TECHNICAL REPORT ON THE MINERAL RESERVE UPDATE AT THE YELLOWHEAD COPPER PROJECT

BRITISH COLUMBIA, CANADA

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DATE AND SIGNATURE PAGE

The effective date of this Technical Report, entitled "Technical Report on the Mineral Reserve Update at the Yellowhead Copper Project, British Columbia, Canada" is January 16, 2020.

"Signed and Sealed"

Richard Weymark, P. Eng., MBA

SECTION 1

SUMMARY

SECTION 1: SUMMARY

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1.1 Executive Summary

This report describes Taseko Mines Limited (Taseko) updated mine plan, Yellowhead Development Plan 1 (YDP-1), for the Yellowhead Copper Project (the Project) in central British Columbia, Canada. YDP-1 proposes a mill throughput of 90,000 tonnes per day (tpd) for 25 years producing an average of 200 million pounds per year for the first five year and 180 million pounds per year of copper in concentrate for the life of mine (LOM). The concentrate will be clean, not complex, and contains no penalty level deleterious elements. The total metal production, as presented in this report, would be 4.4 billion pounds of copper, 440 thousand ounces of gold and 19 million ounces of silver.

After being a major shareholder for eight years, Taseko acquired all of the outstanding common shares of Yellowhead Mining Inc. (YMI) that it did not already own on February 15, 2019, payable in Taseko common shares. The combined value of the common shares issued, acquisition costs and Taseko's previously held position in YMI reflect a total consideration of $16.3 million or 0.4 cents per pound of recoverable copper in reserves. Prior to the acquisition, Taseko identified that significant improvements could be made to the project design including reserves, cut-off grade and mill throughput resulting in increased annual copper production along with improved net present value (NPV) and internal rate of return (IRR).

Changes in design over the 2014 Harper Creek Feasibility Study include a larger open pit mined at a higher cut-off grade resulting in an increase in proven and probable reserves of 100 million tonnes along with increased copper head grades. The concentrator would be relocated and changed from a single line 70,000 tpd to a dual line 90,000 tpd configuration. Several other design changes were made including relocating the crusher 90 meters lower in elevation, utilizing in-pit dumping where possible, significantly reducing the size of the ore stockpile and constructing the tailings storage facility (TSF) main embankment using cycloned sand. Also, changes to the water management strategy would result in a 90% reduction in stored water in the TSF.

A comparison of the 2014 Harper Creek Feasibility Study and YDP-1 is presented in Table 1-1.

The project, as described in this report, has a pre-production capital cost of C$1.3 billion and provides a pre-tax NPV at an 8.0% discount rate of C$1.3 billion using US$3.10/lb Cu, US$1,350/oz Au, US$18/oz Ag and a foreign exchange rate of US$0.80 : C$1.00. The payback period is just over 4 years from start of production.

1.1 Executive Summary – *Cont'd*

Table 1-1: Comparison of 2014 Feasibility Study to YDP-1

	Harper Creek 2014 FS	YDP-1
Reserve Tonnes	716 million	817 million
Copper Cut-off Grade	0.14%	0.17%
Average Copper Grade	0.26% Cu	0.28% Cu
Strip Ratio (w:o)	0.8 : 1	1.4 : 1
Throughput	70,000 tpd	90,000 tpd
Asset Life	28 years	25 years

Performance highlights of the YDP-1 Plan are shown in Table 1-2.

Table 1-2: YDP-1 Plan Performance Highlights

	First 5 Years	LOM
Average Copper Grade	0.32% Cu	0.28% Cu
Average Copper Equivalent Grade	0.35% Cu Eq.	0.29% Cu Eq.
Average Annual Copper Production	200 million lbs	180 million lbs
Average Annual Pre-tax Cashflow	C$330 million	C$270 million
Average Operating Cost/lb Copper*	US$1.43/lb	US$1.67/lb

* Net of byproduct credits

At a 0.35% copper equivalent grade, 90% recovery, US$3.10/lb copper price and an exchange rate of US$0.80 : C$1.00 the average value per tonne of rock milled would be nearly C$26, for the first five years, while onsite operating costs during that period average C$9.76 per tonne milled.

The Yellowhead project is located 125 km north of Kamloops, British Columbia, and 200 km southeast of Taseko's Gibraltar Mine which is similar in size, nature of operations, and mine life. Yellowhead has paved highway, rail, and power access within 10 km of the property. Proximity to the mining hub of Kamloops will provide available services and labour and the project will make a significant contribution to the economy of the region.

The deposit consists of a remobilized polymetallic volcanogenic massive sulphide deposit, comprising lenses of disseminated, fracture-filling and banded iron and copper sulphides with accessory magnetite. Chalcopyrite is the dominant copper mineral representing >98% of the copper species present. Gold and silver are present throughout the deposit and report to the concentrate in payable levels.

1.1 Executive Summary – *Cont'd*

The report details the geography, ownership, geology, mineralization, metallurgy and the design, methods, and economics utilized, in determining a proven and probable mineral reserve of 817 million tonnes grading 0.28% copper, 0.03 grams per tonne (gpt) gold and 1.3 gpt silver at a 0.17% copper cut-off grade. Copper contained in the reserve is five billion pounds. Mineral reserves are contained within a measured and indicated resource totaling 1.3 billion tonnes grading 0.25% copper at a 0.15% copper cutoff grade.

The climate is typical of the central interior of BC, with short warm summers and comparatively mild Canadian winters. The area terrain is characterized by gently sloping upland ridges flanked by steepened valley slopes. The average elevation of the open pit area and plant site is 1,800m while the lowest waste storage area is at 1,400m. The project is covered in coniferous forest and has undergone extensive logging in the past.

The report has been prepared by Taseko, a producing issuer, under the supervision of Richard Weymark, P.Eng., MBA, Chief Engineer of Taseko. Yellowhead Mining Inc. is a wholly owned subsidiary of Taseko. Mr. Weymark is a Qualified Person under the provisions of National Instrument 43-101 published by the Canadian Securities Administrators. Yellowhead is a greenfield project and while federal and provincial regulatory agencies have been engaged, the project is not yet in the formal environmental assessment or permitting processes.

Due to the project's positive and robust economics, the author recommends that the project advance through the environmental assessment process as soon as practical and that identified performance opportunities be more fully evaluated.

1.2 Introduction

The purpose of this report is to summarize the pre-feasibility study and document the mineral reserve estimate announced in Taseko's news release dated January 16, 2020 in the format prescribed in National Instrument 43-101.

The resource and reserve estimation was completed by Taseko staff and contributing consultants under the supervision of Richard Weymark, P. Eng., MBA. Chief Engineer, Taseko and a Qualified Person under National Instrument 43-101.

All costs are in Canadian dollars (C$) and units are metric unless stated otherwise.

1.3 Property Location and Description

The Yellowhead project is located approximately 150 km northeast of Kamloops at latitude 51°30’ north and longitude 119°48’ west in the Kamloops Mining Division. The project has paved highway, rail, and power access at Highway #5 within 10 km of the property.

The property consists of 131 mineral claims covering 42,500 hectares. All mineral claims are in good standing and an application has been submitted to the BC Mineral Titles Office to convert 40 claims to a mining lease. There are three parcels of fee simple land located 2.5 km west of the nearest community, Vavenby, where the rail load-out facility would be located.

Six claims are subject to a 2.5% net smelter returns (NSR) royalty to Xstrata while 31 claims are subject to a 3% NSR royalty to US Steel Corp., capped at C$3.0 million, subject to inflation.

1.4 History

Copper mineralization was discovered in the immediate vicinity of the deposit in the mid-1960s. The initial discovery was followed up by extensive prospecting, line cutting, road building, surface geochemical sampling, geological mapping, geophysics, trenching and diamond drilling programs.

Noranda Exploration Company (Noranda) and Québec Cartier Mining Company (QCM), a 100% wholly owned subsidiary of US Steel, staked claims in the deposit area in 1965 and 1966 respectively. This resulted in the area west of the Harper Creek tributary belonging to Noranda and east of it to QCM. The two companies worked independently on their properties from 1966 until 1970. In late 1970, the companies formed a joint venture, which explored their contiguous properties until 1974.

