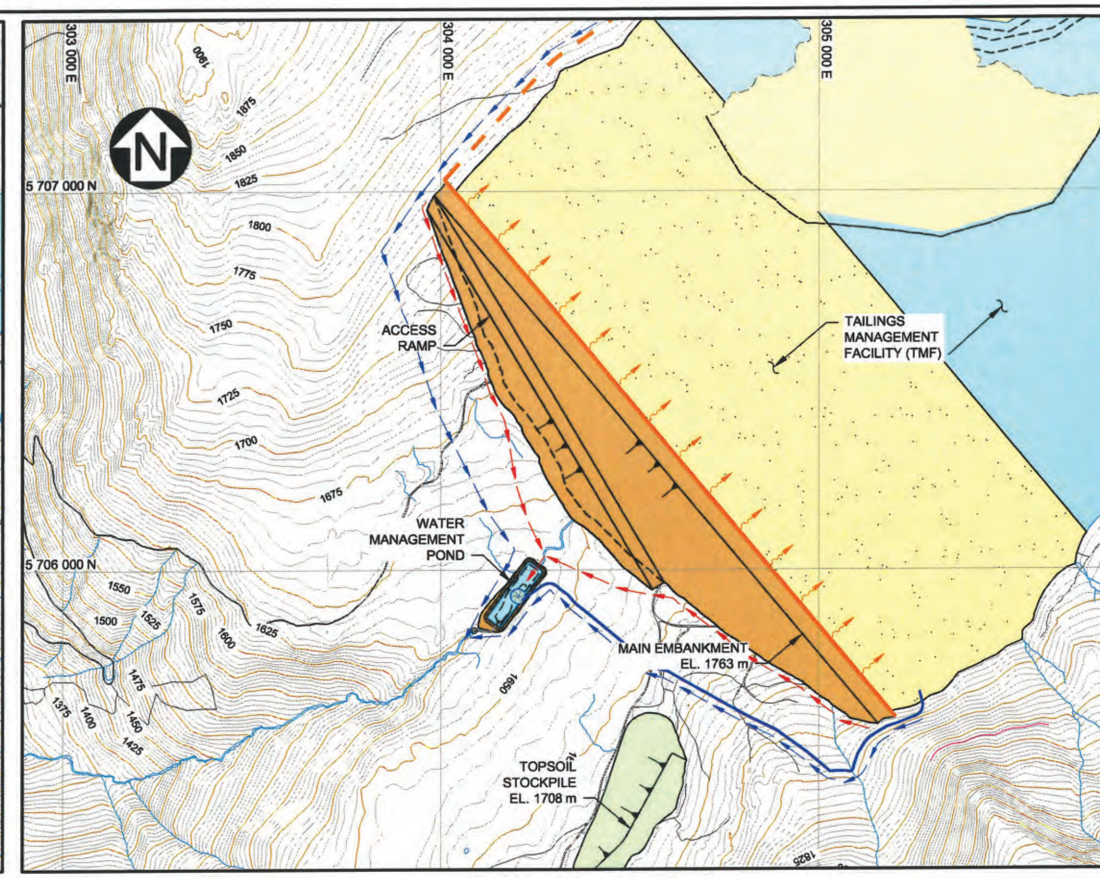
HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
STAGE 1 EMBANKMENT CONSTRUCTION
PLAN AND SECTIONS

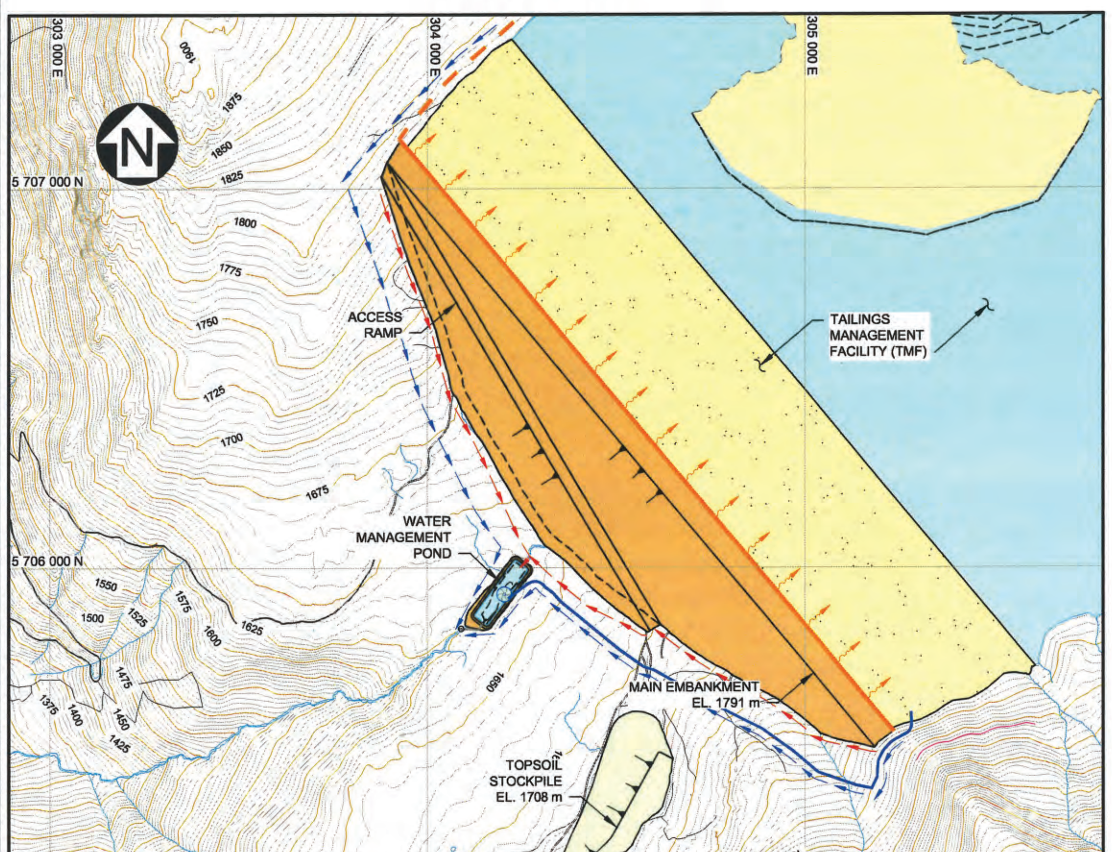
P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0045	0



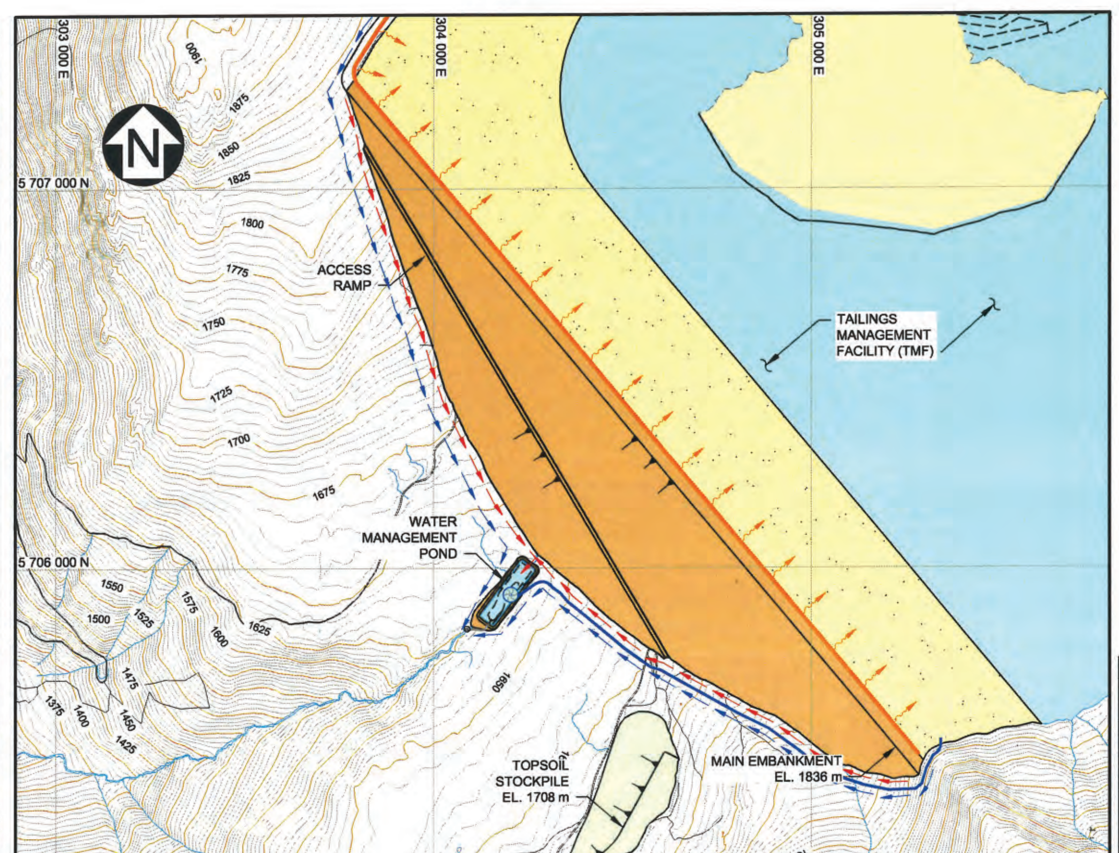
PLAN
EMBANKMENT YEAR 1
SCALE A



PLAN
EMBANKMENT YEAR 5
SCALE A



PLAN
EMBANKMENT YEAR 10
SCALE A



PLAN
EMBANKMENT YEAR 24
SCALE A

LEGEND:

- TAILINGS BEACH
- MINE WATER
- EMBANKMENT FILL
- TOPSOIL
- RECLAIM SYSTEM
- PUMPSTATION
- MINE ACCESS ROAD
- WATER MANAGEMENT PIPELINE
- EXISTING ACCESS TRAILS
- DIVERSION CHANNEL
- COLLECTION CHANNEL
- RECLAIM WATER PIPELINE
- TAILINGS DELIVERY PIPELINE
- TAILINGS DISCHARGE PIPELINE

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

NOT FOR CONSTRUCTION

SCALE A 200 100 0 200 400 600 800 1000 m

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PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
D. D. FONTAINE
36208
Sept 25, 2014

Knight Piésold
CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

WATER MANAGEMENT
TMF EMBANKMENT
PHASED DEVELOPMENT

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0050	0

REVISIONS

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	ABN	BB	KJB

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REVISIONS

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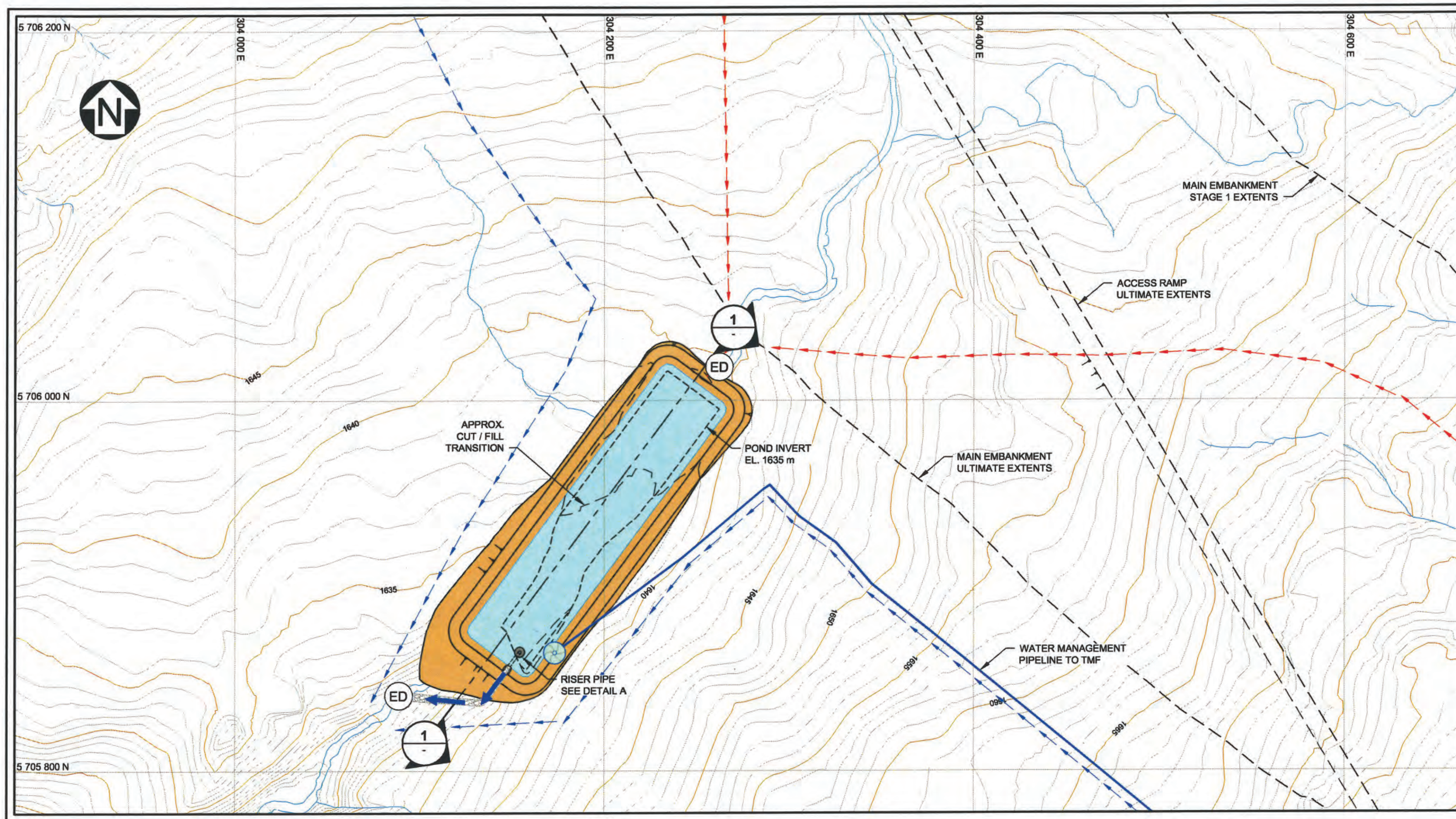
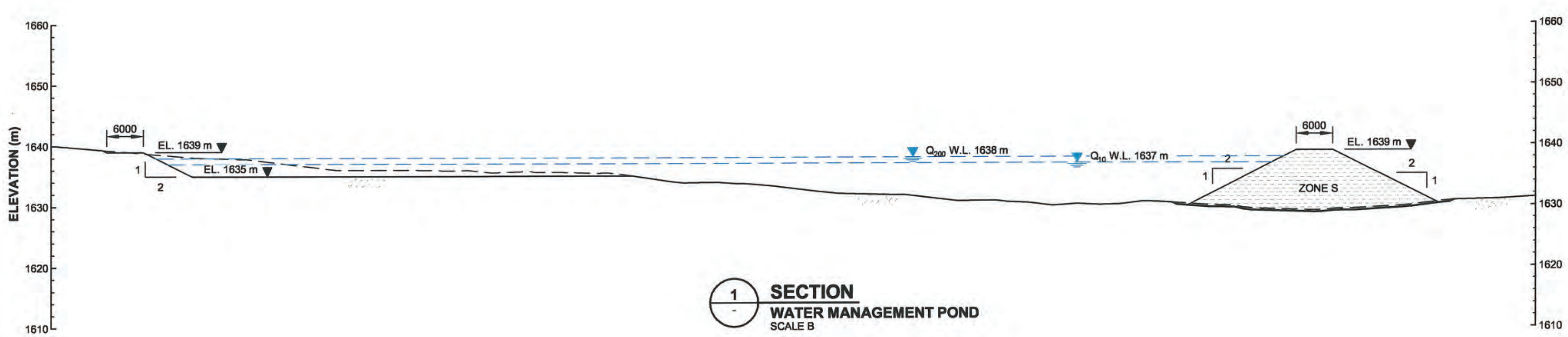
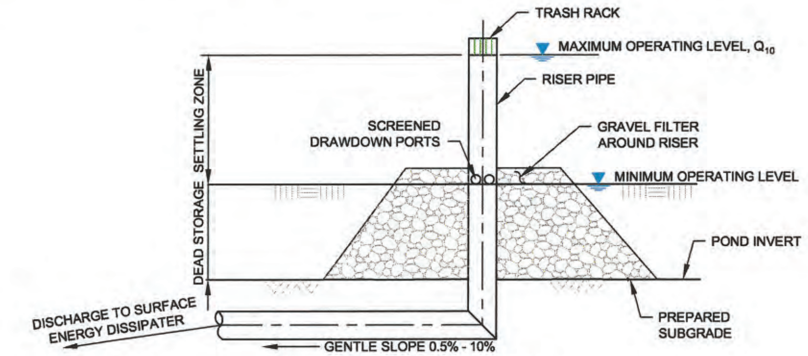


PLAN
WATER MANAGEMENT POND
SCALE A



SECTION
WATER MANAGEMENT POND
SCALE B

BASE DIMENSION (m) (W x L x D)	1 IN 10 STORM EVENT				1 IN 200 STORM EVENT				PUMP SYSTEM
	PEAK INFLOW (m^3/s)	OUTLET TYPE	INVERT ELEVATION (m)	PEAK OUTFLOW (m^3/s)	PEAK INFLOW (m^3/s)	OUTLET TYPE	INVERT ELEVATION (m)	PEAK OUTFLOW (m^3/s)	DESIGN FLOW RATE (m^3/s)
35 x 175 x 4	1.99	1 x 0.24 m Ø RISER PIPE	1637	0.104	5.05	5 m WIDE SPILLWAY	1638	2.213	0.128



DETAIL
RISER PIPE
NTS

LEGEND:

- MINE WATER
- EMBANKMENT FILL
- ZONE S (OVERBURDEN / GLACIAL TILL)
- WATER MANAGEMENT PIPELINE
- COLLECTION CHANNEL
- DIVERSION CHANNEL
- SPILLWAY
- ED ENERGY DISSIPATER
- PUMP

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. CONTOUR INTERVAL IS 1 METRES.
3. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.
4. PEAK EVENT OUTFLOWS ASSUME POND IS FULL TO PRIMARY OUTLET INVERT AT THE TIME OF THE EVENT.

NOT FOR CONSTRUCTION

SCALE B 10 5 0 10 20 30 40 50 m
SCALE A 30 15 0 50 100 150 m

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36208
Sep 25 2014

Knight Piésold
CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

WATER MANAGEMENT
TMF MAIN EMBANKMENT
WATER MANAGEMENT POND
PLAN AND SECTION

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0051	0

REV	DATE	DESCRIPTION	DESIGN	DRAWN	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	DDF	ABN	BB	KJB

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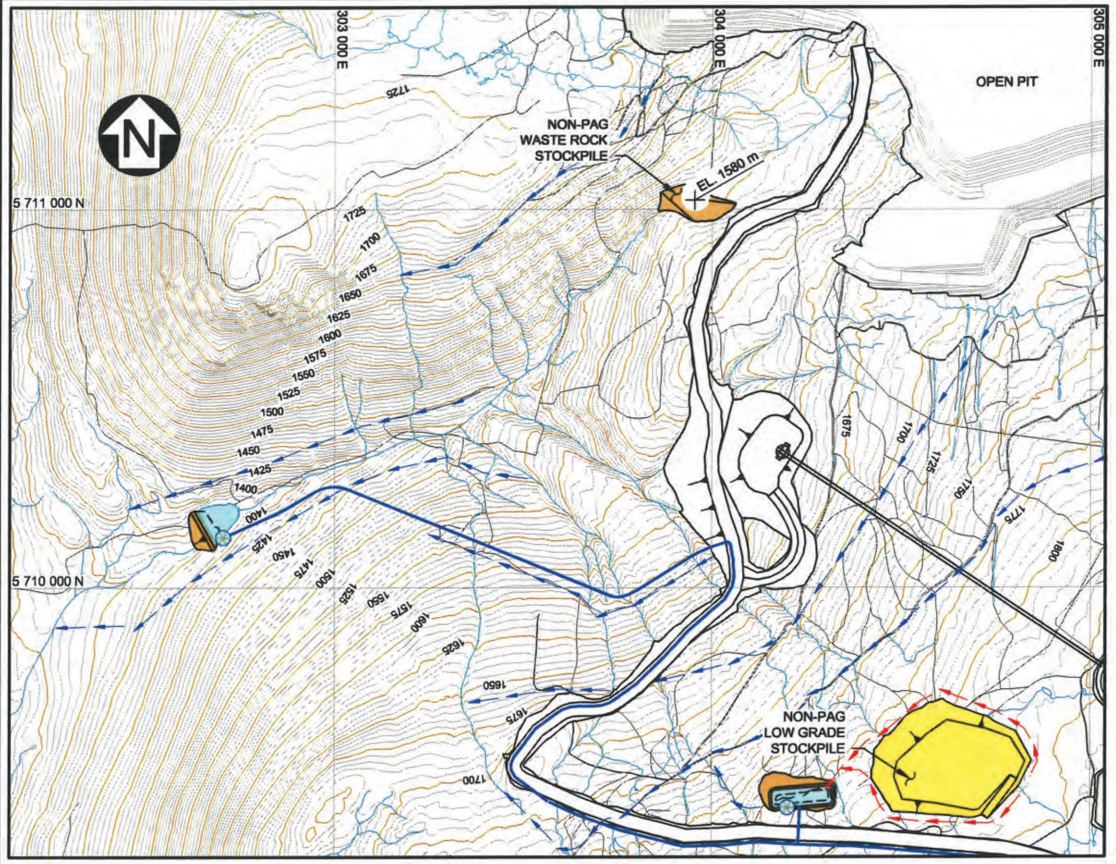
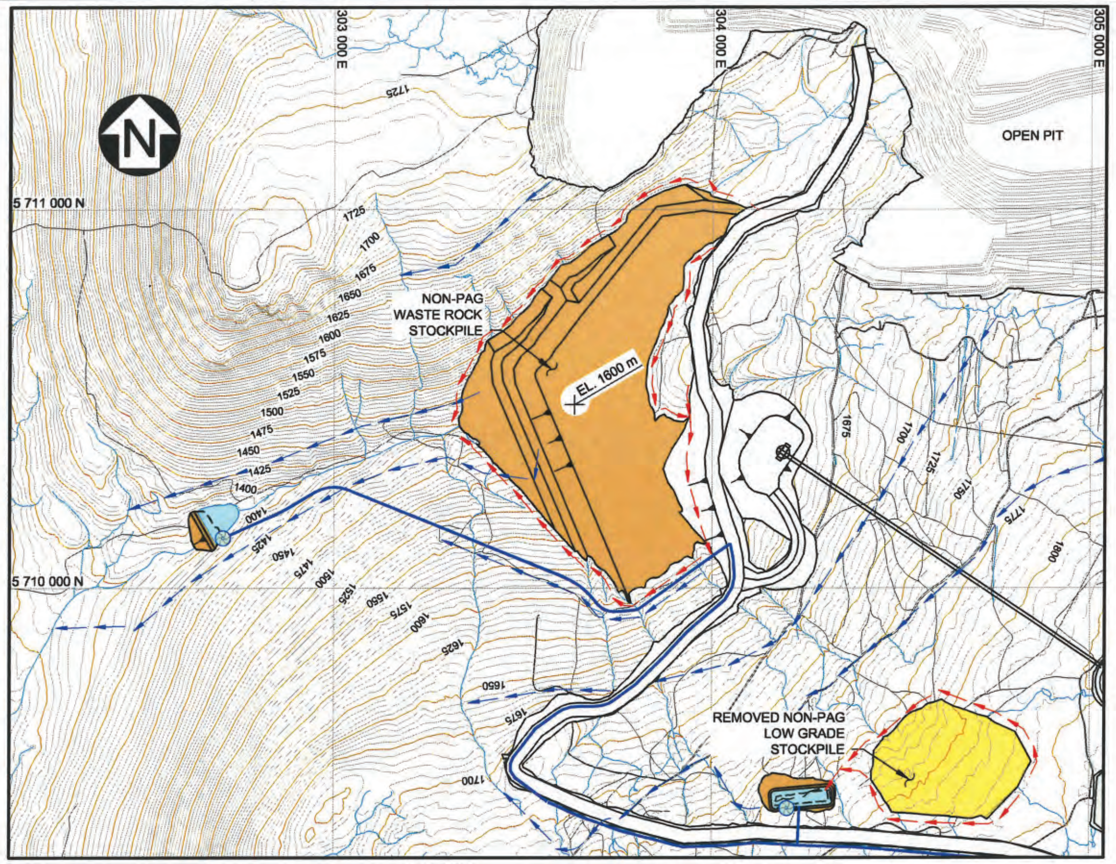
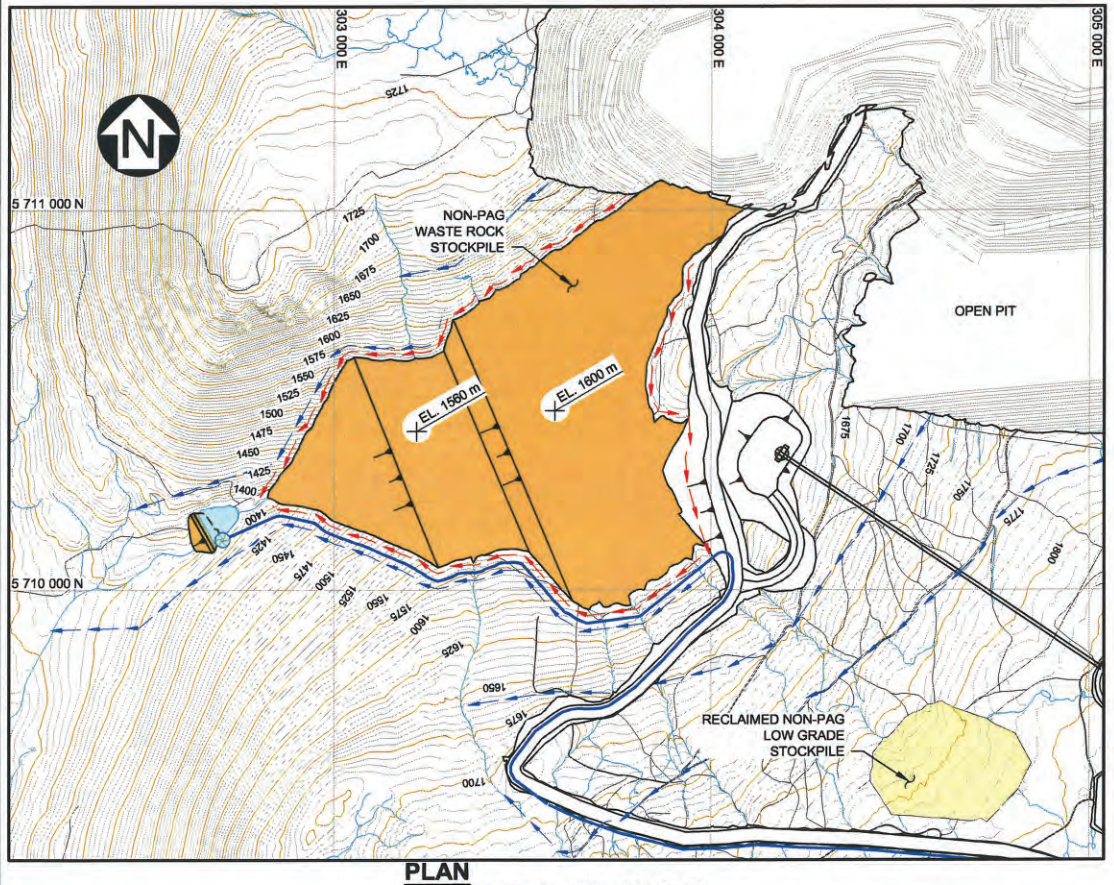


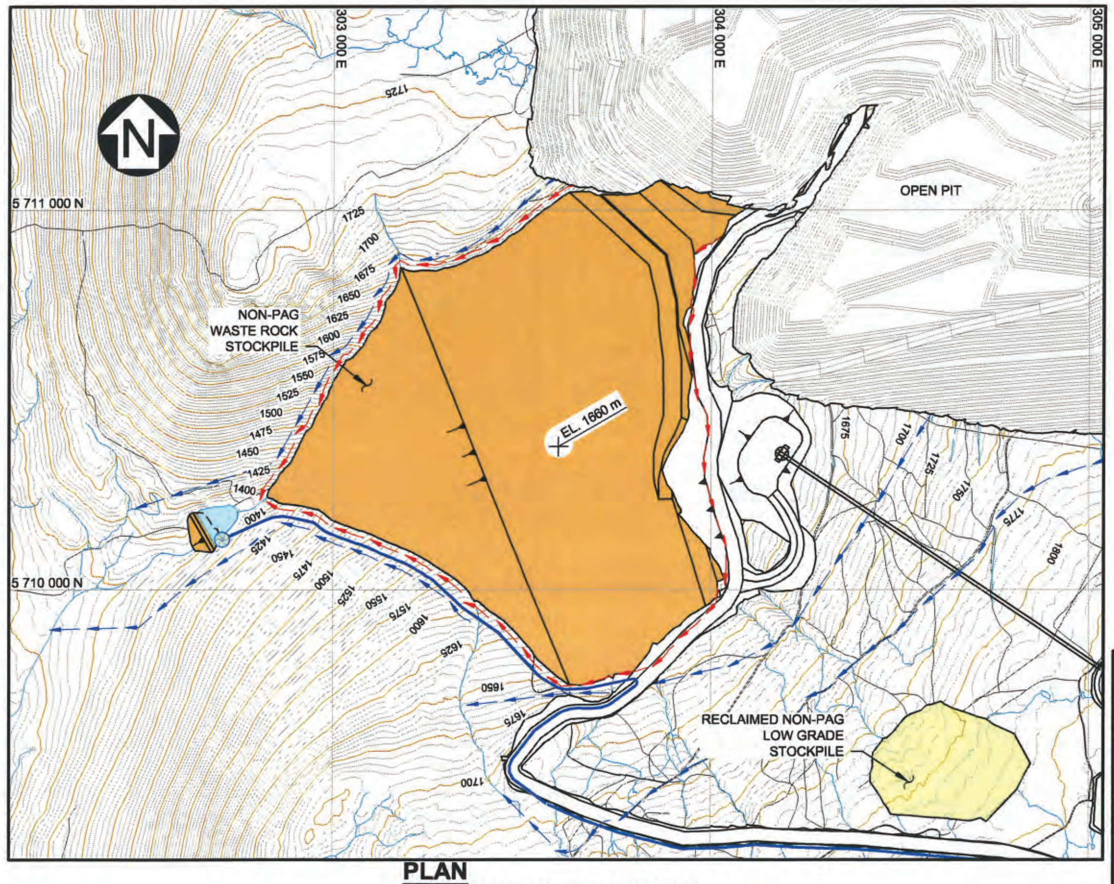
PLAN
NON-PAG WASTE ROCK YEAR 1
SCALE A



PLAN
NON-PAG WASTE ROCK YEAR 5
SCALE A



PLAN
NON-PAG WASTE ROCK YEAR 10
SCALE A



PLAN
NON-PAG WASTE ROCK YEAR 24
SCALE A

LEGEND:

- MINE WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK
- LOW-GRADE ORE
- UPLAND SLOPE
- PUMPSTATION
- WATER MANAGEMENT PIPELINE
- EXISTING ACCESS TRAILS
- DIVERSION CHANNEL
- COLLECTION CHANNEL

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

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SCALE A: 200 100 0 200 400 600 800 1000 m

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HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

WATER MANAGEMENT
NON-PAG WASTE ROCK STOCKPILE
PHASED DEVELOPMENT

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0055	0

DRG. NO.	DESCRIPTION
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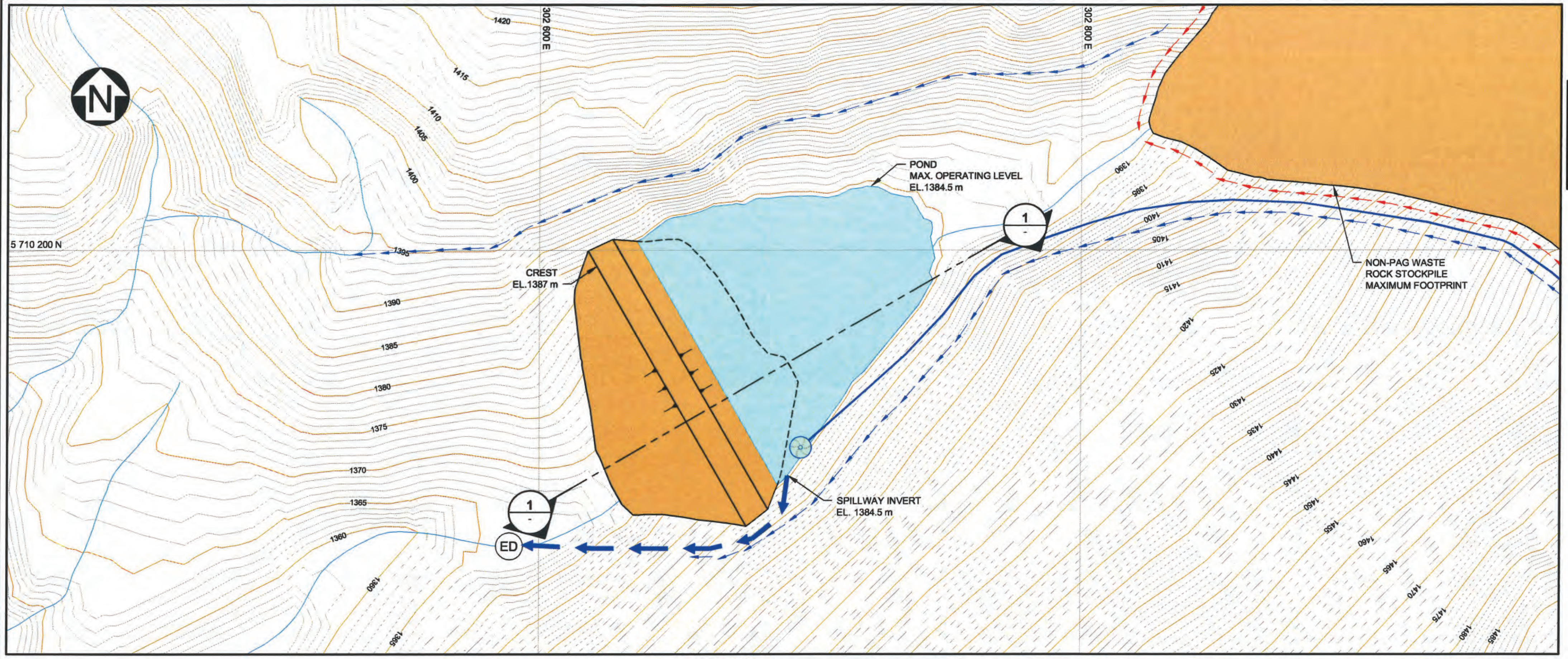
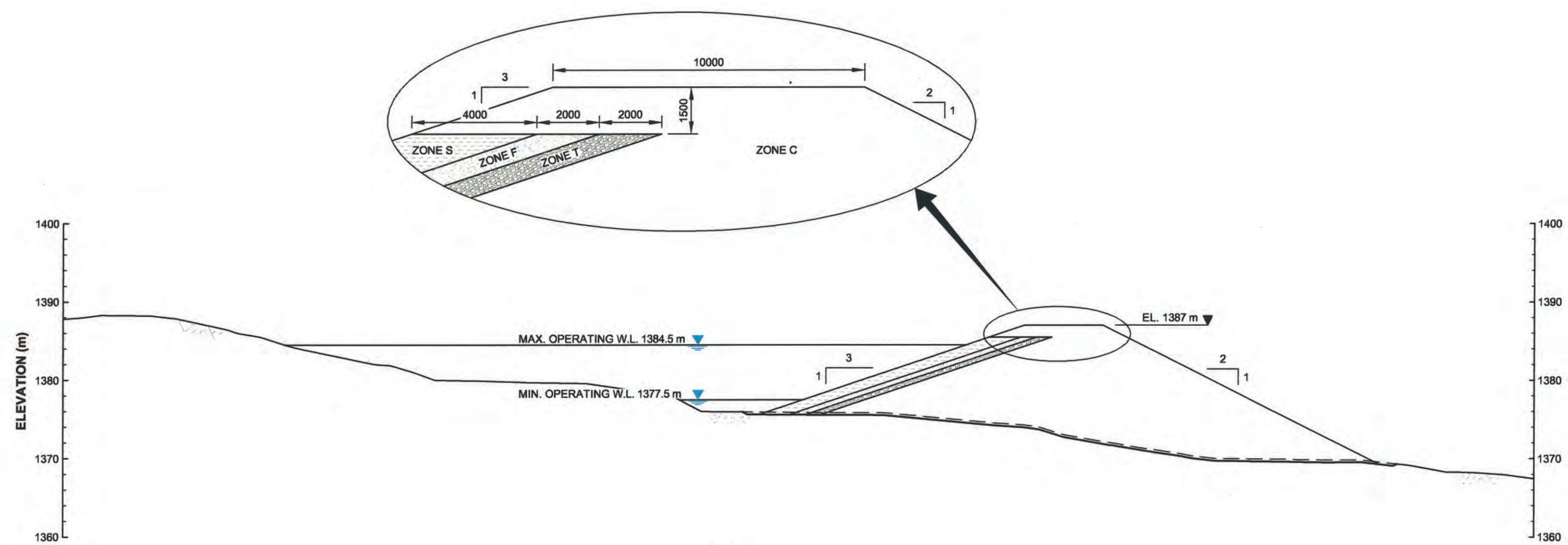


PLAN
WATER MANAGEMENT POND
SCALE A

AVAILABLE OPERATING VOLUME (m³)	1 IN 10 STORM EVENT				1 IN 200 STORM EVENT				PUMP SYSTEM
	PEAK INFLOW (m³/s)	OUTLET TYPE	INVERT ELEVATION (m)	PEAK OUTFLOW (m³/s)	PEAK INFLOW (m³/s)	OUTLET TYPE	INVERT ELEVATION (m)	PEAK OUTFLOW (m³/s)	DESIGN FLOW RATE (m³/s)
24,000	4.34	5 m WIDE SPILLWAY	1384.5	0.16	10.9	5 m WIDE SPILLWAY	1384.5	2.55	0.253

LEGEND:

- MINE WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK
- ZONE S (OVERBURDEN / GLACIAL TILL)
- ZONE F (FILTER)
- ZONE T (TRANSITION)
- ZONE C
- COLLECTION DITCH
- DIVERSION DITCH
- SPILLWAY
- EXISTING ACCESS TRAILS
- ED ENERGY DISSIPATER
- PUMP STATION
- WATER MANAGEMENT PIPELINE



SECTION 1
WATER MANAGEMENT POND
SCALE B

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. CONTOUR INTERVAL IS 1 METRES.
3. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.
4. POND LINING SUBJECT TO ENGINEER'S REVIEW OF FOUNDATION MATERIAL.
5. PEAK EVENT OUTFLOWS ASSUME POND IS FULL TO PRIMARY OUTLET INVERT AT THE TIME OF THE EVENT.
6. THE PUMP SYSTEM WAS DESIGNED TO MAINTAIN THE POND AT THE MINIMUM OPERATING WATER LEVEL DURING NORMAL OPERATIONS.

NOT FOR CONSTRUCTION

SCALE B 8 4 0 10 20 30 40 m
SCALE A 20 10 0 20 40 60 80 100 m

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Sept 25 2014

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HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

WATER MANAGEMENT
NON-PAG WASTE ROCK STOCKPILE
WATER MANAGEMENT POND
PLAN AND SECTION

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0056	0

DRG. NO.	DESCRIPTION
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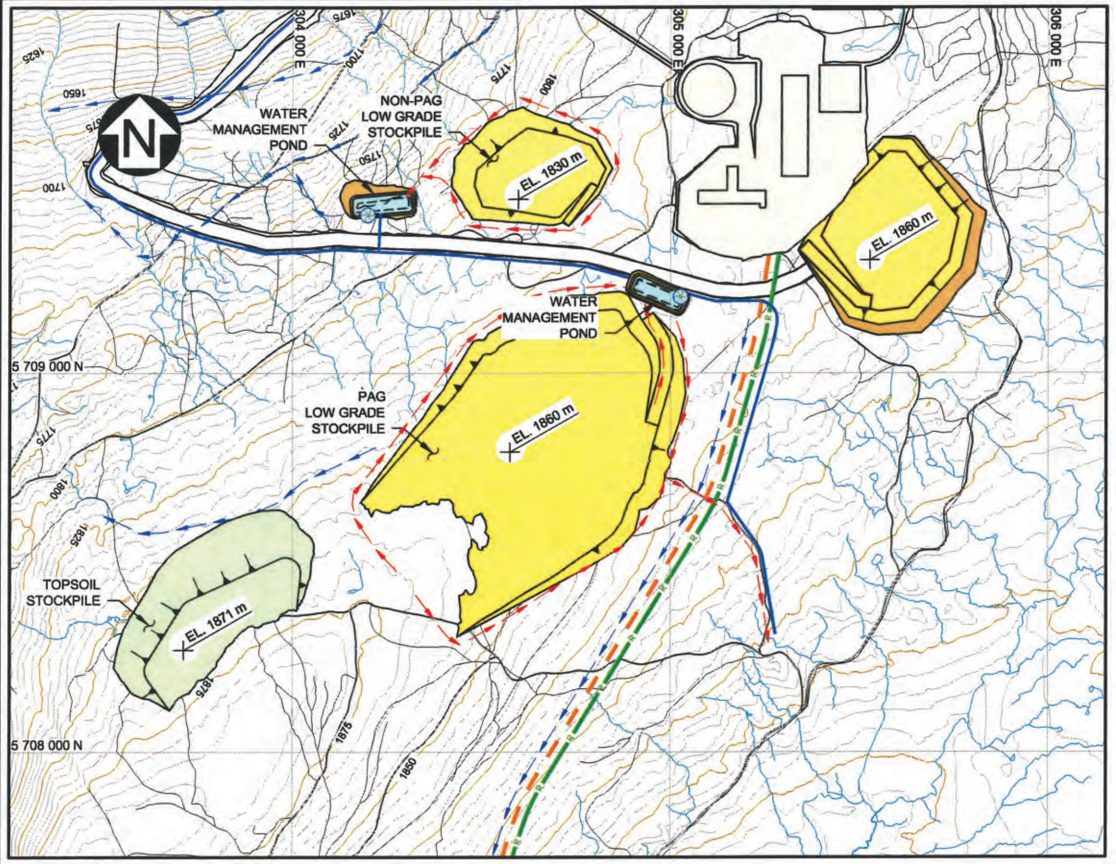
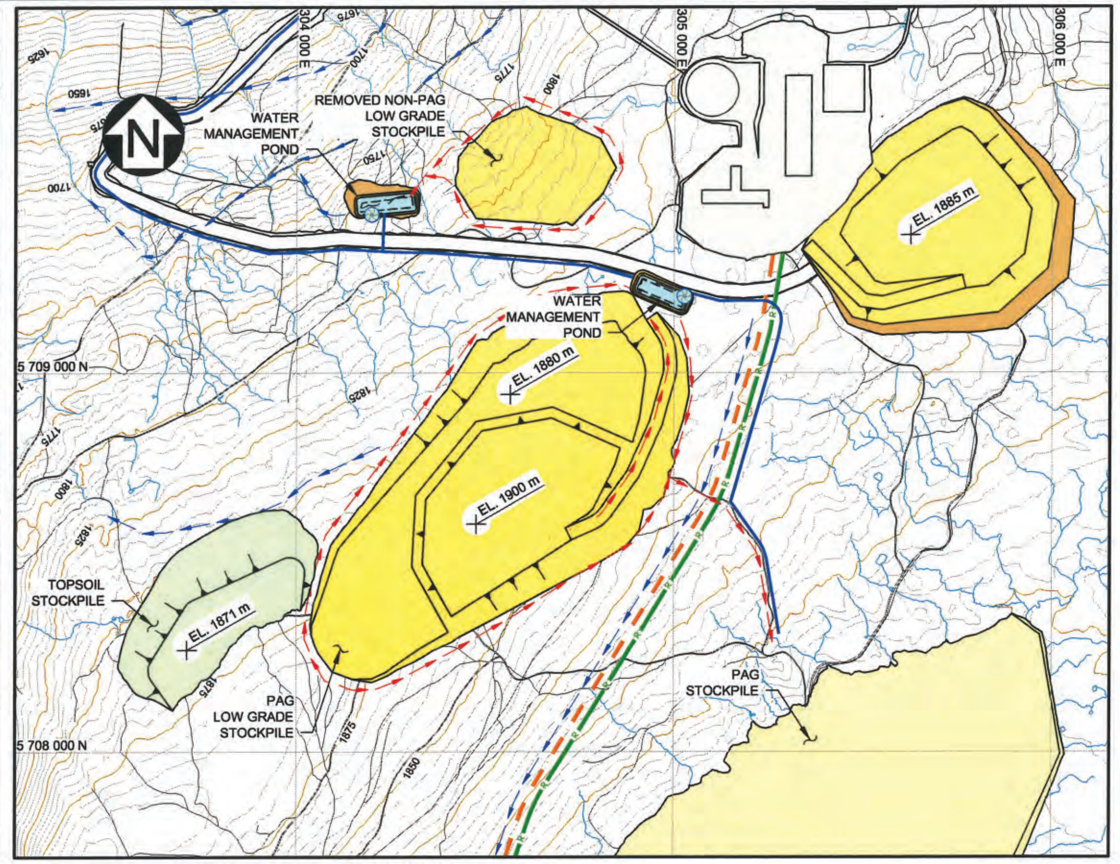
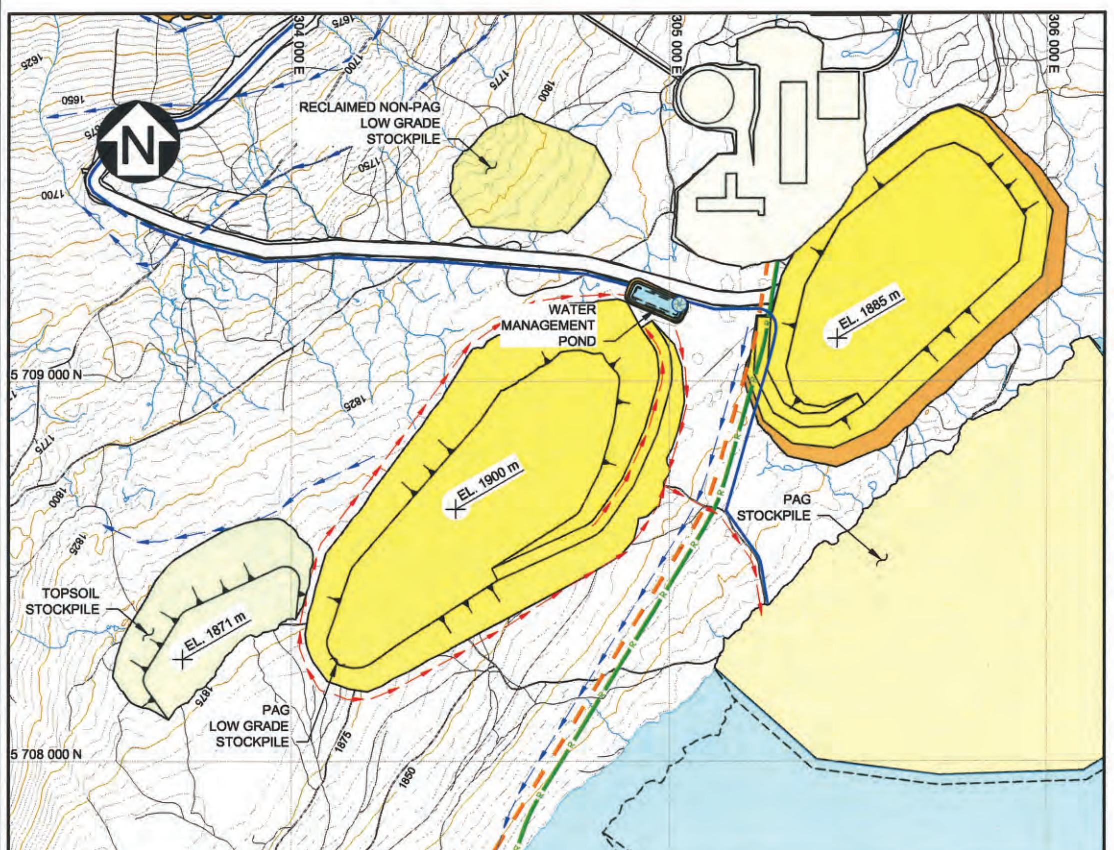


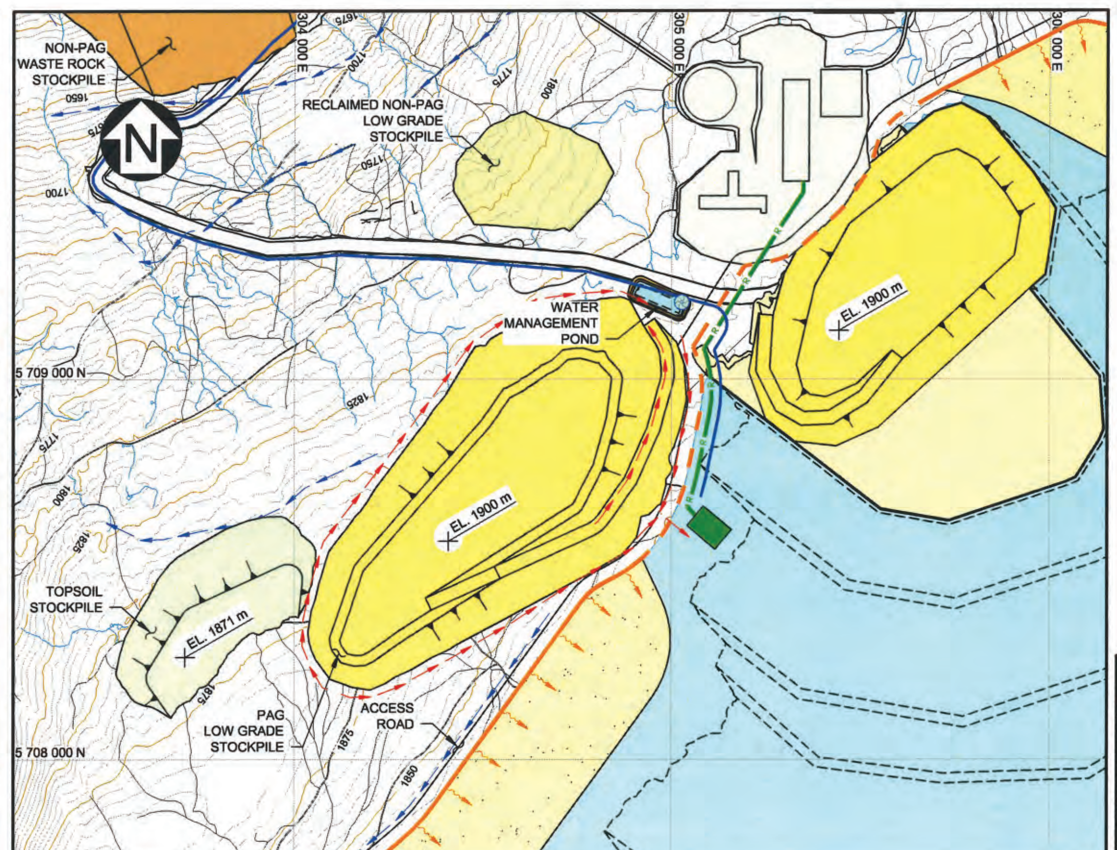
PLAN
LOW-GRADE ORE STOCKPILE YEAR 1
SCALE A



PLAN
LOW-GRADE ORE STOCKPILE YEAR 5
SCALE A



PLAN
LOW-GRADE ORE STOCKPILE YEAR 10
SCALE A



PLAN
LOW-GRADE ORE STOCKPILE YEAR 24
SCALE A

LEGEND:

- TAILINGS BEACH
- MINE WATER
- EMBANKMENT FILL / NON-PAG WASTE ROCK
- PAG WASTE ROCK
- LOW-GRADE ORE
- TOPSOIL
- UPLAND SLOPE
- RECLAIM SYSTEM
- PUMPSTATION
- MINE ACCESS ROAD
- WATER MANAGEMENT PIPELINE
- EXISTING ACCESS TRAILS
- DIVERSION CHANNEL
- COLLECTION CHANNEL
- RECLAIM WATER PIPELINE
- TAILINGS DELIVERY PIPELINE
- TAILINGS DISCHARGE PIPELINE

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

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SCALE A 200 100 0 200 400 600 800 1000 m

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Sept 25, 2014

Knight Piésold
CONSULTING

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

WATER MANAGEMENT
LOW-GRADE ORE STOCKPILE
PHASED DEVELOPMENT

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0060	0

DRG. NO.	DESCRIPTION
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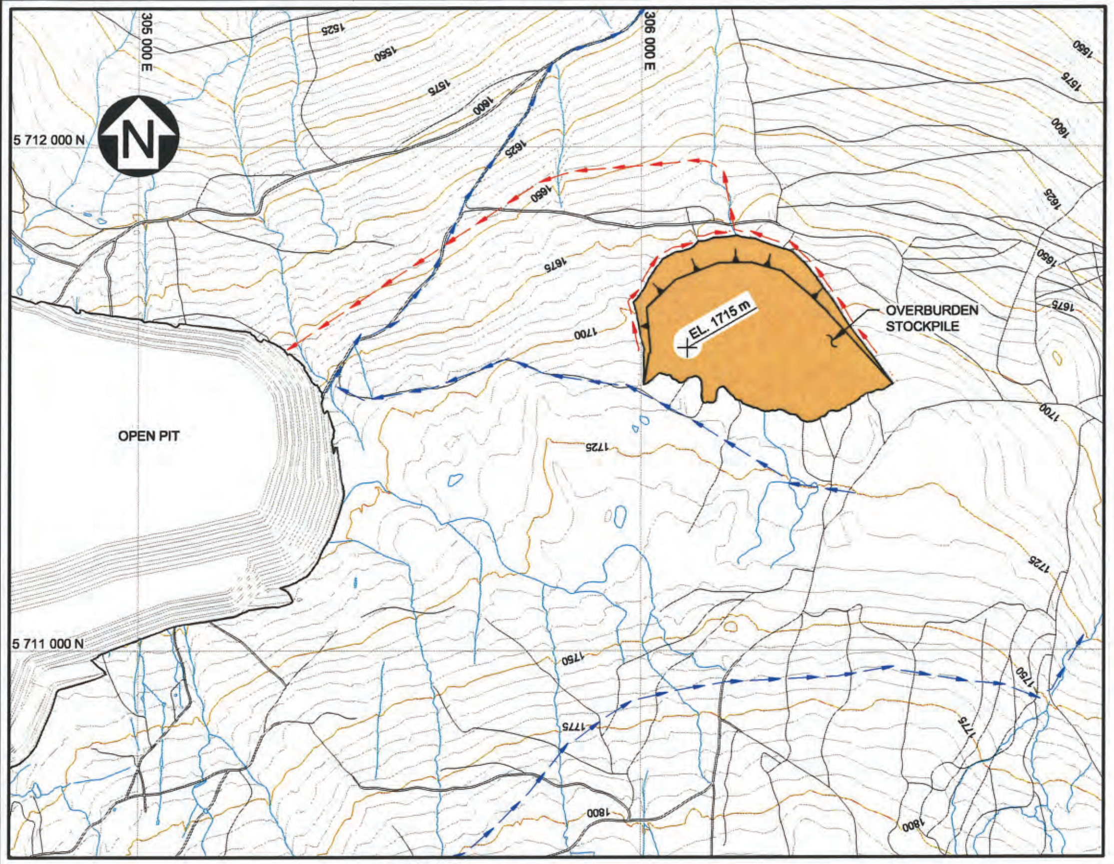
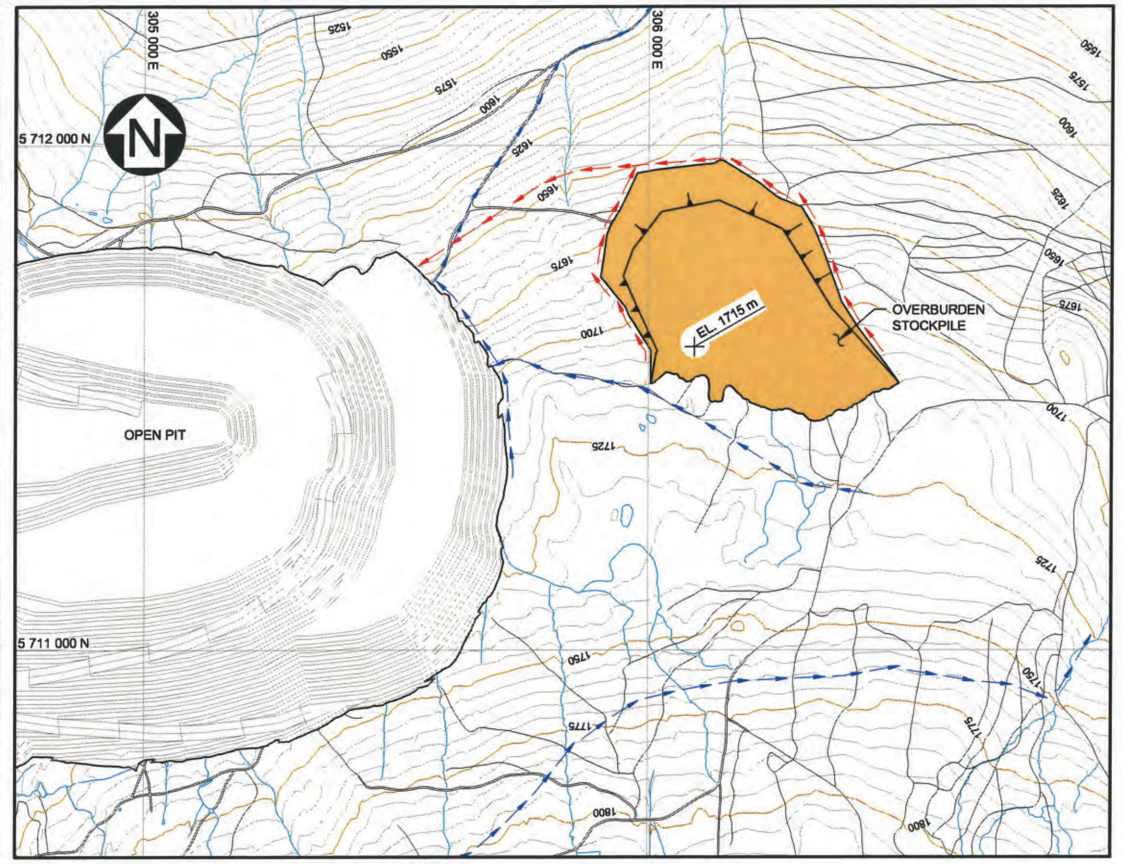
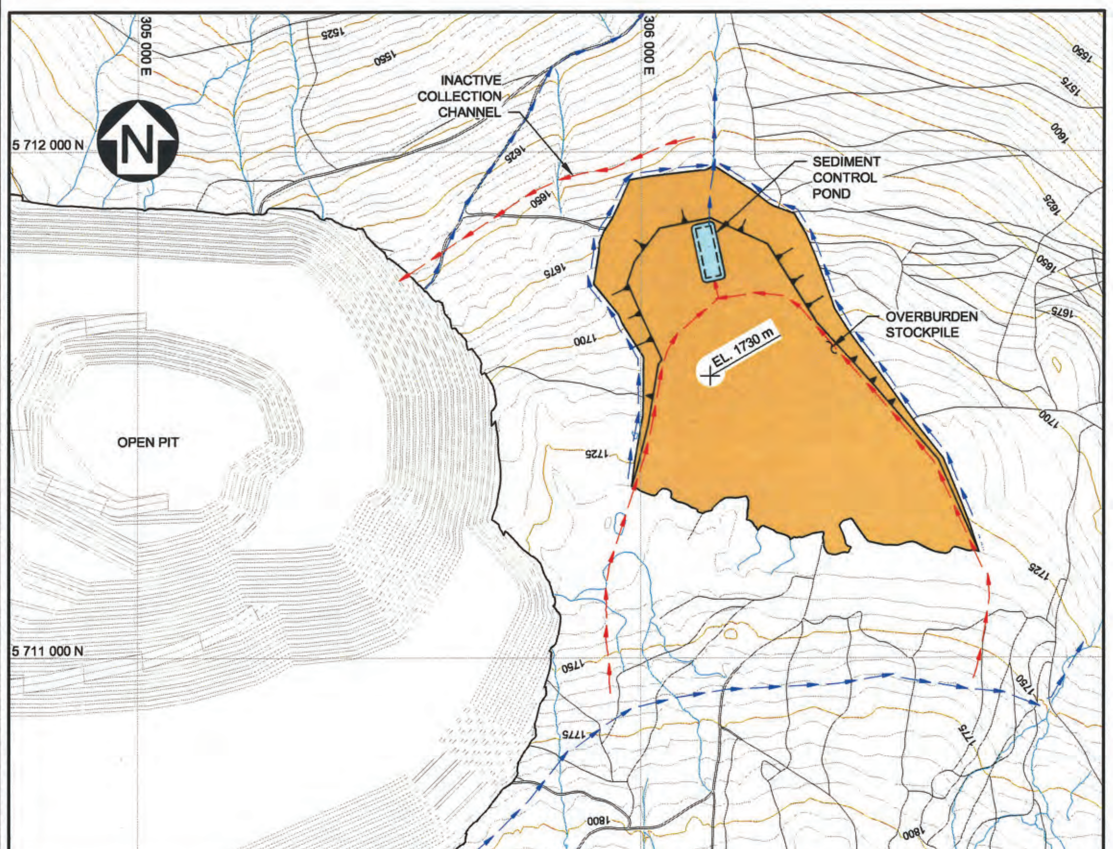


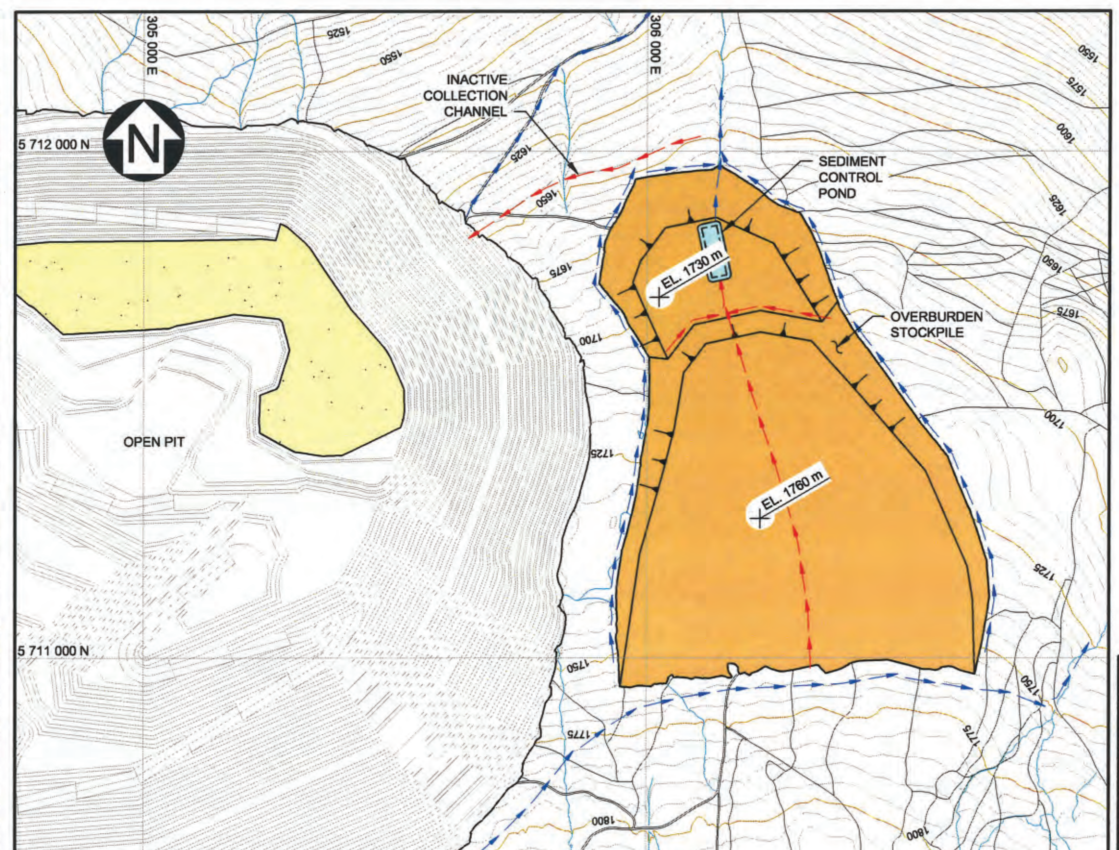
PLAN
OVERBURDEN STOCKPILE YEAR 1
SCALE A



PLAN
OVERBURDEN STOCKPILE YEAR 5
SCALE A



PLAN
OVERBURDEN STOCKPILE YEAR 10
SCALE A



PLAN
OVERBURDEN STOCKPILE YEAR 24
SCALE A

LEGEND:

- TAILINGS BEACH
- MINE WATER
- OVERBURDEN
- TOPSOIL
- MINE ACCESS ROAD
- EXISTING ACCESS TRAILS
- DIVERSION CHANNEL
- COLLECTION CHANNEL

NOTES:

1. COORDINATE GRID IS UTM NAD83 ZONE 11.
2. PLAN BASED ON INFORMATION PROVIDED BY HARPER CREEK MINING CORP. DATED MARCH 11, 2014.
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS ARE IN MILLIMETRES AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

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SCALE A 150 75 0 250 500 750 m

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Sep 25 2014

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HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

WATER MANAGEMENT OVERBURDEN STOCKPILE PHASED DEVELOPMENT

P/A NO.	DRAWING NO.	REVISION
VA101-458/11	C0065	0

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REFERENCE DRAWINGS

APPENDIX B

LABORATORY TESTING OF TAILINGS

APPENDIX B1

TAILINGS LABORATORY TESTING SUMMARY

(Pages B1-1 to B1-16)

Knight Piésold CONSULTING

APPENDIX B1

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT
GEOTECHNICAL LABORATORY TESTING OF TAILINGS

SECTION 1.0 – INTRODUCTION

This appendix presents the details and findings of a laboratory testing program to determine the geotechnical characteristics of the Harper Creek tailings. Two samples of tailings, referred to as ROTL and CuROTL, were tested at solids contents of 35, 45, and 55%. The laboratory testing was carried out at the Knight Piésold laboratory in Denver, Colorado. The following is a list of the geotechnical tests conducted for the two samples:

Index Tests

- Specific Gravity of solids,
- Atterberg Limits (Plastic and Liquid Limits), and
- Particle size distribution by mechanical sieve and hydrometer.

Tailings Slurry Tests

- Undrained Settling for slurries at three different solids contents,
- Drained Settling (with permeability measurement) for three different solids contents,
- Slurry Consolidometer Test (with permeability measurement) for one sample,
- Slurry Consolidation Cylinder (Burette) Test for slurries at three different solids contents, and
- Air drying for slurries at three different solids contents.

The slurry settling (sedimentation) tests provide an estimate of the density to which the tailings slurry will settle in a sub-aqueous environment, under undrained and drained conditions. These tests provide an estimate of expected tailings density in a storage facility after settling and before any significant consolidation or air drying occurs. The slurry consolidometer test is used to determine the consolidation, compressibility and permeability characteristics of the tailings over a wide range of confining pressures (typically corresponding to expected conditions in the field). These parameters can be used to estimate the amount and rate of consolidation, seepage rates, settlement and density increase for the tailings. The slurry consolidation cylinder test provides a measure of the consolidation characteristics of the tailings at very low effective stresses (representing freshly deposited tailings immediately following settling).

The air drying test provides the geotechnical characteristics of the tailings for expected conditions on exposed beaches following deposition. The effect of air drying on moisture loss and tailings density is determined by this test.

SECTION 2.0 – INDEX TESTS

2.1 GENERAL

Index testing was carried out on the tailings solids for both the ROTL and CuROTL samples. The Specific Gravity, Atterberg Limits (Plastic and Liquid Limits), and particle size distribution (screen and hydrometer) of the tailings were determined. These tests provide a measure of the type and condition of the material, specifically the particle density, composition (size and distribution) and plasticity characteristics. The index properties can provide a relationship to material structural properties, including compressibility, permeability and strength.

A summary of the tailings index test results is presented in Table B1.1. Detailed results of the index tests are included in Appendix B2.

2.2 PARTICLE SIZE DISTRIBUTION

Screen and hydrometer particle size analyses were carried out on both tailings samples in accordance with ASTM standard D422 procedures. The hydrometer analysis is used to determine the silt and clay fraction particle sizes. The tailings contain approximately 46-52% fine sand, 44-50% silt, and 4% clay. The particle size distribution of the tailings is shown on Figure B1.1 and B1.2 for both the ROTL and CuROTL samples, respectively.

2.3 SPECIFIC GRAVITY

The specific gravity of the tailings solids was determined in accordance with ASTM standard D854. The average value of the Specific Gravity for both the ROTL and CuROTL samples was 2.79.

2.4 ATTERBERG LIMITS (PLASTICITY)

The Atterberg Limits (Plastic and Liquid Limits) of the tailings were determined in accordance with the ASTM standard D4318 test procedure. Examination of the tailings sample indicated that the tailings material is non-plastic. The Liquid Limit (LL) was determined to be 20% and 28% for the ROTL and CuROTL samples, respectively. The liquid limit is the moisture content at which a soil material starts to exhibit liquid behaviour.

