Appendix 5-A

Technical Report and Feasibility Study

HARPER CREEK PROJECT

Application for an Environmental Assessment Certificate / Environmental Impact Statement

TECHNICAL REPORT & FEASIBILITY STUDY
OF THE
HARPER CREEK COPPER PROJECT

NEAR VAVENBY, BRITISH COLUMBIA

JULY 31, 2014

Qualified Persons

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YELLOWHEAD MINING INC.

MERIT CONSULTANTS INTERNATIONAL INC.

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Prepared by Qualified Persons

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1 EXECUTIVE SUMMARY

1.1 INTRODUCTION

Yellowhead Mining Inc. (YMI) is a public company listed on the Toronto Stock Exchange. The corporate office is located at Suite 730-800 West Pender Street, Vancouver, British Columbia V6C 2V6. YMI has a 100% interest in the Project (subject to the payment of a 3% NSR royalty capped at C$2.5M, adjusted for inflation), and an additional 2.5% NSR royalty on an estimated 1.5Mt of ore within the NI43-101 resource which is expected to be mined beginning in year nine. Throughout the Feasibility Study (FS) narrative all currency references are designated as C$ or US$.

1.1.1 *PREVIOUS STUDIES*

The Harper Creek Copper Project has been the subject of previous studies, notably:

- "2013 Amended & Restated Technical Report & Feasibility Study for the Harper Creek Copper Project." Collins, A.J., Dobbs, M., Simpson, R., Brouwer, K., Fox, J., Nilsson, J., January 25, 2013.
- "2012 Technical Report & Feasibility Study for the Harper Creek Copper Project." Collins, A.J., Dobbs, M., Simpson, R., Brouwer, K., Fox, J., Nilsson, J., March 29, 2012.
- "2011 Technical Report and Preliminary Assessment of the Harper Creek Project for Yellowhead Mining Inc." Narciso, N., Huang, J., Boyle, J.M., Ghaffari, H., Triebel, K., Teymouri, S., Cameron, M., Greenaway, G., and Donaghue, P., March 31, 2011.

1.1.2 *2014 FEASIBILITY STUDY*

This Feasibility Study (FS) was prepared by:

- Merit Consultants International Inc. ("Merit"): study management, capital cost estimating, project scheduling and implementation strategy;
- Knight Piésold Limited ("KP"): geotechnical, mine waste and water management;
- Allnorth Consultants Limited: process plant and facilities design;
- GeoSim Services Inc.: resource estimation;
- Nilsson Mine Services Ltd.: mineral reserve estimation, mine planning and scheduling, mine capital and operating costs; and
- Laurion Consulting Inc.: metallurgy, plant flowsheet design, and process operating costs.

1.1.3 *SITE VISITS*

Qualified Person, Daniel Fontaine visited the Harper Creek Project on October 26 to 27, 2011, July 15 to 17, 2012, and September 30, 2012 to view proposed locations of site facilities, potential construction material borrow areas and to review geotechnical site investigation progress.

All other Qualified Persons visited the Project site on July 11 and 12, 2011. The tour of the property included area inspections of the proposed plant site, crusher, borrow pits, tailings dam, access roads, Vavenby Bridge, Birch Island Bridge, BC Hydro power line, CN rail spur line and the neighboring Canfor and Weyerhaeuser facilities. YMI provided complete access to all relevant areas of the Harper Creek project (Project).

In addition, during the course of the FS, independent visits were made by QPs to further the study of viable infrastructure options including the main access road, Vavenby load-out and administrative facilities and the proposed high voltage power line route.

1.2 KEY PROJECT DATA

The key details about this project are as follows:

- The Project is primarily a copper project that is planned to process 70,000t/d or 25.6Mt/a for a period of 28 years producing a copper concentrate for sale to markets throughout the Pacific Rim countries. Metals to be recovered are copper, gold and silver.
- Estimated mineral reserve:
    - Proven: 457.2Mt @ 0.27% Cu, 0.030 g/t Au and 1.19 g/t Ag
    - Probable: 258.9Mt @ 0.24% Cu, 0.026 g/t Au and 1.16 g/t Ag
    - Proven+Probable: 716.2Mt @ 0.26% Cu, 0.029 g/t Au and 1.18 g/t Ag
- Based on the economic analysis, the Project will produce the following over the life of the mine from flotation:
    - Copper: 3.64bn lb;
    - Gold: 0.368M oz; and
    - Silver: 15.6M oz.
- The mine will be developed using conventional open pit mining utilizing electric hydraulic face shovels and 227t haul trucks to transport ore and waste. The process will include a conventional single-line SAG Ball mill circuit followed by conventional flotation to produce concentrate for sale.
- The Project is currently serviced by two access roads (Jones Creek and Vavenby Mountain Forest Service Road) and is located approximately 12km due south from existing major infrastructure including Highway #5 and the CN rail line. The Vavenby Mountain FSR will be upgraded to provide access during construction and project operation.
- Initial capital cost of C$1bn and LOM sustaining capital of C$336M.
- Life of Mine ("LOM") average annual copper concentrate production is estimated at 231,000 dmt at a cash operating cost, net of precious metals credits, of US$1.82/lb Cu.
- Economic analysis assuming 100% equity and metal prices of US$3.00/lb Cu, US$1,250/oz Au and US$20/oz Ag respectively results in a Net present value$_8$ ("NPV$_8$") before tax of US$684M, and an after tax NPV$_8$ of US$355M. The unlevered internal rate of return ("IRR") before tax is 16.8%, and an after tax IRR of 13.4%. Project payback after-tax is 5.4 years.
- The major milestones in the schedule are as follows:
    - Receipt of EA certificate – third quarter 2015;
    - Notice to Proceed and construction start-up – third quarter 2016;
    - Mechanical completion – second quarter 2018; and
    - Commercial Production – third quarter 2018.

- Key risks to the project include:
    - Receipt of all necessary permits;
    - Financing; and
    - Additional power provided to the existing North Thompson transmission line for operations. This will require the construction of a transmission line by BC Hydro.

1.3 PROJECT LOCATION & DESCRIPTION

The Project is located in the Thompson-Nicola area of British Columbia approximately 150km northeast of Kamloops. Clearwater, the largest community in the project area is 124km north of Kamloops, along the Yellowhead #5 Highway route. Twenty seven kilometers further along the Highway is Vavenby, the closest community to the project area.

Figure 1-1: Project Location

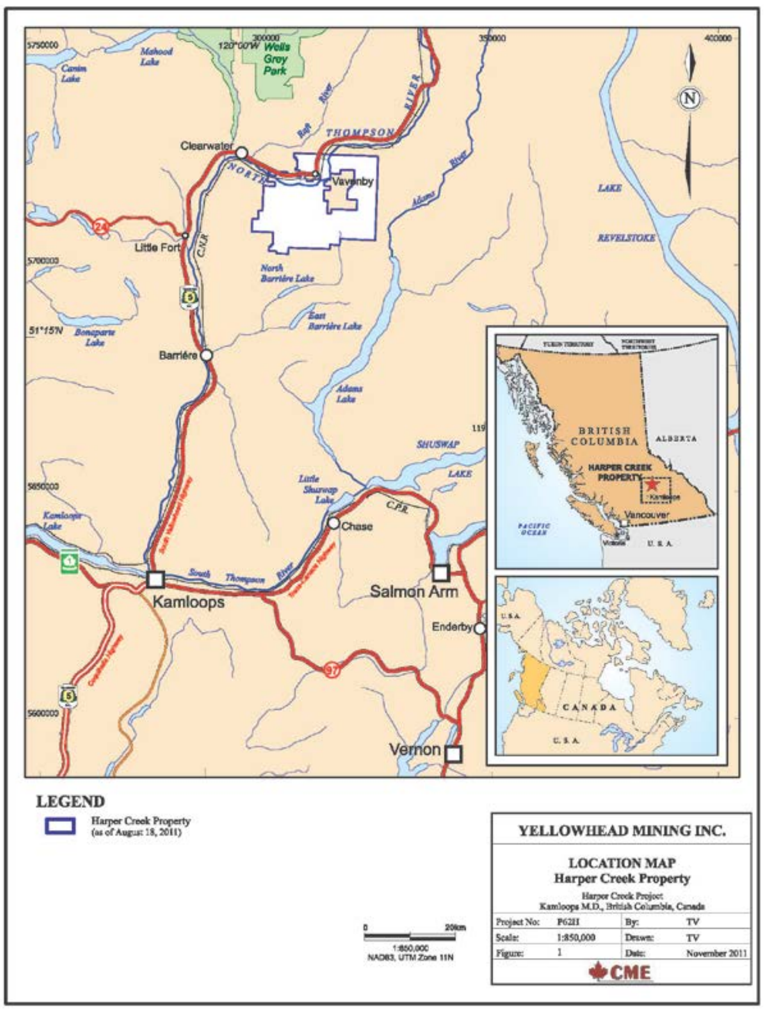
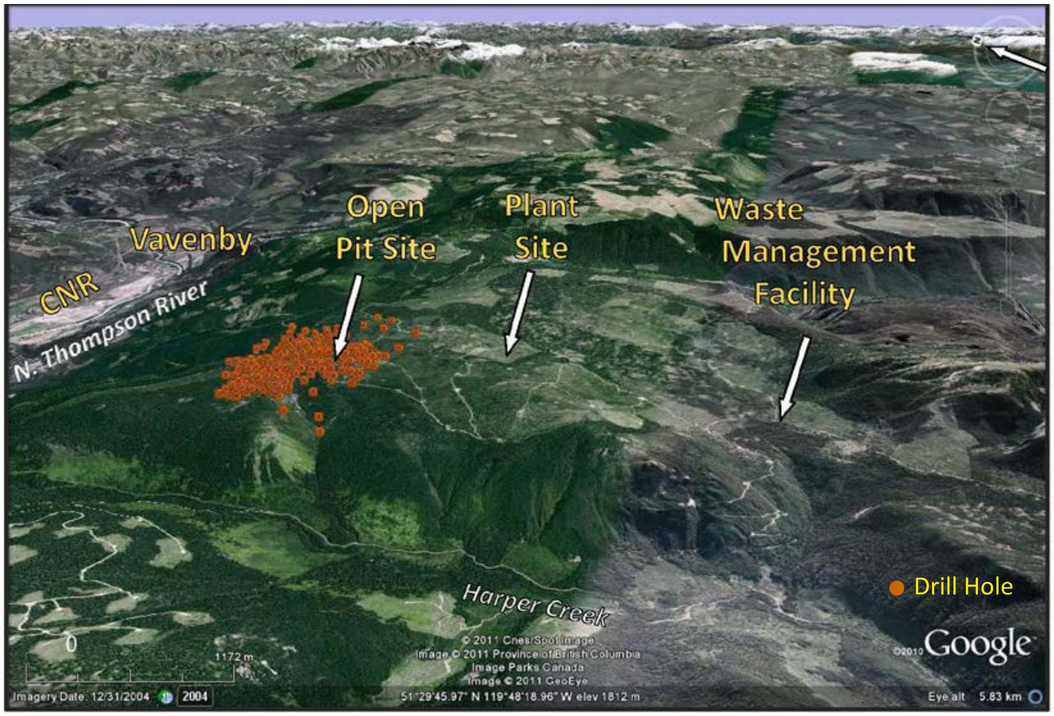


Figure 1-2: Isometric View of Site Location



The project area is hosted within the Shuswap Highlands characterized by gently sloping upland ridges and flanked by steepened valley slopes. These valleys include the Harper Creek Valley to the west and the Barrière River to the East, with the moderately sloped Thompson River Valley to the north. The elevations of the area range from approximately 1,100m at the floor of the Harper Creek Valley to 1,900m at the ridges surrounding the TMF area. The average elevation of the open pit area and plant site is 1,800m. The area has been glaciated and mountain tops are typically rounded. The project is covered in coniferous forest and has undergone extensive logging in the past.

The climate is typical of the BC central interior. The winter season runs from late October to late March. Site climatic conditions are dependent on location and elevation. Temperatures in Vavenby range from a high of +26^{o}C in August to a low of -10^{o}C in January. The estimated annual precipitation is 1,050mm, with approximately 60% falling as snow.

1.4 Early History

In 1966 separate prospecting and stream sediment sampling campaigns by Noranda Exploration and Quebec Cartier (a subsidiary of US Steel) discovered copper mineralization on the Project at the headwaters of Baker Creek and an unnamed tributary of Harper Creek. The two companies worked independently from 1967 until mid-1970 at which time they began a joint exploration program that continued to 1974. Drilling on the main Harper Creek Deposit (the "Deposit") totaled 25,806m in 161 holes. In 1996, American Comstock purchased the Noranda claims and acquired an option on the Quebec Cartier claims (held by Cygnus Mines Limited, a wholly-owned subsidiary of US Steel). American Comstock drilled 2,847m in eight holes. Six hundred eighty six samples were analyzed for copper, molybdenum and silver. Subsequently, American Comstock dropped the Cygnus option (Quebec Cartier claims) but maintained ownership of the Noranda claims.

YMI was formed in 2005 as a private British Columbia company. In 2005 and 2006, five claim groups were acquired or optioned by YMI on the historical drilling area and contiguous parts of the Eagle Bay Assemblage, which includes the Deposit. In 2006, YMI began the company's first phase of field exploration on the Harper Creek claims.

1.5 GEOLOGY & DEPOSIT

The Project is located within structurally complex, low-grade metamorphic rocks of the Eagle Bay Assemblage, part of the Kootenay Terrane on the western margin of the Omineca Belt in south-central British Columbia. The Eagle Bay Assemblage is divided into four northeast-dipping thrust sheets that collectively contain a succession of Lower Cambrian rocks overlain by a succession of Devonian-Mississippian rocks.

The Deposit is hosted by a Devonian-Mississippian succession of mafic to intermediate metavolcanic rocks which are intercalated with and overlain by dark grey phyllite, sandstone and grit. The nature of the structure in the region is a complex sequence of polyphase deformation consisting of a sequence of thrust faulting, intrusion-related folding and faulting, strike-slip and normal faulting all of which impose a complex alteration and metamorphic fabric on the rocks. At the southern edge of the Project, the Eagle Bay succession is cut by the mid-Cretaceous Baldy batholith.

The Deposit has characteristics of a polymetallic volcanogenic sulphide deposit, comprising lenses of disseminated, fracture-filling and banded iron and copper sulphides with accessory magnetite. Mineralization is generally conformable with the host-rock stratigraphy, as it is consistent with the volcanogenic model. Sulphide lenses are observed to measure many tens of metres in thickness with km-scale strike and dip extents.

The Deposit is separated by the northeast trending Harper Creek Fault into a West Domain and East Domain. In the West Domain, chalcopyrite mineralization is primarily observed in three copper bearing horizons. The upper horizon ranges from 60m to 170m in width and is continuous along an east-west strike for some 1,320m dipping approximately 30º north.

In the East Domain, mineralization is characterized by high angle, discontinuous, tension fractures of pyrrhotite, chalcopyrite±bornite which is frequently associated with quartz carbonate gangue. At the near surface areas in the south and down dip to the north, mineralization widths typically range from 120m to 160m. In the central area of the East Domain where thrust/reverse fault stacking has been interpreted, mineralization thicknesses typically range from 220m to 260m with local intersections of up to 290m.

1.6 MINERAL PROCESSING AND METALLURGICAL TESTING

The 2011 FS metallurgical test program achieved its objectives in both confirming and improving upon the results of the Preliminary Economic Assessment (PEA). In particular, the 2011 FS metallurgical test program focused on:

- Potential to increase grind size without adversely affecting concentrate grade and recovery; and
- Potential to simplify the reagent schedule.

The PQ drilling program was undertaken to source samples for the FS metallurgical testwork program. In all, four holes were located to ensure a representative sample of lithologies and grades covering the first 10 years of mine production was collected. A total sample weight of 5,261kg was sent to G&T Metallurgical Services (G&T) for testing, and 752kg was sent to FLSmidth (FLS) for comminution testwork.

Three lithologies dominate the Deposit. Approximately 50% is quartz eye schist with some slight variation in the precise breakdown of minerals, but with serecite-chlorite-quartz dominant. Schists (without quartz eyes) represent 21% and phyllites represent 19%. All other classifications each represent less than 1% of the overall resource with the exception of silica alteration representing 6.5% of the three main ore types. There is some slight variation in the lithological breakdown in different areas of the Deposit and was tested in zonal composites.

Various mineralogical studies were carried out, most recently by G&T. It was determined that chalcopyrite is the dominant copper mineral representing >98% of the copper species in the main grade/lithology composites. The only composite to fall below 98% chalcopyrite is a silica altered species which contains 94.5% chalcopyrite with 2% bornite and the balance equally covellite and chalcocite.

The FS testwork carried out by FLS on the nine samples indicated BMWi values ranging between 10.5 to 19.1kWh/t with an average of 13.19kWh/t, confirming the previous soft mineralization results reported in the PEA.

Preliminary evaluation indicated that the ore would be amenable to processing in either, a SAG mill/ball mill circuit or a SABC circuit (includes pebble crushing). KWM Consulting Inc. (KWM) evaluated the grinding power needs by carrying out seven simulations. The FS team then selected a SAG mill/ball mill grinding circuit, which includes a 21MW SAG mill and 2 parallel 13MW ball mills with a combined power of 26MW for a total grinding mill power of 47MW.

The PEA flowsheet with a finer primary grind (P_{80} 106µm) running essentially at a natural pH (8.5) achieved rougher copper recoveries close to 93% but at a concentrate weight pull to rougher/scavenger concentrates of around 12%. This is very typical of many low grade copper operations. The FS program rougher scavenger recovery was about the same; 93% copper recovery but at a weight pull of almost half (~6.5%) that achieved in the PEA. This was done at a coarser grind (P_{80} 180µm) but perhaps more importantly, at a high pH (11). This is abnormally high for a rougher/scavenger circuit (but quite common in a cleaner circuit). It is notable that the high pH was clearly enabling much more pyrite to be rejected in the rougher circuit, but at no significant loss in copper. It is the fear of copper loss that has generally driven the operation of rougher circuits at lower pH (<10).

The FS regrind is 20-25µm and will be achieved in an inert stirred IsaMill™. Also noteworthy is that with the rejection of more pyrite at the rougher stage, the losses of copper in cleaner tailings is reduced to gain approximately 3% copper recovery to final concentrate over that achieved in the PEA. This may be an effect of the reduced tonnage in the cleaner tails. Table 1-1 provides the summary of the metallurgical results from the FS testing program.

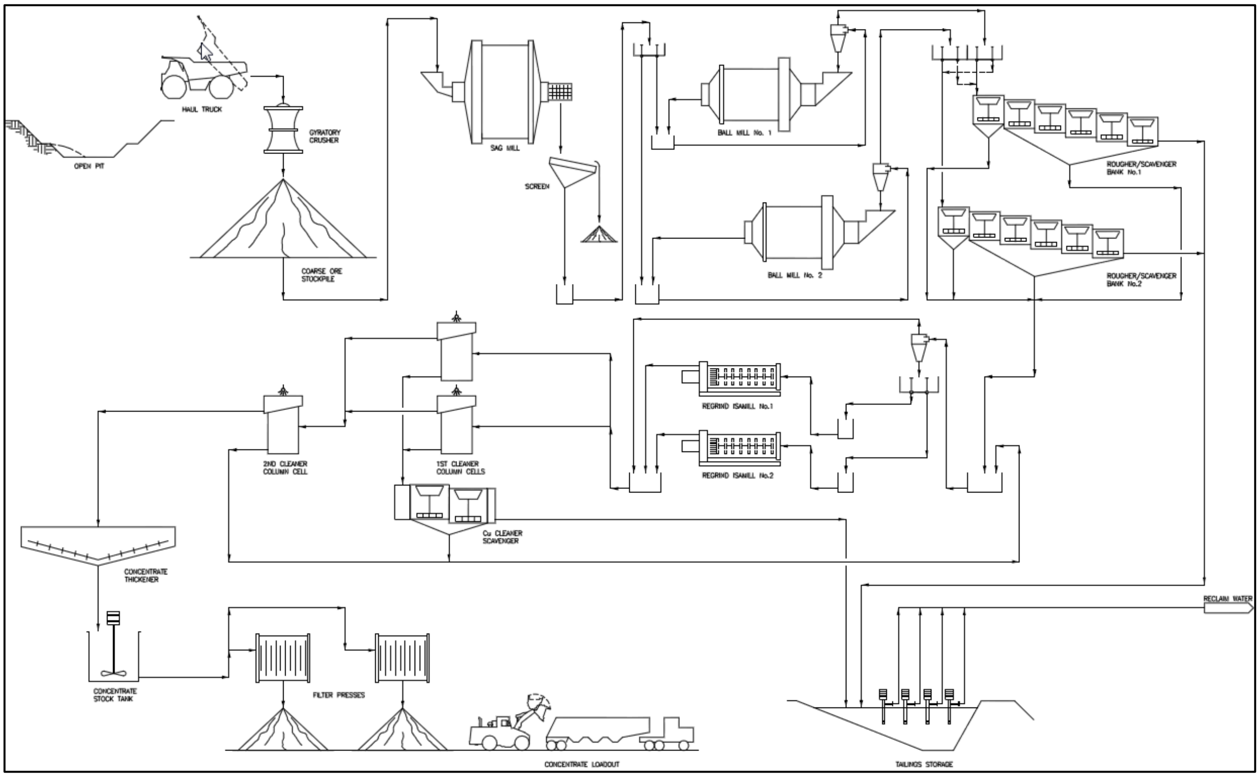
Table 1-1: FS Metallurgical Results Summary

Parameter	Units	FS
Crusher Index	kWh/t	6.6
Bond Ball Mill Work Index	kWh/t	13.2
Bond Rod Mill Work Index	kWh/t	11.8
Grind Size	µm	180
Regrind	µm	20
Recovery	%	89.2
Conc. Grade	%	25.5
Copper grade, year 1 – 5	%	0.31
Note – clean concentrate – no deleterious elements		

The two ore types, representing a very minor percentage of the total ore tonnage, that gave lower recoveries in the variability testing gave acceptable rougher recoveries. The problems encountered were found in the cleaner circuit, with higher cleaner tailings grades than normal. The one lithology (silica alteration of quartz eye schist) was the only sample in which copper minerals other than chalcopyrite were present in any appreciable amounts. It was observed that some of these minerals like chalcocite and covelite are very friable and can often be susceptible to slimes losses. This lithology was included in the master composite and so the discounted recovery is already reflected in the recoveries from the master composite. Also, some of the other composites gave markedly better recoveries, for example 92.9% Cu recovery @ 29.2% Cu concentrate grade from Western Low Grade @ 0.25% Cu head grade.

The results from the metallurgical testing program formed the basis of the FS flowsheet, shown in a simplified format in Figure 1-3. A summary of major equipment required for processing is summarized in Table 1-2.

Figure 1-3: Simplified FS Flowsheet



Allnorth Consultants Limited, November 2011

Table 1-2: Major Processing Circuit Equipment

Equipment	Specifications
Gyratory Crusher	Primary Crushing
	Size: 60" x 89"
	Power: 1,000kW
SAG Mill	Primary Grinding Mill
	Size 38' x 22'
	Power: 21MW
Ball Mills	Secondary Grinding Mills
	Size: 24' x 42'
	Power: 13MW each
Flotation Cells	2 Banks x 6 cells
	Size: 300m^3 each
Regrind	2 x IsaMill™
	Size: M10,000
	Power: 3MW each
Cleaner Circuit	1st Cleaner: 2 x Column Cells
	Size: 170m^3 each
	2nd Cleaner
	Size: 170m^3 each

1.7 Mineral Resource & Mineral Reserve Estimate

1.7.1 *Mineral Resource Estimate*

The mineral resource was estimated within a block model with dimensions of 12m by 12m by 12m. The estimate utilized analytical results from 353 core holes completed on the project between 1967 and 2013. Downhole composites of 6m length were generated from the initial sample intervals and capped at levels of 5% for Cu, 1g/t for Au and 30g/t for Ag prior to compositing.

Bulk densities were assigned by lithology based on the results of 10,739 measurements performed on core samples in 2006 and 2007.

Copper, gold and silver grades were estimated in three passes using an inverse distance squared weighting method (ID^2). The resulting grade model was constrained within a 700ppm Cu grade shell within potentially mineralized stratigraphy. The Harper Creek Fault was used as a hard boundary separating the east and west structural domains.

Table 1-3: Mineral Resource Estimate March 30, 2014 (Geosim)

Measured and Indicated Mineral Resource						Contained Metal		
Category	**Cut-off (Cu %)**	**Tonnes (000's)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Cu lbs (M's)**	**Au oz (000's)**	**Ag oz (000's)**
Measured (M)	0.15	564,361	0.27	0.029	1.2	3,359	526	21,769
Indicated (I)	0.15	735,877	0.24	0.027	1.2	3,894	639	28,385
Total M + I	**0.15**	**1,300,238**	**0.25**	**0.028**	**1.2**	**7,253**	**1,165**	**50,154**
Inferred Mineral Resource								
Category	**Cut-off (Cu %)**	**Tonnes (000's)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Cu lbs (M's)**	**Au oz (000's)**	**Ag oz (000's)**
Inferred	0.15	119,743	0.25	0.025	1.2	660	96	4,619

NB: Mineral resources are amenable to open pit mining methods and have been constrained using a Lerch-Grossman optimized pit. Assumptions include US$3.50/lb Cu with an average recovery of 89%, C$1.84/t mining cost, C$4.20/t process and G&A cost. Exchange rate of C$1-US$0.90. Pit slope angle of 42°.

1.7.1.1 Interpretation of the Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource Estimate include:

- Commodity price assumptions;
- Pit slope angles;
- Metal recovery assumptions; and
- Mining and Process cost assumptions.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the Province of British Columbia with respect to environmental, permitting, taxation, socio-economic, marketing and political factors. There are no known legal or title issues that would materially affect the mineral resource estimate.

1.7.2 MINERAL RESERVE ESTIMATE

The mineral reserve for the Deposit was estimated using a copper price of US$2.25/lb., a gold price of US$1,250.00/oz. and a silver price of US$20.00/oz. An exchange rate of US$0.90:C$1.00 was assumed. The mineral reserve is reported using a 0.14% copper cutoff grade. The proven and probable mineral reserve at Harper Creek is 716.2Mt with an average grade of 0.26% Cu, 0.029g/t Au and 1.18g/t Ag (Table 1-4).

Table 1-4 Harper Creek Estimated Mineral Reserve (Nilsson)

Mineral Reserve					Contained Metal		
Category	**Tonnes (000's)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Cu lbs (M's)**	**Au oz (000's)**	**Ag oz (000's)**
Proven	457,227	0.27	0.030	1.19	2,706	439	17,465
Probable	258,948	0.24	0.026	1.16	1,371	220	9,636
Total	**716,175**	**0.26**	**0.029**	**1.18**	**4,077**	**659**	**27,101**

1.7.2.1 Interpretation of the Mineral Reserve Estimate

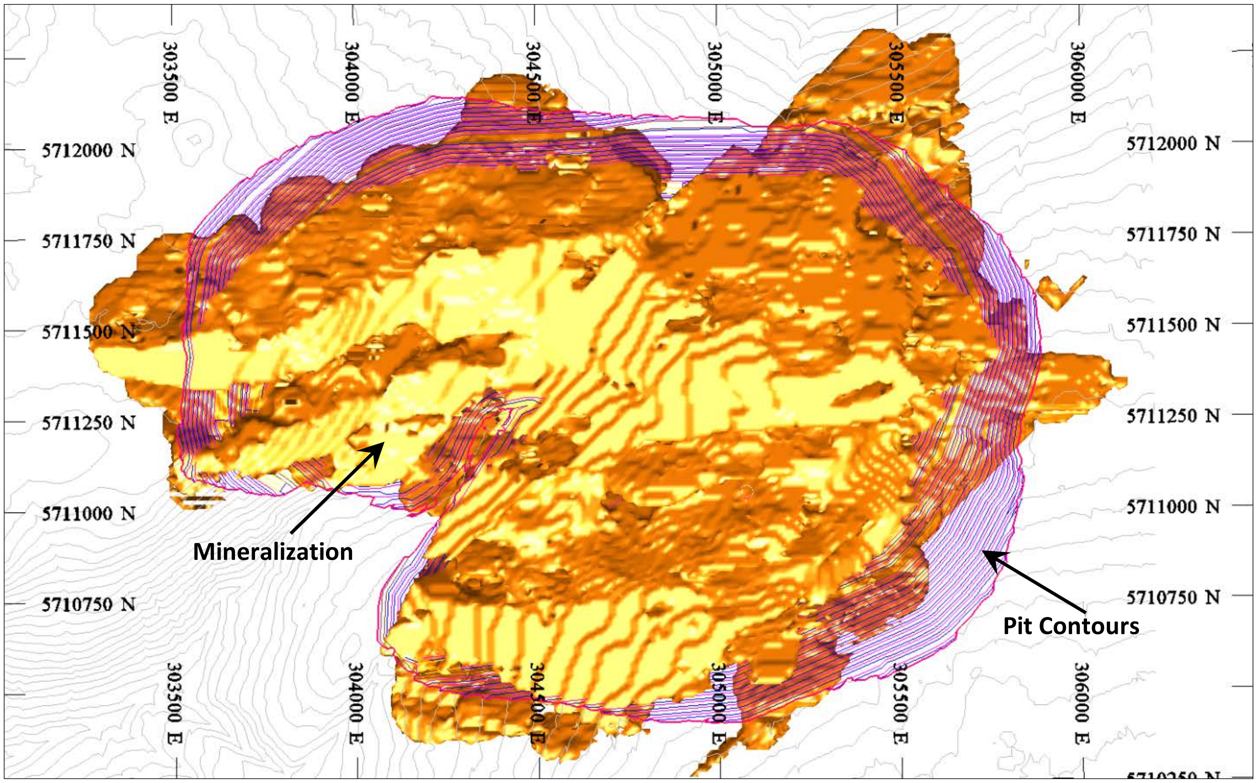
The mineral reserve estimate was based upon economic parameters, geotechnical design criteria and metallurgical recovery assumptions detailed throughout this FS. Changes in these assumptions will impact the in-pit mineral reserve estimate.

In general, increases in operating costs, reductions in revenue assumptions or reductions in metallurgical recovery may result in increased cut-off grades, reductions in in-pit mineral reserve and increasing strip ratios. The converse is also true. Reductions in operating costs, increases in revenue assumptions or increases in metallurgical recovery may result in reduced cut-off grades and increases to in-pit mineral resources.

The mineral reserve estimate is also dependent upon successful completion of the environmental permitting process and provision of electric power to the mine-site.

A pit limit plan view of the distribution of measured and indicated resource above 0.14% Cu is shown in Figure 1-4.

Figure 1-4 Resource Distribution - Pit Limits



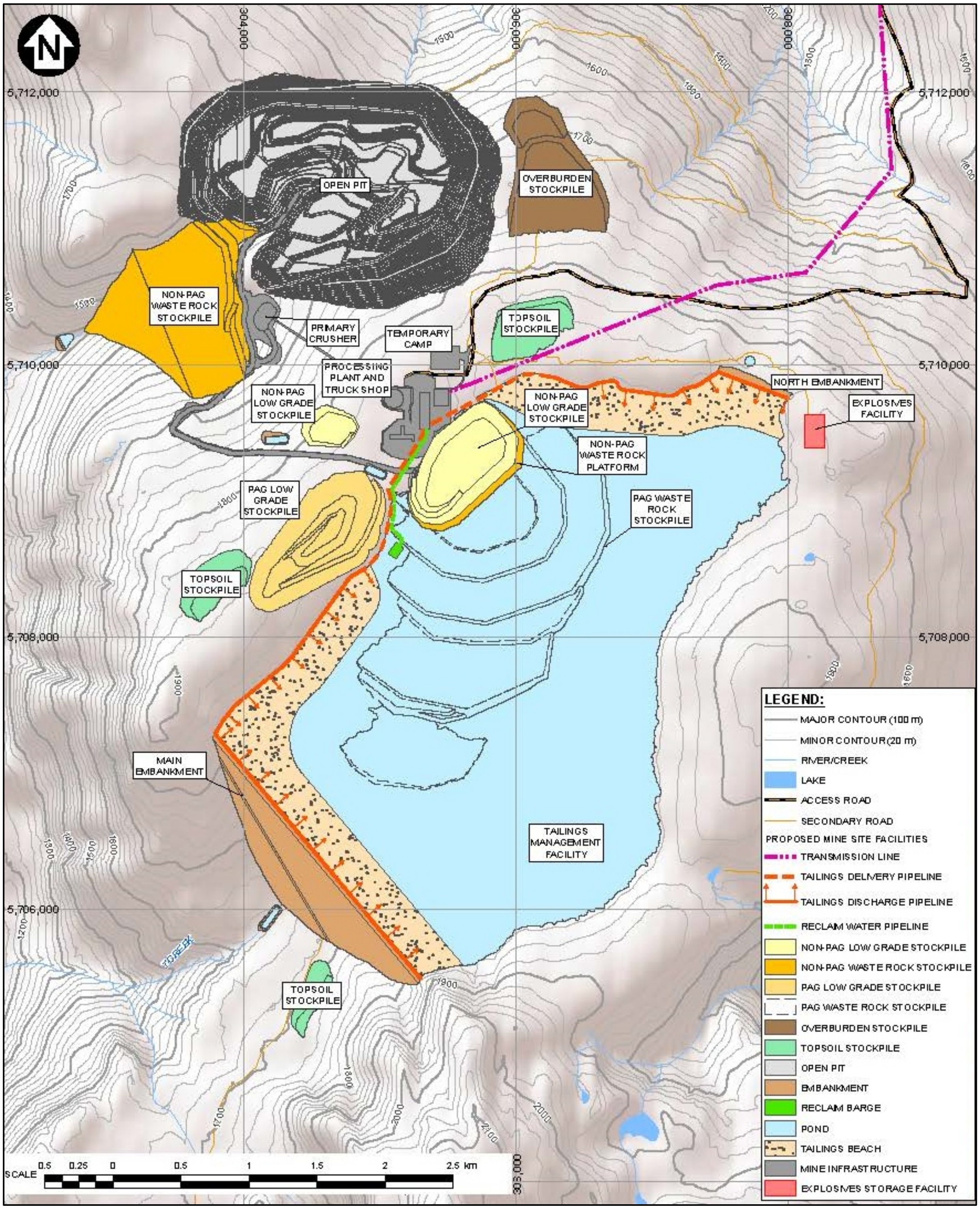
Nilsson Mining Services, July 2014

1.8 MINING METHODS

The Project consists of a combined electric and diesel powered open pit operation and a 70,000t/d conventional copper concentrator. The mineral reserve is estimated to be 716.2Mt with an average grade of 0.26% Cu, 0.029g/t Au and 1.18g/t Ag reported at a 0.14% Cu cutoff grade. The mineral reserve will be mined by open pit methods in five phases of pit development and expansion. The overall strip ratio is 0.76:1. The total in-pit waste is 571.7Mt. The overall mine life is 28 years after start-up of the concentrator.

Mill feed and waste will be drilled by diesel and electric powered 311mm rotary drills and blasted using heavy ammonium nitrate and fuel oil. Mill feed and waste will be loaded into 227t mine trucks by 42.0m^3 electric hydraulic shovels and an 18.5m^3 wheel loader. Potential acid generating waste rock (PAG) will be placed in the tailings management facility (TMF). Non acid generating waste (non-PAG) will be placed in designated disposal sites adjacent to the pit. Low grade will be stockpiled south of the plant site and in the valley south of the open pit. Direct mill feed will be hauled to the primary crusher located south of the pit. Crushed ore will be conveyed to the coarse ore stockpile and subsequently to the crushing, grinding and flotation sections of the process plant (Figure 1-5).

Figure 1-5: Site Layout



Knight Piesold Consulting, May 2014

1.9 RECOVERY METHODS

The Concentrator is of conventional design and is designed for simplicity of operation and to maximize copper recovery. The Run of Mine (ROM) ore will be reduced through three stages of comminution and the copper minerals recovered by flotation, with rougher/scavenger concentrates reground and cleaned to final commercial concentrate grades. The concentrator is designed to process a nominal 70,000t/d of copper sulphide ore and produce marketable copper concentrate.

The flow of ore will be through crushing, grinding, mechanical rougher/scavenger flotation in tank cell banks and the rougher/scavenger concentrate cleaned through two stage column flotation cleaning to increase the quality of the concentrate. The rougher/scavenger concentrate will be sent through a regrind circuit and reprocessed through the cleaners to increase copper grade to commercial levels. Final concentrate from the second cleaner column will be densified through a thickener and dried in filter presses to achieve concentrate moisture of approximately 8%. This concentrate will be trucked offsite for shipping to smelters.

The final flotation tailings, including the rougher/scavenger flotation tailings and the first cleaner scavenger tailings, will be disposed using the conventional tailing storage method. The process water in the TMF will be recycled to the process plant. Fresh water for process water make-up, gland seal service, mill cooling, and reagent preparation will be obtained from various local water sources.

1.10 PROJECT INFRASTRUCTURE

The general arrangement showing the layout of the project at the maximum extent of all facilities is shown on Figure 1-5. The facilities and services that will be required include the following:

- Roads
    - Project site access road upgrade; and
    - Plant site roads, yard areas and parking.
- Power
    - Supply from the BC Hydro grid;
    - Power line to site; and
    - Project site distribution.
- Plantsite Facilities
    - Primary Crusher;
    - Mill building and concentrate loadout;
    - Mine maintenance work shop; and
    - Warehouse
- Onsite Ancillary Buildings
    - Temporary housing facilities for construction personnel;
    - Fuel Storage;
    - Explosives Storage; and
    - Security, safety, and first aid facilities.
- Offsite Ancillary Buildings
    - Administration; and

  - Rail Loadout.
- Waste rock storage and stockpiles
- Tailings management facility
- Water Supply and Management

The plant site layout is detailed in Figure 1-6.

Figure 1-6: Conceptual Rendering of Plant Site Layout

Allnorth Consultants Limited, November 2011

1.11 Market Studies & Contracts

The Project is strategically located for delivery of copper concentrate to Pacific Rim Asian smelters. The concentrate is highly marketable and is considered to be low in deleterious elements. Concentrate will be trucked from the mine site approximately 25km to the rail head at Vavenby, and temporarily stored until loaded onto CN rail for transport to the Port of Vancouver, an approximate rail distance of 526km.

1.12 Environmental, Permitting, Social and Community Impact

1.12.1 *Environmental Studies*

YMI initiated environmental baseline studies on the Project in December 2007. In April 2011, it increased the scope of baseline data collection to fulfill the information requirements of an environmental assessment application submission. There are no known issues identified that would materially affect the ability of YMI to extract minerals as part of developing the Project.

1.12.2 *FIRST NATIONS*

The Project is located within the asserted traditional territory of the Simpcw First Nation and the Adams Lake Indian Band. Adams Lake is a member of the Lakes Division which includes the Little Shuswap Indian Band and Neskonlith Indian Band. All four of these First Nations are members of the Secwepemc Nation and the Shuswap Nation Tribal Council (SNTC). SNTC is a political organization that works on matters of common concern to all its members, including the development of self-government and the settlement of the aboriginal land title question.

Chu Chua is the main reserve of the Simpcw, meaning "the People of the North Thompson River". It is located on the North Thompson River, 20 minutes from Barriere, BC. The Simpcw have 4 other reserves located near Little Fort, Louis Creek and Dunn Lake. Simpcw has approximately 650 members (2011); 250 live on reserve. The Adams Lake Indian Band has approximately 740 members, half of which live on the seven reserves located near Chase and Shuswap Lake.

YMI has initiated a range of consultation activities with stakeholders since 2006. This includes one-one discussions with local landowners. Consultation with local First Nations has been, and continues to be, an important part of these activities. YMI continues to work closely with First Nations on the development of working agreements. YMI signed a Negotiation Agreement with Simpcw First Nation, and a General Services Agreement with both Simpcw and Adams Lake. Both communities had members involved in the baseline studies and fieldwork, including the archaeological impact assessment, for the Project.

1.12.3 *WASTE STOCKPILES, TAILINGS AND WATER MANAGEMENT*

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 effective 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 (ARD) and minimization of metal leaching from reactive tailings and waste rock, and
- staged development of the facility over the life of the project.

1.12.4 *MINE WATER MANAGEMENT*

The overall water management objective is to provide sufficient water to support the mill water requirements and maintain 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 ditches;
- 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 ponds 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 ponds; and
- groundwater and surface water from open pit dewatering.

1.12.5 *Mine Closure and 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 Low Grade Ore (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:

1.12.5.1 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.

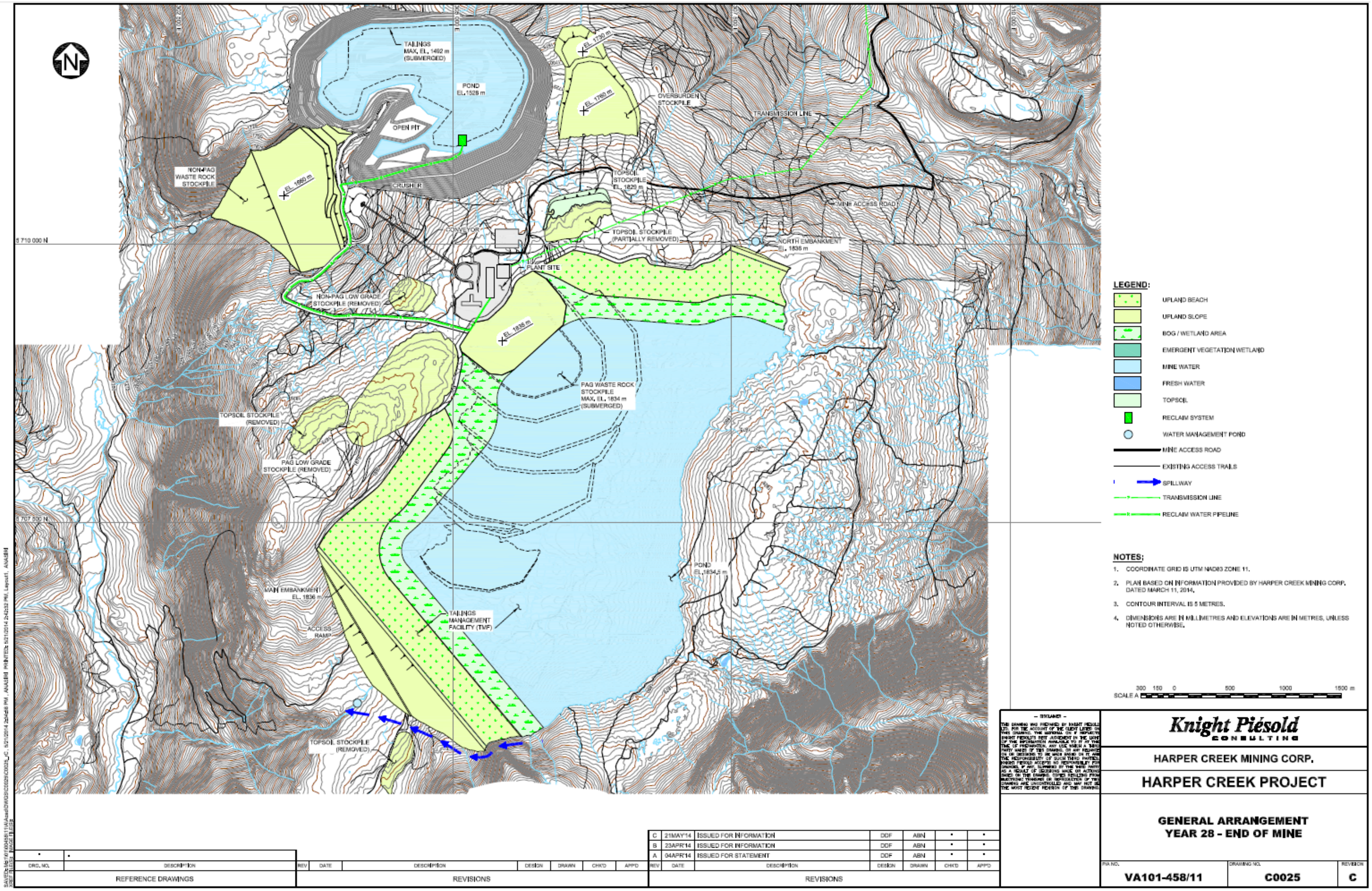
1.12.5.2 Final Reclamation Activities

- 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 300mm 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. Figure 1-7 provides an illustration of the general arrangement of the project in 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 1,530m and 1,545m, 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 1,555m, which allows for 10m of freeboard to manage storm inflows.

Figure 1-7: General Arrangement Year 28



1.13 CAPITAL AND OPERATING COST ESTIMATE

1.13.1 *INITIAL AND SUSTAINING CAPITAL*

The estimated initial capital cost of this Project is C$1,026M (+15%/-5%), including a contingency amount of C$91M, bonding amount of C$8M and an allowance of C$10M for BC Provincial Sales Tax (Table 1-5). All pricing is in Q1 2014 dollars.

Table 1-5: Capital Cost Estimate Summary

Capital Cost Estimate Summary	
Pre Production Capital (Year -2 and -1)	**C$ (M)**
Direct Costs	
Mining & Pre-production Development	298.0
Plant Site Infrastructure	9.5
Site Services & Utilities	13.4
Process	279.6
Ancillaries	27.4
Power Supply & Distribution	47.7
Tailings & Water Reclaim	54.0
Total Direct	**729.5**
Indirect Costs	
Owner's Costs	25.6
Indirects	162.1
Total Indirect	**187.7**
Subtotal	**917.2**
Contingency	90.7
Direct Costs + Indirect Costs	**1,007.9**
Bonding	7.9
BC Provincial Sales Tax	10.0
Total Project	**C$1,025.8**

Sustaining capital costs are estimated at C$335.8M including an allowance for reclamation of C$37.8M (Table 1-6). With regards to the sustaining capital, rock fill for the tailings dam will be sourced from run of mine non-PAG material and the cost for rock fill is included in the mining cost estimate. Replacement haul trucks will be replaced at the original capital cost and where possible older trucks will be sourced for parts as the fleet requirements decline after year 17 of operation.

Table 1-6: Sustaining Capital Summary Estimate

Area	Total Cost (C$)
Mining	203.7
Pit Power	13.6
Tailing Management Facility and Reclaim Water	45.2
Working Capital	35.5
Reclamation Bonding*	37.8
Total	**C$335.8**

NB. A total of C$112.4M is accumulated from Year-1 to Year 24. C$66.7M is recaptured as low grade stockpiles are processed

1.13.2 *Cash Operating Cost*

Total LOM cash operating cost for the project is estimated at C$8.22/t milled (+15%/- 5%). The estimate includes mining, process, G&A, site services, royalties, and excludes prestripping. On average, a total of 426 personnel are projected for the operation, including 272 for mining operation, 102 personnel for process and 52 personnel for site services and general and administration. The unit costs are based on an annual ore production rate of 25,550,000t/a (or 70,000t/d), and operation of 365d/a. Table 1-7 provides a summary of operating costs.

Table 1-7: Operating Cost Estimate

	Year 1 to 5 (C$)	Life of Mine (LOM) (C$)
Mining (per t mined)	1.81	1.97
Mining (per t milled)	4.48	3.75
Process (per t milled)	3.65	3.65
G & A (per t milled)	0.44	0.44
Site Services (per t milled)	0.35	0.35
Royalties	0.02	0.04
Total (per t milled)	**C$8.94**	**C$8.22**

1.13.3 *Net Smelter Return*

The average NSR over the LOM is estimated at US$13.36/t milled or C$14.85/t milled.

1.14 Economic Analysis

The financial analysis in the FS utilized the following assumptions as detailed in Table 1-8:

Table 1-8 Financial Analysis Assumptions

Parameter	Inputs
General Assumptions	
Mine Life	28 Years
Available mill operating days per year	365 days
Production Rate (average)	70,000 t/d
Average Process Recovery (Cu)	89.2%
Average Process Recovery (Ag)	55.8%
Average Process Recovery (Au)	57.7%
LOM Average Annual Copper Concentrate Production	231,000dmt/a
Concentrate Grade	25.5%
Long Term C$:US$ exchange rate	0.90
Market Assumptions	
Equity Basis	100%
Discount Rate	8%
Base Case LOM average copper price	US$3.00
Base Case LOM average gold price	US$1,250.00
Base Case LOM average silver price	US$20.00

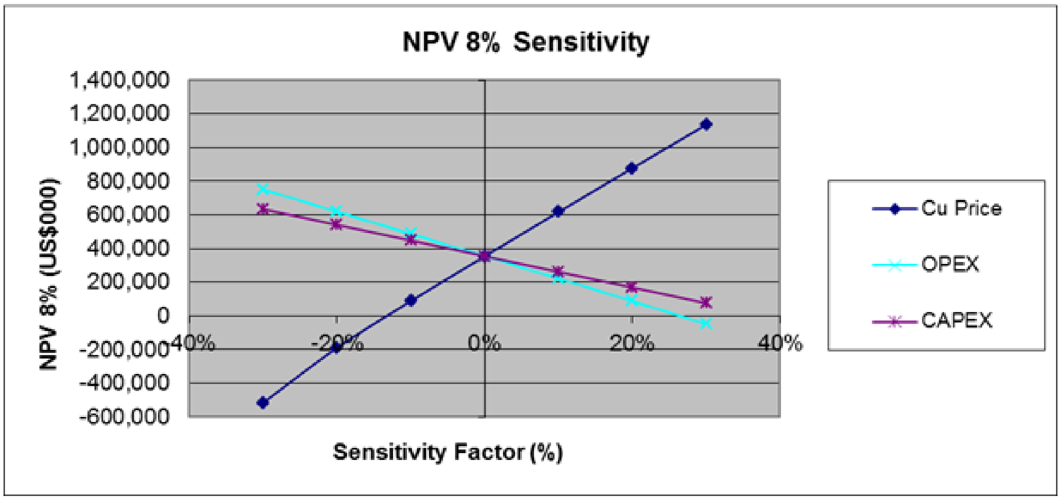
The base case economic analysis (Table 1-9) indicates that the project has a net income after-tax of US$2bn and an after-tax Internal Rate of Return (IRR) of 13.4% with a payback period of 5.4 years. Net present value (NPV) has been calculated to Year -2, Q2 2016.

Table 1-9 and Figure 1-8 compares the base case project financial indicators with the financial indicators for other cases when the sales price, operating cost and the amount of capital expenditure vary from the base case values. The sensitivity study indicates that both the project's NPV and IRR are most sensitive to changes in metal price.

Table 1-9 Financial Analysis

Analysis		NPV @ 0% (US$M)	NPV @ 5% (US$M)	NPV @ 8% (US$M)	NPV @ 10% (US$M)	IRR (%)	Payback Years
Base Case	Pre-tax	3,078.6	1,212.7	683.6	448.5	16.8%	4.9
	After tax	1,977.6	717.0	354.9	192.4	13.4%	5.4
Base Case Sensitivities – After Tax							
Cu Price +10%		2,645.4	1,071.4	617.3	412.8	17.2%	4.1
Cu Price -10%		1,310.2	360.0	89.1	-31.9	9.4%	7.0
Op Costs +10%		1,636.8	536.7	222.2	81.3	11.5%	6.0
Op Costs -10%		2,318.6	896.8	487.1	302.8	15.3%	4.8
Capex Costs +10%		1,871.9	619.8	262.0	102.0	11.7%	6.1
Capex Costs -10%		2,083.4	814.2	447.8	282.8	15.6%	4.5

Figure 1-8 NPV 8% Sensitivity



Yellowhead Mining Inc., July 2014

1.15 Project Execution Plan

Detailed Engineering can begin whenever YMI is ready, but Construction activities will not start in the field until the regulatory authority grants the Mines Act Permit approving initial development (Reference: Project Schedule Section 24).

YMI intends to act as the General Contractor and employ 'sub-contractors' to perform the construction work in an 'Open Shop" environment utilizing the largest pool of construction talent available; CLRA (Construction Labour Relations Association) member companies, CLAC (Christian Labour Association Contractors) and non-union contractors. Contract agreements will protect that strategy and contractors will subscribe to working without dispute in such an environment. An estimated 600 construction persons will be required at the peak (the second construction season) when work is focused on the civil, mechanical and electrical work. A typical modular camp will house the construction workers.

The construction strategy will use the permanent mine production equipment as much as possible to help build the large earthwork structures like haul roads outside of the pit boundaries and the tailings dam. At least one shovel and one drill will operate under temporary power, as main hydro line power will not be available. Three 2000kW generator sets will required to power up the mine equipment and will be installed at the site of the new plant site sub-station. At least two of the units will remain for operations as the emergency generators for the plant. There is approximately 2.5MW of power available on the existing power line near Vavenby. A 25kV power line will be constructed from a junction point on that line up to the plant site to power the camp and other construction activities.

Construction of the starter dam will be completed to the 1,723m elevation by the time of start up. At this point, it will have captured one full freshet and have a filled pond capacity capable of sustaining the fresh water requirements of the operating plant.

Site geotechnical investigations during feasibility determined there are sufficient construction material types available for aggregate materials for general and structural backfill, sand for pipe and electrical cable bedding, filters for the tailings dam, and concrete aggregates. A 150m^3 to 200m^3 per hour concrete batching plant will be located close to the plant site location where the majority of concrete will be required.

Wherever possible the project will make use of pre-engineered buildings, including the concentrator. There are five overhead cranes in the concentrator, with the largest (120t) over the grinding area. It will handle the installation of the SAG Mill. Each section of the wrap around motor for the SAG Mill will weigh approximately 95t.

The project budget will be based on the approved FS capital cost estimate, scope of work, schedule and quality plan, and will form the baseline against which progress and cost will be measured and managed. The Project Manager will be responsible for cost management and reporting and will provide an integrated project management cost (and schedule) database for cost management and reporting.

The project schedule reflects a traditional approach to project execution, with the field construction commencing after engineering tasks are well advanced, in order to accommodate long lead times for the delivery of major equipment. Figure 1-9 summarizes the project schedule milestones.

A list of potential risks will be regularly reviewed and risk mitigation plans implemented through detailed design, construction and commissioning. YMI will ensure that the Environmental Management Plan, Safety Management Plan, and site security are in place before construction begins. Plans will meet the requirements of applicable safety, health, environmental, hygiene and emergency response legislation.

The plant commissioning plan will provide key commissioning definitions, an outline of the facilities to be commissioned, a summary plan, guidelines on risk management, quality assurance and control, and sign off certificates. Proof testing criteria (rates, durations, and quality) will be set well in advance to avoid issues that might delay final acceptance. The completion of a final acceptance certificate at the end of this stage will signify that the completed facility is ready for full ramp-up under Operations ownership. All project execution plan activities are substantially complete when the project meets its production objectives.

Table 1-10: Schedule Milestones

PROJECT OVERVIEW	2016												2017												2018											
	J	F	M	A	M	J	J	A	S	O	N	D	J	F	M	A	M	J	J	A	S	O	N	D	J	F	M	A	M	J	J	A	S	O	N	D
Start Detailed Engineering & Procurement	♦ Start Detailed Engineering & Procurement																																			
Commitments on Long Lead Equipment		♦ Commitments on Long Lead Equipment																																		
Field Construction Starts							• Field Construction Starts																													
Receive Mine Equip for Haul Road & Tailings Starter Dam Construction							• Receive Mine Equipment for Haul Road & Tailings Starter Dam Construction																													
Mine Equipment Assembled											♦ Mine Equipment Assembled																									
BC Hydro Power Required for Commissioning / Bumping Motors																											♦BC Hydro Power Avail for Commissioning									
BC Hydro Power Available																											♦BC Hydro Power Available									
Overall Mechanical Completion (Excl BC Hydro Work)																															♦Mech Completion					

Merit Consultants International Inc., July 2014

1.16 Conclusions

The FS authors are of the opinion that the data is adequate to support a mineral resource and mineral reserve estimate as defined under NI43-101.

The FS authors conclude that a 70,000t/d copper processing facility for the Harper Creek Project is economically viable and can be successfully achieved with proven conventional mining and processing methods under the conditions and assumptions outlined in this report.

Opportunities exist to enhance the project economics including:

- Expansion of the resource which remains open down dip to the north and east;
- Utilization of alternative energy supply including trolley assist and LNG; and
- Optimize the process during the basic and detailed engineering phases.

As such, the FS authors recommend that YMI undertake the following:

- YMI should continue with the environmental studies and permitting efforts now underway (approximately C$3M required).
- YMI should continue with the engineering effort in support of permitting (approximately C$7.5M required).
- Initiate discussions with BC Hydro for increased power supply to the North Thompson Valley (approximately C$0.5M required).

The key risks to successful project development include but are not limited to:

- Permitting;
- Project Financing; and
- Additional power supply to the North Thompson transmission line.

2 INTRODUCTION

Yellowhead Mining Inc. (YMI) is a public company listed on the Toronto Stock Exchange. The corporate office is located at Suite 730-800 West Pender Street, Vancouver, British Columbia V6C 2V6. YMI has a 100% interest in the Project subject to the payment of a 3% NSR royalty capped at C$2.5M (adjusted for inflation), and an additional 2.5% NSR royalty on an estimated 3.3Mt of ore within the NI43-101 resource which is expected to be mined beginning in year nine.

The Project has been the subject of several prior studies, notably:

- “2013 Amended & Restated Technical Report & Feasibility Study for the Harper Creek Copper Project.” Collins, A.J., Dobbs, M., Simpson, R., Brouwer, K., Fox, J., Nilsson, J., January 25, 2013.
- “2012 Technical Report & Feasibility Study for the Harper Creek Copper Project.” Collins, A.J., Dobbs, M., Simpson, R., Brouwer, K., Fox, J., Nilsson, J., March 29, 2012.
- “2011 Technical Report and Preliminary Assessment of the Harper Creek Project for Yellowhead Mining Inc.” Narciso, N., Huang, J., Boyle, J.M., Ghaffari, H., Triebel, K., Teymouri, S., Cameron, M., Greenaway, G., and Donaghue, P., March 31, 2011.

This Technical Report and Feasibility Study (FS) for the Harper Creek Copper Project was prepared to provide Yellowhead Mining Inc. with an up-to-date development plan, capital and operating cost estimate, and financial analysis for the Project.

2.1 TERMS OF REFERENCE

Merit Consultants International Inc. (Merit) managed the FS in compliance with NI43-101 requirements for technical reports. The following independent consultants (FS Team) authored this feasibility study:

Qualified Person	Company
A. Jay Collins, P.Eng.	Merit Consultants International Inc. (Merit)
Mark W. Dobbs, P. Eng.	Allnorth Consultants Ltd. (Allnorth)
Ronald Simpson, P. Geo.	GeoSim Services Inc. (GeoSim)
Daniel Fontaine, P.Eng.	Knight Piésold Ltd. (KP)
John R. W. Fox, P. Eng.	Laurion Consulting Inc. (Laurion)
John Nilsson, P. Eng.	Nilsson Mine Services Ltd. (Nilsson)

2.2 INSPECTION OF PROJECT

Qualified Person, Daniel Fontaine visited the Project site on October 26 to 27, 2011, July 15 to 17, 2012, and September 30, 2012 to view proposed locations of site facilities, potential construction material borrow areas and to review geotechnical site investigation progress.

All other Qualified Persons visited the Project site on July 11 and 12, 2011. The tour of the project included area inspections of the proposed plant site, crusher, borrow pit, tailings dam, access roads, and load out facility. In addition the Vavenby Bridge, Birch Island Bridge, BC Hydro power line, and CN rail line were also inspected. YMI provided complete access to all relevant areas of the Project.

In addition, during the course of the FS, independent visits were made by QPs to further the study of viable infrastructure options including the main access road, Vavenby load-out and administrative facilities and the proposed high voltage power line route.

2.3 SOURCES OF INFORMATION

This report is based in part on the FS Team's corporate knowledge and experience developing copper projects. It also relies on maps, published government reports, YMI letters and memoranda referring to historical work, previously conducted studies, reports and public information, as listed in Section 27 "References". The FS was conducted using recognized international guidelines to determine levels of accuracy and content and was assembled in accordance with the requirements set out in NI43-101. Areas of responsibility of the Qualified Persons are summarized by Section in Table 2-1.

Table 2-1: Sources of Information

Section	Description	Company	Qualified Person	Non-Independent Source
1	Summary	Merit	Jay Collins	
2	Introduction	Merit	Jay Collins	
3	Reliance on Other Experts	Merit	Jay Collins	
4	Project Description & Location	Merit	Jay Collins	
5	Accessibility, Climate, Local Resources, Infrastructure, Physiography	Merit	Jay Collins	
6	History	Merit	Jay Collins	CME Consultants (Chris Naas)
7	Geological Setting & Mineralization	GeoSim	Ron Simpson	CME Consultants (Chris Naas)
8	Deposit Types	GeoSim	Ron Simpson	CME Consultants (Chris Naas)
9	Exploration	GeoSim	Ron Simpson	CME Consultants (Chris Naas)
10	Drilling	GeoSim	Ron Simpson	CME Consultants (Chris Naas)
11	Sample Preparation, Analyses & Security	GeoSim	Ron Simpson	CME Consultants (Chris Naas)
12	Data Verification	GeoSim	Ron Simpson	
13	Mineral Processing & Metallurgical Testing	Laurion	John Fox	
14	Mineral Resource Estimates	GeoSim	Ron Simpson	
15	Mineral Reserve Estimate	Nilsson	John Nilsson	
16	Mining Methods	Nilsson	John Nilsson	
17	Recovery Methods	Laurion	John Fox	
18	Project Infrastructure	Allnorth	Mark Dobbs	
19	Market Studies & Contracts	Merit	Jay Collins	Cliveden A.G. (Richard Latter)
20	Environmental, Permitting, Social & Community Impact	KP	Daniel Fontaine	YMI (Charlene Higgins)
21	Capital & Operating Costs	Merit	Jay Collins	YMI (Alastair Tiver)
22	Economic Analysis	Merit	Jay Collins	YMI (Alastair Tiver)
23	Adjacent Properties	Merit	Jay Collins	
24	Other Relevant Data & Information	Merit	Jay Collins	
25	Interpretation & Conclusions	Merit	Jay Collins	
26	Recommendations	Merit	Jay Collins	
27	References	Merit	Jay Collins	

2.4 Units of Measurement, Currencies

Throughout this document funds are quoted in denomination, i.e., Canadian Dollars (C$) and US Dollars (US$). Measurements are metric, unless noted otherwise. Abbreviations of the units of measurement used in the FS report are provided in Table 2-2.

Table 2-2: Units of Measure

Units of Measure	Abbreviation
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Centimetre	cm
Centipoise	mPa s
Cubic Centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degree Celsius	°C
Diameter	ø
Dollar (US)	US$
Dollar (Cdn)	Cdn$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square	kg/m^2
metre	
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt / Kilovolts	kV
Kilovolt-ampere	kVA
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litre per minute	L/m
Megabytes per second	Mb/s
Megpascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a
Million tones	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimeter	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometer	km^2
Square metre	m^2
Thousand tonnes	kt
Three dimensional	3D
Three dimensional model	3DM
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	A

3 RELIANCE ON OTHER EXPERTS

The FS Team assumed that all the information and technical documents listed in the Reference Section (Section 27) of this report are accurate and complete in all material aspects. While the FS Team reviewed all the available information, it has not audited this work. The FS Team believes qualified professionals performed the work diligently and that the conclusions derived are reasonable. If any significant new information arises, that would have a significant effect on the findings and conclusions contained in this report, the FS Team will revise this report.

The FS Team did not review any licenses, permits, or work contracts, nor perform an independent verification of land title and tenure. The FS Team has not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s), such as royalty agreements, between third parties.

While the FS Team has relied largely on the documents listed in Section 27 for the information in this report, the conclusions and recommendations belong exclusively to the FS Team. The results and opinions outlined in this report are dependent on the aforementioned information being current, accurate and complete as of the date of this report.

The Qualified Persons who prepared this report relied upon information provided by:

- Mr. Alastair Tiver, VP Operations, YMI, representing the issuer, for information related to financial matters (Section 21.2 Operating Cost Estimate, Section 22 Economic Analysis);
- Mr. Chris Naas, P. Geo., CME Consultants Inc., Director of YMI, for matters related to history, geology, exploration, drilling, sample analysis (Section 6 History, Section 7 Geological Setting and Mineralization, Section 8 Deposit Type, Section 9 Exploration, Section 10 Drilling, Section 11 Sample Preparation, Analysis and Security);
- Dr. Charlene Higgins, MSc.,PhD., VP Environment, Community & First Nations Relations, YMI, for matters related to environmental assessment and First Nations (Section20 Environmental Studies, Permitting and Social or Community Impact); and
- Mr. Richard Latter, Cliveden A.G., for advice on matters relating to Marketing and Concentrate off-take terms (Section 19 Market Studies and Contracts).

4 PROJECT LOCATION & DESCRIPTION

4.1 PROJECT LOCATION

The Project is located in the Thompson-Nicola area of British Columbia approximately 150km northeast of Kamloops. Clearwater, the largest community in the Project area is 124km north of Kamloops, along the Yellowhead Highway route (Highway #5). Twenty-seven km further along the Highway is Vavenby, the closest community to the Project area (Figure 4-1).

4.2 PROJECT DESCRIPTION

The Project Area is located on NTS map sheets 82M/12 and 82M/5 and is geographically centered at 51°30'N and 119°48'W (Figure 4-2). It covers a total of 42,636.48 hectares and is comprised of 97 cell claims (41,786.48 ha) and 34 legacy claims (850 ha).

None of the 97 cell claims are subject to any royalties. 3 unconverted legacy claims (mineral tenures 220877, 220878, 220879), and 3 converted legacy claims (mineral tenures 513235, 513237, 513239), are subject to a 2.5% Net Smelter Royalty (NSR) to XStrata. Based on a historical agreement between Noranda and MBI Mining Brokers, XStrata retains the right to back into a 50% working interest in the original Noranda claims. The remaining 31 legacy claims were acquired from Cygnus Mines Ltd. (subsidiary of US Steel Corp.) pursuant to an Option Agreement exercised in July 2010 and are subject to a 3% NSR, capped at C$2.5 million, subject to inflation adjustment (Table 4-1).

Figure 4-1: Project Location (Latitude 51°33' n, Longitude 119°42' w)

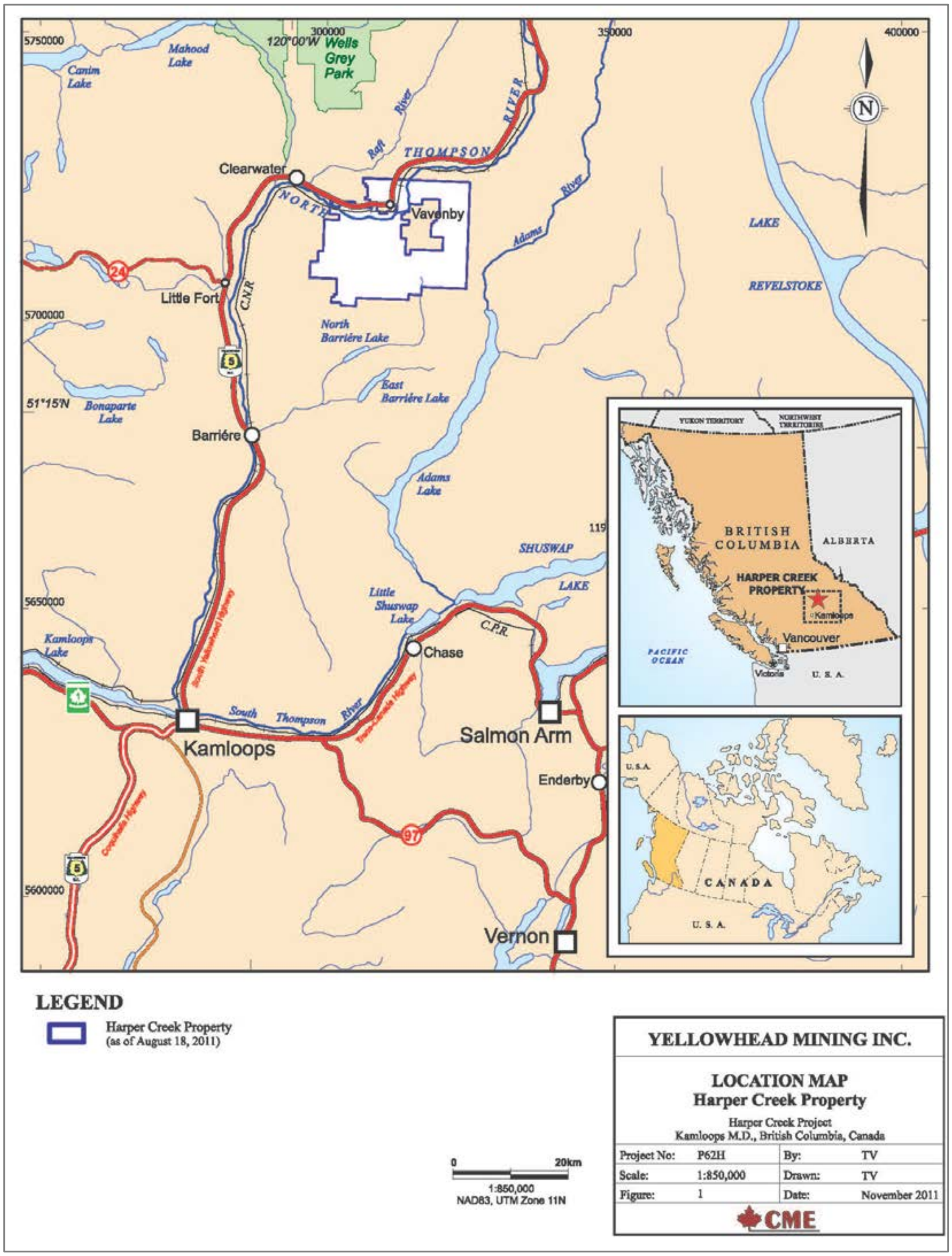


Figure 4-2: Harper Creek Claims Area

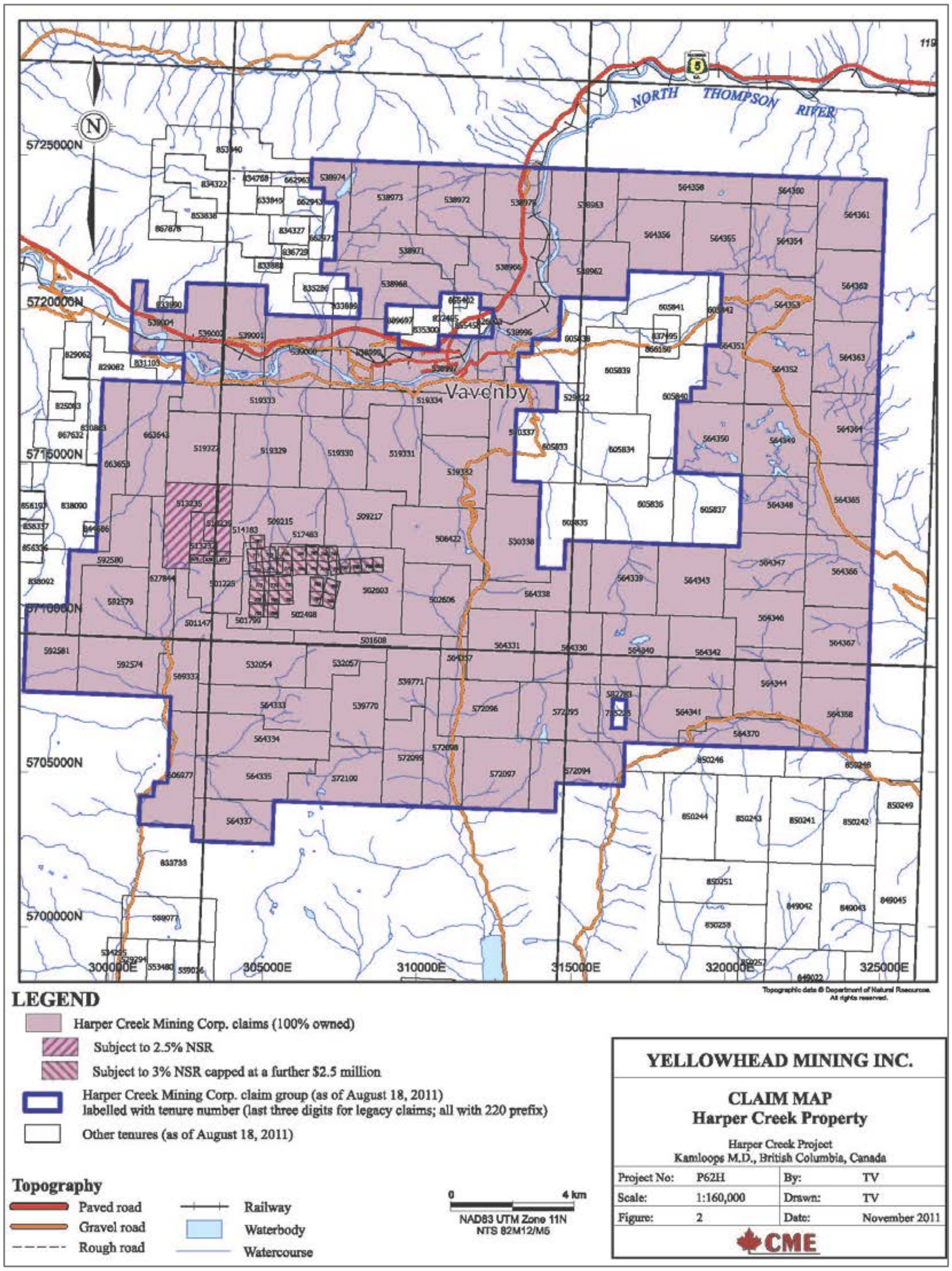


Table 4-1: Harper Creek Mineral Claims

Tenure No.	Area (ha)	Ownership 100%	Good To Date	Tenure Type
220771	25	Yellowhead	2024/Nov/03	Legacy
220772	25	Yellowhead	2024/Nov/03	Legacy
220773	25	Yellowhead	2024/Nov/03	Legacy
220774	25	Yellowhead	2024/Nov/03	Legacy
220775	25	Yellowhead	2024/Nov/03	Legacy
220776	25	Yellowhead	2024/Nov/03	Legacy
220777	25	Yellowhead	2024/Nov/03	Legacy
220778	25	Yellowhead	2024/Nov/03	Legacy
220779	25	Yellowhead	2024/Nov/03	Legacy
220780	25	Yellowhead	2024/Nov/03	Legacy
220781	25	Yellowhead	2024/Nov/03	Legacy
220782	25	Yellowhead	2024/Nov/03	Legacy
220783	25	Yellowhead	2024/Nov/03	Legacy
220784	25	Yellowhead	2024/Nov/03	Legacy
220785	25	Yellowhead	2024/Nov/03	Legacy
220786	25	Yellowhead	2024/Nov/03	Legacy
220787	25	Yellowhead	2024/Nov/03	Legacy
220788	25	Yellowhead	2024/Nov/03	Legacy
220789	25	Yellowhead	2024/Nov/03	Legacy
220790	25	Yellowhead	2024/Nov/03	Legacy
220791	25	Yellowhead	2024/Nov/03	Legacy
220792	25	Yellowhead	2024/Nov/03	Legacy
220793	25	Yellowhead	2024/Nov/03	Legacy
220794	25	Yellowhead	2024/Nov/03	Legacy
220795	25	Yellowhead	2024/Nov/03	Legacy
220796	25	Yellowhead	2024/Nov/03	Legacy
220797	25	Yellowhead	2024/Nov/03	Legacy
220798	25	Yellowhead	2024/Nov/03	Legacy
220799	25	Yellowhead	2024/Nov/03	Legacy
220800	25	Yellowhead	2024/Nov/03	Legacy
220877	25	Yellowhead	2024/Nov/03	Legacy
220878	25	Yellowhead	2024/Nov/03	Legacy
220879	25	Yellowhead	2024/Nov/03	Legacy
220961	25	Yellowhead	2024/Nov/03	Legacy
501147	342.02	Yellowhead	2024/Nov/03	MTO Cell
501225	301.71	Yellowhead	2024/Nov/03	MTO Cell
501608	221.33	Yellowhead	2024/Nov/03	MTO Cell
501799	181.05	Yellowhead	2024/Nov/03	MTO Cell
502498	583.32	Yellowhead	2024/Nov/03	MTO Cell
502603	603.43	Yellowhead	2024/Nov/03	MTO Cell
502606	502.87	Yellowhead	2024/Nov/03	MTO Cell
506422	562.99	Yellowhead	2024/Nov/03	MTO Cell
509215	603.17	Yellowhead	2024/Nov/03	MTO Cell
509217	422.21	Yellowhead	2024/Nov/03	MTO Cell
513235	321.7	Yellowhead	2024/Nov/03	MTO Cell
513237	80.43	Yellowhead	2024/Nov/03	MTO Cell
513239	140.75	Yellowhead	2024/Nov/03	MTO Cell
514183	40.22	Yellowhead	2024/Nov/03	MTO Cell
517483	20.11	Yellowhead	2024/Nov/03	MTO Cell
519327	502.43	Yellowhead	2024/Nov/03	MTO Cell
519329	502.43	Yellowhead	2024/Nov/03	MTO Cell
519330	502.43	Yellowhead	2024/Nov/03	MTO Cell
519331	502.41	Yellowhead	2024/Nov/03	MTO Cell
519332	502.47	Yellowhead	2024/Nov/03	MTO Cell
519333	502.27	Yellowhead	2024/Nov/03	MTO Cell
519334	462.09	Yellowhead	2024/Nov/03	MTO Cell
530337	502.33	Yellowhead	2024/Nov/03	MTO Cell
530338	502.67	Yellowhead	2024/Nov/03	MTO Cell
532054	482.98	Yellowhead	2024/Nov/03	MTO Cell
532057	241.48	Yellowhead	2024/Nov/03	MTO Cell

Tenure No.	Area (ha)	Ownership 100%	Good To Date	Tenure Type
538962	501.81	Yellowhead	2024/Nov/03	MTO Cell
538963	501.61	Yellowhead	2024/Nov/03	MTO Cell
538966	501.81	Yellowhead	2024/Nov/03	MTO Cell
538968	501.88	Yellowhead	2024/Nov/03	MTO Cell
538970	501.61	Yellowhead	2024/Nov/03	MTO Cell
538971	421.49	Yellowhead	2024/Nov/03	MTO Cell
538972	501.61	Yellowhead	2024/Nov/03	MTO Cell
538973	501.61	Yellowhead	2024/Nov/03	MTO Cell
538974	200.63	Yellowhead	2024/Nov/03	MTO Cell
538996	502.01	Yellowhead	2024/Nov/03	MTO Cell
538997	502.14	Yellowhead	2024/Nov/03	MTO Cell
538999	421.77	Yellowhead	2024/Nov/03	MTO Cell
539000	502.11	Yellowhead	2024/Nov/03	MTO Cell
539001	421.73	Yellowhead	2024/Nov/03	MTO Cell
539002	421.73	Yellowhead	2024/Nov/03	MTO Cell
539004	281.14	Yellowhead	2024/Nov/03	MTO Cell
539770	442.84	Yellowhead	2024/Nov/03	MTO Cell
539771	322	Yellowhead	2024/Nov/03	MTO Cell
564330	503.01	Yellowhead	2024/Nov/03	MTO Cell
564331	503.01	Yellowhead	2024/Nov/03	MTO Cell
564333	503.23	Yellowhead	2024/Nov/03	MTO Cell
564334	503.34	Yellowhead	2024/Nov/03	MTO Cell
564335	463.1833	Yellowhead	2024/Nov/03	Mineral Claim
564337	362.5917	Yellowhead	2024/Nov/03	Mineral Claim
564338	502.8196	Yellowhead	2024/Nov/03	Mineral Claim
564339	502.7818	Yellowhead	2024/Nov/03	Mineral Claim
564340	503.0087	Yellowhead	2024/Nov/03	Mineral Claim
564341	442.8144	Yellowhead	2024/Nov/03	Mineral Claim
564342	503.0083	Yellowhead	2024/Nov/03	Mineral Claim
564343	502.7818	Yellowhead	2024/Nov/03	Mineral Claim
564344	503.1017	Yellowhead	2024/Nov/03	Mineral Claim
564346	442.5459	Yellowhead	2024/Nov/03	Mineral Claim
564347	462.5005	Yellowhead	2024/Nov/03	Mineral Claim
564348	402.0263	Yellowhead	2024/Nov/03	Mineral Claim
564349	502.3277	Yellowhead	2024/Nov/03	Mineral Claim
564350	502.3298	Yellowhead	2024/Nov/03	Mineral Claim
564351	461.8769	Yellowhead	2024/Nov/03	Mineral Claim
564352	502.0996	Yellowhead	2024/Nov/03	Mineral Claim
564353	401.5149	Yellowhead	2024/Nov/03	Mineral Claim
564354	501.6872	Yellowhead	2024/Nov/03	Mineral Claim
564355	501.6924	Yellowhead	2024/Nov/03	Mineral Claim
564356	461.5516	Yellowhead	2024/Nov/03	Mineral Claim
564357	120.7333	Yellowhead	2024/Nov/03	Mineral Claim
564358	401.2258	Yellowhead	2024/Nov/03	Mineral Claim
564360	200.6108	Yellowhead	2024/Nov/03	Mineral Claim
564361	501.5948	Yellowhead	2024/Nov/03	Mineral Claim
564362	501.824	Yellowhead	2024/Nov/03	Mineral Claim
564363	502.0528	Yellowhead	2024/Nov/03	Mineral Claim
564364	502.2816	Yellowhead	2024/Nov/03	Mineral Claim
564365	502.5096	Yellowhead	2024/Nov/03	Mineral Claim
564366	502.7379	Yellowhead	2024/Nov/03	Mineral Claim
564367	502.9658	Yellowhead	2024/Nov/03	Mineral Claim
564368	503.1923	Yellowhead	2024/Nov/03	Mineral Claim
564370	322.0876	Yellowhead	2024/Nov/03	Mineral Claim
569337	261.6354	Yellowhead	2024/Nov/03	Mineral Claim
572094	503.3905	Yellowhead	2024/Nov/03	Mineral Claim
572095	483.0856	Yellowhead	2024/Nov/03	Mineral Claim
572096	483.0853	Yellowhead	2024/Nov/03	Mineral Claim
572097	503.417	Yellowhead	2024/Nov/03	Mineral Claim
572098	382.5648	Yellowhead	2024/Nov/03	Mineral Claim
572099	382.5738	Yellowhead	2024/Nov/03	Mineral Claim
572100	463.1775	Yellowhead	2024/Nov/03	Mineral Claim

Tenure No.	Area (ha)	Ownership 100%	Good To Date	Tenure Type
582783	201.2855	Yellowhead	2024/Nov/03	Mineral Claim
592574	503.1198	Yellowhead	2024/Nov/03	Mineral Claim
592579	502.92	Yellowhead	2024/Nov/03	MTO Cell
592580	462.54	Yellowhead	2024/Nov/03	MTO Cell
592581	442.72	Yellowhead	2024/Nov/03	MTO Cell
606977	415.44	Yellowhead	2024/Nov/03	MTO Cell
627844	301.71	Yellowhead	2024/Nov/03	MTO Cell
663643	502.4	Yellowhead	2024/Nov/03	MTO Cell
663658	401.97	Yellowhead	2024/Nov/03	MTO Cell
TOTAL	**42,636.48**			

4.3 Environmental Liabilites

None of the environmental parameters identified to date are considered to have a material impact on the development and operation of the Project. Section 20.4.1 details the Environmental Assessment Review Process in more detail.

4.4 Permits Obtained & To Be Acquired

Table 4-2 provides the list of major permits, licenses, approvals, consents and material authorizations required to occupy, use, construct and operate the Project. However, this list cannot be considered all-inclusive due to the complexity of government regulatory processes and the myriad of minor permits, licenses, approvals, consents and authorizations, and amendments thereto, which will be required throughout the life of the mine.

4.5 Risk Factors

The FS Team is not aware of any risks that may affect access, title, or the right or ability to perform work on the Project.

Table 4-2: Major Permit Approvals

Type	Description	Authority	Legislation	Information Requirements
Certificate	Environmental Assessment	BC	BC Environmental Assessment Act	Terms of Reference, Open House, TEM, Archaeology, Baseline Monitoring, etc.
Permit & Amendment(s)	Work System & Reclamation Program Mine Site; Pre-Production, Bonding, Production	BC	Mines Act	Mine, Reclamation, Materials Handling, ARD Prediction/Prevention; Reclamation Cost Estimate
Approval	Construct & Operate Tailings Impoundment Dam	BC	Mines Act	Tailings Dam Design Report, Water Balance, Closure Plan, Operations Manual
Permit	Work System & Reclamation Program; Gravel Pit / Wash Plant	BC	Mines Act	Aggregate Investigation, Requirements Estimation, Borrow Pit Design, Reclamation Plan
Application	Mining Lease Notice of Intention	BC	Mines Act	Legal Survey
License(s)	Water Use, Water Diversion, Water Storage & Use	BC	Water Act	Plans, Quantities, Design of Diversion Structures
License(s)	To Cut; Mine Site, Tailings Impoundment, Gravel Pit, Access Road, Pump House, Waterline, Power Line	BC	Forest Act	Timber Cruise, Logging Plan
Permit(s)	Road Use, Road Special Use, & Access Road Upgrade	BC	Forest Act	Designs, Terrain Stability, C/L Layout, Archaeology, TEM, Access Management, Road Use Stats, Spill Contingency
Surface Lease & License(s)	Mine Site Facilities Occupation; Load Out, Gravel Pit, Temp Camp, Power Line	BC	Lands Act	Legal Survey Plans, Location Survey, Reclamation Plan
Permit(s) & Approval	Waste Mgmt; Effluent & Sewage Special Waste	BC	Waste Management Act – Special Waste Regulations	Effluent / Waste Characterization (Chemical, Physical); Impact Assessment; WQ Objectives, Sampling, Equip Design & Specs, Flow Rate, Discharge, Volume, Soils Assessment, Plan Design, Disposal, Recycling; Destinations
Permit	Camp Operation; Drinking Water, Sewage Disposal, Sanitation, Food Handling	BC	Health Act	Camp Specifications
Approval	Fuel Storage	BC	Fire Services Act	Design, Storage Capacity, Spill Contingency Plan
Approval	No Shooting Area	BC	Wildlife Act	Not Required by Law, Internal Safety/ Security Issue; Application, Designated Area.
Approval	CEAA	Canada	Canadian Environmental Assessment Act	Comprehensive Study Report- EA Plus: Alternatives, Cumulative Effects, Accidents & Malfunctions
Agreement	Fish Habitat Compensation & Section 35(2) Authorization	Canada	Fisheries Act	Fish Habitat Assessment; Habitat Loss/Gain; Approved Fish Habitat Compensation Plan
Approval	Navigable Waters	Canada	NWPA	River/Creek Hydrology; Intake/Crossing Design
License	Explosives Factory	Canada	Explosives Act	Explosives Details
Resolution	Band Council	Canada	Indian Act	"On-Reserve" Activity Only
License(s)	Radio, Radioisotope; Nuclear Density Gauges/X-Ray Analyzer	Canada	Radio Communication Act; Atomic Energy Control Act	Equipment Details
License / Lot Approval	Business; Load Out, Office, Parking	Municipality		Activity Within Municipal Boundary

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, PHYSIOGRAPHY

5.1 ACCESSIBILITY

Road access to the Project area is via the Yellowhead Highway (Highway #5) north from Kamloops to the Town of Vavenby, then across the North Thompson River and westward along the Birch Island-Lost Creek Forest Service Road (FSR) for approximately 6km to the Jones Creek Forest Service Road (FSR). The Jones Creek FSR provides excellent access to the western part of the Project. The Birch Island-Lost Creek FSR continues westward for 9km to the town of Birch Island, located on the north side of the North Thompson River with access to the Yellowhead Highway.

At approximately the 11.5km mark, the Jones Creek FSR meets the Road 5 Junction. Access to the drilling sites of the Deposit is 2km west of this junction. Access to the tailings management facility (TMF) is 9km southwest of this junction.

Access to the east side of the Project is from Vavenby via the Vavenby Mountain FSR. This road runs along the western side of Chuck Creek for approximately 6km before heading west toward Avery Creek and the southeastern part of the Project. The road meets the Barrière Mountain FSR at approximately 11km. From there, the Saskum Plateau FSR heads southwest to the eastern and central areas of the Project. The Barrière Mountain FSR heads due south, providing access to the most easterly part of the Deposit area.

5.2 CLIMATE

The climate is typical of the central interior of BC, with short warm summers and comparatively mild Canadian winters. The winter season runs from late October to late March. There is significant relief on the Project, and site climatic conditions are dependent on location and elevation.

Temperatures in Vavenby range from a high of +26°C in August to a low of -10°C in January. Precipitation is highest during the months of June and July and lowest during the late winter months of February and March. At the higher site elevations, precipitation falls almost exclusively as snow from November through March, and as rain from June through August. During the shoulder months of April, May, September and October there are often mixed rain and snow conditions. The mean annual wind speed is approximately 1.6m/s, with the wind predominantly blowing from the east-southeast year round, although east-northeast winds are common during the summer. The mean annual relative humidity is approximately 75%.

Table 5-1: Site Meteorological Data (KP Mar 29, 2012)

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Est Long Term Precipitation (mm)	130	75	67	75	57	80	69	66	57	122	117	134	1050
Mean Temp (°C)	-7.3	-7.1	-5.0	-2.1	4.2	7.4	13.1	11.6	7.3	0.7	-4.7	-10.5	0.6
Mean Wind Speed (m/s)	1.8	1.6	2.0	1.7	1.3	1.2	1.2	1.1	1.7	1.6	2.2	1.5	1.6
Mean Relative Humidity %	86.3	83.7	79.2	73.6	68.9	72.0	59.3	66.1	74.3	80.3	85.1	86.9	75.3

5.3 LOCAL RESOURCES

In 2013, the total population of the Thompson-Nicola District numbered 131,166 residents. Kamloops is the largest centre in the area and has a population of 87,705. The nearby towns of Clearwater, Vavenby, Barrière, Blue River, and surrounding district have a combined population of approximately 5,000 (bcstats.gov.bc.ca).

The Project will give employment preference to people from the North Thompson Valley. Vavenby currently serves as the local base for the Project's exploration activities but provides no other significant facilities or services at this time.

5.4 INFRASTRUCTURE

The Yellowhead #5 Highway, the CNR transcontinental main line, and a main BC Hydro 138kVa transmission line all pass approximately 8km due north of the Project area. Other than the existing network of Forest Service Roads, there are no services or utilities currently running to the immediate Project site. The area's established infrastructure preclude the need for any major off-site infrastructure developments to service the Project other than upgrading the existing powerline and adding a new section to the site.

5.5 PHYSIOGRAPHY

The project area is hosted within the Shuswap Highlands characterized by gently sloping upland ridges and flanked by steepened valley slopes. These valleys include the Harper Creek Valley to the west and the Barrière River to the East, with the moderately sloped Thompson River Valley to the north. The elevations of the area range from approximately 1,100m at the floor of the Harper Creek Valley to 1,900m at the ridges surrounding the TMF area. The average elevation of the open pit area and plant site is 1,800m. The area has been glaciated and mountain tops are typically rounded. The project is covered in coniferous forest and has undergone extensive logging in the past.

6 HISTORY

Section 6 summarizes drilling and other exploration work conducted on the Harper Creek deposit before YMI's acquisition of the Project and is largely compiled from the "Technical Report on the Phase VIII Exploration Program of the Harper Creek Property", dated January 31, 2012 prepared for YMI by CME Consultants Inc. (C. Naas). Other reports on the historical work include Reeve (1967, 1968, 1970) and Sanguinetti (1996).

6.1 PRIOR OWNERSHIP (1966 – 2004)

In April 1966, Noranda Exploration Company (Noranda) discovered copper mineralization at the headwaters of Baker Creek and Jones Creek through a program of prospecting and stream sediment sampling. In June 1966, Quebec Cartier (100% wholly owned subsidiary of US Steel) discovered copper mineralization at the headwaters of a tributary of Harper Creek through a similar program of prospecting and stream sediment sampling. Staking by the two companies in 1966 resulted in ground west of Harper Creek tributary belonging to Noranda (Harper Creek Claims) and east of Harper Creek belonging to Quebec Cartier (Hail Claims). The two companies worked independently on their properties from 1967 until mid-1970. In late 1970 the two companies began the joint venture exploration of their contiguous copper deposits until 1974.

The next recorded work program was in 1986, when Aurun Mines Ltd. (Aurun) signed an option agreement with Quebec Cartier (April 22, 1986). On July 16, 1991, Quebec Cartier officially terminated the option agreement with Aurun (at this time insolvent and in receivership).

In 1996, American Comstock purchased the Noranda claims and acquired an option on the Quebec Cartier claims (now held by Cygnus Mines Limited, but still a wholly owned subsidiary of US Steel). Eventually American Comstock dropped the Cygnus option, but maintained ownership of the Noranda group of claims.

In 2005, YMI obtained control of the Harper Creek claims through a series of claim staking, purchase and option agreements. YMI located the historical core from the Noranda drilling campaigns from which selected holes were logged and sampled with the goal of verifying the historical analytical copper results (Naas, 2006). In 2006, YMI undertook the first phase of field exploration on the Harper Creek claims.

6.2 EXPLORATION & DEVELOPMENT BY PREVIOUS OWNERS (1967 – 1996)

In 1966 Noranda and Quebec Cartier discovered copper mineralization at the headwaters of Baker Creek and Jones Creek respectively through a program of prospecting and stream sediment sampling which indicated higher levels of cadmium, copper, aluminum and iron in the stream.

In 1967, Noranda undertook a soil orientation survey followed by the establishment of a survey grid with an east-west oriented baseline and north south oriented cross lines at 243.84m spacing. Soil samples were collected at 60.96m intervals along the cross lines. During 1968 and 1970, extensions to the initial grid with subsequent soil sampling were made to the south and west, as well as in-fill at 121.92m cross line spacing.

In 1967, Quebec Cartier established a grid of 13 lines totaling 128.72km in the area broadly defined by the results of the silt-sampling program. A total of 2,500 soil samples from the B-horizon were collected from this grid and analyzed for copper and zinc. In addition, a local logging road on the Quebec Cartier ground was extended 4.8km to the western side of the Hail claims to allow 7 trenches to be excavated. A total volume of 1,524m^3 was excavated from which 31 channel samples were collected. Samples were taken along 3.048m lengths of bedrock. In that same year, the Company performed a ground magnetic survey. Approximately 9,000 observations of the vertical component of the total field were made with a fluxgate magnetometer at 15.24m intervals totaling 137.16km. All readings were corrected for daily and diurnal variations.

In the fall of 1970, a soil orientation survey on the Quebec Cartier grid revealed check sampling was warranted for comparisons of results from the two grid systems. In 1971, Noranda re-sampled a portion of the Quebec Cartier grid. All soil samples were analyzed for copper and zinc and samples from two lines were analyzed for molybdenum.

Between 1967 and 1971 Noranda undertook geophysical surveys comprising a total of 9 lines (11.5km) of magnetometer, 28 lines (51.49km) of VLF-EM and 8 lines (57.92km) of induced polarization. The IP survey was conducted as a test survey when drilling in the area had been completed prior to the survey. The remaining surveys were carried out on the Noranda ground before drilling and trenching, except for one VLF-EM survey that was carried out on the Quebec Cartier ground during 1971.

The magnetometer used by Noranda included an ES-180 magnetometer, which is a dip needle with a micrometer-adjusted compensating magnet. Accuracy at best would be ±50 gammas. Throughout the survey the vertical component of the magnetic field was read. The IP equipment included an IPL Noranda Model 19 battery powered unit and was used to carry out dipole-dipole electrode configuration. Both electrode spread and dipole separation maintained constant 60.96m spacing. Frequencies used were 0.3 Hz and 10 Hz.

The VLF-EM survey carried out utilized Ronca EM-16 and Crone Radem equipment. The initial survey was conducted in 1967 using the EM-16 receiver and utilizing the signal transmitted from Cutler, Maine. This work was restricted to the Noranda property. During 1971, a VLF-EM survey was carried out on the Quebec Cartier ground using the Radem receiver and utilizing a signal transmitted from Seattle, Washington. In addition, the dip angle field data was manipulated using D. Fraser's filtering technique.

In 1972, exploration expanded out from the main Deposit into the southwest corner of the project area (Area A), the southern part (Area B), and the northern part (Area C). Work in Area A consisted of detailed stream sediment sampling, reconnaissance geological mapping and 16 km of grid establishment, which was soil sampled and surveyed by VLF-EM. An IP survey was run on four lines that returned anomalous copper-in-soil results. Work in Area B consisted of geological mapping and in Area C it consisted of 20.8km of grid establishment, which was soil sampled and surveyed by VLF-EM. Six alternating lines were surveyed by IP.

In 1973, groundwork shifted back to the Deposit, as newly constructed logging roads opened up other areas of the deposit. A total of 20.8km of grid was cut or reestablished from which 22.4km of VLF-EM surveying was undertaken.

6.2.1 *Drilling By Previous Owners (1967 – 1996)*

Quebec Cartier commenced drilling in 1967. They completed 6 NQ sized diamond drill holes totaling 546.19m at the "K" anomaly. A total of 174 samples were collected at 3.048m intervals for copper analysis. No drilling was undertaken in 1968 by Quebec Cartier, but resumed in 1969 with 27 BQ drill holes totaling 4,737.21m from which 1,529 samples were collected for copper analysis. Three of these drill holes targeted the "M" anomaly. No further drilling was conducted on the Quebec Cartier ground until the joint venture exploration program commenced in late 1970.

In 1968 Noranda commenced drilling 2,102.17m in 17 holes (NH-1 to NH-17) and analyzing 710 samples for copper. In 1969, 13 drill holes (NH-18 to NH30) totaling 1,733.56m were drilled with 532 samples analyzed for copper. In 1970, 57 drill holes (NH31 to NH81 and NH-90 to NH-95) totaling 8,314.59m drilled from which 2,504 samples were collected and analyzed for copper. Some samples were selected for additional elements such as zinc (88 samples), lead (12 samples), gold (1 sample) and silver (2 samples). Composite lengths were created for selected drill holes, which were analyzed for copper, gold and silver. Data or maps showing locations of drill holes NH-82 to NH-89 have not been located.

From late 1970 onward, drilling on the Noranda and Quebec Cartier properties was undertaken through a joint venture program, managed by Noranda. In 1970, 12 holes (J-1 to J-12) totaling 2,328.69m were completed and 618 samples were collected. In 1971, 27 holes (J-13 to J-39 and J-5Ext.) totaling 5,593.67m resulted in 1,678 samples. All samples were analyzed for copper. Drill logs for holes J-37 and J-39, which tested the "M" anomaly, did not report the collection of any core samples.

Diamond drilling continued in 1972 and the program completed 4 drill holes totaling 456.74m (J-40 to J-43). In 1973 a limited exploration program of VLF-EM (14 miles/22.53 km) and 5 diamond drill holes totaling 625.45m was undertaken (J-44 to J-48). Drill log J-44 did not report the collection of any core samples. Thirteen samples were collected from the remaining 4 drill holes.

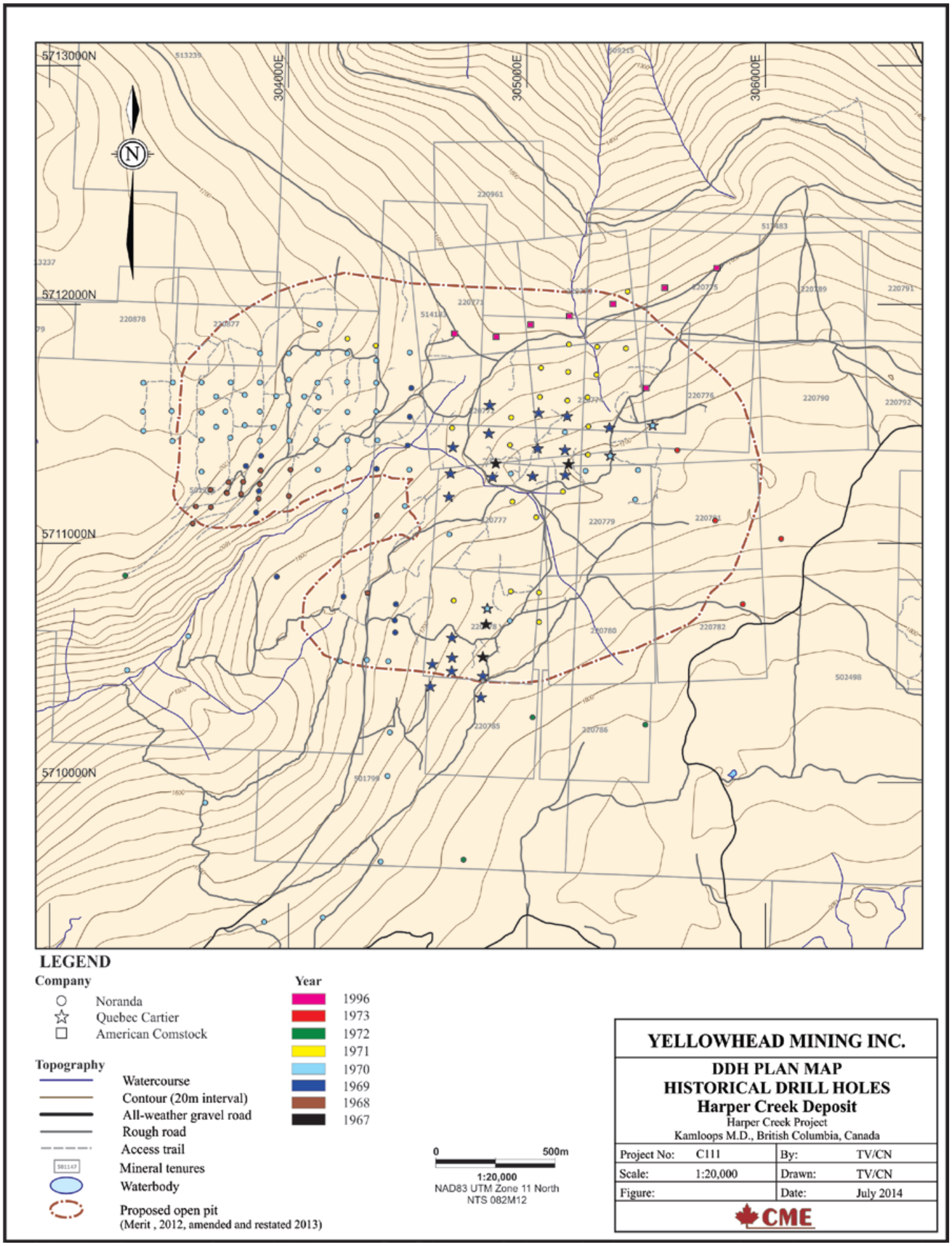
In 1974, only geological mapping of newly cut logging roads and relogging of historical drill core were undertaken.

In April 1986 Aurun investigated the potential for high grade Cu-Mo deposits, checked the possible presence of precious metal content of the massive sulphide layers of the deposit, determined significance of titanium-bearing minerals and investigated leaching possibilities of the low grade copper mineralization. Information was gathered by sampling historical trenches and sampling of selected historical drill core. A total of 14 surface samples and 39 drill core samples were collected and analyzed for gold and silver and TiO_2. Trench samples were collected from Trenches E, F, G, H, I, J, T1, T2, T8, T13, T14 and T17. Drill core samples were collected from J-16 (4 samples), J-8 (8 samples), J-13 (8 samples), J-15 (2 samples), J-17 (8 samples), J-5 (3 samples), J-27 (8 samples) and J-7 (3 samples).

In 1996, American Comstock completed a total of 2,847.44m of NQ diamond drilling in 8 holes. A total of 686 samples were analyzed for copper, molybdenum and silver. Of the 8 holes drilled, 7 holes were completed on the Cygnus portion of the project and 1 hole (96-6) was completed on the Noranda property.

Figure 6-1 summarizes the historical drilling work, representing exploration for the period 1967 – 1996.

Figure 6-1: Historical Drill Holes Harper Creek Deposit (CME)



6.3 HISTORICAL MINERAL RESOURCE & MINERAL RESERVE ESTIMATES

The historical resource estimates referred to in Section 6.3 in part refer to work completed by parties other than YMI and estimates prepared by consultants not engaged by YMI. These historical estimates are not current and most of them do not meet NI 43-101 Definition Standards. All of the historical estimates should not be relied upon. **A new resource estimate prepared in accordance with NI 43-10 Definition Standards is set out in Section 14 of this report.**

In 1986, Aurun commissioned a pre-feasibility study by Phillips Barratt Kaiser Engineering Ltd. (Table 6-1).

Table 6-1: Mineral Resource & Mineral Reserve Estimate PBK (1988)[1]

Geological Resource				
Zone	**Tonnage (t)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
East	42,500,000	0.39	0.043	2.4
West	53,500,000	0.42	0.047	2.6
Total	**96,000,000**	**0.41**	**0.045**	**2.5**
Mineral Resource				
Zone	**Tonnage (t)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
East	42,200,000	0.34	0.037	2.1
West	23,140,000	0.40	0.044	2.4
Total	**65,340,000**	**0.36**	**0.040**	**2.2**

[1]*Historical value only*

In 2007, and prior to the completion of the Phase IV Exploration Program by YMI, Scott Wilson Roscoe Postle Associates Inc. (SWRPA) prepared a 43-101 Mineral Resource Estimate and Technical Report for the Harper Creek Project (Rennie and Scott, 2007). The report utilized drill holes up to HC07-20. Following an additional 12,655.95m of diamond drilling in 34 holes (HC07-21 to HC08-53) the estimate was updated and reported in NI 43-101 Resource - March 2008 (*writ. comm*. Rennie, 2008). Table 6-2 summarizes the two estimates.

Table 6-2: Rennie & Scott 2007 / Rennie, Writ.Comm. 2008[1]

Date	Cut-off Grade (% Cu)	Tonnage (t)	Grade (% Cu)	Cu (t)
Indicated				
Nov-07	0.2	450,900	0.32	1,457,800
Mar-08	0.2	538,000	0.32	1,735,000
Inferred				
Nov-07	0.2	142,000	0.33	463,900
Mar-08	0.2	65,000	0.34	221,000

[1]*Historical value only*

In 2010, SWRPA carried out a third resource estimate (Rennie and Scott, 2010) to provide an updated estimation of the copper resource with the inclusion of a further 23 diamond drill holes (HC08-54 to HC08-75) completed after the 2008 resource estimate (Table 6-3).

Table 6-3: Resource Estimate (after Rennie & Scott, 2010)[1]

Cut-Off (% Cu)	Tonnage (kt)	Cu Grade (%)	Contained Cu (M lb)
Indicated			
0.5	39,800	0.58	509
0.4	102,000	0.49	1,100
0.3	256,000	0.40	2,260
0.2	569,000	0.32	4,010
0.1	973,000	0.25	5,360
Inferred			
0.5	6,810	0.59	88.6
0.4	14,900	0.51	168
0.3	30,100	0.43	285
0.2	62,700	0.33	456
0.1	102,000	0.26	585

[1]*Historical value only*

1. *CIM definitions were followed for Mineral Resources*
2. *Base case Mineral Resources estimated at a cut-off grade of 0.20% Cu*
3. *Block size 15mx15mx5m was used*
4. *Estimate is constrained by a pit shell*
5. *Avg bulk density 2.79 t/m^3*

Wardrop completed a resource estimate in early 2011 that included gold and silver for the first time (Narciso *et al*, 2011). Data for this estimate included YMI drilling up to Hole HC08-75 and re-sampled historical holes up to March 31, 2011 (Table 6-4).

Table 6-4: Mineral Resource Estimate (Wardrop PEA April 1, 2011)[1]

Cut-Off Grade (% Cu)	Resource Tonnage (kt)	Cu Grade (%)	Au Grade 1 (g/t)	Ag Grade 1 (g/t)
Measured				
0.5	3,701.7	0.56	0.055	1.66
0.4	12,391.7	0.47	0.046	1.55
0.3	38,632.4	0.38	0.039	1.37
0.2	89,992.9	0.30	0.033	1.18
0.1	146,402.4	0.24	0.029	1.04
Indicated				
0.5	25,128.2	0.58	0.065	1.54
0.4	72,464.5	0.49	0.051	1.36
0.3	190,133.7	0.39	0.040	1.22
0.2	442,071.1	0.31	0.032	1.06
0.1	847,302.0	0.23	0.026	0.91
Inferred				
0.5	3,316.1	0.56	0.051	1.81
0.4	14,116.7	0.46	0.043	1.65
0.3	47,036.7	0.38	0.037	1.49
0.2	117,236.9	0.29	0.032	1.32
0.1	231,239.0	0.22	0.027	1.09

[1]*Historical value only*

Geosim completed a resource estimate in December 2011 for the Technical Report and Feasibility Study for the Harper Creek Copper Project issued March 29, 2012 (subsequent amendment to the FS dated January 25, 2013).

Table 6-5: Mineral Resource Estimate (Geosim Services Inc. Dec 2011)[1]

Harper Creek Copper Project Mineral Resource Summary - GeoSim Services Inc. December 2011				
Cut-off Grade (% Cu)	**Tonnes (000`s)**	**Cu Grade (%)**	**Au Grade (g/t)**	**Ag Grade (g/t)**
Measured				
0.1	590,790	0.24	0.028	1.1
0.2	348,515	0.31	0.034	1.3
0.3	149,694	0.39	0.044	1.5
0.4	56,753	0.48	0.056	1.7
0.5	18,925	0.58	0.074	2.0
Indicated				
0.1	928,207	0.22	0.026	1.1
0.2	466,482	0.28	0.030	1.3
0.3	144,943	0.38	0.040	1.5
0.4	44,638	0.47	0.051	1.7
0.5	11,687	0.57	0.065	1.9
Measured + Indicated				
0.1	1,518,997	0.23	0.027	1.1
0.2	814,997	0.29	0.032	1.3
0.3	294,637	0.39	0.042	1.5
0.4	101,391	0.48	0.054	1.7
0.5	30,612	0.58	0.071	2.0
Inferred				
0.1	155,251	0.22	0.027	1.1
0.2	80,169	0.30	0.033	1.4
0.3	31,635	0.39	0.037	1.5
0.4	11,360	0.47	0.044	1.8
0.5	3,017	0.57	0.054	2.0

[1] *Historical value only*

6.3.1 *Production from Project*

To date there has been no production from the Project.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Project is located within structurally complex, low-grade metamorphic rocks of the Eagle Bay Assemblage, part of the Kootenay Terrane on the western margin of the Omineca Belt in south-central BC (Fig. 7-1). The Eagle Bay Assemblage is flanked by high-grade metamorphic rocks of the Shuswap Complex immediately to the east, also part of the Kootenay Terrane, and by rocks of the Fennell Assemblage immediately to the west (Naas, 2013). Additionally, the Project lies within the Cretaceous Bayonne plutonic belt (Logan, 2002) represented by two large batholiths, the Baldy batholith to the south and the Raft batholith to the north (Fig. 7-1).

Regional unit names used in this report are referenced from Schiarizza and Preto (1987).

7.1.1 *LOWER CAMBRIAN TO MISSISSIPPIAN EAGLE BAY ASSEMBLAGE*

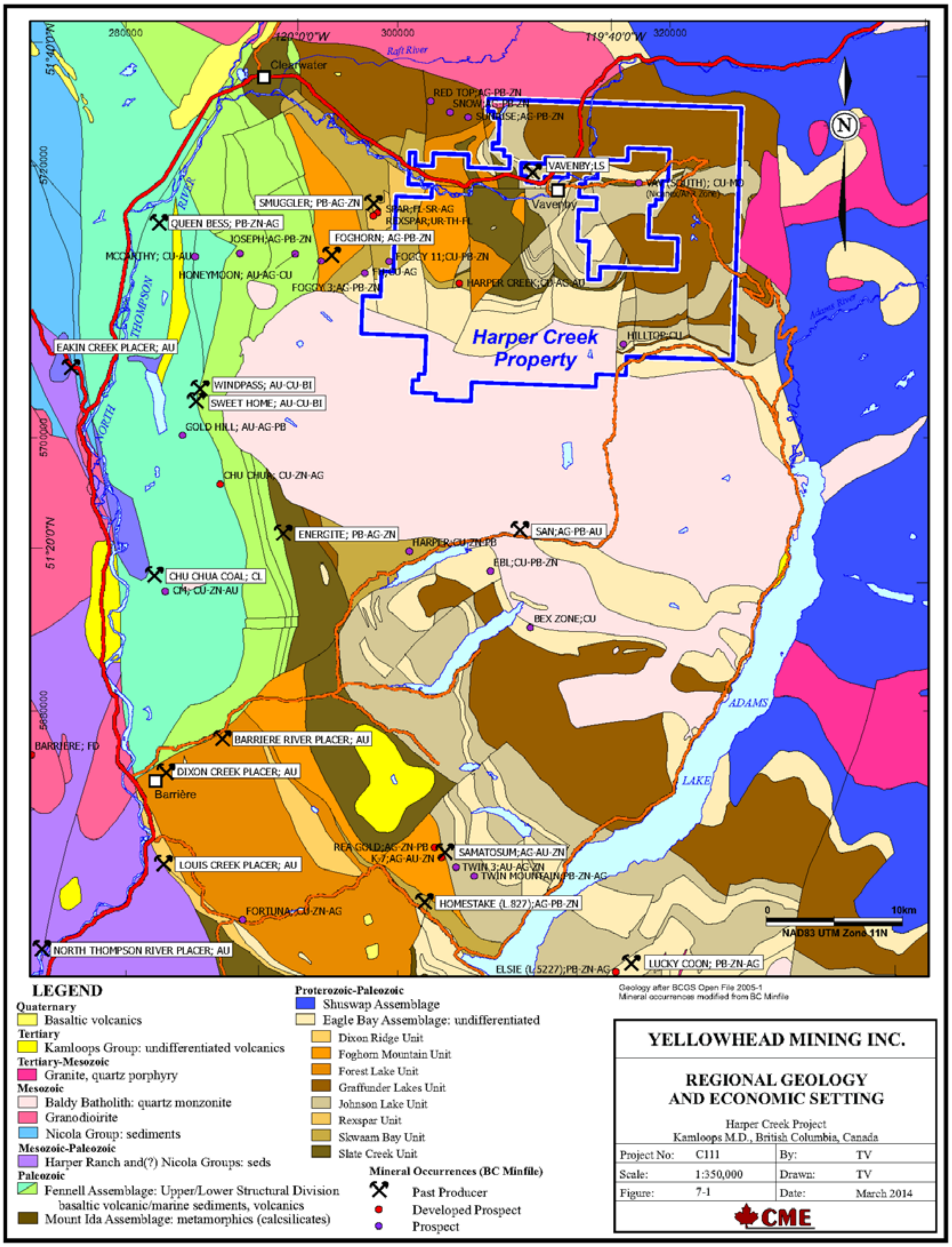
The Eagle Bay Assemblage incorporates Lower Cambrian to Mississippian sedimentary and volcanic rocks deformed and metamorphosed during the Jurassic-Cretaceous orogeny (Schiarizza and Preto, 1987).

The Eagle Bay Assemblage is divided into four northeast-dipping thrust sheets that collectively contain a succession of Lower Cambrian rocks overlain by a succession of Devonian-Mississippian rocks. The Lower Cambrian (and possibly Late Proterozoic) rocks include quartzites, grits and quartz mica schists (Units EBH and EBQ), mafic metavolcanic rocks and limestone (Unit EBG), and overlying schistose sandstones and grits (Unit EBS) with minor calcareous and mafic volcanic units. These older units are overlain by Devonian-Mississippian succession of mafic to intermediate metavolcanic rocks (Units EBA and EBF) intercalated with and overlain by dark grey phyllite, sandstone and grit (Unit EBP) (Schiarizza and Preto, 1987).

The Deposit is hosted by Unit EBA of the Devonian-Mississippian succession (Schiarizza, 1986a, 1986b; Schiarizza and Preto, 1987). To the south Unit EBA is overthrusted by the Lower Cambrian greenstones, chloritic phyllites, quartzitic units and orthogneiss of Unit EBG and to the north by dominantly metasedimentary rocks of Unit EBP (Schiarizza, 1986a; Schiarizza and Preto, 1987).

According to Bailey *et al* (2001), the Devonian volcanic rocks of the Eagle Bay Assemblage (EBA and EBF) belong to bimodal basalt-rhyolite association with alkalic affinity corresponding to a rifted continental marginal setting.

Figure 7-1 Regional Geology and Economic Setting (1:350,000)



7.1.2 *DEVONIAN TO PERMIAN FENNELL FORMATION*

The Fennell Formation is located to the northeast of the Project and is comprised of Devonian to Permian oceanic rocks of the Slide Mountain Terrane. These units have been tectonically emplaced over the Mississippian rocks of the Eagle Bay Assemblage early in the Mesozoic. The Fennell Formation comprises two major divisions. The lower structural division is a heterogeneous assemblage of bedded chert, gabbro, diabase, pillowed basalt, sandstone, quartz-feldspar-porphyry rhyolite and intraformational conglomerate. The upper division consists almost entirely of pillowed and massive basalt, with minor bedded cherts and gabbros. The Fennell Formation is thought to be the deep oceanic basin, distal equivalent to the Eagle Bay Assemblage through striking similarities in sandstone units found in both formations with the sandstone of the Fennell Formation hypothesized as being derived from the sandstones of the Eagle Bay Assemblage. The Devonian quartz-feldspar-porphyry rhyolites found in the Fennell Formation and the Devonian felsic volcanic rocks found in the Eagle Bay Assemblage also bear resemblance to each other and are hypothesized as being an expression of the same igneous activity. As such, the Fennel succession is inferred to comprise an imbricated marginal basin suite that was originally not far removed from the Eagle Bay terrane (Schiarizza and Preto, 1987).

7.1.3 *MID-CRETACEOUS BAYONNE PLUTONIC BELT*

The Mid-Cretaceous Bayonne Plutonic Suite forms a belt that extends roughly north-south and consists of mostly peraluminous, subalkalic hornblende-biotite granodiorite and highly fractionated two-mica granites, aplites and pegmatites (Logan, 2002). In the Project area, this plutonic suite is represented by the Baldy batholith to the south and the Raft batholith to the north.

The Baldy batholith is a west-trending multiphase pluton which covers approximately 650 square kilometres (Schiarizza and Preto, 1987; Calderwood *et al*, 1990; Logan, 2000, 2001). It intrudes Proterozoic to mid-Paleozoic Kootenay Terrane metasedimentary and metavolcanic rocks and postdates most of the penetrative deformation in the area. The pluton incorporates potassium-feldspar megacrystic hornblende-biotite quartz monzonite, biotite monzogranite to granite and biotite-muscovite granite. As summarized by Logan (2000), the main part of the Baldy batholith is interpreted to have the Mid- to Late-Cretaceous age of crystallization of some 129±4 to 99.7±4 Ma. However, quartz monzodiorite of the Honeymoon stock located on the southern margin of the batholith has yielded a Middle Jurassic U-Pb date of 161±7.8 Ma (Logan, 2001).

The Raft batholith is an elongate granitic pluton that extends for about 70 kilometres in a west-northwest direction, and cuts across the boundaries between the Kootenay, Slide Mountain and Quesnel Terranes (Schiarizza *et al*, 2002). It is composed mostly of hornblende-biotite granodiorite to monzogranite intruded by dykes of pegmatite, aplite and quartz-feldspar porphyry. The southern Raft batholith margin dips southward in exposures of deeper structural levels (Okulitch, 1979). The main part of the Raft batholith is interpreted to have the late Early Cretaceous age of crystallization of some 105.5±0.5 Ma (Schiarizza and Boulton, 2006). However, a much older date (168+14/-12 Ma) was obtained from a granodiorite sample (Calderwood *et al*, 1990). Therefore, similar to the Baldy batholith, it is possible that the predominantly Mid-Cretaceous Raft batholith encompasses some older, Middle Jurassic phases (Schiarizza and Boulton, 2006).

7.2 Regional Mineralization

The Eagle Bay Assemblage hosts numerous polymetallic massive sulphide deposits, found mainly within Devonian felsic volcanic rocks (Figure 7-1). These deposits are formed in a volcanic arc environment in response to eastward subduction of a paleo-Pacific ocean (Höy and Goutier, 1986; Höy, 1999; Bailey *et al*, 2000). The general characteristics of these massive sulphide deposits allow the more important ones to be grouped into several types (Schiarizza and Preto, 1987), such as silver-lead-zinc stratabound massive sulphides within metasedimentary rocks (Units EBG and EBQ), copper-zinc-cobalt volcanogenic massive sulphides (Fennell Formation) and gold-silver-zinc-lead-copper-barite volcanogenic massive sulphides (Units EBA and EBF).

A variety of mineral occurrences are related to the Baldy batholith (Schiarizza and Preto, 1987; Cathro and Lefebure, 2000; Logan, 2000, 2001). According to Logan (2000, 2001), copper, copper-molybdenum porphyry and base metal polymetallic vein showings are associated with the hornblende-biotite granite phase of the pluton. The muscovite-biotite granite is associated with pegmatites, aplites and porphyry molybdenum mineralization. Areas encompassing the known intrusive-related deposits extend from the mainly steep-dipping contacts of the Baldy batholith at least as far as 7.5km (Logan, 2001).

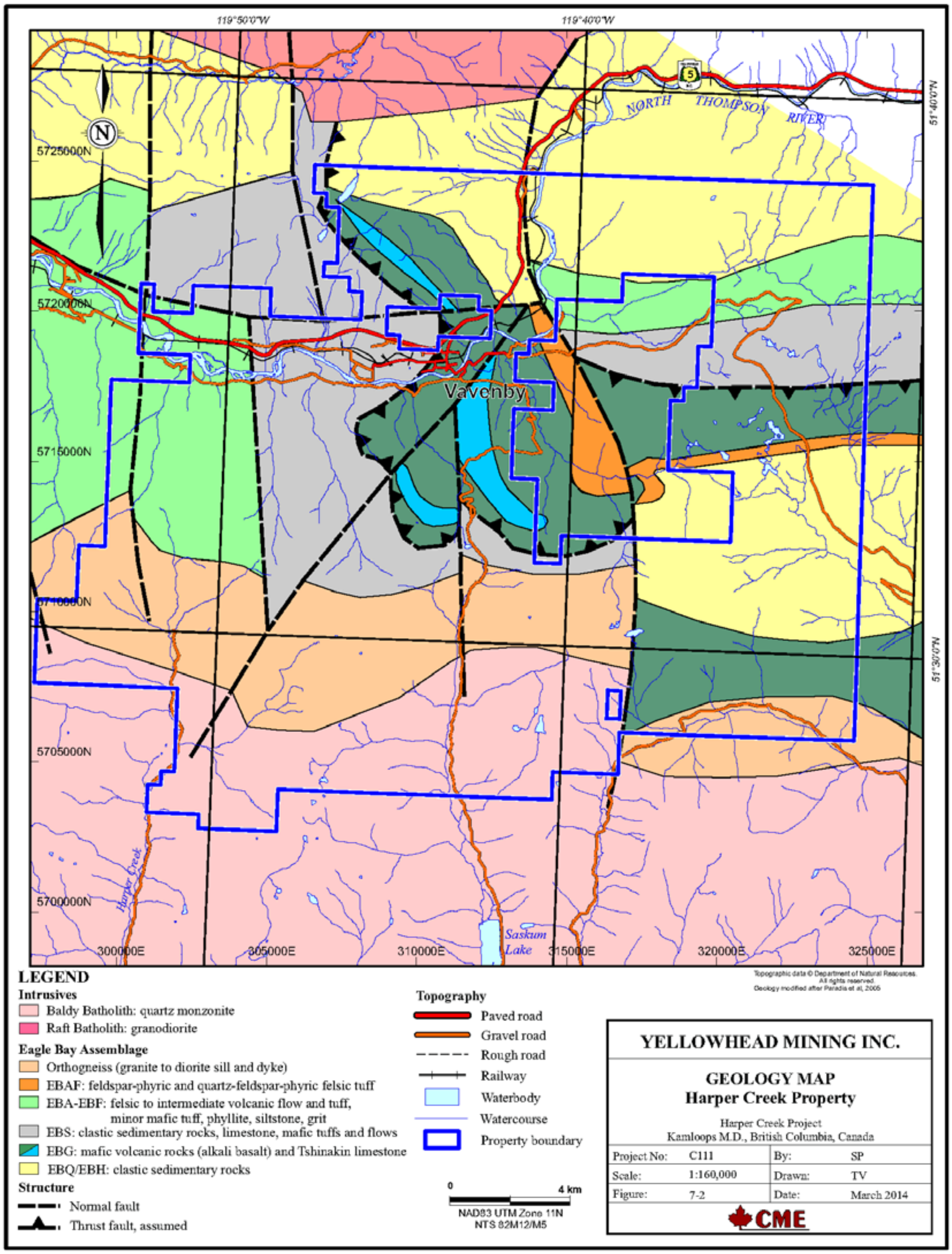
7.3 Project Geology

The Project is primarily underlain by rocks of the Eagle Bay Assemblage with a lithological succession interpreted as belonging to the Dgn, EBQ, EBA, EBF and EBG units of the Eagle Bay Assemblage (Schiarizza and Preto, 1987). This succession consists of a series of orthogneiss, metasediments, metavolcanics and metavolcanic clastics respectively, structurally overlain by the Tshinakin limestone unit belonging to unit EBG. The nature of the structure in the region is a complex sequence of polyphase deformation consisting of a sequence of thrust faulting, intrusion-related folding and faulting, strike-slip and normal faulting all of which impose a complex alteration and metamorphic fabric on the rocks (Naas, 2012a).

At the southern edge of the Project, the Eagle Bay succession is cut by the mid-Cretaceous Baldy batholith. There is a late epidote alteration event related to this intrusion (Armstrong and Hawkins, 2009).

A simplified project-scale geology map modified from Paradis *et al* (2006) is presented in Figure 7-2.

Figure 7-2 Geology Map, Harper Creek Project (1:160,000)



7.4 PROJECT MINERALIZATION

The principal area of mineralization on the Project is the Harper Creek deposit (the "Deposit"). The Deposit is separated by the northeast trending Harper Creek Fault into a West Domain and East Domain (Figure 7-3). In the West Domain, chalcopyrite mineralization is primarily observed in three copper bearing horizons. The upper horizon ranges from 60m to 170m in width and is continuous along an east-west strike for some 1,320m, dipping approximately 30º north. Mineralization within this horizon is hosted within felsic and mafic volcanics and volcaniclastic packages of rocks. The middle horizon is not as well developed and is often fragmented. It is primarily hosted within a graphitic and variably silicified package of rocks that range from 30m to 40m in width at the western extent, increasing up to 90m locally eastward, then gradually appearing to blend into the upper horizon. Of the three horizons, this horizon contains strong to intense silicification and localized fracture-fill tension fractures of mineralization. The lowest or third horizon is even less defined mainly due to a lack of drill intersections. It is commonly hosted within mafic to intermediate volcaniclastics and fragmental rocks and can range from 30m to 90m in width though more typical intersections are at the 30m range. These horizons typically contain foliation-parallel wisps and bands as the dominant style of sulphide mineralization and hosted within felsic and mafic metavolcanics and metavolcaniclastics (Naas, 2013).

In the East Domain, mineralization is characterized by high angle, discontinuous, tension fractures of pyrrhotite, chalcopyrite±bornite which is frequently associated with quartz carbonate gangue. This style is commonly observed within, but not limited to, the metasedimentary rocks and areas of increased pervasive silicification. Mineralization is not selective to individual units and is frequently observed to transgress lithological contacts throughout the area. Due to multiple east-west trending and northward dipping interpreted thrust faults (or possible reverse faults), isolating mineralized horizons in this area has proven difficult. At the near surface areas in the south and down dip to the north, mineralization widths typically range from 120m to 160m. In the central area of the East Domain where thrust/reverse fault stacking has been interpreted, mineralization thicknesses typically range from 220m to 260m with local intersections of up to 290m. Generally the mafic metavolcanics and coarse-grained quartz-rich metasedimentary rocks contain higher grade copper mineralization (Naas, 2013).

Little is known of the mineralization located outside of the Deposit, as the primary focus of exploration has been on the deposit proper. Known showings include M Anomaly, Avery and Northwest. Like the Deposit, the principal mineralization of these areas is copper, however at the Northwest Showing, barite and zinc are noted (Naas, 2013).

7.5 DEPOSIT GEOLOGY AND MINERALIZATION

7.5.1 *GEOLOGICAL LITHOLOGIES AND PACKAGES*

The Deposit is hosted by metamorphic rocks of the Eagle Bay Assemblage. The pervasive alteration and structural deformation of the host rocks has made confident identification of protolith extremely difficult.

The metamorphic rocks have been grouped into 4 primary lithologies, namely phyllites (unit 7), schists with no or minimal quartz content (unit 8), schists with quartz content and/or quartz eyes (unit 9) and orthogneiss (unit 10). These units comprise close to 90% of all drilled lithologies of the Deposit, with the quartz/quartz-eye schists comprising almost half of these lithologies. The phyllites and schists have been further subdivided based on mineral or textural characteristics.

Phyllites of unit 7 have been subdivided into: graphite (unit 7a); sericite-chlorite (7b); calcareous chlorite-sericite (unit 7c); and, sericite-chlorite-quartz (unit 7d). Unit 7d, the sericite-chlorite quartz phyllites is the most common phyllite subunit identified through drilling.

Schists of unit 8 have been subdivided into: sericite-chlorite (unit 8a); sericite-chlorite-fuchsite (unit 8b); and, chlorite sericite fragmental (unit 8c). Of these, the sericite-chlorite schist (unit 8a) is the most common subunit encountered in drilling.