Further work in the deposit area occurred in 1986 and 1996. This included trenching, core resampling and metallurgical testing and additional drilling.

Historical core drilling took place on the property in 11 different years over a 30-year period between 1967 and 1996. The total length of the 191 holes drilled on the property was 30,800 m. Of these holes, 165 targeted what is now known as the Yellowhead Copper Deposit, for a total of 28,200 m or 92% of the overall drilling.

No further drilling on the deposit area took place until 2006.

YMI formed as a private British Columbian company and obtained control of the project through staking, purchase and option agreements in 2005. YMI undertook their first phase of field exploration on the project in 2006 and completed 65,000 m of drilling from 2006 through 2013.

Historical resource estimates date back to 2007. A 2014 feasibility study completed for YMI resulted in the establishment of a proven and probable mineral reserve totalling 716 million tonnes at 0.26% copper, 0.029 gpt gold, and 1.18 gpt silver at a 0.14% copper cut-off grade.

This was contained within a measured and indicated mineral resource of 1.3 billion tonnes at 0.25% copper, 0.028 gpt gold, and 1.2 gpt silver at a 0.15% copper cut-off grade.

In February 2019, Taseko acquired a 100% interest in YMI.

1.5 Geology and 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 BC.

The Eagle Bay Assemblage incorporates Lower Cambrian to Mississippian sedimentary and volcanic rocks subject to deformation and metamorphism. The Eagle Bay Assemblage divides 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 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).

Unit EBA of the Devonian-Mississippian succession hosts the deposit.

The northeast trending Harper Creek Fault separates the deposit into a west domain and east domain. In the west domain, chalcopyrite mineralization is primarily in three copper bearing horizons. The upper horizon ranges from 60 m to 170 m in width and is continuous along an east-west strike for some 1,300 m, dipping approximately 30º north. The middle horizon is not as well developed and is often fragmented. It primarily exists within a graphitic and variably silicified package of rocks that range from 30 m to 40 m in width at the western extent, increasing up to 90 m locally eastward, gradually appearing to blend into the upper horizon. The lowest or third horizon has less definition mainly due to a lack of drill intersections. Commonly hosted within mafic to intermediate volcaniclastics and fragmental rocks, it can range from 30 m to 90 m in width although typical intersections are in the 30 m range. These horizons generally contain foliation-parallel wisps and bands as the dominant style of sulphide mineralization.

In the east domain, mineralization characterized by high angle, discontinuous, tension fractures of pyrrhotite, chalcopyrite ± bornite is frequently associated with quartz carbonate gangue. This style is common within, but not limited to, the metasedimentary rocks and areas of increased pervasive silicification. Mineralization is not selective to individual units and frequently transgresses lithological contacts throughout the area. At the near surface areas in the south and down-dip to the north, widths of mineralization typically range from 120 m to 160 m. In the central area of the east domain where thrust/reverse fault stacking has been interpreted, mineralization thicknesses typically range from 220 m to 260 m with local intersections of up to 290 m.

1.5 Geology and Deposit – *Cont'd*

The deposit type is a remobilized polymetallic volcanogenic massive 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 is consistent with the volcanogenic model. Observed sulphide lenses measure many tens of metres in thickness with km-scale strike and dip extents.

1.6 Mineral Processing and Metallurgical Testing

The basis of process design for the project was informed from feasibility level metallurgical test work program conducted in 2011 and early 2012 at G & T Metallurgical Services Ltd. (G&T), in Kamloops, BC.

This test program consisted of a suite of open circuit batch flotation testing, lock cycle testing, ore hardness testing, a pilot plant campaign, and mineralogical characterization of both a primary master composite representing feed from the earlier phases of mine development along with a suite of composite samples representing variable lithology and discreet spatial zones within the pit. Additional laboratory comminution test work conducted in 2011 at FLSmidth (FLS) of Bethlehem, Pennsylvania, was also used to inform process comminution circuit design and power requirements.

The proposed process for the project consists of a conventional milling circuit to recover copper via grinding, rougher flotation, regrinding of rougher concentrate, followed by a cleaner flotation circuit. All comminution testing conducted to date suggest the ore is soft to moderately soft and very amenable to both SAG milling and ball milling.

Mineralogy characterization on ore samples from the deposit demonstrate that chalcopyrite is the dominant copper bearing mineral making up over 98% of the copper species in majority of the deposit.

Lock cycle testing produced a final copper concentrate grade of 26% copper at about a 90% total copper recovery. The final concentrate produced from lock cycle testing and the pilot plant produced a clean concentrate with deleterious elements below typical penalty limits at smelters, and also containing payable gold and silver values.

1.7 Mineral Resource and Reserve Estimate

(a) Resource Estimate

The last exploration work on the Yellowhead resource was documented in the technical report titled “Technical Report & Feasibility Study of the Harper Creek Copper Project”, dated July 31, 2014, filed on www.sedar.com available under Yellowhead’s profile. There have been no additional relevant exploration results within the resource area nor changes to the resource block model since that time.

The sample database for the project contains results from 353 core holes (90,779 m) drilled between 1967 and the end of 2013.

The mineralized stratigraphy comprises a sequence of phyllites and schists overlying un-mineralized gneiss. Weakly mineralized to barren phyllites overlie the main mineralized horizons. The Harper Creek Fault bisects the deposit in a southwest-northeast direction and dips steeply to the southeast. The three main lithologic domains (gneiss, mineralized meta-sediments and overlying phyllites) were modeled in Gemcom-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. Block dimension are 12 m x 12 m x 12 m. Block volumes in in-situ rock domains use a density factor ranging from 2.71 to 2.85 dependant on lithology while density of overburden was assigned a factor of 2.2.

Copper, gold and silver grades within the northwest and southeast 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.

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 copper. A comparison of global mean values within the grade shell domain shows a reasonably close relationship with samples, composites and block model values.

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.

1.7 Mineral Resource and Reserve Estimate – *Cont'd*

(a) Resource Estimate – *Cont'd*

Delineated mineralization of the Yellowhead deposit is classified as a resource according to the definitions in National Instrument 43-101 and CIM (2014). Blocks were classified as measured if they were estimated in the first pass with a minimum of 4 composites from at least 2 drillholes within 82.5 m of the block centroid corresponding to one third of the maximum variogram range. The blocks meeting these criteria were then examined visually and some blocks were downgraded to indicated if they 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 third pass were also classified as indicated if the closest composite was within 125 m of a block centroid corresponding to half the maximum variogram range. A series of blocks estimated in the third 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.

Measured and indicated mineral resources at a 0.15% copper cut-off grade as of December 31, 2019 are shown in Table 1-3.

The resources presented in Table 1-3 are constrained by a pit shell derived using US $3.25/lb copper, US $1,300/oz gold, US $17.00/oz silver and an exchange rate of US$0.80 : C$1.00.

Table 1-3: Yellowhead Mineral Resources

Category	Tonnes (millions)	Cu (%)	Au (gpt)	Ag (gpt)	Cu Eq. * (%)
Measured	561	0.27	0.029	1.2	0.29
Indicated	730	0.24	0.027	1.2	0.26
M&I	**1,292**	**0.25**	**0.028**	**1.2**	**0.27**
Inferred	109	0.21	0.024	1.2	0.23

Note: totals may not add due to rounding

*Copper Equivalent is based on 90% copper recovery, US$3.10/lb copper price, 56% gold recovery, US$1350/oz gold, 59% silver recovery, and US$18.00/oz silver price.

1.7 Mineral Resource and Reserve Estimate – *Cont'd*

(b) Reserve Estimate

A reserve basis pit shell was selected by evaluating a series of nested pit shells on the basis of a number of metrics including supporting commodity price, approximate cash flow, strip ratio, metal production, equipment requirements, and number of operating years. This pit shell was used as a guide to develop the detailed pit design and subsequent scheduling for input to the cash flow.

The input parameters used to derive the reserve basis pit shell include conservative commodity prices, appropriate metal recoveries and unit costs for mining, processing, water treatment, general and administration (G&A), a sustaining capital allowance, and consultant recommended pit wall slopes.

An optimum cut-off grade was selected by developing a series of mine schedules and corresponding cash flows at various cut-off grades within the reserve basis pit shell. The cash flows were evaluated on the basis of annual cash flow, annual metal production, capital requirements, and NPV. The resulting cut-off grade used is 0.17% copper.