2.5 MATERIAL CLASSIFICATION

The Harper Creek tailings ROTL sample is described as a non-plastic, fine-grained sandy silt with some clay particles, and classifies as ML (inorganic silts) and the CuROTL is a non-plastic, fine grained, silty sand, classifying as SM (silty sand) under the Unified Soil Classification System (USCS).

SECTION 3.0 – TAILINGS SLURRY TESTING

3.1 GENERAL

Slurry settling (sedimentation), consolidation, permeability and air drying tests were carried out on representative samples of the tailings material. Three samples with variable solids contents of 35, 45, and 55% were used for all testing for both the ROTL and CuROTL samples, except for the consolidometer test in which only one slurry sample at 41 and 37% respectively were used for both ROTL and CuROTL samples. The tailings samples were thoroughly mixed to produce a consistent slurry prior to testing.

3.2 SLURRY SETTLING TESTS

3.2.1 General

Undrained and drained slurry settling (sedimentation) tests were carried out on the tailings slurry samples. The undrained settling test estimates the density to which the tailings slurry will settle in an undrained sub-aqueous environment. The drained settling test provides an indication of the dry density achieved from settling with free drainage from the base of the sample. These tests provide an estimate of tailings densities in a storage facility after settling (and completion of supernatant water production) and before any significant consolidation or air-drying occurs. The settled density in the field, prior to commencement of consolidation or air-drying, is likely to be within the range bounded by the densities obtained from the undrained and drained settling tests.

Permeability measurements were taken for the tailings after completion of the drained settling test. Complete results of the slurry settling tests and permeability measurements are included in Appendix B2.

3.2.2 Undrained Settling

Undrained settling tests were performed by placing the slurry into a one litre graduated cylinder and recording the rate of settling and change in volume of the tailings sample as supernatant water bleeds to the surface. The dry density of the settled solids is calculated once the change in settled volume remains essentially constant.

The final settled dry densities of the ROTL samples were approximately 1.2, 1.3, and 1.4 tonnes/m^3 for the solids contents of 35, 45, and 55%, respectively. The measured supernatant water release was 77, 67 and 51% for respective slurry samples at solids contents of 35, 45 and 55%. Similarly, the final settled dry densities of the CuROTL samples were approximately 1.2, 1.3, and 1.3 tonnes/m^3 for the solids contents of 35, 45, and 55%, respectively. The measured supernatant water release was 74, 62 and 43% for these respective slurry samples. The tailings slurry completed drained settling in one to four days.

3.2.3 Drained Settling

The drained settling tests were performed by placing the slurry into a one litre graduated cylinder with provision for bottom drainage and recovery of downward seepage. The rate of settling and change in volume of the sample is recorded with time, as supernatant water bleeds to the surface and drains from the base. Supernatant water was continually decanted from the surface, whenever possible, to minimize development of a vertical seepage gradient across the sample. The dry density of the settled solids is calculated once the change in settled volume remains constant.

The final settled dry densities of the ROTL samples were approximately 1.2, 1.3, and 1.4 tonnes/m^3 for the solids contents of 35, 45, and 55%, respectively. The measured supernatant water release was 77, 69 and 54 % for respective slurry samples at solids content of 35, 45 and 55%. Similarly, the final settled dry densities of the CuROTL samples were approximately 1.2, 1.3, and 1.4 tonnes/m^3 for the solids contents of 35, 45, and 55%, and the measured supernatant water release was 75, 66 and 53% for these slurry samples. The tailings slurry completed drained settling in less than two days.

After completion of the drained settling test, a falling head permeability test was performed on the settled tailings sample. Water was applied to the surface, imposing a vertical gradient across the sample. The drainage rate and drop in water level were recorded with time to provide a value of vertical permeability (hydraulic conductivity). Permeability values range between 1.2×10^{-4} and 8.6×10^{-5} cm/sec for the ROTL sample and 1.4×10^{-4} and 7.8×10^{-5} cm/sec for the CuROTL sample. These tests provide an indication of the vertical permeability of the tailings material at very low effective stresses and corresponding low density (high void ratio). In practice, the permeability will decrease as consolidation reduces the void ratio and increases the density.

3.3 SLURRY CONSOLIDATION TESTS

3.3.1 Slurry Consolidometer Test

A specialized slurry consolidometer device was used to determine the consolidation, compressibility and permeability characteristics of the tailings over a range of tailings densities and effective confining stresses. The slurry consolidometer apparatus is designed to evaluate tailings densities and the consolidation characteristics of slurries that initially have high void ratios and high moisture contents at low effective stresses.

The test was conducted by placing the slurry sample into the consolidometer and allowing the tailings to settle and consolidate under self weight. Confining stresses ranging from very low (about 3 kPa) up to approximately 900 kPa were then applied in incremental loading stages. Routine measurements of settlement with time were recorded during each loading stage. Once settlement ceased or became negligible during loading the confining stress was increased to the next loading stage. The permeability of the tailings was measured at the end of each loading stage. Two-way drainage conditions were facilitated in the test. Detailed results of the testing are provided in Appendix B2.

The results of the slurry consolidation test were used to calculate the coefficients of consolidation (c_v), which is a measure of the consolidation characteristics (rate of consolidation) of a material. A high coefficient of consolidation corresponds to a high rate of consolidation while a low value indicates a slow rate of consolidation. The coefficients of consolidation and void ratios for the ROTL sample determined using the Taylor and Casegrande Methods are presented in Table B1.2 and Table B1.3 for each loading stage, respectively. The corresponding calculated coefficients of volume compressibility (m_v) and measured vertical permeability (k_v) during testing of the samples are also included. Similarly, Tables B1.4 and B1.5 present the same information for the CuROTL tailings sample.

Coefficients of consolidation determined for the tailings generally increase with increasing effective confining stress, ranging from approximately 160 to 1,400 m^2/year. These coefficients of consolidation are within the range of typical values for sandy silt hard rock mine tailings materials, and are higher compared to more fine-grained tailings. The relationship between coefficient of consolidation and effective stress for the tailings samples are shown on Figures B1.3 and B1.4.

The calculated tailings dry density for each loading stage is included in Tables B1.2, B1.3, B1.4, and B1.5. The dry density of the tailings increases with increasing effective stress (load increment), with a value of about 1.6 tonne/m^3 achieved at an effective stress of approximately 1,000 kPa.

Measured vertical permeabilities range from approximately 1 x 10^{-5} cm/sec at very low effective stresses, decreasing to about 3 x 10^{-6} cm/sec at higher stresses. The permeability value at very low stress compares closely to that determined from the falling head test (discussed in Section 3.2.3).

3.3.2 Slurry Consolidation Cylinder (Burette) Test

A slurry consolidation cylinder test was performed to determine the coefficient of consolidation of the tailings at very low effective stresses (high void ratio). Detailed results of the testing are provided in Appendix B2.

The test was carried out by introducing a pre-measured quantity of the tailings slurry sample into a one litre burette with the bottom stopcock closed. After settling of the slurry, the bottom stopcock was opened to permit drainage and dissipation of pore pressures, causing an increase in the effective stress within the sample. Observations of the decrease in slurry volume (settlement) with time were recorded.

The calculated coefficients of consolidation for the ROTL sample were 244, 177 and 188 m^2/year for the solid of contents of 35, 45 and 55%, respectively. The corresponding average effective stress varies between 1.2 and 1.7 kPa.

Similarly, the calculated coefficients of consolidation for the CuROTL sample were 184, 253 and 222 m^2/year for the solid of contents of 35, 45 and 55%, respectively. The corresponding average effective stress varies between 1.1 and 1.8 kPa.

3.4 AIR DRYING TEST

An air drying test was carried out on the tailings to determine the effect of air-drying after initial slurry settling and removal of supernatant water, thereby simulating expected conditions following subaerial exposure of the settled tailings solids.

A sample of the tailings slurry was allowed to settle and air dry under monitored conditions in order to investigate the relationship between density, moisture content, and degree of saturation in a drying environment. Partially saturated conditions were achieved as the amount of moisture loss through evaporation exceeded the reduction in volume of the sample. An absolute relationship between dry density and moisture content exists up to a point at which the degree of saturation falls below 100%. At this stage negative pore pressures (suction pressures) develop and act to further consolidate the sample (reducing the volume). Further drying below a limiting moisture content (the shrinkage limit) produces no further consolidation and the density at this point represents the maximum that can be achieved by air drying of the material.

The air drying test was performed by introducing a sample of tailings slurry into a one litre container with no underdrainage and allowing the slurry to settle while decanting supernatant water. Routine measurements were taken of sample weight and volume. Once the slurry had completely settled and all surface water had been removed, air drying commenced causing moisture loss and consolidation. An evaporation control, subjected to the same drying environment as the slurry sample, was also monitored in order to estimate the rate and amount of evaporation (from a free water surface) applied to the sample.

The final dry density of the air dried samples for both ROTL and CuROTL were estimated to range from approximately 1.5 to 1.6 tonnes/m^3. Air drying of the tailings material produces a moderate increase in dry density over those achieved from the undrained and drained settling tests (approximately 1.3 tonnes/m^3 in average for both conditions).

Complete results of the air drying test are provided in Appendix B2, including plots showing the relationship between tailings dry density, volume reduction, moisture content, degree of saturation and evaporation.

SECTION 4.0 – CONCLUSIONS

Two tailings samples from the expected bulk tailings stream were provided for testing. The test program included index testing to enable geotechnical classification of the materials, and slurry settling, air drying, consolidation and permeability testing to determine the characteristics of the tailings for a range of conditions expected to be representative of field conditions. Test work was completed on tailings samples at solid contents of 35, 45, and 55%. A summary of the test work and results for tailings with a solids content of 35% are provided below.

The specific gravity of the tailings solids was determined to be 2.79 and the material can be described as a non-plastic, fine-grained sandy-silt with traces of clay. The particle size distribution of the tailings sample comprised approximately 46-52% fine sand, 44-50% silt, and 4% clay. The Unified Soil Classification System (USCS) has been used for describing and categorizing soil within groups to allow for the development of distinct soil properties. The tailings can be classified as sand with fines (SM) and a fine-grained soil with very fine sands (ML) depending on the particle size distribution.

Undrained settling, drained settling, and air drying tests were carried out to provide information on the effect of initial slurry solids content on the settling and permeability characteristics of the material and the effect on water recovery and achieved density. Slurry settling (sedimentation) tests provide an estimate of the density to which the tailings slurry will settle in a sub-aqueous environment, under drained and undrained conditions. These tests provide an indication of the tailings dry density achieved in a storage facility after settling and before any significant consolidation occurs. Air drying tests were carried out on the tailings samples to determine the effect of air drying after initial slurry settling and removal of supernatant water.

The tests were performed for a target solids content equal to 35% and the main findings were as follows:

- The settled dry density of the tailings was 1.2 t/m^3 for undrained and drained settling conditions, with a measured supernatant water release of approximately 75%.
- The tailings slurry took up to four days to complete undrained settling and less than two days to complete drained settling.
- A tailings dry density of 1.5 t/m^3 was achieved under air drying conditions.

Laboratory tests carried out to determine the consolidation and permeability characteristics of the tailings included slurry consolidometer, a low stress slurry consolidation test and a falling head permeability test (conducted on settled tailings after completion of drained settling). Relationships between coefficient of consolidation, void ratio and vertical coefficient of permeability to effective stress have been developed for the tailings. The calculated coefficients of consolidation for the tailings range from 20 m^2/year at very low stresses (representing unconsolidated or fresher tailings near surface) to over 1600 m^2/year at high stresses (representing more consolidated or deeper tailings within the deposit). The permeability of the tailings ranged from 1x10^{-4} cm/second at low stresses to 3x10^{-5} cm/second at high stresses.

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MINE WASTE AND WATER MANAGEMENT
LABORATORY TESTING OF TAILINGS

TABLE B1.1 - SUMMARY OF TAILINGS INDEX TESTS

Print Jul/25/14 14:21:08

Type of Tailings	Specific Gravity of Solids	Atterberg Limits			Particle Size Distribution		
		Plastic Limit %	Liquid Limit %	Plasticity Index %	Sand % (4.75 mm to 0.075 mm)	Silt % (0.074 mm to 0.002 mm)	Clay % (< 0.002 mm)
ROTL	2.79	Non-Plastic	20	Non-Plastic	46.7	49.5	3.8
CuROTL	2.79	Non-Plastic	28	Non-Plastic	52.1	43.6	4.3

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix B\Appendix B1\Tables\[Tables.xlsx]Table B1.1

0	26SEP'14	ISSUED WITH REPORT VA101-458/11-1	CM	DDF	KJB
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LABORATORY TESTING OF TAILINGS

TABLE B1.2 - SUMMARY OF SLURRY CONSOLIDATION TEST ON ROTL TAILINGS (TAYLOR METHOD)

Print Sep/26/14 10:10:11

Load Increment (psi)	Load Increment (kPa)	Average Effective Stress (kPa)	Void Ratio (e)	Dry Density (Note 1) (tonne/m^3)	C_v (Note 2) (m^2/year)	m_v (Note 3) (m^2/kN)	k_v (Note 4) (cm/sec)
0.5	3		1.05	1.36			-
		9			-	2.7E-03	
2	14		0.99	1.40			-
		24			46	1.1E-03	
5	34		0.95	1.43			8.2E-05
		52			297	4.3E-04	
10	69		0.92	1.45			7.9E-05
		103			220	3.1E-04	
20	138		0.88	1.49			6.9E-05
		207			258	1.9E-04	
40	276		0.83	1.53			5.7E-05
		414			259	9.7E-05	
80	552		0.78	1.57			4.7E-05
		689			249	5.1E-05	
120	827		0.75	1.59			4.1E-05

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix B\Appendix B1\Tables\[Tables.xlsx]Table B1.2

NOTES:

(1) DRY DENSITY CALCULATED USING VOID RATIO AND MEASURED TAILINGS SPECIFIC GRAVITY OF 2.79.

(2) C_v = COEFFICIENT OF CONSOLIDATION

(3) m_v = COEFFICIENT OF VOLUME COMPRESSIBILITY

(4) k_v = COEFFICIENT OF VERTICAL PERMEABILITY. PERMEABILITY MEASURED AT THE END OF EACH LOADING STAGE.

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LABORATORY TESTING OF TAILINGS

TABLE B1.3 - SUMMARY OF SLURRY CONSOLIDATION TEST ON ROTL TAILINGS (CASAGRANDE METHOD)

Print Sep/26/14 10:10:31

Load Increment (psi)	Load Increment (kPa)	Average Effective Stress (kPa)	Void Ratio (e)	Dry Density (Note 1) (tonne/m^3)	C_v (Note 2) (m^2/year)	m_v (Note 3) (m^2/kN)	k_v (Note 4) (cm/sec)
0.5	3		1.05	1.36			-
		9			-	2.7E-03	
2	14		0.99	1.40			-
		24			58	1.1E-03	
5	34		0.95	1.43			8.2E-05
		52			374	4.3E-04	
10	69		0.92	1.45			7.9E-05
		103			191	3.1E-04	
20	138		0.88	1.49			6.9E-05
		207			193	1.9E-04	
40	276		0.83	1.53			5.7E-05
		414			267	9.7E-05	
80	552		0.78	1.57			4.7E-05
		689			469	5.1E-05	
120	827		0.75	1.59			4.1E-05

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix B\Appendix B1\Tables\[Tables.xlsx]Table B1.3

NOTES:

(1) DRY DENSITY CALCULATED USING VOID RATIO AND MEASURED TAILINGS SPECIFIC GRAVITY OF 2.79.

(2) C_v = COEFFICIENT OF CONSOLIDATION

(3) m_v = COEFFICIENT OF VOLUME COMPRESSIBILITY

(4) k_v = COEFFICIENT OF VERTICAL PERMEABILITY. PERMEABILITY MEASURED AT THE END OF EACH LOADING STAGE.

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
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LABORATORY TESTING OF TAILINGS

TABLE B1.4 - SUMMARY OF SLURRY CONSOLIDATION TEST ON CuROTL TAILINGS (TAYLOR METHOD)

Print Sep/26/14 10:10:40

Load Increment (psi)	Load Increment (kPa)	Average Effective Stress (kPa)	Void Ratio (e)	Dry Density (Note 1) ($tonne/m^3$)	C_v (Note 2) ($m^2/year$)	m_v (Note 3) (m^2/kN)	k_v (Note 4) (cm/sec)
0.5	3		1.23	1.25			-
		9			-	1.4E-03	
2	14		1.19	1.27			-
		24			21	1.2E-03	
5	34		1.14	1.30			9.3E-05
		52			55	4.9E-04	
10	69		1.10	1.33			8.5E-05
		103			190	3.5E-04	
20	138		1.05	1.36			7.1E-05
		207			387	1.9E-04	
40	276		1.00	1.40			5.8E-05
		414			959	1.1E-04	
80	552		0.94	1.44			3.9E-05
		689			1625	6.0E-05	
120	827		0.91	1.46			3.6E-05

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix B\Appendix B1\Tables\[Tables.xlsx]Table B1.4

NOTES:

(1) DRY DENSITY CALCULATED USING VOID RATIO AND MEASURED TAILINGS SPECIFIC GRAVITY OF 2.79.

(2) C_v = COEFFICIENT OF CONSOLIDATION

(3) m_v = COEFFICIENT OF VOLUME COMPRESSIBILITY

(4) k_v = COEFFICIENT OF VERTICAL PERMEABILITY. PERMEABILITY MEASURED AT THE END OF EACH LOADING STAGE.

0	26SEP'14	ISSUED WITH REPORT VA101-458/11-1	CM	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

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MINE WASTE AND WATER MANAGEMENT
LABORATORY TESTING OF TAILINGS

TABLE B1.5 - SUMMARY OF SLURRY CONSOLIDATION TEST ON ROTL TAILINGS (CASAGRANDE METHOD)

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Load Increment (psi)	Load Increment (kPa)	Average Effective Stress (kPa)	Void Ratio (e)	Dry Density (Note 1) (tonne/m^3)	C_v (Note 2) (m^2/year)	m_v (Note 3) (m^2/kN)	k_v (Note 4) (cm/sec)
0.5	3		1.23	1.25			-
		9			-	1.4E-03	
2	14		1.19	1.27			-
		24			22	1.2E-03	
5	34		1.14	1.30			9.3E-05
		52			52	4.9E-04	
10	69		1.10	1.33			8.5E-05
		103			198	3.5E-04	
20	138		1.05	1.36			7.1E-05
		207			342	1.9E-04	
40	276		1.00	1.40			5.8E-05
		414			887	1.1E-04	
80	552		0.94	1.44			3.9E-05
		689			563	6.0E-05	
120	827		0.91	1.46			3.6E-05

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix B\Appendix B1\Tables\[Tables.xlsx]Table B1.5

NOTES:

(1) DRY DENSITY CALCULATED USING VOID RATIO AND MEASURED TAILINGS SPECIFIC GRAVITY OF 2.79.

(2) C_v = COEFFICIENT OF CONSOLIDATION

(3) m_v = COEFFICIENT OF VOLUME COMPRESSIBILITY

(4) k_v = COEFFICIENT OF VERTICAL PERMEABILITY. PERMEABILITY MEASURED AT THE END OF EACH LOADING STAGE.

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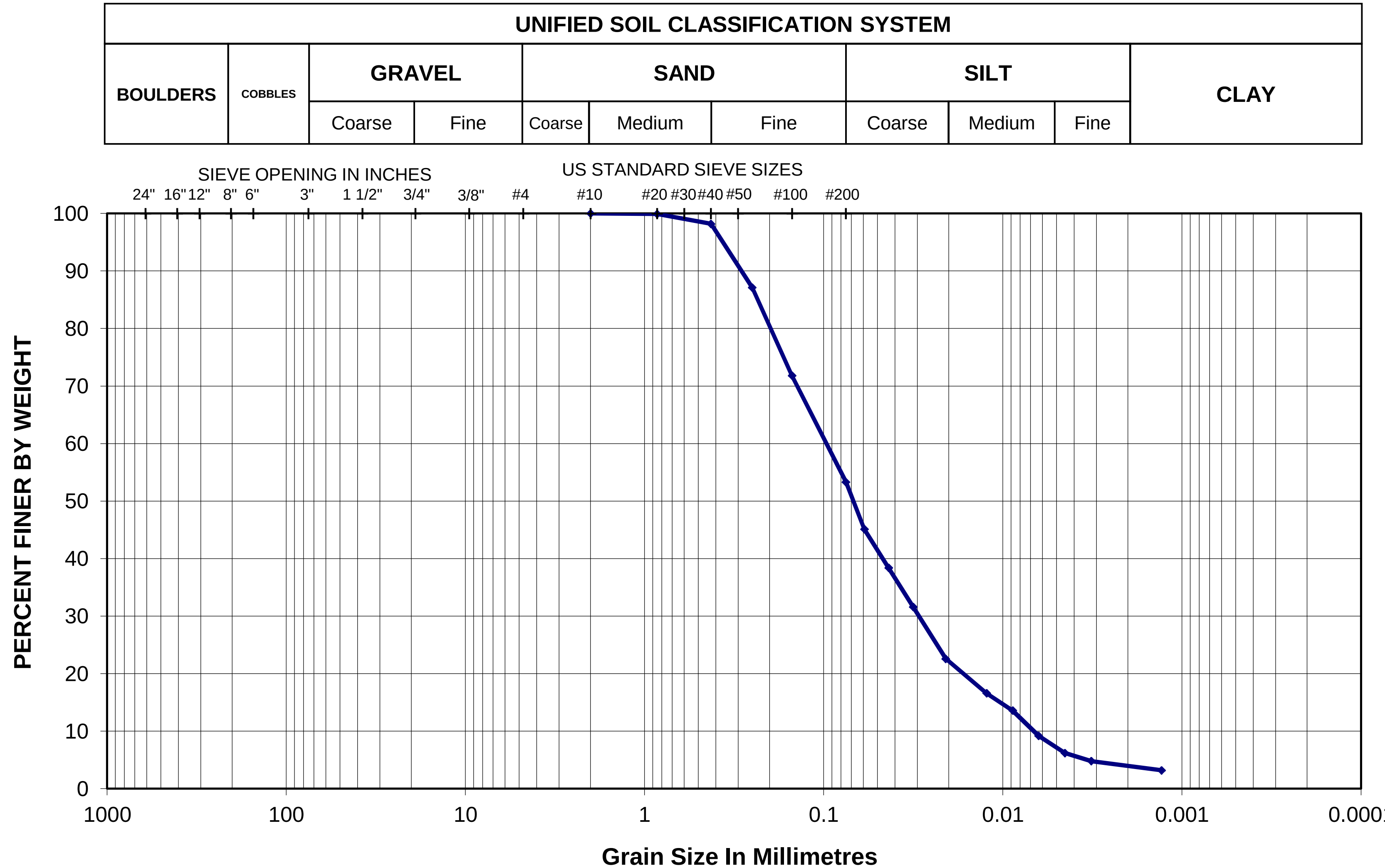
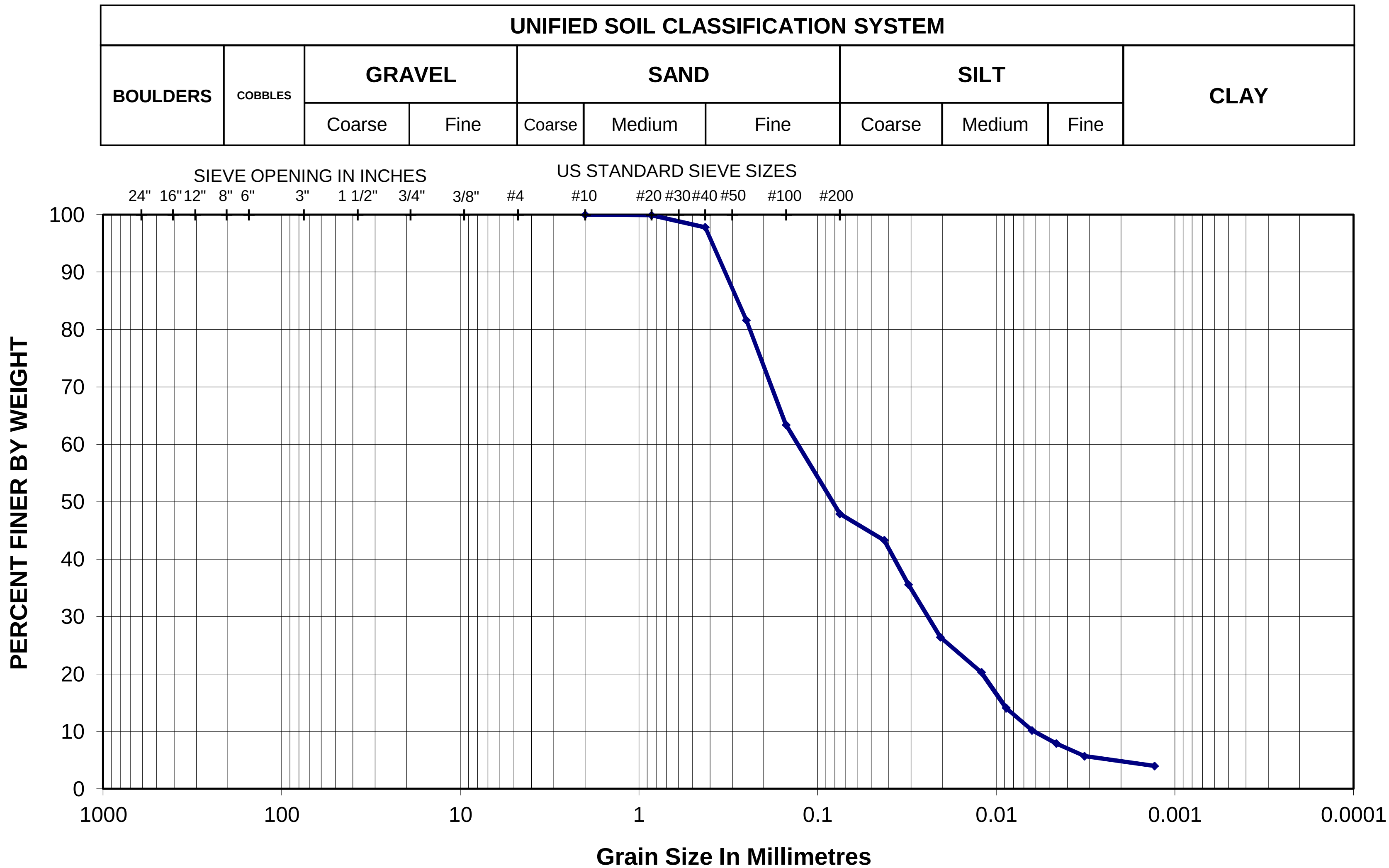


HARPER CREEK MINING CORP.	
HARPER CREEK PROJECT	
PARTICLE SIZE DISTRIBUTION OF TAILINGS SAMPLE (ROTL)	
Knight Piésold CONSULTING	P/A NO. VA101-458/11; REF. NO. 1
	FIGURE B1.1; REV. 0

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	26SEP'14	ISSUED WITH REPORT	CM	DDF	KJB



HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

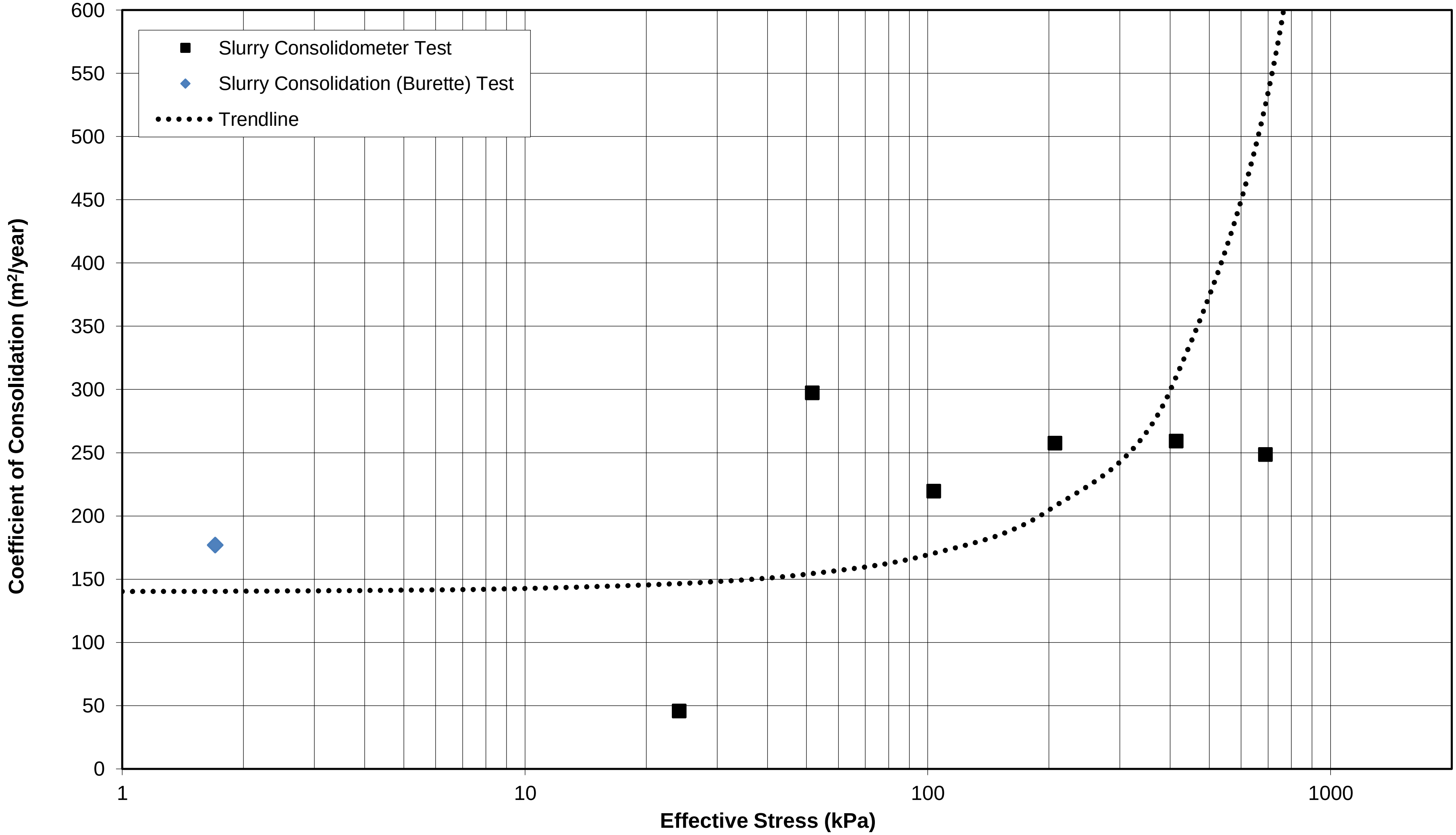
PARTICLE SIZE DISTRIBUTION OF TAILINGS SAMPLE (CuROTL)

Knight Piésold CONSULTING

P/A NO.	REF. NO.
VA101-458/11	1

FIGURE B1.2 REV. 0

0	26SEP'14	ISSUED WITH REPORT	CM	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D



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REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

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RELATIONSHIP BETWEEN TAILINGS COEFFICIENT OF CONSOLIDATION AND EFFECTIVE STRESS ROTL TAILINGS

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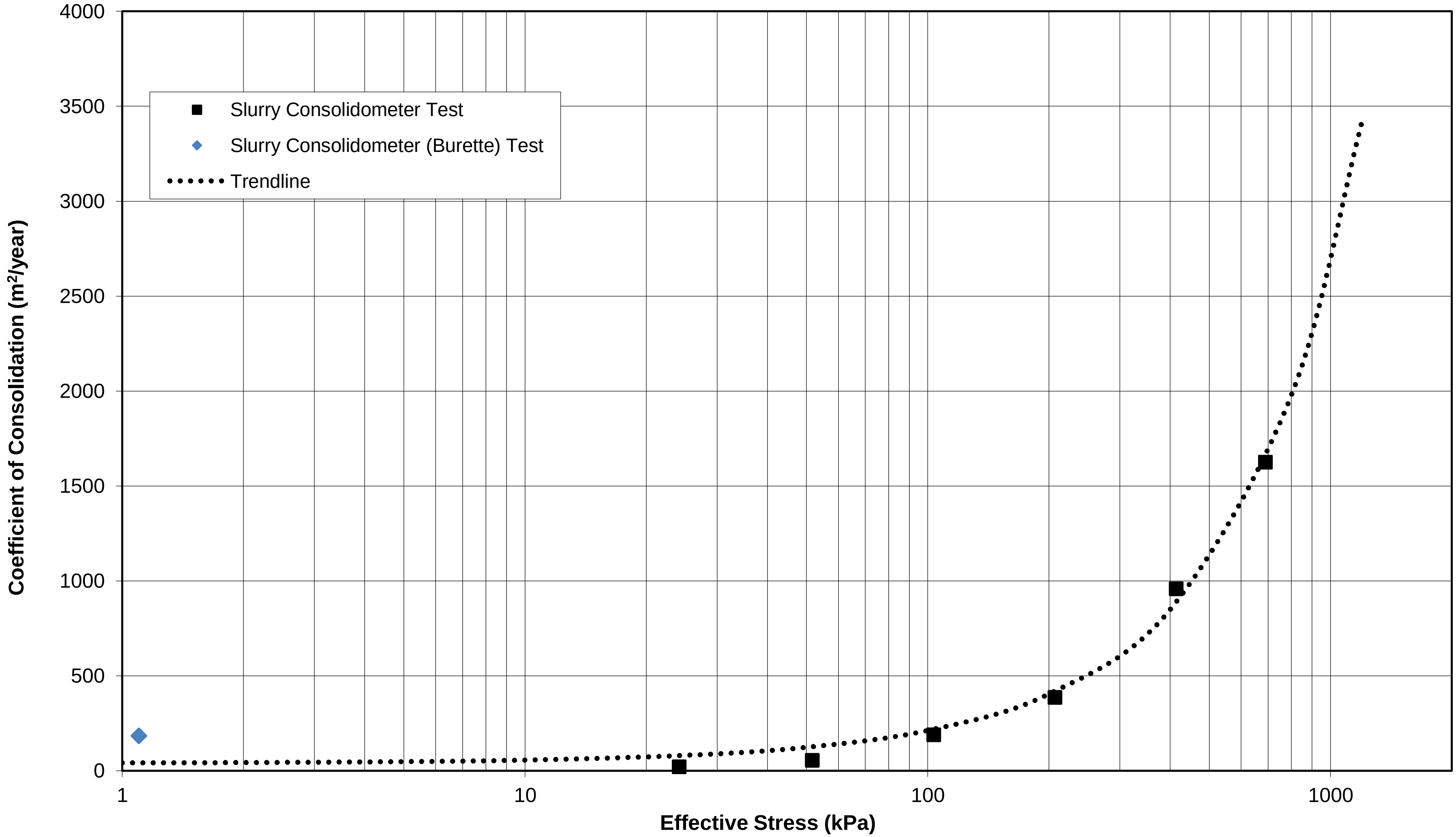
P/A NO. VA101-458/11

REF. NO. 1

FIGURE B1.3

REV 0



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RELATIONSHIP BETWEEN TAILINGS COEFFICIENT OF CONSOLIDATION AND EFFECTIVE STRESS CuROTL TAILINGS

Knight Piésold CONSULTING

P/A NO. VA101-458/11

REF. NO. 1

FIGURE B1.4

REV 0

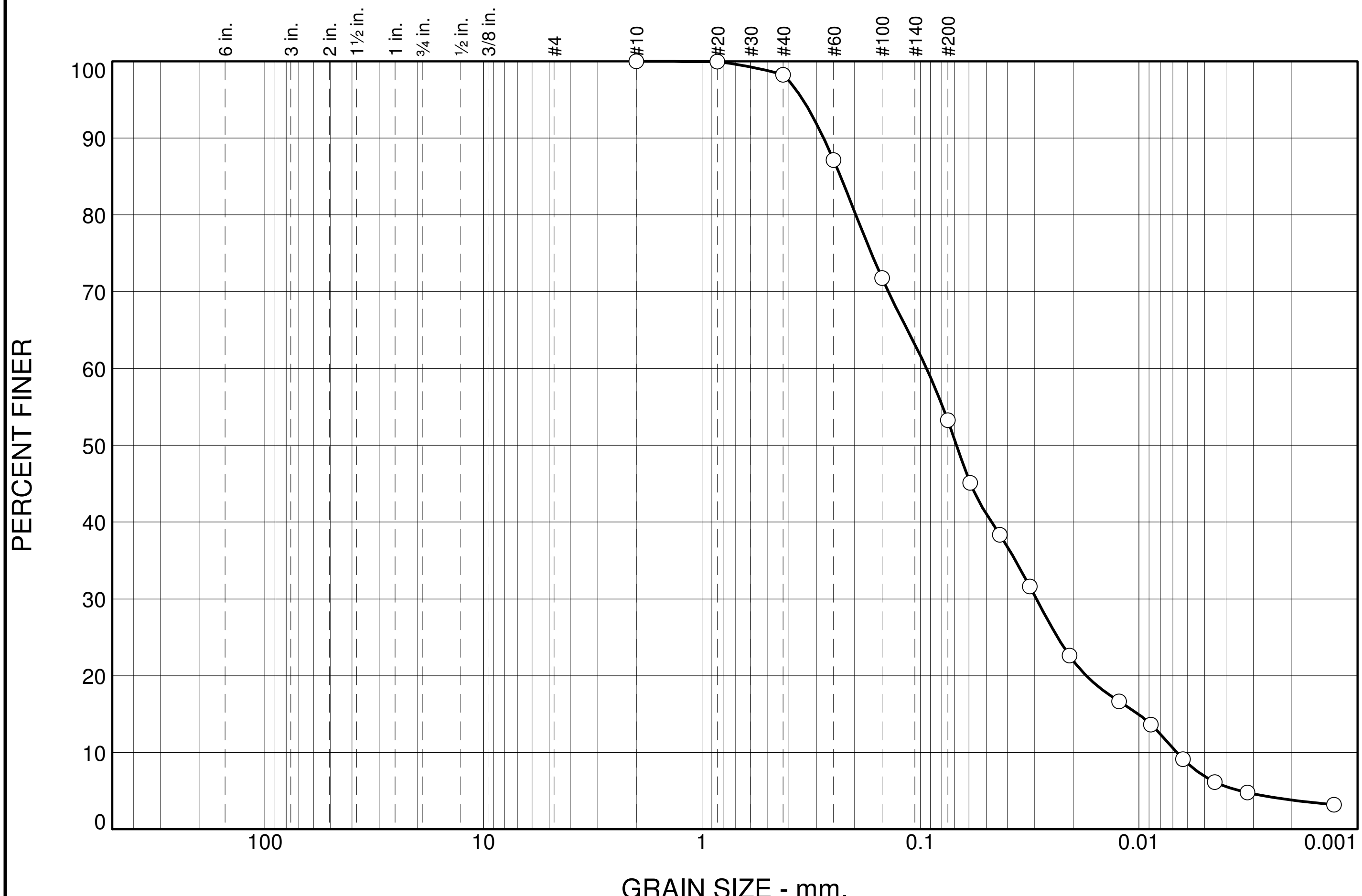
0	26SEP'14	ISSUED WITH REPORT	CM	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

APPENDIX B2

TAILINGS LAB TESTING RESULTS

(Pages B2-1 to B2-77)

Particle Size Distribution Report



% +3"	% Gravel		% Sand			% Fines	
	Coarse	Fine	Coarse	Medium	Fine	Silt	Clay
0.0	0.0	0.0	0.0	1.8	44.9	49.5	3.8

SIEVE SIZE	PERCENT FINER	SPEC.* PERCENT	PASS? (X=NO)
#10	100.0		
#20	99.9		
#40	98.2		
#60	87.1		
#100	71.8		
#200	53.3		
0.0592 mm.	45.1		
0.0433 mm.	38.4		
0.0316 mm.	31.6		
0.0208 mm.	22.6		
0.0123 mm.	16.6		
0.0088 mm.	13.6		
0.0063 mm.	9.2		
0.0045 mm.	6.2		
0.0032 mm.	4.8		
0.0013 mm.	3.2		

* (no specification provided)

Soil Description

sandy silt

Atterberg Limits

PL= NP LL= 20 PI= NP

Coefficients

D_{85}= 0.2325 D_{60}= 0.0939 D_{50}= 0.0684
D_{30}= 0.0295 D_{15}= 0.0101 D_{10}= 0.0067
C_u= 13.97 C_c= 1.38

Classification

USCS= ML AASHTO= A-4(0)

Remarks

Gs=2.79

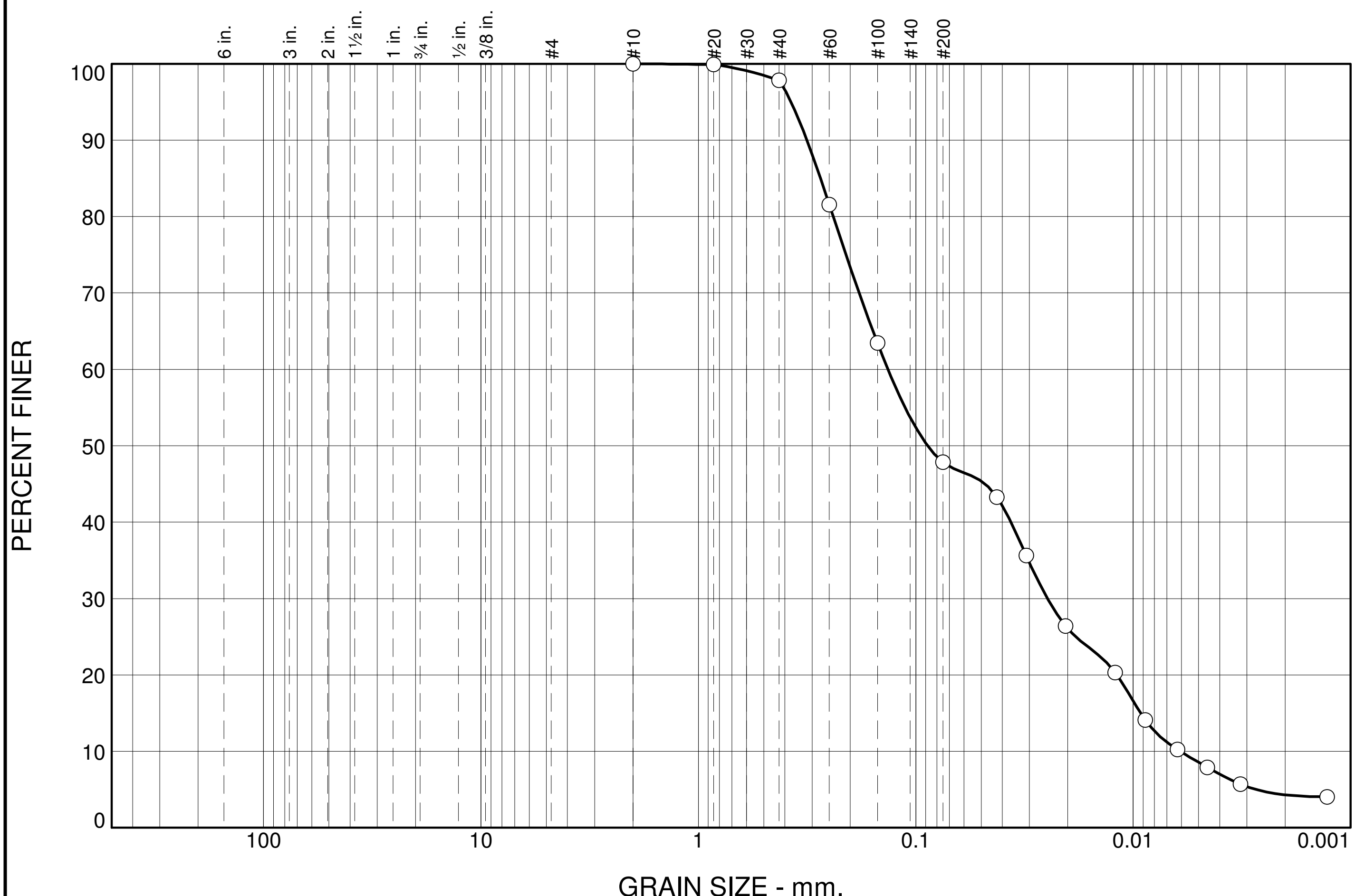
Sample No.: L2011-099-01 **Source of Sample:** **Date:** 10/12/2011
Location: ROTL **Elev./Depth:**

Knight Piésold CONSULTING

Client: YellowHead Mining
Project: Harper Creek

Project No: VA101-458.03 **Fig.**

Particle Size Distribution Report



% +3"	% Gravel		% Sand			% Fines	
	Coarse	Fine	Coarse	Medium	Fine	Silt	Clay
0.0	0.0	0.0	0.0	2.2	49.9	43.6	4.3

SIEVE SIZE	PERCENT FINER	SPEC.* PERCENT	PASS? (X=NO)
#10	100.0		
#20	99.9		
#40	97.8		
#60	81.6		
#100	63.4		
#200	47.9		
0.0423 mm.	43.3		
0.0310 mm.	35.6		
0.0205 mm.	26.4		
0.0121 mm.	20.3		
0.0088 mm.	14.1		
0.0063 mm.	10.2		
0.0046 mm.	7.9		
0.0032 mm.	5.7		
0.0013 mm.	4.0		

* (no specification provided)

Soil Description

silty sand

Atterberg Limits

PL= NP LL= 28 PI= NP

Coefficients

D_{85}= 0.2745 D_{60}= 0.1343 D_{50}= 0.0882
D_{30}= 0.0248 D_{15}= 0.0092 D_{10}= 0.0061
C_u= 22.16 C_c= 0.75

Classification

USCS= SM AASHTO= A-4(0)

Remarks

Gs=2.79

Sample No.: L2011-099-02 **Source of Sample:** **Date:** 10/12/2011
Location: CuROTL **Elev./Depth:**

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Client: YellowHead Mining
Project: Harper Creek
Project No: VA101-458.03 **Fig.**

Knight Piésold CONSULTING	FLOW CONE VISCOSITY & SOLIDS CONTENT DETERMINATION	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	35%	45%	55%
	ROTL	Location:	ROTL		
Test Date:	22-Sep-11 10:57 AM	Tested By:	jhk/jdb		

FLOW CONE WATER DISCHARGE CALIBRATION

Trial No.	1.	2.	3.	Average
Time for complete water discharge (sec)				

FLOW CONE TAILINGS DISCHARGE TESTS

Trial No.	1	2	3	
Time for tailings discharge (sec)				

PERCENT SOLIDS AND MOISTURE CONTENT DETERMINATION

	Trial No.	1.	2.	3.	4.	Average
	Origin of Sample	Target 35%	Target 45%	Target 55%		
	Tare No.					
a.	Tare Weight (g)	113	118	113		
b.	Tare + Wet Sample Weight (g)	959	871	663		
c.	Tare + Dry Sample Weight (g)	402	462	421		
	Drying Time - From	22-Sep-11	22-Sep-11	22-Sep-11		
	- To	23-Sep-11	23-Sep-11	23-Sep-11		
d.	Moisture Loss [b-c] (g)	557	409	242.1		
e.	Dry Sample Weight [c-a] (g)	288.9	344.1	307.8		
	Initial Parameters from Previous Test					
f.	Tare (Cylinder or Beaker) Weight (g)	113	118	113		
g.	Tare + Initial Slurry Weight (g)	959	871	663		
h.	Initial Slurry Weight (g)	846	753	550		
i.	True Moisture Loss [h-e] (g)	557	409	242		
	PERCENT SOLIDS [e/h*100] (%)	34.2	45.7	56.0		
	MOISTURE CONTENT [i/e*100] (%)	192.8	118.9	78.7		

Comments: The tailings settled too quickly to perform the flow cone viscosity tests.

Solids content percentages reported in trials 1, 2 & 3 will vary slightly for each specific test.

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UNDRAINED SETTLING TEST

Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	35%	Test Date:	9/23/2011-10/5/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters

a.	Cylinder (Tare) Weight =	186	g	d. Moisture Content (from drying test) =	**195.8**	%
b.	Initial Slurry Volume =	780	ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.27	g/cm³
c.	Tare + Initial Slurry Weight =	1175	g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	655	g
	Time of Readings 23-Sep-11	08:40 AM		g. Weight of Solids [(c-a)/(1+d/100)] =	335	g

On-going Readings

	Date of Reading	Time of Reading	A. Total Cylinder Weight (g)	B. Total Cylinder Volume (ml)	C. Settled Slurry Volume (ml)	D. Water Recovery [(B-C)/f] (%)	E. Volume Reduction of Solids [1-C/b] (%)	F. Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	G. Slurry Dry Density [g/C] (g/cm³)	H. Moisture Content [(f-(B-C))/g] (%)
1.	23-Sep-11	08:53 AM	1175	780	580	31	26	1.36	0.58	135.99
2.	23-Sep-11	09:07 AM	1175	780	415	56	47	1.50	0.81	86.67
3.	23-Sep-11	10:24 AM	1175	780	295	74	62	1.71	1.13	50.80
4.	23-Sep-11	01:00 PM	1175	780	295	74	62	1.71	1.13	50.80
5.	23-Sep-11	05:48 PM	1175	780	295	74	62	1.71	1.13	50.80
6.	24-Sep-11	11:05 AM	1175	780	292	75	63	1.72	1.15	49.91
7.	25-Sep-11	12:47 PM	1175	780	288	75	63	1.73	1.16	48.71
8.	26-Sep-11	11:16 AM	1175	775	285	75	63	1.75	1.17	49.31
9.	27-Sep-11	10:36 AM	1175	775	270	77	65	1.79	1.24	44.82
10.	28-Sep-11	08:55 AM	1175	775	270	77	65	1.79	1.24	44.82
11.	29-Sep-11	10:28 AM	1174	775	270	77	65	1.79	1.24	44.82

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Knight Piésold CONSULTING	UNDRAINED SETTLING TEST	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	45%	Test Date:	9/23/2011-10/5/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters

a.	Cylinder (Tare) Weight =	184	g	d. Moisture Content (from drying test) =	**123.0**	%
b.	Initial Slurry Volume =	650	ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.39	g/cm³
c.	Tare + Initial Slurry Weight =	1085	g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	497	g
	Time of Readings 23-Sep-11	08:27 AM		g. Weight of Solids [(c-a)/(1+d/100)] =	404	g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Recovery [(B-C)/f] (%)	Volume Reduction of Solids [1-C/b] (%)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)	Moisture Content [(f-(B-C))/g] (%)
1.	23-Sep-11	08:53 AM	1085	650	485	33	25	1.52	0.83	82.15
2.	23-Sep-11	09:07 AM	1085	650	390	52	40	1.64	1.04	58.65
3.	23-Sep-11	10:24 AM	1085	650	340	62	48	1.74	1.19	46.28
4.	23-Sep-11	01:00 PM	1085	650	340	62	48	1.74	1.19	46.28
5.	23-Sep-11	05:48 PM	1085	650	340	62	48	1.74	1.19	46.28
6.	24-Sep-11	11:05 AM	1085	650	340	62	48	1.74	1.19	46.28
7.	25-Sep-11	12:47 PM	1085	650	335	63	48	1.75	1.21	45.04
8.	26-Sep-11	11:15 AM	1085	650	331	64	49	1.76	1.22	44.05
9.	27-Sep-11	10:36 AM	1085	650	315	67	52	1.80	1.28	40.09
10.	28-Sep-11	08:55 AM	1085	650	315	67	52	1.80	1.28	40.09

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Knight Piésold CONSULTING	UNDRAINED SETTLING TEST	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	55%	Test Date:	9/23/2011-10/5/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =	195	g	d. Moisture Content (from drying test) =	**76.5**	%
b. Initial Slurry Volume =	550	ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.57	g/cm³
c. Tare + Initial Slurry Weight =	1057	g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	373	g
Time of Readings 23-Sep-11	08:15 AM		g. Weight of Solids [(c-a)/(1+d/100)] =	488	g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Recovery [(B-C)/f] (%)	Volume Reduction of Solids [1-C/b] (%)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)	Moisture Content [(f-(B-C))/g] (%)
1.	23-Sep-11	08:30 AM	1057	550	500	13	9	1.62	0.98	66.24
2.	23-Sep-11	09:06 AM	1057	550	380	46	31	1.82	1.28	41.66
3.	23-Sep-11	10:24 AM	1057	550	375	47	32	1.83	1.30	40.63
4.	23-Sep-11	01:00 PM	1057	550	375	47	32	1.83	1.30	40.63
5.	23-Sep-11	05:48 PM	1057	550	375	47	32	1.83	1.30	40.63
6.	24-Sep-11	11:05 AM	1057	550	373	47	32	1.84	1.31	40.22
7.	25-Sep-11	12:47 PM	1057	550	373	47	32	1.84	1.31	40.22
8.	26-Sep-11	11:14 AM	1056	550	371	48	33	1.84	1.32	39.81
9.	27-Sep-11	10:36 AM	1056	550	360	51	35	1.86	1.36	37.56
10	28-Sep-11	08:55 AM	1056	550	360	51	35	1.86	1.36	37.56
11	29-Sep-11	10:26 AM	1056	550	360	51	35	1.86	1.36	37.56

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Knight Piésold CONSULTING	DRAINED SETTLING TEST AND FALLING HEAD PERMEABILITY TEST	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	35%	Test Date:	9/23/2011-9/28/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =	278 g		d. Moisture Content (from drying test) =	**184.0** %
b. Initial Slurry Volume =	870 ml		e. Initial Slurry Bulk Density [(c-a)/b] =	1.28 g/cm³
c. Tare + Initial Slurry Weight =	1392 g		f. Weight of Water [(c-a)/(1+1/(d/100))] =	721 g
Time of Readings	08:42 AM	23-Sep-11	g. Weight of Solids [(c-a)/(1+d/100)] =	392 g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (before decant) (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Volume [B-C] (ml)	Drainage Volume Collected (ml)	Decanted Water Volume (ml)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)
1.	23-Sep-11	08:51 AM	1382	855	720	135	10	140	1.35	0.54
2.	23-Sep-11	09:04 AM	1231	705	540	165	20	174	1.46	0.73
3.	23-Sep-11	09:11 AM	1054	530	465	65	24	64	1.53	0.84
4.	23-Sep-11	09:30 AM	981	455	350	105	33	98	1.71	1.12
5.	23-Sep-11	10:23 AM	865	340	325	15	50	11	1.76	1.21
6.	23-Sep-11	12:51 PM	840	320	320	0	64	0	1.76	1.23
7.	23-Sep-11	05:39 PM	840	320	320	0	64	0	1.75	1.23
8.	24-Sep-11	11:00 AM	839	318	318	0	65	0	1.76	1.23
9.	25-Sep-11	12:35 PM	838	315	315	0	66	0	1.78	1.24
10	26-Sep-11	10:44 AM	838	315	315	0	66	0	1.75	1.24

Falling Head Permeability Test

	Data Readings, Ti (hours)	Initial Water Height, hi (cm)	Initial Solids Height, Hi (cm)	Finishing Time, Tf (hours)	Final Water Height, hf (cm)	Final Solids Height, Hf (cm)	Drainage Collected (ml)	Elapsed Time, T (hours)	Ave. Solids Thickness, H (cm)	Permeability k H/3600T*ln(hi/hf) (cm/sec)
1.	0.00	36.3	11.2	0.00	30.0	11.2	185	3.37	11.2	1.8E-04
2.	0.00	30.0	11.2	0.00	23.0	11.2	206	4.52	11.2	1.8E-04
3.	0.00	35.9	11.2	0.00	31.1	11.2	139	2.60	11.2	1.7E-04
4.	0.00	31.1	11.2	0.00	26.0	11.2	152	3.25	11.2	1.7E-04
									AVG.	1.8E-04

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Knight Piésold CONSULTING	DRAINED SETTLING TEST AND FALLING HEAD PERMEABILITY TEST	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	45%	Test Date:	9/23/2011-9/30/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =		185 g	d. Moisture Content (from drying test) =	**125.8**	%
b. Initial Slurry Volume =		655 ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.41	g/cm³
c. Tare + Initial Slurry Weight =		1106 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	513	g
Time of Readings	23-Sep-11	08:31 AM	g. Weight of Solids [(c-a)/(1+d/100)] =	408	g

On-going Readings

	Date of Reading	Time of Reading	A. Total Cylinder Weight (before decant) (g)	B. Total Cylinder Volume (ml)	C. Settled Slurry Volume (ml)	D. Water Volume [B-C] (ml)	E. Drainage Volume Collected (ml)	F. Decanted Water Volume (ml)	G. Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	H. Slurry Dry Density [g/C] (g/cm³)
1.	23-Sep-11	08:46 AM	1098	655	550	105	9	100	1.47	0.74
2.	23-Sep-11	09:01 AM	991	545	450	95	15	88	1.58	0.91
3.	23-Sep-11	09:09 AM	900	455	395	60	18	60	1.66	1.03
4.	23-Sep-11	09:32 AM	833	390	350	40	25	36	1.74	1.17
5.	23-Sep-11	10:20 AM	787	345	325	20	35	15	1.79	1.26
6.	23-Sep-11	12:48 PM	758	320	320	0	49	0	1.79	1.27
7.	23-Sep-11	05:34 PM	757	320	320	0	50	0	1.79	1.27
8.	24-Sep-11	11:00 AM	757	320	320	0	50	0	1.79	1.27
9.	25-Sep-11	12:33 PM	754	315	315	0	53	0	1.81	1.29
10.	26-Sep-11	10:42 AM	754	315	315	0	53	0	1.81	1.29

Falling Head Permeability Test

	Data Readings, Ti (hours)	Initial Water Height, hi (cm)	Initial Solids Height, Hi (cm)	Finishing Time, Tf (hours)	Final Water Height, hf (cm)	Final Solids Height, Hf (cm)	Drainage Collected (ml)	Elapsed Time, T (hours)	Ave. Solids Thickness, H (cm)	Permeability k H/3600T*ln(hi/hf) (cm/sec)
1.	0.00	29.1	12.9	0.00	15.2	12.8	412	17.88	12.9	1.3E-04
2.	0.00	31.1	12.8	0.00	26.5	12.8	138	4.48	12.8	1.3E-04
3.	0.00	26.5	12.8	0.00	16.1	12.8	308	14.30	12.8	1.2E-04
4.	0.00	31.1	12.8	0.00	28.4	12.8	98	3.25	12.8	9.9E-05
									AVG.	1.2E-04

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Knight Piésold CONSULTING	DRAINED SETTLING TEST AND FALLING HEAD PERMEABILITY TEST	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	55%	Test Date:	9/23/2011-9/28/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =	186	g	d. Moisture Content (from drying test) =	**76.7**	%
b. Initial Slurry Volume =	600	ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.55	g/cm³
c. Tare + Initial Slurry Weight =	1118	g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	405	g
Time of Readings 23-Sep-11	08:17 AM		g. Weight of Solids [(c-a)/(1+d/100)] =	528	g

On-going Readings

	Date of Reading	Time of Reading	A. Total Cylinder Weight (before decant) (g)	B. Total Cylinder Volume (ml)	C. Settled Slurry Volume (ml)	D. Water Volume [B-C] (ml)	E. Drainage Volume Collected (ml)	F. Decanted Water Volume (ml)	G. Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	H. Slurry Dry Density [g/C] (g/cm³)
1.	23-Sep-11	08:29 AM	1111	590	550	40	8	36	1.61	0.96
2.	23-Sep-11	08:56 AM	1061	540	420	120	21	107	1.80	1.26
3.	23-Sep-11	10:17 AM	931	410	395	15	44	9	1.85	1.34
4.	23-Sep-11	12:43 PM	908	390	390	0	58	0	1.85	1.35
5.	23-Sep-11	05:31 PM	907	390	390	0	59	0	1.85	1.35
6.	24-Sep-11	11:00 AM	906	387	387	0	60	0	1.86	1.36
7.	25-Sep-11	12:31 PM	901	382	382	0	65	0	1.87	1.38
8.	26-Sep-11	10:40 AM	901	381	381	0	65	0	1.88	1.38

Falling Head Permeability Test

	Data Readings, Ti (hours)	Initial Water Height, hi (cm)	Initial Solids Height, Hi (cm)	Finishing Time, Tf (hours)	Final Water Height, hf (cm)	Final Solids Height, Hf (cm)	Drainage Collected (ml)	Elapsed Time, T (hours)	Ave. Solids Thickness, H (cm)	Permeability k H/3600T*ln(hi/hf) (cm/sec)
1.	0.00	30.4	10.6	0.00	17.1	10.6	380	17.90	10.6	9.5E-05
2.	0.00	32.4	10.6	0.00	28.4	10.6	123	4.50	10.6	8.6E-05
3.	0.00	28.4	10.6	0.00	18.6	10.6	290	14.37	10.6	8.7E-05
4.	0.00	18.6	10.6	0.00	17.3	10.6	39	2.65	10.6	8.1E-05
5.	0.00	17.3	10.6	0.00	15.8	10.6	45	3.27	10.6	8.2E-05
6.	0.00	21.8	10.6	0.00	19.9	10.6	52	3.10	10.6	8.7E-05
									AVG.	8.6E-05

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FIGURE 2.1

YELLOWHEAD MINING, INC.
HARPER CREEK
ROTL

TAILINGS DEPOSITION METHOD VS. DRY DENSITY
TAILINGS COMPOSITE

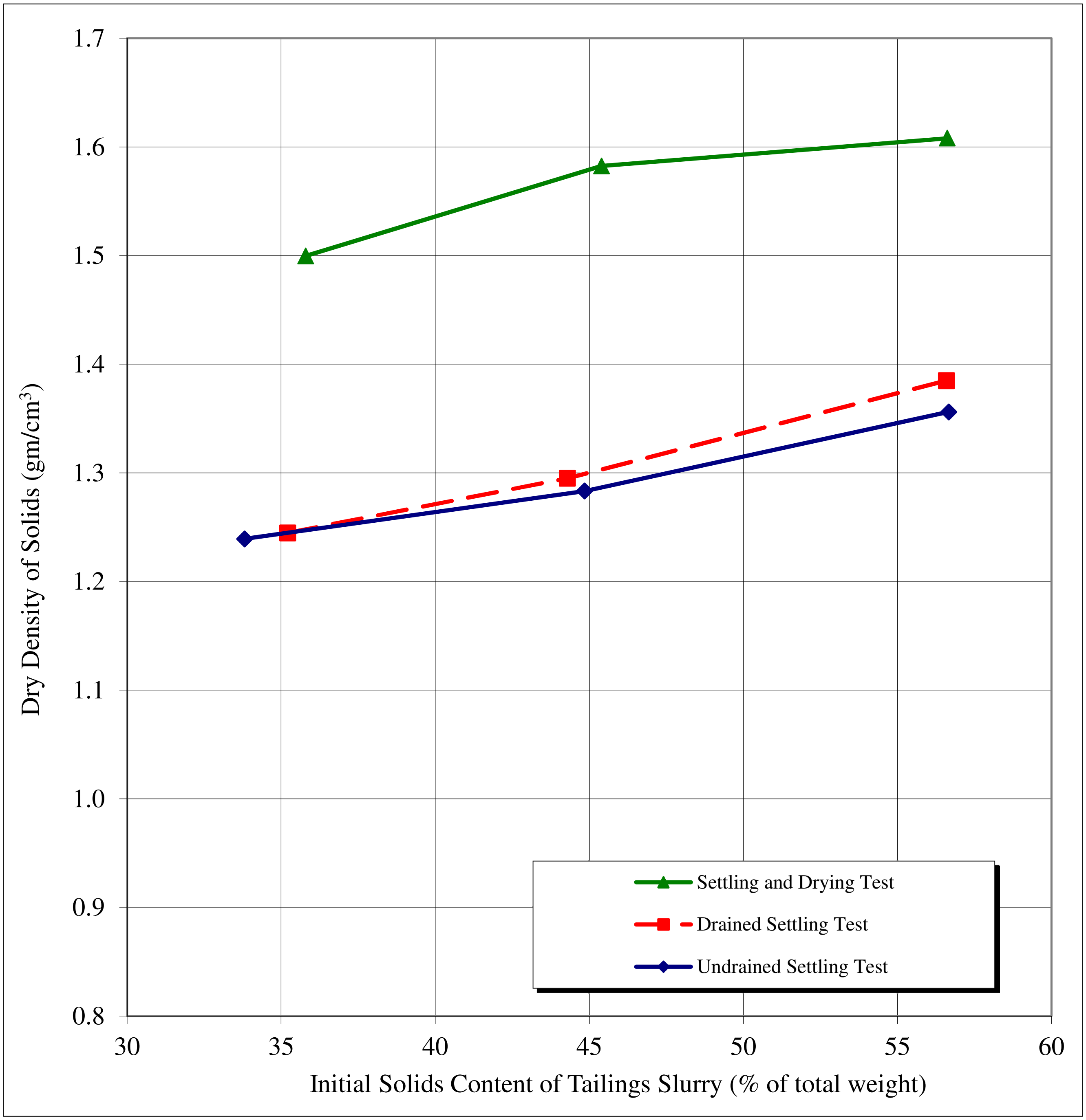

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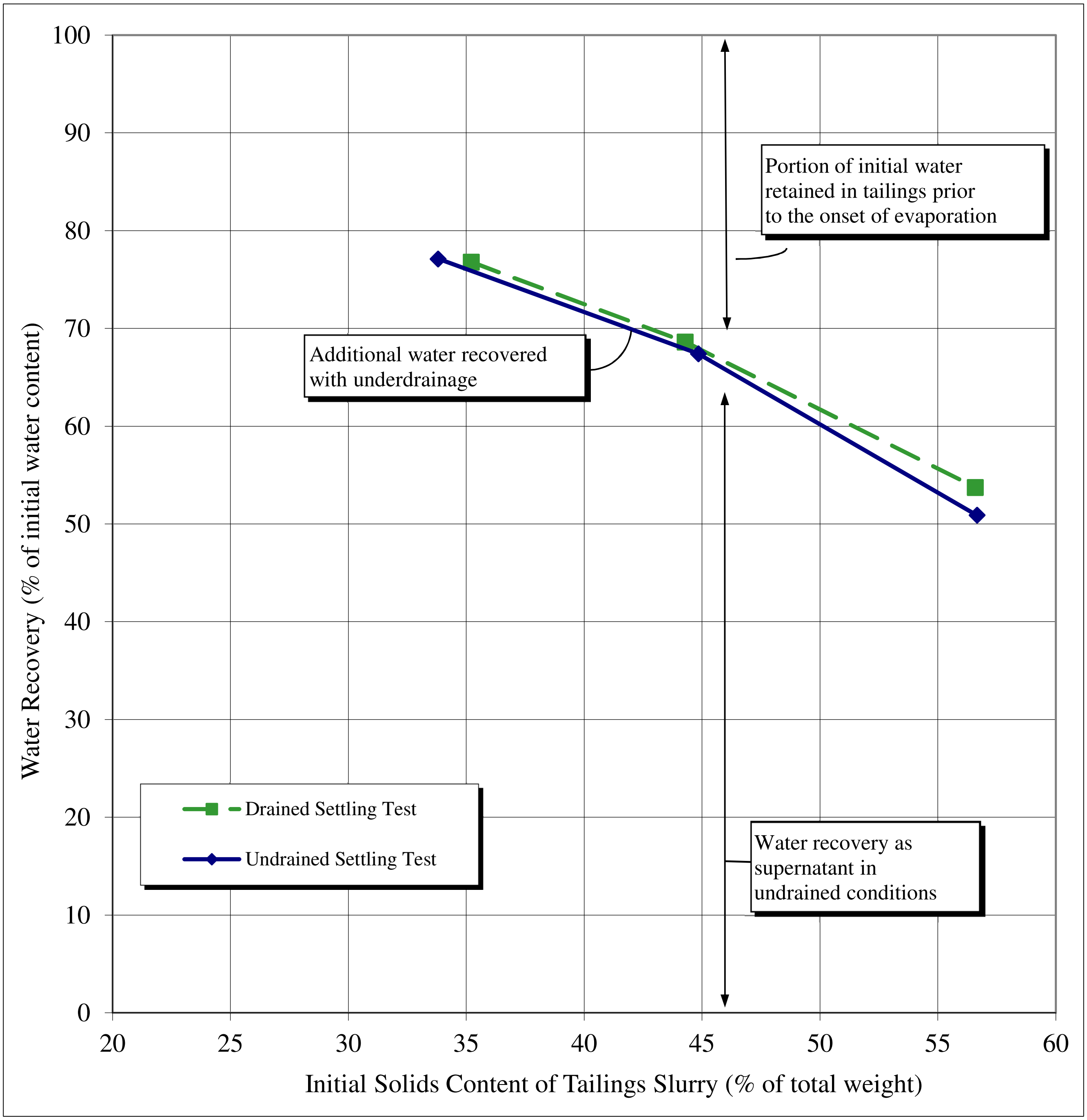
FIGURE 2.2

YELLOWHEAD MINING, INC.
HARPER CREEK
ROTL

TAILINGS DEPOSITION METHOD VS. WATER RECOVERY
TAILINGS COMPOSITE



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Knight Piésold CONSULTING	FLOW CONE VISCOSITY & SOLIDS CONTENT DETERMINATION	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	35%	45%	55%
	CuROTL	Location:	CuROTL		
Test Date:	22-Sep-11 10:57 AM	Tested By:	jhk/jdb		

FLOW CONE WATER DISCHARGE CALIBRATION

Trial No.	1.	2.	3.	Average
Time for complete water discharge (sec)				

FLOW CONE TAILINGS DISCHARGE TESTS

Trial No.	1	2	3	
Time for tailings discharge (sec)				

PERCENT SOLIDS AND MOISTURE CONTENT DETERMINATION

	Trial No.	1.	2.	3.	4.	Average
	Origin of Sample	Target 35%	Target 45%	Target 55%		
	Tare No.					
a.	Tare Weight (g)	375	395	396		
b.	Tare + Wet Sample Weight (g)	1288	1181	1251		
c.	Tare + Dry Sample Weight (g)	707	762	881		
	Drying Time - From	22-Sep-11	22-Sep-11	22-Sep-11		
	- To	23-Sep-11	23-Sep-11	23-Sep-11		
d.	Moisture Loss [b-c] (g)	580.5	418.8	370		
e.	Dry Sample Weight [c-a] (g)	331.9	367.1	485.4		
	Initial Parameters from Previous Test					
f.	Tare (Cylinder or Beaker) Weight (g)	375	395	396		
g.	Tare + Initial Slurry Weight (g)	1288	1181	1251		
h.	Initial Slurry Weight (g)	912	786	855		
i.	True Moisture Loss [h-e] (g)	581	419	370		
	PERCENT SOLIDS [e/h*100] (%)	36.4	46.7	56.7		
	MOISTURE CONTENT [i/e*100] (%)	174.9	114.1	76.2		

Comments: The tailings settled too quickly to perform the flow cone viscosity tests.