Schists of unit 9 have been subdivided into: sericite hornblende-quartz-feldspar (unit 9a), sericite-chlorite-quartz (unit 9b), sericite-chlorite-quartz-feldspar (unit 9c); sericite-augen quartz (unit 9d) and siliceous chlorite-sericite quartz (unit 9e). Within this unit, the sericite-chlorite-quartz schists represent the most significant component, followed by sericite-chlorite-quartz-feldspar.

Areas of pervasive alteration that mask the geological textures have been assigned a unique unit number (unit 11). This unit has been subdivided based on the alteration product. Currently, silica (unit 11a) and chlorite (unit 11b) have been defined.

Areas of massive sulphide mineralization, although not forming a significant portion of the lithologies, have also been assigned a lithological unit (unit 12) with subdivisions based on the dominant sulphide. Currently magnetite (unit 12b), pyrrhotite (unit 12c), pyrite (unit 12d) and chalcopyrite (unit 12e) have been defined. Undivided sulphides has been assigned to unit 12a.

In rare situations where protoliths are identifiable, they have been assigned to specific units such as intrusives (unit 3), volcanic flows or intrusions (unit 4), volcaniclastics (unit 5) and sedimentary (unit 6). This was most notable in drill core from the area immediately southeast of the Deposit where argillites and sandstones were intersected. Rare thin limestones have been also identified in several drill holes. A late-stage series of andesitic dykes and sills (unit 4a) have also been encountered in drill core from various areas of the Deposit. Only one occurrence of an intrusive (granodiorite, unit 3a) has been noted to date in the Deposit, (Naas, 2012b, 2012c).

Due to multiphase deformation and alteration, correlating lithologies between drill holes has been difficult. To maintain the lithological detail, yet simplify the geology for correlation, the lithologic units were grouped into geological packages of common characteristics and affinities. Nine packages have been currently defined and named A, B, C, D, E, Fa, Fb, G and H. Package A represented the lowest stratigraphic unit, moving up-section to Package H.

Descriptions of the individual packages are summarized below from Naas (2013).

A surface map of the geological packages is presented in Figure 7-3. Cross sectional examples of the package stratigraphy from the West and East Domains of the Deposit are presented in Figures 7-4 and 7-5.

7.5.1.1 Package A

Package A comprises the Late Devonian orthogneiss (unit 10a) as well as the strongly to intensely deformed marginal-phase of the orthogneiss intrusion. The latter frequently contains texturally destructive deformation and is classified as a sericite-chlorite-quartz phyllite (unit 7d). This unit is interpreted as strongly deformed felsic intrusives (sericite-augen quartz schists, unit 9d) and are often observed cutting through in the upper sections of this package. Package A has been intersected in the West Domain by three drill holes (HC07-24, HC10-76, HC11-95) with drill thicknesses of 17m to 54m. In the East Domain, four drill holes (HC07-26, HC07-27, HC07-31, HC07-33) with drill thicknesses of 49m to 95m are intersected.

Immediately before encountering the orthogneiss is a zone of intense deformation represented by unit 7d. This unit shows possible relict textures of a metasedimentary unit 9b and may in fact be related to an older sequence of metasediments EBQgn (as defined by Schiarizza and Preto, 1987) proximal to the intrusive body. Historically there has been difficulty defining the rocks proximal to the orthogneiss due to deformation and strong to intense biotite alteration. Unit 7d is intensely foliated and deformed with colour ranging from medium green to dark green to brown, as a function of biotite content. Weak to moderate interstitial calcite is frequently observed and with the textural and compositional change is indicative of the proximity of basement rock.

Figure 7-3: Geology Map, Harper Creek Deposit (1:20,000)

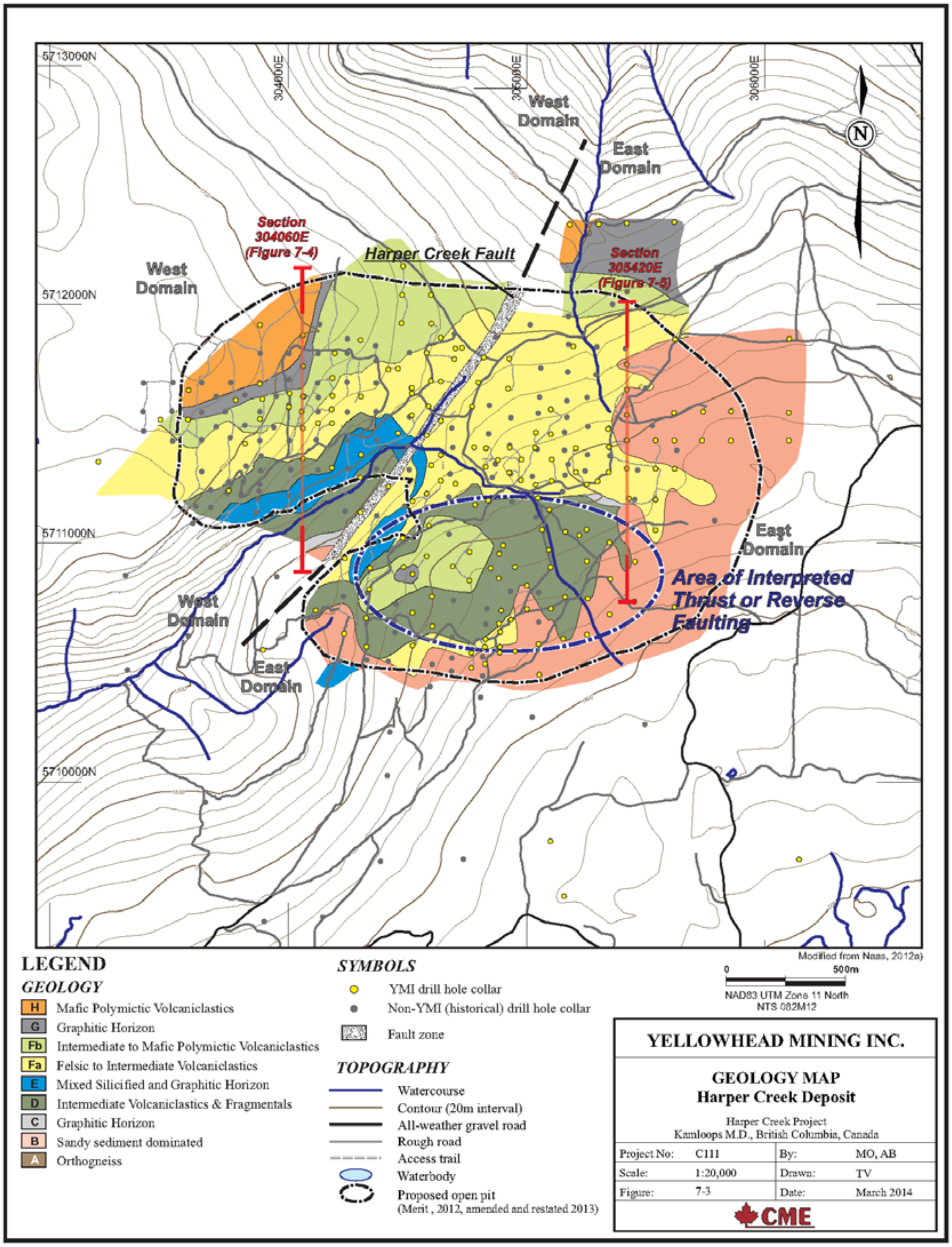


Figure 7-4: Geological Cross Section, 3040 60E West Domain (1:5,000)

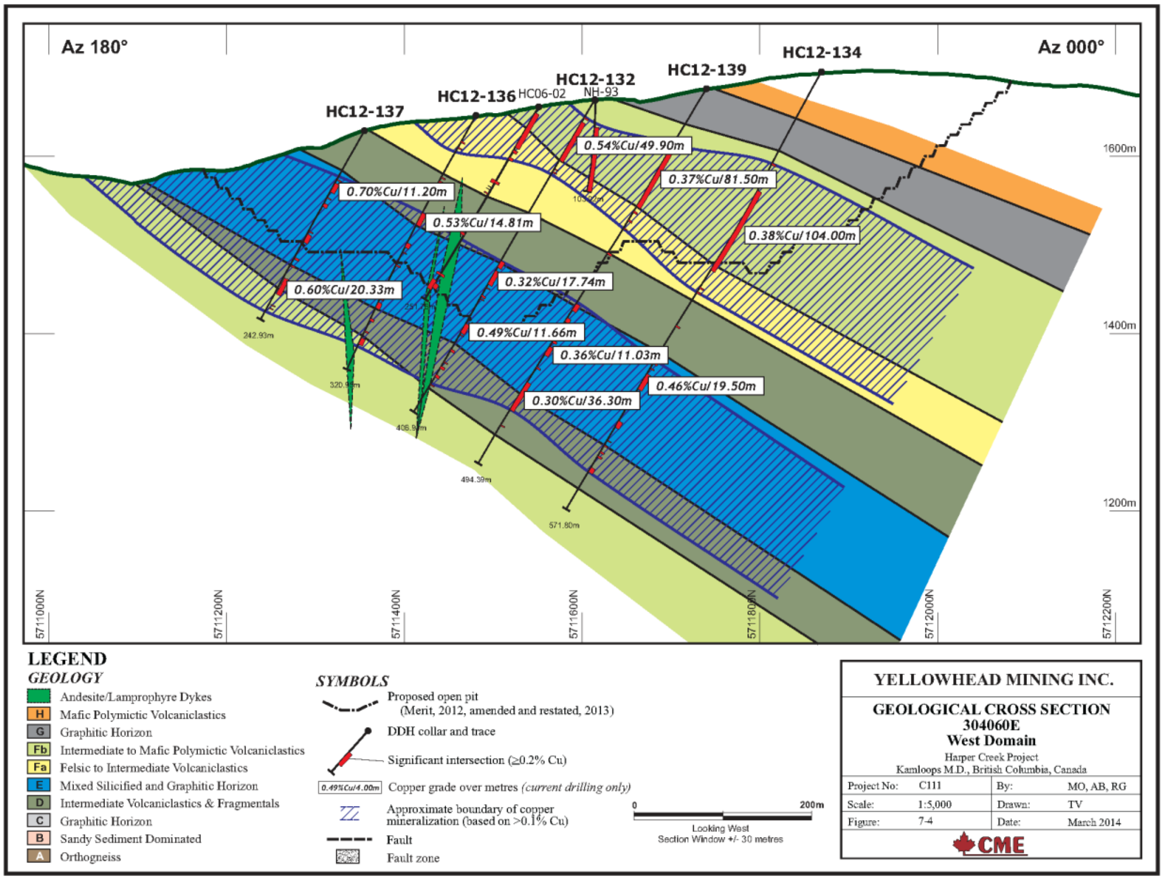
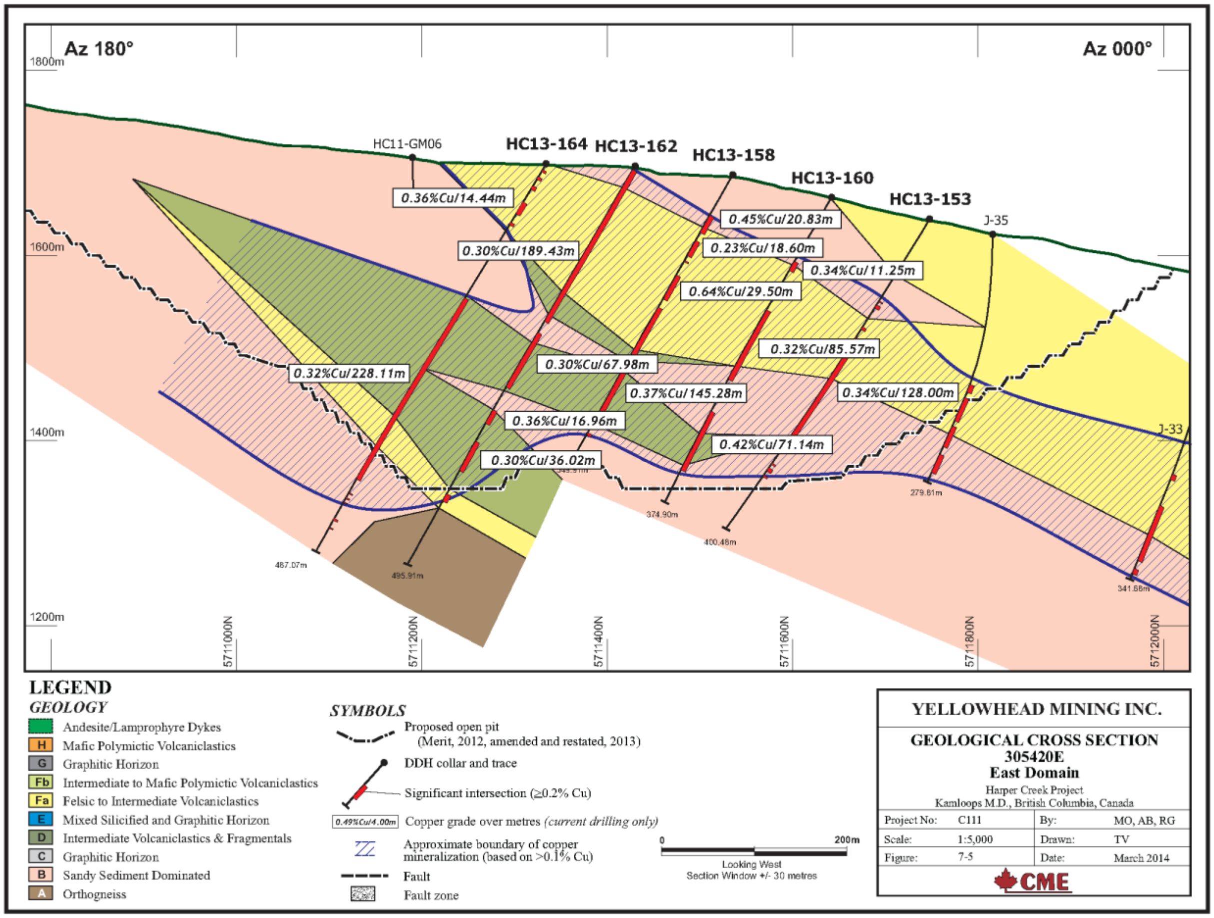


Figure 7-5: Geological Cross Section, 3054 E East Domain (1:5,000)



The unit contains foliation-parallel quartz bands that are commonly milky, boudinaged, 1cm to 15cm wide and internally fractured with iron carbonate and occasionally calcium carbonate infill. Throughout the 10a and 7d units there are felsic dykes represented by unit 9d. These felsic dykes are beige to pale green and show strong to intense foliation. They contain 15% to 20%, grey to translucent, augen shaped quartz eyes up to 1cm in size.

Sulphide mineralization is poor within Package A, consisting predominantly of foliation-parallel bands or disseminations of pyrite with lesser amounts of pyrrhotite and localized fine-grained foliation-parallel disseminations and rare fracture-fill chalcopyrite.

Package A corresponds to the granitic orthogneiss, Unit Dgn, described by Schiarizza and Preto (1987). They note that the orthogneiss is situated on the northern and southeastern portions of the Baldy batholith overlain by, and intruding into, metasedimentary units. The age of the orthogneiss is difficult to discern as metamorphism has reset the original age but uranium-lead on zircons collected within the orthogneiss have yielded a radiometric age of Late Devonian (Schiarizza and Preto, 1987).

7.5.1.2 Package B

This package is a heterogeneous group of rocks consisting primarily of very fine- to coarse-grained clastic metasediments, intercalated with felsic to mafic metavolcaniclastics and silt-sized argillaceous horizons.

In both the western and eastern domains, this package is primarily composed of sandy sequences of the sericite-chlorite-quartz schists (unit 9b), consisting mainly of fine-coarse polycrystalline sand intercalated with thin to thick beds of felsic and mafic silts and metavolcaniclastics.

In the western half of the Deposit area, this package primarily consists of intercalated sericite-chlorite-quartz schists (unit 9b, 30-50%) and sericite-chlorite schists (unit 8a, 5-40%). Other intercalated units that are not present in every succession include: sericite-chlorite-quartz feldspar schists (unit 9c, <1%), graphitic phyllites (unit 7a, approximately 1%), sericite-chlorite-quartz phyllites (unit 7d, approximately 5-30%) and siliceous chlorite-quartz schists (unit 9e, <5%). Drill intercepts for Package B range from 24m in HC07-21 to 218m in HC11-95.

Moving eastward, there is a noticeable increase in the abundance of unit 9c within this package, typically 5-10% and as high as 30%. Unit 7d also increases in abundance, ranging from 5 to 20%. Drill intercepts range from 15m in HC08-70 to 343m in HC11-90.

Both domains have some intensely silicified intervals of unit 11a as well as possible pebble conglomerates (unit 9e) which range up to 5%. Unit 8a intercalations consist of a well-foliated matrix with no visible quartz grains. It is recognized that the 9b units grade in and out of 8a horizons which may indicate a siltstone version of metasediments or mafic metavolcaniclastics.

In the easternmost portion of the Deposit the metasediments become the dominant lithology. Package B is now observed at the top of the stratigraphy with small intervals of Fa, E, or D situated between a second interval of Package B at the bottom (HC11-124 and 126). In the top interval there is a graphitic component to the metasediments previously unobserved to the west. This is evident with intercalations and seams of graphite as well as black to smoky grey quartz grains commonly observed in other graphite-influenced sedimentary intervals. A second section of Package B separated by pinching out intervals of Packages Fa, E, and/or D is intersected in the bottom half of these easterly drilled holes. This zone is strongly intercalated with an increased abundance of unit 9c (up to 50%) while unit 8a decreases and becomes rarely observed. It is unclear whether the graphitic 9b unit and the zones with unit 9c are different geological packages or are just one large sedimentary interval with interfingering volcanic sequences. Moving eastwards indicates a waning of metavolcaniclastic rocks with a large increase in metasediments, possibly indicating this area was previously a sedimentary basin and a greater distance from the volcanic source.

Through most of the Deposit, unless inundated with pervasive secondary silicification, copper mineralization is generally weak within this package of rocks and only occurs as sporadic intervals containing fracture-fill and very fine-grained chalcopyrite disseminations. In the far eastern part of the Deposit, copper mineralization is observed in greater abundances within this package. Following the graphitic portion (unmineralized) it is no longer generally selective to Packages Fa and D, but instead is observed in large intervals throughout. This may be as a result of the increased intervals of unit 9c (which are typically well mineralized) and thus influence mineralization within the surrounding metasediments. Styles of mineralization include very fine-grained disseminations, fracture-fill, and foliation parallel wisps.

7.5.1.3 Package C

This package is observed as a graphitic phyllite (unit 7a) horizon ranging from 2m to 25m in thickness. It is commonly observed as an uppermost mudstone horizon at the top of the Package B sequence, possibly defining an unconformity. Being less competent in relation to the other lithologies, it becomes a preferred horizon for a thrust-fault slip. It is therefore a marker horizon that separates Packages B and D respectively in the West Domain. In the East Domain, this package is interpreted to occur more commonly as intercalations rather than as a distinct horizon. The package often is noted to be absent altogether and Package D is frequently noted to overlie Package B.

Sulphide mineralization within Package C is low. Sulphides are mainly present as pyrite, lesser pyrrhotite and locally trace chalcopyrite. Pyrite and pyrrhotite are precipitated as porphyroblasts up to 1.5cm in size and as fine-grained disseminations. Increased copper mineralization occurs in conjunction with high angle tension fractures of quartz, carbonate, and chalcopyrite.

7.5.1.4 Package D

This package is noted between two graphitic horizons, and is comprised predominantly of intermediate to mafic metavolcaniclastic tuffs and fragmental volcaniclastics frequently containing secondary quartz and calcite alteration which occurs interstitially and as foliation-parallel bands. The dominant lithologies consist of sericite-chlorite schists (unit 8a) and chlorite-carbonate phyllites (unit 7c), similar to the rocks observed within the upper Fb Package (described below).

In the West Domain, Package D transitions eastward from predominantly mafic metavolcaniclastic tuffs and/or silts to a package with increased intercalations of quartz rich metasediments (unit 9b). This is observed in western drill holes HC11-120 (173.21m to 309.98m), HC11-121 (242.94m to 401.42m) and HC11-122 (136.53m to 325.22m) where the package is made up of greater than 95% mafic units, whereas further east the intercalations of metasediments (sericite-chlorite-quartz schists, unit 9b) increase to comprise 20-70% of the package. Sporadic, discontinuous intercalations of sericite-chlorite-quartz-feldspar schists (unit 9c, <1%), sericite-chlorite-quartz phyllites (unit 7d, <5%), siliceous chlorite-sericite-quartz schists (unit 9e, <1%), and pervasive silica alteration (unit 11a, <1%) are also observed within the package and noted to increase moving eastward.

In the East Domain, Package D gradually decreases in thickness and intercalations of more felsics units (units 9b and 9c) increase in abundance as shown in HC11-126 (237.74m to 283.67m) where Package D is present as a lens. This may indicate a shallower marine environment moving distally away from the source of the mafic volcanic rocks. Unit 8a comprises between 25 to 85%, averaging approximately 50%, with a noticeable increase (15-30%) in felsic metavolcaniclastics (unit 9c). Locally, this package may also include discontinuous lenses of units 9b (1-45%), 7d (<1-30%), and 9e (<1-5%). Moving further eastwards (east of 305560E) mafic metavolcaniclastics continue to decrease in abundance as felsic metavolcaniclastics and metasediments are the dominant rock types and the D package is not observed in many of the drill holes. In the West Domain unit 8a is generally present with unit 9b within D and B packages. In the far eastern part of the Deposit it appears unit 9c has replaced the intervals previously occupied by unit 8a. The lithology is now represented by intervals of units 9b and 9c with Package D a decreasing amount.

Sulphide mineralization within Package D is inconsistent with sporadic emplacement of multiple sulphide lenses up to 5m wide. Thick lenses more common with Package Fa are not present here. Zones of sulphide mineralization are present in units 8a, 7c, and 9b and frequently transgress lithological contacts with no preference to one or the other. Chalcopyrite mineralization is mainly observed parallel to foliation as wisps and bands with quartz+calcite and interstitial sulphide disseminations. Locally chalcopyrite is noted as hairline tension fractures bleeding into foliation planes.

7.5.1.5 Package E

Package E consists of pervasive, often texturally destructive silica altered host (unit 11a) that overlies a graphitic phyllite (unit 7a). The silica altered host portion of the package appears to consist mainly of a succession of intercalated fine to medium grained (≤1 mm) sandstone intercalated with siltstone. Within the silica-flooded host, relict opalline-blue quartz grains commonly observed within Package B (unit 9b) are preserved. The impermeable mudstone is preserved as graphitic phyllite (unit 7a), however in many places it also shows strong to intense silicification. This package may be a large thrust fault following the weak graphitic units with silicification resulting from increased fluid movement related to the Harper Creek normal fault. Package E in HC11-123 (9.15m to 91.44m) and HC11-125 (68.90m to 119.67m) could be related to the many structures present in the area including the Harper Creek fault.

Package E is easily traced from west to east throughout the drill holes in the West Domain and ranges from 15m to 91m in thickness. In the East Domain the trend is discontinuous and frequently not observed. Silicified intervals resembling Package E are not confined to the contact between packages D and Fa, but rather as an alteration occurring randomly throughout the stratigraphy. Unsilicified graphitic intervals are also randomly present in the East Domain and may represent mudstone and/or shearing planes. Package E intervals in the East Domain range in width from 4m to 80m with silicified and graphitic intervals generally not associated with one another as observed in the West Domain.

Sulphide mineralization within Package E is strong, with high grade lenses of copper traced throughout. Chalcopyrite (<1-3%) is mainly noted as fracture-fill in tension fractures at 10° to 30° to core axis. Frequently specularite (and locally molybdenite) are present and rare bornite has also been observed. This sulphide assemblage can often be used as a marker within the silicified section. Early interpretation suggests the possibility of an increased temperature gradient moving eastward within the sulphide fluid phase, as specularite appears to decrease while molybdenite and bornite increase.

7.5.1.6 Package Fa

This assemblage is dominated by pale to medium brown to medium greenish grey and green sericite-chlorite-quartz-feldspar schists (unit 9c) mainly derived from felsic volcanic and volcaniclastic rocks. Intercalations include green to dark green mafic volcanics, chlorite-sericite schists (unit 8a), sericite-chlorite-quartz phyllites (unit 7d), graphitic phyllites (unit 7a), sericite-chlorite phyllite (unit 7b) and rare sericite-chlorite-quartz schists (unit 9b) with local intense zones of silica altered host (unit 11a).

In the West Domain, unit 9c comprises 30 to 60% of the package while unit 8a comprises 10 to 40%. Large deformation zones (unit 7d) make up 10 to 50% of the package, with the larger zones often overlying the silica-altered zone stratigraphically below. Argillaceous intervals comprising units 7a and 9b metasediments (without opalline-blue quartz grains) represent less than 5% of the package. Localized pervasive silica altered host intervals (unit 11a) may also be present.

In the East Domain Package Fa becomes intensely convoluted, indistinct and much more difficult to trace across drill holes similar to Package D. A marked decrease in the abundance of unit 9c is noted with its occurrence ranging from 10 to 50%. Also, zones of increased texturally destructive deformation are observed increasing the 7d unit up to 80% in abundance locally. It is possible these zones may have originally been felsic volcanics or unit 9c. Strongly silicified intervals (unit 11a) persist (up to 30%) while mafic units (8a) are generally inconsistent (but up to 50% locally). Metasediments are also incorporated to the Fa package in the east and are variable in abundance (up to 20%).

Moving eastwards Package Fa appears to decrease in size and abundance as metasediments become the dominant lithology. This is evident as Package Fa becomes lenses or pinched out intervals observed in drill holes HC11-124, HC11-126, HC11-128 and HC11-130. Through the removal of Package FD, Fa is now better defined in the East domain.

This package commonly contains the highest percentage of chalcopyrite mineralization within the Deposit. Mineralization is predominantly hosted within the sericite chlorite quartz feldspar schists (unit 9c), interpreted to represent a sequence of felsic volcanics and volcaniclastics intervals. Chalcopyrite, ranging from less than 1 to 3%, commonly occurs as very fine-grained foliation-parallel wisps, on rims of pyritic chain-of-grain bands, interstitial disseminations, and (locally) fracture-fill tension fractures at 10° to 30° to core axis.

7.5.1.7 Package Fb

This package is composed primarily of polymictic fragmental chlorite schists (unit 8c) and chlorite-carbonate phyllites (unit 7c) both of these are interpreted as being derived from mafic volcanic and volcaniclastic rocks. Similar to Package D, these units frequently contain secondary quartz and calcite alteration that occurs interstitially and as foliation parallel bands. Intersections of this package in the southern area of the Deposit predominantly contain secondary dolomite rather than calcite within the same textural variety. Although rare, strong to intense biotite alteration is observed within the chlorite-carbonate phyllites (e.g. HC07-15, HC07-16). The fragmental variety of the package consists of flattened, foliation-parallel fragments that appear to range in composition from mafic to felsic. Locally fine- to coarse-grained pyroxene and amphibole phenocrysts are preserved. Where textures are reasonably well preserved the unit shows a flow-like texture and appears similar to a welded ignimbrite. This package of rocks is reasonably well defined geochemically by a marked increase in titanium and phosphorus that appears consistent throughout the Deposit.

This package is most notably present in the West Domain and is situated in the northern part of the Deposit (HC07-13 to HC07-18, HC11-93) and the western part of the Deposit (HC11-120 to HC11-122, 1.96m to 55.87m in thickness). Unit 8c represents 40 to 90% of this package, along with Unit 8a (20-50%) and unit 7c (up to 60%). Noted locally are intercalations of unit 9c, (5-40%), and unit 9a (40%), the latter specifically in drill hole HC07-15. Unit 9a is of particular note, as it represents a hornblende-quartz phyric tuff, generally only found in the northern part of the West Domain of the Deposit and is likely part of the EBF Unit described by Schiarizza and Preto (1987).

In the East Domain, Fb occurs in two areas: the first is noted near surface in the south, and the second, in the north at depth. Unit 8c comprises 30 to 80% of the package with variable amounts of unit 7c (<60%), unit 8a (10-40%), unit 7d (5-10%), and unit 7a (<5%). These units are inconsistent and not present in all successions. There is the potential that these very similar looking rocks may belong to two different formations. Further work, including comparison of the geochemistry, will be required to resolve this matter.

Sulphide mineralization in Package Fb consists mainly of pyrite as chain-of-grain bands that overprint bands of carbonate. Ranging from less than 1 to 7%, pyrite is also noted as very fine-grained disseminations. Pyrrhotite is present from 1 to 5%, generally appearing as foliation-parallel wisps. Trace chalcopyrite is noted generally on rims of pyrite in chain-of-grain bands and with pyrrhotite wisps. Sulphides appear to be selective to carbonate as either bands or in fractures.

7.5.1.8 Package G

Package G is a graphitic horizon ranging from 6m to 40m in thickness and is interpreted to represent a black mudstone with intercalations of possible mafic tuffs, silts and sandstones. Alternatively the unit may represent a shear zone separating Package Fb and Package H. The package consists primarily of a calcareous graphitic phyllite (unit 7a). It is marked by pale grey to white, moderate to strongly deformed, discontinuous wispy to lensoidal calcite and quartz veining, ranging from less than 1mm, to 11cm in width. It is well foliated, and locally appears fragmental in texture with lenticular to banded fragments parallel to foliation (1mm by 6cm). Intercalations of medium to dark grey limestone (unit 6f) are also observed within the package.

In the East Domain, this package occurs as sporadic lenses, which do not correlate well across the Deposit. It is noted in the southwest (e.g. HC07-27) as calcite-dominant with intercalated graphitic limestone. Centrally, dolomite is the more prominent carbonate and occurs similar to the description above.

Sulphide mineralization in Package G is mainly made up of pyrite (up to 3%) and pyrrhotite (up to 1%) as anhedral to euhedral porphyroblasts and foliation-parallel wisps. Trace chalcopyrite is noted locally as fracture-fill or foliation-parallel wisps.

7.5.1.9 Package H

This is the uppermost package of rocks within the Deposit. Its occurrence thus far is restricted to rocks observed in the far north and west of the Deposit, primarily within drill holes HC10-78 and HC11-84. The base of the package appears to have undergone strong to intense deformation as noted by the presence of thick intersections of sericite chlorite quartz phyllites (unit 7d) that are frequently intercalated within a succession of what resembles felsic volcanic tuffs similar to those identified within the 9c unit. Intercalations of hornblende-feldspar-quartz crystal lithic tuffs are locally identified and likely are representative of the EBF assemblage (Schiarizza and Preto, 1987).

In the West Domain, it is mainly noted in historical holes NH-52, NH-63, and NH-71. The latter two holes consist of unit 9c, while NH-52 shows intercalations of unit 9a.

In the East Domain, Package H was previously interpreted as occurring in HC07-41, and other holes in this area. This interpretation has since been re-evaluated, and still remains uncertain at this time. Historical J-series drill holes are also noted to contain possible occurrences of Package H but interpretation of the historical logs and re-logged historical core is not reliable.

Mineralization is often weak within the package and is dominated by fine to medium-grained pyrite and pyrrhotite, frequently with chlorite, possibly as mafic mineral replacement.

7.5.2 *STRUCTURE*

7.5.2.1 Harper Creek Fault

The Harper Creek Fault is a large fault zone that trends northeast and appears to dip approximately 70° to 75° to the southeast. The structure follows a northeast trending tributary of Harper Creek and marks the separation of the Deposit into the East and West Domains (Figure 7-3). The fault commonly contains several wide zones of pale grey to green gougy faults and localized quartz and iron carbonate-healed fault breccias. Polylithic and often silicified fault breccia fragments are noted within the structure. These fragments often have disseminated and fracture-fill mineralization within. The quartz-iron carbonate breccias are generally barren and have been faulted by a later event, defined by reactivated gougy sections. Strong to intense deformation commonly observed as kink folding, in addition to abundant clay (argillic) alteration, is common within the structure. As the structure is composed of several fault zones, thickness varies from hole to hole, however it generally ranges from 25m to 50m in thickness. Initial interpretation of movement of the structure demonstrates somewhat oblique right lateral offset. There may also be some degree of rotational movement as well, but further work is needed to verify this. The south side appears to have been down dropped from 60m to 100m.

The structure also contains several mafic to andesitic dykes. These are interpreted as late Tertiary dykes and show no regional deformation but may have been affected by the late northerly trending faults as many intersections are gougy and brecciated. It appears that several of these dykes are utilizing the Harper Creek fault structure as a pathway (Naas, 2012a).

7.5.2.2 East Domain Structures

The East Domain of the Deposit appears to be separated by several fault slices related to a structural event. These structures are noted throughout the domain causing offsets in mineralized zones as well as offsets in packages of rocks that appear to range from tens to possibly hundreds of metres. Further research is needed on these structures to confidently measure offsets to mineralized zones.

Originally, the Larry Fault was thought to be an east west trending structure that dipped steeply to south or north. Drilling during this phase of work has shown that the Larry Fault is related to these fault slices and the previously theorized steep dipping structures do not exist.

The orientation of these structures trends west southwest (~250°) with a northwest dip of 20° to 35°. The style of these faults indicates an imbricated thrust fault system with multiple variations in strength and angles. Characteristics of these structures vary with the lithology the structure is passing through. Within interpreted feldspar-dominated hosts, (units 9c, 8a, 8c, and 7c) the host contains abundant foliation-parallel flaking (friable rock) with the core becoming disc shaped along with multiple foliation-parallel gouge zones where back and forth movement has occurred. Within the more silicified and weakly foliated sericite-chlorite-quartz schists (unit 9b and 11a), the host rocks are generally observed as broken up fragments along with abundant hairline fractures with no preference in orientation. Frequently, clay and gouge are noted along fracture surfaces within the silicified areas. In several instances iron carbonate and silica-healed breccias are noted within gouge zones (Naas, 2012a).

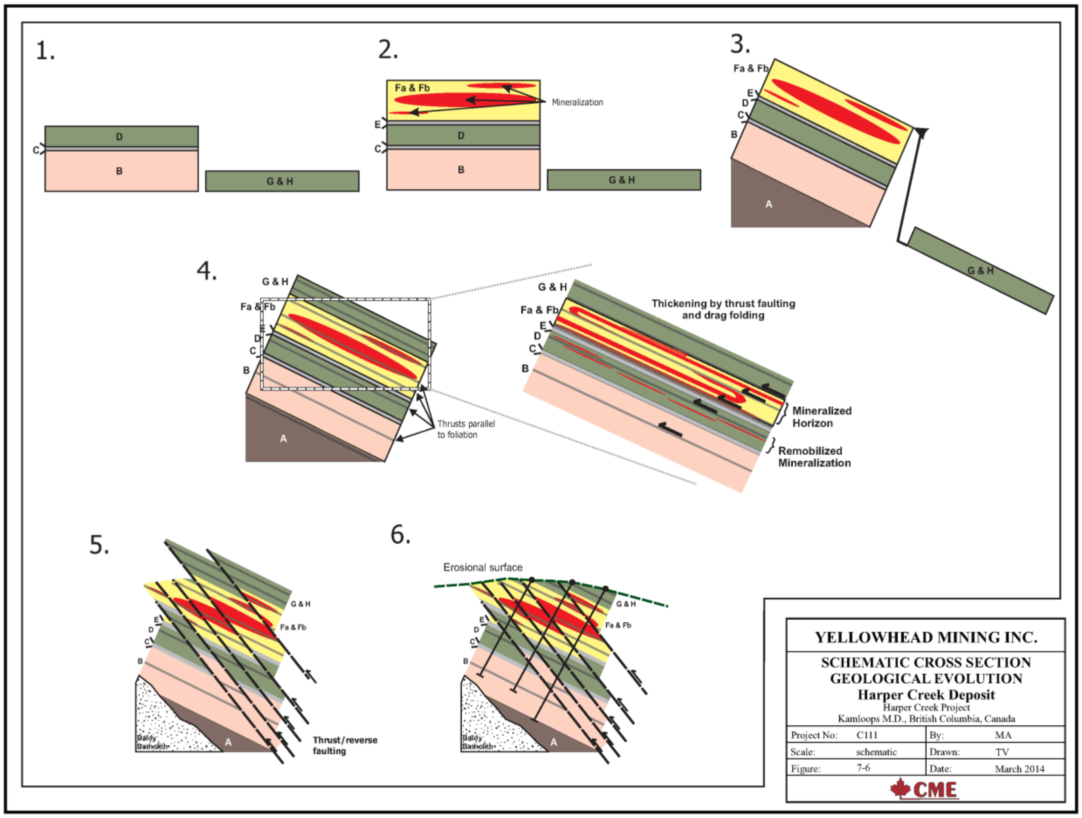
7.5.3 *Geological Interpretation*

The proposed sequence of formation for the Deposit as presented in Table 7-1 is illustrated graphically in Figure 7-6 (Naas, 2012a).

Table 7-1: Harper Creek Deposit Sequence of Formation

1. Lower Cambrian:
Deposition of the B and C package sediments followed by the deposition of mafic volcaniclastics of the D package.
Concurrent deposition, elsewhere, of packages Fb, G, H and I, calcareous volcaniclastics and sediments including limestones.
Mid-Cambrian - Mid-Devonian: Depositional hiatus.
2. Late Devonian-Early Mississippian:
Deposition of the Fa felsic volcaniclastics and Fb mafic volcaniclastic packages with syngenetic volcanogenic sulphide mineralization.
3. Late Devonian:
Intrusion of the orthogneiss, Unit 10a.
Late Triassic-Early Jurassic: 1st regional phase of deformation, this is not directly observed in the immediate Deposit area.
4. Late Jurassic-Early Cretaceous:
Continuous folding accompanied by southwest directed thrust faulting.
Possible repetition of the stratigraphy by thrusting of B, C, D, Fa and Fb packages on top of itself in places on the Project.
Thrusting of the Fb, G, H and I packages on top of the Fa and Fb packages.
Remobilization of the sulphide mineralization along thrust fault planes and foliation.
5.Mid-Cretaceous:
Intrusion of the Baldy batholith to the south.
This is accompanied by contact metamorphism, east-west trending folds and kinks and the west-northwest trending system of reverse faulting system, which reconfigured the stratigraphy of the East Domain and thickened the mineralized zone by repetition.
6. Late Cretaceous:
Southwest-northeast trending Harper Creek Fault separating the West and East structural domains with a strike-slip displacement.
7. Tertiary:
North trending normal faults. This generation of faults occurs in both the West and the East Domains; potentially sub-parallel to the orientation of the drill sections, their emplacement has not been pinpointed with accuracy at this time. Their displacement appears to be minimal.
Intrusion of quartz-feldspar porphyry, andesite, and lamprophyre dykes.
8. Erosion to current topography.

Figure 7-6: Schematic Cross Section, Geological Evolution, Harper Creek Deposit



8 DEPOSIT TYPE

The Deposit is interpreted to be a polymetallic volcanogenic sulphide deposit, comprising lenses of disseminated, fracture-filling and banded iron and copper sulphides with accessory magnetite. Mineralization is generally conformable with the host-rock stratigraphy, as it is consistent with the volcanogenic model. Sulphide lenses are observed to measure many tens of metres in thickness with km-scale strike and dip extents. The current theory is that the Deposit is a remobilized volcanogenic massive sulphide deposit.

9 EXPLORATION

Since 2006, YMI has carried out airborne geophysics (magnetic and electromagnetic), soil sampling, ground geophysics (magnetic, electromagnetic and induced polarization), rock sampling and geological mapping on the Project. Additional studies have included petrographic and whole rock analysis of drill core and surface rock samples.

9.1 AIRBORNE GEOPHYSICS

In 2006, Aeroquest Limited of Milton, ON (Aeroquest) conducted a 1,097.4 line-km helicopter-borne magnetic and electromagnetic survey. Airborne surveys have not been flown since 2006. The geophysical sensor included Aeroquest's exclusive AeroTEM II time domain helicopter electromagnetic system that was employed in conjunction with a high-sensitivity cesium vapour magnetometer. Ancillary equipment included a real-time differential GPS navigation system, radar altimeter, video recorder and a base station magnetometer. Line spacing was 100m except for the northeastern corner of the Project, where line spacing was 200m (Aeroquest, 2006). Colour contoured results of total field magnetics and electromagnetic (AeroTEM Z off-time) are presented in Figures 9-1 and 9-2, respectively.

Geophysical targets were successfully identified and prioritized for follow up ground surveys (Aeroquest, 2009).

Additional processing of the airborne survey data was carried out in 2007 by Insight Geophysics of Toronto, ON, (Insight Geophysics), which supplied analytical signal and reduction to pole maps.

Figure 9-1: Airborne Geophysical Survey, Total Field Magnetics, Harper Creek Project (1:75,000)

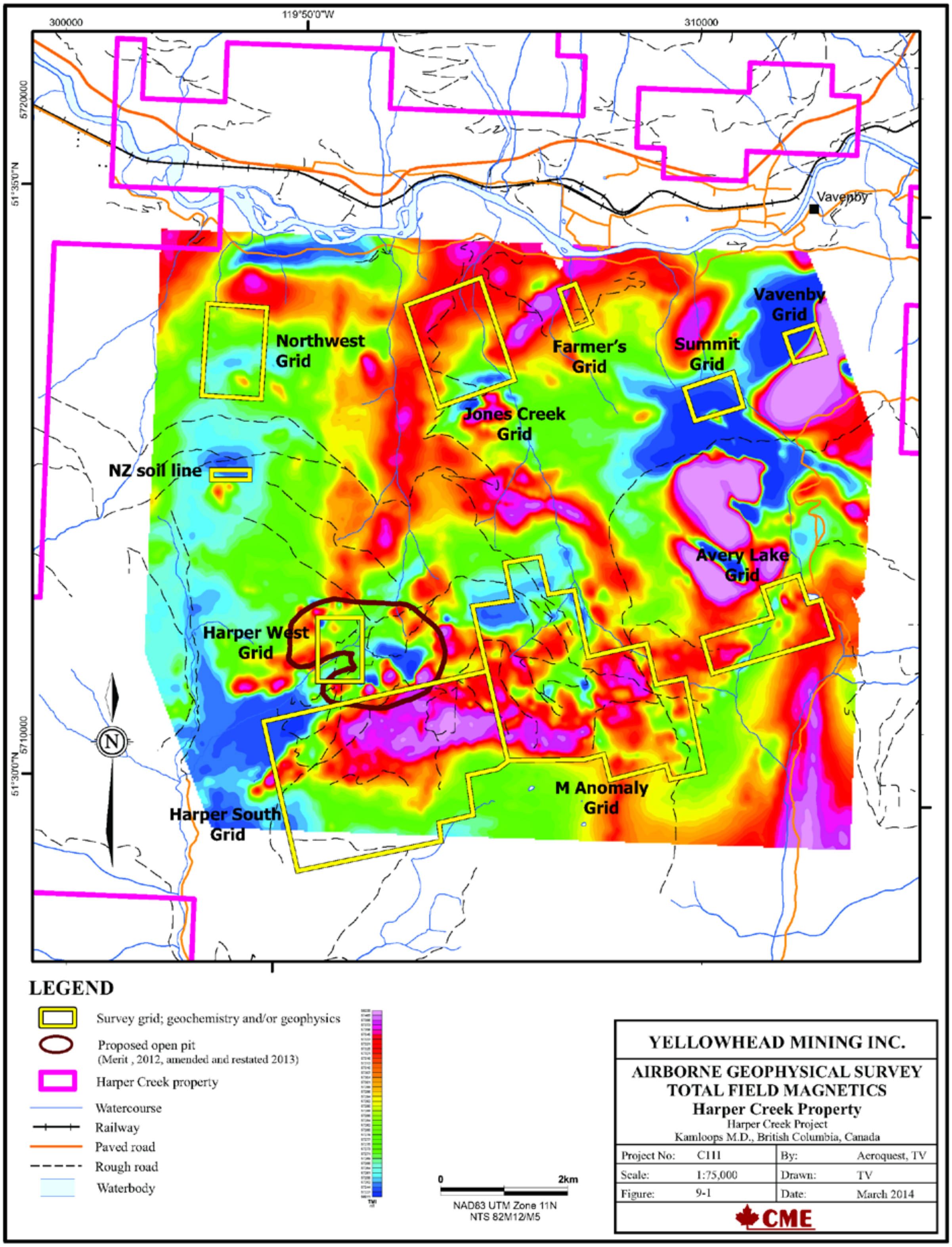
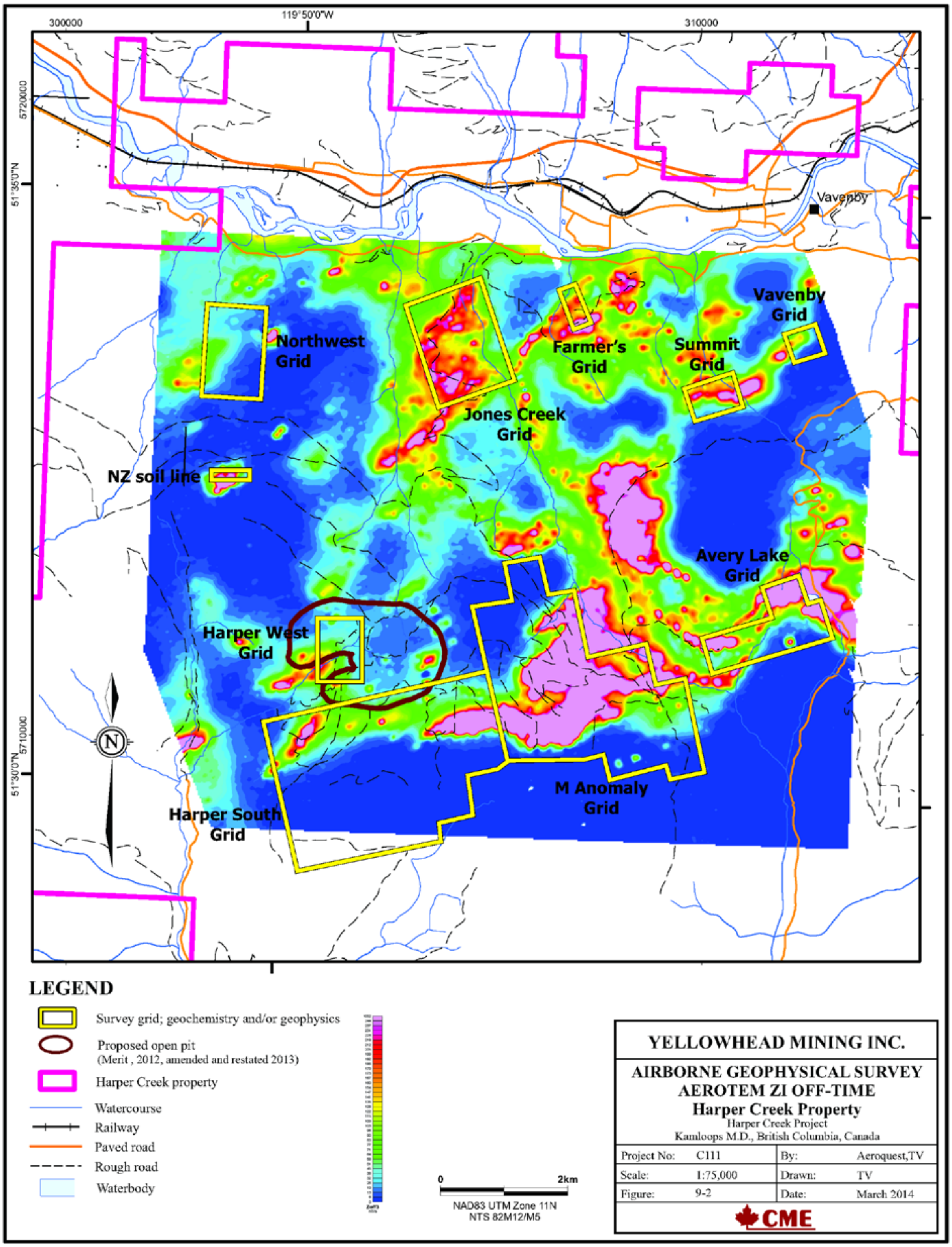


Figure 9-2: Airborne Geophysical Survey, AeroTEM Z1 Off-Time, Harper Creek Project (1:75,000)



9.2 GROUND GEOPHYSICS

Ground-based geophysical surveys were completed on the Project during 2007 and 2008. Survey types include horizontal loop electromagnetic (HLEM), magnetics, and induced polarization (IP). HLEM and ground magnetics were carried out on the Harper West Grid, Jones Creek Grid, Northwest Grid, and the M Anomaly Grid. Only ground magnetic surveying was undertaken out on the Harper South Grid. The IP survey was performed on the southeastern area the M Anomaly Grid. Figures 9-1 and 9-2 illustrate the grid areas discussed.

9.2.1 *HLEM AND GROUND MAGNETICS*

In 2007 Insight Geophysics was contracted to perform HLEM (Maxmin) and ground magnetic surveys over selected survey grids: Harper West; Northwest; Jones Creek; and, the eastern portion of the M Anomaly Grid. These surveys were designed to further test existing anomalous targets defined by both airborne geophysical and soil sampling. The HLEM data is collected using up to four frequencies measuring (220, 880, 3520, 14080Hz) the magnitude of secondary response and phase shift versus the primary field in complex space. Every situation of conductivity thickness has a specific frequency it is best tuned to. On the Project, it was anticipated that better targets will respond best to lower frequencies. Survey equipment consisted of an Apex MaxMin II.

9.2.1.1 Harper West Grid

Three anomalous areas were defined within the Harper West Grid (Insight 2007a) and illustrated in Figure 9-3:

- L4120E and 1950N (304,120E 5,711,950N): The northern portion of this grid is dominated by sub horizontal (dipping NNW) conductivity thickness and a relatively strong magnetic signature.
- L4240E and 1775N (304,240E 5,711,775N): This location is marked by an isolated conductivity thickness high and a strong magnetic contrast.
- L4000E and 1750N (304,000E 5,711,750N): This location is marked by an isolated conductivity thickness high and a strong magnetic contrast.

9.2.1.2 Northwest Grid

Three conductor axes within the Northwest Grid (Figure 9-4) were defined (Insight 2007a):

- L26100E 5100N (302,289E 5,715,895N): Best flat lying conductivity feature within the grid. The amplitude of the anomaly is not large in an absolute sense but the response is well behaved and developed.
- L26100E 5625N (302,322E 5,716,391N): Target keys on a reversal of the imaginary and real components of the HLEM response. Target is flat lying and lesser in conductivity thickness than the previous choice.
- L26600 4800N (302,776E 5,715,566N): There is no conductivity thickness associated with this selection, but there is an isolated magnetic anomaly showing association with persistent soil values.

The western edge of this grid is dominated by a strong and complex conductivity thickness response detected in both the ground and air borne EM surveys. There is also a second conductivity package developing immediately north-northeast of the grid and is outlined in the air borne surveys as well.

9.2.1.3 Jones Creek Grid

The Jones Creek Grid (Figure 9-5) is dominated by large-scale conductivity thickness and soil geochemical anomalies (Insight 2007a). Three targets include:

- L37000E 9725N (305,933E 5,716,449N): This marks the down dip edge of the potential target along the north south direction. Edge is marked in the ground magnetic data as well.
- L37400E 9600N (306,329E 5,716,457N): This area marks the down dip edge of the potential target along the north south direction. Edge is marked in the ground magnetic data as well. Clear association with soil geochemical anomaly.
- L36600E 9500N-9600N (305,625E 5,716,099N – 305,595E 5,716,185N): This area marks the down dip edge of the potential target along the north south direction. Edge is marked in the ground magnetic data as well.

9.2.1.4 M Anomaly Grid

Geophysical plan maps showing HLEM conductivity and total field magnetics are presented in Figures 9-6 and 9-7 respectively. The survey shows numerous profiles that may appear to be shallow depth extent vertically tending responses. For the most part, the majority of these responses are considered to be the edge of a larger horizontal plate (Insight 2007b).

9.2.1.5 Harper South Grid

A ground magnetometer survey was conducted over the Harper South Grid in 2008 by CME. Equipment consisted of a GEM systems GSM-19 Proton magnetometer as the Base Station and a GSM-19 Overhauser as the rover unit. Both units were synchronized daily to facilitate accurate data correction. The base station magnetometer was set up in a three seconds cycle time mode for monitoring diurnal variations in the geomagnetic field. The base station was safe-guarded against external interference throughout the survey period. Magnetic datum for diurnal correction was set at 52,000nT on both units (base and rover) and a two metre ground clearance sensor used on each unit. The base station magnetometer was located at UTM coordinates 306,153.78E, 5,710,688.62N and 1,799.62m elevation.

A total of 39.475 line kms of data were collected. Cut lines (33.53 line kms) were surveyed by continuous reading at 3 second intervals, while the uncut lines were surveyed at 25m intervals.

The total field magnetic data shows better resolution than that of the airborne data. In particular, a prominent boundary between higher mag rocks to the north and a moderate mag unit to the south, corresponds with field observations of the contact between Eagle Bay orthogneiss and the metavolcanic and/or metasedimentary units (Naas, 2009).

Figure 9-3: Ground Geophysical Survey, Total Field Magnetics, Harper West Grid

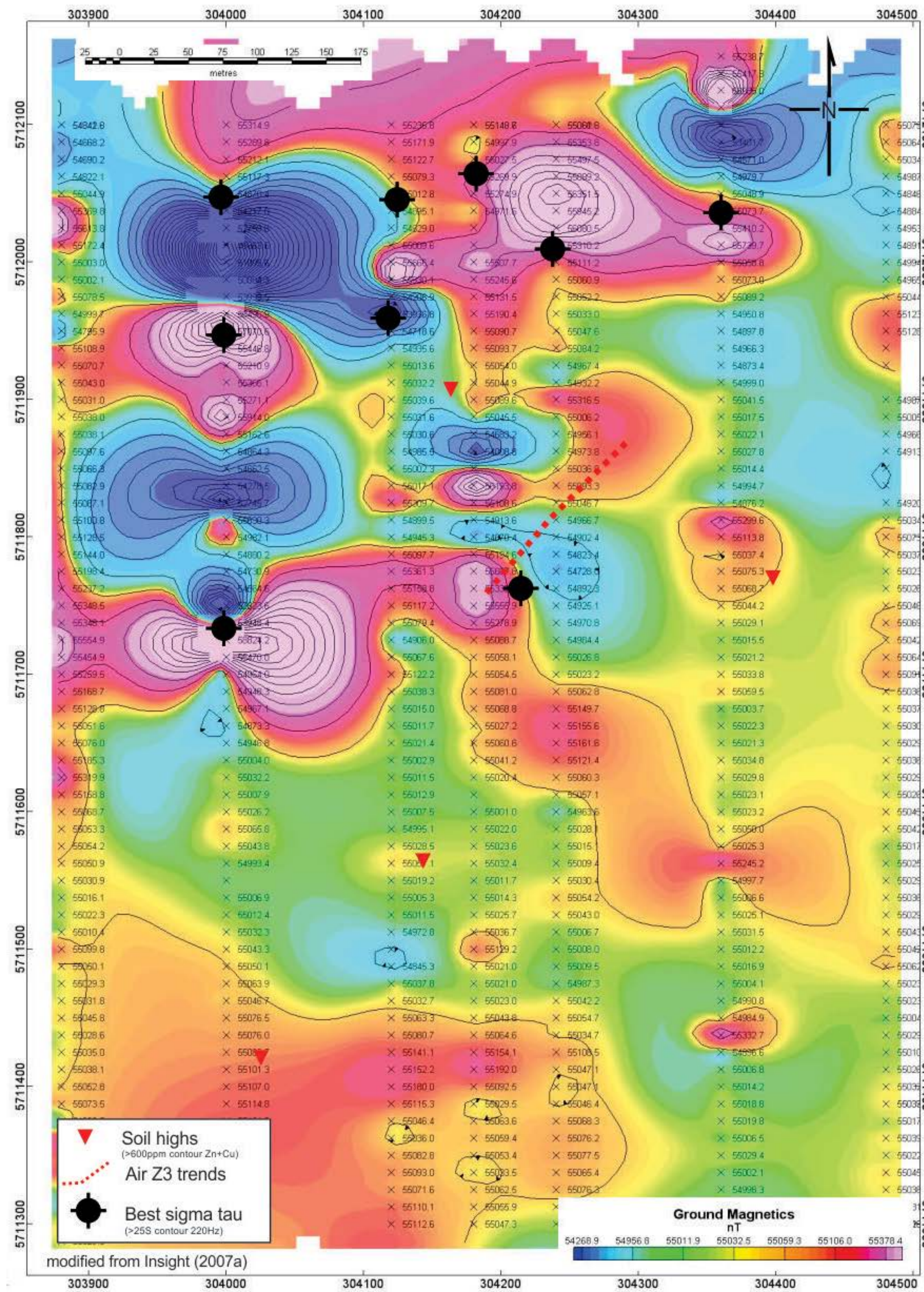


Figure 9-4: Ground Geophysical Survey, Total Field Magnetics Northwest Grid

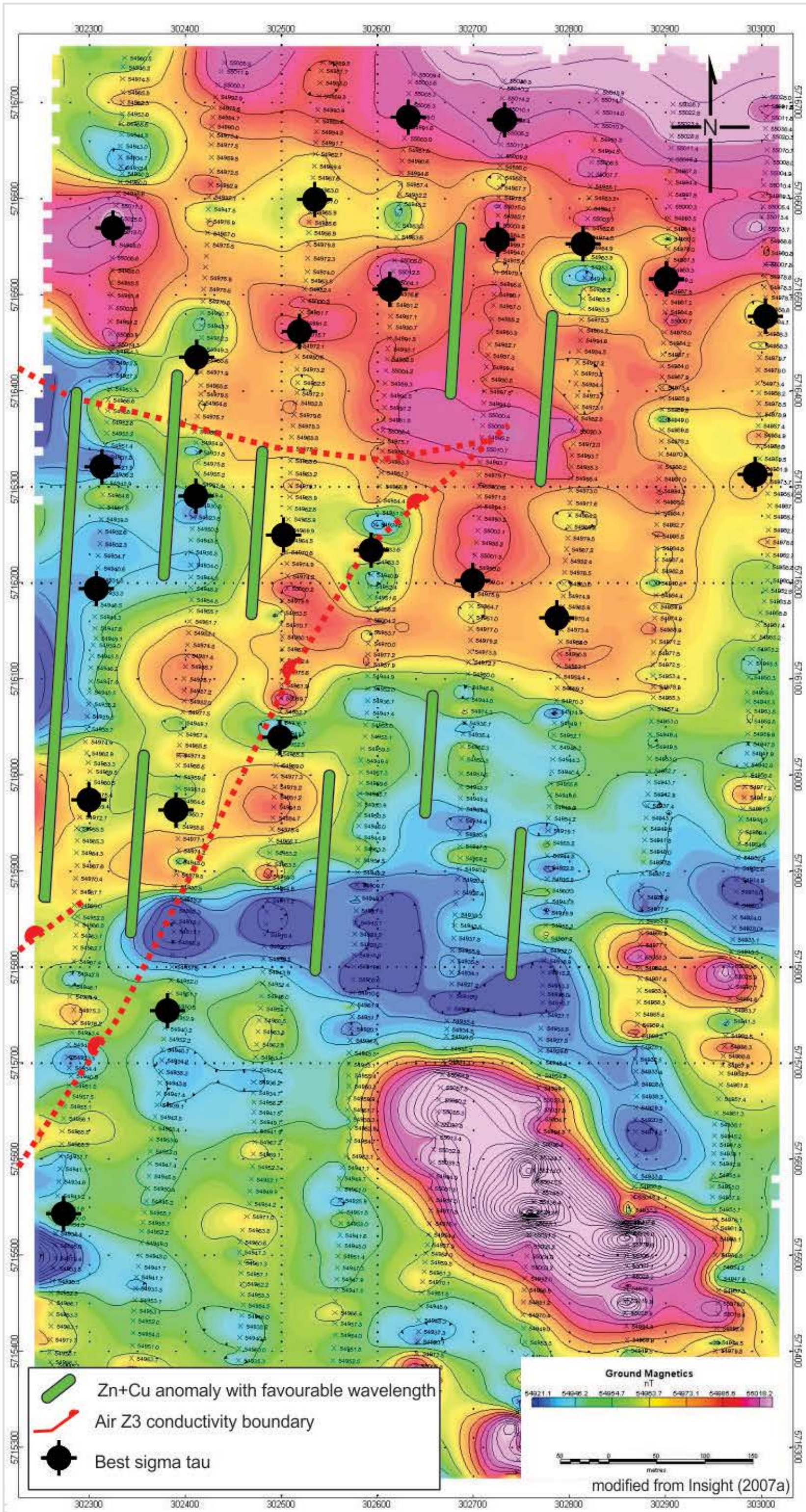


Figure 9-5: Ground Geophysical Survey, Total Field Magnetics, Jones Creek Grid

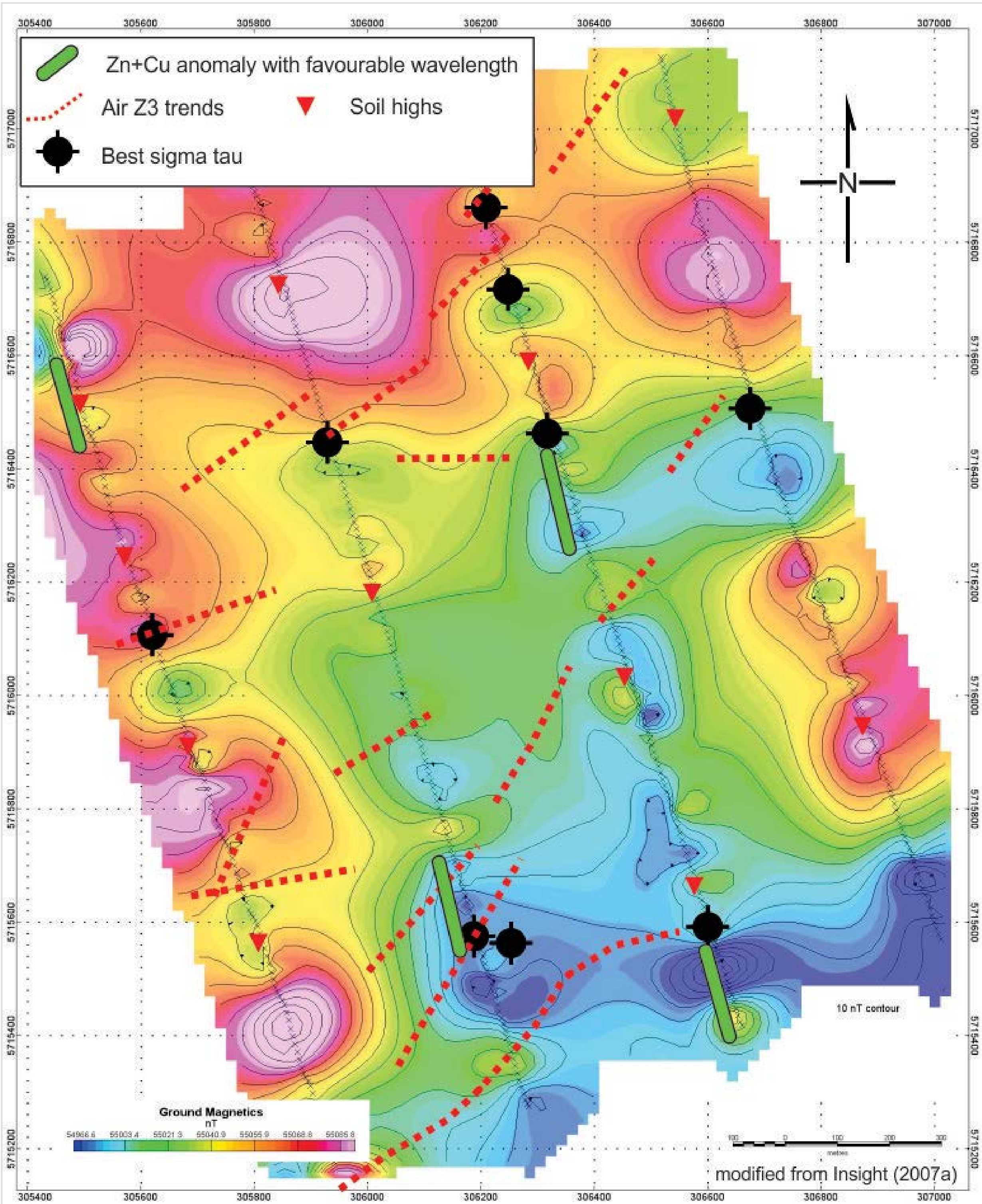
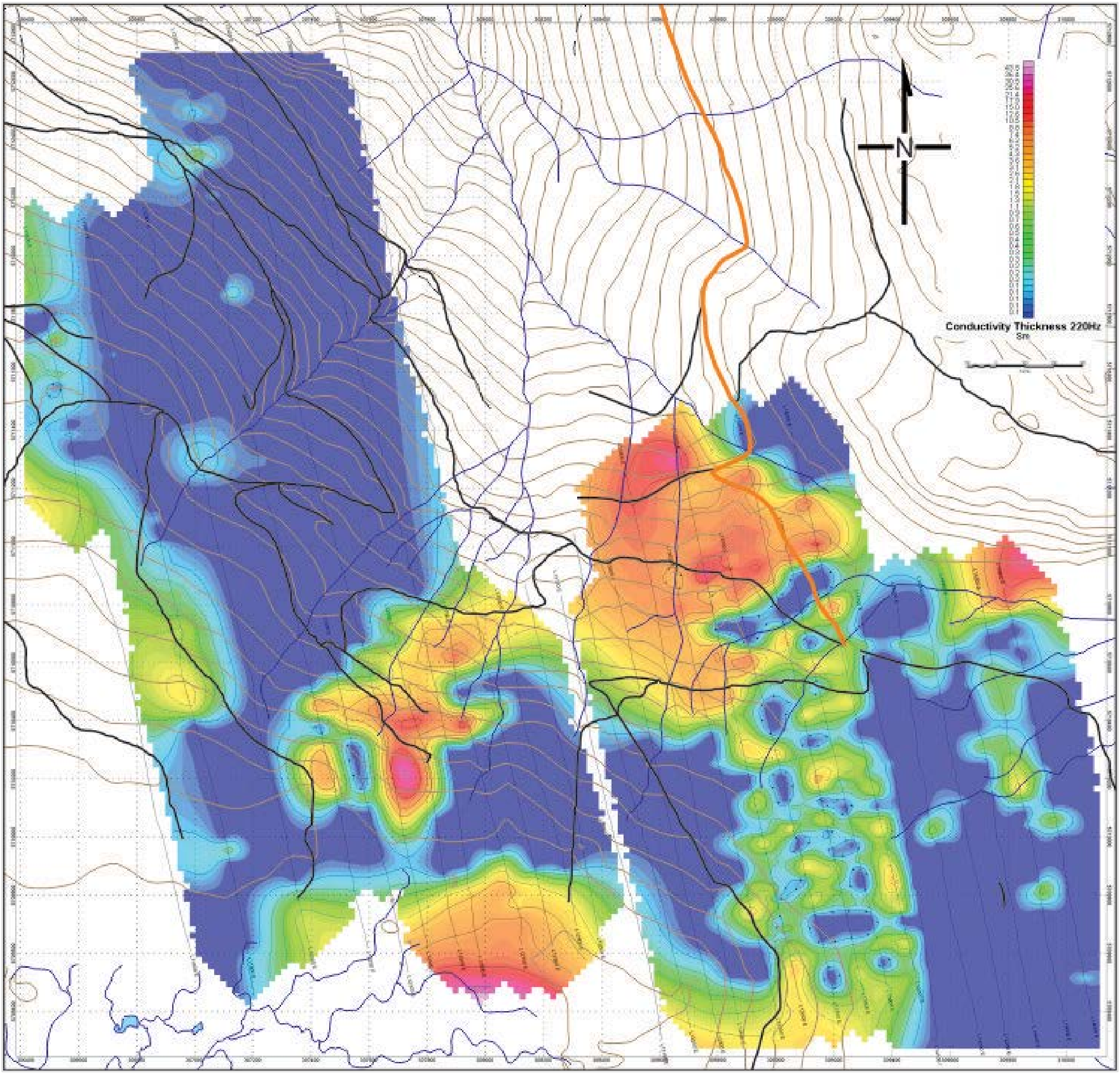
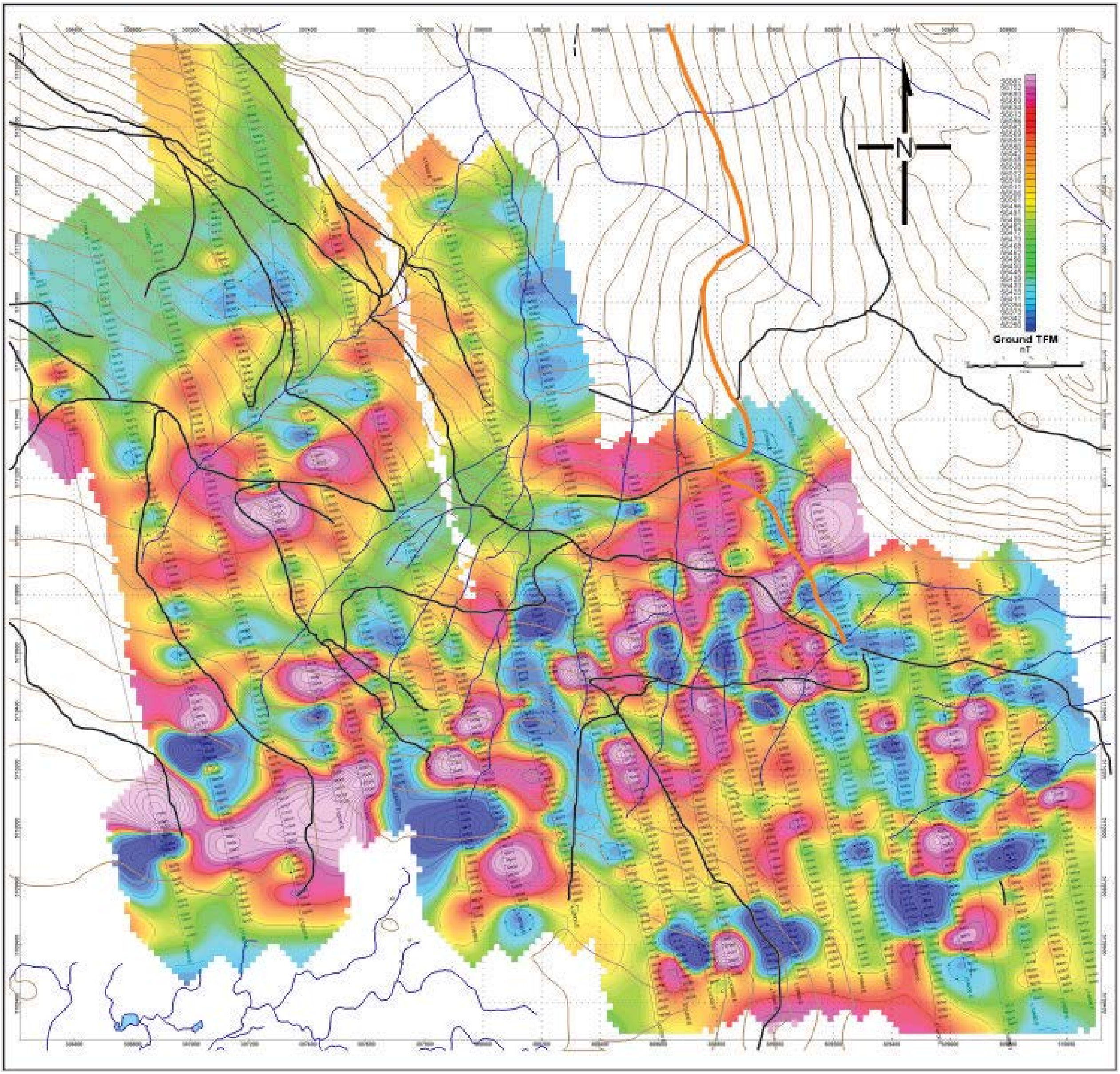


Figure 9-6: Ground Geophysical Survey, Conductivity, M Anomaly Grid



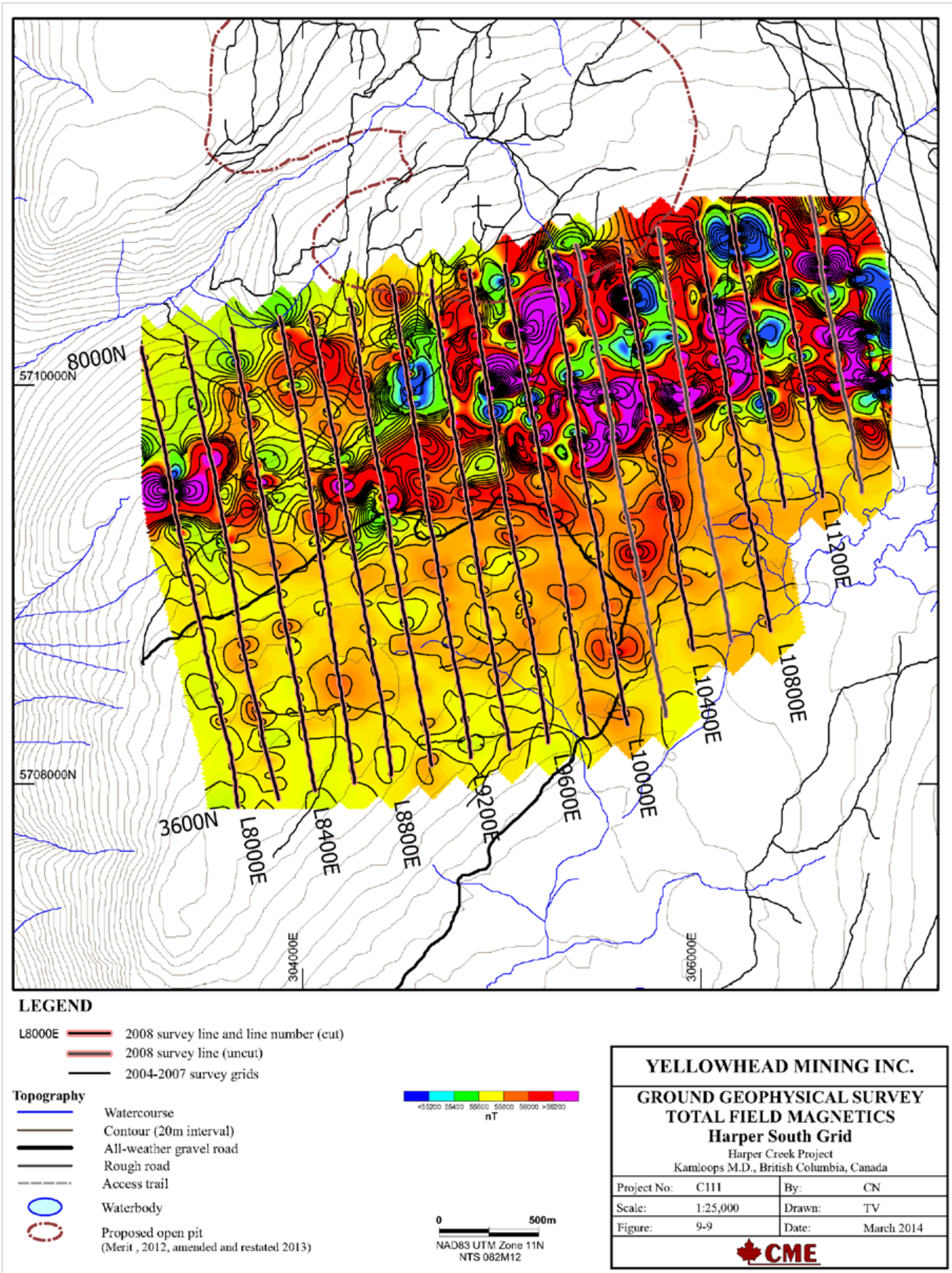
modified from Insight Geophysics Inc., 2007

Figure 9-7: Ground Geophysical Survey, Total Field Magnetics, M Anomaly Grid



modified from Insight Geophysics Inc., 2007

Figure 9-8: Ground Geophysical Survey, Total Field Magnetics, Harper South Grid (1:25,000)



9.2.2 *IP SURVEY*

Insight Geophysics conducted an IP survey over a selected area of the M Anomaly Grid. The objective of this survey was to compliment the ground magnetics and HLEM surveys program in delineating discrete targets potentially associated with massive sulphide mineralization in the area of the M Anomaly geochemical anomalies (Naas and Soloviev, 2008). Equipment used includes a Huntec Mk IV Transmitter and an ELREC PRO Ten channel IP receiver. Approximately 32 line kms of tuned gradient IP/Resistivity surveying was completed. Insight section surveys were conducted on three lines. Depth of exploration was from surface to approximately 250m to 300m depth (Insight 2007c).

Plan maps of the chargeability and apparent resistivity are presented in Figures 9-9 and 9-10 respectively.

Figure 9-9: Ground Geophysical Survey, Chargeability, M Anomaly Grid

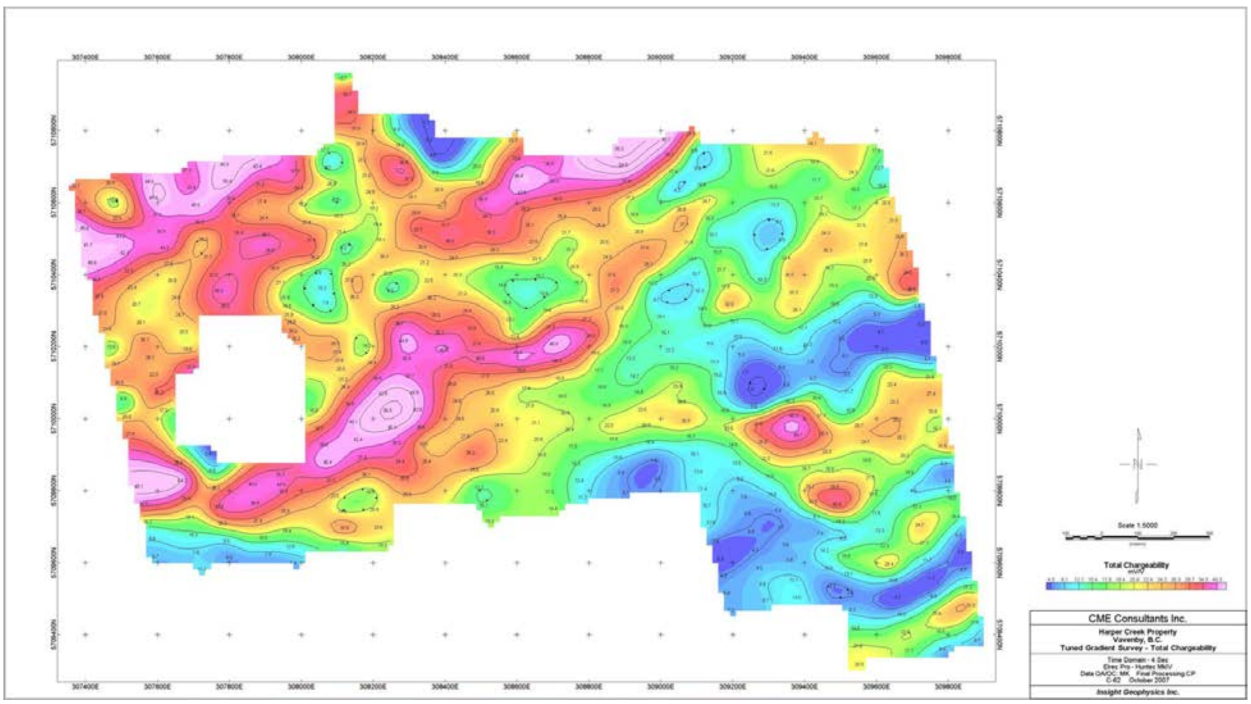
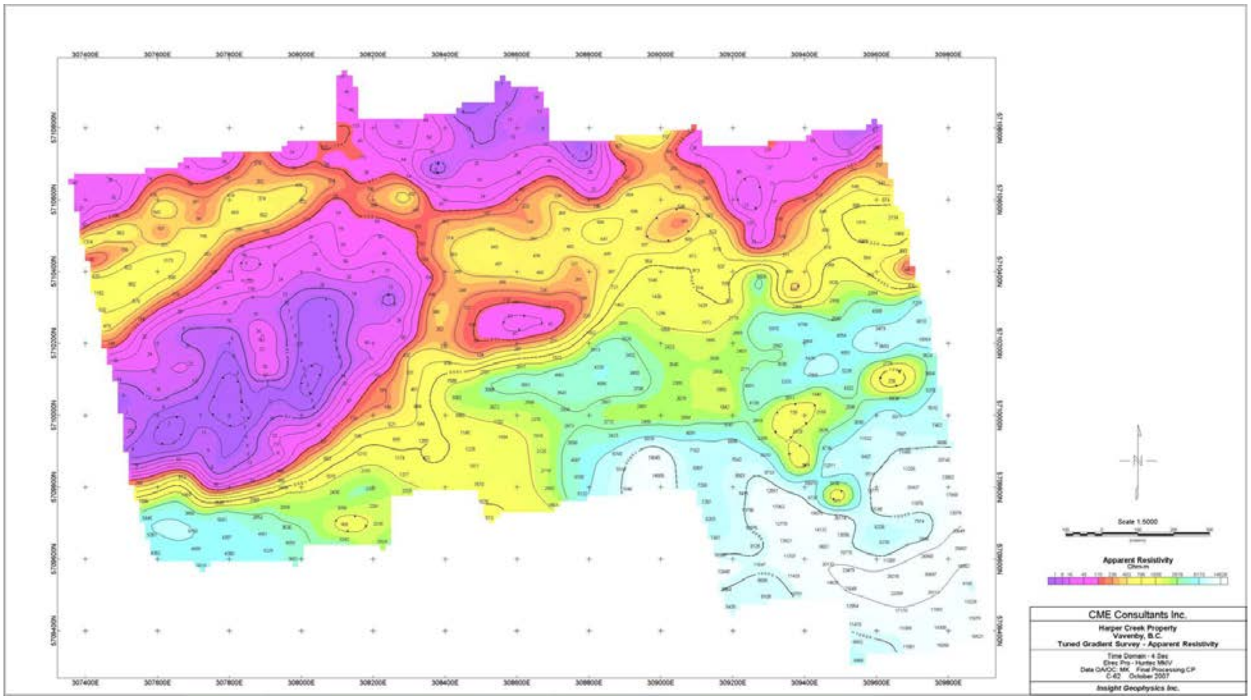


Figure 9-10: Ground Geophysical Survey, Apparent Resistivity, M Anomaly Grid



9.3 Soil Sampling

Between 2006 and 2008, eight soil sample grids and one soil line along the side of a logging road were established over high priority targets identified by the airborne geophysics (Naas, 2006, 2007, 2009). The survey locations are presented in Figure 9-11.

A total of 4,532 soil samples have been collected by YMI from all survey grids and lines.

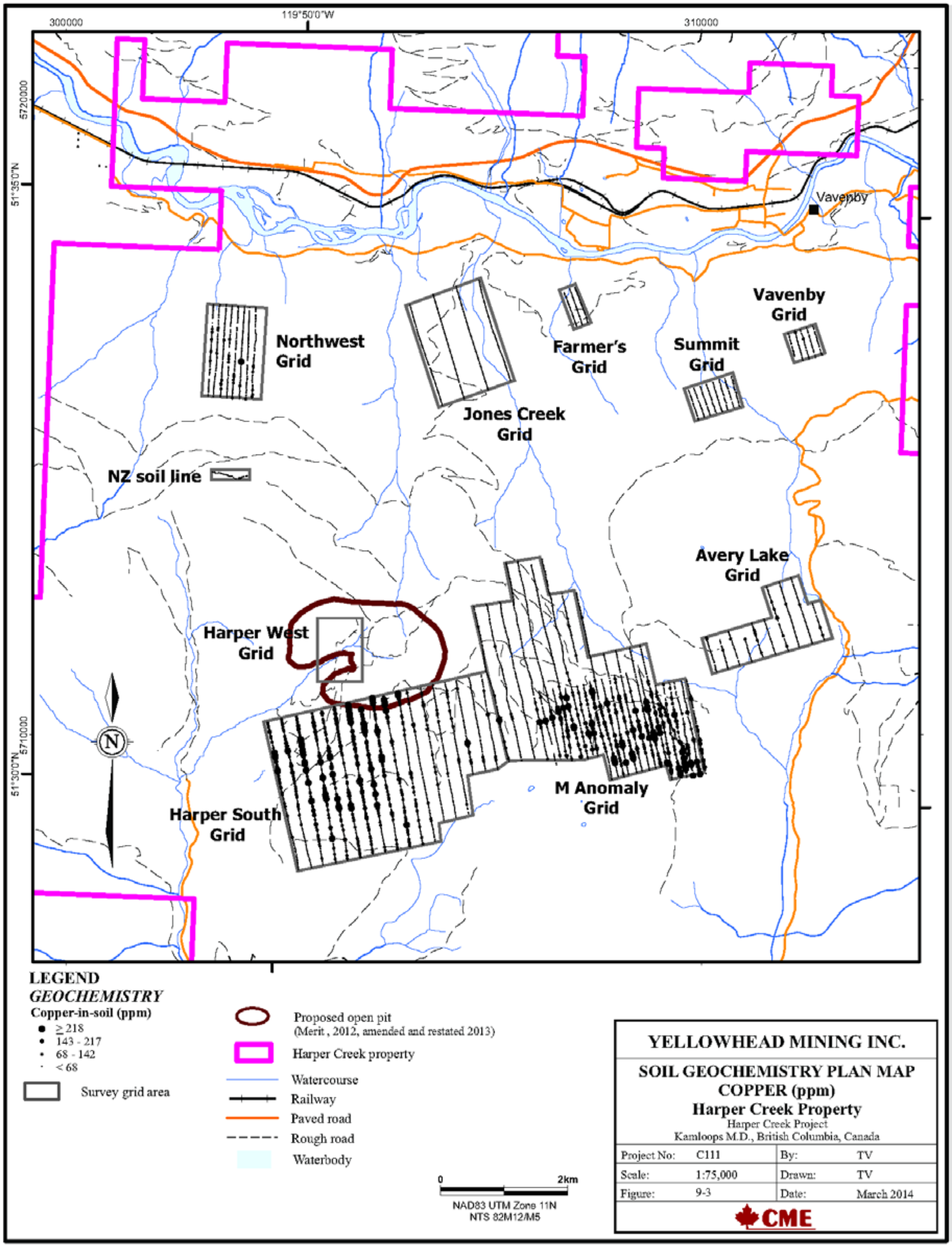
Survey grids were established with cross lines NNW-SSE, perpendicular to the regional trend, except for the Northwest Grid which was oriented close to N-S. Cross line spacings ranged from 100m for detailed sampling to 400m for reconnaissance-scale sampling. Sample stations were established at 25m spacings along all lines. Stations were GPS surveyed using a Trimble GeoExplorer XT rover. The GPS data was differentially corrected using data from the Williams Lake public domain GPS base station.

Samples were collected from the B-horizon, but in areas of poor soil development samples were collected from the C-horizon. Depth of collection ranged from 20cm to 30cm from surface with sample sizes ranging from 200g to 400g. No samples were collected in areas of unsuitable material (i.e. roads, swamp). Sample material was stored in kraft sample bags and labeled by local grid coordinates along with a corresponding bar code.

A brief summary of the results of the soil sampling from each grid is as follows:

- **Northwest Grid:** Sampling returned a zinc anomaly with a weak correlation to copper. The zinc anomaly is 100m to 300m wide and 600m long, trending approximately 045°. The anomaly is open to the west, but narrows quickly to the east at line L26700E. Two high zinc values of 1,253ppm and 1,298ppm are present within this anomalous zone. Copper values are up to 784ppm within the anomaly. To the south of the zinc anomaly, from L26300E to L26700E, a moderate copper anomaly is noted, but lacks continuity from line to line (Naas, 2006).
- **Avery Lake Grid**: Sample results indicate one persistent copper anomaly trending across the entire grid (over 900m). At its eastern end, it appears offset to the south, though it may be due to the nature of the topography should the anomaly be related to a flat-lying or shallow dipping horizon. The anomaly ranges from 50m to 200m wide. Soil samples within the anomaly returned up to 316ppm Cu (Naas, 2006).
- **Vavenby Grid**: Sample results indicate a possible weak copper anomaly approximately 25m to 100m wide and 400m long, trending close to east-west (090°). Copper values range from 61ppm to 122ppm within this zone. This grid covers a very small area and further work would be required to determine the nature of the anomaly (Naas, 2006).
- **NZ Soil Line:** Anomalous results of up to 119ppm Cu and 511ppm Zn were returned from this area (Naas, 2006).
- **Summit Grid**: Results of soil sampling do not show any strongly anomalous sample results. A weakly anomalous zone is possibly present with a sampling high value of 98ppm Cu. A single point anomaly of 243ppm Zn is also present but not as part of the copper anomaly.
- **Jones Grid:** Results of soil sampling do not show any strongly anomalous sample results. The highest single sample is 86ppm Cu.
- **Farmer's Grid:** Results of soil sampling do not show any strongly anomalous sample results. The highest single sample is 43ppm Cu (Naas, 2007).
- **M Anomaly Grid:** Sample results confirmed anomalies identified by the previous owner and further refined their location and orientation. Copper values returned are similar to those previously encountered (Naas 2006). Additional sampling in 2007 confirmed previous anomalies and further refined their location and orientation. Several copper zones have been outlined with widths of up to 200m and lengths of up to 700m, all trending NE-SW (Naas, 2007).
- **Harper South Grid:** An 800m long anomaly along the northern extent of the grid lines returned highly anomalous soil samples up to 3,522ppm Cu. This anomaly occurs within and immediately outside the proposed pit, representative of the surface expression the Deposit (Naas 2011). Another anomalous area of up to 1,150ppm Cu is observed in the western portion of the grid. The anomaly is approximately 450m long and ranges from 100m to 400m wide.

Figure 9-11: Soil Geochemistry Plan Map Copper (ppm), Harper Creek Project (1:75,000)



9.4 ROCK SAMPLING

Rock samples have been collected by YMI from various areas of the Project during work programs in 2006 (15 samples) and 2008 (336 samples). The 2008 rock samples were collected as part of a wider mapping program outside of the main Deposit area. The Harper Creek geodatabase for rock samples contains these 351 samples along with the results from 111 rock samples collected in 2004 and 2005 by the previous project owner.

Field samples were collected in order to capture changes in lithology, alteration and mineralization assemblages. Sample size varied but was large enough to incorporate a representative sample for assay (>100g). Samples were taken within historic trenches as well as from sub-crop, out-crop and float. All samples were marked with a sample number and placed in 8ml 8”x12” poly ore sample bags. Locations of samples were recorded using a Garman handheld GPS accurate to ±3m. Each sample location was marked in the field using orange or pink flagging with the corresponding sample number written on the flagging tape with a permanent black marker. All samples collected were given a lithological and alteration description as well as an estimated modal percentage of mineral and sulphide abundance.

In general, sampling has identified numerous significant copper and other base metal occurrences as well as several significant precious metal occurrences. A tabulation of the anomalous results is presented in Table 9-4.

Of the 351 samples collected by YMI, a total of 71 samples returned copper values greater than 1,000ppm Cu (0.1%) with seven greater than 1.0% Cu and a maximum of 3.98% Cu (sample 34464). This came from a float sample from the eastern part of the M Anomaly Grid area along the Saskum West Forest Service Road (FSR). North of this sample along the road, another sample returned 1.27% Cu (sample 5382). Other notable copper results include 3.18% Cu from a sample collected in the M Anomaly area south of the 1969 drill holes, 69-H-16 and 69-H-17. Further to the west and south of the Deposit, sampling returned significant copper results of 1.44% Cu and 1.35% Cu (samples 6309 and 6253, respectively).

Samples 5339 and 5400 both collected in the M Anomaly Grid area, returned 1.45 and 1.52 g/t Au, respectively. Both are grab samples from outcrop. Sample 5,339 is a narrow (2cm to 5cm) siliceous sulphide vein comprised of up to 10% pyrite with traces of chalcopyrite and malachite. Sample 5400 appears to be a quartzitic material with medium grained disseminated pyrite and arsenopyrite and trace chalcopyrite. Sample 6293 returned 138g/t Ag and 2.4% Pb, collected along the Jones Creek FSR (near the junction of Baker FSR). The sample material is noted as hematite stained quartz vein with chunky clots of galena.

Sample 5383 returned molybdenum values of up to 543ppm along with appreciable amounts of silver and lead (20.7ppm Ag and 2,652ppm Pb) collected in the M Anomaly area approximately 300m from the start of the Saskum West FSR. A sample collected in the western portion of the Harper South Grid area, returned 15ppm W, along with 1,158ppm Pb.

Table 9-1: Selected Anomalous Rock Samples

Sample No.	Location (UTM Zone 11 North)		Sample Type	Cu (ppm unless noted)	Ag (ppm unless noted)	Au (ppb unless noted)	Zn (ppm unless noted)	Other Elements
	Easting	Northing						
34464	308265	5710555	Float	3.89%	38.2 g/t	420	796	
34465	308266	5710556	Grab	4,937	11.1	55	182	
5260	308868	5709608	Grab	3.18%	16.2	55	264	
5307	308635	5709536	Grab	5,240	2.7	20	38	
5338	308774	5709708	Grab	556	48 g/t	135	3,316	2,046ppm Pb
5339	308850	5709808	Grab	9,204	29.6	1.45 g/t	186	
5345	308765	5709868	Grab	4,810	28.1	90	4,730	
5347	308788	5709888	Grab	5,156	4.5	30	102	
5350	308792	5709934	Grab	1,474	34.2 g/t	40	214	
5361	308737	5709948	Float	9,786	36.3 g/t	155	5,794	2,406ppm Pb
5370	309720	5709437	Float	6,178	4.5	20	74	
5372	309756	5709528	Float	3.55%	36.0 g/t	25	158	
5382	309794	5709796	Grab	1.27%	9.7	50	104	
5383	309794	5709796	Float	268	20.7	10	18	534ppm Mo 2,652ppm Pb
5392	309049	5710534	Grab	1,272	4.7	35	46	210ppm Mo
5400	308317	5710259	Grab	194	2.5	1.52 g/t	3,114	
5434	303682	5709183	Float	8,702	15.3	90	206	
5435	303990	5709084	Grab	5,914	5.5	35	90	
5454	304111	5709386	Grab	192	2.2	30	774	1,158ppm Pb 115ppm W
5456	304060	5709320	Float	700	4.2	380	2.80%	
5463	305162	5711323	Grab	2.25%	11.6	125	86	
5468	303546	5710491	Float	5,774	6.6	25	220	
6251	305263	5710191	Grab	4,044	8.8	275	310	
6252	304567	5709252	Grab	5,982	4.4	15	140	
6253	304558	5709331	Grab	1.35%	11.4	85	186	
6259	301060	5713570	Grab	446	42.3 g/t	20	4,884	2.80% Pb 173ppm W
6260	303622	5709333	Grab	980	8.7	90	1.33%	1,284ppm Pb 135ppm W
6263	304101	5710233	Float	1,078	29.9	265	2.16%	1.47% Pb 645ppm W
6264	304175	5710418	Float	7,242	22.3	75	242	
6293	306678	5712413	Float	12	138 g/t	70	4	2.40% Pb 121ppm W
6305	305705	5710545	Grab	5,468	5.5	110	306	
6309	305704	5710268	Grab	1.44%	22.8	185	936	
34068*	308266	5710555	Grab	4.71%	47.5 g/t	160	973	
34069*	308266	5710555	Grab	4,624	7.5	35	170	

*Non-YMI samples

9.5 PETROGRAPHIC STUDIES

During the course of exploration, petrographic studies, including thin and polished section analysis and whole rock analysis were undertaken on various drill core and rock sample specimens as part of an effort to better understand the lithology, alteration and mineralization characters of the Deposit.

As part of the 2007 and 2008 work programs, thin section analysis was conducted by S. Soloviev, *Ph.D.*, and J. Getsinger, *Ph.D.* on selected drill core samples. Twenty one thin sections were reviewed by Soloviev and 18 thin sections were reviewed by Getsinger. An additional 50 samples were selected for polished section analysis and examined by Soloviev (Naas and Soloviev, 2008).

Coupled with this work, 27 samples were submitted to Geo Labs of Sudbury, ON for assaying of major petrogenic oxides, losses on ignition (LOI) and other volatile components, and trace elements. Although all possible attempts were made to collect as much unaltered rock samples as possible, the analytical results indicate quite high degree of alteration. This was evident by high LOI values typically exceeding 4 to 6 weight percent, and as high as 11 to 14 weight percent in some rock varieties. Generally, the assays are “good” if LOI are less than 2 to 3 weight percent for most of the rocks, and less than 4 to 5 weight percent for lamprophyres. Subsequent analysis for CO_2, S and H_2O supports these high LOI values and clarifies as to what kind of alteration is superimposed (CO_2, i.e., carbonatization, contributes the most to the total alteration). Due to this alteration, care was taken with the interpretation of the results (Naas and Soloviev, 2008).

These aforementioned studies were undertaken prior to the development of the current geological model and therefore many of the lithological descriptions do not match the current terminology.

As part of the 2009 work program, 57 samples were sent to Vancouver Petrographics Ltd. of Langley, BC for preparation of thin sections and petrographic descriptions. Dr. Craig Leitch, Ph.D. carried out all petrographic descriptions (Naas, 2010). This work was in support of the development of the geological model. Additionally, whole rock analyses were obtained for all 57 petrographic samples. All samples were sent to Eco-Tech Laboratories Ltd. of Kamloops, BC for analysis (Naas 2010).

9.6 ADDITIONAL STUDIES

During 2009, a study on the Deposit was undertaken by Dr. R. Armstrong *Ph.D.* and T. Hawkins. The study included 7 days of reconnaissance field mapping and sampling, relogging of a drill hole (HC07-15), and re-assessment of the total digestion geochemical dataset (Armstrong and Hawkins, 2009). Additional petrographic analyses on previously prepared thin sections were also completed.

Assessment of field, logging, petrographic and geochemical data sets confirmed the style of mineralization is a volcanic-hosted massive sulphide deposit (VHMS). The key attributes that allow this conclusion are:

- the mineralogy and elemental assemblage - namely chalcopyrite+pyrite+galena;
- the tabular and broadly concordant nature of the mineralization;
- the strong spatial and temporal association of sulphide mineralization with sub-volcanic and volcanic dome complexes and associated volcanic breccias and hyaloclastites. The minor sub-volcanic intrusions

and volcanic domes cross-cut the sedimentary sequence. The recognition of hyaloclastic and jigsaw textures at outcrop and in existing drill core indicate that significant amounts of the volcanic activity took place in an active sedimentary basin;

- the alteration observed with drill hole HC07-15 is characterized by varying degrees of seritization, chloritization and silicification; and,
- the presence of other occurrences of base metal mineralization associated with volcanic rocks within Eagle Bay Assemblage of the Kootenay Terrane.

The presence of black shales within the sequence indicate that sedimentation occurred under anoxic conditions. This may have been a significant in the preservation of the dispersed sulphide mineralization at the Deposit. The sulphide mineralization is dominated by pyrite with pyrrhotite, chalcopyrite, and minor galena. The mineralization occurs in three distinct types:

- as a dispersed sulphide mineralization hosted by meta-sediments and volcanogenic lithologies;
- bedding/foliation parallel pyrrhotite and magnetite rich lenses; and,
- cross cutting chalcopyrite-rich stringers.

10 DRILLING

10.1 DIAMOND DRILLING

From 2006 to 2013 YMI has undertaken 4 main drilling programs; resource; condemnation; metallurgical; and geomechanical/geotechnical drilling. Two hundred seventeen drill holes, totalling 64,989.54m have been completed.

A summary of the drilling programs is presented in Table 10-1. Locations of the drill holes are presented in Figure 10-1 and 10-2.

Table 10-1: Summary of YMI Drilling

Drilling Purpose	Year(s)	No. Holes	No. Metres	Hole Size	Contractor(s)
Resource	2006-08 2010-13	165	58,611.94	NQ/NQ2	Atlas Drilling (2006-11) Matrix Drilling (2012-13)
Condemnation	2011	8	1790.98	NQ	Atlas Drilling
Metallurgical	2011	4	441.04	PQ	Atlas Drilling
Geomechanical / Geotechnical	2011-12	40	4145.58	HQ/HQ3	Atlas Drilling (2011) Westech Drilling (2012)
TOTAL		217	64,989.54		

CME Consultants Inc. of Richmond, BC (CME) was responsible for management of the resource, condemnation, and metallurgical drill programs. Knight Piésold of Vancouver, BC was responsible for management of the geomechanical and geotechnical drilling. Drill core is currently stored at CME's field office in Vavenby, BC.

Surface coordinates of diamond drill hole collars were surveyed by GPS methods. From 2006 to 2008, a Trimble GeoExplorer XT rover was utilized. The GPS data was differentially corrected using data from the Williams Lake public domain GPS base station. Accuracy is sub metre for easting and northing readings and 3 to 5m for elevation readings. Elevations used for all drill holes during this time period were determined by draping the collar coordinates over the one metre contour digital terrain model utilizing Micromine software. From 2008 to 2013, all drill collars except for one geotechnical drill hole (GT12-08), were surveyed using a Trimble GeoExplorer XH rover utilizing a Tornado antenna. The collected point data was differentially corrected using data recorded from Trimble 5700 base station and Zephyr antenna located at YMI's field camp, located 2.5 kilometres up Jones Creek FSR. Accuracy is sub-metre for easting, northing and elevation readings relative to the base station. Elevations used for all drill holes during this time period utilized the GPS readings. Drill hole GT12-08 was surveyed by hand-held GPS (Knight Piésold, 2013).

Figure 10-1: DDH Plan Map Harper Creek Project (1:40,000)

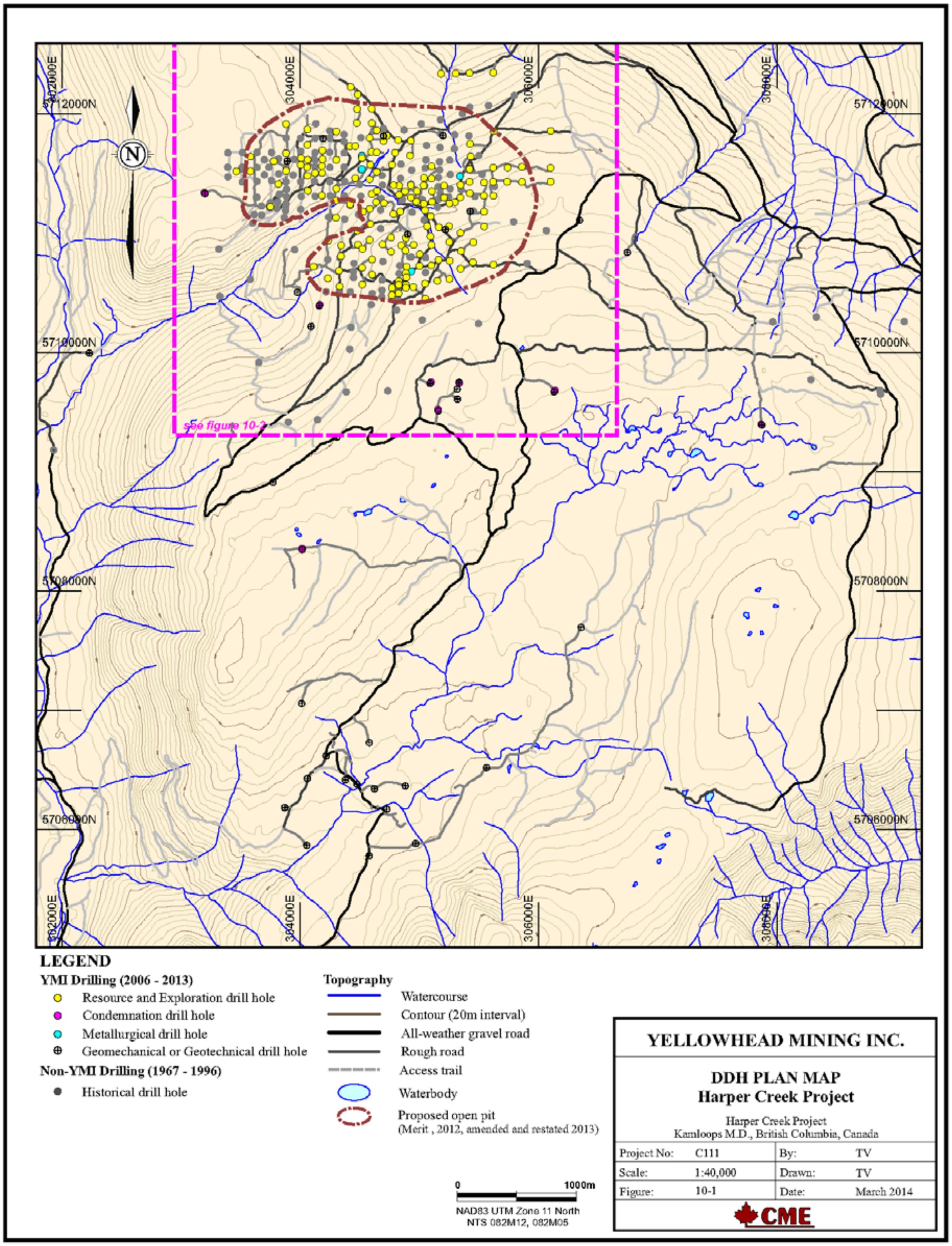
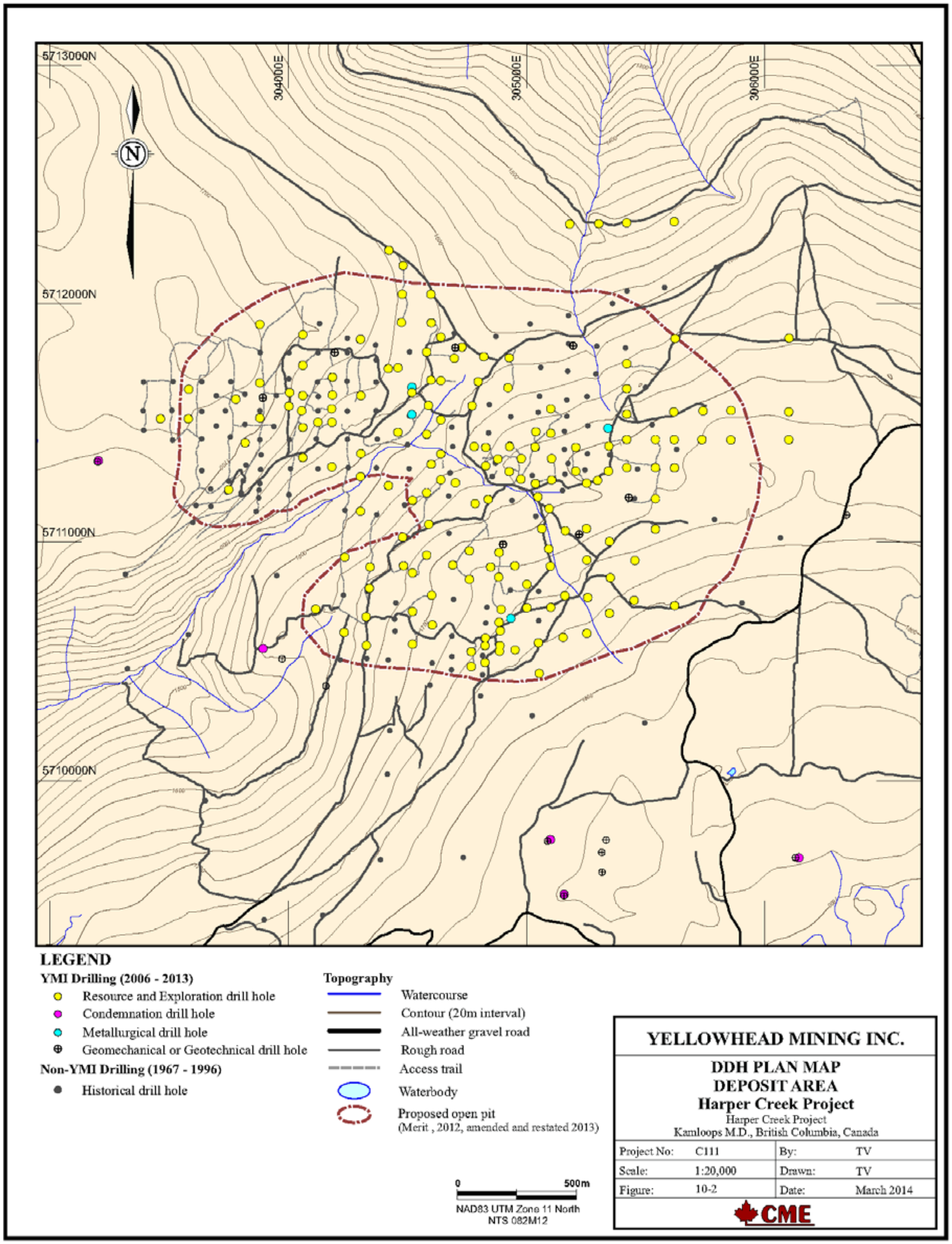


Figure 10-2: DDH Plan Map Deposit Area Harper Creek Project (1:20,000)



Of the 165 resource drill holes completed by YMI from 2006 to 2013, 158 were down hole surveyed for both azimuth and dip using four types of instruments: mechanical (Tropari), camera (Sperry-Sun), digital single-shot and digital multi-shot (Icefield, Reflex, Flex-it). Two drill holes were down hole surveyed for dip only using acid etch-method (HC06-01 and HC06-02) and five drill holes were not surveyed due to instrument failure (HC06-07, HC06-11, HC06-12, HC07-18, HC07-21).

All the condemnation and metallurgical drill holes were down hole surveyed for both azimuth and dip using digital multi-shot or single-shot instruments.

Of the 8 geomechanical drill holes, 5 drill holes were down hole surveyed. Holes HC11-GM03 and HC11-GM07 were not surveyed. HC11-GM01 was abandoned and re-drilled as HC11-GM01A. The geotechnical drill holes were not down hole surveyed.

10.1.1 *Resource Drilling*

Diamond drilling was undertaken at the Deposit, both within the main body of mineralization to increase the confidence of the resource as well as along strike and down dip to expand the resource. Geological and geochemical information obtained from this drilling was also used to develop and improve the geological understanding of the deposit model. This work was undertaken in seven phases from 2006 to 2013. A summary of this drilling by year is presented in Table 10-2.

Table 10-2: Summary of YMI Resource Drilling

Year	Drill Holes	No. Holes	Metres
2006	HC06-01 to HC06-12	7	4,101.40
2007	HC07-13 to HC07-52	5	15,879.94
2008	HC08-53 to HC08-75	26	7,602.92
2010	HC10-76 to HC10-82	37	3,486.92
2011	HC11-83 to HC11-130	12	15,571.31
2012	HC12-131 to HC12-142	12	3,803.29
2013	HC13-143 to HC13-165	23	8,166.16
	TOTAL	**165**	**58,611.94**

A total of 46,275 drill core samples were collected and submitted for geochemical analysis. Core sample lengths were selected to account for differences in lithology, alteration and mineralization. Mean sample length is 1.26m with a median length of 1.23m. Sampled core represents a total of 58,494.19m of the 64,989.54m drilled (90%).

Details of core handling and processing, logging procedures, and analytical techniques are reported in Section 11.

Results from the drilling program were utilized in the mineral resource estimate which is reported in Section 14.

10.1.2 *Condemnation Drilling*

In 2011, a diamond drilling program was undertaken to test for potential mineralization below the proposed mine site infrastructure locations. A total of 1,790.98m of NQ diamond drill core was drilled in eight drill holes, HC11-C01 through HC11-C08, as summarized in Table 10-3. A total of 571 drill core samples were collected and submitted for geochemical analysis (Naas, 2012b).

Table 10-3: Condemnation Drill Hole Specifications

Drill Hole	Location (UTM Zone 11 North, NAD 83)			Azimuth	Dip	Length	Area
	Easting	Northing	Elevation (m)	(degrees)	(degrees)	(m)	
HC11-C01	305332.05	5709748.34	1839.78	179.6	-61.0	203.30	Mill Building
HC11-C02	303156.08	5709520.76	1837.92	178.4	-60.9	200.25	Truck Shop
HC11-C03	305097.63	5709750.76	1833.87	189.2	-60.9	200.25	Coarse Ore Stockpile
HC11-C04	304015.90	5708356.27	1872.52	143.0	-88.4	200.25	Low Grade Storage
HC11-C05	306138.92	5709681.19	1816.88	181.3	-62.0	200.25	Waste Rock Storage
HC11-C06	303194.54	5711337.54	1723.93	177.2	-60.4	340.46	West Waste Storage
HC11-C07	307869.40	5709396.99	1831.99	171.6	-60.9	200.25	Waste Rock Storage
HC11-C08	304163.40	5710398.37	1635.50	306.9	-46.8	245.97	Primary Crusher

10.1.2.1 Geochemical Results

The results of drilling and subsequent assaying of drill holes are summarized in Table 10-4. In general, drill holes HC11-C01 through HC11-C05 and HC11-C07 exhibited no significant copper mineralization. Sulphide mineralization in these holes was primarily pyrite with minor pyrrhotite and rare, sporadic chalcopyrite at best.

Table 10-4: Selected Significant Intersections (≥1 metre)

Drill Hole	From (m)	To (m)	Length (m)	Cu (%)	Ag (ppm)	Au (ppb)	Zn (ppm)
HC11-C01	No significant results						
HC11-C02	No significant results						
HC11-C03	No significant results						
HC11-C04	No significant results						
HC11-C05	No significant results						
HC11-C06	104.70	106.00	1.30	0.25	5.5	374	2,480
	109.00	110.75	1.75	0.21	3.6	173	1,177
	129.95	131.27	1.32	1.04	6.7	49	640
	216.00	217.00	1.00	0.38	9.7	58	4,010
HC11-C07	No significant results						
HC11-C08	17.00	25.00	8.00	0.22	3.5	6	91
	49.00	70.00	21.00	0.31	5.8	78	177
	99.67	111.00	11.33	0.25	3.6	52	229
	209.40	211.40	2.00	0.16	1.4	904	151

Drill hole HC11-C06 was drilled below the West Waste Storage location. In general, sulphide minerals encountered in this hole were mainly disseminated and foliation parallel pyrite, locally up to 15%, with up to 5% patchy, sporadic sphalerite, and trace to 3% pyrrhotite. A combined total of 7.44m of copper mineralization was intersected throughout the entire hole in four narrow (<2m) discrete intersections beginning 104.70m. The highest value returned was 1.04% Cu over 1.32m (129.95-131.27m).

Drill hole HC11-C08, encountered the most significant copper mineralization with up to 0.31% Cu over 21.00m (49.00m-70.00m). As with the other the drill holes, sulphide mineralization encountered consisted primarily pyrite. Mineralization style ranged from disseminated to foliation parallel to fracture fill to massive. A massive sulphide zone was intersected between 187.78m to 189.86m depth, consisting of approximately 40% pyrite, 40% pyrrhotite, 10% magnetite, and 1-2% chalcopyrite with quartz and iron carbonate gangue minerals. Copper concentration within this horizon was 0.21% Cu over 1.00m (188.00m-189.00m). Two gold intersections greater than 1.0g/t were also encountered: 1.37g/t Au over 1.00m (189.00m-190.00m) and 1.14g/t Au over 1.00m (210.40m-211.40m).

10.1.3 *Metallurgical Drilling*

As part of the Feasibility Study in 2011, a diamond drilling program was undertaken to collect drill core for metallurgical and crushing/grinding test-work. Drill holes were designed to twin 4 historical drill holes, NH-27, NH-29, J-4, and 69-H-22. A total of 441.04m of PQ diamond drill coring was completed. Samples collected for crushing/grinding test-work were sent to Dawson Metallurgical Laboratories of Midvale, Utah. All remaining core was sent to G&T Metallurgical Services in Kamloops, BC (Naas, 2011b). Results of metallurgical test-work are discussed in Section 13.0.

In addition to the samples collected for the metallurgical test-work, 137 drill core samples were also collected and submitted for geochemical analysis (all from drill hole HC11-M04).

10.1.4 *Geomechanical and Geotechnical Drilling*

As part Feasibility Study in 2011, a series of geomechanical and geotechnical drill holes were completed as part of Knight Piésold's site investigation studies and reported in Knight Piésold (2012).

Geomechanical drilling was undertaken in the proposed pit area (Figure 10-2) and consisted of 8 HQ drill holes totaling of 2,433.13m (HC11-GM01, HC11-GM01A, and HC11-GM02 to HC11-GM07). In addition to samples selected by Knight Piésold for the geomechanical studies, a total of 1,025 drill core samples were collected and submitted for geochemical analysis from 6 of the 8 geomechanical holes (Naas, 2012c).

Geotechnical drilling was undertaken in various areas of proposed mine infrastructure and consisted of a 24 HQ drill holes totaling of 1,269.95m (HC11-GT01 to HC11-GT24) in 2011. A total of 191 drill core samples were collected and submitted for geochemical analysis from 13 of 24 geotechnical holes (Naas, 2012c).

Additional geotechnical drill holes were completed in 2012 (Knight Piésold, 2013). This consisted of 8 HQ3 drill holes totaling 442.50m (HC12-GT01 to HC12-GT08). No drill core samples were collected for geochemical analysis.

10.2 Drill Core Relogging and Resampling

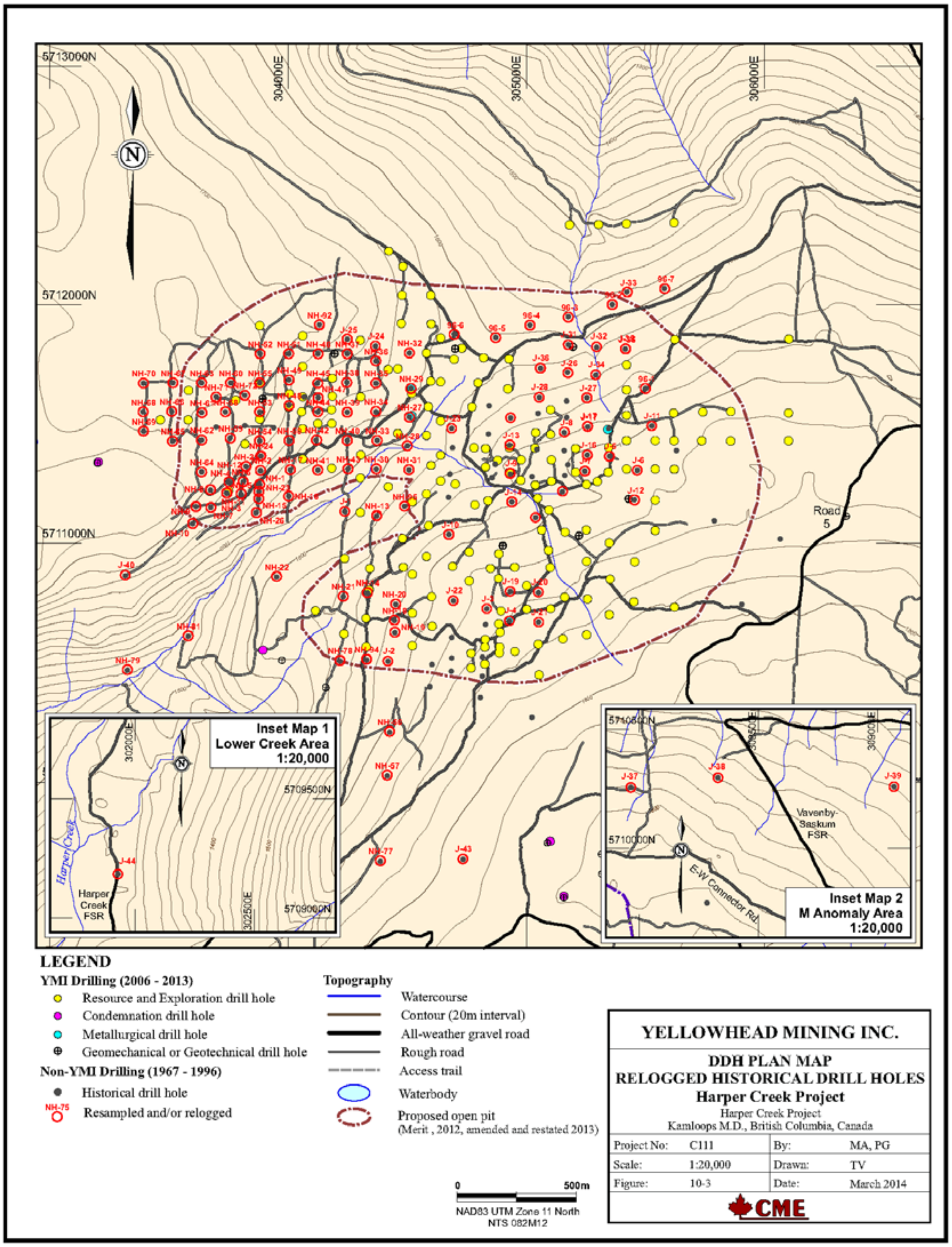
10.2.1 *Historical (pre-YMI) Drill Core*

Drill core from Noranda, US Steel and American Comstock drilling (referred to as historical drilling) was salvaged from the abandoned Noranda camp located on the Project. Sample pulps from the 1996 drilling program by American Comstock were located at Acme Analytical Laboratories Ltd. of Vancouver, BC.

Of the 191 drill holes completed prior to YMI's involvement, a total of 131 drill holes have been resampled and/or relogged. This amounts to 18,874.31m of drill core of the 30,745.12m drilled by the historical operators. A total 9,465 drill core samples were submitted for geochemical analysis. Of the 131 drill holes resampled, 127 are located within, or immediately adjacent to, the Deposit area. Of the remaining four drill holes, three were located in the M Anomaly area and one was located in the Lower Creek area, west of the Deposit (Fig. 10-3).

Resampling efforts were initially undertaken to verify the historically reported copper grades. Historical analyses had rarely reported on other elements, in particular precious metals, so gold and silver analyses were also undertaken on the historical core samples.

Figure 10-3: DDH Plan Map Relogged Historical Drill Holes, Harper Creek Project (1:20,000)



11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 SAMPLE PREPARATION

11.1.1 *FIELD PREPARATION AND SECURITY*

11.1.1.1 Soil

Sample material was bagged in labeled kraft sample bags and dried prior to shipping. Drying was undertaken in the CME core processing facility on plastic flow through shelving. Bar codes of the dried samples were scanned to record sample numbers prior to being placed into rice bags for transport.

11.1.1.2 Rock

Field geologists collected or supervised acquisition of all rock samples. Samples were placed into labeled poly ore bags and stored in the core processing facility. Samples were cut by rock saw for a library reference before shipping in rice bags to the laboratory.

11.1.1.3 YMI Drill Core

At the end of each shift, the drill contractor transported the drill core in secured wooden core boxes to CME's core processing facility in Vavenby, BC. Prior to core analysis and core logging, the drill core was sorted by CME personnel in an effort to find any mislabels and/or obvious defects or damage. Drill core was then washed to remove dirt and grease and refitted so successive pieces of core match.

Start and end depths of each core box were determined and marked on the respective corners with permanent felt marker. Aluminum tags were inscribed with the drill hole name, box number, start and end depths and affixed onto the front of the box. Once labelled, the core was photographed for a permanent record prior to further core analysis. Core analysis has included core recovery and rock quality designation, magnetic susceptibility, density testing, core orientation, geological logging, and geochemical sampling.

Visitors to the core processing facility were under supervision by the senior geologist.

11.1.1.3.1 Core Recovery & Rock Quality Designation

Drill crews record depths determined from imperial length drill rods by placing marked wooden blocks in the core box at the end of each drill run. Lengths were recorded in feet and metres with 10ft (3.05m) incremental interval being the most common. Both recovery and RQD measurements were determined for each drill run.

Recovery is a measure of actual core recovered against the interval drilled and is calculated by dividing the measured sum of all pieces (actual) over a drilled interval by the length of that interval. Recovery is reported as percent of the interval drilled.

Rock Quality Designation (RQD) is the measured sum of all pieces of core exceeding 10cm over an interval divided by the calculated length of the interval. RQD is reported as a decimal. Lengths were measured at right angles to the core axis. Rarely were fractures at right angles; midpoints on angled fractures were selected and measured. Core that appears to be fractured was removed from the box, examined and stressed with hand pressure to determine motion on the fracture plane. With motion or breakage the piece is discounted.

Recovery and RQD measurements were recorded on paper after which the data was entered into spreadsheet that calculated the recovery and RQD values. Average recovery for all drilling is 98%. RQD for all of drilling ranges from 0.00 to 1.00 with an average of 0.39 that translates to a rock quality designation of 'poor'.

11.1.1.3.2 Magnetic Susceptibility

All drill core, except for core from the 2012 geotechnical drilling, was tested for magnetic susceptibility using an Exploranium KT-9 Kappameter (2006-11) and a KT-10 Kappameter (2012-13). Measurements are dimensionless "SI units". Up to 10 readings were taken between the depth markers in the box, then averaged and recorded on paper.

Magnetic susceptibility readings for all phases of drilling range from below detection limit (0.01) up to 744.9 with an average value of 1.795. Very high magnetic susceptibility readings are consistent with occurrences of magnetite and pyrrhotite.

11.1.1.3.3 Density Testing

Density testing was performed on 10,739 core samples marked for geochemical analysis from drill holes HC06-01 to HC07-39 (excluding HC06-08). The method for determining density was the weight of core in and out of water. An Ohaus Scout Pro digital balance (2.0kg capacity, 0.1g sensitivity) was used for all weight determinations. Two pieces of core from each analytical sample were tested with the final density determined by averaging the two values. In a few cases only one piece of core could be tested due to lack of sufficient sample for a second test or inappropriate sampling material (i.e. moist clay gouge material).

Prior to testing, water temperature of the bath was recorded on the test sheet. Also, the balance was calibrated using a 2kg standard weight. The wire hanger was then hung and tared (zeroed) off.

The length of core was measured and reported on the test sheet. Core length was limited by the diameter of the water bath (23cm) though in general size was typically 10cm to 12cm. Weight of each piece was determined by hanging the core in a simple wire hanger and then recorded on the test sheet. The water bath was then placed under the scale and each piece of core was reweighed while hanging directly in the water bath. The "wet" weight was then recorded on the test sheet.

The raw data was entered into a spreadsheet for calculation of the specific gravity (S.G.) using the formula:

$$S.G. = \frac{(Dry\ Weight)}{(Dry\ Weight) - (Wet\ Weight)}$$

Density of the material is then calculated using the formula:

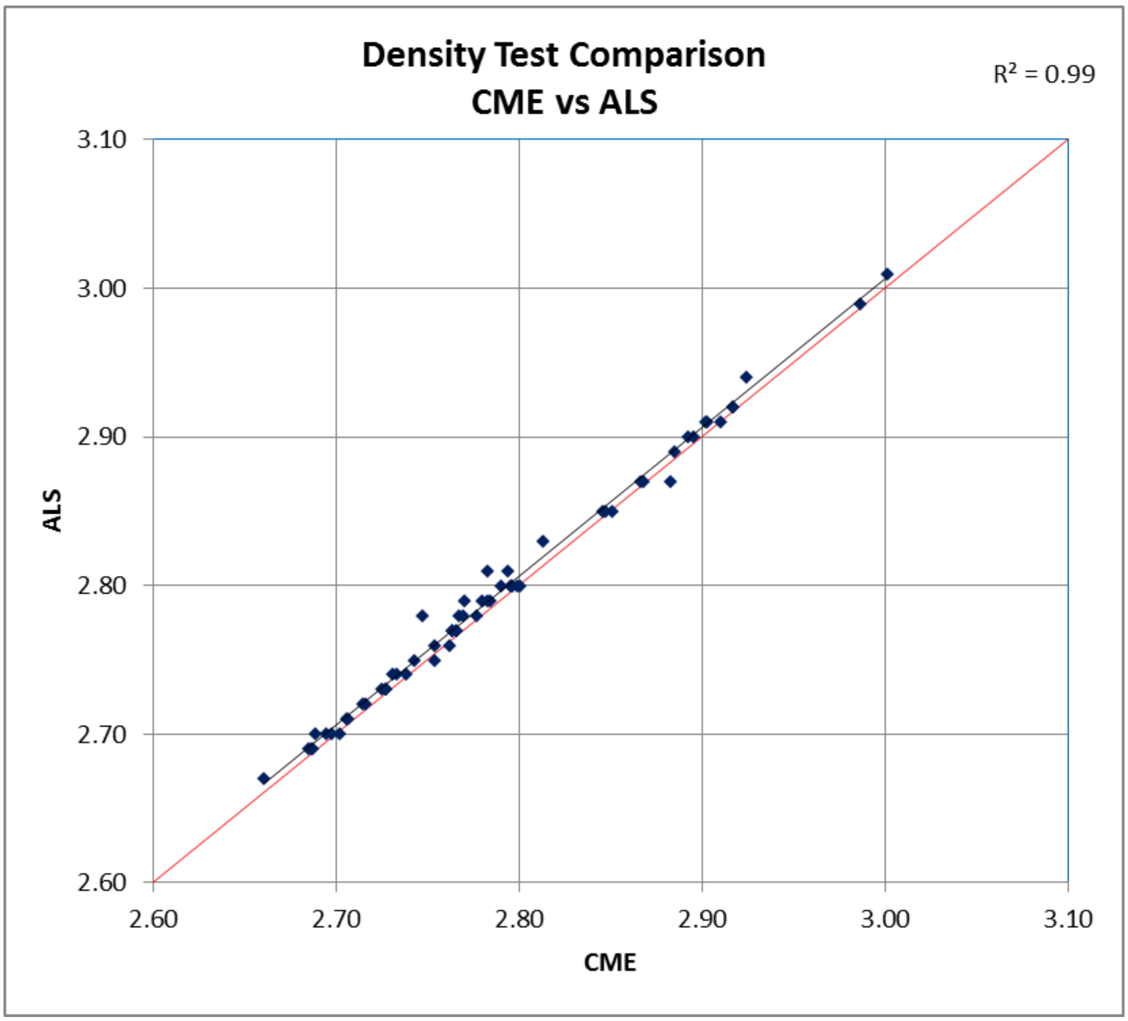
Density = S.G. X Density of water

Water density is sensitive to temperature, so water temperature was recorded at the time of each core sample reading.