Proven and probable reserves are derived from measured and indicated resources respectively, that are contained within the final ultimate design and are above the stated copper cut-off grade. Table 1-4 summarizes the proven and probable mineral reserves as of December 31, 2019.

Table 1-4: Yellowhead Mineral Reserves

Category	Tonnes (millions)	Cu (%)	Au (gpt)	Ag (gpt)	Cu Eq. * (%)
Proven	458	0.29	0.031	1.3	0.31
Probable	359	0.26	0.028	1.2	0.28
Total	**817**	**0.28**	**0.030**	**1.3**	**0.29**

Note: totals may not add due to rounding

*Copper Equivalent is based on 90% copper recovery, US$3.10/lb copper price, 56% gold recovery, US$1350/oz gold, 59% silver recovery, and US$18.00/oz silver price.

1.8 Mining Method

The Yellowhead open pit is designed to be mined utilizing conventional truck and shovel mining techniques. The equipment utilized in this operation would be typical of that found in today's large open pit operations. Open pit operations are planned to supply a conventional copper concentrator with 90,000 tpd of ore at a cut-off grade of 0.17% copper at a strip ratio of 1.4 : 1 for 25 years. Ore would be delivered to a primary crusher located at the southwestern rim of the ultimate pit. An ore stockpile would be established during the first five years of operation to maximize ore grade delivered to the mill during that period and provide operating flexibility. Potentially acid generating (PAG) waste rock would be stored inside the TSF while non-acid generating (NAG) waste and overburden would be placed in conventional waste storage locations proximal to the open pit.

1.9 Recovery Method

The proposed process plant for the project is a conventional sulphide concentrator utilizing three-stages of comminution, sulphide flotation and concentrate dewatering. Process design and equipment sizing undertaken were informed by results obtained from the 2011/2012 feasibility metallurgical test program conducted at G&T.

The concentrator is designed to process a nominal 90,000 tpd of ore and produce a marketable copper concentrate containing silver and gold. The concentrator would consist of a primary gyratory crusher fed run-of-mine (ROM) ore from the pit transported via haul trucks. The product from the crusher would be transported via overland conveyors to a coarse ore stockpile. Ore from the stockpile would then be reclaimed and fed to two parallel SAG-ball mill circuits which produce feed for a single rougher flotation bank. The rougher flotation concentrate would be reground with two parallel vertical stirred mills prior to being reprocessed in a two stage cleaner flotation circuit which includes both tank and column flotation cells. Sulphide minerals are collected with a conventional xanthate collector and pyrite is rejected using lime.

The final concentrate would be dewatered by thickening followed by filtration prior to being conveyed to the final concentrate stockpile. The final concentrate would be trucked off-site to a proximal rail load-out facility for subsequent transport to the Port of Vancouver or direct rail to other North American markets.

Both rougher and first cleaner flotation tailings would be transported separately to the TSF. Process water from the TSF would be reclaimed and recycled back to the process plant for reuse.

1.10 Infrastructure

The infrastructure, services and ancillary facilities required for the project include the following:

- Site access road;
- Rail load-out facility;
- Power supply and site electrical distribution;
- Crusher and conveyor facilities;
- Concentrator building;
- Water management and treatment;
- Tailings distribution and storage facility (TSF);
- Maintenance facilities;
- Warehouse and storage facilities;
- Explosives facilities;
- Construction camp;
- Administrative and dry facilities;
- Site security and first aid;
- Fuel storage and dispensing;
- Sewage collection and treatment.

1.11 Market Studies & Contracts

Copper concentrate produced at Yellowhead is estimated to have a 25.5% copper grade with payable amounts of gold and silver and no deleterious elements at typical smelter penalty levels. While there are currently no contracts in place for the sale of concentrate, it is expected that the clean nature of the concentrate would make it attractive to a large array of smelters globally.

For evaluating the project Taseko has relied on long term street consensus commodity pricing as of December 2019.

The offsite costs associated with concentrate transport, port storage, stevedoring, shipping, treatment and refining have been incorporated into the economic analysis of the project based upon Taseko's current experience at it's Gibraltar Mine.

Standard procurement contracts would be required for construction, materials delivery and some site services.

1.12 Environmental, Permitting, Social and Community Impact

Taseko has engaged with both the British Columbia Environmental Assessment Office (BCEAO) and the Impact Assessment Agency of Canada (IAC) regarding the Yellowhead project but it is not yet formally in the environmental assessment process.

BCEAO is expected to confirm that an assessment is required in order for the project to proceed and an Environmental Assessment (EA) certificate needs to be issued after the review of the EA application.

Federally, the Impact Assessment Act came into effect in August 2019 and applies to projects described in the Physical Activities Regulation. It is expected that the agency will confirm that an impact assessment is required.

Federal permits, licenses or approvals that may be required for the construction, operation, or closure of the project are the following:

- Authorization will be required for explosives storage under the Explosives Act;
- Authorization will be required for aeronautical clearance for the overhead transmission line crossing of the North Thompson River;
- Authorization under the Fisheries Act and Metal Mining Effluent Regulations (MMER) may be required, as water will be discharged from the site during operations and into post closure;
- Authorizations under the Fisheries Act may be required although current field data and presence of downstream barriers suggests that the mine site area is not providing habitat to any fish species, and proposed transmission line crossings will be designed to avoid habitat disruption in riparian areas.

It is expected that during the EA process and after further discussion with federal departments the nature of any federal authorizations will be confirmed.

Provincial permits, licences and approvals that may be required for the project from the following ministries:

- BC Ministry of Energy, Mines and Petroleum Resources (BCMEMPR);
- BC Ministry of Forests, Lands and Natural Resource Operations (BCMFLNRO);
- BC Ministry of Environment (BCMOE);
- Ministry of Transportation (MOT).

It is expected that during the EA process and the exchanges with BC regulatory authorities, specific requirements will be refined.

There are currently no permit applications under review with provincial or federal regulatory bodies.

1.13 Capital and Operating Costs

A summary of the pre-production capital costs estimated for the project is provided in Table 1-5. All costs shown are in Q4, 2019 Canadian dollars.

Table 1-5: Pre-Production Capital Costs

Area	Total Pre-Production Capital ($ billions)
Mining Equipment* / Pre-Production Mining Costs	0.2
Processing Facilities	0.5
Tailings & Water Collection Facilities	0.1
Ancillary Facilities / Infrastructure	0.2
Subtotal Direct Costs	**1.0**
Indirect Costs	**0.4**
Grand Total	**1.3**

*Includes down payment and lease costs in pre-production years only.
Note: totals may not add due to rounding

The sustaining capital estimate includes a water treatment plant (WTP), staged TSF embankment construction, additional water collection systems, additional mining equipment, mining equipment lease payments, and general sustaining capital through the life of the mine. Sustaining capital costs are shown in Table 1-6.

1.13 Capital and Operating Costs – *Cont'd*

Table 1-6: LOM Sustaining Capital Costs

Area	Total Sustaining Capital ($ billions)
Water Treatment, TSF Construction & Water Collection	0.1
Mine Incremental Capital / Equipment Leases	0.3
General Sustaining Capital	0.2
Total	**0.6**

Note: totals may not add due to rounding

Onsite operating costs comprise mining, processing and general and administration. Average onsite costs for the project are summarized in Table 1-7.

Table 1-7: Onsite Operating Costs

Area	Operating Cost ($/t Milled)
Mining	4.53
Processing	4.65
G&A	0.79
Total Onsite	**9.97**

Note: totals may not add due to rounding

Offsite costs include copper concentrate transportation costs, smelter fees and deductions, and royalty payments. Average offsite costs are US$0.39/lb.

1.14 Economic Analysis

Metal prices are based on street consensus metal pricing as of Q4 2019 and long-term foreign exchange rates are based on Taseko's expectations informed by historical exchange rates as shown in Table 1-8. A discounted net present value (NPV) cashflow model using a discount rate of 8% is used for the valuation basis with an effective date of December 31, 2019. Results of the valuation are presented on a 100% basis and assume no debt financing costs except for mining equipment leases. All values are in Canadian dollars unless otherwise noted.