Solids content percentages reported in trials 1, 2 & 3 will vary slightly for each specific test.

Knight Piésold CONSULTING	UNDRAINED SETTLING TEST	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	35%	Test Date:	9/23/2011-10/5/2011
	CuROTL	Location:	CuROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =	217	g	d. Moisture Content (from drying test) =	**186.1**	%
b. Initial Slurry Volume =	775	ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.29	g/cm³
c. Tare + Initial Slurry Weight =	1217	g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	650	g
Time of Readings 23-Sep-11	09:46 AM		g. Weight of Solids [(c-a)/(1+d/100)] =	349	g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Recovery [(B-C)/f] (%)	Volume Reduction of Solids [1-C/b] (%)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)	Moisture Content [(f-(B-C))/g] (%)
1.	23-Sep-11	09:58 AM	1217	775	590	28	24	1.38	0.59	133.10
2.	23-Sep-11	10:09 AM	1217	775	450	50	42	1.50	0.78	93.03
3.	23-Sep-11	10:25 AM	1217	775	345	66	55	1.65	1.01	62.98
4.	23-Sep-11	01:01 PM	1217	775	305	72	61	1.74	1.15	51.53
5.	23-Sep-11	05:49 PM	1217	775	300	73	61	1.75	1.16	50.10
6.	24-Sep-11	11:05 AM	1217	775	300	73	61	1.75	1.16	50.10
7.	25-Sep-11	12:48 PM	1217	775	300	73	61	1.75	1.16	50.10
8.	26-Sep-11	11:19 AM	1216	775	300	73	61	1.75	1.16	50.10
9.	27-Sep-11	10:38 AM	1216	775	295	74	62	1.76	1.18	48.67
10.	28-Sep-11	08:57 AM	1216	775	295	74	62	1.76	1.18	48.67

Knight Piésold CONSULTING

UNDRAINED SETTLING TEST

Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	45%	Test Date:	9/23/2011-10/5/2011
	CuROTL	Location:	CuROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =	186 g		d. Moisture Content (from drying test) =	**115.9** %
b. Initial Slurry Volume =	735 ml		e. Initial Slurry Bulk Density [(c-a)/b] =	1.41 g/cm³
c. Tare + Initial Slurry Weight =	1222 g		f. Weight of Water [(c-a)/(1+1/(d/100))] =	556 g
Time of Readings	23-Sep-11	09:39 AM	g. Weight of Solids [(c-a)/(1+d/100)] =	480 g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Recovery [(B-C)/f] (%)	Volume Reduction of Solids [1-C/b] (%)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)	Moisture Content [(f-(B-C))/g] (%)
1.	23-Sep-11	09:58 AM	1222	735	590	26	20	1.51	0.81	85.72
2.	23-Sep-11	10:09 AM	1222	730	490	43	33	1.63	0.98	65.93
3.	23-Sep-11	10:25 AM	1222	730	435	53	41	1.70	1.10	54.47
4.	23-Sep-11	01:01 PM	1222	730	410	58	44	1.75	1.17	49.26
5.	23-Sep-11	05:49 PM	1222	730	410	58	44	1.75	1.17	49.26
6.	24-Sep-11	11:05 AM	1222	730	400	59	46	1.77	1.20	47.18
7.	25-Sep-11	12:48 PM	1222	730	392	61	47	1.78	1.22	45.51
8.	26-Sep-11	11:18 AM	1222	730	390	61	47	1.78	1.23	45.10
9.	27-Sep-11	10:38 AM	1222	730	383	62	48	1.80	1.25	43.64
10.	28-Sep-11	08:57 AM	1222	730	383	62	48	1.80	1.25	43.64

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Knight Piésold CONSULTING

UNDRAINED SETTLING TEST

Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	55%	Test Date:	9/23/2011-10/5/2011
	CuROTL	Location:	CuROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =	217 g	d. Moisture Content (from drying test) =	**76.0** %
b. Initial Slurry Volume =	505 ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.55 g/cm³
c. Tare + Initial Slurry Weight =	1002 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	339 g
Time of Readings 23-Sep-11	09:26 AM	g. Weight of Solids [(c-a)/(1+d/100)] =	446 g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Recovery [(B-C)/f] (%)	Volume Reduction of Solids [1-C/b] (%)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)	Moisture Content [(f-(B-C))/g] (%)
1.	23-Sep-11	09:36 AM	1002	505	465	12	8	1.60	0.96	67.00
2.	23-Sep-11	10:09 AM	1002	500	365	40	28	1.78	1.22	45.69
3.	23-Sep-11	10:25 AM	1002	500	360	41	29	1.79	1.24	44.57
4.	23-Sep-11	01:01 PM	1002	500	360	41	29	1.79	1.24	44.57
5.	23-Sep-11	05:49 PM	1002	500	360	41	29	1.79	1.24	44.57
6.	24-Sep-11	11:05 AM	1002	500	358	42	29	1.80	1.25	44.12
7.	25-Sep-11	12:48 PM	1002	500	358	42	29	1.80	1.25	44.12
8.	26-Sep-11	11:17 AM	1002	500	355	43	30	1.80	1.26	43.45
9.	27-Sep-11	10:38 AM	1002	500	353	43	30	1.81	1.26	43.00
10	28-Sep-11	08:57 AM	1002	500	353	43	30	1.81	1.26	43.00

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Knight Piésold CONSULTING	DRAINED SETTLING TEST AND FALLING HEAD PERMEABILITY TEST	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	35%	Test Date:	9/23/2011-9/28/2011
	CuROTL	Location:	CuROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =		186 g	d. Moisture Content (from drying test) =	**180.3**	%
b. Initial Slurry Volume =		810 ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.29	g/cm³
c. Tare + Initial Slurry Weight =		1232 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	673	g
Time of Readings	23-Sep-11	09:48 AM	g. Weight of Solids [(c-a)/(1+d/100)] =	373	g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (before decant) (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Volume [B-C] (ml)	Drainage Volume Collected (ml)	Decanted Water Volume (ml)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)
1.	23-Sep-11	09:56 AM	1216	790	635	155	15	169	1.38	0.59
2.	23-Sep-11	10:07 AM	1034	610	465	145	28	143	1.51	0.80
3.	23-Sep-11	10:34 AM	872	450	330	120	48	119	1.72	1.13
4.	23-Sep-11	12:59 PM	730	310	310	0	70	0	1.76	1.20
5.	23-Sep-11	05:47 PM	729	305	305	0	71	0	1.78	1.22
6.	24-Sep-11	11:05 AM	729	305	305	0	72	0	1.78	1.22
7.	25-Sep-11	12:46 PM	727	300	300	0	73	0	1.80	1.24
8.	26-Sep-11	10:58 AM	727	300	300	0	74	0	1.80	1.24

Falling Head Permeability Test

	Data Readings, Ti (hours)	Initial Water Height, hi (cm)	Initial Solids Height, Hi (cm)	Finishing Time, Tf (hours)	Final Water Height, hf (cm)	Final Solids Height, Hf (cm)	Drainage Collected (ml)	Elapsed Time, T (hours)	Ave. Solids Thickness, H (cm)	Permeability k H/3600T*ln(hi/hf) (cm/sec)
1.	0.00	36.1	10.3	0.00	30.5	10.3	171	3.27	10.3	1.5E-04
2.	0.00	30.5	10.3	0.00	23.9	10.3	203	4.52	10.3	1.5E-04
3.	0.00	35.4	10.3	0.00	31.5	10.3	119	2.47	10.3	1.4E-04
4.	0.00	31.5	10.3	0.00	26.8	10.3	141	3.27	10.3	1.4E-04
									AVG.	1.4E-04

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Knight Piésold CONSULTING

DRAINED SETTLING TEST AND FALLING HEAD PERMEABILITY TEST

Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	45%	Test Date:	9/23/2011-9/28/2011
	CuROTL	Location:	CuROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =	272 g		d. Moisture Content (from drying test) =	**116.1** %
b. Initial Slurry Volume =	735 ml		e. Initial Slurry Bulk Density [(c-a)/b] =	1.41 g/cm³
c. Tare + Initial Slurry Weight =	1311 g		f. Weight of Water [(c-a)/(1+1/(d/100))] =	558 g
Time of Readings	23-Sep-11	09:40 AM	g. Weight of Solids [(c-a)/(1+d/100)] =	481 g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (before decant) (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Volume [B-C] (ml)	Drainage Volume Collected (ml)	Decanted Water Volume (ml)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)
1.	23-Sep-11	09:52 AM	1295	715	625	90	15	86	1.49	0.77
2.	23-Sep-11	10:04 AM	1200	620	530	90	25	89	1.58	0.91
3.	23-Sep-11	10:31 AM	1095	520	400	120	42	111	1.76	1.20
4.	23-Sep-11	12:57 PM	958	380	380	0	67	0	1.81	1.26
5.	23-Sep-11	05:44 PM	957	375	375	0	68	0	1.83	1.28
6.	24-Sep-11	11:00 AM	957	370	370	0	68	0	1.85	1.30
7.	25-Sep-11	12:44 PM	955	370	370	0	70	0	1.84	1.30
8.	26-Sep-11	10:54 AM	945	370	370	0	80	0	1.82	1.30

Falling Head Permeability Test

	Data Readings, Ti (hours)	Initial Water Height, hi (cm)	Initial Solids Height, Hi (cm)	Finishing Time, Tf (hours)	Final Water Height, hf (cm)	Final Solids Height, Hf (cm)	Drainage Collected (ml)	Elapsed Time, T (hours)	Ave. Solids Thickness, H (cm)	Permeability k H/3600T*ln(hi/hf) (cm/sec)
1.	0.00	37.2	13.2	0.00	32.1	13.2	149	3.28	13.2	1.6E-04
2.	0.00	32.1	13.2	0.00	26.0	13.2	180	4.50	13.2	1.7E-04
3.	0.00	36.6	13.2	0.00	32.8	13.2	112	2.50	13.2	1.6E-04
4.	0.00	32.8	13.2	0.00	28.3	13.2	131	3.27	13.2	1.7E-04
									AVG.	1.7E-04

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Knight Piésold CONSULTING

DRAINED SETTLING TEST AND FALLING HEAD PERMEABILITY TEST

Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	55%	Test Date:	9/23/2011-9/28/2011
	CuROTL	Location:	CuROTL	Tested By:	jdb/jhk

Initial Parameters

a. Cylinder (Tare) Weight =		186 g	d. Moisture Content (from drying test) =	**77.8**	%
b. Initial Slurry Volume =		605 ml	e. Initial Slurry Bulk Density [(c-a)/b] =	1.56	g/cm³
c. Tare + Initial Slurry Weight =		1128 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	412	g
Time of Readings	23-Sep-11	09:26 AM	g. Weight of Solids [(c-a)/(1+d/100)] =	530	g

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.
	Date of Reading	Time of Reading	Total Cylinder Weight (before decant) (g)	Total Cylinder Volume (ml)	Settled Slurry Volume (ml)	Water Volume [B-C] (ml)	Drainage Volume Collected (ml)	Decanted Water Volume (ml)	Slurry Bulk Density [(A-a-(B-C))/C] (g/cm³)	Slurry Dry Density [g/C] (g/cm³)
1.	23-Sep-11	09:34 AM	1124	600	570	30	4	28	1.59	0.93
2.	23-Sep-11	10:01 AM	1090	560	430	130	11	130	1.80	1.23
3.	23-Sep-11	10:28 AM	955	430	410	20	16	18	1.83	1.29
4.	23-Sep-11	12:54 PM	917	395	395	0	36	0	1.85	1.34
5.	23-Sep-11	05:42 PM	912	390	390	0	41	0	1.86	1.36
6.	24-Sep-11	11:00 AM	911	390	390	0	42	0	1.86	1.36
7.	25-Sep-11	12:41 PM	909	385	385	0	45	0	1.88	1.38
8.	26-Sep-11	10:52 AM	908	385	385	0	45	0	1.87	1.38

Falling Head Permeability Test

	Data Readings, Ti (hours)	Initial Water Height, hi (cm)	Initial Solids Height, Hi (cm)	Finishing Time, Tf (hours)	Final Water Height, hf (cm)	Final Solids Height, Hf (cm)	Drainage Collected (ml)	Elapsed Time, T (hours)	Ave. Solids Thickness, H (cm)	Permeability k H/3600T*ln(hi/hf) (cm/sec)
1.	0.00	32.1	13.1	0.00	21.7	13.1	311	18.00	13.1	7.9E-05
2.	0.00	21.7	13.1	0.00	20.1	13.1	44	3.33	13.1	8.4E-05
3.	0.00	20.1	13.1	0.00	18.2	13.1	57	4.50	13.1	8.0E-05
4	0.00	26.2	13.1	0.00	24.9	13.1	40	2.55	13.1	7.3E-05
5	0.00	24.9	13.1	0.00	23.3	13.1	49	3.27	13.1	7.4E-05
									AVG.	7.8E-05

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FIGURE 2.1

YELLOWHEAD MINING, INC.
HARPER CREEK
CuROTL

TAILINGS DEPOSITION METHOD VS. DRY DENSITY
TAILINGS COMPOSITE

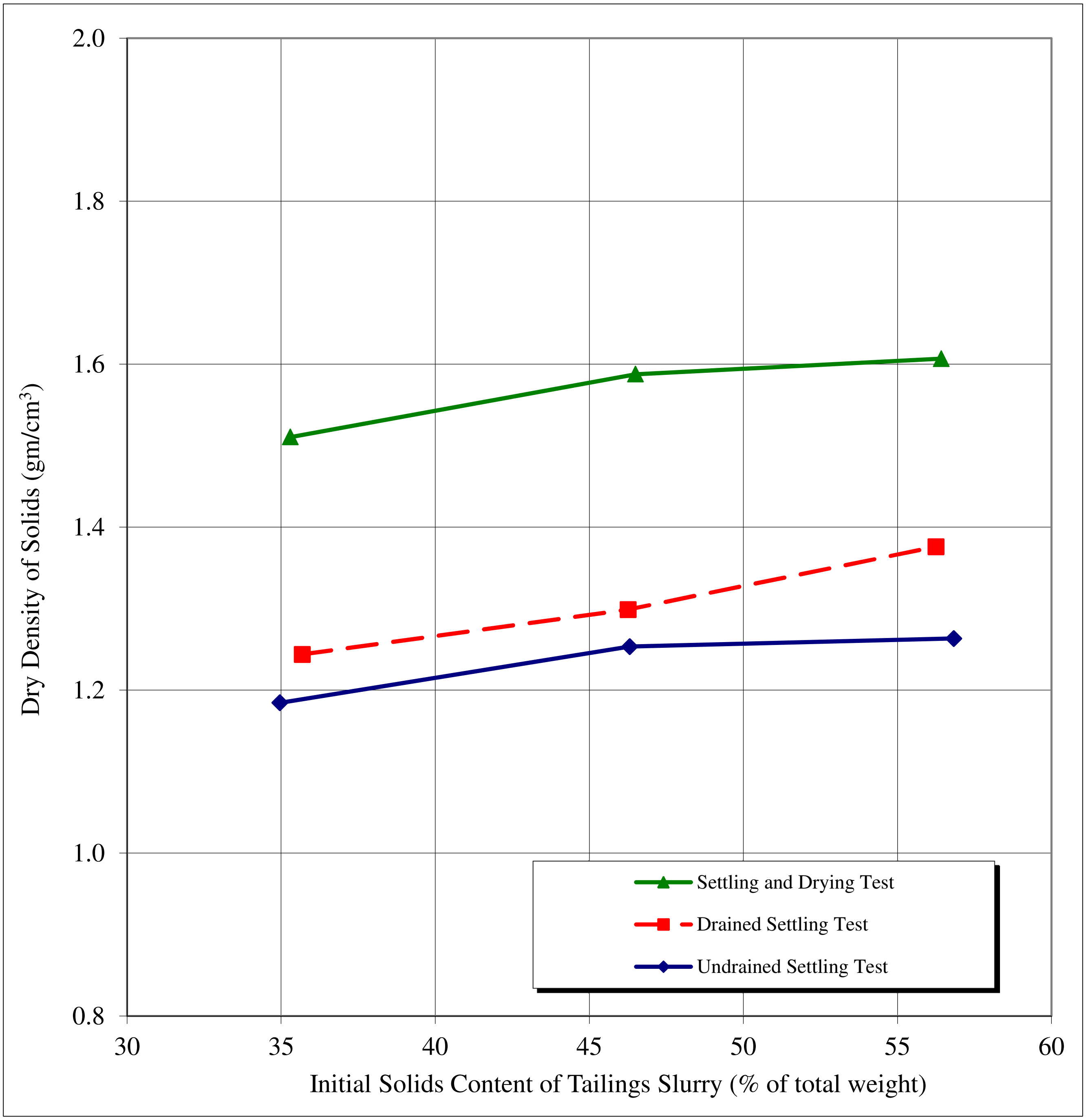

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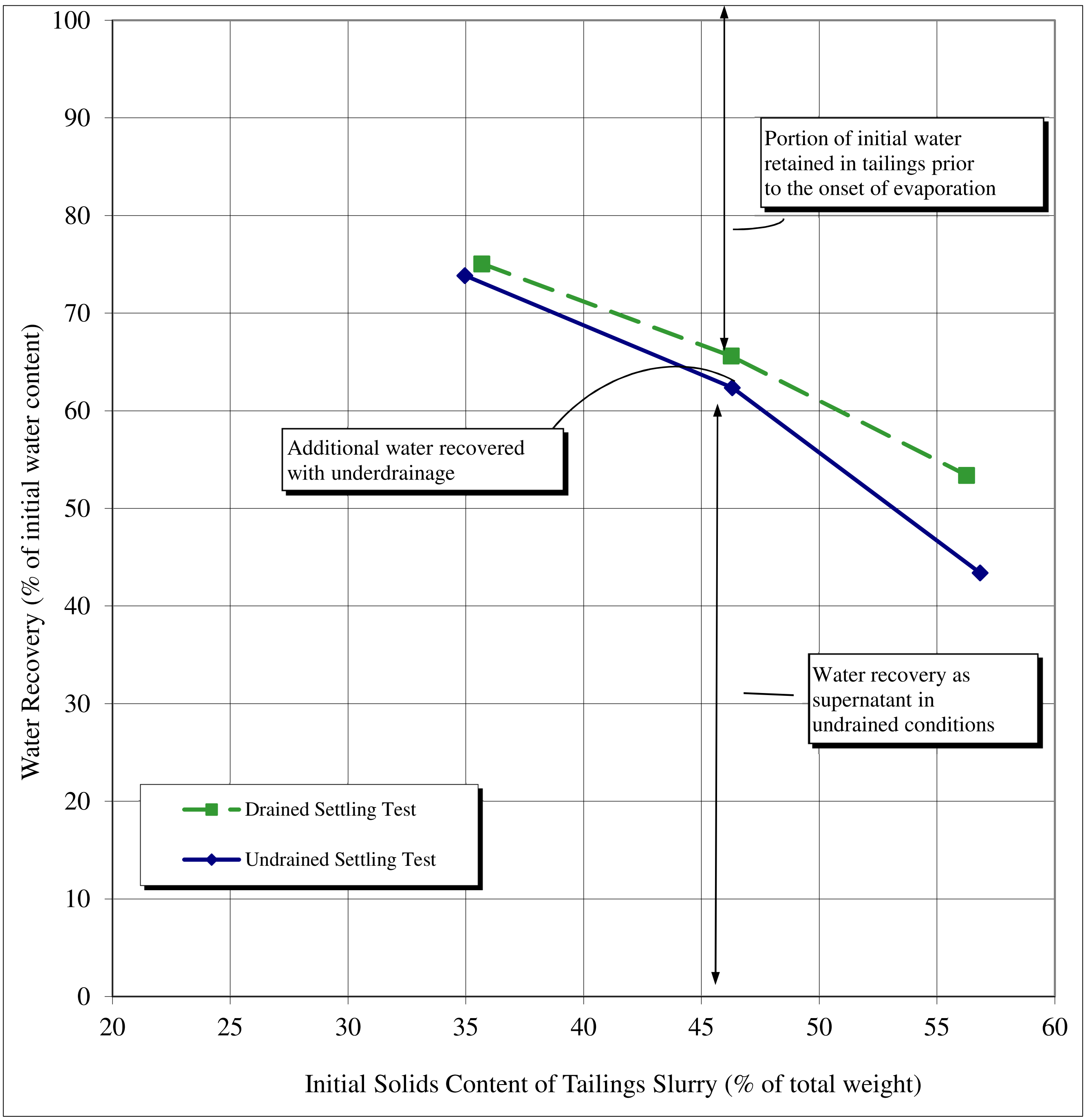
FIGURE 2.2

YELLOWHEAD MINING, INC.
HARPER CREEK
CuROTL

TAILINGS DEPOSITION METHOD VS. WATER RECOVERY
TAILINGS COMPOSITE



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SLURRY CONSOLIDOMETER TEST RESULTS
HARPER CREEK
ROTL
VA101-458.04

		Consolidation Test Data - Double Ended Drainage								
STRESS	psi	Free	0.5	2	5	10	20	40	80	120
	psf		72	288	720	1440	2880	5760	11520	17280
	kPa		3	14	34	69	138	276	552	827
Volume Change, cc		446.8	7.8	8.9	7.3	4.5	6.5	7.6	7.7	3.9
Cumulative Volume Change, cc			454.6	463.5	470.8	475.3	481.8	489.4	497.1	501.0
Consolidated Slurry Volume, cm^3		327.7	319.9	311.0	303.7	299.2	292.8	285.2	277.5	273.6
Settled Void Ratio, e_c		1.101	1.051	0.994	0.947	0.918	0.877	0.828	0.779	0.754
Dry Unit Weight, pcf		82.9	84.9	87.4	89.5	90.8	92.8	95.3	97.9	99.3
Slurry Height, cm		3.931	3.838	3.731	3.643	3.589	3.512	3.421	3.329	3.282
Mean Slurry Height, H^2 mm			377.3	358.0	339.9	327.0	315.2	300.4	284.7	273.1
Total Ht. Change, cm		5.360	5.454	5.560	5.648	5.702	5.779	5.870	5.963	6.010
Cumulative change in height, in.		2.110	2.147	2.189	2.224	2.245	2.275	2.311	2.348	2.366
Individual change in height, in.		2.110	0.037	0.042	0.034	0.021	0.030	0.036	0.036	0.018

SAMPLE PARAMETERS

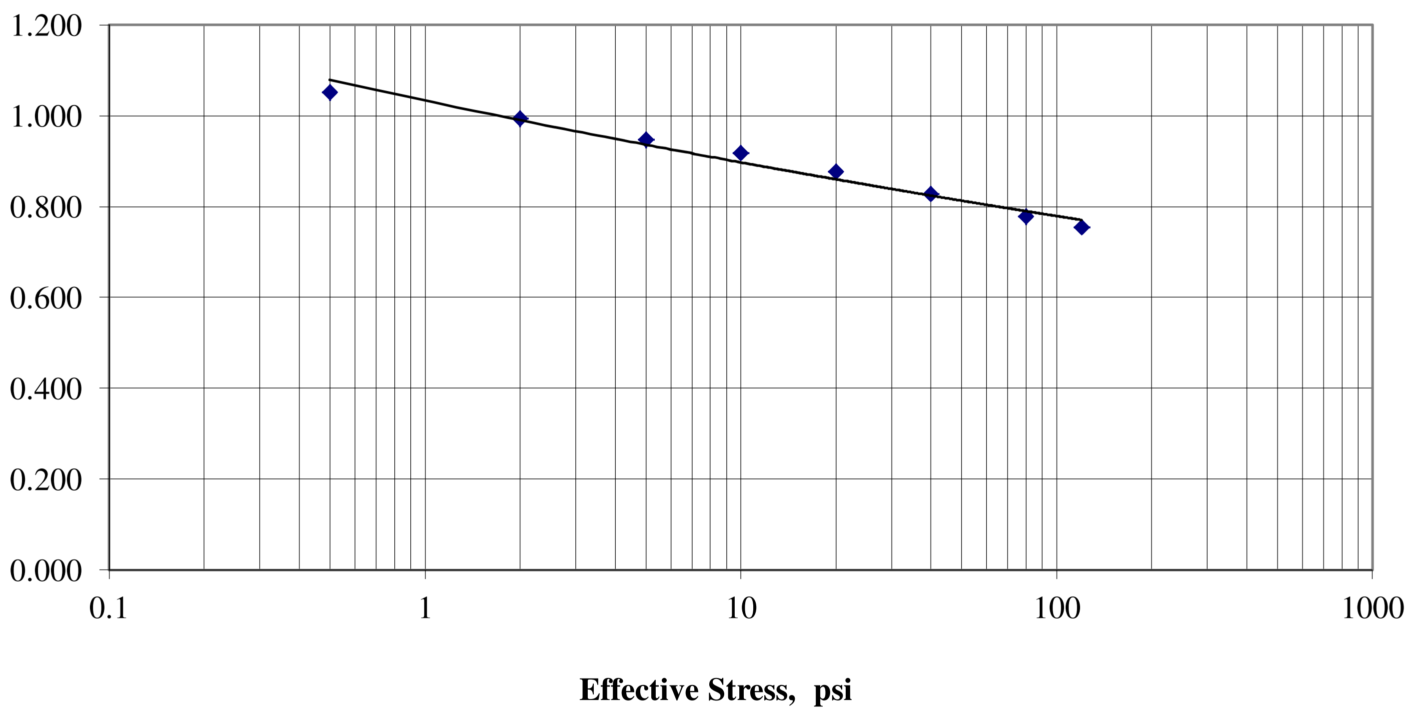
Specific Gravity	2.790	
Slurry Mass w/tare, g	1348.40	
Tare, g	282.10	
Slurry Mass, g	1066.30	
Height of Slurry (in./cm)	3.658	9.29
Volume of Slurry, cc		774.52
Solids Content, %	40.8	
Wt. of Solids, g	435.2	
Ht. of Solids, cm		1.871
Volume of Solids, cc	155.99	
Intial Void Ratio, e	3.965	

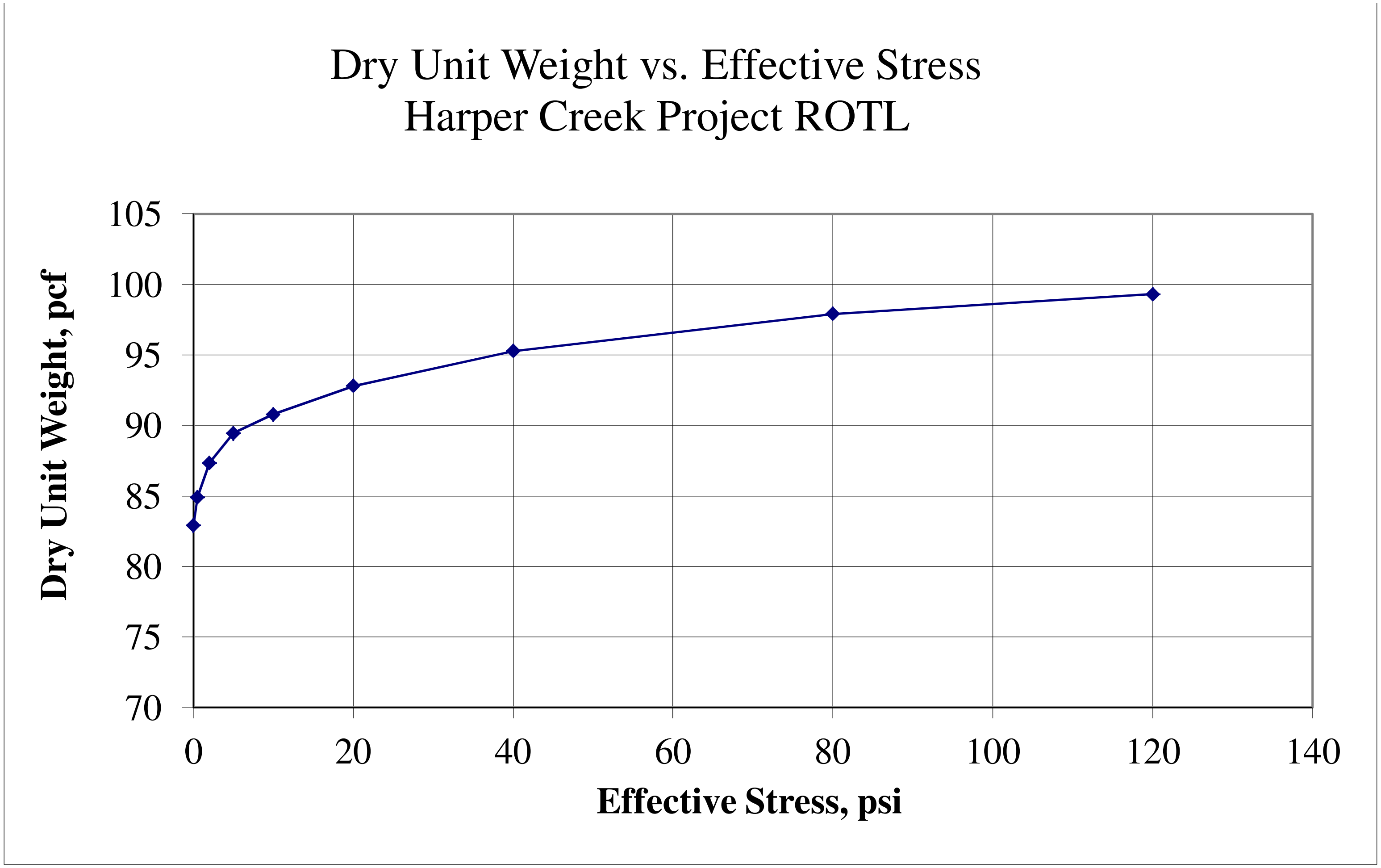
Ring Diameter (in./cm)	4.056	10.30
Ring Area, cm^2		83.359
Ring Height (in./cm)	4.058	10.31
Ring Volume, cm^3		859.21

Post Test Specimen Data

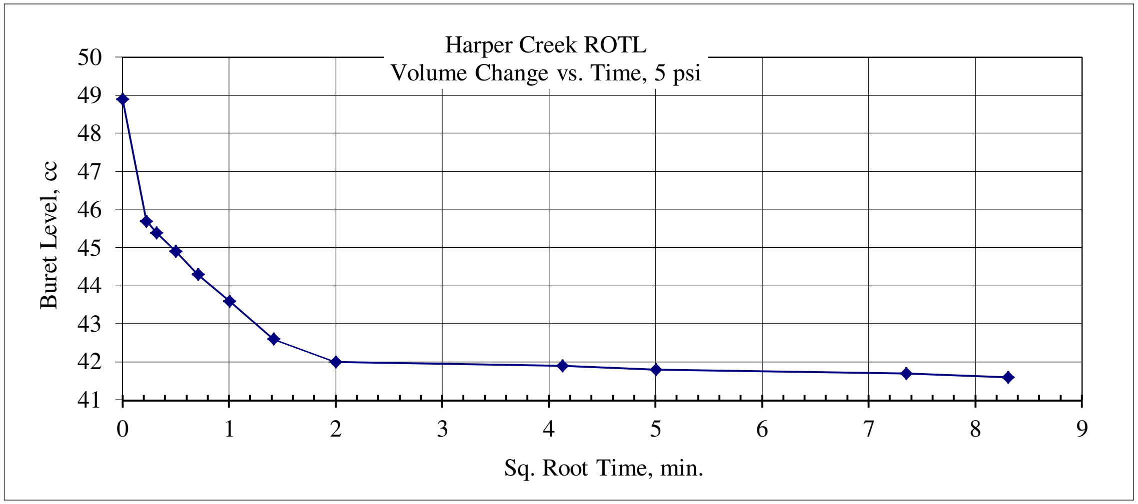
Wet + Tare, g	718.40
Dry + Tare, g	551.20
Tare, g	116.00
Wt. of Water, g	144.80
Wt. of Dry Solids, g	435.20
Moisture Content, %	33.27

ROTL

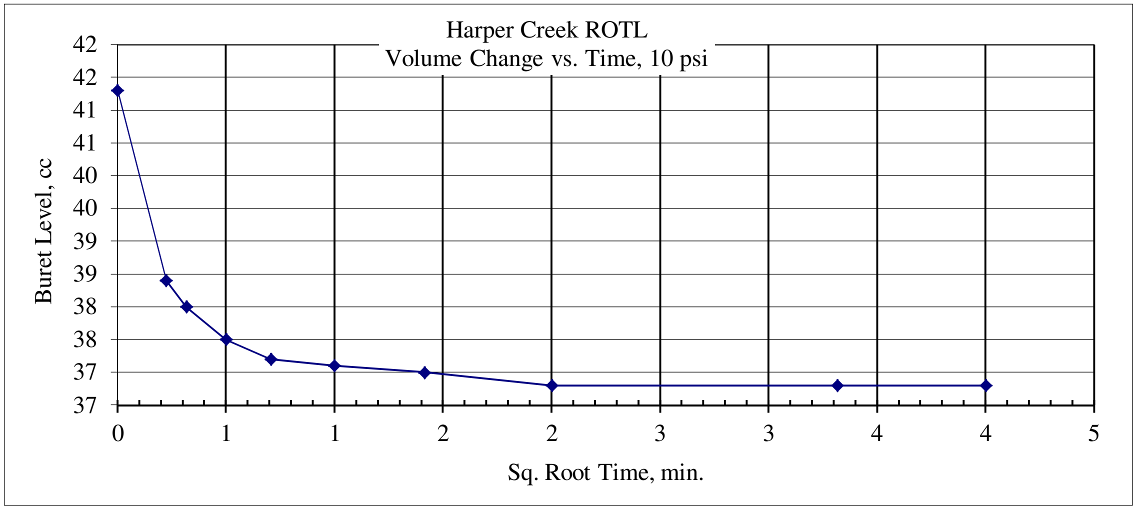

1.200
1.000
0.800
0.600
0.400
0.200
0.000
0.1
1
10
100
1000
Effective Stress, psi


Dry Unit Weight vs. Effective Stress
Harper Creek Project ROTL
Dry Unit Weight, pcf
105
100
95
90
85
80
75
70
0
20
40
60
80
100
120
140
Effective Stress, psi

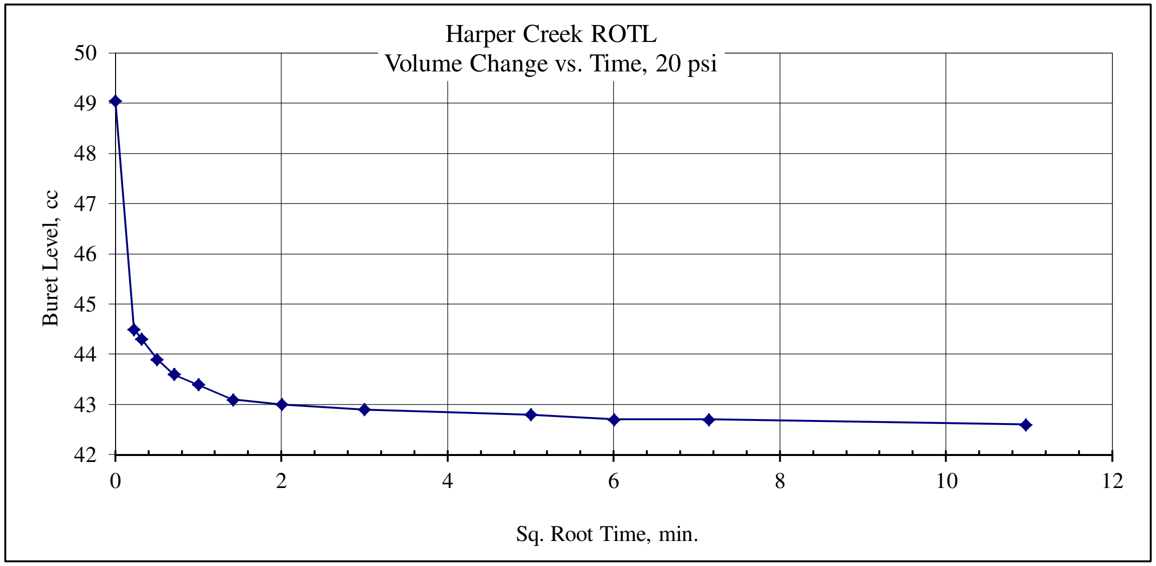
elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	48.9	1.1
0.05	0.22	45.7	4.3
0.10	0.32	45.4	4.6
0.25	0.50	44.9	5.1
0.50	0.71	44.3	5.7
1.00	1.00	43.6	6.4
2.00	1.41	42.6	7.4
4.00	2.00	42.0	8.0
17.00	4.12	41.9	8.1
25.00	5.00	41.8	8.2
54.00	7.35	41.7	8.3
69.00	8.31	41.6	8.4
		41.6	8.4



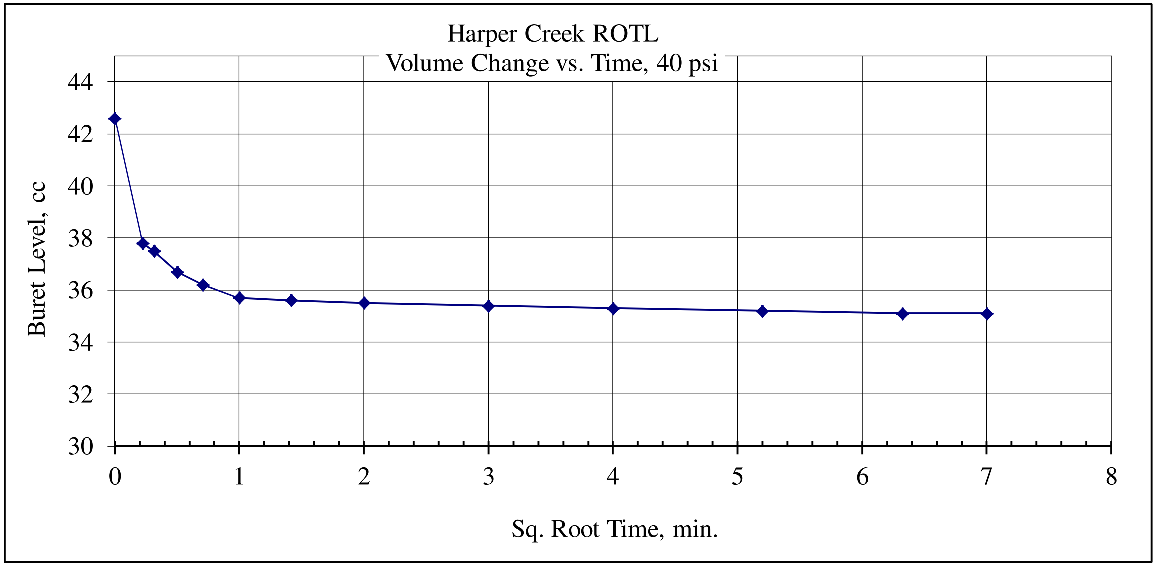
elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	41.3	8.7
0.05	0.22	38.4	11.6
0.10	0.32	38.0	12.0
0.25	0.50	37.5	12.5
0.50	0.71	37.2	12.8
1.00	1.00	37.1	12.9
2.00	1.41	37.0	13.0
4.00	2.00	36.8	13.2
11.00	3.32	36.8	13.2
16.00	4.00	36.8	13.2



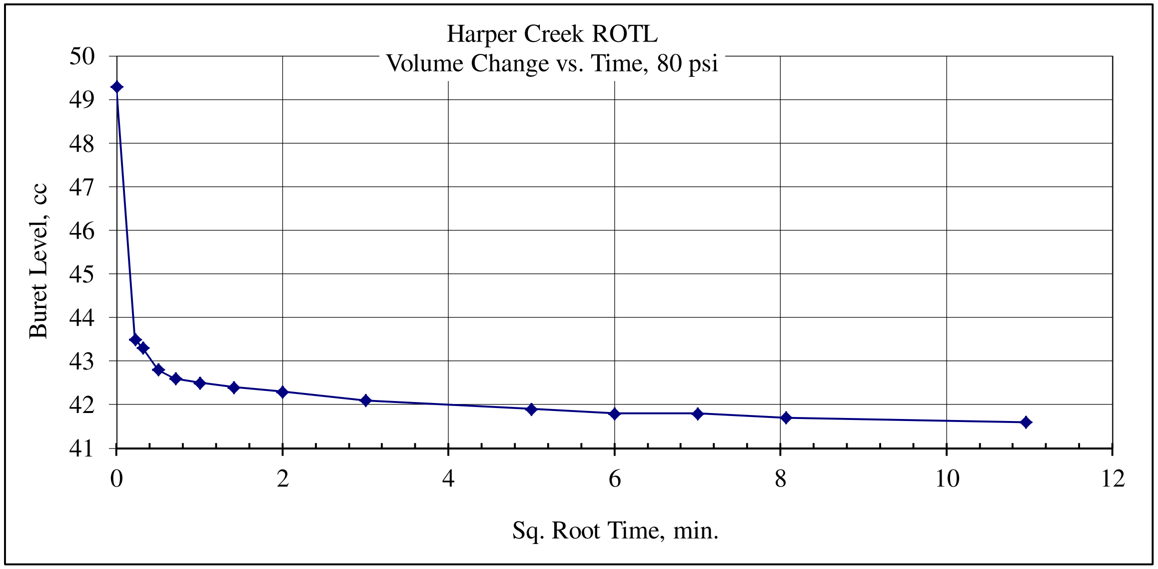
elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	49.1	1.0
0.05	0.22	44.5	5.5
0.10	0.32	44.3	5.7
0.25	0.50	43.9	6.1
0.50	0.71	43.6	6.4
1.00	1.00	43.4	6.6
2.00	1.41	43.1	6.9
4.00	2.00	43.0	7.0
9.00	3.00	42.9	7.1
25.00	5.00	42.8	7.2
36.00	6.00	42.7	7.3
51.00	7.14	42.7	7.3
120.00	10.95	42.6	7.4



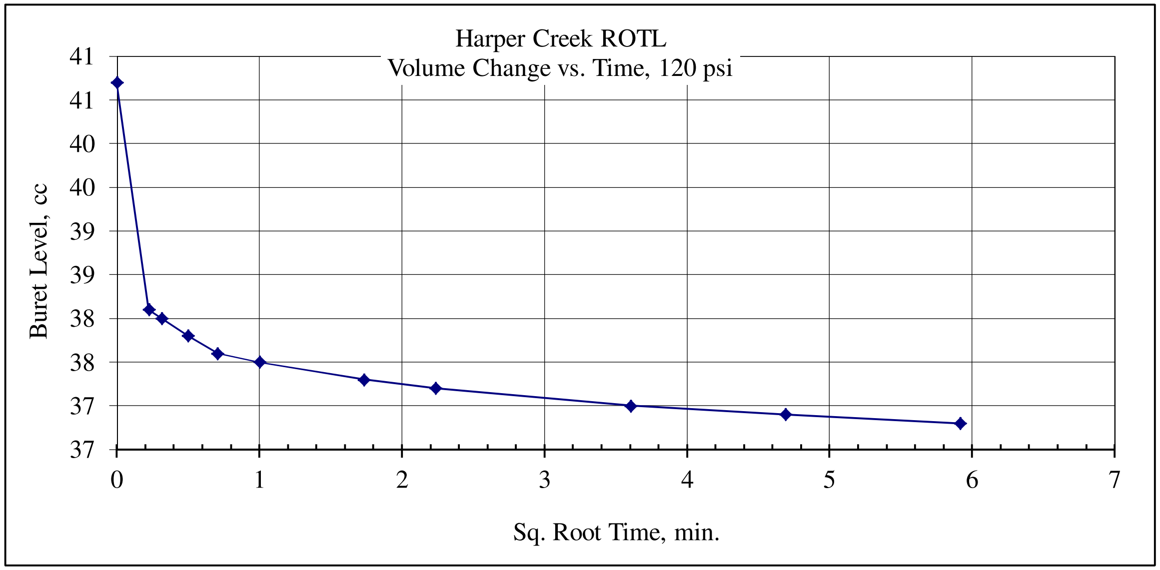
elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	42.6	7.4
0.05	0.22	37.8	12.2
0.10	0.32	37.5	12.5
0.25	0.50	36.7	13.3
0.50	0.71	36.2	13.8
1.00	1.00	35.7	14.3
2.00	1.41	35.6	14.4
4.00	2.00	35.5	14.5
9.00	3.00	35.4	14.6
16.00	4.00	35.3	14.7
27.00	5.20	35.2	14.8
40.00	6.32	35.1	14.9
49.00	7.00	35.1	14.9



elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	49.3	0.7
0.05	0.22	43.5	6.5
0.10	0.32	43.3	6.7
0.25	0.50	42.8	7.2
0.50	0.71	42.6	7.4
1.00	1.00	42.5	7.5
2.00	1.41	42.4	7.6
4.00	2.00	42.3	7.7
9.00	3.00	42.1	7.9
25.00	5.00	41.9	8.1
36.00	6.00	41.8	8.2
49.00	7.00	41.8	8.2
65.00	8.06	41.7	8.3
120.00	10.95	41.6	8.4



elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	40.7	9.3
0.05	0.22	38.1	11.9
0.10	0.32	38.0	12.0
0.25	0.50	37.8	12.2
0.50	0.71	37.6	12.4
1.00	1.00	37.5	12.5
3.00	1.73	37.3	12.7
5.00	2.24	37.2	12.8
13.00	3.61	37.0	13.0
22.00	4.69	36.9	13.1
35.00	5.92	36.8	13.2



RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-12	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-01
SAMPLE ID:	ROTL	TEST STARTED :	09/27/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/04/11
CONF. PRESSURE. (psi)	5	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1066.30	987.40
Wt. Wet Soil & Pan (g)	1066.30	595.50
Wt. Dry Soil & Pan (g)	435.20	435.20
Wt. Moisture Lost (g)	631.10	160.30
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	435.20	435.20
Moisture Content %	145.0	36.8
Wet Density (pcf)	85.9	122.4
Dry Density (pcf)	35.1	89.5

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	3.658	(cm)	9.291
Height Change (in)	2.224	(cm)	5.649
Consol. Height (in)	1.434	(cm)	3.642
Area After Consol. (sq in)	12.924	(sq cm)	83.385

Vol. Before Consol. (cu ft)	0.02735	Specific Gravity	2.79
Vol. Before Consol. (cc)	774.5	Assumed?	No
Change in Vol. (cc)	470.8		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	303.7	Init. Void Ratio	3.965
Vol. After Consol. (cu ft)	0.01073	Final Saturation	100.0
Effective Porosity %	79.86	Final Void Ratio	0.947
Pressure Difference (psi):	0.00		
C =	0.01584	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	2.3	37.7	37.7	
25.53	35.0	7.3	27.7	27.7	8.3E-05
0.00	40.0	1.8	38.2	38.2	
24.78	35.0	6.8	28.2	28.2	8.4E-05
0.00	40.0	2.3	37.7	37.7	
25.22	35.0	7.3	27.7	27.7	8.4E-05
0.00	40.0	2.8	37.2	37.2	
26.09	35.0	7.8	27.2	27.2	8.3E-05
0.00	40.0	2.2	37.8	37.8	
25.84	35.0	7.2	27.8	27.8	8.2E-05

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-12	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-01
SAMPLE ID:	ROTL	TEST STARTED :	09/27/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/04/11
CONF. PRESSURE. (psi)	10	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1066.30	987.40
Wt. Wet Soil & Pan (g)	1066.30	591.00
Wt. Dry Soil & Pan (g)	435.20	435.20
Wt. Moisture Lost (g)	631.10	155.80
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	435.20	435.20
Moisture Content %	145.0	35.8
Wet Density (pcf)	85.9	123.3
Dry Density (pcf)	35.1	90.8

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	3.658	(cm)	9.291
Height Change (in)	2.245	(cm)	5.702
Consol. Height (in)	1.413	(cm)	3.589
Area After Consol. (sq in)	12.922	(sq cm)	83.370

Vol. Before Consol. (cu ft)	0.02735	Specific Gravity	2.79
Vol. Before Consol. (cc)	774.5	Assumed?	No
Change in Vol. (cc)	475.3		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	299.2	Init. Void Ratio	3.965
Vol. After Consol. (cu ft)	0.01057	Final Saturation	100.0
Effective Porosity %	79.86	Final Void Ratio	0.918
Pressure Difference (psi):	0.00		
C =	0.01561	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	2.3	37.7	37.7	
26.50	35.0	7.3	27.7	27.7	7.9E-05
0.00	40.0	2.4	37.6	37.6	
26.91	35.0	7.4	27.6	27.6	7.8E-05
0.00	40.0	2.8	37.2	37.2	
27.18	35.0	7.8	27.2	27.2	7.8E-05
0.00	40.0	3.2	36.8	36.8	
27.19	35.0	8.2	26.8	26.8	7.9E-05
0.00	40.0	3.0	37	37.0	
26.97	35.0	8.0	27.0	27.0	7.9E-05

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-12	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-01
SAMPLE ID:	ROTL	TEST STARTED :	09/27/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/04/11
CONF. PRESSURE. (psi)	20	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1066.30	987.40
Wt. Wet Soil & Pan (g)	1066.30	584.50
Wt. Dry Soil & Pan (g)	435.20	435.20
Wt. Moisture Lost (g)	631.10	149.30
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	435.20	435.20
Moisture Content %	145.0	34.3
Wet Density (pcf)	85.9	124.7
Dry Density (pcf)	35.1	92.8

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	3.658	(cm)	9.291
Height Change (in)	2.275	(cm)	5.779
Consol. Height (in)	1.383	(cm)	3.513
Area After Consol. (sq in)	12.915	(sq cm)	83.328

Vol. Before Consol. (cu ft)	0.02735	Specific Gravity	2.79
Vol. Before Consol. (cc)	774.5	Assumed?	No
Change in Vol. (cc)	481.8		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	292.7	Init. Void Ratio	3.965
Vol. After Consol. (cu ft)	0.01034	Final Saturation	100.0
Effective Porosity %	79.86	Final Void Ratio	0.877
Pressure Difference (psi):	0.00		
C =	0.01528	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	2.3	37.7	37.7	
29.75	35.0	7.3	27.7	27.7	6.9E-05
0.00	40.0	2.4	37.6	37.6	
29.59	35.0	7.4	27.6	27.6	6.9E-05
0.00	40.0	2.8	37.2	37.2	
29.41	35.0	7.8	27.2	27.2	7.1E-05
0.00	40.0	3.2	36.8	36.8	
30.40	35.0	8.2	26.8	26.8	6.9E-05

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-12	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-01
SAMPLE ID:	ROTL	TEST STARTED :	09/27/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/04/11
CONF. PRESSURE. (psi)	40	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1066.30	987.40
Wt. Wet Soil & Pan (g)	1066.30	576.90
Wt. Dry Soil & Pan (g)	435.20	435.20
Wt. Moisture Lost (g)	631.10	141.70
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	435.20	435.20
Moisture Content %	145.0	32.6
Wet Density (pcf)	85.9	126.3
Dry Density (pcf)	35.1	95.3

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	3.658	(cm)	9.291
Height Change (in)	2.311	(cm)	5.870
Consol. Height (in)	1.347	(cm)	3.421
Area After Consol. (sq in)	12.916	(sq cm)	83.334

Vol. Before Consol. (cu ft)	0.02735	Specific Gravity	2.79
Vol. Before Consol. (cc)	774.5	Assumed?	No
Change in Vol. (cc)	489.4		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	285.1	Init. Void Ratio	3.965
Vol. After Consol. (cu ft)	0.01007	Final Saturation	100.0
Effective Porosity %	79.86	Final Void Ratio	0.828
Pressure Difference (psi):	0.00		
C =	0.01489	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	2.1	37.9	37.9	
33.79	35.0	7.1	27.9	27.9	5.9E-05
0.00	40.0	2.0	38.0	38.0	
33.40	35.0	7.0	28.0	28.0	5.9E-05
0.00	40.0	1.8	38.2	38.2	
33.50	35.0	6.8	28.2	28.2	5.9E-05
0.00	40.0	1.9	38.1	38.1	
34.35	35.0	6.9	28.1	28.1	5.7E-05
0.00	40.0	2.1	37.9	37.9	

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-12	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-01
SAMPLE ID:	ROTL	TEST STARTED :	09/27/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/04/11
CONF. PRESSURE. (psi)	80	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1066.30	987.40
Wt. Wet Soil & Pan (g)	1066.30	569.20
Wt. Dry Soil & Pan (g)	435.20	435.20
Wt. Moisture Lost (g)	631.10	134.00
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	435.20	435.20
Moisture Content %	145.0	30.8
Wet Density (pcf)	85.9	128.1
Dry Density (pcf)	35.1	97.9

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	3.658	(cm)	9.291
Height Change (in)	2.348	(cm)	5.964
Consol. Height (in)	1.310	(cm)	3.327
Area After Consol. (sq in)	12.922	(sq cm)	83.373

Vol. Before Consol. (cu ft)	0.02735	Specific Gravity	2.79
Vol. Before Consol. (cc)	774.5	Assumed?	No
Change in Vol. (cc)	497.1		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	277.4	Init. Void Ratio	3.965
Vol. After Consol. (cu ft)	0.00980	Final Saturation	100.0
Effective Porosity %	79.86	Final Void Ratio	0.778
Pressure Difference (psi):	0.00		
C =	0.01447	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	1.0	39.0	39.0	
39.50	35.0	6.0	29.0	29.0	4.7E-05
0.00	40.0	1.9	38.1	38.1	
40.81	35.0	6.9	28.1	28.1	4.7E-05
0.00	40.0	2.1	37.9	37.9	
40.93	35.0	7.1	27.9	27.9	4.7E-05
0.00	40.0	2.1	37.9	37.9	
40.66	35.0	7.1	27.9	27.9	4.7E-05
0.00	40.0	2.0	38.0	38.0	

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-12	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-01
SAMPLE ID:	ROTL	TEST STARTED :	09/27/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/04/11
CONF. PRESSURE. (psi)	120	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1066.30	987.40
Wt. Wet Soil & Pan (g)	1066.30	565.30
Wt. Dry Soil & Pan (g)	435.20	435.20
Wt. Moisture Lost (g)	631.10	130.10
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	435.20	435.20
Moisture Content %	145.0	29.9
Wet Density (pcf)	85.9	129.0
Dry Density (pcf)	35.1	99.3

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	3.658	(cm)	9.291
Height Change (in)	2.366	(cm)	6.010
Consol. Height (in)	1.292	(cm)	3.282
Area After Consol. (sq in)	12.918	(sq cm)	83.346

Vol. Before Consol. (cu ft)	0.02735	Specific Gravity	2.79
Vol. Before Consol. (cc)	774.5	Assumed?	No
Change in Vol. (cc)	501.0		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	273.5	Init. Void Ratio	3.965
Vol. After Consol. (cu ft)	0.00966	Final Saturation	100.0
Effective Porosity %	79.86	Final Void Ratio	0.753
Pressure Difference (psi):	0.00		
C =	0.01428	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	2.7	37.3	37.3	
48.76	35.0	7.7	27.3	27.3	4.0E-05
0.00	40.0	2.0	38.0	38.0	
45.62	35.0	7.0	28.0	28.0	4.2E-05
0.00	40.0	3.2	36.8	36.8	
48.84	35.0	8.2	26.8	26.8	4.0E-05
0.00	40.0	2.0	38.0	38.0	
45.87	35.0	7.0	28.0	28.0	4.1E-05
0.00	40.0	2.1	37.9	37.9	

PROJECT NO. VA101-458.03
PROJECT: Harper Creek ROTL 45% solids 20 11
TESTED BY: jhk Denver
DATE: 03/10/11

S.G. SOLIDS : 2.79
S.G. LIQUOR: 1.0000
pH LIQUOR: 10.00

CONSOLIDATION TESTS

CYLINDER:	A
Weight of Container:	217.7
Weight of Slurry + Cont.:	1590.4
Height to Bottom of Slurry:	0.0
Height to Top of Slurry:	329.0
Diameter of Container:	59.0
Settled Top of Slurry Heights:	206
BEFORE CONSOLIDATION:	206.0
AFTER CONSOLIDATION:	194.5
AFTER COMPLETE DRAINAGE:	193.0

DATE DAY	MONTH	TIME HOUR	MINUTE	SECOND	SLURRY HEIGHT READING	WEIGHT OF WATER + CONTAINER	WEIGHT OF CONTAINER	Water Level
5	10	7	57	0	206.0	33.3	33.3	329.0
5	10	7	57	6	206.0			329.0
5	10	7	57	15	206.0			329.0
5	10	7	57	30	206.0			329.0
5	10	7	58	0	206.0			329.0
5	10	7	59	0	206.0			329.0
5	10	8	2	0	202.0			329.0
5	10	8	7	0	198.0			329.0
5	10	8	17	0	195.0			329.0
5	10	8	27	0	195.0			329.0
5	10	8	57	0	195.0			329.0
5	10	9	27	0	195.0			329.0
5	10	9	57	0	195.0			329.0
5	10	10	27	0	195.0			329.0
5	10	11	57	0	194.5			329.0
5	10	15	57	0	194.5	682.6	33.3	329.0
6	10	8	29	0	193.0			193.0

NOTES: The water drained out completely overnight.

SLURRY CONSOLIDOMETER TEST RESULTS
HARPER CREEK
Cu ROTL
VA101-458.04

		Consolidation Test Data - Double Ended Drainage								
STRESS	psi	Free	0.5	2	5	10	20	40	80	120
	psf		72	288	720	1440	2880	5760	11520	17280
	kPa		3	14	34	69	138	276	552	827
Volume Change, cc		522.0	9.0	5.0	8.4	5.7	8.0	8.5	9.7	5.0
Cumulative Volume Change, cc			531.0	536.0	544.4	550.1	558.1	566.6	576.3	581.3
Consolidated Slurry Volume, cm^3		358.8	349.8	344.8	336.4	330.7	322.7	314.2	304.5	299.5
Settled Void Ratio, e_c		1.282	1.225	1.193	1.139	1.103	1.052	0.998	0.937	0.905
Dry Unit Weight, pcf		76.3	78.3	79.4	81.4	82.8	84.9	87.2	89.9	91.4
Slurry Height, cm		4.304	4.196	4.136	4.036	3.967	3.871	3.769	3.653	3.593
Mean Slurry Height, H^2 mm			451.6	434.0	417.4	400.3	384.0	364.9	344.3	328.1
Total Ht. Change, cm		6.262	6.370	6.430	6.531	6.599	6.695	6.797	6.913	6.973
Cumulative change in height, in.		2.465	2.508	2.531	2.571	2.598	2.636	2.676	2.722	2.745
Individual change in height, in.		2.465	0.043	0.024	0.040	0.027	0.038	0.040	0.046	0.024

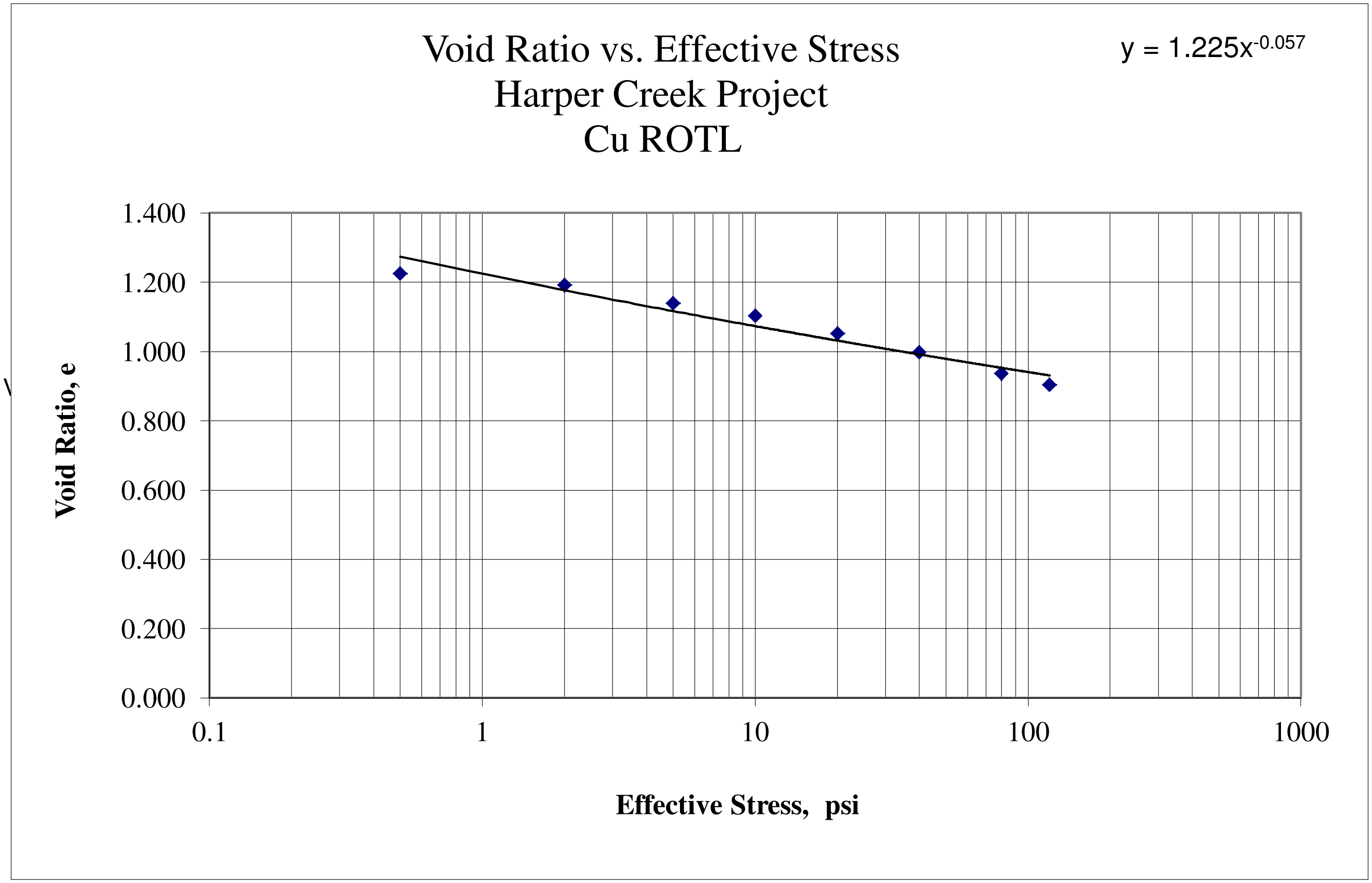
SAMPLE PARAMETERS

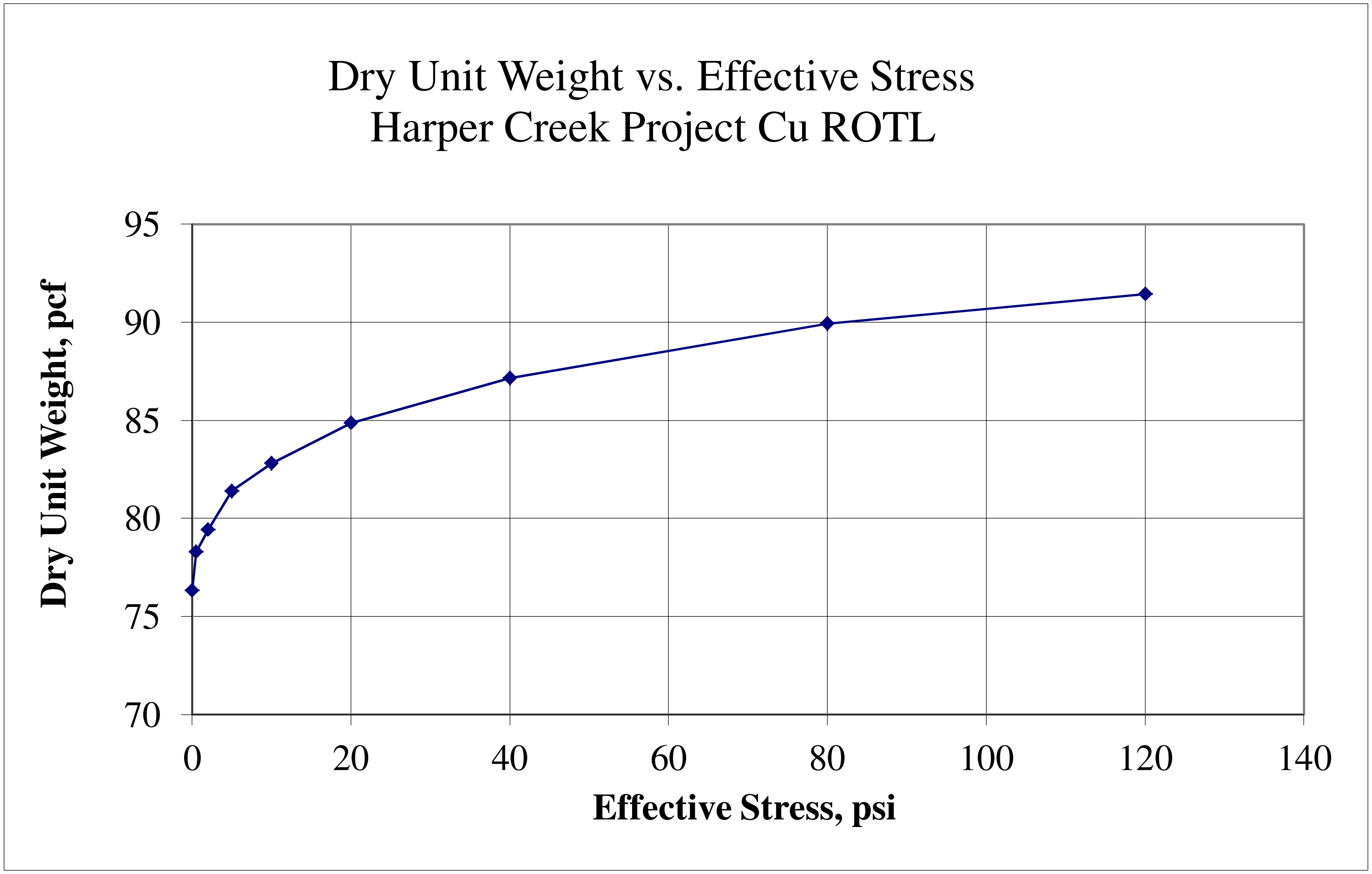
Specific Gravity	2.790	
Slurry Mass w/tare, g	1505.30	
Tare, g	311.20	
Slurry Mass, g	1194.10	
Height of Slurry (in./cm)	4.160	10.57
Volume of Slurry, cc		880.81
Solids Content, %	36.7	
Wt. of Solids, g	438.7	
Ht. of Solids, cm		1.886
Volume of Solids, cc	157.24	
Intial Void Ratio, e	4.602	

Ring Diameter (in./cm)	4.056	10.30
Ring Area, cm^2		83.359
Ring Height (in./cm)	4.058	10.31
Ring Volume, cm^3		859.21

Post Test Specimen Data

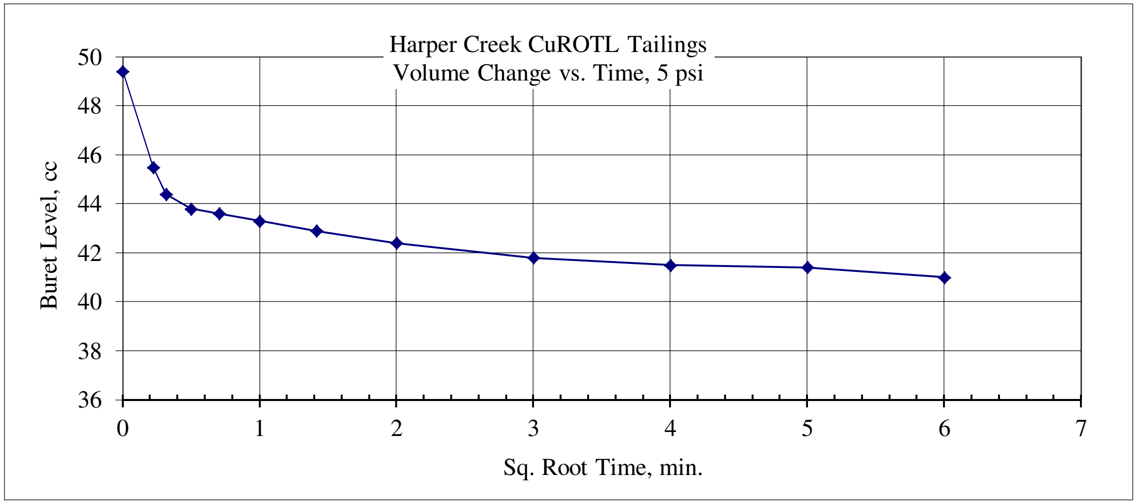
Wet + Tare, g	692.00
Dry + Tare, g	549.20
Tare, g	110.50
Wt. of Water, g	144.80
Wt. of Dry Solids, g	438.70
Moisture Content, %	33.01