Average density of all samples is 2.80t/m^3. The median density is 2.78t/m^3.

Sixty of the tested specimens were re-analyzed at ALS laboratory in 2012 and no significant differences were found (Figure 11-1).

Figure 11-1: Comparison of Specific Gravity Measurements



Geosim Consultants, March 2014

11.1.1.3.4 Core Orientation

Drill core orientation was undertaken on drill core from 26 drill holes during programs in 2006 and 2007. Orientation tests were performed at the end of each drill run in competent zones using an Ezy-Mark orientation tool provided by 2iC Australia Inc. of North Vancouver, BC. This tool consists of a cylinder with a series of spring loaded nails arranged around its lower face aligned down the hole. Three ball bearings are located inside the cylinder. When the tool is engaged, the ball bearings align and mark the lowest point in the cylinder. The spring-loaded nails extend until they contact the surface of the piece of core immediately below, matching its surface shape. Upon removal of the tool and corresponding core, the tool and core are rotated until they fit together, and a line is drawn on the core parallel to the core axis marking the position of the ball bearings. When orienting core at the core processing facility, the core from each drill run is pieced together until a snug fit is achieved. The orientation line made with the Ezy-Mark tool at the end of the drill run is then extended through the length of the core. Wherever possible, orientation is double-checked by working backward from the orientation mark on the subsequent drill run.

A total of 1,933 measurements have been collected to date. An average of the foliation measurements returned an orientation of 259°/30°N which agrees with field observations.

11.1.1.3.5 Geological Logging

Core logging was carried out by the field geologists, conducting both summary and detailed geological descriptions of the drill core. Key geological parameters such as lithology, alteration and mineralization were recorded in non-digital and digital format.

11.1.1.3.6 Geochemical Sampling

Geochemical samples were marked out by coloured lumber crayon on the drill core based on lithology, alteration and mineralization by the geologists. The relevant lithology codes were assigned to each sample. A sample tag with sample number, corresponding bar code, and sample start and end depth was inserted for each marked sample.

Core was cut by 10in electric rock saw operated by both CME and YMI contracted personnel. Half of the core was placed into a poly ore sample bag, with the remaining core returned to the core box.

Sample tags, which included a sample number and a corresponding bar code, accompanied each core sample marked for geochemical analysis. Sample tags consisted of three parts. One part was stapled to the core box at the end of the sampled length and reported the start and end depth of the sample. The remaining two parts contained no sample location information. One tag was placed inside the plastic sample bag containing the core and the other stapled to the top of the same bag, exposing the bar code for later scanning.

The poly ore sample bags containing cut core were passed through a heat sealer to secure the contents prior to shipping.

11.1.1.4 Historical Drill Core

Salvaged core boxes were cross-stacked in the open at the former Noranda camp. No security restrictions were in place with this core, as it had been in public storage since the early 1970's. Initial sampling of the historical core was conducted in the field at the former Noranda camp. In 2008 salvaged core was moved to CME's core processing facility for later analysis.

Only geological logging and geochemical sampling was carried out on the historical drill core. Logging and sampling followed similar procedures as for the YMI drill core described above with some differences as noted below.

Due to the schistosity of the core, quartering of historical core was not possible, so the entire half core was submitted for analysis. Sampling of the drill core was undertaken at a maximum of 10-foot intervals with variations in length made to isolate geological areas of interest. Some core boxes were in very poor condition with some missing sections of core. The missing interval start and end were estimated and reported as "no sample" ("N/S").

11.1.1.5 Laboratory Delivery

For shipping, soil, rock and core samples were sorted and scanned to record sample numbers prior to placing into rice bags for transport Greyhound Courier. From 2008 to 2013, the analytical laboratory took responsibility for transporting samples from the core processing facility.

11.1.2 *Laboratory Sample Preparation*

Eco-Tech Laboratories Ltd. (Eco-Tech) of Kamloops, BC undertook sample preparation from 2005 to 2011. In August 2008, Eco Tech was purchased by Stewart Group. In July 2011, Stewart Group was purchased by ALS. Sample preparation continued in Kamloops, but under ALS.

11.1.2.1 Soil

Soils samples submitted to Eco-Tech (2006-07) and Stewart Group (2008) were prepared by sieving through an 80-mesh screen to obtain a minus 80-mesh fraction (178µm). Samples unable to produce adequate minus 80-mesh material were screened at a coarser fraction. These samples were flagged with the relevant mesh.

11.1.2.2 Rock and Drill Core

11.1.2.2.1 Eco-Tech

Rock and drill core were crushed through a jaw crusher and cone or roll crusher to -10 mesh screen size (1.68mm). The sample was then split through a Jones riffle until a 250g sub sample was achieved. The sub sample was pulverized in a ring and puck pulverizer to 95% passing through a Tyler 150 mesh screen (104µm). The sample was then rolled to homogenize.

11.1.2.2.2 Stewart Group

Rock and drill core were crushed on a Terminator jaw crusher to 10 mesh ensuring that 70% passes through a Tyler 10 mesh screen (1.68mm). Every 35 samples a re-split was taken using a riffle splitter to be tested to ensure the homogeneity of the crushed material. A split of up to 250 grams is taken and pulverized to better than 95% passing a Tyler 150 mesh screen (104µm). The sub sample is rolled, homogenized and bagged in a pre-numbered bag. A barren gravel blank was prepared before each job in the sample prep to be analyzed for trace contamination along with the processed samples.

11.1.2.2.3 ALS

Drill core was crushed to better than 70 % passing a Tyler 9 mesh screen (2mm). A split of up to 250g was taken and pulverized to better than 85% passing a Tyler 200 mesh screen (75µm).

11.1.2.2.4 Geoscience Laboratories

Core samples submitted to Geoscience Laboratories of Sudbury, ON for whole rock and trace element analyses were prepared using a small jaw crusher with steel plates, a riffle to split the samples and pulverization in a 99.8% pure Al_2O_3 planetary ball mill (a minor amount of Al is added to the sample) (Naas and Soloviev, 2008).

11.2 Sample Analysis

All samples were submitted to Eco-Tech for analysis until 2011. In August 2008, Eco Tech was purchased by Stewart Group. Eco-Tech was registered for ISO9001:2008 for the provision of assay, geochemical and environmental analytical services.

In July 2011, Stewart Group was purchased by ALS. At this time, analysis was shifted to the ALS facility in North Vancouver, BC. ALS is an ISO17025 accredited testing laboratory.

As part of the QA/QC protocol for drilling programs, a portion of the core samples were submitted to a second laboratory for check analysis. Samples were submitted to Acme Analytical Laboratories Ltd. of Vancouver BC (Acme) for the 2006 and 2010-13 drill programs and Assayers Canada of Vancouver, BC (now SGS Group) for the 2007-08 drill programs (Ref. Section 11.3.2).

Soil and rock samples were analyzed for multi-elements used an Aqua Regia ICP and for gold by geochem level Aqua Regia/Fire Assay. Overlimit (>30ppm Ag, >10,000ppm Cu, Pb, Zn) ICP analyses of rock samples were subsequently assayed for the appropriate element. Overlimit gold analyses of rock samples (>1,000ppb Au) were subsequently fire assayed.

YMI drill core samples were analyzed for multi-elements by Total Digestion ICP. Samples were analyzed for gold by geochem level Aqua Regia/Fire Assay. Copper assays were undertaken on samples greater than 2,900ppm Cu based on the ICP analysis from 2006 to 2008. From 2010 onward drill programs undertook assays on samples greater than or equal to 2,000ppm Cu based on the ICP analysis except during the latter portion of the 2011 Feasibility Study drill program where all sample batches analyzed by ALS were assayed for copper, regardless of the ICP copper value. Assays of silver, lead and zinc were performed only where ICP values were greater than detection limit. Palladium was also analyzed for during the 2005 work program.

Selected rock and drill core samples were analyzed for whole rock concentrations (Naas, 2008, 2010). Methodologies of the various analytical techniques are presented below.

11.2.1.1 Aqua Regia ICP

11.2.1.1.1 Eco-Tech

A 0.50g sample was digested with a 3:1:2 ($HCl: HN0_3: H_20$) solution in a water bath at 95°C. The sample was then diluted to 10ml with water. All solutions used during the digestion process contain Indium, which acts as an internal standard for the ICP run. The sample was analyzed on a Thermo iCap 6000 ICP unit. Certified reference material was used to check the performance of the machine and to ensure that proper digestion occurred in the wet lab. QC samples were run along with the client samples to ensure no machine drift occurred or instrumentation issues occurred during the run procedure. Repeat samples (every batch of 10 or less) and re-splits (every batch of 35 or less) were also run to ensure proper weighing and digestion occurred.

11.2.1.2 Total Digestion ICP

11.2.1.2.1 Eco-Tech

A prepared 0.50g sample was digested with nitric acid and hydrochloric acid, then hydrofluoric and perchloric acids. The sample was dried and subsequently re-dissolved with 3ml of a 3:1:2 ($HCl, HN0_3:H_20$) which contains beryllium and was then diluted to 10ml with water. The sample was then analyzed on an ICP-AES instrument and reported in ppm and %.

11.2.1.2.2 ALS

A prepared 0.25g sample was digested with perchloric, nitric, hydrofluoric and hydrochloric acids. The residue was topped up with dilute hydrochloric acid and the resulting solution was analyzed by ICP-AES and elements were reported in ppm and %.

11.2.1.3 Gold Geochem

11.2.1.3.1 Eco-Tech

Gold analyses utilized a 30g sample size and were fire assayed using appropriate fluxes. The resultant dore bead was parted, digested with aqua regia, and then analyzed on a atomic absorption instrument and reported in ppb. Historical drill core samples collected in 2005 were analyzed for palladium (Lefebvre, 2006) using this same technique.

11.2.1.3.2 ALS

A prepared sample was fused with a mixture of lead oxide, sodium carbonate, borax , silica and other reagents as required, inquarted with 6mg of gold-free silver and then cupelled to yield a precious metal bead. The bead was digested in 0.5ml dilute nitric acid in the microwave oven, 0.5ml concentrated hydrochloric acid was then added and the bead was further digested in the microwave over at a low power setting. The digested solution was cooled, diluted to a total volume of 4ml with de-mineralized water and analyzed by atomic absorption spectroscopy against matrix matched standards. Results were reported in ppm.

11.2.1.4 Base Metal Assay

11.2.1.4.1 Eco-Tech

Base metal assay samples and standards underwent an oxidizing digestion in 200ml phosphoric flasks with final solution in aqua regia solution. The digested solutions were made to volume with RO water and allowed to settle. An aliquot of the sample was analyzed by atomic absorption instrument and reported in %.

11.2.1.4.2 ALS

A prepared sample was digested with perchloric, nitric, hydrofluoric and hydrochloric acids and then evaporated to incipient dryness. Hydrochloric acid and de-ionized water was added for further digestion and the sample was heated for an additional allotted time. The sample was cooled to room temperature and transferred to a volumetric flask (100ml). The resulting solution was diluted to volume with de-ionized water, homogenized and the solution was analyzed by ICP-AES or by AAS. Results were reported in %.

11.2.1.5 Whole Rock and Trace Element Analyses

11.2.1.5.1 Geoscience Laboratories

Whole rock and trace elements analysis were performed by Geoscience Laboratories of Sudbury, ON. Analyses conducted included:

- X-ray fluorescence (XRF) for major oxides and selected trace elements (Co, Cs, Mo, Sc, Sn, V);
- ICP Mass Spectrometry for rare-earth/high field strength elements/large-ion lithophile elements analysis;
- anion analyzer for chlorine and fluorine;
- total carbon and sulphur,
- ferrous iron; and
- moisture content.

Detailed analytical procedures were not available.

11.2.1.5.2 Eco-Tech

The sample was weighed out with 0.5g of $LiBO_2$ (lithium metaborate) into a graphite crucible. The sample was fused in a furnace and then digested in an HNO_3/HF solution. The solution was then cut with a 5% HCl and a 2% HNO_3 solution. The samples were analyzed on a Thermo Scientific IRIS Intrepid II XSP ICP unit. Synthetic standards prepared by SCP science were used to calibrate the ICP-AES. All synthetic standards were purchased and verified by three independent analysts and were used for instrument calibration before each and every ICP-AES run.

11.3 Quality Control

Quality control protocol was implemented for samples submitted for geochemical analysis. Control samples, consisting of certified reference material (CRM) and blank material, were inserted into the sample sequence prior to submission to the laboratory.

Check samples, consisting of 4 to 5% of the core samples, were submitted to a different analytical laboratory.

11.3.1 *Control Samples*

Certified reference materials (CRM) were obtained from CDN Resource Labs of Langley, BC and Analytical Solutions Ltd. of Toronto, ON as prepackaged 60g to 150g sample sizes. A list of the CRM's employed is presented in Table 11-1.

Table 11-1: List of Certified Reference Material

CRM	Recommended Values	
	Copper	Gold
CDN-CGS-6	0.318 ± 0.018%	0.26 ± 0.037g/t
CDN-CGS-8	0.105 0.008%	0.080 0.012g/t*
CDN-CGS-9	0.473 ± 0.025%	0.34 ± 0.034g/t
CDN-CGS-12	0.265 ± 0.015%	0.29 ± 0.04g/t
CDN-CGS-15	0.451 ± 0.020%	0.57 ± 0.06g/t
CDN-CGS-22	0.725 ± 0.028 %	0.64 ± 0.06g/t
CDN-CGS-24	0.486 ± 0.034 %	0.487 ± 0.050g/t
CDN-CGS-27	0.379 ± 0.015 %	0.432 ± 0.046g/t
CDN-CGS-29	0.585 ± 0.034%	0.228 ± 0.030g/t
CDN-CM-25	0.191 ± 0.006%	0.191 ± 0.030g/t
CDN-CM-27	0.592 ± 0.030%	0.636 ± 0.068g/t
CDN-FCM-1	0.94 ± 0.07%	1.71 ± 0.147g/t
CDN-CM-1	0.853 ± 0.020%	1.85 ± 0.16g/t
CDN-HLLC	1.49 ± 0.06%	0.83 ± 0.12g/t
CDN-GS-P2A		0.229 ± 0.030g/t
Oreas 152a	0.385 ± 0.018%	0.116 ± 0.010g/t

**provisional value only*

The non-certified blank material consisted of either crushed granite tile (2005) or decorative landscape rock (2006-13). Certified blank materials were obtained from Analytical Solutions Ltd. as prepackaged 100 gram (fine blank) to 500g (coarse blank) sample sizes. The certified blank material was only used in the 2012-2013 drilling program.

11.3.1.1 Soil

One CRM was submitted for every 100 soil samples. No blanks were submitted with these sample batches. No quality control issues were noted.

11.3.1.2 Rock

Of the 3 batches of rock samples submitted (2006, 2008), 2 batches contained at least 1 CRM and 1 blank. No quality control issues were noted.

11.3.1.2.1 Drill Core: 2006-2008 Drill Programs

From 2006 to 2007, 1 non-blind reference and 1 non-blind blank were inserted into the sample sequence every 50 samples.

In 2007 and throughout 2008, the insertion of reference and blank samples was modified from non-blind to blind. Control samples were randomly inserted into each group of 50 samples.

Upon receipt of results, control samples were checked for any significant discrepancies from their recommended values (less than two standard deviations from the recommended value). No quality control issues were noted.

11.3.1.2.2 Drill Core: 2011 Drill Programs

In 2011, 1 blind reference and 1 blind blank were randomly inserted into each group of 33 samples.

Upon receipt of results, control samples were checked for any significant discrepancies from their recommended values.

All 87 sample batches analyzed by Eco-Tech returned acceptable values of copper and gold of the inserted SRM (less than two standard deviations from the recommended value).

Of the 103 sample batches analyzed by ALS, all returned acceptable copper values for the inserted CRM. A total of 17 (16.5%) of the ALS sample batches returned gold values of greater than 2 standard deviations (±2σ) from the recommended gold value. Of these 17 batches, 12 returned values within 3 standard deviations (±3σ) while the remaining 5 returned values outside 3 standard deviations (±3σ).

An investigation by ALS into the possible quality control issue involved re-analyzing the CRM along with several samples before and after in the sample sequence. Re-analysis of the CRM's returned acceptable values within the 2 standard deviation range.

An investigation by CME resulted in all samples from the 18 batches (including control samples) being sent to a second laboratory (Acme) for re-analysis of gold. Of the 18 submitted batches, 13 batches returned a CRM gold value within the recommended 2 standard deviation range. Of the remaining 5 batches, 4 returned an SRM gold value within 3 standard deviation range and the remaining batch fell outside of this range.

A comparison of the ALS and Acme gold results for the core samples indicated a positive bias of the Acme gold values with the Acme values reporting, on average, 21% higher than the original ALS values. However, with the exclusion of three samples that reported a very high variance (>1000%), the Acme values report and average of 7% higher than the original gold values. Variance of the sample population (589 samples, excluding standard reference and blanks):

- 60.4% of reported a variance of <100%;
- 32.9% reported a variance of 100 to 300%; and,
- 6.6% reported a variance of greater than 300%.

Graphing the gold variances against the original gold grade showed that the majority of high variance samples occur at the low grades of gold. Of the 39 samples that reported greater than 300%, 36 are a result of a below detection limit value (<0.005ppm; treated as 0.003 for calculations) reanalyzed and returning a detectable gold value of up to 0.023ppm Au. The remaining three reported a fairly large increase in the reported gold grade. These may represent statistical outliers (Naas, 2012a).

11.3.1.2.3 Drill Core: 2012-2013 Drill Program

Upon receipt of results, control samples were checked for any significant discrepancies from their recommended values. If a CRM returned a copper or gold value greater than two standard deviations from the mean value, ALS was notified and would run a re-analysis for the particular element(s) of the CRM that failed as well as several samples above and below it in the sample sequence (for ICP or gold analyses). If the CRM failure was from a copper assay, ALS would re-assay the CRM and all other core samples with a copper assay value.

Of the CRM failures that were re-analyzed by ALS, all but 3 returned values within two standard deviations from the mean value of the CRM. The remaining 3 CRM re-analyses that did not return values within the two standard deviation range were rerun for gold only (the respective original copper assays in all three batches passed QC). An examination of the re-analysis showed that the gold results of the core samples in all 3 sample batches were low-grade, from below detection limit (0.005ppm Au) to a maximum of 0.037ppm Au. At these low-grades, variance of the re-analyses could be quite large (e.g. original value of 0.006, reanalyzed as 0.012 equals 100% variance) (Naas, 2013).

11.3.2 *CHECK SAMPLES*

11.3.2.1 Check Samples: 2006–2008 Drill Programs

For drill programs from 2006 to 2008, approximately 5% of the drill core samples were submitted to either Acme or Assayers Canada.

Results of the check analysis demonstrated very good reproducibility of the original assay values. More detailed information on the results of the check sampling can be found in the various work program reports by Naas (2006, 2007) and Naas and Soloviev (2008).

11.3.2.2 Check Samples: 2010-2013 Drill Programs

For drill programs from 2010 to 2013, approximately 4% of the drill core samples were submitted to Acme.

Results of the check analysis demonstrated very good reproducibility of the original assay values. More detailed information on the results of the check sampling can be found in the various work program reports by Naas (2011, 2012a-d, 2013).

11.3.3 *RESULTS*

Geosim prepared Reduced Major Axis (RMA) plots to quantify the errors between matching data pairs (Table 11-2, Figure 11-2 to Figure 11-4) shows the performance of all duplicates run at Acme, where;

- n is the number of samples in the population
- Bias is the bias between the mean of the original and check assays
- CV_{ARV} is the average coefficient of variation
- Int is the intercept of the RMA linear model
- Slope is the slope of the RMA linear model
- S_{RMA} is the standard deviation of the data points around the RMA line
- R^2 is the Correlation Coefficient squared

Table 11-2 Acme Pulp Re-checks

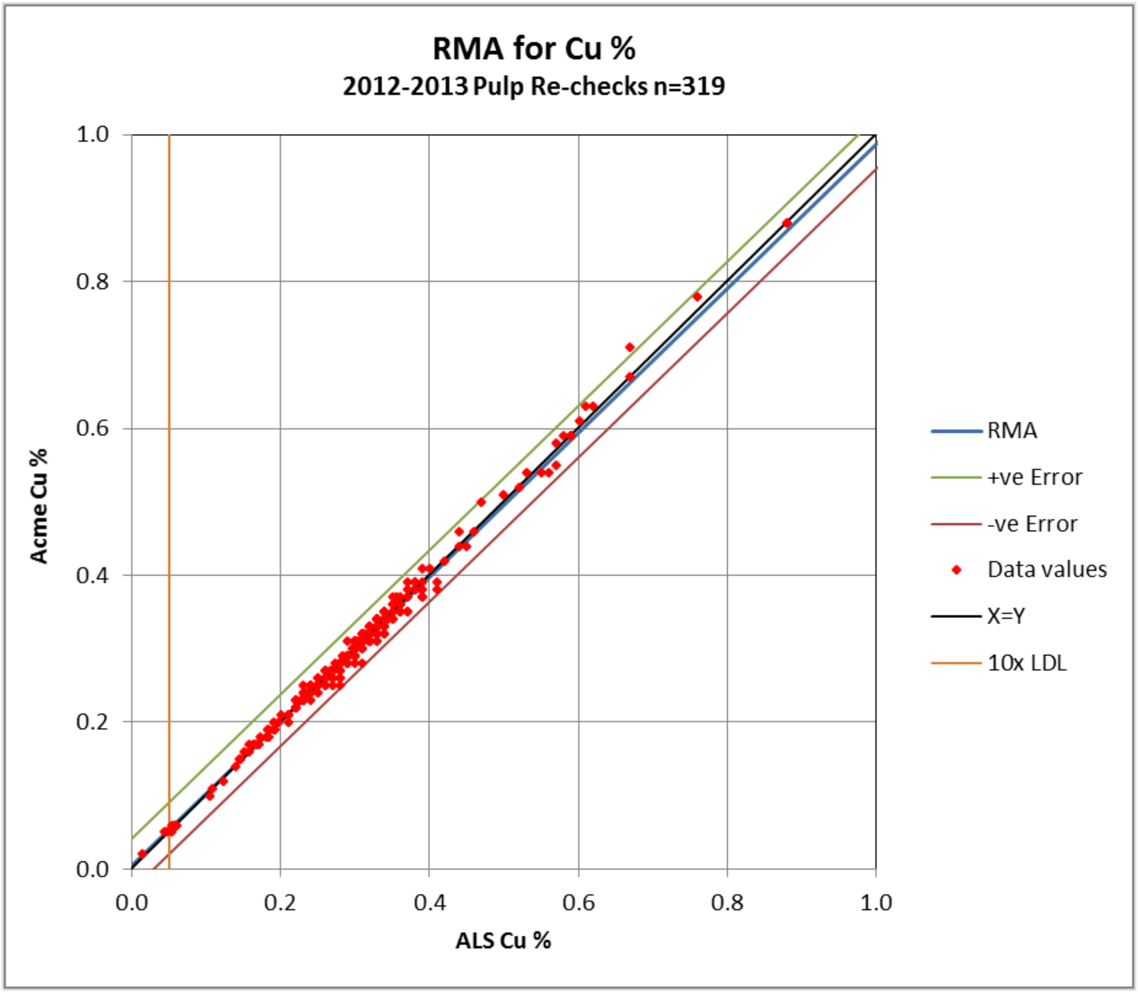
Item	n	Mean Orig	Mean Check	Bias	CV_{ARV} %	Int	Slope	S_{RMA}	R^2
Cu	319	0.411	0.409	0.45%	2.60	0.01	0.98	0.018	0.999
Au	309	0.045	0.058	-30.04%	38.36	0.01	1.00	0.027	0.941
Au > 0.025	166	0.071	0.084	-18.14%	23.59	0.01	1.00	0.036	0.937
Au > 0.05	59	0.138	0.150	-8.78%	16.72	0.01	1.00	0.053	0.936
Ag	310	2.60	2.85	-9.39%	15.99	0.08	1.06	0.585	0.978

Cu results show very good precision with a CV_{ARV} of 2.6% and no significant bias.

Au results overall show poor precision and significant bias but most of the data points were within 10x the lower detection limit. The precision increases significantly and the relative bias decreases above a threshold of 0.05g/t. The low precision associated with very low gold grades is not unusual and is not considered to preclude the use of this data in resource estimation.

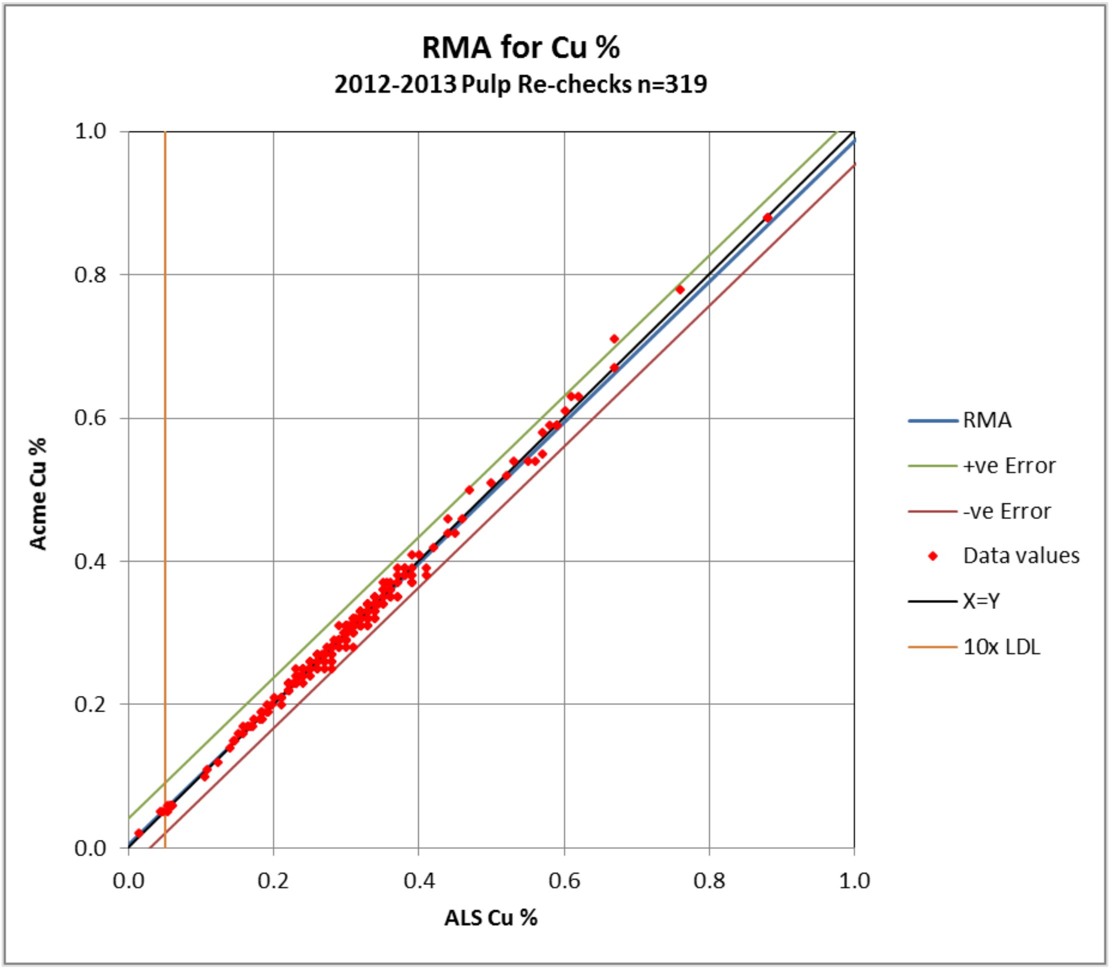
Ag results show similar precision and bias to Au results above the 0.05g/t threshold. In both cases ALS shows a low bias of about 9% compared to ALS.

Figure 11-2 RMA Plot for Cu re-checks at Acme



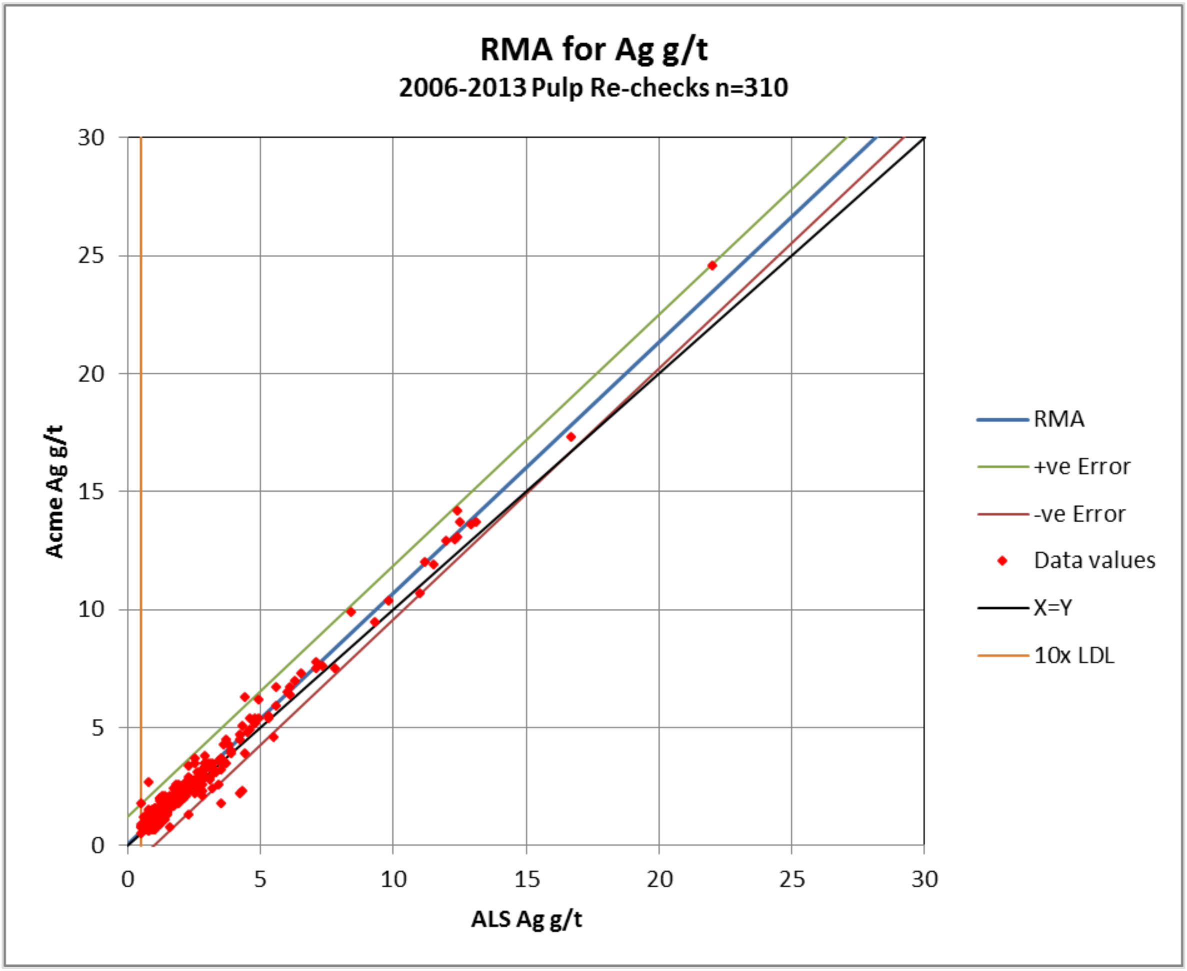
GeoSim Services Inc., March 2014

Figure 11-3 RMA Plot for Au re-checks at Acme



GeoSim Services Inc., March 2014

Figure 11-4 RMA Plot for Ag re-checks at Acme



GeoSim Services Inc., March 2014

12 DATA VERIFICATION

12.1 SITE VISIT

R. Simpson, P.Geo, visited the Project site on July 11 and 12, 2011. A tour of the project included area inspections of the proposed plant site, crusher, borrow pit, tailings dam, access roads and load out facility. YMI provided complete access to all areas of the Project relevant to the FS.

The primary purpose of the QP visit was to review the drilling, sampling, and quality assurance/quality control procedures. The geology and mineralization encountered in the drill holes completed to date were reviewed. During the site visit Mr. Simpson verified:

- Collar locations are reasonably accurate by comparing several drill hole database collar locations with hand-held GPS readings.
- Down-holes surveys are routinely taken at approximately 15m intervals using a Reflex single-shot unit.
- Drill logs compare well with observed core intervals.
- Core recoveries were generally high throughout the mineralized zones.

Specific gravity was determined using a water immersion method where the weight of the sample in air and in water was measured with a balance beam scale.

The QP did not collect independent samples but did visually observe copper mineralization in drill core consistent with reported grades.

12.2 DRILL HOLE DATABASE VERIFICATION

R. Simpson independently audited the sample database for location accuracy, down hole survey errors, interval errors and missing sample intervals. The QP also reviewed the sample QA/QC results.

12.3 OTHER DATA VERIFICATION

Verification of metallurgical, hydrological, environmental baseline and geotechnical data is discussed in the relevant sections of this Report. The data is concluded to be adequate to support the FS.

12.4 CONCLUSION

The QP's are of the opinion that the data is adequate to support a mineral resource and mineral reserve estimate as defined under NI43-101.

13 MINERAL PROCESSING & METALLURGICAL TESTING

13.1 INTRODUCTION

Harper Creek is an extensive volcanogenic hosted sulphide system in a series of sedimentary sandstones and siltstones making it different from other low-grade type copper deposits in British Columbia from a lithological standpoint but not mineralogically.

Following completion of the PEA metallurgical test work program certain aspects of the flowsheet indicated opportunities for improvement, in particular:

- Potential to increase grind size without adversely affecting concentrate grade and recovery; and
- Potential to simplify the reagent schedule.

As a result, these areas were the focus of metallurgical test work for the 2012 FS. In early 2011, a PQ drilling program was undertaken to source metallurgical samples for the program.

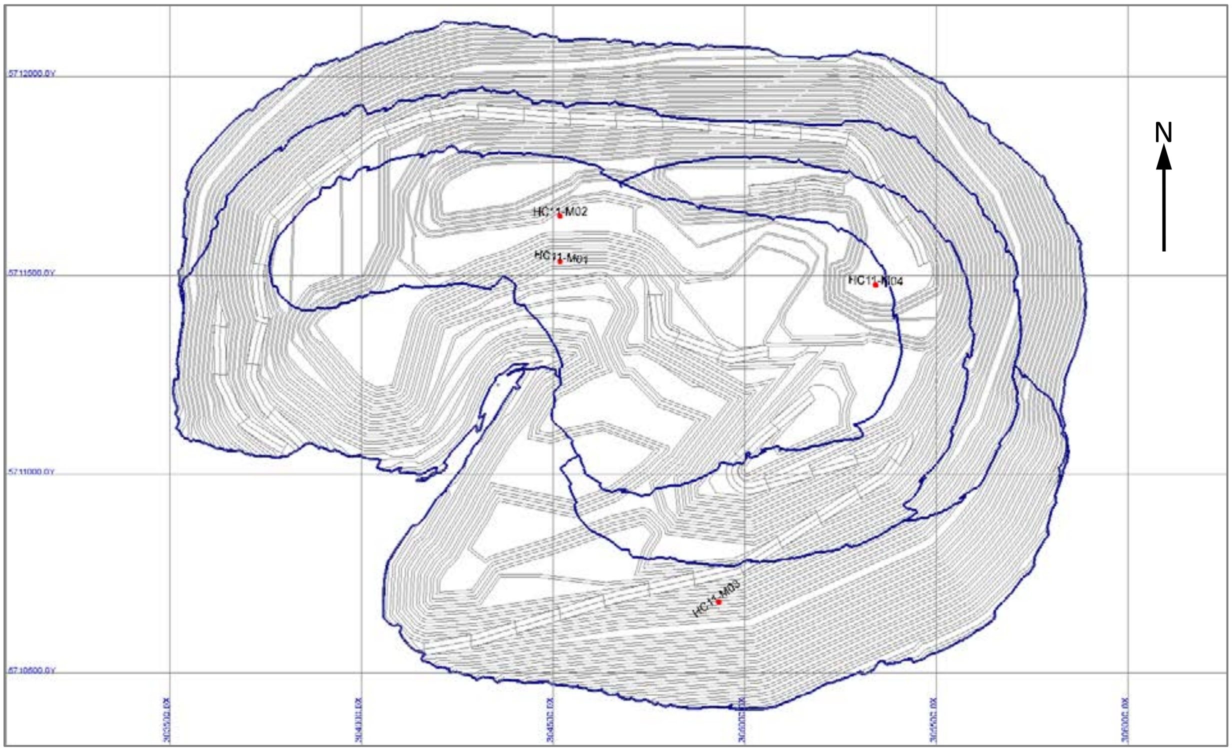
Test work for the 2012 FS was undertaken during 2011 and early 2012 by G & T Metallurgical Services Ltd. (G&T), of Kamloops, BC. In addition, FLSmidth (FLS) of Bethlehem, PA performed laboratory comminution test work, which was further reviewed by KWM Consulting Inc (KWM) of Vancouver, BC to confirm grinding power requirements. (“Results of Crushing and Grinding Index Tests on Nine Core Samples from the Harper Creek Project”, FLSmidth Dawson Metallurgical Laboratories, July 22, 2011. “Compressive Strength, Impact Strength, Ball Mill Bond Mill Work Index and Abrasion Analysis”, FLSmidth Inc., July 2011. “Grinding Circuit Evaluation for the Harper Creek Project”, KWM Consulting Inc., October 7, 2011).

13.2 MINERALIZATION CHARACTERISTICS

13.2.1 *SAMPLES*

The PQ drilling program was undertaken to source samples for the FS metallurgical test work program. In all, 4 holes were located (Figure 13-1) to ensure a representative sample of lithologies and grades covering the first 10 years of mine production was collected. A total sample weight of 5,261kg was sent to G&T for testing, and 752kg was sent to FLS for comminution test work.

Figure 13-1 Metallurgical Test Hole Locations



Yellowhead Mining Inc., July 2014

An estimate on the relative abundance of the major lithologies was made from the drill hole database as detailed in Table 13-1. Intercepts from the four PQ metallurgical holes were made into composites representing ten major lithologies and grades. These grade/lithology composites were partially used to make a Master Composite in proportion to the general lithologies detailed in Table 13-1. Unused composites and individual intercepts were stored and sealed under nitrogen in a freezer for future test work.

Table 13-1 Lithological Breakdown of the Harper Creek Deposit

Lith Unit	Lith Description	Total Metrage	Percentage
1a	Fault Zone	169.18	0.849%
1b	Fault Gouge	17.06	0.086%
1c	Fault Breccia	0.63	0.003%
1e	Fault Breccia and Healed	25.25	0.127%
2a	Quartz-dominant Vein	54.71	0.275%
2b	Carbonate-dominant Vein	1.69	0.008%
2c	Quartz-carbonate dominant Vein	16.09	0.081%
3a	Intrusive: granodiorite	3.74	0.019%
4a	Andesite dyke	58.94	0.296%
5c	Lapilli tuff	23.11	0.116%
6b	Sandstone	62.71	0.315%
6d	Argillite	45.95	0.231%
6f	Limestone	3.62	0.018%
7a	Phyllite: graphitic	855.75	4.295%
7b	Phyllite: sericite-chlorite	420.80	2.112%
7c	Phyllite: calcareous chlorite	679.87	3.412%
7d	Phyllite: sericite-chlorite-quartz	1,827.71	9.174%
8a	Schists (no quartz eyes): sericite-chlorite	3,263.36	16.380%
8b	Schists (no quartz eyes): sericite-chlorite-fuchsite	274.55	1.378%
8c	Schists (no quartz eyes): polymictic fragmental-conglomeratic chlorite	704.48	3.536%
9a	Schists (quartz eyes): sericite-hornblende-quartz-feldspar	95.70	0.480%
9b	Schists (quartz eyes): sericite-chlorite-quartz	5,440.44	27.307%
9c	Schists (quartz eyes): sericite-chlorite-quartz	3,821.53	19.181%
9d	Schists (quartz eyes): sericite-augen quartz	253.83	1.274%
9e	Schists (quartz eyes): siliceous-chlorite-sericite-quartz	302.41	1.518%
10a	Gneiss: orthogneiss	144.19	0.724%
11a	Silica altered host	1,305.75	6.554%
12a	Sulphide-dominant rock, undivided	15.57	0.078%
12b	Magnetite-dominant rock	5.80	0.029%
12c	Pyrrhotite-dominant rock	8.75	0.044%
12d	Pyrite-dominant rock	20.24	0.102%
	Total Metrage	**19,923.41**	**100%**

CME Consultants, August 2011

Three lithologies dominate the Deposit. Approximately 50% is quartz eye schist with some slight variation in the precise breakdown of minerals, but with serecite-chlorite-quartz dominant. Schists (without quartz eyes) represent 21% and phyllites represent 19%. All other classifications each represent less than 1% of the overall resource with the exception of silica alteration representing 6.5% of the three main ore types. There is some slight variation in the lithological breakdown in different areas of the Deposit and these were tested in zonal composites (Table 13-2).

Table 13-2: Met Grade Lithology Composites

Sample						Composite	
Hole	From	To	Width	Lithology	Est Avg % Cu	Kg	Lithology
HC11-M04	147.36	149.96	2.60	Phyllite	0.22	258.3	Phyllite 1
HC11-M03	21.64	38.97	17.33	Phyllite	0.32		
HC11-M04	97.87	101.42	3.55	Phyllite	0.33		
HC11-M03	96.46	100.36	3.90	Qz Eye Schist	0.15	487.7	Qz Eye Sch 1
HC11-M03	81.43	83.91	2.48	Qz Eye Schist	0.16		
HC11-M04	27.14	39.00	11.86	Qz Eye Schist	0.18		
HC11-M02	10.44	23.52	13.08	Qz Eye Schist	0.19		
HC11-M04	89.30	97.87	8.57	Qz Eye Schist	0.19		
HC11-M01	7.92	14.75	6.83	Qz Eye Schist	0.20	1,285.9	Qz Eye Sch2
HC11-M03	8.44	21.64	13.20	Qz Eye Schist	0.20		
HC11-M04	119.48	126.69	7.21	Qz Eye Schist	0.20		
HC11-M02	70.71	80.95	10.24	Qz Eye Schist	0.21		
HC11-M04	39.00	86.54	47.54	Qz Eye Schist	0.21		
HC11-M04	140.37	147.36	6.99	Qz Eye Schist	0.21		
HC11-M01	30.24	40.23	9.99	Qz Eye Schist	0.23		
HC11-M02	95.91	99.29	3.38	Qz Eye Schist	0.24		
HC11-M03	38.97	41.70	2.73	Qz Eye Schist	0.24		
HC11-M03	5.40	8.44	3.04	Qz Eye Schist	0.25	943.7	Qz Eye Sch3
HC11-M04	129.57	135.38	5.81	Qz Eye Schist	0.25		
HC11-M04	135.38	140.37	4.99	Qz Eye Schist	0.25		
HC11-M02	6.01	8.60	2.59	Qz Eye Schist	0.26		
HC11-M02	23.52	26.55	3.03	Qz Eye Schist	0.26		
HC11-M02	28.40	42.80	14.40	Qz Eye Schist	0.26		
HC11-M03	85.81	91.77	5.96	Qz Eye Schist	0.28		
HC11-M02	56.60	64.37	7.77	Qz Eye Schist	0.29		
HC11-M04	101.42	116.20	14.78	Qz Eye Schist	0.29		
HC11-M03	91.77	96.46	4.69	Qz Eye Schist	0.30		
HC11-M02	92.72	95.91	3.19	Qz Eye Schist	0.35	515.4	Qz Eye Sch4
HC11-M03	41.70	70.87	29.17	Qz Eye Schist	0.44		
HC11-M03	72.88	74.20	1.32	Qz Eye Schist	0.46		
HC11-M02	137.40	139.50	2.10	Schist	0.28	112	Schist1
HC11-M02	99.29	101.66	2.37	Schist	0.29		
HC11-M02	42.80	46.47	3.67	Schist	0.30		
HC11-M03	83.91	85.81	1.90	Schist	0.37	36	Schist2
HC11-M03	74.20	76.79	2.59	Schist	0.46		
HC11-M04	86.54	89.30	2.76	Silica Altered	0.15	162.6	Silica Altered 1
HC11-M03	79.38	81.43	2.05	Silica Altered	0.19		
HC11-M02	8.60	10.44	1.84	Silica Altered	0.20		
HC11-M04	116.20	119.48	3.28	Silica Altered	0.20		
HC11-M04	126.69	129.57	2.88	Silica Altered	0.20		
HC11-M02	139.50	149.96	10.46	Silica Altered	0.33	124.7	Silica Altered 2
HC11-M03	70.87	72.88	2.01	Silica Altered	0.33		
HC11-M02	46.47	50.66	4.19	Vein	0.34	234.6	Vein1
HC11-M01	14.75	30.24	15.49	Vein	0.41		

G&T Metallurgical Services Limited, February 2012

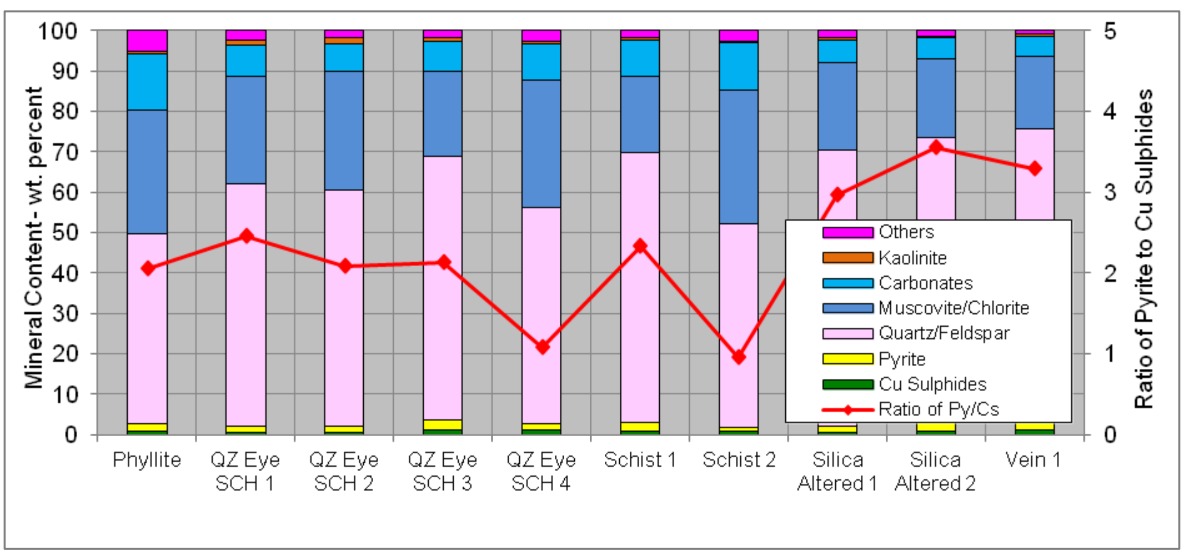
13.3 MINERALOGY

Various mineralogical studies have been carried out, most recently by G&T. It was determined that chalcopyrite is the dominant copper mineral representing >98% of the copper species in the main grade/lithology composites (Figure 13-3). The only composite to fall below 98% chalcopyrite is a silica altered species which contains 94.5% chalcopyrite with 2% bornite and the balance equally covellite and chalcocite. The mineralogical breakdown is shown in Figures 13-2 to 13-4.

The 2012 FS test work expanded the bulk mineral and liberation knowledge included in the PEA and established the information provided in Figures 13.2 and 13.3 based on ten grade lithology composites. Conclusions drawn from the testing were:

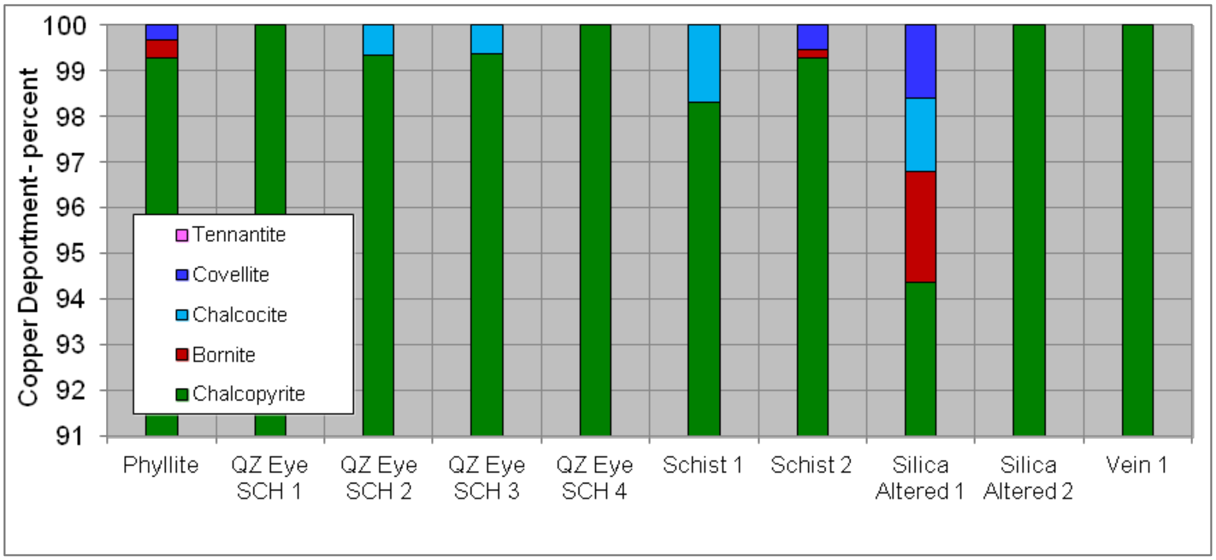
- The sulphide mineral content varied from about 2 to 5 percent across the suite of samples tested;
- Chalcopyrite was the main copper bearing sulphide mineral observed in all the samples;
- The Pyrite:Chalcopyrite ratio is approximately 2.5:1. A Pyrite:Chalcopyrite ratio less than 3:1 is conducive to high copper recovery by flotation;
- The level of copper sulphide liberation should ensure good recovery of copper to a rougher concentrate; and
- Most of the un-liberated copper sulphide mineral was in binary form with non-sulphide gangue minerals.

Figure 13-2: Mineral Content of the Variability Composites



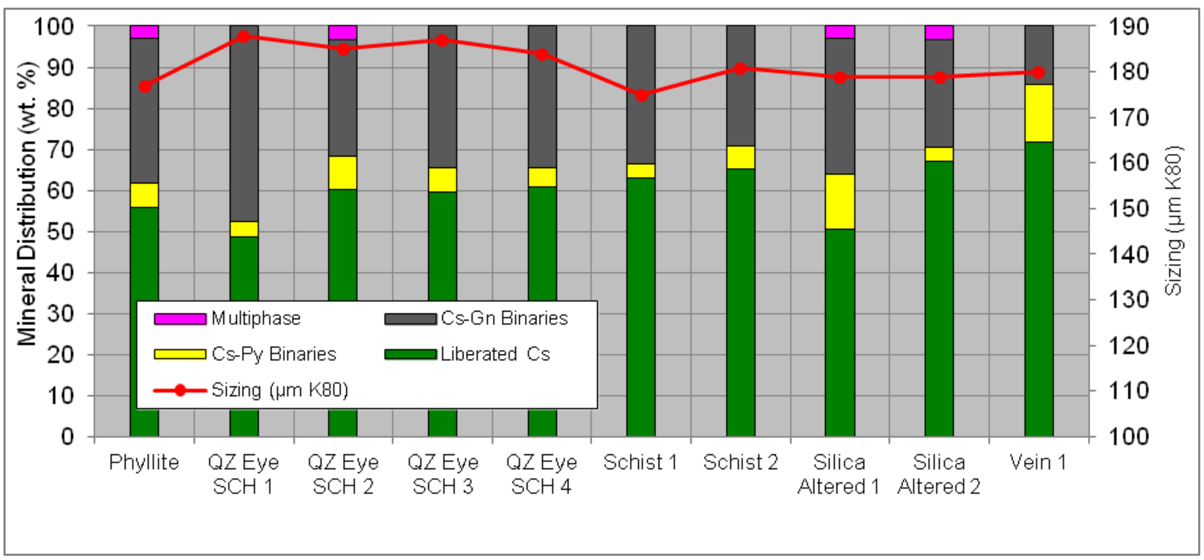
G&T Metallurgical Services Limited, February 2012

Figure 13-3: Cu Deportment by Mineral Species of the Variability Composites



G&T Metallurgical Services Limited, February 2012

Figure 13-4: Cu Sulphides Distribution by Class of the Variability Composites



G&T Metallurgical Services Limited, February 2012

13.4 COMMINUTION

Historical test work included Bond Work Index testing and concluded the ore was soft to medium- soft (BMWi 11-13kWh/t). One of the reasons the 2011 metallurgical drill program went to PQ size core was to enable comprehensive comminution testing on the larger lump sizes required in some tests, especially crusher tests. Selected samples of major lithology types were sent to FLS for testing and those results are summarized in Table 13-3.

Table 13-3: FLS Comminution Results 2011 (1000kg/t)

Sample	Lithology/Code/Hole	Bond Cwi	Bond Ai	Bond RMWi	Bond BMWi
A	Qtz Vein (2a) (M11-01)	5.68	0.3325	10.79	19.1
B	Phyllite (7c) (M11-02)	7.06	0.1729	11.09	11.7
C	Phyllite (7c) (M11-03)	8.76	0.0557	11.56	10.5
D	Schists (8a) (M11-02)	7.45	0.0933	11.97	11.7
E	Silic Alt. QESchists (11a.9b) (M11-02)	8.07	0.4319	10.95	14.0
F	Silic Alt. QESchists (11a.9c) (M11-02)	7.49	0.3214	10.30	12.8
G	Qtz Eye Schist (9c) (M11-01)	6.88	0.0770	11.54	10.7
H	Qtz Eye Schist (9c) (M11-04)	5.23	0.1573	13.41	14.2
I	Qtz Eye Schist (9b) (M11-04)	2.73	0.1878	14.35	14.0
	Average	6.594	0.2033	11.773	13.19

The results generally conform to historical data and the work indices are generally consistent over the wide range of samples analyzed. Sample A (BMWi 19.1 kWh/t or 17.3 kWh/st) is the only sample that was an outlier (quartz vein material). This ore type represents <0.5% of the whole resource and, in any event, would be blended in the mill feed.

It is important to note that conditions inherent in the core samples necessitated some modification to the sample preparation procedure thus leaving the results open to interpretation. To address this issue YMI elected to have a third party, KWM, review the comminution test work before finalizing any conclusions. A summary of KWM's findings is provided below:

- The vast majority of the core has a foliation plane perpendicular to the axis of drilling resulting in fracturing of the core while in the core box creating what has been termed the "poker chip" effect.
- The "poker chip" effect leads to difficulties in preparing samples of an appropriate size for testing. As such, sample preparation procedures for the Drop Weight Test were modified to incorporate sawing because splitting produced samples too small for testing.
- In general, the input into power equations from screen analysis assumes cubical particles passing through the screen. Because of the poker chip effect exhibited by samples tested, interpretation of the result was biased toward a harder ore than what actually exists.

Following KMW's independent review it was concluded that the relatively low work index strongly suggests that the ore is appropriate for a conventional SAG/Ball mill grinding circuit. The observed platey breakage pattern in the core boxes suggested pebble crushing was not needed.

13.5 Grindability Tests

All the test work programs investigated the effect of primary grind size on metallurgical performance. G&T conducted tests at much coarser primary grind sizes compared to the other laboratories. In early test work including that for the PEA, both G&T and PRA test results suggested that the copper metallurgical performances were not very sensitive to the tested primary grind sizes, although finer primary grinding seemed to produce better copper recoveries at rougher flotation.

The 2012 FS test work by FLS on the nine samples indicated BMWi values ranging between 10.5kWh/t to 19.1kWh/t with an average of 13.19kWh/t, confirming the previous soft mineralization results of a soft to medium-soft ore as reported in the PEA. Table 13-4 summarizes the grindability test results.

Table 13-4: Core Samples Test Summary

Core Samples Test Summary – Averages of 9 Samples	
Unconfined Compression Strength (UCS)	46.9MPa (6,801psi)
Bond Crushability Index (CWi)	6.59kWh/t
Abrasion Index (Ai)	0.2033g
Ball Mill Bond Work Index (BMWi)	13.19kWh/t (11.96kWh/st)
Angle of Repose	36.2°

Preliminary evaluation indicated that the ore would be amenable to processing in either a SAG Mill/Ball Mill circuit or a SAG Mill/Ball Mill/Pebble Crusher circuit. KWM evaluated the grinding power needs by carrying out 7 simulations. Subsequently, a SAG mill/ball mill grinding circuit was selected, which includes a 21MW SAG mill and 2 parallel 13MW ball mills with a combined power of 26MW for a total grinding mill power of 47MW.

13.6 Flotation Tests

FS 2012 test work was designed to determine if the grind could be coarsened. Much of the historical test work was done at a fine grind and reflects to some extent “normal” practice at the time the work was performed. Historical studies did not indicate the need for a fine grind and yet the PEA provided for a primary grind of P_{80} of 106µm. The PEA showed very little difference in rougher performance between grinding to 147µm and 204µm.

One of the challenges in copper concentration is to limit the amount of pyrite in flotation concentrate due to its adverse impact on concentrate grade. Pyrite to copper ratio in each of the 10 primary lithology/grade composites varied from 1:1 to 5:1 with a 2.5:1 average. A number of strategies were evaluated to control copper recovery while rejecting pyrite. Three common methods of rejecting pyrite are:

- Elevated pH using lime with general sulphide collectors such as xanthate: Extreme pH may depress copper recovery. A pH>9.8 is usually required to significantly reject pyrite with cleaner circuits running at pH11.
- Special copper specific collectors such as aerophine, dithiophosphate and others. These reagents are significantly more expensive than xanthate although less is usually required.

- Pyrite depressants of which cyanide is one of the most effective and cost efficient. Others such as sulphur dioxide have been used and in excess will depress copper.

The three approaches are applied at either the rougher stage (to a greater or lesser extent) or during cleaning. The PEA ran the rougher/scavenger circuit at neutral to moderately elevated pH but not high enough to have a huge effect on pyrite rejection. The cleaner circuit after regrind was run at an elevated pH up to pH11 with cyanide to further depress pyrite. This resulted in a viable circuit with projected recoveries of ~88% Cu and a concentrate grade of 27% Cu.

G&T FS test work examined a number of alternative conditions of primary grind, rougher/scavenger flotation pH and reagents, regrind grind size, and various conditions in the cleaner circuits with the object of coarsening the primary grind. These alternatives were studied to eliminate the use of cyanide in the cleaning circuit and reduce energy consumption and cost.

The results of the first locked cycle test carried out on the master composite achieved:

- concentrate grade of 26.3% Cu, and
- recovery of 89.6 % Cu (& 66.8% Ag, 57.9% Au).

The test was carried out after a primary grind of 80% passing 189µm, pH11 (with lime) using PAX as the collector. Rougher/scavenger concentrate was reground to 27µm and cleaned at pH11 using PAX as collector. No other depressants (except lime/pH) were used. MIBC was used as the frother throughout. Tests were repeated at a slightly finer regrind and gave similar results (same grade recovery curve), i.e., 25.6% Cu concentrate grade at 90.0% recovery.

Following the establishment of this viable flowsheet and reagent schedule, the flowsheet was tested on the ten grade-lithology composites used in compiling the master composite.

Eight of the ten grade lithology composites gave similar results to the master composite. The two that produced inferior results were the two composites that showed silica alteration of the quartz eye schist and had the highest pyrite-chalcopyrite ratios (approximately 5:1). One of the two composites was low grade (~0.15% Cu) but overall the silica altered material represented approximately 6% of the overall Deposit, and as such may not be of serious concern. This material is represented in the master composite and the overall results from the master composite reflect the inclusion of this material.

Composites representing a different range of grades from different zones of the Deposit were also tested. They were normal and low-grade composites from the south, east and west zones of the Deposit. Broadly, samples from Holes HC11-M01&02 came from the West Zone, HC11-M03 from the South Zone and HC11-M04 from the East Zone. The results conformed to the expected results based on the Master composite. A second master composite #2 was generated to provide enough sample to produce concentrate for smelter acceptability tests. The Py:Cpy ratio in the composite was determined to be 3.2:1, higher than the overall average determined for the Deposit, namely 2.5:1. The pilot plant run was relatively short (10hrs) which is not really long enough to achieve good circuit stability, but during that period 25.5% Cu in concentrate was produced at 91% Cu recovery. Improvements can be expected with stable circuit conditions and lower Py:Cpy ratios expected in the average feed.

The analysis of the concentrate in Table 13-5 indicates payment will be received for copper, gold and silver. No other elements attract a penalty.

Table 13-5: Final Concentrate Analysis - Minor Elements

Element	Symbol	Units	KM2916 LC12	KM2916 LC13	KM3221 WSB Fn Con
Copper	Cu	%	25.6	26.3	25.5
Gold	Au	g/t	1.35	1.55	1.92
Silver	Ag	g/t	74	123	122
Sulphur	S	%	35.2	33.7	30.0
Iron	Fe	%	32.9	31.6	27.3
Aluminium	Al	%		0.25	1.08
Antimony	Sb	%		0.001	0.002
Arsenic	As	g/t		87	104
Bismuth	Bi	g/t		31	13
Cadmium	Cd	g/t		32	34
Calcium	Ca	%		0.36	0.67
Carbon	C	%		0.34	0.83
Cobalt	Co	g/t		0.40	110
Fluorine	F	g/t		101	151
Lead	Pb	%		0.21	0.17
Magnesium	Mg	%		0.24	0.82
Manganese	Mn	%		0.010	0.024
Mercury	Hg	g/t		<1	<1
Molybdenum	Mo	%		0.010	0.020
Nickel	Ni	g/t		206	350
Phosphorous	P	g/t		76	418
Selenium	Se	g/t		127	2
Silicon	Si	%		1.25	3.33
Zinc	Zn	%		0.49	0.35

13.6.1 *CLEANER FLOTATION TESTS*

FS test work carried out by G&T in 2011 demonstrated that concentrates could be upgraded to above 30% Cu but that at 25.5% Cu recoveries were over 90% which represents better economics at current metal prices. This general conclusion is supported by test work carried out in PEA.

13.7 CONCLUSION

The FS achieved its objectives insofar as the historical test work reported in the PEA was consistent with the results of the FS test work carried out in 2011. The rougher scavenger recovery was about the same as achieved in the PEA; 93% copper recovery but at a weight pull of almost half (~6.5%) that achieved in the PEA. This was done at a coarser grind (P_{80} 180µm) but perhaps more importantly, at a high pH (11). This is abnormally high for a rougher/scavenger circuit (but quite common in a cleaner circuit). It is notable that the high pH was clearly enabling much more pyrite to be rejected in the rougher circuit, but at no significant loss in copper. It is the fear of copper loss that has generally driven the operation of rougher circuits at lower pH (~10).

The FS regrind 20-25µm will be achieved in an inert stirred IsaMill™. Also noteworthy is that with the rejection of more pyrite at the rougher stage, the losses of copper in cleaner tailings (because of reduced tonnage in rougher concentrate) is reduced with a gain of approximately 3% copper recovery to final concentrate over that achieved in the PEA. The two ore types, representing a very minor percentage of the total ore tonnage, that gave lower recoveries in the variability testing gave good and acceptable rougher recoveries. The problems that were encountered were found in the cleaner circuit, with higher cleaner tailings grades than normal. The one lithology (silica alteration of quartz eye schist) was the only sample in which copper minerals other than chalcopyrite were present in any appreciable amounts. It was observed that some of these minerals like chalcocite and covelite are very friable and can often be susceptible to slimes losses. This lithology was included in the master composite and so the discounted recovery is reflected in the recoveries from the master composite. Also, some of the other composites gave markedly better recoveries, for example 92.9% Cu recovery @ 29.2% Cu concentrate grade from Western Low Grade: @0.25% Cu head grade.

It should be noted that in laboratory test work, the procedure for grinding samples is by timed rod milling, which is very different to real plant operations using cyclones in closed circuit ball milling and is more likely to produce slimed products of friable minerals.

14 MINERAL RESOURCE ESTIMATE

14.1 EXPLORATORY DATA ANALYSIS

The sample database for the Project contains results from 353 core holes (90,778.52 m) drilled between 1967 and the end of 2013. Of these, 177 were completed since the start of 2006 by YMI and comprise 69% of the total sampled core length. Seven condemnation holes (1,545m) were also drilled in 2011 but were outside of the resource area. A total of 24 geotechnical holes (1,270m) were also completed in 2011. The drilling used to develop the resource model is summarized in Table 14-1.

Table 14-1: Resource Drill Hole Summary (Geosim)

Series	Year	Company	Holes Drilled	Core Diam	Total Metres	Intervals Assayed	Metres Assayed
67-H-1 to 6	1967	Quebec Cartier	6	NQ	546.19	174	526.08
NH-1 to 17	1968	Noranda	17	BQ	2,105.83	709	1,988.16
69-H-1 to 27	1969	Quebec Cartier	27	BQ	4,739.19	1,528	4,578.69
NH-18 to 30	1969	Noranda	13	BQ	1,733.56	532	1,614.52
J-1 to 12	1970	Noranda	12	BQ	2,328.69	617	1,894.04
NH-31 to 95	1970	Noranda	57	BQ	8,315.50	2,503	7,654.08
J-13 to 43	1971	Noranda	27	BQ	5,593.82	1,728	5,353.93
J-40 to 42	1972	Noranda	4	BQ	456.74	39	117.8
J44 to 48	1973	Noranda	5	BQ	625.45	13	39.62
96-1 to 8	1996	American Comstock	8	NQ	2,847.44	686	2,045.96
		Subtotal 1967-1996	**176**		**29,292.41**	**8,529**	**25,812.88**
HC06-01 to 12	2006	YMI	12	NQ2	4,101.40	2,536	4,028.81
HC07-13 to 52	2007	YMI	40	NQ2	15,879.94	12,569	15,602.5
HC08-53 to 75	2008	YMI	23	NQ2	7,602.92	6991	7,496.02
HC10-76 to 82	2010	YMI	7	NQ2	3,486.92	2,637	3,405.93
HC11-83 to 130	2011	YMI	48	NQ2	15,571.31	1,1865	1,4930
HC11-GM01 to GM07	2011	YMI	8	PQ	2,433.13	1,025	1,290.82
HC11-M01 to M04	2011	YMI	4	PQ	441.04	137	142.58
		Subtotal 2006-2011	**142**		**49,516.66**	**37,760.00**	**46,896.66**
HC12-131 to 142	2012	YMI	12		3,803.29	2,547	3,466.34
HC13-143 to 165	2013	YMI	23		8,166.16	5,206	7,259.47
		Subtotal 2012-2013	**35**		**11,969.45**	**7,753.00**	**10,725.81**
Total			**353**		**90,778.52**	**54,042.00**	**83,435.35**

The previous resource estimate data cut-off was hole HC11-119. Data from 36 additional drill holes (16,698m) was used in the present study.

Many of the legacy holes, not assayed for precious metals at the time of drilling, were re-assayed by YMI for copper, gold, and silver. Because the original assay intervals were not always maintained, two independent databases were established; one for copper grades and one for precious metal grades.

Legacy holes were sampled on regular 3.05m (10ft) lengths corresponding to the length of the core barrel and drill rods. YMI drilling was sampled on nominal 3m intervals in 2006, 2m intervals in 2007 and 1m intervals in 2010-2011. YMI also broke sample intervals at lithologic boundaries.

Cumulative frequency distribution for the Cu and Au samples within resource domains are illustrated in Figure 14-1 to Figure 14-3. The sample population for Cu is highly skewed approaching log normal distribution with no significant bimodality evident. Some bi-modality is suggested in the log cumulative frequency distribution of Au and this is attributed to the more irregular distribution of Au in the Deposit.

Cu shows a moderate positive correlation with Au and a weaker positive correlation with Ag with correlation coefficients of 0.53 and 0.36 respectively (Figure 14-4).

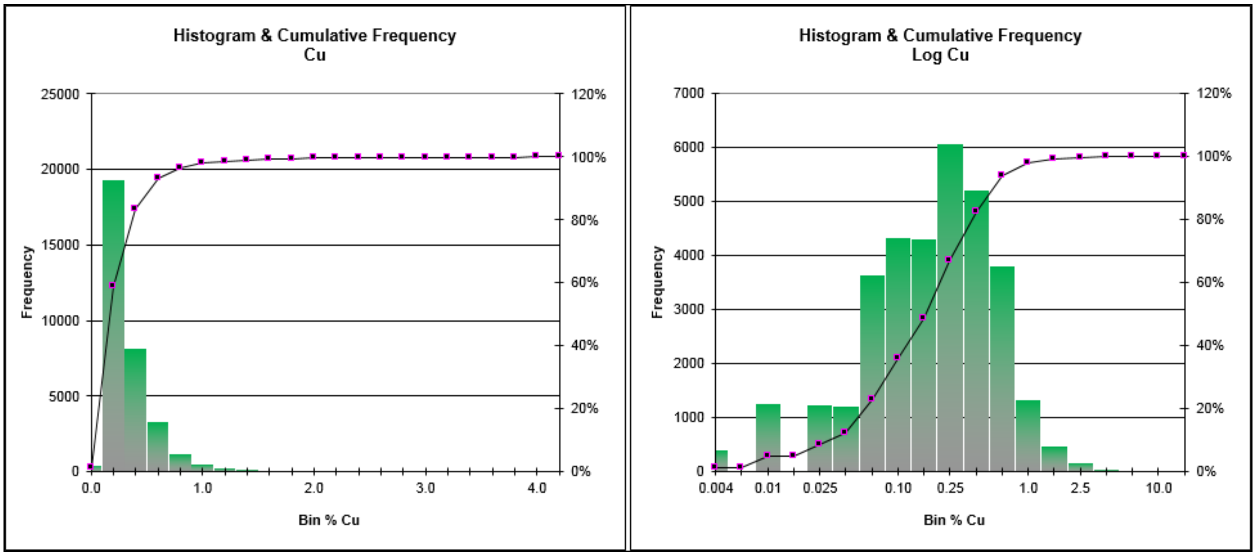
Au and Ag show a weak positive correlation (correlation coefficient = 0.2) and a linear regression yields a low R^2 value of 0.03 (Figure 14-5).

Basic statistics for samples falling within the resource domains are shown in Table 14-2.

Table 14-2: Sample Statistics (Geosim)

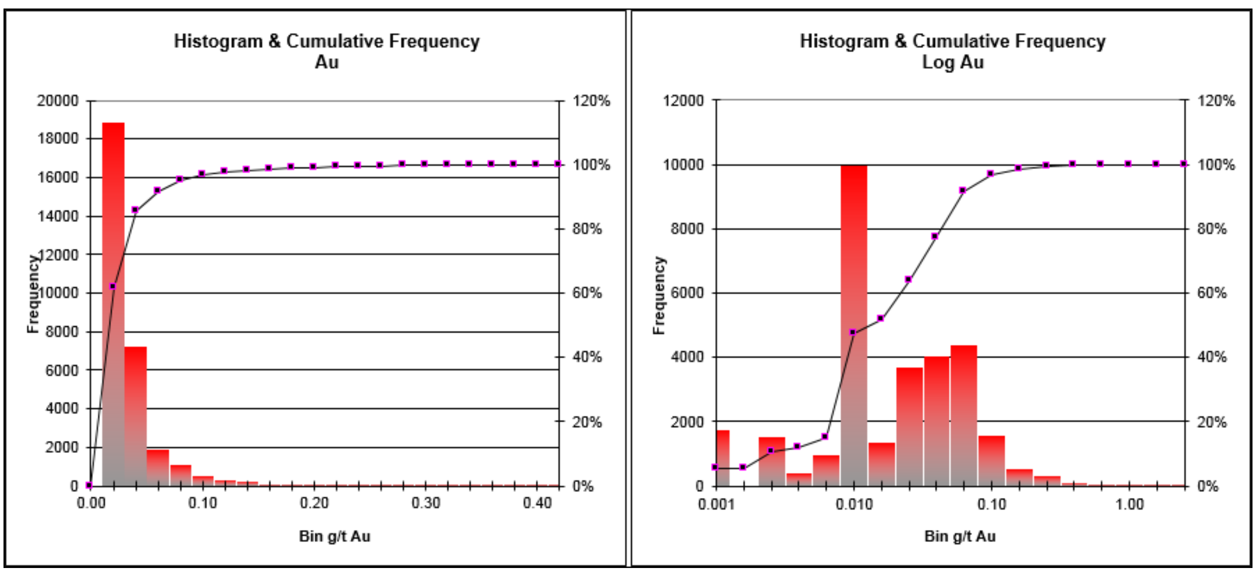
	Cu	Au	Ag
n	33,452	30,539	30,477
min	0.00	0.001	0.0
max	10.50	1.940	410.0
Median	0.16	0.013	0.8
Mean	0.24	0.027	1.3
Wt Avg	0.23	0.026	1.2
Variance	0.09	0.002	17.2
Std Dev	0.31	0.044	4.1
CV	1.27	1.59	3.10

Figure 14-1: Frequency Distribution of Cu



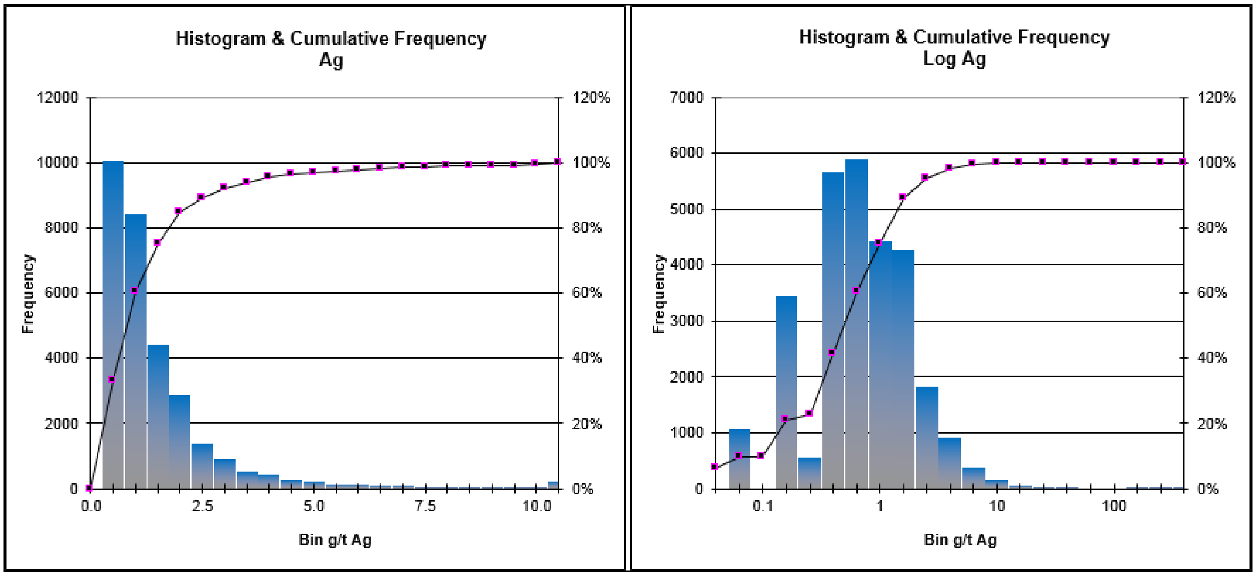
GeoSim Services Inc., March 2014

Figure 14-2: Frequency Distribution of Au



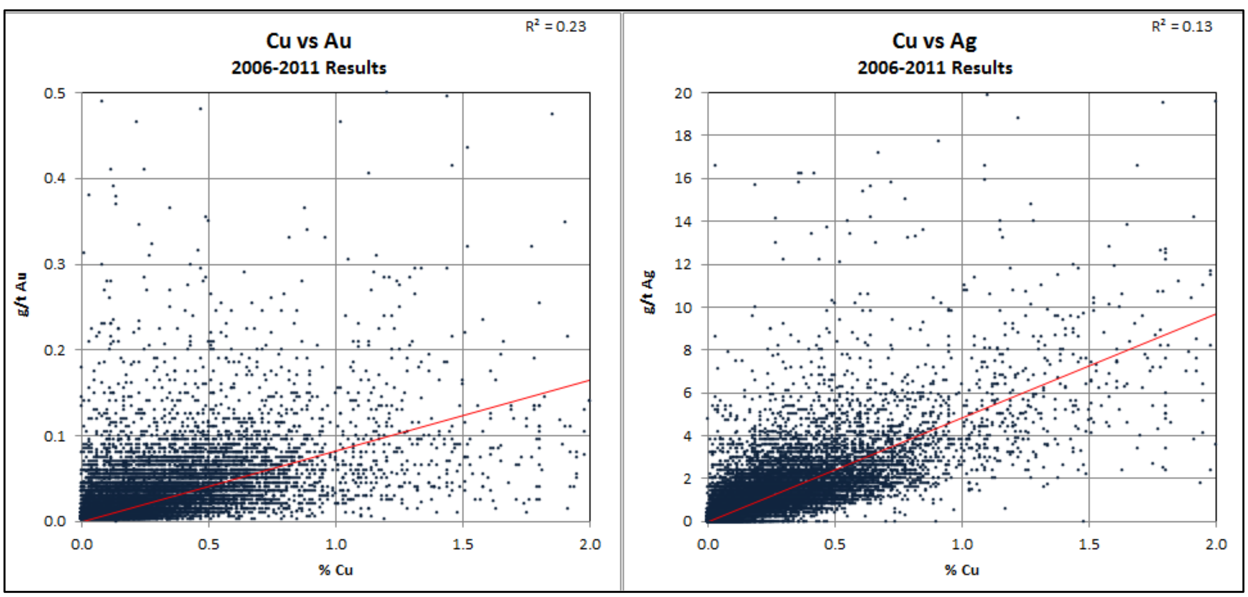
GeoSim Services Inc., march 2014

Figure 14-3: Frequency Distribution of Ag



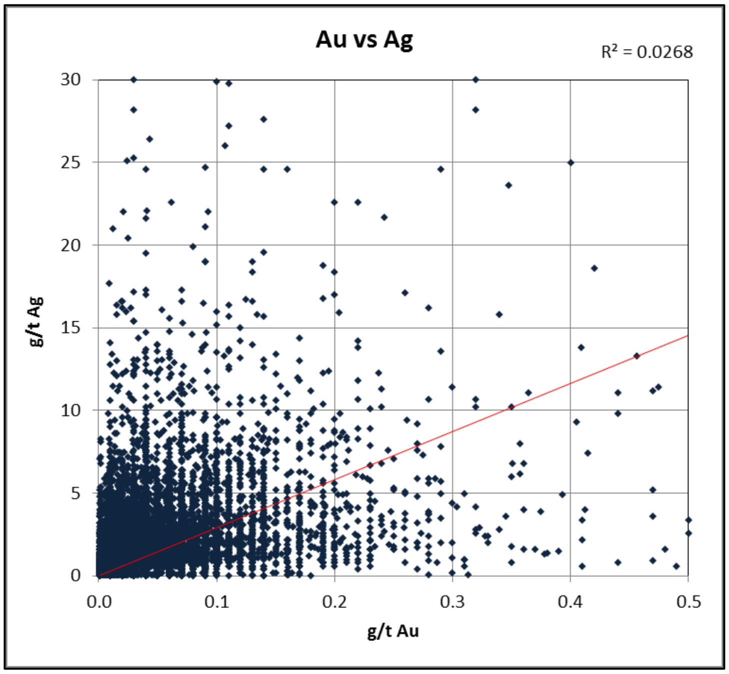
GeoSim Services Inc., March 2014

Figure 14-4: Scatterplot of Cu vs Au and Ag Sample Data



GeoSim Services Inc., March 2014

Figure 14-5: Scatterplot of Au vs Ag Sample Data



GeoSim Services Inc., March 2014

14.2 Outlier Analysis

Before compositing, grade distribution in the raw sample data was examined to determine if grade capping or special treatment of high outliers was warranted. Cumulative log probability plots were examined for outlier populations and decile analyses was performed for Cu, Au and Ag within the resource constraint domains. As a general rule, the cutting of high grades is warranted if:

- the last decile (upper 10% of samples) contains more than 40% of the metal; or
- the last decile contains more than 2.3 times the metal of the previous decile; or
- the last centile (upper 1%) contains more than 10% of the metal; or
- the last centile contains more than 1.75 times the next highest centile.

None of these criteria were met by this sample population suggesting that capping or special treatment of outliers is not warranted. However, examination of CPP plots did reveal a few scattered outliers that could have a local impact on block grades and it was decided to cap grades as shown in Table 14-3.

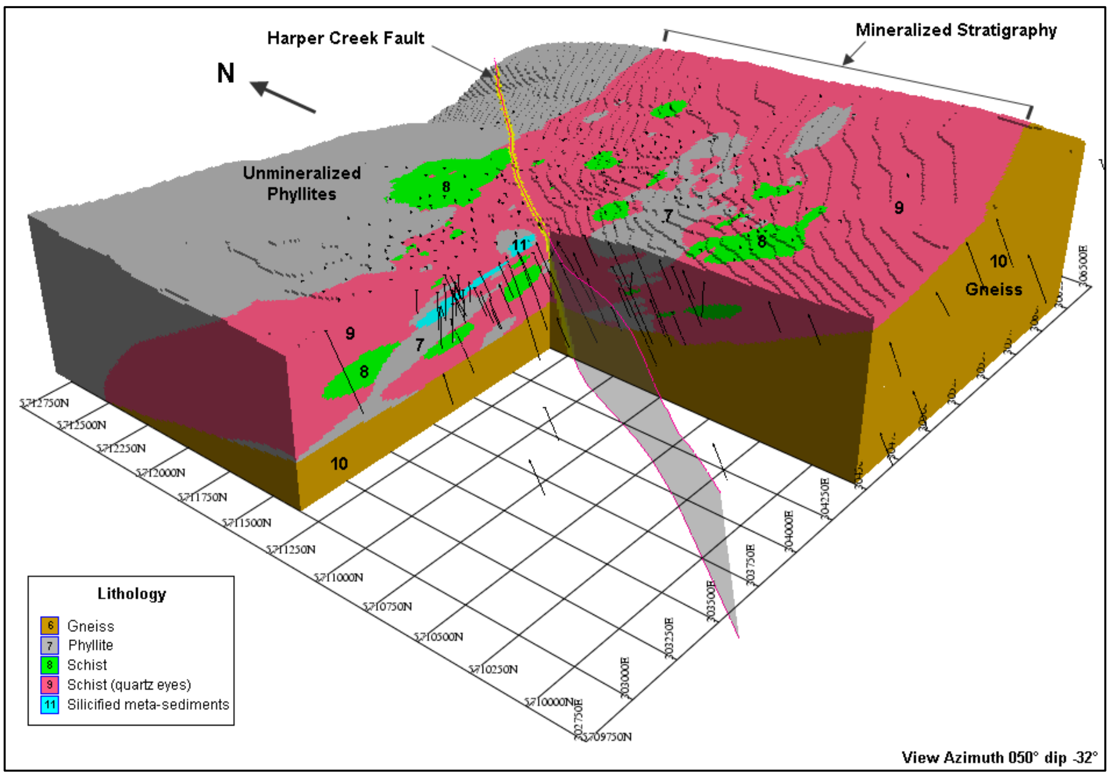
Table 14-3: Grade Capping (Geosim)

Item	Cap Level	Unit	Samples Affected
Cu	5	%	15
Au	1	g/t	4
Ag	30	g/t	10

14.3 Deposit Modeling

The mineralized stratigraphy comprises a sequence of phyllites and schists (units 7-9) overlying un-mineralized gneiss (unit 10). Weakly mineralized to barren phyllites overlie the main mineralized horizons. The Harper Creek fault bisects the Deposit in a SW-NE direction and dips steeply to the SE. The three main lithologic domains (gneiss, mineralized meta-sediments and overlying phyllites) were modeled in Surpac Vision software as 3d wireframes. The Harper Creek Fault was modeled as a surface and acts as a hard boundary for both the lithologic and grade models. The final lithology assigned to the block model is illustrated in Figure 14-6.

Figure 14-6: Block Model Lithology



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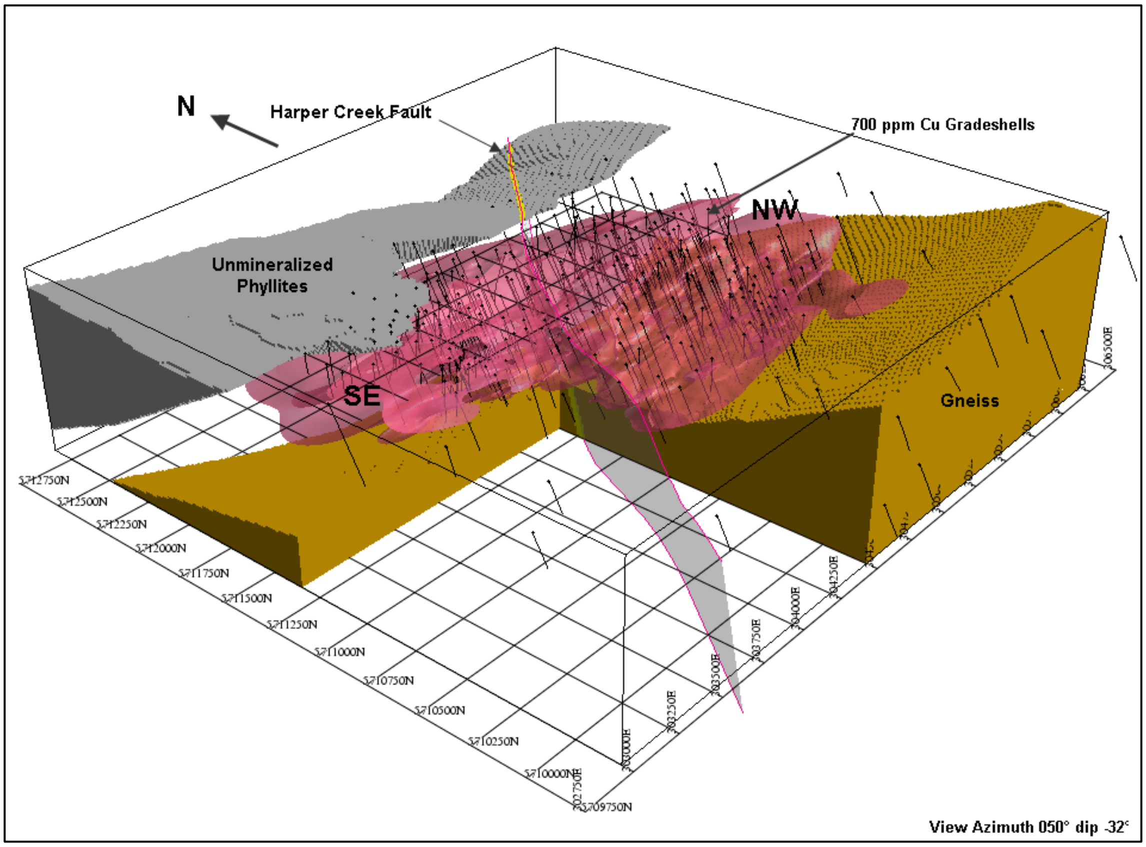
In order to further constrain the block model grade estimation, gradeshells based on a 700ppm Cu cut-off were generated by modeling log transformed data using Leapfrog3d© software. Separate zones were modeled on either side of the Harper Creek Fault (Figure 14-7) and are referred to as the NW and SE zones.

A bedrock surface DEM was constructed in Surpac based on drill hole data and projected to the edges of the resource model.

Figure 14-7: Gradeshell Constraints



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14.4 Compositing

Best fit downhole composites of Cu, Au and Ag were generated using 6m intervals within the zone domains. All samples within the domain constraints were capped prior to compositing at levels of 5% Cu, 1g/t Au and 30g/t Ag. Statistics for composites are summarized Table 14-4. The combination of capping and compositing reduce the coefficient of variation for Cu from 1.27 in the raw sample data to 0.75. The CV for Au was reduced from 1.59 to 1.04 and Ag dropped from 3.1 to 0.84.

Table 14-4: Composite Statistics (Geosim)

	Cu in 700ppm Cu grade shells			Au in 700ppm Cu grade shells			Ag in 700ppm Cu grade shells		
	NW	SE	COMB	NW	SE	COMB	NW	SE	COMB
n	2,810	5,676	8,486	2,408	4,437	6,844	2,408	4,437	6,845
Min	0.00	0.00	0.00	0.000	0.000	0.000	0.0	0.0	0.0
Max	1.62	2.38	2.38	0.541	0.453	0.541	12.6	11.4	12.6
Median	0.17	0.19	0.18	0.035	0.019	0.019	1.5	1.0	0.9
Q3	0.30	0.30	0.30	0.010	0.032	0.033	0.5	1.4	1.5
Mean	0.23	0.23	0.23	0.029	0.025	0.026	1.2	1.2	1.2
Variance	0.04	0.03	0.03	0.001	0.000	0.001	1.2	0.9	1.0
Std Dev	0.20	0.16	0.17	0.034	0.022	0.027	1.1	0.9	1.0
CV	0.84	0.70	0.75	1.17	0.90	1.03	0.91	0.80	0.84

14.5 Density

A total of 10,739 bulk density measurements were made on core sampled between 2006 and 2007. After removal of outliers, the median bulk density values for each modeled lithology were assigned to the corresponding blocks in the resource model. Density of overburden was assumed to be 2.2.

Table 14-5: Bulk Density Statistics for Modeled Lithologies (Geosim)

Material	Code	No. of Measurements	Model Density
HC Fault	1	51	2.72
Phyllite	7	1,588	2.80
Schist	8	1,493	2.85
Schist	9	2,742	2.76
Gneiss	10	142	2.74
Silicified	11	745	2.71

14.6 Variogram Analysis

Directional pairwise relative semi-variograms for Cu, Au and Ag were modeled using composites falling within the domain constraint in order to determine search parameters and anisotropy. Maximum ranges for Cu in both zones were 250m while Au and Ag had modeled ranges of 250m in the SE zone and 200m in the NW zone. Variogram model parameters for Cu, Au and Ag are shown in Table 14-6.

Table 14-6: Semi-Variogram Model Parameters (Geosim)

Item Zone	Type	Axis	Azim	Dip	co	c1	a1	c2	a2
Cu NW	Pairwise Relative Spherical	major	0	-30	0.007	0.0219	80	0.0138	250
		semi-major	90	0	0.007	0.0219	80	0.0138	250
		minor	180	-60	0.007	0.0219	15.6	0.0138	49
Cu SE	Pairwise Relative Spherical	major	47.1	-21.4	0.007	0.0087	80	0.0087	250
		semi-major	306.6	-25	0.007	0.0087	80	0.0087	250
		minor	352.8	56	0.007	0.0087	15.5	0.0087	48.5
Au NW	Pairwise Relative Spherical	major	0	-30	0.0004	0.000	75	0.000448	200
		semi-major	90	0	0.0004	0.000	75	0.000448	200
		minor	180	-60	0.0004	0.000	17.8	0.000448	40.5
Au SE	Pairwise Relative Spherical	major	47.1	-21.4	0.000156	0.000115	80	0.000155	250
		semi-major	306.6	-25	0.000156	0.000115	80	0.000155	250
		minor	352.8	56	0.000156	0.000115	25	0.000155	70
Ag NW	Pairwise Relative Spherical	major	0	-30	0.464	0.547	75	0.203	200
		semi-major	90	0	0.464	0.547	75	0.203	200
		minor	180	-60	0.464	0.547	15	0.203	55
Ag SE	Pairwise Relative Spherical	major	47.1	-21.4	0.327	0.179	80	0.1558	250
		semi-major	306.6	-25	0.327	0.179	80	0.1558	250
		minor	352.8	56	0.327	0.179	17	0.1558	80

14.7 Block Model and Grade Estimation Procedures

A block model was created in Gemcom-Surpac Vision© software using a block size 12m x12m x12m. Block model extents are summarized in Table 14-7.

Table 14-7: Block Model Parameters (Geosim)

	East	North	Elev
Minimum	303,000	5,709,850	1,000
Maximum	306,000	5,712,850	1,816
Extent	3,000	3,000	816
Block Size (m)	12	12	12
No. of Blocks	250	250	68

The model blocks were first coded by the partial percent within the zone domain and below topography. Lithologic codes and SG values were then assigned as described in Sections 14.3 and 14.5.

Cu, Au and Ag grades within the NW and SE zone domains were estimated in three passes using the inverse distance squared weighting method (ID^2). The second pass used an octant search in order to differentiate interpolated from extrapolated block grade estimates for classification. Search parameters are outlined in Table 14-8. The frequency distributions of block grades are shown in Figures 14-8 to 14-10. Block model grade distribution are shown in Figures 14-11 to 14-16.

Table 14-8: Grade Model Search Parameter (Geosim)

Zone	Pass	Search Type	Max Search Dist (m)	Min # Composites	Max # Composites	Min Octants Required	Max per Hole
NW	1	Ellipsoidal	82.5	4	24		3
	2	Octant	250	4	24	5	
	3	Ellipsoidal	250	4	24		3
SE	1	Ellipsoidal	82.5	4	24		3
	2	Octant	250	4	24	5	
	3	Ellipsoidal	250	4	24		3

Figure 14-8: Frequency Distribution of Cu Grades in Block Model

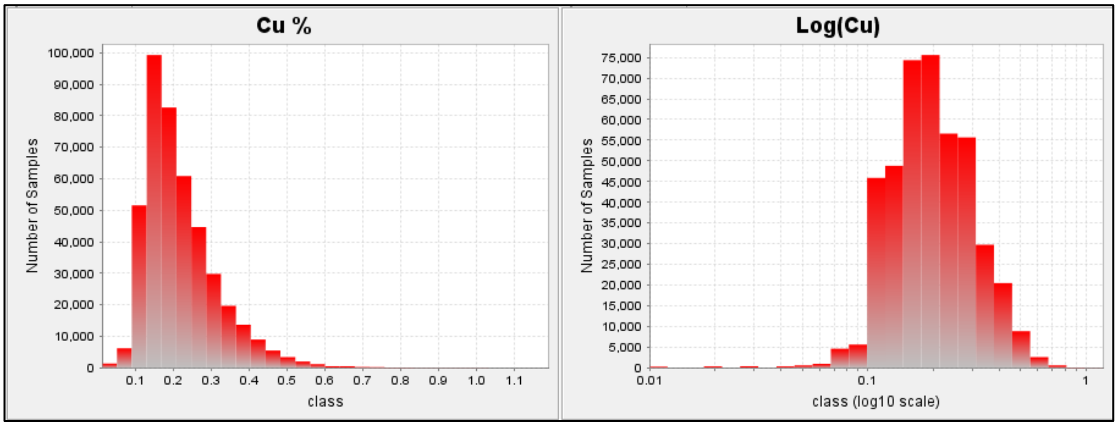

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Figure 14-9: Frequency Distribution of Au Grades in Block Model

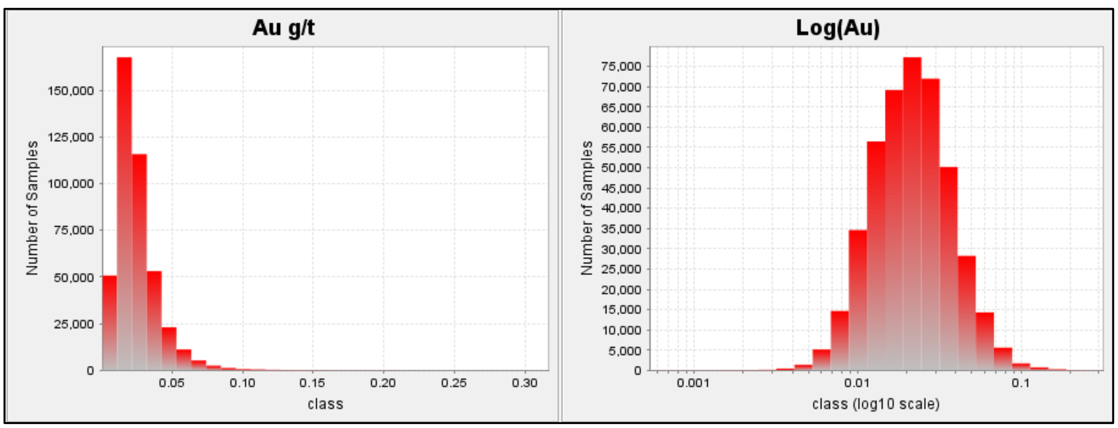

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Figure 14-10: Frequency Distribution of Ag Grades in Block Model

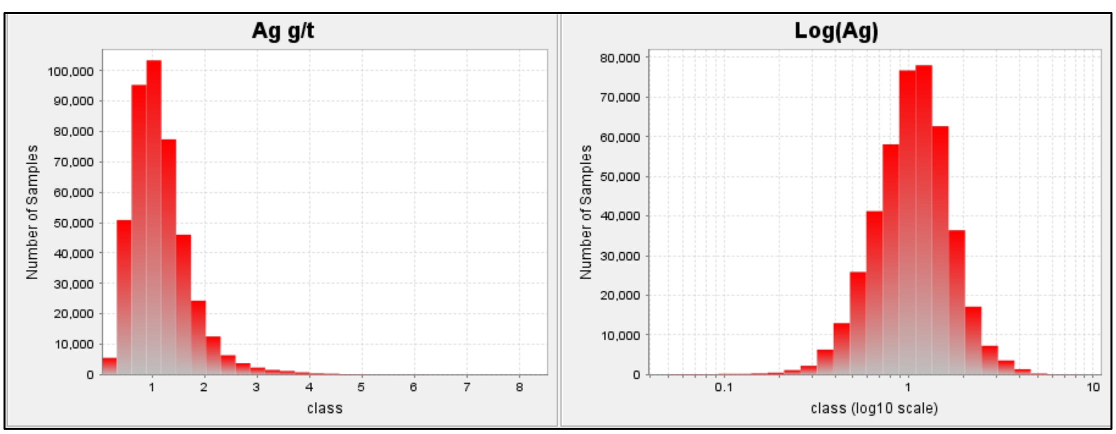

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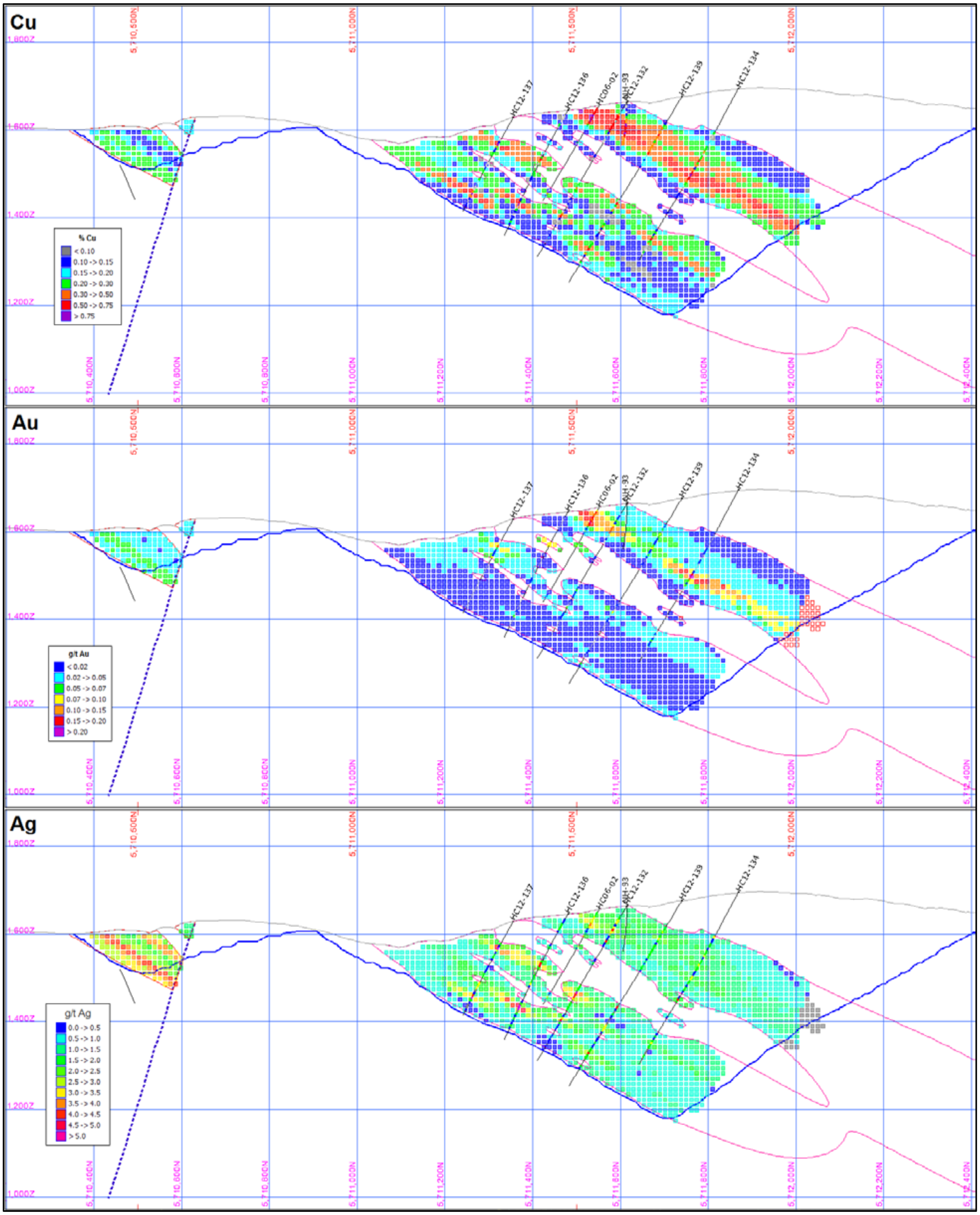
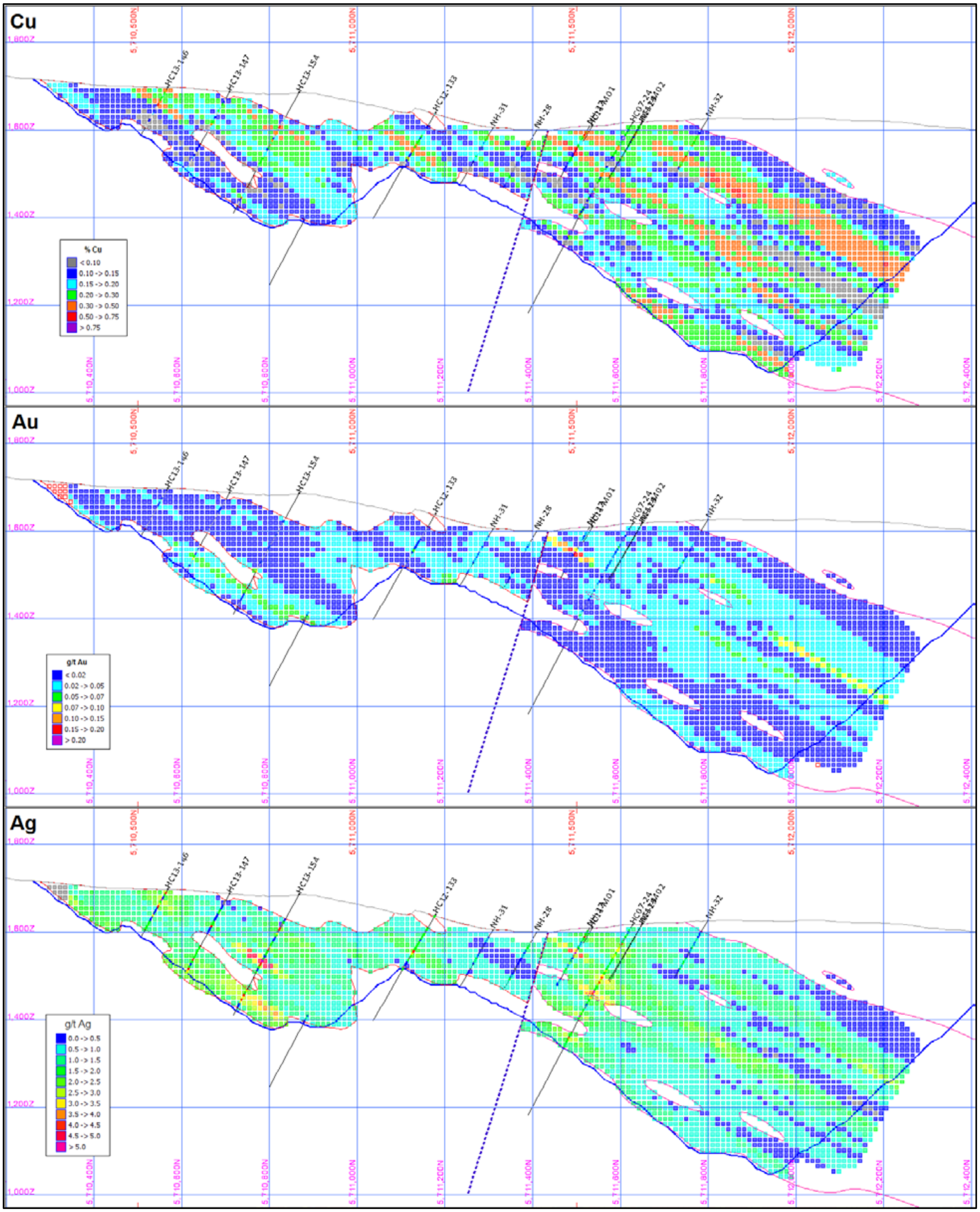
Figure 14-11: Block Grade Distribution – Section 304060E



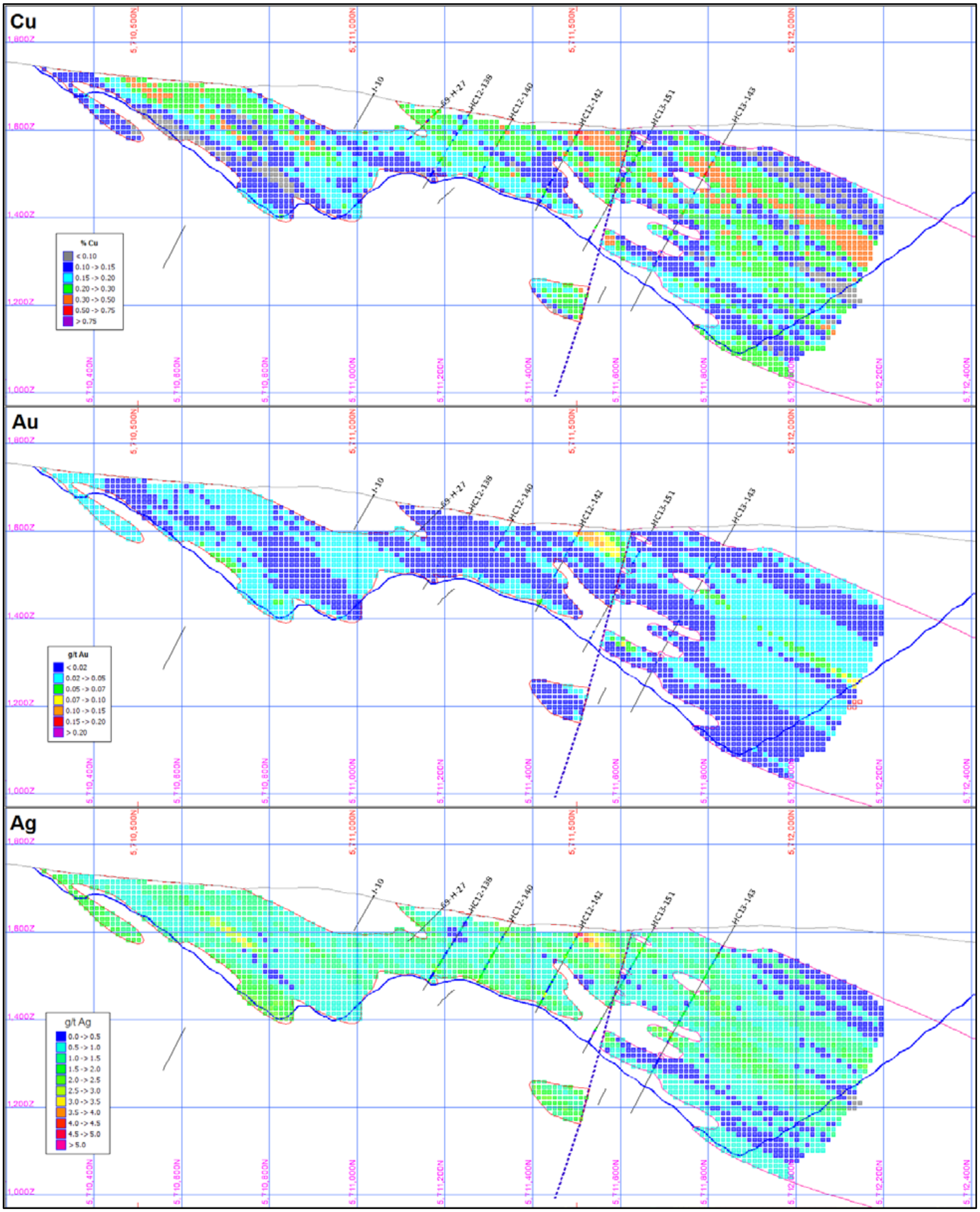
GeoSim Services Inc., March 2014

Figure 14-12: Block Grade Distribution – Section 304518E



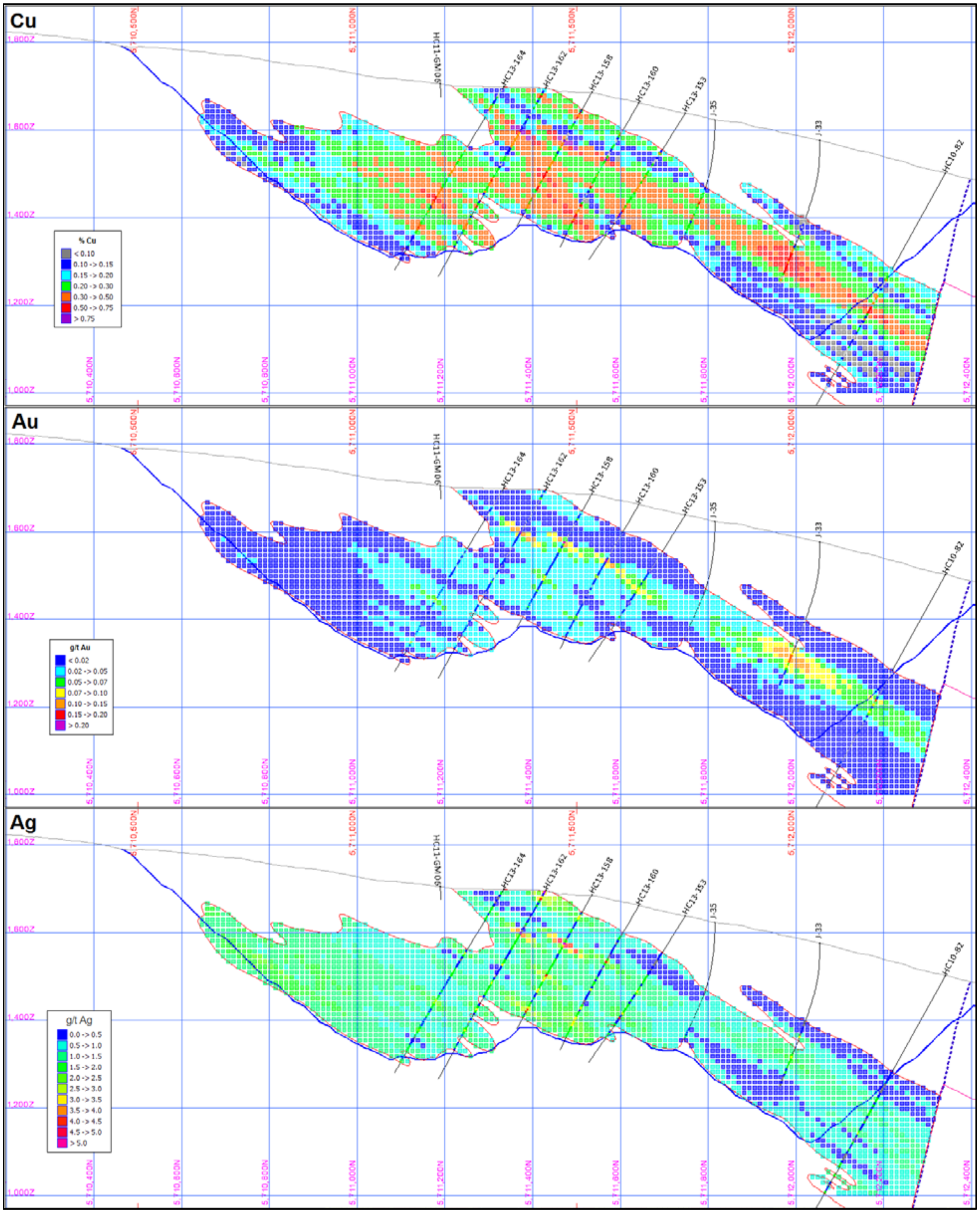
GeoSim Services Inc., March 2014

Figure 14-13: Block grade distribution – Section 304650E



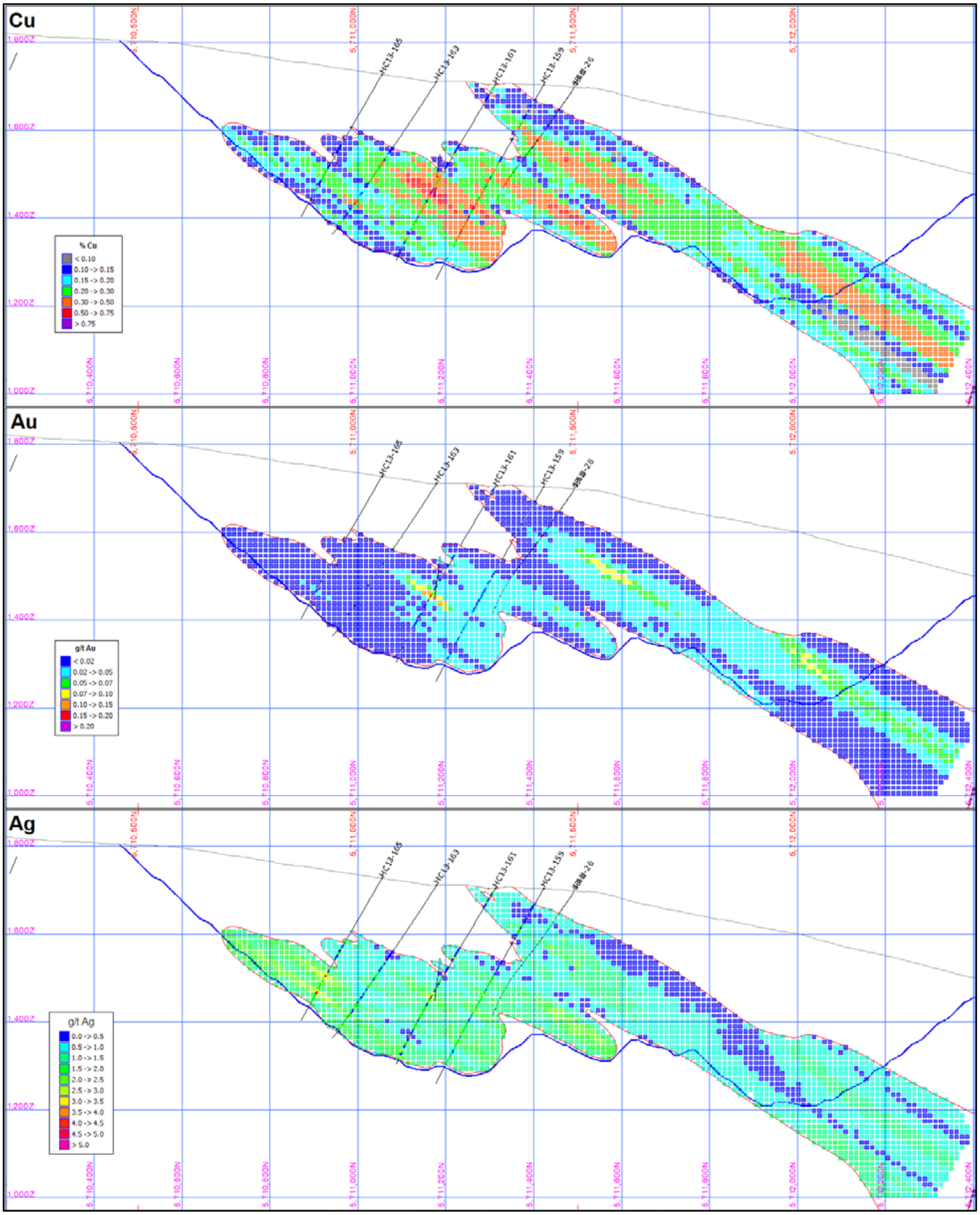
GeoSim Services Inc., March 2014

Figure 14-14: Block grade distribution – Section 305418E



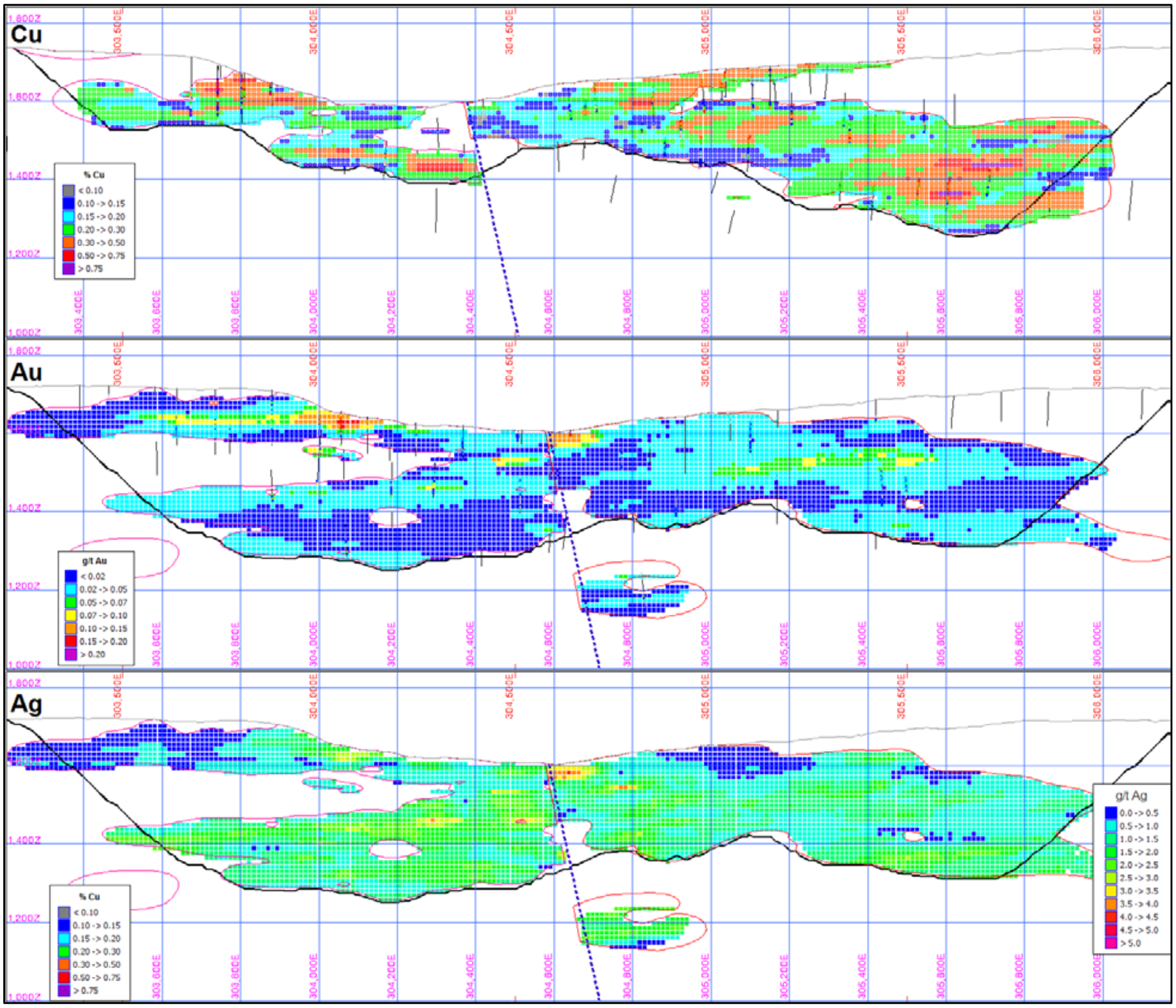
GeoSim Services Inc., March 2014

Figure 14-15:Block grade distribution – Section 305538E



GeoSim Services Inc., March 2014

Figure 14-16: Block Grade Distribution – Section 5711528N



GeoSim Services Inc., March 2014

14.8 Mineral Resource Classification

Resource classifications used in this study conform to the definitions of Mineral Resource, Measured Mineral Resource, Indicated Mineral Resource, and Inferred Mineral Resource prescribed by NI43-101. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

14.8.1 *Lerchs-Grossman Pit*

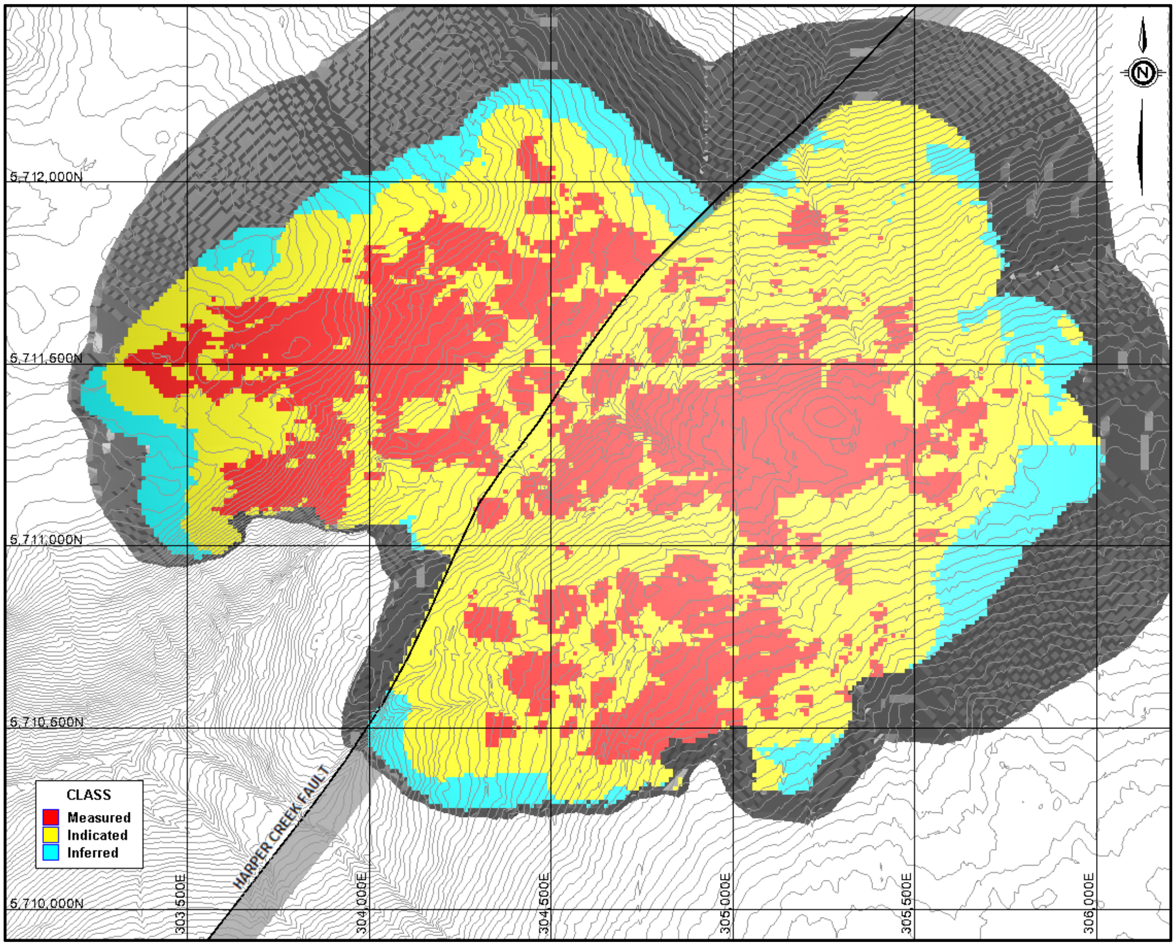
In order to meet the requirements of NI43-101 with respect to reasonable prospects of economic extraction, by open pit mining methods, a Lerchs-Grossman pit was generated to constrain the resource within the block model. Metal price used was US$3.50/lb Cu with an average recovery of 89%. Combined processing and G & A costs were set at C$4.20/t milled. Ore and waste mining costs were C$1.84/t mined with a bench increment of C$0.025/t mined. The pit slope was set at 42°. Profiles of the pit are included in Figures 14-11 to 14-16.

Blocks were initially classified as measured if they were estimated in the 1st pass with a minimum of 4 composites from at least 2 drill holes within 82.5m of the block centroid corresponding to 1/3 of the maximum variogram range. The blocks meeting these criteria were then examined visually and some blocks were downgraded to 'indicated' if there were in areas missing precious metal assays or in isolated clusters.

Remaining unclassified blocks were flagged as indicated if they were estimate in the 2nd pass which used an octant search to limit extrapolation. Some extrapolated estimates from the 3rd pass were also classified as 'indicated' if the closest composite was within 125m of a block centroid corresponding to half the maximum variogram range. A series of blocks estimated in the 3rd pass that were adjacent to the Harper Creek Fault and not estimated in the octant search due to the imposed hard boundary were also classified as 'indicated'.

All other estimated blocks were classified as inferred. Block classification is illustrated in Figure 14-17.

Figure 14-17: Block Classification - Plan View



GeoSim Services Inc., March 2014

14.9 Model Validation

Model verification was initially carried out by visual comparison of blocks and sample grades in plan and section views. The estimated block grades showed reasonable correlation with adjacent composite grades.

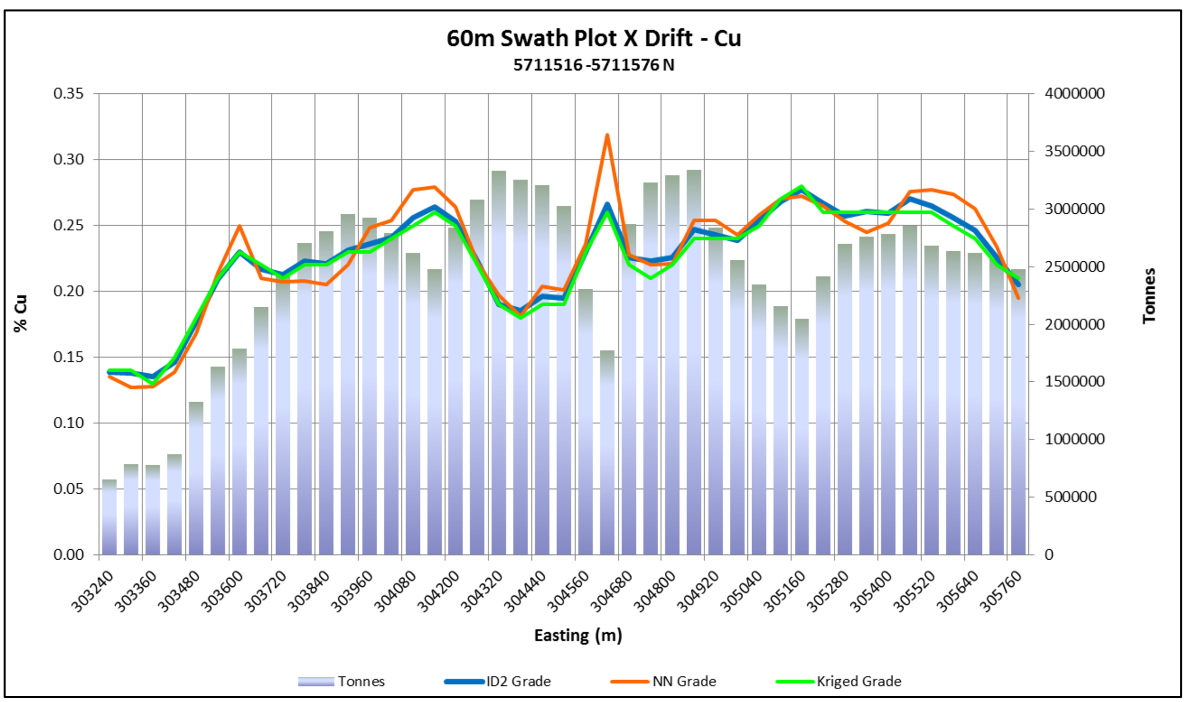
Block grades were also estimated using the nearest neighbour method and separate kriging runs were carried out for Cu. A comparison of global mean values within the grade shell domain shows a reasonably close relationship with samples, composites and block model values (Table 14-9).