Table 1-8: Street Consensus Long-Term Metal Pricing and Foreign Exchange Rate

Long-Term Forecasts	Metal Price
Copper Price	US$3.10/lb
Gold Price	US$1,350/oz
Silver Price	US$18.00/oz
Foreign Exchange	US$0.80 : CAD$1.00

Pre-tax economic indicators for the project are presented in Table 1-9.

Table 1-9: Pre-Tax Economic Valuation

Economic Indicator	Value
Average Annual Pre-Tax Cash Flow	$270 million
Pre-Tax NPV at 8%	$1.3 billion
Internal Rate of Return	18%
Payback Period	4.2 years

The project after-tax economic indicators, assuming current federal and provincial tax laws are in force are presented in Table 1-10.

Table 1-10: After-Tax Economic Valuation

Economic Indicator	Value
After-Tax NPV at 8%	$0.7 billion
After-Tax IRR	14%

1.15 Conclusions and Recommendations

Proven and probable mineral reserves total 817 million tonnes grading 0.29% Cu Eq. The reserve pit design is based on a copper price of US$2.40/lb, gold price of US$1,000/oz, silver price of US$13.50/oz, exchange rate of US$0.80 : C$1.00, and a 0.17% Cu cut-off.

The mineral reserve supports 25 years of operation at a design milling rate of 90,000 tpd with average annual production of approximately 180 million pounds of copper and total gold and silver production of 440 thousand ounces and 19 million ounces respectively. The average strip ratio is 1.4:1.

The concentrate is clean, not complex, and contains no penalty level deleterious elements.

The project has a pre-production capital cost of C$1.3 billion and provides a pre-tax NPV at an 8.0% discount rate of C$1.3 billion using US$3.10/lb Cu, US$1,350/oz Au, US$18/oz Ag and a foreign exchange rate of US$0.80 : C$1.00. The payback period is just over 4 years from start of production.

In the author's opinion the geological data, project design, capital and operating cost estimates and marketing assumptions used are appropriate for purposes of defining and demonstrating resources and reserves as prescribed by National Instrument 43-101.

The author recommends that additional engineering opportunities be evaluated, specifically the opportunity to reduce lime consumption and optimize grind and reagent selection. Independent of this work the author recommends that the project advance through the environmental assessment and permitting processes.

SECTION 2

INTRODUCTION

SECTION 2: INTRODUCTION

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2.1 Introduction

This technical report has been prepared by Taseko Mines Limited (Taseko). Taseko was incorporated on April 15, 1966, pursuant to the Company Act of the Province of British Columbia. This corporate legislation was superseded in 2004 by the British Columbia Business Corporations Act which is now the corporate law statute that governs Taseko.

The Company's principal subsidiaries are listed in Table 2-1.

Table 2-1: Subsidiaries of Taseko Mines Limited

	Jurisdiction of Incorporation	Ownership
Gibraltar Mines Ltd. [1]	British Columbia	100%
Aley Corporation	British Columbia	100%
Yellowhead Mining Inc.	British Columbia	100%
Florence Copper Inc. [2]	Arizona	100%

1. Taseko owns 100% of Gibraltar Mines Ltd, which in turn owns 75% of the Gibraltar Joint Venture.
2. Taseko owns 100% of Curis Resources Ltd, which owns 100% of Curis Holdings (Canada) Ltd which owns 100% of Florence Copper Inc.

Taseko Mines Limited owns 100% of the New Prosperity Project and Gibraltar Mines Ltd. owns 100% of the Harmony Gold Project.

On March 31, 2010, the Company established a joint venture with Cariboo Copper Corp. ("Cariboo") over the Gibraltar mine, whereby Cariboo acquired a 25% interest in the mine and Gibraltar retained a 75% interest. On November 20, 2014, the Company acquired a 100% interest in the Florence Copper Project though the acquisition of Curis Resources Ltd. On February 15, 2019, the Company acquired a 100% interest in Yellowhead Mining Inc.

The head office of Taseko is located at 15th Floor, 1040 West Georgia Street, Vancouver, British Columbia, Canada V6E 4H1, telephone (778) 373-4533, facsimile (778) 373-4534. The Company's legal registered office is in care of its Canadian attorneys McMillan LLP, Suite 1500, 1055 West Georgia Street, Vancouver, British Columbia, Canada V6E 4N7, telephone (604) 689-9111, facsimile (604) 685-7084.

The purpose of this report is to summarize the pre-feasibility study and document the mineral reserve estimate announced in Taseko's news release dated January 16, 2020 in the format prescribed in National Instrument 43-101, Form 43-101F1.

The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Taseko at the time of preparation of this report;
- Assumptions, conditions, and qualifications as set forth in this report;
- Data, reports, and opinions supplied by Taseko and other third party sources listed as references.

2.1 Introduction – *Cont'd*

Contributing consultants, Allnorth, Knight Piesold, BQE Water, Geosim Services Inc., G&T Metallurgical Services Ltd., FLSmidth, and Cohesion are independent of both Yellowhead Mining Inc. and Taseko Mines Limited and have no beneficial interest in the Yellowhead Copper Project. Fees for technical input are not dependent in whole or in part on any prior or future engagement or understanding resulting from the conclusions of resulting reports. Taseko has relied upon technical reports from these consultants to derive relevant aspects of this report. Reports developed by each consultant have been supplied to each of the other consultants as appropriate to support their own work and help derive the information, data and results that make up the content of this report.

Cohesion Consulting provided a review of oversight on the sampling, chain of custody, assaying and geological database management of this project. GeoSim Services Inc. carried out the geostatistics, built the geological block model and estimated the mineral resource. Taseko relied on the geological block model supplied by GeoSim Services Inc. in order to carry out pit design and mine planning in support of the mineral reserve estimate. Knight Piesold supplied the geotechnical parameters used in the pit and plant design, completed the tailings storage facility design, the water balance and the water management layouts. BQE Water completed the water treatment plant design and capital and operating cost estimates. Allnorth completed the concentrator and plant site infrastructure and rail load-out design and capital cost estimate. Metallurgical test work programs that have contributed to the performance predictions have been complete by G & T Metallurgical Services Ltd. (G&T), of Kamloops, BC. FLSmidth (FLS) of Bethlehem, PA, USA performed laboratory comminution test work, which was further reviewed by KWM Consulting Inc (KWM) of Vancouver, BC to confirm grinding power requirements.

Richard Weymark, P.Eng., MBA has provided oversight for this study and supervised the preparation of this full report as the qualified person (QP). Other authors contributing sections of this report Adil Cheema, P.Eng., Jeremy Guichon, P.Eng., Eric Titley, P. Geo., and Ronald G. Simpson, P.Geo.

Mr. Weymark has supervised the preparation of all sections of this report with a primary focus on sections 1 through 5, 18, 19, 20, and 23 through 26 of this report and has reviewed the methods used to determine the pit design, the long range mine plan, capital and operating cost estimates, and directed the updated economic evaluation. Mr. Weymark's current position is Chief Engineer and he has direct knowledge of the project, having been employed by Taseko Mines since July 2018. Mr. Weymark visited the site on September 11, 2019 to review the general site topography, drill core storage and Vavenby rail load-out site.

2.1 Introduction – *Cont'd*

Mr. Cheema has supervised the preparation of sections 13 and 17 of this report and has reviewed the laboratory analytical methods as well as the test work methodology used to determine the metallurgical and recovery projections used in the economic analysis accompanying this report. Mr. Cheema's current position is Senior Process Engineering and he has been employed by Taseko Mines since March of 2019.

Mr. Guichon has supervised the preparation of sections 15, 16, 21 and 22 of this report, and has reviewed the mine operating costs, mine equipment capital costs, the mineral resource estimate and the economic analysis. Mr. Guichon's current position is Senior Mine Engineer, and he has been employed by Taseko Mines since June 2012.

Mr. Titley has supervised the preparation of sections 6 through 12 of this report. Mr. Titley is a consultant with Cohesion Consulting.

Mr. Simpson has supervised the preparation of section 14 of this report. He conducted a site visit to the project on July 11 and 12, 2011. The purpose of the visit was to review the geology and mineralization encountered in the drillholes completed to date. In addition, drilling, sampling, quality assurance/quality control (QA/QC), sample preparation and analytical protocols and procedures, and database structure were reviewed.