Void Ratio vs. Effective Stress
Harper Creek Project
Cu ROTL
y = 1.225x-0.057
Void Ratio, e
1.400
1.200
1.000
0.800
0.600
0.400
0.200
0.000
0.1
1
10
100
1000
Effective Stress, psi

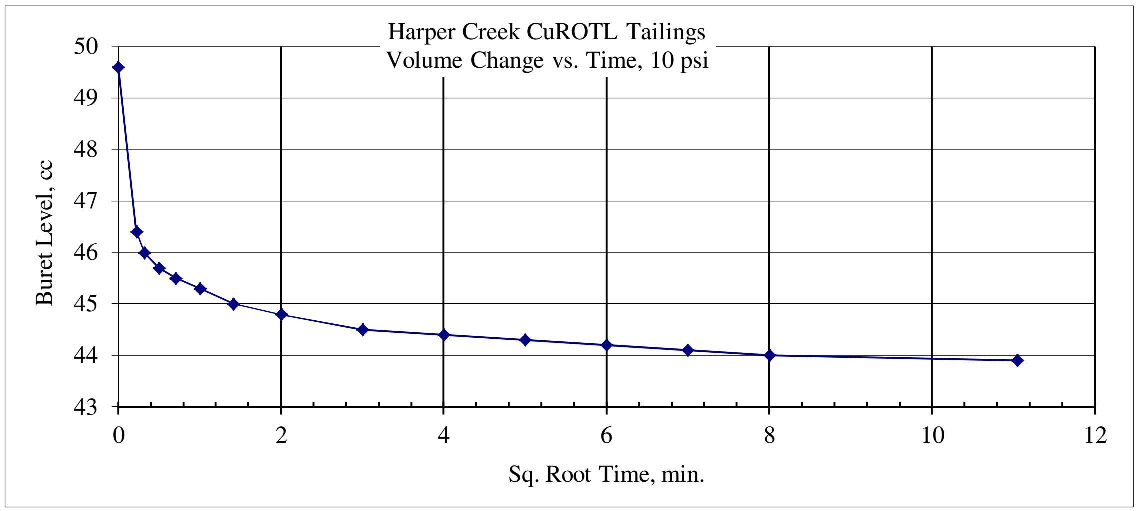

Dry Unit Weight vs. Effective Stress
Harper Creek Project Cu ROTL
Dry Unit Weight, pcf
95
90
85
80
75
70
0
20
40
60
80
100
120
140
Effective Stress, psi

Harper Creek Project
Cu ROTL
High Stress Consolidation Test

elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	49.4	0.6
0.05	0.22	45.5	4.5
0.10	0.32	44.4	5.6
0.25	0.50	43.8	6.2
0.50	0.71	43.6	6.4
1.00	1.00	43.3	6.7
2.00	1.41	42.9	7.1
4.00	2.00	42.4	7.6
9.00	3.00	41.8	8.2
16.00	4.00	41.5	8.5
25.00	5.00	41.4	8.6
36.00	6.00	41.0	9.0

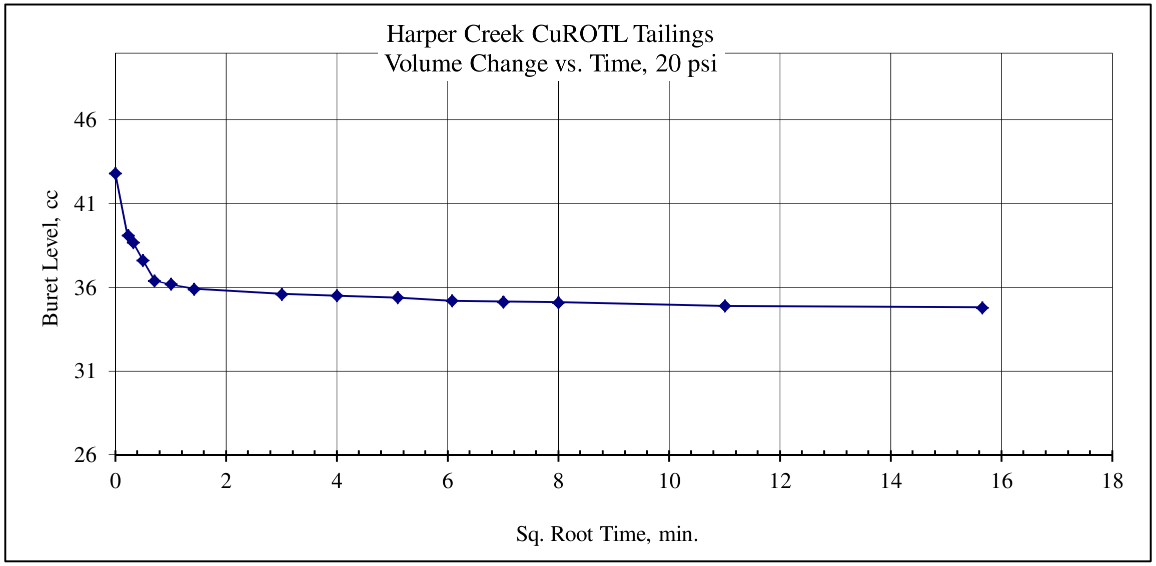


elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	49.6	0.4
0.05	0.22	46.4	3.6
0.10	0.32	46.0	4.0
0.25	0.50	45.7	4.3
0.50	0.71	45.5	4.5
1.00	1.00	45.3	4.7
2.00	1.41	45.0	5.0
4.00	2.00	44.8	5.2
9.00	3.00	44.5	5.5
16.00	4.00	44.4	5.6
25.00	5.00	44.3	5.7
36.00	6.00	44.2	5.8
49.00	7.00	44.1	5.9
64.00	8.00	44.0	6.0
122.00	11.05	43.9	6.1

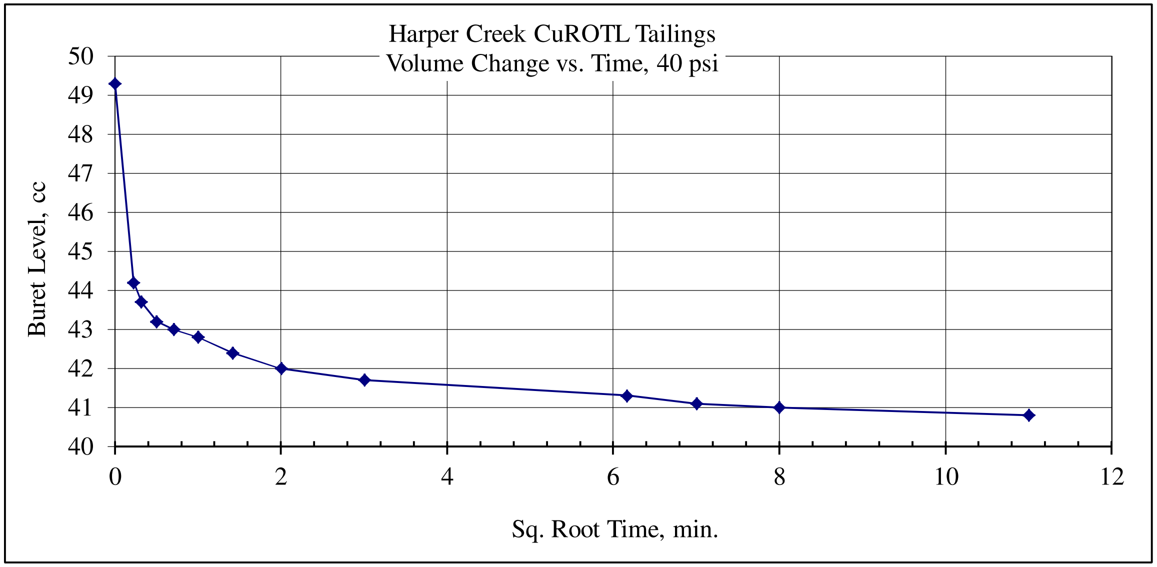


Harper Creek Project
Cu ROTL
High Stress Consolidation Test

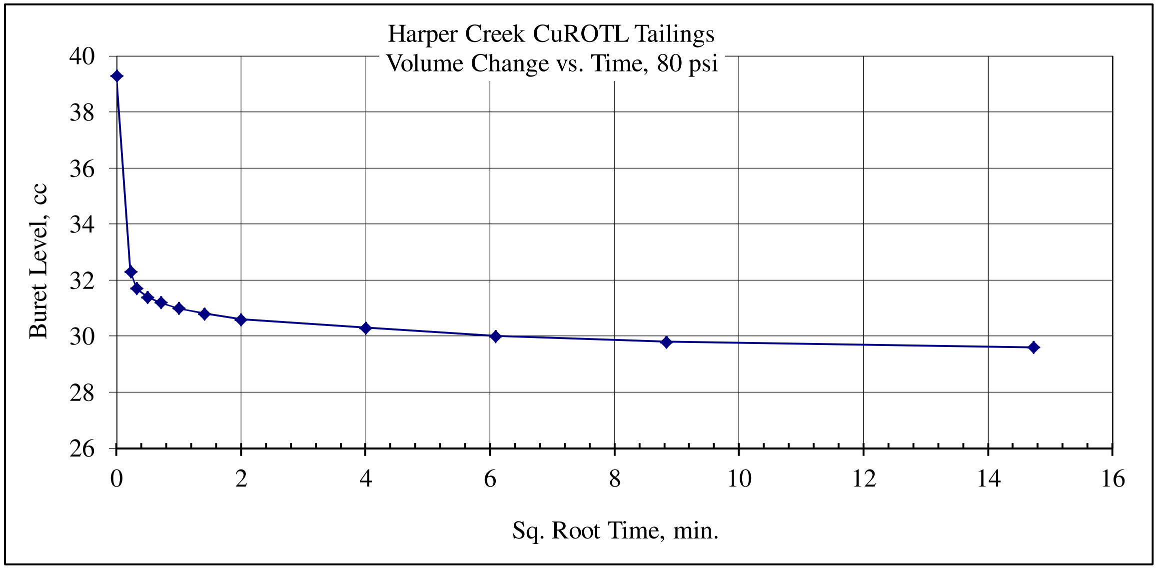
elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	42.8	7.2
0.05	0.22	39.1	10.9
0.10	0.32	38.7	11.3
0.25	0.50	37.6	12.4
0.50	0.71	36.4	13.6
1.00	1.00	36.2	13.8
2.00	1.41	35.9	14.1
9.00	3.00	35.6	14.4
16.00	4.00	35.5	14.5
26.00	5.10	35.4	14.6
37.00	6.08	35.2	14.8
49.00	7.00	35.2	14.9
64.00	8.00	35.1	14.9
121.00	11.00	34.9	15.1
245.00	15.65	34.8	15.2



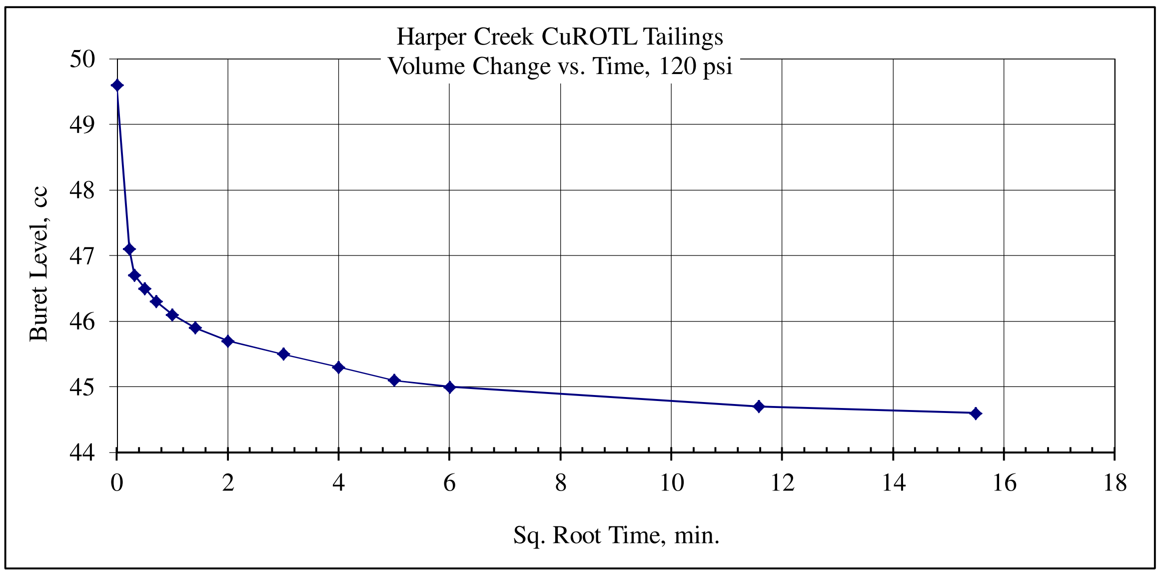
elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	49.3	0.7
0.05	0.22	44.2	5.8
0.10	0.32	43.7	6.3
0.25	0.50	43.2	6.8
0.50	0.71	43.0	7.0
1.00	1.00	42.8	7.2
2.00	1.41	42.4	7.6
4.00	2.00	42.0	8.0
9.00	3.00	41.7	8.3
38.00	6.16	41.3	8.7
49.00	7.00	41.1	8.9
64.00	8.00	41.0	9.0
121.00	11.00	40.8	9.2



elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	39.3	10.7
0.05	0.22	32.3	17.7
0.10	0.32	31.7	18.3
0.25	0.50	31.4	18.6
0.50	0.71	31.2	18.8
1.00	1.00	31.0	19.0
2.00	1.41	30.8	19.2
4.00	2.00	30.6	19.4
16.00	4.00	30.3	19.7
37.00	6.08	30.0	20.0
78.00	8.83	29.8	20.2
217.00	14.73	29.6	20.4



elapsed time min.	sq. root time min.	Change in Volume (50 - cc)	Cell Buret Level cc
0.00	0.00	49.6	0.4
0.05	0.22	47.1	2.9
0.10	0.32	46.7	3.3
0.25	0.50	46.5	3.5
0.50	0.71	46.3	3.7
1.00	1.00	46.1	3.9
2.00	1.41	45.9	4.1
4.00	2.00	45.7	4.3
9.00	3.00	45.5	4.5
16.00	4.00	45.3	4.7
25.00	5.00	45.1	4.9
36.00	6.00	45.0	5.0
134.00	11.58	44.7	5.3
240.00	15.49	44.6	5.4



RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-13	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-02
SAMPLE ID:	CuROTL	TEST STARTED :	10/03/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/11/11
CONF. PRESSURE. (psi)	5	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1194.10	987.40
Wt. Wet Soil & Pan (g)	1194.10	649.70
Wt. Dry Soil & Pan (g)	438.70	438.70
Wt. Moisture Lost (g)	755.40	211.00
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	438.70	438.70
Moisture Content %	172.2	48.1
Wet Density (pcf)	84.6	120.6
Dry Density (pcf)	31.1	81.4

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	4.160	(cm)	10.566
Height Change (in)	2.571	(cm)	6.530
Consol. Height (in)	1.589	(cm)	4.036
Area After Consol. (sq in)	12.918	(sq cm)	83.350

Vol. Before Consol. (cu ft)	0.03111	Specific Gravity	2.79
Vol. Before Consol. (cc)	880.8	Assumed?	No
Change in Vol. (cc)	544.4		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	336.4	Init. Void Ratio	4.602
Vol. After Consol. (cu ft)	0.01188	Final Saturation	100.0
Effective Porosity %	82.15	Final Void Ratio	1.139
Pressure Difference (psi):	0.00		
C =	0.01756	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	1.8	38.2	38.2	
24.50	35.0	6.8	28.2	28.2	9.4E-05
0.00	40.0	2.0	38.0	38.0	
25.00	35.0	7.0	28.0	28.0	9.3E-05
0.00	40.0	2.0	38.0	38.0	
25.19	35.0	7.0	28.0	28.0	9.2E-05
0.00	40.0	1.9	38.1	38.1	
25.06	35.0	6.9	28.1	28.1	9.3E-05
0.00	40.0	1.9	38.1	38.1	
24.96	35.0	6.9	28.1	28.1	9.3E-05

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-13	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-02
SAMPLE ID:	CuROTL	TEST STARTED :	10/03/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/11/11
CONF. PRESSURE. (psi)	10	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1194.10	987.40
Wt. Wet Soil & Pan (g)	1194.10	644.00
Wt. Dry Soil & Pan (g)	438.70	438.70
Wt. Moisture Lost (g)	755.40	205.30
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	438.70	438.70
Moisture Content %	172.2	46.8
Wet Density (pcf)	84.6	121.6
Dry Density (pcf)	31.1	82.8

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	4.160	(cm)	10.566
Height Change (in)	2.598	(cm)	6.599
Consol. Height (in)	1.562	(cm)	3.967
Area After Consol. (sq in)	12.919	(sq cm)	83.354

Vol. Before Consol. (cu ft)	0.03111	Specific Gravity	2.79
Vol. Before Consol. (cc)	880.8	Assumed?	No
Change in Vol. (cc)	550.1		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	330.7	Init. Void Ratio	4.602
Vol. After Consol. (cu ft)	0.01168	Final Saturation	100.0
Effective Porosity %	82.15	Final Void Ratio	1.103
Pressure Difference (psi):	0.00		
C =	0.01726	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	1.8	38.2	38.2	
26.84	35.0	6.8	28.2	28.2	8.5E-05
0.00	40.0	2.0	38.0	38.0	
27.59	35.0	7.0	28.0	28.0	8.3E-05
0.00	40.0	2.0	38.0	38.0	
27.46	35.0	7.0	28.0	28.0	8.3E-05
0.00	40.0	1.9	38.1	38.1	
27.16	35.0	6.9	28.1	28.1	8.4E-05
0.00	40.0	1.9	38.1	38.1	
26.87	35.0	6.9	28.1	28.1	8.5E-05

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-13	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-02
SAMPLE ID:	CuROTL	TEST STARTED :	10/03/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/11/11
CONF. PRESSURE. (psi)	20	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1194.10	987.40
Wt. Wet Soil & Pan (g)	1194.10	636.00
Wt. Dry Soil & Pan (g)	438.70	438.70
Wt. Moisture Lost (g)	755.40	197.30
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	438.70	438.70
Moisture Content %	172.2	45.0
Wet Density (pcf)	84.6	123.0
Dry Density (pcf)	31.1	84.9

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	4.160	(cm)	10.566
Height Change (in)	2.636	(cm)	6.695
Consol. Height (in)	1.524	(cm)	3.871
Area After Consol. (sq in)	12.921	(sq cm)	83.366

Vol. Before Consol. (cu ft)	0.03111	Specific Gravity	2.79
Vol. Before Consol. (cc)	880.8	Assumed?	No
Change in Vol. (cc)	558.1		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	322.7	Init. Void Ratio	4.602
Vol. After Consol. (cu ft)	0.01140	Final Saturation	100.0
Effective Porosity %	82.15	Final Void Ratio	1.052
Pressure Difference (psi):	0.00		
C =	0.01684	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.0	1.8	38.2	38.2	
31.75	35.0	6.8	28.2	28.2	7.0E-05
0.00	40.0	2.0	38.0	38.0	
31.85	35.0	7.0	28.0	28.0	7.0E-05
0.00	40.0	2.0	38.0	38.0	
31.75	35.0	7.0	28.0	28.0	7.0E-05
0.00	40.0	1.9	38.1	38.1	
31.81	35.0	6.9	28.1	28.1	7.0E-05
0.00	40.0	1.9	38.1	38.1	
31.41	35.0	6.9	28.1	28.1	7.1E-05

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-13	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-02
SAMPLE ID:	CuROTL	TEST STARTED :	10/03/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/11/11
CONF. PRESSURE. (psi)	40	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1194.10	987.40
Wt. Wet Soil & Pan (g)	1194.10	627.50
Wt. Dry Soil & Pan (g)	438.70	438.70
Wt. Moisture Lost (g)	755.40	188.80
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	438.70	438.70
Moisture Content %	172.2	43.0
Wet Density (pcf)	84.6	124.7
Dry Density (pcf)	31.1	87.2

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	4.160	(cm)	10.566
Height Change (in)	2.676	(cm)	6.797
Consol. Height (in)	1.484	(cm)	3.769
Area After Consol. (sq in)	12.920	(sq cm)	83.358

Vol. Before Consol. (cu ft)	0.03111	Specific Gravity	2.79
Vol. Before Consol. (cc)	880.8	Assumed?	No
Change in Vol. (cc)	566.6		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	314.2	Init. Void Ratio	4.602
Vol. After Consol. (cu ft)	0.01110	Final Saturation	100.0
Effective Porosity %	82.15	Final Void Ratio	0.998
Pressure Difference (psi):	0.00		
C =	0.01639	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	40.4	1.6	38.8	38.8	
15.00	38.3	4.0	34.3	34.3	5.9E-05
15.00	36.3	5.9	30.4	30.4	5.7E-05
15.00	34.6	7.3	27.3	27.3	5.1E-05
15.00	33.0	8.9	24.1	24.1	5.9E-05
15.00	31.8	10.1	21.7	21.7	5.0E-05
15.00	30.5	11.3	19.2	19.2	5.8E-05

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-13	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-02
SAMPLE ID:	CuROTL	TEST STARTED :	10/03/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/11/11
CONF. PRESSURE. (psi)	80	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1194.10	987.40
Wt. Wet Soil & Pan (g)	1194.10	617.80
Wt. Dry Soil & Pan (g)	438.70	438.70
Wt. Moisture Lost (g)	755.40	179.10
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	438.70	438.70
Moisture Content %	172.2	40.8
Wet Density (pcf)	84.6	126.7
Dry Density (pcf)	31.1	89.9

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	4.160	(cm)	10.566
Height Change (in)	2.722	(cm)	6.914
Consol. Height (in)	1.438	(cm)	3.653
Area After Consol. (sq in)	12.921	(sq cm)	83.369

Vol. Before Consol. (cu ft)	0.03111	Specific Gravity	2.79
Vol. Before Consol. (cc)	880.8	Assumed?	No
Change in Vol. (cc)	576.3		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	304.5	Init. Void Ratio	4.602
Vol. After Consol. (cu ft)	0.01075	Final Saturation	100.0
Effective Porosity %	82.15	Final Void Ratio	0.937
Pressure Difference (psi):	0.00		
C =	0.01588	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	46.6	5.6	41.0	41.0	
15.00	44.8	7.7	37.1	37.1	4.6E-05
15.00	43.3	9.3	34.0	34.0	4.0E-05
15.00	41.8	10.5	31.3	31.3	3.8E-05
15.00	40.6	11.8	28.8	28.8	3.8E-05
15.00	39.3	13.8	25.5	25.5	5.6E-05
30.00	37.0	15.5	21.5	21.5	3.9E-05

RIGID WALL PERMEABILITY TEST

ASTM D 5856-02

Falling Head / Increasing Tailwater Pressure

CLIENT:	Yellowhead Mining, Inc.		
PROJECT:	Harper Creek	PROJECT NO. :	VA101-458.04
SAMPLE NO.	2916-13	LAB NO. :	L2011-099
DEPTH		SAMPLE ID:	2011-099-02
SAMPLE ID:	CuROTL	TEST STARTED :	10/03/11
SAMPLE TYPE	Slurry	TEST FINISHED :	10/11/11
CONF. PRESSURE. (psi)	120	SATURATED TEST:	YES

MOISTURE/DENSITY DATA	BEFORE TEST	AFTER TEST
Wt. Soil + Moisture (g)	1194.10	987.40
Wt. Wet Soil & Pan (g)	1194.10	612.80
Wt. Dry Soil & Pan (g)	438.70	438.70
Wt. Moisture Lost (g)	755.40	174.10
Wt. of Pan Only (g)	0.00	0.00
Wt. of Dry Soil (g)	438.70	438.70
Moisture Content %	172.2	39.7
Wet Density (pcf)	84.6	127.7
Dry Density (pcf)	31.1	91.4

Init. Diameter (in)	4.056	(cm)	10.302
Init. Area (sq in)	12.921	(sq cm)	83.359
Init. Height (in)	4.160	(cm)	10.566
Height Change (in)	2.745	(cm)	6.972
Consol. Height (in)	1.415	(cm)	3.594
Area After Consol. (sq in)	12.916	(sq cm)	83.333

Vol. Before Consol. (cu ft)	0.03111	Specific Gravity	2.79
Vol. Before Consol. (cc)	880.8	Assumed?	No
Change in Vol. (cc)	581.3		
Cell Exp. (cc)	0.0	Init. Saturation	100.0
Vol. After Consol. (cc)	299.5	Init. Void Ratio	4.602
Vol. After Consol. (cu ft)	0.01058	Final Saturation	100.0
Effective Porosity %	82.15	Final Void Ratio	0.905
Pressure Difference (psi):	0.00		
C =	0.01564	Buret Constant, a	0.315
k, cm/s = C/t*log(h1/h2)		Buret Stand	20

Permeability Test Trials

Time Sec.	Cap Elevation cm	Pedestal Elevation cm	Elevation Head cm	Total Head cm	Permeability k cm/sec
0.00	45.8	1.5	44.3	44.3	
15.00	44.0	3.0	41.0	41.0	3.5E-05
15.00	42.3	5.0	37.3	37.3	4.3E-05
15.00	40.9	6.3	34.6	34.6	3.4E-05
15.00	39.6	7.8	31.8	31.8	3.8E-05
30.00	37.3	10.1	27.2	27.2	3.5E-05
30.00	35.3	12.1	23.2	23.2	3.6E-05

PROJECT NO. VA101-458.03
PROJECT: Harper Creek ROTL 35% solids 20 11
TESTED BY: jhk Denver
DATE: 27/09/11

S.G. SOLIDS : 2.79
S.G. LIQUOR: 1.0000
pH LIQUOR: 10.00

CONSOLIDATION TESTS

CYLINDER:	B
Weight of Container:	222.1
Weight of Slurry + Cont.:	1385.6
Height to Bottom of Slurry:	0.0
Height to Top of Slurry:	305.0
Diameter of Container:	58.8
Settled Top of Slurry Heights:	137
BEFORE CONSOLIDATION:	137.0
AFTER CONSOLIDATION:	128.0
AFTER COMPLETE DRAINAGE:	126.0

DATE		TIME			SLURRY	WEIGHT	WEIGHT	Water
					HEIGHT	OF WATER +	OF	Level
DAY	MONTH	HOUR	MINUTE	SECOND	READING	CONTAINER	CONTAINER	
29	9	8	59	0	137.0	33.5	33.5	305.0
29	9	8	59	6	137.0			305.0
29	9	8	59	15	137.0			305.0
29	9	8	59	30	135.0			305.0
29	9	9	0	0	134.0			305.0
29	9	9	1	0	132.5			305.0
29	9	9	4	0	130.0			305.0
29	9	9	9	0	128.5			305.0
29	9	9	19	0	128.0			305.0
29	9	9	29	0	128.0			305.0
29	9	9	59	0	128.0			305.0
29	9	10	29	0	128.0			305.0
29	9	11	59	0	128.0			305.0
29	9	11	29	0	128.0			305.0
29	9	13	4	0	128.0			305.0
29	9	16	5	0	128.0			305.0
30	9	9	29	0	128.0	808.7	33.5	305.0

NSOLIDATION TEST

	PROJECT	Harper Creek ROTL 35% solids				DATE	:	27-Sep-11
	SAMPLE	:				TESTED BY	:	jhk Denver
*	Weight of Container	:	222.1			PULP DENSITY	:	1.407
*	Weight of Slurry + Container	:	1385.6		*	S.G. SOLIDS	:	2.79
	Weight of Slurry	:	1163.5		*	S.G. LIQUOR	:	1.00
*	Height to Bottom of Slurry	:	0.0			% WT. SOLIDS	:	45.07
*	Height to Top of Slurry	:	305.0			Weight of Solids	:	524.3
	Height of Slurry	:	305.0			Volume of water	:	639.2
*	Internal dia. of Container	:	58.8					
	Volume of Slurry	:	827.1					

	BEFORE CONSOLIDATION				AFTER CONSOLIDATION				AFTER COMPLETE DRAINAGE		
*	Settled Slurry Ht	:	137.0	*	Settled Slurry Ht	:	128.0	*	Settled Slurry Ht	:	126.0
	Ht of Water	:	168.0		Ht of Water	:	177.0		Ht of Water	:	-
	Vol of Slurry	:	371.5		Vol of Slurry	:	347.1		Vol of Slurry	:	341.7
	Vol of Water	:	455.6		Vol of Water	:	480.0		Vol of Water	:	-
	Dry Density	:	1.411		Dry Density	:	1.511		Dry Density	:	1.535
	Pulp Density	:	1.906		Pulp Density	:	1.969		Pulp Density	:	1.985
	Void Ratio	:	0.977		Void Ratio	:	0.847		Void Ratio	:	0.818
	Total Stress at Base	:	4.208		Total Stress at Ba	:	4.208		Total Stress at Ba	:	2.452
	Eff. Stress at Base	:	1.217		Eff. Stress at Bas	:	4.208		Eff. Stress at Bas	:	3.688
	Average Eff. Stress	:	0.608		Average Eff. Stres	:	2.104		Average Eff. Stres	:	1.844

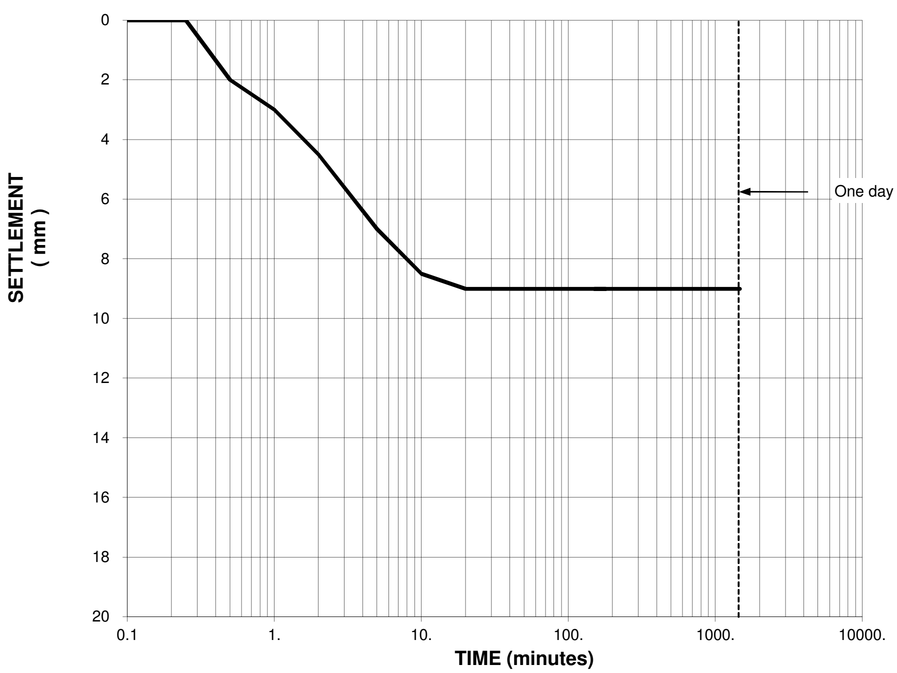
DAY	TIME	ELAPSED TIME (hours)	(min)	(sec)	* SLURRY HEIGHT READING	SETTLEMENT OF SLURRY	* WEIGHT OF WATER & CONT	* WEIGHT OF CONT	VOLUME OF WATER	CUMUL. VOLUME OF WATER	HEIGHT OF SLURRY	VOLUME OF SLURRY	DRY DENSITY t/m^3	i VALUE (av ht)	PERMEABILITY (m/s)
29/09/11	08:59:00	< START TIME		0	137.0	0.00	33.5	33.5	0.0	0.0	137.0	371.5	1.411		
29/09/11	08:59:06	0.00	0.10	6	137.0	0.00	0.0	0.0	0.0	0.0	137.0	371.5	1.411	2.23	0.000E+00
29/09/11	08:59:15	0.00	0.25	15	137.0	0.00	0.0	0.0	0.0	0.0	137.0	371.5	1.411	2.23	--
29/09/11	08:59:30	0.01	0.50	30	135.0	2.00	0.0	0.0	0.0	0.0	135.0	366.1	1.432	2.24	--
29/09/11	09:00:00	0.02	1.00	60	134.0	3.00	0.0	0.0	0.0	0.0	134.0	363.4	1.443	2.27	--
29/09/11	09:01:00	0.03	2.00	120	132.5	4.50	0.0	0.0	0.0	0.0	132.5	359.3	1.459	2.29	--
29/09/11	09:04:00	0.08	5.00	300	130.0	7.00	0.0	0.0	0.0	0.0	130.0	352.5	1.487	2.32	--
29/09/11	09:09:00	0.17	10.00	600	128.5	8.50	0.0	0.0	0.0	0.0	128.5	348.5	1.505	2.36	--
29/09/11	09:19:00	0.33	20.00	1200	128.0	9.00	0.0	0.0	0.0	0.0	128.0	347.1	1.511	2.38	--
29/09/11	09:29:00	0.50	30.00	1800	128.0	9.00	0.0	0.0	0.0	0.0	128.0	347.1	1.511	2.38	--
29/09/11	09:59:00	1.00	60.00	3600	128.0	9.00	0.0	0.0	0.0	0.0	128.0	347.1	1.511	2.38	--
29/09/11	10:29:00	1.50	90.00	5400	128.0	9.00	0.0	0.0	0.0	0.0	128.0	347.1	1.511	2.38	--
29/09/11	11:59:00	3.00	180.00	10800	128.0	9.00	0.0	0.0	0.0	0.0	128.0	347.1	1.511	2.38	--
29/09/11	11:29:00	2.50	150.00	9000	128.0	9.00	0.0	0.0	0.0	0.0	128.0	347.1	1.511	2.38	--
29/09/11	13:04:00	4.08	245.00	14700	128.0	9.00	0.0	0.0	0.0	0.0	128.0	347.1	1.511	2.38	--
29/09/11	16:05:00	7.10	426.00	25560	128.0	9.00	0.0	0.0	0.0	0.0	128.0	347.1	1.511	2.38	--
30/09/11	09:29:00	24.50	1470.00	88200	128.0	9.00	808.7	33.5	775.2	775.2	128.0	347.1	1.511	2.38	1.360E-06

ENT OF CONSOLIDATION

U value	Tv	Measured Value	ROW	X values	Y values	L values	H values	A	B	C	Time(h)	Cv	
													Mv
0.5	0.197	4.50	5	4.500	-1.477			0.010	0.040	-1.867	0.033	235.01	0.0439
				7.000	-1.079	11.500	0.159						Kv
				8.500	-0.778	13.000	0.175						3.334E-06
													Cc
0.6	0.287	5.40	5	4.500	-1.477			0.010	0.040	-1.867	0.045	253.10	0.2410
				7.000	-1.079	11.500	0.159						
				8.500	-0.778	13.000	0.175					244.05	

ELAPSED TIME (minutes)	SETTLEMENT	PERMEABILITY m/s	DRY DENSITY t/m^3		
0	0	0.000E+00	1.41138259	Cv	2.441E+02
0.1	0	0.000E+00	1.41138259	Mv	4.393E-02
0.25	0		1.41138259	Kv	3.334E-06
0.5	2	--	1.432291961	Cc	2.410E-01
1	3	--	1.442980707		
2.000000001	4.5	--	1.459316338		
5	7	--	1.487380114		
10	8.5	--	1.504742528		
20	9	--	1.510620428		
30	9	--	1.510620428		
60	9	--	1.510620428		
90	9	--	1.510620428		
180	9	--	1.510620428		
150	9	--	1.510620428		
245	9	--	1.510620428		
426	9	--	1.510620428		
1470	9	1.360E-06	1.510620428		

Harper Creek Low Stress Consolidation ROTL 35% Solids



PROJECT NO. VA101-458.03
PROJECT: Harper Creek ROTL 45% solids 20 11
TESTED BY: jhk Denver
DATE: 03/10/11

S.G. SOLIDS : 2.79
S.G. LIQUOR: 1.0000
pH LIQUOR: 10.00

CONSOLIDATION TESTS

CYLINDER:	A
Weight of Container:	217.7
Weight of Slurry + Cont.:	1590.4
Height to Bottom of Slurry:	0.0
Height to Top of Slurry:	329.0
Diameter of Container:	59.0
Settled Top of Slurry Heights:	206
BEFORE CONSOLIDATION:	206.0
AFTER CONSOLIDATION:	194.5
AFTER COMPLETE DRAINAGE:	193.0

DATE		TIME			SLURRY HEIGHT	WEIGHT OF WATER +	WEIGHT OF	Water Level
DAY	MONTH	HOUR	MINUTE	SECOND	READING	CONTAINER	CONTAINER	
5	10	7	57	0	206.0	33.3	33.3	329.0
5	10	7	57	6	206.0			329.0
5	10	7	57	15	206.0			329.0
5	10	7	57	30	206.0			329.0
5	10	7	58	0	206.0			329.0
5	10	7	59	0	206.0			329.0
5	10	8	2	0	202.0			329.0
5	10	8	7	0	198.0			329.0
5	10	8	17	0	195.0			329.0
5	10	8	27	0	195.0			329.0
5	10	8	57	0	195.0			329.0
5	10	9	27	0	195.0			329.0
5	10	9	57	0	195.0			329.0
5	10	10	27	0	195.0			329.0
5	10	11	57	0	194.5			329.0
5	10	15	57	0	194.5	682.6	33.3	329.0
6	10	8	29	0	193.0			193.0

NOTES: The water drained out completely overnight.

ɴSOLIDATION TEST	PROJECT	Harper Creek ROTL 45% solids	DATE	:	3-Oct-11
	SAMPLE	:	TESTED BY	:	jhk Denver

*	Weight of Container	:	217.7		PULP DENSITY	:	1.526
*	Weight of Slurry + Container	:	1590.4	*	S.G. SOLIDS	:	2.79
	Weight of Slurry	:	1372.7	*	S.G. LIQUOR	:	1.00
*	Height to Bottom of Slurry	:	0.0		% WT. SOLIDS	:	53.73
*	Height to Top of Slurry	:	329.0		Weight of Solids	:	737.6
	Height of Slurry	:	329.0		Volume of water	:	635.1
*	Internal dia. of Container	:	59.0				
	Volume of Slurry	:	899.5				

	BEFORE CONSOLIDATION				AFTER CONSOLIDATION				AFTER COMPLETE DRAINAGE		
*	Settled Slurry Ht	:	206.0	*	Settled Slurry Ht	:	194.5	*	Settled Slurry Ht	:	193.0
	Ht of Water	:	123.0		Ht of Water	:	134.5		Ht of Water	:	-
	Vol of Slurry	:	563.2		Vol of Slurry	:	531.8		Vol of Slurry	:	527.7
	Vol of Water	:	336.3		Vol of Water	:	367.7		Vol of Water	:	-
	Dry Density	:	1.310		Dry Density	:	1.387		Dry Density	:	1.398
	Pulp Density	:	1.840		Pulp Density	:	1.890		Pulp Density	:	1.897
	Void Ratio	:	1.130		Void Ratio	:	1.011		Void Ratio	:	0.996
	Total Stress at Base	:	4.924		Total Stress at Ba:	:	4.924		Total Stress at Ba:	:	3.590
	Eff. Stress at Base	:	1.697		Eff. Stress at Bas	:	4.924		Eff. Stress at Bas	:	5.483
	Average Eff. Stress	:	0.849		Average Eff. Stres	:	2.462		Average Eff. Stres	:	2.741

DAY	TIME	ELAPSED TIME (hours)	(min)	(sec)	* SLURRY HEIGHT READING	SETTLEMENT OF SLURRY	* WEIGHT OF WATER & CONT	* WEIGHT OF CONT	VOLUME OF WATER	CUMUL. VOLUME OF WATER	HEIGHT OF SLURRY	VOLUME OF SLURRY	DRY DENSITY t/m^3	i VALUE (av ht)	PERMEABILITY (m/s)
05/10/11	07:57:00	‹ START TIME		0	206.0	0.00	33.3	33.3	0.0	0.0	206.0	563.2	1.310		
05/10/11	07:57:06	0.00	0.10	6	206.0	0.00	0.0	0.0	0.0	0.0	206.0	563.2	1.310	1.60	0.000E+00
05/10/11	07:57:15	0.00	0.25	15	206.0	0.00	0.0	0.0	0.0	0.0	206.0	563.2	1.310	1.60	--
05/10/11	07:57:30	0.01	0.50	30	206.0	0.00	0.0	0.0	0.0	0.0	206.0	563.2	1.310	1.60	--
05/10/11	07:58:00	0.02	1.00	60	206.0	0.00	0.0	0.0	0.0	0.0	206.0	563.2	1.310	1.60	--
05/10/11	07:59:00	0.03	2.00	120	206.0	0.00	0.0	0.0	0.0	0.0	206.0	563.2	1.310	1.60	--
05/10/11	08:02:00	0.08	5.00	300	202.0	4.00	0.0	0.0	0.0	0.0	202.0	552.3	1.336	1.61	--
05/10/11	08:07:00	0.17	10.00	600	198.0	8.00	0.0	0.0	0.0	0.0	198.0	541.3	1.363	1.65	--
05/10/11	08:17:00	0.33	20.00	1200	195.0	11.00	0.0	0.0	0.0	0.0	195.0	533.1	1.384	1.67	--
05/10/11	08:27:00	0.50	30.00	1800	195.0	11.00	0.0	0.0	0.0	0.0	195.0	533.1	1.384	1.69	--
05/10/11	08:57:00	1.00	60.00	3600	195.0	11.00	0.0	0.0	0.0	0.0	195.0	533.1	1.384	1.69	--
05/10/11	09:27:00	1.50	90.00	5400	195.0	11.00	0.0	0.0	0.0	0.0	195.0	533.1	1.384	1.69	--
05/10/11	09:57:00	2.00	120.00	7200	195.0	11.00	0.0	0.0	0.0	0.0	195.0	533.1	1.384	1.69	--
05/10/11	10:27:00	2.50	150.00	9000	195.0	11.00	0.0	0.0	0.0	0.0	195.0	533.1	1.384	1.69	--
05/10/11	11:57:00	4.00	240.00	14400	194.5	11.50	0.0	0.0	0.0	0.0	194.5	531.8	1.387	1.69	--
05/10/11	15:57:00	8.00	480.00	28800	194.5	11.50	682.6	33.3	649.3	649.3	194.5	531.8	1.387	1.69	4.875E-06
06/10/11	08:29:00	24.53	1472.00	88320	193.0	13.00	0.0	0.0	0.0	649.3	193.0	527.7	1.398	1.70	0.000E+00

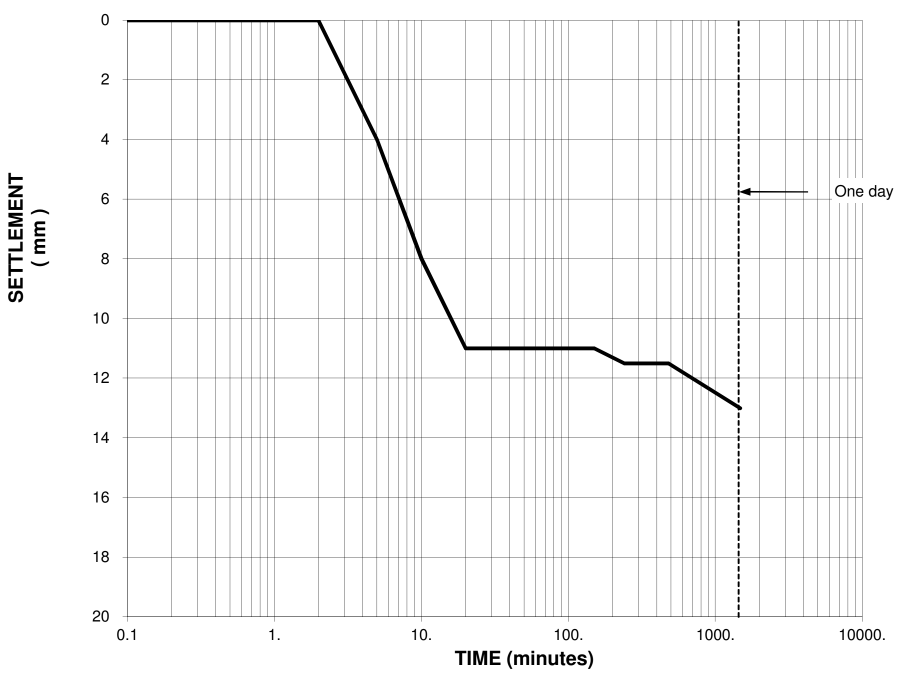
ENT OF CONSOLIDATION

U value	Tv	leasured Value	ROW	X values	Y values	L values	H values	A	B	C	Time(h)	Cv	
													Mv
0.5	0.197	5.75	6	4.000	-1.079			0.004	0.032	-1.266	0.109	162.94	0.0346
				8.000	-0.778	12.000	0.075						Kv
				11.000	-0.477	15.000	0.086						1.911E-06
													Cc
0.6	0.287	6.90	6	4.000	-1.079			0.004	0.032	-1.266	0.134	192.20	0.2571
				8.000	-0.778	12.000	0.075						
				11.000	-0.477	15.000	0.086					177.57	

ELAPSED TIME (minutes)	SETTLEMENT	PERMEABILITY m/s	DRY DENSITY t/m^3
0	0	0.000E+00	1.309653434
0.1	0	0.000E+00	1.309653434
0.25	0		1.309653434
0.500000001	0	--	1.309653434
1	0	--	1.309653434
2	0	--	1.309653434
5	4	--	1.335587166
10	8	--	1.362568725
20	11	--	1.38353132
30	11	--	1.38353132
60	11	--	1.38353132
90	11	--	1.38353132
120	11	--	1.38353132
150	11	--	1.38353132
240	11.5	--	1.387087956
480	11.5	4.875E-06	1.387087956
1472	13	0.000E+00	1.397868433

Cv	1.776E+02
Mv	3.460E-02
Kv	1.911E-06
Cc	2.571E-01

Harper Creek Low Stress Consolidation ROTL 45% Solids



PROJECT NO. VA101-458.03
PROJECT: Harper Creek ROTL 55% solids 20 11
TESTED BY: jhk Denver
DATE: 07/10/11

S.G. SOLIDS : 2.79
S.G. LIQUOR: 1.0000
pH LIQUOR: 10.00

CONSOLIDATION TESTS

CYLINDER:	A
Weight of Container:	852.0
Weight of Slurry + Cont.:	2225.0
Height to Bottom of Slurry:	0.0
Height to Top of Slurry:	326.0
Diameter of Container:	59.0
Settled Top of Slurry Heights:	219
BEFORE CONSOLIDATION:	219.0
AFTER CONSOLIDATION:	208.0
AFTER COMPLETE DRAINAGE:	206.0

DATE		TIME			SLURRY HEIGHT READING	WEIGHT OF WATER + CONTAINER	WEIGHT OF CONTAINER	Water Level
DAY	MONTH	HOUR	MINUTE	SECOND				
10	10	8	43	0	219.0	33.4	33.4	329.0
10	10	8	43	6	219.0			329.0
10	10	8	43	15	219.0			329.0
10	10	8	43	30	218.0			329.0
10	10	8	44	0	217.5			329.0
10	10	8	45	0	216.0			329.0
10	10	8	48	0	214.5			329.0
10	10	8	53	0	212.0			329.0
10	10	9	3	0	209.0			329.0
10	10	9	13	0	208.5			329.0
10	10	9	43	0	208.0			329.0
10	10	10	13	0	208.0			329.0
10	10	10	43	0	208.0			329.0
10	10	11	13	0	208.0			329.0
10	10	12	43	0	208.0			329.0
10	10	16	43	0	208.0	494.6	33.4	329.0
11	10	8	43	0	206.0			193.0

NOTES: The water drained completely overnight

NSOLIDATION TEST

PROJECT	Harper Creek ROTL 55% solids	DATE :	7-Oct-11
SAMPLE	:	TESTED BY :	jhk Denver

*	Weight of Container	:	852.0		PULP DENSITY	:	1.543
*	Weight of Slurry + Container	:	2225.0	*	S.G. SOLIDS	:	2.79
	Weight of Slurry	:	1373.0	*	S.G. LIQUOR	:	1.00
*	Height to Bottom of Slurry	:	0.0		% WT. SOLIDS	:	54.86
*	Height to Top of Slurry	:	326.0		Weight of Solids	:	753.2
	Height of Slurry	:	326.0		Volume of water	:	619.8
*	Internal dia. of Container	:	59.0				
	Volume of Slurry	:	889.8				

BEFORE CONSOLIDATION			AFTER CONSOLIDATION			AFTER COMPLETE DRAINAGE		
* Settled Slurry Ht	:	219.0	* Settled Slurry Ht	:	208.0	* Settled Slurry Ht	:	206.0
Ht of Water	:	107.0	Ht of Water	:	118.0	Ht of Water	:	-
Vol of Slurry	:	597.7	Vol of Slurry	:	567.7	Vol of Slurry	:	562.2
Vol of Water	:	292.0	Vol of Water	:	322.1	Vol of Water	:	-
Dry Density	:	1.260	Dry Density	:	1.327	Dry Density	:	1.340
Pulp Density	:	1.808	Pulp Density	:	1.851	Pulp Density	:	1.859
Void Ratio	:	1.214	Void Ratio	:	1.103	Void Ratio	:	1.083
Total Stress at Base	:	4.933	Total Stress at Ba:	:	4.933	Total Stress at Ba:	:	3.757
Eff. Stress at Base	:	1.736	Eff. Stress at Bas	:	4.933	Eff. Stress at Bas	:	5.777
Average Eff. Stress	:	0.868	Average Eff. Stres	:	2.467	Average Eff. Stres	:	2.888

DAY	TIME	ELAPSED TIME (hours)	(min)	(sec)	* SLURRY HEIGHT READING	SETTLEMENT OF SLURRY	* WEIGHT OF WATER & CONT	* WEIGHT OF CONT	VOLUME OF WATER	CUMUL. VOLUME OF WATER	HEIGHT OF SLURRY	VOLUME OF SLURRY	DRY DENSITY t/m^3	i VALUE (av ht)	PERMEABILITY (m/s)
10/10/11	08:43:00	START TIME		0	219.0	0.00	33.4	33.4	0.0	0.0	219.0	597.7	1.260		
10/10/11	08:43:06	0.00	0.10	6	219.0	0.00	0.0	0.0	0.0	0.0	219.0	597.7	1.260	1.49	0.000E+00
10/10/11	08:43:15	0.00	0.25	15	219.0	0.00	0.0	0.0	0.0	0.0	219.0	597.7	1.260	1.49	--
10/10/11	08:43:30	0.01	0.50	30	218.0	1.00	0.0	0.0	0.0	0.0	218.0	595.0	1.266	1.49	--
10/10/11	08:44:00	0.02	1.00	60	217.5	1.50	0.0	0.0	0.0	0.0	217.5	593.6	1.269	1.50	--
10/10/11	08:45:00	0.03	2.00	120	216.0	3.00	0.0	0.0	0.0	0.0	216.0	589.5	1.278	1.50	--
10/10/11	08:48:00	0.08	5.00	300	214.5	4.50	0.0	0.0	0.0	0.0	214.5	585.4	1.287	1.51	--
10/10/11	08:53:00	0.17	10.00	600	212.0	7.00	0.0	0.0	0.0	0.0	212.0	578.6	1.302	1.53	--
10/10/11	09:03:00	0.33	20.00	1200	209.0	10.00	0.0	0.0	0.0	0.0	209.0	570.4	1.320	1.55	--
10/10/11	09:13:00	0.50	30.00	1800	208.5	10.50	0.0	0.0	0.0	0.0	208.5	569.1	1.324	1.56	--
10/10/11	09:43:00	1.00	60.00	3600	208.0	11.00	0.0	0.0	0.0	0.0	208.0	567.7	1.327	1.57	--
10/10/11	10:13:00	1.50	90.00	5400	208.0	11.00	0.0	0.0	0.0	0.0	208.0	567.7	1.327	1.57	--
10/10/11	10:43:00	2.00	120.00	7200	208.0	11.00	0.0	0.0	0.0	0.0	208.0	567.7	1.327	1.57	--
10/10/11	11:13:00	2.50	150.00	9000	208.0	11.00	0.0	0.0	0.0	0.0	208.0	567.7	1.327	1.57	--
10/10/11	12:43:00	4.00	240.00	14400	208.0	11.00	0.0	0.0	0.0	0.0	208.0	567.7	1.327	1.57	--
10/10/11	16:43:00	8.00	480.00	28800	208.0	11.00	494.6	33.4	461.2	461.2	208.0	567.7	1.327	1.57	3.743E-06
11/10/11	08:43:00	24.00	1440.00	86400	206.0	13.00	0.0	0.0	0.0	461.2	206.0	562.2	1.340	1.57	0.000E+00

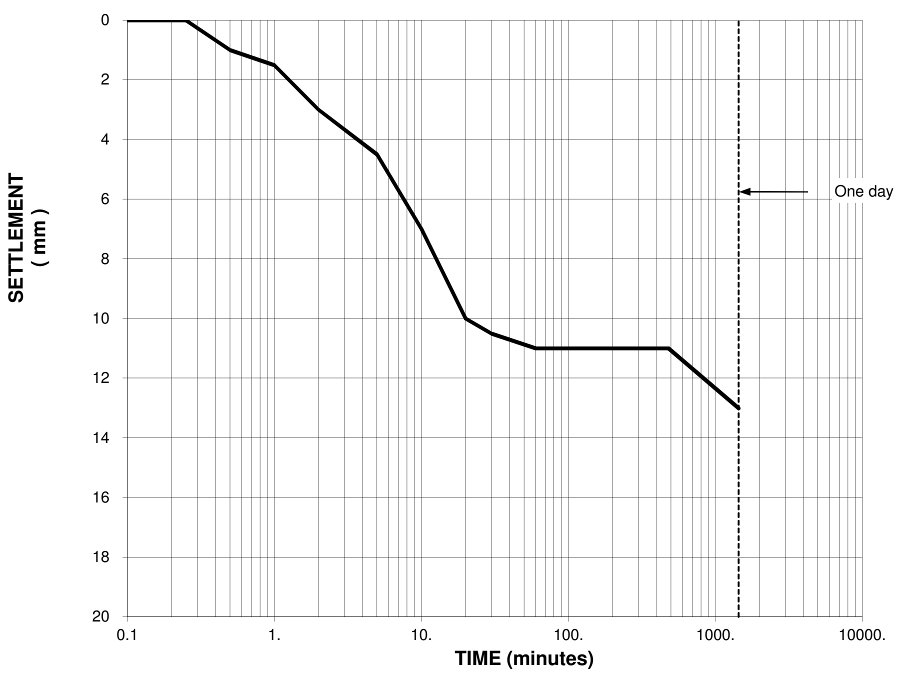
ENT OF CONSOLIDATION

U value	Tv	Measured Value	ROW	X values	Y values	L values	H values	A	B	C	Time(h)	Cv	
													Mv
0.5	0.197	5.50	6	4.500	-1.079			-0.004	0.162	-1.736	0.111	181.18	0.0314
				7.000	-0.778	11.500	0.120						Kv
				10.000	-0.477	14.500	0.109						1.836E-06
													Cc
0.6	0.287	6.60	6	4.500	-1.079			-0.004	0.162	-1.736	0.150	194.66	0.2452
				7.000	-0.778	11.500	0.120						
				10.000	-0.477	14.500	0.109					187.92	

ELAPSED TIME (minutes)	SETTLEMENT	PERMEABILITY m/s	DRY DENSITY t/m^3
0	0	0.000E+00	1.260109659
0.099999999	0	0.000E+00	1.260109659
0.25	0		1.260109659
0.5	1	--	1.265889979
1	1.5	--	1.268800071
2	3	--	1.277611182
5	4.5	--	1.286545526
9.999999999	7	--	1.301717054
20	10	--	1.320401987
30	10.5	--	1.323568419
60	11	--	1.326750074
90	11	--	1.326750074
120	11	--	1.326750074
150	11	--	1.326750074
240	11	--	1.326750074
480	11	3.743E-06	1.326750074
1440	13	0.000E+00	1.339631142

Cv	1.879E+02
Mv	3.142E-02
Kv	1.836E-06
Cc	2.452E-01

Harper Creek Low Stress Consolidation ROTL 55% Solids



PROJECT NO. VA101-458.03
PROJECT: Harper Creek CuROTL 35% solids 20 11
TESTED BY: jhk Denver
DATE: 27/09/11

S.G. SOLIDS : 2.79
S.G. LIQUOR: 1.0000
pH LIQUOR: 10.00

CONSOLIDATION TESTS

CYLINDER:	A
Weight of Container:	220.7
Weight of Slurry + Cont.:	1441.0
Height to Bottom of Slurry:	0.0
Height to Top of Slurry:	335.0
Diameter of Container:	59.0
Settled Top of Slurry Heights:	146
BEFORE CONSOLIDATION:	146.0
AFTER CONSOLIDATION:	135.5
AFTER COMPLETE DRAINAGE:	133.0

DATE		TIME			SLURRY HEIGHT READING	WEIGHT OF WATER + CONTAINER	WEIGHT OF CONTAINER	Water Level
DAY	MONTH	HOUR	MINUTE	SECOND				
29	9	9	1	0	146.0	34.1	34.1	335.0
29	9	9	1	6	145.0			335.0
29	9	9	1	15	144.5			335.0
29	9	9	1	30	144.0			335.0
29	9	9	2	0	143.0			335.0
29	9	9	3	0	142.0			335.0
29	9	9	6	0	139.0			335.0
29	9	9	11	0	136.5			335.0
29	9	9	21	0	136.0			335.0
29	9	9	31	0	135.5			335.0
29	9	10	0	0	135.5			335.0
29	9	10	31	0	135.5			335.0
29	9	11	1	0	135.5			335.0
29	9	11	31	0	135.5			335.0
29	9	13	5	0	135.5			335.0
29	9	16	5	0	135.5	664.7	34.1	335.0
30	9	9	29	0	133.0			133.0

NOTES: The water drained completely overnight

NSOLIDATION TEST

PROJECT	Harper Creek CuROTL 35% solid		DATE	:	27-Sep-11
SAMPLE	:		TESTED BY	:	jhk Denver

*	Weight of Container	:	220.7		PULP DENSITY	:	1.335
*	Weight of Slurry + Container	:	1441.0	*	S.G. SOLIDS	:	2.79
	Weight of Slurry	:	1220.3	*	S.G. LIQUOR	:	1.00
*	Height to Bottom of Slurry	:	0.0		% WT. SOLIDS	:	39.08
*	Height to Top of Slurry	:	335.0		Weight of Solids	:	476.9
	Height of Slurry	:	335.0		Volume of water	:	743.4
*	Internal dia. of Container	:	59.0				
	Volume of Slurry	:	914.3				

BEFORE CONSOLIDATION			AFTER CONSOLIDATION			AFTER COMPLETE DRAINAGE		
* Settled Slurry Ht	:	146.0	* Settled Slurry Ht	:	135.5	* Settled Slurry Ht	:	133.0
Ht of Water	:	189.0	Ht of Water	:	199.5	Ht of Water	:	-
Vol of Slurry	:	398.5	Vol of Slurry	:	369.8	Vol of Slurry	:	363.0
Vol of Water	:	515.8	Vol of Water	:	544.5	Vol of Water	:	-
Dry Density	:	1.197	Dry Density	:	1.290	Dry Density	:	1.314
Pulp Density	:	1.768	Pulp Density	:	1.827	Pulp Density	:	1.843
Void Ratio	:	1.331	Void Ratio	:	1.164	Void Ratio	:	1.124
Total Stress at Base	:	4.385	Total Stress at Ba	:	4.385	Total Stress at Ba	:	2.404
Eff. Stress at Base	:	1.099	Eff. Stress at Bas	:	4.385	Eff. Stress at Bas	:	3.708
Average Eff. Stress	:	0.550	Average Eff. Stres	:	2.192	Average Eff. Stres	:	1.854

DAY	TIME	ELAPSED TIME (hours)	(min)	(sec)	* SLURRY HEIGHT READING	SETTLEMENT OF SLURRY	* WEIGHT OF WATER & CONT	* WEIGHT OF CONT	VOLUME OF WATER	CUMUL. VOLUME OF WATER	HEIGHT OF SLURRY	VOLUME OF SLURRY	DRY DENSITY t/m^3	i VALUE (av ht)	PERMEABILITY (m/s)
29/09/11	09:01:00	< START TIME		0	146.0	0.00	34.1	34.1	0.0	0.0	146.0	398.5	1.197		
29/09/11	09:01:06	0.00	0.10	6	145.0	1.00	0.0	0.0	0.0	0.0	145.0	395.8	1.205	2.30	0.000E+00
29/09/11	09:01:15	0.00	0.25	15	144.5	1.50	0.0	0.0	0.0	0.0	144.5	394.4	1.209	2.31	--
29/09/11	09:01:30	0.01	0.50	30	144.0	2.00	0.0	0.0	0.0	0.0	144.0	393.0	1.213	2.32	--
29/09/11	09:02:00	0.02	1.00	60	143.0	3.00	0.0	0.0	0.0	0.0	143.0	390.3	1.222	2.33	--
29/09/11	09:03:00	0.03	2.00	120	142.0	4.00	0.0	0.0	0.0	0.0	142.0	387.6	1.231	2.35	--
29/09/11	09:06:00	0.08	5.00	300	139.0	7.00	0.0	0.0	0.0	0.0	139.0	379.4	1.257	2.38	--
29/09/11	09:11:00	0.17	10.00	600	136.5	9.50	0.0	0.0	0.0	0.0	136.5	372.6	1.280	2.43	--
29/09/11	09:21:00	0.33	20.00	1200	136.0	10.00	0.0	0.0	0.0	0.0	136.0	371.2	1.285	2.46	--
29/09/11	09:31:00	0.50	30.00	1800	135.5	10.50	0.0	0.0	0.0	0.0	135.5	369.8	1.290	2.47	--
29/09/11	10:00:00	0.98	59.00	3540	135.5	10.50	0.0	0.0	0.0	0.0	135.5	369.8	1.290	2.47	--
29/09/11	10:31:00	1.50	90.00	5400	135.5	10.50	0.0	0.0	0.0	0.0	135.5	369.8	1.290	2.47	--
29/09/11	11:01:00	2.00	120.00	7200	135.5	10.50	0.0	0.0	0.0	0.0	135.5	369.8	1.290	2.47	--
29/09/11	11:31:00	2.50	150.00	9000	135.5	10.50	0.0	0.0	0.0	0.0	135.5	369.8	1.290	2.47	--
29/09/11	13:05:00	4.07	244.00	14640	135.5	10.50	0.0	0.0	0.0	0.0	135.5	369.8	1.290	2.47	--
29/09/11	16:05:00	7.07	424.00	25440	135.5	10.50	664.7	34.1	630.6	630.6	135.5	369.8	1.290	2.47	3.674E-06
30/09/11	09:29:00	24.47	1468.00	88080	133.0	13.00	0.0	0.0	0.0	630.6	133.0	363.0	1.314	2.50	0.000E+00

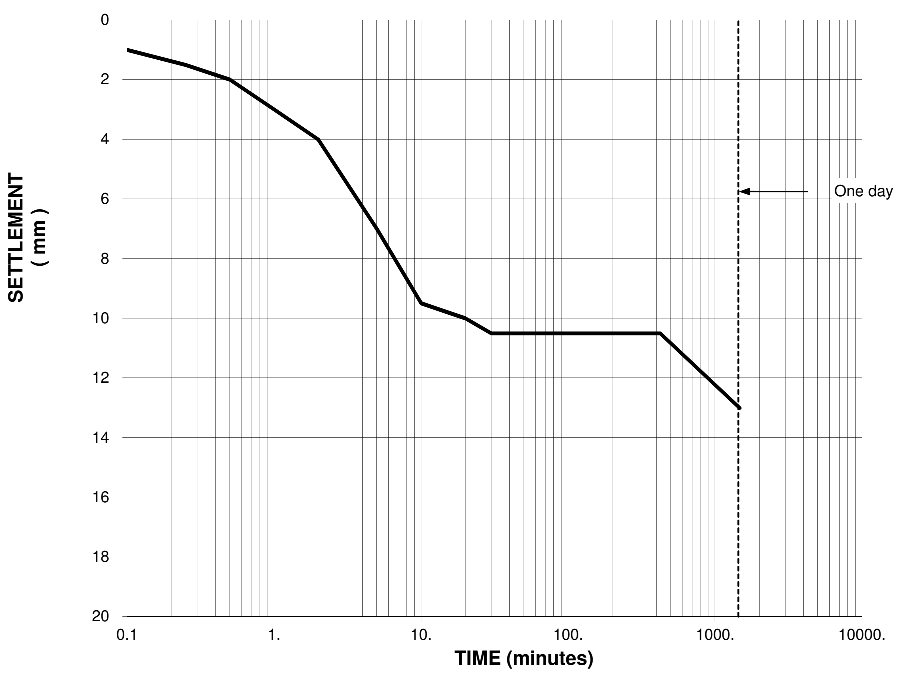
ENT OF CONSOLIDATION

U value	Tv	Measured Value	ROW	X values	Y values	L values	H values	A	B	C	Time(h)	Cv	
													Mv
0.5	0.197	5.25	5	4.000	-1.477			-0.002	0.157	-2.070	0.049	179.60	0.0438
				7.000	-1.079	11.000	0.133						Kv
				9.500	-0.778	13.500	0.127						2.509E-06
													Cc
0.6	0.287	6.30	5	4.000	-1.477			-0.002	0.157	-2.070	0.068	189.03	0.2791
				7.000	-1.079	11.000	0.133						
				9.500	-0.778	13.500	0.127					184.32	

ELAPSED TIME (minutes)	SETTLEMENT	PERMEABILITY m/s	DRY DENSITY t/m^3
0	0	0.000E+00	1.19680017
0.1	1	0.000E+00	1.205053964
0.25	1.5		1.209223701
0.5	2	--	1.213422394
1	3	--	1.221907865
2.000000001	4	--	1.23051285
5	7	--	1.257070682
10	9.5	--	1.280093954
20	10	--	1.284800182
30	10.5	--	1.289541142
59	10.5	--	1.289541142
90	10.5	--	1.289541142
120	10.5	--	1.289541142
150	10.5	--	1.289541142
244	10.5	--	1.289541142
424	10.5	3.674E-06	1.289541142
1468	13	0.000E+00	1.313780637

Cv	1.843E+02
Mv	4.378E-02
Kv	2.509E-06
Cc	2.791E-01

Harper Creek Low Stress Consolidation CuROTL 35% Solids



PROJECT NO. VA101-458.03
PROJECT: Harper Creek CuROTL 45% solids 20 11
TESTED BY: jhk Denver
DATE: 03/10/11

S.G. SOLIDS : 2.79
S.G. LIQUOR: 1.0000
pH LIQUOR: 10.00

CONSOLIDATION TESTS

CYLINDER:	B
Weight of Container:	218.9
Weight of Slurry + Cont.:	1567.7
Height to Bottom of Slurry:	0.0
Height to Top of Slurry:	345.0
Diameter of Container:	58.8
Settled Top of Slurry Heights:	189
BEFORE CONSOLIDATION:	189.0
AFTER CONSOLIDATION:	178.0
AFTER COMPLETE DRAINAGE:	176.0

DATE DAY	MONTH	TIME HOUR	MINUTE	SECOND	SLURRY HEIGHT READING	WEIGHT OF WATER + CONTAINER	WEIGHT OF CONTAINER	Water Level
5	10	7	58	0	189.0	33.4	33.4	345.0
5	10	7	58	6	189.0			345.0
5	10	7	58	15	188.0			345.0
5	10	7	58	30	187.0			345.0
5	10	7	59	0	186.5			345.0
5	10	8	0	0	185.0			345.0
5	10	8	3	0	182.5			345.0
5	10	8	8	0	180.0			345.0
5	10	8	18	0	178.0			345.0
5	10	8	27	0	178.0			345.0
5	10	8	58	0	178.0			345.0
5	10	9	28	0	178.0			345.0
5	10	9	58	0	178.0			345.0
5	10	10	28	0	178.0			345.0
5	10	11	58	0	178.0			345.0
5	10	16	58	0	178.0	655.2	33.4	345.0
6	10	8	30	0	176.0			176.0

NOTES: The water drained completely overnight

NSOLIDATION TEST

	PROJECT	Harper Creek CuROTL 45% solid			DATE	:	3-Oct-11
	SAMPLE	:			TESTED BY	:	jhk Denver
*	Weight of Container	:	218.9		PULP DENSITY	:	1.442
*	Weight of Slurry + Container	:	1567.7	*	S.G. SOLIDS	:	2.79
	Weight of Slurry	:	1348.8	*	S.G. LIQUOR	:	1.00
*	Height to Bottom of Slurry	:	0.0		% WT. SOLIDS	:	47.75
*	Height to Top of Slurry	:	345.0		Weight of Solids	:	644.1
	Height of Slurry	:	345.0		Volume of water	:	704.7
*	Internal dia. of Container	:	58.8				
	Volume of Slurry	:	935.6				

	BEFORE CONSOLIDATION				AFTER CONSOLIDATION				AFTER COMPLETE DRAINAGE		
*	Settled Slurry Ht	:	189.0	*	Settled Slurry Ht	:	178.0	*	Settled Slurry Ht	:	176.0
	Ht of Water	:	156.0		Ht of Water	:	167.0		Ht of Water	:	-
	Vol of Slurry	:	512.5		Vol of Slurry	:	482.7		Vol of Slurry	:	477.3
	Vol of Water	:	423.0		Vol of Water	:	452.9		Vol of Water	:	-
	Dry Density	:	1.257		Dry Density	:	1.334		Dry Density	:	1.350
	Pulp Density	:	1.806		Pulp Density	:	1.856		Pulp Density	:	1.866
	Void Ratio	:	1.220		Void Ratio	:	1.091		Void Ratio	:	1.067
	Total Stress at Base	:	4.878		Total Stress at Ba	:	4.878		Total Stress at Ba	:	3.220
	Eff. Stress at Base	:	1.494		Eff. Stress at Bas	:	4.878		Eff. Stress at Bas	:	4.947
	Average Eff. Stress	:	0.747		Average Eff. Stres	:	2.439		Average Eff. Stres	:	2.473