Table 14-9: Global Mean Grade Comparison (Geosim)

	Cu %	Au g/t	Ag g/t
Samples (Wt Avg)	0.231	0.027	1.3
Samples Capped	0.230	0.027	1.3
Composites	0.229	0.026	1.2
ID^2 Blocks	0.215	0.025	1.2
Nearest Neighbour	0.215	0.025	1.2
Kriged Blocks	0.210		

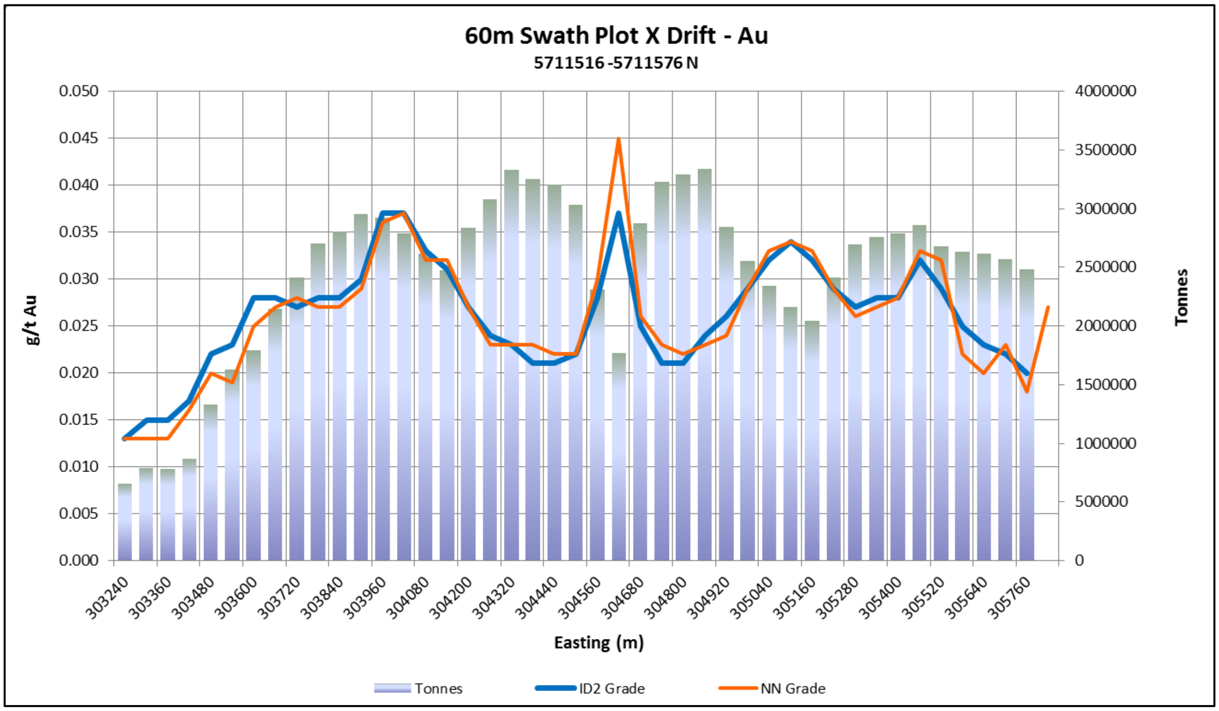
Swath plots were generated to assess the model for global bias by comparing Kriged, ID^2 and nearest neighbour estimates on panels through the Deposit. Results show a reasonable comparison between the methods, particularly in the main portions of the Deposit indicated by the bar charts (Figure 14-18 to Figure 14-20).

Figure 14-18: Cu Swath Plot (E-W) at 5711516-571156 NORTH



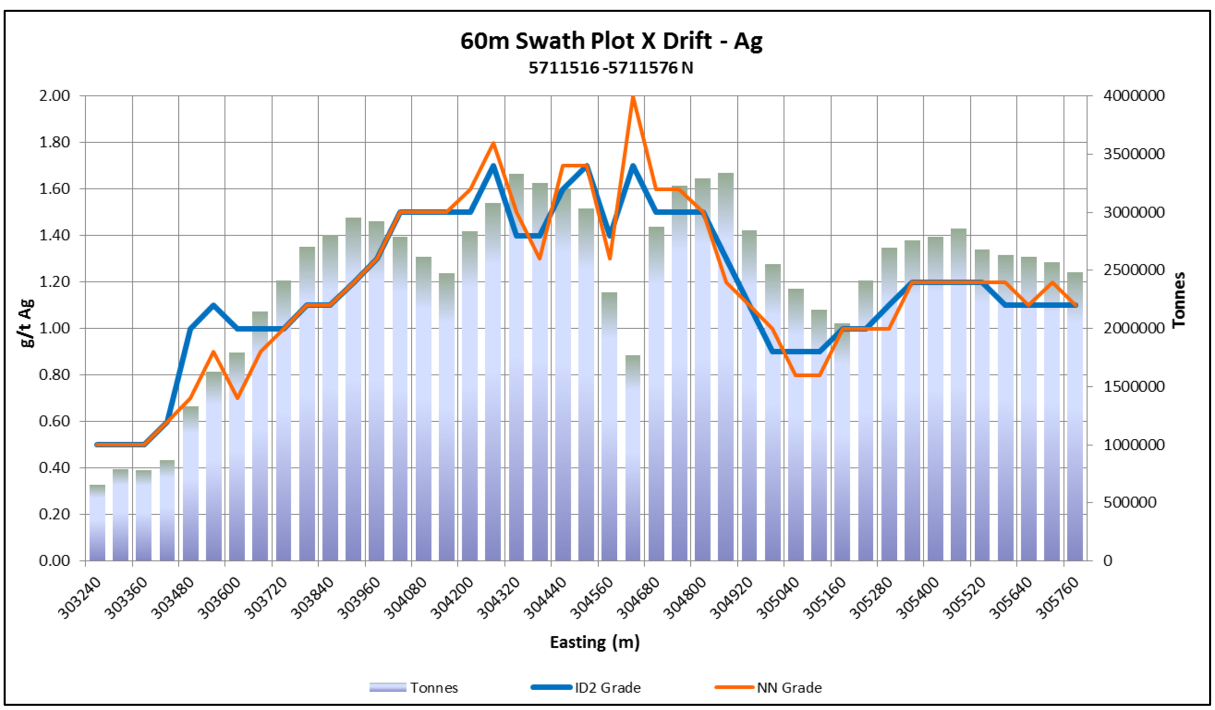
GeoSim Services Inc., March 2014

Figure 14-19: Au Swath Plot (N-S) at 5711516-571156 NORTH



GeoSim Services Inc., March 2014

Figure 14-20: Ag Swath Plot (N-S) at 5711516-571156 NORTH



GeoSim Services Inc., March 2014

14.10 MINERAL RESOURCE SUMMARY

Table 14-10 present the mineral resource estimate for the Project at a range of cut-off grades with the base case highlighted. The selected base case cut-off grade of 0.2% Cu is considered consistent with other mineral deposits of similar characteristics, scale and location.

Table 14-10: Mineral Resource Estimate March 30, 2014

Mineral Resource Estimate (Mar 30, 2014 (Geosim Services Inc.)				
Cut-off Grade (% Cu)	**Tonnes (000`s)**	**Cu Grade (%)**	**Au Grade (g/t)**	**Ag Grade (g/t)**
Measured				
0.10	707,577	0.24	0.027	1.2
0.15	564,361	0.27	0.029	1.2
0.20	405,242	0.31	0.033	1.3
0.25	270,582	0.35	0.037	1.4
0.30	174,357	0.39	0.043	1.6
Indicated				
0.10	938,948	0.21	0.025	1.2
0.15	735,877	0.24	0.027	1.2
0.20	449,111	0.28	0.031	1.3
0.25	265,150	0.33	0.036	1.4
0.30	144,310	0.37	0.041	1.5
Measured + Indicated				
0.10	1,646,525	0.22	0.026	1.2
0.15	1,300,238	0.25	0.028	1.2
0.20	854,353	0.29	0.032	1.3
0.25	535,732	0.34	0.037	1.4
0.30	318,667	0.38	0.042	1.6
Inferred				
0.10	166,251	0.21	0.023	1.1
0.15	119,743	0.25	0.025	1.2
0.20	76,884	0.29	0.029	1.3
0.25	46,492	0.33	0.032	1.4
0.30	28,615	0.38	0.034	1.5

Notes to accompany Mineral Resource Estimate:

Mineral resources are amenable to open pit mining methods and have been constrained using a Lerch-Grossman optimized pit. Assumptions include US$3.50/lb Cu with an average recovery of 89%, C$1.84/t mining cost, C$4.20/t process and G&A cost, US Exchange rate of C$1:US$0.90, Pit slope angle of 42°, No allowances were made for mining losses and dilution.

14.11 FACTORS WHICH COULD AFFECT THE MINERAL RESOURCE ESTIMATE

Areas of uncertainty that may materially impact the Mineral Resource Estimate include:

- Commodity price assumptions;
- Pit slope angles;
- Metal recovery assumptions; and
- Mining and Process cost assumptions.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the Province of British Columbia with respect to environmental, permitting, taxation, socio-economic, marketing and political factors. There are no known legal or title issues that would materially affect the mineral resource estimate.

There is a degree of uncertainty to the estimation of Mineral Reserves and Mineral Resources and corresponding grades being mined or dedicated to future production. The estimating of mineralization is a subjective process and the accuracy of estimates is a function of the accuracy, quantity, and quality of available data, the accuracy of statistical computations, and the assumptions used and judgments made in interpreting engineering and geological information. There is significant uncertainty in any Mineral Resource/Mineral Reserve estimate, and the actual deposits encountered and the economic viability of mining a deposit may differ significantly from these estimates until Mineral Reserves or Mineral Resources are actually mined and processed, the quantity of Mineral Resources/Mineral Reserves and their respective grades must be considered as estimates only. In addition, the quantity of Mineral Reserves and Mineral Resources may vary depending on, among other things, metal prices.

Any material change in quantity of Mineral Reserves, Mineral Resources, grade, or stripping ratio may affect the economic viability of a property. In addition, there can be no assurance that recoveries in small scale laboratory tests will be duplicated in larger scale tests under on-site conditions or during production. Fluctuation in metal or commodity prices, results of additional drilling, metallurgical testing, receipt of new information, and production and the evaluation of mine plans subsequent to the date of any estimate may require revision of such Mineral Resources may be materially affected by mining, infrastructure, or other relevant factors.

15 MINERAL RESERVE ESTIMATE

This Section describes the open pit optimization process including key assumptions and economic considerations leading to pit limit selection and the reporting of the mineral reserve used for mine planning and scheduling as described in Section 16.

15.1 OPEN PIT OPTIMIZATION

15.1.1 *INTRODUCTION*

The Harper Creek Copper/Gold Project pit optimization was carried out using Minesight® mine planning software for large scale open pit mining. A series of unsmoothed pit shells were created using a Lerchs Grossman algorithm with revenue factors declining from unity. These pit shells were used as a basis for selecting an ultimate pit and to develop pit phase detailed designs to be used in production scheduling.

15.1.2 *ECONOMIC PARAMETERS APPLIED TO MINE DESIGN*

15.1.2.1 Metal Prices

Base case pit optimization metal prices were provided by Yellowhead Mining Incorporated (YMI). Metal prices used in the resource value determination are defined in Table 15-1.

Table 15-1: Base Case Metal Prices (YMI)

Base Case Pit Optimization Metal Prices	
Copper	US$2.25/lb
Gold	US$1,250/oz
Silver	US$20.00/oz
Exchange Rate	US$0.90

15.1.2.2 Smelter Terms and Offsite Costs

Copper, gold and silver will report to a copper concentrate. The basis for pit optimization was the net minegate revenue/t calculated for each block in the resource model.

Offsite costs used in the resource value determination to develop pit limits are defined in Table 15-2. Minor adjustments made subsequent to pit design and scheduling are summarized in the financial analysis section of the report.

Table 15-2: Off Site Estimate (YMI)

Resource Valuation Off Site Costs		
Item	**Estimate C$**	**Estimate US$**
Concentrate transportation; truck and rail to Vancouver	40.00/wmt	
Stevedoring (Vancouver)	35.00/wmt	
Ocean Freight	55.56/wmt	50.00/wmt
Total Concentrate Handling, including losses and insurance	142.88/wmt or 155.31/dmt	
Treatment charges		85.00/dmt
Refining charges		0.085/payable lb of copper
1 unit deduction assumed for 96.1% pay factor		

The Net Smelter Return (NSR) calculations allow for the accounting of:

- ore grades (Cu, Au, and Ag) thus taking into account the variability in the metal content of the Deposit;
- ore mill recoveries;
- contained metal in concentrate;
- deductions and payable metal value;
- metal prices;
- freight smelting and refining charges.

Royalty charges were not applied at this level of resource valuation. However, a royalty payable to Noranda is applicable to a minor amount of the mineral reserve and was included in the cashflow model of the economic analysis section.

The NSR calculation, smelter terms and offsite costs as summarized in Table 15-3 used the C$1:00:US$0.90 exchange rate. The table shows the development of conversion factors for recoverable grades to net minegate revenue for copper, gold and silver.

Table 15-3 Net Smelter Return

Test Block NSR Calculation		
Copper Head Grade	%	0.270
Gold Head Grade	g/t	0.030
Silver Head Grade	g/t	1.110
Molybdenum Head Grade	%	-
Recoverable Copper Head Grade	%	0.242
Recoverable Gold Head Grade	g/t	0.017
Recoverable Silver Head Grade	g/t	0.627
Metallurgical Recovery		
Copper Recovery	%	89.55
Gold Recovery	%	56.67
Silver Recovery	%	56.48
Metal Pricing		
Copper Price Price Participation Level	US$/lb	$3.00
Copper Price Participation	US$/lb	5.0%
Copper Price Participation Cap	US$/lb	$4.00
Copper Price	US$/lb	$2.25
Price Participation	US$/lb	$0.00
Copper Price Realized Net of PP	US$/lb	$2.25
Gold Price	US$/ounce	$1,250.00
Silver Price	US$/ounce	$20.00
USD:CDN Exchange		$0.90
Copper Concentrate		
Copper Concentrate Grade		25.50
Moisture Content	%	8.0%
Contained Copper	lb/dmt	562.02
Contained Gold	g/dmt	1.79
Contained Silver	g/dmt	66.12
Payable Copper	lb/dmt	539.98
Payable Gold	g/dmt	1.61
Payable Silver	g/t	59.50
Concentrate - Recovery Based	dmt/t ore	0.00948
Gross Value Concentrate	C$/dmt	**$1,464.52**
Gross Value Concentrate	C$/wmt	**$1,347.36**
	C$/t ore	**$13.89**
Copper Concentrate Handling		
Truck Haul & Rail to Vancouver	C$/wmt	$40.00
Stevedoring in Vancouver	C$/wmt	$35.00
Ocean Freight	C$/wmt	$55.56
Umpiring & Sampling	C$/wmt	$1.40
Marketing	C$/wmt	$11.00
Insurance	C$/wmt	$2.02
Losses	C$/wmt	$1.35
Total Concentrate Handling	C$/wmt	$146.32
Total	C$/dmt	**$159.05**
	C$/t ore	**$1.51**
Copper Concentrate Treatment and Refining		
Deduction for Copper	unit	1.00
Treatment Charges	US$/dmt	$85.00
Gold Payment	%	90.0%
Silver Payment	%	90.0%
Copper Refining Cost	US$/payable lb	$0.0850
Gold Refining Cost	US$/payable oz	$6.500
Silver Refining	US$/payable oz	$0.550
Total Treatment and Refining	C$/dmt	**$146.99**
	C$/t ore	**$1.39**
Copper Net Smelter Return		
Net Smelter Return	C$/dmt	**$1,158.48**
	C$/payable lb Cu	$2.15
NSR Before Royalty	**NSR C$/t**	**$10.98**
Royalty	**CDN$/t**	**$0.00**
NSR After Royalty	**NSR C$/t**	**$10.98**
NSR for Copper	**NSR C$/t**	**$10.02**
NSR for Gold	**NSR C$/t**	**$0.61**
NSR for Silver	**NSR C$/t**	**$0.35**
Check	**NSR C$/t**	**$10.98**
Factor for Copper	**$/% Recoverable**	**$41.45**
Factor for Gold	**$/g Recoverable**	**$35.89**
Factor for Silver	**$/g Recoverable**	**$0.56**

15.1.2.3 Onsite Operating Costs and Increments

The onsite operating costs used for pit limit analysis include general and administration (G&A), surface facilities, processing and mining costs.

The G&A, surface facilities and processing cost was estimated at C$0.42/t, C$0.31/t and C$3.25/t milled respectively. The preliminary operating costs for mining ore and waste was estimated at C$1.85/t mined. An incremental haulage cost of C$0.025/t/bench was added for each bench below the pit entrance at 1,660m elevation.

15.1.2.4 Sustaining Capital Consideration

Sustaining capital consideration was made for tailings and mining equipment. Tailings dam construction is an ongoing cost related directly to waste and tailings placement in the tailings management facility. In a similar context, mining equipment is consumed in direct relationship to mined quantities. An allowance of C$0.20/t mined was made for equipment sustaining capital. An allowance of C$0.30/t milled was made for tailings operation and dam construction. The onsite costs used for pit limit definition are summarized in Table 15-4.

Table 15-4 Pit Optimization Parameters (C$)

Processing Operating Costs		**Ore**	
General & Administration	$/t milled	$0.420	
Surface Facilities	$/t milled	$0.310	
Mill Operating	$/t milled	$3.250	
Subtotal	$/t milled	**$3.980**	
Tailings Operating Costs		$0.030	
Tailings Sustaining Capital	$/t milled	$0.270	
Total Processing	$/t milled	**$4.280**	
Recoverable Copper Equivalent	CuEq%	0.103	

Base Mine Operating Costs		**Ore**	**Waste**
Mining	$/t mined	$1.850	$1.850
Sustaining Capital	$/t mined	$0.200	$0.200
Total Base Mining	$/t mined	**$2.050**	**$2.050**
Entrance Bench	bench	18	18
Increment	$/t/bench	$0.025	$0.025
Total Processing + Base Mining	$/t milled	**$6.330**	
Direct Process + Mining	$/t milled	**$5.830**	
Stockpile Reclaim	$/t Stockpile	$0.750	
Minimum Profit	$/t Stockpile	$0.000	
Stockpile Marginal Breakeven	$/t Stockpile	**$5.030**	
Recoverable Copper Equivalent	CuEq%	0.121	
Stockpile Reclaim	$/t Stockpile	$0.750	
Minimum Profit	$/t Stockpile	$0.250	
Stockpile Marginal Breakeven	$/t Stockpile	**$5.280**	
Recoverable Copper Equivalent	CuEq%	0.127	

15.1.3 *METALLURGICAL PARAMETERS*

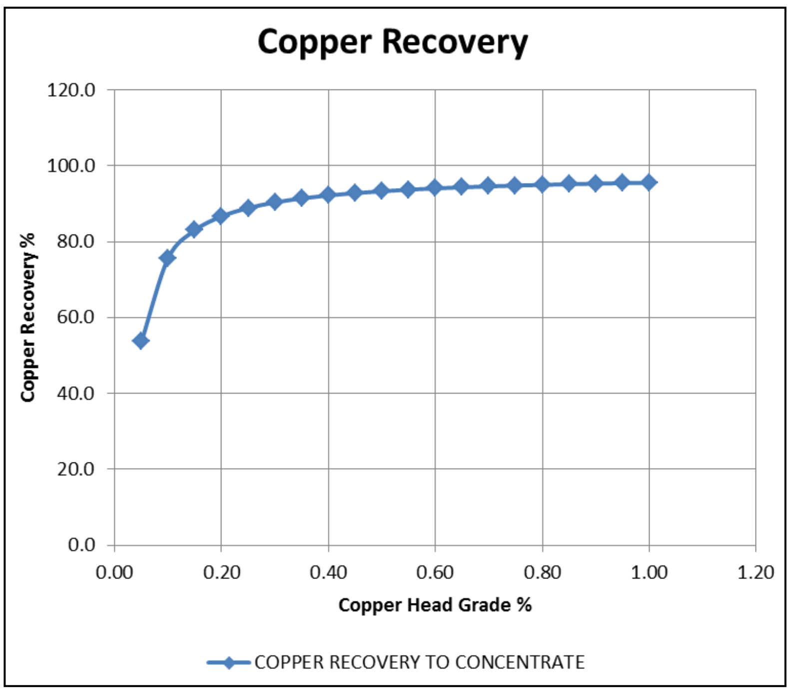
15.1.3.1 Process Selection

A single mineral processing option is considered for the Project; i.e., primary crushing followed by grinding and conventional flotation of copper concentrate that will be smelted and refined off-site.

15.1.3.2 Process Recovery

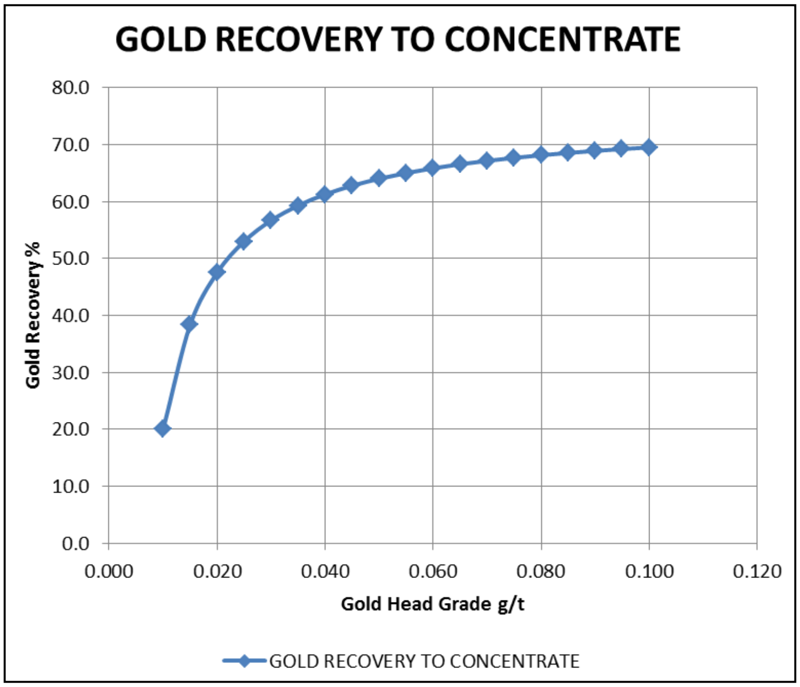
Metallurgical recovery estimates were provided by Laurion Consulting Inc. The recoveries of copper, gold and silver to copper concentrate were based upon predictions of process tailings grade. Equations were developed to estimate metallurgical recovery and recoverable copper, gold and silver for each block in the resource model. This is an intermediate step in the development of the mine plan. The final production schedule was used to provide the basis for annual recovery estimates. The estimated metallurgical recoveries to concentrate is shown in Figures 15-1 to 15-3.

Figure 15-1 Copper Recovery



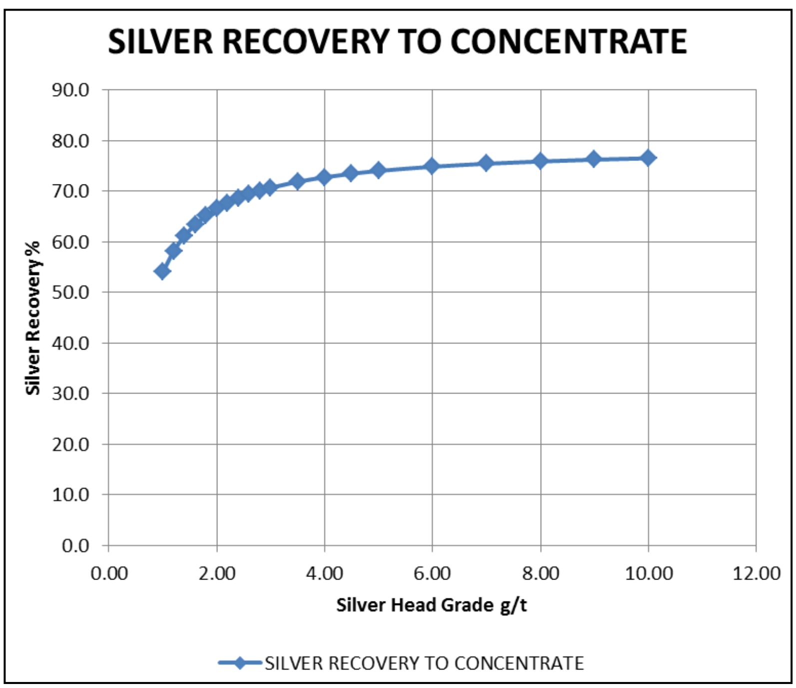
Laurion Consulting Inc., February 2012

Figure 15-2 Gold Recovery



Laurion Consulting Inc., February 2011

Figure 15-3 Silver Recovery



Laurion Consulting Inc., February 2011

15.1.3.3 Concentrate Grade

The copper concentrate grade for copper was estimated by Laurion Consulting Inc. to be 25.5% Cu with 8.0% moisture content. Gold and silver concentrate grades were calculated based upon the gold and silver head grades and the estimated recovery to concentrate.

15.1.4 *Block Model*

15.1.4.1 General

The resource block model developed by Geosim Services Inc. using Surpac software is described in Section 14. The block model and surfaces for topography and the hardrock/overburden interface were imported to a Minesight® mine planning system model. The block model limits and block dimensions are shown in Table 15-5.

Table 15-5 Block Model Limits

Model Limits 2014				
	Meters/Blocks	**Minimum**	**Maximum**	**Length**
Limits X	m	303,000.0	306,252.0	3,252.0
Limits Y	m	5,709,800.0	5,712,860.0	3,060.0
Limits Z	m	1,000.0	1,864.0	864.0
block x	m	12.0		
block y	m	12.0		
block z	m	12.0		
nblock x	blocks	271.0		
nblock y	blocks	255.0		
nblock z	blocks	72.0	4,975,560.0	

Key block model items transferred from Geosim included density, lithology, copper, gold and silver grades, resource class, acid potential, neutralization potential, calcium, sulphur, calcium carbonate and neutralization potential ratio.

Additional items were included in the Minesight® Model to store slope codes for design purposes, net value and scheduling destinations.

15.1.4.2 Resource Classification

15.1.4.2.1 Resource Class

The resource model includes measured, indicated and inferred resources. Measured and indicated resources were used to define the pit limits and for reporting of the mineral reserve for scheduling. Inferred resources were not used for the purpose of mineral resource estimation.

15.1.4.2.2 Mining Recovery

Mining Recovery was assumed to be 100%. No mining losses were applied to the mineral reserve for the following reasons:

- The Deposit shows good lateral and vertical continuity at the cutoff grades applied for scheduling;
- There is a broad width to the ore zones on individual benches; and
- A detailed grade control program will be implemented.

15.1.4.2.3 Mining Dilution

Internal dilution is incorporated in the resource model by virtue of the compositing and interpolation method used to obtain the block grades. No additional dilution was applied in optimization.

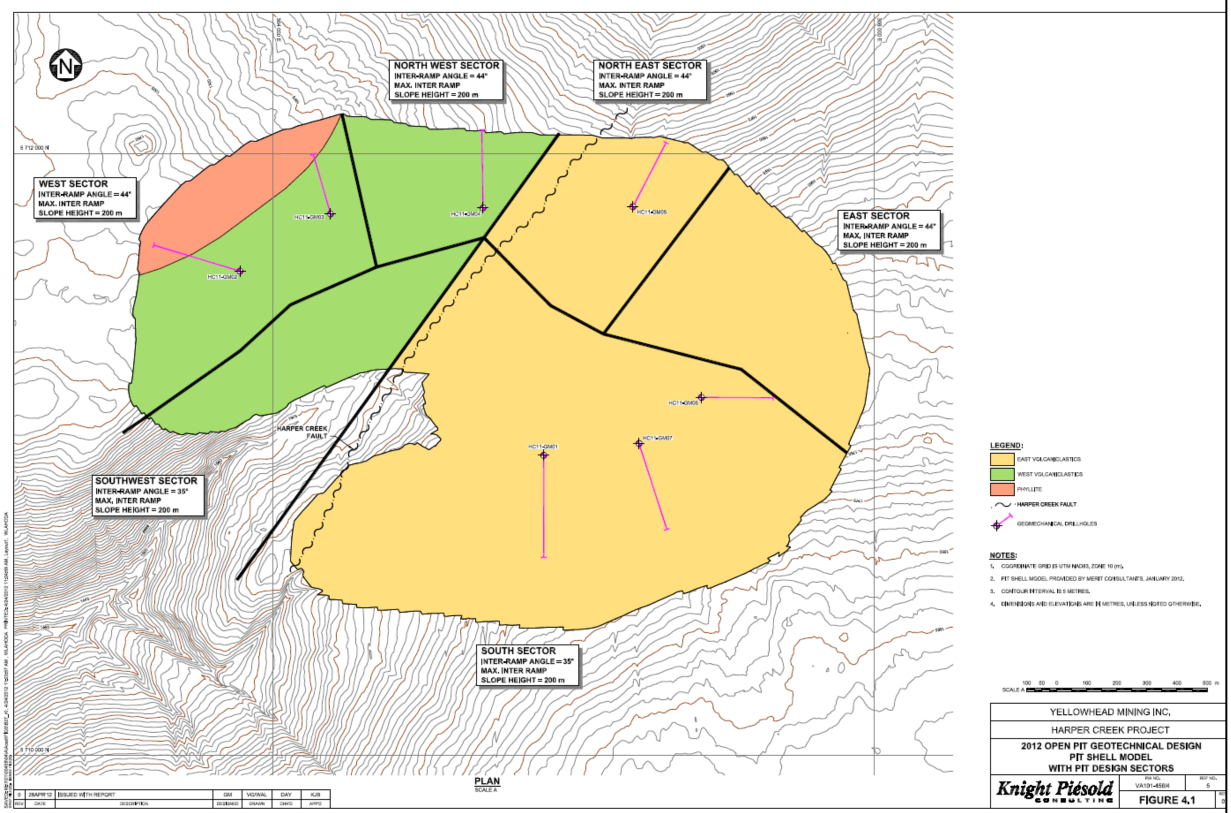
15.1.5 *Wall Slope Design*

Wall slopes design angles were assigned by sector. Wireframes were constructed to represent six design sectors. The block model was then coded using these wireframes and a seventh sector was then assigned for overburden. Design sector definitions are shown in Figure 15-4. Wall slope design sectors are described in more detail in Section 16.

Figure 15-4 Pit Slope Design Sectors

SLOPE SECTORS

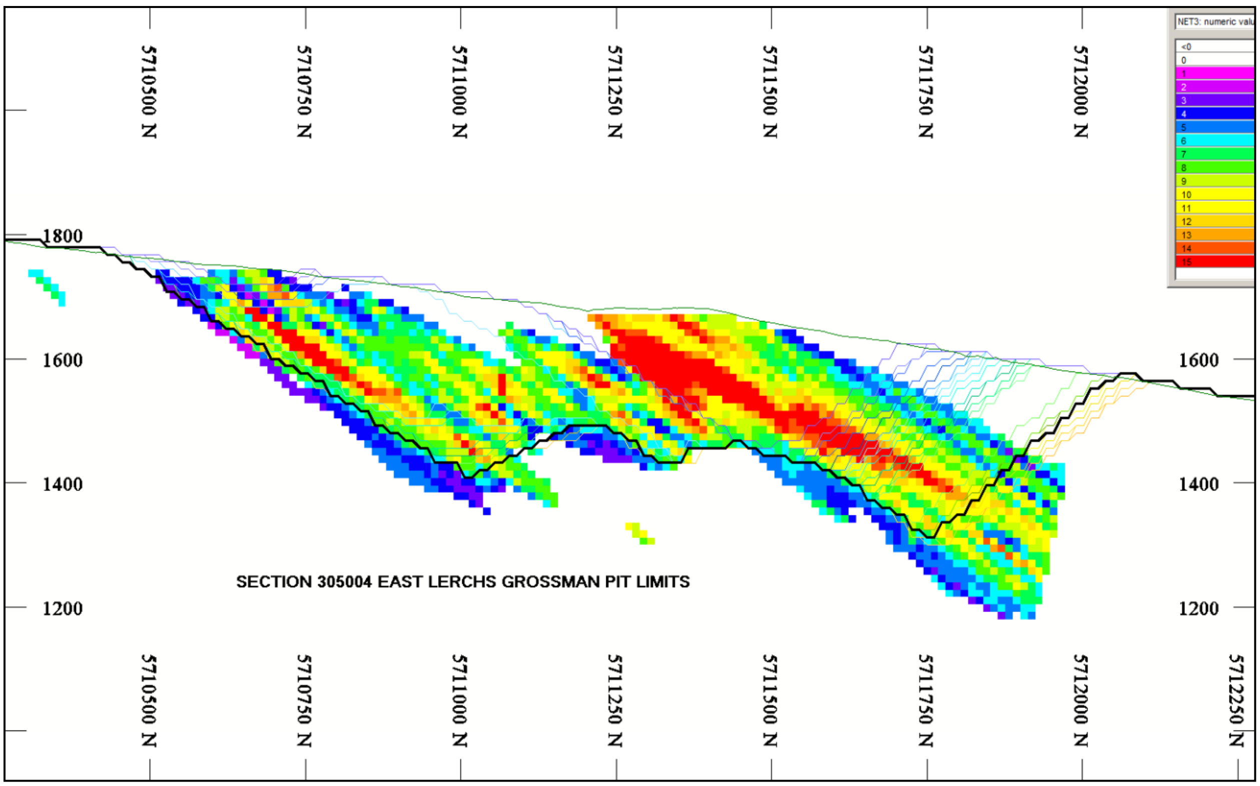
Sector	Slope Code	Wall Geology	Inter-ramp Slope	Bench Height	Bench Face Angle	Berm Width	Berm Interval	Pit Optimization Slope
NORTHEAST	1	East Volcaniclastics	44.0	12.0	70.0	8.0	1.0	42.0
EAST	2	East Volcaniclastics	44.0	12.0	70.0	8.0	1.0	42.0
SOUTH	3	East Volcaniclastics	35.0	12.0	60.0	10.0	1.0	33.0
SOUTHWEST	4	East Volcaniclastics	35.0	12.0	60.0	10.0	1.0	33.0
WEST	5	Phyllite	44.0	12.0	70.0	8.0	1.0	40.0
NORTHWEST	6	West Volcaniclastics	44.0	12.0	70.0	8.0	1.0	40.0
OVERBURDEN	7	Lith Code 99	38.0	12.0	38.0	8.0	1.0	27.0



15.1.6 *PIT LIMIT ANALYSIS*

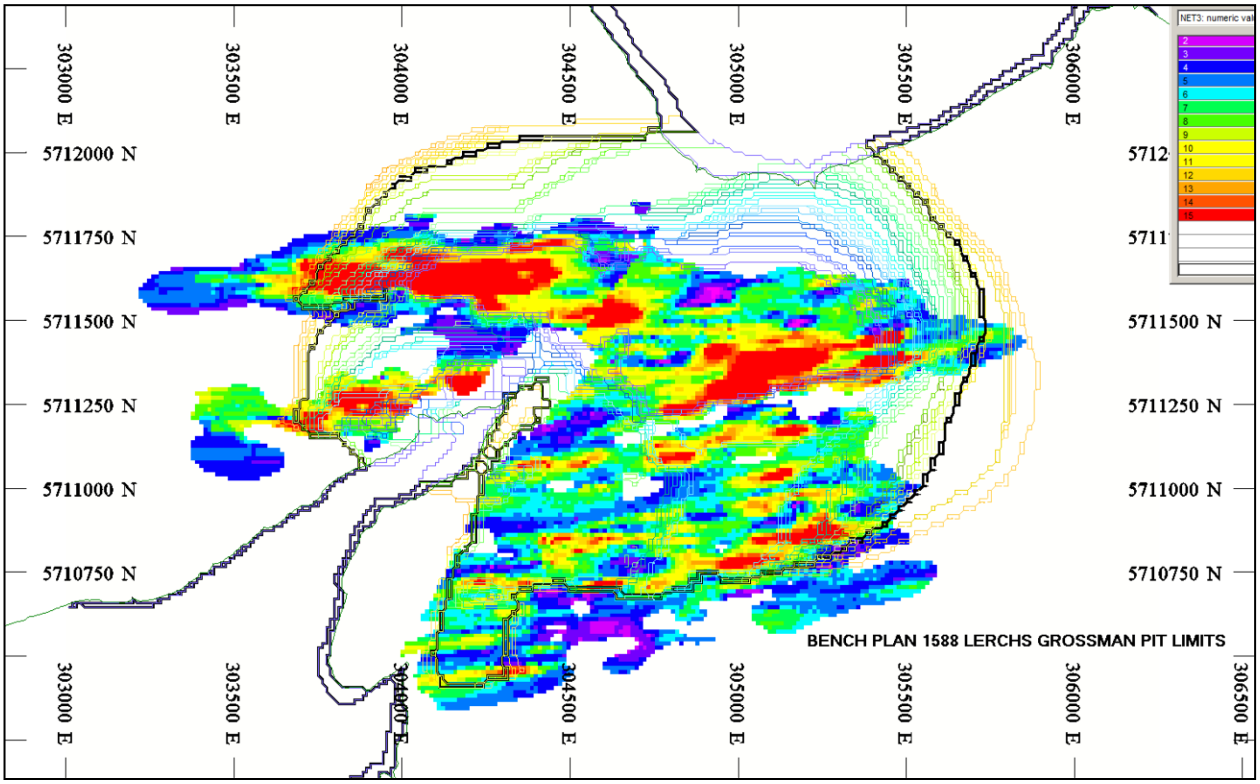
Unsmoothed pit limits were developed using a Minesight® variable slope Lerchs Grossman Algorithm. The preliminary net mine gate revenue and operating costs were used to estimate the value of each regular block in the model. A series of 40 nested pit limits were defined using revenue factors between 0.20 and 1.00. The NSR values and nested pit limits used as a guide for pit design and are shown on the section and plan below.

Figure 15-5 Cross Section NSR Lerchs Grossman Pit Limits



Nilsson Mining Services Inc., June 2014

Figure 15-6 Bench Plan NSR Lerch Grossman Pit Limits



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The resources within the unsmoothed nested Lerchs Grossman pit limits are summarized in Table 15-6 at 0.14% Cu cutoff grade that corresponds to approximately C$5.30/t NSR Cutoff.

Table 15-6 Lerchs Grossman In-Pit Mineral Resources (Nilsson)

Pit Limit Analysis – Cutoff 0.14% Cu- approximately C$/tonne Net Minegate Revenue Base Case C$2.25/lb Cu using Net3

Pit	Factor	Ore tonnes	Waste tonnes	Total tonnes	SR	Grade Cu%	Grade Au g/t	Grade Ag g/t	Net Minegate $/tonne
1	0.200	146,834	33,048	179,882	0.23	0.289	0.033	1.17	11.81
2	0.221	156,765	36,999	193,764	0.24	0.288	0.033	1.18	11.76
3	0.241	174,146	44,359	218,505	0.25	0.287	0.033	1.17	11.68
4	0.262	177,653	45,693	223,346	0.26	0.286	0.033	1.17	11.65
5	0.282	188,253	51,463	239,716	0.27	0.285	0.032	1.18	11.63
6	0.303	199,103	57,352	256,455	0.29	0.285	0.032	1.18	11.59
7	0.323	209,918	64,414	274,332	0.31	0.284	0.032	1.18	11.58
8	0.344	213,187	66,739	279,926	0.31	0.284	0.032	1.18	11.57
9	0.364	221,352	71,536	292,888	0.32	0.283	0.032	1.18	11.53
10	0.385	231,584	77,068	308,652	0.33	0.282	0.032	1.17	11.46
11	0.405	276,808	111,924	388,732	0.40	0.280	0.032	1.16	11.36
12	0.426	343,293	148,297	491,590	0.43	0.271	0.031	1.14	10.98
13	0.446	356,614	159,551	516,165	0.45	0.271	0.031	1.14	10.97
14	0.467	371,686	169,969	541,655	0.46	0.270	0.031	1.14	10.93
15	0.487	403,168	192,671	595,839	0.48	0.269	0.030	1.14	10.86
16	0.508	422,124	207,215	629,339	0.49	0.268	0.030	1.15	10.82
17	0.528	433,154	217,963	651,117	0.50	0.268	0.030	1.15	10.82
18	0.549	441,651	225,121	666,772	0.51	0.268	0.030	1.14	10.81
19	0.569	460,013	242,862	702,875	0.53	0.267	0.030	1.14	10.79
20	0.590	464,391	247,379	711,770	0.53	0.267	0.030	1.15	10.79
21	0.610	571,219	327,121	898,340	0.57	0.261	0.029	1.17	10.54
22	0.631	590,931	340,108	931,039	0.58	0.260	0.029	1.18	10.49
23	0.651	631,344	386,835	1,018,179	0.61	0.260	0.029	1.17	10.50
24	0.672	637,644	393,266	1,030,910	0.62	0.260	0.029	1.18	10.50
25	0.692	654,088	416,353	1,070,441	0.64	0.260	0.029	1.18	10.51
26	0.713	657,183	419,669	1,076,852	0.64	0.260	0.029	1.18	10.51
27	0.733	678,402	440,303	1,118,705	0.65	0.259	0.029	1.19	10.47
28	0.754	697,397	466,444	1,163,841	0.67	0.260	0.029	1.20	10.48
29	0.774	703,193	474,463	1,177,656	0.67	0.259	0.029	1.20	10.48
30	0.795	714,176	488,804	1,202,980	0.68	0.259	0.029	1.20	10.48
31	0.815	727,252	505,655	1,232,907	0.70	0.259	0.029	1.20	10.48
32	0.836	731,676	511,498	1,243,174	0.70	0.259	0.029	1.20	10.48
33	0.856	765,761	571,534	1,337,295	0.75	0.260	0.029	1.20	10.51
34	0.877	767,230	574,762	1,341,992	0.75	0.260	0.029	1.20	10.51
35	0.897	782,868	594,224	1,377,092	0.76	0.260	0.029	1.20	10.50
36	0.918	793,254	608,288	1,401,542	0.77	0.260	0.029	1.20	10.50
37	0.938	798,263	614,323	1,412,586	0.77	0.260	0.029	1.20	10.49
38	0.959	822,063	655,934	1,477,997	0.80	0.260	0.029	1.21	10.50
39	0.979	868,972	694,093	1,563,065	0.80	0.258	0.028	1.23	10.39
40	1.000	869,798	694,965	1,564,763	0.80	0.258	0.028	1.23	10.39

Preliminary scheduling of unsmoothed pit resources indicated that significant additional discounted value was not realized beyond Pit 30 which captured 96.7% of the total potential value of the ultimate pit limit while processing 83.4% of the ultimate pit contained resources.

Figure 15-7 Cumulative Present Value Pit Shell

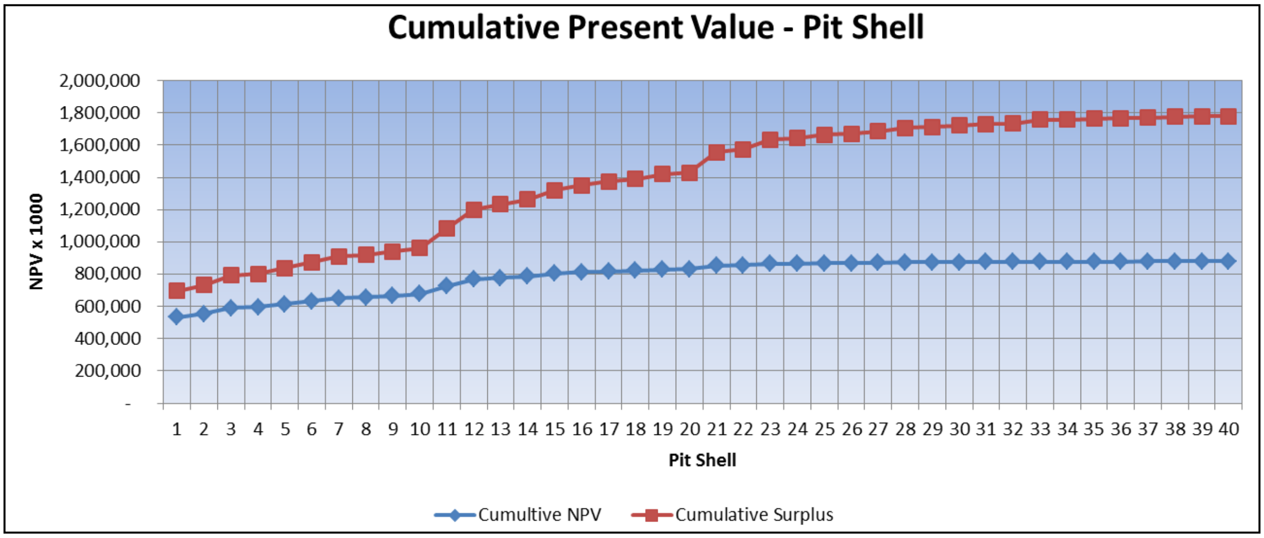

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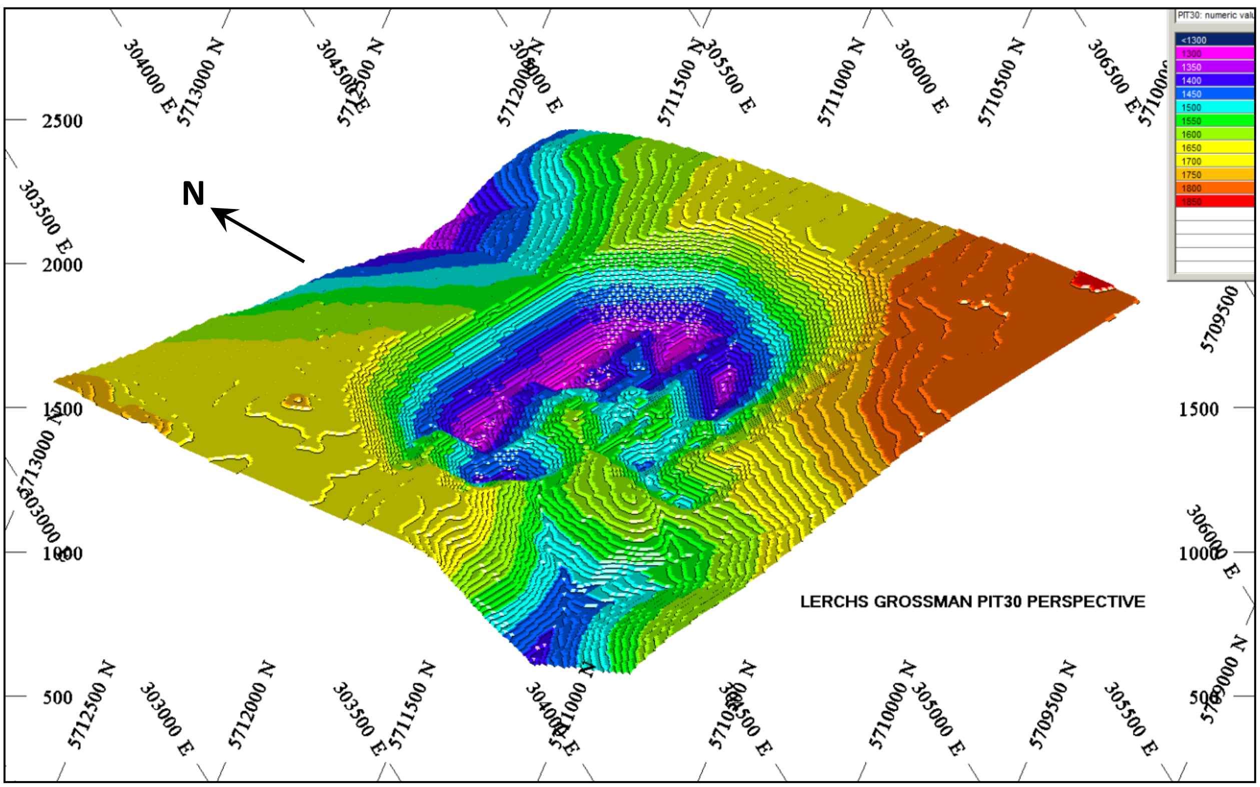
Figure 15-8 Rendered Isometric View-Pit 30



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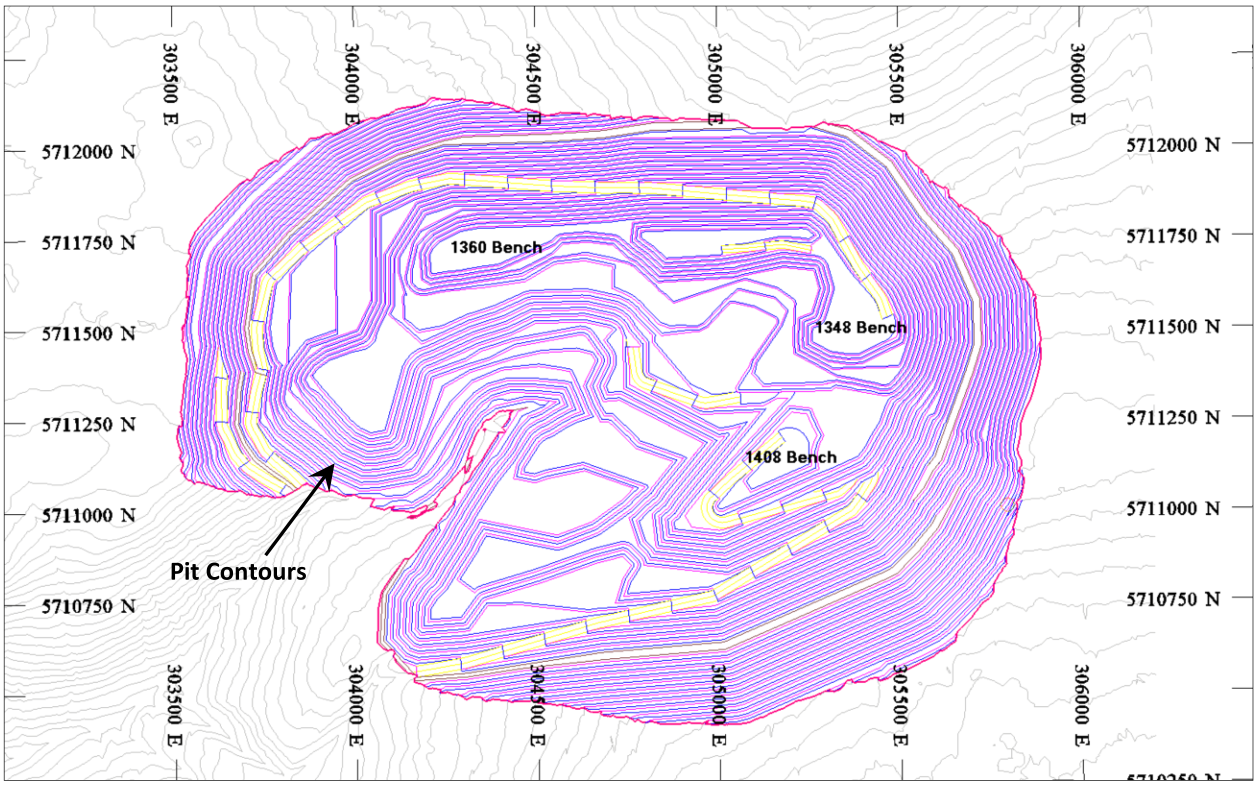
15.2 PIT DESIGN

The pit design detailed description is provided in Section 16. The open pit was designed to be developed in 5 phases. The ultimate pit configuration is shown in Figure 15-9.

Figure 15-9 Final Pit Limits



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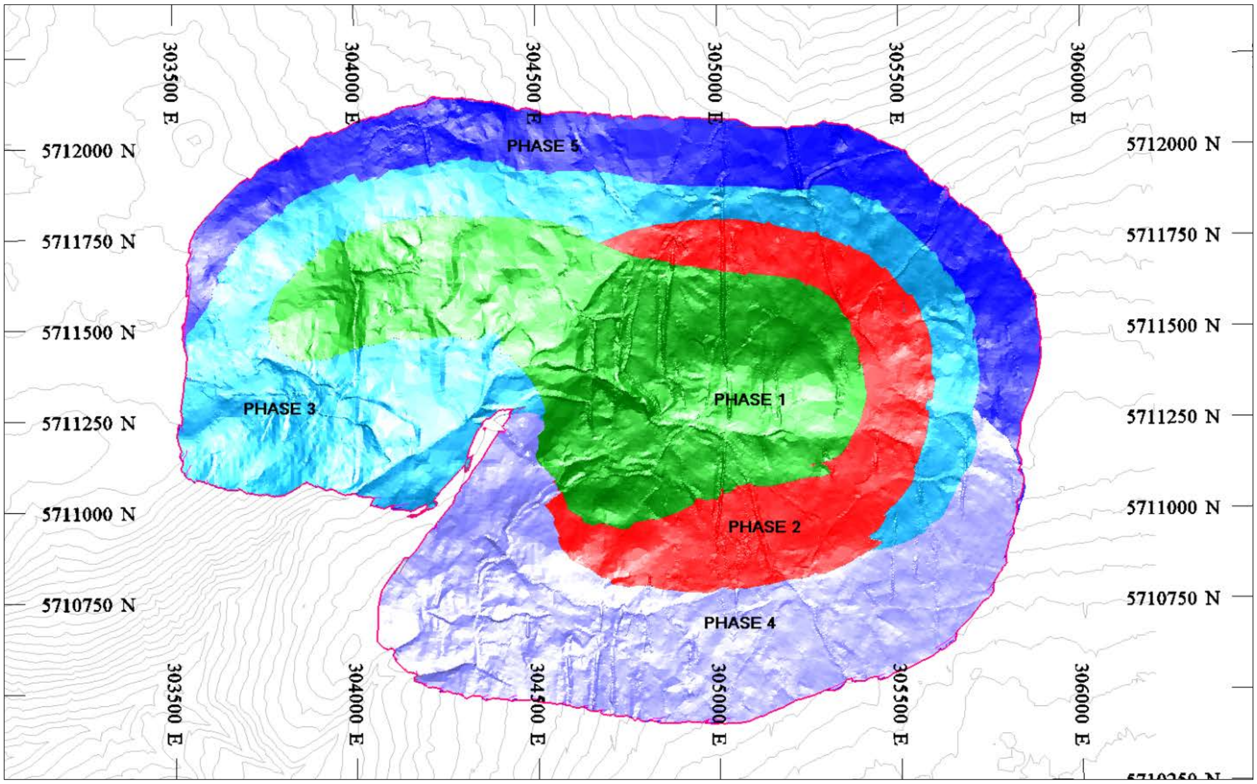
The five development phases are shown as 3D solids in Figure 15-10. Phase 1 is located in the central part of the Deposit. Phase 2 expands the pit to the north, east and south. Phase 3 expands the pit to the north and west and to depth. Phase 4 is an expansion to the south. Phase 5 expands the pit to the north and to depth.

Figure 15-10 Rendering of Mine Development Phase Solids



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15.3 Mineral Reserve

The mineral reserve for the Deposit was estimated using a copper price of US$2.25/lb., a gold price of US$1,250.00/oz. and a silver price of US$20.00/oz. An exchange rate of C$1.00:US$0.90 was assumed. The mineral reserve is reported using a 0.14% copper cutoff grade. The proven and probable mineral reserve at the Project is 716.2Mt with an average grade of 0.26% Cu, 0.029g/t Au and 1.18g/t Ag (Table 15-7).

Table 15-7 Harper Creek Mineral Reserve (Nilsson)

Mineral Reserve					Contained Metal		
Phase 1	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**	**Cu t x 1000**	**Au oz x 1000**	**Ag oz x 1000**
Proven	107.3	0.29	0.034	1.17	310.8	116.2	4,022.8
Probable	12.7	0.24	0.030	1.24	30.7	12.0	503.2
Total	**119.9**	**0.28**	**0.033**	**1.17**	341.5	128.3	4,526.0
Phase 2	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**	**Cu t x 1000**	**Au oz x 1000**	**Ag oz x 1000**
Proven	67.3	0.26	0.029	1.08	174.4	63.7	2,337.8
Probable	27.6	0.23	0.027	1.03	64.9	23.9	914.0
Total	**95.0**	**0.25**	**0.029**	**1.07**	**239.4**	**87.6**	**3,251.7**
Phase 3	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**	**Cu t x 1000**	**Au oz x 1000**	**Ag oz x 1000**
Proven	114.9	0.27	0.032	1.34	309.0	117.8	4,941.6
Probable	58.2	0.25	0.028	1.22	146.6	52.8	2,275.2
Total	**173.1**	**0.26**	**0.031**	**1.30**	**455.6**	**170.6**	**7,216.8**
Phase 4	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**	**Cu t x 1000**	**Au oz x 1000**	**Ag oz x 1000**
Proven	110.0	0.24	0.023	1.15	266.3	82.8	4,068.8
Probable	82.4	0.22	0.023	1.23	179.6	60.6	3,257.2
Total	**192.4**	**0.23**	**0.023**	**1.18**	**445.9**	**143.4**	**7,326.0**
Phase 5	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**	**Cu t x 1000**	**Au oz x 1000**	**Ag oz x 1000**
Proven	57.7	0.29	0.032	1.13	166.9	58.7	2,094.0
Probable	78.1	0.26	0.028	1.07	199.9	71.0	2,686.3
Total	**135.8**	**0.27**	**0.030**	**1.09**	366.8	129.7	4,780.3
Summary	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**	**Cu t x 1000**	**Au oz x 1000**	**Ag oz x 1000**
Proven	457.2	0.27	0.030	1.19	1,227.4	439.2	17,465.0
Probable	258.9	0.24	0.026	1.16	621.7	220.4	9,636.0
Total	**716.2**	**0.26**	**0.029**	**1.18**	**1,849.2**	**659.6**	**27,101.0**

15.3.1 *INTERPRETATION*

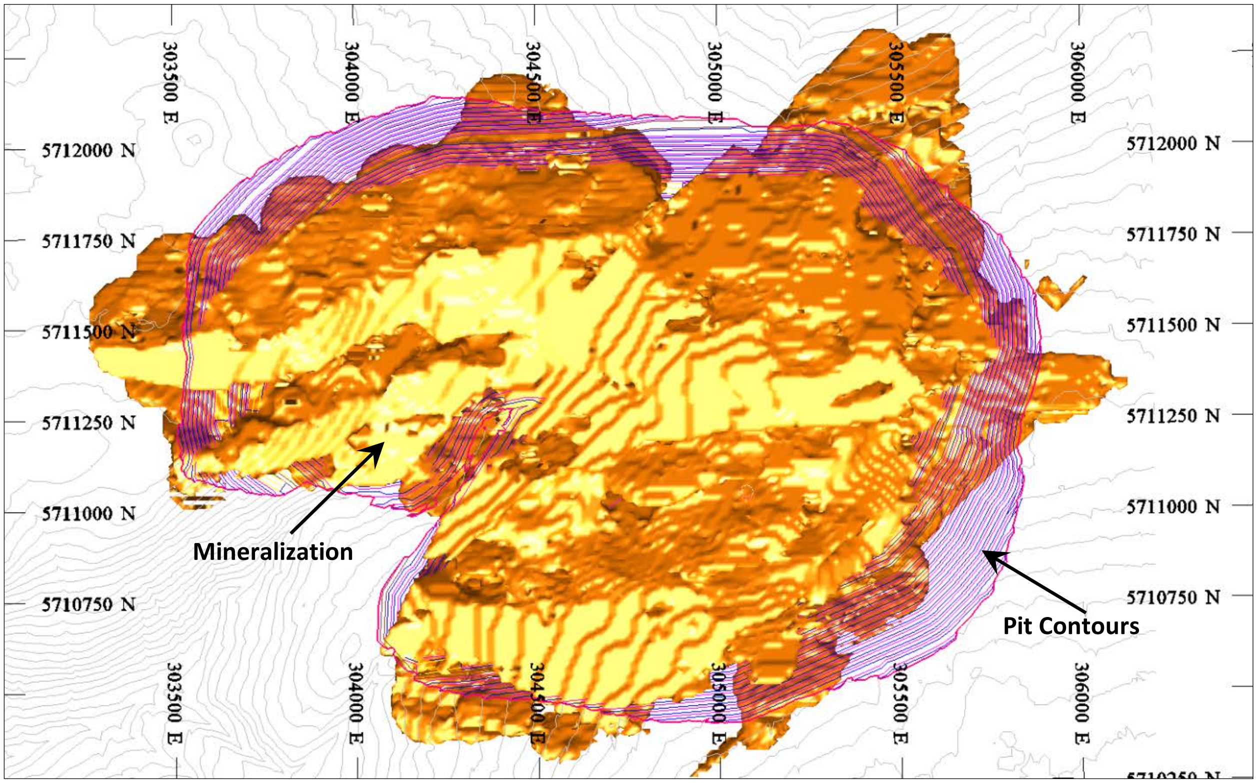
The mineral reserve estimate was based upon economic parameters, geotechnical design criteria and metallurgical recovery assumptions detailed throughout this FS. Changes in these assumptions will impact the in-pit mineral reserve estimate.

In general, increases in operating costs, reductions in revenue assumptions or reductions in metallurgical recovery may result in increased cut-off grades, reductions in in-pit mineral reserve and increasing strip ratios. The converse is also true. Reductions in operating costs, increases in revenue assumptions or increases in metallurgical recovery may result in reduced cut-off grades and increases to in-pit mineral resources.

The mineral reserve estimate is also dependent upon successful completion of the environmental permitting process and provision of electric power to the mine-site.

A pit limit plan view of the distribution of measured and indicated resource above 0.14% Cu is shown in Figure 15-11.

Figure 15-11 Resource Distribution - Pit Limits



Nilsson Mining Services Inc., June 2014

16 MINING METHODS

John Nilsson, P.Eng., Nilsson Mine Services Ltd. (NMS) visited the project location in July 2011. Following the site visit, pit designs and production schedules were developed for a five phase open pit mining plan. The results of which were detailed in the "Technical Report and Feasibility Study for the Harper Creek Copper Project", March 2012 and restated in January 2013. Since that time, additional drilling was undertaken on the project and an updated resource was developed, hence Section 16 reflects the changes described and summarizes the design process, the mine schedule, the mine operations plan supporting the operating and capital cost estimates for the open pit.

16.1 SUMMARY

16.1.1 *PROJECT DESCRIPTION*

The Project consists of a 70,000t/d conventional copper concentrator and a combined electric and diesel powered open pit operation. The mineral reserve is estimated to be 716.2Mt with an average grade of 0.26% Cu, 0.029g/t Au and 1.18g/t Ag reported at a 0.14% Cu cutoff grade. The mineral reserve will be mined by open pit methods in five phases of open pit development and expansion. The overall strip ratio is 0.76:1. The total in-pit waste is 541.7Mt. The overall mine life is 28 years after start-up of the concentrator.

Mill feed and waste will be drilled by diesel and electric powered rotary drills and blasted using heavy ammonium nitrate and fuel oil. Mill feed and waste will be loaded into 227t end dump mine trucks by 42.0m^3 electric hydraulic shovels and an 18.5m^3 wheel loader. Potential acid generating waste rock (PAG) will be placed in the tailings management facility (TMF). Non acid generating waste (non-PAG) will be placed in designated disposal sites adjacent to the pit. Low grade will be stockpiled south of the plant site in designated PAG and NAG low grade stockpiles. Direct mill feed will be hauled to the primary crusher located south of the pit. Crushed ore will then be conveyed to the coarse ore stockpile and subsequently to the crushing, grinding and flotation sections of the process plant.

In general, the concentrator design is conventional with primary crushing followed by SAG mill & ball milling grinding and flotation. The simple copper gold concentrate will be dewatered and transported by truck and rail for shipment to treatment facilities overseas. Resources will be mined from the open pits and hauled directly to the crusher for 23.6 years. The implementation of an elevated cutoff grade strategy results in a low grade stockpile containing 114.9Mt at the end of the open pit life. This material will be reclaimed and processed for another 4.3 years.

Tailings will be impounded behind a constructed dam some distance from the open pit. Overburden and non-PAG waste rock from the mine will be used to construct the dam.

16.1.2 *Mineral Resource and Mineral Reserve*

The resource model for the Project was developed using conventional block modeling techniques previously described in Section 14 of this report. Measured and Indicated resources used to develop the mining plan and to report the mineral reserve as summarized in Table 16-1.

Table 16-1: Mineral Reserve (Nilsson)

Phase 1	kt x 1000	Cu%	Au g/t	Ag g/t
Proven	107.3	0.29	0.034	1.17
Probable	12.7	0.24	0.030	1.24
Total	**119.9**	**0.28**	**0.033**	**1.17**
Phase 2	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**
Proven	67.3	0.26	0.029	1.08
Probable	27.6	0.23	0.027	1.03
Total	**95.0**	**0.25**	**0.029**	**1.07**
Phase 3	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**
Proven	114.9	0.27	0.032	1.34
Probable	58.2	0.25	0.028	1.22
Total	**173.1**	**0.26**	**0.031**	**1.30**
Phase 4	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**
Proven	110.0	0.24	0.023	1.15
Probable	82.4	0.22	0.023	1.23
Total	**192.4**	**0.23**	**0.023**	**1.18**
Phase 5	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**
Proven	57.7	0.29	0.032	1.13
Probable	78.1	0.26	0.028	1.07
Total	**135.8**	**0.27**	**0.030**	**1.09**
Summary	**kt x 1000**	**Cu%**	**Au g/t**	**Ag g/t**
Proven	457.2	0.27	0.030	1.19
Probable	258.9	0.24	0.026	1.16
Total	**716.2**	**0.26**	**0.029**	**1.18**

16.1.3 *Open Pit Mine Plan*

The mine plan provides mill feed at a rate of 70,000t/d or 25,550kt/a with an allowance for ramping up of production in Year 1 to 90% capacity. The overall mining rate will be 60,000kt/a.

The overall mine production has been scheduled by bench and development phase on an annual basis. A copper cutoff grade of 0.14% Cu was applied for differentiating ore from waste. Material above 0.14% Cu will be stockpiled in separate PAG and non-PAG low grade stockpiles located southeast and southwest of the plant site for processing at the end of the mine life. In general, material above 0.16% Cu will be directed to the primary crusher on the basis of highest grade to meet the requirements of the mill schedule requirement and the balance was stockpiled recovered as required and processed at the end of the mine life. The mine production forecast is summarized in Table 16-2 and shown graphically in Figures 16-1 and 16-2.

Table 16-2: Summary Production Schedule

Summary Production Schedule		Year -1	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	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	Year 29	Year 30	Total
PHASE I OPEN PIT																																	
Potential Mill Feed	kt	9,538	37,203	39,312	22,720	13,144	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**121,917**
Waste	kt	21,312	22,447	4,740	1,870	1,547	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**51,916**
Total	kt	30,850	59,650	44,052	24,590	14,691	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**173,833**
PHASE II OPEN PIT																																	
Potential Mill Feed	kt	-	-	1,238	11,458	21,104	18,782	20,947	17,729	3,701	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**94,959**
Waste	kt	-	-	14,210	23,952	11,522	3,114	1,161	1,950	1,275	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**57,184**
Total	kt	-	-	15,448	35,410	32,626	21,896	22,108	19,679	4,976	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**152,143**
PHASE III OPEN PIT																																	
Potential Mill Feed	kt	-	-	-	-	852	7,567	9,655	14,276	24,743	23,109	16,390	22,568	18,484	18,705	9,464	3,749	3,502	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**173,064**
Waste	kt	-	-	-	-	11,831	30,537	28,237	26,045	19,985	11,043	5,784	7,655	4,476	3,017	1,063	659	615	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**150,947**
Total	kt	-	-	-	-	12,683	38,104	37,892	40,321	44,728	34,152	22,174	30,223	22,960	21,722	10,527	4,408	4,117	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**324,011**
PHASE IV OPEN PIT																																	
Potential Mill Feed	kt	-	-	-	-	-	-	-	-	163	5,544	17,262	6,868	12,981	11,157	15,838	33,475	17,661	21,923	19,082	15,877	8,895	5,679	-	-	-	-	-	-	-	-	-	**192,405**
Waste	kt	-	-	-	-	-	-	-	-	10,133	20,304	20,564	5,225	9,269	8,691	11,244	11,398	3,868	3,657	1,737	1,459	1,473	1,263	-	-	-	-	-	-	-	-	-	**110,285**
Total	kt	-	-	-	-	-	-	-	-	10,296	25,848	37,826	12,093	22,250	19,848	27,082	44,873	21,529	25,580	20,819	17,336	10,368	6,942	-	-	-	-	-	-	-	-	-	**302,690**
PHASE V OPEN PIT																																	
Potential Mill Feed	kt	-	-	-	-	-	-	-	-	-	-	-	115	339	1,118	2,752	1,780	5,361	2,984	5,426	9,167	16,384	21,243	26,852	26,103	16,197	-	-	-	-	-	-	**135,821**
Waste	kt	-	-	-	-	-	-	-	-	-	-	-	17,569	14,451	17,312	19,639	8,939	23,993	11,436	13,755	13,497	9,248	6,815	8,148	3,897	2,656	-	-	-	-	-	-	**171,355**
Total	kt	-	-	-	-	-	-	-	-	-	-	-	17,684	14,790	18,430	22,391	10,719	29,354	14,420	19,181	22,664	25,632	28,058	35,000	30,000	18,853	-	-	-	-	-	-	**307,176**
TOTAL OPEN PIT																																	
Potential Mill Feed	kt	9,538	37,203	40,550	34,178	35,100	26,349	30,602	32,005	28,607	28,653	33,652	29,551	31,804	30,980	28,054	39,004	26,524	24,907	24,508	25,044	25,279	26,922	26,852	26,103	16,197	-	-	-	-	-	-	**718,166**
Waste	kt	21,312	22,447	18,950	25,822	24,900	33,651	29,398	27,995	31,393	31,347	26,348	30,449	28,196	29,020	31,946	20,996	28,476	15,093	15,492	14,956	10,721	8,078	8,148	3,897	2,656	-	-	-	-	-	-	**541,687**
Total	kt	30,850	59,650	59,500	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	55,000	40,000	40,000	40,000	36,000	35,000	35,000	30,000	18,853	-	-	-	-	-	-	**1,259,853**
TOTAL STOCKPILE RECOVERY																																	
TOTAL	tonnes	**-**	**-**	**-**	**5,663**	**-**	**5,765**	**-**	**-**	**1,329**	**-**	**-**	**218**	**-**	**-**	**807**	**-**	**-**	**643**	**1,042**	**506**	**270**	**-**	**-**	**-**	**10,418**	**26,615**	**26,615**	**26,615**	**24,625**	**-**	**-**	**131,131**
OPEN PIT MILL FEED - Including stockpile recovery			90%																														
PHASE 1	kt	-	22,995	25,346	21,033	11,200	3,744	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	10,986	24,625	-	-	**119,929**
Cu	%	-	0.341	0.349	0.336	0.274	0.224	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.201	0.173	-	-	**0.285**
Au	g/t	-	0.040	0.042	0.039	0.029	0.025	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.023	0.021	-	-	**0.033**
Ag	g/t	-	1.157	1.317	1.390	1.267	1.079	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.979	0.916	-	-	**1.174**
PHASE 2	kt	-	-	204	4,517	13,781	16,914	18,621	14,732	4,508	-	-	-	-	-	807	-	-	-	-	-	-	-	-	-	-	-	5,245	15,629	-	-	-	**94,958**
Cu	%	-	-	0.274	0.266	0.268	0.258	0.294	0.309	0.245	-	-	-	-	-	0.184	-	-	-	-	-	-	-	-	-	-	-	0.173	0.157	-	-	-	**0.252**
Au	g/t	-	-	0.023	0.028	0.033	0.032	0.032	0.030	0.024	-	-	-	-	-	0.022	-	-	-	-	-	-	-	-	-	-	-	0.021	0.020	-	-	-	**0.029**
Ag	%	-	-	1.492	1.279	1.149	1.042	1.122	1.203	1.052	-	-	-	-	-	0.917	-	-	-	-	-	-	-	-	-	-	-	0.879	0.825	-	-	-	**1.065**
PHASE 3	kt	-	-	-	-	569	4,893	6,929	10,818	20,887	20,553	12,281	19,166	15,393	16,334	8,961	2,968	2,547	643	1,042	506	270	-	-	-	10,418	17,888	-	-	-	-	-	**173,064**
Cu	%	-	-	-	-	0.303	0.354	0.331	0.297	0.263	0.260	0.268	0.267	0.298	0.320	0.301	0.312	0.296	0.186	0.186	0.175	0.165	-	-	-	0.163	0.144	-	-	-	-	-	**0.263**
Au	g/t	-	-	-	-	0.036	0.039	0.043	0.041	0.033	0.032	0.032	0.030	0.032	0.032	0.031	0.031	0.024	0.021	0.021	0.021	0.020	-	-	-	0.020	0.019	-	-	-	-	-	**0.031**
Ag	%	-	-	-	-	0.813	1.163	1.631	1.510	1.332	1.298	1.306	1.283	1.387	1.508	1.430	1.463	1.313	1.117	1.117	1.058	1.000	-	-	-	0.988	0.897	-	-	-	-	-	**1.297**
PHASE 4	kt	-	-	-	-	-	-	-	-	155	4,997	13,269	6,299	9,877	8,342	13,645	21,657	17,642	21,923	19,082	15,877	8,895	5,679	-	-	-	8,727	16,340	-	-	-	-	**192,406**
Cu	%	-	-	-	-	-	-	-	-	0.185	0.246	0.260	0.236	0.245	0.247	0.234	0.251	0.223	0.233	0.245	0.256	0.246	0.240	-	-	-	0.166	0.148	-	-	-	-	**0.231**
Au	g/t	-	-	-	-	-	-	-	-	0.015	0.021	0.023	0.022	0.023	0.023	0.023	0.024	0.022	0.023	0.024	0.026	0.026	0.026	-	-	-	0.021	0.019	-	-	-	-	**0.023**
Ag	%	-	-	-	-	-	-	-	-	1.010	0.988	1.141	1.133	1.159	1.180	1.184	1.290	1.241	1.288	1.274	1.199	1.164	1.096	-	-	-	1.047	0.977	-	-	-	-	**1.185**
PHASE 5	kt	-	-	-	-	-	-	-	-	-	-	-	86	281	874	2,137	925	5,361	2,984	5,426	9,167	16,384	19,871	25,550	25,550	16,197	-	5,030	-	-	-	-	**135,823**
Cu	%	-	-	-	-	-	-	-	-	-	-	-	0.284	0.266	0.307	0.281	0.324	0.255	0.262	0.247	0.244	0.261	0.278	0.272	0.295	0.288	-	0.148	-	-	-	-	**0.270**
Au	%	-	-	-	-	-	-	-	-	-	-	-	0.027	0.029	0.034	0.033	0.041	0.031	0.032	0.028	0.027	0.030	0.031	0.030	0.032	0.028	-	0.018	-	-	-	-	**0.030**
Ag	%	-	-	-	-	-	-	-	-	-	-	-	0.439	0.378	0.423	0.621	0.790	0.905	0.961	0.954	0.953	0.983	1.041	1.119	1.320	1.304	-	0.844	-	-	-	-	**1.095**
TOTAL FEED	kt	-	22,995	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	26,615	26,615	26,615	26,615	24,625	-	-	**716,180**
Cu	%	-	0.341	0.348	0.324	0.271	0.271	0.304	0.304	0.259	0.257	0.264	0.260	0.277	0.296	0.260	0.261	0.237	0.235	0.243	0.250	0.255	0.269	0.272	0.295	0.239	0.151	0.153	0.175	0.173	-	-	**0.258**
Au	g/t	-	0.040	0.042	0.037	0.031	0.032	0.035	0.035	0.031	0.029	0.027	0.028	0.028	0.029	0.027	0.026	0.024	0.024	0.025	0.026	0.028	0.030	0.030	0.032	0.025	0.020	0.019	0.021	0.021	-	-	**0.029**
Ag	g/t	-	1.157	1.318	1.371	1.193	1.071	1.260	1.333	1.281	1.237	1.220	1.243	1.288	1.364	1.215	1.292	1.178	1.245	1.200	1.108	1.046	1.054	1.119	1.320	1.180	0.946	0.933	0.889	0.916	-	-	**1.177**
Copper Recovery	%	0.0%	90.4%	90.7%	89.9%	88.5%	88.5%	89.4%	89.4%	88.2%	88.1%	88.3%	88.2%	88.7%	89.2%	88.2%	88.2%	87.5%	87.5%	87.7%	87.9%	88.1%	88.4%	88.5%	89.1%	87.6%	82.3%	82.5%	84.6%	84.4%	0.0%	0.0%	
Gold Recovery	%	0.0%	66.6%	67.0%	65.6%	62.9%	63.4%	64.9%	64.7%	62.9%	62.1%	60.6%	61.1%	61.4%	62.0%	60.2%	59.4%	58.4%	58.1%	58.9%	60.1%	61.4%	62.3%	62.3%	63.3%	59.0%	53.9%	53.0%	55.3%	55.1%	0.0%	0.0%	
Silver Recovery	%	0.0%	40.4%	44.7%	45.8%	41.5%	37.6%	43.3%	45.0%	43.8%	42.7%	42.2%	42.8%	43.9%	45.7%	42.1%	44.0%	41.0%	42.9%	41.7%	38.9%	36.8%	37.0%	39.2%	44.7%	41.1%	32.7%	32.1%	30.0%	31.3%	0.0%	0.0%	
Recoverable Copper	lbs x 1000	-	156,166.0	177,881.1	164,019.3	135,336.8	135,335.4	153,088.7	152,982.8	128,776.7	127,682.0	131,373.7	128,920.3	138,465.6	148,695.6	128,922.5	129,611.1	116,684.2	115,830.0	119,943.9	123,727.0	126,542.7	134,174.8	135,859.0	147,932.3	122,877.9	73,010.9	74,009.1	86,918.6	79,446.2	-	-	**3,594,214.4**
Recoverable Gold	ounces	-	19,880	22,870	20,129	16,074	16,749	18,849	18,603	16,022	15,037	13,538	14,032	14,293	15,008	13,135	12,458	11,631	11,453	12,035	13,044	14,302	15,306	15,354	16,599	12,606	9,205	8,730	9,984	9,112	-	-	**406,036**
Recoverable Silver	ounces	-	345,820	483,688	515,817	406,552	331,164	447,757	492,787	460,424	433,782	423,221	437,398	465,056	511,845	419,925	467,212	397,117	438,538	410,494	354,167	315,820	320,487	360,621	484,605	415,295	264,825	256,286	228,197	227,420	-	-	**11,116,321**
Cumulative Waste	kt	21,312	43,759	62,709	88,531	113,431	147,082	176,480	204,475	235,868	267,215	293,563	324,012	352,208	381,228	413,174	434,170	462,646	477,739	493,231	508,187	518,908	526,986	535,134	539,031	541,687	541,687	541,687	541,687	541,687	541,687	541,687	
Stockpile Addition	kt	9,538	14,208	15,000	8,628	9,550	799	5,052	6,455	3,057	3,103	8,102	4,001	6,254	5,430	2,504	13,454	974	-	-	-	-	1,372	1,302	553	-	-	-	-	-	-	-	
Stockpile Balance	kt	9,538	23,746	38,746	47,374	56,924	57,723	62,775	69,230	72,287	75,390	83,492	87,493	93,747	99,177	101,681	115,135	116,109	115,466	114,424	113,918	113,647	115,019	116,321	116,874	106,456	79,841	53,226	26,611	1,986	1,986	1,986	
Overburden	kt	5,567	5,306	3,743	726	3,076	1,872	2,024	2,707	3,145	1,505	969	2,575	1,260	1,197	1,352	856	952	50	-	-	-	-	-	-	-	-	-	-	-	-	-	38,882
Unknown	kt	(1)	0	5	0	(2)	1	2	(0)	1	1	2	-	0	1	0	1	0	-	1	(1)	-	(1)	-	(1)	-	-	-	-	-	-	-	10
PAG Waste	kt	6,401	7,712	4,885	10,013	8,386	15,777	19,098	18,041	15,612	11,088	9,803	11,510	10,879	10,600	11,686	9,964	12,644	8,284	8,438	7,262	3,290	3,259	6,298	3,774	2,646	-	-	-	-	-	-	237,349
NAG Waste	kt	9,346	9,428	10,316	15,083	13,441	16,000	8,273	7,248	12,634	18,753	15,573	16,364	16,056	17,222	18,908	10,175	14,881	6,759	7,053	7,695	7,430	4,820	1,850	123	10	-	-	-	-	-	-	265,442

Figure 16-1 Material Movement Schedule

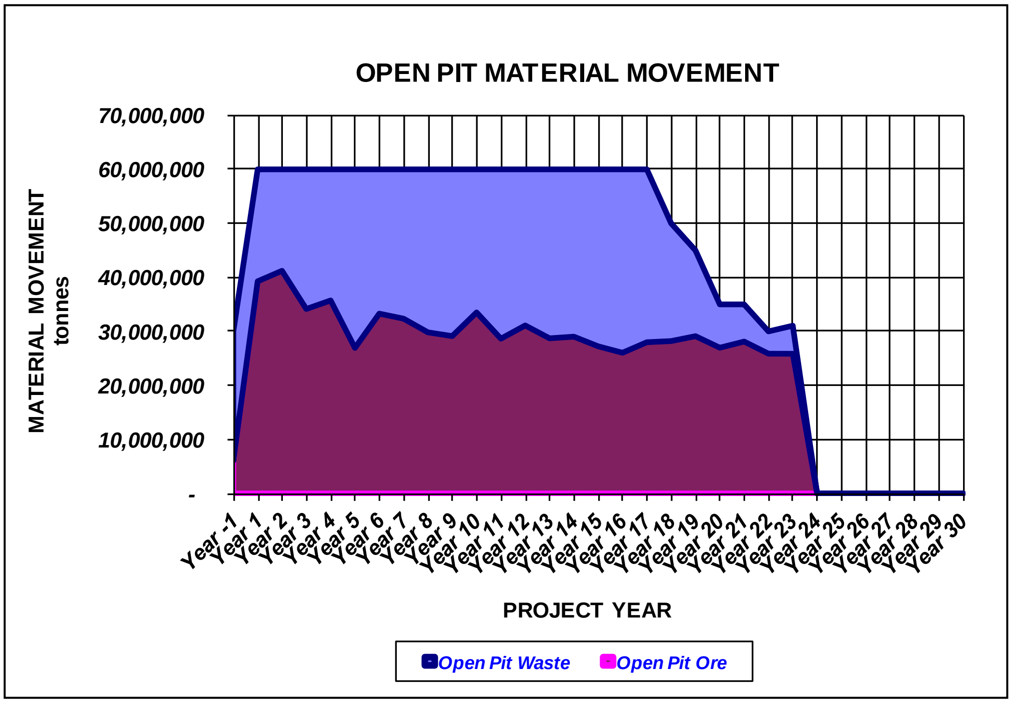


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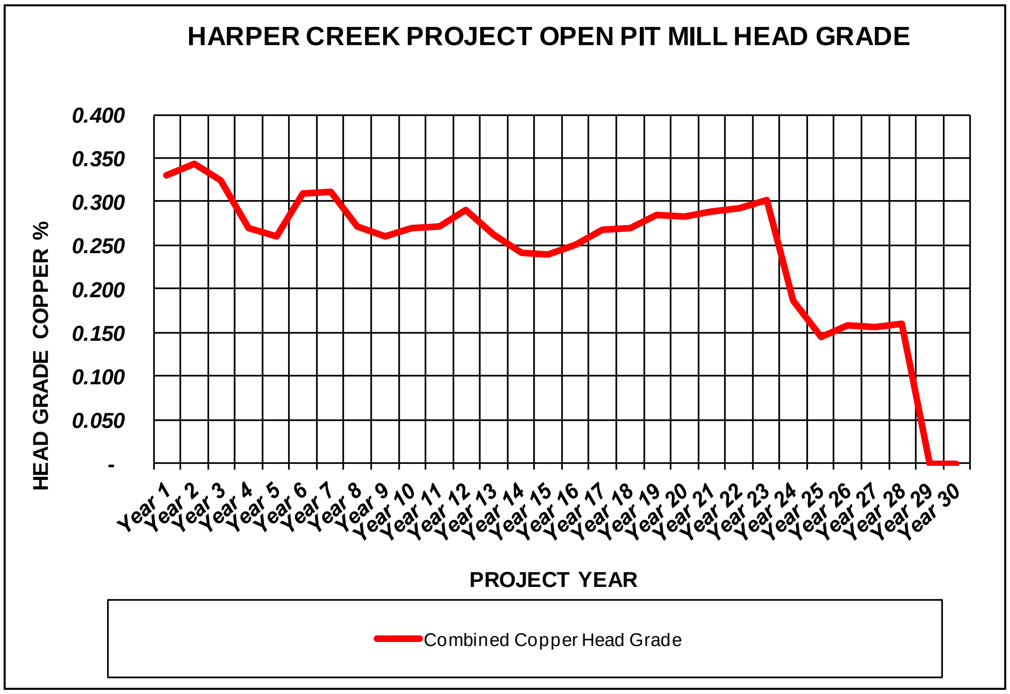
Figure 16-2 Head Grades



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16.1.4 *WASTE STORAGE AND STOCKPILE PLAN*

Total waste to be mined from the open pit is 541.7Mt including 38.9Mt of overburden. Some overburden will be used for road and dam construction and the balance will be placed in stockpiles located to the east of the open pit. There will be a total of 265.4Mt of non-PAG waste. Some non-PAG waste will be used for dam and road construction and for creating bases below low grade stockpiles. The balance will be placed in a stockpile located west of the open pit. PAG and unclassified waste total 237.4Mt. This material will be placed within the tailings management facility.

16.1.5 *MINING EQUIPMENT*

The mine will operate one diesel rotary dill, two electric rotary drills, three electric hydraulic shovels, one wheel loader, up to twenty eight haulage trucks, and a fleet of support equipment to maintain roads, and stockpiles. The equipment fleet will incorporate large scale units which have been well proven in existing operations.

16.2 OPEN PIT DESIGN

16.2.1 *GENERAL*

The open pit design is based upon the following key considerations:

- Overall and inter-ramp slope recommendations provided by Knight Piésold Ltd.;
- Waste stockpiles overall final slopes of 2H:1V;
- Operating constraints of the equipment selected for mining:
- Minimum mining width defined by double side loading of trucks with allowance for an access ramp;
- Bench height achievable and within the safe operating reach of the primary loading units;
- Minimum haulage road operating width and maximum effective grade within the operating limitations of the primary haulage units;
- Logical and efficient scheduling of material movement from multiple phases of pit expansion to the crusher, the stockpiles and to final waste material placement sites; and
- Minimum footprint for disturbance of the surrounding area.

16.2.2 *WALL SLOPE DESIGN SECTORS*

16.2.2.1 Site Investigation

KP undertook a geotechnical investigation program that provided the basis for open pit slope design recommendations for the Feasibility Study. Details of the open pit slope design for the study are contained in the report by Knight Piésold Ltd. “2012 Open Pit Geotechnical Design” (Ref. No. VA101-458/4-5). A summary of the recommendations is provided in the following sections.

KP completed a Site Investigation (SI) program from June to October of 2011, to collect geomechanical and hydrogeological information to support a FS pit slope design based on the pit shell model provided by YMI. The geomechanical portion of the program consisted of seven oriented core holes drilled within the open pit area.

Detailed geomechanical logging, Point Load Tests (PLT), and in situ packer hydrogeological testing were completed, and standpipe piezometers were installed in each hole. The locations of the drillholes are illustrated in Figure 16-3.

Representative rock samples were collected for laboratory testing. A testing schedule was developed to determine the Unconfined Compressive Strength (UCS) and shear strength of the rock mass.

16.2.2.2 Geotechnical Characterization

The Deposit is an extensive volcanogenic sulphide system hosted in the metamorphic rocks of the Eagle Bay Assemblage, specifically within the Lower Paleozoic and Greenstone Belts. Multiple lithological units were defined within the open pit by Project geologists, including phyllite, schist, quartz eye schist, silica altered host rock and zones of faulting.

The individual lithological units are complex due to multiphase deformation and alteration sequences, and correlating lithologies across drill holes and sections is difficult and may produce an unreliable geological model. Regrouping the individual lithologies into packages of common characteristics allowed for an easier understanding of the geology and ability to correlate between drill holes across the Deposit area. A series of geological rock packages were defined by Project geologists to allow for greater confidence in geological interpretation between drill holes and geological sections. The geological packages, as defined by Project geologists that were logged in the geomechanical drillholes are as follows:

- **H –** Mafic Polymictic Volcaniclastics, frequently calcareous.
- 'FAIR' quality rock with a mean RMR value of 60.
- Average RQD value of 61%.
- 'AVERAGE' rock strength classification with UCS laboratory test results ranging from 29MPa to 60MPa with a mean of 44MPa for all failures. It should be noted that all lab samples tested failed along pre-existing foliation planes, which are ubiquitous throughout the open pit rock mass, and that the intact rock strength may be higher.
- **Fa –** Felsic to Intermediate Volcaniclastics.
- 'GOOD' quality rock with a mean RMR value of 64.
- Average RQD value of 84%.
- 'AVERAGE' rock strength classification with UCS laboratory test results ranging from 1MPa to 100MPa with a mean of 29MPa for failures along foliation planes and 68MPa for intact rock failures. The low-end values typically represent breaks along foliation, as opposed to through intact rock.
- **Fb -** Intermediate to Mafic Polymictic Volcaniclastics.
- 'GOOD' quality rock with a mean RMR value of 63.
- Average RQD value of 76%.
- 'SOFT' rock strength classification with UCS laboratory test results ranging from 2MPa to 42MPa with a mean of 20MPa for failures along foliation planes and 40MPa for intact rock failures. The low-end values typically represent breaks along foliation, as opposed to through intact rock.
- **E –** Graphitic Horizon, typically comprised of Silica Altered Host and graphitic Phyllite.
- 'GOOD' quality rock with a mean RMR value of 68.
- Average RQD value of 77%.

- 'AVERAGE' rock strength classification with UCS laboratory test results ranging from 10MPa to 75MPa with a mean of 33MPa for failures along foliation planes and 38MPa for intact rock failures. The low-end values typically represent breaks along foliation, as opposed to through intact rock.
- **D –** Intermediate Volcaniclastics & Fragmentals, primarily calcareous Phyllite and fragmental Schist.
- 'GOOD' quality rock with a mean RMR value of 64.
- Average RQD value of 80%.
- 'SOFT' rock strength classification with UCS laboratory test results ranging from 13MPa to 32MPa with a mean of 20MPa for failures along foliation planes. The low-end values typically represent breaks along foliation, as opposed to through intact rock.
- **B –** Sandy Sediment dominated, comprised of calcareous Phyllite, Quartz Eye Schist, and mafic sediments.
- 'GOOD' quality rock with a mean RMR value of 68.
- Average RQD value of 74%.
- 'HARD' rock strength classification with UCS laboratory test results ranging from 1MPa to 175MPa with a mean of 47MPa for failures along foliation planes and 101MPa for intact rock failures. The low-end values typically represent breaks along foliation, as opposed to through intact rock.
- **FD –** Combines elements of D and Fa packages as found on the east side of the Harper Creek Fault.
- 'GOOD' quality rock with a mean RMR value of 64.
- Average RQD value of 81%.
- 'HARD' rock strength classification with UCS laboratory test results ranging from 7MPa to 110MPa with a mean of 28MPa for failures along foliation planes and 97MPa for intact rock failures. The low-end values typically represent breaks along foliation, as opposed to through intact rock.

The intact rock Unconfined Compressive Strength (UCS) average values calculated from rock strength laboratory testing data. Rock mass quality was characterized using several well-accepted geomechanical indices, including Rock Quality Designation (RQD), and Rock Mass Rating (RMR, Bieniawski, 1989). Rock mass quality is typically GOOD, with certain areas exhibiting zones of FAIR quality rock near the surface. There is no significant trend demonstrating a variation of rock quality with lithology (excepting Fault Zone rock). The mean values for RMR and RQD were calculated to a single standard deviation for each geotechnical unit.

Geological packages were only defined within or near the open pit area. Creating a geotechnical model utilizing the numerous geological packages identified by the Project geologists is impractical due to the relative similarities between units, the lack of information to delineate packages throughout the pit area, and because such a model would be too complex to be practical. The geomechanical data collected from the seven oriented core holes logged by KPL was used to define rock mass characteristics. Simplified geotechnical domains were delineated by incorporating the Project geologist's interpretation for slope stability analyses.

A mineralization model was provided by GeoSim Services Inc. (GeoSim) in October 2011, which illustrates three units: Phyllites, located at the northwest corner of the pit, the West Mineralized Zone, and the East Mineralized Zone. These mineralized zones are divided by the Harper Creek Fault, which bisects the pit area along a northeast-southwest strike. The distribution of the mineralized zone was correlated with the distribution of geological packages with similar rock mass strength and quality parameters to delineate the geotechnical domains used in the stability analyses. It was assumed that the alteration assemblages used to define the Phyllite and Western and Eastern Mineralized Zones are related to the alteration phases that characterize the geological packages. Based on this correlation, three geotechnical domains were defined for the purposes of the rock mass stability analysis, defined as follows:

- **Phyllite –** The Phyllite unit is geological package H, and is located in the northwest corner of the pit.
- 'GOOD' quality rock with a mean RMR value of 63.
- Average RQD value of 80%.
- 'AVERAGE' rock strength classification with UCS laboratory test results ranging from 29MPa to 60MPa with a mean of 44MPa.
- **West Volcaniclastics –** Comprised of geological packages Fa, Fb, E and D. These units are located along the west side of the Harper Creek Fault.
- 'GOOD' quality rock with a mean RMR value of 63.
- Average RQD value of 76%.
- 'HARD' rock strength classification with UCS laboratory test results ranging from 1MPa to 100MPa with a mean of 26MPa for failures along foliation planes and 65MPa for intact rock failures. The low-end values typically represent breaks along foliation, as opposed to through intact rock.
- **East Volcaniclastics –** Comprised of geological packages FD and B. These units are located on the east side of the Harper Creek Fault. The rock mass strength of these units are relatively higher compared to the Western Volcaniclastics, due to the silicification of the rock mass in that area, as confirmed by Project geologists.
- 'GOOD' quality rock with a mean RMR value of 64.
- Average RQD value of 81%.
- 'HARD' rock strength classification with UCS laboratory test results ranging from 1MPa to 175MPa with a mean of 45MPa for failures along foliation planes and 98MPa for intact rock failures. The low-end values typically represent breaks along foliation, as opposed to through intact rock.

The distribution of the geotechnical domains within the pit area is shown in Figure 16-3.

The orientation and characteristics of the small scale structural features, including joints and schistosity, were determined from seven oriented drill holes completed during the 2011 Site Investigation program. The preliminary structural data review utilizes the available oriented data and the inferred orientation of the large scale Harper Creek Fault which bisects the open pit area. The oriented core logging data was presented in stereographic plots to determine structural trends within the open pit area. There is a dominant structural trend, influence by the schistosity and bedding of the metamorphic rocks within the open pit, dipping at 25-35° towards the north.

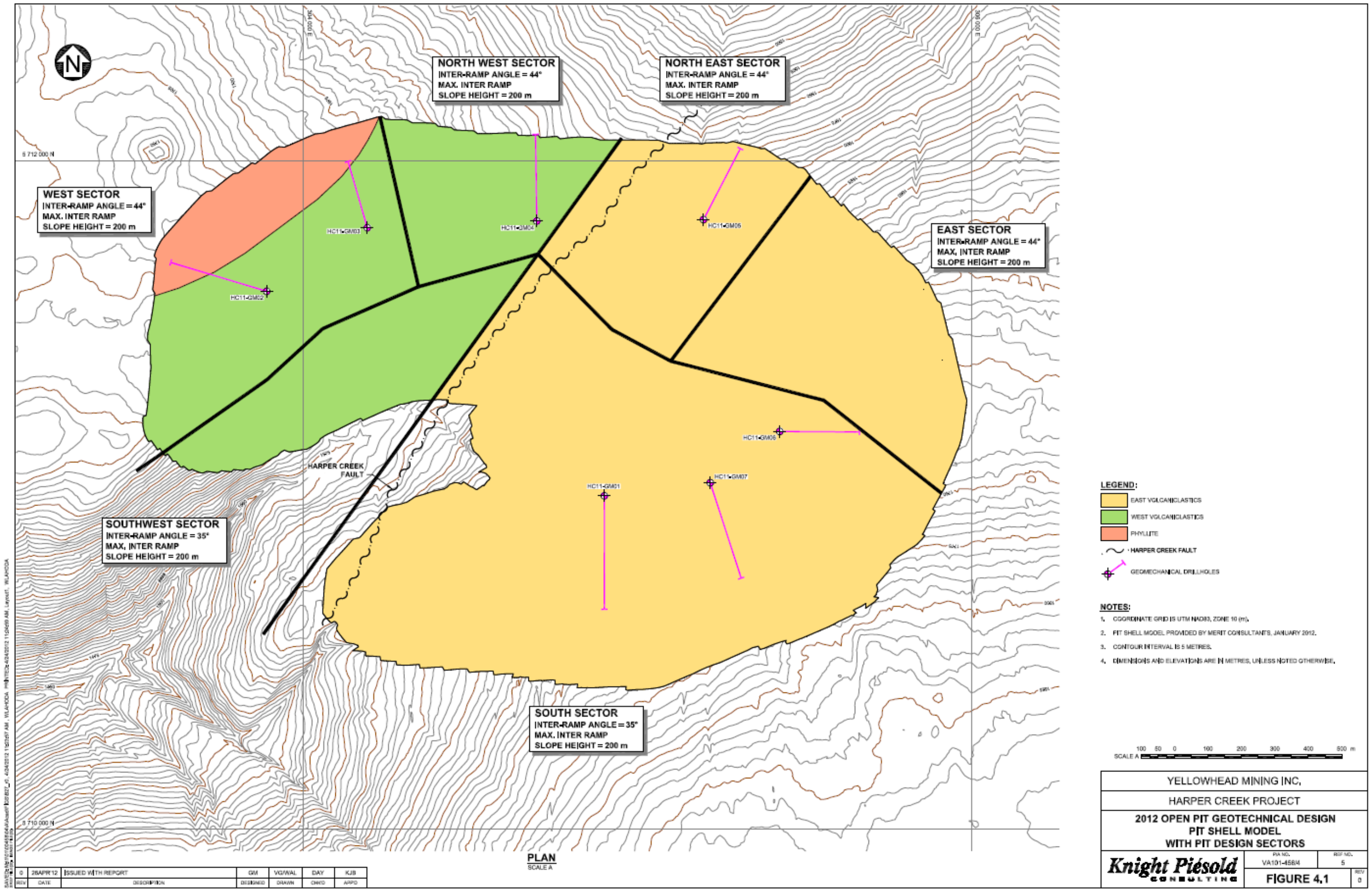
Groundwater levels vary throughout the open pit, from artesian conditions observed within the south, northeast and east regions of the pit to 12m deep in the northwest. The hydraulic conductivity of the rock varies from $1x10^{-9}$ to $5x10^{-6}$m/sec. There is no significant correlation between hydraulic conductivity and lithology or geological package; however conductivity generally decreases with depth. The average hydraulic conductivity of bedrock is in the order of $4x10^{-8}$m/sec. This value is based on the geometric mean of the permeability values measured during the 2011 Geotechnical Site Investigation. There is no significant variation in hydraulic conductivity by depth or rock type, so a single value has been deemed appropriate for all geotechnical domains in the pit.

16.2.2.3 Pit Design Sectors

Six design sectors were delineated for stability analysis based on pit wall orientations and rock structures. The design sectors as shown on Figure 16-3 are described as follows:

- Northeast Sector
    - The north hanging wall dips at an azimuth of 180°.
    - Comprised of West Volcaniclastics geotechnical domain.
    - Pit walls approximately 270m high.
    - Characterized by drillholes HC11-GM03 to GM05.
- East Sector
    - Dips towards the west at a nominal pit wall dip direction of 270°.
    - Comprised of East Volcaniclastic geotechnical domain.
    - Pit walls approximately 375m high.
    - Characterized by drill hole HC11-GM06.
- South Sector
    - This sector contains the northward dipping south foot wall of the pit. The slope angle of the foot wall is influenced by the orientation of the foliation.
    - Comprised of East Volcaniclastic geotechnical domain.
    - Pit walls in this sector are approximately 445m high.
    - Characterized by drillholes HC11-GM01 and HC11-GM07.
- Southwest Sector
    - This sector contains the northward dipping south foot walls of the western arm of the pit. The slope angle of this foot wall is influenced by the orientation of the foliation.
    - Comprised of West Volcaniclastics geotechnical domain.
    - Pit walls are approximately 210m high.
    - Characterized by drill hole HC11-GM02.
- West Sector
    - Comprised of southeast and northeast dipping walls of the western area of the pit.
    - Both Phyllite and West Volcaniclastic geotechnical domains are present in this sector.
    - Pit walls approximately 210m high.
    - Characterized by drill hole HC11-GM02.
- Northwest Sector
    - Continuation of the north hanging wall which dips at an azimuth of 180°, rotates towards a 220° dip direction at the eastern end of the sector.
    - Comprised of East Volcaniclastic geotechnical domain.
    - Pit walls approximately 270m to 300m high.
    - Characterized by drillholes HC11-GM03 to GM05.

Figure 16-3 Pit Shell Model 2014



16.2.2.4 Pit Slope Configuration

The primary components for open pit slope design include bench geometry, inter-ramp slope angle, and overall slope angles. Preliminarily recommended pit slope configurations are detailed below:

Bench Geometry

The bench design was developed based on bench geometry specifications provided by YMI and adjusted based the geology, geomechanical and geometrical characteristics of each main design sector. The bench face angles derived from the kinematic analyses are as steep as reasonably can be expected given the characteristics of the rock masses and mine requirements. As such, the potential for planar or wedge failures still exists within some design sectors, but the majority of these are expected to be manifested as small bench-scale ravelling type failures that will be removed during initial excavation or controlled through a normal bench maintenance program.

Optimum bench configurations will be determined during initial open pit development. However, recommendations for bench design will be provided for design and costing. Bench face angles of 60° are required in all northward dipping slopes within the South and Southwest Sectors to allow the requisite inter-ramp angle of 35° to be achieved. A 70° bench face angle is considered appropriate in all other sectors of the pit. Recommended bench geometries are summarized in Table 16-3.

Inter-Ramp Slope Angles

The inter-ramp slope angle is typically dictated by the bench geometry and controlled by large-scale structural features. It is assumed that a 12m high single bench configuration will be used for pit development. The recommended inter-ramp slope angles for each of the design sectors are summarized in Table 16-3.

The critical wall of the open pit is the foot wall within the South Sector of the pit. This wall is oriented parallel to the foliation and primary jointing of the rock mass. Therefore, it is recommended that the inter-ramp angle of the South sector be 35° or less to mitigate the risk of multiple-bench planar failures. This is achievable by developing 12m high single benches, with a minimum width of 10m and a bench face angle of 60°. The Southwest Sector inter-ramp slopes should utilize the same geometry to reduce the risk of multiple-bench planar failures.

The remaining design sectors contain no significant kinematic controls, and slope angles are therefore primarily determined by bench geometry. The Northeast, East, West and Northwest sectors of the open pit will utilize 12m high single benches, 8m bench widths, and a maximum bench face angle of 70°, such that an inter-ramp angle of 44° is attainable.

The inter-ramp slopes of the design sectors shown in Table 16-3 were used for the Lerchs Grossman pit optimization and pit design.

Table 16-3 Slope Design Sectors

Design Sector	Bench Face Angle (degree °)	Bench Height (m)	Bench Width (m)	Inter-ramp Angle (degree°)
Northeast	70	12	8	44
East	70	12	8	44
South	60	12	10	35
Southwest	60	12	10	35
West	70	12	8	44
Northwest	70	12	8	44

16.3 General Design

16.3.1 *Design Summary*

The mining equipment will operate on a 12m high bench. Berms will be left on every bench. KP recommended 70° bench face angles on the north, east and west walls and 60° bench face angles on the south wall. Inter-ramp slopes were designed at 35° on all northward dipping walls and 44° on all other walls.

The open pit is designed for five phases of development. The mining will be by conventional truck and shovel methods. The overall life of mine strip ratio is relatively low at 0.76:1.00 waste to ore. However, the overall mining rate will be 60,000kt/a for most of the mine life. The mine has been designed for operations with hydraulic shovels in the 42m^3 range and trucks in the 227t class with typical support equipment associated with this type of primary mining equipment. The Phase 1 pit is approximately 1,600m long in the east-west direction and 700m wide in the north south direction with a depth of approximately 230m. Expansion pits are typically 120m to 200m wide. The ultimate pit is 2,400m long and 1,670m wide with a depth of approximately 375m.

16.3.2 *Material Types*

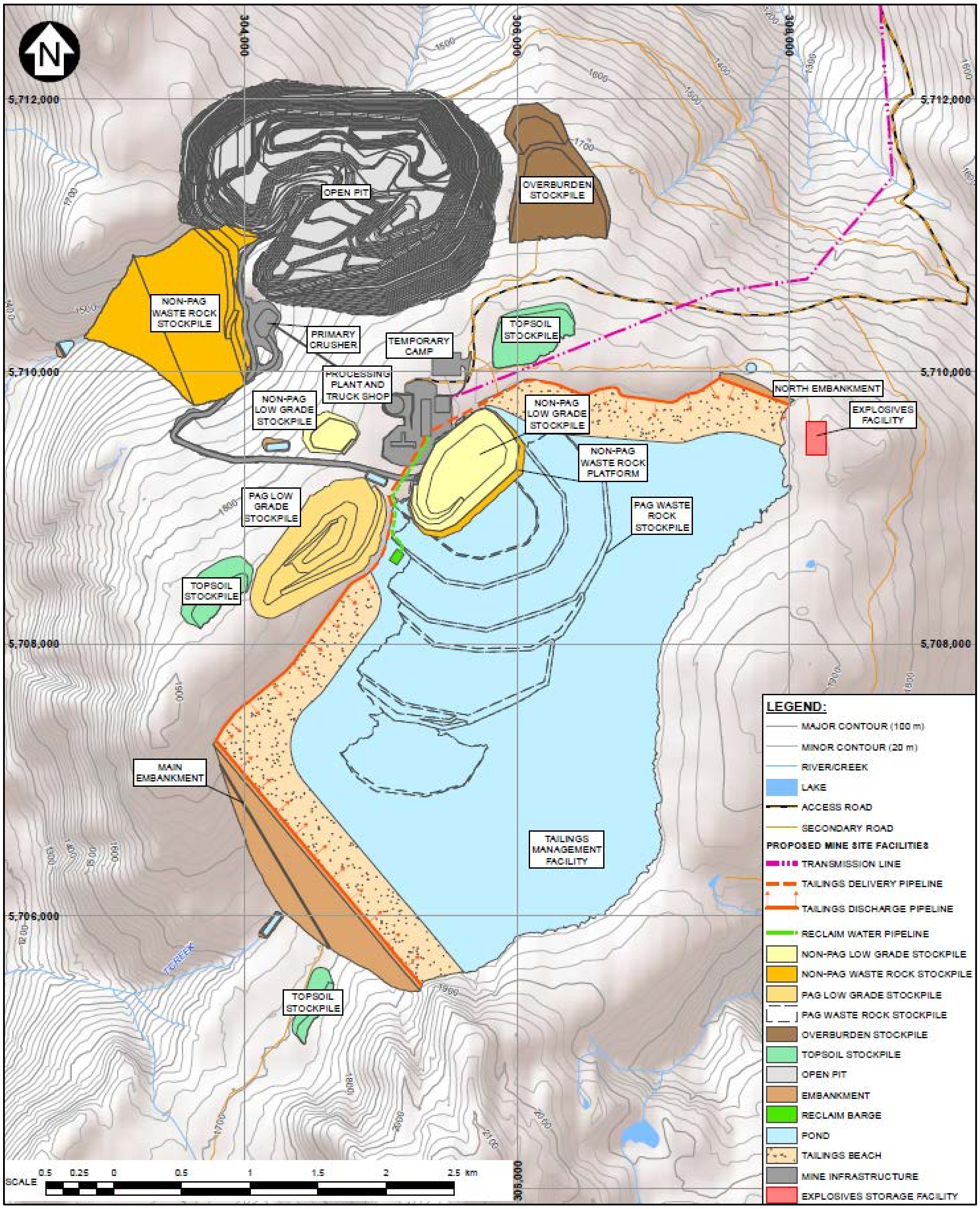
The material types included in the mine material movement schedule are subdivided into overburden, waste rock including non-PAG & PAG and ore. Ore has been scheduled according to six grade bins. These bins begin at 0.14% Cu and increase in 0.02% Cu increments. All material above 0.14% Cu is processed. There is a low grade stockpile plan that increases effective head grades during earlier years of the mine plan. PAG and non-PAG waste rock was classified on the basis of NPR. Material with an NPR greater than 2.00 was classified as non-PAG. Topsoil was estimated as a subset of overburden for the purposes of stockpiling.

16.3.3 *Waste and Low Grade Storage*

Waste, overburden and low grade stockpiles are located peripheral to the open pit and within the TMF as required to accommodate the scheduled quantity of each material type. Proposed dump and stockpile locations are shown in Figure 16-4. PAG waste will be placed in the upper TMF. Non-PAG waste will be placed in the Non-PAG waste rock stockpile, as well as used to construct the TMF embankments. Overburden will be placed in the overburden stockpile and used for road and embankment construction.

The dumps and stockpiles have been placed outside of a spoil line reflecting a larger pit limit including inferred resources at an elevated copper price. The Phase 1 pit is shown as a solid and the expansion pit limits are shown as traces in Figure 16-4.

Figure 16-4 Waste Dump and Stockpile Locations



Knight Piesold Consulting, May 2014

16.3.4 *Haulage Roads*

Allowances in the pit designs were made for 30m wide roads including ditches and berms. Roads will have a maximum gradient of 10%.

Surface haulage roads will connect the pit ramps to the crusher, low grade stockpile and waste dumps. When possible these roads will be constructed using non-PAG waste rock and overburden. As in the pit, roads will have a running surface three times the width of the largest haulage truck with allowance for ditches and berms. Roads will have a maximum grade of 10% but may be constructed to 8% to improve haulage cycle times and reduce truck component wear.

The main haulage roads to the crusher, plant site/TMF and a construction haulage road to the dam site will be built in pre-production. Other roads will be constructed during the normal course of mine operations.

16.4 Mine Production Schedule

16.4.1 *Summary*

The open pit mine development plan consists of five pit development phases expanding to a single large open pit. These five phases will be mined sequentially with overlap of up to three phases. A total of 30.8Mt will be mined in preproduction. A total of 9.5Mt of ore will be mined and stockpiled and 21.3Mt of waste will be mined and placed on dumps or used for construction. During the first 15 years of operations the total mining rate will be 60,000kt/a. During Years 16 through 25 the mining rates decline from 55,000kt/a to 30,000kt/a as final benches of Phase 5 are mined to completion. During Years 24 through 28 the low grade stockpiles are recovered and processed. The mill will process 716.2Mt of ore with an average grade of 0.26% Cu 0.03g/t Au and 1.2g/t Ag. The total effective waste mined will be 541.7Mt with an additional 2.0Mt of low grade remaining unrecovered in the stockpile base. The effective overall strip ratio will be 0.76:1.00 waste to ore.

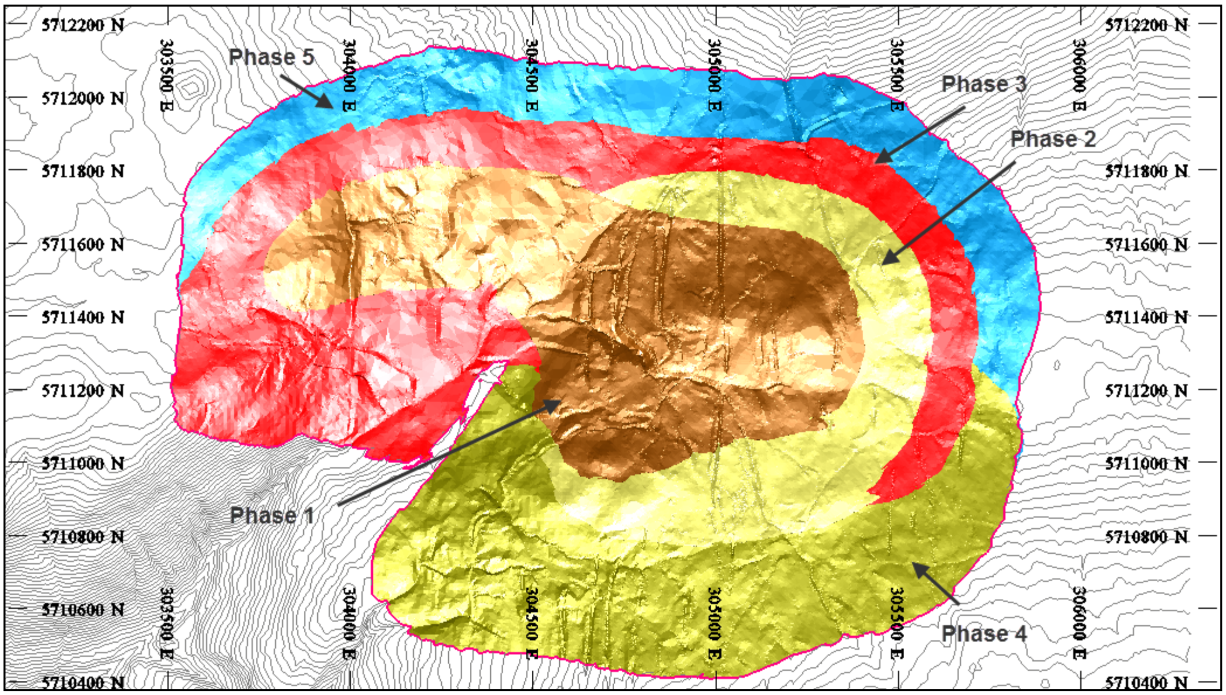
16.4.2 *Cutoff Grade Selection*

The total onsite operating costs for general and administration, surface operations, mill operating, tailings operating and tailings sustaining capital were estimated to be C$4.28/t milled. With an additional C$0.75/t for stockpile reclaim and a minimum profit of C$0.25/t a marginal stockpile cutoff grade would be C$5.28/t NSR This represents a recoverable copper grade of 0.115% at US$2.25/lb copper with co-product credits for gold and silver adding 10% to the average NSR. Recovery of copper at the estimated ore/waste cutoff grade level was estimated to be approximately 82%. The insitu copper cutoff grade was therefore estimated to be 0.14% Cu for the purposes of establishing a cutoff for ore and waste.

16.4.3 *PIT SEQUENCING*

The pit development phases are shown as three dimension solids in the plan view in Figure 16-5. The development sequence of the current schedule is sequential Phase 1 through Phase 5.

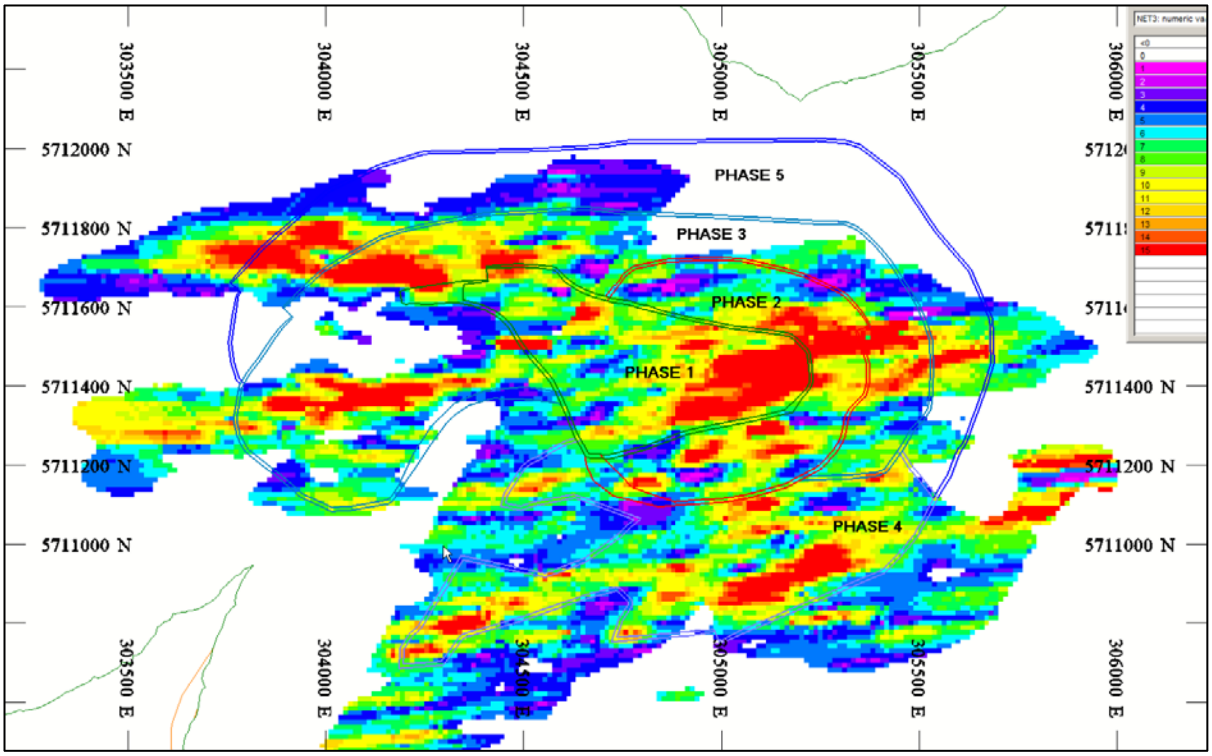
Figure 16-5 Rendered View of Pit Phase Solids



Nilsson Mining Services Inc., June 2014

A typical bench plan showing the development phase limits and the block model NSR values are shown in Figure 16-6.

Figure 16-6 Pit Phases and NSR Bench Plan 1540



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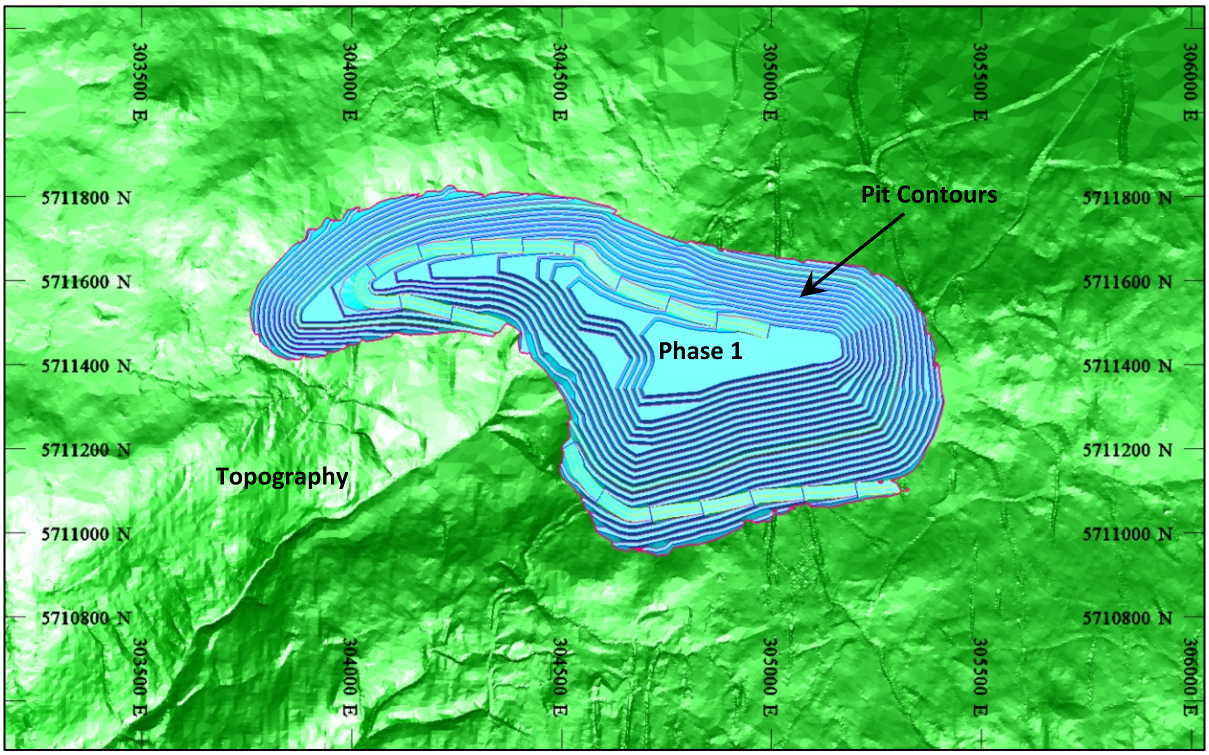
16.4.4 *MINE DEVELOPMENT AND ACCESS*

16.4.4.1 Phase 1 Open Pit

The Phase 1 pit is shown in Figure 16-7. Ramp access will be developed to depth from 1,612m elevation in a clockwise direction. A full width ramp will be left in the south wall design to access the upper benches for the Phase 2 expansion.

Figure 16-7 Rendered View of Phase 1 Pit Design

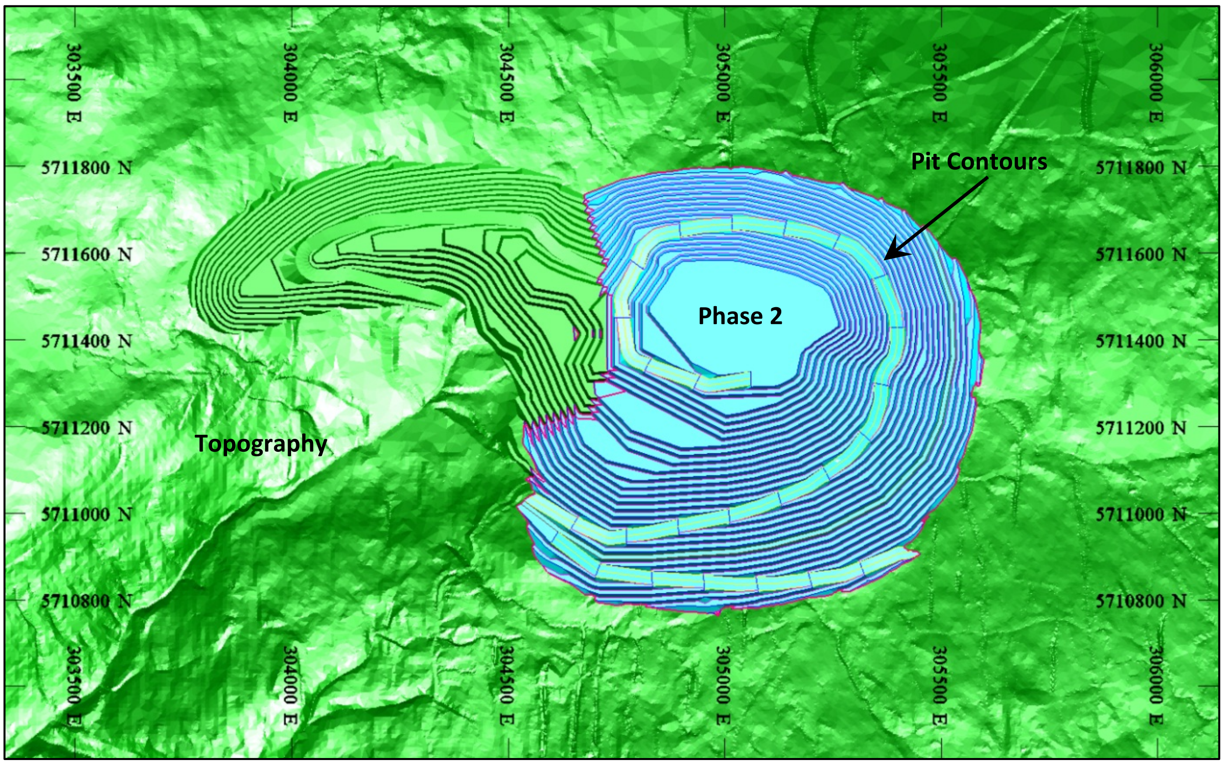


Nilsson Mining Services Inc., June 2014

16.4.4.2 Phase 2 Open Pit

The Phase 2 pit (Figure 16-8) is an easterly expansion of the Phase 1 pit that also expands the mine to depth. Ramp access will be developed from the south side 1,648m elevation in a counter clockwise direction. A full width ramp will be left on the south side to the upper elevation Phase 4 expansion pit.

Figure 16-8 Rendered View Phase 2 Pit Design

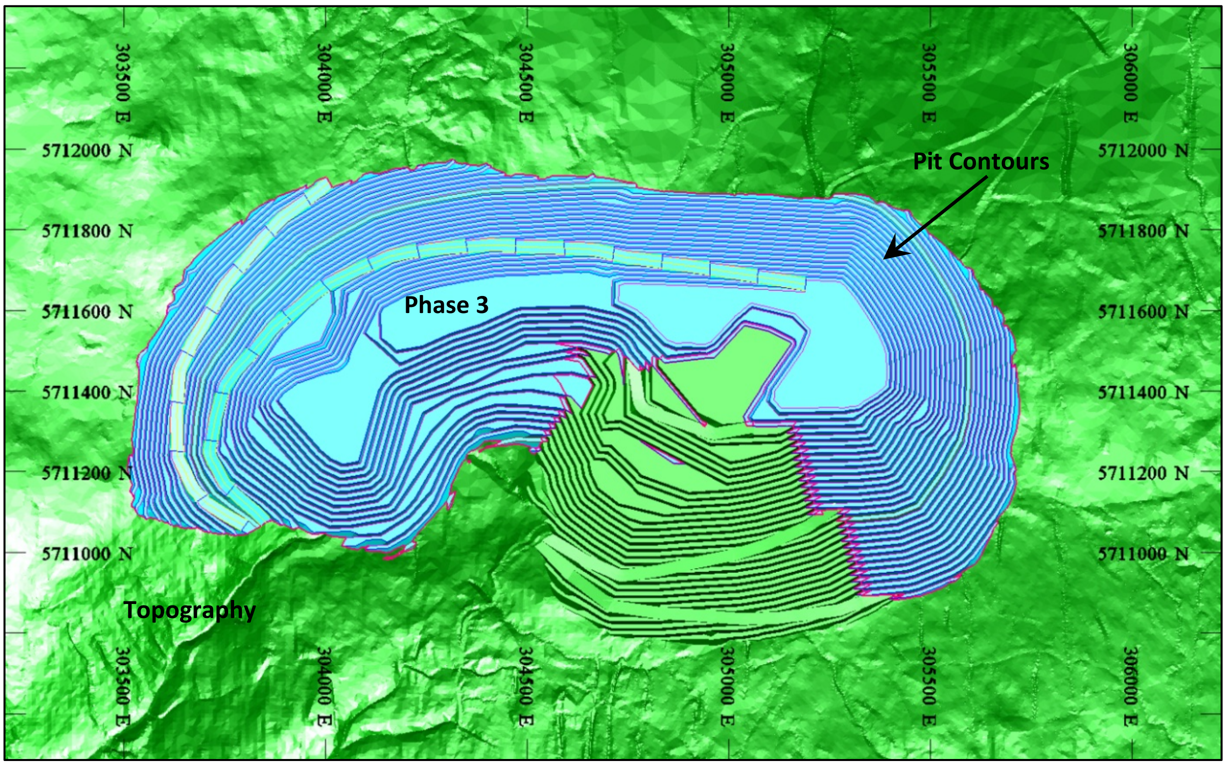


Nilsson Mining Services Inc., June 2014

16.4.4.3 Phase 3 Open Pit

The Phase 3 pit (Figure 16-9) is an easterly expansion of the Phase 1 and Phase 2 pits that also expands the mine to depth. Ramp access will be developed from the south side 1,600m elevation in a clockwise direction. A full width ramp will be left on the west side to the upper elevation Phase 5 expansion pit.