All measurement units used in this report are metric, and currency is expressed in Canadian dollars unless stated otherwise.

SECTION 3

RELIANCE ON OTHER EXPERTS

SECTION 3: RELIANCE ON OTHER EXPERTS

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3.1 Reliance on Other Experts

Standard professional procedures have been followed in the preparation of this technical report. Data used in this report has been verified where possible and the authors have no reason to believe that data was not collected in a professional manner and no information has been withheld that would affect the conclusions of this report. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Taseko as of the effective date of this report;
- Assumptions, conditions, and qualifications as stated in this report.

For the purposes of this report, the authors have relied on title and property ownership provided by the Mineral Titles Branch, Mines and Mineral Resources Division of the BC Ministry of Energy and Mines and Petroleum Resources as of December 31, 2019 to confirm Taseko's internal tenure tracking system. The Mineral Title Branch system is an on-line viewer accessible by anyone with a free miner's certificate. This tenure information applies to section 4.2 of this report.

Standard tax calculations for BC based mining projects were reviewed internally in December 2019 by Taseko's CFO Bryce Hamming CFA, CPA, CA, an accountant with knowledge in Canadian mining taxation, and were incorporated into the cashflow and tax related information referenced in section 22.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

SECTION 4

PROPERTY DESCRIPTION AND LOCATION

SECTION 4: PROPERTY DESCRIPTION AND LOCATION

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4.1 Property Description and Location

The Yellowhead claims are located in the Thompson-Nicola area of BC, approximately 150 km northeast of Kamloops and are centered at latitude 51°30’ north and longitude 119°48’ west (Figure 4-1) in the Kamloops Mining Division. Clearwater, the largest community in the project area is 124 km north of Kamloops, along the Yellowhead Highway route (Highway #5). Vavenby, the closest community to the project area, is 27 km west of Clearwater along Highway #5.

4.1 Property Description and Location – *Cont'd*

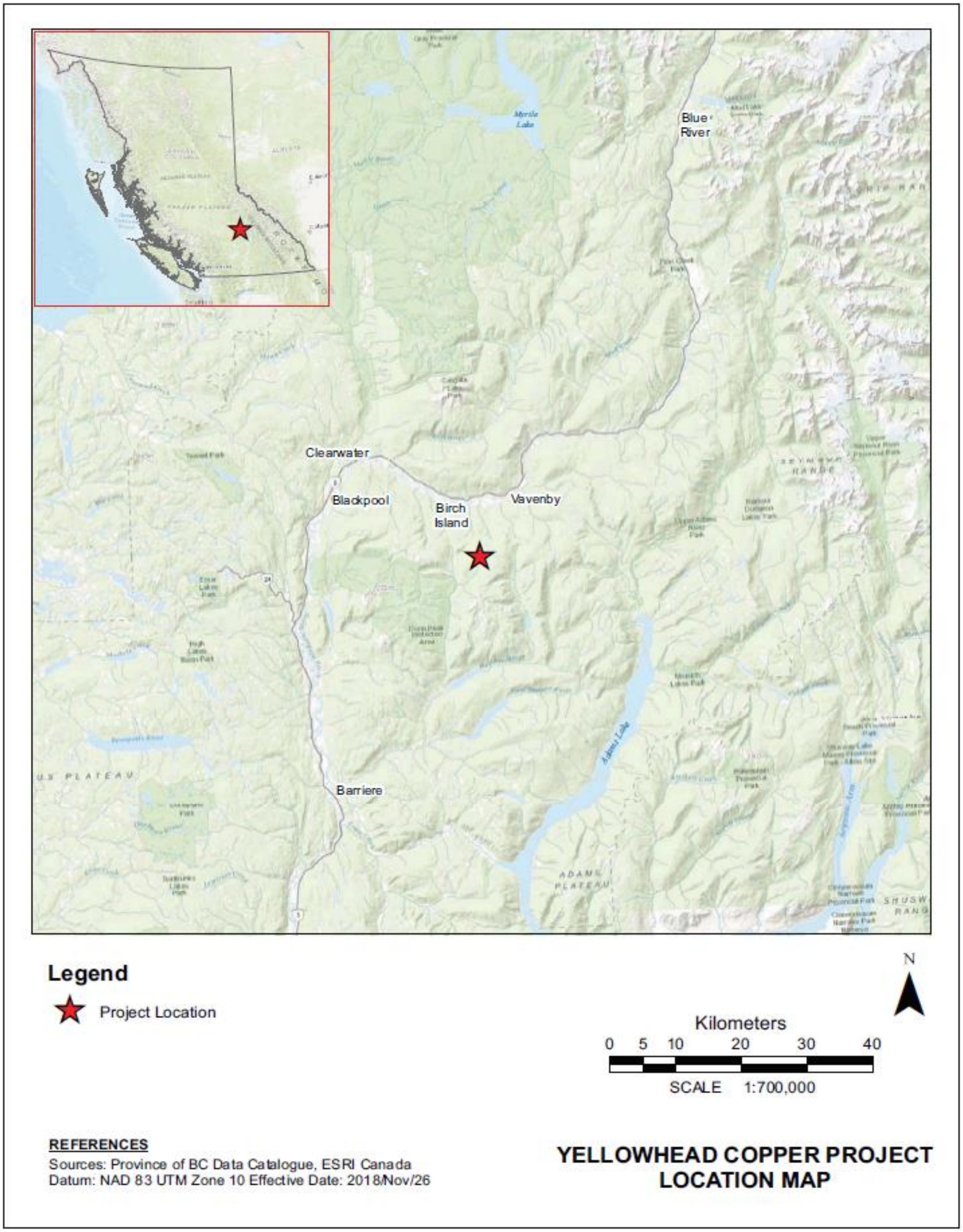

Figure 4-1: Project Location

4.2 Land Tenure

Taseko, through its wholly owned subsidiary Yellowhead Mining Inc. (FMC 285998), is the 100% owner of the Yellowhead mineral claims.

The property consists of 131 mineral claims (97 cell claims and 34 legacy claims) covering 42,636 hectares as summarized in Table 4-1 and shown in Figure 4-2.

Table 4-1: Mineral Titles

Tenure Type	Number	Area (ha)
Claims	131	42,636
Leases	0	0
Total	131	42,636

All mineral claims are in good standing and details of each claim are provided in Table 4-2. There are three parcels of fee simple land located 2.5 km west of Vavenby where the rail load-out facility will be located.

None of the 97 cell claims are subject to any royalties. 3 unconverted legacy claims (220877, 220878, 220879), and 3 converted legacy claims (513235, 513237, 513239), are subject to a 2.5% NSR royalty to XStrata. 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 royalty, capped at C$3 million, subject to inflation.

An application has been submitted to the BC Mineral Titles Office to convert 40 claims to a mining lease. These claims are outlined in Figure 4-2 and encompass the deposit area and mill footprint.