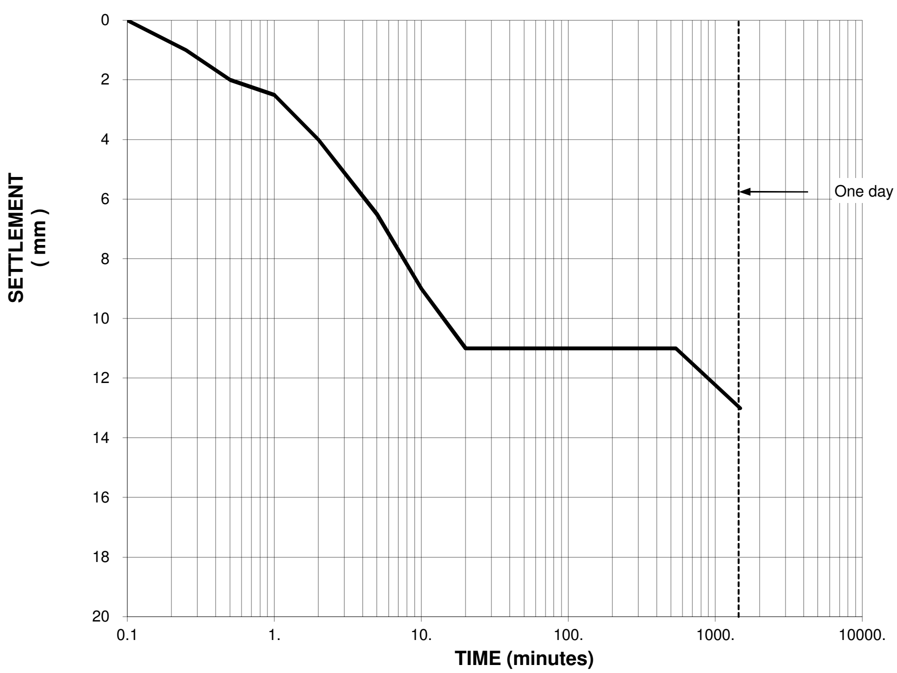
DAY	TIME	ELAPSED TIME (hours)	(min)	(sec)	* SLURRY HEIGHT READING	SETTLEMENT OF SLURRY	* WEIGHT OF WATER & CONT	* WEIGHT OF CONT	VOLUME OF WATER	CUMUL. VOLUME OF WATER	HEIGHT OF SLURRY	VOLUME OF SLURRY	DRY DENSITY t/m^3	i VALUE (av ht)	PERMEABILITY (m/s)
05/10/11	07:58:00	< START TIME		0	189.0	0.00	33.4	33.4	0.0	0.0	189.0	512.5	1.257		
05/10/11	07:58:06	0.00	0.10	6	189.0	0.00	0.0	0.0	0.0	0.0	189.0	512.5	1.257	1.83	0.000E+00
05/10/11	07:58:15	0.00	0.25	15	188.0	1.00	0.0	0.0	0.0	0.0	188.0	509.8	1.263	1.83	--
05/10/11	07:58:30	0.01	0.50	30	187.0	2.00	0.0	0.0	0.0	0.0	187.0	507.1	1.270	1.84	--
05/10/11	07:59:00	0.02	1.00	60	186.5	2.50	0.0	0.0	0.0	0.0	186.5	505.7	1.274	1.85	--
05/10/11	08:00:00	0.03	2.00	120	185.0	4.00	0.0	0.0	0.0	0.0	185.0	501.7	1.284	1.86	--
05/10/11	08:03:00	0.08	5.00	300	182.5	6.50	0.0	0.0	0.0	0.0	182.5	494.9	1.301	1.88	--
05/10/11	08:08:00	0.17	10.00	600	180.0	9.00	0.0	0.0	0.0	0.0	180.0	488.1	1.320	1.90	--
05/10/11	08:18:00	0.33	20.00	1200	178.0	11.00	0.0	0.0	0.0	0.0	178.0	482.7	1.334	1.93	--
05/10/11	08:27:00	0.48	29.00	1740	178.0	11.00	0.0	0.0	0.0	0.0	178.0	482.7	1.334	1.94	--
05/10/11	08:58:00	1.00	60.00	3600	178.0	11.00	0.0	0.0	0.0	0.0	178.0	482.7	1.334	1.94	--
05/10/11	09:28:00	1.50	90.00	5400	178.0	11.00	0.0	0.0	0.0	0.0	178.0	482.7	1.334	1.94	--
05/10/11	09:58:00	2.00	120.00	7200	178.0	11.00	0.0	0.0	0.0	0.0	178.0	482.7	1.334	1.94	--
05/10/11	10:28:00	2.50	150.00	9000	178.0	11.00	0.0	0.0	0.0	0.0	178.0	482.7	1.334	1.94	--
05/10/11	11:58:00	4.00	240.00	14400	178.0	11.00	0.0	0.0	0.0	0.0	178.0	482.7	1.334	1.94	--
05/10/11	16:58:00	9.00	540.00	32400	178.0	11.00	655.2	33.4	621.8	621.8	178.0	482.7	1.334	1.94	3.651E-06
06/10/11	08:30:00	24.53	1472.00	88320	176.0	13.00	0.0	0.0	0.0	621.8	176.0	477.3	1.350	1.95	0.000E+00

ENT OF CONSOLIDATION

U value	Tv	Measured Value	ROW	X values	Y values	L values	H values	A	B	C	Time(h)	Cv	
													Mv
0.5	0.197	5.50	5	4.000	-1.477			-0.008	0.241	-2.315	0.059	252.25	0.0344
				6.500	-1.079	10.500	0.159						Kv
				9.000	-0.778	13.000	0.140						2.708E-06
													Cc
0.6	0.287	6.60	6	6.500	-1.079			0.007	0.017	-1.471	0.085	253.92	0.2515
				9.000	-0.778	15.500	0.120						
				11.000	-0.477	17.500	0.134					253.08	

ELAPSED TIME (minutes)	SETTLEMENT	PERMEABILITY m/s	DRY DENSITY t/m^3		
				Cv	2.531E+02
				Mv	3.440E-02
				Kv	2.708E-06
0	0	0.000E+00	1.256713312	Cc	2.515E-01
0.099999999	0	0.000E+00	1.256713312		
0.25	1		1.263397957		
0.5	2	--	1.270154096		
1	2.5	--	1.273559335		
2	4	--	1.283885492		
5	6.5	--	1.301472964		
9.999999999	9	--	1.319548978		
20	11	--	1.334375371		
29	11	--	1.334375371		
60	11	--	1.334375371		
90	11	--	1.334375371		
120	11	--	1.334375371		
150	11	--	1.334375371		
240	11	--	1.334375371		
540	11	3.651E-06	1.334375371		
1472	13	0.000E+00	1.349538727		

Harper Creek Low Stress Consolidation CuROTL 45% Solids



PROJECT NO. VA101-458.03
PROJECT: Harper Creek CuROTL 55% solids 20 11
TESTED BY: jhk Denver
DATE: 07/10/11

S.G. SOLIDS : 2.79
S.G. LIQUOR: 1.0000
pH LIQUOR: 10.00

CONSOLIDATION TESTS

CYLINDER:	B
Weight of Container:	865.0
Weight of Slurry + Cont.:	2271.0
Height to Bottom of Slurry:	0.0
Height to Top of Slurry:	337.0
Diameter of Container:	58.8
Settled Top of Slurry Heights:	228
BEFORE CONSOLIDATION:	228.0
AFTER CONSOLIDATION:	218.0
AFTER COMPLETE DRAINAGE:	218.0

DATE		TIME			SLURRY	WEIGHT	WEIGHT	Water
					HEIGHT	OF WATER +	OF	Level
DAY	MONTH	HOUR	MINUTE	SECOND	READING	CONTAINER	CONTAINER	
10	10	8	45	0	228.0	33.4	33.4	337.0
10	10	8	45	6	228.0			337.0
10	10	8	45	15	228.0			337.0
10	10	8	45	30	227.0			337.0
10	10	8	46	0	226.5			337.0
10	10	8	47	0	225.0			337.0
10	10	8	50	0	223.5			337.0
10	10	8	55	0	221.5			337.0
10	10	9	5	0	219.0			337.0
10	10	9	15	0	218.0			337.0
10	10	9	45	0	218.0			337.0
10	10	10	15	0	218.0			337.0
10	10	10	45	0	218.0			337.0
10	10	11	45	0	218.0			337.0
10	10	12	45	0	218.0			337.0
10	10	16	45	0	218.0	508.3	33.4	337.0
11	10	8	45	0	218.0			218.0

NOTES: The water drained completely overnight

NSOLIDATION TEST

PROJECT	Harper Creek CuROTL 55% solids		DATE	:	7-Oct-11
SAMPLE	:		TESTED BY	:	jhk Denver

*	Weight of Container	:	865.0		PULP DENSITY	:	1.539
*	Weight of Slurry + Container	:	2271.0	*	S.G. SOLIDS	:	2.79
	Weight of Slurry	:	1406.0	*	S.G. LIQUOR	:	1.00
*	Height to Bottom of Slurry	:	0.0		% WT. SOLIDS	:	54.56
*	Height to Top of Slurry	:	337.0		Weight of Solids	:	767.1
	Height of Slurry	:	337.0		Volume of water	:	638.9
*	Internal dia. of Container	:	58.8				
	Volume of Slurry	:	913.9				

BEFORE CONSOLIDATION			AFTER CONSOLIDATION			AFTER COMPLETE DRAINAGE		
* Settled Slurry Ht	:	228.0	* Settled Slurry Ht	:	218.0	* Settled Slurry Ht	:	218.0
Ht of Water	:	109.0	Ht of Water	:	119.0	Ht of Water	:	-
Vol of Slurry	:	618.3	Vol of Slurry	:	591.2	Vol of Slurry	:	591.2
Vol of Water	:	295.6	Vol of Water	:	322.7	Vol of Water	:	-
Dry Density	:	1.241	Dry Density	:	1.298	Dry Density	:	1.298
Pulp Density	:	1.796	Pulp Density	:	1.832	Pulp Density	:	1.832
Void Ratio	:	1.249	Void Ratio	:	1.150	Void Ratio	:	1.150
Total Stress at Base	:	5.085	Total Stress at Ba	:	5.085	Total Stress at Ba	:	3.918
Eff. Stress at Base	:	1.780	Eff. Stress at Bas	:	5.085	Eff. Stress at Bas	:	6.056
Average Eff. Stress	:	0.890	Average Eff. Stres	:	2.542	Average Eff. Stres	:	3.028

DAY	TIME	ELAPSED TIME (hours)	(min)	(sec)	* SLURRY HEIGHT READING	SETTLEMENT OF SLURRY	* WEIGHT OF WATER & CONT	* WEIGHT OF CONT	VOLUME OF WATER	CUMUL. VOLUME OF WATER	HEIGHT OF SLURRY	VOLUME OF SLURRY	DRY DENSITY t/m^3	i VALUE (av ht)	PERMEABILITY (m/s)
10/10/11	08:45:00	< START TIME		0	228.0	0.00	33.4	33.4	0.0	0.0	228.0	618.3	1.241		
10/10/11	08:45:06	0.00	0.10	6	228.0	0.00	0.0	0.0	0.0	0.0	228.0	618.3	1.241	1.48	0.000E+00
10/10/11	08:45:15	0.00	0.25	15	228.0	0.00	0.0	0.0	0.0	0.0	228.0	618.3	1.241	1.48	--
10/10/11	08:45:30	0.01	0.50	30	227.0	1.00	0.0	0.0	0.0	0.0	227.0	615.6	1.246	1.48	--
10/10/11	08:46:00	0.02	1.00	60	226.5	1.50	0.0	0.0	0.0	0.0	226.5	614.2	1.249	1.49	--
10/10/11	08:47:00	0.03	2.00	120	225.0	3.00	0.0	0.0	0.0	0.0	225.0	610.1	1.257	1.49	--
10/10/11	08:50:00	0.08	5.00	300	223.5	4.50	0.0	0.0	0.0	0.0	223.5	606.1	1.266	1.50	--
10/10/11	08:55:00	0.17	10.00	600	221.5	6.50	0.0	0.0	0.0	0.0	221.5	600.7	1.277	1.51	--
10/10/11	09:05:00	0.33	20.00	1200	219.0	9.00	0.0	0.0	0.0	0.0	219.0	593.9	1.292	1.53	--
10/10/11	09:15:00	0.50	30.00	1800	218.0	10.00	0.0	0.0	0.0	0.0	218.0	591.2	1.298	1.54	--
10/10/11	09:45:00	1.00	60.00	3600	218.0	10.00	0.0	0.0	0.0	0.0	218.0	591.2	1.298	1.55	--
10/10/11	10:15:00	1.50	90.00	5400	218.0	10.00	0.0	0.0	0.0	0.0	218.0	591.2	1.298	1.55	--
10/10/11	10:45:00	2.00	120.00	7200	218.0	10.00	0.0	0.0	0.0	0.0	218.0	591.2	1.298	1.55	--
10/10/11	11:45:00	3.00	180.00	10800	218.0	10.00	0.0	0.0	0.0	0.0	218.0	591.2	1.298	1.55	--
10/10/11	12:45:00	4.00	240.00	14400	218.0	10.00	0.0	0.0	0.0	0.0	218.0	591.2	1.298	1.55	--
10/10/11	16:45:00	8.00	480.00	28800	218.0	10.00	508.3	33.4	474.9	474.9	218.0	591.2	1.298	1.55	3.934E-06
11/10/11	08:45:00	24.00	1440.00	86400	218.0	10.00	0.0	0.0	0.0	474.9	218.0	591.2	1.298	1.55	0.000E+00

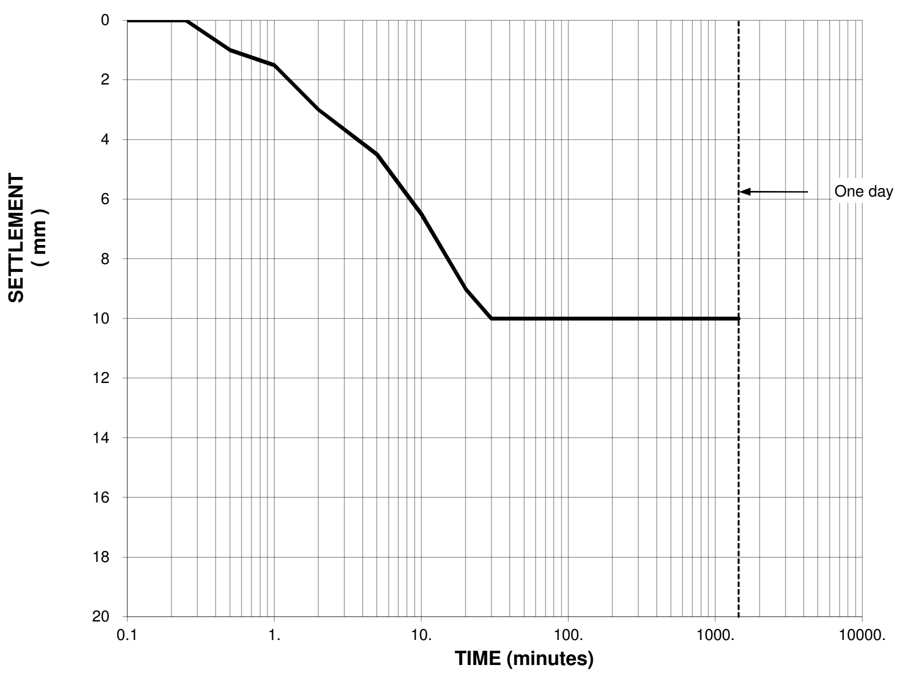
ENT OF CONSOLIDATION

U value	Tv	leasured Value	ROW	X values	Y values	L values	H values	A	B	C	Time(h)	Cv	
													Mv
0.5	0.197	5.00	6	4.500	-1.079			-0.007	0.224	-1.952	0.100	218.83	0.0265
				6.500	-0.778	11.000	0.151						Kv
				9.000	-0.477	13.500	0.134						1.829E-06
													Cc
0.6	0.287	6.00	6	4.500	-1.079			-0.007	0.224	-1.952	0.142	224.43	0.2163
				6.500	-0.778	11.000	0.151						
				9.000	-0.477	13.500	0.134					221.63	

ELAPSED TIME (minutes)	SETTLEMENT	PERMEABILITY m/s	DRY DENSITY t/m^3
0	0	0.000E+00	1.240636878
0.099999999	0	0.000E+00	1.240636878
0.25	0		1.240636878
0.5	1	--	1.246102238
1	1.5	--	1.248853016
2	3	--	1.257178703
5	4.5	--	1.265616144
9.999999999	6.5	--	1.277043829
20	9	--	1.291621955
30	10	--	1.297546826
60	10	--	1.297546826
90	10	--	1.297546826
120	10	--	1.297546826
180	10	--	1.297546826
240	10	--	1.297546826
480	10	3.934E-06	1.297546826
1440	10	0.000E+00	1.297546826

Cv	2.216E+02
Mv	2.654E-02
Kv	1.829E-06
Cc	2.163E-01

Harper Creek Low Stress Consolidation CuROTL 55% Solids



Knight Piésold CONSULTING

SETTLING AND DRYING TEST (including Evaporation Control)

Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	35%	Test Date:	9/23/2011-10/5/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters for Settling and Drying Test

a. Beaker (Tare) Weight =	399.17 g	d. Moisture Content (from drying test) =	**179.4** %		
b. Initial Slurry Volume =	875 cm³	e. Initial Slurry Bulk Density [(c-a)/b] =	1.29 g/cm³		
c. Tare + Initial Slurry Weight =	1524.3 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	722 g		
Time of Readings	23-Sep-11	g. Weight of Solids [(c-a)/(1+d/100)] =	403 g		
	08:43 AM	h. Tailings Solids Specific Gravity =	2.79		
		i. Solids Volume [g/h] =	144.4 cm³		

Initial Parameters for Evaporation Control

- x. Beaker Tare Weight = 391 g
- y. Initial Weight of Beaker = 1028 g
- z. Beaker Cross-Sectional Area = 84.13 cm²

On-going Readings

	Date of Reading	Time of Reading	A. Total Remaining Weight (g)	B. Total Remaining Volume (cm³)	C. Settled Slurry Volume (cm³)	D. Decanted Water Volume (if any) (cm³)	E. Shrinkage Crack Volume (estimated) (cm³)	F. Net. Slurry Volume [C-E] (cm³)	G. Volume Reduction [(b-F)/b] (%)	H. Slurry Dry Density [g/F] (g/cm³)	I. Moisture Content [(A-a)/g]-1 (%)	J. Saturation (A-a-g)/(B-i) (%)	Comments	Evaporation Control: Total Weight After Decant (g)	Decanted Weight (if any) (g)	Evap. (mm)
1	23-Sep-11	8:48 AM	1524	875.0	650.0	246.3		650.0	25.7	0.62	179.3	100.0	Water Decanted	1028	0	0
2	23-Sep-11	9:02 AM	1278	630.0	400.0	210.4		400.0	54.3	1.01	118.1	100.0	Water Decanted	1028	0	0
3	23-Sep-11	10:21 AM	1066	425.0	315.0	88.7		315.0	64.0	1.28	65.7	100.0	Water Decanted	1027	0	0
4	23-Sep-11	12:49 PM	976	340.0	300.0	13.7		300.0	65.7	1.34	43.3	100.0	Water Decanted	1025	0	0
5	23-Sep-11	5:37 PM	960	325.0	300.0	3.7		300.0	65.7	1.34	39.1	100.0	Water Decanted	1021	0	1
6	24-Sep-11	11:00 AM	944	295.0	295.0	0.0		295.0	66.3	1.37	35.4	94.6	No free water	1005	0	3
7	25-Sep-11	12:34 PM	927	290.0	290.0	0.0		290.0	66.9	1.39	31.0	85.7	No free water	980	0	6
8	26-Sep-11	10:44 AM	912	280.0	280.0	0.0		280.0	68.0	1.44	27.3	81.1	No free water	961	0	8
9	27-Sep-11	10:21 AM	893	280.0	280.0	0.0		280.0	68.0	1.44	22.6	67.2	No free water	937	0	11
10	28-Sep-11	8:39 AM	875	280.0	280.0	0.0		280.0	68.0	1.44	18.2	54.1	No free water	915	0	13
11	29-Sep-11	10:04 AM	855		280.9		1.6	279.3	68.1	1.44	13.2	39.5	Speciman pulling from sides	890	0	16
12	30-Sep-11	10:36 AM	835		280.9		7.5	273.4	68.8	1.47	8.2	25.7	Measured in jar	864	0	19
13	02-Oct-11	9:49 AM	806		280.9		11.7	269.1	69.2	1.50	0.9	3.0	Measured in jar	825	0	24
14	03-Oct-11	9:03 AM	803		280.9		12.3	268.6	69.3	1.50	0.2	0.5	Measured in jar	804	0	27
15	05-Oct-11	9:34 AM	803		280.9		12.3	268.6	69.3	1.50	0.2	0.5	Measured in jar	760	0	32

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Notes:

Knight Piésold CONSULTING

SETTLING AND DRYING TEST (including Evaporation Control)

Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	45%	Test Date:	9/23/2011-10/5/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters for Settling and Drying Test

a.	Beaker (Tare) Weight =	399.67 g	d. Moisture Content (from drying test) =	**120.3**	%
b.	Initial Slurry Volume =	710 cm³	e. Initial Slurry Bulk Density [(c-a)/b] =	1.41	g/cm³
c.	Tare + Initial Slurry Weight =	1402.6 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	548	g
	Time of Readings	23-Sep-11	g. Weight of Solids [(c-a)/(1+d/100)] =	455	g
		08:32 AM	h. Tailings Solids Specific Gravity =	2.79	
			i. Solids Volume [g/h] =	163.2	cm³

Initial Parameters for Evaporation Control

x. Beaker Tare Weight =	391	g
y. Initial Weight of Beaker =	1028	g
z. Beaker Cross-Sectional Area =	84.13	cm²

On-going Readings

	Date of Reading	Time of Reading	A. Total Remaining Weight (g)	B. Total Remaining Volume (cm³)	C. Settled Slurry Volume (cm³)	D. Decanted Water Volume (if any) (cm³)	E. Shrinkage Crack Volume (estimated) (cm³)	F. Net. Slurry Volume [C-E] (cm³)	G. Volume Reduction [(b-F)/b] (%)	H. Slurry Dry Density [g/F] (g/cm³)	I. Moisture Content [(A-a)/g]-1 (%)	J. Saturation (A-a-g)/(B-i) (%)	Comments	Evaporation Control: Total Weight After Decant (g)	Evaporation Control: Decanted Weight (if any) (g)	Evaporation Control: Evap. (mm)
1	23-Sep-11	8:43 AM	1403	710.0	490.0	214.8		490.0	31.0	0.93	120.3	100.0	Water Decanted	1028	0	0
2	23-Sep-11	8:58 AM	1187	500.0	390.0	105.2		390.0	45.1	1.17	73.0	100.0	Water Decanted	1028	0	0
3	23-Sep-11	10:18 AM	1081	400.0	325.0	52.0		325.0	54.2	1.40	49.7	100.0	Water Decanted	1027	0	0
4	23-Sep-11	12:46 PM	1028	350.0	320.0	12.8		320.0	54.9	1.42	38.0	100.0	Water Decanted	1025	0	0
5	23-Sep-11	5:32 PM	1012	325.0	305.0	3.2		305.0	57.0	1.49	34.5	100.0	Water Decanted	1022	0	1
6	24-Sep-11	11:00 AM	997	300.0	300.0	0.0		300.0	57.7	1.52	31.2	100.0	No free water	1005	0	3
7	25-Sep-11	12:32 PM	979	300.0	300.0	0.0		300.0	57.7	1.52	27.3	90.7	No free water	980	0	6
8	26-Sep-11	10:41 AM	964	300.0	300.0	0.0		300.0	57.7	1.52	24.0	79.9	No free water	961	0	8
9	27-Sep-11	10:18 AM	945	300.0	300.0	0.0		300.0	57.7	1.52	19.8	66.0	No free water	937	0	11
10	28-Sep-11	8:31 AM	928		296.7		1.7	295.0	58.4	1.54	15.9	55.0	Speciman pulling from sides	915	0	13
11	29-Sep-11	9:56 AM	907		295.1		5.1	290.0	59.2	1.57	11.5	41.1	measured in jar	890	0	16
12	30-Sep-11	10:31 AM	887		295.1		6.8	288.3	59.4	1.58	7.0	25.5	measured in jar	864	0	19
13	02-Oct-11	9:46 AM	859		295.1		7.3	287.7	59.5	1.58	0.8	3.0	measured in jar	825	0	24
14	03-Oct-11	9:01 AM	856		295.1		7.3	287.7	59.5	1.58	0.1	0.5	measured in jar	804	0	27
15	05-Oct-11	9:33 AM	855		295.1		7.3	287.7	59.5	1.58	0.1	0.4	measured in jar	760	0	32

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Notes:

Knight Piésold CONSULTING	SETTLING AND DRYING TEST (including Evaporation Control)	Project No. VA101-458.03

Project:	Harper Creek	Sample No.:	55%	Test Date:	9/23/2011-10/9/2011
	ROTL	Location:	ROTL	Tested By:	jdb/jhk

Initial Parameters for Settling and Drying Test

a.	Beaker (Tare) Weight =	396.11 g	d. Moisture Content (from drying test) =	**76.6**	%
b.	Initial Slurry Volume =	650 cm³	e. Initial Slurry Bulk Density [(c-a)/b] =	1.59	g/cm³
c.	Tare + Initial Slurry Weight =	1427.1 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	447	g
	Time of Readings	23-Sep-11	g. Weight of Solids [(c-a)/(1+d/100)] =	584	g
		08:19 AM	h. Tailings Solids Specific Gravity =	2.79	
			i. Solids Volume [g/h] =	209.2	cm³

Initial Parameters for Evaporation Control

x. Beaker Tare Weight =	391	g
y. Initial Weight of Beaker =	1028	g
z. Beaker Cross-Sectional Area =	84.13	cm²

On-going Readings

	Date of Reading	Time of Reading	A. Total Remaining Weight (g)	B. Total Remaining Volume (cm³)	C. Settled Slurry Volume (cm³)	D. Decanted Water Volume (if any) (cm³)	E. Shrinkage Crack Volume (estimated) (cm³)	F. Net. Slurry Volume [C-E] (cm³)	G. Volume Reduction [(b-F)/b] (%)	H. Slurry Dry Density [g/F] (g/cm³)	I. Moisture Content [(A-a)/g]-1 (%)	J. Saturation (A-a-g)/(B-i) (%)	Comments	Evaporation Control: Total Weight After Decant (g)	Evaporation Control: Decanted Weight (if any) (g)	Evaporation Control: Evap. (mm)
1	23-Sep-11	8:22 AM	1427	650.0	610.0	72.1		610.0	6.2	0.96	76.6	100.0	Water Decanted	1028	0	0
2	23-Sep-11	8:54 AM	1354	590.0	430.0	136.0		430.0	33.8	1.36	64.2	100.0	Water Decanted	1028	0	0
3	23-Sep-11	10:15 AM	1217	450.0	400.0	33.7		400.0	38.5	1.46	40.7	100.0	Water Decanted	1027	0	0
4	23-Sep-11	12:41 PM	1182	425.0	390.0	16.9		390.0	40.0	1.50	34.7	100.0	Water Decanted	1025	0	0
5	23-Sep-11	5:29 PM	1162	400.0	380.0	0.0		380.0	41.5	1.54	31.2	100.0	No free water observed	1022	0	1
6	24-Sep-11	11:00 AM	1149	375.0	375.0	0.0		375.0	42.3	1.56	29.1	100.0	No free water observed	1005	0	3
7	25-Sep-11	12:29 PM	1131	375.0	375.0	0.0		375.0	42.3	1.56	25.9	91.2	No free water observed	981	0	6
8	26-Sep-11	10:37 AM	1116	375.0	375.0	0.0		375.0	42.3	1.56	23.3	81.9	No free water observed	961	0	8
9	27-Sep-11	10:14 AM	1097	375.0	375.0	0.0		375.0	42.3	1.56	20.0	70.5	No free water observed	937	0	11
10	28-Sep-11	8:27 AM	1080	375.0	375.0	0.0		375.0	42.3	1.56	17.1	60.1	No free water observed	915	0	13
11	29-Sep-11	9:53 AM	1060		375.2		1.4	373.8	42.5	1.56	13.7	48.6	Speciman pulling from sides	890	0	16
12	30-Sep-11	10:28 AM	1040		374.4		2.2	372.2	42.7	1.57	10.3	37.0	measured in jar	864	0	19
13	02-Oct-11	9:43 AM	1009		372.7		5.0	367.7	43.4	1.59	4.9	18.2	measured in jar	825	0	24
14	03-Oct-11	8:57 AM	993		370.1		6.4	363.7	44.0	1.60	2.3	8.6	measured in jar	804	0	27
15	05-Oct-11	9:30 AM	982		370.1		7.1	363.0	44.1	1.61	0.4	1.6	measured in jar	760	0	32
16	07-Oct-11	10:43 AM	981		370.1		7.1	363.0	44.1	1.61	0.2	0.6	measured in jar	714	0	37
17	09-Oct-11	12:07 PM	981		370.1		7.1	363.0	44.1	1.61	0.2	0.6	measured in jar	714	0	37

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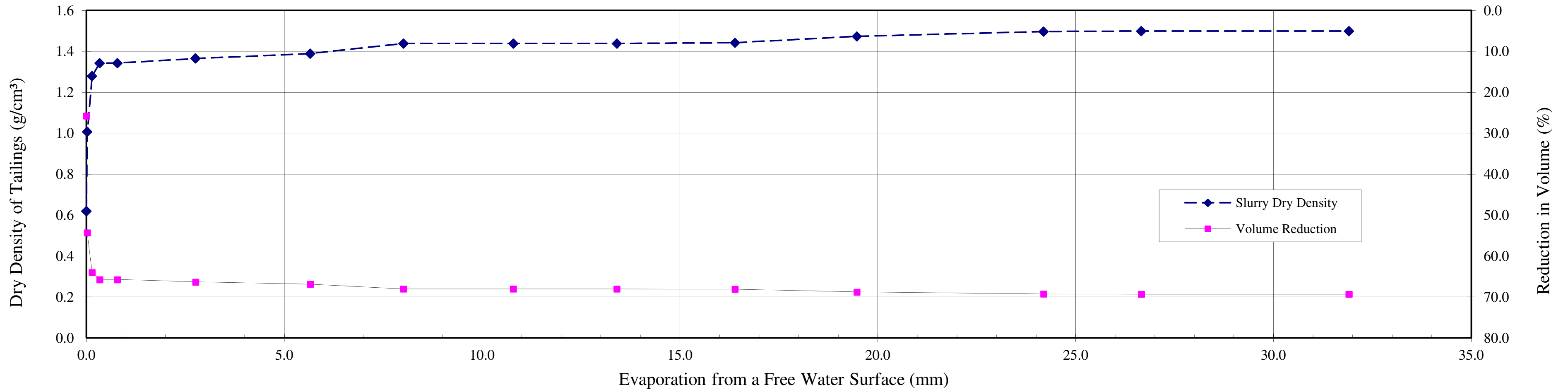
Notes:

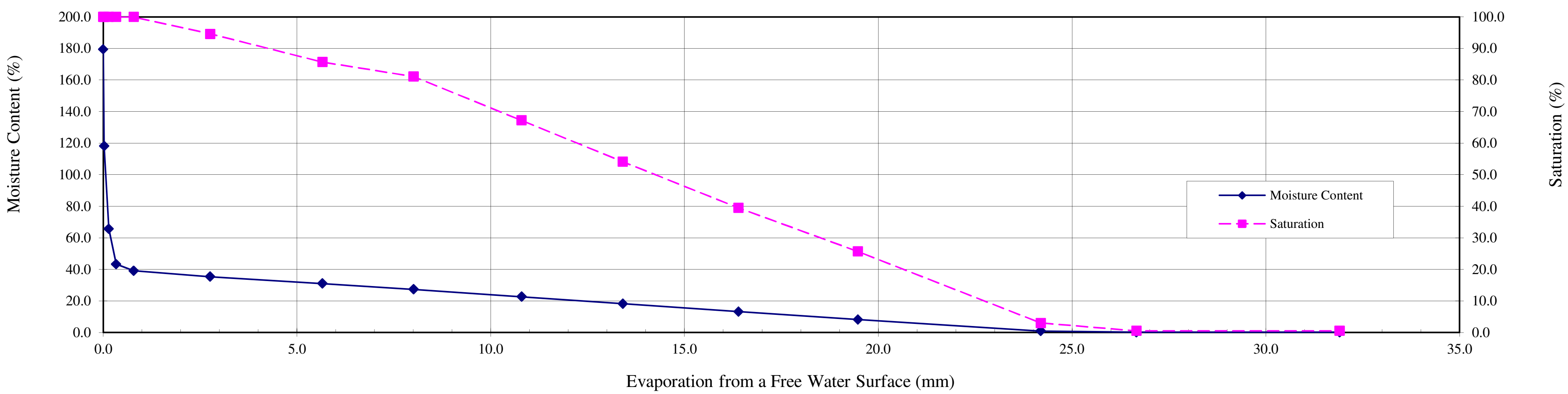
Knight Piésold CONSULTING

FIGURE 2.3

YELLOWHEAD MINING, INC.
HARPER CREEK
ROTL

VARIATION OF TAILINGS PARAMETERS WITH ONGOING EVAPORATION
0.35





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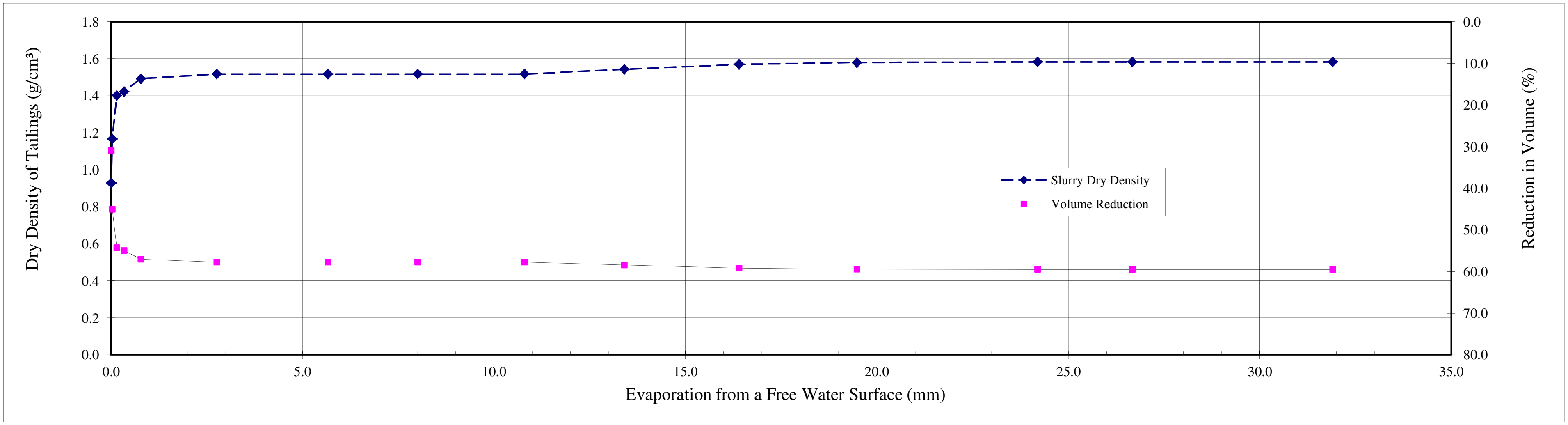
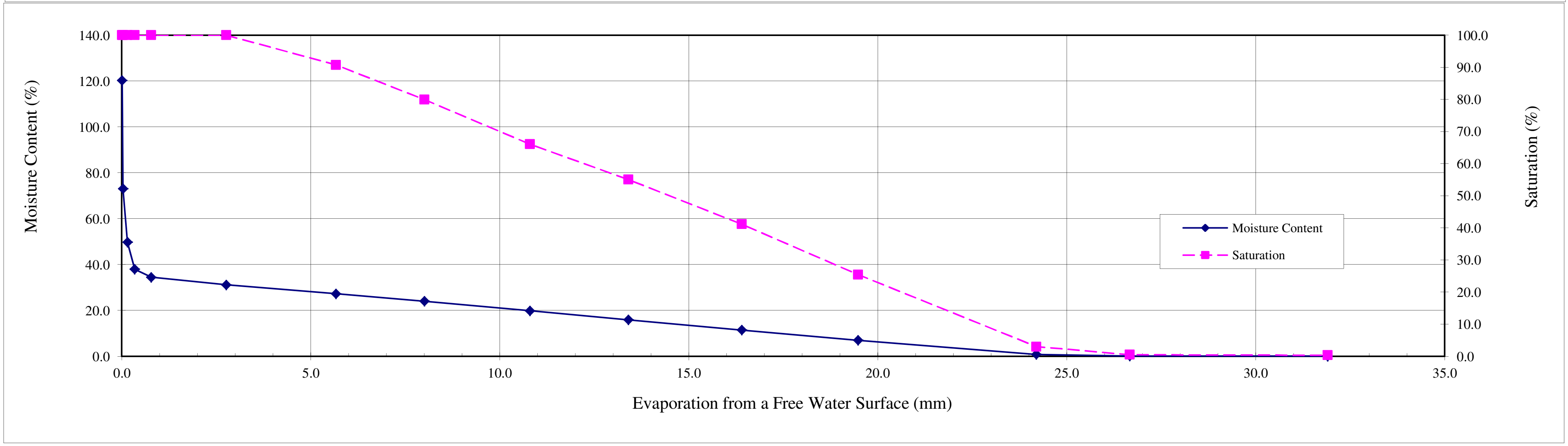
Knight Piésold CONSULTING

FIGURE 2.4

YELLOWHEAD MINING, INC.
HARPER CREEK
ROTL

VARIATION OF TAILINGS PARAMETERS WITH ONGOING EVAPORATION
0.45





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Note:

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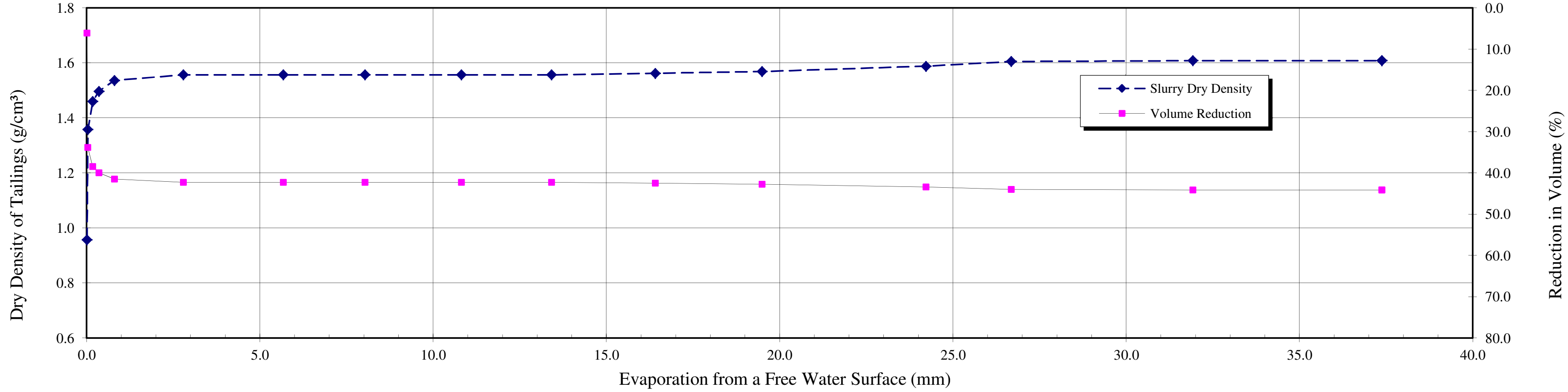
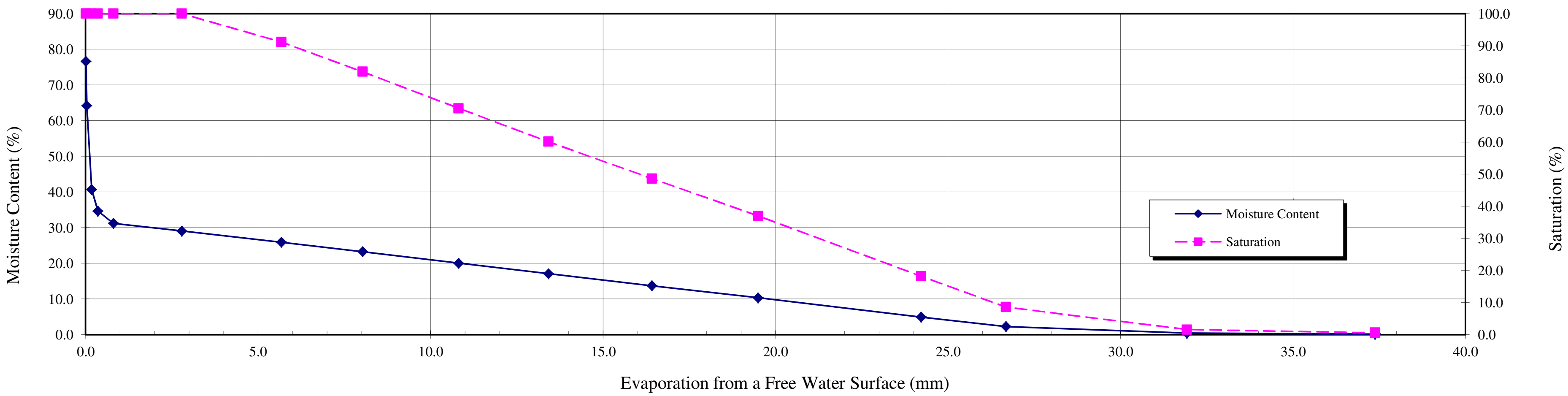
FIGURE 2.5

YELLOWHEAD MINING, INC.
HARPER CREEK
ROTL

VARIATION OF TAILINGS PARAMETERS WITH ONGOING EVAPORATION
0.55





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Note:

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TABLE 2.0

YELLOWHEAD MINING, INC.
HARPER CREEK
ROTL

SUMMARY OF TAILINGS SEDIMENTATION TEST RESULTS
TAILINGS COMPOSITE

Undrained Settling Test				Drained Settling Test					Settling and Drying Test			Additional
Solids Content (%)	Slurry Dry Density (g/cm³)	Total Water Recovery (%)	Portion of Initial Water Retained in Tailings prior to Onset of Evaporation (%)	Solids Content (%)	Slurry Dry Density (g/cm³)	Total Water Recovery (%)	Portion of Initial Water Retained in Tailings prior to Onset of Evaporation (%)	Average Permeability (cm/sec)	Solids Content (%)	Slurry Dry Density (g/cm³)	Total Evaporation (mm)	Water Recovered in Drained Test (%)
33.8	1.24	77.1	22.9	35.2	1.24	76.8	23.2	1.8E-04	35.8	1.50	31.9	-0.3
44.8	1.28	67.4	32.6	44.3	1.29	68.6	31.4	1.2E-04	45.4	1.58	31.9	1.2
56.7	1.36	50.9	49.1	56.6	1.38	53.7	46.3	8.6E-05	56.6	1.61	37.4	2.8

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Knight Piésold CONSULTING	SETTLING AND DRYING TEST (including Evaporation Control)	Project No. VA101-458.03

Project: Harper Creek, CuROTL | Sample No.: 35% | Location: CuROTL | Test Date: 9/23/2011-10/5/2011 | Tested By: jdb/jhk

Initial Parameters for Settling and Drying Test

a. Beaker (Tare) Weight =	398.03 g	d. Moisture Content (from drying test) =	**183.3** %		
b. Initial Slurry Volume =	780 cm³	e. Initial Slurry Bulk Density [(c-a)/b] =	1.29 g/cm³		
c. Tare + Initial Slurry Weight =	1406.8 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	653 g		
Time of Readings	23-Sep-11	g. Weight of Solids [(c-a)/(1+d/100)] =	356 g		
	09:49 AM	h. Tailings Solids Specific Gravity =	2.79		
		i. Solids Volume [g/h] =	127.6 cm³		

Initial Parameters for Evaporation Control

x. Beaker Tare Weight =	391 g
y. Initial Weight of Beaker =	1027 g
z. Beaker Cross-Sectional Area =	84.13 cm²

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.	I.	J.		Evaporation Control		
	Date of Reading	Time of Reading	Total Remaining Weight (g)	Total Remaining Volume (cm³)	Settled Slurry Volume (cm³)	Decanted Water Volume (if any) (cm³)	Shrinkage Crack Volume (estimated) (cm³)	Net. Slurry Volume [C-E] (cm³)	Volume Reduction [(b-F)/b] (%)	Slurry Dry Density [g/F] (g/cm³)	Moisture Content [(A-a)/g]-1 (%)	Saturation (A-a-g)/(B-i) (%)	Comments	Total Weight After Decant (g)	Decanted Weight (if any) (g)	Evap. (mm)
1	23-Sep-11	9:54 AM	1407	780.0	510.0	302.7		510.0	34.6	0.70	183.3	100.0	Water Decanted	1027	0	0
2	23-Sep-11	10:05 AM	1104	490.0	355.0	121.5		355.0	54.5	1.00	98.2	100.0	Water Decanted	1027	0	0
3	23-Sep-11	10:32 AM	982	380.0	300.0	58.9		300.0	61.5	1.19	64.0	100.0	Water Decanted	1027	0	0
4	23-Sep-11	12:58 PM	921	305.0	285.0	16.1		285.0	63.5	1.25	47.0	100.0	Water Decanted	1025	0	0
5	23-Sep-11	5:45 PM	902	300.0	275.0	5.5		275.0	64.7	1.29	41.5	100.0	Water Decanted	1021	0	1
6	24-Sep-11	11:05 AM	882	260.0	260.0	0.0		260.0	66.7	1.37	35.9	96.6	No free water	1005	0	3
7	25-Sep-11	12:45 PM	862	255.0	255.0	0.0		255.0	67.3	1.40	30.2	84.6	No free water	980	0	6
8	26-Sep-11	10:57 AM	845	255.0	255.0	0.0		255.0	67.3	1.40	25.6	71.7	No free water	961	0	8
9	27-Sep-11	10:33 AM	827	255.0	255.0	0.0		255.0	67.3	1.40	20.4	56.9	No free water	937	0	11
10	28-Sep-11	8:50 AM	809		253.6		1.5	252.2	67.7	1.41	15.3	43.8	Specimen pulling from sides	915	0	13
11	29-Sep-11	10:13 AM	788		253.6		2.4	251.2	67.8	1.42	9.6	27.7	Measured in jar	890	0	16
12	30-Sep-11	10:50 AM	768		253.6		13.0	240.6	69.1	1.48	4.0	12.6	Measured in jar	864	0	19
13	02-Oct-11	10:01 AM	755		251.9		15.3	236.7	69.7	1.50	0.1	0.4	Measured in jar	825	0	24
14	03-Oct-11	9:16 AM	755		251.9		16.2	235.7	69.8	1.51	0.1	0.4	Measured in jar	804	0	27
15	05-Oct-11	9:38 AM	755		251.9		16.2	235.7	69.8	1.51	0.1	0.4	Measured in jar	760	0	32

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Notes:

Knight Piésold CONSULTING	**SETTLING AND DRYING TEST (including Evaporation Control)**	Project No. VA101-458.03

Project: Harper Creek CuROTL | Sample No.: 45% | Location: CuROTL | Test Date: 9/23/2011-10/5/2011 | Tested By: jdb/jhk

Initial Parameters for Settling and Drying Test

a.	Beaker (Tare) Weight =	401.84 g	d. Moisture Content (from drying test) =	**115.1** %
b.	Initial Slurry Volume =	650 cm³	e. Initial Slurry Bulk Density [(c-a)/b] =	1.42 g/cm³
c.	Tare + Initial Slurry Weight =	1326.4 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	495 g
	Time of Readings	23-Sep-11	g. Weight of Solids [(c-a)/(1+d/100)] =	430 g
		09:41 AM	h. Tailings Solids Specific Gravity =	2.79
			i. Solids Volume [g/h] =	154.1 cm³

Initial Parameters for Evaporation Control

x. Beaker Tare Weight =	391 g
y. Initial Weight of Beaker =	1027 g
z. Beaker Cross-Sectional Area =	84.13 cm²

On-going Readings

	Date of Reading	Time of Reading	A. Total Remaining Weight (g)	B. Total Remaining Volume (cm³)	C. Settled Slurry Volume (cm³)	D. Decanted Water Volume (if any) (cm³)	E. Shrinkage Crack Volume (estimated) (cm³)	F. Net. Slurry Volume [C-E] (cm³)	G. Volume Reduction [(b-F)/b] (%)	H. Slurry Dry Density [g/F] (g/cm³)	I. Moisture Content [(A-a)/g]-1 (%)	J. Saturation (A-a-g)/(B-i) (%)	Comments	Evaporation Control: Total Weight After Decant (g)	Evaporation Control: Decanted Weight (if any) (g)	Evaporation Control: Evap. (mm)
1	23-Sep-11	9:50 AM	1326	650.0	445.0	212.4		445.0	31.5	0.97	115.0	100.0	Water Decanted	1027	0	0
2	23-Sep-11	10:02 AM	1114	450.0	375.0	50.9		375.0	42.3	1.15	65.6	100.0	Water Decanted	1027	0	0
3	23-Sep-11	10:29 AM	1063	400.0	340.0	45.1		340.0	47.7	1.26	53.7	100.0	Water Decanted	1027	0	0
4	23-Sep-11	12:55 PM	1016	350.0	315.0	20.0		315.0	51.5	1.36	42.8	100.0	Water Decanted	1025	0	0
5	23-Sep-11	5:43 PM	992	325.0	300.0	5.6		300.0	53.8	1.43	37.3	100.0	Water Decanted	1021	0	1
6	24-Sep-11	11:05 AM	972	300.0	300.0	0.0		300.0	53.8	1.43	32.7	96.3	No free water	1005	0	3
7	25-Sep-11	12:42 PM	952	295.0	295.0	0.0		295.0	54.6	1.46	27.9	85.0	No free water	980	0	6
8	26-Sep-11	10:54 AM	936	295.0	295.0	0.0		295.0	54.6	1.46	24.1	73.6	No free water	961	0	8
9	27-Sep-11	10:28 AM	917		292.5		1.7	290.8	55.3	1.48	19.9	62.4	Specimen pulling from sides	937	0	11
10	28-Sep-11	8:46 AM	899		290.8		4.5	286.4	55.9	1.50	15.7	51.1	measured in jar	915	0	13
11	29-Sep-11	10:10 AM	879		288.3		5.0	283.3	56.4	1.52	11.1	36.8	measured in jar	890	0	16
12	30-Sep-11	10:45 AM	859		286.6		15.8	270.8	58.3	1.59	6.4	23.5	specimen measured	864	0	19
13	02-Oct-11	9:56 AM	835		286.6		15.8	270.8	58.3	1.59	0.8	2.9	specimen measured	825	0	24
14	03-Oct-11	9:12 AM	833		286.6		15.8	270.8	58.3	1.59	0.3	1.0	specimen measured	804	0	27
15	05-Oct-11	9:37 AM	832		286.6		15.8	270.8	58.3	1.59	0.1	0.5	specimen measured	760	0	32

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Notes:

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SETTLING AND DRYING TEST (including Evaporation Control)

Project No. VA101-458.03

Project:	Harper Creek CuROTL	Test Date:	9/23/2011-10/7/2011
Sample No.:	55%	Tested By:	jdb/jhk
Location:	CuROTL		

Initial Parameters for Settling and Drying Test

a. Beaker (Tare) Weight =	395.19 g	d. Moisture Content (from drying test) =	**77.2**	%
b. Initial Slurry Volume =	590 cm³	e. Initial Slurry Bulk Density [(c-a)/b] =	1.58	g/cm³
c. Tare + Initial Slurry Weight =	1325.1 g	f. Weight of Water [(c-a)/(1+1/(d/100))] =	405	g
Time of Readings	23-Sep-11	g. Weight of Solids [(c-a)/(1+d/100)] =	525	g
	09:27 AM	h. Tailings Solids Specific Gravity =	2.79	
		i. Solids Volume [g/h] =	188.1	cm³

Initial Parameters for Evaporation Control

x. Beaker Tare Weight =	391 g
y. Initial Weight of Beaker =	1027 g
z. Beaker Cross-Sectional Area =	84.13 cm²

On-going Readings

			A.	B.	C.	D.	E.	F.	G.	H.	I.	J.		Evaporation Control		
	Date of Reading	Time of Reading	Total Remaining Weight (g)	Total Remaining Volume (cm³)	Settled Slurry Volume (cm³)	Decanted Water Volume (if any) (cm³)	Shrinkage Crack Volume (estimated) (cm³)	Net. Slurry Volume [C-E] (cm³)	Volume Reduction [(b-F)/b] (%)	Slurry Dry Density [g/F] (g/cm³)	Moisture Content [(A-a)/g]-1 (%)	Saturation (A-a-g)/(B-i) (%)	Comments	Total Weight After Decant (g)	Decanted Weight (if any) (g)	Evap. (mm)
1	23-Sep-11	9:33 AM	1325	590.0	500.0	62.3		500.0	15.3	1.05	77.2	100.0	Water Decanted	1027	0	0
2	23-Sep-11	9:59 AM	1262	540.0	390.0	120.6		390.0	33.9	1.35	65.3	100.0	Water Decanted	1027	0	0
3	23-Sep-11	10:26 AM	1141	410.0	360.0	27.1		360.0	39.0	1.46	42.2	100.0	Water Decanted	1027	0	0
4	23-Sep-11	12:52 PM	1113	390.0	350.0	12.8		350.0	40.7	1.50	36.7	100.0	Water Decanted	1025	0	0
5	23-Sep-11	5:40 PM	1096	365.0	350.0	2.7		350.0	40.7	1.50	33.6	100.0	Water Decanted	1021	0	1
6	24-Sep-11	11:00 AM	1080	348.0	348.0	0.0		348.0	41.0	1.51	30.5	99.9	No free water observed	1005	0	3
7	25-Sep-11	12:38 PM	1061		361.8		13.8	347.9	41.0	1.51	27.0	88.5	Specimen pulling from sides	980	0	6
8	26-Sep-11	10:47 AM	1046		361.8		15.2	346.6	41.3	1.51	24.1	79.8	measured in jar	961	0	8
9	27-Sep-11	10:24 AM	1028		361.8		15.9	345.9	41.4	1.52	20.5	68.2	measured in jar	937	0	11
10	28-Sep-11	8:41 AM	1010		361.8		19.3	342.5	42.0	1.53	17.1	58.1	measured in jar	915	0	13
11	29-Sep-11	10:06 AM	989		361.8		21.3	340.4	42.3	1.54	13.2	45.4	measured in jar	890	0	16
12	30-Sep-11	10:38 AM	969		361.8		24.1	337.7	42.8	1.55	9.3	32.7	Specimen measured	864	0	19
13	02-Oct-11	9:51 AM	937		359.3		25.9	333.4	43.5	1.57	3.2	11.5	Specimen measured	825	0	24
14	03-Oct-11	9:07 AM	927		355.0		26.3	328.8	44.3	1.60	1.3	5.0	Specimen measured	804	0	27
15	05-Oct-11	9:35 AM	921		353.4		26.8	326.6	44.6	1.61	0.2	0.8	Specimen measured	760	0	32
16	07-Oct-11	10:42 AM	921		353.4		26.8	326.6	44.6	1.61	0.1	0.4	Specimen measured	714	0	37

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Notes:

FIGURE 2.3

YELLOWHEAD MINING, INC.
HARPER CREEK
CuROTL

VARIATION OF TAILINGS PARAMETERS WITH ONGOING EVAPORATION
0.35

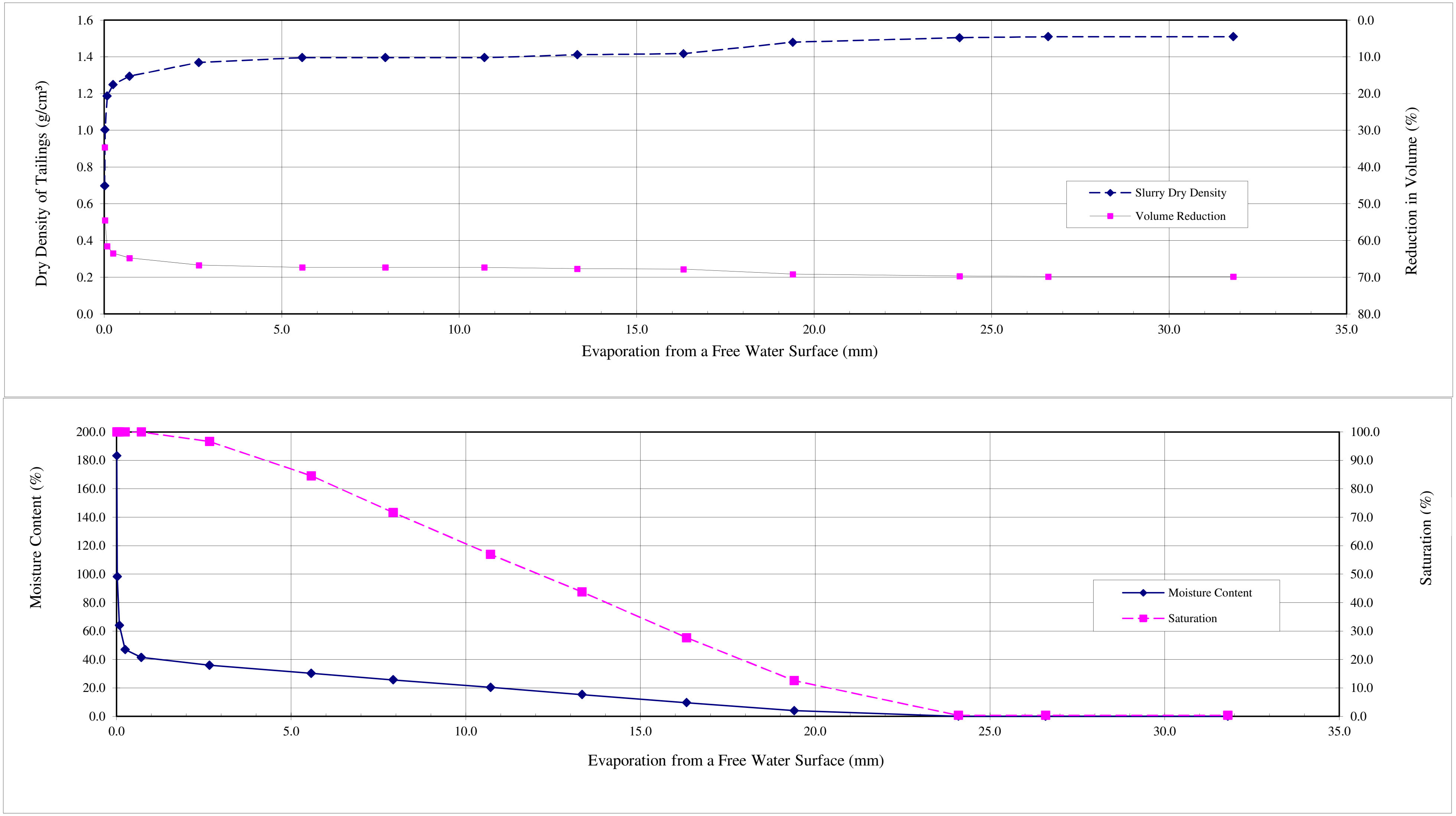

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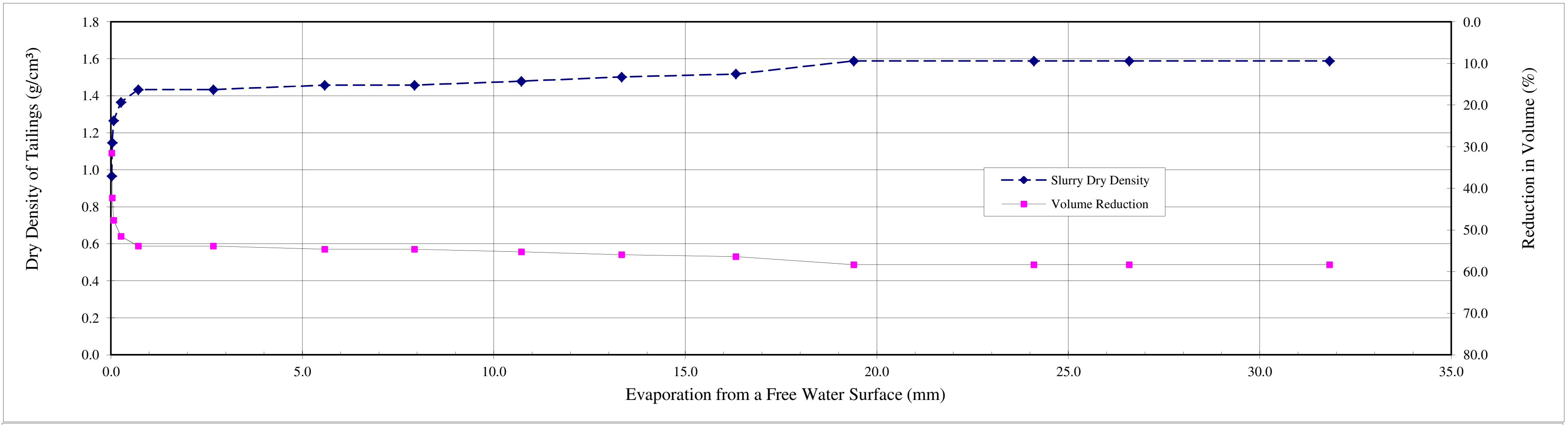
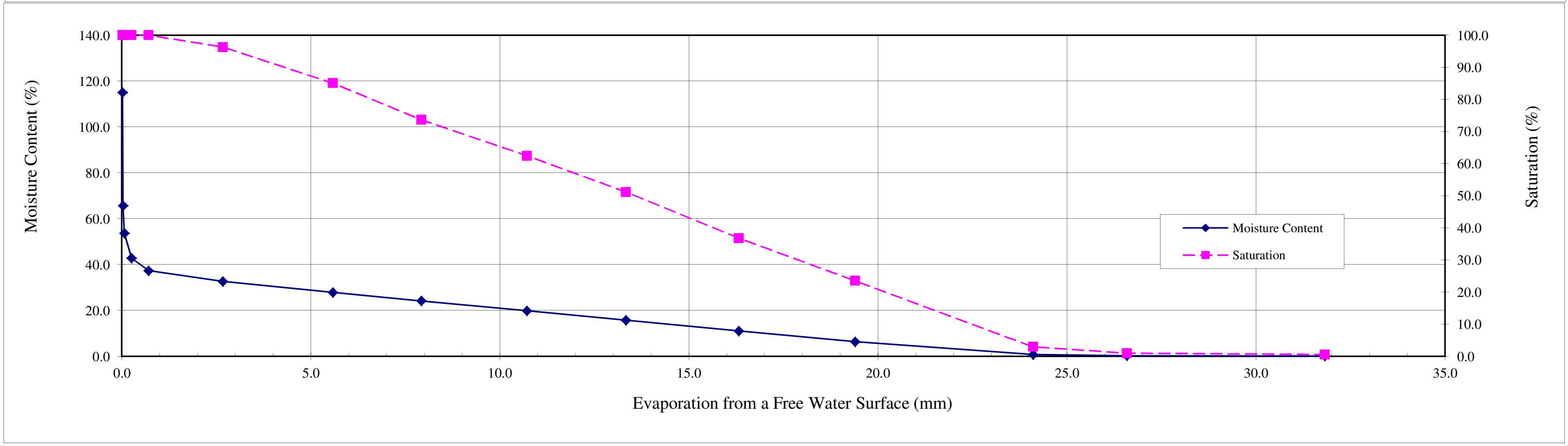
FIGURE 2.4

YELLOWHEAD MINING, INC.
HARPER CREEK
CuROTL

VARIATION OF TAILINGS PARAMETERS WITH ONGOING EVAPORATION
0.45





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Note:

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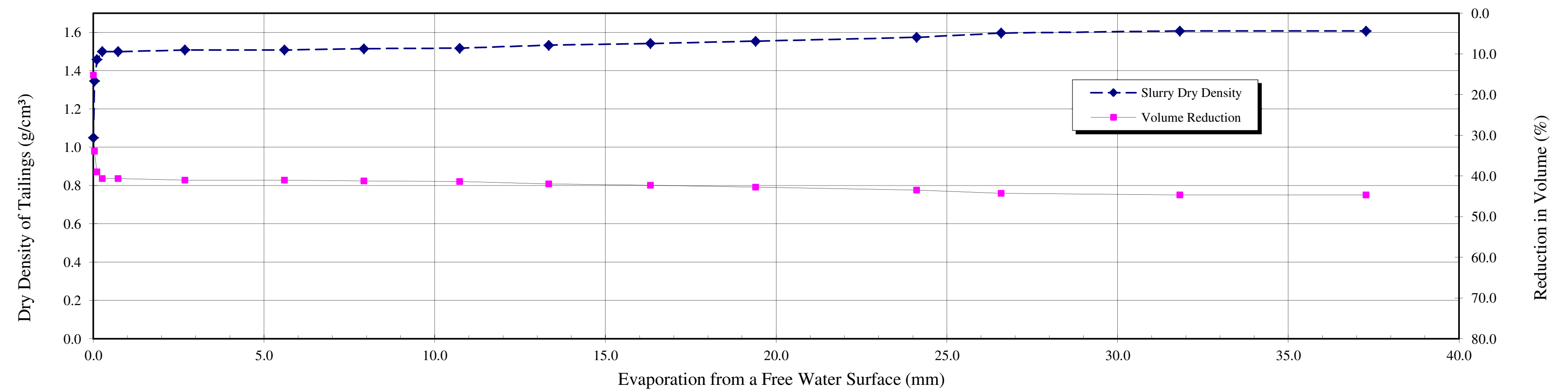
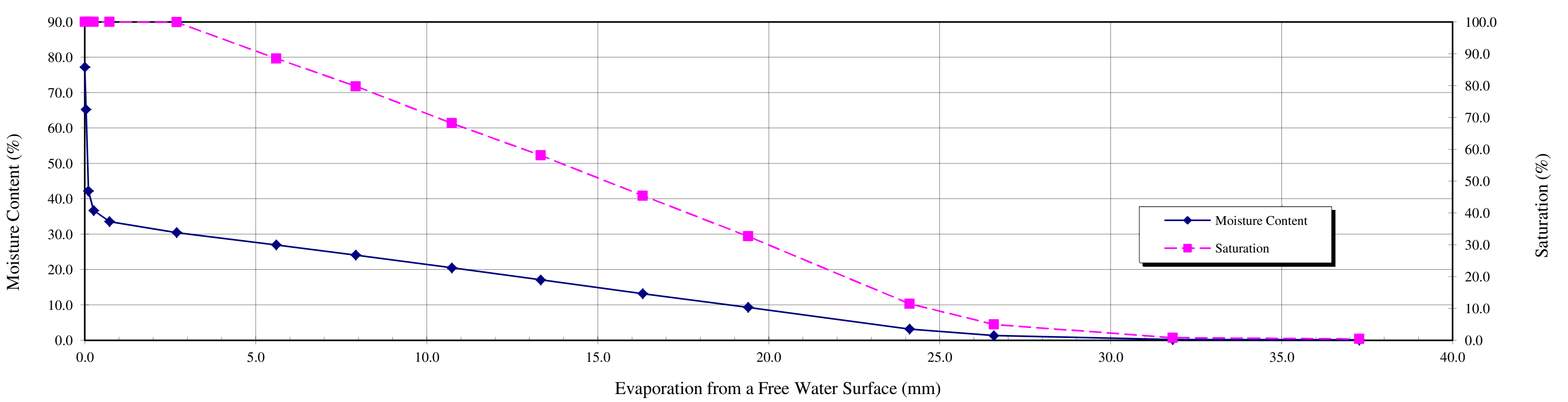
FIGURE 2.5

YELLOWHEAD MINING, INC.
HARPER CREEK
CuROTL

VARIATION OF TAILINGS PARAMETERS WITH ONGOING EVAPORATION
0.55





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Note:

Knight Piésold CONSULTING

TABLE 2.0

YELLOWHEAD MINING, INC.
HARPER CREEK
CuROTL

SUMMARY OF TAILINGS SEDIMENTATION TEST RESULTS
TAILINGS COMPOSITE

Undrained Settling Test				Drained Settling Test					Settling and Drying Test			Additional
Solids Content (%)	Slurry Dry Density (g/cm³)	Total Water Recovery (%)	Portion of Initial Water Retained in Tailings prior to Onset of Evaporation (%)	Solids Content (%)	Slurry Dry Density (g/cm³)	Total Water Recovery (%)	Portion of Initial Water Retained in Tailings prior to Onset of Evaporation (%)	Average Permeability (cm/sec)	Solids Content (%)	Slurry Dry Density (g/cm³)	Total Evaporation (mm)	Water Recovered in Drained Test (%)
35.0	1.18	73.8	26.2	35.7	1.24	75.0	25.0	1.4E-04	35.3	1.51	31.8	1.2
46.3	1.25	62.4	37.6	46.3	1.30	65.6	34.4	1.7E-04	46.5	1.59	31.8	3.3
56.8	1.26	43.4	56.6	56.3	1.38	53.4	46.6	7.8E-05	56.4	1.61	37.3	10.0

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APPENDIX C

SEISMICITY ASSESSMENT

(Pages C-1 to C-8)

Knight Piésold
CONSULTING

MEMORANDUM

To:	Mr. Ken Brouwer	Date:	March 8, 2012
Copy To:		File No.:	VA101-458/4-A.01
From:	Graham Greenaway	Cont. No.:	VA12-00565
Re:	Harper Creek Project – Seismicity Assessment		

A seismicity assessment has been carried out for the Harper Creek Project, including a review of the regional seismicity and a seismic hazard analysis. The results of the seismic hazard analysis are required to provide seismic design parameters for the design of the Tailings Management Facility and for other geotechnical structures at the project site.