Figure 16-9 Rendered View Phase 3 Design

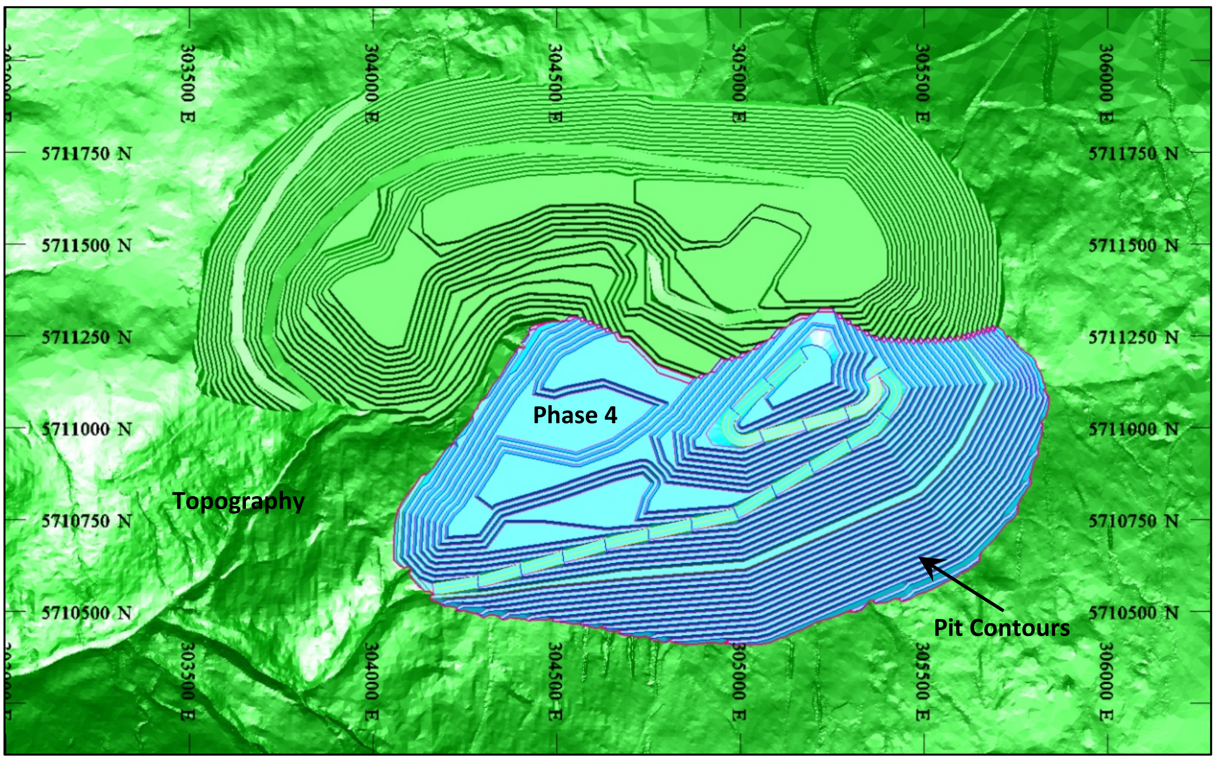


Nilsson Mining Services Inc., June 2014

16.4.4.4 Phase 4 Open Pit

The Phase 4 pit (Figure 16-10) is a southern up-dip expansion of the Phase 1, Phase 2 and Phase 3 pits. Ramp access will be developed from the south side 1,600m elevation in a counter clockwise direction. A geotechnical berm will be placed half way down the wall on the south side of the pit.

Figure 16-10 Rendered View Phase 4 Pit Design

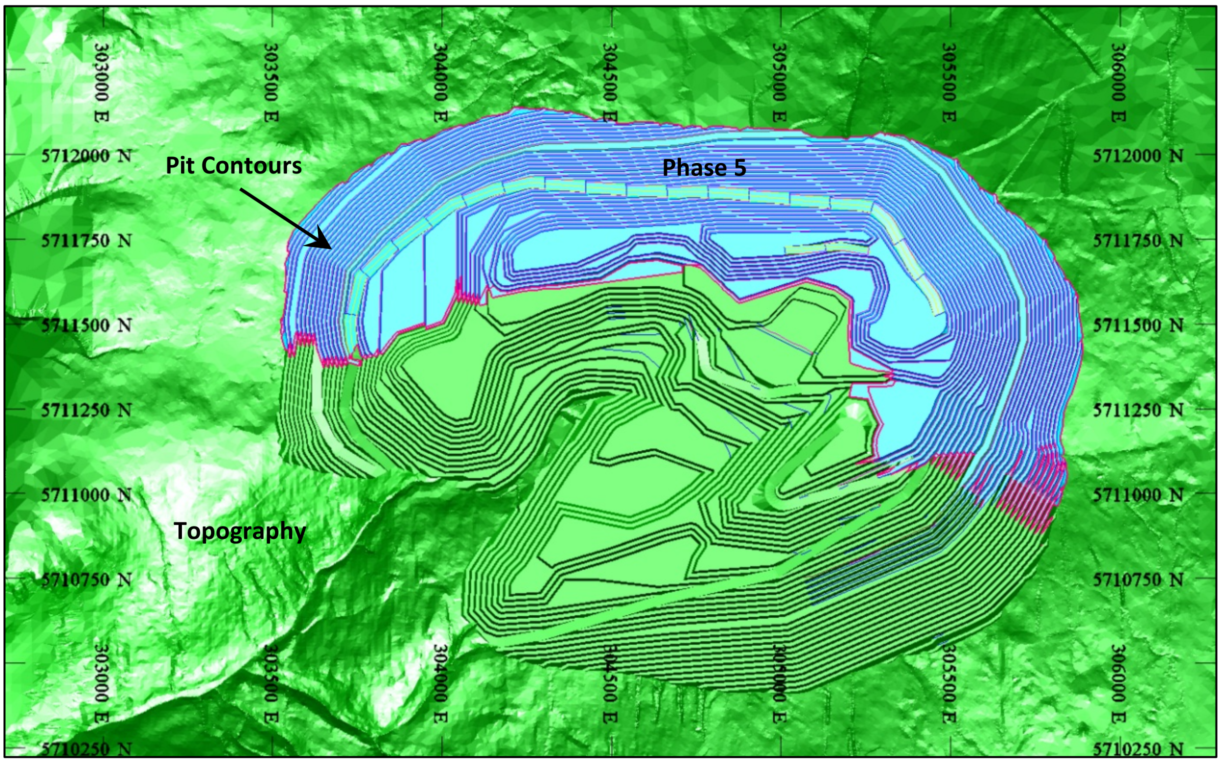


Nilsson Mining Services Inc., June 2014

16.4.4.5 Phase 5 Open Pit

The Phase 5 pit (Figure 16-11) is a northern down-dip expansion of the Phase 1, Phase 2 and Phase 3 pits. Ramp access will be developed from the southwest side linking to the Phase 3 ramp declining in a clockwise direction. Geotechnical berms will be located on the north wall and the east wall.

Figure 16-11 Rendered View Phase 5



Nilsson Mining Services Inc., June 2014

16.4.5 *MINE DEVELOPMENT*

Pre-production development will take place during the two years prior to concentrator start up. In Year 2 access road construction from the pit to the plant site and TMF will take place and the crusher and conveyor excavations will commence.

Figures 16-12 to 16-19 illustrate the progressive development of the pit, stockpiles and TMF throughout the mine life. In general development proceeds as follows:

- Year -1 (Figure 16-12) -the Phase 1 overburden stripping and waste pre-stripping will be completed to open up the mineral reserve and to provide construction material for the TMF starter dam. A total of 30.9Mt will be mined and the Phase 1 pit will be developed to the 1648 Bench. Power will not be available until startup and as such mining in pre-production will be undertaken with one diesel powered blasthole drill, a wheel loader and one hydraulic shovel using generated power. Overburden not required for construction is stockpiled to the east of the pit while non-PAG waste not required for construction is stockpiled in the valley to the west of the pit. PAG waste is transported to the TMF. A temporary non-PAG low grade stockpile is established to the south of the pit as a permanent pad for non-PAG low grade is constructed within the TMF. A PAG low grade stockpile is also established.
- Year 1 (Figure 16-13) - Mining continues in the Phase 1 pit to the 1588 Bench. Additional shovels and drills will be delivered as production and haul distances increase. The process facility will reach full production at the end of the year and the overall mine rate will be 59,650kt/a. The Non-PAG waste stockpile continues development down the valley while a PAG waste stockpile is established at the southern end of the TMF. The Non-PAG low grade is moved to the TMF location and a total of 23.7Mt of non-PAG and PAG low grade is stockpiled by the end of Year1.
- Year 3 (Figure 16-14) – Mining is focused in Phase 1 and Phase 2 with development down to the 1528 Bench and 1612 respectively by the end of the year. The non-PAG waste stockpile continues development to the 1,600m levels while the PAG waste stockpile is developed upstream to the 1,751m level. By the end of Year 3 a total of 47.3Mt of low grade (non-PAG + PAG) is stockpiled.
- Year 5 (Figure 16-15) – Phase 1 has been completed and mining is focused in Phase 2 and Phase 3 (west side) with development to the 1540 and 1624 Bench respectively. Non-PAG waste continues development to the west while the PAG waste stockpile is progressively moved upstream within the TMF. A total of 57.7Mt of low grade (non-PAG + PAG) has been stockpiled by the end of Year 5.
- Year 10 (Figure 16-16) - Phase 2 has been completed and mining is focused in Phase 3 and Phase 4 (west side) with development to the 1540 and 1672 Bench respectively. Non-PAG waste continues development to the west while the PAG waste stockpile has been moved upstream within the TMF to 1,785m level. A total of 83.5Mt of LG (non-PAG + PAG) has been stockpiled by the end of Year 10.
- Year 15 (Figure 16-17) – By the end of Year 15 Phase 3 has been mined to 1420 Bench, Phase 4 has been mined to 1648 Bench and Phase 5 has been developed on the north side to 1624 Bench. The non-PAG waste stockpile has reached its western limit and is being progressively raised to its final height of 1,680m. The PAG waste stockpile has been developed to 1,805m level. A total of 83.5Mt of LG (non-PAG + PAG) is stockpiled by the end of Year 15.
- Year 20 (Figure 16-18) – Phase 3 is completed by Year 20 and mining continues in Phase 4 and Phase 5 with development down to the 1456 and 1480 Bench's respectively. The non-PAG waste stockpile has reached its western limit and is being progressively raised. By the end of Year 20 it has been raised to a

height of 1,660m. The PAG waste stockpile has been developed to 1,825m level. A total of 115.1Mt of low grade (non-PAG + PAG) is stockpiled by the end of Year 20.

- Year 24 (Figure 16-19) – Mining within the pit has been completed and reclaiming of low grade stockpiles commenced. A total of 265.4Mt of non-PAG waste material has been stockpiled to the west of the pit to a height of 1,680m. A total of 237.4Mt of PAG waste is stored sub-aqueously within the TMF. Low grade (non-PAG + PAG) stored in stockpiles totals 106.5Mt. This material will be progressively reclaimed over the 4 years to Year 28 of operation.

Figure 16-12 Year-1 Pre-Production Mine Development

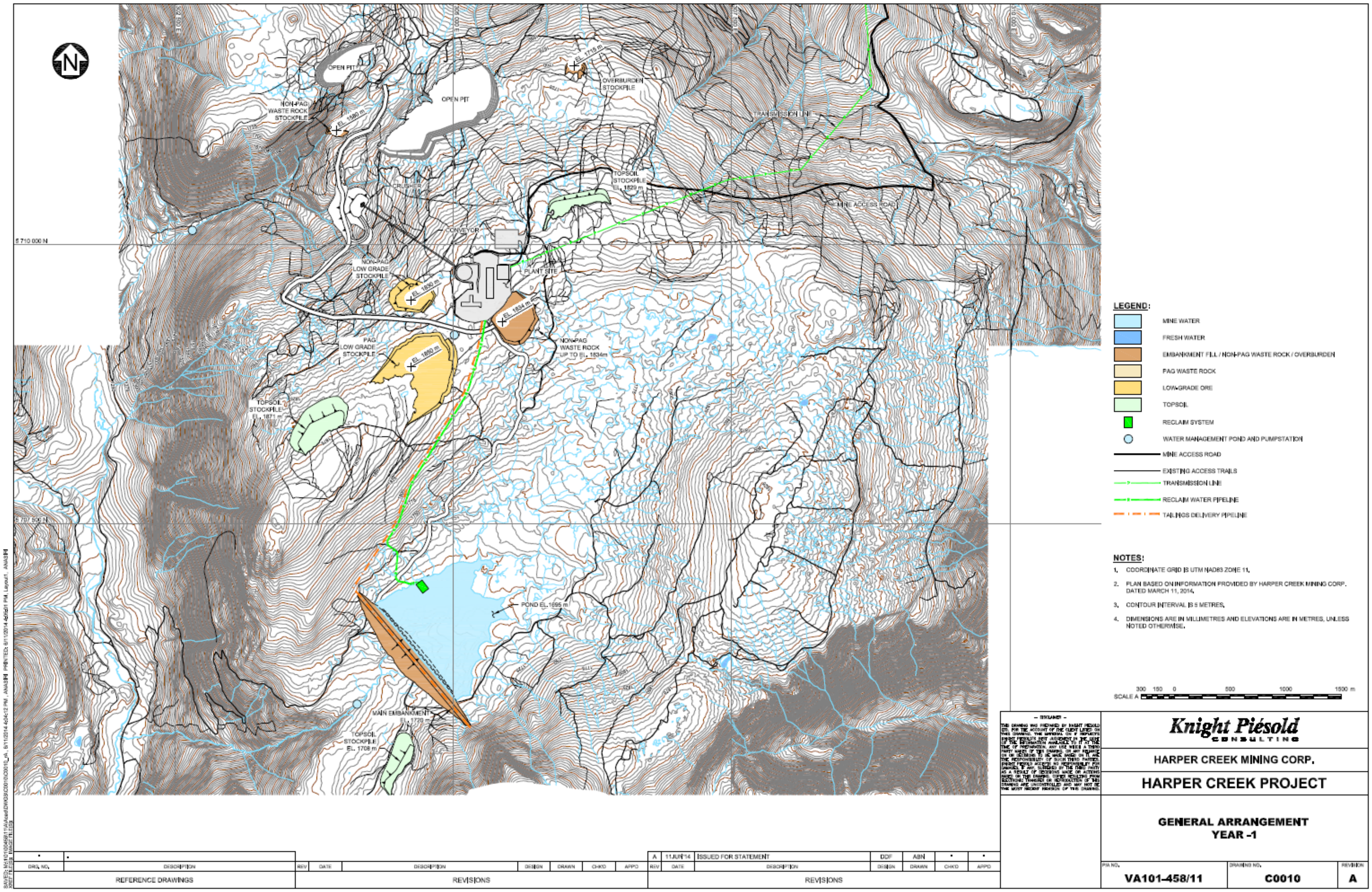


Figure 16-13 Year 1 Development

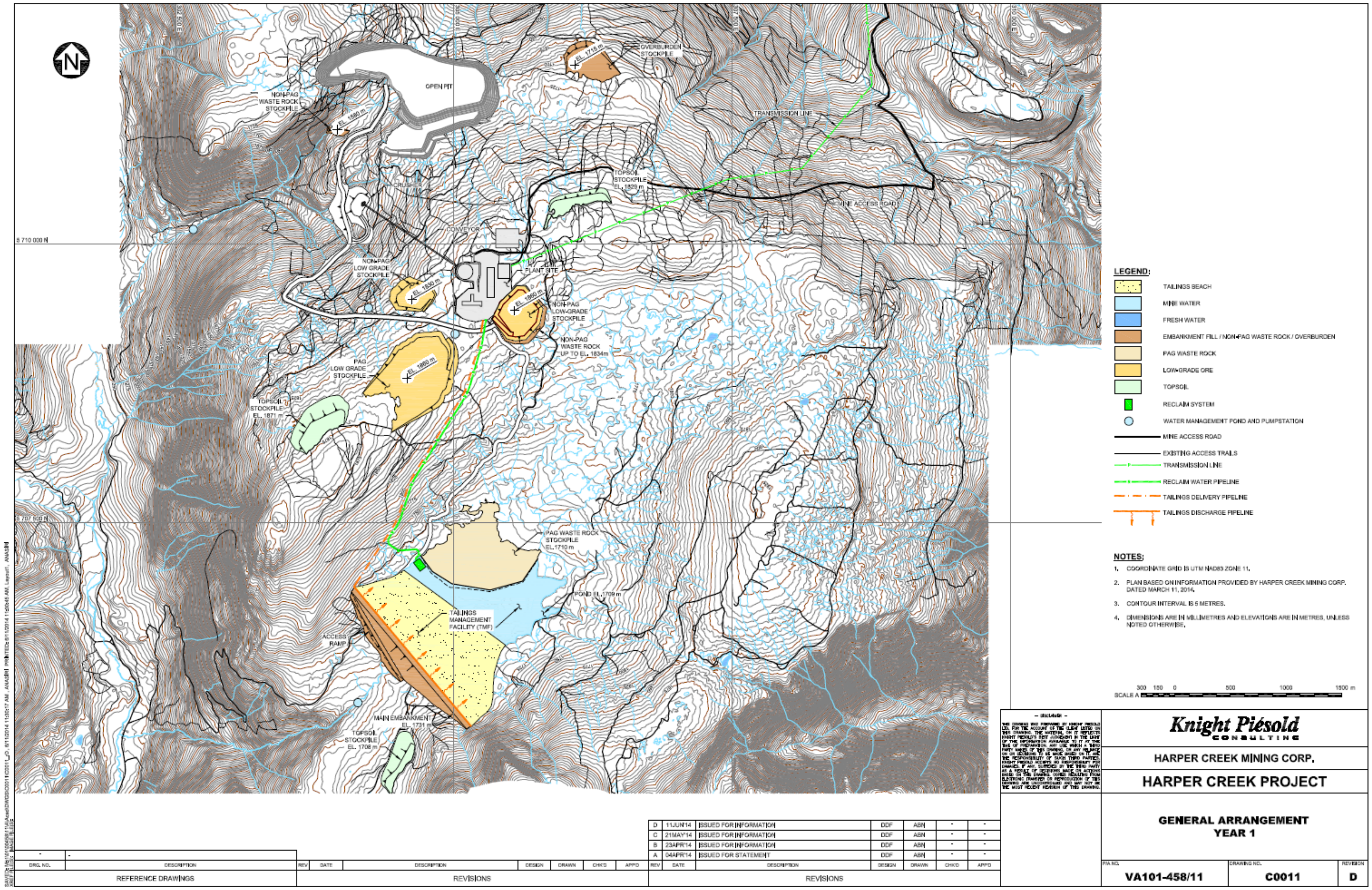


Figure 16-14 Year 3 Development

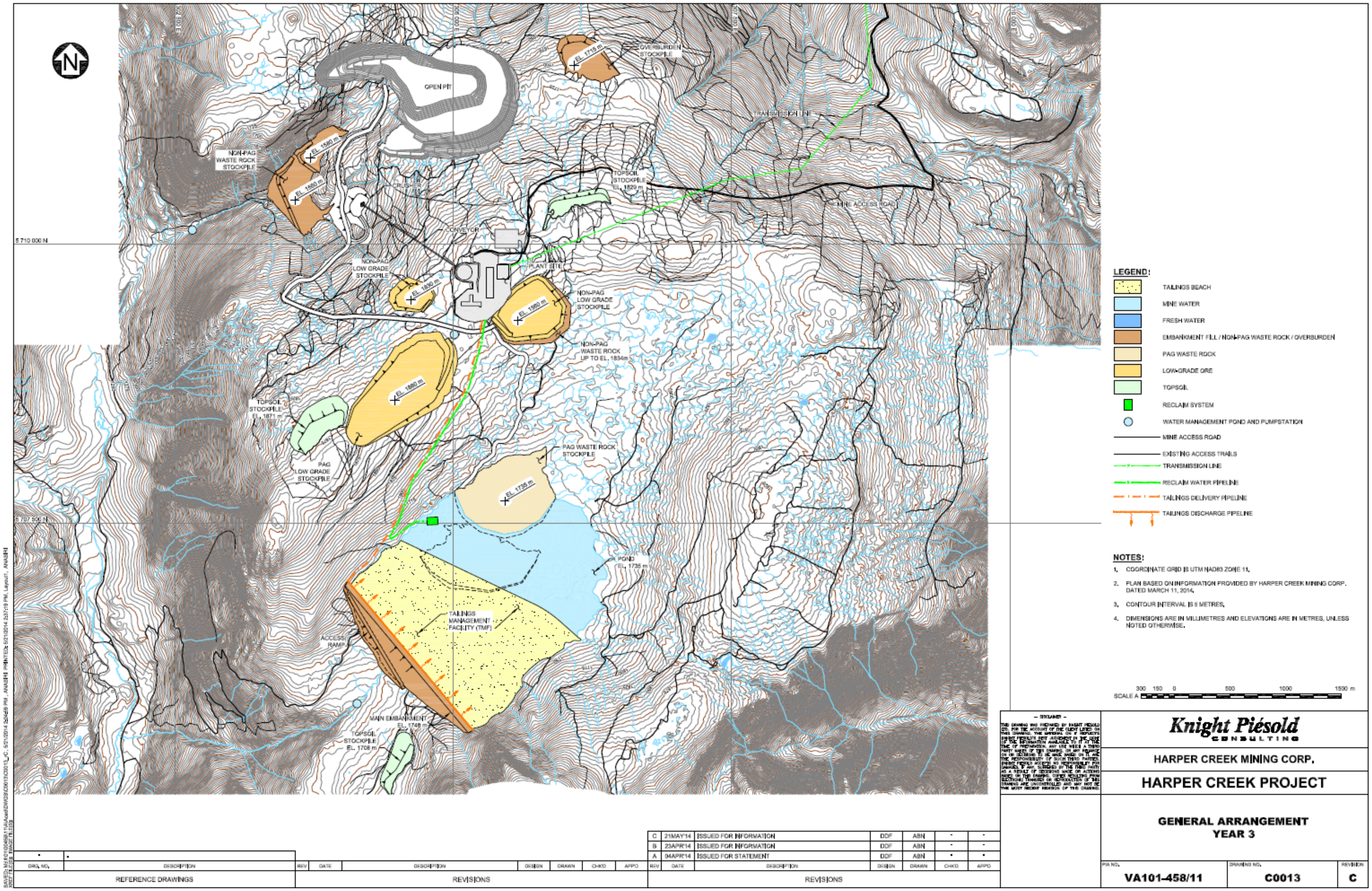


Figure 16-15 Year 5 Development

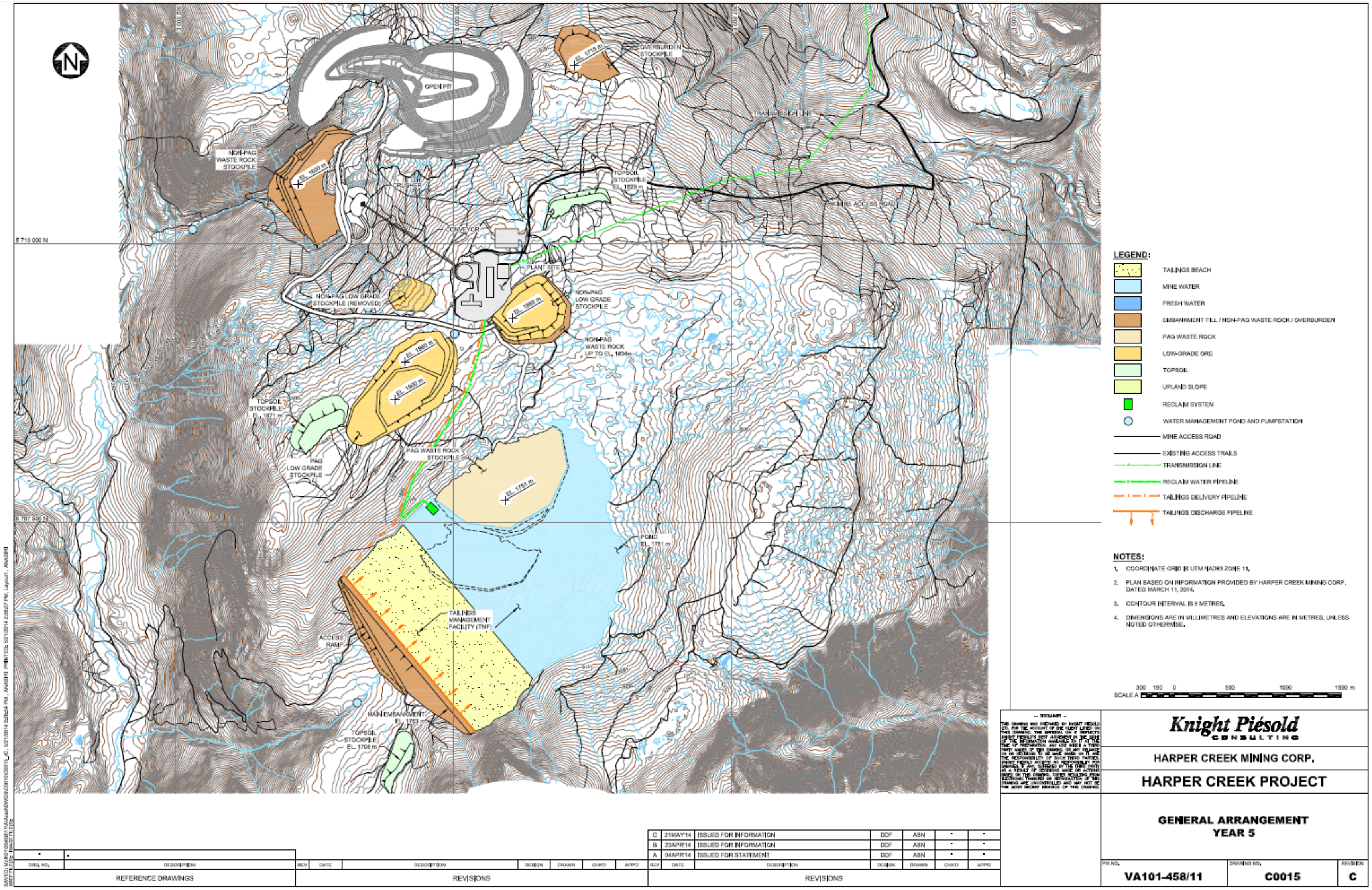


Figure 16-16 Year 10 Development

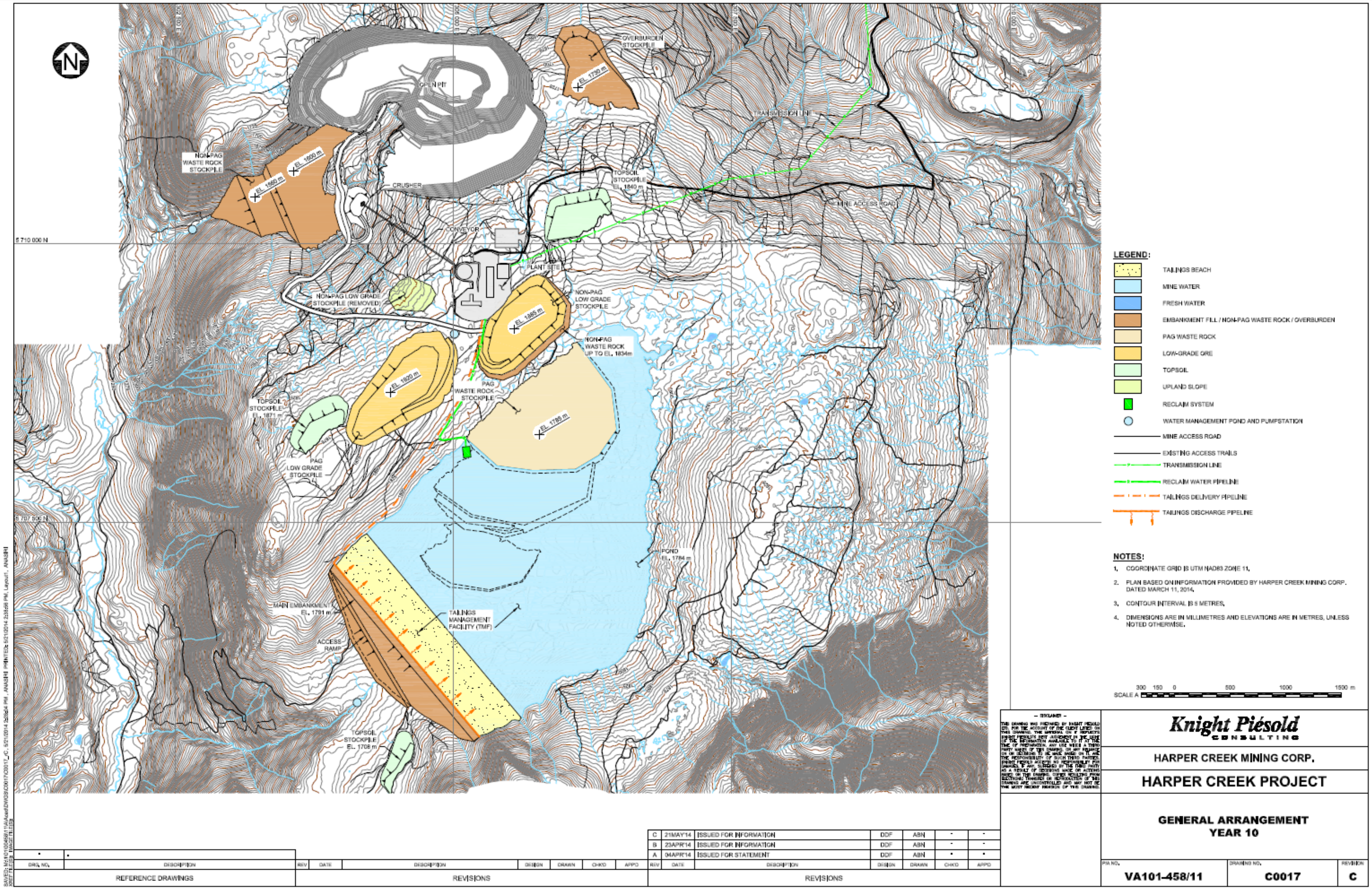


Figure 16-17 Year 15 Development

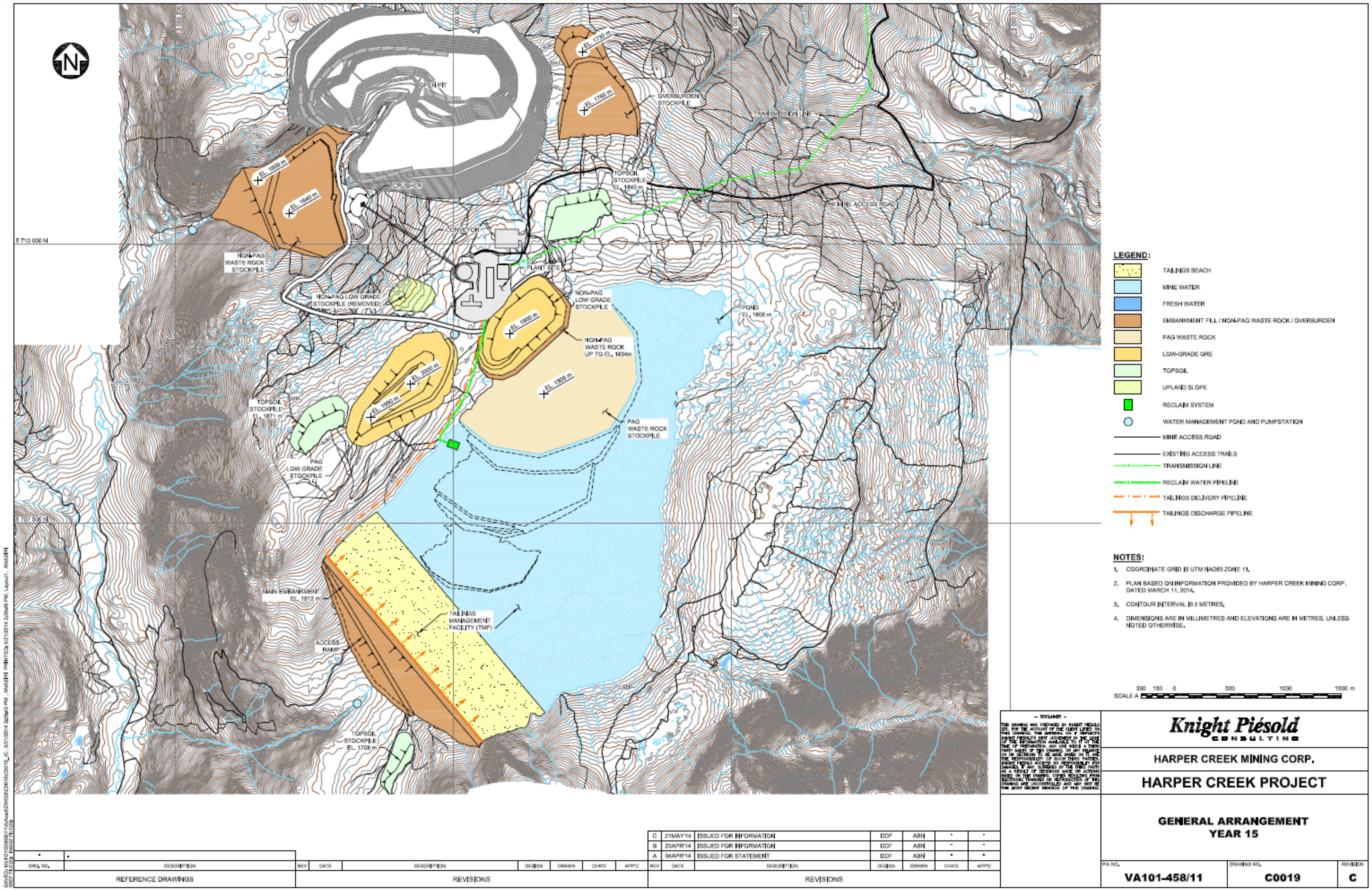


Figure 16-18 Year 20 Development

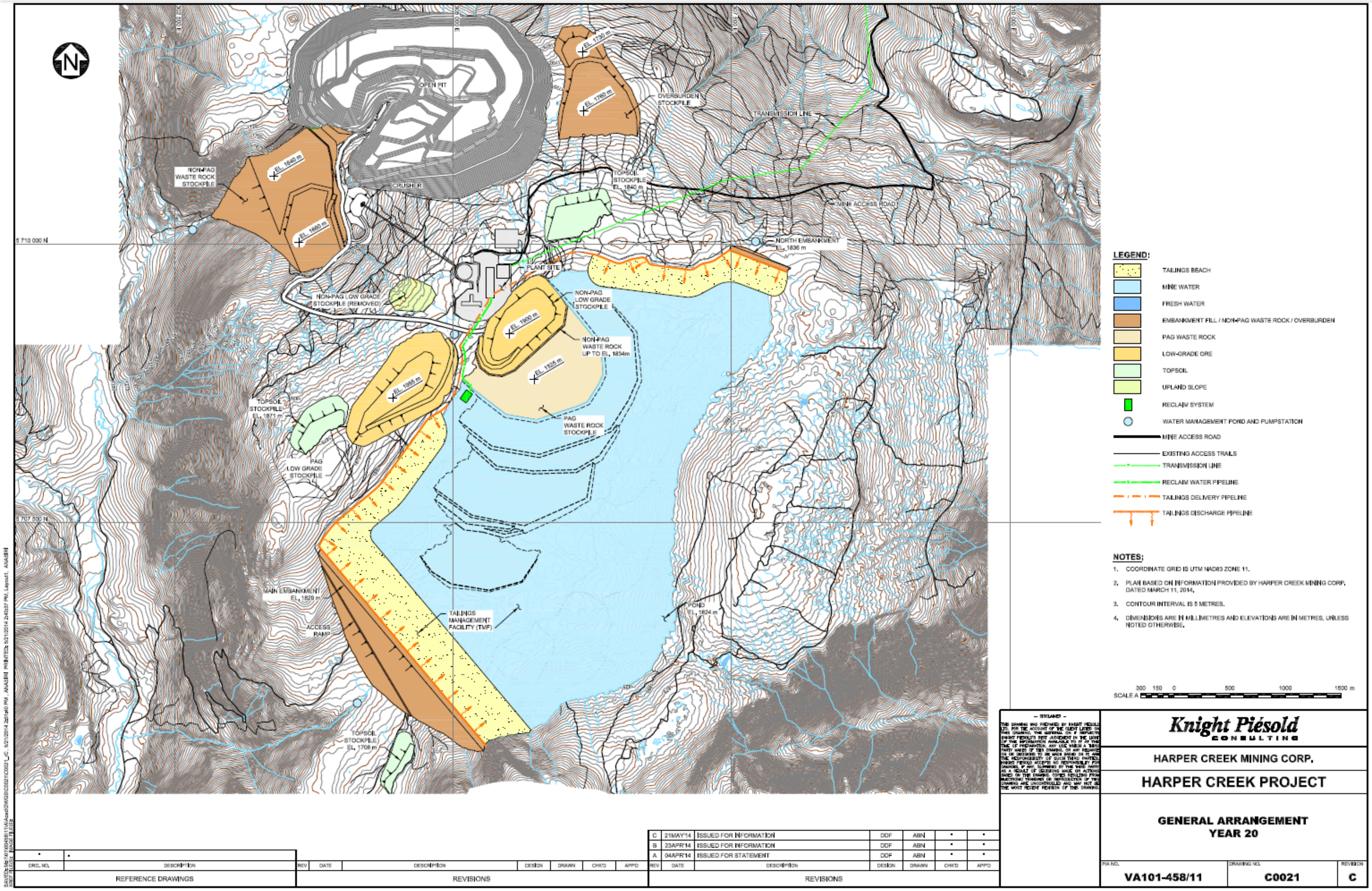
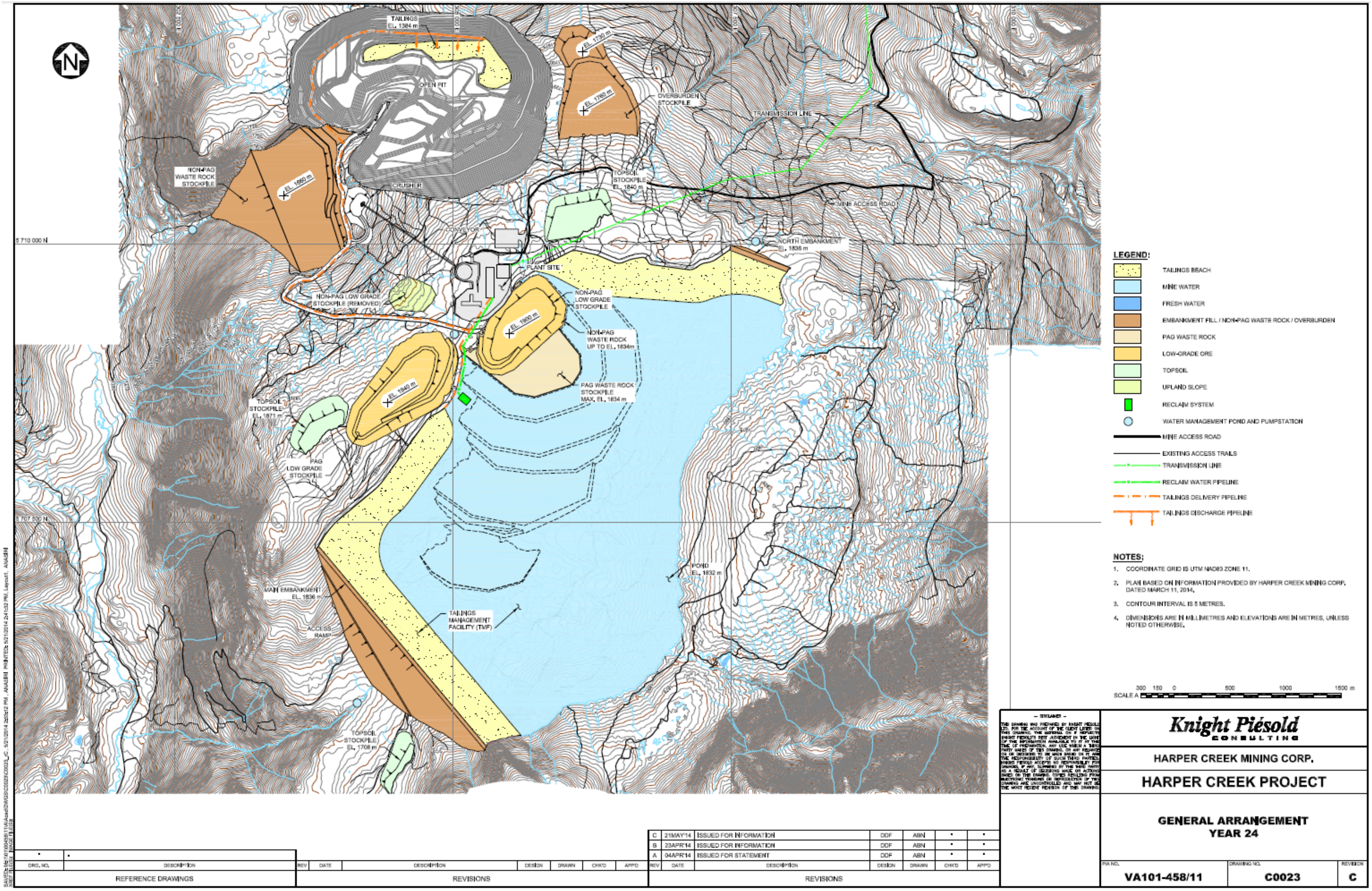


Figure 16-19 Year 24 Development - Final Pit



16.5 Mine Equipment

16.5.1 *Summary*

The mine will operate as a conventional truck and shovel operation. The typical production cycle will be drilling, blasting, grade control, loading and hauling. Primary loading units will be three electric hydraulic shovels, one drill will be diesel powered, two drills will be electric and the balance of the equipment will be diesel powered with support equipment providing development access, road maintenance and equipment servicing capability.

16.5.2 *Major Mine Equipment Operating Parameters*

The mine will operate 24 hours per day 365 days per year. Shift employees will work 12 hour shifts on a 28 day cycle. In general, it is expected that major equipment will have 85% to 90% availability initially, declining with age. Detailed equipment productivity calculations were made on an annual basis for drills, shovels and trucks. Support equipment operating time was factored on an annual utilization basis.

16.5.3 *Drilling and Blasting*

The primary blasthole drills will be rotary machines capable of single pass drilling 311mm holes for a 12m bench height. These drills will be used for production and wall control drilling. One diesel powered drill will be mobilized during pre-production and additional drills will be electric. A secondary hydraulic drill will be available for pioneering and oversize reduction. This drill will also be used for drilling sub-horizontal drain holes for wall slope depressurization.

The fleet will initially consist of one unit during the pre-production period. Drill additions will be made in Year 1 with the fleet peaking at three blasthole drills. Blasthole drilling requirements have been estimated on an annual basis according to the production schedule and wall control drilling requirements for trim blasting.

Material will be drilled on a 12m bench using on a 10.0m x 11.5m pattern. Subgrade drilling will be 2.0m to allow even breakage to the design bench elevation. Blasthole cuttings will be sampled and assayed for grade control.

The wall control blasting will consist of trim rows at reduced burden and spacing. These will be drilled with the production drill. The sub-grade drilling depth will be reduced in areas of final berm locations.

Blasting will be carried out with heavy AN/FO. The overall production blasting agent consumption is expected to be 0.22kg/t of material. Blastholes will be single primed and initiated using non-electric methods. An explosive supply contractor will deliver bulk explosives to the borehole. The mine blasting crew supervised by the Mine Foreman will work closely with the Drill & Blast Engineer.

16.5.4 *Loading*

The loading fleet will consist of three 42.0m^3 electric hydraulic shovels, and one 18.5m^3 wheel loader. The wheel loader will be available to work in stockpile areas, low face conditions, and where required to meet production objectives during periods of unscheduled shovel downtime. The first hydraulic shovel and the wheel loader will be required in Year 2 pre- production for pre-stripping and road construction.

The loading equipment will operate two twelve hour shifts per day. Initial loading equipment availability is expected to be 85%, declining at a rate of 1%/year as equipment ages. The productivity calculations assume good digging conditions, three pass loading of the trucks with an overall cycle time of 3.15 minutes.

16.5.5 *HAULAGE*

Diesel electric drive rear dump haul trucks of 227t capacity will be used to move material to the crusher and to the various dumps and stockpiles. The haulage trucks will operate two twelve hour shifts per day. Initial equipment availability is expected to be 90% declining at a rate of 1% per year to 80% as equipment ages.

The cycle times for ore or waste were calculated for each year of production based on haul distances and road grades. These were then used to calculate haul truck productivities and fleet requirements. A total of fifteen 227t haulage units are required during pre-production with additions to the fleet made continuously until the fleet total reaches 28 trucks in Year 13.

16.5.6 *MINE SUPPORT*

The mining support equipment includes track dozers, wheel dozers, graders, a water truck, and a sand truck required for road, bench and dump maintenance. Miscellaneous ancillary equipment is also required to service, maintain the major equipment and support ongoing pit operations.

Track dozers will operate on active benches pushing back break and performing heavy dozer operations around operating shovels. In the open pit they will also build roads, prepare sinking cut faces, clean berms, scale walls and rip hard toes. On waste dumps and stockpiles the track dozers will maintain positive grades on the bench surfaces near the crest and provide safe berms for truck dumping.

Road graders and rubber tire dozers will maintain road, dump and bench surfaces to provide level running surfaces and move snow. Water tucks and sand trucks will be used in the road maintenance program to provide dust control and safe winter running conditions.

A complement of ancillary equipment as listed in Table 16-4 will also be available to perform service functions including fueling, provide work area lighting, excavation capability for ditching etc. as required to ensure a safe self-sufficient mine operation.

Pick-up trucks and crew-cabs will be required for transportation of supervisors, technical staff and maintenance personnel.

Explosives will be delivered to the blasthole. The blasting crew will require support equipment to pump wet holes, deliver blasting accessories and stem holes. The bulk delivery truck and storage facilities will be provided by the explosives contractor.

Table 16-4 Mine Equipment Fleet

MINE EQUIPMENT REQUIREMENT																															
Year		-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28
Rotary Blasthole Drill	311 mm .	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3				
Hydraulic Drill	150 mm	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1					
Hydraulic Shovel	2900 kW 42 m^3	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	1	1	1	1	1
Wheel Loader	1176 kW 18 m^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Haul Truck	1976 kW 227 tonne	15	20	20	20	20	22	24	24	24	26	26	26	26	28	28	28	28	20	18	18	15	15	15	14	11	6	6	6	6
Track Dozer	433 kW 66 tonne	6	6	6	6	6	6	6	5	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4	4	2	1	1	1	1
Wheel Dozer	512 kW 15,9 m^3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1
Grader	221 kW 4.9 m	2	2	2	3	3	3	4	4	4	4	4	4	4	4	4	4	4	3	3	3	3	3	2	2	1	1	1	1	1
Water Truck	1082 kW 30,000 gal	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Sand Truck	758 kW 90 tonne	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Blasthole Stemmer	1.5 m^3 5.6 tonne	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3					
Wheel Loader	414 kW 6.9 m^3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1
Haul Truck	552 kW 55 tonne	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	2	2	2					
Excavator	4 m^3 86 tonne	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1	1					
Tire Manipulator	Large Tire	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Vibratory Compactor	116 kW 2134 mm	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe	75 kW 10 tonne	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cable Reeler	261 kW	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel and Lube Truck	18,000 liter	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tractor and Low Bed	160 tonne	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Flatbed Hiab Truck		2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Snow Plow and Sand Truck		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Rough Terrain Crane	80 tonne	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Rough Terrain Forklift	30 tonne	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Shop Forklift	16 tonne	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mechanics Truck		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Welding Truck		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Engineering Pickup	1 tonne Pickup	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	6	6	6	6	6
Pit Services Pickup	1 tonne Pickup	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	6	6	6	6	6
Engineering Pickup	1 tonne Crewcab	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4	4	4
Pit Services Pickup	1 tonne Crewcab	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pit Services Bus	10 Passenger	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pit Services Bus	10 Paseenger	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shovel Crew Flat Deck		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shovel Crew Hiab		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surface Crew Hiab		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surface Crew Stinger		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Tower		5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	2	2	2	2	2
Hydraulic Hammer		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Rescue Vehicle		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

16.6 MINE ANCILLARY FACILITIES

16.6.1 *MINE MAINTENANCE SHOP*

The Mine Maintenance Shop (or Truckshop or Service Complex) is intended to be a pre-engineered steel building located close to the plant site where the mine trucks pass on route to the majority of the proposed dump areas – whether they be at the tailings dam, low grade storage stockpile, or PAG dump.

The Truck Shop will have five regular mine fleet service bays, two welding bays and one preventative maintenance (PM) bay. While the mine fleet trucks are sized for 227t capacity, the working bays have been sized to accommodate 292t trucks in the event that the operations are expanded and a larger fleet is needed. The main vehicle bays are provided with 12m x 8.5m vertical fabric folding doors. The truck shop and other bays will be serviced by a 50/15t bridge crane. Section 18 provides a more complete description and plan view of the facility that will also house a Dry, ambulance and fire truck bay, some mine offices (the admin offices are in Vavenby), a warehouse located in a fabric housing connected to the pre-engineered building, and truck wash facilities.

16.6.2 *MINE ELECTRICAL POWER*

The mining equipment uses about 11.8% of the operating power with a connected load of about 15,000kW and an average demand of about 9,700kW. Power will be provided through the plant site sub- station via an overhead line to the pit rim where the pit will be serviced by portable 25kV/7.2 kV substations with trailing cables as required for the equipment.

During the construction stage when a part of the mine fleet, including a diesel drill and electric shovel, will be used to build the haul road to the tailings, backfill for the crusher and most of the Stage 1A tailings dam, power will be provided via leased generator sets complementing the two generator sets that make up the emergency power for the permanent plant.

16.6.3 *MINE DISPATCH*

A mine dispatch system with be implemented. Provisions have been made for software and infrastructure as well as hardware installation on all major equipment.

16.6.4 *MINE WATER MANAGEMENT*

Water sources across the project site will be managed in such a manner as to reuse contact water, while diverting non-contact water around the various mine facilities as much as possible. Potable water will be derived from groundwater sources near the plant site, while fresh water requirements for the process plant will be sourced from the pit dewatering and/or waste rock stockpile seepage collection ponds. The largest water requirements exist for the milling process, and this water will come from the reclaim water in the TMF.

16.6.5 *Fuel Storage*

Adequate fuel cost allowance has been incorporated in the capital cost estimate to cover what will be needed for the running of the generator sets providing power to the drill and shovel during construction. This means building the fuel farm near the plant site early in the construction program to accommodate storage. However, with the plant site being located so close to the communities in the area, bringing fuel into the site as needed does not pose a problem.

16.7 Mine Staff and Hourly Employees

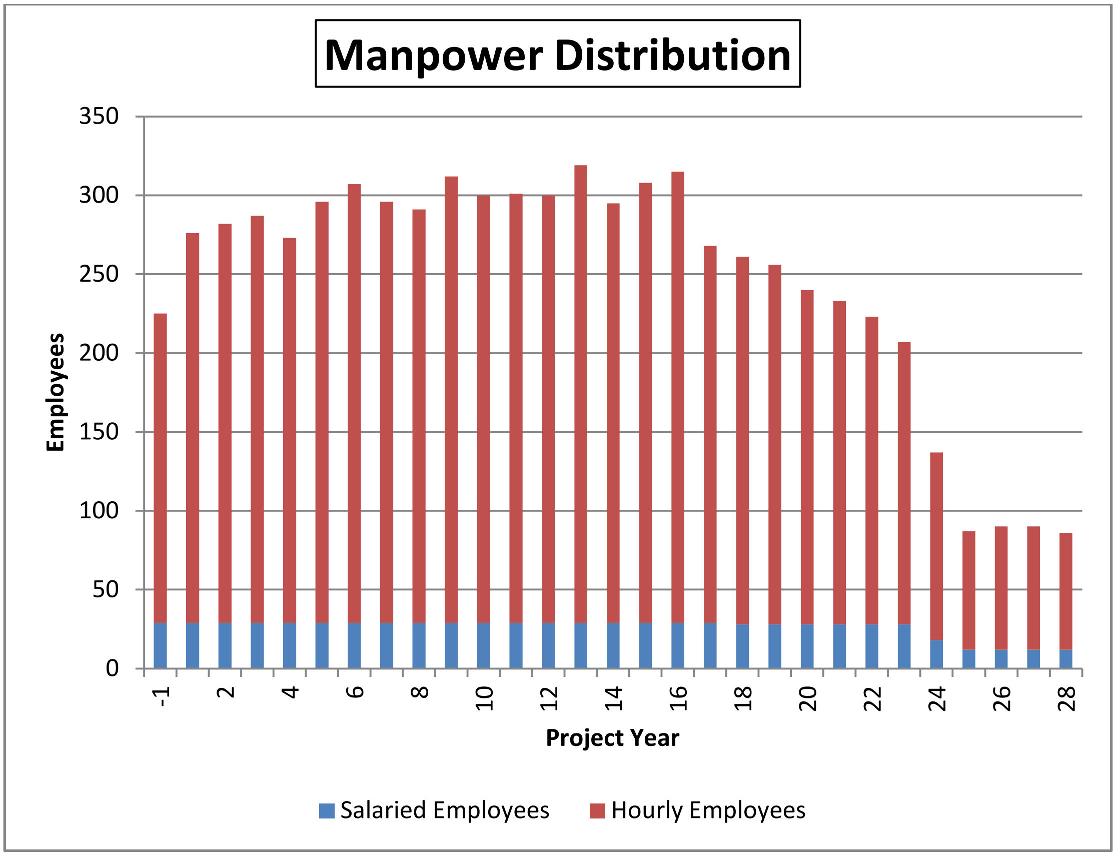
The mine staff and hourly employee requirements have been estimated for the life of the mine. There will be a total of 29 salaried employees for mine supervision, mine maintenance and engineering. The hourly employee level will vary with time as the haulage fleet changes in response to increasing haulage profiles and overall mining quantities. Initially, 209 hourly employees will be required. Hourly employee requirements will increase to a maximum of 290 in Year 13 then decline as the mining rate drops and stockpile recovery commences.

The anticipated mine staff and hourly employee levels are shown in Table 16-5. Manpower distribution is shown in Figure 16-20.

Table 16-5 Manpower Distribution

Project Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28
Salaried Employees																													
Mine Supervision	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	7	5	5	5	5
Mine Maintenance	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	7	5	5	5	5
Engineering & Geolo	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	11	11	11	11	11	11	4	2	2	2	2
Subtotal	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**29**	**28**	**28**	**28**	**28**	**28**	**28**	**18**	**12**	**12**	**12**	**12**
Hourly Employees																													
Operations	135	171	173	177	163	184	194	184	179	194	182	183	182	201	177	191	198	163	157	154	143	139	132	121	79	49	49	49	46
Maintenance	61	76	80	81	81	83	84	83	83	89	89	89	89	89	89	88	88	76	76	74	69	66	63	58	40	26	29	29	28
Subtotal	**196**	**247**	**253**	**258**	**244**	**267**	**278**	**267**	**262**	**283**	**271**	**272**	**271**	**290**	**266**	**279**	**286**	**239**	**233**	**228**	**212**	**205**	**195**	**179**	**119**	**75**	**78**	**78**	**74**
Total	**225**	**276**	**282**	**287**	**273**	**296**	**307**	**296**	**291**	**312**	**300**	**301**	**300**	**319**	**295**	**308**	**315**	**268**	**261**	**256**	**240**	**233**	**223**	**207**	**137**	**87**	**90**	**90**	**86**

Figure 16-20 Manpower Distribution



Nilsson Mining Services Inc., June 2014

17 RECOVERY METHODS

17.1 PROCESS PLANT

Harper Creek ore is a VMS hosted by a sedimentary series of sandstone and siltstones and thus differs somewhat from the typical BC low-grade copper deposit.

The proposed ore process plant (Concentrator) for the Project is conventional for a large tonnage, low grade copper deposit in BC, but differs slightly from some of the designs in earlier studies (PEA). The differences are the result of ongoing metallurgical test work carried out on core samples specifically procured for metallurgical test work as discussed in Section 13 of this report.

One of the major changes from earlier studies is the primary grind size. The PEA had a primary grind of 105 microns but for the 2012 FS superior results were achieved with a primary grind of 180 microns. This coarser grind is more typical of BC copper projects and has a number of benefits including reduced operating costs. In addition, the control of unwanted pyrite into the primary rougher/scavenger concentrate was achieved by operating this circuit at an elevated pH (11.0) compared with earlier test work carried out at a more natural pH (9.0). The copper recovery to this concentrate was equal to (or better) than the earlier test work and the weight recovery was reduced (from 12% to 6.5%, because of the rejection of pyrite), thus reducing the size of the regrind and cleaner circuits. A second benefit of this reduced weight recovery was the easier rejection of Pyrite in cleaning (because there is not as much): This can now be achieved after a fine (20-25 micron) regrind but without the use of any special depressants (e.g., Cyanide).

The ore will be handled conventionally through a primary gyratory crusher and then transported to a coarse ore stockpile. Ore from the stockpile is reclaimed and then ground through a SAG-Ball mill circuit. The option of HPGR to replace the SAG mill was rejected because of the relatively low metric Bond BM Work index (13.2) and the modest/high abrasion index (~0.35 on the most significant ore types). Comminution test work carried out by FLS also suggested that pebble crushing in the SAG circuit would not be required: This conclusion was reviewed by an independent consultant, as it was atypical, and appears to be related to the soft to moderately soft sedimentary lithology of the Deposit. The primary grind to 80% passing 180μm is moderately coarse but typical of large tonnage low grade copper operations.

Concentration is by flotation at elevated pH to depress pyrite. Rougher/scavenger concentrates are very finely reground ahead of cleaning using a stirred mill (IsaMill™). This newer technology is now widely used and makes fine regrinding feasible. Cleaning takes place in column cells, a relatively recent technology, but is widely used and is the standard in cleaning circuits.

The final copper concentrate is thickened and then filtered in an automated pressure filter. This type of filtration makes possible the production of concentrates sufficiently low in % moisture by weight so that they can be shipped by bulk ocean freight without the need for thermal drying.

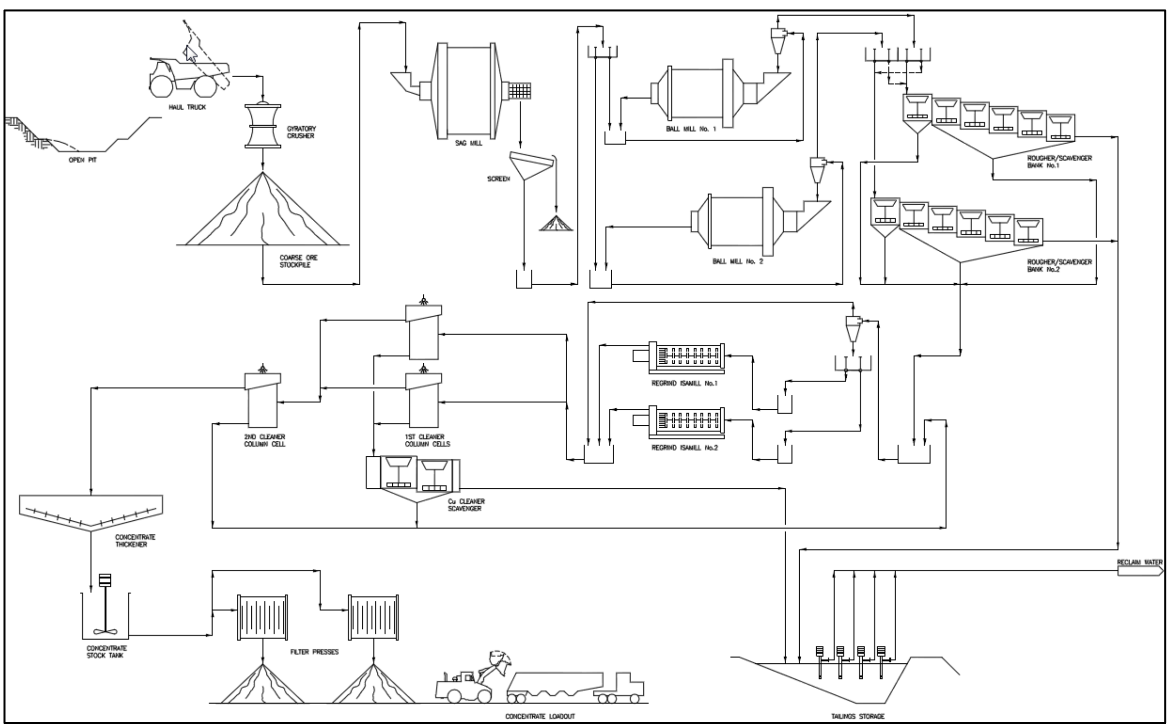
17.2 PLANT DESIGN & EQUIPMENT

The Concentrator is of conventional design and is designed for simplicity of operation and to maximize recovery. The ROM ore is reduced through three stages of comminution and the copper minerals recovered by flotation, with rougher/scavenger concentrates reground and cleaned to final commercial concentrate grades. The concentrator is designed to process a nominal 70,000t/d of copper sulphide ore and produce marketable copper concentrate.

The flow of ore will be through crushing, grinding, mechanical rougher/scavenger flotation tank cell banks and the rougher/scavenger concentrate is cleaned through two stage column flotation cleaning to increase the quality of the concentrate. The rougher/scavenger concentrate will be sent through a regrind circuit and reprocessed through the cleaners to increase copper grade to commercial levels. Final concentrate from the second cleaner column will be densified through a thickener and dewatered in filter presses to achieve concentrate moisture of approximately 8%. This concentrate will be trucked offsite for shipping to smelters.

The final flotation tailings, including the rougher/scavenger flotation tailings and the first cleaner/scavenger tailings, will be disposed using the conventional tailing storage method. The process water in the TMF will be recycled to the process plant. Fresh water for process water make-up, gland seal service, mill cooling, and reagent preparation will be obtained from various local water sources. A simplified FS flowsheet of the process is shown in Figure 17.1.

Figure 17-1: Simplified FS Flowsheet



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The process plant will consist of the following unit operations and facilities:

- Primary crushing;
- Overland conveying;
- Crushed material stockpile and reclaim;
- Primary grinding circuit, including a SAG mill, two ball mills, incorporating hydrocyclones for classification;
- Copper rougher and scavenger flotation;
- Rougher and scavenger concentrate regrinding;
- Copper cleaner flotation;
- Copper concentrate thickening, filtration and stockpiling, including off-site;
- Concentrate handling; and
- Tailing disposal to the tailings pond.

17.2.1 *MAJOR DESIGN CRITERIA*

The concentrator will process the mineralization at a nominal rate of 70,000t/d, or 25.55Mt/a. The major criteria used in the design of the concentrator are summarized in Table 17.1 and the equipment list is summarized in Table 17-2 (Section 17.2.12).

Table 17-1: Major Design Criteria

Criteria	Unit	Value
Operating Days	d	365
Operating Time	h/d	24
Annual Throughput	t/a	25,550,000
Daily Process Rate	t/d	70,000
Crushing Availability	%	70
Grinding and Flotation Availability	%	92
Ore Specific Gravity	-	2.7
Crushed Ore Bulk Density	t/m^3	1.6 (est)
Crusher Work index	-	6.5
Primary Crushing Rate	t/h	4,167
Grinding & Flotation Process Rate	t/h	3,170
SAG Mill Feed Size, 80% Passing	mm	200
SAG Mill Transfer Size, 80% Passing	mm	1.25
Ball Mill Product Size, 80% Passing	µm	180
Bond Ball Mill Work Index (metric)	kWh/t	13.2
Concentrate Regrind Size, 80% Passing	µm	20
Concentrate Bulk Density	t/m^3	2.2
Filter Press Availability	%	90

The SAG and ball mills were sized based on the Bond grindability work index. The regrind mill size was based on XStrata applying an efficiency factor estimated from similar IsaMill™ installations.

The flotation cells were sized and selected based on the estimated slurry flow rates and the flotation retention times as determined from the laboratory tests. Typical scale-up factors were applied for sizing flotation cells and a minimum number based on experience was applied to avoid bypassing.

17.2.2 *Primary Crushing*

The conventional gyratory crusher facility will crush the ROM at an average rate of 4,167t/h for the downstream grinding process. The major equipment and facilities include:

- One gyratory crusher: 1,524mm x 2,261mm (60” x 89”);
- One apron feeder: 2,438mm x 8,600mm;
- One hydraulic rock breaker;
- One sacrificial collecting conveyor;
- One belt conveyor to transport the crushed material to the stockpile; and
- Dust suppression systems.

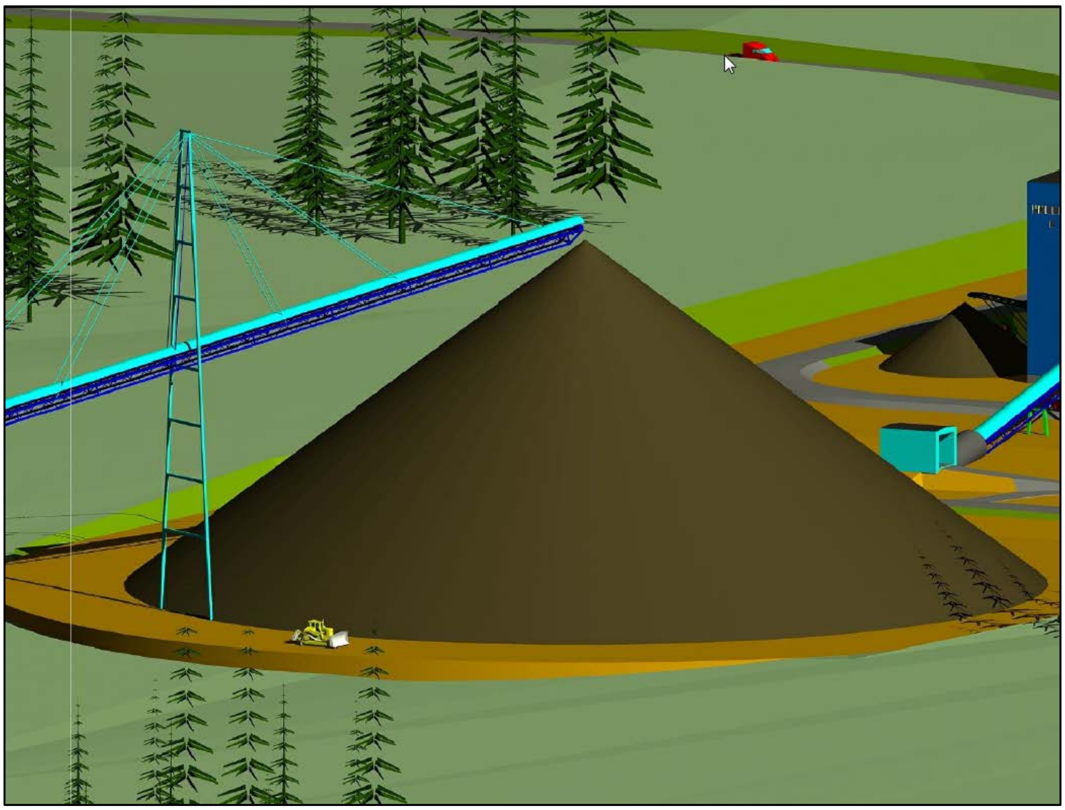
The ROM will be trucked from the open pit to the primary crusher by haul trucks. The ROM will be reduced to a product size of 80% of it finer than 200mm using the primary crusher. A rock breaker will be installed to break any oversize rocks that may clog the dump pocket. The crushed material will be discharged underneath the crusher and then onto the apron feeder. The apron feeder will convey the crushed materials onto a sacrificial conveyor and then to the overland conveyor which transports the crushed material to the stockpile.

The crushing facility will be equipped with a dust suppression/collection system to control fugitive dust that will be generated during crushing, material loading, and related operations.

17.2.3 *STOCKPILE AND RECLAIM*

The stockpile for the crushed material will have a live capacity of 70,000t/d. The material will be reclaimed from this stockpile by three 1829mm x 6,135mm apron feeders (two operating, one standby) at a nominal rate of 3,170t/h. The apron feeders will feed a 1,524mm wide belt conveyor at a controlled rate. The conveyor will transport the crushed material to the SAG mill. The conveyor belt will be equipped with a belt scale. The reclaim area will be equipped with a dust collection system to control fugitive dust that will be generated during the loading and the transportation of the crushed material (Figure 17-2).

Figure 17-2: Conceptual Rendering of Stockpile and SAG Feed Conveyor



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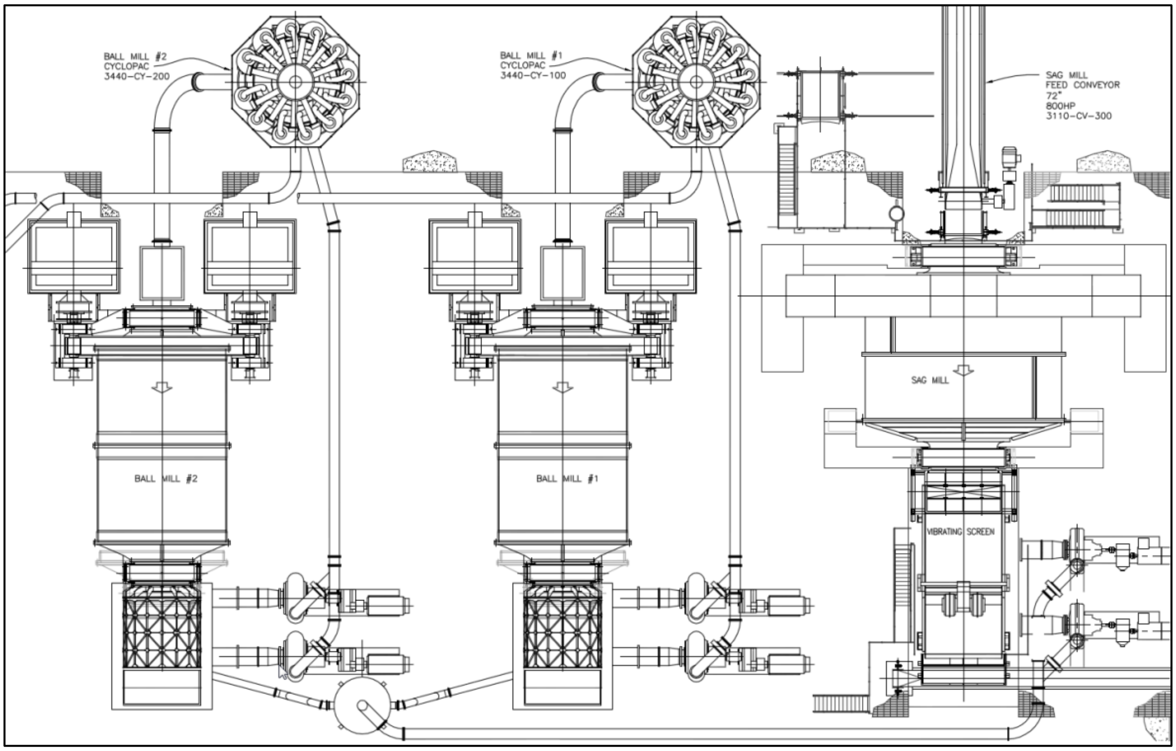
17.2.4 *PRIMARY GRINDING AND CLASSIFICATION*

Figure 17-3: Grinding Area Layout



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The primary grinding circuit will be a SAB circuit encompassing a two stage grinding process which incorporates a SAG mill and two ball mills in closed circuit with classifying hydrocyclones (Figure 17-3). The grinding will be conducted as a wet process at a nominal rate of 3,170t/h of circuit feed. Lime will be added to the SAG mill feedbelt to raise the pH to 11, and aid selectively in flotation. The circuit has been designed for the future installation of a pebble crusher should it be required, however, as stated previously comminution test work and analysis suggest that pebble crushing is unlikely to be required. The grinding circuit will include:

- One SAG mill – 11.6m ø x 6.7m long (38ft x 22ft), 21MW;
- One SAG mill discharge pulp distributor;
- Two ball mills – 7.3m ø x 12.8m long (24ft x 42ft), 13MW each;
- Two 4,300mm x 10,000mm vibrating screens (one operation, one standby);
- Four hydrocyclone feed slurry pumps (two operation, two standby); and
- Two hydrocyclone clusters, each with eight 800mm hydrocyclones.

The crushed material from the stockpile will be reclaimed at a controlled feed rate and fed to the SAG mill. The mill will discharge onto a 4,300mm x 10,000mm vibrating screen with the oversize being conveyed to a bunker outside the grinding building for disposal, or manual re-entry into the process. The screen undersize will be pumped to a distribution box which will split the flow into two streams. The split slurry will report separately to two Ball mill discharge/hydrocyclone feed pump boxes. Each ball mill will be operated independently in closed circuit with a hydrocyclone cluster. The feed to each of the ball mills will be the underflow of each mill's cyclone cluster. The hydrocyclone overflow advances downstream to the rougher/scavenger flotation process with a particle size of 80% passing 180µm and contain approximately 35% solids by weight. The hydrocyclone underflow will report back to the ball mill. The circulating load of the ball mill circuit will be approximately 250% of the circuit new feed. Ball charge systems will be provided to add grinding media to the mills to maintain grinding charge.

17.2.5 *Flotation and Regrinding Circuits*

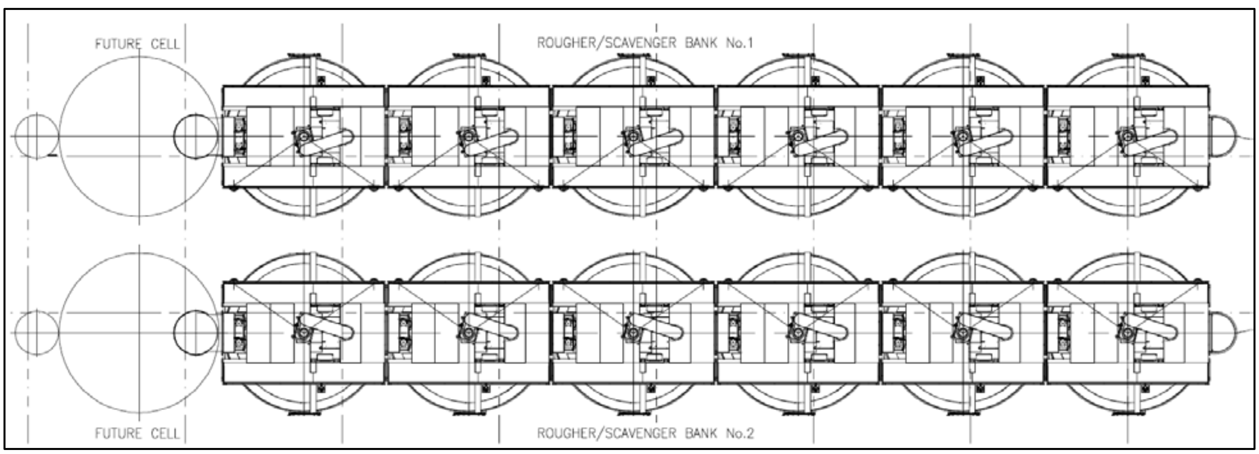
The hydrocyclone overflow will gravity flow to the flotation circuits to recover the copper minerals. The recovery process will consist of rougher/scavenger mechanical flotation, concentrate regrinding, and two stages of cleaner column flotation. A final cleaner scavenger train on the first cleaner cell will process the cleaner tails to recover residual copper minerals.

17.2.5.1 Copper Rougher/Scavenger Flotation Circuit

There will be two rougher/scavenger flotation banks taking the cyclone overflow product of each of the two ball mills. The hydrocyclone overflow will gravity flow to each of the rougher/scavenger flotation trains. The resulting scavenger tailings will be the final tailings which will gravity flow to the tailing storage facility. The rougher/scavenger concentrate will be pumped to the regrinding circuit. The rougher/scavenger flotation circuits will include twelve (12) 300m^3 rougher/scavenger flotation tank cells; six cells per each train.

Flotation reagents added to the rougher/scavenger flotation will be PAX as collector, and MIBC as a frother. Provision has been made for additional reagents should they be required.

Figure 17-4: Rougher Flotation Layout



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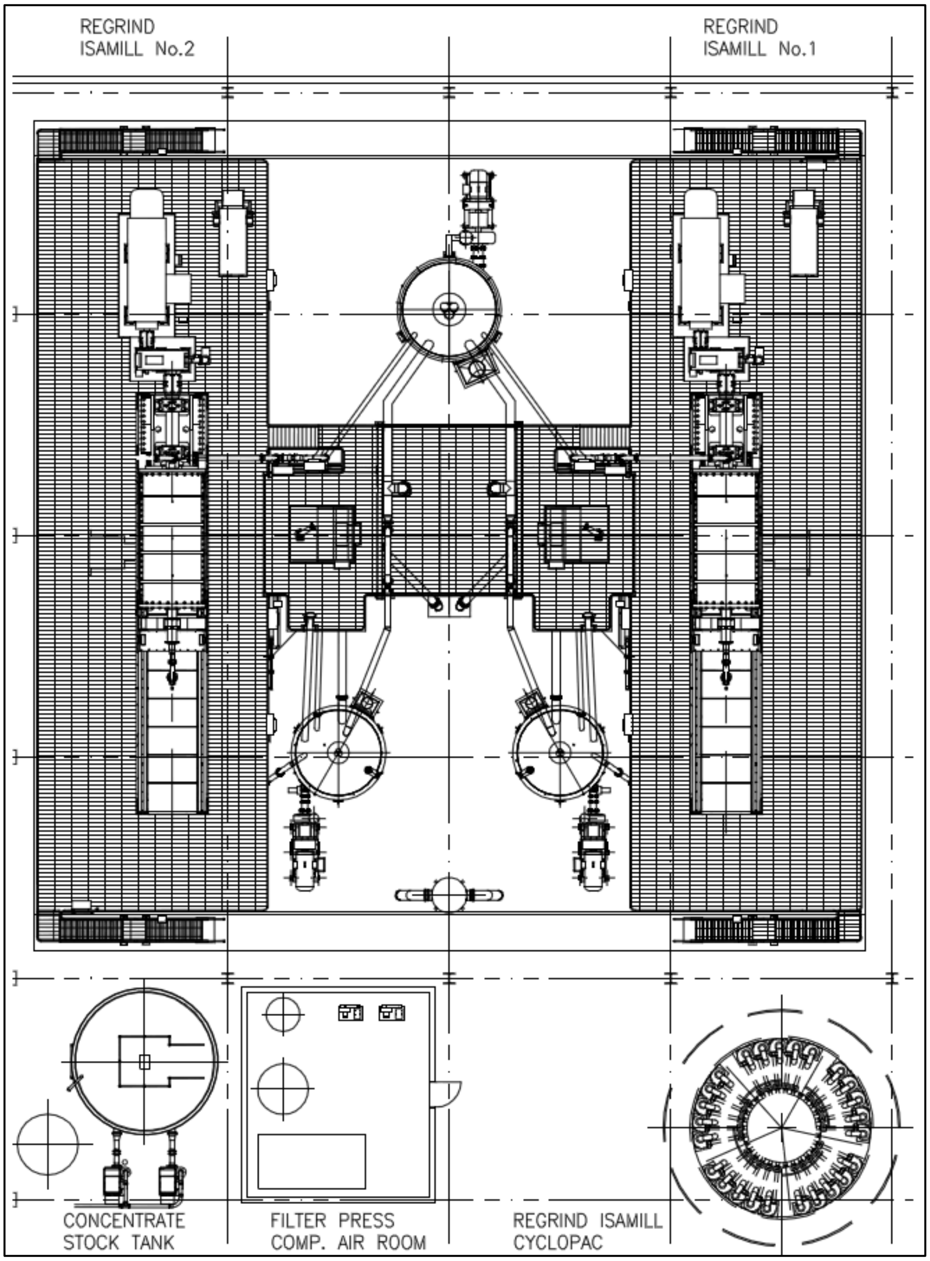
17.2.5.2 Regrind Circuit

The rougher/scavenger flotation concentrate together with the copper cleaner/scavenger concentrate and the second cleaner tails will be reground to 80% passing 20µm to 25µm to improve the liberation of the target minerals prior to the subsequent upgrading processes. The equipment used for the regrind will include:

- Two IsaMill™ M10000 horizontal grinding mills;
- One hydrocyclone cluster consisting of ten 250mm hydrocyclones, nominally for dewatering; and
- Two hydrocyclone feed pumps (one operation and one standby).

The feed slurry will be pumped from the regrind pump box through the regrind cyclone to densify the slurry with the underflow being split into two flows and then fed into the two IsaMills. The regrind cyclones are not meant for classification as this will be done within the IsaMills, however finished material from the rougher circuit will be removed in these cyclones and advanced to the cleaner circuit. Slurry lime may be added to the regrinding hydrocyclone feed pumpbox to maintain slurry pH to approximately 11 for the downstream cleaner flotation (Figure 17-5).

Figure 17-5: Regrind Area Layout



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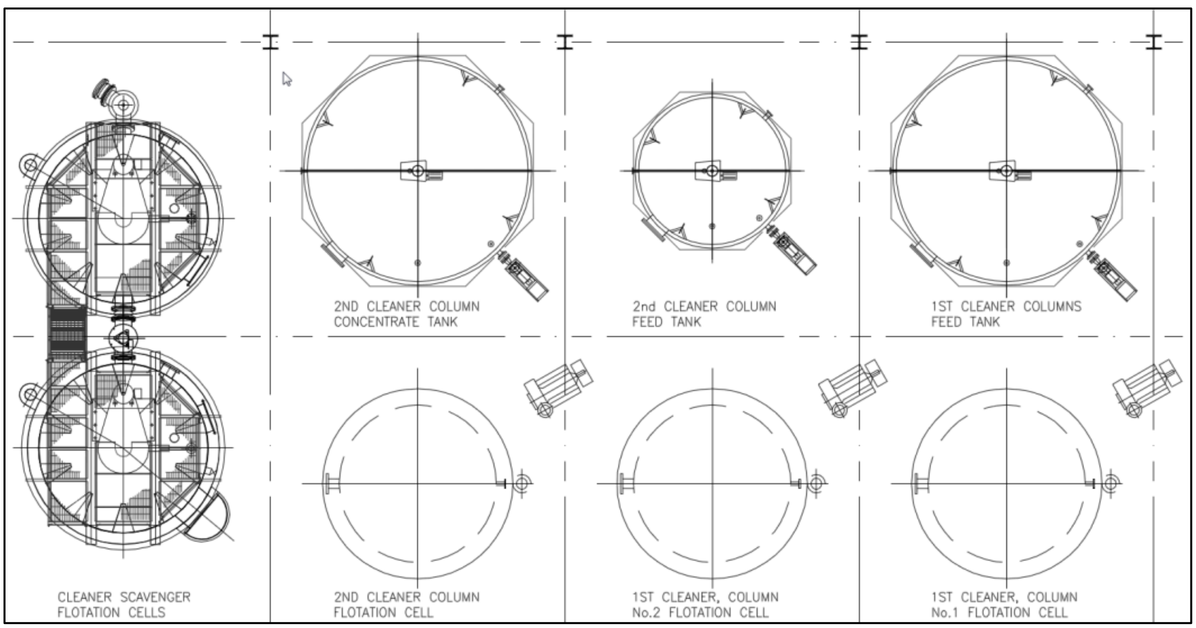
17.2.6 *CLEANER FLOTATION CIRCUIT*

The reground rougher/scavenger concentrate will be pumped to the first cleaner flotation column cells. The concentrate from the first cleaner columns will be sent to the second cleaner column. The concentrate from the second cleaner column will be the final copper concentrate and will be directly pumped to the copper concentrate thickener.

The tailings from the first cleaner will be fed through a cleaner scavenger bank of mechanical flotation cells with the concentrate being pumped back to the regrind circuit for further processing. The tailings of the second cleaner will also be pumped to the regrind circuit for further processing. The equipment used in the cleaner scavenger circuit will include:

- Two 170m^3 column cells operated in parallel (First cleaner column cells);
- One 170m^3 column cell (Second cleaner column cell); and
- Two 50m^3 tank cells (Cleaner scavenger cells).

Figure 17-6: Cleaner Flotation Area Layout



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17.2.7 CONCENTRATE DEWATERING AND HANDLING

The final cleaner flotation concentrate will be dewatered by high rate thickening and subsequently by pressure filtration, and then stockpiled prior to shipment to the smelters.

The final concentrate will be pumped to the concentrate thickener. Flocculent will be added to the thickener feed to aid the settling process. The thickened concentrate with 60% solids will be pumped to the concentrate stock tank and then be fed to the pressure filters for further dewatering. The filter cakes from the filter will contain less than 8% water and will be stockpiled before trucking to the off-site concentrate handling facility in Vavenby for shipping to the smelters.

The filtrate will be sent to the concentrate thickener as dilution water. The concentrate thickener overflow will be collected and sent to the process water pond as process make-up water. The concentrate handling facility will include the following key equipment:

- One 18m high rate thickener;
- Two 100m^2 concentrate filter presses; and
- Slurry pumps, including the high head pumps for the pressure filters.

17.2.8 TAILINGS

The rougher/scavenger and cleaner scavenger flotation tailings will be the final tailings and will be pumped separately to the TMF. The water from the pond will be reclaimed by the pumps installed on the reclaim water barge. The reclaimed water will be pumped to the process water pond for distribution to the points of usage. The tailings handling, including the tailings pond dam construction, is detailed in Section 20.

17.2.9 REAGENT HANDLING AND STORAGE

Various chemical reagents will be added to the process slurry streams to facilitate the copper flotation process. Reagents used in the process will include:

- PAX;
- Provision for second collector;
- Lime;
- MIBC;
- Flocculant; and
- Anti-scalant.

Reagent solutions will be stored in separate holding tanks and added to the addition points as required by processes using metering pumps. PAX will be added to the grinding and flotation circuits to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles into the various concentrate products. Most lime will be added dry to the SAG mill feed belt but some quicklime will be slaked on site and the lime slurry will be added to the primary grinding and regrinding circuits to depress pyrite flotation. MIBC will be used as a frother. Fresh water will be used for the preparation of the solid reagents, including PAX, lime and flocculant to the required solution strength. The strength of the diluted reagent solutions for PAX will be 20% by weight while the lime content will be 15% by weight. Each reagent will have its own preparation system, including a bulk handling system and mixing and holding tanks. A lime silo has been designed to store the lime required by the process for at least seven days. Lime will be delivered in bulk and will be off-loaded pneumatically into the silos. A lime slaking system has been provided for preparing Milk of Lime which will be pumped to the points of addition using a closed loop system. Flocculant will be prepared in the standard manner as a dilute solution of less than 0.5% solution strength for conditioning and further diluted prior to use. The liquid reagents, including MIBC, 3418A (if required) and anti-scalant, will not be diluted and will be pumped directly from the bulk containers to the points of addition using metering pumps. The mixing and holding tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, eye-wash stations, safety showers, fire and safety protection, and Material Safety Data Sheet stations will be provided at the reagent preparation areas.

17.2.10 *Assay and Metallurgical Laboratory*

The assay laboratory will be equipped with the necessary analytical instruments to provide all routine assays for the mine, the process plant, and the environmental departments. The most important of these instruments includes:

- Atomic absorption spectrophotometer (AAS);
- X-ray fluorescence spectrometer (XRF);
- Fire assay equipment;
- Sulphur and carbon determination furnace (Leco); and
- ICP_MS.

The metallurgical laboratory will undertake all necessary test work to monitor metallurgical performance and, more importantly, to improve process flowsheet unit operations and efficiencies. The laboratory will be equipped with laboratory crushers, ball and stirred mills, particle size analyzers, test sieves and shakers, flotation cell filtering and settling equipment, balances, and pH and Oxidation Reduction Potential meters.

17.2.11 *Water Supply*

Two separate water supply systems for fresh water and process water will support the operation. Fresh water required for process will be sourced from the TMF and sent to a fresh/fire water storage tank. Potable fresh water will be supplied from local wells. Fresh water will be used primarily for the following:

- Fire water for emergency use;
- Cooling water for mill motors and mill lubrication systems;
- Gland seal service for the slurry pumps;
- Reagent preparation;
- Process water make-up; and
- Potable water supply.

Fresh water for use in the process and for fire will be pre-treated to ensure suitability. The fresh/fire water tank will be equipped with a standpipe, which will ensure that the tank is always holding at least a two-hour supply of firewater.

The potable water from the fresh water source will be treated and stored in the potable water storage tank prior to delivery to various service points.

All process water will be distributed to the plant site from the process water pond. The majority of the process water will be reclaimed water from the tailings pond including the water from the proposed mine sites (pit water and run-off water). The balance of the required process water will be from the fresh water tank. The concentrate thickener overflow will be reused in the process circuit.

17.2.12 *PROCESSING EQUIPMENT*

Major equipment incorporated into the process circuit is summarized in Table 17-2:

Table 17-2 Processing Equipment Summary

Equipment	Specifications
Gyratory Crusher	Primary Crushing
	Size: 60” x 89”
	Power: 1,000kW
SAG Mill	Primary Grinding Mill
	Size 38’ x 22’
	Power: 21MW
Ball Mills	Secondary Grinding Mills
	Size: 24’ x 42’
	Power: 13MW each
Flotation Cells	2 Banks x 6 cells
	Size: 300m^3 each
Regrind	2 x IsaMill™
	Size: M10,000
	Power: 3MW each
Cleaner Circuit	1st Cleaner: 2 x Column Cells
	Size: 170m^3 each
	2nd Cleaner
	Size: 170m^3 each

A complete equipment list is provided in Section 17.5.

17.3 ENERGY, WATER AND PROCESS MATERIALS

17.3.1 *ENERGY LOAD*

The electrical power requirement of the major process areas is provided in Table 17-3. The grinding circuit will consume the largest proportion of energy. The annual power consumption (MW/a) is based on the mine operating 24/7 and with an availability of 92%.

Table 17-3: Energy Requirements by Area (Allnorth)

Description	Connected Load (kW)	Maximum Demand (kW)	Average Demand (kW)	Annual Consumption (MW h/a)	% of Total
RECLAIM WATER (TAILINGS POND)	5,635	4,508	3,606	29,061	4.4
WATER COLLECTION (SITE RUNOFF)	3,015	2,412	1,930	15,554	2.4
MINING EQUIPMENT	15,160	12,127	9,702	78,190	11.8
PIT DEWATERING	1,890	1,512	1,210	9,752	1.5
CRUSHING	1,334	1,067	853	6,874	1.0
CONVEYOR , STOCKPILE RECLAIM	5,412	4,315	3,452	27,820	4.2
CONCENTRATOR BUILDING	1,473	1,271	1,001	8,067	1.2
GRINDING	63,802	61,303	48,552	391,290	58.8
AIR SYSTEM	5,079	4,063	3,250	26,192	4.0
FLOTATION	4,063	3,251	2,600	20,954	3.2
REGRIND	7,586	6,057	4,845	39,047	5.9
CONCENTRATE THICKENING & LOADOUT	565	452	362	2,917	0.4
CONCENTRATE REAGENTS	266	212	170	1,370	0.2
TRUCK SHOP	516	412	314	2,531	0.4
TAILINGS PUMPING	714	572	457	3,683	0.6
TOTAL	**116,510**	**103,534**	**82,304**	**663,302**	**100**

17.4 INSTRUMENTATION & CONTROL SYSTEM

17.4.1 *CONTROL SYSTEM*

The process control system will be a PLC based system. The PLC will be used to control and monitor all the operations of the plant. The plant is broken into different process areas. Each process area is controlled by a single PLC system. The PLC will be tied together to form a plant wide control system by the use of an Ethernet communication system. The motor starters, VFDs and some of the field devices will be controlled by the PLC via a Devicenet communication system. Process control and monitoring for the facility will be performed in a two operator control room utilizing Graphical Operator stations. These stations will contain the graphical representation of the process equipment and will interface to the PLC via the ethernet network. There will be two operator control rooms located in the Primary Crusher Area and the Concentrator Building.

17.4.2 *Primary Crushing Area*

A Graphical Operator station will be installed in the primary crushing control room. This station will be used to control and monitor all crushing and conveying from the primary crusher to the coarse ore stock pile. The primary crushing graphical screens can also be viewed in the main control room in the Concentrator building.

17.4.3 *Concentrator*

There will be four Graphical Operator stations located in the concentrator control room and will be used to monitor and control the following process areas:

- Grinding Area;
- Flotation Area;
- Regrind Area; and
- Thickener and Concentrator Area.

17.4.4 *Process Communication System*

The process communication system will be a 1000/100 based Ethernet system. All PLC and Graphical Operator stations will be connected to the Ethernet system. There will be a plant wide fiber optic backbone interconnecting the PLC and the Graphical systems.

17.5 EQUIPMENT LIST

Table 17-4: FS Equipment List

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
2000			
2000	PLANT PROCESS WATER POND	HOLD THIS ENTIRE AREA	
2000	PLANT PROCESS WATER PUMP No.1		
2000	PLANT PROCESS WATER PUMP No.1		500
2000	PLANT PROCESS WATER PUMP No.2		
2000	PLANT PROCESS WATER PUMP No.2		500
2000	PLANT PROCESS WATER PUMP No.3		
2000	PLANT PROCESS WATER PUMP No.3		500
2000	PLANT PROCESS WATER PUMP No.4		
2000	PLANT PROCESS WATER PUMP No.4		500
2000	WASHDOWN BOOSTER PUMP		
2000	WASHDOWN BOOSTER PUMP		75
2000	FRESH WATER TANK	2600 m3, 600 m3 Fire water	
2000	FRESH WATER PUMP No.1		
2000	FRESH WATER PUMP No.1		250
2000	FRESH WATER PUMP No.2		
2000	FRESH WATER PUMP No.2		250
2000	POTABLE WATER WELL PUMP No.1	Vertical pump	
2000	POTABLE WATER WELL PUMP No.1		10
2000	POTABLE WATER WELL PUMP No.2	Vertical pump	
2000	POTABLE WATER WELL PUMP No.2		10
2000	POTABLE WATER TANK		
2000	POTABLE WATER SUPPLY PUMP No.1		
2000	POTABLE WATER SUPPLY PUMP No.1		10
2000	POTABLE WATER SUPPLY PUMP No.2		
2000	POTABLE WATER SUPPLY PUMP No.2		10
			2615
2010			
2010	FIRE WATER PUMP No.1	ELECTRIC	
2010	FIRE WATER PUMP No.1	ELECTRIC	75
2010	FIRE WATER PUMP No.2	DIESEL	
2010	FIRE WATER PUMP No.2	DIESEL	115
2010	FIRE WATER JOCKEY PUMP No.1		
2010	FIRE WATER JOCKEY PUMP No.1		20
2010	FIRE WATER JOCKEY PUMP No.2		
2010	FIRE WATER JOCKEY PUMP No.2		20
			230
2100			
2100	PLANT AIR COMPRESSOR No. 1	HOLD THIS ENTIRE AREA	
2100	PLANT AIR COMPRESSOR No. 1	HP'S TO COME	400
2100	PLANT AIR COMPRESSOR No. 2		
2100	PLANT AIR COMPRESSOR No. 2	HP'S TO COME	400
2100	PLANT AIR COMPRESSOR No. 3		
2100	PLANT AIR COMPRESSOR No. 3	HP'S TO COME	400
2100	PLANT AIR COMPRESSOR No. 4		
2100	PLANT AIR COMPRESSOR No. 4	Standby HP'S TO COME	400
2100	PLANT AIR RECEIVER No.1	To Concentrate filters	
2100	PLANT AIR RECEIVER No.2	To Concentrate filters	
2100	PLANT AIR RECEIVER No.3	To plant air loop	
2100	INSTRUMENT AIR FILTER		
2100	INSTRUMENT SURGE AIR RECEIVER		
2100	INSTRUMENT AIR DRYER		

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
2100	INSTRUMENT DRY AIR RECEIVER	To plant instruments	
			1600
2200			
2200	FLOTATION BUILDING SEWAGE SYSTEM	RBC UNIT	3
2200	SEWAGE INFLUENT GRINDER PUMP No.1		2
2200	SEWAGE INFLUENT GRINDER PUMP No.2		2
			7
2300			
2300	DIESEL STORAGE TANK	75,000 Liters	
2300	DIESEL STORAGE TANK	75,000 Liters	
2300	DIESEL STORAGE TANK	75,000 Liters	
2300	DIESEL STORAGE TANK	75,000 Liters	
2300	DIESEL DISPENSING PUMP		5
2300	DIESEL DISPENSING PUMP		5
2300	LIGHT VEIHCLE DIESEL DISPENSING PUMP		5
2300	GASOLINE STORAGE TANK		
2300	GASOLINE DISPENSING PUMP		5
2300	TRUCK SHOP REFUELING PAD OIL/WATER SEPARATOR		
			20
3000			
3000	ROCK BREAKER	12.5 m Reach	
3000	ROCK BRAKER HYDRAULIC UNIT		
3000	ROCK BRAKER HYDRAULIC UNIT PUMP		150
3000	ROCK BRAKER HYDRAULIC UNIT TANK HEATER		6.7
3000	ROCK BRAKER HYDRAULIC UNIT HEAT EXCHANGER		4
3000	DUMP POCKET No.1	500m3	
3000	DUMP POCKET No.2		
3000	PRIMARY CRUSHER	60" x 89"	
3000	PRIMARY CRUSHER		1349
3000	PRIMARY CRUSHER DUST SEAL BLOWER		1.5
3000	PRIMARY CRUSHER DUST COLLECTOR		
3000	PRIMARY CRUSHER DUST COLLECTOR MOTOR No.1		30
3000	PRIMARY CRUSHER DUST COLLECTOR MOTOR No.2		30
3000	PRIMARY CRUSHER LUBE UNIT		
3000	PRIMARY CRUSHER SPIDER LUBE UNIT		
3000	PRIMARY CRUSHER SPIDER LUBE UNIT PUMP		2
3000	PRIMARY CRUSHER LUBE UNIT PUMP		7.5
3000	PRIMARY CRUSHER LUBE UNIT PUMP		7.5
3000	PRIMARY CRUSHER LUBE UNIT PUMP		7.5
3000	PRIMARY CRUSHER LUBE UNIT PUMP		7.5
3000	PRIMARY CRUSHER POSITIONING SYSTEM		
3000	PRIMARY CRUSHER POSITIONING SYSTEM PUMP		7.5
3000	PRIMARY CRUSHER LUBE UNIT TANK HEATER		2.7
3000	PRIMARY CRUSHER LUBE UNIT TANK HEATER		2.7
3000	PRIMARY CRUSHER LUBE UNIT TANK HEATER		2.7
3000	PRIMARY CRUSHER LUBE UNIT TANK HEATER		2.7
3000	PRIMARY CRUSHER LUBE UNIT TANK HEATER		2.7
3000	PRIMARY CRUSHER LUBE UNIT TANK HEATER		2.7
3000	PRIMARY CRUSHER POSITIONING SYSTEM TANK HEATER		6.7
3000	PRIMARY CRUSHER LUBE UNIT HEAT EXCHANGER		15
3000	PRIMARY CRUSHER LUBE UNIT HEAT EXCHANGER		20
3000	PRIMARY CRUSHER DISCHARGE BIN	(2) Trucks, 720 t, 520 m3	
3000	PRIMARY CRUSHER DISCHARGE CHUTE		
3000	PRIMARY CRUSHER DISCHARGE FEEDER FEED CHUTE		
3000	PRIMARY CRUSHER DISCHARGE APRON FEEDER	2.4m (94") x 8.6 m	250
3000	PRIMARY CRUSHER DISCHARGE FEEDER DRIBBLE CHUTE		
3000	PRIMARY CRUSHER DISCHARGE FEEDER HEAD CHUTE		
3000	PRIMARY CRUSHER CRANE	Mobile Crane	
3000	PRIMARY CRUSHER DRIVE MONORAIL HOIST	5 t	7.5
3000	PRIMARY CRUSHER DISCHARGE FEEDER DRIVE MONORAIL HOIST	7.5 t	5
3000	PRIMARY CRUSHER DISCHARGE FEEDER TAIL MONORAIL HOIST	5 t	5

YMIHPC-001	Equipment Name	Issued for Feasibility Study	
AREA		Description / Performance	Power
		/Specification	HP
3000	PRIMARY CRUSHER ECCENTRIC REMOVAL CART		
3000	PRIMARY CRUSHER ECCENTRIC REMOVAL CART TROLLEY DRIVE		13.4
3000	PRIMARY CRUSHER AREA COMPRESSOR	200 CFM @ 100 PSI	
3000	PRIMARY CRUSHER AREA COMPRESSOR		40
3000	PRIMARY CRUSHER AREA AIR DRYER	HOLD	13.4
3000	PRIMARY CRUSHER AREA AIR RECEIVER		
3000	PRIMARY CRUSHER AREA SUMP PUMP		
3000	PRIMARY CRUSHER AREA SUMP PUMP		30
3000	HVAC - PRIMARY CRUSHER CONTROL ROOM		13.4
3000	HVAC - PRIMARY CRUSHER ELECTRICAL ROOM		26.8
			1524.1
3100			
3100	COARSE ORE STOCKPILE OVERLAND FEED CONVEYOR FEED CHUTE		
3100	COARSE ORE STOCKPILE OVERLAND FEED CONVEYOR	84" 35 Degree Idler	
3100	COARSE ORE STOCKPILE OVERLAND FEED CONVEYOR DRIVE		125
3100	COARSE ORE STOCKPILE OVERLAND FEED CONVEYOR MAGNET		
3100	COARSE ORE STOCKPILE OVERLAND FEED CONVEYOR RECTIFIER		10
3100	COARSE ORE STOCKPILE OVERLAND FEED CONVEYOR BELT SCALE		
3100	COARSE ORE STOCKPILE OVERLAND FEED CONVEYOR METAL DETECTOR		
3100	COARSE ORE STOCKPILE OVERLAND FEED CONVEYOR RIP DETECTION		
3100	COARSE ORE STOCKPILE OVERLAND CONVEYOR	72" 35 Degree Idler	
3100	COARSE ORE STOCKPILE OVERLAND CONVEYOR DRIVE No.1		2000
3100	COARSE ORE STOCKPILE OVERLAND CONVEYOR DRIVE No.2		2000
3100	COARSE ORE STOCKPILE OVERLAND CONVEYOR DRIVE No.3		2000
			6135
3110			
3110	RECLAIM APRON FEEDER No.1 FEED CHUTE		
3110	RECLAIM APRON FEEDER No.1	1.8 m x 6.1 m	100
3110	RECLAIM APRON FEEDER No.1 DISCHARGE CHUTE		
3110	RECLAIM APRON FEEDER 1 DRIBBLE CHUTE		
3110	RECLAIM APRON FEEDER No.2 FEED CHUTE		
3110	RECLAIM APRON FEEDER No.2	1.8 m x 6.1 m	100
3110	RECLAIM APRON FEEDER No.2 DISCHARGE CHUTE		
3110	RECLAIM APRON FEEDER 2 DRIBBLE CHUTE		
3110	RECLAIM APRON FEEDER No.3 FEED CHUTE		
3110	RECLAIM APRON FEEDER No.3	1.8 m x 6.1 m	100
3110	RECLAIM APRON FEEDER No.3 DISCHARGE CHUTE		
3110	RECLAIM APRON FEEDER 3 DRIBBLE CHUTE		
3110	SAG MILL FEED CONVEYOR	60" 35 Degree Idler	
3110	SAG MILL FEED CONVEYOR DRIVE		800
3110	SAG MILL FEED CONVEYOR BELT SCALE		
3110	SAG MILL FEED CONVEYOR HEAD CHUTE		
3110	SAG MILL BALL CHUTE		
3110	SAG MILL BALL FEEDER		5.4
3110	RECLAIM TUNNEL VENTILATION FAN No.1		13.4
3110	RECLAIM TUNNEL VENTILATION FAN No.2		13.4
3110	ESCAPE TUNNEL VENTILATION FAN		13.4
3110	RECLAIM TUNNEL UNIT HEATER No.1		67
3110	RECLAIM TUNNEL UNIT HEATER No.2		67
3110	RECLAIM TUNNEL SUMP PUMP No.1		30
3110	RECLAIM TUNNEL SUMP PUMP No.2	HOLD	30
3110	RECLAIM APRON FEEDER No.1 HOIST		2.7
3110	RECLAIM APRON FEEDER No.2 HOIST		2.7
3110	RECLAIM APRON FEEDER No.3 HOIST		2.7
3110	RECLAIM AREA BAGHOUSE	10,000 CFM	
3110	RECLAIM AREA BAGHOUSE		30
3110	RECLAIM AREA BAGHOUSE		30
			1407.7
3400			
3400	SEAL WATER BOOSTER PUMP No.1		
3400	SEAL WATER BOOSTER PUMP No.1		10
3400	SEAL WATER BOOSTER PUMP No.2		
3400	SEAL WATER BOOSTER PUMP No.2		10
3400	GRINDING AREA SUMP PUMP No.1		

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
3400	GRINDING AREA SUMP PUMP No.1		101
3400	GRINDING AREA SUMP PUMP No.2		
3400	GRINDING AREA SUMP PUMP No.2		101
			222
3410			
3410	SAG MILL	38'D x 22' EGL (11.6x6.7 m EGL)	
3410	SAG MILL RING MOTOR		28161
3410	SAG MILL RING MOTOR COOLING FAN No.1		15
3410	SAG MILL RING MOTOR COOLING FAN No.2		15
3410	SAG MILL RING MOTOR COOLING FAN No.3		15
3410	SAG MILL RING MOTOR COOLING FAN No.4		15
3410	SAG MILL RING MOTOR COOLING FAN No.5		15
3410	SAG MILL RING MOTOR COOLING FAN No.6		15
3410	SAG MILL RING MOTOR COOLING FAN No.7		15
3410	SAG MILL RING MOTOR COOLING FAN No.8		15
3410	SAG MILL RING MOTOR WINDING HEATER		15
3410	SAG MILL FEED CHUTE		
3410	SAG MILL FEED CHUTE WINCH		13.4
3410	SAG MILL FEED CHUTE TRAVERSE WINCH		13.4
3410	SAG MILL LUBRICATION UNIT		
3410	SAG MILL LUBRICATION UNIT HP PUMP No.1		150
3410	SAG MILL LUBRICATION UNIT HP PUMP No.2		150
3410	SAG MILL LUBRICATION UNIT HP PUMP No.3		150
3410	SAG MILL LUBRICATION UNIT LP PUMP No.1		50
3410	SAG MILL LUBRICATION UNIT LP PUMP No.2		50
3410	SAG MILL LUBRICATION UNIT TANK HEATER No.1		13.4
3410	SAG MILL LUBRICATION UNIT TANK HEATER No.2		13.4
3410	SAG MILL LUBRICATION UNIT TANK HEATER No.3		13.4
3410	SAG MILL LUBRICATION UNIT TANK HEATER No.4		13.4
3410	SAG MILL LUBRICATION UNIT HEAT EXCHANGER		20
3410	SAG MILL RING MOTOR LUBRICATION UNIT		
3410	SAG MILL RING MOTOR LUBRICATION UNIT HP PUMP No.1		13.4
3410	SAG MILL RING MOTOR LUBRICATION UNIT HP PUMP No.2		13.4
3410	SAG MILL RING MOTOR LUBRICATION UNIT HP PUMP No.3		13.4
3410	SAG MILL RING MOTOR LUBRICATION UNIT HP PUMP No.4		13.4
3410	SAG MILL RING MOTOR LUBRICATION UNIT TANK HEATER		2.7
3410	SAG MILL RING MOTOR SPACE HEATER No.1		4
3410	SAG MILL RING MOTOR SPACE HEATER No.2		4
3410	SAG MILL RING MOTOR FILTER VENTILATION FAN		29.5
3410	SAG MILL BRAKE No.1		
3410	SAG MILL BRAKE No.2		
3410	SAG MILL JACKING CRADLE		
3410	SAG MILL PORTABLE JACKING SYSTEM		
3410	SAG MILL DISCHARGE TROMMEL SCREEN		
3410	SAG MILL DISCHARGE HOOD/CHUTE		
3410	SAG MILL DISCHARGE SCREEN PEBBLE CHUTE		
3410	CHILLED WATER SUPPLY TANK		
3410	CHILLED WATER PUMP No.1		5.36
3410	CHILLED WATER PUMP No.2		5.36
3410	CHILLED WATER PUMP No.3		5.36
3410	CHILLED WATER PUMP No.4		5.36
3410	SAG MILL CHILLER No.1		54
3410	SAG MILL CHILLER No.2		54
3410	SAG MILL LINER HANDLER	RME 7 Axis 6t Cap: 13m Lg.	50
3410	SAG MILL LINER HANDLER RECOILLESS HAMMER		
3410	SAG MILL RELINE HOIST	7.5t	3.7
3410	SAG MILL RECOILLESS HAMMER HOIST	1t	5.4
3410	SAG MILL HYTORC WRENCH		
			27877.74
3420			
3420	SAG MILL DISCHARGE VIBRATING SCREEN	Double Deck 14x32.8' (4.3x10 m)	75

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance /Specification	Power HP
3420	SAG MILL SPARE DISCHARGE SCREEN	Double Deck 14x32.8' (4.3x10 m)	
3420	VIBRATING SCREEN MAGNET		5
3420	VIBRATING SCREEN MAGNET RECTIFIER		
3420	VIBRATING SCREEN OVERSIZE CHUTE		
3420	SAG MILL DISCHARGE PUMPBOX	230 m3	
3420	SAG MILL DISCHARGE PUMP No.1		
3420	SAG MILL DISCHARGE PUMP No.1		1500
3420	SAG MILL DISCHARGE PUMP No.2		
3420	SAG MILL DISCHARGE PUMP No.2	Standby	1500
3420	SCREEN OVERS CONVEYOR	36" 35 Degree Idler	
3420	SCREEN OVERS CONVEYOR		40
			3120
3440			
3440	BALL MILL FEED DISTRIBUTOR		
3440	BALL MILL No.1 FEED LAUNDER		
3440	BALL MILL No.1 FEED CHUTE		
3440	BALL MILL No.1 FEED CHUTE WINCH		5
3440	BALL MILL FEED CHUTE PORTABLE HYDRAULIC UNIT		5
3440	BALL MILL No.1 BALL ADDITION HOPPER & CHUTE		
3440	BALL MILL No.1	24'Dx42' EGL (7.3x12.8 m EGL)	
3440	BALL MILL No.1 MILL MOTOR No.1	c/w Inching Drive	8713
3440	BALL MILL No.1 MILL MOTOR No.2	c/w Inching Drive	8713
3440	BALL MILL No.1 GEAR SPRAY SYSTEM	Air Operated	
3440	BALL MILL No.1 LUBRICATION UNIT		
3440	BALL MILL No.1 LUBRICATION UNIT HP PUMP No.1		150
3440	BALL MILL No.1 LUBRICATION UNIT HP PUMP No.2		150
3440	BALL MILL No.1 LUBRICATION UNIT HP PUMP No.3		150
3440	BALL MILL No.1 LUBRICATION UNIT LP PUMP No.1		50
3440	BALL MILL No.1 LUBRICATION UNIT LP PUMP No.2		50
3440	BALL MILL No.1 LUBRICATION UNIT TANK HEATER No.1		13.4
3440	BALL MILL No.1 LUBRICATION UNIT TANK HEATER No.2		13.4
3440	BALL MILL No.1 LUBRICATION UNIT TANK HEATER No.3		13.4
3440	BALL MILL No.1 LUBRICATION UNIT TANK HEATER No.4		13.4
3440	BALL MILL No.1 LUBRICATION UNIT HEAT EXCHANGER		20
3440	BALL MILL No.1 MILL MOTOR LUBRICATION UNIT		
3440	BALL MILL No.1 MILL MOTOR LUBRICATION UNIT HP PUMP No.1		13.4
3440	BALL MILL No.1 MILL MOTOR LUBRICATION UNIT HP PUMP No.2		13.4
3440	BALL MILL No.1 MILL MOTOR LUBRICATION UNIT HP PUMP No.3		13.4
3440	BALL MILL No.1 MILL MOTOR LUBRICATION UNIT HP PUMP No.4		13.4
3440	BALL MILL No.1 MILL MOTOR LUBRICATION UNIT TANK HEATER		2.7
3440	BALL MILL No.1 MILL MOTOR SPACE HEATER No.1		4
3440	BALL MILL No.1 MILL MOTOR SPACE HEATER No.2		4
3440	BALL MILL No.1 MILL MOTOR FILTER VENT FAN		29.5
3440	BALL MILL No.1 BRAKE No.1		
3440	BALL MILL No.1 BRAKE No.2		
3440	BALL MILL No.1 CLUTCH No.1	Required? Depends on vendor	
3440	BALL MILL No.1 CLUTCH No.2	Required? Depends on vendor	
3440	BALL MILL No.1 CLUTCH AIR CONTROL SKID	Required? Depends on vendor	
3440	BALL MILL No.1 CLUTCH AIR RECEIVER No.1	Required? Depends on vendor	
3440	BALL MILL No.1 CLUTCH AIR RECEIVER No.2	Required? Depends on vendor	
3440	BALL MILL No.1 JACKING CRADLE		
3440	BALL MILL No.1 PORTABLE JACKING SYSTEM		
3440	BALL MILL No.1 DISCHARGE TROMMEL SCREEN		
3440	BALL MILL No.1 DISCHARGE HOOD & CHUTE		
3440	BALL MILL No. 1 DISCHARGE MAGNET		
3440	BALL MILL No.1 SCATS CHUTE		
3440	BALL MILL No.1 DISCHARGE PUMPBOX	230 m3	

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
3440	BALL MILL No.1 CYCLONE FEED PUMP No.1	__" x __"	
3440	BALL MILL No.1 CYCLONE FEED PUMP No.1		3000
3440	BALL MILL No.1 CYCLONE FEED PUMP No.2	__" x __"	
3440	BALL MILL No.1 CYCLONE FEED PUMP No.2		3000
3440	BALL MILL No.1 CHILLER No.1		54
3440	BALL MILL No.1 CHILLER No.2		54
3440	BALL MILL No.1 CYCLOPAC	0.8 m cyclones - Confirm	
3440	BALL MILL No.1 CYCLONE No.101		
3440	BALL MILL No.1 CYCLONE No.102		
3440	BALL MILL No.1 CYCLONE No.103		
3440	BALL MILL No.1 CYCLONE No.104		
3440	BALL MILL No.1 CYCLONE No.105		
3440	BALL MILL No.1 CYCLONE No.106		
3440	BALL MILL No.1 CYCLONE No.107		
3440	BALL MILL No.1 CYCLONE No.108		
3440	BALL MILL No.1 CYCLONE No.109		
3440	BALL MILL No.1 CYCLONE No.110		
3440	BALL MILL No.1 CYCLOPAC UNDERFLOW LAUNDER		
3440	BALL MILL No.1 CYCLOPAC OVERFLOW LAUNDER		
3440	BALL MILL No.2 FEED LAUNDER		
3440	BALL MILL No.2 FEED CHUTE		
3440	BALL MILL No.2 FEED CHUTE WINCH		5
3440	BALL MILL No.2	24'Dx42' EGL (7.3x12.8 m EGL)	
3440	BALL MILL No.2 MILL MOTOR No.1	c/w Inching Drive	8713
3440	BALL MILL No.2 MILL MOTOR No.2	c/w Inching Drive	8713
3440	BALL MILL No.2 GEAR SPRAY SYSTEM	Air Operated	
3440	BALL MILL No.2 LUBRICATION UNIT		
3440	BALL MILL No.2 LUBRICATION UNIT HP PUMP No.1		150
3440	BALL MILL No.2 LUBRICATION UNIT HP PUMP No.2		150
3440	BALL MILL No.2 LUBRICATION UNIT HP PUMP No.3		150
3440	BALL MILL No.2 LUBRICATION UNIT LP PUMP No.1		50
3440	BALL MILL No.2 LUBRICATION UNIT LP PUMP No.2		50
3440	BALL MILL No.2 LUBRICATION UNIT TANK HEATER No.1		13.4
3440	BALL MILL No.2 LUBRICATION UNIT TANK HEATER No.2		13.4
3440	BALL MILL No.2 LUBRICATION UNIT TANK HEATER No.3		13.4
3440	BALL MILL No.2 LUBRICATION UNIT TANK HEATER No.4		13.4
3440	BALL MILL No.2 LUBRICATION UNIT HEAT EXCHANGER		20
3440	BALL MILL No.2 MILL MOTOR LUBRICATION UNIT		
3440	BALL MILL No.2 MILL MOTOR LUBRICATION UNIT HP PUMP No.1		13.4
3440	BALL MILL No.2 MILL MOTOR LUBRICATION UNIT HP PUMP No.2		13.4
3440	BALL MILL No.2 MILL MOTOR LUBRICATION UNIT HP PUMP No.3		13.4
3440	BALL MILL No.2 MILL MOTOR LUBRICATION UNIT HP PUMP No.4		13.4
3440	BALL MILL No.2 MILL MOTOR LUBRICATION UNIT TANK HEATER		2.7
3440	BALL MILL No.2 MILL MOTOR SPACE HEATER No.1		4
3440	BALL MILL No.2 MILL MOTOR SPACE HEATER No.2		4
3440	BALL MILL No.2 MILL MOTOR FILTER VENT FAN		29.5
3440	BALL MILL No.2 BRAKE No.1		
3440	BALL MILL No.2 BRAKE No.2		
3440	BALL MILL No.2 CLUTCH No.1	Required? Depends on vendor	
3440	BALL MILL No.2 CLUTCH No.2	Required? Depends on vendor	
3440	BALL MILL No.2 CLUTCH AIR CONTROL SKID	Required? Depends on vendor	
3440	BALL MILL No.2 CLUTCH AIR RECEIVER No.1	Required? Depends on vendor	
3440	BALL MILL No.2 CLUTCH AIR RECEIVER No.2	Required? Depends on vendor	
3440	BALL MILL No.2 JACKING CRADLE		
3440	BALL MILL No.2 PORTABLE JACKING SYSTEM		
3440	BALL MILL No.2 DISCHARGE TROMMEL SCREEN		
3440	BALL MILL No.2 DISCHARGE HOOD & CHUTE		

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
3440	BALL MILL No. 2 DISCHARGE MAGNET		
3440	BALL MILL No.2 SCATS CHUTE		
3440	BALL MILL No.2 DISCHARGE PUMPBOX	230 m3	
3440	BALL MILL No.2 CYCLONE FEED PUMP No.1	__" x __"	
3440	BALL MILL No.2 CYCLONE FEED PUMP No.1		3000
3440	BALL MILL No.2 CYCLONE FEED PUMP No.2	__" x __"	
3440	BALL MILL No.2 CYCLONE FEED PUMP No.2		3000
3440	BALL MILL No.2 CHILLER No.1		54
3440	BALL MILL No.2 CHILLER No.2		54
3440	BALL MILL No.2 CYCLOPAC	0.8 m cyclones - Confirm	
3440	BALL MILL No.2 CYCLONE No.201		
3440	BALL MILL No.2 CYCLONE No.202		
3440	BALL MILL No.2 CYCLONE No.203		
3440	BALL MILL No.2 CYCLONE No.204		
3440	BALL MILL No.2 CYCLONE No.205		
3440	BALL MILL No.2 CYCLONE No.206		
3440	BALL MILL No.2 CYCLONE No.207		
3440	BALL MILL No.2 CYCLONE No.208		
3440	BALL MILL No.2 CYCLONE No.209		
3440	BALL MILL No.2 CYCLONE No.210		
3440	BALL MILL No.2 CYCLOPAC UNDERFLOW LAUNDER		
3440	BALL MILL No.2 CYCLOPAC OVERFLOW LAUNDER		
3440	BALL MILLS LINER HANDLER	RME 7 Axis 2.5t Cap: 13.7m Lg.	50
3440	BALL MILLS LINER HANDLER RECOILLESS HAMMER		
3440	BALL MILLS RELINE HOIST	7.5t	3.7
3440	BALL MILLS RECOILLESS HAMMER HOIST	1t	5.4
3440	BALL MILLS HYTORC WRENCH		
			48576.9
3450			
3450	SAG MILL BALL BIN		
3450	BALL MILL BALL BIN		
3450	BALL MILL BALL BIN MAGNET	Suspended Electromagnet	
3450	BALL MILL BALL KIBBLE		
3500			
3500	FLOTATION BLOWER No.1		
3500	FLOTATION BLOWER No.1		800
3500	FLOTATION BLOWER No.2		
3500	FLOTATION BLOWER No.2		800
3500	FLOTATION BLOWER No.3		
3500	FLOTATION BLOWER No.3		800
3500	SEAL WATER BOOSTER PUMP No.3		
3500	SEAL WATER BOOSTER PUMP No.3		10
3500	SEAL WATER BOOSTER PUMP No.4		
3500	SEAL WATER BOOSTER PUMP No.4		10
3500	SEAL WATER BOOSTER PUMP No.5		
3500	SEAL WATER BOOSTER PUMP No.5		10
3500	SEAL WATER BOOSTER PUMP No.6		
3500	SEAL WATER BOOSTER PUMP No.6		10
3500	LOW GRADE MULTIPLEXER	6 Stream	
3500	LOW GRADE MULTIPLEXER	6 Stream (FUTURE)	
3500	HIGH GRADE MULTIPLEXER	6 Stream	
3500	DEMULTIPLEXER		
3500	ON STREAM ANALYZER		
3500	PARTICLE SIZE ANALYZER	HOLD	
3500	REGRIND PARTICLE SIZE MONITOR		
3500	ANALYZER FEED PUMP		
3500	ANALYZER FEED PUMP		5
3500	HIGH GRADE SAMPLE RETURN PUMP		
3500	HIGH GRADE SAMPLE RETURN PUMP		5
			2450
3510			

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
3510	FLOTATION FEED No.1 PRIMARY SAMPLER	Ball Mill No.1	
3510	FLOTATION FEED No.2 PRIMARY SAMPLER	Ball Mill No.2	
3510	FLOTATION CELLS DISTRIBUTOR		
3510	ROUGHER BANK No.1 FEED LAUNDER		
3510	ROUGHER BANK No.2 FEED LAUNDER		
3510	ROUGHER/SCAVENGER BANK No.1		
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.11	300 m3	
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.11 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.12	300 m3	
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.12 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.13	300 m3	
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.13 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.14	300 m3	
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.14 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.15	300 m3	
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.15 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.16	300 m3	
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.16 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.17	300 m3, FUTURE	
3510	ROUGHER/SCAVENGER BANK No.1 FLOTATION CELL No.17 ROTOR	FUTURE	
3510	ROUGHER/SCAVENGER BANK No.1 TAILS SAMPLER		
3510	ROUGHER/SCAVENGER BANK No.1 TAILS SAMPLE PUMP		10
3510	ROUGHER No.1 CONCENTRATE SAMPLER		
3510	ROUGHER No.1 CONCENTRATE SAMPLE PUMP		10
3510	ROUGHER/SCAVENGER BANK No.2		
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.21	300 m3	
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.21 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.22	300 m3	
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.22 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.23	300 m3	
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.23 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.24	300 m3	
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.24 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.25	300 m3	
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.25 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.26	300 m3	
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.26 ROTOR		350
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.27	300 m3, FUTURE	
3510	ROUGHER/SCAVENGER BANK No.2 FLOTATION CELL No.27 ROTOR	FUTURE	
3510	ROUGHER/SCAVENGER BANK No.2 TAILS SAMPLER		
3510	ROUGHER/SCAVENGER BANK No.2 TAILS SAMPLE PUMP		10
3510	ROUGHER No.2 CONCENTRATE SAMPLER		
3510	ROUGHER No.2 CONCENTRATE SAMPLE PUMP		10
3510	ROUGHER 1ST CELL BYPASS CONCENTRATE SAMPLER		
3510	ROUGHER 1ST CELL BYPASS CONCENTRATE SAMPLE PUMP		10
3510	FINAL TAILS SAMPLER		
3510	FINAL TAILS SAMPLER PUMP		10
3510	ROUGHER/SCAVENGER AREA SUMP PUMP No.1		
3510	ROUGHER/SCAVENGER AREA SUMP PUMP No.1		75
3510	ROUGHER/SCAVENGER AREA SUMP PUMP No.2		
3510	ROUGHER/SCAVENGER AREA SUMP PUMP No.2		75
			4410
3520			
3520	1st CLEANER COLUMN FEED TANK	170 m3, 6m D x 6m H	
3520	1st CLEANER COLUMN FEED TANK AGITATOR		
3520	1st CLEANER COLUMN FEED TANK AGITATOR		200
3520	1st CLEANER, COLUMN No.1 FEED PUMP		
3520	1st CLEANER, COLUMN No.1 FEED PUMP		30
3520	1st CLEANER, COLUMN No.1 FLOTATION CELL	170 m3	
3520	1st CLEANER, COLUMN No.1 SPARGING RECIRC PUMP		
3520	1st CLEANER, COLUMN No.1 SPARGING RECIRC PUMP		200
3520	1st CLEANER, COLUMN No.1 TAILS PUMP		
3520	1st CLEANER, COLUMN No.1 TAILS PUMP		30

YMIHPC-001	Equipment Name	Issued for Feasibility Study	
AREA		Description / Performance	Power
		/Specification	HP
3520	1st CLEANER, COLUMN No.2 FEED PUMP		
3520	1st CLEANER, COLUMN No.2 FEED PUMP		30
3520	1st CLEANER, COLUMN No.2 FLOTATION CELL	170 m3	
3520	1st CLEANER, COLUMN No.2 SPARGING RECIRC PUMP		
3520	1st CLEANER, COLUMN No.2 SPARGING RECIRC PUMP		200
3520	1st CLEANER, COLUMN No.2 TAILS PUMP		
3520	1st CLEANER, COLUMN No.2 TAILS PUMP		30
3520	2nd CLEANER COLUMN FEED TANK	50 m3, 4m D x 4m H	
3520	2nd CLEANER COLUMN FEED TANK AGITATOR		
3520	2nd CLEANER COLUMN FEED TANK AGITATOR		100
3520	2nd CLEANER COLUMN FEED PUMP		
3520	2nd CLEANER COLUMN FEED PUMP		15
3520	2nd CLEANER COLUMN FLOTATION CELL	170 m3	
3520	2nd CLEANER COLUMN SPARGING RECIRC PUMP		
3520	2nd CLEANER COLUMN SPARGING RECIRC PUMP		200
3520	2nd CLEANER COLUMN TAILS PUMP		
3520	2nd CLEANER COLUMN TAILS PUMP		7.5
3520	2nd CLEANER COLUMN CONCENTRATE TANK	170 m3, 6m D x 6m H	
3520	2nd CLEANER COLUMN CONCENTRATE TANK AGITATOR		
3520	2nd CLEANER COLUMN CONCENTRATE TANK AGITATOR		200
3520	2nd CLEANER COLUMN CONCENTRATE PUMP		
3520	2nd CLEANER COLUMN CONCENTRATE PUMP		7.5
3520	COPPER CLEANER/SCAVENGER FLOTATION CELL No.51	50 m3	
3520	COPPER CLEANER/SCAVENGER FLOTATION CELL No.1 ROTOR		75
3520	COPPER CLEANER/SCAVENGER FLOTATION CELL No.52	50 m3	
3520	COPPER CLEANER/SCAVENGER FLOTATION CELL No.2 ROTOR		75
3520	CLEANER/SCAVENGER CONCENTRATE VERTICAL PUMP		
3520	CLEANER/SCAVENGER CONCENTRATE VERTICAL PUMP		15
3520	FINAL CONCENTRATE SAMPLER		
3520	FINAL CONCENTRATE SAMPLER PUMP		10
3520	COPPER SCAVENGER TAILS SAMPLER		
3520	COPPER SCAVENGER TAILS SAMPLER PUMP		10
3520	CLEANER AREA SUMP PUMP No.1		
3520	CLEANER AREA SUMP PUMP No.1		30
3520	CLEANER AREA SUMP PUMP No.2		
3520	CLEANER AREA SUMP PUMP No.2		30
			1495
3530			
3530	REGRIND CYCLONE FEED PUMPBOX	___m 3	
3530	REGRIND CYCLONE FEED PUMP No.1		
3530	REGRIND CYCLONE FEED PUMP No.1		200
3530	REGRIND CYCLONE FEED PUMP No.2		
3530	REGRIND CYCLONE FEED PUMP No.2		200
3530	REGRIND ISAMILL CYCLOPAC	Densifying & Desliming 0.25m Cyclones	
3530	REGRIND CYCLONE No. 401		
3530	REGRIND CYCLONE No. 402		
3530	REGRIND CYCLONE No. 403		
3530	REGRIND CYCLONE No. 404		
3530	REGRIND CYCLONE No. 405		
3530	REGRIND CYCLONE No. 406		
3530	REGRIND CYCLONE No. 407		
3530	REGRIND CYCLONE No. 408		
3530	REGRIND CYCLONE No. 409		
3530	REGRIND CYCLONE No. 410		
3530	REGRIND CYCLONE No. 411		
3530	REGRIND CYCLONE No. 412		
3530	REGRIND ISAMILL FEED DISTRIBUTOR		
3530	REGRIND ISAMILL No.1 FEED PUMPBOX	___m 3	
3530	REGRIND ISAMILL No.1 FEED PUMP		
3530	REGRIND ISAMILL No.1 FEED PUMP		15
3530	REGRIND ISAMILL No.1	Isamill M10000	