4.2 Land Tenure – *Cont'd*

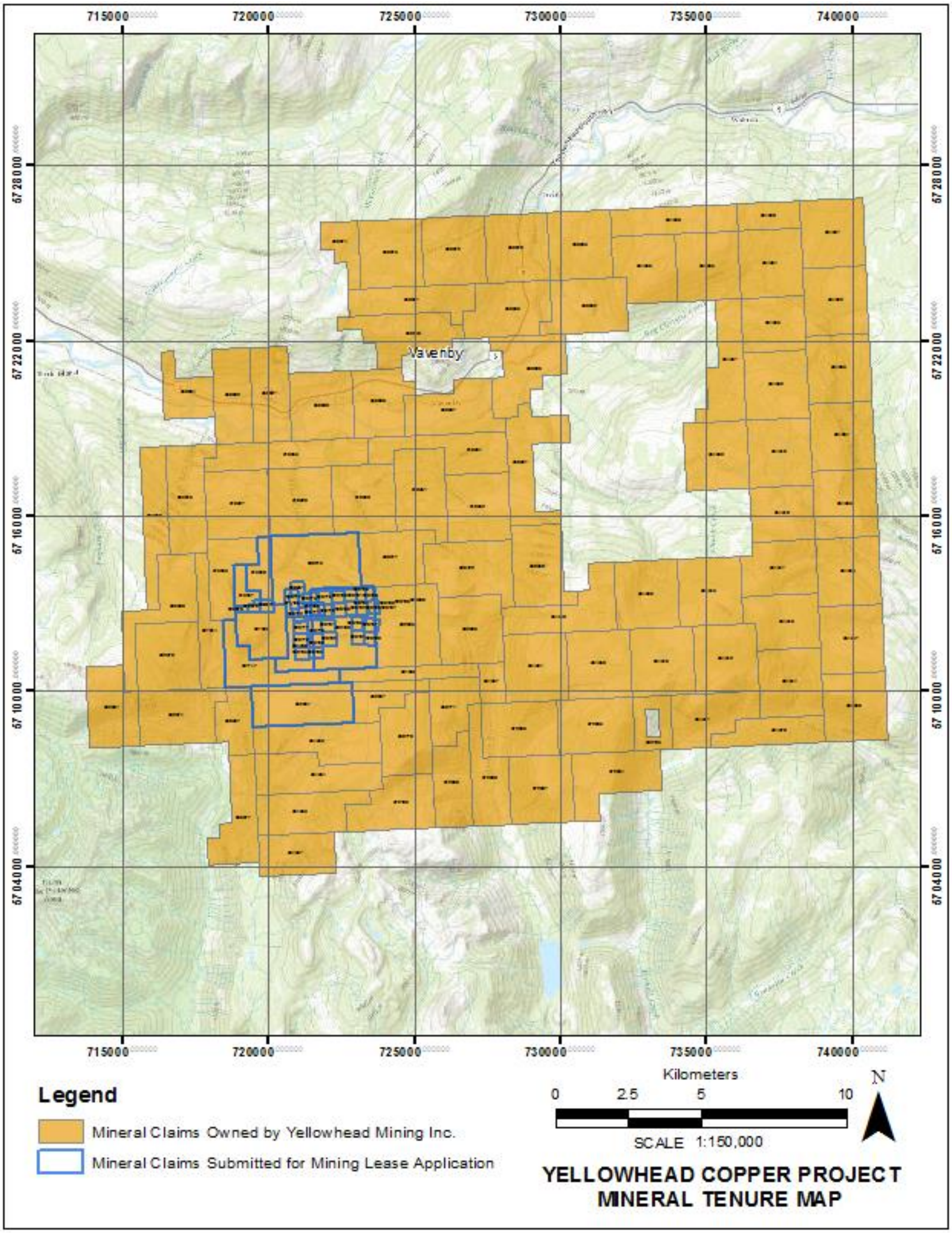

Figure 4-2: Mineral Claims

4.2 Land Tenure – *Cont'd*

Table 4-2: Yellowhead Mineral Claims

Title #	Name	Type	Issue Date	Good To Date	Area (Ha)
220771	HAIL 1	Claim	1966/Jul/13	2024/Nov/03	25.0
220772	HAIL 2	Claim	1966/Jul/13	2024/Nov/03	25.0
220773	HAIL 3	Claim	1966/Jul/13	2024/Nov/03	25.0
220774	HAIL 4	Claim	1966/Jul/13	2024/Nov/03	25.0
220775	HAIL 5	Claim	1966/Jul/13	2024/Nov/03	25.0
220776	HAIL 6	Claim	1966/Jul/13	2024/Nov/03	25.0
220777	HAIL 7	Claim	1966/Jul/13	2024/Nov/03	25.0
220778	HAIL 8	Claim	1966/Jul/13	2024/Nov/03	25.0
220779	HAIL 9	Claim	1966/Jul/13	2024/Nov/03	25.0
220780	HAIL 10	Claim	1966/Jul/13	2024/Nov/03	25.0
220781	HAIL 11	Claim	1966/Jul/13	2024/Nov/03	25.0
220782	HAIL 12	Claim	1966/Jul/13	2024/Nov/03	25.0
220783	HAIL 15	Claim	1966/Jul/13	2024/Nov/03	25.0
220784	HAIL 16	Claim	1966/Jul/13	2024/Nov/03	25.0
220785	HAIL 17	Claim	1966/Jul/13	2024/Nov/03	25.0
220786	HAIL 19	Claim	1966/Jul/13	2024/Nov/03	25.0
220787	HAIL 29	Claim	1966/Jul/13	2024/Nov/03	25.0
220788	HAIL 30	Claim	1966/Jul/13	2024/Nov/03	25.0
220789	HAIL 31	Claim	1966/Jul/13	2024/Nov/03	25.0
220790	HAIL 32	Claim	1966/Jul/13	2024/Nov/03	25.0
220791	HAIL 33	Claim	1966/Jul/13	2024/Nov/03	25.0
220792	HAIL 34	Claim	1966/Jul/13	2024/Nov/03	25.0
220793	HAIL 35	Claim	1966/Jul/13	2024/Nov/03	25.0
220794	HAIL 36	Claim	1966/Jul/13	2024/Nov/03	25.0
220795	HAIL 99	Claim	1966/Jul/22	2024/Nov/03	25.0
220796	HAIL 100	Claim	1966/Jul/22	2024/Nov/03	25.0
220797	HAIL 102	Claim	1966/Jul/22	2024/Nov/03	25.0
220798	HAIL 104	Claim	1966/Jul/22	2024/Nov/03	25.0
220799	HAIL 106	Claim	1966/Jul/22	2024/Nov/03	25.0
220800	HAIL 108	Claim	1966/Jul/22	2024/Nov/03	25.0
220877	SUE #19	Claim	1967/Sep/28	2024/Nov/03	25.0
220878	SUE #20	Claim	1967/Sep/28	2024/Nov/03	25.0
220879	SUE #21	Claim	1967/Sep/28	2024/Nov/03	25.0
220961	HAIL 590	Claim	1968/Jul/31	2024/Nov/03	25.0

4.2 Land Tenure – *Cont'd*

Table 4-2: Yellowhead Mineral Claims – *Cont'd*

Title #	Name	Type	Issue Date	Good To Date	Area (Ha)
501147	HARPER 1	Claim	2005/Jan/12	2024/Nov/03	342.0
501225		Claim	2005/Jan/12	2024/Nov/03	301.7
501608	HARPER 2	Claim	2005/Jan/12	2024/Nov/03	221.3
501799		Claim	2005/Jan/12	2024/Nov/03	181.0
502498		Claim	2005/Jan/12	2024/Nov/03	583.3
502603		Claim	2005/Jan/12	2024/Nov/03	603.4
502606		Claim	2005/Jan/12	2024/Nov/03	502.9
506422		Claim	2005/Feb/09	2024/Nov/03	563.0
509215		Claim	2005/Mar/18	2024/Nov/03	603.2
509217		Claim	2005/Mar/18	2024/Nov/03	422.2
513235		Claim	2005/May/24	2024/Nov/03	321.7
513237		Claim	2005/May/24	2024/Nov/03	80.4
513239		Claim	2005/May/24	2024/Nov/03	140.7
514183		Claim	2005/Jun/09	2024/Nov/03	40.2
517483		Claim	2005/Jul/12	2024/Nov/03	20.1
519327	TOM1	Claim	2005/Aug/25	2024/Nov/03	502.4
519329	TOM2	Claim	2005/Aug/25	2024/Nov/03	502.4
519330	TOM3	Claim	2005/Aug/25	2024/Nov/03	502.4
519331	TOM4	Claim	2005/Aug/25	2024/Nov/03	502.4
519332	TOM5	Claim	2005/Aug/25	2024/Nov/03	502.5
519333	TOM6	Claim	2005/Aug/25	2024/Nov/03	502.3
519334	TOM7	Claim	2005/Aug/25	2024/Nov/03	462.1
530337	SUN 1	Claim	2006/Mar/20	2024/Nov/03	502.3
530338	SUN 2	Claim	2006/Mar/20	2024/Nov/03	502.7
532054	HAR1	Claim	2006/Apr/13	2024/Nov/03	483.0
532057	HAR2	Claim	2006/Apr/13	2024/Nov/03	241.5
538962		Claim	2006/Aug/09	2024/Nov/03	501.8
538963		Claim	2006/Aug/09	2024/Nov/03	501.6
538966		Claim	2006/Aug/09	2024/Nov/03	501.8
538968		Claim	2006/Aug/09	2024/Nov/03	501.9
538970		Claim	2006/Aug/09	2024/Nov/03	501.6
538971		Claim	2006/Aug/09	2024/Nov/03	421.5
538972		Claim	2006/Aug/09	2024/Nov/03	501.6
538973		Claim	2006/Aug/09	2024/Nov/03	501.6