This memo presents the findings of the seismicity review and the methodology and results of the seismic hazard analysis. Design ground motion parameters provided by the seismic hazard analysis include peak ground acceleration, spectral acceleration (defining the uniform hazard spectrum) and design earthquake magnitude.

1.0 REGIONAL TECTONICS AND SEISMICITY

The Harper Creek project is situated within south-eastern B.C., where the level of historical seismic activity has been low. Figure 1 shows the regional tectonics and historical seismicity of southern B.C. and the location of the Harper Creek project.

The level of seismicity in the interior of B.C. and the Rocky Mountains region drops off rapidly with distance from the west coast and to the north. The largest earthquake recorded in the southern Cordillera region was an event of about Magnitude 6.0 in 1918, located in the Valemount area of the Rocky Mountain trench. More recently, a Magnitude 5.4 earthquake occurred near Prince George in 1986 causing minor damage, and a Magnitude 5.3 earthquake occurred in 2001 east of Dawson Creek. The maximum earthquake magnitude for the region of south-eastern B.C. is estimated to be about Magnitude 7.0, with an upper bound estimate of Magnitude 7.3, based on historical earthquake data and the regional tectonics (Adams and Halchuk, 2003).

The seismic hazard along the west coast of B.C. is significant due to subduction zone earthquakes along offshore faults and within the subducting oceanic tectonic plate. There is potential for very large earthquakes of Magnitude 8.0 to 9.0+ along this Cascadia subduction zone. Geological evidence indicates that these great subduction earthquakes occur on average approximately every 500 years, but this interval varies from about 300 to 800 years. The last great Cascadia earthquake occurred over 300 years ago, in 1700. However, such an event would be located over 450 km southwest of the project site, and therefore the amplitude of ground motions experienced at the site would be very low due to attenuation over such a large distance. Peak ground accelerations on rock at the project site from a great subduction earthquake would likely be less than 0.05g. There is also potential for intraslab (inslab) earthquakes, occurring deep within the subducted Juan de Fuca plate that extends eastwards beneath the North American plate. These events, which have potential to be as large as about Magnitude 7.5, would likely occur over 300 km to the southwest, at a depth of over 40 km. Ground motions on rock experienced at the project site for this type of subduction earthquake are likely to be less than 0.1g. The seismic hazard at the Harper Creek project is predominantly from potential shallow crustal earthquakes occurring closer to the site.

Suite 1400 – 750 W. Pender Street, Vancouver, BC V6C 2T8
Phone: 604-685-0543 Facsimile: 604-685-0147

2.0 SEISMIC HAZARD ANALYSIS

The seismic hazard for the Harper Creek project has been defined using probabilistic methods of analysis. This method requires an examination of historical earthquake data and the regional tectonics to identify potential seismic sources and to determine the maximum earthquake magnitude for each seismic source. Appropriate relationships defining the attenuation of earthquake ground motion with distance are also required.

Design ground motion parameters have been determined for the Harper Creek project site using information provided by the probabilistic seismic hazard database of Natural Resources Canada (NRC) (http://earthquakescanada.nrcan.gc.ca/hazard-alea/zoning/haz-eng.php). The results are summarized in Table 1 in terms of earthquake return period, probability of exceedance (for a 23 year design operating life) and the corresponding peak ground acceleration (median and mean hazard values). However, the NRC database only provides ground motion parameters up to a return period of approximately 2500 years (corresponding to a 2% probability of exceedance in 50 years). Higher return periods will need to be considered for dam design if the classification is defined as Very High or Extreme (based on the requirements of the Canadian Dam Safety Guidelines). Therefore, a probabilistic seismic hazard analysis has been conducted to provide ground motion parameters beyond 2500 years, specifically for return periods of 5000 and 10,000 years.

The methodology used to complete the site-specific probabilistic seismic hazard analysis and the results of the analysis are described in the following sections.

2.1 Ground Motion Attenuation

Appropriate attenuation models defining the relationship between earthquake magnitude, source to site distance and peak ground motion (acceleration) are required to carry out a probabilistic seismic hazard analysis. The ground motions experienced at the project site are dependent on the regional ground motion attenuation characteristics and the earthquake source mechanism.

For shallow crustal earthquakes a set of four ground motion attenuation models, known as the New Generation Attenuation (NGA) relations was used (Earthquake Spectra, 2008). These include the ground motion relationships of Abrahamson and Silva, Boore and Atkinson, Campbell and Bozorgnia and Chiou and Youngs. These ground motion attenuation relationships are applicable to shallow crustal earthquakes in western North American and similar tectonic regions of the world. The predicted peak ground accelerations for shallow crustal earthquakes are average values calculated using the four attenuation relationships (equal weighting).

The peak ground accelerations and spectral accelerations predicted using the attenuation relationships are for soft rock/very dense soil site conditions, assuming an average shear wave velocity in the upper 30 meters (defined as the Vs30 value) of 560 m/sec (range of 360 to 760 m/sec). This corresponds to Site Class C, as defined by the National Building Code of Canada (NBCC 2010).

Attenuation relationships provided by Youngs (1997) were used for interface subduction and intraslab subduction earthquake source zones. These relationships were developed specifically for oceanic subduction zone earthquakes. These relationships are also used in the seismic hazard model developed for the NRC probabilistic seismic hazard database.

2.2 Probabilistic Analysis

A probabilistic seismic hazard analysis is carried out to define a unique probability of occurrence for each possible level of ground acceleration experienced at a site. The methodology used for the probabilistic analysis is based on that presented by Cornell (1968). The likelihood of occurrence of earthquakes within defined seismic source zones is determined by examining seismicity data. Using historical earthquake records for the region, magnitude-frequency recurrence relationships are established for potential earthquake source zones. The magnitude recurrence relationships are of the form derived by Gutenberg-Richter (1944):

$$\log(N) = a - b(M)$$

where, M = Earthquake magnitude
N =Annual frequency of occurrence for earthquakes exceeding magnitude M
(1/N = Return Period)

The computer program EZ-FRISK (Risk Engineering, Inc., 2008) was used to develop a seismic hazard model for southern British Columbia and the surrounding regions. The seismic hazard analysis module available with EZ-FRISK includes a database provided by Risk Engineering Inc. of faults and areal seismic sources for the pertinent regions of western Canada. Seismic sources defined in the hazard model include South-eastern B.C., Puget Sound and the Cascadia Subduction Zone. The project site is located within an areal seismic source that defines the seismicity in south-eastern B.C. A maximum earthquake of Magnitude 7.3 was assigned to this source zone, which is characterized by shallow crustal earthquakes.

Magnitude-frequency recurrence relationships and the corresponding maximum earthquake magnitude for each seismic source are prepared by Risk Engineering from consideration of historical seismicity, fault characteristics and the regional tectonics, using information obtained from the Geological Survey of Canada and proprietary studies. For calculation of peak ground accelerations a minimum magnitude of 5.0 was used in the analysis for all seismic source zones. Earthquakes of lower magnitude are not considered to be a risk to engineered facilities. Appropriate ground motion attenuation relationships were assigned to each seismic source, as discussed in Section 2.1 above.

The seismic hazard model developed using EZ-FRISK was used to determine the relationship between peak ground acceleration and annual frequency of occurrence for the project site. Median hazard values of peak ground acceleration have been determined for return periods up to 10,000 years. Predicted values for the project site are included in Table 1 for return periods of 5000 and 10,000 years. Predicted values for lower return periods were very similar to those provided by the NRC seismic hazard database.

The probabilistic seismic hazard analysis has also been used to calculate spectral acceleration values (5% damping). These have been used to develop site-specific uniform hazard spectra corresponding to return periods of 5000 and 10,000 years. The uniform hazard spectra are shown on Figure 2. Tabulated values of these uniform hazard spectra are provided in Table 2.

Deaggregation of the probabilistic seismic hazard results has been carried out to provide the relative contributions of all potential seismic sources, and to more accurately define the characteristics of potential earthquakes contributing to the seismic hazard. The findings indicate that the seismic hazard for the project site is predominantly from shallow crustal earthquakes in the region of south-eastern B.C.

Conservative design earthquake magnitudes of 7.0 and 7.3 have been selected for earthquake return periods of 5000 years and 10,000 years respectively, based on the review of regional tectonics and historical seismicity, and the findings of deaggregation off the probabilistic seismic hazard.

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CONSULTING

3.0 REFERENCES

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Gutenberg, B. and Richter, C.F., (1944), "Frequency of Earthquakes in California", Bulletin of the Seismological Society of America, Vol. 34, p.185-188.

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Signed: G.R. Greenaway

Graham Greenaway, P.Eng. – Specialist Engineer/Project Manager

Approved:

Ken Brouwer, P.Eng. – Managing Director

Attachments:

Table 1 Rev 0	Summary of Probabilistic Seismic Hazard Analysis
Table 2 Rev 0	Uniform Hazard Spectra for 1/5000 and 1/10,000 Year Earthquakes
Figure 1 Rev 1	Regional Tectonics and Historical Seismicity
Figure 2 Rev 0	Uniform Hazard Spectra for 1/5000 and 1/10,000 Year Earthquakes

/grg

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TABLE 1

YELLOWHEAD MINING INC.
HARPER CREEK PROJECT

FEASIBILITY DESIGN STUDIES
SUMMARY OF PROBABILISTIC SEISMIC HAZARD ANALYSIS

Print 08/Mar/12 9:42:59

Return Period (Years)	Probability of Exceedance[1] (%)	Peak Ground Acceleration (PGA)[2]	
		Median PGA[3,4] (g)	Estimate Mean PGA[5] (g)
100	21	0.03	0.04
500	4	0.07	0.08
1,000	2	0.10	0.11
2,500	1	0.14	0.16
5,000	0.5	0.16	0.19
10,000	0.2	0.23	0.26

M:\1\01\00458\04\A\Data\0200 - Tailings Management Facility\Seismicity\[Harper Creek - Seismic Hazard.xlsx]PSHA Table 1
Rev 0

NOTES:

1. PROBABILITY OF EXCEEDANCE CALCULATED FOR A DESIGN LIFE OF 23 YEARS.

$$q = 1^{-(-L/T)}$$

WHERE: q = PROBABILITY OF EXCEEDANCE
L = DESIGN LIFE IN YEARS
T = RETURN PERIOD IN YEARS

2. PEAK GROUND ACCELERATIONS ARE FOR SOFT ROCK/VERY DENSE SOIL (Vs30 = 560 M/SEC)
3. MEDIAN PEAK GROUND ACCELERATIONS FOR RETURN PERIOD UPTO 2,500 YEARS OBTAINED FROM THE SEISM
4. MEDIAN PEAK GROUND ACCELERATIONS FOR RETURN PERIODS OF 5,000 AND 10,000 YEARS OBTAINED FROM S
5. MEAN PGA VALUES ESTIMATED AS 1.15 X MEDIAN VALUES.

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	7MAR'12	ISSUED WITH MEMO VA12-00565	GRG	DDF	KJB

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CONSULTING

TABLE 2

YELLOWHEAD MINING INC.
HARPER CREEK PROJECT

UNIFORM HAZARD SPECTRA FOR 1/5000 AND 1/10,000 YEAR EARTHQUAKES

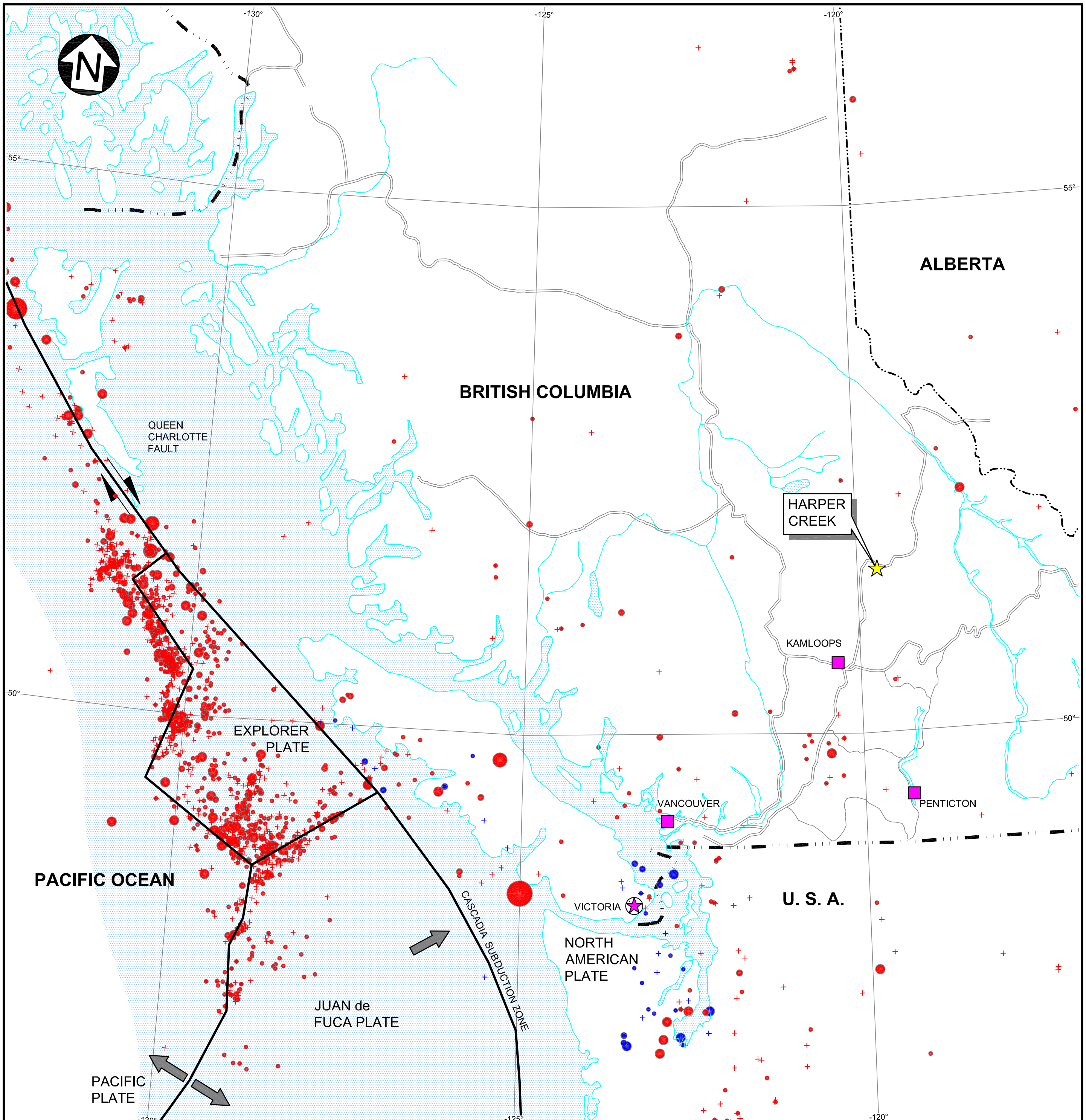
Spectral Period (seconds)	Spectral Accelerations for 1/5,000 Year Earthquake		Spectral Accelerations for 1/10,000 Year Earthquake	
	Median (g)	Estimated Mean (g)	Median (g)	Estimated Mean (g)
PGA	0.16	0.19	0.23	0.26
0.02	0.17	0.19	0.23	0.26
0.03	0.18	0.21	0.25	0.29
0.05	0.22	0.25	0.30	0.35
0.1	0.33	0.38	0.46	0.53
0.2	0.40	0.46	0.55	0.63
0.3	0.35	0.40	0.47	0.55
0.4	0.29	0.34	0.40	0.45
0.5	0.25	0.29	0.33	0.38
0.75	0.22	0.26	0.28	0.32
1.0	0.17	0.20	0.22	0.25
2.0	0.09	0.11	0.12	0.14
3.0	0.05	0.06	0.06	0.07
4.0	0.03	0.04	0.04	0.05

M:\1\01\00458\04\A\Data\0200 - Tailings Management Facility\Seismicity\[Harper Creek - Seismic Hazard.xlsx]UHS Table 2 Rev 0

NOTES:

1. SPECTRAL ACCELERATIONS ARE MEAN HAZARD VALUES FOR SOFT ROCK/ VERY DENSE SOIL SITE CONDITIONS (Vs30 = 560 m/sec).
2. ESTIMATED MEAN VALUES ESTIMATED AS 1.15 X MEDIAN VALUES.

0	7MAR'12	ISSUED WITH MEMO VA12-00565	GRG	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D



NOTES:

1. GLOBAL COORDINATE SYSTEM IS UTM WITH NAD83 DATUM, ZONE 10 m, CENTRAL MERIDIAN.
2. HISTORICAL EARTHQUAKE DATA FROM NRCAN, USGS, AND NGDC DATABASES.
3. INCLUDES RECORDED EARTHQUAKES FROM 1600 TO 1973 (MAGNITUDE > 6.0) AND FROM 1973 TO 2009 (MAGNITUDE > 3.5).

SCALE A: 50 0 100 200 300 km

SYMBOL SIZE PROPORTIONAL TO MAGNITUDE

- 3.5 - 3.9
- 4.0 - 4.9
- 5.0 - 5.9
- 6.0 - 6.9
- 7.0 - 7.9
- 8.0 - 8.9
- 9.0 +

EARTHQUAKE FOCAL DEPTH

- 35 or LESS km
- > 35 km
- DIRECTION OF TECTONIC PLATE MOVEMENT

YELLOWHEAD MINING INC.

HARPER CREEK PROJECT

REGIONAL TECTONICS AND HISTORICAL SEISMICITY

Knight Piésold CONSULTING

P/A NO.	REF NO.
VA101-458/4	VA12-00565
FIGURE 1	REV 0

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	07MAR'12	ISSUED WITH MEMO	GRG	NSD	DDF	KJB

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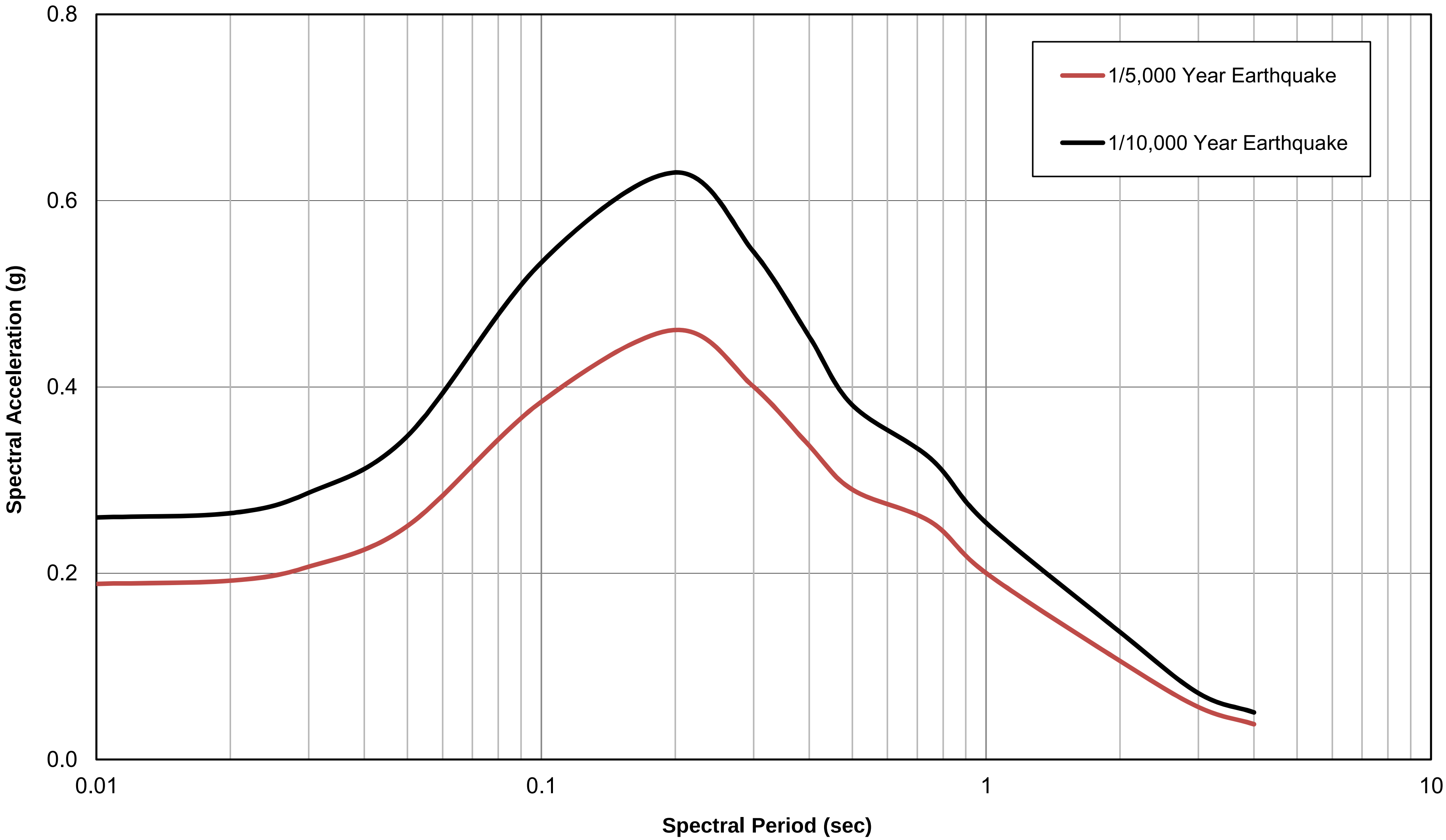
XREF FILE(S): IMAGE FILE(S): BC_map_All Platemap



NOTES:
1. SPECTRAL ACCELERATIONS ARE MEAN HAZARD VALUES FOR SOFT ROCK/ VERY DENSE SOIL SITE CONDITIONS (Vs30 = 560 m/sec).

0	7MAR'12	ISSUED WITH MEMO	GRG	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

YELLOWHEAD MINING INC..

HARPER CREEK PROJECT

UNIFORM HAZARD SPECTRA
FOR 1/5,000 AND 1/10,000 YEAR EARTHQUAKES

Knight Piésold CONSULTING

P/A. NO. VA101-458/4

REF NO. VA12-00565

FIGURE 2

REV 0

APPENDIX D

SEEPAGE AND STABILITY MODELLING

(Pages D-1 to D-42)

August 19, 2014

File No.:VA101-458/14-A.01
Cont. No.:VA14-00865

Mr. Alastair Tiver
Vice President Operations
Harper Creek Mining Corp
730 - 800 West Pender Street
Vancouver, BC V6C 2V6

Dear Alastair,

Re: Harper Creek Project - Seepage and Stability Modelling

1 – INTRODUCTION

Harper Creek Mining Corporation (HCMC) proposes to construct and operate the Harper Creek Project (the Project), an open pit copper mine near Vavenby, British Columbia (BC). HCMC is a wholly owned subsidiary of Yellowhead Mining Inc. (YMI), which is a public BC junior mineral development company trading on the Toronto Stock Exchange. The Project has an estimated 28-year mine life based on a process plant throughput of 70,000 tonnes per day (25 million tonnes per year). Ore will be processed on site through a conventional crushing, grinding and flotation process to produce a copper concentrate, with gold and silver by-products, which will be trucked from the Project site along approximately 24km of existing access roads to a rail load-out facility located at Vavenby. The concentrate will be transported via the existing Canadian National Railway network to the existing Vancouver Wharves storage, handling and loading facilities located at the Port of Vancouver for shipment to overseas smelters.

The Project consists of an open pit mine, on-site processing facility, tailings management facility (TMF) (for tailings solids, subaqueous storage of Potentially Acid Generating (PAG) waste rock, and recycling of water for processing), waste rock stockpiles, low grade and overburden stockpiles, a temporary construction camp, ancillary facilities, mine haul roads, sewage and waste management facilities, a 24km access road between the Project site and a rail load-out facility located on private land owned by HCMC in Vavenby, and a 12km power line connecting the Project site to the BC Hydro transmission line corridor in Vavenby.

2 – SCOPE OF REPORT AND KEY REFERENCE DOCUMENTS

In 2012, YMI commissioned Merit Consultants International Inc., Knight Piésold Ltd. (KP), Nilsson Mine Services Ltd., All North Consultants, and other specialist consultants to undertake a Feasibility Study (FS) for the Project. The Technical Report for the FS was filed on SEDAR on March 29, 2012 (Merit, 2012). The FS included technical modelling of seepage potential and stability analyses for the tailings management facility (TMF).

In 2014, KP was retained by HCMC to complete engineering studies and to update the design of the mine waste and water management facilities to contribute to an updated FS for the project. KP revised the technical modeling for the project, including updates to the 2 Dimensional (2D) stability and seepage analyses for the following:

- Tailings Management Facility (TMF)
- Non-PAG Waste Rock Stockpile

This letter presents the results of the revised 2D seepage and stability modeling for the project, and supersedes the findings discussed in the previous study (Knight Piésold, 2012a). This letter discusses the technical modelling approach and findings, and should be read in conjunction with other comprehensive reports that have been developed for the project. The following KP reports are essential to developing a complete understanding of the project mine waste and water management design and predicted project effects:

- **Mine Waste and Water Management Design** – KP report *Mine Waste and Water Management Design Report*, Ref. No. VA101-458/11-1. (Knight Piésold, 2014a)
- **Watershed Modelling** – KP report *Watershed Modelling*, Ref. No. VA101-458/14-1. (Knight Piésold, 2014b)
- **Numerical Groundwater Modelling** – KP report *Numerical Groundwater Modelling*, Ref. No. VA101-458/14-2. (Knight Piésold, 2014c)
- **Water Quality Predictions** – KP report *Water Quality Predictions*, Ref. No. VA101-458/14-3. (Knight Piésold, 2014d)

3 – TAILINGS MANAGEMENT FACILITY SEEPAGE ANALYSES

3.1 MODELLING APPROACH

Steady state seepage analyses were carried out for the main and north embankments to provide preliminary estimates of the seepage through the embankments and foundation materials for the final embankment configuration.

In order to determine the potential for seepage flow along the northwestern and southeastern flanks of the TMF, seepage analyses were completed at two sections of low topography (denoted east Saddle and west Saddle).

The analysed sections for the TMF are identified on Figure 1 and are described as follows:

- Main Embankment: Sections 1, 2 & 3
- North Embankment: Section 6
- East Saddle: Section 4
- West Saddle: Section 5

The seepage analyses were conducted using the 2D finite element computer program SEEP/W (Geostudio, 2007). Sensitivity analyses were also carried out to assess the range of the predicted seepage rates to variation in the saturated hydraulic conductivity of the foundation and embankment materials and variation in the model boundary conditions.

The seepage rate through foundation materials and embankment fill zones will be influenced by the following factors:

- Permeability of the natural glacial till materials that blanket the basin
- Permeability of the Orthogneiss bedrock foundation
- Thickness and permeability of the tailings stored within the TMF
- Permeability of the embankment core zones
- Seepage gradients in the embankment and foundation zones, and
- Seepage area (increases during operations).

The seepage flow rate is expected to vary over the life of the TMF as it is gradually filled with tailings, PAG waste rock materials and supernatant water. The tailings deposit will increase in thickness during operations and the tailings mass will also decrease in permeability due to on-going self-weight consolidation.

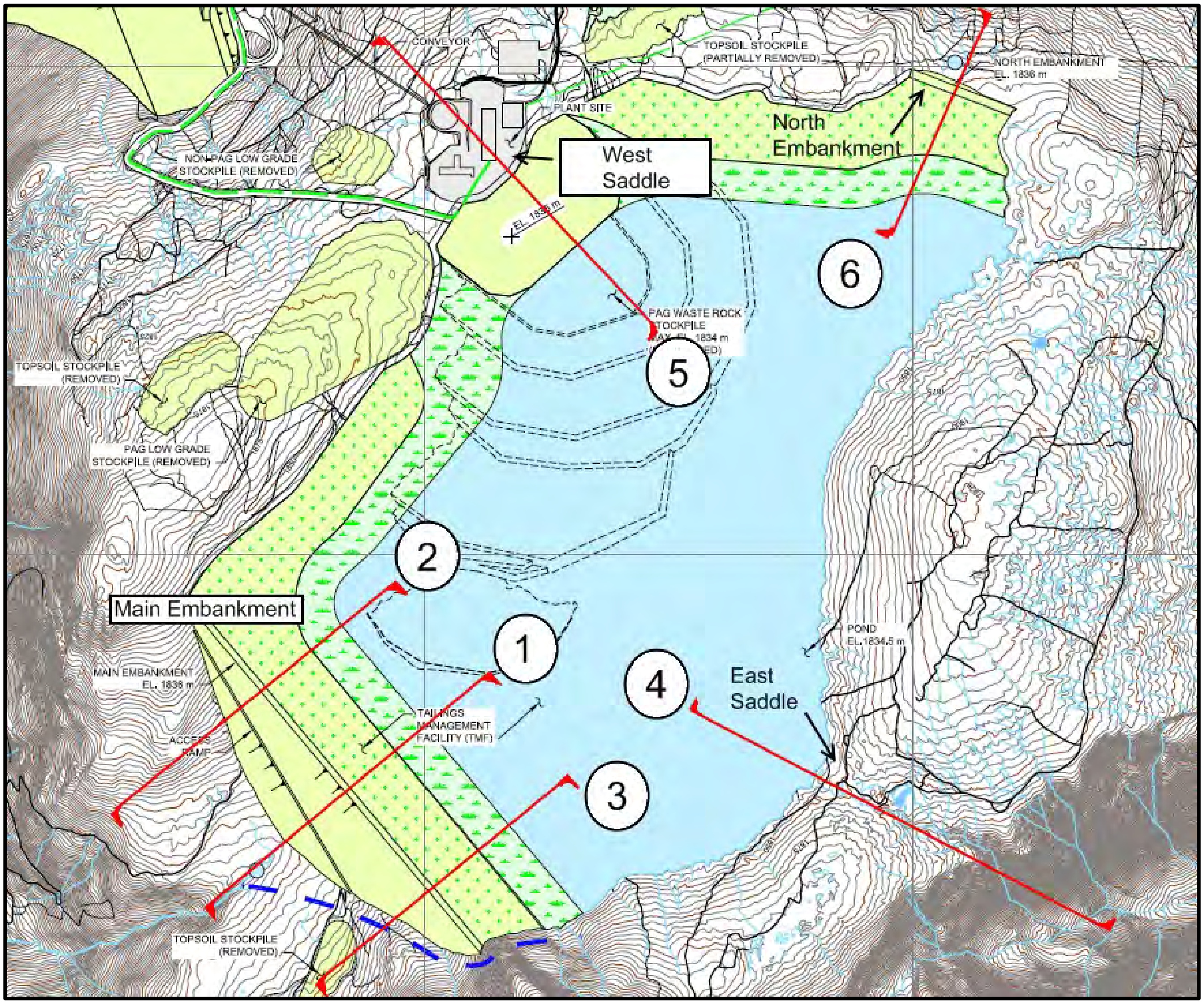


Figure 1 General Arrangement of TMF at Closure with 2D Analysis Sections Identified

3.2 SUMMARY OF MATERIAL PARAMETERS

The following sections provide a description of materials that have been included in the seepage analysis. The saturated hydraulic conductivity of each of the materials was based on published values for anticipated material types and compared with existing in-situ permeability testing or laboratory test results wherever possible to derive a best estimate value. Where the material permeability is expected to be variable, or is expected to have a significant impact on the estimated seepage rates, the sensitivity of the total seepage rates has been assessed by varying the saturated hydraulic conductivity within a reasonable range. Hydraulic conductivity functions for partially saturated soils were estimated based on material type.

The material parameters used in the seepage analyses are summarized in Table 1.

Table 1 Summary of Seepage Analysis Material Parameters

Unit	Saturated or Unsaturated	Horizontal Saturated Hydraulic Conductivity (m/sec)			Anisotropy Ratio (KV:KH)
		Lower Bound	Base Case	Upper Bound	
Embankment Materials					
Zone S (Core)	Saturated or Unsaturated	1E-08	5E-08	1E-07	1
Zone F (Filter)	Saturated or Unsaturated		5E-05		1
Zone T (Transition)	Saturated or Unsaturated		1E-04		1
Zone C (Waste Rock / Shell)	Saturated or Unsaturated		1E-04		1
Waste Materials					
Tailings Beach	Saturated or Unsaturated	1E-07	5E-07	1E-06	0.1
Consolidated Tailings	Saturated or Unsaturated	1E-08	5E-08	1E-07	0.1
Unconsolidated Tailings	Saturated or Unsaturated		5E-07		0.1
PAG Waste Rock	Saturated or Unsaturated		1E-04		1
Foundation Materials					
Overburden (SEE NOTE 1)	Saturated or Unsaturated		5E-07		1
Glacial Till (SEE NOTE 1)	Saturated or Unsaturated	5E-08	1E-07	5E-07	1
Orthogneiss Bedrock (to 30m depth)	Saturated or Unsaturated	5E-08	1E-07	1E-06	1
Orthogneiss Bedrock (30 to 50m depth)	Saturated or Unsaturated	2E-08	5E-08	2E-07	1
Orthogneiss Bedrock (50 to 200m depth)	Saturated or Unsaturated		1E-08		1
Orthogneiss Bedrock (200 to >500m depth)	Saturated		1E-10		1

NOTES:

1. 'Overburden' refers to the moderately permeable foundation material that is expected to comprise a combination of glacial till and colluvium in the vicinity of the non PAG waste rock stockpile and seepage collection dam, whilst 'Glacial Till' refers to the foundation material in the vicinity of the TMF.

3.2.1 Embankment Materials

The materials used in the construction of the embankments will be excavated and/or processed from the open pit and local borrow areas. The embankments will comprise the following zones:

- The core zone (Zone S) will be constructed from low-permeability glacial till from nearby external borrows and from pit stripping. The material will consist of well-graded silty sand with some gravel with a fines content of 20% to 60% passing the #200 sieve. The material will be compacted to 95% standard proctor maximum dry density (SPMDD).
- The filter zone (Zone F) will be processed material and will comprise clean, fine to coarse sand. Zone F will be placed and spread in maximum 600 mm lifts loose and compacted by four to six passes with smooth drum vibratory rollers.
- The transition zone (Zone T) will be processed material and will clean, sand and gravel. Zone T will be placed and spread in maximum 600 mm lifts loose and compacted by four to six passes with smooth-drum vibratory rollers.
- The shell zone (Zone C) will comprise random fill consisting of overburden and specific waste rock material types from the open pit. The material will be compacted by truck traffic in maximum lifts between 1 to 2 m depending on the equipment utilised.

3.2.2 Tailings and Waste Rock Materials

Laboratory testing has been completed on the tailings samples produced during lock cycle metallurgical test work. The tested tailings materials can be described as a non-plastic, fine-grained sandy-silt with traces of clay. The particle size distribution of the tailings sample comprised approximately 46-52% fine sand, 44-50% silt, and 4% clay. The Unified Soil Classification System (USCS) has been used for describing and categorizing soil within groups to allow for the development of distinct soil properties. The tailings can be classified as sand with fines (SM) and a fine-grained soil with very fine sands (ML) depending on the particle size distribution. The tailings material was grouped into three separate units for the purposes of the seepage analysis;

- The 'tailings beach' unit represents the higher permeability coarser grained fraction of the tailings that is expected to settle into the tailings basin over the length of the beach as the tailings slurries migrate towards the TMF pond
- The 'consolidated tailings' unit represents the tailings materials that have consolidated under considerable self-weight over the life of the project. A clear boundary between consolidated and unconsolidated tailings will not exist, however for modelling purposes this has been approximated to half the depth of the tailings impoundment.
- The 'unconsolidated tailings' unit represents the portion of tailings that are undergoing ongoing self-weight consolidation.

The PAG waste rock from the open pit will be placed in the TMF impoundment for subaqueous disposal. For the purposes of the seepage analysis, the PAG waste rock material is assigned the same saturated hydraulic conductivity as the shell zone (Zone C) waste rock.

3.2.3 Foundation Materials

<u>Overburden Materials</u>

The overburden thickness in the vicinity of the embankments is a glacial till material that is found to range in thickness from scarce to approximately 10 m. An average thickness was chosen to represent the overburden layer in the numerical models. The glacial till material was characterized through visual classification and laboratory particle size analysis testing. The details of the site investigation and laboratory program were presented in the 2011 Site Investigation Report (Knight Piésold Ltd., 2012a). The overburden typically consisted of silty-sand with some gravel, and is classified by the USCS as a coarse grained soil with gravel and fines (SM-SC and GM-GC).

The USCS classification group allows for comparison of anticipated geotechnical properties of the soil with published typical ranges of these properties. These properties include permeability, shear strength, compaction characteristics, workability and volume change potential of a soil, and how it will be affected by water, frost and other physical conditions. The range of material parameters was verified with respect to the expected hydraulic conductivity ranges published in Freeze and Cherry (1979).

Orthogneiss Bedrock

The bedrock unit in the vicinity of the TMF footprint comprises orthogneiss. Bedrock characterization undertaken during the 2011 site investigation program (Knight Piésold, 2012b) identified that the orthogneiss has a mean RMR of 68, a mean RQD of 74%, and a mean intact Uniaxial Compressive Strength of approximately 130 MPa. No distinct weathering profile was observed. During the site program, hydrogeological testing was completed in order to estimate the in situ hydraulic conductivity of the orthogneiss. Lugeon testing (single packer) was completed in all geotechnical and geomechanical drillholes, and falling head response testing was conducted following standpipe piezometer or monitoring well installation. The hydraulic conductivity of the orthogneiss was shown to generally decrease with depth. A plot of hydraulic conductivity values measured during the testing compared with test interval depth is shown on Figure 2.

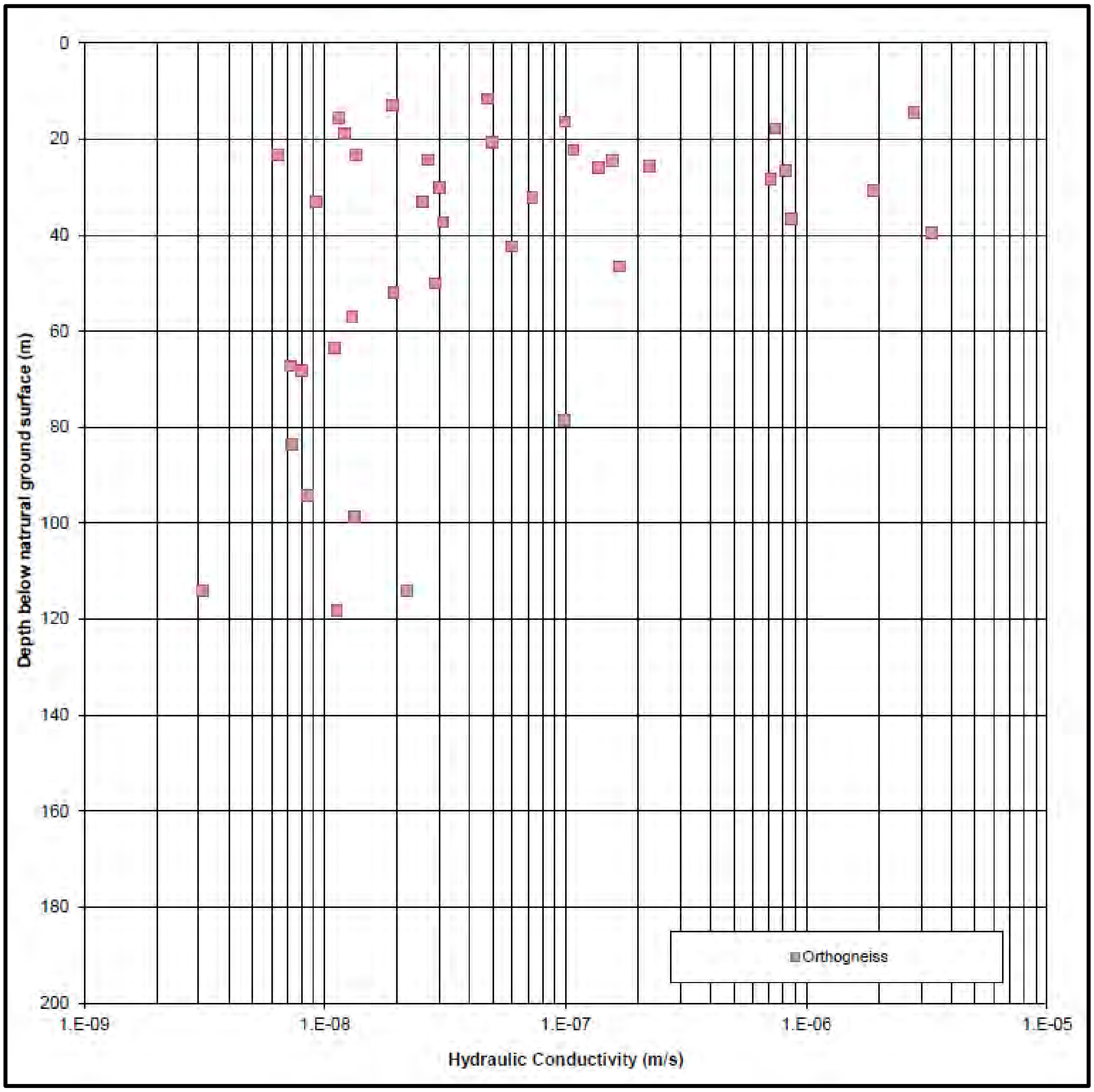


Figure 2 Hydraulic Conductivity Testing Summary – Orthogneiss Bedrock

3.3 BOUNDARY CONDITIONS AND FLUX SECTIONS

Boundary conditions used in the seepage analyses were selected to represent the hydrogeological conditions expected during operation of the TMF. The boundary conditions used in the analyses are summarized as follows:

- A total head boundary was used to represent the phreatic surface at the upstream side of the embankment for the final embankment elevations. A final embankment pond elevation of 1,834 m was modelled with a 300 m tailings beach as the base case condition.
- A total head boundary was used to represent the phreatic surface at the downstream extent of the models. The downstream phreatic surface was set at approximately 2 m below natural ground surface.
- A seepage face boundary condition was applied to the downstream face of the dam and the downstream natural ground surface to estimate the seepage flow expected to exit the ground within the model extents. The seepage flowing out of the embankment dam face was recovered and returned to the tailings pond via the seepage collection pond whilst the seepage exiting the ground downslope of the embankment dam lost to the watershed.
- A seepage face boundary condition was applied to the base of the transition zone to model the presence of a longitudinal PVC drain. The seepage flow exiting the model via this drain was recovered and returned to the tailings pond via the seepage collection pond.
- As a sensitivity case, a recharge value of 1×10^{-8} m/sec (315 mm/year) was applied to the beach of the main embankment dam sections to assess the effect of tailings water (transport water) and precipitation infiltration on the total seepage flow rates.
- As a sensitivity case, a recharge value of 1×10^{-9} m/sec (31.5 mm/year) was applied to the downslope ground surface of the saddle sections to assess the effect of precipitation infiltration on the total seepage flow rates.

Flux sections were located in key areas of the seepage models to estimate total, recovered and potentially unrecoverable seepage flows.

3.4 SEEPAGE FLOW CALCULATION METHODOLOGY

The seepage models provided an estimate of the unit seepage rate (per lineal metre of embankment) through each representative section.

The main embankment was divided into three sections (Sections 1, 2 and 3 as identified on Figure 1) and the unit seepage rates were estimated for each section. The total seepage flow was estimated by establishing a linear function between unit flow rate and dam height across the length of the dam from the three representative sections.

The seepage rates for the north embankment (Section 6), east saddle (Section 4) and the west saddle (Section 5) were estimated using a single representative cross section at each location. The total seepage flow was calculated by establishing a linear function between unit flow rate, section height, and a representative length for each section. The seepage estimate is reported by means of the following metrics:

- Total Seepage (l/s) – Indicates the total tailings seepage estimated to permeate through the TMF embankments and foundation for each section.
- Unrecoverable Seepage (l/s) – Indicates the total tailings seepage estimated to be unrecoverable and could reach the watershed downstream of the TMF with the planned seepage controls in place.
- Unrecoverable Seepage as a Percentage of Total Seepage (%) – Indicates the proportion of unrecoverable seepage relative to the total seepage originating from the TMF. This is considered a useful metric to evaluate the effectiveness of the water management features.

Representative cross sections through the main embankment, north embankment, east saddle and west saddle are shown on Figures A-1 through A-5 in Appendix A.

3.5 BASE CASE SEEPAGE RESULTS

3.5.1 Base Case Seepage Estimates

The base case seepage was estimated using the base case parameters identified in Table 1.

The base case total seepage through the main embankment and foundation was predicted to be approximately 14 l/s at the end of operations at a final embankment crest elevation of 1836 m. Approximately 1 l/s (7%) was estimated to be unrecoverable and lost to the watershed. The remaining amount was recovered in the seepage collection system and returned to the TMF.

The base case total seepage through the north embankment and foundation was predicted to be approximately 0.10 l/s at the end of operations at a final embankment crest elevation of 1836 m. The analysis indicated that the majority of this seepage will infiltrate into the foundation and will be unrecoverable, however in practice it is expected a portion of this total seepage will be recovered in the downslope seepage collection system and will be returned to the TMF.

The base case total seepage through the foundation in the vicinity of the east saddle and west saddle was estimated to be 0.11 l/s and 0.07 l/s respectively and at the end of operations and at a final pond elevation of 1834 m.

In practice, precipitation recharge on the downslope side of TMF the embankment is expected to reduce the hydraulic gradient across these saddles and the net total seepage is expected to be negligible.

3.6 MATERIAL PARAMETER SENSITIVITY ANALYSIS SEEPAGE RESULTS

A material parameter sensitivity analysis was completed for each of the sections. The sensitivity analyses were undertaken by investigating the change in total seepage estimate when the saturated hydraulic conductivity of a single material was varied in isolation. Hydraulic conductivity parameters were varied for the following materials:

- Zone S (core zone material)
- Tailings Beach Material (coarse grained tailings)
- Consolidated Tailings
- Glacial Till
- Orthogneiss Bedrock to 30 m depth, and
- Orthogneiss Bedrock from 30 m to 50 m depth.

The following sections describe the results of the material parameter sensitivity analysis completed for each of the analysis sections. Plots of the sensitivity analysis results are provided in Appendix B.

3.6.1 Main Embankment (Sections 1, 2 & 3)

The results of the sensitivity analysis for the main embankment are presented in Table 2 (below) and Figure B-1 and Figure B-2 (Appendix B). The results indicate that within the range of saturated hydraulic conductivity values selected, the main embankment dam seepage estimate is particularly sensitive to the saturated hydraulic conductivity of the 'Tailings Beach' material and the uppermost layer of the orthogneiss bedrock (<30 m depth below natural ground level (ngl)). The unrecoverable seepage is shown to be most notably sensitive to the saturated hydraulic conductivity of the uppermost layer of the orthogneiss bedrock (<30 m depth below ngl).

Table 2 Upper, Lower Bound and Base Case Seepage Estimates – Main Embankment

Sensitivity Analysis NOTE 1	Lower Bound	Base Case	Upper Bound
Material	**Total Seepage (l/s)**		
Zone C	13	14	15
Tailings Beach	9		17
Consolidated Tailings	13		15
Glacial Till	14		15
Orthogneiss Bedrock (to 30 m depth)	14		19
Orthogneiss Bedrock (30 to 50 m depth)	14		15
Material	**Unrecoverable Seepage (l/s)**		
Zone C	1	1	1
Tailings Beach	1		1
Consolidated Tailings	1		1
Glacial Till	1		1
Orthogneiss Bedrock (to 30 m depth)	1		4
Orthogneiss Bedrock (30 to 50 m depth)	1		2
Material	**Unrecoverable Seepage as a percentage of Total (%)**		
Zone C	9	7	7
Tailings Beach	11		6
Consolidated Tailings	8		7
Glacial Till	7		9
Orthogneiss Bedrock (to 30 m depth)	7		18
Orthogneiss Bedrock (30 to 50 m depth)	6		12

NOTES:

1. The Base Case seepage estimate was completed as a single case using the Base Case material parameters as identified in Table 1. The Lower Bound and Upper Bound seepage estimates were completed using the Lower and Upper bound seepage parameters as identified in Table 1, with the sensitivity of each material varied in isolation for each respective case.

3.6.2 North Embankment (Section 6)

The results of the sensitivity analysis for the north embankment are presented in Figure B-3 and Figure B-4 (attached). The results indicate that within the range of saturated hydraulic conductivity values selected, the north embankment seepage estimate is particularly sensitive to the saturated hydraulic conductivity of the uppermost layer of the orthogneiss bedrock (<30 m depth below ngl) and the second layer of orthogneiss bedrock (30 to 50 m depth below ngl). For each case, the estimate of unrecoverable seepage is expected to be over 95% of the total seepage estimate.

3.6.3 East Saddle (Section 4)

The results of the sensitivity analysis for the east saddle are presented in Figure B-5. The results indicate that that within the range of saturated hydraulic conductivity values selected, the east saddle seepage estimate is sensitive to the saturated hydraulic conductivity of the uppermost layer of the Orthogneiss bedrock (<30 m depth below ngl) and the second layer of orthogneiss bedrock (30 to 50 m depth below ngl) with an upper bound total seepage estimate of 0.20 l/s.

3.6.4 West Saddle (Section 5)

The results of the sensitivity analysis for the west saddle are presented in Figure B-6. The results indicate that within the range of saturated hydraulic conductivity values selected, the North embankment dam seepage estimate is sensitive to the saturated hydraulic conductivity of the uppermost layer of the Orthogneiss bedrock (<30 m depth below ngl) with an upper bound total seepage estimate of 0.39 l/s.

3.7 BOUNDARY CONDITIONS SEEPAGE SENSITIVITY ANALYSIS

3.7.1 Effect of Recharge Water on Tailings Beach

A recharge boundary condition of 1 x 10^{-8} m/sec (315 mm/year) was applied to the tailings beach at the main embankment to assess the effect of tailings transport water and precipitation on the total seepage rates. The total seepage estimate for the main embankment was found to increase to 19 l/s (132% of the base case estimate) with 1 l/s unrecovered seepage (unchanged).

3.7.2 Effect of Tailings Beach

In normal operating conditions, the tailings beach is expected to extend approximately 300 m from the main embankment crest. A scenario was modelled to determine an upper bound seepage estimate assuming the supernatant pond was allowed to reach the embankment dam (i.e. no tailings beach). The result was an increase in total seepage by an order of magnitude, with a total seepage of approximately 160 L/s. Unrecoverable seepage did not increase in this scenario, indicating in this upper bound case, seepage could still be captured at the downstream water management pond and recycled back to the TMF for long-term storage.

4 – TAILINGS MANAGEMENT FACILITY STABILITY ANALYSES

4.1 MODELLING APPROACH

Stability analyses of the TMF embankment were carried out to investigate the slope stability under both static and seismic loading conditions. The following cases were evaluated:

- Static conditions during operations and post-closure.
- Earthquake loading from the Operating Basis Earthquake (OBE), the Maximum Design Earthquake (MDE), and Earthquake loading from the 1:10,000 year earthquake event.
- Post-earthquake conditions using residual (post-liquefaction) tailings strengths.

Representative cross sections through the main and north embankments were based on the geotechnical foundation conditions and the maximum section for each embankment. The analyses were carried out for the following embankment configurations:

- Final embankment (crest elevation 1836 m) with full tailings storage and pond elevation at 1834 m.
- Stage 1 embankment (crest elevation 1720 m) with no tailings deposition and no retained water (main embankment only – upstream failure mode).
- Stage 1 embankment (crest elevation 1720 m) with no tailings deposition and pond water level at 1718 m (main embankment only – downstream failure mode).

The stability analyses were carried out using the limit equilibrium computer program SLOPE/W (Geostudio, 2007). In this program a systematic search is performed to obtain the minimum factor of safety from a number of potential slip surfaces. Factors of safety have been computed using the Morgenstern-Price Method.

In accordance with international recommendations (ICOLD, 1995) and standard industry practice, the minimum acceptable factor of safety for the tailings embankment under static conditions is 1.5 for normal operating conditions and for long-term (post-closure) of the TMF. A factor of safety of less than 1.0 is acceptable for earthquake loading conditions provided that calculated embankment deformations resulting from seismic loading are not significant and that the post-earthquake stability of the embankment maintains a factor of safety greater than 1.2, to ensure there is no potential for a flow-slide failure following liquefaction. Limited deformation of the embankment is acceptable under seismic loading from the MDE, provided that the overall stability and integrity of the TMF is maintained and that there is no release of stored tailings or water. Some remediation may be required following the MDE.

4.2 MATERIAL PARAMETERS AND ASSUMPTIONS

The following parameters and assumptions were incorporated into the stability analyses:

- Bulk unit weights for the embankment and foundation materials were based on laboratory testing or typical values for similar materials.
- An undrained shear strength was adopted to represent the tailings material strength for the static, seismic and post-earthquake cases, as described by the following relation:
  - $S_u/p' = 0.25$ (static and seismic loading)
  - Su/p' = 0.10 (post liquefaction residual strength), where;
    - S_u = undrained shear strength, and
    - p' = effective vertical stress.
- Effective strength parameters for the embankment fill and foundation materials were estimated based on typical values for similar materials.
- The shear strength for Zone C was defined using a conservative strength function that defines the variation with shear strength with normal stress. This strength function is based on published information on the shear strength properties of rockfill (Leps, 1970).
- A piezometric line was used to represent the predicted phreatic surface in the stability analysis as determined from the seepage analysis.

The material strength parameters adopted for the stability analyses are summarized in Table 3.

The embankment geometries analyzed for the main embankment are shown on Figures C-1 and C-2 (Appendix C) for the Stage 1A embankment and final embankment, respectively. The geometry of the final north embankment used in the stability analyses is shown in Figure C-3.

Table 3 Material Strength Parameters

Unit	Unit Weight (kN/m^3)	Friction Angle (deg)	Cohesion (kPa)
Embankment Materials			
Zone S (Core)	22	34	0
Zone F (Filter)	21	36	0
Zone T (Transition)	21	36	0
Zone C (Waste Rock / Shell)	23	See Note 1	
Tailings Materials			
Tailings Beach	18	See Note 2	
Consolidated Tailings	18	See Note 2	
Unconsolidated Tailings	18	See Note 2	
Waste Rock			
Non PAG Waste Rock	23	See Note 1	
PAG Waste Rock	23	See Note 1	
Foundation Materials			
Overburden (See Note 1)	22	36	0
Glacial Till (See Note 1)	22	36	0
Orthogneiss Bedrock	Impenetrable		

NOTES:

1. A relationship for friction angle and effective stress was developed for the rockfill materials, based on published information on the shear strength properties of rockfill (Leps, 1970).
2. A relationship for shear stress and effective normal stress (S_u/p') was used to model the tailings strength. The (S_u/p') values used for the analyses were 0.25 for static and seismic loading and 0.1 for liquefied tailings.

4.3 RESULTS OF STABILITY ANALYSIS

4.3.1 Static Analyses

The calculated Factors of Safety (FOS) for each of the dam sections considered in this study exceed the minimum Factor of Safety requirement of 1.5 for static normal operating (steady-state) conditions. In addition, calculated FOS for short term stability of the upstream starter embankment dam (prior to tailings deposition) exceeds the minimum Factor of Safety of 1.3 for static operating (steady-state) conditions. It should be further noted that the critical surface identified for each static analysis does not result in any loss of freeboard as the critical failure surface is shown not to pass through the dam crest. A summary of the Factors of Safety (FOS) for the cases analysed are presented in Table 4.

Table 4 Static Analyses Results Summary

Description	Minimum FOS	Comments
TMF Main Embankment at EL 1836 m (Final Height)		
Normal Operating Conditions	1.56	-
TMF Main Embankment at EL 1720 m (Starter Embankment – Stage 1A)		
Normal Operating Conditions	1.71	-
Normal Operating Conditions – Failure of upstream slope	1.42	-
Normal Operating Conditions – Pond at EL 1718 m	1.63	No tailings deposition, water in impoundment to EL 1718 m
TMF North Embankment at EL 1836 m (Final Height)		
Normal Operating Conditions	2.04	-

NOTES:
1. Only slip surfaces with a minimum of 2 m depth have been considered in the analysis.

4.3.2 Seismic Stability and Deformation Analyses

A seismic stability assessment of the TMF has included estimation of earthquake induced deformation of the embankment from the OBE, MDE, and the 1:10,000 event. The design ground motion parameters for the design earthquake events have been provided by the seismic hazard analysis completed for the project (Knight Piésold, 2012c).

The OBE has been defined as the 1 in 475 year earthquake with a mean Peak Ground Acceleration (PGA) of 0.08g. A design earthquake magnitude of 7 was adopted for the OBE.

The MDE has been assessed to correspond with the Earthquake Design Ground Motion (EDGM) as per table 6-1B of the 2013 revision to the 2007 CDA Dam Safety Guidelines. The guidelines revision states that the EDGM for a Dam Class 'Very High' should be selected based on the mean PGA corresponding to halfway between the PGA for the 1 in 2,475 year earthquake and the PGA for the 1 in 10,000 year earthquake. This corresponds to a PGA of 0.21g. A design earthquake magnitude of 7.3 was adopted for the MDE.

The PGA acceleration for the 1:10,000 year event has also been considered to demonstrate the robustness of the embankment design in closure to seismic loading. The 1 in 10,000 year earthquake corresponds with a PGA of 0.26g. A design earthquake magnitude of 7.3 was adopted for the 1:10,000 year event.

Embankment stability during earthquake loading from the OBE, MDE and 1:10,000 year event has been assessed by performing pseudo-static analysis, whereby a horizontal force (seismic coefficient) is applied to the embankment to simulate earthquake loading. The yield acceleration required to reduce the factor of safety to 1.0 was determined by iterative stability analyses. Deformation of the embankment is predicted to occur if the yield acceleration is lower than the average maximum ground acceleration along the potential slip surface from the earthquake.

Potential deformations under earthquake loading from the design earthquake events have been estimated using the simplified methods of Newmark (1965) and Makdisi-Seed (1977). These two methods estimate displacement of the potential sliding mass based on the average maximum ground acceleration along the slip surface and the yield acceleration.

The more recently published method of Bray (2007) was also used to predict seismically induced slide displacement of the embankment. In addition to the yield acceleration, this method considers the predominant period of response (Ts) of the embankment under seismic loading and the corresponding spectral ground acceleration (Sa). The predominant period is related to the stiffness characteristics of the embankment fill and to the height of the embankment. Spectral acceleration values were provided by the uniform hazard spectrum defined for each design earthquake event. The uniform hazard spectra for the design earthquake events were defined from the results of the site specific probabilistic seismic hazard analysis (Knight Piésold, 2012c).

The estimated yield acceleration is 0.2g for the Main Embankment at final height, between 0.18g and 0.23g for the Main Embankment at the starter height (elevation 1720 m) and 0.35g for the North Embankment at final height. Predicted embankment deformations under seismic loading are negligible, if any, as the calculated yield acceleration either exceeds, or is only slightly lower than the estimated average PGA values for the OBE and MDE events. For the 1:10,000 event, the estimated deformations are very small (<0.03 m) and do not impact the embankment freeboard or result in any loss of embankment integrity.

Some deformation of the embankment is expected to result from settlement of the fill materials during earthquake shaking. Potential settlement of the embankment crest has been estimated using the empirical relationship provided by Swaisgood (2003). This relationship was developed from an extensive review of case histories of embankment dam behaviour due to earthquake loading. Required inputs to the relationship are the earthquake magnitude, the maximum acceleration on rock at the site, the depth to rock (overburden thickness) and the embankment height. The predicted maximum crest settlements for the Main Embankment at final height are approximately 0.05 m for the OBE, 0.14 m for the MDE and 0.19 m for the 1:10,000 year event. The predicted maximum crest settlements for the North Embankment at final height are minor (<0.02) for all design earthquake events.

The calculated yield accelerations and corresponding estimated embankment deformations and crest settlements for each of the methods described above are presented in Table 5.

The predicted maximum embankment displacements and potential crest settlements under seismic loading from the OBE and MDE are acceptable and would not significantly impact embankment freeboard or result in any loss of embankment stability or integrity. The performance and integrity of the embankment core, drainage and filter zones would not be impacted by the predicted deformations.

The findings of the seismic stability analyses indicate that the TMF would remain stable and function normally after the OBE, MDE and 1:10,000 year event.

Table 5 TMF Seismic Displacement Results Summary

Description	Design PGA[1] (g) Mean[2]	Design Earthquake Magnitude	Calculated Yield Acceleration (K_Y)[3]	Displacement Along Slip Surface (m)			Crest Settlement (m)
				Newmark[4]	Makdisi-Seed (Average)[4]	Bray ($D_{84\%}$)[5]	Swaisgood[6]
TMF Main Embankment at EL 1836 m (Final Height)							
OBE	0.08	7	0.20	0.00	0.00	0.00	0.05
MDE	0.21	7.3	0.20	0.00	0.02	0.00	0.14
1:10,000 event	0.26	7.3	0.20	0.01	0.03	0.02	0.19
TMF Main Embankment at EL 1720 m (Starter Embankment – Stage 1A)							
OBE – Full tailings height volume	0.08	7	0.18	0.00	0.00	0.00	0.02
MDE – Empty Impoundment	0.21	7.3	0.18	0.01	0.04	0.01	0.05
MDE – Pond at EL 1718 m	0.21	7.3	0.23	0.00	0.00	0.01	0.05
TMF North Embankment at EL 1836 m (Final Height)							
OBE	0.08	7	0.35	0.00	0.00	0.00	0.01
MDE	0.21	7.3	0.35	0.00	0.01	0.01	0.02
1:10,000 Event	0.26	7.3	0.35	0.00	0.00	0.01	0.02

NOTES

1. The design maximum acceleration is for site class C conditions (defined as soft rock or very dense soils).
2. Mean acceleration values are conservatively estimated by multiplying the median acceleration value by 1.15. Mean acceleration values are recommended for dam design by the Canadian Dam Association "Dam Safety Guidelines" (2007).
3. The yield acceleration (ky) corresponds to the horizontal seismic coefficient (acceleration) required to reduce the factor of safety to 1.0
4. The Newmark (1965) and Makdisi-Seed (1977) methods estimate potential displacement along the critical slip surface.
5. The Bray (2007) method estimates potential displacement taking into consideration the fundamental period of the structure (Ts) and the ground motion's spectral acceleration at a degraded period equal to 1.5Ts.
6. The Swaisgood (2003) method estimates the predicted vertical settlement of the dam crest
7. Slip surfaces are a minimum of 2 m depth

4.3.3 Post-Liquefaction Stability Analysis

A stability assessment of the TMF has been undertaken to assess the static stability of the embankments following an earthquake event. The calculated Factors of Safety (FOS) for each of the dam sections considered in this study exceed the minimum Factor of Safety requirement of 1.2 for post liquefaction stability.

The post-earthquake condition conservatively assumes complete liquefaction of the tailing deposit and assumes a post-liquefaction residual strength for the entire tailings deposit. For each of the dam sections the calculated minimum factors of safety are the same as the static factor of safety as the critical potential slip surface does not pass through the liquefied tailing deposit. This indicates that the TMF embankment is not dependent on tailing strength to maintain stability and is not susceptible to a flow slide or large deformations resulting from earthquake-induced liquefaction of the tailing deposit.

A summary of the Factors of Safety (FOS) for the cases analysed are presented in Table 6.

Table 6 Post-Liquefaction Analyses Results Summary

Description	Minimum FOS	Comments
TMF Main Embankment at EL 1836 m (Final Height)		
Post Liquefaction Stability - Reduced Tailings Strength	1.56	Failure does not propagate into tailings (see Note 2)
TMF North Embankment at EL 1836 m (Final Height)		
Post Liquefaction Stability - Reduced Tailings Strength	2.04	Failure does not propagate into tailings (see Note 2)

NOTES:

1. Only slip surfaces with a minimum of 2 m depth have been considered in the analysis.
2. The post liquefaction Factor of Safety is the same as the pre earthquake static case as critical potential slip surfaces do not pass through the tailings deposit.

5 – NON PAG WASTE STOCKPILE STABILITY

The non PAG waste stockpile was assessed against the Dump Stability Rating (DSR) scheme from the Investigation and Design Manual Interim Guidelines (BC MWRPRC, 1991). A stability analysis was also undertaken to determine the factors of safety for the stockpile.

5.1 WASTE STOCKPILE STABILITY RATING SCHEME

The Investigation and Design Manual Interim Guidelines (BC MWRPRC, 1991) provides recommendations for stability assessment of mine waste piles. These guidelines include a Dump Stability Rating (DSR) scheme. The DSR system provides a semi-quantitative method for assessing the relative potential of dump stability and recommends the appropriate level of investigation and design. This is based on individual point ratings for each of the main factors affecting dump stability. Each factor is given a point rating based on qualitative and/or quantitative descriptions accounting for the possible range of conditions. An overall DSR is calculated as the sum of the individual ratings for each of the various factors. Copies of Table 5.1 “Dump Stability Rating Scheme” and Table 5.2 “Dump Stability Classes and Recommended Level of Effort” from the waste dump research committee guidelines are included in Appendix D.

The dump rating guidelines were used to classify the Non PAG Waste Stockpile. A summary of the results are presented in Table 7. The Non-PAG Waste Stockpile is classified as Class III, Moderate Hazard. The Moderate Hazard classification recommends that additional site investigations, including laboratory testing and a detailed stability analysis be completed for the next level of detailed design.

Table 7 Non-PAG Waste Rock Stockpile Stability Classification

Key Factors Affecting Stability[(1)]	Condition	Point Rating
Dump Height	100 - 200 m	100
Dump Volume	Large	100
Dump Slope	Moderate	50
Foundation Slope	Moderate	50
Degree of Confinement	Confined	0
Foundation Type	Intermediate	100
Dump Material Quality	Moderate	100
Method of Construction	Mixed	100
Piezometric & Climatic Conditions	Intermediate	100
Dumping Rate	Moderate	100
Seismicity	Moderate	50
DUMP STABILITY RATING		**850**
	Class	**Failure Hazard**
Dump Stability Class[(2)]	III	Moderate

In general, the dump stability classification indicates a basic stability analysis is required. In accordance with provincial guidelines (BC MWRPRC, 1991) and standard industry practice, the minimum acceptable factor of safety for waste dumps under static conditions is 1.3 for short-term operating conditions, 1.5 after reclamation and abandonment and 1.0 for a pseudo-static analysis. The BC Mine Waste Rock Pile Research Committee (MWRPRC) interim guidelines for design factors of safety are presented in Appendix D (Table 6.4).

5.2 NON-PAG WASTE STOCKPILE STABILITY ANALYSES

Slope stability analyses for the non PAG Waste Stockpile were carried out for the final design height of the stockpile (closure condition). The stability analyses were carried out using the 2D finite element software SLOPE/W (Geostudio, 2007) along the section identified in plan on Figure 3. The analysis was undertaken to assess the stability of the maximum height of the stockpile slope. The effect of the interaction of the waste stockpile on the open pit slope stability was not assessed for this study.