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AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
3530	REGRIND ISAMILL No.1		4021.44
3530	REGRIND ISAMILL No.1 MOTOR LUBE UNIT		
3530	REGRIND ISAMILL No.1 MOTOR LUBE PUMP No.1		7.5
3530	REGRIND ISAMILL No.1 MOTOR LUBE PUMP No.2		7.5
3530	REGRIND ISAMILL No.1 MOTOR LUBE UNIT COOLING FAN		
3530	REGRIND ISAMILL No.1 MOTOR BRUSH LIFTER		
3530	REGRIND ISAMILL No.1 MOTOR LRS STARTER UNIT		
3530	REGRIND ISAMILL No.1 LRS COOLING FAN No.1		
3530	REGRIND ISAMILL No.1 LRS COOLING FAN No.2		
3530	REGRIND ISAMILL No.1 GEARBOX LUBE UNIT		
3530	REGRIND ISAMILL No.1 GEARBOX LUBE PUMP		5
3530	REGRIND ISAMILL No.1 GEARBOX LUBE UNIT COOLING FAN		
3530	REGRIND ISAMILL No.1 BEARING LUBE UNIT		
3530	REGRIND ISAMILL No.1 BEARING LUBE PUMP		
3530	REGRIND ISAMILL No.1 BEARING LUBE PUMP (Standby)		
3530	REGRIND ISAMILL No.1 BEARING LUBE UNIT COOLING FAN		
3530	REGRIND ISAMILL No.1 MEDIA HOPPER		
3530	REGRIND ISAMILL No.1 ISACHARGER PUMP		
3530	REGRIND ISAMILL No.2 FEED PUMPBOX	___m 3	
3530	REGRIND ISAMILL No.2 FEED PUMP		
3530	REGRIND ISAMILL No.2 FEED PUMP		15
3530	REGRIND ISAMILL No.2	Isamill M10000	
3530	REGRIND ISAMILL No.2		4021.44
3530	REGRIND ISAMILL No.2 MOTOR LUBE UNIT		
3530	REGRIND ISAMILL No.2 MOTOR LUBE PUMP No.1		7.5
3530	REGRIND ISAMILL No.2 MOTOR LUBE PUMP No.2		7.5
3530	REGRIND ISAMILL No.2 MOTOR LUBE UNIT COOLING FAN		
3530	REGRIND ISAMILL No.2 MOTOR BRUSH LIFTER		
3530	REGRIND ISAMILL No.2 MOTOR LRS STARTER UNIT		
3530	REGRIND ISAMILL No.2 LRS COOLING FAN No.1		
3530	REGRIND ISAMILL No.2 LRS COOLING FAN No.2		
3530	REGRIND ISAMILL No.2 GEARBOX LUBE UNIT		
3530	REGRIND ISAMILL No.2 GEARBOX LUBE PUMP		5
3530	REGRIND ISAMILL No.2 GEARBOX LUBE UNIT COOLING FAN		
3530	REGRIND ISAMILL No.2 BEARING LUBE UNIT		
3530	REGRIND ISAMILL No.2 BEARING LUBE PUMP		
3530	REGRIND ISAMILL No.2 BEARING LUBE PUMP (Standby)		
3530	REGRIND ISAMILL No.2 BEARING LUBE UNIT COOLING FAN		
3530	REGRIND ISAMILL No.2 MEDIA HOPPER		
3530	REGRIND ISAMILL No.2 ISACHARGER PUMP		
3530	REGRIND PRODUCT PUMPBOX	___m 3	
3530	REGRIND PRODUCT PUMP No.1		
3530	REGRIND PRODUCT PUMP No.1		100
3530	REGRIND PRODUCT PUMP No.2		
3530	REGRIND PRODUCT PUMP No.2		100
3530	REGRIND PRODUCT SAMPLER		
3530	REGRIND PRODUCT SAMPLE PUMP		10
3530	REGRIND ISAMILL No.1 SUMP PUMP		
3530	REGRIND ISAMILL No.1 SUMP PUMP		30
3530	REGRIND ISAMILL No.2 SUMP PUMP		
3530	REGRIND ISAMILL No.2 SUMP PUMP		30
			8782.89
3600			
3600	PRIMARY COLLECTOR HOPPER	SIPX SUPER SACS	
3600	PRIMARY COLLECTOR MIXING TANK	3 m DIA x 3 m, 20 m3	
3600	PRIMARY COLLECTOR MIXING TANK AGITATOR		
3600	PRIMARY COLLECTOR MIXING TANK AGITATOR		7.5
3600	PRIMARY COLLECTOR FUME EXHAUST FAN		
3600	PRIMARY COLLECTOR FUME EXHAUST FAN		4
3600	PRIMARY COLLECTOR TRANSFER PUMP		
3600	PRIMARY COLLECTOR TRANSFER PUMP		10
3600	PRIMARY COLLECTOR STORAGE TANK	4 m DIA x 4 m, 50 m3	

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
3600	PRIMARY COLLECTOR METERING PUMP No. 1		
3600	PRIMARY COLLECTOR METERING PUMP No. 1		5
3600	PRIMARY COLLECTOR METERING PUMP No. 2		
3600	PRIMARY COLLECTOR METERING PUMP No. 2		5
3600	PRIMARY COLLECTOR METERING PUMP No. 3		
3600	PRIMARY COLLECTOR METERING PUMP No. 3		5
3600	PRIMARY COLLECTOR METERING PUMP No. 4		
3600	PRIMARY COLLECTOR METERING PUMP No. 4		5
3600	PRIMARY COLLECTOR METERING PUMP No. 5		
3600	PRIMARY COLLECTOR METERING PUMP No. 5		5
3600	PRIMARY COLLECTOR METERING PUMP No. 6		
3600	PRIMARY COLLECTOR METERING PUMP No. 6		5
3600	PRIMARY COLLECTOR METERING PUMP No. 7		
3600	PRIMARY COLLECTOR METERING PUMP No. 7		5
3600	PRIMARY COLLECTOR METERING PUMP No. 8		
3600	PRIMARY COLLECTOR METERING PUMP No. 8		5
3600	SECONDARY COLLECTOR METERING PUMP No. 1	3418A TOTES	
3600	SECONDARY COLLECTOR METERING PUMP No. 1		5
3600	SECONDARY COLLECTOR METERING PUMP No. 2		
3600	SECONDARY COLLECTOR METERING PUMP No. 2		5
3600	SECONDARY COLLECTOR METERING PUMP No. 3		
3600	SECONDARY COLLECTOR METERING PUMP No. 3		5
3600	SECONDARY COLLECTOR METERING PUMP No. 4		
3600	SECONDARY COLLECTOR METERING PUMP No. 4		5
3600	SECONDARY COLLECTOR METERING PUMP No. 5		
3600	SECONDARY COLLECTOR METERING PUMP No. 5		5
3600	SECONDARY COLLECTOR METERING PUMP No. 6		
3600	SECONDARY COLLECTOR METERING PUMP No. 6		5
3600	SECONDARY COLLECTOR METERING PUMP No. 7		
3600	SECONDARY COLLECTOR METERING PUMP No. 7		5
3600	SECONDARY COLLECTOR METERING PUMP No. 8		
3600	SECONDARY COLLECTOR METERING PUMP No. 8		5
3600	FROTHER UNLOADING PUMP	MIBC TANKER DELIVERY	
3600	FROTHER UNLOADING PUMP		10
3600	FROTHER STORAGE TANK	4 m DIA x 8 m, 100 m3	
3600	FROTHER METERING PUMP No. 1		
3600	FROTHER METERING PUMP No. 1		5
3600	FROTHER METERING PUMP No. 2		
3600	FROTHER METERING PUMP No. 2		5
3600	FROTHER METERING PUMP No. 3		
3600	FROTHER METERING PUMP No. 3		5
3600	FROTHER METERING PUMP No. 4		
3600	FROTHER METERING PUMP No. 4		5
3600	FROTHER METERING PUMP No. 5		
3600	FROTHER METERING PUMP No. 5		5
3600	DEPRESSANT HOPPER	NaCN SUPER SACS	
3600	DEPRESSANT MIXING TANK	4 m DIA x 6 m, 75 m3	
3600	DEPRESSANT MIXING TANK AGITATOR		
3600	DEPRESSANT MIXING TANK AGITATOR MOTOR		7.5
3600	DEPRESSANT FUME EXHAUST FAN		
3600	DEPRESSANT FUME EXHAUST FAN		2
3600	DEPRESSANT TRANSFER PUMP		
3600	DEPRESSANT TRANSFER PUMP		10
3600	DEPRESSANT STORAGE TANK	4 m DIA x 8 m, 100 m3	
3600	DEPRESSANT METERING PUMP No. 1		
3600	DEPRESSANT METERING PUMP No. 1		5
3600	DEPRESSANT METERING PUMP No. 2		
3600	DEPRESSANT METERING PUMP No. 2		5
3600	LIME STORAGE SILO	500 tonne	
3600	LIME STORAGE BIN VENT FILTER		

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AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
3600	LIME SILO DUST COLLECTOR		
3600	LIME SILO DUST COLLECTOR		1
3600	LIME FEEDER		
3600	LIME FEEDER		3
3600	LIME SLAKER TANK		
3600	LIME SLAKER MILL		40
3600	LIME MILK TRANSFER PUMPBOX		
3600	LIME TRANSFER PUMP		
3600	LIME TRANSFER PUMP		10
3600	LIME STORAGE TANK	4.5 m DIA x 5.5 m, 87 m3	
3600	LIME STORAGE TANK AGITATOR		
3600	LIME STORAGE TANK AGITATOR		40
3600	LIME DISTRIBUTION PUMP No.1		
3600	LIME DISTRIBUTION PUMP No.1		20
3600	LIME DISTRIBUTION PUMP No.2		
3600	LIME DISTRIBUTION PUMP No.2		20
3600	FLOCCULANT HOPPER		
3600	FLOCCULANT MIXING EDUCTOR		
3600	FLOCCULANT MIXING TANK	2 m DIA x 2 m, 5 m3	
3600	FLOCCULANT MIXING TANK AGITATOR		
3600	FLOCCULANT MIXING TANK AGITATOR		3
3600	FLOCCULANT TRANSFER PUMP		
3600	FLOCCULANT TRANSFER PUMP		5
3600	FLOCCULANT STORAGE TANK	3 m DIA x 3 m, 20 m3	
3600	FLOCCULANT METERING PUMP		
3600	FLOCCULANT METERING PUMP		2
3600	FLOCCULANT INLINE STATIC MIXER		
3600	REAGENTS AREA SUMP PUMP		
3600	REAGENTS AREA SUMP PUMP		7.5
3600	LIME AREA SUMP PUMP		
3600	LIME AREA SUMP PUMP		10
3600	FLOCCULANT AREA SUMP PUMP		
3600	FLOCCULANT AREA SUMP PUMP		7.5
			335
3800			
3800	SAFETY SHOWER c/w INLINE HEATER No. 1		
3800	SAFETY SHOWER c/w INLINE HEATER No. 2		
3800	SAFETY SHOWER c/w INLINE HEATER No. 3		
3800	SAFETY SHOWER c/w INLINE HEATER No. 4		
3800	GRINDING BUILDING		
3800	GRINDING BUILDING HVAC		
3800	GRINDING BUILDING OFFICES HVAC		
3800	GRINDING BUILDING VENTILATION FAN No.1	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.1		10
3800	GRINDING BUILDING VENTILATION FAN No.2	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.2		10
3800	GRINDING BUILDING VENTILATION FAN No.3	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.3		10
3800	GRINDING BUILDING VENTILATION FAN No.4	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.4		10
3800	GRINDING BUILDING VENTILATION FAN No.5	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.5		10
3800	GRINDING BUILDING VENTILATION FAN No.6	Wall Axial Exhaust	

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
		Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.6		10
3800	GRINDING BUILDING VENTILATION FAN No.7	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.7		10
3800	GRINDING BUILDING VENTILATION FAN No.8	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.8		10
3800	GRINDING BUILDING VENTILATION FAN No.9	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.9		10
3800	GRINDING BUILDING VENTILATION FAN No.10	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.10		10
3800	GRINDING BUILDING VENTILATION FAN No.11	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.11		10
3800	GRINDING BUILDING VENTILATION FAN No.12	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	GRINDING BUILDING VENTILATION FAN No.12		10
3800	GRINDING BUILDING UNIT HEATER No.1	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.2	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.3	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.4	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.5	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.6	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.7	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.8	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.9	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.10	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING BUILDING UNIT HEATER No.11	Electric Unit Heater, Model HUAA-A10	13.4
3800	GRINDING AREA CRANE	110 tonne	
3800	GRINDING AREA CRANE BRIDGE		50
3800	GRINDING AREA CRANE TROLLEY		
3800	GRINDING AREA CRANE HOIST		
3800	CYCLOPAC AREA CRANE	25 tonne	
3800	CYCLOPAC AREA CRANE BRIDGE		15
3800	CYCLOPAC AREA CRANE TROLLEY		
3800	CYCLOPAC AREA CRANE HOIST		
3800	FLOTATION BUILDING		
3800	FLOTATION BUILDING HVAC		
3800	FLOTATION BUILDING OFFICES & CONTROL ROOM HVAC		
3800	FLOTATION BUILDING VENTILATION FAN No.1	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.1		10
3800	FLOTATION BUILDING VENTILATION FAN No.2	Wall Axial Exhaust Fan, Model SBCE-	

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
		3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.2		10
3800	FLOTATION BUILDING VENTILATION FAN No.3	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.3		10
3800	FLOTATION BUILDING VENTILATION FAN No.4	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.4		10
3800	FLOTATION BUILDING VENTILATION FAN No.5	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.5		10
3800	FLOTATION BUILDING VENTILATION FAN No.6	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.6		10
3800	FLOTATION BUILDING VENTILATION FAN No.7	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.7		10
3800	FLOTATION BUILDING VENTILATION FAN No.8	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.8		10
3800	FLOTATION BUILDING VENTILATION FAN No.9	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.9		10
3800	FLOTATION BUILDING VENTILATION FAN No.10	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	FLOTATION BUILDING VENTILATION FAN No.10		10
3800	FLOTATION BUILDING UNIT HEATER No.1	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION BUILDING UNIT HEATER No.2	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION BUILDING UNIT HEATER No.3	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION BUILDING UNIT HEATER No.4	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION BUILDING UNIT HEATER No.5	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION BUILDING UNIT HEATER No.6	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION BUILDING UNIT HEATER No.7	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION BUILDING UNIT HEATER No.8	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION BUILDING UNIT HEATER No.9	Electric Unit Heater, Model HUAA-A10	13.4
3800	FLOTATION AREA CRANE	15 tonne	
3800	FLOTATION AREA CRANE BRIDGE		15
3800	FLOTATION AREA CRANE TROLLEY		
3800	FLOTATION AREA CRANE HOIST		
3800	CLEANER FLOTATION AREA CRANE	10 tonne	
3800	CLEANER FLOTATION AREA CRANE BRIDGE		10
3800	CLEANER FLOTATION AREA CRANE TROLLEY		
3800	CLEANER FLOTATION AREA CRANE HOIST		
3800	REGRIND BUILDING VENTILATION FAN No.1	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	REGRIND BUILDING VENTILATION FAN No.1		10
3800	REGRIND BUILDING VENTILATION FAN No.2	Wall Axial Exhaust Fan, Model SBCE-	

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
		3L60-100	
3800	REGRIND BUILDING VENTILATION FAN No.2		10
3800	REGRIND BUILDING VENTILATION FAN No.3	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	REGRIND BUILDING VENTILATION FAN No.3		10
3800	REGRIND BUILDING VENTILATION FAN No.4	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	REGRIND BUILDING VENTILATION FAN No.4		10
3800	REGRIND BUILDING VENTILATION FAN No.5	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	REGRIND BUILDING VENTILATION FAN No.5		10
3800	REGRIND BUILDING VENTILATION FAN No.6	Wall Axial Exhaust Fan, Model SBCE-3L60-100	
3800	REGRIND BUILDING VENTILATION FAN No.6		10
3800	REGRIND BUILDING UNIT HEATER No.1	Electric Unit Heater, Model HUAA-A10	13.4
3800	REGRIND BUILDING UNIT HEATER No.2	Electric Unit Heater, Model HUAA-A10	13.4
3800	REGRIND BUILDING UNIT HEATER No.3	Electric Unit Heater, Model HUAA-A10	13.4
3800	REGRIND BUILDING UNIT HEATER No.4	Electric Unit Heater, Model HUAA-A10	13.4
3800	REGRIND BUILDING UNIT HEATER No.5	Electric Unit Heater, Model HUAA-A10	13.4
3800	FILTER PRESS BUILDING VENTILATION FAN No.1	Wall Axial Exhaust Fan, Model SBCE-3H36-50	
3800	FILTER PRESS BUILDING VENTILATION FAN No.1		5
3800	FILTER PRESS BUILDING VENTILATION FAN No.2	Wall Axial Exhaust Fan, Model SBCE-3H36-50	
3800	FILTER PRESS BUILDING VENTILATION FAN No.2		5
3800	FILTER PRESS BUILDING VENTILATION FAN No.3	Wall Axial Exhaust Fan, Model SBCE-3H36-50	
3800	FILTER PRESS BUILDING VENTILATION FAN No.3		5
3800	FILTER PRESS BUILDING VENTILATION FAN No.4	Wall Axial Exhaust Fan, Model SBCE-3H36-50	
3800	FILTER PRESS BUILDING VENTILATION FAN No.4		5
3800	FILTER PRESS BUILDING VENTILATION FAN No.5	Wall Axial Exhaust Fan, Model SBCE-3H36-50	
3800	FILTER PRESS BUILDING VENTILATION FAN No.5		5
3800	FILTER PRESS BUILDING VENTILATION FAN No.6	Wall Axial Exhaust Fan, Model SBCE-3H36-50	
3800	FILTER PRESS BUILDING VENTILATION FAN No.6		5
3800	FILTER BUILDING UNIT HEATER No.1	Electric Unit Heater, Model HUAA-A10	13.4
3800	FILTER BUILDING UNIT HEATER No.2	Electric Unit Heater, Model HUAA-A10	13.4
3800	FILTER BUILDING UNIT HEATER No.3	Electric Unit Heater, Model HUAA-A10	13.4
3800	FILTER BUILDING UNIT HEATER No.4	Electric Unit Heater, Model HUAA-A10	13.4
3800	FILTER BUILDING UNIT HEATER No.5	Electric Unit Heater, Model HUAA-A10	13.4
3800	MILL BUIILDING AIR HANDLING UNIT PACKAGE No.1	Heating and Cooling (grinding area, lunch room, meeting room,	60

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance /Specification	Power HP
		offices). Air condensing outside, evaporation inside	
3800	MILL BUIILDING AIR HANDLING UNIT PACKAGE No. 2	Heating and Cooling (electrical room cooling). Air condensing outside, evaporation inside	160
3800	FILTER PRESS AREA CRANE	10 tonne	
3800	FILTER PRESS AREA CRANE - BRIDGE		10
3800	FILTER PRESS AREA CRANE - TROLLEY		
3800	FILTER PRESS AREA CRANE - HOIST		
			1032
3900			
3900	SEAL WATER BOOSTER PUMP No. 7		
3900	SEAL WATER BOOSTER PUMP No. 7		10
3900	SEAL WATER BOOSTER PUMP No. 8		
3900	SEAL WATER BOOSTER PUMP No. 8		10
3900	FILTER PRESS HP COMPRESSOR No.1	Piston type, 35 bar (500 psi)	
3900	FILTER PRESS HP COMPRESSOR No.1		5
3900	FILTER PRESS HP COMPRESSOR No.2	Piston type, 35 bar (500 psi)	
3900	FILTER PRESS HP COMPRESSOR No.2		5
3900	FILTER PRESS AIR RECEIVER	2 m3	
			30
3910			
3910	CONCENTRATE THICKENER FEED LAUNDER		
3910	CONCENTRATE THICKENER	18 m DIA.	
3910	CONCENTRATE THICKENER RAKE		10
3910	CONCENTRATE THICKENER RAKE LIFT		
3910	CONCENTRATE THICKENER UNDERFLOW PUMP No.1		
3910	CONCENTRATE THICKENER UNDERFLOW PUMP No.1		30
3910	CONCENTRATE THICKENER UNDERFLOW PUMP No.2		
3910	CONCENTRATE THICKENER UNDERFLOW PUMP No.2		30
3910	CONCENTRATE THICKENER OVERFLOW PUMPBOX	___ m 3	
3910	CONCENTRATE THICKENER OVERFLOW PUMP		
3910	CONCENTRATE THICKENER OVERFLOW PUMP		20
3910	CONCENTRATE STOCK TANK	5.0 m DIA x 10.0 m, __ m3	
3910	CONCENTRATE STOCK TANK AGITATOR		
3910	CONCENTRATE STOCK TANK AGITATOR		100
3910	FILTER FEED PUMP No. 1		
3910	FILTER FEED PUMP No. 1		60
3910	FILTER FEED PUMP No. 2		
3910	FILTER FEED PUMP No. 2		60
3910	FILTER PRESS No. 1	___ m2, __ Plates x _ m2	
3910	FILTER PRESS No.1 HYDRAULIC UNIT		20
3910	FILTER PRESS No. 1 PLATE SHIFT		2
3910	FILTER PRESS No. 1 LOAD CELL		0.5
3910	FILTER PRESS No. 1 LOAD CELL		0.5
3910	FILTER PRESS No. 1 DRIP TRAY		3
3910	FILTER PRESS No. 2	___ m2, __ Plates x _ m2	
3910	FILTER PRESS No.2 HYDRAULIC UNIT		20
3910	FILTER PRESS No.2 PLATE SHIFT		2
3910	FILTER PRESS No.2 LOAD CELL		0.5
3910	FILTER PRESS No.2 LOAD CELL		0.5
3910	FILTER PRESS No. 2 DRIP TRAY		3
3910	CLOTH WASH PUMP No. 1		
3910	CLOTH WASH PUMP No. 1		25
3910	CLOTH WASH PUMP No.2		
3910	CLOTH WASH PUMP No.2		25
3910	AIR RELEASE TANK	2.2m DIA x1.7m, __	

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AREA	Equipment Name	Description / Performance	Power
		/Specification	HP
		m3	
3910	FILTRATE TANK	3.0 m DIA x 3.5 m, __ m3	
3910	FILTRATE TANK AGITATOR		
3910	FILTRATE TANK AGITATOR		7.5
3910	FILTRATE PUMP No.1		
3910	FILTRATE PUMP No.1		100
3910	FILTRATE PUMP No.2		
3910	FILTRATE PUMP No.2		100
3910	THICKENER AREA SUMP PUMP		
3910	THICKENER AREA SUMP PUMP		15
			634.5
3920			
3920	GLYCOL TANK	REQUIRED?	
3920	GLYCOL PUMP		
3920	GLYCOL PUMP		7.5
3920	GLYCOL SPRAY		
3920	TIRE WASH STATION		
3920	TIRE WASH PUMP		
3920	TIRE WASH PUMP		10
3920	TIRE WASH SUMP PUMP		
3920	TIRE WASH SUMP PUMP		30
3920	TIRE WASH OIL/WATER SEPARATOR		
			47.5
5000			
5000	LUBE OIL TANK No.1		
5000	LUBE OIL PUMP No.1		
5000	LUBE OIL PUMP No.1		5
5000	LUBE OIL TANK No.2		
5000	LUBE OIL PUMP No.2		
5000	LUBE OIL PUMP No.2		5
5000	LUBE OIL TANK No.3		
5000	LUBE OIL PUMP No.3		
5000	LUBE OIL PUMP No.3		5
5000	LUBE OIL TANK No.4		
5000	LUBE OIL PUMP No.4		
5000	LUBE OIL PUMP No.4		5
5000	LUBE OIL TANK No.5		
5000	LUBE OIL PUMP No.5		
5000	LUBE OIL PUMP No.5		5
5000	WASTE OIL TANK		
5000	SHOP AIR COMPRESSOR No.1		
5000	SHOP AIR COMPRESSOR No.1		100
5000	SHOP AIR RECEIVER		
5000	SHOP AIR DRYER		
5000	REPAIR BAY CRANE		
5000	REPAIR BAY CRANE BRIDGE		
5000	REPAIR BAY CRANE TROLLEY		
5000	REPAIR BAY CRANE HOIST No.1		
5000	REPAIR BAY CRANE HOIST No.2		
5000	TRUCK WASH MONITORS		
5000	TRUCK WASH SPRAY PUMP		
5000	TRUCK WASH SPRAY PUMP		20
5000	LIGHT VEHICLE WASH SPRAY PUMP		
5000	LIGHT VEHICLE WASH SPRAY PUMP		10
5000	TRUCK WASH SUMP PUMP		
5000	TRUCK WASH SUMP PUMP		30
5000	TRUCK WASH OIL/WATER SEPARATOR		
5000	TRUCK SHOP BUILDING		
5000	TRUCK SHOP BUILDING HVAC		
5000	TRUCK SHOP OFFICES HVAC		
5000	WASH BAY HVAC		
5000	TRUCK SHOP BUILDING HEATED PAD		

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance /Specification	Power HP
5000	TRUCK SHOP BUILDING VENTILATION FAN No.1	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.1		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.2	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.2		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.3	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.3		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.4	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.4		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.5	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.5		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.6	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.6		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.7	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.7		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.8	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.8		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.9	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.9		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.10	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.10		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.11	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.11		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.12	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.12		5
5000	TRUCK SHOP BUILDING VENTILATION FAN No.13	Wall Axial Exhaust Fan, Model SBCE-3L36-50	
5000	TRUCK SHOP BUILDING VENTILATION FAN No.13		5
5000	TRUCK SHOP UNIT HEATER No.1	Electric Unit Heater, Model HUAA-A10	13.4
5000	TRUCK SHOP UNIT HEATER No.2	Electric Unit Heater, Model HUAA-A10	13.4
5000	TRUCK SHOP UNIT HEATER No.3	Electric Unit Heater, Model HUAA-A10	13.4
5000	TRUCK SHOP UNIT HEATER No.4	Electric Unit Heater, Model HUAA-A10	13.4
5000	TRUCK SHOP UNIT HEATER No.5	Electric Unit Heater, Model HUAA-A10	13.4
5000	TRUCK SHOP UNIT HEATER No.6	Electric Unit Heater, Model HUAA-A10	13.4
5000	TRUCK SHOP UNIT HEATER No.7	Electric Unit Heater, Model HUAA-A10	13.4

YMIHPC-001		Issued for Feasibility Study	
AREA	Equipment Name	Description / Performance /Specification	Power HP
5000	TRUCK SHOP UNIT HEATER No.8	Electric Unit Heater, Model HUAA-A10	13.4
5000	TRUCK SHOP UNIT HEATER No.9	Electric Unit Heater, Model HUAA-A10	13.4
5000	TRUCK SHOP UNIT HEATER No.10	Electric Unit Heater, Model HUAA-A10	13.4
3800	TRUCK SHOP AIR HANDLING UNIT PACKAGE	Heating and Cooling (elec room, lunch room, training room, mtg room, offices, locker room). Air condensing outside, evaporation inside	60
5000	WAREHOUSE FACILITY		
			444
5300			
5300	GATE HOUSE BUILDING		
5300	GATE HOUSE HVAC		
5300	TRUCK SCALE	AT GATE HOUSE	
5400			
5400	ASSAY LAB FACILITY		
6100			
6100	EMERGENCY POWER GENERATOR No. 1	DIESEL	2681
6100	EMERGENCY POWER GENERATOR No. 2	DIESEL	2681
	LIGHTING AND PROCESS LOADS		
			5362
7010			
7010	TAILINGS PUMPBOX	110 m 3	
7010	TAILINGS PUMP No. 1		
7010	TAILINGS PUMP No. 1		150
7010	TAILINGS PUMP No. 2		
7010	TAILINGS PUMP No. 2		150
7010	TAILINGS PUMP No. 3	FUTURE	
7010	TAILINGS PUMP No. 3	FUTURE	150
7010	TAILINGS PUMP No. 4	FUTURE	
7010	TAILINGS PUMP No. 4	FUTURE	150
7010	CLEANER SCAVENGER TAILINGS PUMPBOX	9 m 3	
7010	CLEANER SCAVENGER TAILINGS PUMP No. 1	FUTURE	
7010	CLEANER SCAVENGER TAILINGS PUMP No. 1	FUTURE	20
7010	CLEANER SCAVENGER TAILINGS PUMP No. 2	FUTURE	
7010	CLEANER SCAVENGER TAILINGS PUMP No. 2	FUTURE	20
			640
7020			
7020	RECLAIM WATER BARGE		
7020	RECLAIM WATER BARGE VERTICAL PUMP No. 1	Vertical pump	
7020	RECLAIM WATER BARGE VERTICAL PUMP No. 1		1750
7020	RECLAIM WATER BARGE VERTICAL PUMP No. 2	Vertical pump	
7020	RECLAIM WATER BARGE VERTICAL PUMP No. 2		1750
7020	RECLAIM WATER BARGE VERTICAL PUMP No. 3	Vertical pump	
7020	RECLAIM WATER BARGE VERTICAL PUMP No. 3		1750
7020	RECLAIM WATER BARGE VERTICAL PUMP No. 4	Vertical pump	
7020	RECLAIM WATER BARGE VERTICAL PUMP No. 4		1750
7020	TAILINGS SEEPAGE COLLECTION VERTICAL PUMP No. 1		
7020	TAILINGS SEEPAGE COLLECTION VERTICAL PUMP No. 1		250
7020	TAILINGS SEEPAGE COLLECTION VERTICAL PUMP No. 2		
7020	TAILINGS SEEPAGE COLLECTION VERTICAL PUMP No. 2		250
7020	NORTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 1		
7020	NORTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 1		600
7020	NORTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 2		
7020	NORTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 2		600
7020	NORTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 3		
7020	NORTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 3		600
7020	SOUTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 1		
7020	SOUTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 1		500

YMIHPC-001	Equipment Name	Issued for Feasibility Study	
AREA		Description / Performance	Power
		/Specification	HP
7020	SOUTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 2		
7020	SOUTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 2		500
7020	SOUTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 3		
7020	SOUTH SITE RUN OFF SEEPAGE COLLECTION VERTICAL PUMP No. 3		500
			10800
8300			
8300	CAMP FACILITY		
8300	CAMP POWER GENERATOR No. 1	DIESEL	
8300	CAMP POWER GENERATOR No. 2	DIESEL	
8300	CAMP POWER GENERATOR No. 3	DIESEL	
8300	CAMP HVAC	DIESEL	
			129798.33

18 PROJECT INFRASTRUCTURE

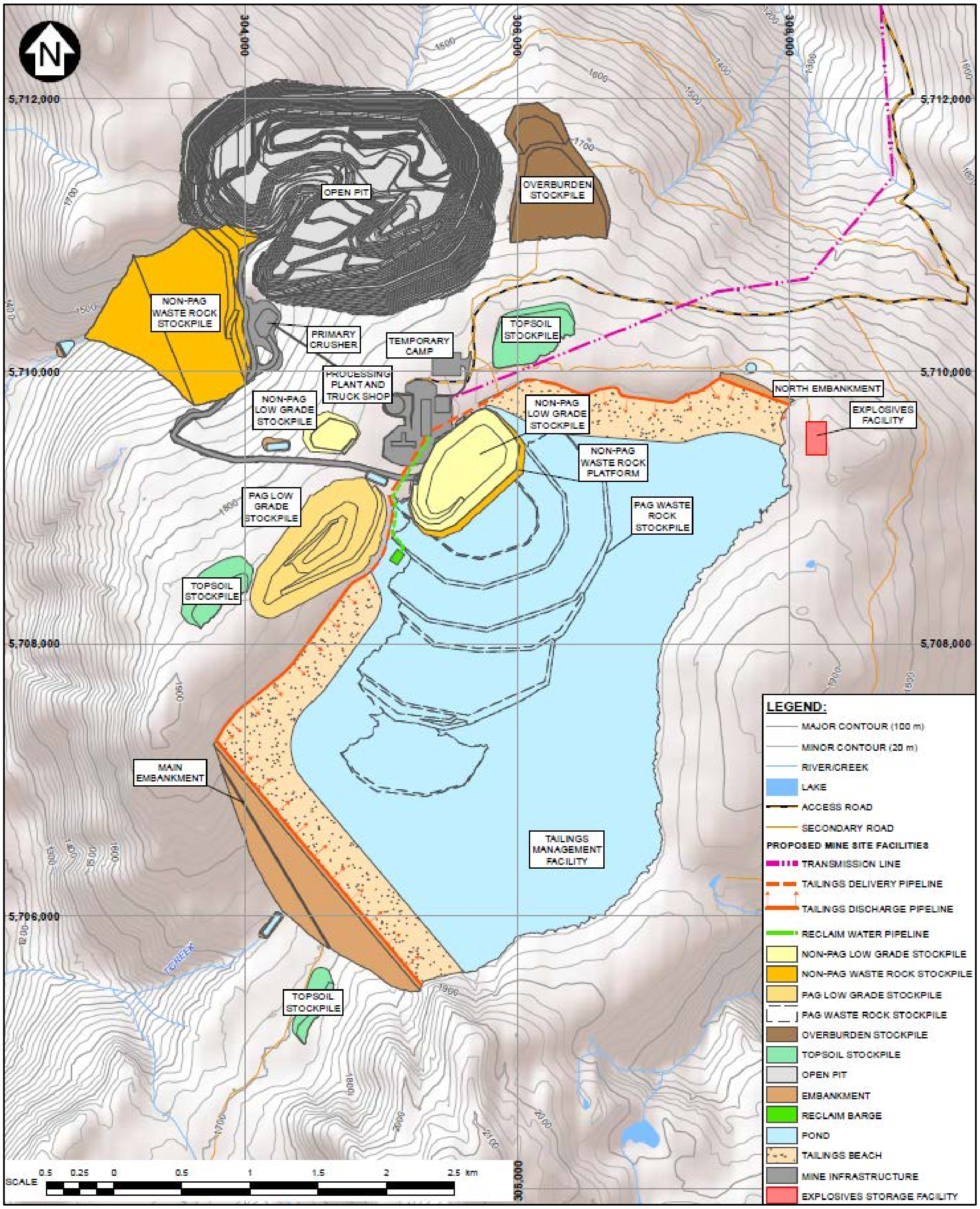
18.1 OVERVIEW

The services and ancillary facilities that will be required include the following:

- Project site access road upgrade;
- Power supply from the BC Hydro grid, transmission to site, and project site distribution;
- Permanent building structures;
- Fresh water supply, fire/fresh water storage and distribution, recycled water collection/storage/distribution, fuel storage and dispensing, sewage collection and treatment, drainage and runoff settling ponds;
- Temporary housing facilities for construction personnel;
- Plant site roads, yard areas and parking;
- Waste rock storage and stockpiles;
- Tailings management facility
- Security, safety, and first aid facilities.

Figures 18-1 and 18-2 provide the layout of the proposed facilities.

Figure 18-1 Plant Site Layout



Knight Piesold Consulting, May 2014

Figure 18-2: Conceptual Rendering of Plant Site Layout

Allnorth Consultants Limited, November 2011

18.2 ACCESS ROADS

18.2.1 *SITE ACCESS*

In general, access to site from Highway #5 is via the Vavenby Bridge Road through Vavenby and across the North Thompson River to the Birch Island Lost Creek Road (BILCR). From there, access is via an existing 18.5km network of Forest Service Roads (FSR) that climb up to the plant site from its junction at Birch Island Lost Creek Road south of Vavenby. The FSRs that comprise the access road will be the:

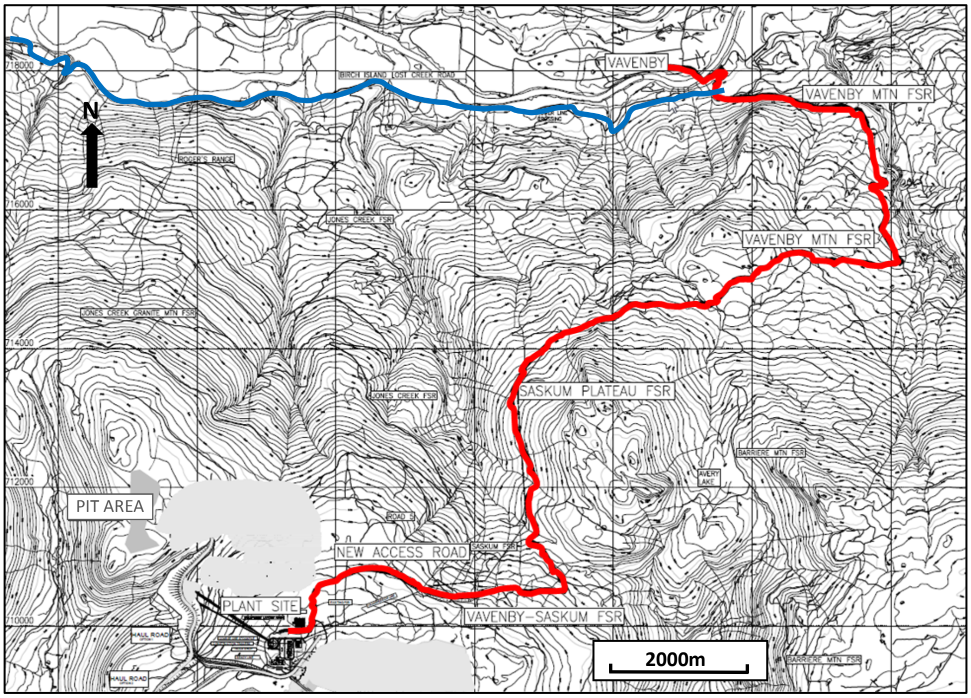
- Vavenby Mountain FSR;
- Saskum Plateau FSR; and
- Vavenby-Saskum FSR.

In order to improve access for both construction and mining the FSRs will be upgraded as required. Upgrades will include:

- widening where necessary;
- Improvements to alignment where practical;
- Improvements to the BILCR/Vavenby Mountain FSR junction; and
- Signage improvement.

In addition, a new section road, approximately 2km in length, will connect the upgraded Vavenby/Saskum FSR to the plant site. Local borrow sources will be created along the route to provide material for the necessary upgrades. Figure 18.3 illustrates the layout of the site access road highlighted in red.

Figure 18-3: Site Access Road



Yellowhead Mining Inc., July 2014

18.2.2 *TEMPORARY CONSTRUCTION ACCESS*

During the course of construction, oversized loads (overweight and/or over length/width), will require an alternative access across the North Thompson River as the Vavenby Bridge has not been designed to cater for such loads safely. The proposed temporary construction route access for oversize loads will be:

- Yellowhead Highway #5 (from both north and south bound);
- Birch Island Lost Creek Road (BILCR);
- Vavenby Mountain FSR;
- Saskum Plateau FSR; and
- Vavenby-Saskum FSR.

This proposed route crosses the North Thompson River at the BILCR Bridge which has been design for heavier loads. Figure 18.3 illustrates the temporary access route for oversize loads (highlighted in blue). Some upgrades will be required, in particular, areas to be examined at the permitting stage include:

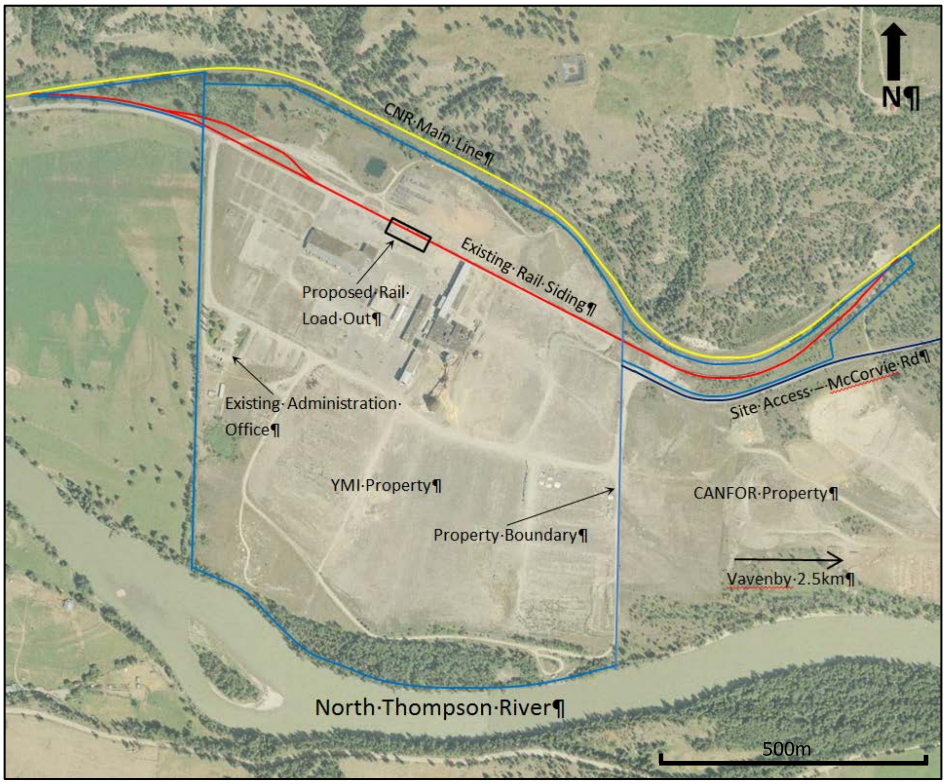
- BILCR Bridge;
- BILCR flood zone (200m);
- BILCR swithchback;
- BC Hydro and Telus overhead communication lines on BILCR; and
- Intersection BILCR and Vavenby Mountain FSR.

For the purpose of this FS an allowance was made in initial capital for proposed upgrades. In addition, suitable traffic management plans will be developed to ensure the safe transportation of these loads.

18.3 Rail

In 2011 YMI acquired an abandoned 79.3 hectare facility owned by Weyerhaeuser Company Limited, 2.5km west of Vavenby. The acquisition included the rail siding, buildings, offices and statutory rights of way. The rail load-out area will be located here, approximately 25km by road from the Project mill site (Figure 18-4).

Figure 18-4: Aerial View of Proposed Rail Load Out Facility Location



Yellowhead Mining Inc., July 2014

Allnorth conducted a site inspection of the existing rail siding (Figure 18-5, 18-6). The inspection consisted of walking from the previous switch-line location at the CN mainline, on the east side of the property, through the mill yard along the siding and back up to the CN mainline on the west side of the property along Kp Road. Attention was given to the quality of materials present, materials found to be missing or of sub-par quality. Initial recommendations for the upgrading, new construction, and preliminary maintenance that would be required in order to make the track a usable siding are as follows:

The Canadian National Railways (CNR) transcontinental mainline passes through the Vavenby area. YMI's property has an existing rail siding in place. The existing siding track is a standard 100lb AR, 3 dot, siding rail. It is visually passable, with some curl on the head of the rail and is good tangent, not curve, rail. The bulk of the existing material was manufactured in the mid-fifties. The track is joined lengthwise by four-hole joint-bars, using 11" tie-plates and 5-12” spikes on ties with a 24” center-to-center spacing. The rails are set to a standard gauge of 4’ 8-1/2”.

There are two sections, along the siding location, where the existing track has been almost completely removed, separated by a remaining middle section of track. The middle section of track appears to be in good condition. The existing roadbed for all three sections is overgrown with vegetation and brush.

The main CN line is composed of 132lb rail, welded in continuous sections, on good crush ballast. There is a posted track speed, to the west of where the existing siding would have terminated at the main line, of 30mph MAS for passenger trains, and 25mph MAS for freight trains.

18.3.1 *Rail Requirements & Assessment*

Given modern rail engineering practices, and the good condition of the existing 100lb past worn rail, there is no reason to suspect that it will not satisfy current CN requirements. If not satisfactory, the rail can be upgraded to 115lb rail for the entire siding and would allow the use of existing ties, tie-plates, and anchors. Any track grade beyond that would require additional upgrading of present material. In summary:

- Two 132lb switches will be required for tie-in to the existing main line.
- Siding at west and east end requires re-grading to make a suitable road-bed for the reintroduced siding tie-ins.
- Ballast should be added to stabilize track along the length of the sections of track where the rail for the siding has been removed and possibly, the remaining section of track in between, to stabilize the track and ensure proper drainage requirements.
- Three switches remain on the track and are in good condition but will require updating and refurbishment. If any of the switches are not required they will be removed from the siding to prevent the necessity of additional maintenance.
- Where the track has been flooded, or in-filled, with earthen material or excess ballast, and if it is not required, the material should be removed. This can be accomplished by either a ballast regulator or by manual labor and heavy equipment.
- The existing road-bed requires brush cutting and the application of a soil sterilant for maintenance. Prior to commencement of any construction, the existing road-bed requires extensive brush cutting to remove overgrown vegetation from the existing track and from the areas where the track has been removed.

In order to ensure conformance to policy, procedure, and adherence to CN’s Maintenance-of-Way guidelines, standard practices, and to analyze the requirements for construction, a post assessment review of the current findings will be scheduled with a consultant and a qualified CN representative as a next step moving forward.

Given the state of the presently existing track, and the measures necessary to replace the missing track, along the KP road siding on the former Weyerhaeuser mill site, it is our opinion that this site could viably be used as a loading site for mineral concentrate loading purposes. Certain subsequent information and requirements, as outlined in this report, would have to be gathered in order to fully assess an estimated cost of reinstating this siding for use. Figures 18-4 and 18-5 illustrate the current condition of the existing track.

Figure 18-5: Rail Siding West to Old Mill Building

Allnorth Consultants Limited, December 2011

Figure 18-6: Rail Siding East to Main CN Line

Allnorth Consultants Limited, December 2011

18.4 Power Supply

BC Hydro will supply power to the Project via a 14km, 138kV overhead transmission line from the Vavenby Substation, crossing the North Thompson River to the Project's main substation located adjacent to the processing plant. The power line will be constructed using wooden poles in a configuration that will be a combination of single pole towers and H-frame structures. The average power demand will be approximately 82MW.

18.4.1 *Main Substation*

The incoming power line will terminate on the dead end structure within the main substation. The main substation will contain three transformers rated at 138kV-25kV, 50/66.77/83.3MVA. The main transformers are sized to allow for full operation in the event that one of the transformers is off line for maintenance or has failed. The main transformers will feed power to a 25kV outdoor switchgear unit located inside a prefab building within the substation area.

18.4.2 *Site Power Distribution*

The plant will have an operating load of 82MVA and a peak load of 85MVA. The power from the 25kV outdoor switchgear will be distributed throughout the plant site at 25kV. The secondary power distribution will consist of 25kV-4160V transformers and 25kV – 600V transformers. These transformers will be oil filled, outdoor rated transformers. The mine pit will be serviced by portable 25kV/7.2kV substations.

18.4.3 *Emergency Power Distribution*

The permanent emergency power distribution system will consist of two (2) MVA standby generators. These generators will be located in the main substation and are connected to the 25kV outdoor switchgear. During the construction period these generators will be installed early in the program to power one of the pit drills and a shovel that will be used to move overburden for use as construction materials for haul roads and the starter tailings dam.

18.4.4 *Onsite Power Lines*

A 25kV overhead power line will supply electrical loads outside the process building area to the tailings area, coarse ore storage area, primary crushing area, and mine pit area. A 25kV power line will also be installed from a tap point on the existing BC Hydro grid near Vavenby up to the site for construction power use.

18.5 PLANT SITE FACILITIES

The plant site and truck shop are located on relatively flat terrain between the open pit and tailings pond. It has been established that rock is close to the existing ground surface at this location to adequately support equipment foundations. The facilities contained in the plant site area. It consists of a stock pile fed from the crusher, the mill building and the Truck Shop/Mine Dry building (Figure 18-7).

Figure 18-7: Conceptual Rendering of Plant Site Facilities

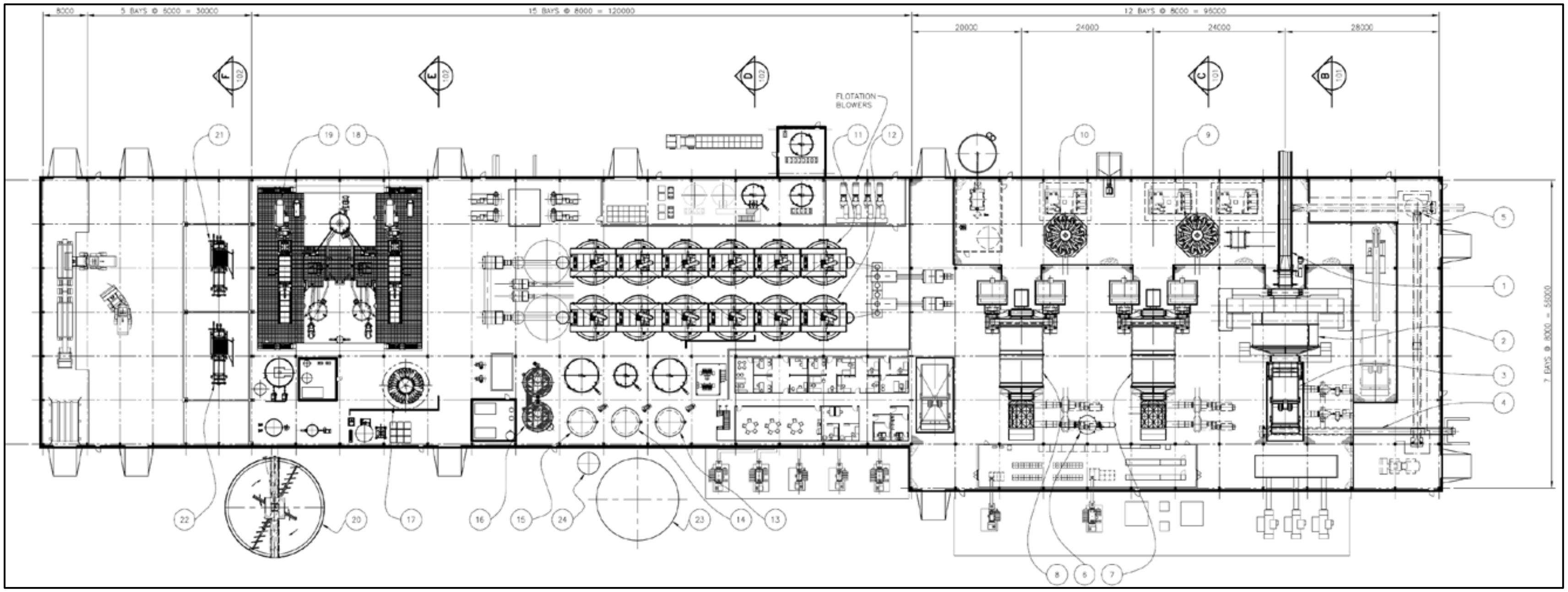
Allnorth Consultants Limited, November 2011

18.5.1 *MILL BUILDING*

The Grinding/Concentrate equipment will be housed in a pre-engineered metal building with two distinctive sections. The grinding area (Figure 18.6) will contain the SAG and ball mill equipment, including the associated equipment for the grinding circuits. It will be approximately 96m x 56m and will have several platforms at various levels to support equipment and allow access where necessary. The majority of the Grinding area will be serviceable by one of two heavy-duty overhead cranes. The mills will be serviced by a 110t overhead crane and the cyclones and ball bins will be serviced by a 25t overhead crane.

The concentrate area will be 120m x 48m, and will be located to the south of the Grinding Area. This area will be accessible by 3 overhead cranes. The Flotation area will have a 15t overhead crane while the regrind area will be serviced with a 10t overhead crane. The filter press will have its own 10t crane. Adjacent to this part of the process building are the Concentrate Thickener and the fresh water tank. It is from the South end of the concentrate area that the final concentrate will be shipped by truck to the Concentrate Storage and Transportation Facilities near Vavenby. A four-floor structure will be located within the grinding area, containing the Assay Laboratory, Electrical Rooms, Offices, and the Control Room (Figure 18-8 Grinding / Concentrate Plant Layout).

Figure 18-8 Proposed Grinding Concentrate Plant Layout

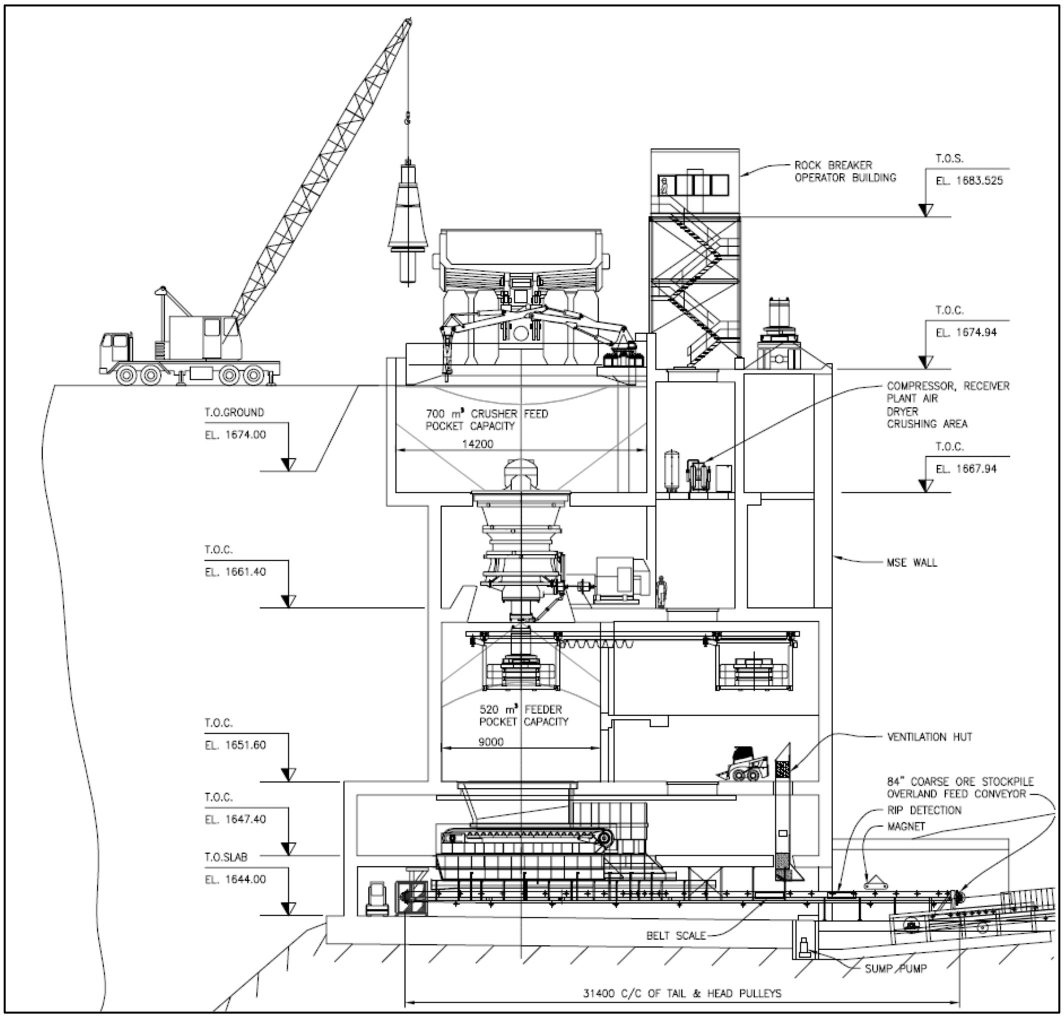


Allnorth Consultants Limited, November 2011

18.5.2 *PRIMARY CRUSHER*

The crusher (Figure 18-9) will be located to the north-west of the plant site, and a 30m wide haul road ramp constructed by the mine fleet using overburden will be constructed from the pit to the crusher pad. A mechanically stabilized earth (MSE), or a Hilfiker style, wall will be required at the crusher pad. Figure 18.8 illustrates a cross section through the crusher. The geotechnical nature of the site has been established as sound, uniform and able to support the intended equipment although more geotechnical information is needed to establish the extent of the rock.

Figure 18-9: Primary Crusher Cross Section



Allnorth Consultants Limited, November 2011

18.5.3 *Assay Laboratory*

The laboratory will be located inside the process plant, and will contain a sample preparation area, a chemical laboratory for standard ore analysis (including gold fire assay) and an environmental analysis facility.

18.5.4 *Mine Maintenance Shop and Mine Dry*

The main purpose of the Mine Maintenance Shop is to provide maintenance and servicing to mining equipment. The truck shop itself will be comprised of five regular service bays, two welding bays and one preventative maintenance (PM) bay. The main vehicle bays are sized to accommodate 292t haul trucks and other mobile equipment with 12m x 8.5m vertical fabric folding doors. The truck shop and other bays will be serviced by a 50/15t bridge crane.

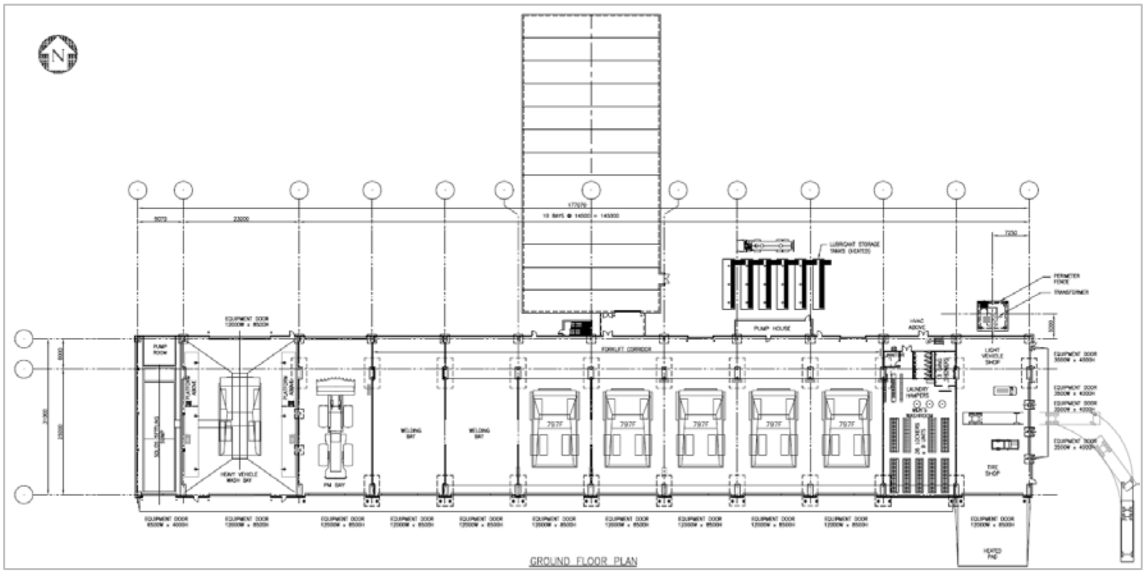
Attached to the west side of the building will be a heavy vehicle wash bay, this bay will be equipped with firefighting type monitors, heated high pressure washers and hose reels as required to ensure truck are thoroughly cleaned. The used wash water will drain to a central sump where it will flow to a settling pond for treatment prior to being discharged to the tailings facility.

Along the north side of the truck shop will be a corridor for the safe movement of parts and material along the length of the building without having to encroach into the truck bays. Above this corridor is a mezzanine with offices, lunch and meeting rooms for the maintenance staff.

Also contained within the maintenance shop building is the mine dry, which will provide lockers and showers for the workers at the beginning and end of each shift.

The truck shop, wash and mine dry will all be housed in a pre-engineered metal building approximately 31m x 177m x 21m high (Figure 18-10).

Figure 18-10: Mine Maintenance Shop & Mine Dry Building Layout



Allnorth Consultants Limited, November 2011

18.5.5 *Ancillary Buildings and Other Structures*

The warehouse will be contained within a Stressed Fabric building near the truck maintenance shop and connected via a passageway. It will be approximately 59m x 27m and will be used for storage of parts and materials needed for mine and plant operations.

The gatehouse, scale house and first aid building will be combined at the entrance to the mine site and will also house the ambulance for the site.

18.5.6 *Fuel Storage*

Diesel fuel for the mining, process and ancillary facilities will be supplied from a diesel fuel storage facility, consisting of four above-ground 75,000 litre capacity diesel fuel storage tanks, suitable for four days of on-site usage, and one gasoline storage tank, and loading and dispensing equipment. The facility will be located near the truck shop. A dedicated fuel truck (bowser) will transport diesel to the mining equipment operating in the pit and fuel replacement will be a daily occurrence from an off-site terminal.

18.5.7 *Construction Camp*

It is anticipated that a peak of 600 construction personnel, not including pre-production operations persons who will be housed in the local communities, or the crews that will upgrade the main access road and\or build the incoming high voltage power lines, will need to be accommodated onsite during construction. To accommodate 600, a construction camp of adequate size will be leased for the duration of construction and removed afterwards. The site will be located adjacent to the mill area on the access road. The camp will include construction offices, a mine dry, a kitchen, a dining room, and recreational facilities. A 25kV power line will be constructed from the BC

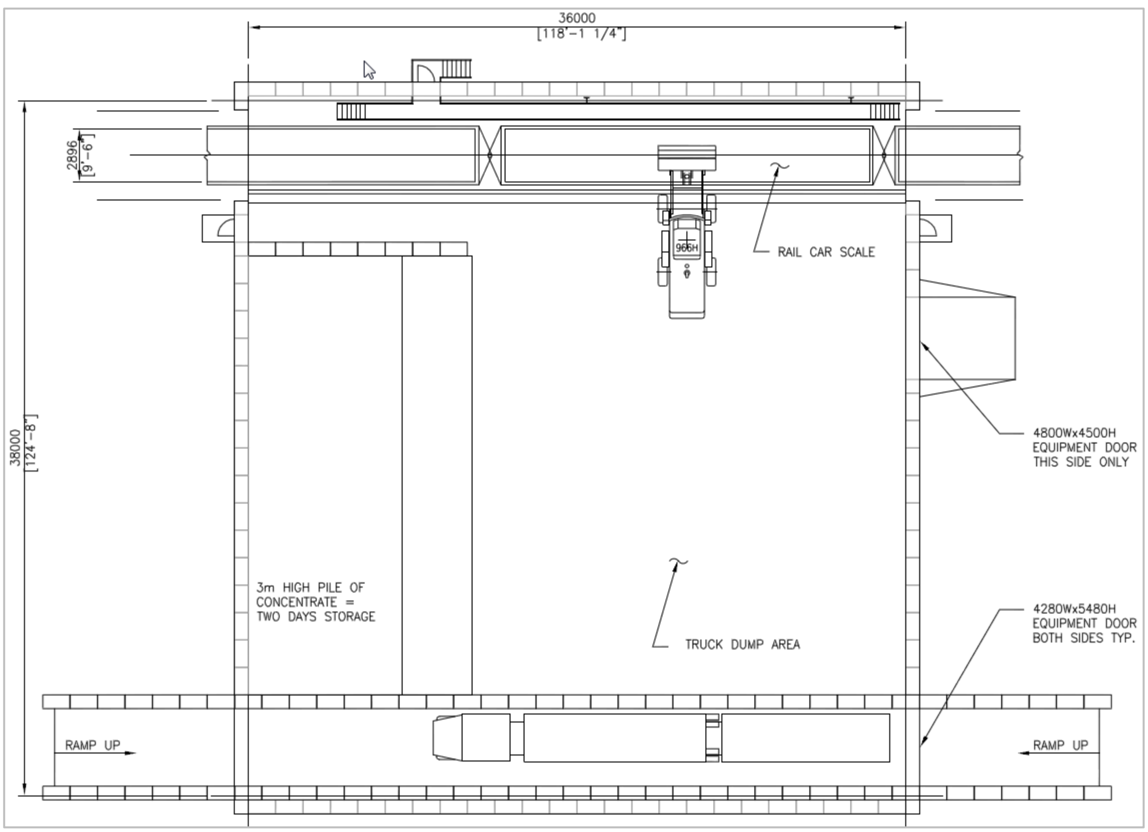
Hydro grid close to Vavenby that has about 2.5MW of spare capacity, up to the camp area to provide power to the camp and for construction.

18.5.8 *Offsite Load Out & Administration*

The copper concentrate will be trucked to nearby Vavenby, BC and stored in an off-site facility capable of storing two days worth of concentrate production at a time. The copper concentrate will be then transported by train to Vancouver for shipment to overseas smelters. The facility will consist of a 36m x 38m building with a raised dumping area for concentrate trucks and bunker walls to contain the concentrate. A rail scale and mechanism for cover removal will be constructed along one wall and a wheel loader, provided by the load out contractor, will be used to load the train cars (Figure 18-11).

It is planned to utilize the existing administration facilities located on the "old" Weyerhauser property to house administration personnel for the project. The existing building has been maintained and monitored from a security standpoint since the closure of the mill. Minimal upgrading is required to bring the facility to an operational status.

Figure 18-11: Rail Load out Layout



Allnorth Consultants Limited, November 2011

18.6 Tailing Management Facility

In general, tailings and potentially acid generating (PAG) waste rock will be deposited in an impoundment located to the south east of the process plant. The Tailings Management Facility (TMF) has been designed to provide sufficient capacity to store approximately 585Mt of tailings and co-disposal of up to 237Mt of PAG waste rock and an anticipated surplus water volume of up to 180Mm^3. The balance of the tailings (LG processed in Years 24 to 28) will be deposited into the open pit.

Design details are provided in Section 20.4.3 of this report, however, in general the main embankment will be developed in stages throughout the life of the project using a combination of suitable non-reactive overburden and waste rock from the open pit and/or local borrow sources. The Main Embankment will be expanded on an annual basis.

A preliminary water balance model was developed by Knight Piésold Ltd. for the TMF to assess the various flows to and from the facility, to predict the variation in supernatant pond volume during operations and to estimate make up water requirements for processing. The water balance predicts that there is a net surplus of water for the entire operating period of the mine, indicating that water falling on and/or flowing into the TMF will supply more than enough water to meet the mill process water requirements.

Seepage water losses from the TMF will be collected in water management ponds constructed downstream of the embankments. The seepage will be pumped back into the TMF throughout the mine life.

Details of the site characteristics, geotechnical and water management considerations for the tailings facility design, seepage collection and reclamation and closure are contained in the Knight Piésold Ltd. “Mine Waste and Water Management Design Report” (Ref. No. VA101-458/11-1).

18.7 Waste Rock Storage And Stockpiles

A total of approximately 542Mt of waste rock and overburden will be mined from the open pit. The results of a waste characterization study and 3D modeling indicate that approximately 237Mt of the waste material is PAG. The current plan is to store this potentially reactive material subaqueously within the Tailings Management Facility.

Of the remaining material, it is anticipated that approximately 24Mt of Non-Acid Generating (NAG) waste rock (including overburden) will be produced during pre-production mining. This material will be used in the construction of the TMF Starter (Stage 1) embankment and mine site infrastructure. A further 100Mt will be required for dam raises throughout the mine life. All additional waste material in excess of that required for construction will be stockpiled to the east and west of the open pit respectively.

Details of the site characteristics, geotechnical conditions, waste rock dump stability classification, and stability assessment are contained in the Knight Piésold report “Mine Waste and Water Management Design Report” (Ref. No. VA101-458/11-1).

During mine operations approximately 116Mt of Low Grade ore will be stored in temporary stockpiles and processed later in the mine life. Of this figure, approximately 74Mt is potentially reactive material and will be stockpiled adjacent to the TMF to maximize both surface and seepage run-off into the TMF.

18.8 PIPELINES

Several pipeline systems will support the waste and water management requirements of the project. Generally, the pipelines will be used for either tailings distribution or site water management. The pipelines include the following:

- Bulk Tailings Pipeline;
- Cleaner Tailings Pipeline;
- Reclaim Water Pipeline;
- Pit Dewatering Pipeline; and
- Water Management Pipelines.

Each system will have a dedicated pump station and pipeline. All water management pipelines will generally report to the TMF supernatant pond for long-term storage of site water. The reclaim water pipeline will draw water from the supernatant pond to the process plant pond for use in the milling process, as required. The two tailings distribution pipelines will deliver tailings waste from the process plant to the TMF. The design of the tailings distribution, reclaim water, pit dewatering, and site water management pipelines are described in the following sections.

18.8.1 *TAILINGS DISTRIBUTION PIPELINES*

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. 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% dry by weight, with a solids density of 2.66t/m^3. The cleaner tailings slurry concentration was estimated to be 32.7% dry by weight, with a solids density of 3.11t/m^3.

Lock cycle metallurgical testwork produced one sample each of cleaner and bulk tailings. The cleaner tailings contain high levels of sulphur and are PAG, while the bulk tailings contain lower levels of sulphur and are non-PAG. The geochemical characteristics of both tailings types have been evaluated by SRK Consulting Inc. (SRK) as part of the feasibility design studies.

The two tailings streams will be transported in separate pipelines to the TMF. These pipelines have been identified as the Bulk Tailings Pipeline and the Cleaner Tailings Pipeline. Both pipelines will follow a pipeline service road from the plant site towards the TMF at an approximate grade of 2%. This arrangement allows for approximately 10 years of gravity fed tailings deposition before relocation of the road and pipeline.

The Bulk Tailings Pipeline will consist of 32” to 36” piping. Bulk tailings will be transported to the TMF Embankment and discharged from the embankment crest using spigots to build tailings beaches.

The Cleaner Tailings Pipeline will consist of 14” DR17 HDPE pipe. The cleaner tailings will be transported to a location within the TMF near to the reclaim barge. The cleaner tailings will be deposited in an area that maintains the tailings solids in a subaqueous state perpetually.

18.8.2 *RECLAIM WATER PIPELINE*

The Reclaim Water Pipeline will consist of 30” DR11 HDPE and a 30” carbon steel pipe. The reclaim system is designed to deliver the process water requirements for a nominal throughput of 70,000t/d. The water will be pumped from the TMF supernatant pond to a process water holding pond at the mill for reuse in the process. The pumps will be mounted on a floating pump station (reclaim) barge. The average elevation of the tailings supernatant pond will increase steadily over the life of the project, with seasonal variations due to snowmelt, runoff, precipitation, evaporation, and consumption of water in the tailings and waste rock voids. The rising elevation will reduce the pumping head requirements over the life of the project.

The proposed reclaim barge (floating pump station) will be located on the northwest side of the TMF supernatant pond, approximately halfway between the plant site and the TMF embankment. The pump station will include two running pumps and one identical installed standby pump (spare). The barge will be periodically relocated closer to shore as the elevation of the supernatant pond rises. This relocation will typically decrease the length of the reclaim water pipeline.

18.8.3 *PIT DEWATERING PIPELINE*

The dewatering system for the open pit will pump all seepage and precipitation inflows out of the pit. The system will keep the pit bottom dry during normal operating conditions. Water removed from the open pit will be pumped into the TMF to allow for sediment settling before being used for mill process water.

The design capacity for the dewatering system is controlled by the pit inflows during the 1 in 10 year, 24-hour storm event. It was assumed that the water will be removed over a ten-day period, during which time mining operations can continue in other active areas of the pit. The peak operational design capacity of the pumping system ranges from 100 L/s during the first phase of the pit, to 400 L/s for the final pit.

The pit dewatering pump system will use 16” and 20” HDPE piping with a DR11 or DR13.5 rating, depending on the location, to convey the water from the bottom of the pit to the TMF. The design of the pit dewatering system was staged to allow for expansion of the system as the pit depth and pit area increases, and consequently design flows and design heads increase. Water will be pumped from the base of the pit to the pit rim using skid-mounted diesel drive pumps. There will initially be one in-pit pump station consisting of one active pump and one installed standby pump. The in-pit system will progressively expand during operations to include three in-pit pump stations with three pumps each. The pit dewatering system will lift water from the base of the pit to the pit rim following access ramps constructed for mining. The pipeline will exit the pit on the southwest side near the main haul road. There will be two permanent pump stations outside of the pit to convey the water to the TMF for long term storage and reuse. The permanent stations will consist of electric drive pumps. The pump stations will initially consist of one active pump and one installed standby pump. The pump stations will expand progressively during operations to include three active pumps and one installed standby pump.

18.8.4 *WATER MANAGEMENT PIPELINES*

Water management ponds will be constructed downstream of the non-PAG waste rock stockpiles, the LGO stockpiles and the TMF embankments. The water management ponds downstream of the non-PAG waste stockpiles and the LGO stockpiles will provide collection points for any surface runoff and infiltration from the stockpiles because of precipitation in these catchment areas. This water will be recycled using dedicated pump stations and pipelines to the TMF supernatant pond for use as reclaim water for the milling process.

The water management pipelines will be the following nominal diameters, materials and lengths:

- TMF Main Embankment – 14" diameter HDPE DR11 and 1,400m long
- Non-PAG Waste Rock Stockpile – 20" diameter HDPE DR11 and 4,200m long
- Non-PAG LGO Stockpile – 12" diameter HDPE DR11 and 2,100m long
- PAG LGO Stockpile – 12" diameter HDPE DR11 and 650m long
- TMF North Embankment – 4" diameter HDPE DR11 and 400m long

19 MARKET STUDIES AND CONTRACTS

19.1 COMMODITY PRICES

In general, commodity prices have risen as a result of the falling US dollar value, rising costs and increasing demand. As such, a higher plateau of long-term metal pricing has now been established compared to previous years. With respect to copper prices, the general consensus is for long term pricing forecasts at a low range of US$2.50/lb to a high of US$3.00/lb. Like copper, the gold price over the same period has also risen substantially as a result of economic concerns. Today the consensus for gold long term prices range from a low of US$900 to a high of approximately US$1,325. For project evaluation the following long term metal prices have been assumed:

Table 19-1 Long Term Metal Price Assumptions

Metal	Currency (US$)
Copper	$3.00/lb
Gold	$1,250/oz
Silver	$20/oz

19.2 COPPER SMELTER TERMS

YMI retained Cliveden AG to provide a view on smelting terms. These terms were used as a basis for the financial model and sensitivity analyses contained in Section 22. Although there are no established contracts in place for the sale of concentrate it is important to note that the terms provided assume that the copper concentrate will be delivered to Asian smelters.

Key copper smelter charges that vary significantly with the market are the Treatment Charge (TC) and Refining Charge (RC). Forecasts by Wood Mackenzie, as at October 2013, suggest a long term average for TC of US$93/dmt concentrate and a RC of US$0.93/lb Cu based on a standard 25% clean copper concentrate and an average copper price of US$3.50/lb (Table 19-3). However, it is Cliveden's view that at these prices additional smelting capacity would be encouraged into the market putting downward pressure on costs. As such, Cliveden believe that a Treatment Charge of US$80/dmt concentrate and a Refining Charge of US$0.08/lb Cu based on a long term average copper price of US$2.75 - US$3.00/lb would be more appropriate. These figures would be closer to the average seen during the last decade which has seen a significant expansion in smelting capacity. As such, for the purpose of this study a long term market cost is assumed:

Table 19-2 Treatment/Refining Assumptions

Charge	Currency (US$)
Treatment Charge	$80/dmt concentrate
Refining Charge	$0.08/lb Cu

Table 19-3: Outlook Long Term TCRC

Long term TCRCs (nominal and constant 2013$). Concentrate grade 25% assumed.

	MOD TC ($/t)	MOD RC (c/lb)	MOD PP (c/lb)	MOD Combined (c/lb)	MOD Price (c/lb)	2013$ TC ($/t)	2013$ RC (c/lb)	2013$ PP (c/lb)	2013$ Combined (c/lb)	2013$ Price (c/lb)	TCRC as % of Cu price
2010	46.5	4.7	0.0	13.4	342	50	5.0	0.0	14.4	366	3.9%
2011	56	5.6	0.0	16.2	400	58	5.8	0.0	16.8	415	4.0%
2012	63.5	6.4	0.0	18.4	361	64.6	6.5	0.0	18.7	367	5.1%
2013	70	7.0	0.0	20.2	328	70	7.0	0.0	20.2	328	6.2%
2014	85	8.5	0.0	24.6	310	83	8.3	0.0	24.0	303	7.9%
2015	84	8.4	0.0	24.3	295	80	8.0	0.0	23.1	280	8.3%
2016	86	8.6	0.0	24.8	300	80	8.0	0.0	23.1	280	8.3%
2017	91	9.1	0.0	26.3	328	83	8.3	0.0	24.0	300	8.0%
2018	95	9.5	0.0	27.4	363	85	8.5	0.0	24.6	325	7.6%
2019	108	10.8	0.0	31.3	382	95	9.5	0.0	27.5	335	8.2%
2020	108	10.8	0.0	31.2	407	93	9.3	0.0	26.9	350	7.7%
2021	110	11.0	0.0	31.8	415	93	9.3	0.0	26.9	350	7.7%
2022	112	11.2	0.0	32.5	423	93	9.3	0.0	26.9	350	7.7%
2023	115	11.5	0.0	33.1	431	93	9.3	0.0	26.9	350	7.7%
2024	117	11.7	0.0	33.8	440	93	9.3	0.0	26.9	350	7.7%
1975 - 2011	65	7.4	1.2	21.0	113	113	13.0	2.2	36.6	197	18.6%
2002 - 2011	63	6.3	2.8	21.0	71	226	22.6	10.1	75.4	255	29.6%
2012 - 2024	96	9.6	0.0	27.7	368	85	8.5	0.0	24.6	328	7.5%

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19.3 Payable Metals

Payment terms have been assumed as follows:

Table 19-4 Metal Deductions

Payment Terms		
Copper	Pay 96.50%, over 30g/dmt, pay 90%; Subject to minimum deduction of 1 unit	
Gold	<1 g/dmt, no payment Over 1g/dmt, pay 90%	Refining charge US$5/oz
Silver	<30 g/dmt, no payment over 30g/dmt, pay 90%	Refining charge US$0.35/oz

19.4 Concentrate Marketability

YMI engaged G&T to run a pilot program January 2012 to produce concentrate in sufficient quantity for potential buyers to perform their own analysis (in addition to performing a comprehensive metallurgical test work program to confirm the metallurgical response of the Harper Creek ore).

A total of 860kg at a grade of 0.31% Cu was treated to produce 7.8kg of concentrate packaged in 500g parcels. Overall recovery was 91% producing a 25.5% copper grade in the concentrate. The concentrate is considered clean with no element approaching typical smelter penalty levels. Table 19-5 details the minor element analysis of this concentrate.

To date, concentrate samples have been sent to 3 independent smelters. Their analysis confirms all elements are below typical penalty levels.

Table 19-5: Estimated Average Concentrate Quantity & Quality Characteristics

Description	Units	Value
Copper Concentrate Production*	t/a	231
Copper Grade in Concentrate	% Cu	25.5
Gold Grade in Concentrate	g/t Au	1.79
Silver Grade in Concentrate	g/t Ag	70.98
Copper in Concentrate	lbs x 1,000	3,360,546
Gold in Concentrate	oz x 1,000	372.1
Silver in Concentrate	oz x 1,000	14,736
Minor Elements		
Aluminum	%	1.08
Antimony	%	0.002
Arsenic	g/t	104
Bismuth	g/t	13
Cadmium	g/t	34
Calcium	%	0.67
Carbon	%	0.83
Cobalt	g/t	110
Copper	%	25.5
Fluorine	g/t	151
Gold	g/t	1.92
Iron	%	27.3
Lead	%	0.17
Magnesium	%	0.82
Manganese	%	0.024
Mercury	g/t	<1
Molybdenum	%	0.020
Nickel	g/t	350
Phosphorus	g/t	418
Selenium	g/t	2
Silicon	%	3.33
Sulphur	%	30.0
Silver	g/t	122
Zinc	%	0.35

*Note: Average life of mine annual production

19.5 Realization Costs

19.5.1 *Transportation*

Concentrate from the mine site will be transported by truck to Vavenby where it will be transferred to rail for delivery to the Port of Vancouver. Transportation charges were based on proposals received by qualified transportation companies and rail charges were provided by CN Rail. Ocean transport cost was provided by YMI.

Table 19-6 Transportation Assumptions

Description	Cost (C$ / US$) Percentage (%)
Truck transport (site to Vavenby)	C$13.77/wmt concentrate
Rail handling and transport	C$26.99/wmt concentrate
Port storage and handling	C$27.00/wmt concentrate
Ocean transport	US$42.00/wmt concentrate
Moisture content	8%

19.6 Insurance

An insurance rate of 0.15% was applied to the provisional invoice value of the concentrate to cover transport from the mine site to the smelter.

19.7 Owners Representation

An Owners representation rate of US$0.10/wmt concentrate will be applied to the provisional invoice value of the concentrate to cover attendance during unloading at the smelter, supervising the taking of samples for assaying, and determining moisture content.

19.8 Losses

Concentrate losses are estimated at 0.5% during shipment from the mine to smelter.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL STUDIES

YMI initiated environmental baseline studies on the Project in December 2007. In April 2011, it increased the scope of baseline data collection to fulfill the information requirements of an environmental assessment application submission.

Climate and hydrology information has been collected for the project site for input to the FS site water balance. The hydrology study component is strengthened by a long term data set from a local Water Survey of Canada hydrological monitoring station located on Harper Creek. Ongoing climate and hydrology studies are intended to refine the runoff estimates for the project site for input to the environmental assessment application.

Geochemistry studies were initiated in 2011 to assess acid rock drainage and metals leaching potential of the tailings and waste rock. Barrels containing representative samples of ore and waste rock were placed at the project site during spring 2012 to collect leachate samples under in-situ conditions. The results of ongoing geochemistry studies will be used for water quality modeling and waste management in the environmental assessment application.

Hydrogeology studies were initiated in fall 2010 with additional groundwater wells installed in 2011 and 2012. Several rounds of quarterly groundwater quality sampling were completed with additional sampling planned throughout 2014. Ongoing hydrogeology studies are intended for input to the environmental assessment application and future monitoring activities.

Surface water quality sampling has been ongoing since 2007 with additional sampling locations added in 2011. The lab water quality sample results show elevated levels of cadmium, copper, aluminum, and iron. The surface water quality data will be used for input to water quality modeling and to inform future monitoring activities.

Fisheries studies were initiated in 2008 followed by a comprehensive sampling program in 2011. The project footprint was confirmed to be fishless. Fish were identified in the lower reaches of streams draining the project footprint. The project will reduce flows in two tributaries of upper Harper Creek used by bull trout for summer rearing and overwintering. These impacts to bull trout habitat will require approval under the federal Fisheries Act and will be addressed by providing offset for loss of fish habitat. Ongoing fisheries studies are intended to collect detailed fish and fish habitat information in upper Harper Creek to inform the environmental assessment application.

Vegetation and wildlife studies were initiated in 2008 followed by additional studies in 2011. The project area and the region surrounding it are heavily impacted by forest harvesting activities although pockets of remnant forest remain. Rare plants were confirmed in two separate areas within the project footprint. Two provincially listed bat species were detected using acoustic recording equipment. Ongoing vegetation and wildlife studies will collect additional information on rare plants and bats for input to the environmental assessment application and to identify appropriate mitigation measures.

Baseline studies related to human health are being conducted for air quality and noise due to the proximity of the project to Vavenby. The proponent is also required to assess the human health effects of the project on country food use. Visual quality assessment is being conducted to model the visual impacts of the Project.

In summary, there are no known issues identified that would materially affect the ability of YMI to extract minerals as part of developing the Project.

20.2 PROJECT PERMITTING REQUIREMENTS

Many permits and authorizations will be required for the development of the Project to proceed. Two key permits that will be required are the Mines Act Permit, enabled under the Mines Act of BC, and the Waste Management Permit, enabled under the Environmental Management Act of BC. A preliminary list of Provincial and Federal permits, authorizations and licenses are listed in Table 4-2 (Section 4).

Under the Mines Act, there will be a need to post a reclamation bond. Bonding costs will be developed as part of the permitting phase, and the bond will be posted prior to the commencement of construction activities. There may be a need to post a performance bond associated with fish compensation works, if required. As with the reclamation bond, this will be evaluated during the permitting phase of the Project.

20.3 SOCIAL AND COMMUNITY RELATIONS

20.3.1 *ENVIRONMENTAL ASSESSMENT REVIEW PROCESS*

Major mining projects in BC are subject to an environmental assessment as part of the legislated certification and permitting process and depending on the scope, may be subject to a federal review. The Project is undergoing a harmonized provincial/federal environmental assessment.

The Environmental Assessment Office (EAO) manages the assessment process for proposed projects in BC. The Section 10 Order, issued by the EAO in September 2008, determined that the Project must undergo a provincial Environmental Assessment (EA). The Section 11 Order outlining the scope, procedures and methods for conducting the EA was issued by the EAO in September 2009. The Section 11 Order identified the Simpcw First Nation and the Adams Lake Indian Band as the main First Nations within the Project Area. A Section 13 Order was issued by the EAO in October 2012, amending the Section 11 Order and adding the Neskonlith Indian Band and the Little Shuswap Indian Band.

The updated project description was accepted by the EAO and the Canadian Environmental Assessment Agency (CEAA) in January 2011 and posted on the EAO website. Based on this information the CEAA determined that an EA was required. The CEAA comprehensive study commenced in April 2011. The federal Major Projects Management Office posted the signed Project Agreement for the Project in June 2011. The Project Agreement with the federal Deputy Ministers outlines the federal review process and timeline for the federal review.

The EA Application Information Requirements for the Project were accepted by the EAO and CEAA in October 2011.

The EA process includes opportunities to involve the public and consultation with First Nations, technical studies to identify and examine potential adverse effects, the development of strategies to avoid, mitigate or reduce potential adverse effects, and the development of comprehensive reports summarizing input and findings.

Environmental baseline studies for the Project have been ongoing since 2007. In 2011 additional baseline studies were added. Studies include surface and ground water quality and quantity, climate and air quality, geochemistry, hydrogeology, fisheries and aquatics, terrestrial vegetation, plants and wildlife, archaeology and traditional use. An Open House was held in Clearwater BC in June 2011 to share information with the public and First Nations about the Application Information Requirements. Technical Working Groups were established that include provincial and federal government regulators, municipal representatives, and First Nations. To date four technical working group meetings have been held to discuss the information needed in the Environmental Assessment (EA) Application. None of the environmental parameters identified to date are considered to have a material impact on the development and operations of the Project.

20.3.2 *Social and Community Requirements*

The Project is located in the area known as the North Thompson Valley within the Thompson Nicola Regional District. The nearest communities to the Project are Vavenby, Birch Island and Clearwater. Some of the mine-related infrastructure, including the rail load-out facility will be located in Vavenby. Overall, these communities are expected to benefit directly and indirectly from the Project. Locally there is much support for the development of the Project. Economic development is needed to offset the economic downturn of the forestry sector, and closing of several mills in the North Thompson Valley.

YMI is committed to hiring local people. During the construction period the Project is expected to employ a peak workforce of approximately 600 people. When fully operational, the Project will support about 450 direct jobs and approximately 800 to 1,000 indirect jobs in the area.

20.3.3 *First Nations*

The Project is located within the asserted traditional territory of the Simpcw First Nation and the Adams Lake Indian Band. From information collected to date, YMI understands that Adams Lake is a member of the Lakes Division which includes the Little Shuswap Indian Band and Neskonlith Indian Band. All four of these First Nations are members of the Secwepemc Nation and the Shuswap Nation Tribal Council (SNTC). SNTC is a political organization that works on matters of common concern to all its members, including the development of self-government and the settlement of the aboriginal land title question.

Chu Chua is the main reserve of the Simpcw, meaning “the People of the North Thompson River”. It is located on the North Thompson River, 20 minutes from Barriere, BC. The Simpcw have 4 other reserves located near Little Fort, Louis Creek and Dunn Lake. Simpcw has approximately 650 members (2011); 250 live on reserve. The Adams Lake Indian Band has approximately 740 members, half of which live on the seven reserves located near Chase and Shuswap Lake.

YMI has initiated a range of consultation activities with stakeholders since 2006. This includes one-one discussions with local landowners. Consultation with local First Nations has been, and continues to be, an important part of these activities. The YMI team is maintaining a detailed record of all communications including telephone calls, emails, letters, and meetings with First Nations and other stakeholders. These records document the specific issues identified and discussed.

YMI continues to work closely with First Nations on the development of working agreements. YMI signed a Negotiation Agreement with Simpcw First Nation, and a General Services Agreement with both Simpcw and Adams Lake. Both communities had members involved in the baseline studies and field work, including the archaeological impact assessment, for the Project.

20.4 Waste Stockpiles, Tailings And Water Management

20.4.1 *Mine Waste Management*

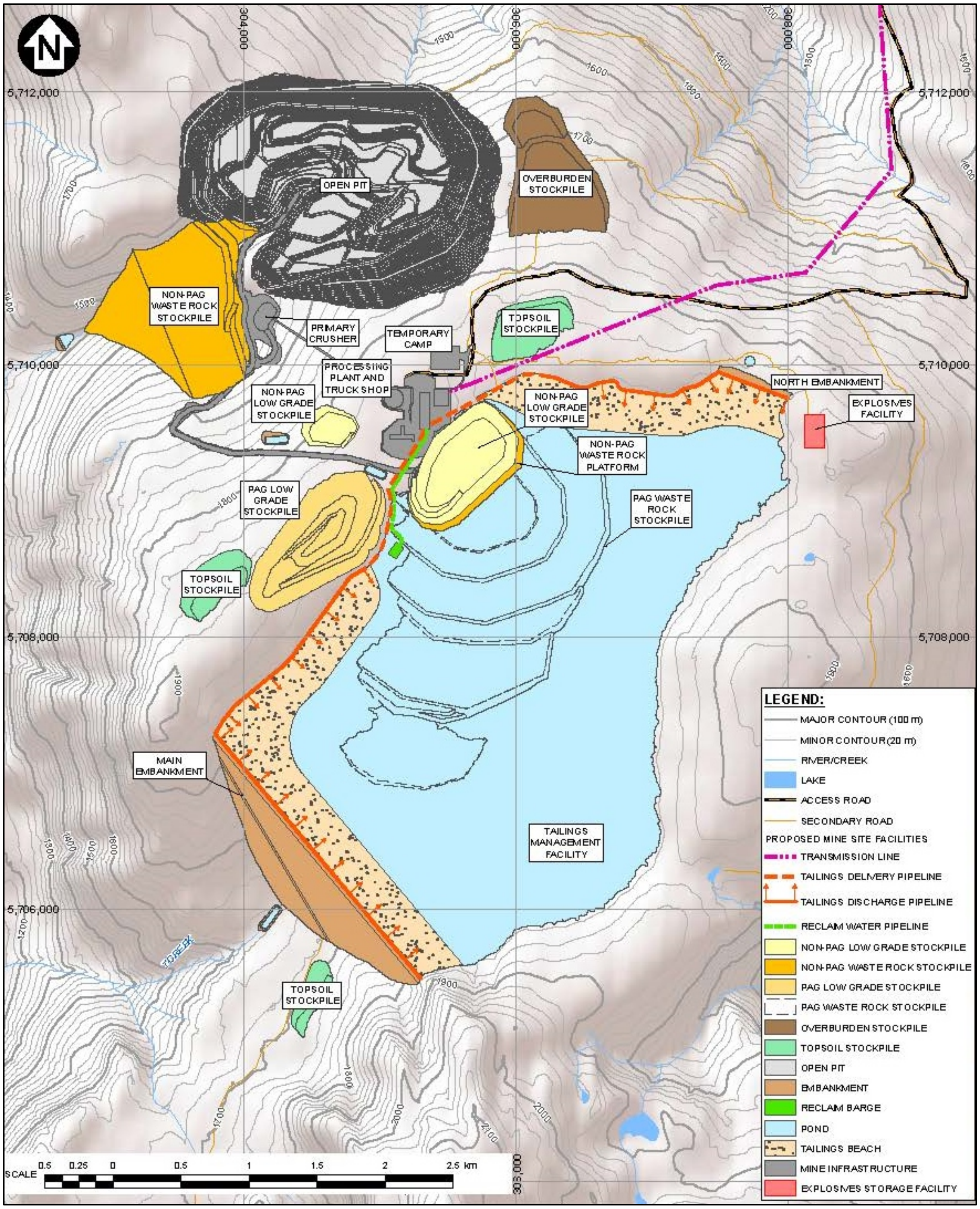
20.4.1.1 Introduction

The principle design objectives for the waste rock stockpiles and tailings management facility (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 effective 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 (ARD) and minimization of metal leaching from reactive tailings and waste rock, and
- staged development of the facility over the life of the project.

The general arrangement showing the layout of the project at the maximum extent of all facilities is shown on Figure 20-1.

Figure 20-1 Project Layout Maximum Footprint



Knight Piesold Consulting, May 2014

20.4.2 *Design and Operating Criteria*

Design and operating criteria have been developed for the FS to facilitate preparation of drawings and material take-offs to support an economic comparison with +15%/-5% accuracy. 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 TMF design provided to KP 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 12Mm^3, and storage of the inflow design flood (IDF) with at least 1m of freeboard for wave run-up.
- Annual staging of the TMF dam 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 1m of freeboard for wave run-up.
- Conventional slurry tailings disposal with tailings solids approximately 35% by weight.
- Water for the process plant sourced from TMF supernatant pond at a flow rate of 5,520m^3/hour.

The Canadian Dam Association (CDA) Dam Safety Guidelines (2007) 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 design of the project:

- IDF – 2/3 between 1/1,000 year and probable maximum flood (PMF)
- EDGM – 1/5,000 year return period.

A draft bulletin released by the CDA in 2012, entitled Application of 2007 Dam Safety Guidelines to Mining Dams – Design Considerations, suggests that in closure of the TMF, a mining dam should be designed for the PMF and the 1/10,000 year return period EDGM regardless of dam classification. These design event levels were adopted for closure of the TMF.

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 upstream zone of the TMF main embankment during first five years.
    - Surplus co-disposed of within the TMF in such a manner 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 Low-grade Ore (LGO)
    - Up to 7.5Mt 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.
    - Surface water and infiltration collected in a water management pond and directed to the TMF.
- Tailings
    - Two tailings streams, consisting of approximately 93% rougher scavenger (bulk) tailings and 7% cleaner scavenger (cleaner) tailings.
    - Bulk Tailings to be conveyed by pipeline and discharged from embankment crests.
    - Cleaner tailings to be conveyed by pipeline and discharged subaqueously into the TMF pond.
    - Tailings during low-grade processing will be discharged into the open pit.

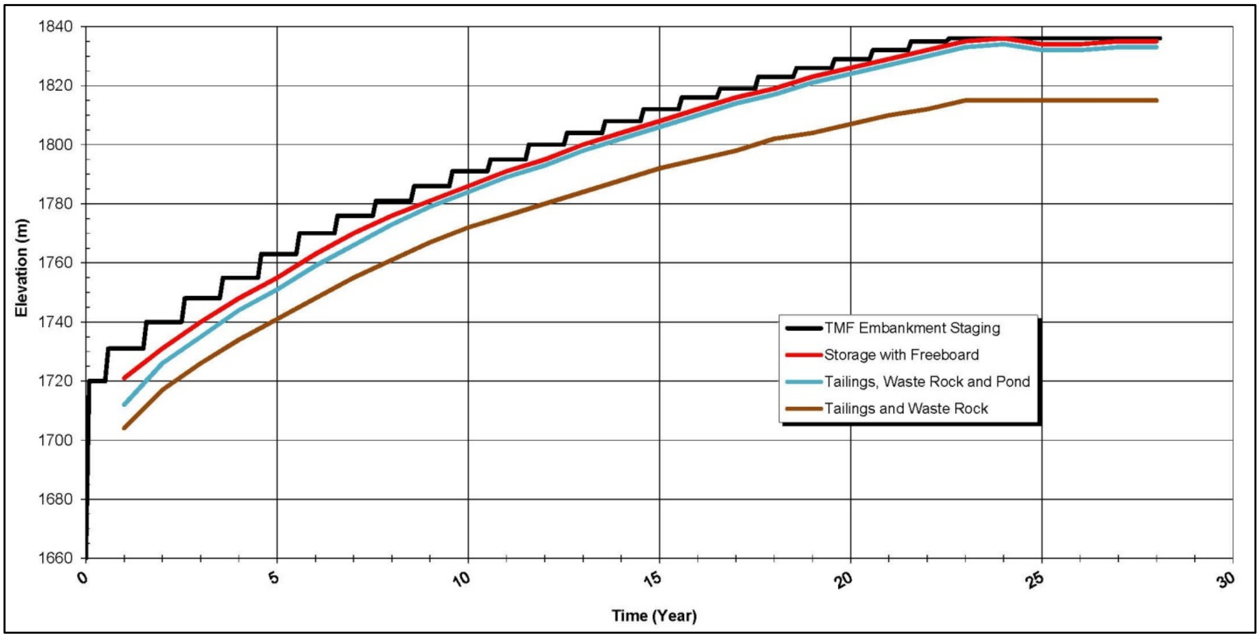
20.4.3 *TAILINGS MANAGEMENT FACILITY DESIGN*

The TMF was designed to permanently store tailings and PAG waste rock generated during operation of the mine (Figure 20-1). Specific overall features of the TMF are listed below:

- cofferdams and sediment 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 storage 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;
- tailings distribution system;
- tailings beaches;
- reclaim water system; and
- supernatant water pond.

The filling schedule for the TMF was based on the detailed mine schedule for the project. 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 20-2.

Figure 20-2: TMF Filling Curve



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The earth-rockfill dams will comprise the following zones:

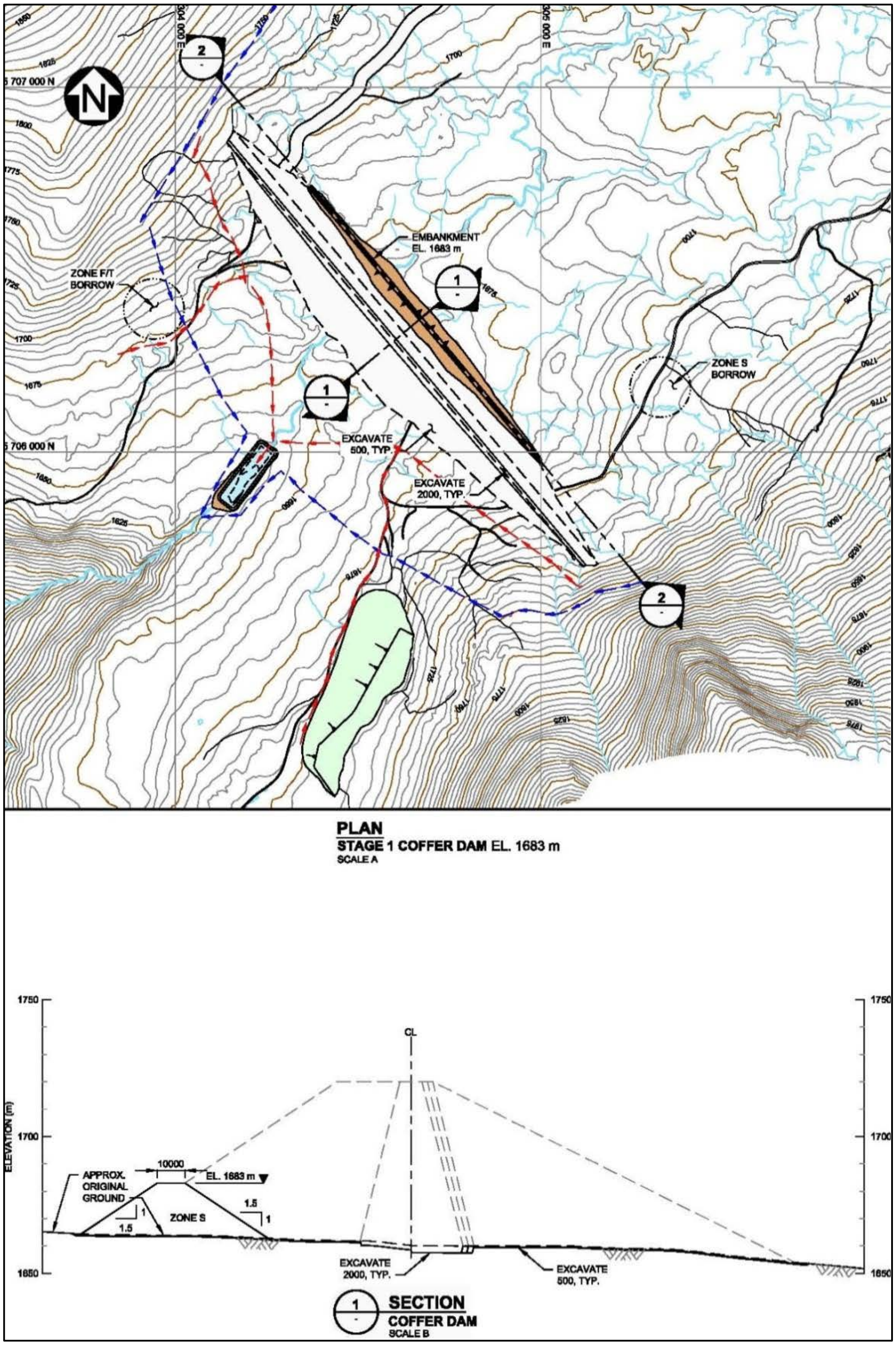
- The core zone (Zone S) will be constructed from low-permeability glacial till from nearby external borrows and from pit stripping. The material will consist of well-graded silty sand with some gravel with a fines content of 20% to 60% passing the #200 sieve. This material will generally require no processing except for the removal of oversized particles. The material will be placed in maximum 300mm 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 600mm 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 600mm 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:

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

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 1,683m with an embankment crest 10m-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 2km of the dam. The total volume of the cofferdam was estimated to be 400,000m^3. The site plan and typical cross section for the cofferdam are shown on Figure 20-3.

Figure 20-3 Cofferdam Plan and Section



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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 1,683m 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.

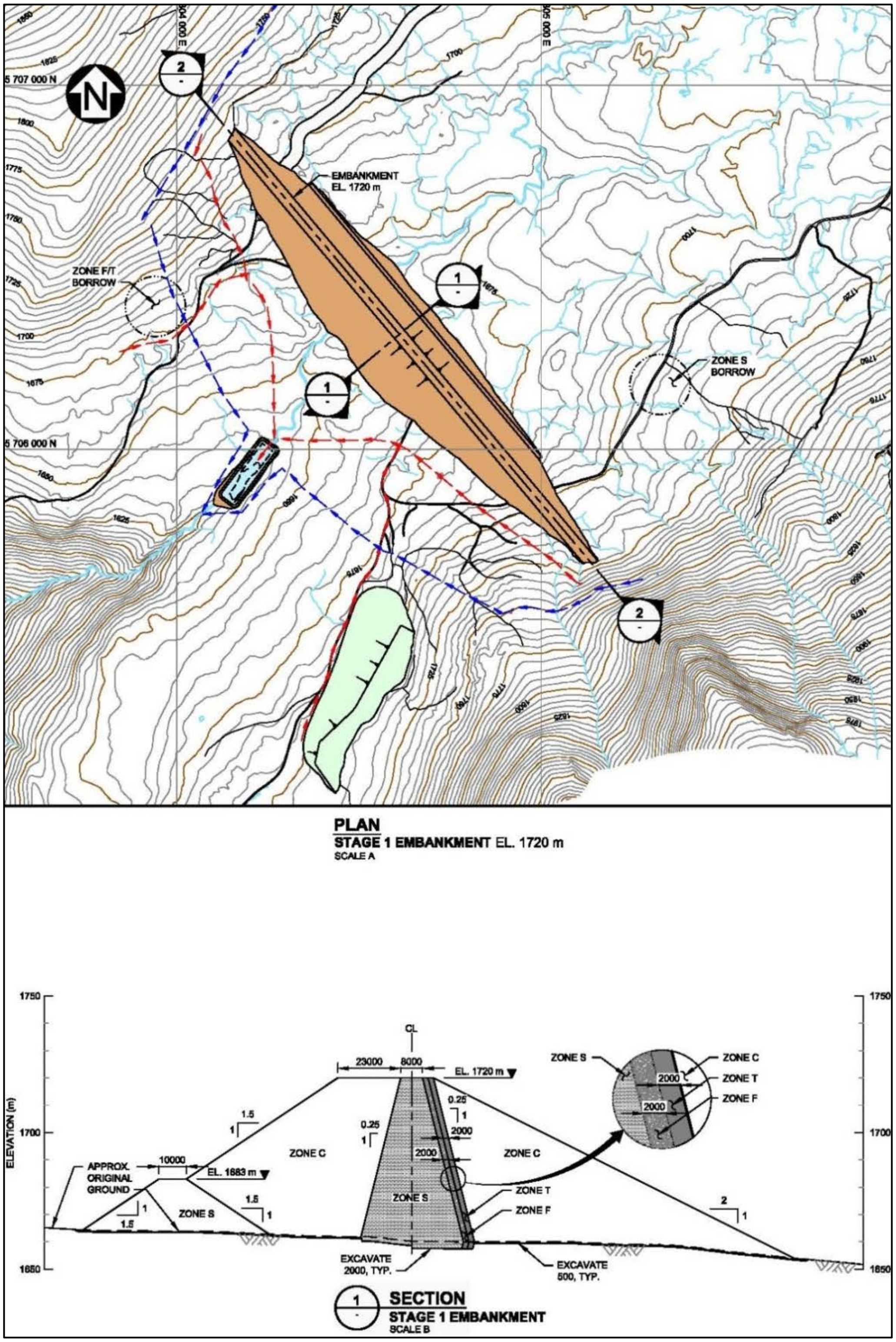
Initial impoundment of water behind the cofferdam will be planned to occur in August following the annual freshet, which generally provides the vast majority of the run-off at the project site. The cofferdam has been sized to provide storage capacity for four months (August through November) of statistically 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.

The main embankment will reach a Stage 1 elevation of 1,720m (approximately 70m in height at the maximum dam section) prior to start-up 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 1m of freeboard for wave run-up. The site plan and typical cross section for the Stage 1 embankment are shown on Figure 18-4.

The Stage 1 main embankment 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 commence immediately following completion of the cofferdam to reach an elevation of 1,700m by May, which will provide the required storage capacity for the maximum pond volume of 12Mm^3 in time to collect and store the summer freshet. The summer 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 1,720m prior to the start of operations.

Figure 20-4: Stage 1 Main Embankment Plan and Section



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Construction of Stage 1 will require approximately 7.35Mm^3 of construction material, which will be provided from pit stripping (5.55Mm^3) and external borrow sources (1.8Mm^3). The volumes of each material zone required for the cofferdam and the Stage 1 embankment are presented below in Table 20-1.

Table 20-1: Stage 1 Main Embankment Volumes

Zone - Material Type	Stage 1 Embankment Volumes		
	Phase		Total
(units)	Cofferdam	Stage 1	
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**

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 30m-wide using non-PAG waste rock from the open pit. An access ramp will be built into each step-out to allow ongoing 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 11m to 3m depending on storage characteristics of the TMF and the volume of waste to be managed in the upcoming year.

The final stage of the main embankment was designed to reach an elevation of 1,836m, which is approximately 185m in height at the maximum dam section. It will be capable of securely storing over 585Mt of process tailings, 237Mt of PAG waste rock, site contact water, and the IDF with at least 1 m of freeboard for wave run-up. A site plan of the final stage of the main embankment and a typical section showing the staged raises of the main embankment are provided as Figure 20-5 and Figure 20-6, respectively.

Figure 20-5: Main Embankment Site Plan – Final Stage

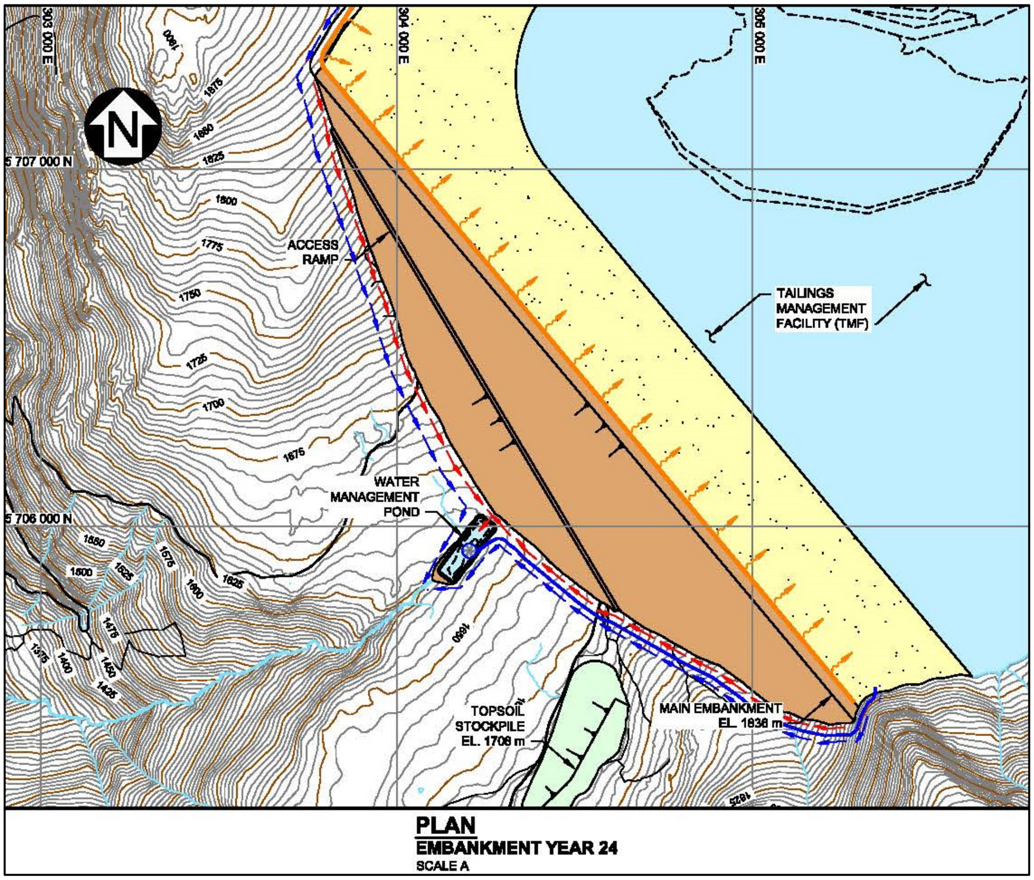

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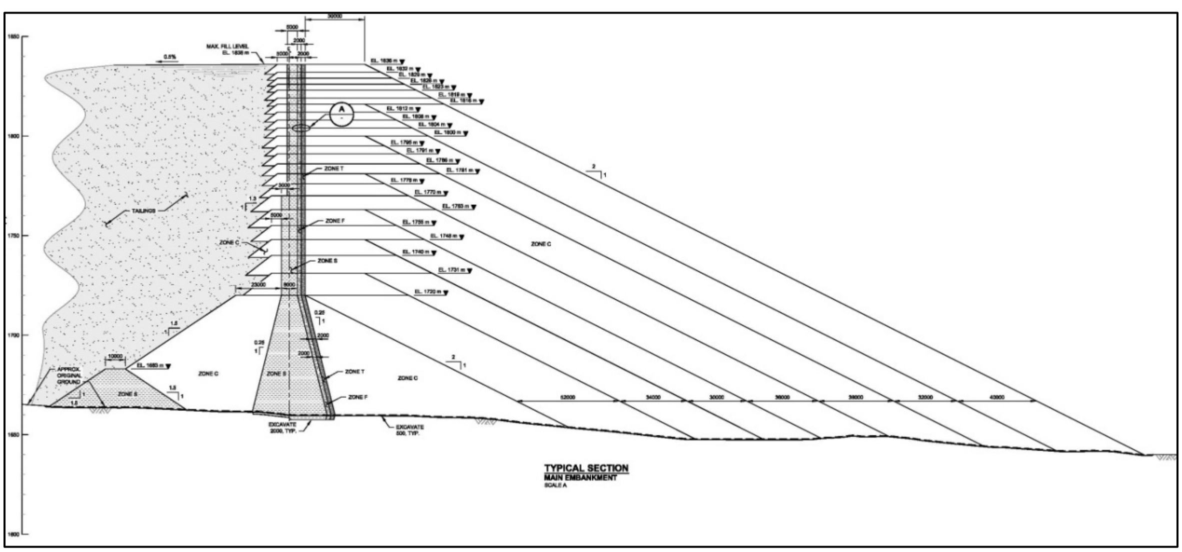
Figure 20-6: Main Embankment Typical Section – Final Stage



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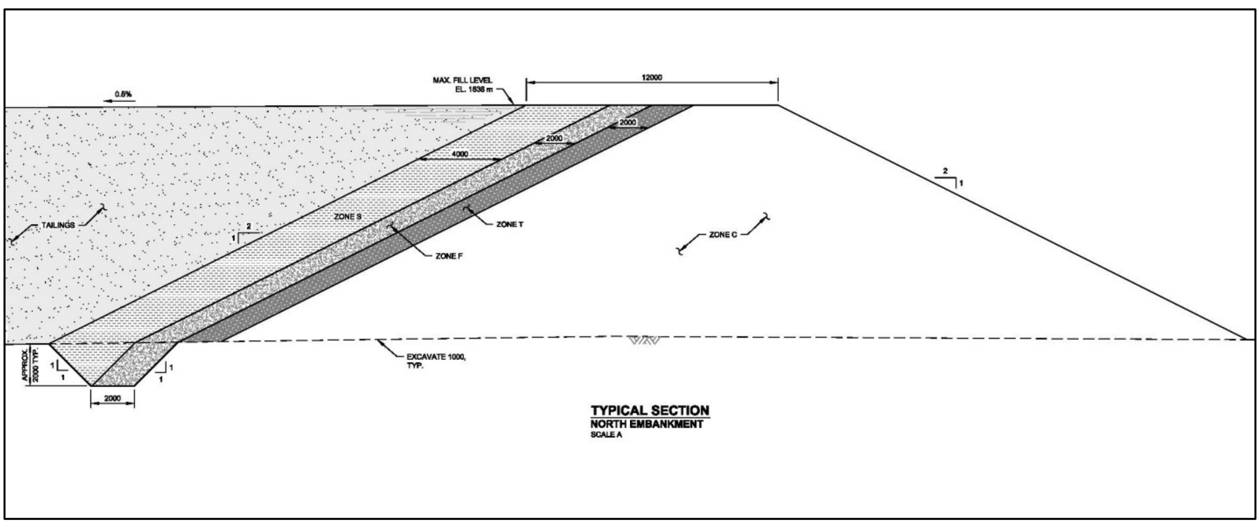
The total fill requirement for the main embankment is 58.4Mm^3 of construction material, which will be provided from pit stripping (55.7Mm^3) and external borrow sources (2.7Mm^3). The volume of each material zone that is required for the main embankment is presented below in Table 20-2.

Table 20-2: Sustaining Embankment Volumes

Zone - Material Type	Main Embankment Volumes		
	Phase		Total
(Units)	Stage 1 Total	Sustaining	
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**

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 11m high. The embankment was designed with a 12m 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 4m thick and two downstream filter/transition layers (Zones F and T) to maintain the integrity of the Zone S layer. The typical cross section for the north embankment is shown on Figure 20-7.

Figure 20-7: TMF North Embankment Cross Section



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Mining operations will cease in the open pit during Year 24 and the mine will begin processing low-grade ore from the site stockpiles. The tailings deposition will continue to occur during this period, but the tailings will be directed towards the open pit for long-term storage.

20.4.4 *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. This includes construction of the Mt. Milligan Tailings Storage Facility, 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.

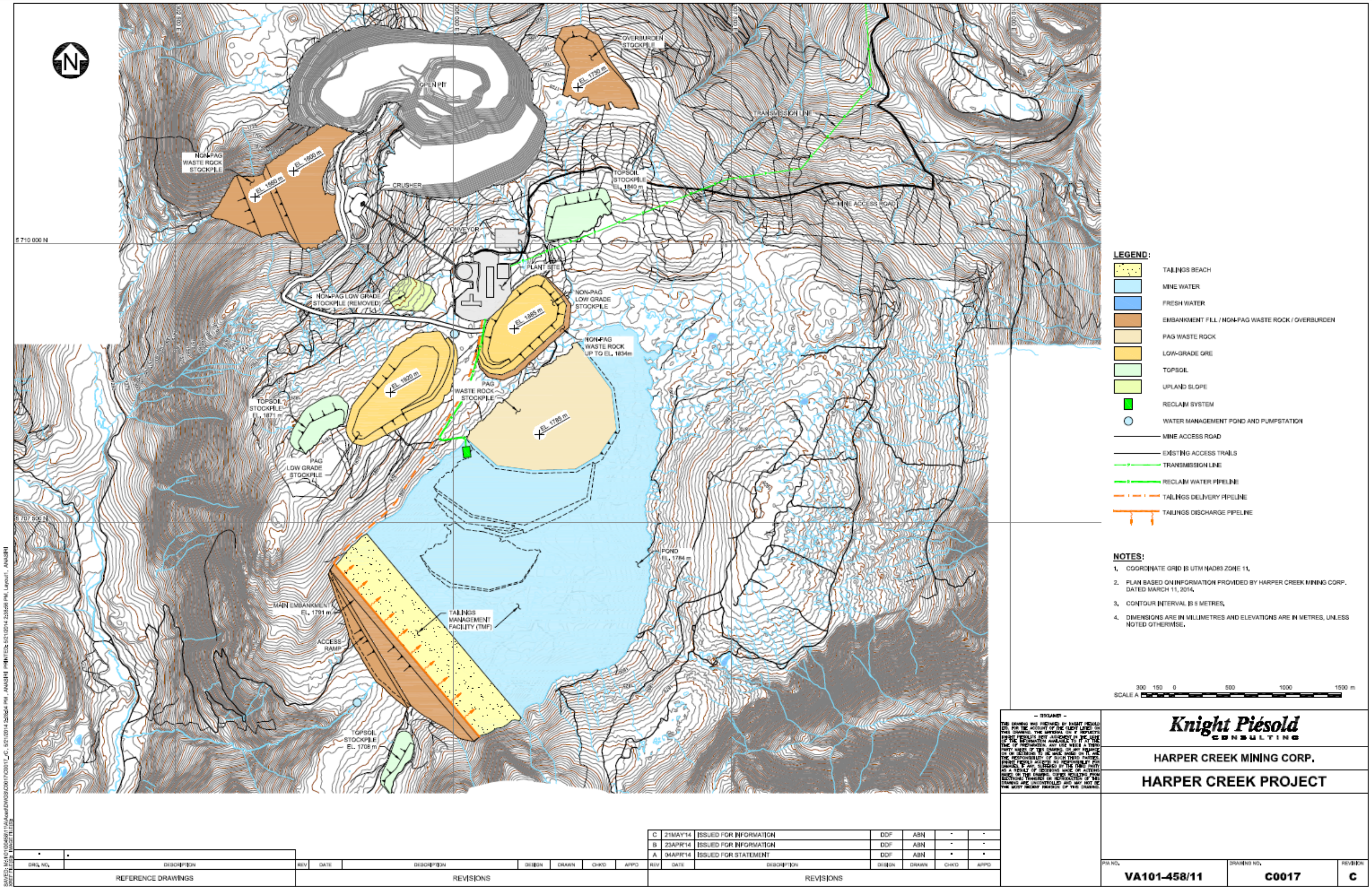
Haul roads will require extra attention in winter to maintain safety and prevent haul time from increasing dramatically. Extra equipment such as sand trucks and graders will be used, along with a supply of road sand.

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.

20.4.5 *PAG Waste Rock Stockpile*

The PAG waste rock disposal 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 TMF filling level but will be several metres higher 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 waste storage 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 20-8.

Figure 20-8: TMF Filling Year 10



20.4.6 *Mine Water Management*

20.4.6.1 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 ditches;
- 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 ponds 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 ponds; and
- 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.

20.4.7 *CONSTRUCTION WATER MANAGEMENT*

Eight discrete areas of development have been identified within the Project boundary that will require a sediment and erosion control plan to be prepared during the detailed design 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).

Specific surface water control elements and measures will be implemented in these areas to minimize erosion and prevent sediment discharge into surrounding areas. 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 re-vegetation;
- mulching;
- rolled erosion control products;
- surface roughening;
- re-contouring;
- 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 0.5m of freeboard and to settle out sediment particles sized 0.01mm (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.

20.4.8 *Operational and Closure Water Management*

The operations and closure water management strategies for the Project have been developed by identifying the size and position of the planned mine site facilities and establishing estimated catchment area boundaries based on the mine site development concept.

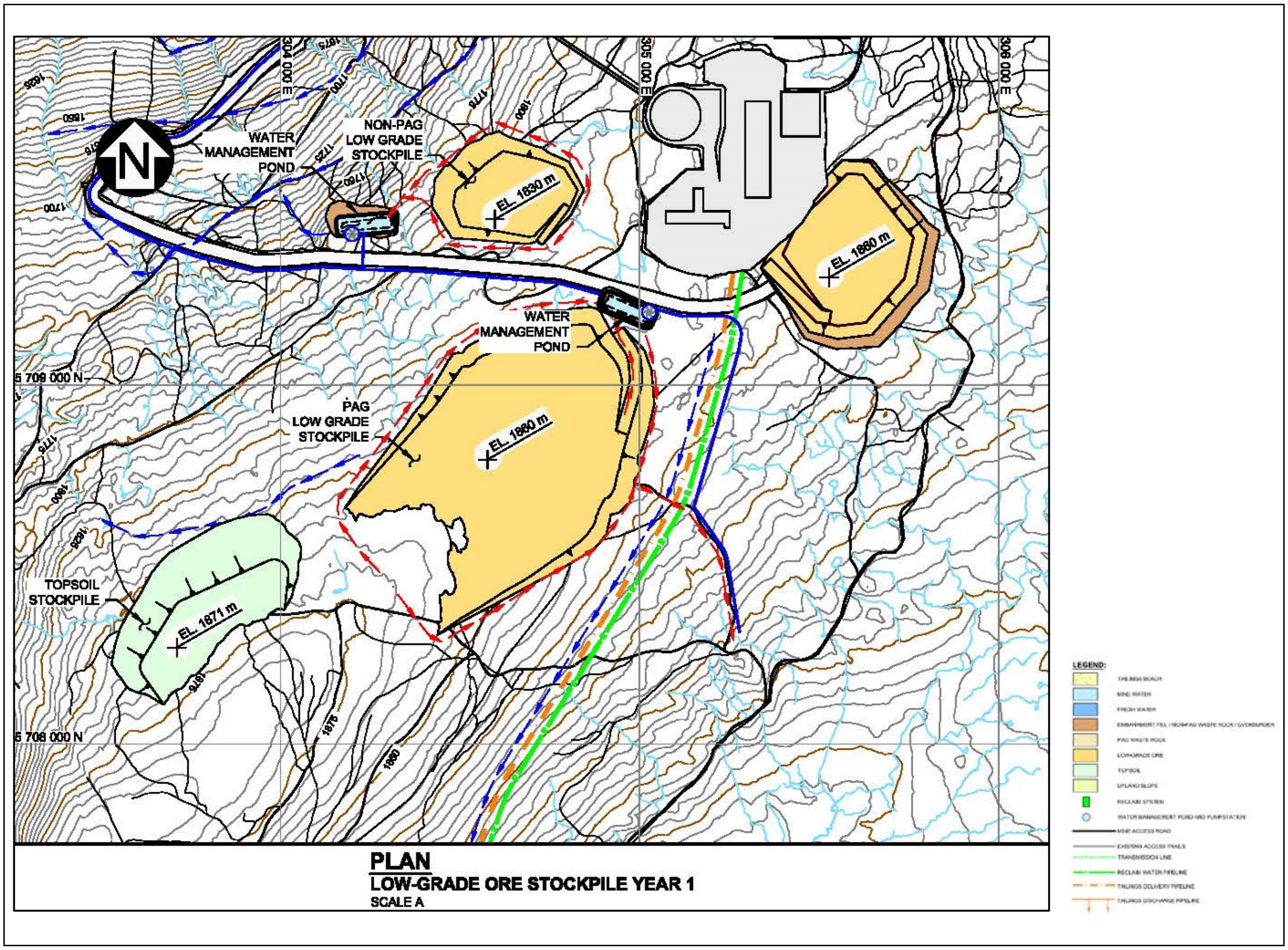
The Project site is in an area of high annual precipitation with a mean annual precipitation (MAP) of approximately 1,050mm. The project site is anticipated to have a surplus of water over the life of the project. Fresh water diversion ditches and mine water collection ditches were designed to separate mine site contact water from fresh water, and to minimize collection of fresh water in the mine site area. A large portion of site drainage during operations and closure will drain by gravity directly to the TMF. The majority of all seepage from the TMF and waste rock stockpiles will be collected and directed to the TMF. Mine contact water will be collected and water quality monitored at water management ponds downstream of the following mine facilities:

- open pit;
- non-PAG waste rock stockpile;
- low-grade ore stockpiles;
- overburden stockpile;
- TMF main embankment;
- TMF north embankment, and

- plant site.

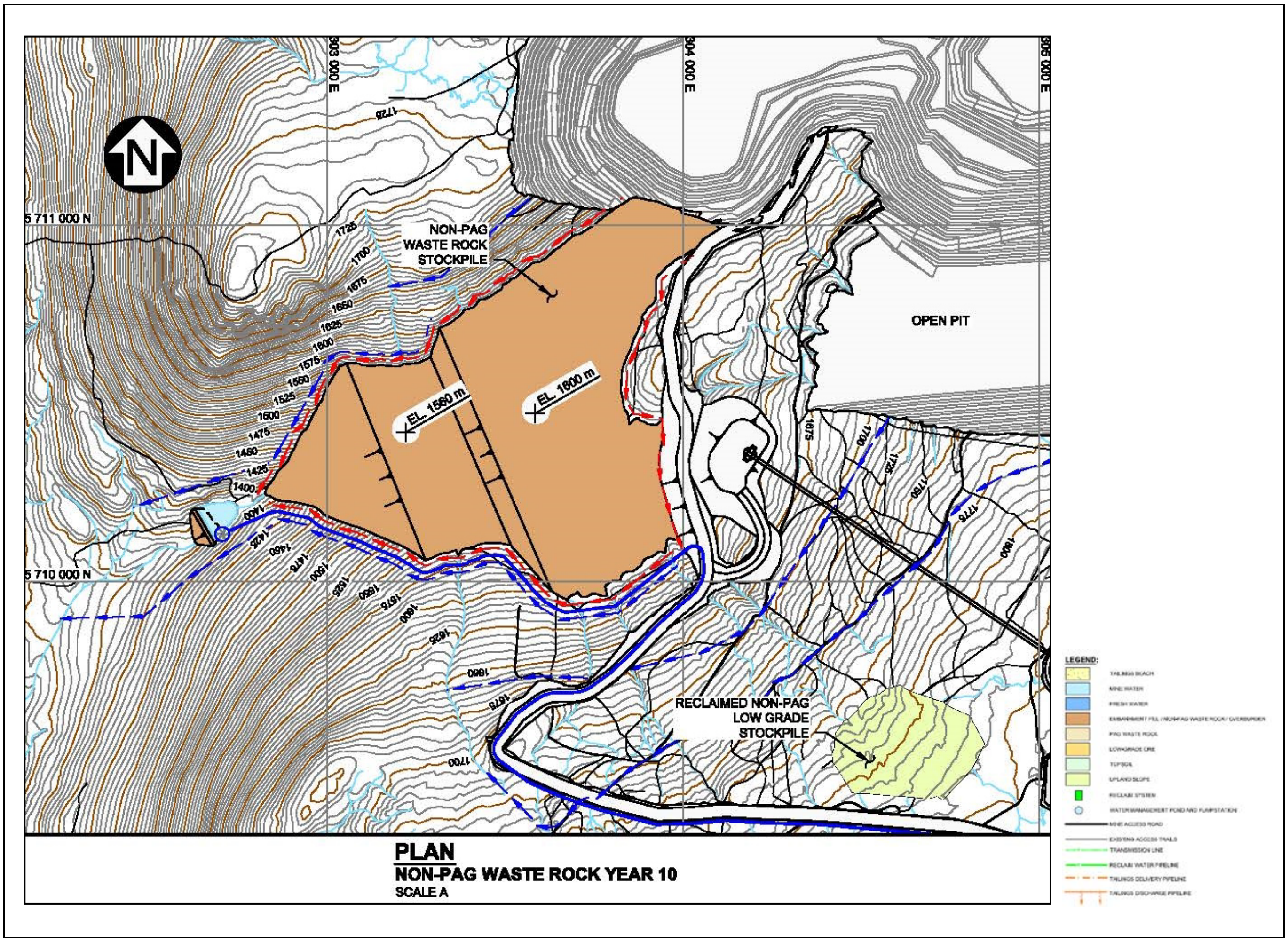
Mine site contact water with out of compliance water quality will be conveyed to the TMF for long-term storage, and reuse in the mill process. Water will be pumped back to the TMF using dedicated pump stations and pipelines to the TMF supernatant pond. The water management ponds and pump system layouts for the low-grade stockpiles and non-PAG waste rock stockpile are shown on Figure 20-9 and 20-10, respectively.