4.2 Land Tenure – *Cont'd*

Table 4-2: Yellowhead Mineral Claims – *Cont'd*

Title #	Name	Type	Issue Date	Good To Date	Area (Ha)
538974		Claim	2006/Aug/09	2024/Nov/03	200.6
538996		Claim	2006/Aug/09	2024/Nov/03	502.0
538997		Claim	2006/Aug/09	2024/Nov/03	502.1
538999		Claim	2006/Aug/09	2024/Nov/03	421.8
539000		Claim	2006/Aug/09	2024/Nov/03	502.1
539001		Claim	2006/Aug/09	2024/Nov/03	421.7
539002		Claim	2006/Aug/09	2024/Nov/03	421.7
539004		Claim	2006/Aug/09	2024/Nov/03	281.1
539770		Claim	2006/Aug/22	2024/Nov/03	442.8
539771		Claim	2006/Aug/22	2024/Nov/03	322.0
564330	GRAF1	Claim	2007/Aug/09	2024/Nov/03	503.0
564331	GRAF2	Claim	2007/Aug/09	2024/Nov/03	503.0
564333	DUNN1	Claim	2007/Aug/09	2024/Nov/03	503.2
564334	DUNN2	Claim	2007/Aug/09	2024/Nov/03	503.3
564335	DUNN3	Claim	2007/Aug/09	2024/Nov/03	463.2
564337	DUNN4	Claim	2007/Aug/09	2024/Nov/03	362.6
564338	GRAF3	Claim	2007/Aug/09	2024/Nov/03	502.8
564339	GRAF4	Claim	2007/Aug/09	2024/Nov/03	502.8
564340	GRAF5	Claim	2007/Aug/09	2024/Nov/03	503.0
564341	GRAF6	Claim	2007/Aug/09	2024/Nov/03	442.8
564342	GRAF7	Claim	2007/Aug/09	2024/Nov/03	503.0
564343	GRAF8	Claim	2007/Aug/09	2024/Nov/03	502.8
564344	GRAF9	Claim	2007/Aug/09	2024/Nov/03	503.1
564346	GRAF10	Claim	2007/Aug/09	2024/Nov/03	442.5
564347	GRAF11	Claim	2007/Aug/09	2024/Nov/03	462.5
564348	GRAF12	Claim	2007/Aug/09	2024/Nov/03	402.0
564349	GRAF13	Claim	2007/Aug/09	2024/Nov/03	502.3
564350	GRAF14	Claim	2007/Aug/09	2024/Nov/03	502.3
564351	GRAF15	Claim	2007/Aug/09	2024/Nov/03	461.9
564352	GRAF16	Claim	2007/Aug/09	2024/Nov/03	502.1
564353	GRAF17	Claim	2007/Aug/09	2024/Nov/03	401.5
564354	GRAF18	Claim	2007/Aug/09	2024/Nov/03	501.7
564355	GRAF19	Claim	2007/Aug/09	2024/Nov/03	501.7
564356	GRAF20	Claim	2007/Aug/09	2024/Nov/03	461.6

4.2 Land Tenure – *Cont'd*

Table 4-2: Yellowhead Mineral Claims – *Cont'd*

Title #	Name	Type	Issue Date	Good To Date	Area (Ha)
564357	DUNN5	Claim	2007/Aug/09	2024/Nov/03	120.7
564358	GRAF21	Claim	2007/Aug/09	2024/Nov/03	401.2
564360	GRAF22	Claim	2007/Aug/09	2024/Nov/03	200.6
564361	GRAF23	Claim	2007/Aug/09	2024/Nov/03	501.6
564362	GRAF24	Claim	2007/Aug/09	2024/Nov/03	501.8
564363	GRAF25	Claim	2007/Aug/09	2024/Nov/03	502.1
564364	GRAF26	Claim	2007/Aug/09	2024/Nov/03	502.3
564365	GRAF27	Claim	2007/Aug/09	2024/Nov/03	502.5
564366	GRAF28	Claim	2007/Aug/09	2024/Nov/03	502.7
564367	GRAF29	Claim	2007/Aug/09	2024/Nov/03	503.0
564368	GRAF30	Claim	2007/Aug/09	2024/Nov/03	503.2
564370	GRAF31	Claim	2007/Aug/09	2024/Nov/03	322.1
569337		Claim	2007/Nov/04	2024/Nov/03	261.6
572094	SANDRA1	Claim	2007/Dec/18	2024/Nov/03	503.4
572095	SANDRA2	Claim	2007/Dec/18	2024/Nov/03	483.1
572096	SANDRA3	Claim	2007/Dec/18	2024/Nov/03	483.1
572097	SANDRA4	Claim	2007/Dec/18	2024/Nov/03	503.4
572098	CHELSEA	Claim	2007/Dec/18	2024/Nov/03	382.6
572099	STEPHANIE	Claim	2007/Dec/18	2024/Nov/03	382.6
572100	ISABEL	Claim	2007/Dec/18	2024/Nov/03	463.2
582783		Claim	2008/Apr/25	2024/Nov/03	201.3
592574		Claim	2008/Oct/05	2024/Nov/03	503.1
592579		Claim	2008/Oct/05	2024/Nov/03	502.9
592580		Claim	2008/Oct/05	2024/Nov/03	462.5
592581		Claim	2008/Oct/05	2024/Nov/03	442.7
606977	DUNN	Claim	2009/Jul/03	2024/Nov/03	415.4
627844	HARP	Claim	2009/Sep/03	2024/Nov/03	301.7
663643		Claim	2009/Nov/02	2024/Nov/03	502.4
663658		Claim	2009/Nov/02	2024/Nov/03	402.0

4.3 Environmental Liabilities

The Yellowhead property is subject to environmental liabilities related to the rehabilitation of drill sites and exploration access roads associated with the work permits received for previous exploration drilling programs. Funds to cover the expense of these reclamation activities are held in trust and are fully recoverable once the site has been rehabilitated to the satisfaction of the Inspector of Mines. There are no other environmental liabilities to which the property is subject.

4.4 Permits Obtained and To Be Acquired

Section 20 provides the list of major permits, licenses, approvals, consents and material authorizations required to occupy, use, construct and operate the project.

SECTION 5

ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

SECTION 5: ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

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5.1 Accessibility

The Yellowhead Copper Project is accessed via the Yellowhead Highway (Highway #5), 150 km by road north of Kamloops, which is serviced with daily flights from Vancouver and Calgary.

As shown in Figure 5-1, the proposed operational 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 18.5 km network of Forest Service Roads (FSRs) that climb up to the mine site from the Vavenby Bridge crossing the North Thompson River at Vavenby. A 2.5 km road extension will need to be constructed from the FSR network to the mine site.

During the construction phase, oversized loads will require alternate access across the North Thompson River as the Vavenby Bridge has not been designed to accommodate such loads. This route crosses the North Thompson River at the BILCR bridge then follows the previously described access.

The FSRs will be upgraded where required. The road will be in frequent use during the operations phase for the transport of concentrate from the mine site to the rail load out facility and transportation of personnel, goods and services.