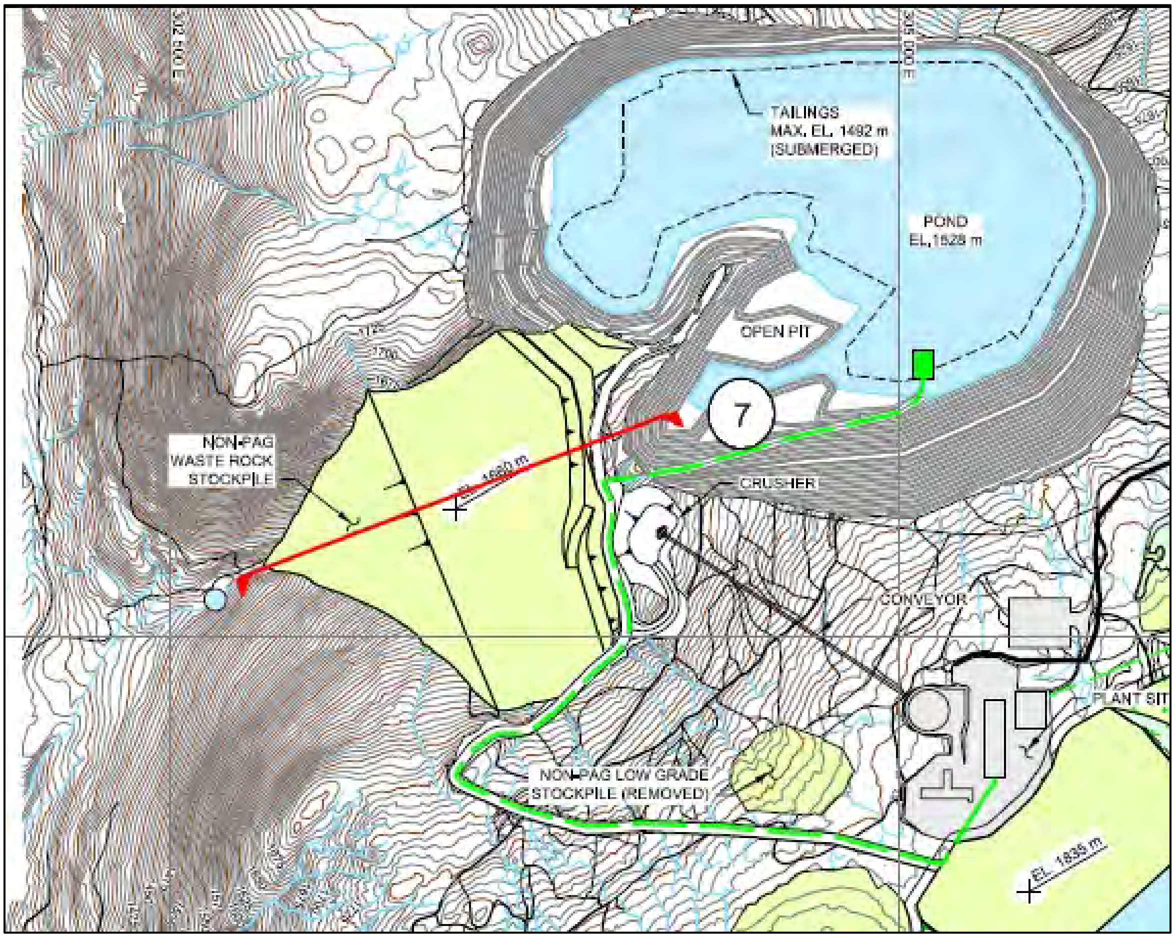


Figure 3 Non PAG Waste Stock Pile General Arrangement at closure with 2D analysis section (Section 7) identified

The static Factor of Safety against failure is 1.52 and the pseudo-static Factor of Safety against failure from an applied PGA corresponding to the 1:475 event (defined as the event which has a 10% probability of exceedance in 50 years) was determined to be 1.38. Both the static and pseudo-static Factors of Safety exceed the minimum design Factors of Safety as presented in Table 6.4 of the BC MWRPRC (1991) and included in Appendix D. The critical potential failure surface and factor of safety for the static condition is shown on Figure C-4.

In order to demonstrate the robustness of the design, seismic displacements were estimated according to the methods of Newmark (1965), Makdisi and Seed (1977), Bray (2007) and Swaisgood (2003) (described in detailed in Section 4.3.2). The ground motion parameters for the 1:10,000 year events as identified in the TMF stability analysis were used to estimate the seismic displacements for the waste stockpile. The estimated yield acceleration is 0.19g. Predicted displacements under seismic loading for the 1:10,000 event are shown to be negligible and estimated crest settlement is 0.29 m.

6 – REFERENCES

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7 – CLOSURE

This letter report presents a summary of the stability and seepage analyses undertaken for the Harper Creek mining project to date.

We trust the information contained herein meets your needs at this time. Should you required additional information please contact the undersigned.

Yours truly,
KNIGHT PIESOLD LTD.

PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA
A. C. ROBB
174229
19.08.2014.

Signed: Angus Robb, P.Eng.
Project Engineer

Reviewed: Daniel Fontaine, P.Eng
Senior Engineer

Approved: Bruno Borntrager for
Ken Brouwer, P.Eng.
President

Attachments:

Appendix A	Seepage Analysis Figures
Appendix B	Seepage Sensitivity Analysis Plots
Appendix C	Stability Analysis Plots
Appendix D	Selected Tables from the Investigation and Design Manual Interim Guidelines (BC MWRPRC, 1991)

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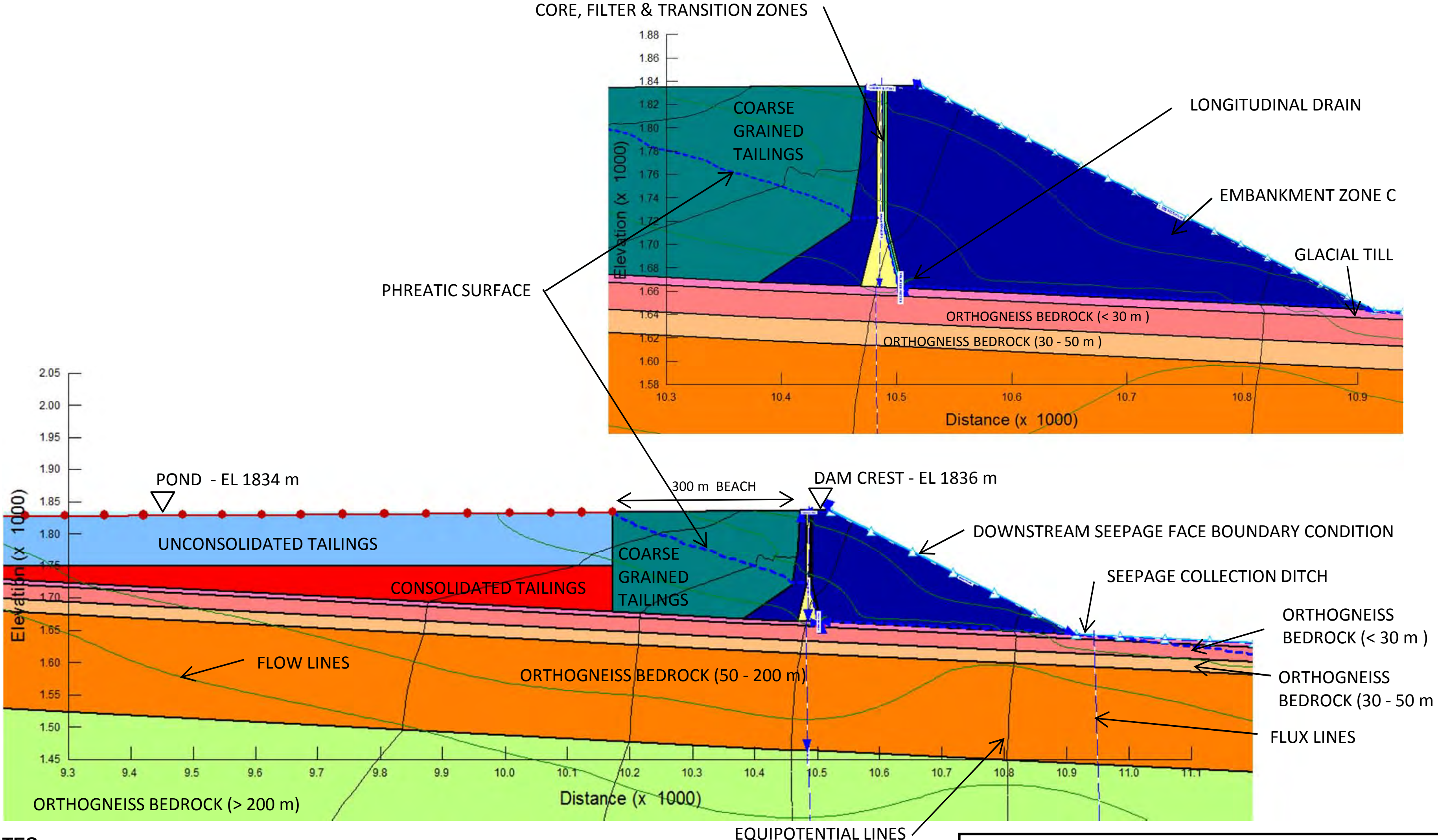
APPENDIX A

SEEPAGE ANALYSIS FIGURES

(Figures A-1 to A-5)



NOTES:

1. LOCATION OF PHREATIC SURFACE PREDICTED FROM STEADY STATE SEEPAGE ANALYSIS.

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
SEEPAGE ANALYSIS
MAIN EMBANKMENT - SECTION 1

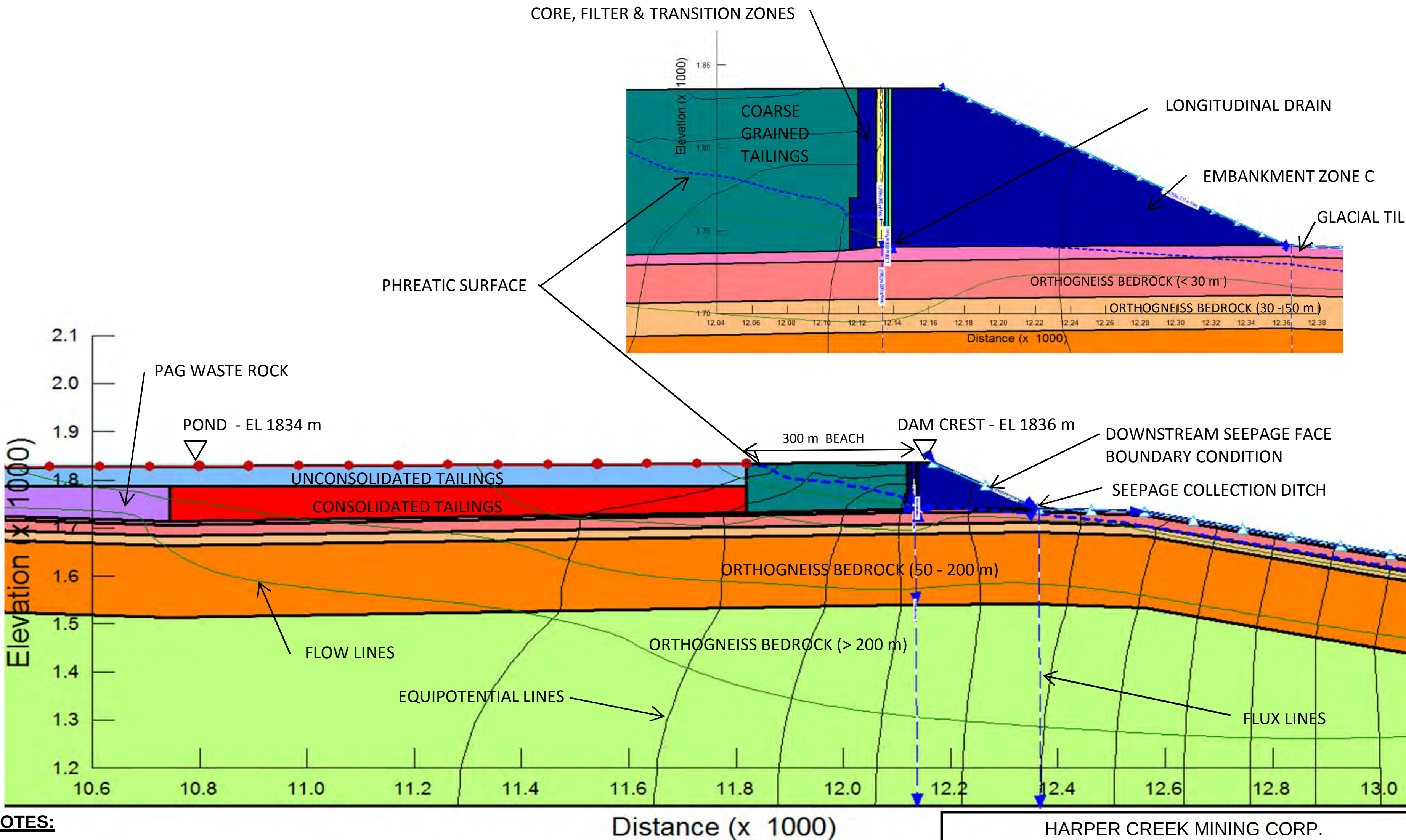
Knight Piésold CONSULTING

P/A. NO. VA101-458/14

REF NO. VA14-00865

FIGURE A-1

REV 0



NOTES:

1. LOCATION OF PHREATIC SURFACE PREDICTED FROM STEADY STATE SEEPAGE ANALYSIS.

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
SEEPAGE ANALYSIS
MAIN EMBANKMENT - SECTION 2 & 3

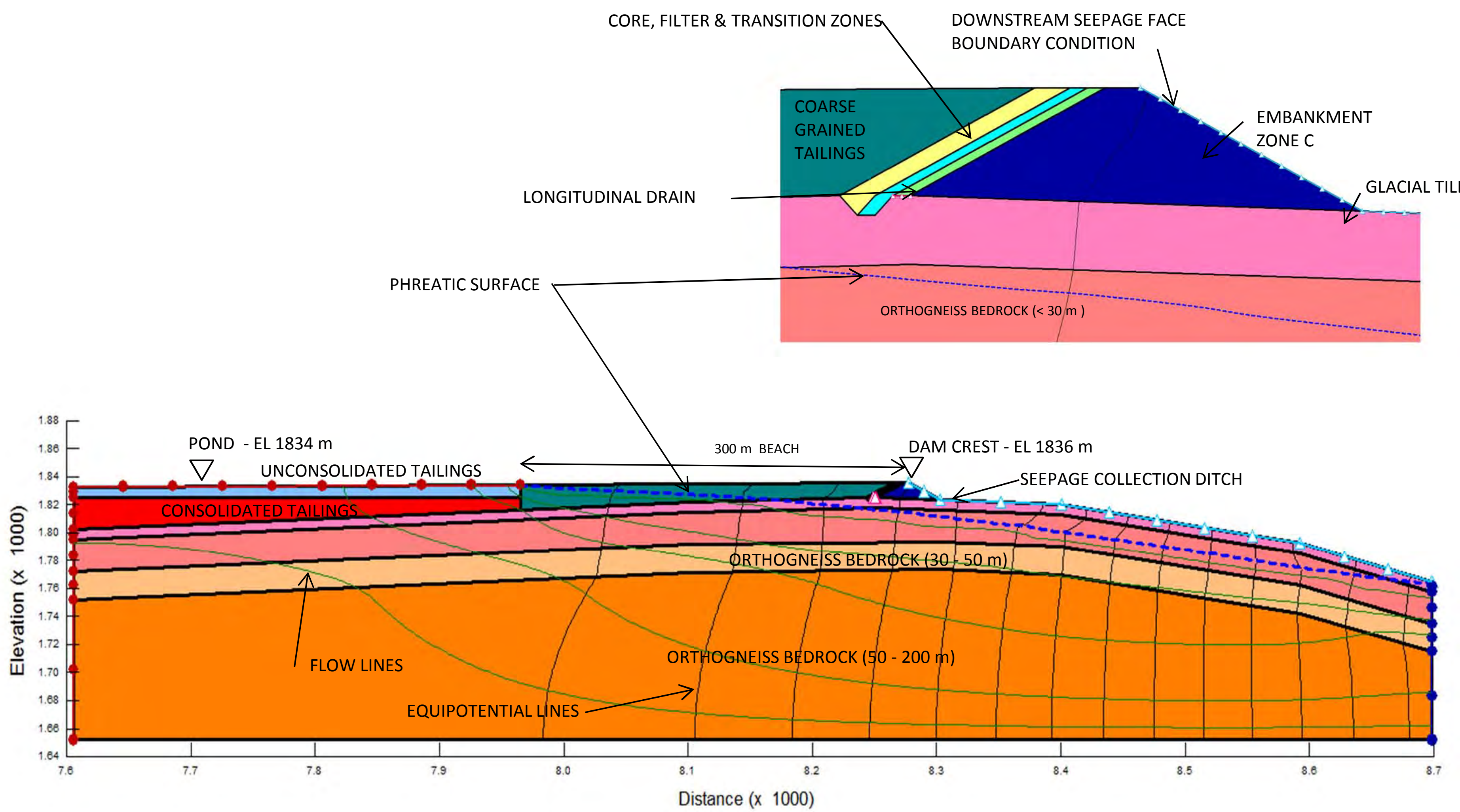
Knight Piésold CONSULTING

P/A. NO. VA101-458/14

REF NO. VA14-00865

FIGURE A-2

REV 0



NOTES:

1. LOCATION OF PHREATIC SURFACE PREDICTED FROM STEADY STATE SEEPAGE ANALYSIS.

0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
SEEPAGE ANALYSIS
NORTH EMBANKMENT - SECTION 6

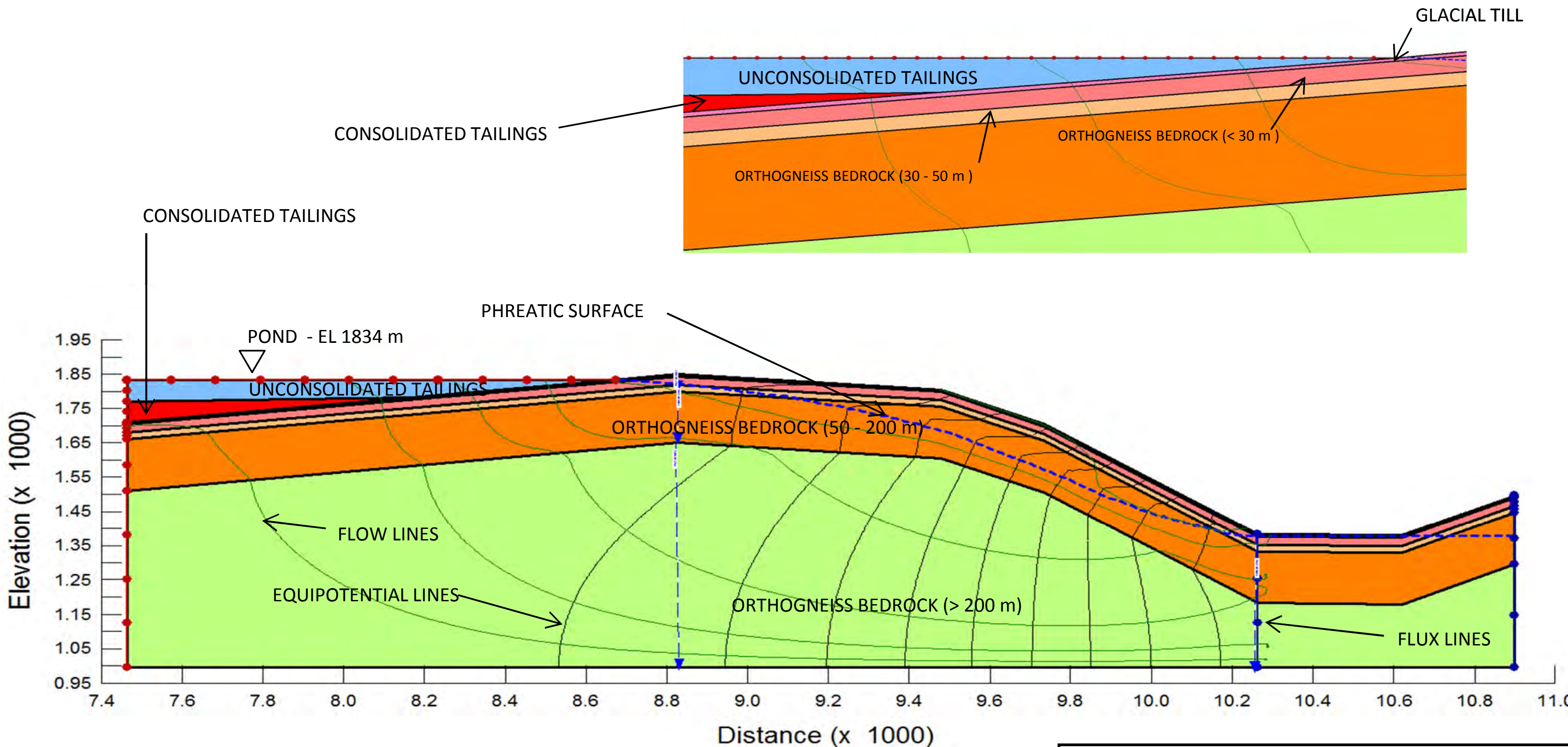
Knight Piésold CONSULTING

P/A. NO. VA101-458/14

REF NO. VA14-00865

FIGURE A-3

REV 0



NOTES:

1. LOCATION OF PHREATIC SURFACE PREDICTED FROM STEADY STATE SEEPAGE ANALYSIS.

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
SEEPAGE ANALYSIS
EAST SADDLE - SECTION 4

Knight Piésold CONSULTING

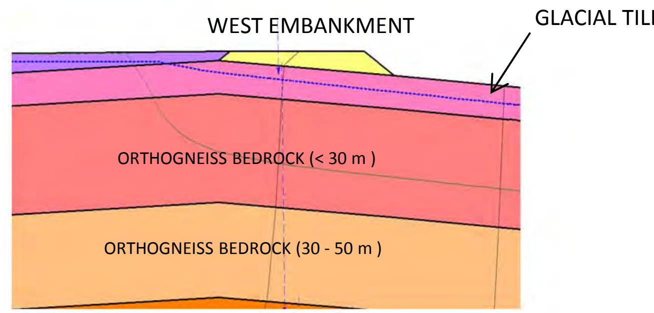
P/A. NO. VA101-458/14

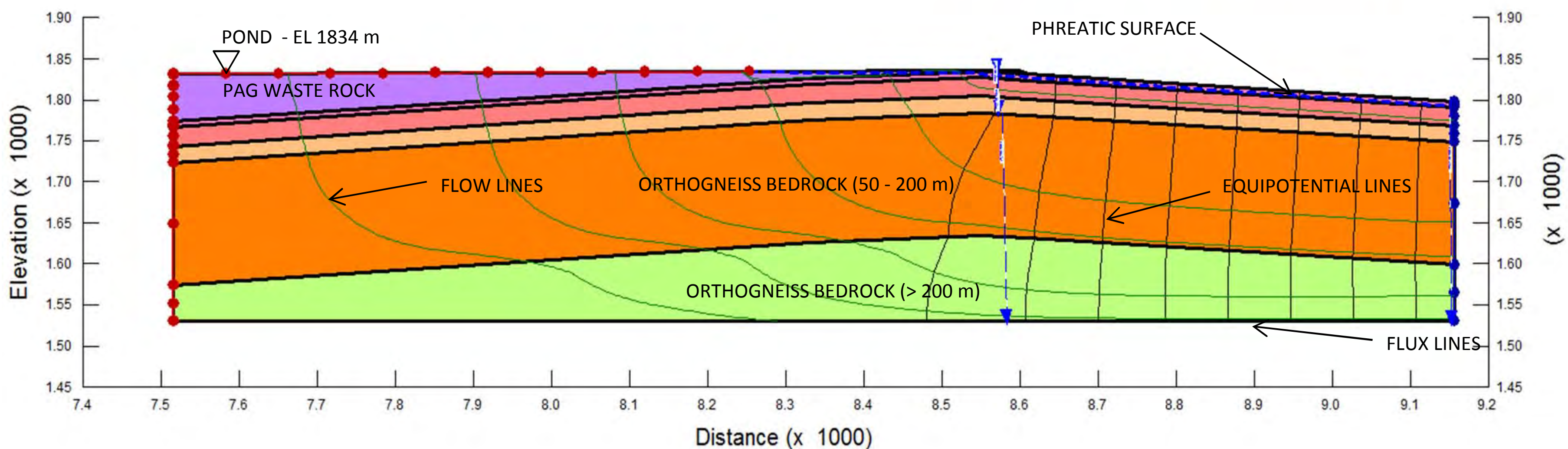
REF NO. VA14-00865

FIGURE A-4

REV 0

0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D





NOTES:

1. LOCATION OF PHREATIC SURFACE PREDICTED FROM STEADY STATE SEEPAGE ANALYSIS.

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
SEEPAGE ANALYSIS
WEST SADDLE - SECTION 5

Knight Piésold CONSULTING

P/A. NO. VA101-458/14

REF NO. VA14-00865

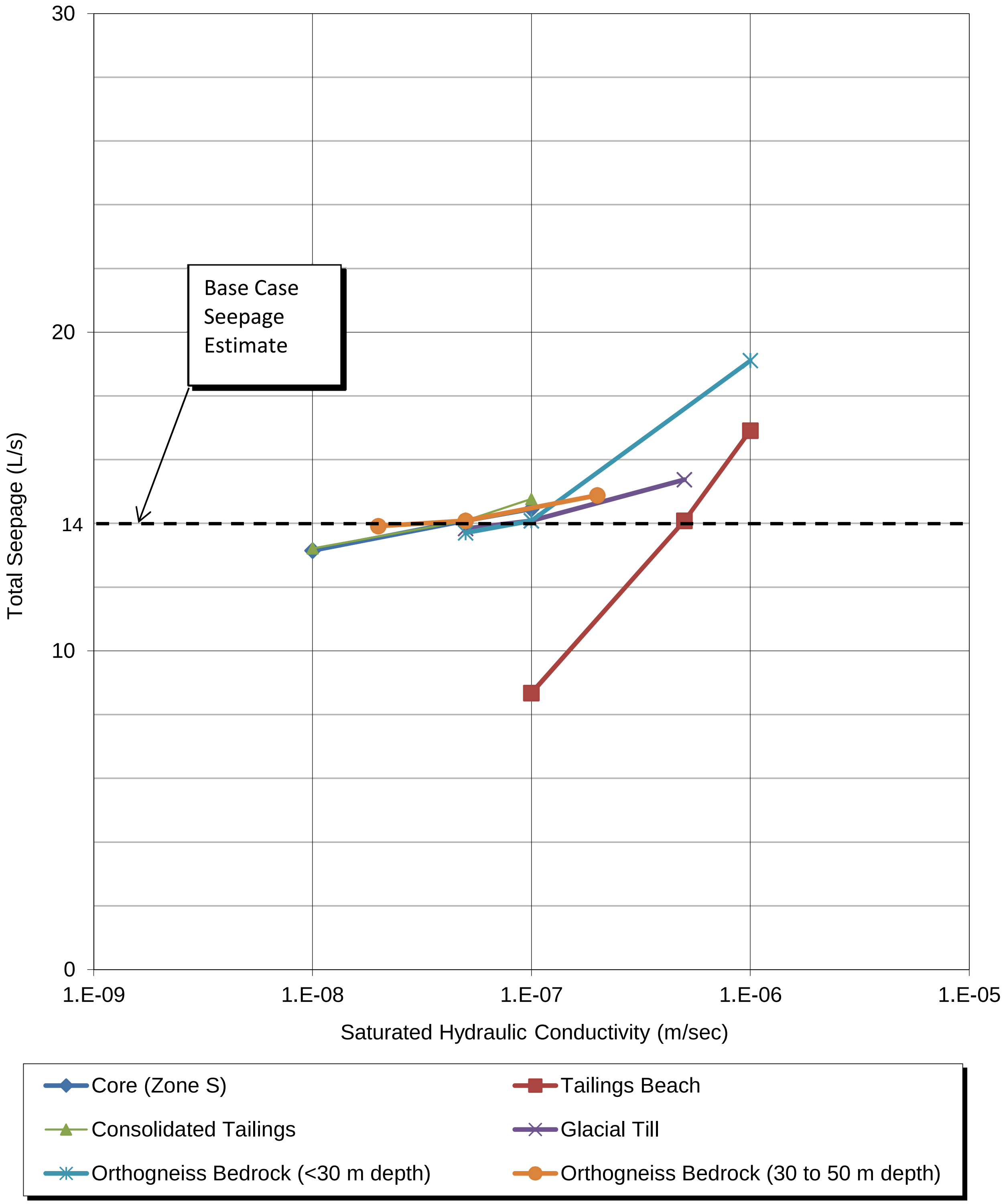
FIGURE A-5

REV 0

APPENDIX B

SEEPAGE SENSITIVITY ANALYSIS PLOTS

(Figures B-1 to B-6)



NOTES:

1. EACH SERIES REPRESENTS THE SENSITIVITY ANALYSIS PERFORMED FOR THE MATERIAL IDENTIFIED IN THE SERIES TITLE, WITH ALL OTHER MATERIAL PARAMETERS AS PER THE BASE CASE PARAMETERS

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

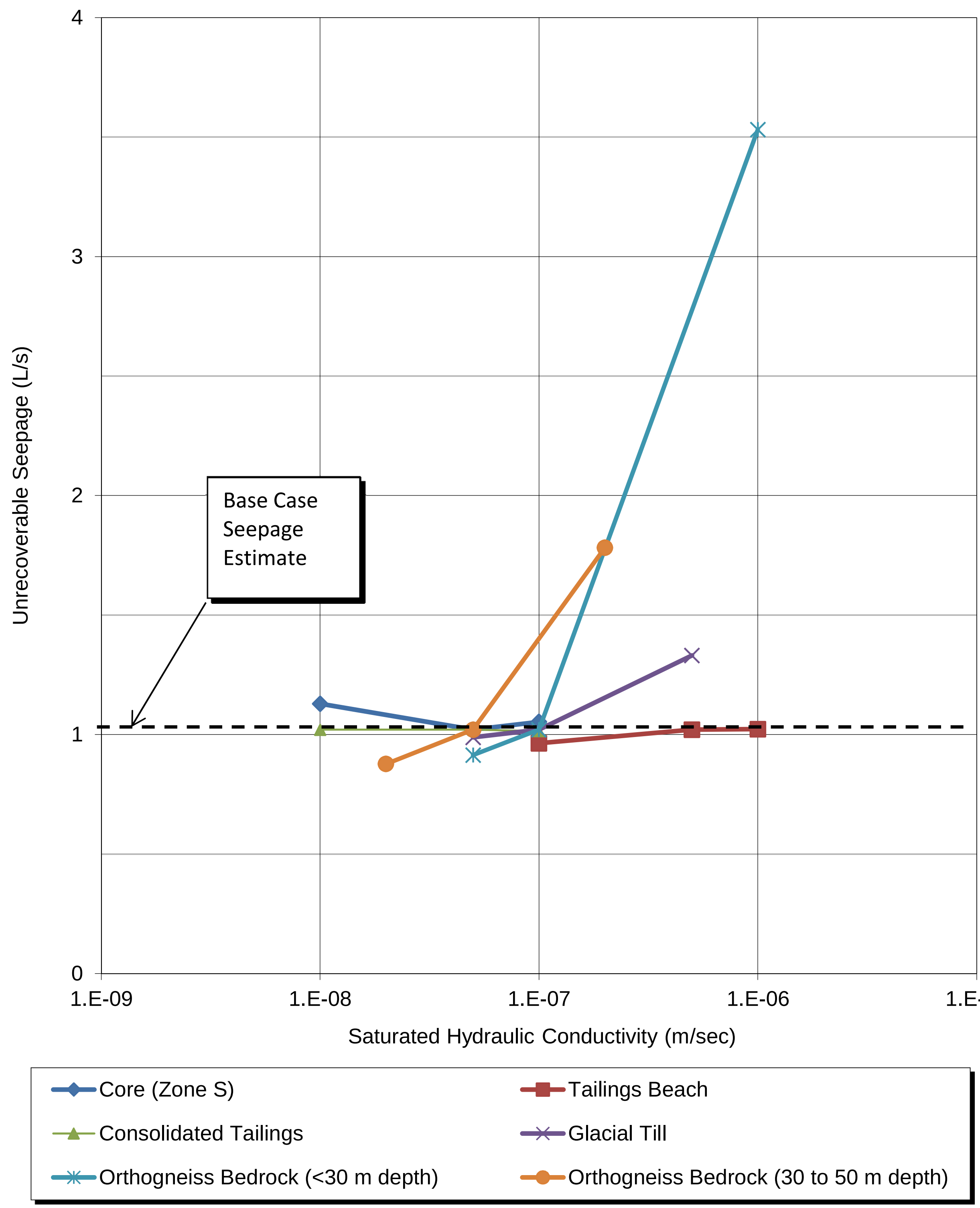
SECTIONS 1,2, & 3 - MAIN EMBANKMENT TMF SEEPAGE SENSITIVITY ANALYSIS TOTAL SEEPAGE

Knight Piésold CONSULTING

P/A NO.	REF. NO.
VA101-458/14	VA14-00865

FIGURE B-1 REV 0

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB



NOTES:

1. EACH SERIES REPRESENTS THE SENSITIVITY ANALYSIS PERFORMED FOR THE MATERIAL IDENTIFIED IN THE SERIES TITLE, WITH ALL OTHER MATERIAL PARAMETERS AS PER THE BASE CASE PARAMETERS

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

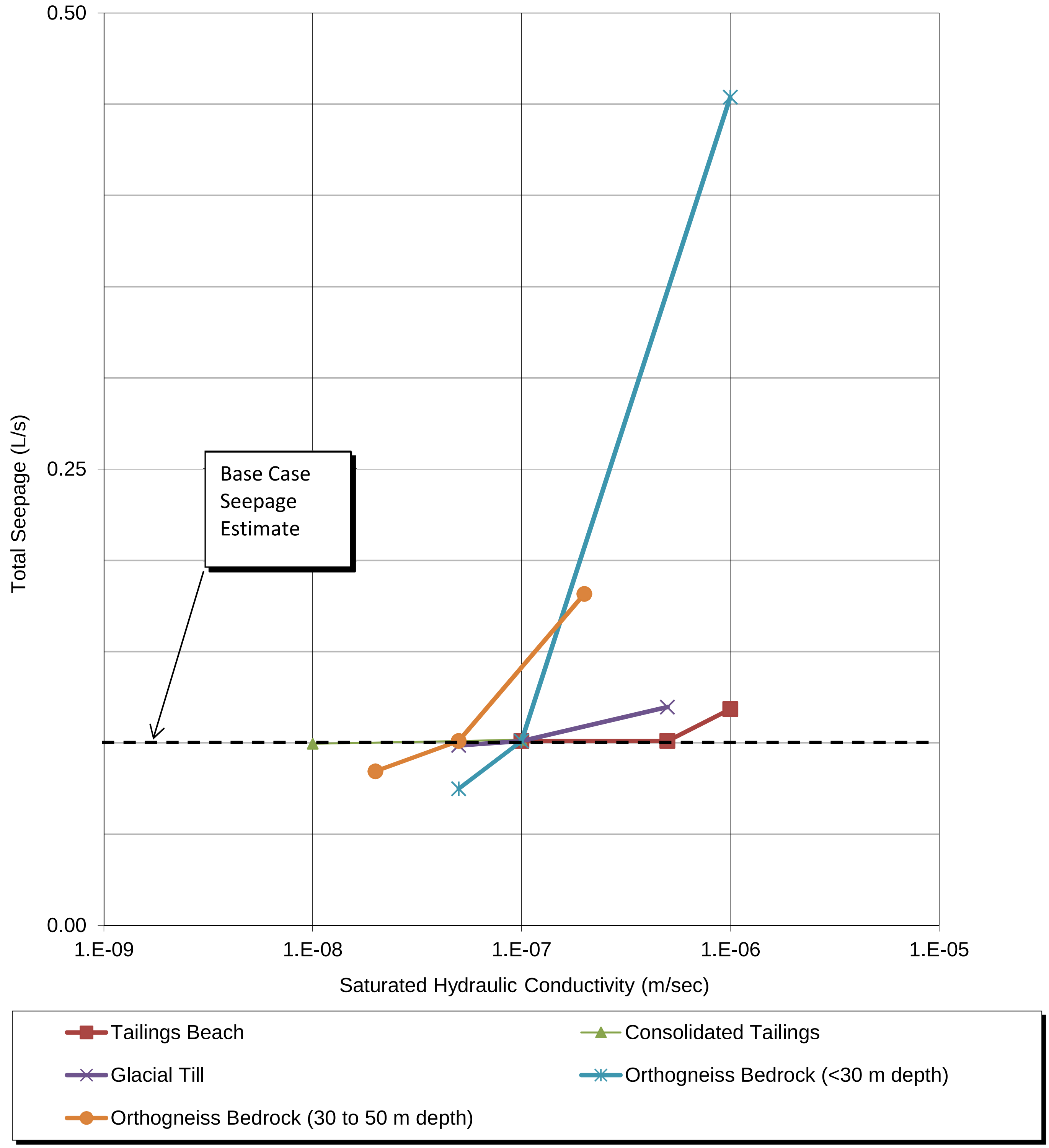
SECTIONS 1, 2 & 3 MAIN EMBANKMENT TMF SEEPAGE SENSITIVITY ANALYSIS UNRECOVERABLE SEEPAGE

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB

Knight Piésold CONSULTING

P/A NO.	REF. NO.
VA101-458/14	VA14-00865

FIGURE B-2 REV 0



NOTES:
1. EACH SERIES REPRESENTS THE SENSITIVITY ANALYSIS PERFORMED FOR THE MATERIAL IDENTIFIED IN THE SERIES TITLE, WITH ALL OTHER MATERIAL PARAMETERS AS PER THE BASE CASE PARAMETERS
2. THE SENSITIVITY OF THE CORE ZONE MATERIAL WAS NOT INVESTIGATED FOR THE NORTH EMBANKMENT SECTION

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

SECTION 6 - NORTH EMBANKMENT
TMF SEEPAGE SENSITIVITY ANALYSIS
TOTAL SEEPAGE

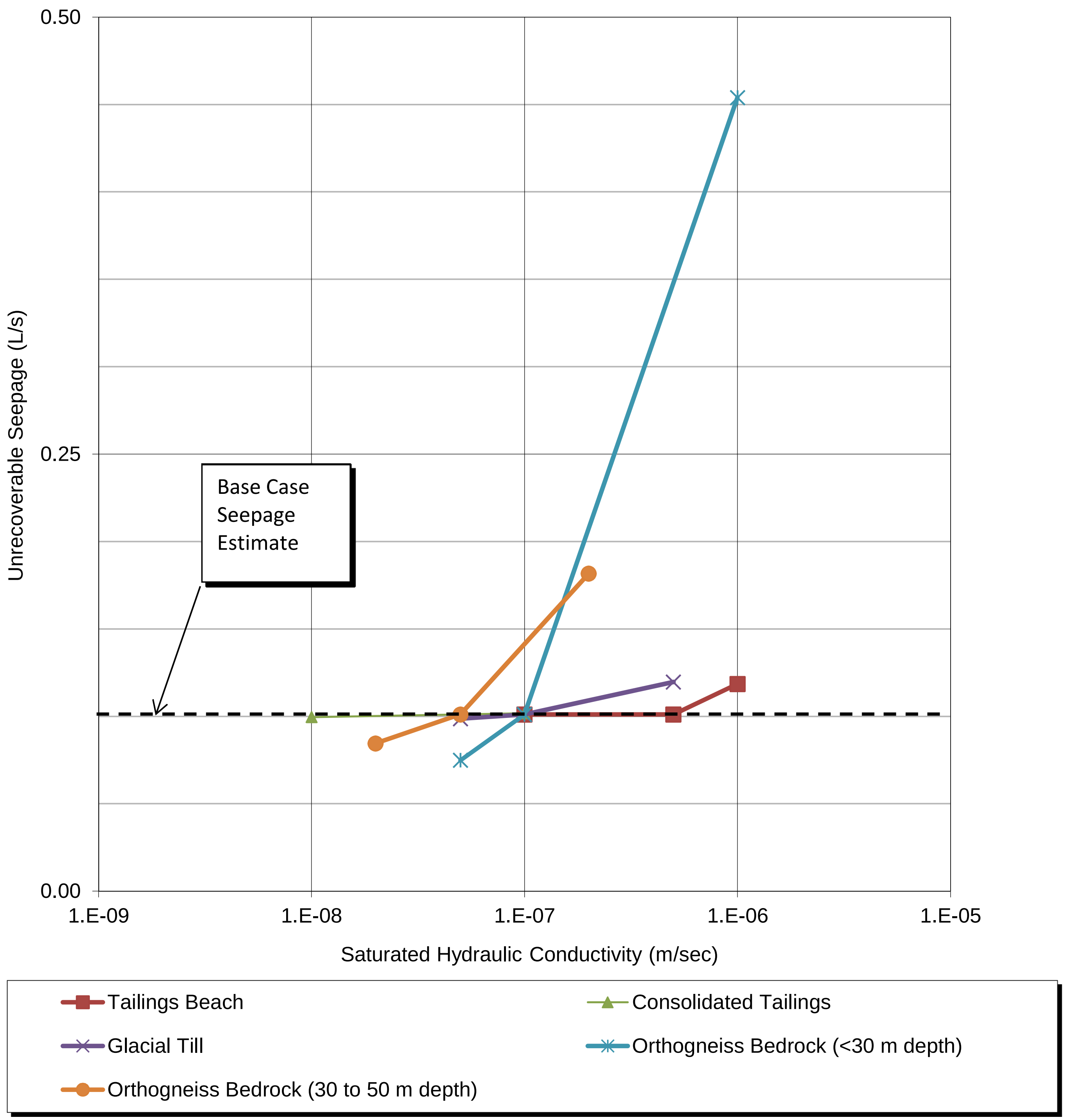
Knight Piésold CONSULTING

P/A NO. VA101-458/14

REF. NO. VA14-00865

FIGURE B-3

REV 0



NOTES:

1. EACH SERIES REPRESENTS THE SENSITIVITY ANALYSIS PERFORMED FOR THE MATERIAL IDENTIFIED IN THE SERIES TITLE, WITH ALL OTHER MATERIAL PARAMETERS AS PER THE BASE CASE PARAMETERS
2. THE SENSITIVITY OF THE CORE ZONE MATERIAL WAS NOT INVESTIGATED FOR THE NORTH EMBANKMENT SECTION
3. LOST SEEPAGE ESTIMATES EQUAL THE TOTAL SEEPAGE ESTIMATES AS THE PHREATIC SURFACE IS FOUND TO DROP BELOW THE EMBANKMENT CORE AND NO SEEPAGE IS RECOVERABLE

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

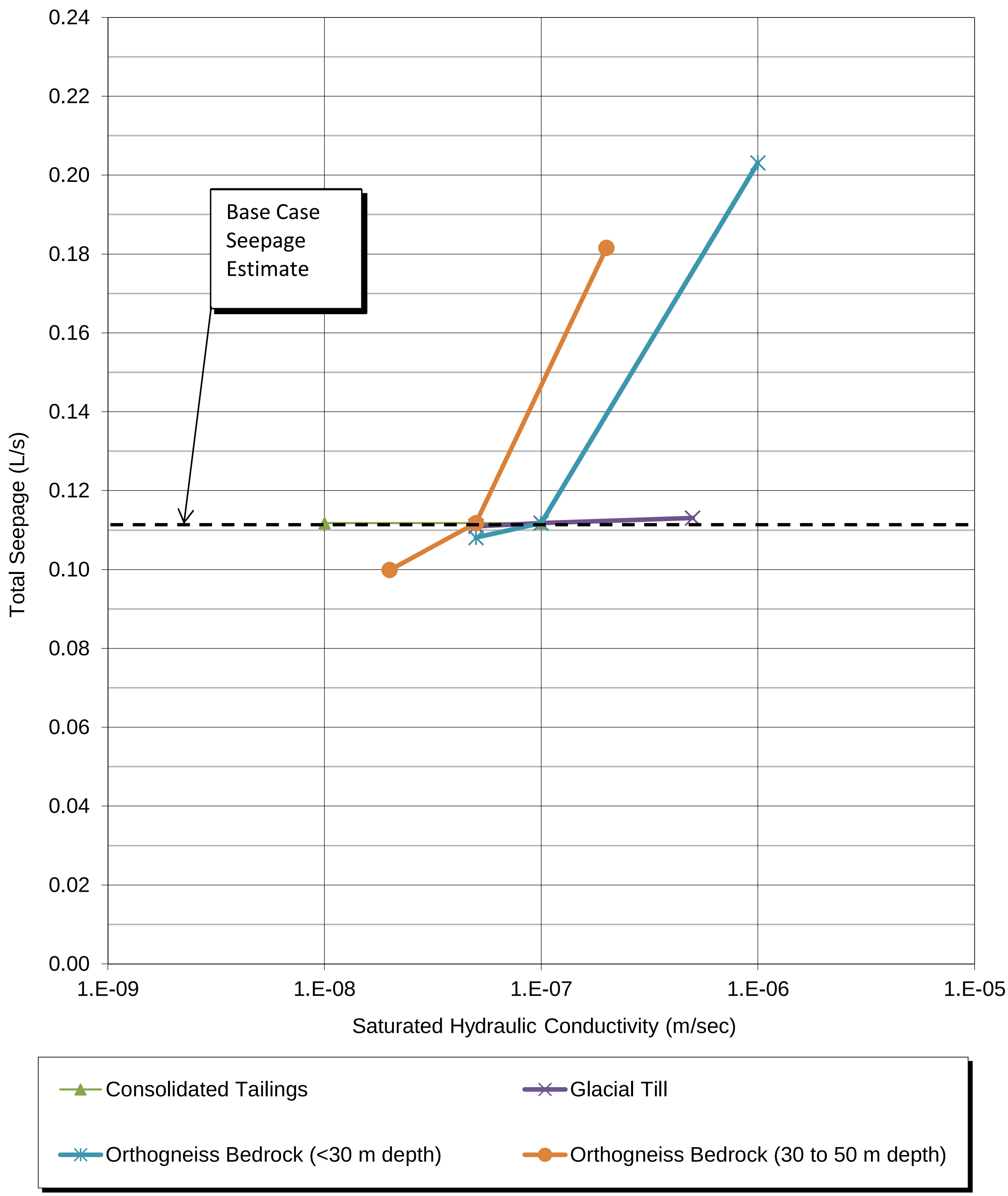
SECTION 6 - NORTH EMBANKMENT
TMF SEEPAGE SENSITIVITY ANALYSIS
UNRECOVERABLE SEEPAGE

Knight Piésold CONSULTING

P/A NO. VA101-458/14 | REF. NO. VA14-00865

FIGURE B-4 REV 0

0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D



NOTES:
1. EACH SERIES REPRESENTS THE SENSITIVITY ANALYSIS PERFORMED FOR THE MATERIAL IDENTIFIED IN THE SERIES TITLE, WITH ALL OTHER MATERIAL PARAMETERS AS PER THE BASE CASE PARAMETERS
2. ALL SEEPAGE IS ASSUMED TO BE UNRECOVERABLE

HARPER CREEK MINING CORP.

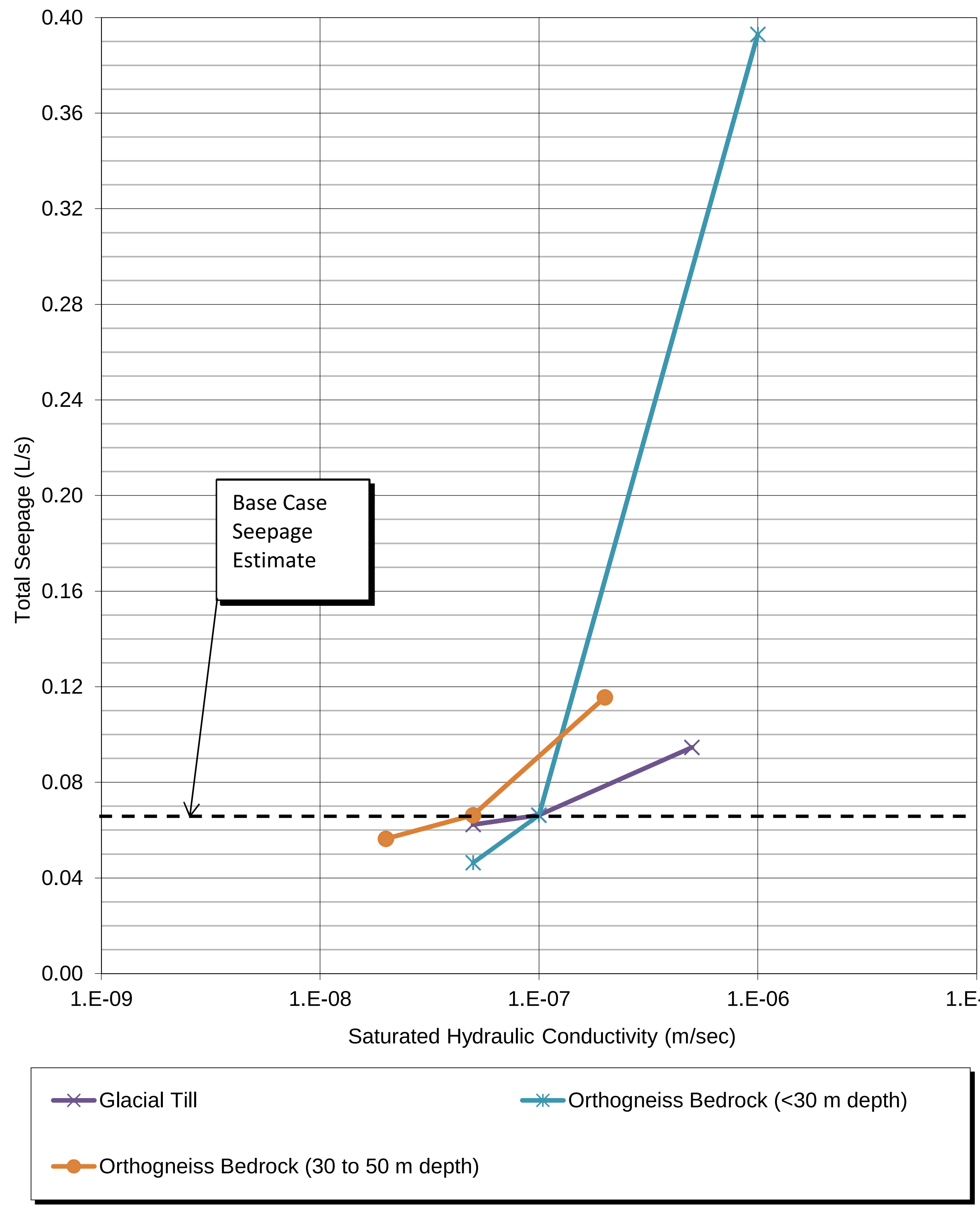
HARPER CREEK PROJECT

SECTION 4 - EAST SADDLE
TMF SEEPAGE SENSITIVITY ANALYSIS
TOTAL SEEPAGE

Knight Piésold CONSULTING

0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

P/A NO.	REF. NO.	
VA101-458/14	VA14-00865	
FIGURE B-5		REV 0



NOTES:

1. EACH SERIES REPRESENTS THE SENSITIVITY ANALYSIS PERFORMED FOR THE MATERIAL IDENTIFIED IN THE SERIES TITLE, WITH ALL OTHER MATERIAL PARAMETERS AS PER THE BASE CASE PARAMETERS
2. ALL SEEPAGE IS ASSUMED TO BE UNRECOVERABLE

0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

SECTION 5 - WEST SADDLE
TMF SEEPAGE SENSITIVITY ANALYSIS
TOTAL SEEPAGE

Knight Piésold CONSULTING

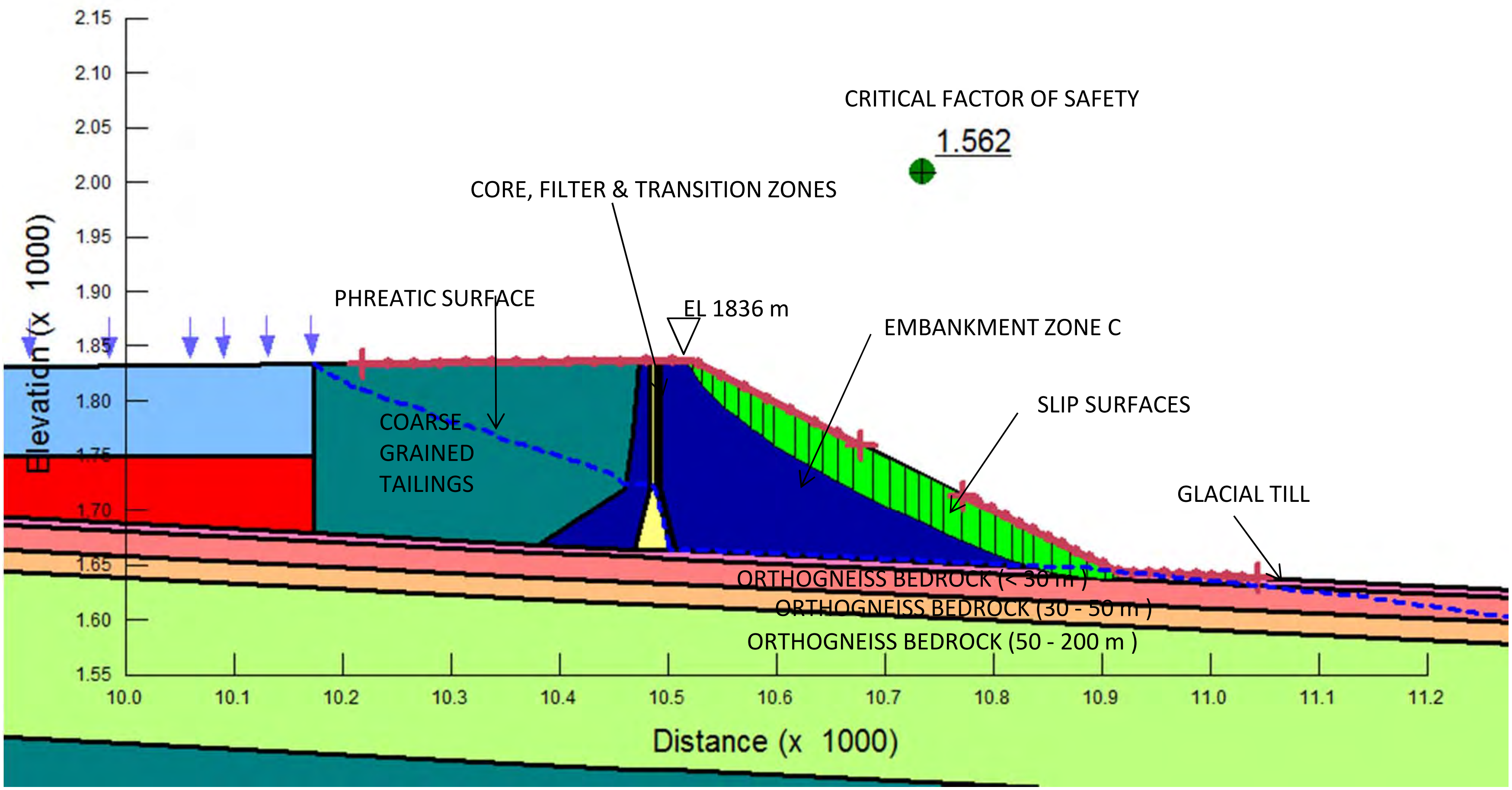
P/A NO.	REF. NO.
VA101-458/14	VA14-00865

FIGURE B-6 REV 0

APPENDIX C

STABILITY ANALYSIS FIGURES

(Figures C-1 to C-4)



NOTES:

1. LOCATION OF PHREATIC SURFACE PREDICTED FROM STEADY STATE SEEPAGE ANALYSIS.

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB

HARPER CREEK MINING CORP

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
MAIN EMBANKMENT STABILITY ANALYSIS
NORMAL OPERATING CONDITIONS AT

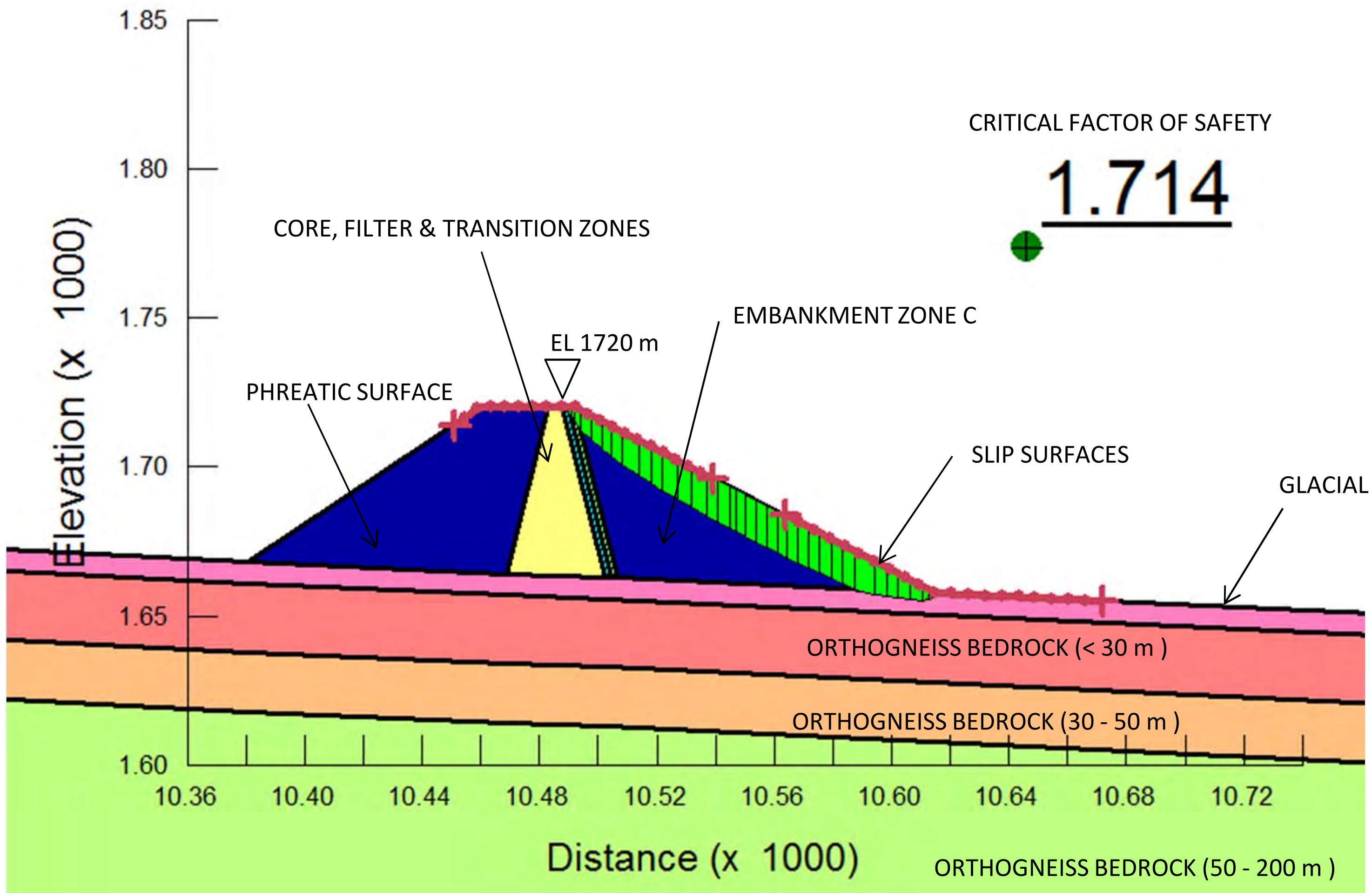
Knight Piésold CONSULTING

P/A. NO. VA101-458/14

REF NO. VA14-00865

FIGURE C-1

REV 0



NOTES:

1. LOCATION OF PHREATIC SURFACE ASSUMED TO BE AT NATURAL GROUND LEVEL

0	19AUG'14	ISSUED WITH LETTER VA14-00865	ACR	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

HARPER CREEK MINING CORP

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
MAIN EMBANKMENT STABILITY ANALYSIS
NORMAL OPERATING CONDITIONS AT STAGE

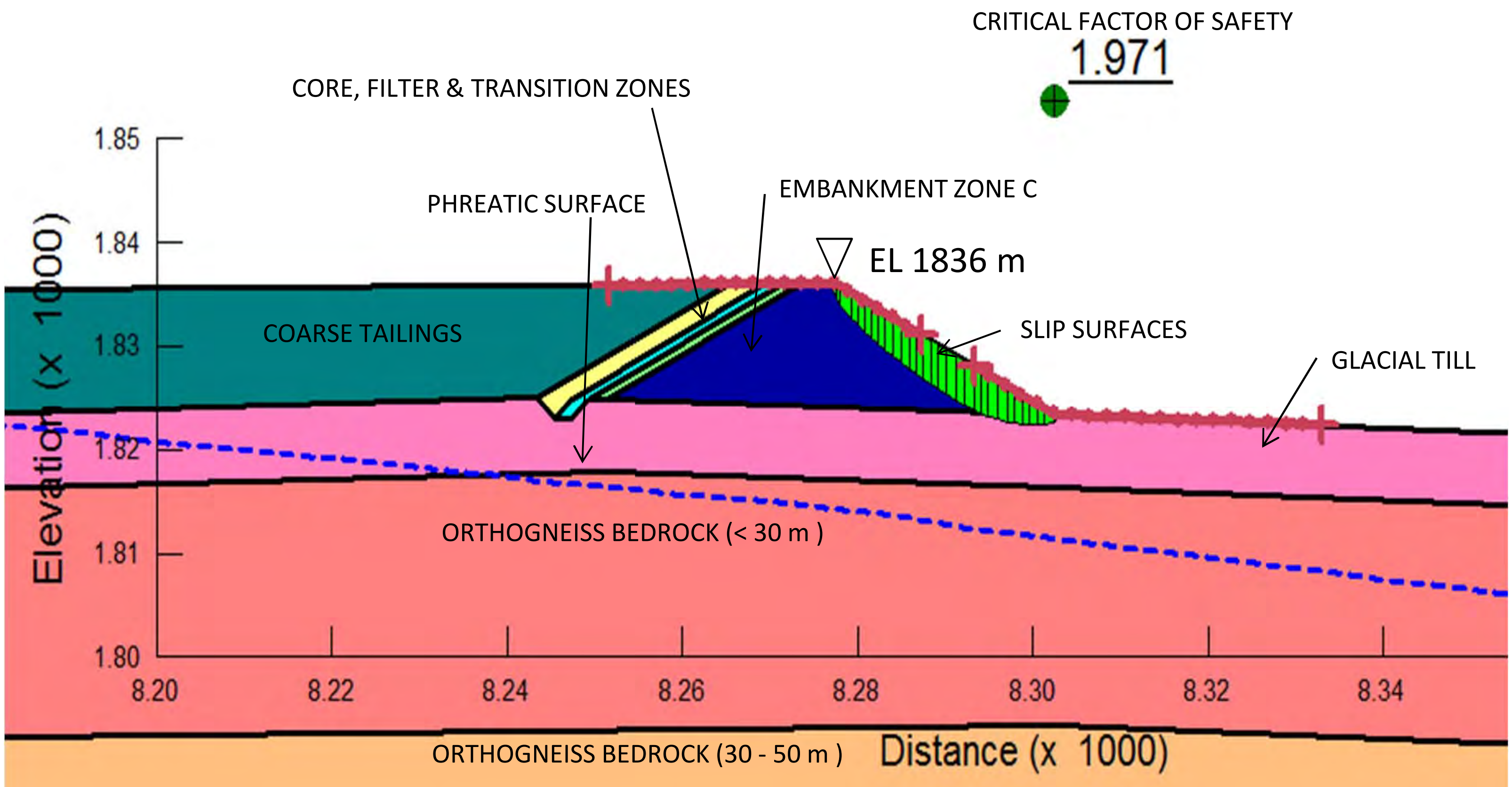
Knight Piésold CONSULTING

P/A. NO. VA101-458/14

REF NO. VA14-00865

FIGURE C-2

REV 0



NOTES:

1. LOCATION OF PHREATIC SURFACE PREDICTED FROM STEADY STATE SEEPAGE ANALYSIS

0	19AUG'14	ISSUED WITH VA14-00865	ACR	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

HARPER CREEK MINING CORP

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
NORTH EMBANKMENTSTABILITY ANALYSIS
NORMAL OPERATING CONDITIONS AT

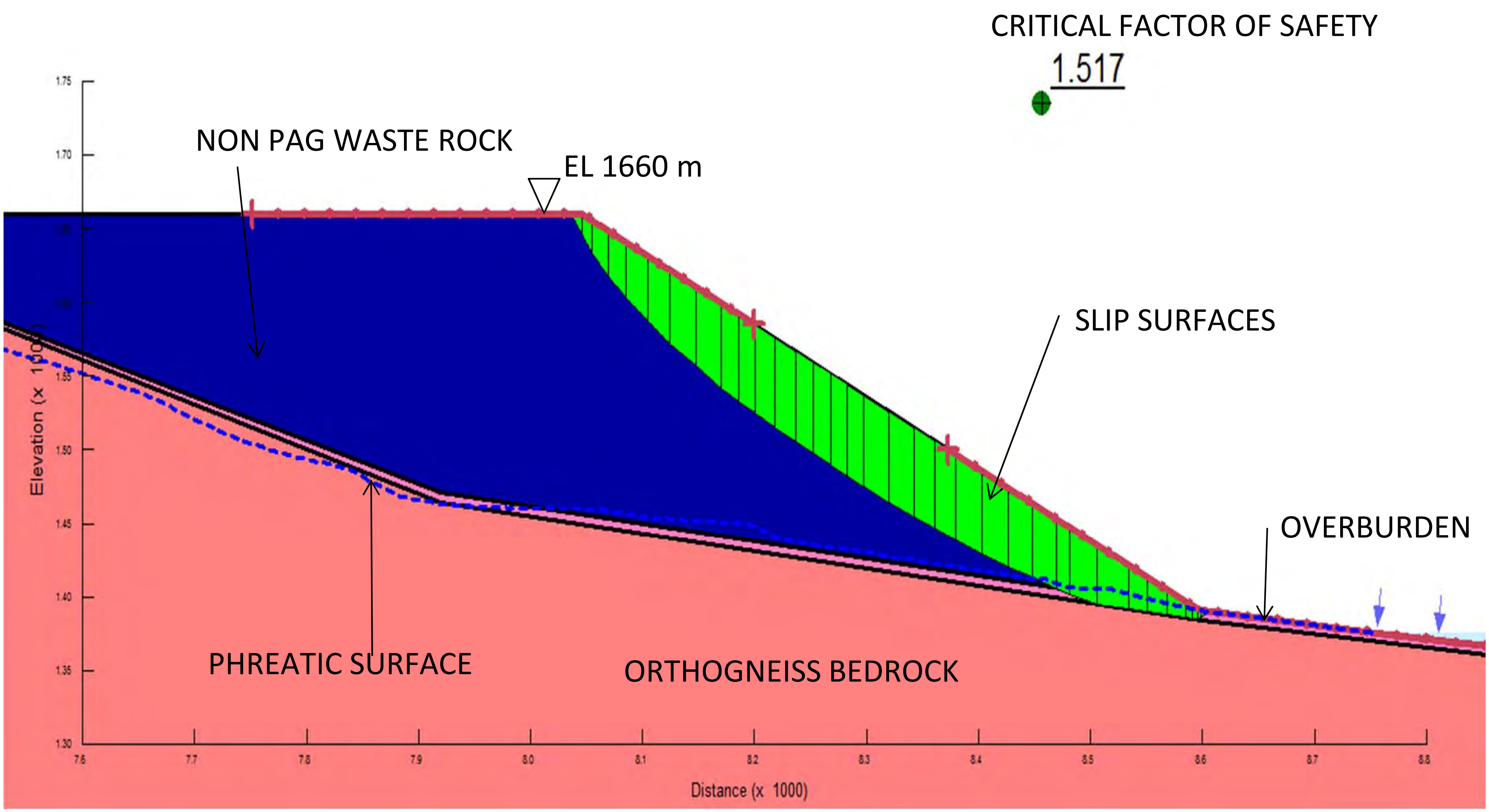
Knight Piésold CONSULTING

P/A. NO. VA101-458/14

REF NO. VA14-00865

FIGURE C-3

REV 0



NOTES:

1. LOCATION OF PHREATIC SURFACE PREDICTED FROM STEADY STATE SEEPAGE ANALYSIS

0	19AUG'14	ISSUED WITH VA14-00865	ACR	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

HARPER CREEK MINING CORP

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY
NON PAG WASTE STOCKPILE STABILITY
OPERATING CONDITIONS AT CLOSURE

Knight Piésold CONSULTING

P/A. NO. VA101-458/14

REF NO. VA14-00865

FIGURE C-4

REV 0

APPENDIX D

SELECTED TABLES FROM THE INVESTIGATION AND DESIGN MANUAL INTERIM GUIDELINES (BC MWRPRC, 1991)

(Pages D-1 to D-4)

TABLE 5.1
DUMP STABILITY RATING SCHEME

KEY FACTORS AFFECTING STABILITY	RANGE OF CONDITIONS OR DESCRIPTION		POINT RATING
DUMP CONFIGURATION		< 50m	0
		50m - 100m	50
DUMP HEIGHT		100m - 200m	100
		> 200m	200
DUMP VOLUME	Small	< 1 million BCM's	0
	Medium	1 - 50 million BCM's	50
	Large	> 50 million BCM's	100
DUMP SLOPE	Flat	< 26°	0
	Moderate	26° - 35°	50
	Steep	> 35°	100
FOUNDATION SLOPE	Flat	< 10°	0
	Moderate	10° - 25°	50
	Steep	25° - 32°	100
	Extreme	> 32°	200
DEGREE OF CONFINEMENT	Confined	-Concave slope in plan or section -Valley or Cross-Valley fill, toe butressed against opposite valley wall -Incised gullies which can be used to limit foundation slope during development	0
	Moderately Confined	-Natural benches or terraces on slope -Even slopes, limited natural topographic diversity -Heaped, Sidehill or broad Valley or Cross-Valley fills	50
	Unconfined	-Convex slope in plan or section -Sidehill or Ridge Crest fill with no toe confinement -No gullies or benches to assist development	100
FOUNDATION TYPE	Competent	-Foundation materials as strong or stronger than dump materials -Not subject to adverse pore pressures -No adverse geologic structure	0
	Intermediate	-Intermediate between competent and weak -Soils gain strength with consolidation -Adverse pore pressures dissipate if loading rate controlled	100
	Weak	-Limited bearing capacity, soft soils -Subject to adverse pore pressure generation upon loading -Adverse groundwater conditions, springs or seeps -Strength sensitive to shear strain, potentially liquefiable	200
DUMP MATERIAL QUALITY	High	-Strong, durable -Less than about 10% fines	0
	Moderate	-Moderately strong, variable durability -10 to 25% fines	100
	Poor	-Predominantly weak rocks of low durability -Greater than about 25% fines, overburden	200

Continued..