Figure 20-9: Water Management - Low-grade Ore Stockpiles Year 1



modified from Knight Piésold Consulting, May 2014

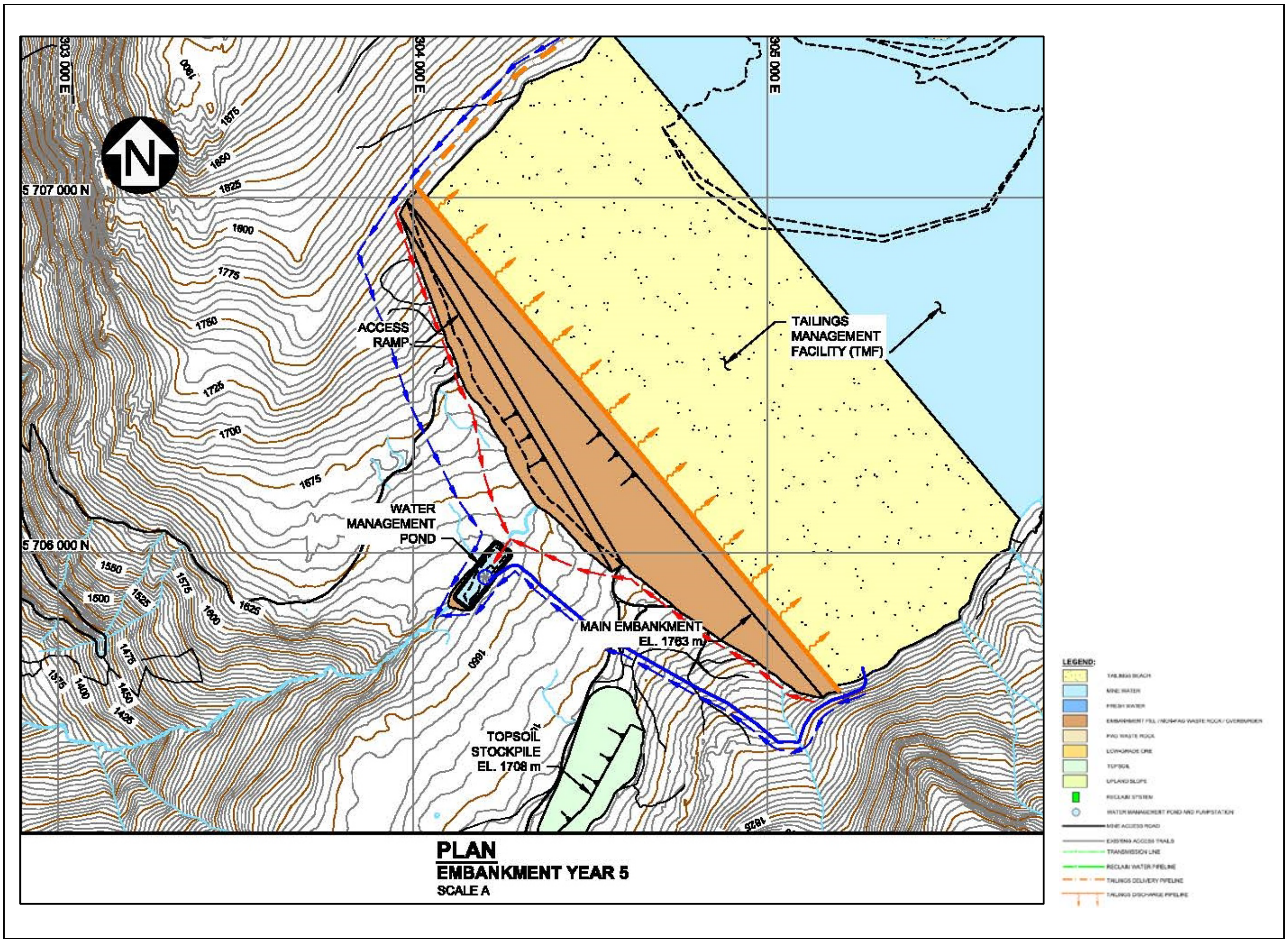
Figure 20-10: Water Management - Non-PAG Waste Rock Stockpile Year 10



modified from Knight Piésold Consulting, May 2014

Multiple levels of seepage control have been included in the design of the TMF to minimize seepage loses. These measures include the development of extensive tailings beaches (thereby isolating the supernatant pond from the embankment), filter zones downstream of the embankment, foundation and embankment drains to reduce seepage gradients and the water management pond downstream of the embankment. Water collected in the water management pond will be recycled to the TMF for storage and used as reclaim water. The layout of water management facilities downstream of the TMF main embankment is shown on Figure 20-11.

Figure 20-11: Water Management - TMF Main Embankment Year 5



modified from Knight Piésold Consulting, May 2014

Water reclaimed from the tailings ponds will be delivered to the reclaim water tank at the mill. The water will consist of supernatant from the settled tailings and runoff from precipitation and snowmelt within the reporting catchment areas. The reclaim water system will utilize a barge-mounted pump station equipped with vertical turbine pumps sized to deliver 5,520m^3/hour of reclaim water. The reclaim barge will be anchored within the TMF during Years 1 through 25 of operations and then moved to the open pit for the remainder of the low-grade ore processing period of operations and closure.

Mine site contact water during closure will be pumped to the TMF from the non-PAG waste rock stockpile and TMF water management ponds. This pumpback period will continue until water quality is suitable for direct release to the downstream receiving environment. The TMF pond will fill with fresh water runoff from the surrounding catchment area until it reaches the closure spillway invert elevation and begins to spill to T-creek, which is predicted to happen one year after cessation of operations. The open pit will begin to fill with runoff and direct precipitation until it reaches an elevation of 1,530m, at which time the pit water pond will be maintained at this elevation and the annual surplus inflow to the open pit will be pumped to the TMF using the relocated reclaim water system and subsequently released to the downstream receiving environment.

20.4.9 *Mine Closure and 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:

20.4.9.1 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.

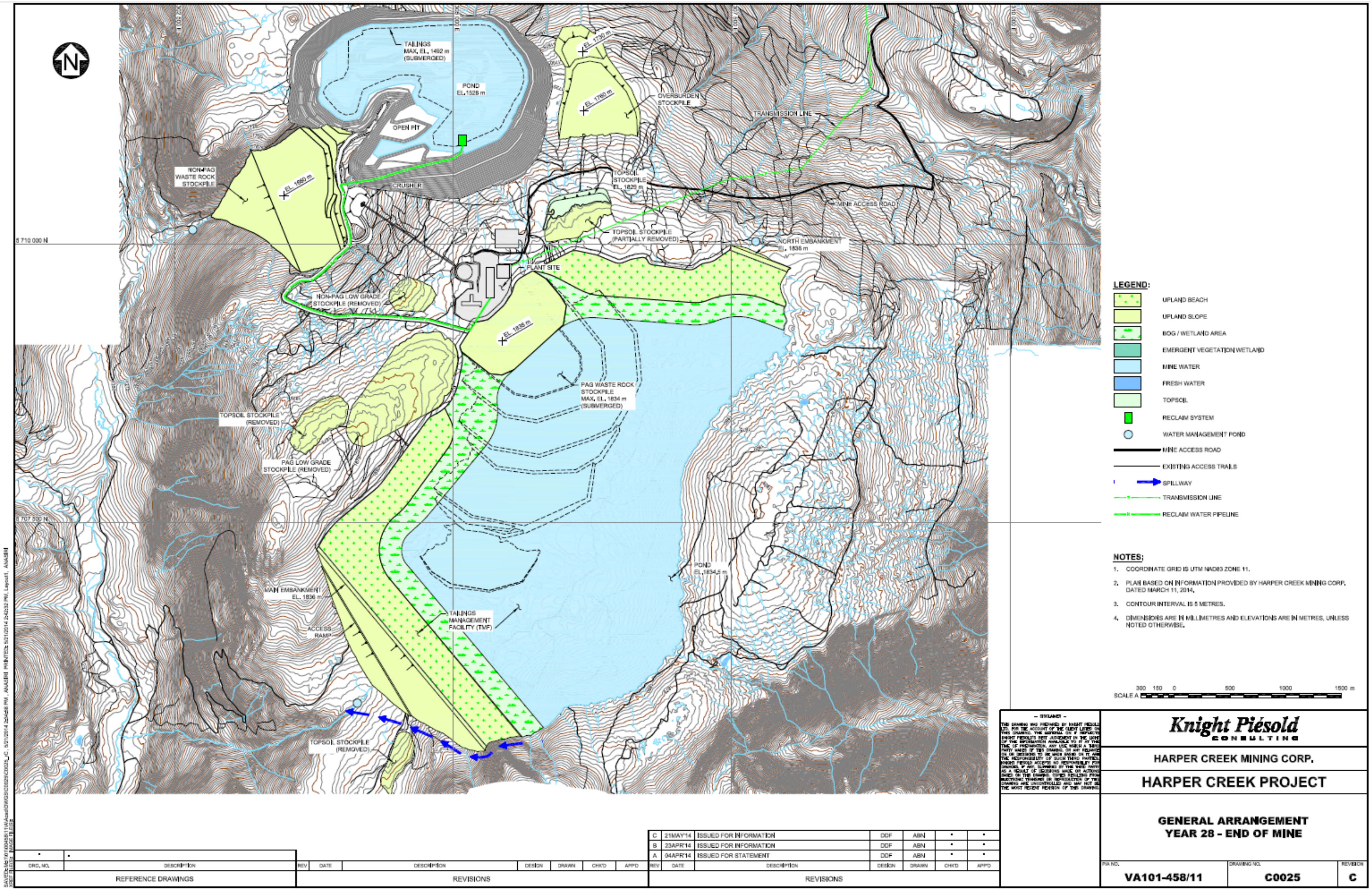
20.4.9.2 Final Reclamation Activities

- 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 300mm 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. Figure 18-12 provides an illustration of the general arrangement of the project in 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 1,530m and 1,545m, 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 1,555m, which allows for 10m of freeboard to manage storm inflows.

Figure 20-12: General Arrangement Year 28



21 CAPITAL AND OPERATING COSTS

21.1 INITIAL & SUSTAINING CAPITAL

The Capital Cost Estimate (CAPEX) for the Project's process plant, power line, infrastructure and mine was prepared in accordance with standard industry best practice for this level of study and to a level of definition and intended accuracy of +15%/-5%. In some cases, the FS improved on this level of accuracy because of the level of detail examined. The plant is a typical processing plant with few, if any, unusual features that are unknown or difficult to define.

The principal consultants, namely, Merit, Allnorth, KP, together with consultant specialists in mine planning, geoscience, metallurgy and environmental management provided input for the CAPEX. Each consultant provided a detailed set of quantities, assisted with the costs, were involved in the scheduling and strategy for execution, and met regularly to discuss the evolvement of the study. Full participation and open communication amongst all parties occurred during the FS process.

With the help of the engineering and consulting groups, Merit talked to contractors, freight forwarders, vendors and service suppliers to establish today's market cost to as great a degree as possible. The exercise reviewed items in detail such as:

- availability and cost of construction and operating personnel;
- productivities to be expected using information from a recently completed project in the same general area;
- locally available materials and support facilities;
- construction materials availability at site;
- work schedules and accommodations;
- use of some of the mining fleet for construction activities; and
- areas of risk.

There are approximately 1.7M direct and indirect man hours associated with the construction of the Project excluding pre-production personnel, vendor representatives and other visitors. Manpower at the project site will peak at approximately 600, excluding pre-production mine personnel, main access road and power line contractors, as well as the contractors that will construct the facilities at the off-site Vavenby administration and load out facility.

The cost estimate covers the direct and indirect costs encompassing all of the traditional items that are standard to any project, and incorporates thinking based on similar projects that have been, or are being built over the last several years in and around the Province of British Columbia, and with some reference to other off-shore projects that have similar process plant sizes. In addition, the FS consultants are currently involved with a number of similar size and type of other feasibility projects and therefore, much information was made available by way of group experience and knowledge. For example, Merit provided Construction Management services for the Copper Mountain Project located near Princeton, BC. This first-hand knowledge was invaluable when verifying construction costs and methods.

21.1.1 *SUMMARY*

Four main categories comprise the CAPEX:

- Direct costs;
- Indirect costs;
- Contingency; and
- Owner's costs.

The basis for the CAPEX for both Initial and Sustaining Capital was developed as follows:

- Merit (process plant, tailing facility, infrastructure and site utilities/services);
- Nilsson (mine development, mine equipment and mine infrastructure);
- Allnorth (process equipment pricing and power line);
- Laurion (process equipment selection);
- KP (Tailings and Water Management); and
- YMI (Owner's Costs and Working Capital).

The CAPEX for the Project is C$1,025.8M as at 1Q 2014 evaluation subject to qualifications, assumptions and exclusions, all of which are detailed herein (Table 21-1).

Table 21-1: Capital Cost Summary Estimate

Capital Cost Estimate Summary	
Pre Production Capital (Year -2 and -1)	**Currency* (C$M)**
Direct Costs	
Mining & Pre-production Development	298.0
Plant Site Infrastructure	9.5
Site Services & Utilities	13.4
Process	279.6
Ancillaries	27.4
Power Supply & Distribution	47.7
Tailings & Water Reclaim	54.0
Total Direct	**729.5**
Indirect Costs	
Owner's Costs	25.6
Indirects	162.1
Total Indirect	187.7
Subtotal	917.2
Contingency	90.7
Subtotal Direct + Indirect	**1,007.9**
Bonding	7.9
PST	10.0
Total Project	**C$1,025.8**

*Note: Totals may not add due to rounding

21.1.2 *Basis of Estimate*

Direct Costs were based on the following information:

- Process flow diagrams, site layout and general arrangement drawings, equipment list, electrical single line diagrams, piping and instrumentation diagrams, and drawings from similar projects.
- Budget submissions for the design/supply of new major and secondary equipment provided by vendors in accordance with specifications and/or datasheets developed by the engineering consultants.
- Prices for permanent materials based on supplier quotations, in-house data, and current market conditions.
- Material Quantity Take-offs provided by Allnorth and KP.
- Labour rates provided by local and regional construction contractors. The rates used were derived from extensive research combining strategy of labour pool research and input from experienced union (CLRA) and alternative union contractors (CLAC) with rates that reflect the most likely situation where a combination work force would be invited to bid on the work.
- Productivities for installing equipment and materials were provided by local and regional construction contractors familiar with the project location and local conditions. Productivities that can be expected for this location have been based on measured results from other projects as well as in-house data.

- Supply and installation prices from experienced vendors of pre-engineered and modular buildings.
- Freight costs for all major components, especially those from offshore and for those extra large pieces of equipment that require multiple handling to get to site, have been determined from experienced freight forwarders. Allowances were made for packaging by the vendors and included in their proposals.
- Topographic data and geotechnical information and recommendations provided by the YMI and KP.
- An engineering, procurement and construction management (EPCM) project execution strategy and project schedule. The project schedule is more detailed than one might typically expect for a FS. There are more than 500 line items included, and the schedule was continually updated as information was received that changed durations or dates. The project schedule was the linchpin for the cash flow calculations.
- Camp and catering prices were derived from quotations received from multiple companies, some of whom are currently performing the same work for Merit on other projects. An average price per man day for accommodations for construction personnel was calculated in conjunction with the contractor of choice.
- Project site accommodation will not be provided for the power line and main access road crews. They will supply their own, best positioned, camps. In addition, site accommodation will not be provided for the mining personnel and they will be housed in the nearby communities. These costs are carried in Owner's Costs.

Escalation is not included in the CAPEX and there are no allowances for currency fluctuations. The rates of exchange are provided in Table 21-2.

21.1.2.1 Direct Costs

Direct Costs include all new equipment, new materials, and installation of all permanent facilities associated with:

- Crushing, material handling and processing facilities;
- Process building;
- Infrastructure roads and site preparation;
- Power supply and distribution;
- Remote concentrate load out;
- Pre-production mining;
- Tailings Management Facility;
- Assay lab;
- Warehousing;
- Administration;
- Truck shop;
- Yard services and other utilities;
- Control and communications system;
- Plant mobile equipment;
- Fuel storage;
- Cold storage; and
- Explosives production.

21.1.2.2 Indirect Costs

Indirect costs include the following:

- Temporary construction facilities including worker's camp, secure lay-down areas, warehouses, etc.;
- Temporary construction services including contractor's mobilization/demobilization cost;
- Construction equipment;
- Freight;
- Vendor representatives;
- First fills and capital spares;
- Engineering, procurement and construction management services (including travel expenses);
- Third party engineering;
- Pre-operational testing services including associated materials;
- Quality assurance;
- Surveying;
- Owner's costs; and
- Start-up and commissioning allowance.

21.1.2.3 Contingency Allowance

The overall contingency for project development was estimated at the rate of 9.89%. The contingency allowance was included to cover unforeseeable costs within the scope of the estimate. Table 21.3 sets out the contingencies for various categories of work (Section 21.6).

21.1.2.4 Qualifications and Assumptions

The following qualifications and assumptions were made:

- Construction work is based on unit and fixed price contracts (no cost plus or time and materials arrangements).
- Budget quotes from vendors for equipment and materials are valid to within +/- 5% of the purchase price.
- Concrete aggregate and suitable backfill material will be available locally and suitable areas were identified by the Owner's geotechnical consultants.
- Soil conditions will be adequate for foundation bearing pressures.
- Other than the construction of the TMF, where weather plays an important role, construction activities will be carried out in a continuous program.
- Labour productivities are valid and were adjusted for northern locations and established with input from experienced contractors and Merits' in-house database of current projects.
- Bulk materials such as cement, rebar, structural steel and plate, cable, cable tray, and piping are all readily available in the scheduled timeframe.
- Capital equipment is available in the timeframes shown and delivery has been verified by the requisite supplier.

21.1.3 *Pricing*

Pricing for commodities or the design/supply of equipment was not based on binding quotations. Budgetary quotations were obtained from vendors and contractors for major equipment and unit rates. "Budgetary quotations" generally mean that indicative pricing was provided for specified equipment, materials and productivity but no commitment was made to secure the equipment or materials at this price for a future date.

21.1.4 *Taxes*

HST is excluded from the capital cost estimate. An estimate of applicable PST has been made and is included in the capital cost estimate.

21.1.5 *Currency, Estimate Base Date & Foreign Exchange*

All project capital costs are in Canadian Dollars (C$) with the following provisions:

- Costs are based on 1Q 2014 market conditions with no provision for escalation beyond this date;
- Costs submitted in other currencies have been converted to Canadian;
- Foreign currency exchange rates applied to the capital cost exercise are set out in Table 21-2. No provision was made for variations in the currency exchange rates from those indicated. No provision was made for any taxes or fees applicable to currency exchanges.

Table 21-2: Foreign Currency FX Rate

Exchange	C$
C to US	1.111
C to Euro	1.524

21.1.6 *Accuracy and Monte Carlo Analysis*

The Capex, including contingency, for the mine, process plant, tailing storage facility and infrastructure has been prepared to an accuracy level of +15%/-5%.

Merit performed a Monte Carlo analysis of the capital cost, and defined the limits of all the major cost line items excluding the contingency, bonding and PST, on approximately C$918M. It was established that the chance of executing the project for less than C$918M is 17.2%, with a 90% chance of executing the project for C$955M or less, and a 95% chance of executing the project for C$961M or less. The maximum value (100 percentile) is C$990M. The most probable value is between C$930M and C$936M. The analysis indicates that the current total Capex, including contingency, of about C$1.008bn is higher than the maximum value predicted for the cost of the project by approximately C$18M.

The Monte Carlo analysis shows that the items that most affect capital costs are the indirects and currency exchange, followed by mining, mechanical, tailings and earthworks.

The Monte Carlo model deals with the "known unknowns" where Merit assumed maximum and minimum values or variations of the things Merit was certain will change. The analysis did not consider the "unknown unknowns", the hidden costs or surprises during execution or engineering changes. Nor did the analysis consider the other risks external to the execution that could affect the final execution costs such as delays, extreme weather, strikes or lockouts, etc.

21.1.7 *PROJECT EXECUTION PLAN*

The Capex is based on the assumption that YMI will follow the project execution plan described in Section 24. Any deviation from this plan may have an impact on both project schedule and costs.

21.1.8 *EXCLUSIONS*

The Capex excludes allowances for the following:

- Escalation during construction;
- Warehouse inventories other than initial fills;
- Interest during construction;
- Schedule delays and associated costs such as those caused by;
- Scope changes;
- Unidentified ground conditions;
- Extraordinary climatic events, force majeure;
- Labour disputes;
- Insurance, bonding, permits and legal costs;
- Receipt of information beyond the control of EPCM contractors;
- Schedule recovery or acceleration;
- Cost of financing;
- Customs duties;
- Community relations;
- Sunk costs;
- Research and exploration drilling;
- Sustaining capital;
- Working capital – is a part of Sustaining Costs, with any pre-operations costs included in Owner's Costs;
- Closure costs; and
- Salvage values.

21.1.9 *PROJECT DIRECT COSTS*

21.1.9.1 Quantities and Unit Pricing

Engineering material take-offs were based on quantities derived by the engineering groups from project drawings, sketches and similar projects.

21.1.9.2 Earthworks

Unit prices were based on recent prices from regional civil contractors who have knowledge of the conditions in the area. Quantities were developed by the engineering groups and based on topographic drawings at 1m to 2m contour intervals.

The earthmoving unit rates were calculated based on data obtained from local contractors and Merit's historic information for similar projects. The rates include the rental of earthmoving equipment, operators, fuel. Mobilization/demobilization costs are included in the indirect costs.

It has been assumed, from geotechnical information based on actual site conditions, that concrete aggregates, structural backfill, granular base, road base and sub-base will be supplied from the borrow pits established at the open pit and tailings facility impoundment locations in particular. The unit costs associated with these materials include borrow pit development (crushing and screening) and transport costs.

21.1.9.3 Concrete, Formwork and Reinforcing Steel

Concrete quantities were determined by the engineering groups using FS level drawings and experience from previous projects of a similar nature. The unit rates for concrete placement and finishing were derived from Merit's in-house data of similar projects and the rates were cross-checked against unit rates provided by regional industrial contractors. Most construction aggregates for structural fill and concrete will be sourced in the open pit for use with an on-site batch plant located close to the new plant site. Materials used for the tailings dam core will be sourced in the tailings dam area. Prices for the processing of the aggregates were solicited from local contractors.

Formwork was estimated for each type of concrete classification and includes local supply, form oil, accessories and shoring, and stripping. Prices include one and half times for reuse of the forms where practical.

Reinforcing steel quantities were developed based on estimated weight per m^3 of concrete for each type of classification based on projects of a very similar nature. The unit price includes the local supply of material, cutting, accessories and installation.

The unit price includes supply and installation of locally available carbon steel material including sleeves and anchors.

21.1.9.4 Structural Steel

Quantities were determined by the engineering groups from FS drawings. The unit rates were developed from in-house information while costs for supplying and installing pre-engineered buildings were solicited from specialized contractors. The weights include allowances for connections and base plates. The steel unit costs include:

- Material supply, fabrication and surface treatment where required.
- Erection at site based on estimated installation man-hours and unit labour costs and includes final touch-up of surface coating.
- Connection steel, weldments, and bolts.
- Steel supply and erection rates were developed based on in-house historical data with supply/erection rates checked against data provided by local contractors.

21.1.9.5 Mechanical Equipment

All large capital equipment shown on the flowsheets was itemized and budget-priced by the engineering consultants, with the final selection based on a FS decision that included consideration of commercial terms, technical acceptance, cost and delivery. Installation hours were estimated by a regional industrial contractor with similar North American project experience.

Vendor representatives will be engaged to monitor and advise on the installation of the mills, crusher, SAG Mill and Ball Mill motors and the IsaMills™.

Where budget quotations were not solicited for equipment, estimates were drawn from other similar projects.

21.1.9.6 Mechanical (Plate Work and Tanks)

Plate work weights were provided by Allnorth and calculated to include allowances for stiffeners, weirs, launders, etc. The prices include locally available plate purchase, detailing, fabrication and installation.

21.1.9.7 Piping

Piping material quantities, and some pricing, was provided by Allnorth for supply (not freight), shop and field fabrication. Overland pipeline quantities for the sediment control ponds related to the dumps and tailings dam were estimated by KP and priced by Merit. Installation hours were estimated and priced by a regional industrial contractor with similar North American project experience. Building Services piping was determined on a percentage basis based on building areas and allowances where no or little information was provided by the engineer.

21.1.9.8 Electrical & Instrumentation

Major electrical equipment and electrical material prices were based on budget quotations. Quantities for all electrical materials were described in detail and were not calculated using a percentage of the areas. Allnorth and Merit worked on installation productivities for in-plant electrical systems, including material supply (not freight) and installation. Lengths for overhead lines and high voltage cable were estimated from the overall site plan. Quantities for the incoming high voltage power line were estimated by a local electrical company. Electrical distribution includes for a power line to the open pit, a closed loop at the pit and 2km of trailing cables for the electric operated equipment. Instrumentation equipment and material prices were provided by Allnorth. Installation rates and productivities were sourced from Merit's in-house data and other similar projects.

21.1.9.9 Direct Field Labour

There are approximately 1.7M man hours of direct and indirect labour associated with construction, including approximately 1.5M man hours of direct construction activity.

Labour rates were studied in depth and rates were calculated based on blended union (CLRA) and alternative union information (such as CLAC, CISSIWU and non-union companies). Rates for each trade were calculated; an average rate was not used. Labour rates include:

- Base labour wage rate;
- Overtime premiums;
- Benefits and burdens;
- Workers compensation premiums;
- Travel allowances (included in Contractor's Travel Area 8500);
- Transportation to and from accommodations;
- Appropriate crew mixes;
- Small tools and consumables allowance;
- Field office overheads (included separately in contractors indirects);
- Home office overheads; and
- Contractors' profit.

A trades' incentive of C$3 per hour is included for tradesmen in specific trades. Note: The average labour rate calculated at the end of the estimate exercise is in the order of C$98 per hour; based on a 70 hour per week / 3 week in / 1 week out work schedule.

21.1.9.10 Direct Field Materials

Bulk materials components constitute locally available and imported quantities priced as FOB manufacturer. Freight costs to transport materials to site are included in the Indirect Costs. Pricing was based on quantities derived by the FS engineering consultants.

21.1.9.11 Off-Site Infrastructure

Pricing for the rail spur repairs and modifications at Vavenby, and the access road upgrade from Highway #5 near Vavenby to the site was provided by Allnorth and has been reviewed by Merit. Pricing for the power line design and construction was developed by Allnorth in conjunction with a local power line company.

21.1.9.12 Architectural Finishes

Estimates for the architectural finishes were based on the floor area of the finished space. Pricing was sourced from Merit's in-house data.

21.1.10 *PROJECT INDIRECT COSTS*

21.1.10.1 Temporary Construction Facilities and Services

All costs for construction not included in the direct costs as unit rates, productivities, material costs and labour are included in the indirect cost estimate for the project and include:

- Some construction equipment to be used by the Owner;
- Construction Management field offices, furnishings, equipment;
- Temporary power supply;
- Temporary water supply;
- Temporary heating and hoarding;
- Warehouse and lay down costs;
- Temporary toilets;
- Temporary communications;
- On-going and final clean-up;
- Yard maintenance;
- Janitorial services;
- Owner's site safety; personnel and training;
- Construction camp for all workers excluding operations, main access and incoming HV line construction workers.

It is important to note that contractor costs to construct the project are included in the Direct Costs. Only the costs associated to manage the contract are included in Indirect Costs. There are some items included in the Indirect costs that are supplemental to the contractors' costs including the 225t capacity crane that will be allocated by the Owner on a managed bases, and other construction equipment that is intended for use by the Owner in and around the plant for maintenance and eventual inclusion in the operations. Unit prices submitted by contractors are "all-in" rates, which include contractor's construction equipment, operators, insurance, overhead and profit.

21.1.10.2 Construction Camp and Catering

There is an existing 36-person camp located on the existing access road to the site currently serving the YMI exploration program. It will be expanded to a 100-person camp for the initial stage of construction. Additional bunkhouse and expansion of the dining and kitchen facilities will be required to allow for a quick start of the site construction program. During the initial stages of construction, an area will be prepared at the plant site for a leased temporary 600-person construction camp to be partially constructed in the first year of construction and increased in capacity during the second year to allow for the estimated peak work force requirements. Capacity will average 285-person over an 18-month time span. The camp growth will be mobilized when needed and will accommodate construction workers, construction management staff and associated visitors such as vendor representatives and other project team members visiting on a casual basis.

Power line and access road subcontractors will provide their own camps along the route of construction and associated costs included in their rates.

All mine operating personnel for pre-production and operations will be housed in the local community and bussed to the site.

An average catering cost of C$74 per camp man-day is based on prices provided by experienced national catering contractors who provided a scale of man day costs based on various levels of camp occupancy. The average takes into account the higher prices for low camp occupancy and lower costs for high levels, with the average cost calculated using the construction manpower schedule.

Upon completion of the construction program, likely a month before Mechanical Completion the temporary camp will be demobilized and the workers will use the 100-person camp.

21.1.10.3 First Fill and Spare Parts

The cost for spares was based on vendor recommendations when available. Where vendor information wasn't available, an allowance of 3% of the equipment purchase value was used. Industry standard allowances were included for first fills for item such as start-up grinding media, reagents and fuel. The cost of first fills for mining and process were provided by YMI. The costs for mining equipment spare parts before plant start-up was provided by YMI.

21.1.10.4 Start-up and Commissioning

The requirements for vendor representatives to supervise the installation of equipment or to conduct a checkout of the equipment prior to start-up of the equipment as deemed necessary for equipment performance warranties has been calculated and included in the estimate. An allowance of 1% of the equipment purchase value was used for commissioning and start-up spares.

An allowance was made for vendor representatives to be available at site during start-up, as well as a team of twenty-five people comprised of representation from the contractors' crews and the construction management staff for a period of three months.

21.1.10.5 Freight

The freight costs for materials and equipment for the project were examined in detail in consultation with several major international forwarders. Where actual freight costs were not obtained an allowance of 10% was included in the estimate. Freight for the large and/or particularly heavy pieces of equipment obtained quote prices for shipping directly from the manufacturers' fabrication plant. Some equipment pricing includes all freight charges to site and all pricing includes packing and crating.

21.1.10.6 Engineering, Procurement and Construction Management (EPCM)

Engineering and procurement costs were based on information from the engineering consultants involved in the FS who developed costs from their expected list of deliverables of drawings and specifications to produce a facility as established (i.e., no substantial changes).

Construction management costs were based on the development of an organization chart, the estimate of personnel required, the project schedule, and includes extended workweeks, transportation, supplies and communications. Charge-out rates are similar to other projects where the turnaround schedules are the same.

21.1.11 *Owner's Cost*

Owner's Costs developed by YMI include the following item:

- Owner's project management staff;
- Insurances;
- Corporate office staff assigned to the project;
- Environmental testing and monitoring;
- Owners allowances for field operations offices and supplies;
- Owner's travel costs during construction;
- Relocation allowance for staff; no working capital allowance was provided for housing.

21.1.12 *Contingency*

The contingency amount of approximately 10% of the total costs of Direct and Indirects covers unforeseeable costs within the scope of the design.

Contingency was estimated by category, taking into account items quoted, estimated or factored. Contingency is subjective and is based on the degree of confidence the FS team deems reasonable. It covers items included in the scope of work as described in this FS, but which cannot be adequately defined at this time due to lack of more accurate detail (Table 21-3).

Table 21-3: Contingency Rates by Category

Contingency by Category	Rate (%)
Direct Civil Earthworks	25%
Direct Power Line	10%
Pre-Production Mining	10%
Mine Equipment	5%
Capital Equipment	5%
Direct Labour less Civil Earthworks, Power Line & Pre-Production	10%
Other Directs	10%
Indirects	15%

Contingency is not intended to describe the accuracy of the estimate, nor should it be used to cover scope changes or project exclusions that would otherwise be added or subtracted from the budget. Nor is contingency intended to cover labour disputes, currency fluctuations, escalation, force majeure, or other uncontrollable risk factors. It should be assumed that the contingency will be spent over the engineering and construction period.

Earthworks are normally considered to represent the greatest risk of unknowns to project cost and schedule. Consequently, an allowance of 25% was assigned to earthworks. It is worth noting that the quality of geotechnical information available both from historical information as well as that developed during the FS, provides a good degree of confidence in what will be encountered during the execution of the work. In addition, the amount of design and confidence level in the quantities provided leads us to believe that the contingency provided here is more than sufficient. However, this is typically an area of a project of greatest concern with unknowns associated with sub surface conditions that are not totally apparent to the geotechnical investigation, such as extent of borrow sources, underground running water, unsuitable material, borrow source quality and so on.

The quantities provided by the engineers are in the order of what we would expect for a plant of this size, capacity, location, and account for items such as frost depth, in-ground water, structural loads and climatic conditions.

Intangible items are in the indirect costs, but are limited to items such as fuel, accommodation, and travel rather than spares, first fills, and the like.

21.1.13 *Escalation*

Escalation through the period of project development has not been included in the Capex.

21.1.14 *SUSTAINING CAPITAL*

The sustaining capital estimate was prepared based on input from:

- Nilsson: Additional and replacement mine equipment;
- Merit: General and Administrative Costs and Site Services;
- KP: Tailings and Water Management; and
- Allnorth: Site Power Requirements.

The sustaining capital estimate includes costs associated with:

- additional and replacement mine equipment;
- placement of core and transition material for tailings dam construction;
- pit power; and
- reclamation bonding.

An allowance was not made for the following:

- Escalation in costs beyond Q1 2014 for labour or materials;
- Exchange rate fluctuations; and
- Taxes.

The accuracy range of the sustaining capital cost estimate is +15%/-5% and all estimates are presented in Q1 2014 dollars. The currency exchange rate used for the estimate is C$1.00:US$0.90. A summary is provided in Table 21.4.

Table 21-4: Sustaining Capital Summary Estimate

Sustaining Capital Summary Estimate*	
Area	**Cost (C$M)**
Mining	203.7
Pit Power	13.6
Tailing Management Facility and Reclaim Water	45.2
Working Capital	35.4
Reclamation Bonding	37.8
Total	**C$335.7**

*Note: Totals may not add due to rounding

21.1.14.1 Tailings Management

For the Tailings Management Facility it is important to note that rock fill for the dam will be sourced from run of mine non-PAG material and the cost of this material is included in the mining cost estimate (not in sustaining capital). An over-haul cost of approximately C$104M has been included in the mining costs for 121.8Mt of non-PAG material over the life of the mine. In addition, where possible older trucks will be sourced for parts as the fleet requirements decline after Year 17 of operation (Table 21-4).

21.1.14.2 Working Capital

Operating working capital is allowed at two months of Year 1 operating costs to provide cash to meet operating expenses prior to receipt of sales revenue. All the working capital is recaptured at the end of the mine life and the final value of the account is C$0.

21.1.14.3 Reclamation Bonding

A maximum of C$112.4M is allocated to reclamation bonding from the project cash flow and is incurred to year 24 of operations to cover possible reclamation commitments in the event of premature mine closure. From year 25 to year 28 approximately C$66.7M will be recaptured as the PAG LG stockpile is processed. Assuming an annualized growth rate of 1.5% after tax for the first 5 years and 2% thereafter, the total reclamation bond remaining in place upon mine closure is C$89.9M. This figure plus salvage value associated with equipment is expected to cover the estimated reclamation cost of C$70M.

21.2 Operating Cost Estimate

The operating cost estimate for the Project was prepared by and based on inputs from the following principal consultants:

- Nilsson: Mining operating costs;
- Laurion: Process operating costs;
- Merit: General and Administrative and Site Services;
- KP: Tailings and Water Management; and
- Allnorth: Site power requirements.

The operating cost estimate assumes mine staff and operators, in the main, will be sourced locally and includes all the costs associated with mining, processing and infrastructure activities for the Project. Unit rates calculated were rounded and reported to two decimal places. The scope includes:

- Mining:
    - general mine expense, grade control, drilling, blasting, loading, hauling, roads and dumps.
- Process:
    - plant labour, and consumables for operations and maintenance for crushing, grinding, flotation, and concentrate dewatering;
    - supply of electrical power, including demand charges.
- Site Services:
    - site water management, tailings disposal and reclaim water;
- General and administration for the operations:
    - site and head office costs

No allowance was made for the following:

- Escalation in costs beyond Q1, 2014 for labour or materials;
- Exchange rate fluctuations;
- Taxes;
- Exploration costs;
- Offsite transportation of concentrates (classified as part of the net smelter return and deducted from gross revenue).

The accuracy range of the operating cost estimate is +15%/-5% and is presented in Q1 2014 dollars. The currency exchange rate used for the estimate is C$1.00:US$0.90.

21.2.1 *METHODOLOGY*

In general, the operating cost estimate was prepared by synthesis of operating and maintenance labour productivities, supplies consumption and energy consumption based on industry experience, and by benchmarking against other similar operations, where appropriate.

Operating and maintenance supply costs were based on in house data and vendor quotations, and are exclusive of taxes. Consumables quantities (fuel, explosives, tires, blasting accessories, etc.) were determined from expected unit consumption rates (per hour, per tonne).

21.2.1.1 Key Commodity Prices

The following power and fuel costs were used in preparing the operating cost estimate as provided in Table 21-5.

Table 21-5: Key Commodity Prices

Key Commodity Prices		
Commodity	**Source**	**Unit Rate**
Electrical Energy		
Energy Charge	BC Hydro (2014 rates)	C$0.03724/kWhr
Demand		C$6.353/kWhr
Fuel		
Diesel	Petro Canada spot price (historical 12 mo. avg. rack rate)	C$1.06/litre

21.2.1.2 Labour Rates and Work Schedule

Labour salaries, wages and benefits were determined from industry experience and from standard salary scales for similar mines in British Columbia. The work schedules assume that mine will operate 24h/d, 7d/wk, 365d/a. All hourly personnel assigned to positions that form part of continuous operations will work a four-days-on/four-days-off rotation. The burden was calculated separately for salaried and hourly employees. The assumptions used for the calculation of the benefits are presented in Table 21-6.

Table 21-6: Payroll Burden

Payroll Burden		
	Wages (C$)	**Salaried (C$)**
Stat Holidays	4.7%	
RSP Program		5.0%
Over-time	12.8%	
EI	1.4% to a maximum of C$1,279.15	1.4% to a maximum of C$1,279.15
CPP	4.95% to a maximum of C$2,425.50	4.95% to a maximum of C$2,425.50
WCB	1.4% to a maximum of C$1,090.6	1.4% to a maximum of C$1,090.6%
Payroll Services Charge	145/yr	C$145/yr
Health Insurance	C$5,445.36/yr + 0.2% LTD & CI	C$5,445.36/yr + 0.2% LTD & CI
Average Burden	32%	15%

21.2.2 *OPERATING COST SUMMARY*

Total LOM operating cost for the project is estimated at C$8.22/t milled, including stockpile rehandle but excluding pre-stripping costs. The estimate includes mining, process, G&A, site services, and royalties. On average, a total of 426 personnel are projected for the operation, including 272 for mining operation, 102 personnel for process and 52 personnel for site services and general and administration. The unit costs are based on an annual ore production rate of 25,550,000t/a (or 70,000t/d), and operation of 365d/a. A summary of the operating cost estimate is presented in Table 21-7.

Table 21-7: Operating Cost Estimate

Category		**Year 1-5 C$/t milled**	**Life of Mine C$/t milled**
G & A	G&A Labour	0.07	0.07
	Direct	0.37	0.37
	Subtotal	**0.44**	**0.44**
Site Services	Site Services Labour	0.13	0.13
	Direct	0.22	0.22
	Subtotal	**0.35**	**0.35**
Mining	Labour	1.11	0.95
	Consumables[1]	1.30	1.10
	Fuel/Power	2.07	1.69
	Subtotal	**4.48**	**3.75**
Processing	Process Labour	0.41	0.41
	Consumables	1.86	1.86
	Power	1.04	1.04
	Parts and Supplies	0.34	0.34
	Subtotal	**3.65**	**3.65**
Royalties		0.02	0.04
Total Mine Site		**C$8.94**	**C$8.22**

NB: Totals may not add due to rounding

1. *Higher mining consumable costs in Y1 to Y5 are a result of clearing and grubbing allowance*

21.2.3 *MINING OPERATING COSTS*

Open pit mining costs were estimated by applying equipment productivities, equipment hourly operating costs, labour requirements, and wages and salaries to the proposed production forecast. The estimate includes costs for equipment operators, staff, contractors, and operating and maintenance supplies for each year of mine operation and subsequent stockpile re-handling. For the purpose of cost estimating the operating costs for the mine were divided into the following areas:

- General Mine Expense;
- Grade Control
- Drilling;
- Blasting;
- Loading;
- Hauling;
- Roads and Dumps;

A summary of annual mining costs are shown in Table 21-8 and Table 21-9. The average mine operating cost is estimated at C$1.97/t of material moved or C$3.75/t ore milled over the life of the mine, excluding pre-stripping but including low grade rehandle. These figures increase to C$1.99/t of material moved or C$3.86/t ore milled over the life of the mine with the inclusion of pre-stripping costs. In-pit mining ceases towards the end of Year 24 and operations shift to re-handling the low-grade stockpile to feed the mill. The average cost to rehandle the low-grade stockpile is estimated at C$1.21/t. The total estimated low grade tonnage is 116.9Mt.

Manning levels average 280 personnel throughout open pit mining with a maximum manning level of 315 in year 16. Levels decrease in Year 24 as operations shift to re-handling the low-grade stockpile to feed the mill. A summary of annual manning requirements is detailed in Table 21-10 with an average annual cost estimate provided in Table 21-11.

An estimated average fuel and power consumption throughout the life of mine is detailed in Table 21-12.

Included in mining costs is the transportation of run of mine rock (ROM) to the tailings dam for construction of the tailings dam during the pre-strip stage and throughout the mine life as the dam is progressively raised. Rockfill for the TMF raising throughout the mine life will be delivered by the mine fleet to the TMF. The total material delivered throughout the mine life to the TMF is estimated at 121.8Mt. Annual movements are provided in Table 21-13. The total cost of the overhaul is estimated at C$104M or C$0.85/t of material hauled to the TMF.

Table 21-8: Mine Operating Cost Summary Estimate

Project Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mine Operating Cost (C$/t mined)[1]															
General Mine Expense	0.48	0.13	0.11	0.13	0.11	0.10	0.11	0.14	0.11	0.11	0.13	0.11	0.11	0.11	0.11
Grade Control	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02
Drilling	0.05	0.05	0.07	0.07	0.09	0.08	0.09	0.09	0.08	0.09	0.09	0.09	0.09	0.08	0.06
Blasting	0.19	0.20	0.20	0.19	0.20	0.19	0.21	0.21	0.20	0.21	0.21	0.20	0.21	0.21	0.21
Loading	0.16	0.11	0.13	0.18	0.24	0.21	0.19	0.24	0.24	0.23	0.21	0.18	0.20	0.28	0.21
Hauling	1.12	0.84	0.90	0.85	0.76	0.83	1.06	1.02	0.88	1.02	0.93	0.96	0.96	1.18	0.88
Roads and Dumps	0.61	0.37	0.39	0.36	0.44	0.37	0.42	0.40	0.40	0.42	0.41	0.42	0.42	0.40	0.40
Total C$/t mined	**2.64**	**1.72**	**1.83**	**1.81**	**1.85**	**1.80**	**2.10**	**2.11**	**1.94**	**2.11**	**2.00**	**1.99**	**2.02**	**2.28**	**1.88**
Mine Operating Cost (C$/t milled)		4.45	4.25	4.66	4.36	4.64	4.94	4.96	4.66	4.95	4.70	4.68	4.74	5.35	4.48
Project Year	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**Avg**
Mine Operating Cost (C$/t mined)															
General Mine Expense	0.11	0.12	0.15	0.15	0.15	0.17	0.17	0.17	0.19	0.15	0.13	0.13	0.13	0.14	0.13
Grade Control	0.02	0.02	0.02	0.02	0.02	0.03	0.03	0.03	0.03	0.02	0.00	0.00	0.00	0.00	0.02
Drilling	0.07	0.08	0.08	0.08	0.08	0.09	0.09	0.10	0.12	0.06	0.00	0.00	0.00	0.00	0.07
Blasting	0.21	0.21	0.22	0.22	0.22	0.23	0.23	0.23	0.22	0.15	0.00	0.00	0.00	0.00	0.19
Loading	0.16	0.20	0.22	0.22	0.18	0.17	0.18	0.20	0.18	0.20	0.21	0.25	0.24	0.24	0.20
Hauling	0.89	1.02	1.10	1.03	1.04	1.08	1.04	0.99	1.04	0.96	0.52	0.48	0.49	0.48	0.93
Roads and Dumps	0.41	0.43	0.57	0.58	0.56	0.60	0.57	0.55	0.64	0.39	0.35	0.35	0.34	0.34	0.44
Total C$/t mined	**1.87**	**2.08**	**2.37**	**2.30**	**2.26**	**2.35**	**2.31**	**2.27**	**2.42**	**1.93**	**1.22**	**1.22**	**1.20**	**1.20**	**1.99**
Mine Operating Cost (C$/t milled)	4.39	4.48	3.77	3.70	3.59	3.34	3.17	3.11	2.84	2.13	1.22	1.22	1.20	1.20	3.86

NB: Totals may not add due to rounding

Table 21-9: Mine Operating Cost Summary Estimate

Project Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mine Operating Cost (C$/t mined)[1]															
Salaries and Wages	0.72	0.45	0.46	0.43	0.45	0.44	0.50	0.48	0.46	0.51	0.49	0.49	0.49	0.52	0.47
Energy	0.85	0.56	0.56	0.50	0.47	0.53	0.62	0.58	0.55	0.63	0.56	0.57	0.57	0.67	0.54
Consumables (R&M + Clearing & Grubbing)	1.07	0.71	0.81	0.89	0.93	0.83	0.99	1.05	0.93	0.97	0.95	0.93	0.96	1.10	0.87
Total C$/t mined	**2.64**	**1.72**	**1.83**	**1.81**	**1.85**	**1.80**	**2.10**	**2.11**	**1.94**	**2.11**	**2.00**	**1.99**	**2.02**	**2.28**	**1.88**
Mine Operating Cost (C$/t milled)		4.45	4.25	4.66	4.36	4.64	4.94	4.96	4.66	4.95	4.70	4.68	4.74	5.35	4.48
Project Year	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**Avg**
Mine Operating Cost (C$/t mined)															
Salaries and Wages	0.50	0.56	0.65	0.62	0.62	0.66	0.66	0.63	0.69	0.48	0.35	0.36	0.36	0.37	0.51
Energy	0.61	0.71	0.74	0.68	0.68	0.66	0.68	0.66	0.69	0.54	0.35	0.36	0.35	0.34	0.58
Consumables (R&M + Clearing & Grubbing)	0.76	0.81	0.98	1.00	0.97	1.03	0.98	0.97	1.04	0.91	0.52	0.51	0.49	0.49	0.90
Total C$/t mined	**1.87**	**2.08**	**2.37**	**2.30**	**2.26**	**2.35**	**2.31**	**2.27**	**2.42**	**1.93**	**1.22**	**1.22**	**1.20**	**1.20**	**1.99**
Mine Operating Cost (C$/t milled)	4.39	4.48	3.77	3.70	3.59	3.34	3.17	3.11	2.84	2.13	1.22	1.22	1.20	1.20	3.86

NB: Totals may not add due to rounding

Table 21-10: Annual Mine Manpower Estimate

Project Year	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Salaried Employees															
Mine Supervision	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Mine Maintenance	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Engineering & Geology	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Subtotal	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29
Hourly Employees															
Operations	135	171	173	177	163	184	194	184	179	194	182	183	182	201	177
Maintenance	61	76	80	81	81	83	84	83	83	89	89	89	89	89	89
Subtotal	**196**	**247**	**253**	**258**	**244**	**267**	**278**	**267**	**262**	**283**	**271**	**272**	**271**	**290**	**266**
Total	**225**	**276**	**282**	**287**	**273**	**296**	**307**	**296**	**291**	**312**	**300**	**301**	**300**	**319**	**295**
Project Year	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**Ave**
Salaried Employees															
Mine Supervision	8	8	8	8	8	8	8	8	8	7	5	5	5	5	8
Mine Maintenance	9	9	9	9	9	9	9	9	9	7	5	5	5	5	8
Engineering & Geology	12	12	12	11	11	11	11	11	11	4	2	2	2	2	10
Subtotal	**29**	**29**	**29**	**28**	**28**	**28**	**28**	**28**	**28**	**18**	**12**	**12**	**12**	**12**	**26**
Hourly Employees															
Operations	191	198	163	157	154	143	139	132	121	79	49	49	49	46	150
Maintenance	88	88	76	76	74	69	66	63	58	40	26	29	29	28	71
Subtotal	**279**	**286**	**239**	**233**	**228**	**212**	**205**	**195**	**179**	**119**	**75**	**78**	**78**	**74**	**221**
Total	**308**	**315**	**268**	**261**	**256**	**240**	**233**	**223**	**207**	**137**	**87**	**90**	**90**	**86**	**247**

NB: Totals may not add due to rounding

Table 21-11: Average Annual Cost Mining Manpower Yr1-23 Estimate

Area	Number Req'd	Unit Cost C$/t mined	Annual payroll (C$)
Operations Staff	8	0.02	1,208,700
Maintenance Staff	9	0.02	1,239,900
Engineering & Geology	12	0.02	1,292,200
Subtotal	**29**	**0.07**	**3,740,800**
Operations Hourly	171	0.30	15,742,000
Maintenance Hourly	80	0.14	7,462,700
Subtotal	**251**	**0.44**	**23,204,700**
Total	**280**	**C$0.51**	**C$26,945,500**

NB: Totals may not add due to rounding

Table 21-12 Annual Power Consumption Estimate

Project Year	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Power Consumption															
Power (MWhr/yr)	19,045	45,479	46,572	50,424	47,194	50,001	46,826	46,517	46,465	46,264	46,173	47,178	46,781	47,068	47,187
Diesel (L x 1,000/yr)	23.7	28.9	28.8	28.3	24.4	30.5	32.5	30.6	29.2	33.2	29.6	30.1	29.7	35.3	28.4
Project Year	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**Avg**
Power Consumption															
Power (MWhr/yr)	46,105	42,528	31,237	31,634	31,168	27,827	27,361	28,121	25,114	20,401	14,179	14,179	14,179	13,119	35,684
Diesel (L x 1,000/yr)	32.3	34.9	26.9	24.8	24.2	21.3	20.9	20.5	18.4	13.9	7.9	8.1	7.9	7.0	24.4

NB: Totals may not add due to rounding

Table 21-13 TMF Rockfill Requirements Estimate

Quantity	**Year Elevation**	**-1 1,720**	**1 1,731**	**2 1,740**	**3 1,748**	**4 1,755**	**5 1,763**	**6 1,770**	**7 1,776**	**8 1,781**	**9 1,786**	**10 1,791**	**11**
Overburden	(m^3)		139,000	120,000	110,000	99,000	116,000	105,000	58,000	49,000	50,000	51,000	41,000
Non-PAG	(m^3)	2,638,205	4,266,000	3,406,000	517,000	3,753,000	2,175,488	3,760,512	636,000	433,000	6,872,000	626,000	431,000
PAG	(m^3)	2,908,795	211,000	181,000	158,000	134,000	167,000	142,000					
Overburden	tonnes		278,000	240,000	220,000	198,000	232,000	210,000	116,000	98,000	100,000	102,000	82,000
Non-PAG	tonnes	5,804,051	9,385,200	7,493,200	1,137,400	8,256,600	4,786,074	8,273,126	1,399,200	952,600	15,118,400	1,377,200	948,200
PAG	tonnes	6,399,349	464,200	398,200	347,600	294,800	367,400	312,400					
Quantity	**Year Elevation**	**12 1,798**	**13 1,802**	**14 1,806**	**15 1,810**	**16 1,814**	**17 1,818**	**18 1,822**	**19 1,824**	**20 1,827**	**21 1,830**	**22 1,833**	**Total**
Overburden	(m^3)	52,000	42,000	43,000	44,000	44,000							1,163,000
Non-PAG	(m^3)	460,000	6,989,000	503,000	912,939	6,763,995	3,072,066	622,000	419,000	382,000	343,000	402,000	50,383,205
PAG	(m^3)												3,901,795
Overburden	tonnes	104,000	84,000	86,000	88,000	88,000							2,326,000
Non-PAG	tonnes	1,012,000	15,375,800	1,106,600	2,008,466	14,880,789	6,758,545	1,368,400	921,800	840,400	754,600	884,400	110,843,051
PAG	tonnes												8,583,949

NB: Totals may not add due to rounding

21.2.4 *Process Operating Costs*

The process cost estimate include costs associated with primary crushing, grinding, flotation, thickening and filtration to produce a copper concentrate. A planned throughput of 70,000t/d with a mill availability of 92% operating 365d/a was assumed. The annual process operating cost estimate was derived from the following:

- operating and maintenance manpower requirements ;
- power consumption based on the estimated power draw of major equipment;
- reagent consumption with costing based on recent prices received from reagent suppliers;
- maintenance costs based on 5% of the equipment capital cost estimate; and
- power demand charges based on a 30 day pay period

The annual processing operating cost summary is provided in Table 21-14. The average process operating cost is estimated at C$3.65/t of material milled over the LOM. A total of 102 employees will be required to operate the Processing Plant for a total estimated annual payroll of C$10.5M.

Table 21-14: Process Operating Cost Summary Estimate

Area	Description	Manpower	Annual Cost (C$)	Unit Cost (C$/t Milled)
Manpower	Operations Staff	20	2,405,730	0.09
	Operations Labour	40	3,550,253	0.14
	Maintenance labour	42	4,543,882	0.18
Subtotal		**102**	**10,499,864**	**0.41**
Supplies	Operating		47,582,555	1.86
	Maintenance		8,648,400	0.34
Subtotal			**56,230,955**	**2.20**
Power	Energy Charge		19,885,974	0.778
	Demand		6,579,167	0.258
Subtotal			**26,465,141**	**1.04**
	Total		**C$93,195,959**	**C$3.65**

NB: Totals may not add due to rounding

Tables 21-15, 21-16 and 21-17 details power requirements and supplies for processing on an annual basis.

Table 21-15: Power Requirements Estimate

Description	Connected Load (kW)	Maximum Demand (kW)	Average Demand (kW)	Unit Cost (C$/t Milled)
Crushing and Grinding				
Gyratory Crusher	1,334	1,067	853	0.01
Conveyor and Stockpile Reclaim	5,412	4,315	3,452	0.04
Grinding	63,802	61,303	48,552	0.58
Subtotal	**70,548**	**66,685**	**52,857**	**0.63**
Balance				
Conc. Building	1,473	1,271	1,001	0.01
Air System	5,079	4,063	3,250	0.04
Flotation	4,063	3,251	2,600	0.03
Regrind	7,586	6,057	4,845	0.06
Conc. Thickening & Loadout	565	452	362	0.00
Reagents	266	212	170	0.00
Tailings Pumping	714	572	457	0.01
Subtotal	**19,746**	**15,878**	**12,685**	**0.15**
Total	**90,294**	**82,563**	**65,542**	**C$0.78**

NB: Totals may not add due to rounding

Table 21-16: Process Plant Operating Supplies Estimate

Supplies	Consumption (kg/t Milled)	Unit Cost (C$/t)	Annual Cost (C$)	Unit Cost (C$/t Milled)
Gyratory Crusher				
Liners	0.015	4,111	1,575,583	0.06
SAG Mill				
Balls	0.306	1,403	10,972,550	0.43
Liners	0.0272	2,556	1,776,009	0.07
Ball Mills				
Balls	0.549	1358	19,047,040	0.75
Liners	0.0488	2556	3,186,369	0.13
IsaMills™				
Grinding Media	0.01	7,222	1,845,278	0.07
Discs			750,000	0.03
Subtotal			**39,152,828**	**1.53**
Reagents				
PAX	0.020	2,610	1,372,603	0.05
MIBC	0.010	2,660	697,657	0.03
Lime	1.200	185	5,672,100	0.22
HCl	0.001	722	21,008	0.00
Flocculent	0.001	6,110	158,694	0.01
Subtotal			**7,922,061**	**0.31**
Supplies				
Assay			252,555	0.01
Miscellaneous			255,110	0.01
Subtotal			**507,665**	**0.02**
Total			**C$47,582,555**	**C$1.86**

NB: Totals may not add due to rounding

Table 21-17: Maintenance Supplies Estimate

	Annual Cost (C$)	Unit Cost (C$/t Milled)
Crushing & Stockpile	1,294	0.05
Grinding	3,990	0.16
Flotation & Regrind	2,716	0.10
Concentrate & Filtration	648	0.03
Total	**C$8,648**	**C$0.34**

NB: Totals may not add due to rounding

21.2.5 *G&A and Site Services Operating Cost*

G&A and site services costs are estimated to be C$0.44/t milled and C$0.35/t milled respectively, over the life of mine including 52 employees. G&A costs include:

- labour cost for administrative and laboratory personnel;
- expenses and services related to general administration, travelling, human resources, safety and security;
- allowances for insurance, regional taxes and licenses allowance; and
- sustainability, including environment, community liaison and engineering consulting.

A summary of costs and manning levels are provided in Table 21-18.

Table 21-18: G&A Operating Cost Estimate

Expense	Annual Expense (C$)	Unit Cost (C$/t Milled)
Head Office Expenses	100,000	0.00
Office Supplies	100,000	0.00
Insurances	2,000,000	0.08
Project Taxes	75,000	0.00
Leases/Permits	600,000	0.02
Environmental Monitoring	600,000	0.02
Warehouse	1,000,000	0.04
Training	250,000	0.01
Security Supplies/First Aid	250,000	0.01
Legal Services - Allowance	500,000	0.02
Regulatory Compliance - Allowance	500,000	0.02
Consulting - Allowance	500,000	0.02
Small Vehicles	300,000	0.01
HR Recruitment	250,000	0.01
Communications	200,000	0.01
Computer Services	192,000	0.01
Travel & Expenses	320,000	0.01
Professional Associations	55,200	0.00
Community Engagement/Public Relations	250,000	0.01
Safety & Training Supplies	508,800	0.02
Safety Incentives	254,400	0.01
Miscellaneous	534,284	0.02
Subtotal	**8,339,684**	**0.33**
Manning	1,790,853	0.07
Power	150,000	0.01
Total	**C$10,280,537**	**C$0.44**

NB: Totals may not add due to rounding

The site service cost estimate as provided in Table 21-19 and include:

- labour costs for site service personnel;
- water management including pit dewatering and reclaim water;
- potable water and waste management;
- general maintenance including yards, roads, fences, and building maintenance;
- off-site operation expenses;
- building heating;
- on-site and off-site transportation services for personnel; and
- surface mobile equipment and light vehicle operations.

Table 21-19: Site Services Operating Cost Estimate

Expense	Total Cost (C$)	Unit Cost (C$/t Milled)
Potable Water & Waste Management	250,000	0.01
Building Maintenance	200,000	0.01
Road/Power Line Maintenance	1,250,000	0.05
Supplies	250,000	0.01
Bus Transportation	1,000,000	0.04
Small Vehicles/Equipment	250,000	0.01
Subtotal	**3,200,000**	**0.13**
Manning	3,262,201	0.13
Power		
Pit Dewatering and Seepage Collection	2,129,122	0.04
Reclaim Water	1,094,013	0.04
Tailings Pumping	138,648	0.01
Truck Shop	95,263	0.00
Subtotal	**3,457,046**	**0.09**
Total	**C$9,919,248**	**C$0.35**

NB: Totals may not add due to rounding

22 ECONOMIC ANALYSIS

Merit considers the following assumptions to be reasonable and consistent with current projections used by industry:

- Project Equity 100%
- Working Capital 2 months
- Discount rate: 8%
- Federal Corporate Income Tax Rate 15%
- Metal prices Copper US$3.00 /lb
- Exchange Rate US$: C$0.90

The financial evaluation presents an estimate of the Net Present Value (NPV), the Internal Rate of Return (IRR) and payback period (time in years to recapture the initial capital investment), for the project. A discounted cash flow approach with annual cash flow projections estimated over the life of the mine (LOM) has been used. Projections used were based on estimates of capital expenditures, production cost, and sales revenue from the production of copper concentrate with gold and silver credits. The estimates of capital expenditures and site production costs were discussed in earlier sections of this report and are summarized here for completeness.

For the purpose of the financial analysis it is expected that the copper concentrate will be sold to markets throughout the Pacific Rim countries.

22.1 MINE PRODUCTION

Annual production figures used in the cash flow are based on the mine plan as reported in Section 16 of this report. In general, the production schedule reflects the stockpiling of lower grade material early in LOM with subsequent processing of this material at the end of the mine life. LOM ore, ore grades and waste quantities scheduled are presented in Table 22-1.

Table 22-1 Mineral Reserve (0.14% Cu cut-off)

	Tonnage (Mt)	Copper (%)	Gold (g/t)	Silver (g/t)
Proven	457.2	0.27	0.03	1.19
Probable	258.9	0.24	0.03	1.16
Proven _ Probable	**716.2**	**0.26**	**0.03**	**1.18**
Waste	541.7			
Total	1,259.9			
Strip Ratio	0.76:1	After reprocessing stockpiled material		

22.2 Plant Production

Ore will be processed at a rate of 70,000t/d (25.55Mt/a) over a planned period of approximately 24 years at which point throughput rate is increased marginally to 72,920t/d (26,62Mt/a) for the processing of low grade material. Production from the flotation plant will produce a copper-gold and silver concentrate. The estimated concentrate production for the life of the operation is 6.5Mt containing 3.64bn lb of copper, 368,000 oz of gold and 15.6M oz of silver. Concentrate production, estimated recoveries and metal production is summarized in Table 22-2.

Table 22-2 Metallurgical Recoveries and Copper Concentrate Production

	Units	Year 1 to 5 Avg	LOM
Copper Recovery	%	90.6	89.2
Gold Recovery	%	59.9	55.8
Silver Recovery	%	58.4	57.7
Tonnes of Concentrate	dmt	276.3	230.9
Copper Concentrate Grade	%	25.5	25.5
Gold Concentrate Grade	g/t	1.98	1.77
Silver Concentrate Grade	g/t	64.7	75.2
Copper Recovered	lbs x 1,000	776,768	3,635,165
Gold Recovered	oz x 1,000	87.8	368.0
Silver Recovered	oz x 1,000	2,874.9	15,632.1

22.3 Capital Costs

22.3.1 *Initial Capital*

The total capital carried in the financial model for new construction is C$1,025.8M to be expended over a two-year period prior to commercial operation. The initial capital includes Owner's costs, contingency, reclamation bonding and an estimate of Provincial Sales Tax (PST) payable (Table 22-3).

Table 22-3 Initial Capital Cost Estimate

Pre Production Capital (Year -2 and -1)	(C$M)
Direct Costs	
Mining & Pre-production Development	298.0
Plant Site Infrastructure	9.5
Site Services & Utilities	13.4
Process	279.6
Ancillaries	27.4
Power Supply & Distribution	47.7
Tailings & Water Reclaim	54.0
Total Direct	**729.5**
Indirect Costs	
Owner's Costs	25.6
Indirects	162.1
Total Indirect	187.7
Subtotal	917.2
Contingency	90.7
Subtotal Direct + Indirect	**1,007.9**
Bonding	7.9
PST	10.0
Total Project	**C$1,025.8**

NB: Totals may not add due to rounding

22.3.2 *Working Capital*

Operating working capital is assumed at two months of the first year's operating costs to meet operating expenses prior to receipt of sales revenue. The working capital will be recaptured at the end of mine life and the final value of the account is C$0.00.

22.3.3 *Owners Costs*

Owner's Costs were developed by YMI and include the following:

- Owner's project management staff;
- Training program for operations staff;
- Insurances;
- Property tax;
- Corporate office staff assigned to the project;

- Environmental testing and monitoring;
- Owners allowances for field operations offices and supplies;
- Owner's travel costs during construction; and
- Relocation allowance for staff, no working capital allowance was included for housing.

22.3.4 *Reclamation Bonding*

A maximum of C$112.4M is allocated to reclamation bonding from the project cash flow and is incurred to year 24 of operations to cover possible reclamation commitments in the event of premature mine closure. From years 25 to year 28 approximately C$66.7M will be recaptured as the PAG LG stockpile is processed. Assuming an annualized growth rate of 1.5% after tax for the first 5 years and 2% thereafter, the total reclamation bond remaining in place upon mine closure is C$89.9M. This figure plus salvage value associated with equipment is expected to cover the estimated reclamation cost of C$70M.

22.3.5 *Provincial Services Tax*

An estimate of applicable PST payable on initial capital has been made and included in the financial analysis.

22.3.6 *Sustaining Capital*

A schedule of capital cost expenditures during the production period was included in the financial analysis as part of the sustaining capital estimate. These expenditures cover additional and replacement mine equipment, tailings dam raises (core material only; rock fill will be placed by the mine fleet and is included in operating costs) and expansion of site power lines throughout the LOM. The total LOM sustaining capital estimate is C$262.5M to be expended during a 23 year period (Table 22-4).

Table 22-4 Sustaining Capital Estimate

Area	(C$M)
Additional & Replacement Equip	203.7
Tailings and Water Reclaim	45.2
Site Power	13.6
Total	**C$262.5**

NB: Totals may not add due to rounding

22.3.7 *Salvage Value*

No allowance for salvage value was made in the cash flow analysis.

22.4 Total Cash Production Costs

22.4.1 *Cash Operating Costs*

The average site cash operating cost is C$8.22/t milled. Included in this cost are mining operations, concentrator, general administration and site services (Table 22-5).

Table 22-5 Site Operating Cost Estimate

	Year 1 to 5	Life of Mine (C$)
Mining (C$/t mined))	1.81	1.97
Mining (C$/t milled))	4.48	3.75
Process (C$/t milled))	3.65	3.65
G & A (C$/t milled)	0.40	0.44
Site Services (C$/t milled)	0.35	0.35
Royalties	0.00	0.04
Total (per t milled)	**C$8.88**	**C$8.22**

NB: Totals may not add due to rounding

22.4.2 *Royalties*

There are two royalties payable on production from the Project. A 3% NSR royalty capped at C$2.5M, adjusted for inflation is payable to US Steel. A 2.5% NSR royalty on an estimated 1.49Mt of ore expected to be mined beginning in Year 16 with an NPV8 of C$5.4M, is payable to XStrata. The average cash operating cost for the royalty payments is C$0.04/t milled.

22.5 Revenue

22.5.1 *Gross Revenue*

The base case financial evaluation uses long-term prices of US$3.00/lb of copper, US$1,250.00/oz of gold, US$20.00/oz of silver. The gross revenue was determined by applying selected metal prices to the annual metal contained in concentrate for each operating year. Metal prices have been applied without escalation or hedging to estimate total gross revenue.

22.5.2 *Deductions*

Applicable metal deductions applied in the cash flow are summarized in Table 22-6. In addition, a 0.5% loss has been applied to final concentrate delivered.

Table 22-6 Metal Deductions

Payable Metal	
Copper Concentrate Pay Factor	96.5% or minimum 1.0 unit deduction
Gold Pay Factor (if Au> 1 g/t)	90%
Silver Pay Factor (if Ag > 30 g/t)	90%
Losses	0.5%

22.5.3 *Concentrate Transportation Costs*

The base case includes concentrate transportation charges such as:

- Trucking from mine to the railhead in Vavenby;
- Temporary storage and load out at Vavenby to CN Rail;
- Rail from Vavenby to the Port of North Vancouver;
- Port loading and environmental charges;
- Umpire sampling;
- Ocean freight;
- Losses; and
- Insurance.

The aggregate rate used to represent the above charges is C$114.5/wmt. Wet concentrate tonnage estimates are based on a moisture content of 8.0%.

22.5.4 *Treatment and Refining Charges*

The following treatment and refining charges have been used in the financial analysis (Table 22-7). These charges were applied to the net revenue along with transportation costs to determine a net smelter return on an annual basis. The concentrate is clean with no elements above penalty limits.

Table 22-7 Treatment and Refining Charges

Treatment and Refining Charges	
Smelter Terms	
Base Treatment Charge	US$80/dmt
Copper Refining Charge	US$0.08/payable lb
Gold Refining Charge	US$6.00/payable troy oz
Silver Refining Charge	US$0.45/oz

22.5.5 *Net Smelter Return*

A Net Smelter return (NSR) was determined by subtracting metal deductions, transportation costs and treatment and refining charges from the gross revenue. The average NSR over the LOM was estimated at US$13.37/t milled or C$14.86/t milled.

22.6 Taxes

The Project will be subject to income and/or revenue taxes:

- 15% Canadian federal corporate income tax;
- 10% British Columbia corporate income tax;
- 2% Net Current Proceeds tax;
- 13% Net Revenue tax; and
- British Columbia Mineral tax*.

**British Columbia Mineral tax is applied to an amount different than taxable income, as defined for federal and provincial income tax purposes, and is assumed deductible in arriving at taxable income.*

The initial provincial current proceeds tax at 2% is a form of minimum tax and is deductible in full against the 13% provincial net revenue tax.

The majority of capital items are subject to a 25% depreciation rate for both federal and provincial corporate tax purposes. However, under the following circumstances the Company can use accelerated rates of up to a maximum of 100% reducing taxable income to a minimum of $nil:

- Capital acquisitions made before the commencement of commercial production;
- Capital acquisitions made during the production phase in excess of 5% of gross revenues from the mine for that particular year.

A goods and services tax (GST) is applied on all goods and services purchased. This tax is recoverable.

22.7 Project Financing

The base case economic analysis was run on a basis of 100% equity.

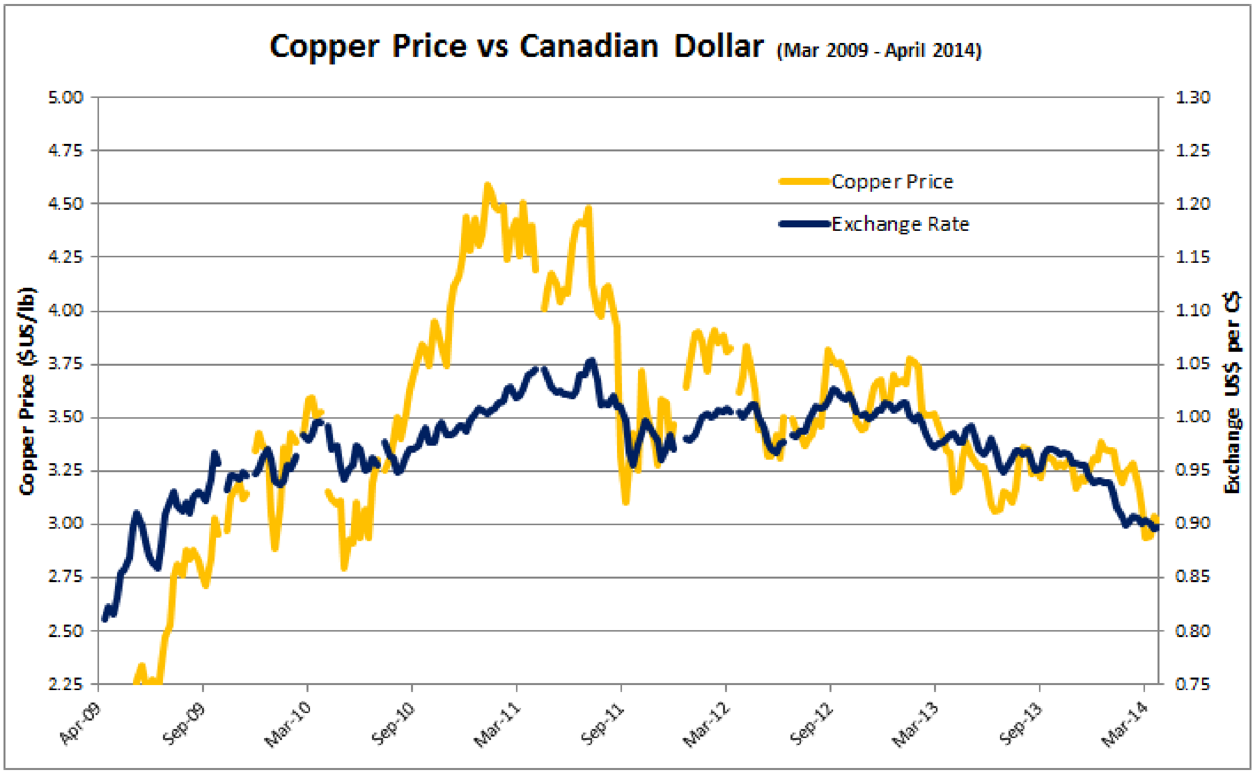
22.8 Inflation/Escalation

The base case economic analysis was run with no inflation (constant dollar basis). Capital and operating costs are expressed in 1Q 2014 Canadian dollars.

22.9 EXCHANGE RATE

Historically there is a very strong correlation between the US:C exchange rate ("FX") and copper price as shown in Figure 22.1. At a long term copper price of US$3.00/lb the FX rate is 0.90. This FX rate was used in the base case financial analysis.

Figure 22-1 Historical Copper Price vs US$:C$ FX Rate



Yellowhead Mining Inc., July 2014 (www.investing.com/commodities/copper-historical-data; www.oanda.com/currency/historical -rates)

22.10 DISCOUNTING

Estimated annual net cash flows were discounted to the beginning of project Year -2, Q2 2016 at real discount rates of 8%.

22.11 NET CASH FLOW AFTER TAX

Net Cash Flow after Tax amounts to US$2.0bn for the LOM.

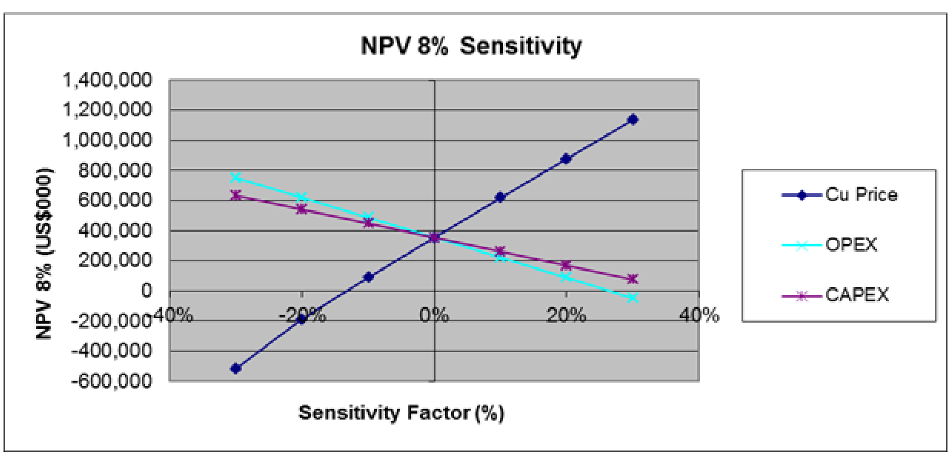
22.12 NPV AND IRR

The base case economic analysis (Table 22-8) indicates that the project has an after-tax Internal Rate of Return (IRR) of 13.4% with a payback period of 5.4 years. Table 22-8 and Figure 22-3 compares the base case project financial indicators with the financial indicators for other cases when the sales price, operating cost and the amount of capital expenditure vary from the base case values. The sensitivity study indicates that both the project's NPV and IRR are most sensitive to changes in metal price. Variation of operating cost and capital cost result in similar sensitivities. The annual cash flows are summarized in Table 22-9 to Table 22-11.

Table 22-8 Financial Analysis

Analysis		NPV @ 0% (US$M)	NPV @ 5% (US$M)	NPV @ 8% (US$M)	NPV @ 10% (US$M)	IRR (%)	Payback Years
Base Case	Pre-tax	3,078.6	1,212.7	683.6	448.5	16.8%	4.9
	After tax	1,977.6	717.0	354.9	192.4	13.4%	5.4
Base Case Sensitivities – After Tax							
Cu Price +10%		2,645.4	1,071.4	617.3	412.8	17.2%	4.1
Cu Price -10%		1,310.2	360.0	89.1	-31.9	9.4%	7.0
Op Costs +10%		1,636.8	536.7	222.2	81.3	11.5%	6.0
Op Costs -10%		2,318.6	896.8	487.1	302.8	15.3%	4.8
Capex Costs +10%		1,871.9	619.8	262.0	102.0	11.7%	6.1
Capex Costs -10%		2,083.4	814.2	447.8	282.8	15.6%	4.5

Figure 22-2 NPV 8% Sensitivity



Yellowhead Mining Inc., July 2014

Table 22-9: Harper Creek Cash Flow Projection Year -2 to 9

MINE PRODUCTION	Units	-2	-1	1	2	3	4	5	6	7	8	9
Ore	dmt x 1000		9,538	37,203	39,312	22,720	13,144					
Waste	dmt x 1000		21,312	22,447	18,950	25,822	24,900	33,651	29,398	27,995	31,393	31,347
Total	dmt x 1000		30,850	59,650	59,500	60,000	60,000	60,000	60,000	60,000	60,000	60,000
Stockpile Balance	dmt x 1000		9,538	23,746	38,746	47,374	56,924	57,723	62,775	69,230	72,287	75,390
MILL PRODUCTION	Units	-2	-1	1	2	3	4	5	6	7	8	9
Mill Feed	dmt x 1000			22,995	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550
Cu	%			0.34	0.35	0.32	0.27	0.27	0.30	0.30	0.26	0.26
Au	g/t			0.04	0.04	0.04	0.03	0.03	0.04	0.03	0.03	0.03
Ag	g/t			1.16	1.32	1.37	1.19	1.07	1.26	1.33	1.28	1.24
Concentrate	dmt x 1000			280	319	295	244	244	276	275	232	230
Copper Grade	%			25.50	25.50	25.50	25.50	25.50	25.50	25.50	25.50	25.50
Gold Grade	g/t			2.03	2.06	1.95	1.87	1.95	1.95	1.92	1.96	1.85
Silver Grade	g/t			54.3	63.1	71.8	72.4	62.5	68.8	74.2	83.6	80.6
Copper	lbs x 1000			157,600	179,361	165,795	137,007	137,005	154,946	154,840	130,290	129,166
Gold	oz x 1000			18.3	21.1	18.5	14.6	15.3	17.3	17.0	14.6	13.6
Silver	oz x 1000			489.6	647.3	681.1	567.0	489.8	609.7	656.9	623.0	595.2
REVENUE	Units	-2	-1	1	2	3	4	5	6	7	8	9
Payable Copper	lbs x 1000			150,634	171,433	158,467	130,951	130,949	148,097	147,996	124,531	123,457
Payable Gold	oz x 1000			16.4	18.9	16.6	13.1	13.7	15.5	15.2	13.0	12.2
Payable Silver	oz x 1000			438.2	579.4	609.6	507.4	438.4	545.7	587.9	557.6	532.7
Gross Revenue	US$ x 1000			505,515	577,434	534,133	440,629	439,887	498,620	498,951	421,538	416,442
Deductions	US$ x 1000			21,805	25,028	23,172	19,073	18,999	21,600	21,652	18,389	18,084
Loses	US$ x 1000			2,528	2,887	2,671	2,203	2,199	2,493	2,495	2,108	2,082
Off-site Costs	US$ x 1000			66,856	76,128	70,399	58,172	58,142	65,780	65,754	55,361	54,868
Net Smelter Return	US$ x 1000			414,326	473,390	437,892	361,182	360,546	408,747	409,049	345,681	341,407
Net Smelter Return	US$/t milled			18.02	18.53	17.14	14.14	14.11	16.00	16.01	13.53	13.36
OPERATING EXPENSES	Units	-2	-1	1	2	3	4	5	6	7	8	9
Mining	C$ x 1000			102,483	108,738	119,258	111,401	118,776	126,246	126,856	119,174	126,510
Milling	C$ x 1000			83,877	93,196	93,196	93,196	93,196	93,196	93,196	93,196	93,196
Site Services	C$ x 1000			7,979	8,866	8,866	8,866	8,866	8,866	8,866	8,866	8,866
General & Admin	C$ x 1000			10,164	11,293	11,293	11,293	11,293	11,293	11,293	11,293	11,293
US Steel NSR	C$ x 1000			2,668								
Noranda Royalty	C$ x 1000											
Total Operating Costs	C$ x 1000			207,171	222,093	232,613	224,756	232,131	239,601	240,211	232,529	239,865
Total Operating Costs	US$ x 1000			186,454	199,884	209,352	202,280	208,918	215,641	216,190	209,276	215,878
Operating Margin	US$ x 1000			227,872	273,507	228,540	158,901	151,628	193,106	192,860	136,405	125,529
Total operating cost/t milled	C$/t milled			9.01	8.69	9.10	8.80	9.09	9.38	9.40	9.10	9.39
PRE-TAX CASH FLOW	Units	-2	-1	1	2	3	4	5	6	7	8	9
Capital Costs	US$ x 1000	-403,107	-513,046	-50,837	-3,426	-3,275	-9,993	-16,004	-3,517	-4,703	-19,383	-5,445
Reclamation Bonding	US$ x 1000		-7,110	-7,560	-8,820	-5,400	-4,410		-720	-3,600	-1,350	-810
Net Cash Flow	US$ x 1000	-403,107	-520,156	169,475	261,261	219,865	144,498	135,624	188,869	184,557	115,672	119,274
Cumulative Cash Flow	US$ x 1000	-403,107	-923,263	-753,788	-492,526	-272,662	-128,164	7,461	196,329	380,886	496,559	615,832
Cash Cost per lb. (excl. Cap.)	(US $/lb)			1.18	1.11	1.26	1.48	1.52	1.39	1.40	1.61	1.67
TAXES	Units	-2	-1	1	2	3	4	5	6	7	8	9
Federal Income Tax	US$ x 1000						4,298	16,262	22,623	21,390	14,837	13,521
Provincial Income Tax	US$ x 1000						3,843	11,925	16,591	15,686	10,881	9,916
BC Mining Tax	US$ x 1000			4,605	5,470	4,571	3,178	3,033	9,129	24,456	15,193	15,605
Total Taxes	US$ x 1000			4,605	5,470	4,571	11,319	31,220	48,343	61,532	40,911	39,042
AFTER-TAX CASH FLOW	Units	-2	-1	1	2	3	4	5	6	7	8	9
Net Cash Flow	US$ x 1000	-403,107	-520,156	164,870	255,791	215,294	133,179	104,405	140,526	123,024	74,761	80,231
Cumulative Cash Flow	US$ x 1000	-403,107	-923,263	-758,393	-502,602	-287,308	-154,129	-49,724	90,801	213,826	288,587	368,818

Table 22-10: Harper Creek Cash Flow Projection Year 10–20

MINE PRODUCTION	Units	10	11	12	13	14	15	16	17	18	19	20
Ore	dmt x 1000	33,652	29,551	31,804	30,980	28,054	39,004	26,524	24,907	24,508	25,044	25,279
Waste	dmt x 1000	26,348	30,449	28,196	29,020	31,946	20,996	28,476	15,093	15,492	14,956	10,721
Total	dmt x 1000	60,000	60,000	60,000	60,000	60,000	60,000	55,000	40,000	40,000	40,000	36,000
Stockpile Balance	dmt x 1000	83,492	87,493	93,747	99,177	101,681	115,135	116,109	115,466	114,424	113,918	113,647
MILL PRODUCTION	Units	10	11	12	13	14	15	16	17	18	19	20
Mill Feed	dmt x 1000	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550	25,550
Cu	%	0.26	0.26	0.28	0.30	0.26	0.26	0.24	0.24	0.24	0.25	0.26
Au	g/t	0.03	0.03	0.03	0.03	0.03	0.03	0.02	0.02	0.02	0.03	0.03
Ag	g/t	1.22	1.24	1.29	1.36	1.21	1.29	1.18	1.25	1.20	1.11	1.05
Concentrate	dmt x 1000	236	232	249	268	232	233	210	208	216	223	228
Copper Grade	%	25.50	25.50	25.50	25.50	25.50	25.50	25.50	25.50	25.50	25.50	25.50
Gold Grade	g/t	1.61	1.70	1.61	1.58	1.59	1.49	1.55	1.53	1.56	1.64	1.77
Silver Grade	g/t	76.8	80.3	78.3	78.6	77.9	84.0	82.7	89.7	82.4	71.7	64.8
Copper	lbs x 1000	132,953	130,437	140,195	150,543	130,440	131,146	117,841	116,960	121,201	125,098	127,995
Gold	oz x 1000	12.2	12.7	12.9	13.6	11.8	11.2	10.4	10.3	10.8	11.8	12.9
Silver	oz x 1000	584.2	599.0	627.8	677.0	580.8	630.1	557.2	600.1	571.0	513.2	474.4
REVENUE	Units	10	11	12	13	14	15	16	17	18	19	20
Payable Copper	lbs x 1000	127,077	124,672	133,999	143,889	124,674	125,349	112,632	111,791	115,844	119,569	122,337
Payable Gold	oz x 1000	10.9	11.4	11.6	12.2	10.6	10.0	9.3	9.2	9.7	10.5	11.6
Payable Silver	oz x 1000	522.9	536.1	561.9	605.9	519.8	563.9	498.7	537.1	511.1	459.3	424.6
Gross Revenue	US$ x 1000	425,818	419,146	449,305	482,174	417,735	420,047	377,704	375,713	388,534	400,251	409,645
Deductions	US$ x 1000	18,331	18,123	19,359	20,758	17,981	18,084	16,276	16,238	16,746	17,207	17,618
Loses	US$ x 1000	2,129	2,096	2,247	2,411	2,089	2,100	1,889	1,879	1,943	2,001	2,048
Off-site Costs	US$ x 1000	56,452	55,399	59,532	63,925	55,386	55,701	50,049	49,695	51,476	53,102	54,317
Net Smelter Return	US$ x 1000	348,906	343,528	368,168	395,079	342,279	344,161	309,490	307,901	318,370	327,941	335,662
Net Smelter Return	US$/t milled	13.66	13.45	14.41	15.46	13.40	13.47	12.11	12.05	12.46	12.84	13.14
OPERATING EXPENSES	Units	10	11	12	13	14	15	16	17	18	19	20
Mining	C$ x 1000	120,083	119,789	121,125	136,796	114,637	112,157	114,606	96,410	94,566	91,777	85,396
Milling	C$ x 1000	93,196	93,196	93,196	93,196	93,196	93,196	93,196	93,196	93,196	93,196	93,196
Site Services	C$ x 1000	8,866	8,866	8,866	8,866	8,866	8,866	8,866	8,866	8,866	8,866	8,866
General & Admin	C$ x 1000	11,293	11,293	11,293	11,293	11,293	11,293	11,293	11,293	11,293	11,293	11,293
US Steel NSR	C$ x 1000											
Noranda Royalty	C$ x 1000							4,716	4,978	13,595		
Total Operating Costs	C$ x 1000	233,438	233,144	234,481	250,151	227,992	225,512	232,678	214,743	221,516	205,132	198,751
Total Operating Costs	US$ x 1000	210,094	209,830	211,033	225,136	205,193	202,960	209,410	193,269	199,365	184,619	178,876
Operating Margin	US$ x 1000	138,812	133,698	157,135	169,943	137,086	141,201	100,080	114,633	119,005	143,322	156,786
Total operating cost/t milled	C$/t milled	9.14	9.13	9.18	9.79	8.92	8.83	9.11	8.40	8.67	8.03	7.78
PRE-TAX CASH FLOW	Units	10	11	12	13	14	15	16	17	18	19	20
Capital Costs	US$ x 1000	-14,238	-12,052	-15,565	-8,467	-3,285	-6,738	-33,825	-18,828	-20,394	-4,306	-2,837
Reclamation Bonding	US$ x 1000	-2,880	-1,890	-2,790	-2,610	-540	-5,670					
Net Cash Flow	US$ x 1000	121,694	119,756	138,780	158,867	133,261	128,793	66,255	95,805	98,611	139,016	153,949
Cumulative Cash Flow	US$ x 1000	737,527	857,282	996,063	1,154,929	1,288,190	1,416,983	1,483,238	1,579,043	1,677,654	1,816,670	1,970,619
Cash Cost per lb. (excl. Cap.)	(US $/lb)	1.58	1.61	1.51	1.50	1.57	1.55	1.78	1.65	1.64	1.48	1.40
TAXES	Units	10	11	12	13	14	15	16	17	18	19	20
Federal Income Tax	US$ x 1000	15,805	15,253	18,473	20,140	16,082	16,968	11,689	12,790	13,118	16,395	18,587
Provincial Income Tax	US$ x 1000	11,590	11,185	13,547	14,770	11,794	12,443	8,572	9,380	9,620	12,023	13,631
BC Mining Tax	US$ x 1000	16,180	15,802	18,388	20,983	17,391	17,473	9,131	13,018	14,389	18,068	20,011
Total Taxes	US$ x 1000	43,576	42,239	50,408	55,894	45,267	46,885	29,392	35,188	37,128	46,485	52,229
AFTER-TAX CASH FLOW	Units	10	11	12	13	14	15	16	17	18	19	20
Net Cash Flow	US$ x 1000	78,119	77,516	88,373	102,973	87,994	81,909	36,863	60,617	61,484	92,531	101,721
Cumulative Cash Flow	US$ x 1000	446,937	524,453	612,826	715,799	803,792	885,701	922,564	983,181	1,044,664	1,137,195	1,238,915

Table 22-11: Harper Creek Cash Flow Projection Year 21-28

MINE PRODUCTION	Units	21	22	23	24	25	26	27	28	TOTAL
Ore	dmt x 1000	26,922	26,852	26,103	16,197					718,166
Waste	dmt x 1000	8,078	8,148	3,897	2,656					541,687
Total	dmt x 1000	35,000	35,000	30,000	18,853					1,259,853
Stockpile Balance	dmt x 1000	115,019	116,321	116,874	106,456	79,841	53,226	26,611	1,986	1,986
MILL PRODUCTION	Units	21	22	23	24	25	26	27	28	TOTAL
Mill Feed	dmt x 1000	25,550	25,550	25,550	26,615	26,615	26,615	26,615	24,620	716,175
Cu	%	0.27	0.27	0.29	0.24	0.15	0.15	0.18	0.17	0.26
Au	g/t	0.03	0.03	0.03	0.02	0.02	0.02	0.02	0.02	0.03
Ag	g/t	1.05	1.12	1.32	1.18	0.95	0.93	0.89	0.92	1.18
Concentrate	dmt x 1000	242	245	266	221	132	134	156	142	6,466
Copper Grade	%	25.50	25.50	25.50	25.50	25.50	25.50	25.50	25.50	25.50
Gold Grade	g/t	1.79	1.77	1.76	1.60	1.91	1.78	1.77	1.76	1.77
Silver Grade	g/t	61.7	66.1	75.7	82.0	101.4	98.2	78.8	83.2	75.2
Copper	lbs x 1000	135,820	137,540	149,775	124,124	74,201	75,147	87,625	80,102	3,635,151
Gold	oz x 1000	13.9	13.9	15.1	11.3	8.1	7.7	8.9	8.1	368.0
Silver	oz x 1000	479.1	519.8	648.3	582.1	430.4	422.1	394.8	381.0	15,632.0
REVENUE	Units	21	22	23	24	25	26	27	28	TOTAL
Payable Copper	lbs x 1000	129,816	131,460	143,155	118,638	70,921	71,825	83,752	76,562	3,474,477
Payable Gold	oz x 1000	12.4	12.5	13.5	10.1	7.3	6.9	7.9	7.2	329.4
Payable Silver	oz x 1000	428.8	465.2	580.2	521.0	385.2	377.7	353.4	341.0	13,990.7
Gross Revenue	US$ x 1000	434,400	440,431	481,187	398,169	241,361	243,467	281,844	258,023	11,678,102
Deductions	US$ x 1000	18,666	18,956	20,800	17,177	10,602	10,640	12,202	11,192	504,759
Loses	US$ x 1000	2,172	2,202	2,406	1,991	1,207	1,217	1,409	1,290	58,391
Off-site Costs	US$ x 1000	57,628	58,372	63,599	52,718	31,560	31,952	37,215	34,029	1,543,567
Net Smelter Return	US$ x 1000	355,934	360,901	394,382	326,284	197,993	199,658	231,018	211,513	9,571,386
Net Smelter Return	US$/t milled	13.93	14.13	15.44	12.26	7.44	7.50	8.68	8.59	13.36
OPERATING EXPENSES	Units	21	22	23	24	25	26	27	28	TOTAL
Mining	C$ x 1000	81,004	79,403	72,714	56,665	32,462	32,546	32,056	29,610	2,764,586
Milling	C$ x 1000	93,196	93,196	93,196	97,081	97,081	97,081	97,081	89,805	2,612,321
Site Services	C$ x 1000	8,866	8,866	8,866	9,235	9,235	9,235	9,235	8,543	248,513
General & Admin	C$ x 1000	11,293	11,293	11,293	11,764	11,764	11,764	11,764	10,882	316,549
US Steel NSR	C$ x 1000									2,668
Noranda Royalty	C$ x 1000					1,506				24,796
Total Operating Costs	C$ x 1000	194,359	192,758	186,069	174,745	152,048	150,626	150,136	138,841	5,888,090
Total Operating Costs	US$ x 1000	174,923	173,482	167,462	157,271	136,844	135,563	135,123	124,957	5,299,281
Operating Margin	US$ x 1000	181,011	187,419	226,919	169,013	61,149	64,094	95,895	86,556	4,272,105
Total operating cost/t milled	C$/t milled	7.61	7.54	7.28	6.57	5.71	5.66	5.64	5.64	8.22
PRE-TAX CASH FLOW	Units	21	22	23	24	25	26	27	28	TOTAL
Capital Costs	US$ x 1000	-1,165	-2,234	-7,666					31,976	1,152,359
Reclamation Bonding	US$ x 1000				-45,000	6,538	15,182	11,656	26,611	-41,173
Net Cash Flow	US$ x 1000	179,845	185,185	219,253	124,013	67,687	79,276	107,551	145,142	3,078,573
Cumulative Cash Flow	US$ x 1000	2,150,465	2,335,650	2,554,902	2,678,915	2,746,603	2,825,879	2,933,431	3,078,573	3,078,573
Op. Cash Cost per lb	(US $/lb)	1.29	1.26	1.12	1.27	1.84	1.80	1.54	1.56	1.46
TAXES	Units	21	22	23	24	25	26	27	28	TOTAL
Federal Income Tax	US$ x 1000	22,123	23,297	28,674	21,096	7,233	7,824	12,109	10,992	401,582
Provincial Income Tax	US$ x 1000	16,224	17,085	21,028	15,470	5,304	5,738	8,880	8,061	295,185
BC Mining Tax	US$ x 1000	23,379	24,072	28,495	21,972	8,126	8,332	12,466	11,252	404,168
Total Taxes	US$ x 1000	61,726	64,454	78,197	58,537	20,663	21,894	33,455	30,304	1,100,935
AFTER-TAX CASH FLOW	Units	21	22	23	24	25	26	27	28	TOTAL
Net Cash Flow	US$ x 1000	118,120	120,731	141,055	65,476	47,024	57,383	74,096	114,838	1,977,638
Cumulative Cash Flow	US$ x 1000	1,357,035	1,477,766	1,618,822	1,684,297	1,731,322	1,788,704	1,862,800	1,977,638	1,977,638

23 ADJACENT PROPERTIES

There are no adjacent operational mining properties that would lead to a better understanding of this project. See Section 8.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 BC HYDRO

Currently the North Thompson Valley is serviced by a long radial 138kV transmission system supplied by BC Hydro's Brocklehurst Substation (BKL) at the southern end of the system and extending to the customer-owned Rearguard Substation (REG) approximately 325km north of BKL. As part of BC Hydro's Growth Capital plan a transmission project has been identified to reinforce the North Thompson transmission system in order to meet future area load growth and provide additional system reliability. Originally proposed to enter service in October 2014 the project has been deferred as a result of forecast load requirements. However, the development of the Harper Creek Project would facilitate the need for additional power to the North Thompson Valley.

Preliminary studies undertaken by BC Hydro and funded inpart by YMI have identified 3 possible options for a transmission line to supply additional power to the North Thompson, these being:

- Construct a 230kV transmission line from Hundred Mile House Substation (HMH) to Clearwater Substation (CLW).
- Construct a 230kV transmission line from a new 500/230kV substation located at the approximate mid-point of 500kV Mica-Nicola transmission line to the Vavenby Substation (VBY).
- Construct a second 138kV transmission line from Brocklehurst Substation (BKL) to Vavenby Substation (VBY).

Work required to advance the project includes the following:

- Study Phase – review existing options
- Definition Phase – option selection
  - System Planning
  - Regulatory – British Columbia Utilities Commission (BCUC)
  - Consultation – Public, First Nation
  - Environmental
  - Basic Engineering – Identify Right of Way
- Implementation Phase
  - Detailed Engineering
  - Land acquisition
  - Construction
  - Commissioning

Approximate timing for this work is estimated at 3 years which fits within YMI's proposed development timeline. YMI has allocated C$0.5M to initiate the study phase early in 2015 to ensure power is available for operations in 2018.

24.2 PROJECT EXECUTION PLAN

24.2.1 *PLAN OBJECTIVE*

The development of a practical Project Execution Plan (PEP, the Plan) at the feasibility stage of the Project is integral to the success of detailed engineering. It helps form the basis for the work in so far as it provides the blueprint upon which assumptions were made during feasibility that would be used to understand what the FS team were thinking when they developed the study. This basis led to many of the cost and schedule conclusions reached by the FS Team. While many assumptions were made, they were based on experience and knowledge of how a project such as Harper Creek would be developed.

In general, the PEP provides a platform to support the successful interaction of engineering, procurement and construction activities. This interaction embraces communications, technical and practical issues, safety, environmental, government, social issues, and all other facets of the project that end when mine operation starts and the Owner takes over a plant.

The importance of the PEP for moving forward is paramount for the next phase of the development. Strategizing now establishes a jumping off point for the execution team who should be encouraged to study the PEP and follow it with a view to maintaining budget and schedule. There will always be unforeseen changes that occur during the detailed stage of the Project but a good PEP will remain as a thread to successful completion and handover to operations.

The PEP is intended to take the reader through project development generalities in a way that provides a description of how the Project should move forward through to Mechanical Completion, after which the Owner will take control of plant operations. The PEP includes discussion of how the following activities will be managed:

- Detailed engineering;
- Long lead delivery equipment;
- Freight;
- Construction field requirements;
- Ordering bulk materials;
- Site environmental requirements;
- Site safety requirements;
- Site security requirements;
- Construction resources;
- Accommodation for construction and operating work force; and
- Commissioning the plant and handover to Owner.

24.2.2 *PLAN EXECUTION STRATEGY*

24.2.2.1 Pre-Construction Phase

The Pre-Construction Phase includes activities between the period leading up to the financing and permitting approvals such as:

- Continuation of resource drilling;
- Additional geotechnical drilling and test pitting to support final design;
- Continuation of environmental monitoring;
- Permit application support;
- Project design and construction optimization;
- Researching local resource availability;
- Negotiation with long delivery vendors;
- Establishing the availability and suitability of contractors;
- Establishing the Owner in the fabrication line-up for long delivery equipment;
- Preparing the project management administration office at the Vavenby lay down area;
- Finalizing Owner commitments to the project including the mine plan; and
- Improving site access.

24.2.2.2 Basic Engineering Phase

Basic Engineering work commences when the Owner releases sufficient funding. It is envisaged that certain items will be finalized with a view to prepare for the Detailed Engineering and Construction stage, including:

- Developing the project management control document;
- Flowsheet finalization;
- Long delivery equipment ordering; crusher, mills, main transformers and large mill motors;
- Water balance finalization;
- General arrangement drawings fixed;
- Constructability reviews;
- Bulk earthwork drawings brought up to a level for construction;
- Tailings dam Stage 1a drawings brought up to a level for construction;
- Overhead power line design route surveyed and fixed;
- Mobilizing a small project team to the new Vavenby offices;
- Searching out site aggregate processing and concrete batching plants;
- Tendering for the clearing and grubbing;
- Tendering for the road upgrade work;
- Searching out and arranging for the construction camp units and catering services;
- Establishing the boiler plate for contracts and purchase orders;
- Finalizing the project schedule based on all the information gathered to that point;
- Establishing the cost reporting and control system;
- Establishing the field survey contract;
- Establishing the quality assurance contract;

- Arranging the freight forwarding contract; and
- Arranging the temporary construction facilities including fuel and water.

24.2.2.3 Detailed Engineering and Construction Phase

Construction activities may not start in the field until the Mine Permit for initial development is approved and granted. Other permits will be required during construction for some of the special trades such as electrical, and later an operating permit for the plant.

24.2.2.4 Detailed Engineering (Design) and Procurement

Once the Owner has established the Project Engineering and Procurement (EP) team, and sufficient financing is in place, detailed engineering and procurement may start. It is expected that some Basic Engineering, as generally described previously, will be undertaken before full financing is available so that the detailed work can start in earnest to produce construction drawings for the earthworks and civil phases of the project when the time comes.

Capital equipment purchases will be based on the flowsheet and performance specifications for all items that have been assigned an equipment number, right down to instruments.

The EP team will develop packages of drawings (deliverables) into groupings that follow the construction contracting strategy; concrete, buildings, structural steel, and piping, etc. For Harper Creek it is intended that work packages will be bid and awarded based on availability of engineering information, in a progressive situation.

24.2.2.5 Construction

YMI intends to act as the General Contractor and employ the services of ‘sub-contractors’ to perform the construction work in an ‘Open Shop” environment. This means the services of the largest pool of construction talent will be made available from CLRA (Construction Labour Relations Association) membership and alternative unions, such as CLAC (Christian Labour Association Contractors) and non-union contractors. Construction contracts will be worded to protect this strategy and all contractors will need to subscribe to working in such an environment.

The construction effort will be ‘construction driven’ such that the scheduled site work will dictate the engineering and procurement needs to meet the target completion date. Approximately 600 construction persons will be required at the peak which occurs in the second construction season when work is focused on the civil, mechanical and electrical work.

Operations personnel will be housed offsite in Vavenby, Clearwater and the surrounding area. Construction personnel will be housed on-site in a temporary modular camp. The camp will provide single occupancy rooms but workers will share common laundry, washroom, dining and recreational facilities. The current plan is that the construction camp will be leased for the term of the project and removed once construction is completed.

The majority of contracts will be “Unit Price”, i.e. engineer established quantities will form the basis for the tendered prices to establish an estimated lump sum for each package of work. The following packages are anticipated:

- Clearing and grubbing;
- Bulk earthworks and main access road improvements;
- High voltage power line;
- Main substation;
- Concrete and detailed earthworks;
- Yard services;
- Pre-engineered buildings supply and installation;
- Structural steel supply and installation;
- Mechanical and piping;
- Electrical and instrumentation;
- Tanks;
- Tailings starter dam Stage 1a;
- Overland conveying system;
- Overland piping system;
- Rail siding repairs;
- Concentrate load out building and cold storage; and
- Miscellaneous construction packages.

24.2.2.6 Earthworks

As soon as detailed engineering begins, some of the early design will be focused on the site civil works including tailings, crusher and process plant areas to allow contracts for the earthworks in those areas to be tendered to start in the field early in the construction season. However, the construction strategy is to also make use of the permanent open pit mine production equipment as much as possible to help build the large earthwork structures like haul roads outside of the pit boundaries, the backfill for the primary crusher and the tailings dam. The FS capital cost estimate is based on using Non-Potentially Acid Generating construction material (Non-PAG) from the open pit to as great an extent as possible during construction. Sufficient material that has to be removed and wasted will be available during the construction period.

The construction strategy, project schedule and capital cost estimate is based on mine equipment being used to construct:

- haul roads to the crusher, dumps and tailings area;
- bulk fill behind the primary crusher ME or Hilfiker style wall; and
- the bulk fill (not the core material) for the starter tailings dam.

The material used for these major earth structures will come from Non-PAG material from the northern area of the open pit and from within the tailings impoundment. As a consequence of this strategy, a portion of the permanent mine fleet will be made operational early in the construction program with at least one of the shovels and one drill operating under temporary diesel generated power, since the main hydro line power will not be available until well into the mechanical equipment installation phase. It is expected that three 2000kW generator sets will be needed to power up the mine equipment and that they will be installed at the site of the new plant site sub-station. At least two of the units will remain for operations and act as the emergency generators for the plant. An overhead line will be constructed to the open pit boundary from where a closed loop power line will be constructed around that part of the open pit to be initially developed. Trailing cables will lead from the loop to the equipment. To this end, the fuel requirements for this additional mine equipment work is included in the CAPEX outside of the pre-production costs.

The use of the mine fleet in the locations identified is critical to keeping construction costs down and maintaining schedule. Also, as a consequence of using the mine fleet to build certain areas, the roads will be built to mine standards for mine equipment, i.e., three times the width of the haul truck for two-way traffic, or two times the width of the mine truck for one-way traffic. The schedule anticipates that the assembly of the mine fleet to be used for construction and the associated construction of the haul roads will take a total of about 8 months. This is conservative but takes some account of any adverse weather conditions that may be experienced. It has been proven that the use of such large equipment on heavy earthwork projects is a really productive advantage in terms of not only large volume movements but those that can also move forward during the winter months when the high volumes are sufficient in size to keep the material from freezing before it is graded and compacted by the mine fleet.

The plan for the construction of the starter dam is to have it completed to the 1,723m elevation by the time of start up. At this point it will have captured at least one full freshet and have a filled pond capacity capable of sustaining the fresh water requirements of the operating plant. While the mine fleet will construct the major part of the dam shell, both upstream and downstream, a contractor will be needed to construct the core with materials available from local borrows. The core is too narrow to use the mine fleet equipment.

The capital cost advantage of using this strategy is that the cost per unit of material moved from the mine to the tailings area represents only the premium attributed to any overhaul required to otherwise take the mine waste to the waste dump. Either way, the cost of pit overburden removal is included in the mining component of the Capital Costs.

While the use of the mine fleet will mean constructing haul roads to mine standards to the dam, they must be constructed regardless since:

- mine equipment will be used for future dam raises, and
- the PAG dump is located within the tailings impoundment also necessitating mine fleet haulage access.

The use of a partial mine fleet for some of the construction effort will mean bringing the equipment onto the site earlier than may otherwise be planned if it was to only handle the pre-production mining. It is expected that at least two of the new haul trucks, and their supporting loading and drilling equipment, will be needed for the construction of the crusher backfill and haul roads to the crusher and the tailings starter dam. The project schedule has accounted for the early purchase and assembly on site of these mine components.

The existing access to the tailings area can be used by contractors for developing the sources in the tailings area for the core materials.

Bulk earthworks for the plant site and crusher foundations will be undertaken by a contractor. It is planned as a balanced cut and fill program. Site geotechnical investigations during feasibility determined there are sufficient construction material types available for:

- aggregate materials for general and structural backfill;
- sand for pipe and electrical cable bedding; and
- filters for the tailings dam.

It is planned to establish aggregate crushing and screening plant at the site to stockpile the various sizes and types of material needed for construction.

24.2.2.7 Concrete

Geotechnical investigations performed during the FS determined there are suitable concrete aggregates available within the mine license area; the best location being the grano-diorite to the south of tailings. The intent is to locate a 150m^3 to 200m^3 per hour concrete batching plant somewhere close to the new plant site where the majority of concrete will be required. Allowances will be made for heating the aggregate stockpiles during cold weather. It is expected that several large pours of 16 to 24 hour durations will be required, including the crusher mat foundation, mill foundations and mill piers. While the construction plan requires two or three concrete trucks with 5m^3 to 8m^3 capacity on a year-round basis, there will be a requirement to bring in additional trucks from local plants when the large pours are scheduled.

Most of the concrete will be placed in the first full year of construction so that buildings can be erected and closed in before the first winter when internal installations will commence.

24.2.2.8 Steel and Buildings

Wherever possible the project will make use of pre-engineered buildings, including the concentrator. There are five overhead cranes in the concentrator, with the largest over the grinding area. It is sized as a 120t capacity to handle the largest single piece of the equipment to be installed during construction. Each section of the wrap around motor for the SAG Mill will weigh approximately 95t.

In addition to the pre-engineered buildings, there will be a textile building used for the cold storage warehouse and perhaps the concentrate load out area in Vavenby, and modular buildings for various offices around the site where needed. At this point, the Vavenby load-out building is priced as pre-engineered but the option to move to a textile building could be considered. The majority of the administration staff will use existing offices at the Vavenby location thus reducing the need for many offices and training rooms at the site Truck Shop.

24.2.2.9 Mechanical and Piping

The majority of capital equipment will be shipped via Edmonton or Vancouver, particularly if sourced from off-shore destinations. A major freight forwarder will manage freight from the vendor’s plant (domestic or international) to a temporary lay down area in Vavenby for all containerized equipment. Other than the major break-bulk equipment such as the mill and crusher parts, the equipment will be relocated to site as needed rather than have it taken directly to site where lay down areas would have to be prepared, and where special transportation equipment may be needed to get the equipment up the hill from the main highway. There will be a contract with a local, private, freight contractor to bring the freight up to the site from Vavenby as required.

Heavy loads include transformers, crusher parts, mill and motor parts. They will likely be break-bulk loads. The local bridges are capable of allowing these loads to pass over them safely as long as they are high enough to clear the railings and loaded onto multiple axle low-boys. A cost allowance has been made to improve the public road close to the bridge to permit heavy loads on multiple axles to maneuver more readily.

The bulk piping will be purchased directly by YMI's representative. Long radius rubber lined slurry lines and chutes will be pre-spooled and have the linings installed off-site. The rest of the pipe will be spooled at site by the contractor who will also prepare the isometrics. All manual valves will be purchased as bulk orders by YMI and all automatic valves, considered long delivery, will be purchased by EP on behalf of the YMI.

24.2.2.10 Electrical and Instrumentation

Electrical construction starts with the electrification of the construction camp. Approximately 2.5MW of power is available on the existing power line near Vavenby. A 25kV power line will be constructed from a junction point on that line up to the plant site to power the camp and other construction activities. It will not be used for the mine equipment.

All permanent secondary distribution including overhead lines to remote areas from the main sub-station will be contractor performed including process and material handling electrical work. The electrical contractor will be responsible for the installation of MCCs and switchgear as well as the lock-out procedure needed as the plant is connected to live power.

The electrical contractor responsible for the secondary distribution will install the instrumentation and control system. The Owner's engineer will program the control system and commission it in conjunction with YMI.

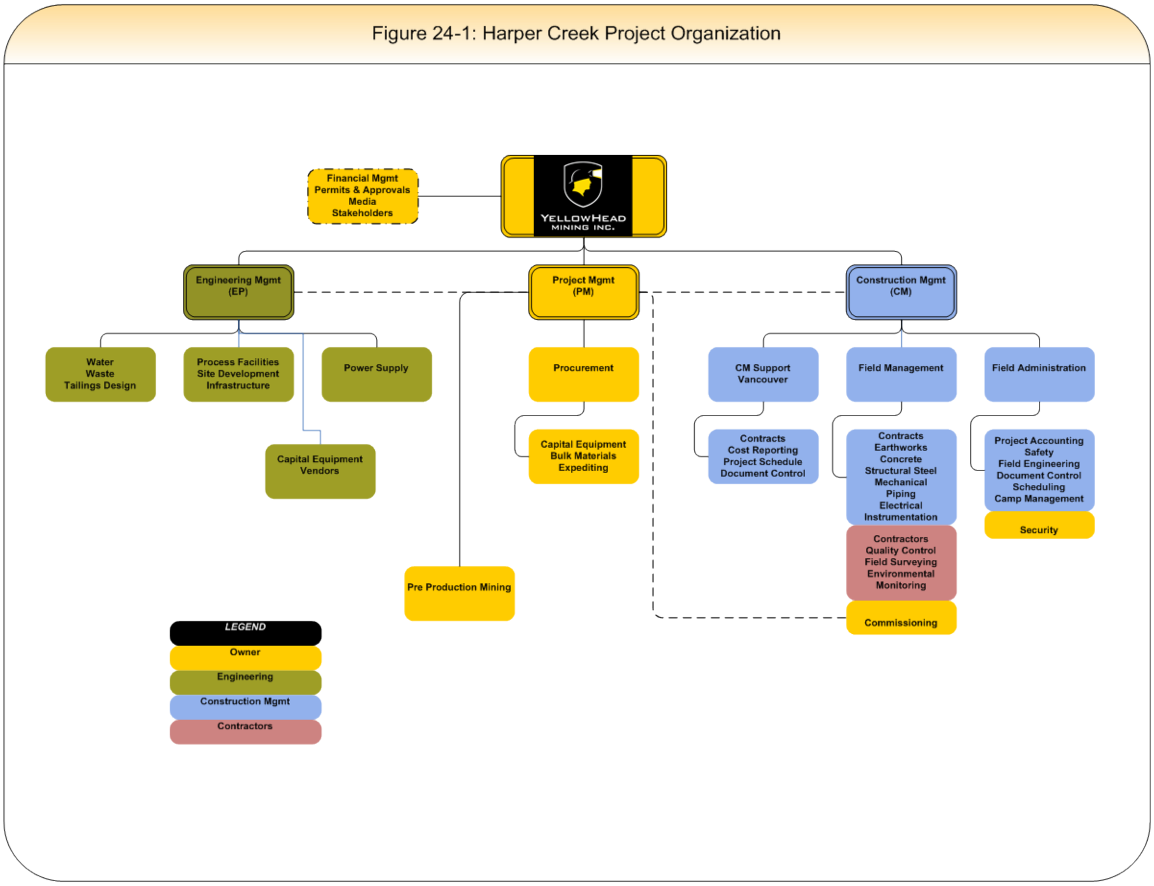
24.2.3 *Management Approach*

Under the administration of YMI's Project Manager (PM), the Engineering (E), Procurement (P) and Construction Management (CM) team will manage the project in accordance with the project schedule, capital cost, health and safety, environmental and quality targets (Figure 24-1 Project Organization Chart).

YMI will be responsible for safety, security, permits and licensing, mine planning and pre-production mining, communication and interaction with the local community and media, financing, accounting and invoice payment, operation staffing, operator training, wet commissioning and start-up.

The EPCM team will provide detailed design, management of the construction program, reporting and controlling project costs, scheduling of engineering and construction, and purchasing all capital equipment on behalf of YMI. Known equipment vendors will be requested to supply capital equipment in a competitive environment.

Figure 24-1: Project Organization



24.2.3.1 Responsibility Matrix

YMI, Engineers (Primary and subs), and Construction Management will comprise the Project Team.

The Primary Engineer will have oversight for:

- Mine design;
- Processing, material handling and infrastructure;
- High voltage power and plant distribution; and
- Tailings dam, water and waste management.

Other consultants such as surveying, material testing and environmental for the purposes of the PEP are considered sub-consultants rather than prime consultants. The full scope of YMI (Owner), EP and CM responsibilities are summarized in Table 24-1.

Table 24-1: Draft Responsibility Matrix

Draft Responsibility Matrix			
Legend: X = Primary Role O = Support Role			
Description	Owner	EP	CM
PROJECT MANAGEMENT			
Camp – Management	O		X
Engineering Controls		X	
Geology	X		
Geotechnical - Mine & Waste Dumps	X		
Geotechnical - Plant Site, Tailings, Infrastructure	O	X	
Hydrological Investigations - U/G	X		
Hydrological Investigations - Surface	O	X	
Government Interface	X		
Insurance - WCB, Gen Liability, 3rd Party & Automobile	X		X
Labour Relations Plan & Site Work Rules	O		X
Legal Surveys	X		
Freight Logistics, Deliveries		X	O
Maintenance & Operating Manuals		X	
Material Balance		X	
Metallurgical Design		X	
Metallurgical Testwork	X	O	
Overall Project Cost System			X
Permits and Approvals (Operation and Construction)	X		
Plant Mobile Equipment Specification, Supply	X		
Process, Plant Site & Utilities Design Interface Coordination		X	
Mine Equipment Specification & Supply	X	O	
Reclamation	X		
Schedule Engineering and Procurement		X	
Scheduling - Overall Project			X
Site Offices for PCM Team	X		O
Site Topographical Survey	X	O	O
Site Utilities for PCM offices (tel, fax, email, lighting, heating, water, etc.)	X		O
Site Vehicles for PCM team	X		O
Coordination with Underwriters	X	O	
Environmental Monitoring	X		
DESIGN			
Overall Site Layout	O	X	O
Mine Planning	O	X	
Mine Services	O	X	
Rough Excavation		X	
Detailed Site Layouts, Design Incl Roads		X	O
Flowsheets		X	
P&IDs		X	
Piping Specifications and Line Lists		X	
Valve Commodity Codes and Specifications		X	
HVAC incl Balancing Requirements		X	
Noise and Vibration Control		X	
Engineered Pipe Supports and Non-engineered Standards		X	
Mechanical Equipment List		X	
Electrical Equipment List		X	
Electrical Cable Cut List		X	
Site Grounding and Cathodic Protection		X	
Site Drainage and Yard Services		X	
Air Strip Extension		X	
Foundations		X	
Concrete Mix Design		O	X
Concrete Aggregate suitability		X	
Structural Steelwork		X	
Structural Steelwork Fabrication Drawings Review		X	

Draft Responsibility Matrix			
Legend: X = Primary Role O = Support Role			
Description	**Owner**	**EP**	**CM**
Architectural Finishes, Painting Specs and Colours		X	
Heat Tracing		X	
Thermal and Stress Analysis		X	
Insulation and Fireproofing Specifications and Details		X	
Admin, Dry, Warehouse	X		
Capital Equipment Specifications		X	
Vendor Drawing Review		X	
Shop Inspections As Needed		X	
Crushing and Conveying System		X	
Ore Processing Facilities		X	
Laboratory Building	X	O	
Laboratory Equipment and Internal fixtures	O	X	
Tailings Dam, Pond and Diversions	X	O	
Tailings Delivery Pipeline and Reclaim Systems		X	
Project Building Utilities		X	
Power Plant		X	
Power Distribution		X	
Water and Waste Management		X	
Construction and Permanent Camp - incl Utilities	O	X	O
Local Power Generation Facilities		X	
Vendor Manuals		X	
Project Control Philosophy and System	O	X	
Plant Automation	O	X	
Loop Drawings		X	
Instrument Lists & Specifications		X	
Protection and Metering Design (if needed)		X	
Programming		X	
Piping Specifications		X	
Commissioning	X	O	O
Fire Protection		X	
Site Communications System		X	
Water Supply System (Process, Fire & Domestic)		X	
Septic System		X	
Diesel Fuel Storage and Dispensing Facility		X	
Explosives Storage N/A		X	
Collection Pond(s)		X	
Mine Dewatering, Settling Ponds, Water Diversions & Impound Dams	O	X	
Communications System (for Construction)	X		O
Document Control – Design		X	
Document Control - General Project & Construction			X
Water Balance	O	X	
Constructability Reviews	O	O	X
Permit Support Information		X	
Bills of Materials		X	
Trend Forecasting Form Quantities, Design Changes, EP Costs		X	
PURCHASING AND EXPEDITING			
Capital Equipment Specifications/Purchasing/Dwg Expediting		X	O
Purchase Orders incl Terms and Conditions for Capital Equipment		X	
Bulk Materials Quantity Take-offs		X	
Purchase Orders incl Terms and Conditions for Bulk Materials and Field Purchases		O	X
Bulk Materials Purchase and Deliveries		O	X
Freight - Capital Equipment	O	X	
Mine Equipment	X	O	
Expediting Capital Equipment		X	O
Equipment Inspection - Vendor Shops		X	
Spare Parts - Selections and Ordering	X	O	
Spare Parts - Start-up & Commissioning	X		O

Draft Responsibility Matrix			
Legend: X = Primary Role O = Support Role			
Description	**Owner**	**EP**	**CM**
Vendor Reps (Erection Support & Commissioning)		O	X
Vendor Reps – Coordination			X
CONSTRUCTION			
Site Construction Management			X
Mining (Contract)	X		
Field Engineering liaison		X	0
Warehouse / Lay Down Area			X
Security Personnel	X		O
Safety Personnel	X		X
Safety Management	O		X
Construction Technical Specifications		X	
Contracting Plan	O	O	X
Contract Bid Documents			X
Contract Tendering, Post Tender Meetings, Recommendations	O		X
Contract Execution & Administration			X
Earthworks and Civil Site Supervision			X
Mechanical and Piping Site Supervision			X
Electrical and Instrumentation Site Supervision			X
Commissioning (as required)	X	O	O
On-site Monitoring of Construction Equip Condition & Safe Operating Capability			X
Survey and Layout (Contract)	X		O
Site Quality Control (Contract)			X
Environmental Monitoring	X		O
Cost Reporting and Controls	O	O	X
"As-Built Drawings" (by Contractors)			X

24.2.3.2 Staffing Plan

YMI will solicit competitive bids from qualified companies to perform the engineering, procurement and construction management activities for the project (regardless of who developed the FS and despite the fact that the EPCM costs carried in the Capital Cost estimate were derived by the FS team). The competition will consider the following issues that would contribute to the selection of the development team for the EP and CM:

- Project team;
- Project management;
- Understanding of salient issues and areas of project risk;
- Understanding possibilities for project improvements;
- Track record in completing similar projects;
- Experience in designing for a central BC climate;
- Experience and resources for construction;
- Experience on construction projects in central BC;
- Proposed execution / work plan which must be submitted with the bid;
- Project engineering and procurement schedule;
- List of deliverables;
- Engineering resources and technical capabilities;
- Environmental stewardship;
- Procurement plan and capabilities;
- Documentation control;
- Project control plan and capabilities;
- Sub-consultants (if applicable);
- Cost and cash flow;
- Project guarantees; and
- Commercial terms.

Management will be supplied by the project team members as follows:

- Owner
- Project Manager
- Metallurgist
- Mine Manager
- Plant Superintendent
- Environmental and Permit Manager
- Safety Management
- Site Security
- Engineering Procurement Consultant
- Engineering Manager
- Procurement Manager
- Metallurgist
- Mine Planner

- Engineering Leads
- Project Engineering Controls
- Construction Management
- Construction Director
- Construction Manager
- Contract Administrator
- Trade Managers
- Field Engineering
- Accounting/Cost Manager
- Scheduler
- Field Expediting/Procurement
- Warehousing/Receiving
- Camp Management
- Site Specific Consultants
- Site Surveying
- Site Quality Assurance
- Vendor Representatives: crusher, mills, lime, main motors, conveyors

24.2.3.3 Project Procedures Manual

The project development strategy must include the development of comprehensive Project Procedures encompassing the requirements and guidelines for the EPCM work. It will be established early, during the first months of consultant selections, and will include the following:

- Project organization, key names, and communication procedures;
- Identification of the division of responsibilities between the project stakeholders;
- Integration of the activities of other designers on the project;
- Reporting requirements, including project systems, project meetings and minutes;
- Project data management (e.g.: format, distribution, filing system);
- Engineering and drawing preparation and transmittal procedures;
- Site procedures, including safety, environmental, and quality requirements;
- Construction work item procedures;
- Procurement and contract procedures;
- Site and office personnel rules and regulations;
- Cost reporting;
- Scheduling; and
- Commissioning.

24.2.4 *ENGINEERING*

24.2.4.1 Engineering (E) Strategy

Main engineering work will be divided into the following design work categories:

- Mine design;
- Power supply;
- Water, waste and tailing design; and
- Process facilities, site development and infrastructure.

Detailed engineering will be completed by the selected engineering group (the "engineer-of-record") and experienced sub-consultants that provide specialist design-supply packages. Engineering will provide in-house discipline design criteria and standard specifications.

Design-supply packages will be developed where possible, allowing specialist groups to input to the main engineering effort (Table 24-2).

Table 24-2: Design / Supply Packages

Design / Supply Packages (anticipated)
Pre-Engineered Buildings
Sewage Treatment Plant
Assay Laboratory
Overland Conveyor, Stacker and SAG Mill Feed Conveyor
Potable Water Treatment Plant
HVAC System
Reclaim Water Barge
Main Substation
Dust Collection

Shop drawings will be provided to the engineer-of-record from contractors providing permanent facilities requiring engineered drawings (rebar bending schedules, wall forms, special crane lifts). Suppliers of structural steel will be expected to engineer and stamp their shop drawings and detailed designs to match the Engineer's design drawings and specifications. The Engineer will sign off the structural shop drawings. Regardless of the Owner's engineering representative sign off on shop drawings from contractors and vendors, the contractor remains responsible and the sign-off does not relieve him of that obligation.

For the purposes of the FS, associated planning and cost estimating, it has been assumed that the division of responsibilities of the FS Team specialist consultants be made up of:

- Process and infrastructure;
- Tailings, water and waste;
- Electrical and controls;
- Mine planning;
- Metallurgy;
- Geosciences; and

- Construction management.

24.2.4.2 Basic Engineering

Basic engineering work scheduled before full construction starts, would include:

- Process Flow Diagrams (PFDs) finalization to reflect any modification made since the FS was issued.
- Specifications will be developed to obtain firm price bids for major equipment, starting with long delivery equipment. The specifications will be issued for bid and submissions will be evaluated.
- Requests for Quotation and associated Purchase Orders for long lead items and major equipment will be prepared for Owner approval and issue.
- Develop the freight contract for equipment delivery to site using an international freight forwarding company.
- Equipment and Load lists will be updated to include information from the firm price bids for each selected vendor for major long delivery equipment.
- Piping & Instrumentation Diagrams (P&IDs) will be updated to include any revisions to equipment information.
- Instrument List will be updated based on the P&IDs.
- General Arrangement Drawings for new facilities will be advanced to allow for the first of the earthwork drawings to be advanced to ‘construction issue’ stage.
- Preliminary primary and secondary distribution Electrical Single Line Diagrams (SLDs) will be updated or prepared for the project.
- Capex will be updated to include firm pricing for equipment, bulk materials and construction bid information.

24.2.4.3 Detailed Engineering

Calculations, specifications, drawings, material requisitions and other items related to detailed engineering will be completed in accordance with the engineering procedures. Items include:

- Complete detailed drawings and documentation associated with the construction of the new facilities.
- Produce design and construction drawings based on site investigations, including surveys, inspections, etc.
- Develop auxiliary equipment specifications and material requisitions for the purchase of bulk materials based on material take-offs.
- Develop technical bid evaluations and purchase recommendations based on proposals for the supply of equipment and materials.
- Develop the technical portion of construction packages, including pertinent drawing package assembly and construction specification development. Prepare scope definitions and schedules.

24.2.5 *Constructability Reviews*

Concurrent with the start of detailed engineering, construction planning and constructability reviews of engineering and procurement may begin. Drawings and documents will be reviewed to determine more effective construction methods and to establish parameters for a prefabrication or preassembly program. Once field work commences, construction will strive for continuous improvement based on activities such as team building, supervisory training, craft participation, problem solving and iterative planning.

Engineering support during construction will be provided from the home offices of the EPCM under the general direction of the Field Engineering Manager. Support activities will include:

- Ensuring that packages of documents issued for construction are complete and up to date;
- Ensuring that technical data and manuals are received and approved;
- Resolving technical questions from the field;
- Ensuring quality is achieved in the field and in fabrication shops;
- Helping to expedite supplier information;
- Checking that commissioning, normal and insurance spares are ordered and delivered;
- Provide a resident engineer if required;
- Collecting as-built records.

24.2.6 *Procurement*

A plan for expediting will be prepared based on the project schedule and equipment lists. The extent to which purchase orders are expedited will be based on complexity, manufacturing cycle time and schedule criticality. Equipment, materials not provided by contractors, certain facility and equipment rentals and consumables will be purchased by the procurement group on behalf of YMI using the Owner's standard terms and conditions, modified as required to meet project specific requirements.

In general, suppliers will be selected based on quality, price and delivery and preference will be given to local suppliers when practical. The CM will organize bulk material purchases such as pipe, cable, cable trays, etc. Pre engineered buildings will be tendered on a lump sum design, fabricate and erect basis including crane rails, doors, windows and insulation.

24.2.7 *Construction Management*

The Construction Manager (CM) manages all activities on the construction site to deliver the safe completion of the project in accordance with the agreed scope of work, budget constraints, schedule and defined quality and safety standards.

The CM provides team leadership and motivation and manages the overall on-site construction effort by identifying priorities, setting goals, duties and objectives. Regular meetings with the Owner, EP and contractor site staff will be convened to clearly communicate the best construction outcome for YMI.

24.2.7.1 CM Objectives & Responsibilities

Key objectives for CM include:

- Employ site hazard management tools and programs to achieve zero accident / no harm HSE objectives;
- Apply contracting and construction infrastructure strategies to support the project execution requirements;
- Develop and implement a construction-sensitive and cost-effective master project schedule;
- Establish a field project control system to ensure effective cost and schedule control, including a cost trending program;
- Establish a field contract administration system to effectively manage, control and coordinate the work performed by the contractors;
- Apply an effective field constructability program, as a continuation of the constructability reviews performed in the design office;
- Develop a detailed field logistics and material control plan to maintain the necessary flow and control of material and equipment to support construction operations;
- Meet the schedule for handover of the plant to the commissioning team.

The CM team is responsible for:

- Field engineering;
- Technical inspection of material and equipment delivered to site;
- Contract supervision;
- Surveys;
- Managing site direct and indirect costs, cost system reporting & project controls;
- Approval of contractors’ payment certificates;
- Materials management and logistics;
- Construction QA/QC; and
- Inspection, commissioning, and system turnover.

24.2.8 *Contracts*

The contracting approach will optimize the use of the local labour force, create a responsible and sustainable relationship within the community, and provide the right mix of management and specialist skills to support the safety, quality, and schedule and cost objectives of the project. Contract packages will be designed to take advantage of a staggered work schedule, some of which may commence before full scope definition is complete, for example, power line, truck fuel station and truck shop. The strategy will be based on the following principles:

- Fixed price contracts are preferred, requiring design engineering to be complete before tendering.
- Incentive based, open book contracts will be used when a contract must be let before engineering is substantially completed.
- Fixed unit prices will be established at the time of tender for undefined work or changes.
- Normally invite tenders from among three to five contractors.
- Let design-supply-and-erect contracts where it makes sense to do so.

- Minimize the number of contractors and sub contractors to minimize administrative costs, interfaces and field overheads.
- Offer incentives for early completion of project-critical tasks.

YMI approved contract templates will be used for all equipment purchase orders, construction and service contracts. Table 24-3 highlights the major construction contract scope outlines and pricing strategies for the Project.

Table 24-3: Major Contract Summary

Major Contract Summary		
Contract Description	Type	Comment
Site Preparation & Tailings Construction	Unit Price	
138 kV Powerline Design/Build to mine site	Lump Sum	
Truck Fuel Station Design/Build	Lump Sum	
Concentrator and Truck Shop Design/Build	Lump Sum	Pre-engineered
Concrete Supply	Unit Price	
Concrete Installation	Unit Price	
Structural Steel Supply and Installation	Unit Price	
Mechanical & Piping	Unit Price +LS	
Electrical & Instrumentation	Unit Price + LS	
Process Control	Fixed Price	
Camp Supply and Installation	Lump Sum	
Yard Services	Unit Prices	
Tanks	Lump Sum	

24.2.9 *COST MANAGEMENT*

24.2.9.1 General

The project budget will be based on the approved FS capital cost estimate, scope of work, schedule and quality plan and will form the baseline against which progress and cost will be measured and managed.

The EP/CM Project Manager will be responsible for cost management and reporting and will provide an integrated project management cost (and schedule) database for cost management and reporting.

24.2.9.2 Budget Allocation & Management

The budget for each work package is the estimated cost, scope and time allocated from the FS WBS. The aggregate of all work package scopes and allocated estimates forms the baseline for the measurement of performance and the baseline for change control.

The Project will be managed by commitments and the budget will be reconciled whenever significant cost commitments are made; including allowances for future variations and minor scope changes. The total projected value of the package will be compared to the budget for that package. Possible requirements to access contingency funds will also be considered. The estimate to complete for each package will be updated and reported each month. Project Costs will be accrued against commitments. The project will recognize two types of costs: incurred (completed but unpaid) and paid (cash transfer initiated).

An AFE (Approved for Expenditure) procedure will be incorporated into the cost system. In this system each responsible management party from the Owner, engineer's or construction management side will be required to sign off on major expenditures (e.g., >$100,000) to signify their awareness that the expenditure will be made. The AFE format will contain the budget and description of the item to be purchased whether equipment, materials or services. Any overrun of the budget is reviewed at this level and before the purchase can be made.

24.2.9.3 Contingency Management

Approval of transfers to and from the project contingency account will be by the authority of YMI. The account will include provisions for escalation and foreign exchange fluctuations and will be managed on a global basis to fund approved cost increases within the project scope. Contingency funds will not be used for project scope changes.

Change management procedures will be used to transfer forecast funds from the contingency account to specific packages. When the revised budget (based on the definitive estimate at 90-95% engineering) exceeds the original baseline budget, a forecast change will be prepared and upon approval, will authorize transfer of funds from contingency to the package forming the current forecast. Alternatively, when the definitive estimate is less than the baseline budget, excess funds will be transferred to contingency, via approved forecast change. Only a Project Change Order can alter the baseline budget. The residual unallocated contingency fund will be updated, compared to plan and reported monthly.

24.2.9.4 Foreign Exchange

Most equipment and supplies will be purchased in Canada. Whenever feasible, imported items will be purchased from local agents, quoting Canadian dollar amounts to minimize exposure to exchange rate risks.

At the time a definitive estimate is prepared for a package, changes in the relevant exchange rate will be compared to those in effect on the FS estimate base date. In preparing the current budget for a work package, an element of the transfer to or from the contingency account may be used to compensate for exchange rate changes.

24.2.10 *Project Schedule*

The overall project execution period from start to mechanical completion is approximately 24 months; and assumes financing is in place to allow all phases of the project to proceed at their projected start times (Figure 24-2).

The project schedule will be continually revised and updated. More detailed schedules will be developed for each work package and will be used to revise the master baseline project schedule. The detailed package schedules will consider interfaces, resource constraints, delivery times, contract scopes, detailed engineering and procurement times as well as inputs from contractors. The resulting detailed project schedule will be used to manage performance. Deviations from detailed schedules will be rolled up monthly and used to measure impacts on the overall schedule.

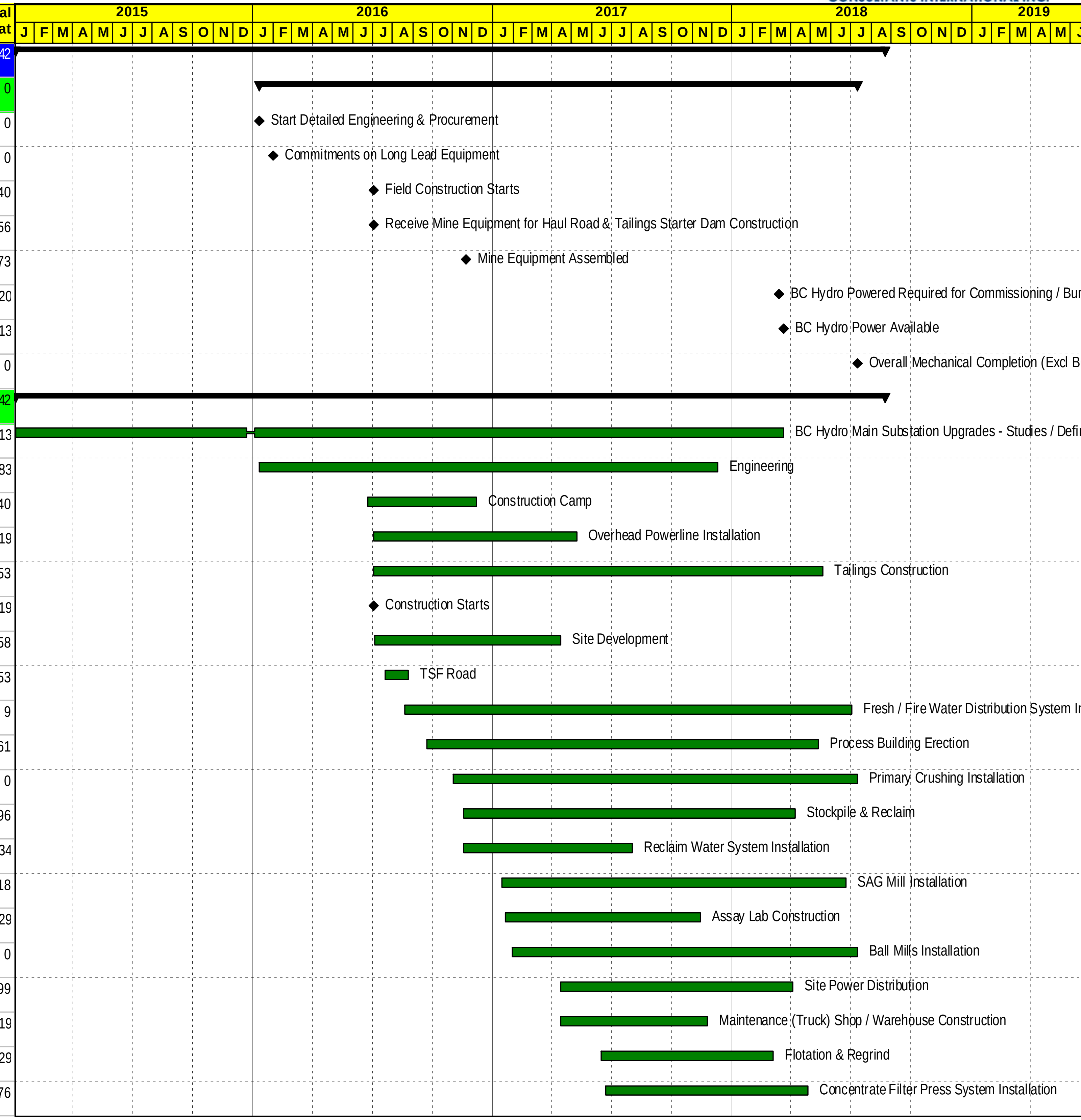
Construction manpower is based on a 70 hour construction contractor work week with crew rotations established by the contractors but which will, generally be 3 weeks on site and 1 week off. Manpower loading indicates a peak requirement for 600 construction workers on site.

Figure 24-2: Project Schedule

YELLOWHEAD MINING
HARPER CREEK - COPPER PROJECT

MERIT CONSULTANTS INTERNATIONAL INC.

Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
411- HARPER CREEK PROJECT SCHEDULE		1295	05-Jan-15	22-Aug-18	-42
PROJECT OVERVIEW		894	11-Jan-16	11-Jul-18	0
A4215	Start Detailed Engineering & Procurement	0	11-Jan-16*		0
A157	Commitments on Long Lead Equipment	0	01-Feb-16		0
A110	Field Construction Starts	0	04-Jul-16*		40
A4235	Receive Mine Equipment for Haul Road & Tailings Starter Dam Construction	0	04-Jul-16		56
A4245	Mine Equipment Assembled	0		20-Nov-16	73
A405	BC Hydro Powered Required for Commissioning / Bumping Motors	0		14-Mar-18	120
A3615	BC Hydro Power Available	0		20-Mar-18	113
A3620	Overall Mechanical Completion (Excl BC Hydro Work)	0		11-Jul-18	0
PROJECT SUMMARY		1295	05-Jan-15	22-Aug-18	-42
SUM-0005	BC Hydro Main Substation Upgrades - Studies / Definition Phases & Implementation Phase	1140	05-Jan-15	20-Mar-18	113
SUM-8600	Engineering	690	11-Jan-16	09-Dec-17	183
SUM-8300	Construction Camp	165	25-Jun-16	06-Dec-16	40
SUM-6000	Overhead Powerline Installation	300	04-Jul-16	08-May-17	419
SUM-7010	Tailings Construction	666	04-Jul-16	19-May-18	53
SUM-0010	Construction Starts	0	04-Jul-16		719
SUM-1000	Site Development	275	04-Jul-16	14-Apr-17	58
SUM-1010	TSF Road	35	21-Jul-16	24-Aug-16	53
SUM-2000	Fresh / Fire Water Distribution System Installation	664	19-Aug-16	02-Jul-18	9
SUM-0030	Process Building Erection	577	23-Sep-16	11-May-18	61
SUM-3000	Primary Crushing Installation	598	02-Nov-16	11-Jul-18	0
SUM-3110	Stockpile & Reclaim	487	17-Nov-16	06-Apr-18	96
SUM-7020	Reclaim Water System Installation	248	17-Nov-16	01-Aug-17	334
SUM-3410	SAG Mill Installation	515	15-Jan-17	23-Jun-18	18
SUM-5400	Assay Lab Construction	299	19-Jan-17	14-Nov-17	229
SUM-3440	Ball Mills Installation	518	30-Jan-17	11-Jul-18	0
SUM-6100	Site Power Distribution	344	15-Apr-17	03-Apr-18	99
SUM-5010	Maintenance (Truck) Shop / Warehouse Construction	224	15-Apr-17	24-Nov-17	219
SUM-3500	Flotation & Regrind	253	15-Jun-17	05-Mar-18	129
SUM-3900	Concentrate Filter Press System Installation	298	23-Jun-17	26-Apr-18	76



Start Date: 05-Jan-15
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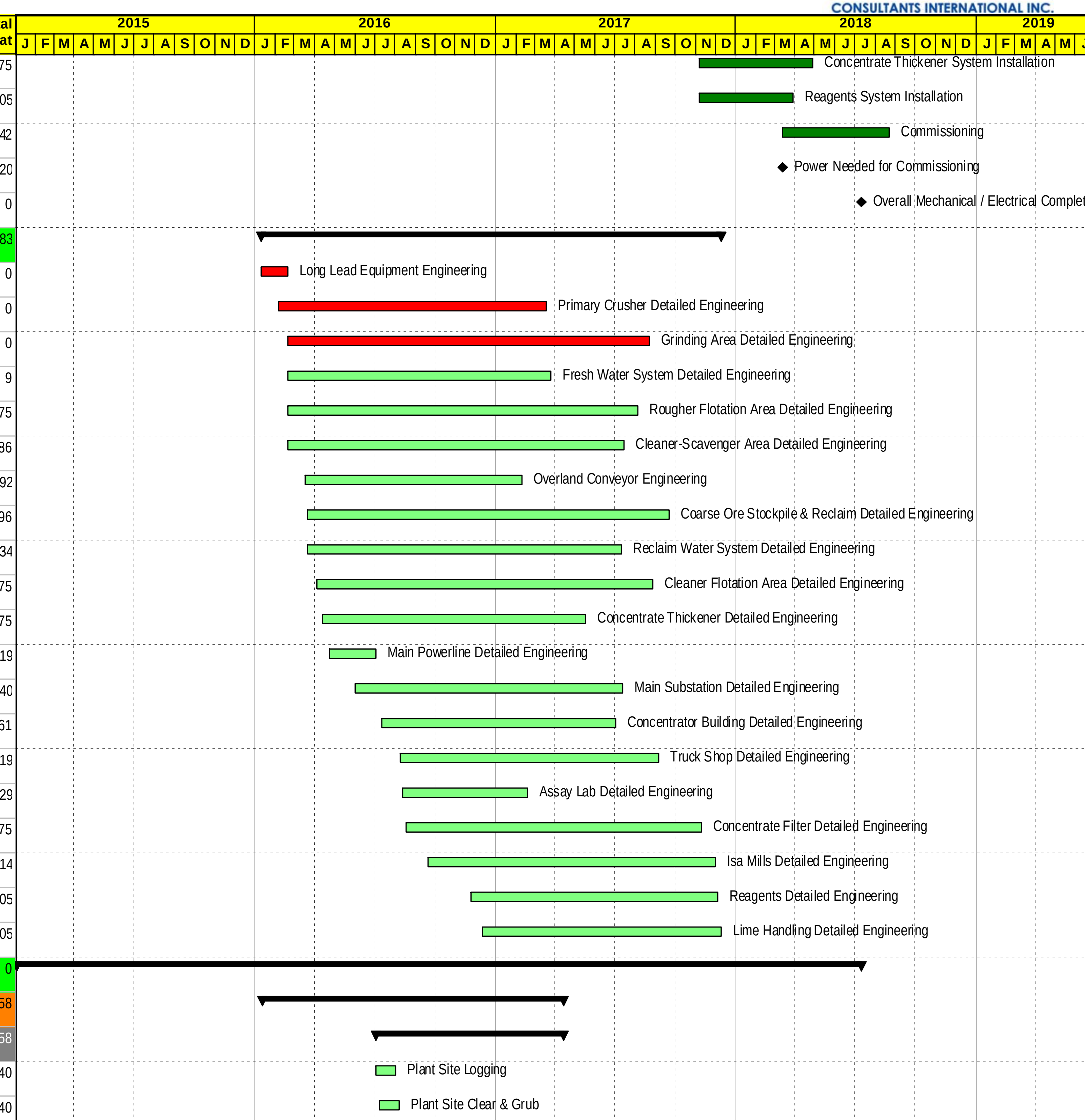
MERIT CONSULTANTS INTERNATIONAL INC.

Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
SUM-3910	Concentrate Thickener System Installation	163	06-Nov-17	28-Apr-18	75
SUM-3600	Reagents System Installation	133	06-Nov-17	28-Mar-18	105
SUM-8700	Commissioning	162	14-Mar-18	22-Aug-18	-42
SUM-0020	Power Needed for Commissioning	0		14-Mar-18	120
SUM-MEC	Overall Mechanical / Electrical Completion	0		11-Jul-18	0
ENGINEERING		690	11-Jan-16	09-Dec-17	183
A155	Long Lead Equipment Engineering	41	11-Jan-16	20-Feb-16	0
A495	Primary Crusher Detailed Engineering	398	06-Feb-16	18-Mar-17	0
A2360	Grinding Area Detailed Engineering	540	21-Feb-16	22-Aug-17	0
A1500	Fresh Water System Detailed Engineering	391	21-Feb-16	26-Mar-17	9
A1710	Rougher Flotation Area Detailed Engineering	522	21-Feb-16	04-Aug-17	75
A1810	Cleaner-Scavenger Area Detailed Engineering	501	21-Feb-16	14-Jul-17	86
A3240	Overland Conveyor Engineering	320	18-Mar-16	09-Feb-17	292
A1100	Coarse Ore Stockpile & Reclaim Detailed Engineering	540	22-Mar-16	21-Sep-17	96
A1550	Reclaim Water System Detailed Engineering	468	22-Mar-16	11-Jul-17	334
A3255	Cleaner Flotation Area Detailed Engineering	501	05-Apr-16	27-Aug-17	75
A2090	Concentrate Thickener Detailed Engineering	392	13-Apr-16	18-May-17	75
A410	Main Powerline Detailed Engineering	72	23-Apr-16	03-Jul-16	419
A480	Main Substation Detailed Engineering	398	02-Jun-16	13-Jul-17	240
A2930	Concentrator Building Detailed Engineering	348	11-Jul-16	03-Jul-17	61
A3130	Truck Shop Detailed Engineering	385	09-Aug-16	06-Sep-17	219
A3080	Assay Lab Detailed Engineering	181	13-Aug-16	18-Feb-17	229
A2000	Concentrate Filter Detailed Engineering	440	18-Aug-16	09-Nov-17	75
A3470	Isa Mills Detailed Engineering	428	20-Sep-16	30-Nov-17	114
A2180	Reagents Detailed Engineering	368	24-Nov-16	05-Dec-17	105
A2270	Lime Handling Detailed Engineering	353	13-Dec-16	09-Dec-17	105
CONSTRUCTION		1253	05-Jan-15	11-Jul-18	0
EARTHWORKS & SITE SERVICES		449	13-Jan-16	14-Apr-17	58
PLANT SITE EARTHWORKS		276	04-Jul-16	14-Apr-17	58
A370	Plant Site Logging	30	04-Jul-16	02-Aug-16	40
A365	Plant Site Clear & Grub	30	09-Jul-16	07-Aug-16	40



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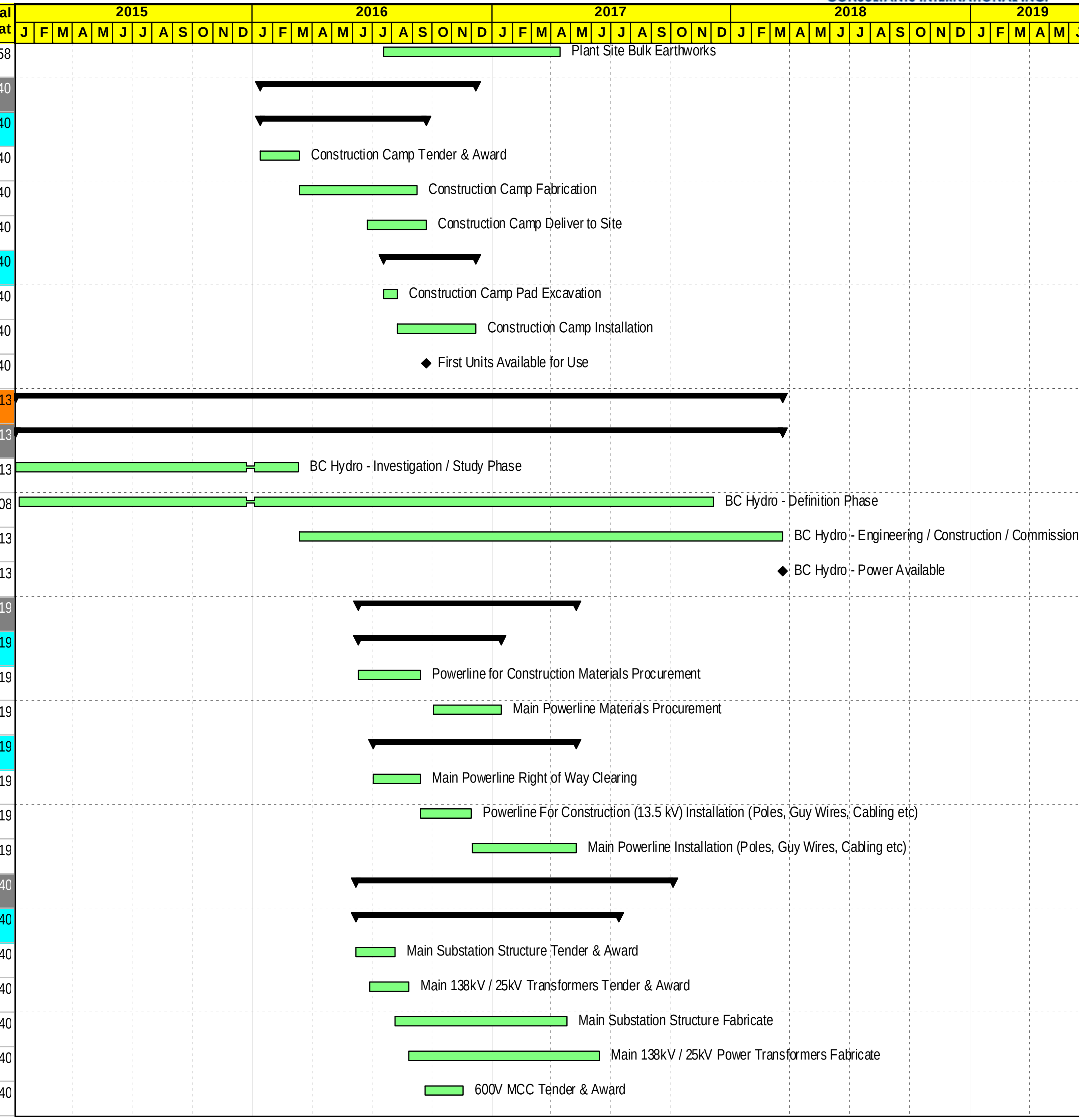
Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
A360	Plant Site Bulk Earthworks	261	19-Jul-16	14-Apr-17	58
	SITE SERVICES (CAMP, COMMUNICATIONS, ACCESS ROADS)	329	13-Jan-16	06-Dec-16	40
	SITE SERVICES (CAMP, COMMUNICATIONS, ACCESS ROADS) - PRO	254	13-Jan-16	22-Sep-16	40
A385	Construction Camp Tender & Award	60	13-Jan-16	12-Mar-16	40
A390	Construction Camp Fabrication	180	13-Mar-16	08-Sep-16	40
A395	Construction Camp Deliver to Site	90	25-Jun-16	22-Sep-16	40
	SITE SERVICES (CAMP, COMMUNICATIONS, ACCESS ROADS) - CON	141	19-Jul-16	06-Dec-16	40
A380	Construction Camp Pad Excavation	21	19-Jul-16	08-Aug-16	40
A400	Construction Camp Installation	120	09-Aug-16	06-Dec-16	40
A490	First Units Available for Use	0	23-Sep-16		40
	MAIN SUBSTATION & POWERLINE	1140	05-Jan-15	20-Mar-18	113
	BC HYDRO MAIN SUBSTATION UPGRADES	1140	05-Jan-15	20-Mar-18	113
A3550	BC Hydro - Investigation / Study Phase	420	05-Jan-15*	11-Mar-16	113
A3580	BC Hydro - Definition Phase	1040	10-Jan-15	05-Dec-17	208
A3610	BC Hydro - Engineering / Construction / Commissioning & Close-Out of Main Substation	720	12-Mar-16	20-Mar-18	113
A475	BC Hydro - Power Available	0		20-Mar-18	113
	OVERHEAD POWERLINE	324	10-Jun-16	08-May-17	419
	OVERHEAD POWERLINE - PROCUREMENT	210	10-Jun-16	14-Jan-17	419
A4250	Powerline for Construction Materials Procurement	96	10-Jun-16	13-Sep-16	419
A415	Main Powerline Materials Procurement	96	02-Oct-16	14-Jan-17	419
	OVERHEAD POWERLINE - CONSTRUCTION	300	04-Jul-16	08-May-17	419
A420	Main Powerline Right of Way Clearing	72	04-Jul-16	13-Sep-16	419
A4200	Powerline For Construction (13.5 kV) Installation (Poles, Guy Wires, Cabling etc)	78	14-Sep-16	30-Nov-16	419
A425	Main Powerline Installation (Poles, Guy Wires, Cabling etc)	150	01-Dec-16	08-May-17	419
	MAIN SUBSTATION	476	07-Jun-16	04-Oct-17	240
	MAIN SUBSTATION - PROCUREMENT	393	07-Jun-16	13-Jul-17	240
A3795	Main Substation Structure Tender & Award	60	07-Jun-16	05-Aug-16	240
A345	Main 138kV / 25kV Transformers Tender & Award	60	28-Jun-16	26-Aug-16	240
A3800	Main Substation Structure Fabricate	254	06-Aug-16	25-Apr-17	240
A350	Main 138kV / 25kV Power Transformers Fabricate	282	27-Aug-16	13-Jun-17	240
A3870	600V MCC Tender & Award	60	20-Sep-16	18-Nov-16	240



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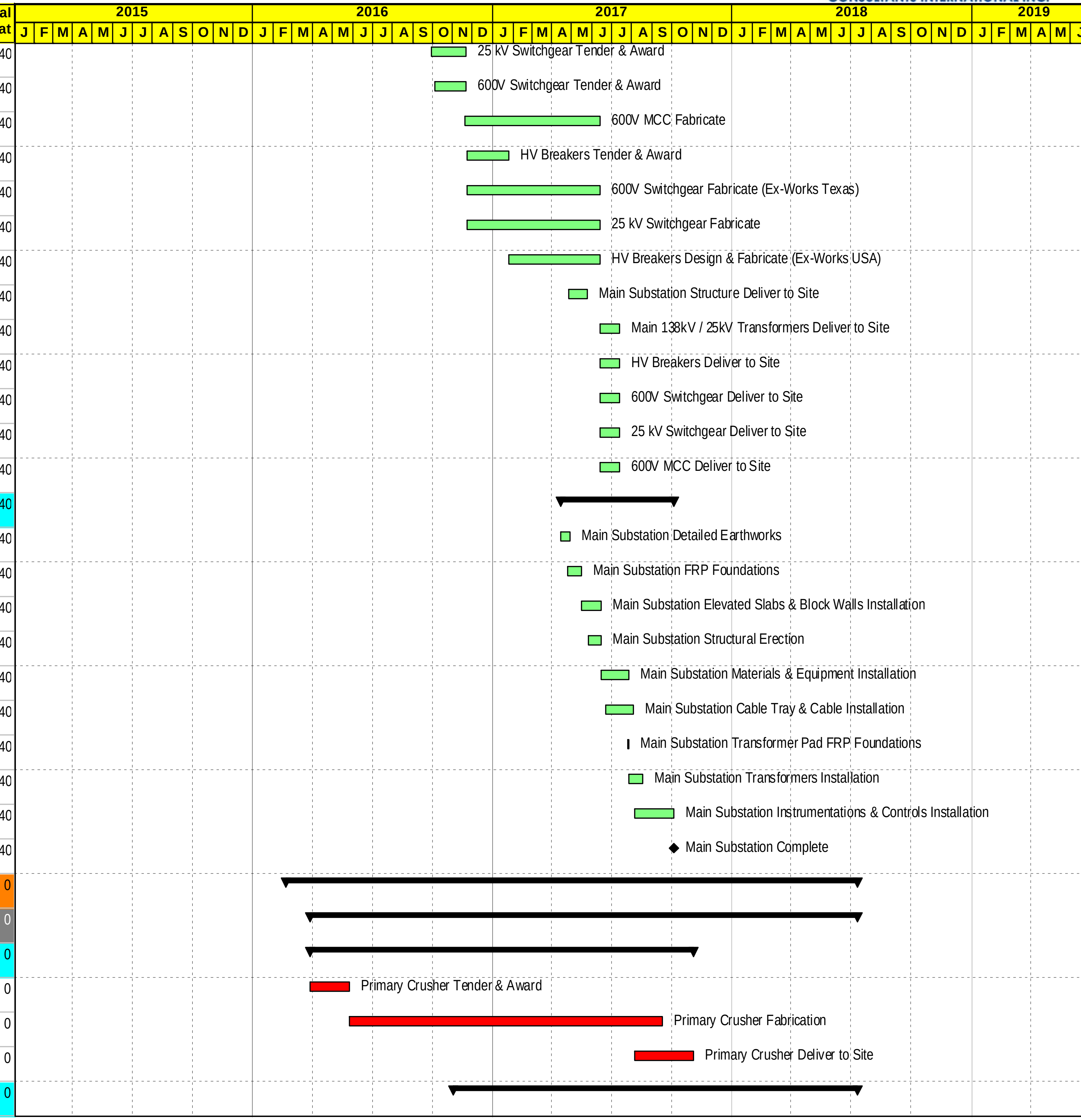
MERIT CONSULTANTS INTERNATIONAL INC.

Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
A3840	25 kV Switchgear Tender & Award	53	30-Sep-16	21-Nov-16	240
A3825	600V Switchgear Tender & Award	49	04-Oct-16	21-Nov-16	240
A3875	600V MCC Fabricate	198	19-Nov-16	13-Jun-17	240
A3810	HV Breakers Tender & Award	55	22-Nov-16	24-Jan-17	240
A3830	600V Switchgear Fabricate (Ex-Works Texas)	195	22-Nov-16	13-Jun-17	240
A3845	25 kV Switchgear Fabricate	195	22-Nov-16	13-Jun-17	240
A3815	HV Breakers Design & Fabricate (Ex-Works USA)	140	25-Jan-17	13-Jun-17	240
A3805	Main Substation Structure Deliver to Site	30	26-Apr-17	25-May-17	240
A355	Main 138kV / 25kV Transformers Deliver to Site	30	14-Jun-17	13-Jul-17	240
A3820	HV Breakers Deliver to Site	30	14-Jun-17	13-Jul-17	240
A3835	600V Switchgear Deliver to Site	30	14-Jun-17	13-Jul-17	240
A3850	25 kV Switchgear Deliver to Site	30	14-Jun-17	13-Jul-17	240
A3880	600V MCC Deliver to Site	30	14-Jun-17	13-Jul-17	240
MAIN SUBSTATION - CONSTRUCTION		173	15-Apr-17	04-Oct-17	240
A430	Main Substation Detailed Earthworks	14	15-Apr-17	28-Apr-17	240
A435	Main Substation FRP Foundations	22	25-Apr-17	16-May-17	240
A445	Main Substation Elevated Slabs & Block Walls Installation	30	17-May-17	15-Jun-17	240
A440	Main Substation Structural Erection	21	26-May-17	15-Jun-17	240
A450	Main Substation Materials & Equipment Installation	42	16-Jun-17	27-Jul-17	240
A460	Main Substation Cable Tray & Cable Installation	42	23-Jun-17	03-Aug-17	240
A4010	Main Substation Transformer Pad FRP Foundations	2	26-Jul-17	27-Jul-17	240
A455	Main Substation Transformers Installation	21	28-Jul-17	17-Aug-17	240
A465	Main Substation Instrumentations & Controls Installation	60	06-Aug-17	04-Oct-17	240
A470	Main Substation Complete	0		04-Oct-17	240
CRUSHING / CONVEYING / ORE STORAGE		853	21-Feb-16	11-Jul-18	0
PRIMARY CRUSHING		816	29-Mar-16	11-Jul-18	0
PRIMARY CRUSHING - PROCUREMENT		576	29-Mar-16	03-Nov-17	0
A180	Primary Crusher Tender & Award	60	29-Mar-16	27-May-16	0
A185	Primary Crusher Fabrication	468	28-May-16	16-Sep-17	0
A190	Primary Crusher Deliver to Site	90	06-Aug-17	03-Nov-17	0
PRIMARY CRUSHING - CONSTRUCTION		598	02-Nov-16	11-Jul-18	0



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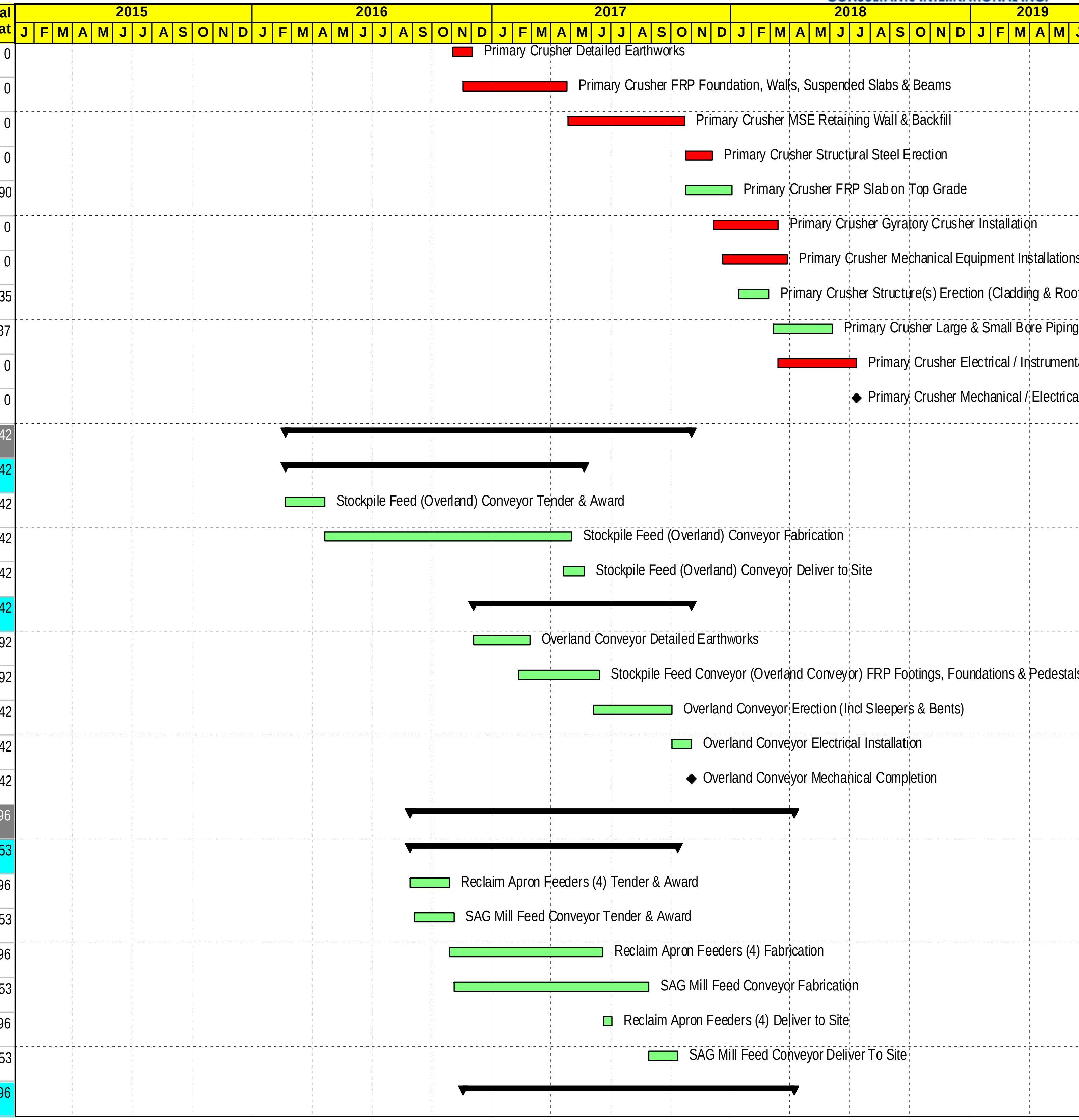
MERIT CONSULTANTS INTERNATIONAL INC.

Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
A500	Primary Crusher Detailed Earthworks	31	02-Nov-16	02-Dec-16	0
A505	Primary Crusher FRP Foundation, Walls, Suspended Slabs & Beams	150	18-Nov-16	25-Apr-17	0
A515	Primary Crusher MSE Retaining Wall & Backfill	180	26-Apr-17	22-Oct-17	0
A520	Primary Crusher Structural Steel Erection	42	23-Oct-17	03-Dec-17	0
A517	Primary Crusher FRP Slab on Top Grade	62	23-Oct-17	02-Jan-18	190
A525	Primary Crusher Gyratory Crusher Installation	90	04-Dec-17	13-Mar-18	0
A530	Primary Crusher Mechanical Equipment Installations	90	18-Dec-17	27-Mar-18	0
A527	Primary Crusher Structure(s) Erection (Cladding & Roofing, Control Room)	45	13-Jan-18	26-Feb-18	135
A540	Primary Crusher Large & Small Bore Piping Installations	90	07-Mar-18	04-Jun-18	37
A535	Primary Crusher Electrical / Instrumentation Installations	120	14-Mar-18	11-Jul-18	0
A550	Primary Crusher Mechanical / Electrical Completion	0		11-Jul-18	0
PRIMARY CRUSHING CONVEYANCE		611	21-Feb-16	02-Nov-17	242
PRIMARY CRUSHING CONVEYANCE - PROCUREMENT		447	21-Feb-16	22-May-17	242
A200	Stockpile Feed (Overland) Conveyor Tender & Award	60	21-Feb-16	20-Apr-16	242
A195	Stockpile Feed (Overland) Conveyor Fabrication	368	21-Apr-16	02-May-17	242
A205	Stockpile Feed (Overland) Conveyor Deliver to Site	33	19-Apr-17	22-May-17	242
PRIMARY CRUSHING CONVEYANCE - CONSTRUCTION		325	03-Dec-16	02-Nov-17	242
A1180	Overland Conveyor Detailed Earthworks	78	03-Dec-16	27-Feb-17	292
A1190	Stockpile Feed Conveyor (Overland Conveyor) FRP Footings, Foundations & Pedestals	123	10-Feb-17	13-Jun-17	292
A1200	Overland Conveyor Erection (Incl Sleepers & Bents)	120	05-Jun-17	03-Oct-17	242
A1210	Overland Conveyor Electrical Installation	30	03-Oct-17	02-Nov-17	242
A1220	Overland Conveyor Mechanical Completion	0		02-Nov-17	242
CRUSHED ORE STOCKPILE & RECLAIM / SAG MILL FEED CONVEYOF		567	29-Aug-16	06-Apr-18	96
CRUSHED ORE STOCKPILE & RECLAIM / SAG MILL FEED CONVEY(		400	29-Aug-16	11-Oct-17	153
A210	Reclaim Apron Feeders (4) Tender & Award	60	29-Aug-16	27-Oct-16	96
A225	SAG Mill Feed Conveyor Tender & Award	60	05-Sep-16	03-Nov-16	153
A215	Reclaim Apron Feeders (4) Fabrication	226	28-Oct-16	19-Jun-17	96
A230	SAG Mill Feed Conveyor Fabrication	288	04-Nov-16	27-Aug-17	153
A220	Reclaim Apron Feeders (4) Deliver to Site	14	20-Jun-17	03-Jul-17	96
A235	SAG Mill Feed Conveyor Deliver To Site	45	28-Aug-17	11-Oct-17	153
CRUSHED ORE STOCKPILE & RECLAIM / SAG MILL FEED CONVEY(		487	17-Nov-16	06-Apr-18	96



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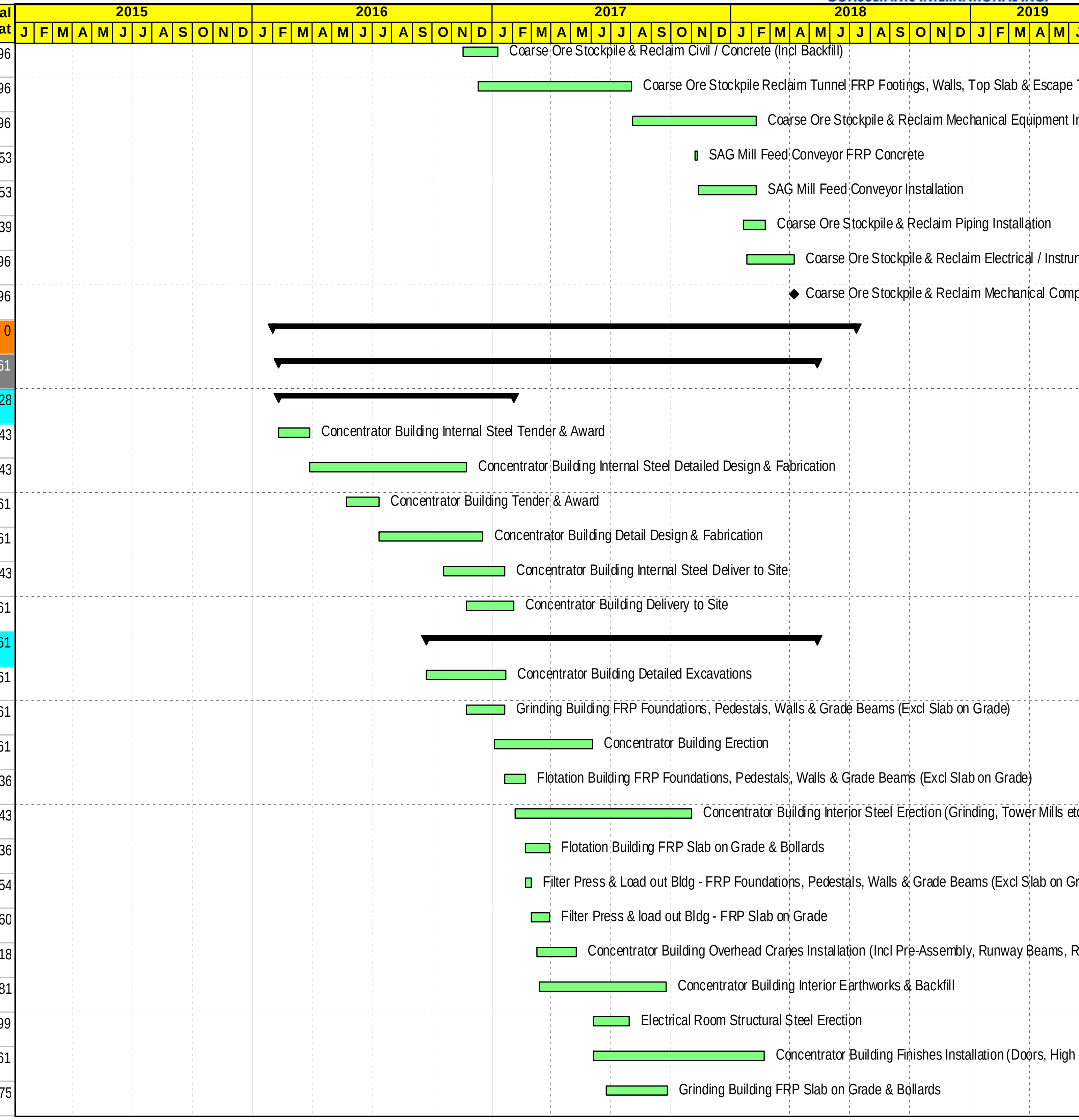
MERIT CONSULTANTS INTERNATIONAL INC.

Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
A1120	Coarse Ore Stockpile & Reclaim Civil / Concrete (Incl Backfill)	45	17-Nov-16	09-Jan-17	96
A4025	Coarse Ore Stockpile Reclaim Tunnel FRP Footings, Walls, Top Slab & Escape Tunnel	226	11-Dec-16	02-Aug-17	96
A1130	Coarse Ore Stockpile & Reclaim Mechanical Equipment Installation	180	03-Aug-17	08-Feb-18	96
A4020	SAG Mill Feed Conveyor FRP Concrete	5	06-Nov-17	10-Nov-17	153
A3235	SAG Mill Feed Conveyor Installation	80	11-Nov-17	08-Feb-18	153
A1150	Coarse Ore Stockpile & Reclaim Piping Installation	35	19-Jan-18	22-Feb-18	139
A1140	Coarse Ore Stockpile & Reclaim Electrical / Instrumentation Installation	71	25-Jan-18	06-Apr-18	96
A1160	Coarse Ore Stockpile & Reclaim Mechanical Completion	0		06-Apr-18	96
PROCESSING		873	01-Feb-16	11-Jul-18	0
PROCESS BUILDING		803	10-Feb-16	11-May-18	61
PROCESS BUILDING - BUILDING & INTERNAL STEEL PROCUREMEN		351	10-Feb-16	03-Feb-17	228
A3720	Concentrator Building Internal Steel Tender & Award	48	10-Feb-16	28-Mar-16	243
A2950	Concentrator Building Internal Steel Detailed Design & Fabrication	240	29-Mar-16	23-Nov-16	243
A3680	Concentrator Building Tender & Award	48	24-May-16	11-Jul-16	61
A2940	Concentrator Building Detail Design & Fabrication	160	11-Jul-16	18-Dec-16	61
A3725	Concentrator Building Internal Steel Deliver to Site	84	19-Oct-16	19-Jan-17	243
A3685	Concentrator Building Delivery to Site	64	23-Nov-16	03-Feb-17	61
PROCESS BUILDING & ANCILLARIES		577	23-Sep-16	11-May-18	61
A3250	Concentrator Building Detailed Excavations	112	23-Sep-16	21-Jan-17	61
A3935	Grinding Building FRP Foundations, Pedestals, Walls & Grade Beams (Excl Slab on Grade)	50	22-Nov-16	19-Jan-17	61
A3010	Concentrator Building Erection	150	05-Jan-17	03-Jun-17	61
A3945	Flotation Building FRP Foundations, Pedestals, Walls & Grade Beams (Excl Slab on Grade)	33	19-Jan-17	21-Feb-17	136
A3305	Concentrator Building Interior Steel Erection (Grinding, Tower Mills etc)	270	04-Feb-17	31-Oct-17	243
A3950	Flotation Building FRP Slab on Grade & Bollards	37	21-Feb-17	30-Mar-17	136
A3955	Filter Press & Load out Bldg - FRP Foundations, Pedestals, Walls & Grade Beams (Excl Slab on Grade)	9	21-Feb-17	02-Mar-17	154
A3960	Filter Press & load out Bldg - FRP Slab on Grade	27	02-Mar-17	29-Mar-17	160
A3020	Concentrator Building Overhead Cranes Installation (Incl Pre-Assembly, Runway Beams, Rail Install)	60	11-Mar-17	10-May-17	418
A2980	Concentrator Building Interior Earthworks & Backfill	194	14-Mar-17	23-Sep-17	281
A1660	Electrical Room Structural Steel Erection	55	04-Jun-17	29-Jul-17	99
A3030	Concentrator Building Finishes Installation (Doors, High Bay Lighting, Roof Ice Armour, Trim etc)	251	05-Jun-17	20-Feb-18	61
A3940	Grinding Building FRP Slab on Grade & Bollards	94	24-Jun-17	25-Sep-17	275



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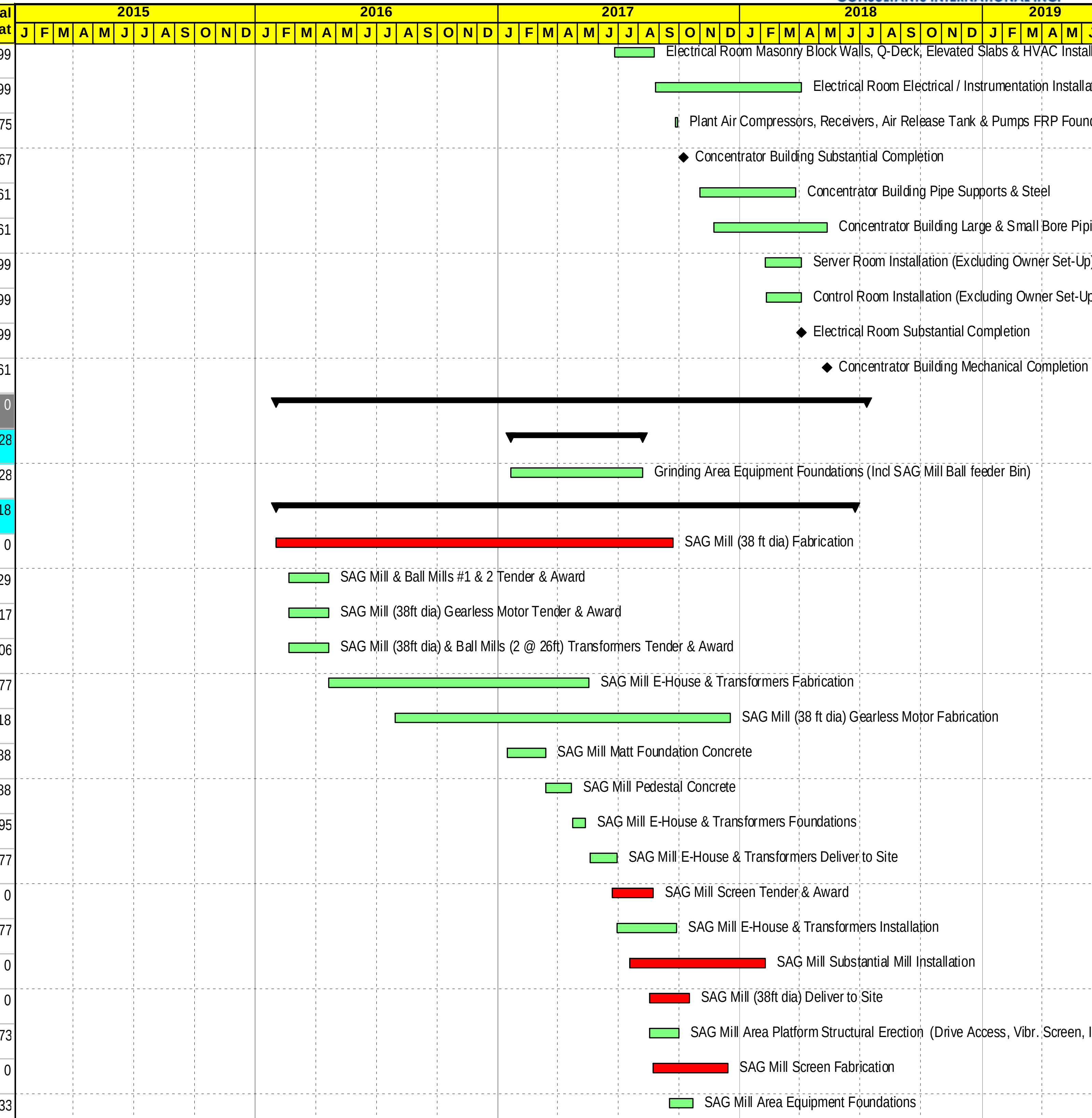
YELLOWHEAD MINING INC.

MERIT CONSULTANTS INTERNATIONAL INC.

Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
A1800	Electrical Room Masonry Block Walls, Q-Deck, Elevated Slabs & HVAC Installation	62	25-Jun-17	25-Aug-17	99
A1690	Electrical Room Electrical / Instrumentation Installations (Incl Grounding, Cables & Cable Tray)	211	26-Aug-17	03-Apr-18	99
A3985	Plant Air Compressors, Receivers, Air Release Tank & Pumps FRP Foundations	4	26-Sep-17	29-Sep-17	275
A3675	Concentrator Building Substantial Completion	0		08-Oct-17	267
A3040	Concentrator Building Pipe Supports & Steel	135	02-Nov-17	26-Mar-18	61
A3050	Concentrator Building Large & Small Bore Piping Installation	160	23-Nov-17	11-May-18	61
A1680	Server Room Installation (Excluding Owner Set-Up)	55	08-Feb-18	03-Apr-18	99
A1670	Control Room Installation (Excluding Owner Set-Up)	54	09-Feb-18	03-Apr-18	99
A1700	Electrical Room Substantial Completion	0		03-Apr-18	99
A3070	Concentrator Building Mechanical Completion	0		11-May-18	61
GRINDING & CLASSIFICATION		873	01-Feb-16	11-Jul-18	0
GRINDING AREA INSTALLATIONS		200	19-Jan-17	07-Aug-17	328
A2510	Grinding Area Equipment Foundations (Incl SAG Mill Ball feeder Bin)	200	19-Jan-17	07-Aug-17	328
SAG MILL (38 ft by 22 ft) SYSTEM		855	01-Feb-16	23-Jun-18	18
A160	SAG Mill (38 ft dia) Fabrication	600	01-Feb-16	22-Sep-17	0
A135	SAG Mill & Ball Mills #1 & 2 Tender & Award	60	21-Feb-16	20-Apr-16	29
A140	SAG Mill (38ft dia) Gearless Motor Tender & Award	60	21-Feb-16	20-Apr-16	117
A330	SAG Mill (38ft dia) & Ball Mills (2 @ 26ft) Transformers Tender & Award	60	21-Feb-16	20-Apr-16	206
A335	SAG Mill E-House & Transformers Fabrication	384	21-Apr-16	18-May-17	277
A170	SAG Mill (38 ft dia) Gearless Motor Fabrication	498	29-Jul-16	17-Dec-17	18
A2440	SAG Mill Matt Foundation Concrete	58	15-Jan-17	13-Mar-17	88
A2470	SAG Mill Pedestal Concrete	40	14-Mar-17	22-Apr-17	88
A2530	SAG Mill E-House & Transformers Foundations	20	23-Apr-17*	12-May-17	195
A340	SAG Mill E-House & Transformers Deliver to Site	42	19-May-17	29-Jun-17	277
A240	SAG Mill Screen Tender & Award	61	23-Jun-17	22-Aug-17	0
A3360	SAG Mill E-House & Transformers Installation	90	30-Jun-17	27-Sep-17	277
A2650	SAG Mill Substantial Mill Installation	194	19-Jul-17	07-Feb-18	0
A165	SAG Mill (38ft dia) Deliver to Site	60	17-Aug-17	16-Oct-17	0
A2640	SAG Mill Area Platform Structural Erection (Drive Access, Vibr. Screen, Infeed)	45	18-Aug-17	01-Oct-17	273
A245	SAG Mill Screen Fabrication	114	23-Aug-17	14-Dec-17	0
A2520	SAG Mill Area Equipment Foundations	35	17-Sep-17	21-Oct-17	133



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Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
A2670	SAG Mill Large & Small Bore Piping Installation	120	22-Oct-17	28-Feb-18	133
A250	SAG Mill Screen Deliver to Site	45	15-Dec-17	07-Feb-18	0
A175	SAG Mill (38ft dia) Gearless Motor Deliver to Site	42	18-Dec-17	07-Feb-18	18
A2690	SAG Mill Electrical / Instrumentation Installation	151	24-Jan-18	23-Jun-18	18
A2655	SAG Mill Wrap Around Motor Installation	115	08-Feb-18	02-Jun-18	18
A4255	SAG Mill Remainder of Mill Installation	14	03-Jun-18	16-Jun-18	25
A2700	SAG Mill Mechanical Completion	0		23-Jun-18	18
BALL MILL #1 (25 ft by 38 ft) SYSTEM		837	21-Feb-16	25-Jun-18	16
A3885	Ball Mills #1 & 2 Mills Dual Pinion Drives Tender & Award	60	21-Feb-16	20-Apr-16	191
A3385	Ball Mill #1 E-House & Transformers Fabrication	384	06-May-16	02-Jun-17	192
A3320	Ball Mill #1(25 ft dia) Fabrication (FLS)	464	20-Jun-16	05-Oct-17	29
A3330	Ball Mill #1 (25 ft dia) Dual Pinion Drives (2) Fabrication (Supplier Estimate - 60 wks)	408	13-Oct-16	03-Dec-17	16
A2450	Ball Mill #1 Matt Foundation Concrete	59	30-Jan-17	29-Mar-17	85
A2490	Ball Mill #1 Pedestal Concrete	27	30-Mar-17	25-Apr-17	114
A2540	Ball Mill #1 E-House & Transformers Foundations	80	26-Apr-17	14-Jul-17	192
A3375	Ball Mill #1 E-House & Transformers Deliver to Site	42	03-Jun-17	14-Jul-17	192
A3365	Ball Mill #1 E-House & Transformers Installation	160	15-Jul-17	21-Dec-17	192
A2570	Ball Mill #1 Mill Install	180	18-Aug-17	23-Feb-18	0
A2580	Ball Mill #1 Mechanical Equipment Installations	180	05-Sep-17	13-Mar-18	120
A3310	Ball Mill #1 (25 ft dia) Deliver to Site	42	06-Sep-17	17-Oct-17	29
A2495	Ball Mill #1 Area Equipment Foundations	35	13-Oct-17	16-Nov-17	46
A2590	Ball Mill #1 Large & Small Bore Piping Installation	121	06-Nov-17	16-Mar-18	117
A3340	Ball Mill #1 (25 ft dia) Dual Pinion Drives (2) Deliver to Site	42	04-Dec-17	24-Jan-18	16
A3350	Ball Mill #1 Dual Pinion Motors Install	131	25-Jan-18	04-Jun-18	16
A2610	Ball Mill #1 Electrical / Instrumentation Installation	151	26-Jan-18	25-Jun-18	16
A2560	Ball Mill #1 Area Platform Erection - Operating Floor	46	24-Feb-18	10-Apr-18	92
A2620	Ball Mill #1 Mechanical Completion	0		25-Jun-18	16
BALL MILL #2 (25 ft by 38 ft) SYSTEM		793	21-Apr-16	11-Jul-18	0
A3335	Ball Mill #2 (25ft dia) Dual Pinion Drives (2) Fabrication (Supplier Estimate - 60 wks)	408	21-Apr-16	11-Jun-17	191
A3325	Ball Mill #2(25ft dia) Fabrication (FLS)	464	20-Jun-16	05-Oct-17	29
A3390	Ball Mill #2 E-House & Transformers Fabrication	384	04-Jul-16	31-Jul-17	132

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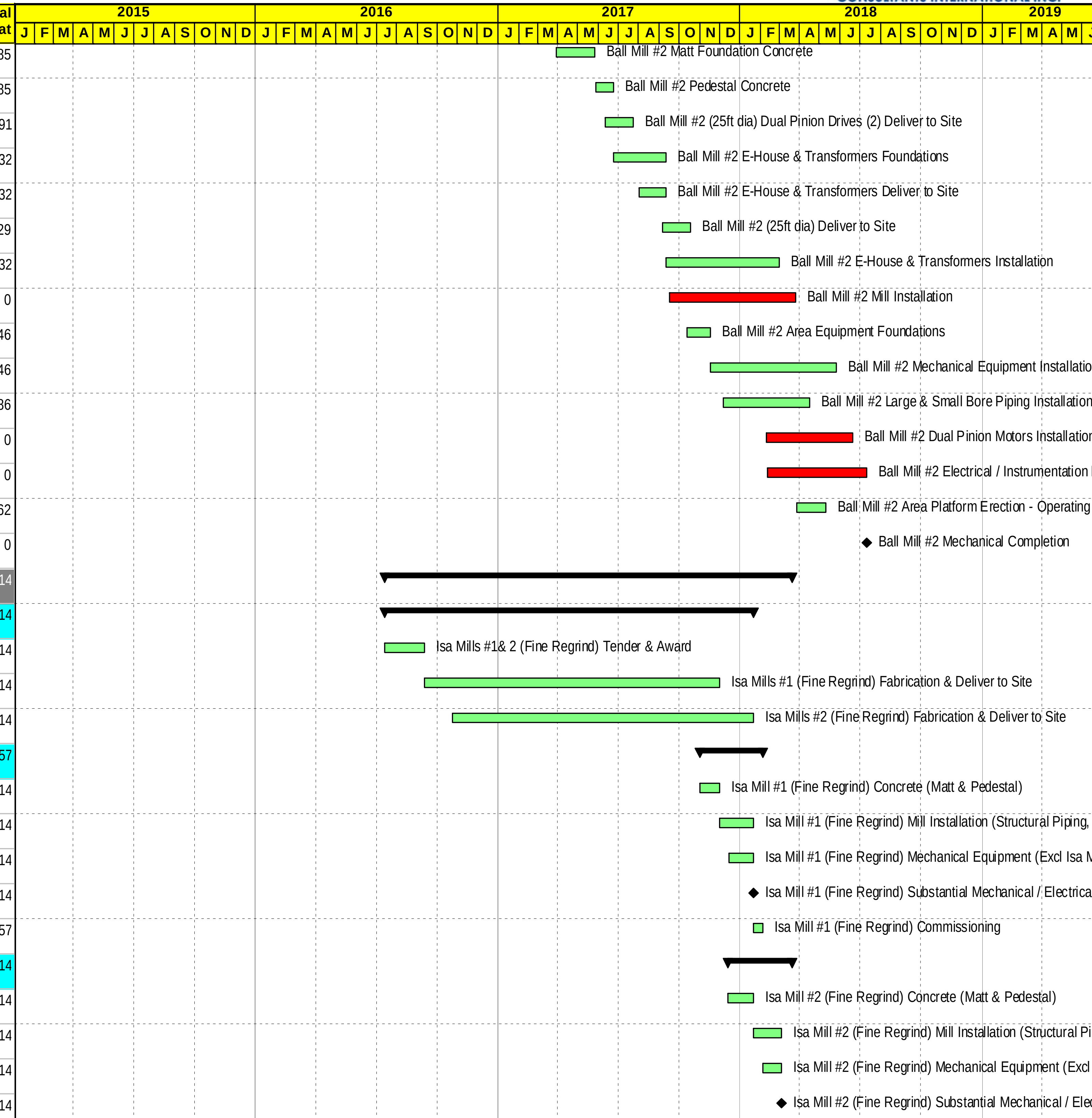
A2460	Ball Mill #2 Matt Foundation Concrete	59	30-Mar-17	27-May-17	85
A2500	Ball Mill #2 Pedestal Concrete	27	28-May-17	23-Jun-17	85
A3345	Ball Mill #2 (25ft dia) Dual Pinion Drives (2) Deliver to Site	42	12-Jun-17	23-Jul-17	191
A2550	Ball Mill #2 E-House & Transformers Foundations	80	24-Jun-17	11-Sep-17	132
A3380	Ball Mill #2 E-House & Transformers Deliver to Site	42	01-Aug-17	11-Sep-17	132
A3315	Ball Mill #2 (25ft dia) Deliver to Site	42	06-Sep-17	17-Oct-17	29
A3370	Ball Mill #2 E-House & Transformers Installation	161	12-Sep-17	01-Mar-18	132
A2730	Ball Mill #2 Mill Installation	181	17-Sep-17	26-Mar-18	0
A3405	Ball Mill #2 Area Equipment Foundations	35	13-Oct-17	16-Nov-17	46
A2740	Ball Mill #2 Mechanical Equipment Installations	181	17-Nov-17	26-May-18	46
A2750	Ball Mill #2 Large & Small Bore Piping Installation	121	07-Dec-17	16-Apr-18	86
A3355	Ball Mill #2 Dual Pinion Motors Installation	131	10-Feb-18	20-Jun-18	0
A2770	Ball Mill #2 Electrical / Instrumentation Installation	151	11-Feb-18	11-Jul-18	0
A2720	Ball Mill #2 Area Platform Erection - Operating Floor	45	27-Mar-18	10-May-18	62
A2780	Ball Mill #2 Mechanical Completion	0		11-Jul-18	0
	REGRIND - ISAMILL #1 & 2 (FINE REGRIND)	595	14-Jul-16	20-Mar-18	114
	REGRIND - ISAMILL #1 & 2 (FINE REGRIND) - PROCUREMENT	538	14-Jul-16	21-Jan-18	114
A3785	Isa Mills #1& 2 (Fine Regrind) Tender & Award	60	14-Jul-16	11-Sep-16	114
A3635	Isa Mills #1 (Fine Regrind) Fabrication & Deliver to Site	436	12-Sep-16	30-Nov-17	114
A3780	Isa Mills #2 (Fine Regrind) Fabrication & Deliver to Site	436	24-Oct-16	21-Jan-18	114
	REGRIND - ISAMILL #1 (FINE REGRIND) - CONSTRUCTION	85	02-Nov-17	05-Feb-18	157
A3645	Isa Mill #1 (Fine Regrind) Concrete (Matt & Pedestal)	29	02-Nov-17	30-Nov-17	114
A3650	Isa Mill #1 (Fine Regrind) Mill Installation (Structural Piping, Mechanical, Piping & Electrical)	42	01-Dec-17	21-Jan-18	114
A3655	Isa Mill #1 (Fine Regrind) Mechanical Equipment (Excl Isa Mills) Installations	28	15-Dec-17	21-Jan-18	114
A3670	Isa Mill #1 (Fine Regrind) Substantial Mechanical / Electrical Completion	0		21-Jan-18	114
A3660	Isa Mill #1 (Fine Regrind) Commissioning	14	22-Jan-18	05-Feb-18	157
	REGRIND - ISAMILL #2 (FINE REGRIND) - CONSTRUCTION	86	14-Dec-17	20-Mar-18	114
A3690	Isa Mill #2 (Fine Regrind) Concrete (Matt & Pedestal)	29	14-Dec-17	21-Jan-18	114
A3695	Isa Mill #2 (Fine Regrind) Mill Installation (Structural Piping, Mechanical, Piping & Electrical)	42	22-Jan-18	04-Mar-18	114
A3700	Isa Mill #2 (Fine Regrind) Mechanical Equipment (Excl Isa Mills) Installations	28	05-Feb-18	05-Mar-18	114
A3715	Isa Mill #2 (Fine Regrind) Substantial Mechanical / Electrical Completion	0		05-Mar-18	114



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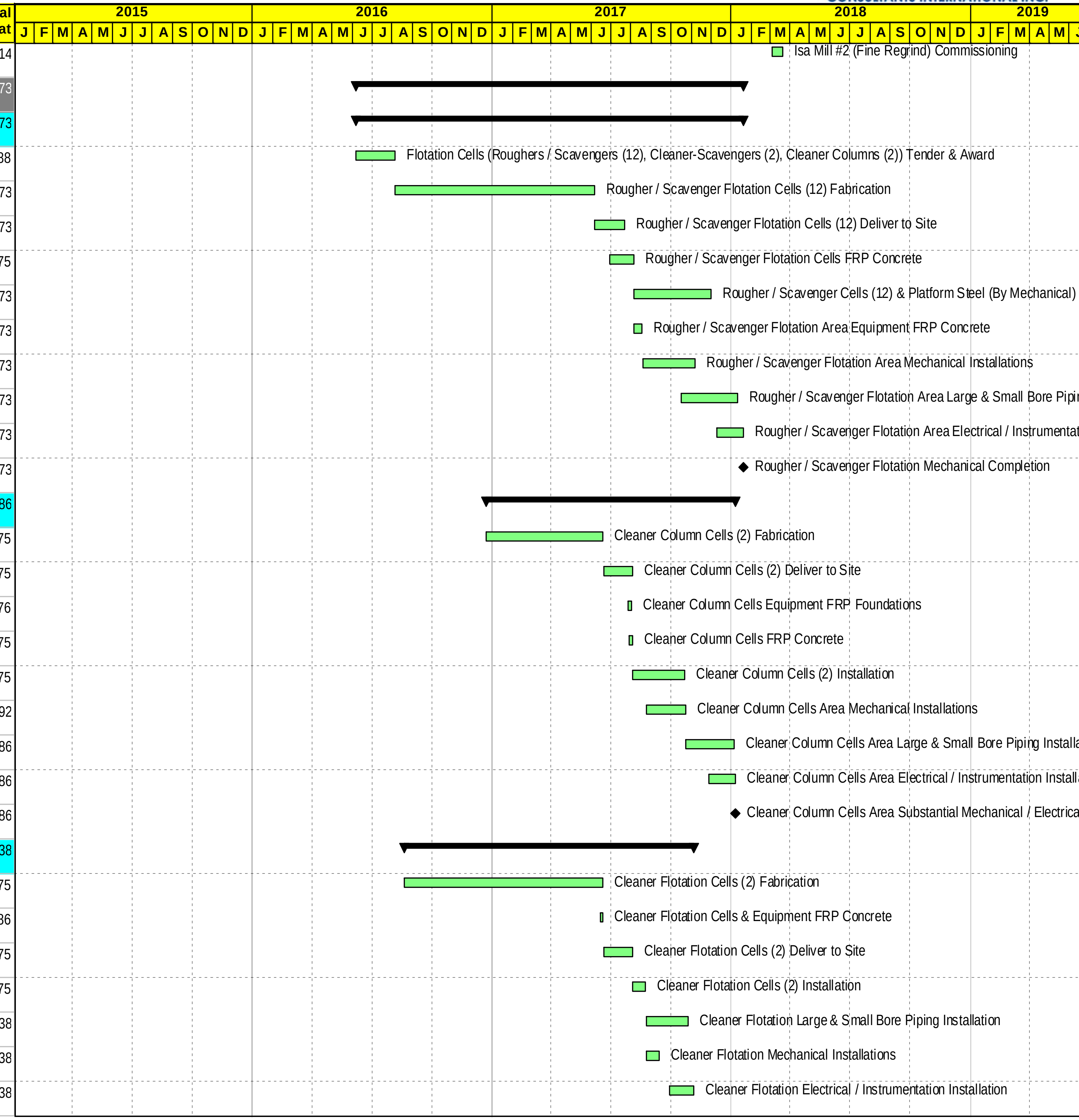
MERIT CONSULTANTS INTERNATIONAL INC.

Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
A4210	Isa Mill #2 (Fine Regrind) Commissioning	15	05-Mar-18	20-Mar-18	114
FLOTATION SYSTEM		573	07-Jun-16	20-Jan-18	173
ROUGHER / SCAVENGER FLOTATION		573	07-Jun-16	20-Jan-18	173
A270	Flotation Cells (Roughers / Scavengers (12), Cleaner-Scavengers (2), Cleaner Columns (2)) Tender & Award	60	07-Jun-16	05-Aug-16	88
A275	Rougher / Scavenger Flotation Cells (12) Fabrication	296	06-Aug-16	06-Jun-17	173
A280	Rougher / Scavenger Flotation Cells (12) Deliver to Site	45	07-Jun-17	21-Jul-17	173
A1730	Rougher / Scavenger Flotation Cells FRP Concrete	36	30-Jun-17	04-Aug-17	75
A1740	Rougher / Scavenger Cells (12) & Platform Steel (By Mechanical) Erection	119	05-Aug-17	01-Dec-17	173
A3965	Rougher / Scavenger Flotation Area Equipment FRP Concrete	14	05-Aug-17	18-Aug-17	173
A1750	Rougher / Scavenger Flotation Area Mechanical Installations	79	19-Aug-17	06-Nov-17	173
A1760	Rougher / Scavenger Flotation Area Large & Small Bore Piping Installation	75	17-Oct-17	10-Jan-18	173
A1780	Rougher / Scavenger Flotation Area Electrical / Instrumentation Installation	31	10-Dec-17	20-Jan-18	173
A1790	Rougher / Scavenger Flotation Mechanical Completion	0		20-Jan-18	173
CLEANER COLUMNS		361	23-Dec-16	06-Jan-18	186
A3425	Cleaner Column Cells (2) Fabrication	170	23-Dec-16	19-Jun-17	75
A3410	Cleaner Column Cells (2) Deliver to Site	45	20-Jun-17	03-Aug-17	75
A3970	Cleaner Column Cells Equipment FRP Foundations	6	28-Jul-17	02-Aug-17	76
A3265	Cleaner Column Cells FRP Concrete	6	29-Jul-17	03-Aug-17	75
A3270	Cleaner Column Cells (2) Installation	80	04-Aug-17	22-Oct-17	75
A3280	Cleaner Column Cells Area Mechanical Installations	60	25-Aug-17	23-Oct-17	192
A3285	Cleaner Column Cells Area Large & Small Bore Piping Installation	65	23-Oct-17	05-Jan-18	186
A3295	Cleaner Column Cells Area Electrical / Instrumentation Installation	31	27-Nov-17	06-Jan-18	186
A3300	Cleaner Column Cells Area Substantial Mechanical / Electrical Completion	0		06-Jan-18	186
CLEANER FLOTATION		435	19-Aug-16	05-Nov-17	238
A3435	Cleaner Flotation Cells (2) Fabrication	296	19-Aug-16	19-Jun-17	75
A1830	Cleaner Flotation Cells & Equipment FRP Concrete	4	15-Jun-17*	18-Jun-17	86
A3420	Cleaner Flotation Cells (2) Deliver to Site	45	20-Jun-17	03-Aug-17	75
A1840	Cleaner Flotation Cells (2) Installation	20	04-Aug-17	23-Aug-17	75
A1870	Cleaner Flotation Large & Small Bore Piping Installation	65	24-Aug-17	27-Oct-17	238
A1860	Cleaner Flotation Mechanical Installations	20	25-Aug-17	13-Sep-17	238
A1890	Cleaner Flotation Electrical / Instrumentation Installation	39	28-Sep-17	05-Nov-17	238



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A1900	Cleaner Flotation Substantial Mechanical / Electrical Completion	0		05-Nov-17	238
CONCENTRATE HANDLING & LOADOUT (THICKENER, FILTER PRESS		576	10-Sep-16	28-Apr-18	75
CONCENTRATE FILTER PRESS		478	16-Dec-16	26-Apr-18	76
A300	Copper Concentrate Filter Press Tender & Award	48	16-Dec-16	10-Feb-17	76
A305	Copper Concentrate Filter Press Fabrication	210	11-Feb-17	08-Sep-17	76
A2020	Concentrate Filter FRP Concrete	54	23-Jun-17	15-Aug-17	75
A3975	Concentrate Stock Tank, Filtrate Tank & Equipment FRP Concrete	8	16-Aug-17	23-Aug-17	75
A2030	Concentrate Filter Stock Tank Erection	90	24-Aug-17	21-Nov-17	75
A310	Copper Concentrate Filter Press Deliver to Site	14	09-Sep-17	22-Sep-17	76
A3475	Concentrate Filter Installation	21	23-Oct-17	12-Nov-17	76
A2040	Concentrate Filter Structural / Mechanical Installation	85	23-Oct-17	25-Jan-18	76
A2050	Concentrate Filter Large & Small Bore Piping Installation	131	22-Nov-17	11-Apr-18	76
A2070	Concentrate Filter Electrical / Instrumentation Installation	76	10-Feb-18	26-Apr-18	76
A2080	Concentrate Filter Mechanical Completion	0		26-Apr-18	76
CONCENTRATE THICKENER		576	10-Sep-16	28-Apr-18	75
A315	Concentrate Thickener Tank Tender & Award	60	10-Sep-16	08-Nov-16	75
A320	Concentrate Thickener Tank Fabrication	288	09-Nov-16	01-Sep-17	75
A325	Concentrate Thickener Tank Deliver to Site	60	02-Sep-17	31-Oct-17	75
A2110	Concentrate Thickener Concrete	9	06-Nov-17	14-Nov-17	75
A2120	Concentrate Thickener Tank Erection	116	15-Nov-17	20-Mar-18	75
A2130	Concentrate Thickener Structural / Mechanical Installation	46	15-Nov-17	09-Jan-18	115
A2140	Concentrate Thickener Large & Small Bore Piping Installation	83	15-Dec-17	18-Mar-18	116
A2160	Concentrate Thickener Electrical / Instrumentation	68	19-Feb-18	28-Apr-18	75
A2170	Concentrate Thickener Substantial Mechanical / Electrical Completion	0		28-Apr-18	75
REAGENTS & LIME HANDLING AREA		133	06-Nov-17	28-Mar-18	105
REAGENTS		125	06-Nov-17	20-Mar-18	113
A2200	Reagents Concrete	9	06-Nov-17	14-Nov-17	105
A2220	Reagents Mechanical / Tanks Installation	116	15-Nov-17	20-Mar-18	105
A2210	Reagents Structural Steel Erection	21	27-Nov-17	17-Dec-17	132
A2240	Reagents Large & Small Bore Piping Installation	71	11-Dec-17	01-Mar-18	132
A2250	Reagents Electrical / Instrumentation Installation	50	08-Jan-18	26-Feb-18	135

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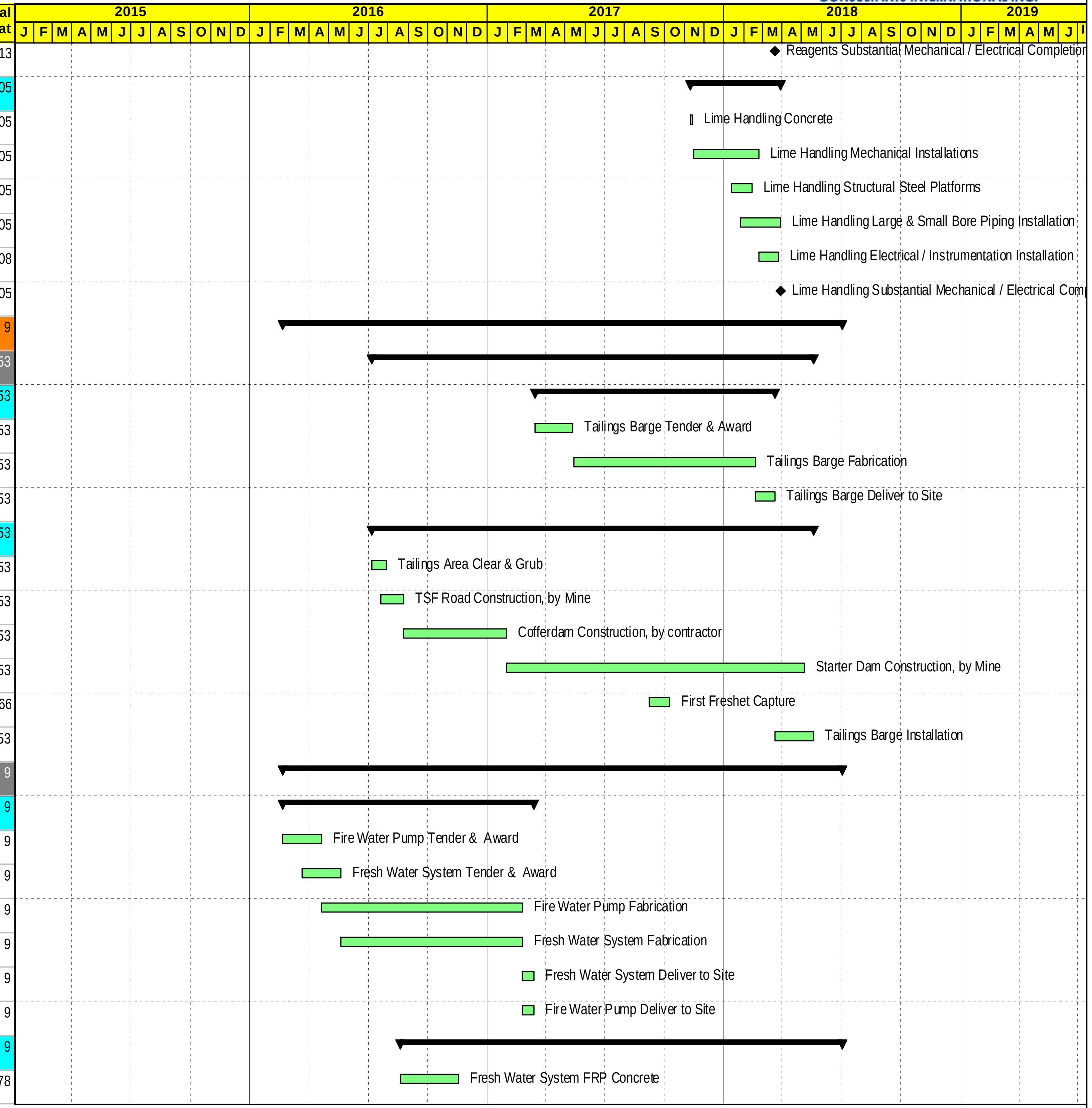
A2260	Reagents Substantial Mechanical / Electrical Completion	0		20-Mar-18	113
LIME HANDLING		129	10-Nov-17	28-Mar-18	105
A2290	Lime Handling Concrete	5	10-Nov-17	14-Nov-17	105
A2310	Lime Handling Mechanical Installations	91	15-Nov-17	23-Feb-18	105
A2300	Lime Handling Structural Steel Platforms	31	13-Jan-18	12-Feb-18	105
A2320	Lime Handling Large & Small Bore Piping Installation	61	27-Jan-18	28-Mar-18	105
A2340	Lime Handling Electrical / Instrumentation Installation	30	24-Feb-18	25-Mar-18	108
A2350	Lime Handling Substantial Mechanical / Electrical Completion	0		28-Mar-18	105
WATER & TAILINGS SYSTEMS		844	21-Feb-16	02-Jul-18	9
TAILINGS		664	06-Jul-16	19-May-18	53
TAILINGS - PROCUREMENT		361	15-Mar-17	20-Mar-18	53
A3905	Tailings Barge Tender & Award	60	15-Mar-17	13-May-17	53
A3910	Tailings Barge Fabrication	271	14-May-17	18-Feb-18	53
A3915	Tailings Barge Deliver to Site	30	19-Feb-18	20-Mar-18	53
TAILINGS STARTER DAM & HAUL ROAD		664	06-Jul-16	19-May-18	53
A100	Tailings Area Clear & Grub	25	06-Jul-16	30-Jul-16	53
A115	TSF Road Construction, by Mine	35	21-Jul-16	24-Aug-16	53
A125	Cofferdam Construction, by contractor	150	25-Aug-16	30-Jan-17	53
A4265	Starter Dam Construction, by Mine	450	31-Jan-17	05-May-18	53
A560	First Freshet Capture	31	08-Sep-17*	09-Oct-17	266
A3900	Tailings Barge Installation	60	21-Mar-18	19-May-18	53
FRESH / FIRE WATER DISTRIBUTION		844	21-Feb-16	02-Jul-18	9
FRESH / FIRE WATER DISTRIBUTION - PROCUREMENT		378	21-Feb-16	13-Mar-17	9
A3920	Fire Water Pump Tender & Award	60	21-Feb-16	20-Apr-16	9
A570	Fresh Water System Tender & Award	60	22-Mar-16	20-May-16	9
A3925	Fire Water Pump Fabrication	300	21-Apr-16	23-Feb-17	9
A3770	Fresh Water System Fabrication	270	21-May-16	23-Feb-17	9
A3775	Fresh Water System Deliver to Site	18	24-Feb-17	13-Mar-17	9
A3930	Fire Water Pump Deliver to Site	18	24-Feb-17	13-Mar-17	9
FRESH / FIRE WATER DISTRIBUTION - CONSTRUCTION		664	19-Aug-16	02-Jul-18	9
A3245	Fresh Water System FRP Concrete	91	19-Aug-16*	17-Nov-16	78



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A3980	Fresh Water System Tank Erection	82	17-Nov-16	16-Feb-17	78
A1530	Fresh Water System Mechanical / Piping Installation	452	14-Mar-17	18-Jun-18	9
A4030	Fresh Water System Electrical / Instrumentation Installation	90	04-Apr-18	02-Jul-18	9
A1540	Fresh Water Available	0		02-Jul-18	9
	RECLAIM WATER	489	22-Mar-16	01-Aug-17	334
	RECLAIM WATER - PROCUREMENT	378	22-Mar-16	12-Apr-17	334
A510	Reclaim Water System Tender & Award	60	22-Mar-16	20-May-16	334
A3750	Reclaim Water System Fabrication	288	21-May-16	13-Mar-17	334
A3755	Reclaim Water System Deliver to Site	30	14-Mar-17	12-Apr-17	334
	RECLAIM WATER - CONSTRUCTION	248	17-Nov-16	01-Aug-17	334
A1580	Process Water Pond Bulk Earthworks & Grading	90	17-Nov-16*	24-Feb-17	376
A3995	Process Water Pond FRP Bottom & Grade Slab, Walls	10	14-Feb-17	24-Feb-17	376
A3890	Reclaim Water System Process Water Pond Lining Installation???	60	24-Feb-17	25-Apr-17	376
A1590	Reclaim Water System Mechanical / Piping Installation	90	13-Apr-17	11-Jul-17	334
A4000	Process Water Pond Pump House FRP Foundation	3	24-Apr-17*	27-Apr-17	381
A4005	Process Water Pond Pump House Erection	14	27-Apr-17	11-May-17	381
A3895	Reclaim Water System Process Water Pond Pump Installation	14	11-May-17	25-May-17	381
A4035	Reclaim Water System Electrical / Instrumentation Installation	61	02-Jun-17	01-Aug-17	334
A1600	Reclaim Water System Complete	0		01-Aug-17	334
	MAINTENANCE SHOP / WAREHOUSE	393	19-Oct-16	24-Nov-17	219
	MAINTENANCE SHOP / WAREHOUSE - PROCUREMENT	330	19-Oct-16	22-Sep-17	238
A3150	Truck Shop Building Tender & Award	48	19-Oct-16	05-Dec-16	219
A3140	Truck Shop Mechanical Equipment Tender & Award	36	04-Dec-16	17-Jan-17	238
A3760	Truck Shop Building Detailed Design & Fabrication	148	06-Dec-16	11-May-17	219
A3740	Truck Shop Mechanical Equipment Fabrication	224	18-Jan-17	29-Aug-17	238
A3765	Truck Shop Building Deliver to Site	48	18-Apr-17	04-Jun-17	219
A3745	Truck Shop Mechanical Equipment Deliver to Site	48	06-Aug-17	22-Sep-17	238
	MAINTENANCE SHOP / WAREHOUSE - CONSTRUCTION	224	15-Apr-17	24-Nov-17	219
A3160	Truck Shop FRP Footings, Pedestals, Walls & Grade Beams	51	15-Apr-17	04-Jun-17	219
A3170	Truck Shop Building Erection (Incl Cladding & Roofing)	66	05-Jun-17	09-Aug-17	219
A3190	Truck Shop Overhead Crane Installation	20	26-Jul-17	14-Aug-17	321

Start Date: 05-Jan-15
Finish Date: 22-Aug-18
Data Date: 01-Sep-13
Run Date: 30-Jul-14

Summary — Remaining Work
Milestone — Critical Work
Actual Work — Activities Summary

PROJECT DEVELOPMENT SCHEDULE

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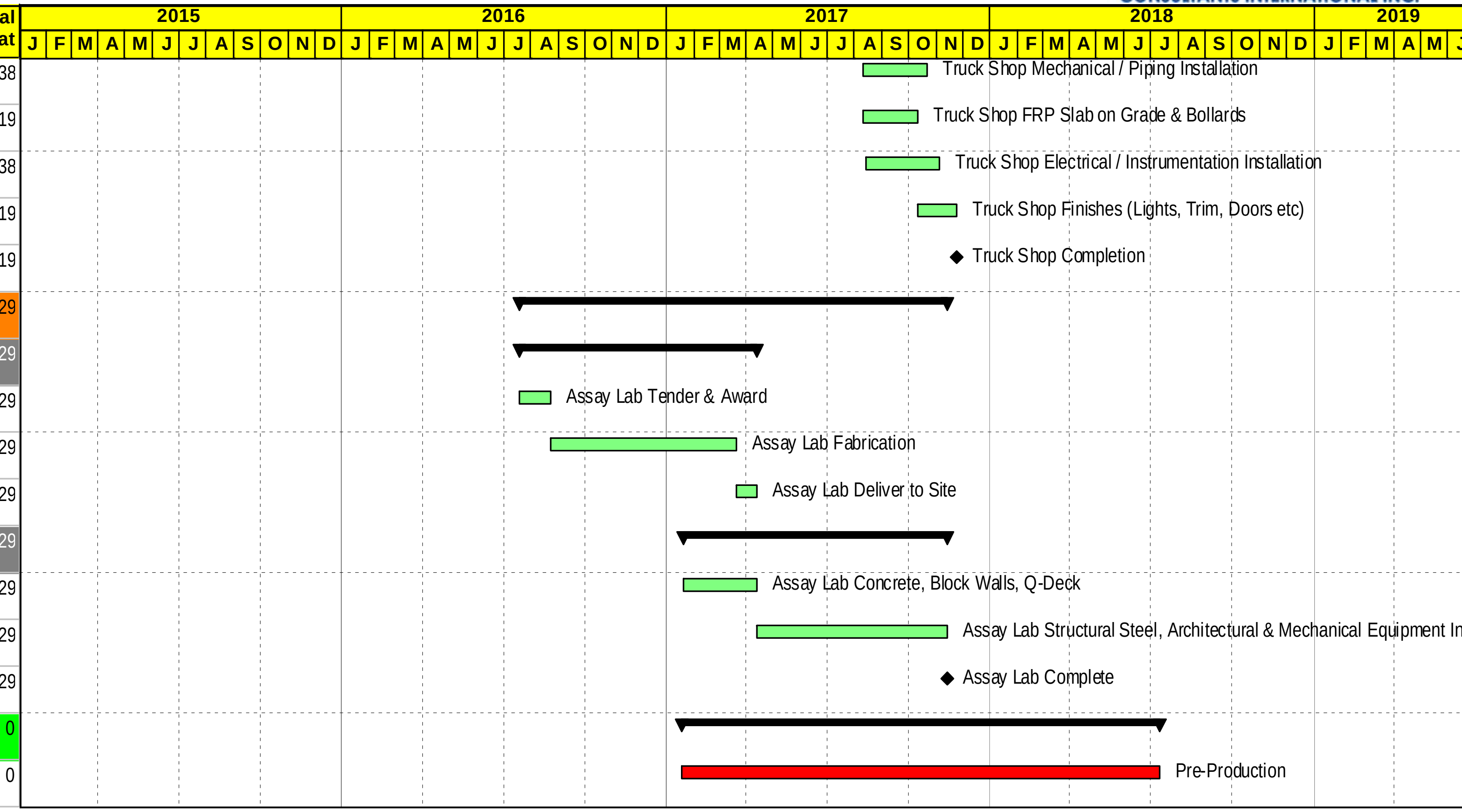
YELLOWHEAD MINING
HARPER CREEK - COPPER PROJECT

Activity ID	Activity Name	Dur (d)	Start	Finish	Total Float
A3200	Truck Shop Mechanical / Piping Installation	74	10-Aug-17	22-Oct-17	238
A3990	Truck Shop FRP Slab on Grade & Bollards	62	10-Aug-17	10-Oct-17	219
A3210	Truck Shop Electrical / Instrumentation Installation	84	14-Aug-17	05-Nov-17	238
A3180	Truck Shop Finishes (Lights, Trim, Doors etc)	45	11-Oct-17	24-Nov-17	219
A3230	Truck Shop Completion	0		24-Nov-17	219
ASSAY LAB		475	18-Jul-16	14-Nov-17	229
ASSAY LAB - PROCUREMENT		260	18-Jul-16	13-Apr-17	229
A3090	Assay Lab Tender & Award	36	18-Jul-16	23-Aug-16	229
A3730	Assay Lab Fabrication	200	23-Aug-16	20-Mar-17	229
A3735	Assay Lab Deliver to Site	24	20-Mar-17	13-Apr-17	229
ASSAY LAB - CONSTRUCTION		299	19-Jan-17	14-Nov-17	229
A3100	Assay Lab Concrete, Block Walls, Q-Deck	84	19-Jan-17	13-Apr-17	229
A3110	Assay Lab Structural Steel, Architectural & Mechanical Equipment Installation (Excl. Owner Supply & Install)	215	13-Apr-17	14-Nov-17	229
A3120	Assay Lab Complete	0		14-Nov-17	229
MINING		529	19-Jan-17	11-Jul-18	0
A4260	Pre-Production	529	19-Jan-17	11-Jul-18	0



Start Date: 05-Jan-15
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Summary | Remaining Work | Milestone | Critical Work | Actual Work | Activities Summary

PROJECT DEVELOPMENT SCHEDULE

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24.2.10.1 Project Milestones

The project schedule reflects a traditional approach to project execution, with the field construction commencing after engineering tasks are well advanced in order to accommodate long lead times for the delivery of major equipment. Project milestones do not change once the project is authorized. Progress is measured and reported compared to the fixed milestones, even if the schedule is adjusted (Table 24-4).

Table 24-4: Key Project Milestones

Key Project Milestones	
Milestone	**Date**
Detail Engineering Starts	11 Jan 2016
Mines Act Permit Issued; Project Released for Construction	26 Feb 2016
Construction Starts	04 Jul 2016
Mills Delivered	17 Aug 2017
BC Hydro Power Required	14 Mar 2018
Mechanical Completion	11 Jul 2018

24.2.11 *Construction Permitting*

The Owner's team is responsible for all regulatory components of project approval and will be the primary liaison with all regulatory bodies for permanent plant and equipment permitting. The EPCM will provide engineering support to YMI to assist with permitting and YMI will develop a schedule and communicate status regularly to the EPCM.

Individual contractors will be responsible for providing permits required for their construction operations.

24.2.12 *Quality Control & Quality Assurance*

24.2.12.1 Manufactured Items

Equipment suppliers will be required to provide details of their quality control systems at the time of bidding. Quality systems will be considered in the bid comparison and vendor selection process. The level of detail and the level of involvement of the project team will depend on the complexity and cost of the equipment involved. As a minimum, inspection and or test certificates complying with vendor programs and proving compliance with specifications will be required before shipment to the site is permitted. With larger and more complex equipment (e.g., mills and their motors, main transformers, standby generator) one or more third party inspections will be arranged at the point of manufacture before shipment is permitted.

24.2.12.2 Construction

Contractors will be required to nominate their quality control plans at the time of bidding. The field engineer, who has overall responsibility for quality, will review contractor quality plans and direct revisions where necessary. Approved quality control plans will become part of contract documents. Contractors will be responsible for their own quality control. The EPCM (on site engineers and construction superintendents) will ensure that contractor quality plans are carried out, i.e., quality assurance. In the event of non-conformance, the engineering manager will be responsible for directing and approving remedial measures or permitting a variance.

24.2.13 *Risk Management*

The EPCM project manager will ensure that the list of possible risks is periodically reviewed and that the agreed risk reduction plans are implemented through detailed design, construction and commissioning. Some risks are operational risks and are the responsibility of the Owner's operational team. All risks will be entered in the Risk Register together with agreed mitigation plans and person responsible. Whenever possible, risks will be addressed through design.

An engineering risk identification and reduction session for each work package will be held and recorded when engineering is 30 to 50% complete. Representatives of the Owner, the design engineering team, consultant (where applicable) and the project manager will be present. Results will be added to the risk register.

A construction safety risk session will be held and recorded for each work package before work begins. Representatives of the work force, the contractors' supervision and project management will be present. Results will be recorded in the risk register.

24.2.14 *Health, Safety, Environment*

24.2.14.1 Safety Management Plan

The Project Manager, via the Site Safety Manager, is responsible for ensuring that a safety management plan is in place before construction begins. The plan will meet the requirements of applicable safety, health, environmental, hygiene and emergency response legislation.

Key features of the plan are:

- Contractor plans must at minimum comply with the requirements set in the project safety plan;
- Contractors will be held accountable for enforcing their plans, including discipline;
- Workers will be primarily responsible for their own safety and will be provided information and education to ensure such;
- Site specific induction will be given before the first working shift, i.e., upon arrival;
- Zero tolerance;
- Contractors will be responsible for providing properly trained personnel to perform the work. The safety manager will inspect proof of training, licenses and qualifications and maintain records submitted during the contract negotiation stage and before a contractor is permitted to mobilize; and
- The EPCM PM or safety manager will have the right to eject for cause any contractor's employee from the site and demand a replacement at the contractor's expense.

Contractors will be required to conduct monthly safety inspections, hold monthly crew safety meetings and daily toolbox meetings to reinforce the safety message. Records of these meetings and inspections will be given to the Project Manager via the Site Safety Manager.

24.2.14.2 Safety Inductions

All personnel arriving on site will attend site induction upon arrival. Induction will include an overview of the project, site rules and emphasize the projects' commitment to safety and environmental protection.

24.2.14.3 Health and Hygiene

The project will provide a first aid facility, showers, washing and toilet facilities to meet regulations. Temporary sanitary facilities (chemical toilets) will be provided close to work places until the permanent facilities are operational.

General standards for personal protective equipment (PPE) will be established in the safety management plan. Contractors will be responsible for ensuring that their people have all the required PPE. YMI will issue PPE to its employees.

The Safety Manager will ensure standards are met for monitoring and compliance for noise, diesel emissions, dust, heat and radiation (since nuclear instruments will be used).

EPCM procurement and contractors will ensure that material safety data sheets (MSDS) accompany all products coming on site. The safety office will ensure that MSDS are properly maintained and kept available according to the Workplace Hazardous Material Information System (WHMIS).

24.2.14.4 Emergency Preparedness and Response

Trained and certified personnel will provide first aid for the project. First aid facilities will be provided at the temporary construction office until the permanent facility in the service complex is available. Ambulance service will be available from the mine site. A mine rescue room will be established and rescue equipment purchased.

Surface fire protection requirements will be met initially by fire extinguishers. Hot work permits will be part of the program. As a matter of priority, the fire hydrants system will be installed and made operational. Grass fires are a particular hazard in the hot dry summer season and is characteristic of the area. Safety superintendent will be required to institute specific precautions.

An emergency response plan for environmental spills will be established upon project start according to established environmental practices.

24.2.14.5 Environmental Management

The Owner is responsible for providing a comprehensive Management Plan for items such as:

- Air quality;
- Water;
- Waste;
- Waste rock and tailings;
- Ecosystems and vegetation;
- Wildlife;

- Aquatic resources;
- ML-ARD prevention and mitigation;
- Surface subsidence;
- Dust control;
- Archaeology; and
- Noise.

The EPCM will ensure that these plans are enforced, periodically reviewed and updated by the project team. Project organization includes an environmental coordinator and a technician (both part time) reporting to the Manager of Safety and Environment.

24.2.15 *COMMUNICATIONS & REPORTS*

Progress review meetings and planning sessions will be held regularly. The CM will provide weekly “flash” reports and comprehensive monthly reports. Flash reports will contain brief summaries of progress compared to plan, plan for the week just started, and highlight issues. Weekly reports will include statistics for safety and on site work hours compared to plan. “Monthly” reports will include the most precise statement of cost and status available and will include an executive summary and sections on health and safety, environment, engineering, procurement, construction, schedule, costs, owner’s responsibilities (preparation of operations) and issues being attended to.

24.2.16 *TEMPORARY & SITE FACILITIES*

There will be several temporary components used for the construction and pre-production periods that will be removed once plant operations commence and the construction team leave site. Some temporary works will be incorporated into the permanent facility. Overall, the project development commitment is to return the environment, areas not disturbed for the permanent plant, back to a condition that is environmentally acceptable.

Once construction begins, site offices and communication for the project development team will be set up close to the work. Contractors will bring their support facilities with them, and the construction camp will be installed. These areas will be cleared, grubbed and the topsoil removed and stockpiled, to be used later for reclamation. Support services will be established for power, water and sewage.

The 600 man construction camp and associated site construction offices and lay down areas will be located within walking distance of the process plant. Workers will be bused to the more remote locations such as the crusher and tailings locations. The camp will gradually be expanded from approximately 100 person capacity at the start of construction to accommodate the peak force sometime in the second year of construction.

Power for the camp and offices will be provided from the current spare capacity on the grid. A temporary power line will be constructed from the Vavenby area to site along the main access road. Diesel generators will be used for some of the remote areas for construction, such as the tailings dam. Potable water will be supplied from local wells. Sewage will be gravity fed to holding tanks that will be periodically emptied by local community services. The solid waste from the offices and camps will be incinerated and solid, non-flammable materials will be dumped into pits for later burial.

Telus is the telephone service provider for Vavenby and the surrounding area. Cell phone service became operational in February 2012. If required, a repeater station and receiver will be installed at the project site to boost cell phone reliability and internet service. In addition, two way radios will be used to supplement site communication.

A substantial lay down area will be needed for capital equipment and materials. It is expected that these materials will be unloaded and stored at the Vavenby load out area. Break-bulk loads will be delivered to the project site thereby minimizing the need for an all purpose built lay down area at the plant site.

Construction fuel tanks will be installed in suitable lined containments at site. Fuel will be delivered as required from the nearby community and a site fuel bowser will fuel remote day tanks.

The transportation, storage, dispensing and use of fuels at the site will be conducted in compliance with all relevant government laws and regulations. Prior to transporting or positioning fuel tanks at the project site, the fuel supplier(s) will be required to provide a copy of their fuel spill contingency plan. At a minimum, re-fuelling operations will incorporate the following:

- Fuel tanks to be self-dyked or be positioned over an impervious mat surrounded by an impervious dyke. The tanks are to be positioned where spills, should they occur, are least likely to flow towards water courses, water bodies, or streams. The storage facilities will be located away from main-frame construction activities and be inspected on a regular basis to ensure compliance with regulations.
- Re-fuelling hoses will have a design pressure rating of at least 150% of the maximum head of the system.
- Re-fuelling operators will be in attendance for the duration of the fuelling operation.
- Fuel oil drums will be limited to a quantity that supports only the current activity and minor equipment maintenance.
- A fuel metering system will be implemented as a control measure and contractors will be required to use card access and report fuel usage to the EPCM team on a weekly basis.
- Spill Protection Kits.
- No Smoking.

24.2.17 *Security*

Overall security of the project construction site is the responsibility of YMI. A security service contractor will be retained and an entry gate will be built to ensure the physical integrity of the facilities as well as control and record the access of people to construction and restricted areas. Vehicles and people will be inspected upon entry (for fitness to work and authority to enter) and upon exit to check for unauthorized removal of materials from the construction site.

Project security relating to owner supplied materials, plant and equipment brought onto the site for construction purposes will be the responsibility of YMI until the material is turned over to the contractor. Upon withdrawal by a contractor of material, plant or equipment from the Owner's warehouse or lay down area, the contractor will take custody and assume the responsibility for the proper care and security of such equipment and materials.

The security of the contractor's designated areas and facilities, including material, plant and equipment, will be the responsibility of each contractor.

24.2.18 *COMMISSIONING*

In this context, commissioning applies to fixed plant and equipment but not mobile mining equipment. Commissioning of specific systems will be carried out by coordinated team from:

- Project construction;
- YMI operations and maintenance;
- Project design engineering;
- Vendors; and
- Construction contractors.

A commissioning program will be developed by the Commissioning Manager containing key commissioning definitions, an outline of the facilities to be commissioned, a summary plan, guidelines on risk management and quality assurance and control and samples of a number of sign off certificates. The Commissioning Manager will ensure that individual system commissioning plans are started when design engineering is about 75% complete. A standard plan template will be used for each system plan.

The Commissioning Manager will establish and maintain the overall commissioning program, support development of system commissioning plans by engineers, provide support for commissioning and help ensure the consistent application of the program.

The main responsibilities of a system commissioning team (several teams for different systems at different times) are:

- Prepare the commissioning plan for the specific system;
- Track the resolution of items on the defect list, including scope defects;
- Organize supplier commissioning assistance, Vendor representatives;
- Participate in inspection, testing and trials;
- Participate in safety checks;
- Prepare turn over packages to client of commissioned systems;
- Sign certificates of acceptance as systems are released from the contractor for cold commissioning dry runs;
- Coordinate the transition to operations personnel during hot commissioning and ramp-up.

24.2.18.1 Pre-Commissioning of Equipment

This stage of commissioning consists of the complete inspection, testing and operation of each piece of equipment individually, checking that electrical control and power wiring has been connected to the equipment correctly and checking the configuration and calibration of each instrument loop. The constructor is responsible for directing this stage. The CM construction supervisor will witness tests, approve inspections and countersign the checklists and data sheets.

24.2.18.2 Cold Commissioning of Systems

This stage consists of testing and operating the equipment, grouped together into systems or facilities without product, e.g., the water system without water, or crushing and screening without rock. The contractor is responsible for directing this stage. At the end of the stage, the contractor will have corrected those defects identified by the construction supervisor deemed necessary to proceed with hot commissioning. Remaining defects will be listed and remedies agreed to in writing by the contractor. Upon completion of cold commissioning, the contractor, YMI operations and maintenance senior representative, and the project construction supervisor will sign a certificate accompanied by a defect list that hands the system to the CM team.

24.2.18.3 Hot Commissioning Start Up

In this stage, YMI operations personnel will put the facility into operation under the direction of the project team. The object is to prove that the system will operate acceptably under realistic conditions. Assistance from the contractor at this stage will be considered extra work and is covered under a separate contract.

24.2.18.4 Facility Ramp-Up Performance Testing

The objective of this stage is to perform and document production performance tests using normal feed to produce final product. Proof testing criteria (rates, durations, and quality) will be set well in advance and no later than during the commissioning plan, to avoid issues that might delay final acceptance. The completion of a final acceptance certificate at the end of this stage will signify the closure of this stage and the completed facility is now ready for full ramp-up under Operations.

24.2.18.5 Acceptance and Production Ramp-Up

Ramp-up of the entire mine-mill complex will be achieved under the direction of the Operations management team, once the final acceptance certificates for individual systems have been signed off and ownership is transferred. Operations will assume responsibility for detailed planning and execution of the ramp up together with correction of any remaining defects. Detailed planning for ramp -up will begin before commissioning is under way.

24.2.19 *PROJECT COMPLETION*

The Project Manager will be responsible for preparing a project close out plan that includes:

- Transfer of care custody and control;
- Issuing project closeout notices to all vendors;
- Declaring project personnel redundant with appropriate notice periods according to an agreed schedule;
- Obtaining waivers and lien releases;
- Processing and resolving all remaining change orders, claims, disputes, back charges and final payments. Releasing holdbacks;
- Confirming that the Owner has a complete set of:
- Warranty records;
- QA records;

- As-built record drawings;
- Operations and maintenance manuals;
- Parts lists;
- Compliance documents (permits, licenses, inspection reports, etc.);
- Closing the project accounting system and cancelling the delegation of authority; and
- Issuing the project closeout report, including the final cost report.

All project completion activities are expected to be substantially completed when the project meets its production objectives. Care of outstanding disputes, if any, would be transferred to the Owner at that time and the project team would be wound down.

25 INTERPRETATION AND CONCLUSIONS

The FS authors are of the opinion that the data is adequate to support a mineral resource and mineral reserve estimate as defined under NI 43-101.

The FS authors conclude that a 70,000t/d copper processing facility for the Harper Creek Project is economically viable and can be successfully achieved with proven conventional mining and processing methods under the conditions and assumptions outlined in this report. The greatest risk to successful project development is metal pricing.

26 RECOMMENDATIONS

Opportunities exist to enhance the project economics including:

- Expansion of the resource which remains open down dip to the north and east;
- Utilization of alternative energy supply including trolley assist and LNG;
- Optimize the process during the basic and detailed engineering phases; and

As such, the FS authors recommend that YMI undertake the following:

- YMI should continue with the environmental studies and permitting efforts now underway (approximately C$1.5M required).
- YMI should continue with the engineering effort in support of permitting (approximately C$6.0M required).
- Initiate discussions with BC Hydro for increased power supply to the North Thompson Valley (approximately C$0.5M required).

27 REFERENCES

Aeroquest Ltd., 2006. *Report on a Helicopter-Borne AeroTEM II Electromagnetic & Magnetometer Survey.*

Aeroquest Ltd., 2009. *Interpretation of Helicopter-Borne AeroTEM II Electromagnetic & Magnetometer Survey*.

Allnorth Consultants Limited, 2011. *Harper Creek Mine Vavenby Rail Siding Report, Lyle Prescott, December 14, 2011*.

Allnorth Consultants Limited, 2011. *Harper Creek Project Access Investigation, Craig Smith, June 22, 2011*.

Armstrong, R.N., and Hawkins, T.D., 2009. *Lithological, Structural and Geochemical Aspects of the Harper Creek Property, British Columbia, unpublished company report for Yellowhead Mining Inc*.

Ash, C.H., and Riveros, C.P., 2001. *Geology of the Gibraltar Copper-Molybdenite Deposit, East-Central British Columbia (93B/9); in Geological Fieldwork 2000, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 2001-1, p.1 19-133*.

Bailey, S.L., Paradis, S., and Johnston, S.T., 2001. *New Insights Into Metavolcanic Successions and Geochemistry of the Eagle Bay Assemblage, South-Central British Columbia; in Geological Survey of Canada, Current Research 2001-A8, 25 p*.

Bailey, S.L., Paradis, S., Johnston, S.T., and Höy, T., 2000. *Geologic Setting of the Devonian-Mississippian, Rea and Samatosum VMS Deposits of the Eagle Bay Assemblage, Adams Lake Area, South Central British Columbia; in Geological Fieldwork 1999, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 2000-1, p.287-296*.

British Columbia Provincial Government, 2013. *Sub-Provincial Population Estimates and Thompson-Nicola Regional District Community Facts*.

Bysouth, G.D., Campbell, K.V., Barker, G.E., and Gagnier, G.K., 1995. *Tonalite-Trondhjemite Fractionation of Peraluminous Magma and the Formation of the Syntectonic Porphyry Copper Mineralization, Gibraltar Mine, Central British Columbia; in Porphyry Deposits of the Northwestern Cordillera of North America, Schroeter, T.G., Editor, Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46, p.201-213.*

Calderwood, A.R., van der Heyden, P., and Armstrong, R.L., 1990. *Geochronometry of the Thuya, Takomkane, Raft and Baldy Batholiths, South- Central British Columbia; Geological Association of Canada – Mineralogical Association of Canada, Joint Annual Meeting, Vancouver, BC, Program with Abstracts, Volume 15, p.A19*.

Campbell, R.B., and Tipper, H.W., 1971. *Bonapart Lake Map-Area, British Columbia, Geological Survey of Canada, Memoir 363.*

Casselman, M.J., McMillan, W.J., and Newman, K.M., 1995. *Highland Valley Porphyry Copper Deposits Near Kamloops, British Columbia: A Review and Update with Emphasis on the Valley Deposit: In Porphyry Deposits of the*

Northwestern Cordillera of North America, Schroeter, T.G., Editor, Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46, p.161-191.

Cathro, M.S., and Lefebure, D.V., 2000. *Several New Plutonic-Related Gold, Bismuth and Tungsten Occurrences in Southern British Columbia: in Geological Fieldwork 1999, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 2000-1, p.207-223.*

FLSmidth Dawson Metallurgical Laboratories, 2011. *Results of Crushing and Grinding Index Tests on Nine Core Samples from the Harper Creek Project, Philip Thompson, prepared for Laurion Consulting Inc., July 22, 2011*.

FLSmidth Inc., 2011. *Compressive Strength, Impact Strength, Ball Mill Bond Mill Work Index and Abrasion Analysis performed for Yellowhead Mining Inc. (#2991150261), L.S. Dutt, July 2011*.

Fraser, T.M., Stanley, C.R., Nikic, Z.T., Pesalj, R., and Gore, D., 1995. *The Mount Polley Alkalic Porphyry Copper-Gold Deposit, South-Central British Columbia; in Porphyry Deposits of the Northwestern Cordillera of North America, Schroeter, T.G., Editor, Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46, p.609-622.*

G&T Metallurgical Services Ltd., 2012. *Feasibility Metallurgical Testing Harper Creek Project Yellowhead Mining Inc. (KM2916), February 24, 2012.*

G&T Metallurgical Services Ltd., 2012. *Concentrate Generation for the Harper Creek Project Yellowhead Mining (KM3221), January 5, 2012.*

G&T Metallurgical Services Ltd., 2012. *Amenability of Harper Creek Ores to Pre-Concentration by Size Yellowhead Mining (KM3234), February 6, 2012*.

G&T Metallurgical Services Ltd., 2011. *Supplemental Metallurgical Testing Harper Creek Project Yellowhead Mining Inc. (KM2877), March 3, 2011.*

G&T Metallurgical Services Ltd., 2010. *Metallurgical Testing Master Composite 1 Harper Creek Project Yellowhead Mining Inc. (KM2756), December 3, 2010*.

Holland, S.S., 1976. *Landforms of British Columbia, a Physiographic Outline; Ministry of Energy, Mines and Petroleum Resources, Bulletin 48*.

Höy, T., 1996. *Harper Creek: a Volcanogenic Sulphide Deposit Within the Eagle Bay Assemblage, Kootenay Terrane, Southern British Columbia: in Geological Fieldwork 1996, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 1997-1, p.199-209*.

Höy, T., 1999. *Massive Sulphide Deposits of the Eagle Bay Assemblage, Adams Plateau, South Central British Columbia: in Geological Fieldwork 1998, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 1999-1, p.223-245*.

Höy, T., and Goutier, F., 1986. *Rea Gold (Hilton) and Homestake Volcanogenic Sulphide-Barite Deposits, Southeastern British Columbia; in Geological Fieldwork 1985, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 1986-1, p.59-68.*

Hughes, N.D., Paradis, S., Sears, J.W., and Pope, M., 2001. *Lithology, Tectonstratigrahy, and Paleogeography of the Vavenby Area, Eagle Bay Assembalge, South-Central British Columbia, a Possible Constraint for the Timing of Rifting of Laurentia. Current Research 2001 -A9 Geological Survey of Canada, 1-8*.

Insight Geophysics Inc., 2007a. *Geophysical Survey Summary Report, Horizontal Loop Electromagnetic and Ground Magnetic Surveys, Jones Grid, Northwest Grid and Harper West Grid, Northwest and Jones Grids, Harper Creek Property, May 2007*.

Insight Geophysics Inc., 2007b. *Geophysical Survey Summary Report, Horizontal Loop Electromagnetic and Ground Magnetic Surveys, M Anomaly Grid, Harper Creek Property, June 2007*.

Insight Geophysics Inc., 2007c. *Geophysics Logistical Report, Tuned Gradient and Insight Section Induced Polarization and Resistivity Surveys, Harper Creek Property, November 2007*.

Knight Piésold Ltd., 2012*. 2012 Open Pit Geotechnical Design (VA101-458/4-5 Rev. 0). dated April 30, 2012).*

Knight Piésold Ltd., 2014. *Mine Waste and Water Management Design Report (VA101-458/11-1 Rev. 0)*, undated.

Kraft, J.E., 1972., *Target Evaluation, Harper Creek Deposits Joint Venture, Private Report, Noranda Exploration Co. Ltd*.

Kraft, J.E., 1974. *Evaluation Review of Harper Creek Joint Venture, Private Report, Noranda Exploration Co. Ltd.*

KWM Consulting Inc., 2011. *Grinding Circuit Evaluation for the Harper Creek Project, prepared for Yellowhead Mining Inc., Ken Major, October 7, 2011.*

Lefebvre, J.J., 2006. *Data Compilation of the Harper Creek Property, unpublished report for Yellowhead Mining Inc., by CME Managing Consultants Inc. February 28, 2006*.

Logan, J.M., 2000. *Plutonic-Related Gold-Quartz Veins in Southern British Columbia; in Geological Fieldwork 1999, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 2000-1, p.193-206*.

Logan, J.M., 2001. *Prospective Areas for Intrusion-Related Gold-Quartz Veins in Southern British Columbia; in Geological Fieldwork 2000, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 2001-1, p.231-252*.

Logan, J.M., 2002. *Intrusion-Related Mineral Occurrences of the Cretaceous Bayonne Magmatic Belt, Southeast British Columbia; British Columbia Ministry of Energy, Mines and Petroleum Resources, Geoscience Map 2002-1*.

Logan, J.M. and Mihalynuk, M.G., 2005. *Regional Geology and Setting of the Cariboo, Bell, Springer and Northeast Porphyry Cu-Au Zones at Mount Polley, South-Central British Columbia; in Geological Fieldwork 2004, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 2005-1, p. 249-270*.

Merit Consultants International Inc., 2013. *Amended & Restated Technical Report and Feasibility Study for the Harper Creek Copper Project. Prepared by Jay Collins, P.Eng., Mark Dobbs, P.Eng, Ronald Simpson, P. Geo., Kenneth Brouwer, P. Eng., John Fox, P. Eng., John Nilsson, M Sc., P.Eng., January 25, 2013*.

Merit Consultants International Inc., 2012. *Technical Report and Feasibility Study for the Harper Creek Copper Project. Prepared by Jay Collins, P.Eng., Mark Dobbs, P.Eng, Ronald Simpson, P. Geo., Kenneth Brouwer, P. Eng., John Fox, P. Eng., John Nilsson, M Sc., P.Eng., March 29, 2012.*

Mortimer, N., 1987. *The Nicola Group: Late Triassic and Early Jurassic Subduction-Related Volcanism in British Columbia; Canadian Journal of Earth Sciences, Volume 24, p.2521-2536*.

Mortensen, J.K., Ghosh, D.K., and Ferri, F., 1995. *U-Pb Geochronology of Intrusive Rocks Associated with Copper-Gold Porphyry Deposits in the Canadian Cordillera, in Schroeter, T.G., ed., Porphyry Deposits in the Northwestern Cordillera of North America: Canadian Institute of Mining, Metallurgy and Petroleum Special Volume 46, p. 142-158*.

Naas, C.O., 2013. *Technical Report on the Phase XI Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., May 31, 2013*.

Naas, C.O., 2012a. *Technical Report on the Phase VIII Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., January 31, 2012*.

Naas, C.O., 2012b. *Report on the Condemnation Diamond Drill Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., March 13, 2012, revised June 22, 2012*.

Naas, C.O., 2012c. *Report on the Core Logging, Geochemical Sampling and GPS Surveying of the Geomechanical and Geotechnical Drilling Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., March 14, 2012*.

Naas, C.O., 2012d. *Report on the Phase X Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., March 20, 2012, revised June 22, 2012*.

Naas, C.O., 2011a. *Technical Report on the Phase VII Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., 8 vol., October 15, 2011.*

Naas, C.O., 2011b. *Metallurgical Drilling and Sample Collection Program on the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., May 24, 2011*.

Naas, C.O., 2010. *Technical Report on the Phase VI Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., 4 vol., January 15, 2010.*

Naas, C.O., 2009. *Technical Report on the Phase V Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., 3 vol., June 11, 2009*.

Naas, C.O., 2008. *Technical Report on the Phase III Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., 2 vol., March 10, 2008*.

Naas, C.O., 2007. *Technical Report on the Phase II Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Managing Consultants Inc. 2 vol., July 20, 2007*.

Naas, C.O., 2006. *Technical Report on the Phase I Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., 3 vol., November 30, 2006*.

Naas C.O. and Soloviev, S., 2008. *Technical Report on the Phase IV Exploration Program of the Harper Creek Property, unpublished report for Yellowhead Mining Inc. by CME Consultants Inc., 9 vol., June 30, 2008*.

Nelson, J.A., and Friedman, R., 2004. *Superimposed Quesnel (late Paleozoic-Jurassic) and Yukon-Tanana (Devonian-Mississippian) Arc Assemblages, Cassiar Mountains, Northern British Columbia: Field, U-Pb, and Igneous Petrochemical Evidence: Canadian Journal of Earth Sciences, 41(10), p.1201-1235*.

Okulitch, A.V., 1979. *Lithology, Stratigraphy, Structure and Mineral Occurrences of the ThompsonShuswap-Okanagan Area, British Columbia; Geological Survey of Canada, Open File 637*.

Paradis, S., Bailey, S.L., Creaser, R.A., Piercey, S.J. and Schiarizza, P., 2006. *Paleozoic Magmatism And Syngenetic Massive Sulphide Deposits Of The Eagle Bay Assemblage, Kootenay Terrane, Southern British Columbia, in Colpron, M. and Nelson, J.L., eds., Paleozoic Evolution and Metallogeny of Pericratonic Terranes at the Ancient Pacific Margin of North America, Canadian and Alaskan Cordillera: Geological Association of Canada, Special Paper 45, p. 3 83-414.*

Panteleyev, A., Bailey, D.G., Bloodgood, M.A., and Hancock, K.D., 1996. *Geology and Mineral Deposits of the Quesnel River-Horsefly Map Area, Central Quesnel Trough, British Columbia; British Columbia Ministry of Energy, Mines and Petroleum Resources, Bulletin 97, 155 p.*

Phillips, Barrat, Kaiser Engineering Ltd., 1988. *Geological Evaluation Report, Pre-feasibility Study, Hail – Harper Creek Copper Prospect; B.C. Ministry of Energy, Mines and Petroleum Resources, Assessment Report 17, 650p*.

Quinn, Colin, 2011. *Quotation for the Supply of Transtank Self Contained Hydrocarbons Storage and Dispensing Equipment, Petro-Canada, Transtank Canada Inc., prepared for Yellowhead Mining Inc. November 22, 2011*.

Rennie, D.W. and Scott K.C., 2010. *Technical Report on the Harper Creek Project, Clearwater, British Columbia, Canada for Yellowhead Mining Inc. prepared by Scott Wilson Roscoe Postle Associates, August 16, 2010*.

Rennie, D.W. and Scott K.C., 2007. *Technical Report on the Harper Creek Project, British Columbia, Canada prepared for Yellowhead Mining Inc. by Scott Wilson Roscoe Postle Associates Inc., November 1, 2007*.

Ross, K.V., Godwin, C.I., Bond, L., and Dawson, K.M., 1995. *Geology, Alteration and Mineralization of the Ajax East and Ajax West Copper- Gold Alkalic Porphyry Deposits, Southern Iron Mask Batholith, Kamloops, British Columbia: in Porphyry Deposits of the Northwestern Cordillera of North America, Schroeter, T.G., Editor, Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46, p.565-580.*

Sanguinetti, M.H. and Lefebvre, J.J., 2006. *Technical Report on the Harper Creek Property, unpublished report for Yellowhead Mining Inc., March 16, 2006.*

Schiarizza, P., 1986a. *Geology of the Eagle Bay Formation Between the Raft and Baldy Batholiths (82M5, 11, 12); in: Geological Fieldwork 1985; Ministry of Energy Mines and Petroleum Resources Paper 1986-1, p. 89-94.*

Schiarizza, P., 1986b. *Geology of the Vavenby Area (82M5, 11, 12) in Geological Fieldwork 1985, British Columbia Ministry of Energy, Mines and Petroleum Resources, Open File Map 1986-5, scale 1:50,000.*

Schiarizza, P. & Boulton, A., 2006. *Geology and Mineral Occurrences of the Quesnel Terrane, Canim Lake Area (NTS 092P/15), South-Central British Columbia; in Geological Fieldwork 2005, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 2006-1, p.163-184.*

Schiarizza, P., Hefferman, S., and Zuber, J., 2002. *Geology of Quesnel and Slide Mountain Terranes West of Clearwater, South- Central British Columbia (NTS 92P/9, 10, 15, 16) in Geological Fieldwork 2001, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 2002-1, p.83-108.*

Schiarizza P. and Preto V.A., 1987. *Geology of the Adams Plateau-Clearwater-Vavenby Area, British Columbia Ministry of Energy Mines and Petroleum Resources Paper 1987-2, 88p.*

SRK Consulting (Canada) Inc., 2012. *Harper Creek ML/ARD Memorandum, Stephen Day, report prepared for Yellowhead Mining Inc. February 1, 2012.*

Wardrop, A Tetra Tech Company, 2011. *Technical Report and Preliminary Assessment of the Harper Creek Project for Yellowhead Mining Inc., prepared by Narciso, N., Huang, J., Boyle, J.M., Ghaffari, H., Triebel, K., Teymouri, S., Cameron, M., Greenaway, G., and Donaghue, P., March 31, 2011.*

28 SIGNATURE PAGE & CERTIFICATES

The effective date of this Technical Report, titled "Technical Report and Feasibility Study of the Harper Creek Copper Project", is July 31, 2014.

Signed by,

"Original document signed by
A. Jay Collins, P.Eng." July 31, 2014
Merit Consultants International Inc. Date

"Original document signed by
Mark W. Dobbs, P. Eng." July 31, 2014
Allnorth Consultants Limited Date

"Original document signed by
Ronald Simpson, P. Geo." July 31, 2014
GeoSim Services Inc.

"Original document signed by
Daniel Fontaine, P. Eng." July 31, 2014
Knight Piésold Ltd. Date

"Original document signed by
John R. W. Fox, P. Eng." July 31, 2014
Laurion Consulting Inc. Date

"Original document signed by
John Nilsson, M. Sc., P. Eng." July 31, 2014
Nilsson Mine Services Ltd. Date

CERTIFICATE OF QUALIFIED PERSON
A. Jay Collins, P.Eng.

I, A Jay Collins, P. Eng., of White Rock, British Columbia, do hereby certify:

- I am the President of Merit Consultants International Inc., with a business address at #401 – 750 West Pender Street, Vancouver, British Columbia V6C 2T8.
- This certificate applies to the Technical Report titled "Technical Report and Feasibility Study of the Harper Creek Copper Project, dated July 31, 2014 (the "Technical Report").
- I graduated in 1974 from Portsmouth Polytechnic with a Bachelor of Science Degree in Civil/Structural Engineering.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #12741).
- My relevant experience includes more than 30 years experience in mining and plant operation, project studies, management, and engineering.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I made a personal inspection of the Property as a member of the Feasibility Study Team on July 11 and 12, 2011; a duration of 8 hours.
- I am responsible for Section(s) 2, 3, 4, 5, 6, 19, 22, 23, 24, 25, 26, 27 and as co-author of Sections 1 and 21 of the Technical Report.
- I am independent of Yellowhead Mining Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property that is the subject of this Technical Report was as co-author of the Technical Report and Feasibility Study for the Harper Creek Copper Project dated March 29, 2012, and subsequent amendment dated January 25, 2013.
- I have read the Instrument and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 31st day of July 2014 at Vancouver, British Columbia.

"Original document signed & sealed"
A. Jay Collins, P.Eng.
President
Merit Consultants International Inc.

CERTIFICATE OF QUALIFIED PERSON
Mark W. Dobbs, P.Eng.

I, Mark W. Dobbs of Surrey, British Columbia, do hereby certify:

- I am the Director, Special Projects with Allnorth Consultants Ltd., with a business address at 312 – 7485 – 130th Street, Surrey, BC V3W 1H8.
- This certificate applies to the Technical Report titled "Technical Report and Feasibility Study of the Harper Creek Copper Project, dated July 31, 2014 (the "Technical Report").
- I am a graduate of the University of British Columbia.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #22906).
- I am a member in good standing of Professional Engineers Ontario (License #100115013)
- My relevant experience includes 8 years experience in mining and plant operation, project studies, management, and engineering.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I made a personal inspection of the Property as a member of the Feasibility Study Team on July 11 and 12, 2011; a duration of 8 hours.
- I am responsible for Section 18, and as co-author of Sections 1, 17, 25, and 26 of the Technical Report.
- I am independent of Yellowhead Mining Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property that is the subject of this Technical Report was as co-author of the Technical Report and Feasibility Study for the Harper Creek Copper Project dated March 29, 2012, and subsequent amendment dated January 25, 2013.
- I have read the Instrument and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 31st day of July 2014 at Surrey, British Columbia.

"Original document signed & sealed"
Mark W. Dobbs, P.Eng.
Director, Special Projects
Allnorth Consultants Ltd.

CERTIFICATE OF QUALIFIED PERSON
Ronald G. Simpson, P.Geo.

I, Ronald G. Simpson, P.Geo, of Vancouver, British Columbia, do hereby certify:

- I am the President of GeoSim Services Inc., with a business address at 1975 Stephens St., Vancouver, British Columbia, V6K 4M7.
- This certificate applies to the Technical Report titled "Technical Report and Feasibility Study of the Harper Creek Copper Project, dated July 31, 2014 (the "Technical Report").
- I graduated with an Honours Degree of Bachelor of Science in Geology from the University of British Columbia in 1975.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License No. 19513).
- My relevant experience includes 37 years' experience in mining and mineral exploration and 25 years' experience in mineral resource estimation.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I made a personal inspection of the Property as a member of the Feasibility Study Team on July 11 and 12, 2011; a duration of 8 hours.
- I am responsible for Sections 7, 8, 9, 10, 11, 12 and Section 14, and as co-author of Section 1, 25, and 26 of the Technical Report.
- I am independent of Yellowhead Mining Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property that is the subject of this Technical Report was as co-author of the Technical Report and Feasibility Study for the Harper Creek Copper Project dated March 29, 2012, and subsequent amendment dated January 25, 2013.
- I have read the Instrument and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 31st day of July 2014 at Vancouver, British Columbia.

"Original document signed & sealed"
Ronald G. Simpson, P.Geo.
President
GeoSim Services Inc.

CERTIFICATE OF QUALIFIED PERSON
Daniel Fontaine, P.Eng.

I, Daniel Fontaine, of Vancouver, British Columbia, do hereby certify:

- I am employed as a Senior Engineer with Knight Piésold Ltd., with a business address at 1400 – 750 West Pender Street, Vancouver, British Columbia, V6C 2T8.
- This certificate applies to the Technical Report titled “Technical Report and Feasibility Study of the Harper Creek Copper Project, dated July 31, 2014 (the “Technical Report”).
- I am a graduate of the McGill University, (B.Eng., 2006).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License 36208).
- I have practiced my profession for over 7 years since graduation in geotechnical engineering, tailings management, water management and environmental studies. I have been directly involved in geotechnical aspects of the project and the mine waste and water management studies.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- I visited the Harper Creek Project on October 26 to 27, 2011, July 15 to 17, 2012, and September 30, 2012 to view proposed locations of site facilities, potential construction material borrow areas and review geotechnical site investigation progress.
- I am responsible for Section 20 and provided the quantities and recommendations related to Tailings and Water Management that were relied upon in preparation of Sections 21.1.9, 21.1.14 and 21.2.3.
- I am a co-author of Section 1, 18, 25, and 26.
- I am independent of Yellowhead Mining Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 31st day of July 2014 at Vancouver, British Columbia.

“Original document signed & sealed”
Daniel Fontaine, P.Eng.
Senior Engineer
Knight Piésold Ltd.

CERTIFICATE OF QUALIFIED PERSON
John R. W. Fox, P. Eng.

I, John R.W. Fox, P. Eng. of North Vancouver, British Columbia, do hereby certify:

- I am a consulting metallurgical engineer with Laurion Consulting Inc. with a business address at #302-304 West Cordova Street, Vancouver, British Columbia, V6B 1E8.
- This certificate applies to the Technical Report titled "Technical Report and Feasibility Study of the Harper Creek Copper Project, dated July 31, 2014 (the "Technical Report").
- I graduated with a Bachelors of Science in Applied Mineral Sciences from the University of Leeds in 1971.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License No. 12578).
- My relevant experience includes more than 40 years of experience working throughout the world as a metallurgical engineer.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I made a personal inspection of the Property as a member of the Feasibility Study Team on July 11 and 12, 2011; a duration of 8 hours.
- I am responsible for Sections 13 and 17 and as co-author of Sections 1, 21, 25, and 26 of the Technical Report.
- I am independent of Yellowhead Mining Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property that is the subject of this Technical Report was as co-author of the Technical Report and Feasibility Study for the Harper Creek Copper Project dated March 29, 2012, and subsequent amendment dated January 25, 2013.
- I have read the Instrument and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 31st day of July 2014 at Vancouver, British Columbia.

"Original document signed & sealed"
John R.W. Fox, P.Eng.
Laurion Consulting Inc.

CERTIFICATE OF QUALIFIED PERSON
John W. Nilsson, P. Eng.

I, John W. Nilsson , P. Eng. of Pitt Meadows, British Columbia, do hereby certify:

- I am the President of Nilsson Mine Services Ltd with a business address at 20263 Mountain Place, Pitt Meadows, B.C., V3Y 2T9.
- This certificate applies to the Technical Report titled "Technical Report and Feasibility Study of the Harper Creek Copper Project, dated July 31, 2014 (the "Technical Report").
- I graduated with a Bachelors degree in Geology from the Queen's University in 1977. In addition, I obtained a Masters degree in Mining Engineering from the Queen's University in 1990.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License No. 20697).
- I have worked as a geologist and mining engineer since graduation from university.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I made a personal inspection of the Property as a member of the Feasibility Study Team on July 11 and 12, 2011; a duration of 8 hours.
- I am responsible for Sections 15 and 16 and as co-author of Sections 1, 21, 25, and 26 of the Technical Report.
- I am independent of Yellowhead Mining Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property that is the subject of this Technical Report was as co-author of the Technical Report and Feasibility Study for the Harper Creek Copper Project dated March 29, 2012, and subsequent amendment dated January 25, 2013.
- I have read the Instrument and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 31st day of July 2014 at Vancouver, British Columbia.

"Original document signed & sealed"
John W. Nilsson, P.Eng.
President
Nilsson Mine Services Ltd.