TABLE 5.1 (Continued)

DUMP STABILITY RATING SCHEME

KEY FACTORS AFFECTING STABILITY	RANGE OF CONDITIONS OR DESCRIPTION		POINT RATING
METHOD OF CONSTRUCTION	Favourable	-Thin lifts (<25m thick), wide platforms -Dumping along contours -Ascending construction -Wrap-arounds or terraces	0
	Mixed	-Moderately thick lifts (25m - 50m) -Mixed construction methods	100
	Unfavourable	-Thick lifts (> 50m), narrow platform (sliver fill) -Dumping down the fall line of the slope -Descending construction	200
PIEZOMETRIC AND CLIMATIC CONDITIONS	Favourable	-Low piezometric pressures, no seepage in foundation -Development of phreatic surface within dump unlikely -Limited precipitation -Minimal infiltration into dump -No snow or ice layers in dump or foundation	0
	Intermediate	-Moderate piezometric pressures, some seeps in foundation -Limited development of phreatic surface in dump possible -Moderate precipitation -High infiltration into dump -Discontinuous snow or ice lenses or layers in dump	100
	Unfavourable	-High piezometric pressures, springs in foundation -High precipitation -Significant potential for development of phreatic surface or perched water tables in dump -Continuous layers or lenses of snow or ice in dump or foundation	200
DUMPING RATE	Slow	-< 25 BCM's per lineal metre of crest per day -Crest advancement rate < 0.1m per day	0
	Moderate	-25 - 200 BCM's per lineal metre of crest per day -Crest advancement rate 0.1m - 1.0m per day	100
	High	-> 200 BCM's per lineal metre of crest per day -Crest advancement > 1.0m per day	200
SEISMICITY	Low	Seismic Risk Zones 0 and 1	0
	Moderate	Seismic Risk Zones 2 and 3	50
	High	Seismic Risk Zones 4 or higher	100

MAXIMUM POSSIBLE DUMP STABILITY RATING: 1800

TABLE 5.2

DUMP STABILITY CLASSES AND RECOMMENDED LEVEL OF EFFORT

DUMP STABILITY CLASS	FAILURE HAZARD	RECOMMENDED LEVEL OF EFFORT FOR INVESTIGATION, DESIGN AND CONSTRUCTION	RANGE OF DUMP RATING (DSR)
I	Negligible	-Basic site reconnaissance, baseline documentation -Minimal lab testing -Routine check of stability, possibly using charts -Minimal restrictions on construction -Visual monitoring only	< 300
II	Low	-Thorough site investigation -Test pits, sampling may be required -Limited lab index testing -Stability may or may not influence design -Basic stability analysis required -Limited restrictions on construction -Routine visual and instrument monitoring	300-600
III	Moderate	-Detailed, phased site investigation -Test pits required, drilling or other subsurface investigations may be required -Undisturbed samples may be required -Detailed lab testing, including index properties, shear strength and durability likely required -Stability influences and may control design -Detailed stability analysis, possibly including parametric studies, required -Stage II detailed design report may be required for approval/permitting -Moderate restrictions on construction (eg. limiting loading rate, lift thickness, material quality, etc.) -Detailed instrument monitoring to confirm design, document behaviour and establish loading limits	600-1200
IV	High	-Detailed, phased site investigation -Test pits, and possibly trenches, required -Drilling, and possible other subsurface investigations probably required -Undisturbed sampling probably required -Detailed lab testing, including index properties, shear strength and durability testing probably required -Stability considerations paramount. -Detailed stability analyses, probably including parametric studies and full evaluation of alternatives probably required -Stage II detailed design report probably required for approval/permitting -Severe restrictions on construction (eg. limiting loading rates, lift thickness, material quality, etc.) -Detailed instrument monitoring to confirm design, document behaviour and establish loading limits	> 1200

TABLE 6.4

INTERIM GUIDELINES FOR MINIMUM DESIGN FACTOR OF SAFETY [1]

STABILITY CONDITION	SUGGESTED MINIMUM DESIGN VALUES FOR FACTOR OF SAFETY	
	CASE A	CASE B
STABILITY OF DUMP SURFACE		
–Short Term (during construction)	1.0	1.0
–Long Term (reclamation – abandonment)	1.2	1.1
OVERALL STABILITY (DEEP SEATED STABILITY)		
–Short Term (static)	1.3 – 1.5	1.1 – 1.3
–Long Term (static)	1.5	1.3
–Pseudo–Static (earthquake) [2]	1.1 – 1.3	1.0

CASE A:
- –Low level of confidence in critical analysis parameters
- –Possibly unconservative interpretation of conditions, assumptions
- –Severe consequences of failure
- –Simplified stability analysis method (charts, simplified method of slices)
- –Stability analysis method poorly simulates physical conditions
- –Poor understanding of potential failure mechanism(s)

CASE B:
- –High level of confidence in critical analysis parameters
- –Conservative interpretation of conditions, assumptions
- –Minimal consequences of failure
- –Rigorous stability analysis method
- –Stability analysis method simulates physical conditions well
- –High level of confidence in critical failure mechanism(s)

NOTES:
1. A range of suggested minimum design values are given to reflect different levels of confidence in understanding site conditions, material parameters, consequences of instability, and other factors.
2. Where pseudo–static analyses, based on peak ground accelerations which have a 10% probability of exceedance in 50 years, yield F.O.S. < 1.0, dynamic analysis of stress–strain response, and comparison of results with stress–strain characteristics of dump materials is recommended.

APPENDIX E

MINE SITE WATER BALANCE MODEL

(Pages E-1 to E-12)

July 24, 2014

File No.:VA101-458/14-A.01
Cont. No.:VA14-00700

Mr. Alastair Tiver
Feasibility Study Director
Harper Creek Mining Corp
Suite 1800, Two Bentall Centre
555 Burrard Street
Vancouver, BC V7X 1M9

Dear Alastair

Re: Harper Creek Project – Updated Feasibility Study Water Balance Model

1 – GENERAL

A monthly operational and closure water balance was developed by Knight Piésold Ltd. (KP) for the Harper Creek Project using the GoldSim© software package. This letter provides results for the updated monthly water balance and the updated model reflects the most up to date mine waste management concepts and water management routing assumptions (KP, 2014).

The intent of the modelling was to estimate the magnitude and extent of any water surplus and/or deficit conditions in the tailings management facility (TMF) based on a range of possible climatic conditions. The modelling timeline included:

- One year of pre-production (Year -1)
- 28 years of operations (Year 1 to 28) at a nominal milling rate of 70,000 dry metric tonnes per day, and
- 17 years of closure.

The model incorporates the following major project components:

- Open Pit
- Mill
- Tailings Management Facility (TMF)
- Non-Potentially Acid Generating (non-PAG) Waste Rock Stockpiles
- Potentially Acid Generating (PAG) Waste Rock Stockpile to be stored within the TMF, and
- Non-PAG and PAG Low-Grade Ore (LGO) Stockpiles.

The water balance model is illustrated schematically on Figure 1 and the key model assumptions are summarized below in Table 1.

Table 1 Water Balance Input Parameters

Component	Assumption
Total Tailings Production (million tonnes)	718
Total Tailings stored in TMF (Years 1 to 23) (million tonnes)	585
Total Tailings stored in Open pit (Years 24 to 28) (million tonnes)	133
Waste Rock (million tonnes stored in TMF Years 1 to 25)	237
Mine Life (years)	28
Tailings slurry solids content (% by weight)	34.5%
Tailings dry density (tonnes/m^3)	1.3
Bulk tailings specific gravity	2.66
Waste Rock dry density (tonnes/m^3)	2.2
Waste Rock specific gravity	2.7
TMF total embankment seepage (total) (L/s) – Year 1	0
TMF total embankment seepage (total) (L/s) – Year 28	15
Open Pit Groundwater inflows (L/s) - Year 1	0
Open Pit Groundwater inflows (L/s) - Year 23	21
Open Pit Groundwater inflows (L/s) - Year 38	2.5
TMF tailings consolidation seepage (L/s) – Year 1	20
TMF tailings consolidation seepage (L/s) – Year 23	82
TMF tailings consolidation seepage (L/s) – Year 24	45
TMF tailings consolidation seepage (L/s) – Year 53	10
TMF tailings consolidation seepage (L/s) – Year 123	0

NOTES:

1. THE OPEN PIT GROUNDWATER INFLOWS WERE ASSUMED TO INCREASE LINEARLY FROM 0 L/S AT THE BEGINNING OF YEAR 1 TO A MAXIMUM OF 21 L/S AT THE END OF YEAR 23. ONCE THE OPEN PIT IS FULL, THE GROUNDWATER INFLOWS ARE ASSUMED TO BE AT A CONSTANT OF 2.5 L/S.
2. THE TAILINGS CONSOLIDATION SEEPAGE IS ASSUMED TO CONTRIBUTE THE TMF SUPERNATANT POND VOLUME UNTIL 100 YEARS (YEAR 123) AFTER THE END OF TAILNGS DEPOSITION IN THE TMF (YEAR 23).
3. THE TMF EMBANKMENT SEEPAGE (FROM BOTH THE MAIN AND NORTH EMBANKMENTS) IS ASSUMED TO INCREASE LINEARLY FROM 0 L/S AT THE BEGINNING OF YEAR 1 TO A MAXIMUM OF 15 L/S AT THE END OF YEAR 28.

2 – OVERVIEW OF SITE WATER MANAGEMENT

A schematic illustration of the components of the water balance model for the Harper Creek Project is shown on Figure 1. The water management plan for the project is summarized below.

2.1 SITE WATER MANAGEMENT: START-UP AND OPERATIONS DURING OPEN PIT MINING

The water management plan for Years 1 to 23 of operations is summarized below:

- The open pit will be mined and tailings will be stored in the TMF during the first 23 years of the mine life.
- All runoff from the open pit walls and upslope catchment areas will be collected by the open pit dewatering system and pumped to the TMF pond.

- The TMF will be the primary source of water to the mill for the first 25.5 years of operations. The TMF pond is assumed to collect runoff for one year prior to mill start-up.
- TMF embankment seepage and runoff will be collected in water management ponds situated at low points downstream of the Main and North embankments. The recycled seepage will be pumped back to the TMF during operations.
- Seepage and runoff from the non-PAG waste rock stockpile, PAG LGO stockpile and the non-PAG LGO stockpile outside the TMF will be collected in water management ponds and pumped to the TMF pond throughout operations.
- Seepage and surface runoff from the overburden stockpile adjacent to the open pit will be directed to the open pit until Year 10 from where it will be pumped to the TMF by the open pit dewatering system. After Year 10, the collected runoff seepage and runoff will be routed through a sediment pond and discharged to the receiving environment.
- The current mine plan includes approximately 237 million tonnes of PAG waste rock over the course of the mine life from Years 1 to 25. The PAG will be deposited in the footprint of the TMF. The PAG will be inundated by water and tailings as the TMF rises over the mine life.

2.2 SITE WATER MANAGEMENT: OPERATIONS DURING LGO PROCESSING

The water management plan for Years 24 to 28 of operations is summarized below:

- Starting in Year 24, LGO will be processed through the mill and tailings will be deposited in the open pit until the end of operations in Year 28.
- The open pit dewatering system will be decommissioned once the LGO tailings deposition in the open pit commences.
- Reclaim water from the TMF will continue to supply the mill for the first 18 months of the LGO processing (Years 24 to 25.5), while the open pit is filling. Starting in Year 25.5, the open pit will be the primary source of water to supply the mill until the end of operations in Year 28.
- Seepage and runoff from waste rock stockpiles (outside of the TMF) will continue to be pumped to the TMF.
- The PAG waste rock stored in the TMF is assumed to be completely encapsulated within the tailings and TMF supernatant pond at the end of operations.

2.3 SITE WATER MANAGEMENT: CLOSURE AND POST-CLOSURE

Closure commences at the end of operations, once the mill operations cease. Post-closure starts once the open pit is full and water is being pumped to the TMF. The water management plan for closure and post-closure is summarized below:

- The mill will be decommissioned and LGO stockpile footprints will be revegetated where practical.
- The open pit will be allowed to fill naturally to elevation 1530 m. Once full, any surplus water will be pumped to the TMF for long-term storage.
- All tailings distribution pipeworks and the water reclaim pump and pipeline will be removed from the TMF.
- A permanent spillway channel will be excavated at the southeastern end of the TMF. The TMF pond will be allowed to fill and spill to the downstream receiving environment.
- The seepage recycle pumping system from the water management ponds to the TMF, will be decommissioned approximately 12 years after the end of operations.
- Seepage and runoff from the waste rock stockpiles outside of the TMF will continue to be pumped to the TMF for long-term storage.

3 – WATER BALANCE MODEL ASSUMPTIONS

3.1 AVERAGE HYDROMETEOROLOGICAL CONDITIONS

The hydrometeorological inputs to the water balance model were based on the baseline watershed model (KP, 2014), which uses long-term data series for both temperature and precipitation. The 96 year long-term data series for the project was developed by correlating the concurrent climate record from the regional climate station at Vavenby operated by Environment Canada (EC) with available measured project site data. Details of the development of the precipitation and temperature record for the project site are included in the Watershed Modelling Report (KP, 2014).

The baseline watershed model was developed separately from the operational water balance to assess the baseline surface and groundwater flow patterns in the project area. The baseline watershed model was calibrated by translating inputs of regional long-term precipitation into corresponding streamflow values for the project area. The hydrologic inputs were adjusted until best fits were reached between calculated and reliable measured site streamflow values.

The mean annual precipitation (MAP) for the project area was estimated to be 1264 mm, at a reference elevation of 1800 m, with 32% of the annual precipitation falling as rain and the remainder as snow. The mean monthly values for precipitation, rainfall, snowfall and the resulting surplus water volumes are summarized in Table 2.

Table 2 Average Hydrometeorological Inputs

Parameter	Monthly Value (mm)												Annual
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	(mm)
Precipitation	189	103	97	51	84	76	66	61	53	117	170	197	1264
Rainfall	0	0	0	30	84	76	66	61	53	35	0	0	405
Snowfall	189	103	97	21	0	0	0	0	0	82	170	197	859
Sublimation	11	10	11	11	11	11	11	0	0	11	11	11	109
Snowmelt	0	0	0	0	200	364	186	0	0	0	0	0	750
Available Precipitation	0	0	0	30	284	440	252	61	53	35	0	0	1155
Lake Evaporation	0	0	0	11	60	86	103	88	49	0	0	0	397
Undisturbed Surplus Water	0	0	0	21	232	367	165	0	0	35	0	0	820

NOTES:
1. THE PRECIPITATION VALUES WERE ESTIMATED FOR THE PROJECT SITE REFERENCE ELEVATION OF 1800 m.
2. THE LAKE EVAPORATION VALUES WERE APPLIED TO OPEN WATER SURFACES TO ESTIMATE EVAPORATIVE LOSSES.
3. AVAILABLE SURPLUS WATER VALUES WERE APPLIED TO UNDISTURBED AREAS WITHIN THE MINE FOOTPRINT TO ESTIMATE RUNOFF.

The available precipitation and surplus water values shown in Table 2 were estimated based on results of the baseline watershed model. The available precipitation values are rainfall plus snowmelt minus sublimation losses. The surplus water values represent the water available for runoff or groundwater recharge, once evapotranspiration and soil moisture losses have been removed for natural (undisturbed) catchment areas.

Available precipitation (mm) = rainfall + snowfall - sublimation

Surplus water for undisturbed areas (mm) = available precipitation - actual evapotranspiration - soil moisture losses

3.2 EVAPOTRANSPIRATION

Potential evapotranspiration (PET) is defined as the amount of evapotranspiration that would occur from a full vegetation cover given an infinite supply of water (ideal conditions); these values are believed to reasonably represent lake evaporation conditions (Ponce, 1989 and Maidment, 1993). The estimated annual average lake evaporation for the project site is 397 mm, which was applied to open water surfaces in the project area (e.g. the TMF pond area and pit lake in closure). The monthly evaporation values are summarized in Table 2.

Site-specific evaporation data are not available for the project area. Accordingly, the PET for the project site was estimated based on the empirical Thornthwaite equation (Thornthwaite, 1948) that requires monthly temperature as an input:

$$ET_0 = \begin{cases} 0, T < 0 \deg C \\ 16\left(\frac{10T_i}{I}\right)^a, 0 \le T \le 26.5 \deg C \\ -415.85 + 32.24T_i - 0.43T_i^2, T \ge 26.5 \deg C \end{cases}$$

Where:

PET_0 = Potential evapotranspiration (mm/month)

T_i = Mean monthly temperature (°C)

I = Heat index, sum of 12 monthly index values (i)

i = Monthly heat index

a = Empirically derived exponent, which is a function of I

And:

$$i = \left(\frac{T}{5}\right)^{1.514}$$

$$a = 6.75 * 10^{-7} I^3 - 7.71 * 10^{-5} I^2 + 1.79 * 10^{-2} I + 0.49$$

Actual evapotranspiration (AET) from undisturbed catchment areas and bare rock surfaces (e.g. TMF embankments, open pit walls, waste rock piles and low grade stockpiles) in the model are assumed to be limited by the availability of water. Therefore evapotranspiration loss for these surfaces was estimated by applying a reduction factor to the calculated PET values to account for non-ideal conditions for evapotranspiration. The AET factors for undisturbed and bare rock areas were assumed to be 0.85 and 0.7, respectively, based on the calibrated baseline watershed model (KP, 2014).

3.3 DISTURBED FOOTPRINT AREA RUNOFF

The surplus water values for the disturbed areas within the mine footprint were estimated on the basis of rainfall and snowmelt estimates (available precipitation) minus the estimated corresponding actual evapotranspiration and soil moisture losses. For bare rock surfaces the soil moisture losses were considered to be negligible. A runoff coefficient was then applied to estimate the runoff component of the surplus water to account for the water lost to groundwater recharge. Generally, for all modelled areas in the water balance (undisturbed and disturbed), the model tracks only the surface water component (runoff) of the surplus water and assumes that any water that goes to ground (groundwater recharge) is lost from the system. Some exceptions include the groundwater recharge for the PAG waste rock stored in the TMF and the undisturbed area within the TMF catchment. For these areas, the groundwater recharge component is assumed to be captured and stored in the TMF itself and therefore the corresponding runoff coefficients are assumed to be 1.0.

Runoff from disturbed areas (mm) = (available precipitation - actual evapotranspiration) x runoff coefficient

The assumed runoff coefficients for the mine site areas are:

- PAG waste rock stored in TMF (exposed): 1.0
- Undisturbed TMF area: 1.0
- Non-PAG LGO stockpile: 0.85
- PAG LGO stockpile: 0.85
- Non-PAG waste rock stockpile: 0.75
- Overburden stockpile: 0.70
- Open pit walls: 0.65
- TMF embankments: 0.75
- Undisturbed area contributing to TMF seepage pond: 0.75

3.4 CLIMATIC VARIABILITY

The potential variability of climate conditions over the project life was addressed by systematically varying climatic inputs to the water balance based on the 96 year historical precipitation and temperature record developed for the project site (KP, 2014). The model was run with 96 iterations for each year of simulated mine life, enabling a large number of combinations of resulting wet, dry, and median months and years of precipitation and corresponding temperature values to be considered. Additionally, this approach maintains the inherent cyclical nature of the climate record. Model outputs, in particular flow volumes, were then compiled as distributions for each month in each year from which probabilities of occurrence could be determined. The probabilities of occurrence presented in the water balance results represent the following conditions:

- Median scenario – 50% chance of the value being equaled or exceeded in any given month or year
- 95th percentile scenario –5% chance that the water volume or flow rate will be equaled or exceeded in any given month or year (also referred to as the 95th percentile wet), and
- 5th percentile scenario – 95% chance that a water volume or flow will be equaled or exceeded in any given month or year (also referred to as the 95th percentile dry).

3.5 TMF EMBANKMENT SEEPAGE AND RECYCLE

Steady-state seepage analyses were completed using the finite element computer program SEEP/W to estimate the amount of seepage through the embankments. The total embankment seepage was estimated to be approximately 15 L/s in Year 28, with 98% being lost through the Main Embankment and the remaining 2% through the North Embankment. It was assumed that approximately 12.5 L/s (85%) of seepage can be captured by the Main Embankment seepage collection system. Therefore, a maximum of approximately 2.5 L/s (15%) of total seepage is assumed to bypass the seepage collection system to the environment downstream of the Main and North Embankments. Recycle from the Main Embankment seepage pond is assumed to continue until Year 40.

3.6 GROUNDWATER INFLOW TO OPEN PIT AND PIT DEWATERING SYSTEM

The total groundwater inflows to the open pit were estimated to be approximately 21 L/s by Year 23. The water pumped from the open pit by the dewatering system includes groundwater inflows, pit wall runoff, and undisturbed pit catchment runoff. Water from the open pit is assumed to be sent to the TMF until the end of Year 23, at which time the dewatering system will be decommissioned temporarily for 1.5 years when tailings deposition to the open pit commences. The dewatering system will be used to supply process water to the mill starting in Year 25.5.

The open pit is assumed to fill naturally from pit wall runoff, direct precipitation on the pit lake surface, and groundwater inflows starting in Year 29, once mill operations and tailings deposition to the open pit has ceased. The groundwater inflows are assumed to decrease as the open pit fills from a maximum of 21 L/s to 2.5 L/s once the open pit lake is full (at elevation 1530 m). Once full, the open pit surplus is assumed to be pumped to the TMF pond for long-term storage.

3.7 WATER RETAINED IN TAILINGS AND WASTE ROCK VOIDS

The amount of water retained in the tailings voids is a function of the mine production schedule, and the dry density and specific gravity of the tailings, as summarized in Table 1. The PAG waste rock stored in the TMF will also retain water in its void spaces as it becomes inundated.

3.8 PROCESS WATER REQUIREMENTS

The amount of water required for ore processing at the mill was based on the mine production schedule and average mill throughput. The modelled mine production rate is 70,000 tpd for 28 years of the mine life. The expected solids content of the tailings slurry is 34.5% by weight. The volume of water available for reclaim to the mill was estimated using the TMF (Years 1 to 25.5) and open pit (Years 25.5 to 28) water balances. Process water will be supplied by the TMF reclaim system to the mill from Years 1 to 23 while tailings are being deposited in the TMF, and for an additional 1.5 years (Years 24 to 25.5) once tailings from LGO processing are being deposited in the open pit. The mill process water requirements will be supplied by the open pit dewatering system from Year 25.5 until the end of Year 28.

The primary TMF inflows are:

- Water in the tailings slurry (Years 1 to 23 only),
- Direct precipitation and runoff to the TMF, which includes runoff from the upslope catchments, and
- Runoff pumped directly to the TMF from the Non-PAG waste rock and LGO stockpiles and exposed PAG waste rock in the TMF.

The primary TMF water losses are:

- Water retained in the tailings voids,
- Water retained in the PAG waste rock voids,
- Evaporation, and
- Unrecoverable seepage.

The primary open pit inflows are:

- Water in the tailings slurry (Years 25.5 to 28),
- Direct precipitation and runoff to the open pit, which includes runoff from the upslope catchments, and
- Groundwater inflows to the open pit.

The primary open pit water losses are:

- Water retained in the tailings voids, and
- Pit lake evaporation.

The water available for process use was assumed to be the difference between these inflows and losses. Any shortfall in the water available for milling will be made up from an external source.

4 – RESULTS

4.1 OPERATIONS

The water balance model results were used to estimate the likelihood of having a water surplus or deficit in the TMF. The TMF pond is predicted to be in a net surplus condition for the entire operating life of the mine, indicating that the system (including the TMF and contributing catchment) is able to supply more than enough water to meet the mill process water requirements. The TMF pond volume throughout operations (Years 1 to 28) is shown on Figure 2.

4.2 CLOSURE

Mining of the open pit will be complete at the end of Year 23, at which time the LGO will be processed through the mill and tailings will be deposited in the open pit until the end of operations in Year 28. Figure 3 illustrates the water accumulated in the open pit, on a monthly basis, as of Year 24 onwards. The initial water volume in the open pit is from the tailings slurry and water trapped in the tailings void spaces (Years 24 to 25.5), when the TMF reclaim system is still in operations. Reclaim water is then supplied to the mill from the open pit, which draws down the open pit water volume from Years 25.5 to 28. The open pit begins to fill naturally to elevation 1530 m starting in Year 29.

The model shows that under the median condition the pit will require 1.5 years to reach its maximum pond capacity of 37 Mm^3. The total pit volume is approximately 139 Mm^3, which includes 102 Mm^3 of stored tailings.

5 – CONCLUSIONS

The results of the monthly water balance model indicate that:

- The TMF pond is predicted to be in a surplus condition throughout operations and is able to supply all the process water required to support mill processing from Years 1 to 25.5. As of Year 25.5, when LGO is processed through the mill, the open pit is able to supply all the process water required for the mill to the end of operations in Year 28.
- The TMF pond ranges from a minimum of 12 Mm^3 at start-up to a maximum of 196 Mm^3 at the end of operations, under median conditions.
- The open pit is predicted to be full as of Year 30 under median conditions, approximately 1.5 years after the end of operations. The excess water from the open pit will be pumped to the TMF for long-term storage.

6 – REFERENCES

Knight Piesold (KP) (2014). *Harper Creek Mining Corp., Harper Creek Project: Watershed Modelling* (Ref. no. VA101-458/14-1, Rev A) July 22, 2014.

Maidment, D.R. (1993). Handbook of Hydrology. McGraw-Hill Inc., Washington, DC, USA.

Ponce, M.P. (1989). Engineering Hydrology – Principles and Practices. Prentice-Hall Inc., New Jersey, USA.

Thornthwaite, C.W., 1948. *An approach toward a rational classification of climate,* Geographical Review (American Geographical Society) 38 (1): 55–94.

We trust that this letter meets the current needs of the project team. Please contact the undersigned with any questions or comments.

Yours truly,

KNIGHT PIESOLD LTD.

PROFESSIONAL PROVINCE OF E. E. RAINEY # 34952 BRITISH COLUMBIA ENGINEER

July 24/14

Signed: ______________________
Erin Rainey, P.Eng.
Project Engineer

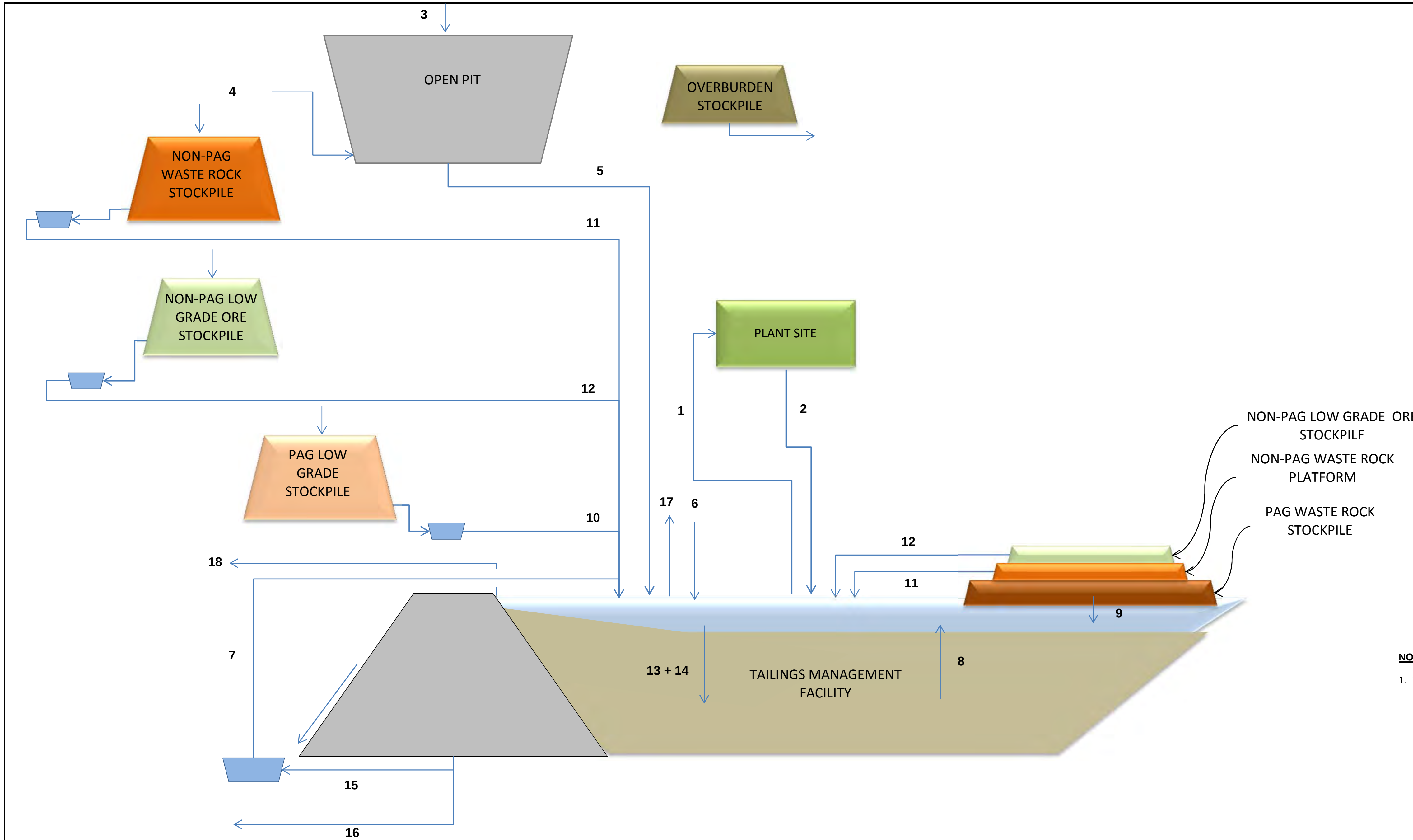
Reviewed: ______________________
For: Daniel Fontaine, P.Eng.
Senior Engineer

Approved: ______________________
Ken Brouwer, P.Eng.
President

Attachments:

Figure 1 Rev 0	Mine Site Water Balance Model Schematic
Figure 2 Rev 0	Monthly TMF Pond Volume Average
Figure 3 Rev 0	Closure Pit Filling Average

/er



Number	Description
	Plant Site Balance
1	TMF Reclaim
2	Water in Tailings Slurry
	Open Pit Balance
3	Direct Precip and Runoff
4	Groundwater Inflows
5	Dewatering to TMF
	Tailings Management Facility Balance
2	Water in Tailings Slurry
5	Pit Dewatering to TMF
6	Direct precipitation and runoff (pond, beach and undisturbed area)
7	Seepage Recycle (including TMF embankment runoff)
1	TMF Reclaim
8	Tailings Consolidation Seepage
9	Exposed PAG waste rock stockpile runoff into TMF
10	PAG LGO stockpile runoff
11	Non-PAG Waste Rock runoff
12	Non-PAG LGO stockpile runoff
13	Water trapped in tailings voids
14	Water trapped in PAG waste rock voids
15	Recoverable embankment seepage
16	Unrecoverable embankment seepage
17	TMF pond and beach evaporation
18	TMF spillway overflow

NOTES:

1. WATER BALANCE SCHEMATIC IS NOT DRAWN TO SCALE.

0	22JUL'14	ISSUED WITH LETTER	EER	DDF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

HARPER CREEK MINING CORP.

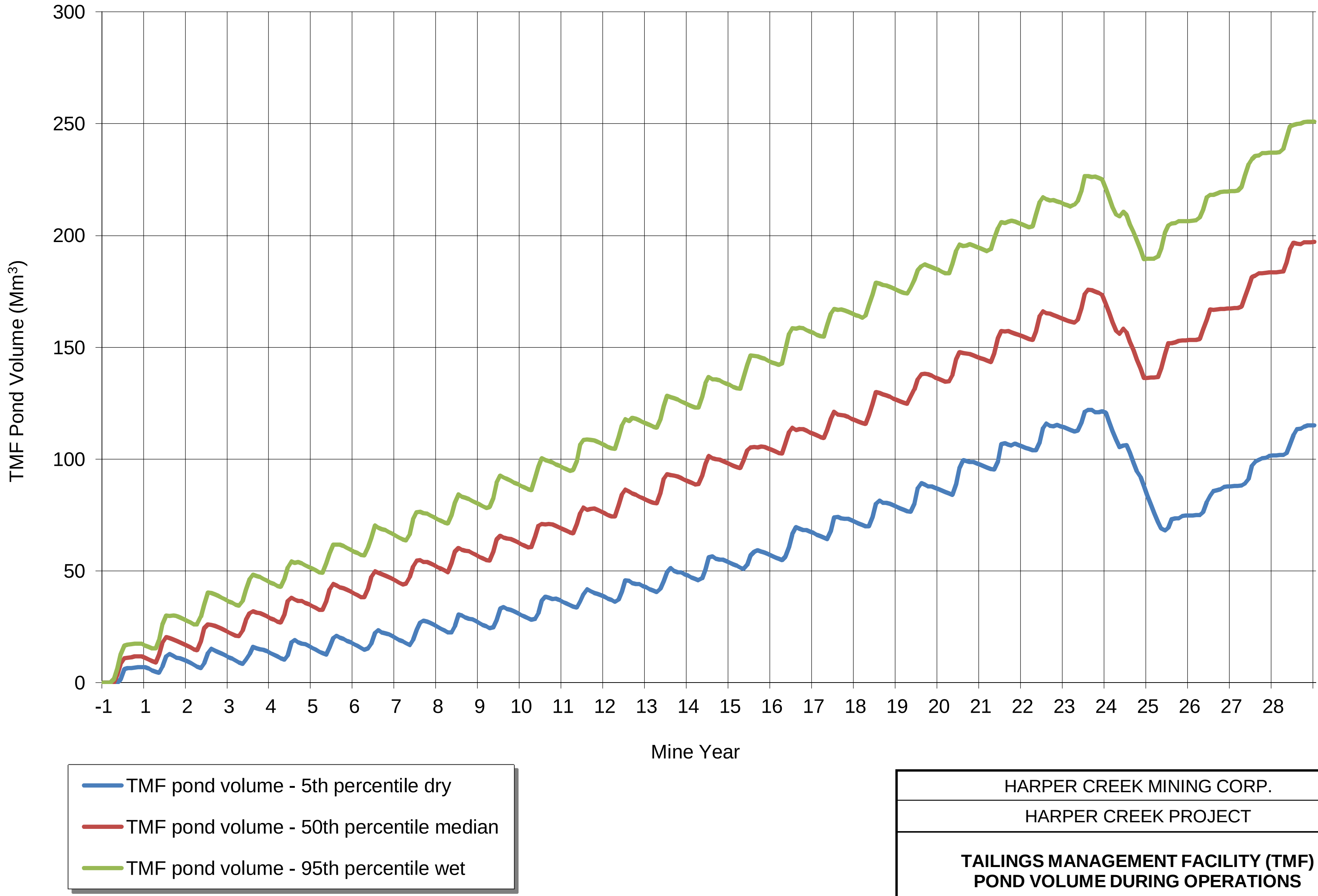
HARPER CREEK PROJECT

MINE SITE WATER BALANCE SCHEMATIC

Knight Piésold CONSULTING

P/A NO.	REF. NO.
VA101-458/14	VA14-00700

FIGURE 1 REV 0



0	23JUL'14	ISSUED WITH LETTER	EER	DF	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

TAILINGS MANAGEMENT FACILITY (TMF) POND VOLUME DURING OPERATIONS

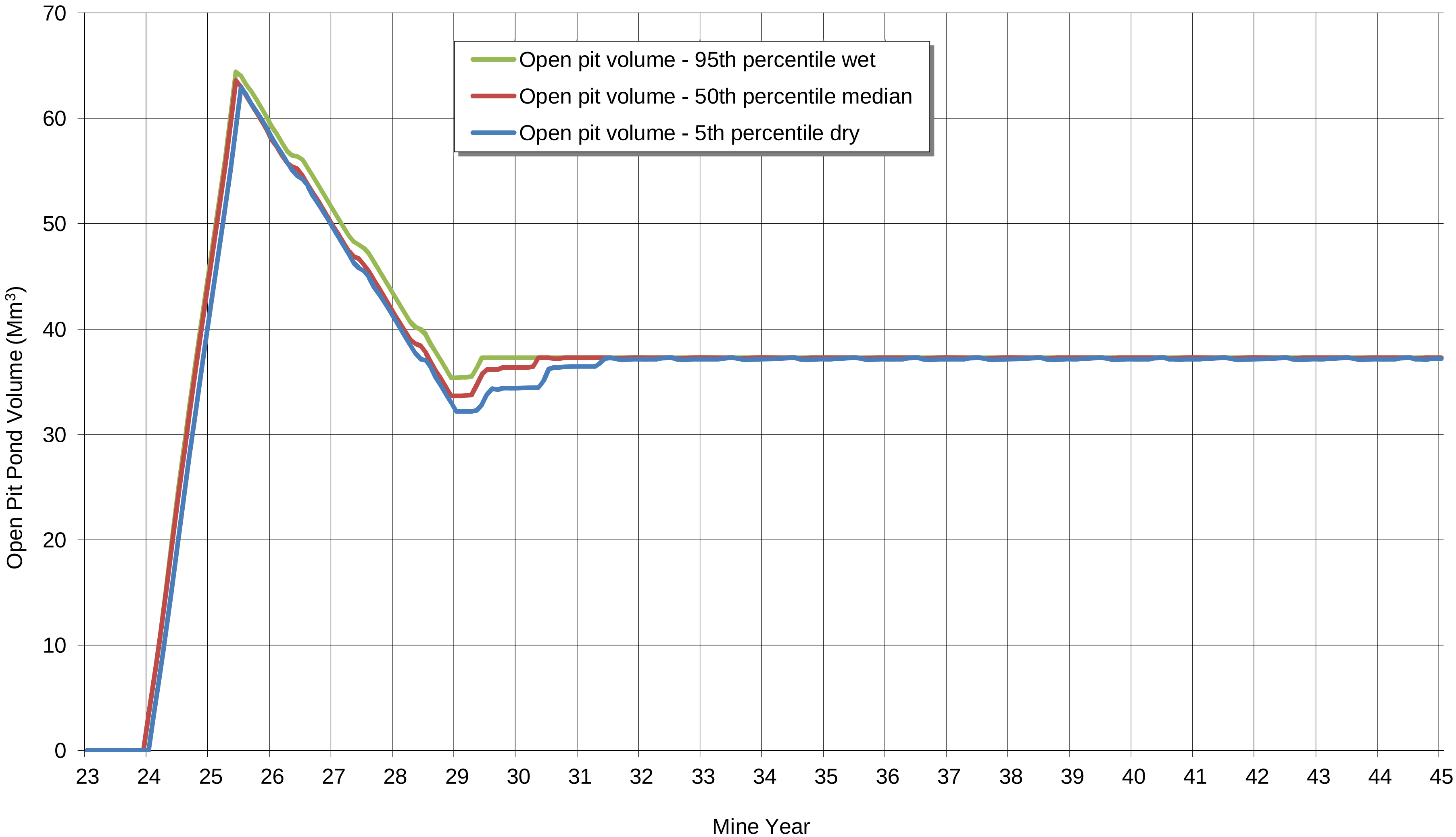
Knight Piésold CONSULTING

P/A NO. VA101-458/14

REF. NO. VA14-00700

FIGURE 2

REV 0



NOTES:

1. THE OPEN PIT FILLING IN YEAR 24 IS BASED ON WATER TRAPPED IN THE VOIDS AND TAILINGS SLURRY WATER THAT IS DISCHARGED DURING THE LGO PROCESSING.

2. AS OF YEAR 25.5, PROCESS WATER WILL BE RECLAIMED FROM THE OPEN PIT AND THE OPEN PIT POND IS DRAWN DOWN AS A RESULT.

3. ONCE TAILING DEPOSITION CEASES IN YEAR 29, THE OPEN PIT BEGINS TO FILL NATURALLY UNTIL IT REACHES AN ULTIMATE ELEVATION OF 1530 m.

0	23JUL'14	ISSUED WITH LETTER	EER	DF	KJB
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HARPER CREEK MINING CORP.

HARPER CREEK PROJECT

OPEN PIT VOLUME
FILLING DURING OPERATIONS AND CLOSURE

Knight Piésold CONSULTING

P/A NO. VA101-458/14

REF. NO. VA14-00700

FIGURE 3

REV 0

APPENDIX F

MATERIAL TAKEOFFS

(Pages F-1 to F-6)

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TABLE F.1

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT DESIGN REPORT
INITIAL CAPITAL COST MATERIAL TAKE-OFF SUMMARY
TAILINGS MANAGEMENT FACILITY

Print Jul/28/14 11:56:47

			QUANTITY	
ITEM NO.	**PRIMARY ITEM DESCRIPTION**	**SECONDARY ITEM DESCRIPTION**	**UNITS**	**PRE-PRODUCTION**
100 - TAILINGS MANAGEMENT FACILITY				
110	**TMF MAIN EMBANKMENT**			
110.01	Sediment and erosion control BMP's	Silt fences, sediment basins, temporary pumping, exfiltration, etc.	L.S.	1
110.02	Upgrade existing logging roads in starter dam area	Excludes road from plant site to TMF (see mining)	m	4,150
110.03	Logging and bunching timber	Stage 1 TMF footprint	ha	209
110.04	Clear and grub footprint (cofferdam)	Stockpile and burn, or haul and dump in TMF within 1000 m.	m^2	45,500
110.05	Remove topsoil to stockpile (cofferdam)	Assume 1000 m haul distance each way	m^3	22,750
110.06	Load, haul, place, compact - Zone S (cofferdam)	1500 m haul distance each way from local borrow	m^3	402,000
110.07	Clear and grub footprint (Stage 1 embankment)	Stockpile and burn, or haul and dump in TMF within 1000 m.	m^2	224,000
110.08	Remove topsoil to stockpile (Stage 1 embankment)	Assume 1000 m haul distance each way	m^3	112,000
110.09	Sub-excavate 2 m key trench for Zone S	Haul to unsuitable pile or use as embankment fill	m^3	22,500
110.10	Load, haul, place, compact - Zone S (key trench)	1500 m haul distance each way from local borrow	m^3	22,500
110.11	Load, haul, place, compact - Zone C (Stage 1)	Costs from mining, compaction by selective routing of trucks	m^3	5,547,000
110.12	Load, haul, place, compact - Zone S (Stage 1)	1500 m haul distance each way from local borrow	m^3	1,169,000
110.13	Load, haul, place, compact - Zones F and T (Stage 1)	1000 m haul distance each way from local quarry	m^3	236,000
110.14	Embankment outlet drain - Zone F	1000 m haul distance each way from local quarry	m^3	1,600
110.15	Embankment outlet drain - Zone D	1000 m haul distance each way from local quarry	m^3	4,250
110.16	Longitudinal embankment drain - Zone D	1000 m haul distance each way from local quarry	m^3	150
110.17	Foundation drain - Zone D	1000 m haul distance each way from local quarry	m^3	2,100
110.18	200 mm perforated CPT (Type SP) Pipe		m	425
110.19	150 mm perforated CPT (Type SP) Pipe		m	2,460
110.20	Seepage collection monitoring sump	Suggest allowance of $50,000 for seepage monitoring sump.	L.S.	1
120	**TMF WATER MANAGEMENT**			
120.01	Construct new roads to sediment control pond		m	1,000
120.02	Sediment and erosion control BMP's	Silt fences, sediment basins, temporary pumping, exfiltration, etc.	L.S.	1
120.03	Logging and bunching timber	Sediment control pond footprint	ha	1.2
120.04	Clear and grub footprint	Stockpile and burn, or haul and dump in TMF within 1000 m.	m^2	12,000
120.05	Remove topsoil to stockpile	Assume 1000 m haul distance each way	m^3	6,000
120.06	Mass excavation, load, haul, dump	Haul to unsuitable pile or use as embankment fill	m^3	7,000
120.07	Load, haul, place, compact - Zone S	1700 m haul distance each way from local borrow	m^3	24,000
120.08	Allowance for pond lining (60 MIL HDPE)	Pond base 35 m x 175 m x 4 m deep with 2H:1V slopes	m^2	8,955
120.09	Construct diversion ditches (Stage 1 embankment)	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m	2,400
120.10	Construct collection ditches (Stage 1 embankment)	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m	1,900
120.11	14" HDPE DR11 pipeline	Main embankment water management pipeline	m	1,410
120.12	Pump Station Pumps (suggest Pioneer PP64C21)	200 hp motor - 153 m^3/hr/pump - 130 m TDH	ea.	4
120.13	Booster Station 1 pumps (suggest Pioneer SC64C21)	200 hp motor - 153 m^3/hr/pump - 130 m TDH	ea.	4
120.14	Pump Station - civil works	Buildings, foundation works, etc. for 10 m x 12 m building	L.S.	2
120.15	Pump Station - mechanical works	Control, isolation and air release valves, flowmeters, etc.	L.S.	2
120.16	Pump Station - electrical works	Electrical and instrumentation	L.S.	2

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix F\[Material Takeoffs - rC.xlsx]Table 1 - CAPEX - TMF

NOTES:

1. QUANTITIES PROVIDED ARE NEAT-LINE MEASURED OR CALCULATED QUANTITIES WITH NO ALLOWANCE FOR DESIGN GROWTH OR WASTE DURING CONSTRUCTION.
2. BULK EARTHWORKS QUANTITIES DO NOT INCLUDE AN ALLOWANCE FOR BULKING AND SHRINKAGE DURING CONSTRUCTION.
3. ALLOWANCES ARE PROVIDED BASED ON RECENT EXPERIENCE WHERE THE LEVEL OF DESIGN DETAIL WAS NOT SUFFICIENT FOR QUANTITY MEASUREMENT OR CALCULATION.

0	27MAY'14	ISSUED WITH REPORT VA101-458/11-1	DDF	BB	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

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TABLE F.2

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT DESIGN REPORT
INITIAL CAPITAL COST MATERIAL TAKE-OFF SUMMARY
NON-PAG WASTE ROCK AND OVERBURDEN STOCKPILES

Print Jul/28/14 11:57:40

			QUANTITY	
ITEM NO.	PRIMARY ITEM DESCRIPTION	SECONDARY ITEM DESCRIPTION	UNITS	PRE-PRODUCTION
200 - NON-PAG WASTE ROCK AND OVERBURDEN STOCKPILES				
210	**NON-PAG WASTE ROCK STOCKPILE WATER MANAGEMENT**			
210.01	Upgrade existing logging roads		m	1,400
210.02	Construct new roads to sediment control pond		m	2,100
210.03	Sediment and erosion control BMP's	Silt fences, sediment basins, temporary pumping, exfiltration, etc.	L.S.	1
210.04	Logging and bunching timber	Sediment control pond footprint	ha	1.2
210.05	Clear and grub footprint	Stockpile and burn, or haul and dump in final footprint area	m^2	12,000
210.06	Remove topsoil to stockpile	Assume 1000 m haul and dump within final footprint area	m^3	6,000
210.07	Mass excavation, load, haul, dump for cut-off trench	Haul to unsuitable pile or use as embankment fill	m^3	2,400
210.08	Load, haul, place, compact - Zone C	Instream water management pond	m^3	38,000
210.09	Load, haul, place, compact - Zone S	Instream water management pond	m^3	5,400
210.10	Load, haul, place, compact - Zone F and T	Instream water management pond	m^3	3,000
210.11	Construct diversion ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m	4,700
210.12	20" HDPE DR11 pipeline	Water management pipeline	m	4,210
210.13	Pump Station Pumps (suggest Pioneer PP86C21)	300 hp motor - 303 m^3/hr/pump - 105 m TDH	ea.	4
210.14	Booster Station 1 pumps (suggest Pioneer SC86C21)	300 hp motor - 303 m^3/hr/pump - 105 m TDH	ea.	4
210.15	Booster Station 2 pumps (suggest Pioneer SC86C21)	300 hp motor - 303 m^3/hr/pump - 105 m TDH	ea.	4
210.16	Booster Station 3 pumps (suggest Pioneer SC86C21)	300 hp motor - 303 m^3/hr/pump - 105 m TDH	ea.	4
210.17	Booster Station 4 pumps (suggest Pioneer SC86C21)	300 hp motor - 303 m^3/hr/pump - 105 m TDH	ea.	4
210.18	Pump Station - civil works	Buildings, foundation works, etc. for 10 m x 12 m building	L.S.	5
210.19	Pump Station - mechanical works	Control, isolation and air release valves, flowmeters, etc.	L.S.	5
210.20	Pump Station - electrical works	Electrical and instrumentation	L.S.	5
220	**OVERBURDEN STOCKPILE WATER MANAGEMENT**			
220.01	Sediment and erosion control BMP's	Silt fences, sediment basins, temporary pumping, exfiltration, etc.	L.S.	1
220.02	Construct diversion ditches	1 m tradezoidal ditch along existing forestry roads	m	2,000
220.03	Construct diversion ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m	1,300
220.04	Construct collection ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m	1,900

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix F\[Material Takeoffs - rC.xlsx]Table 2 - CAPEX - NONPAG

NOTES:

1. QUANTITIES PROVIDED ARE NEAT-LINE MEASURED OR CALCULATED QUANTITIES WITH NO ALLOWANCE FOR DESIGN GROWTH OR WASTE DURING CONSTRUCTION.
2. BULK EARTHWORKS QUANTITIES DO NOT INCLUDE AN ALLOWANCE FOR BULKING AND SHRINKAGE DURING CONSTRUCTION.
3. ALLOWANCES ARE PROVIDED BASED ON RECENT EXPERIENCE WHERE THE LEVEL OF DESIGN DETAIL WAS NOT SUFFICIENT FOR QUANTITY MEASUREMENT OR CALCULATION.

0	27MAY'14	ISSUED WITH REPORT VA101-458/11-1	DDF	BB	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

Knight Piésold CONSULTING

TABLE F.3

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT DESIGN REPORT
INITIAL CAPITAL COST MATERIAL TAKE-OFF SUMMARY
LOW-GRADE ORE STOCKPILES

Print Jul/28/14 11:58:07

				QUANTITY
ITEM NO.	**PRIMARY ITEM DESCRIPTION**	**SECONDARY ITEM DESCRIPTION**	**UNITS**	**PRE-PRODUCTION**
300 - LOW-GRADE ORE (LGO) STOCKPILES				
310	**PAG LGO STOCKPILE WATER MANAGEMENT**			
310.01	Sediment and erosion control BMP's	Silt fences, sediment basins, temporary pumping, exfiltration, etc.	L.S.	1
310.02	Load, haul, place, compact - Zone S	From mining, place 500 mm layer from pit overburden stripping	m^3	282,750
310.03	Logging and bunching timber	Sediment control pond footprint	ha	0.9
310.04	Clear and grub footprint	Stockpile and burn, or haul and dump in TMF	m^2	8,500
310.05	Remove topsoil to stockpile	Assume 1000 m haul distance each direction	m^3	4,250
310.06	Mass excavation, load, haul, dump	Haul to unsuitable pile or use as embankment fill	m^3	12,000
310.07	Load, haul, place, compact - Zone S	From local mass excavation or pit pre-stripping	m^3	11,000
310.08	Allowance for pond lining (60 MIL HDPE)	Pond base 28 m x 125 m x 5 m deep with 2H:1V slopes	m^2	5,870
310.09	Construct collection ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m	3,200
310.10	12" HDPE DR11 pipeline	Water management pipeline	m	650
310.11	Pump Station Pumps (suggest Pioneer PP86C10)	40 hp motor - 395 m^3/hr/pump - 20 m TDH	ea.	2
310.12	Pump Station - civil works	Buildings, foundation works, etc. for 10 m x 12 m building	L.S.	1
310.13	Pump Station - mechanical works	Control, isolation and air release valves, flowmeters, etc.	L.S.	1
310.14	Pump Station - electrical works	Electrical and instrumentation	L.S.	1
320	**NON-PAG LGO STOCKPILE WATER MANAGEMENT**			
320.01	Sediment and erosion control BMP's	Silt fences, sediment basins, temporary pumping, exfiltration, etc.	L.S.	1
320.02	Logging and bunching timber	Sediment control pond footprint	ha	1.0
320.03	Clear and grub footprint	Stockpile and burn, or haul and dump in TMF	m^2	10,000
320.04	Remove topsoil to stockpile	Assume 1000 m haul distance each direction	m^3	5,000
320.05	Mass excavation, load, haul, dump	Haul to unsuitable pile or use as embankment fill	m^3	13,000
320.06	Load, haul, place, compact - Zone S	From local mass excavation or pit pre-stripping	m^3	50,000
320.07	Allowance for pond lining (60 MIL HDPE)	Pond base 28 m x 140 m x 5 m deep with 2H:1V slopes	m^2	6,530
320.08	Construct diversion ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m	800
320.09	Construct collection ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m	2,500
320.10	12" HDPE DR11 pipeline	Water management pipeline	m	2,100
320.11	Pump Station Pumps (suggest Pioneer PP64C21)	250 hp motor - 187.5 m^3/hr/pump - 141 m TDH	ea.	3
320.12	Pump Station - civil works	Buildings, foundation works, etc. for 10 m x 12 m building	L.S.	1
320.13	Pump Station - mechanical works	Control, isolation and air release valves, flowmeters, etc.	L.S.	1
320.14	Pump Station - electrical works	Electrical and instrumentation	L.S.	1

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix F\[Material Takeoffs - rC.xlsx]Table 3 - CAPEX - LGO

NOTES:

1. QUANTITIES PROVIDED ARE NEAT-LINE MEASURED OR CALCULATED QUANTITIES WITH NO ALLOWANCE FOR DESIGN GROWTH OR WASTE DURING CONSTRUCTION.
2. BULK EARTHWORKS QUANTITIES DO NOT INCLUDE AN ALLOWANCE FOR BULKING AND SHRINKAGE DURING CONSTRUCTION.
3. ALLOWANCES ARE PROVIDED BASED ON RECENT EXPERIENCE WHERE THE LEVEL OF DESIGN DETAIL WAS NOT SUFFICIENT FOR QUANTITY MEASUREMENT OR CALCULATION.

0	27MAY'14	ISSUED WITH REPORT VA101-458/11-1	DDF	BB	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

Knight Piésold CONSULTING

TABLE F.4

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT DESIGN REPORT
SUSTAINING CAPITAL COST MATERIAL TAKE-OFF SUMMARY

Print Jul/28/14 11:59:54

				QUANTITY													
ITEM NO.	PRIMARY ITEM DESCRIPTION	SECONDARY ITEM DESCRIPTION	UNITS	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14
130	MAIN EMBANKMENT (SUSTAINING)																
130.01	Logging and bunching timber	On-going expansion of TMF footprint	ha	73	73	46	46	47	47	47	47	46	37	37	37	37	39
130.02	Clear and grub embankment footprint	Stockpile and burn, or haul and dump in TMF within 1000 m.	m^2	20,000	30,000	65,000		75,000		75,000			90,000				30,000
130.03	Remove topsoil to stockpile in embankment footprint	Assume 1000 m haul distance each way	m^3	10,000	15,000	32,500		37,500		37,500			45,000				15,000
130.04	Load, haul, place, compact - Zone C	Costs from mining, compaction by selective routing of trucks	m^3	4,477,000	3,587,000	675,000	3,887,000	2,342,500	3,902,500	636,000	433,000	6,872,000	626,000	431,000	460,000	6,989,000	503,000
130.05	Load, haul, place, compact - Zone S	Costs from mining, spread with dozer and compact	m^3	139,000	120,000	110,000	99,000	116,000	105,000	58,000	49,000	50,000	51,000	41,000	52,000	42,000	43,000
130.06	Load, haul, place, compact - Zones F and T	1000 m haul distance each way from local quarry	m^3	70,000	60,000	55,000	50,000	58,000	52,000	46,000	40,000	40,000	40,000	33,000	42,000	34,000	34,000
130.07	Construct diversion ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m					800					900				
130.08	Construct collection ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m					2,000					2,200				
130.09	Remove topsoil to north stockpile	Remove best material from TMF for beach reclamation	m^3							300,000	300,000	300,000					
330	PAG LGO STOCKPILE WATER MANAGEMENT (SUSTAINING)																
330.01	Load, haul, place, compact - Zone S	From mining, place 500 mm layer from pit overburden stripping	m^3	40,750	40,750												
410	OPEN PIT WATER MANAGEMENT (SUSTAINING)																
410.01	16" HDPE DR13.5 pipeline	Water management pipeline (Pit Base to Pit Crest)	m	520				510									
410.02	20" HDPE DR11 pipeline	Water management pipeline (Pit Crest to Discharge)	m	3,500													
410.03	Pit Intake Pump Station (suggest Godwin HL225M)	300 hp motor - 400 m^3/hr/pump - 21 m TDH (skid mounted)	ea.	1				1					1				
410.04	Booster Station 1 pumps (suggest Pioneer SC86C21)	250 hp motor - 400 m^3/hr/pump - 130 m TDH	ea.	2				1					1				
410.05	Booster Station 2 pumps (suggest Pioneer SC86C21)	250 hp motor - 400 m^3/hr/pump - 130 m TDH	ea.	2				1					1				
410.06	Pump Station - civil works	Buildings, foundation works, etc. for 10 m x 12 m building	ea.	2													
410.07	Pump Station - mechanical works	Control, isolation and air release valves, flowmeters, etc.	ea.	5				5					4				
410.08	Pump Station - electrical works	Electrical and instrumentation	ea.	2													
410.09	In-Pit Booster 1 pumps (suggest Godwin HL225M)	300 hp motor - 400 m^3/hr/pump - 69 m TDH (skid mounted)	ea.					2					1				
410.10	20" HDPE DR13.5 pipeline	Water management pipeline (Pit Base to Pit Crest)	m										1,030				

				QUANTITY													
ITEM NO.	PRIMARY ITEM DESCRIPTION	SECONDARY ITEM DESCRIPTION	UNITS	YEAR 15	YEAR 16	YEAR 17	YEAR 18	YEAR 19	YEAR 20	YEAR 21	YEAR 22	YEAR 23	YEAR 24	YEAR 25	YEAR 26	YEAR 27	YEAR 28
130	TMF MAIN EMBANKMENT (SUSTAINING)																
130.01	Logging and bunching timber	On-going expansion of TMF footprint	ha	41	41	41	41	41	18	18	18	17					
130.04	Load, haul, place, compact - Zone C	Costs from mining, compaction by selective routing of trucks	m^3	913,000	6,764,000	3,072,000	622,000	419,000	382,000	343,000	402,000						
130.05	Load, haul, place, compact - Zone S	Costs from mining, spread with dozer and compact	m^3	44,000	44,000												
130.06	Load, haul, place, compact - Zones F and T	1000 m haul distance each way from local quarry	m^3	35,000	35,000	27,000	36,000	28,000	28,000	28,000	38,000						
130.07	Construct diversion ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m		600												
130.08	Construct collection ditches	Assume 6 m dozer constructed trail, with 1 m trapezoidal ditch	m		2,500												
130.10	Pump Station pumps (suggest Pioneer PP64C21)	Replacement of original pumps, same specifications	ea.	4													
130.11	Booster Station 1 pumps (suggest Pioneer SC64C21)	Replacement of original pumps, same specifications	ea.	4													
130.12	Load, haul, place, compact - Zone S	Double handle from overburden stockpile, or local borrow	m^3			34,000	45,000	34,000	35,000	35,000	47,000						
140	TMF NORTH EMBANKMENT (SUSTAINING)																
140.01	Clear and grub embankment and pond footprint	Stockpile and burn, or haul and dump in TMF within 1000 m.	m^2				33,000										
140.02	Remove topsoil to stockpile in embankment footprint	Assume 1000 m haul distance each way	m^3				16,500										
140.03	Load, haul, place, compact - Zone C	Costs from mining	m^3					77,000									
140.04	Load, haul, place, compact - Zone S	Costs from mining or double handle from overburden stockpile	m^3					15,000									
140.05	Load, haul, place, compact - Zones F and T	3000 m haul distance each way	m^3					15,000									
140.06	Pond - Mass excavation, load, haul, dump	Haul to unsuitable pile or use as embankment fill	m^3				4,200										
140.07	Pond - Load, haul, place, compact - Zone S	2000 m haul distance each way from overburden stockpile	m^3				2,000										
140.08	Allowance for pond lining (60 MIL HDPE)	Pond base 10 m x 50 m x 3.5 m deep with 2H:1V slopes	m^2				1,104										
140.09	4" HDPE DR11 pipeline	Water management pipeline	m				410										
140.10	Pump Station Pumps (suggest Pioneer PP53C14)	25 hp motor - 35 m^3/hr/pump - 35 m TDH	ea.				2										
140.11	Pump Station - civil works	Buildings, foundation works, etc. for 10 m x 12 m building	L.S.				1										
140.12	Pump Station - mechanical works	Control, isolation and air release valves, flowmeters, etc.	L.S.				1										
140.13	Pump Station - electrical works	Electrical and instrumentation	L.S.				1										
230	NON-PAG WASTE ROCK STOCKPILE WATER MANAGEMENT (SUSTAINING)																
230.01	Pump Station Pumps (suggest Pioneer PP86C21)	Replacement of original pumps, same specifications	ea.	4													
230.02	Booster Station 1 pumps (suggest Pioneer SC86C21)	Replacement of original pumps, same specifications	ea.	4													
230.03	Booster Station 2 pumps (suggest Pioneer SC86C21)	Replacement of original pumps, same specifications	ea.	4													
230.04	Booster Station 3 pumps (suggest Pioneer SC86C21)	Replacement of original pumps, same specifications	ea.	4													
230.05	Booster Station 4 pumps (suggest Pioneer SC86C21)	Replacement of original pumps, same specifications	ea.	4													
330	PAG LGO STOCKPILE WATER MANAGEMENT (SUSTAINING)																
330.02	Pump Station Pumps (suggest Pioneer SC86C10)	Replacement of original pumps, same specifications	ea.	2													
340	NON-PAG LGO STOCKPILE WATER MANAGEMENT (SUSTAINING)																
340.01	Pump Station Pumps (suggest Pioneer SC64C21)	Replacement of original pumps, same specifications	ea.	3													
410	OPEN PIT WATER MANAGEMENT (SUSTAINING)																
410.07	Pump Station - mechanical works	Control, isolation and air release valves, flowmeters, etc.	ea.	3													
410.11	In-Pit Booster 2 pumps (suggest Godwin HL225M)	300 hp motor - 400 m^3/hr/pump - 69 m TDH (skid mounted)	ea.	3													

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix F\[Material Takeoffs - rC.xlsx]Table 4 - SUSTAINING

NOTES:

1. QUANTITIES PROVIDED ARE NEAT-LINE MEASURED OR CALCULATED QUANTITIES WITH NO ALLOWANCE FOR DESIGN GROWTH OR WASTE DURING CONSTRUCTION.
2. BULK EARTHWORKS QUANTITIES DO NOT INCLUDE AN ALLOWANCE FOR BULKING AND SHRINKAGE DURING CONSTRUCTION.
3. ALLOWANCES ARE PROVIDED BASED ON RECENT EXPERIENCE WHERE THE LEVEL OF DESIGN DETAIL WAS NOT SUFFICIENT FOR QUANTITY MEASUREMENT OR CALCULATION.

0	27MAY'14	ISSUED WITH REPORT VA101-458/11-1	DDF	BB	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

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TABLE F.5

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT DESIGN REPORT
OPERATING COST ESTIMATE - ANNUAL POWER CONSUMPTION SUMMARY

Print Jul/28/14 12:00:15

				QUANTITY													
ITEM NO.	PRIMARY ITEM DESCRIPTION	SECONDARY ITEM DESCRIPTION	UNITS	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14
100	TMF Main Embankment	Water Management Pond - Pump System Energy	MWhr/year	632	651	669	688	706	711	716	720	725	730	734	739	743	747
100	TMF North Embankment	Water Management Pond - Pump System Energy	MWhr/year														
200	NON-PAG Waste Rock Stockpile	Water Management Pond - Pump System Energy	MWhr/year	3,065	3,119	3,173	3,226	3,280	3,257	3,234	3,211	3,188	3,165	3,165	3,165	3,165	3,165
300	PAG LGO Stockpile	Water Management Pond - Pump System Energy	MWhr/year	50	50	50	50	52	52	52	52	52	52	52	52	52	52
300	NON-PAG LGO Stockpile	Water Management Pond - Pump System Energy	MWhr/year	374	374	374	374	390	390	390	390	390	390	390	390	390	390
400	Open Pit Dewatering	Pump System Energy	MWhr/year	1,020	1,250	1,470	1,700	1,930	2,120	2,300	2,500	2,680	2,870	2,980	3,080	3,180	3,290
400	Open Pit Dewatering	Pump System Diesel Consumption	L/year	208,000	362,000	517,000	671,000	826,000	904,000	983,000	1,060,000	1,140,000	1,220,000	1,270,000	1,325,000	1,375,000	1,430,000
				QUANTITY													
ITEM NO.	PRIMARY ITEM DESCRIPTION	SECONDARY ITEM DESCRIPTION	UNITS	YEAR 15	YEAR 16	YEAR 17	YEAR 18	YEAR 19	YEAR 20	YEAR 21	YEAR 22	YEAR 23	YEAR 24	YEAR 25	YEAR 26	YEAR 27	YEAR 28
100	TMF Main Embankment	Water Management Pond - Pump System Energy	MWhr/year	752	756	760	765	769	773	778	782	782	782	782	782	782	782
100	TMF North Embankment	Water Management Pond - Pump System Energy	MWhr/year					13	13	13	13	13	13	13	13	13	13
200	NON-PAG Waste Rock Stockpile	Water Management Pond - Pump System Energy	MWhr/year	3,165	3,180	3,180	3,180	3,180	3,180	3,180	3,180	3,180	3,180	3,180	3,180	3,180	3,180
300	PAG LGO Stockpile	Water Management Pond - Pump System Energy	MWhr/year	52	52	52	52	52	52	52	52	52	52	52	52	52	52
300	NON-PAG LGO Stockpile	Water Management Pond - Pump System Energy	MWhr/year	390	390	390	390	390	390	390	390	390	389	389	389	389	389
400	Open Pit Dewatering	Pump System Energy	MWhr/year	3,400	3,500	3,600	3,710	3,820	3,920	4,030	4,130	4,240	4,340	0	0	0	0
400	Open Pit Dewatering	Pump System Diesel Consumption	L/year	1,480,000	1,535,000	1,590,000	1,640,000	1,700,000	1,750,000	1,800,000	1,850,000	1,905,000	1,960,000	0	0	0	0

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix F\[Material Takeoffs - rC.xlsx]Table 5 - OPEX

NOTES:

1. POWER CONSUMPTION QUANTITIES PROVIDED ARE BASED ON AVERAGE ANNUAL PRECIPITATION.
2. POWER CONSUMPTION WAS ESTIMATED FOR EACH SYSTEM FOR YEARS 1, 5, 10, AND 24, AND THE ANNUAL POWER CONSUMPTION IS A LINEAR INTERPOLATION BETWEEN THOSE YEARS.

0	27MAY'14	ISSUED WITH REPORT VA101-458/11-1	DDF	BB	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

Knight Piésold CONSULTING

TABLE F.6

HARPER CREEK MINING CORP.
HARPER CREEK PROJECT

MINE WASTE AND WATER MANAGEMENT DESIGN REPORT
CAPITAL COST MATERIAL TAKE-OFF SUMMARY
SUGGESTED PUMP PURCHASE COSTS

Print Jul/28/14 11:59:26

			QUANTITY	
ITEM NO.	PRIMARY ITEM DESCRIPTION	SECONDARY ITEM DESCRIPTION	UNIT	$ CAD / each
120 / 130	**TMF WATER MANAGEMENT**			
120 / 130	Pioneer PP64C21	200 hp motor - 153 m^3/hr/pump - 130 m TDH	ea.	$40,000
120 / 130	Pioneer SC64C21	200 hp motor - 153 m^3/hr/pump - 130 m TDH	ea.	$32,000
210 / 230	**NON-PAG WASTE ROCK STOCKPILE WATER MANAGEMENT**			
210 / 230	Pioneer PP86C21	300 hp motor - 303 m^3/hr/pump - 105 m TDH	ea.	$55,000
210 / 230	Pioneer SC86C21	300 hp motor - 303 m^3/hr/pump - 105 m TDH	ea.	$46,000
310 / 330	**PAG LGO STOCKPILE WATER MANAGEMENT**			
310 / 330	Pioneer PP86C10)	40 hp motor - 395 m^3/hr/pump - 20 m TDH	ea.	$18,000
320 / 340	**NON-PAG LGO STOCKPILE WATER MANAGEMENT**			
320 / 340	Pioneer PP64C21	250 hp motor - 187.5 m^3/hr/pump - 141 m TDH	ea.	$46,000
140	**TMF NORTH EMBANKMENT (SUSTAINING)**			
140	Pioneer PP53C14	25 hp motor - 35 m^3/hr/pump - 35 m TDH	ea.	$16,000
410	**OPEN PIT WATER MANAGEMENT SYSTEM (SUSTAINING)**			
410	Godwin HL225M (Diesel powered)	300 hp motor - 400 m^3/hr/pump - 21 m TDH (skid mounted)	ea.	$140,000
410	Godwin HL225M (Diesel powered)	300 hp motor - 400 m3/hr/pump - 69 m TDH (skid mounted)	ea.	$140,000
410	Pioneer SC86C21	250 hp motor - 400 m3/hr/pump - 130 m TDH	ea.	$46,000

M:\1\01\00458\11\A\Report\1 - Mine Waste and Water Management Design Report\Appendices\Appendix F\[Material Takeoffs - rC.xlsx]Table 6 - PUMP COSTS

NOTES:

1. SC = STANDARD CENTRIFUGAL.
2. PP = PIONEER PRIME (SC WITH VACUUM ASSISTED PRIMING, CAN PULL UP TO 23 mm Hg.
3. DUCTILE IRON CONSTRUCTION, SS IMPELLER AND WEAR RINGS, HORIZONTAL BEAR-SHAFT LONG COUPLED, OIL LUBRICATED BEARING FRAME.
4. SUGGESTED COSTS ARE POINT OF SALE ESTIMATED COSTS FROM THE SUPPLIER, AND EXCLUDE TRANSPORTATION AND INSTALLATION.

0	27MAY'14	ISSUED WITH REPORT VA101-458/11-1	DDF	BB	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D