5.1 Accessibility – *Cont'd*

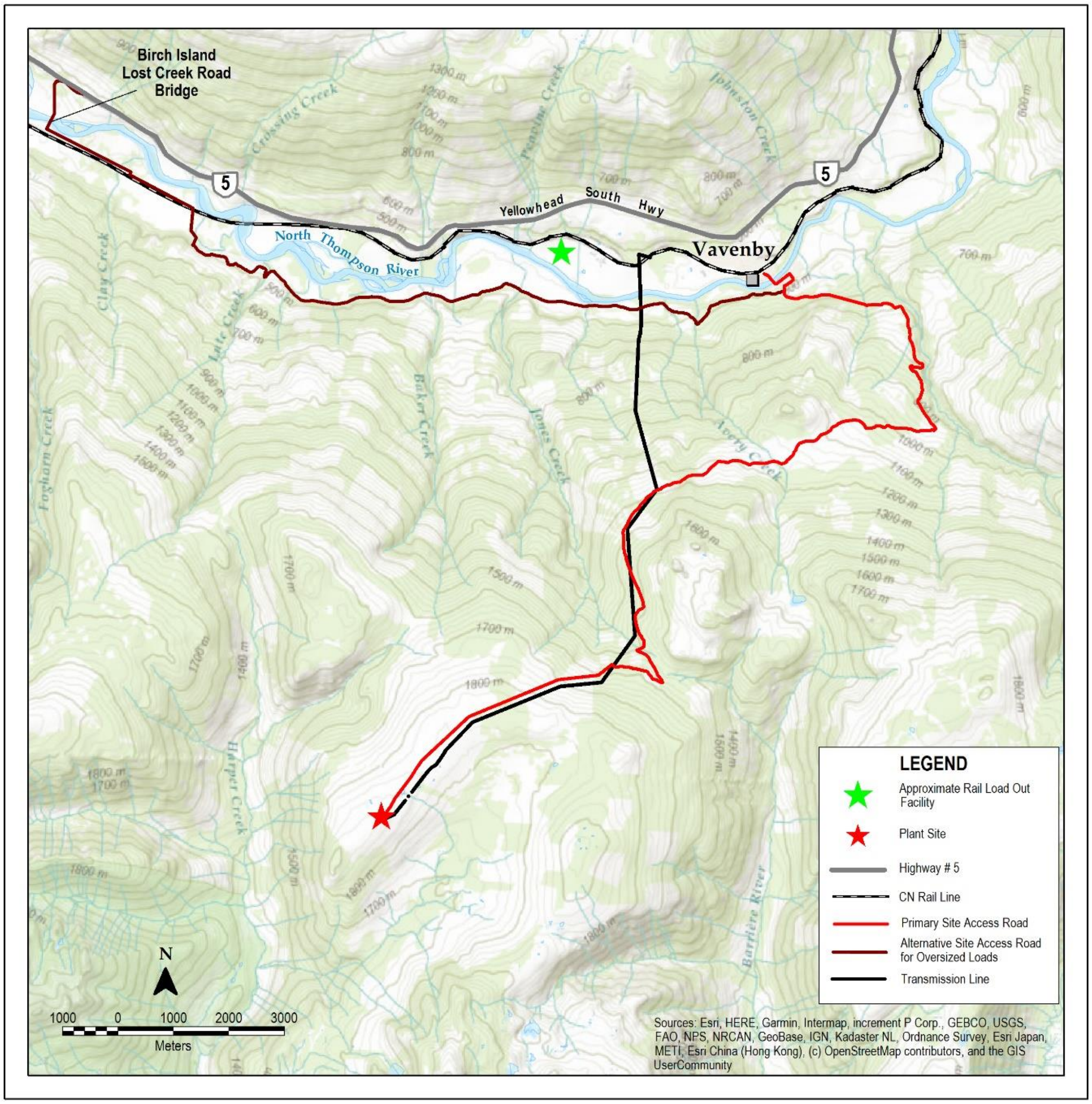

Figure 5-1: Site Access

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 on site range from highs of +26°C to lows of -35°C. The mean annual precipitation is 1,259 mm at an elevation of 1,837 masl, with about 40% falling as snow and 60% falling as rain. 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. The mean annual wind speed is approximately 1.6 m/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%.

5.3 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 1,100 masl at the floor of the Harper Creek Valley to 1,900 masl at the ridges surrounding the TSF area.

The average elevation of the open pit area and plant site is 1,800 masl. The area has been glaciated and mountain tops are typically rounded. The project is covered in coniferous forest and has undergone extensive logging.

5.4 Local Resources

In 2016, the population of the Thompson-Nicola Regional District numbered 133,000 residents. Kamloops is the largest centre in the area and has a population of 90,000. With several operating mines in the area Kamloops is a regional mining hub home to many suppliers, consultants, and contractors that service the mining industry.

Accommodation for mine employees is available in the nearby towns of Clearwater, Vavenby, Barrière, and surrounding district which have a combined population of approximately 6,000. With the recent decline in the forestry sector and the closure of several mills in the North Thompson Valley there is a local workforce with industrial experience in need of economic development.

The project will give employment preference to people from the North Thompson Valley. Vavenby has served as the local base for the project's exploration activities but provides limited facilities or services at this time. Industrial activities within the regional area include forestry and the CN rail line passing through Vavenby. A sawmill in Vavenby has shutdown as of the writing of this report.

5.5 Infrastructure

The Yellowhead Highway, the CNR transcontinental main line, and a main BC Hydro 138 kVA transmission line all pass approximately 8 km north of the project area.

Other than the existing network of FSRs, there are no services or utilities currently routed to the immediate project site.

The area's established infrastructure precludes the need for any major off-site infrastructure developments to service the project other than upgrading and tying into the existing BC Hydro 138 kVa powerline in Vavenby and upgrading and adding a new section to the site access road.

YMI holds sufficient mineral tenure to accommodate mining operations, tailings storage areas, waste disposal areas, processing facilities and site infrastructure.

YMI owns a property with a rail siding 2.5 km west of Vavenby. The rail load-out facility will be located here, approximately 25 km by road from the project site.

SECTION 6

HISTORY

SECTION 6: HISTORY

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6.1 Introduction

This section describes the historical mineral exploration work conducted on the property prior to YMI obtaining control of the property in late 2005. It also includes a summary of mineral resource and mineral reserve estimates that pre-date the current mineral resource and mineral reserve estimate detailed in this report.

Prospecting and geochemical reconnaissance led to the discovery of copper mineralization in the immediate vicinity of the deposit in 1966. In 1967, the initial discovery was followed-up by extensive prospecting, line cutting, road building, surface geochemical sampling, geological mapping, geophysics, trenching and diamond drilling programs.

Noranda Exploration Company (Noranda) and Québec Cartier Mining Company (QCM), a 100% wholly owned subsidiary of US Steel, staked claims in the deposit area in 1965 and 1966 respectively. This resulted in the area west of the Harper Creek tributary belonging to Noranda (Harper Creek claims) and east of it to QCM (Hail claims). The two companies worked independently on their properties from 1966 until 1970. In late 1970, the companies formed a joint venture, which explored their contiguous copper deposits until 1974.

Work on the property continued for nine consecutive years and included extensive drilling on the deposit, a number of expanded geophysical and geochemical surveys and some drilling of other targets on the property. By the end of 1974, work was curtailed on the original showing. Sporadic prospecting, geochemical, geophysical and geological work by a number of operators continued in other outlying areas of the current property.

Additional work in the deposit area occurred in 1986 and 1996. This included trenching, core resampling and metallurgical testing and additional drilling. No further drilling on the deposit area took place until 2006.

6.2 Historical Surface Exploration

(a) Noranda Exploration Company Ltd.

Noranda discovered copper mineralization at the headwaters of Baker Creek and Jones Creek on the Harper Creek claims in 1966 by prospecting and stream sediment sampling which had indicated higher levels of cadmium, copper, aluminum and iron in the stream sediments. Upon completion of an orientation survey the following year, Noranda surveyed a soil sample grid. Extension of the soil grid to the south and west and cross line infilling took place in 1968 and 1970.

Between 1967 and 1971, Noranda undertook geophysical surveys comprising 11.5 km in 9 lines of magnetometer, 51.5 km in 28 lines of very low frequency – electromagnetic (VLF-EM), and 58 km in 8 lines of induced polarization (IP). An IP survey conducted after completion of drilling informed as a test survey.

6.2 Historical Surface Exploration – *Cont'd*

(b) Québec Cartier Mining Company

In 1966, QCM discovered copper mineralization at the headwaters of a tributary of Harper Creek on the Hail Claims through a program of prospecting and stream sediment sampling similar to that undertaken by Noranda.

In 1967, QCM established a 13-line grid totaling 129 km in an area broadly defined by the results of the silt-sampling program. Analysis of 2,500 B-horizon soil samples collected on this grid was for copper and zinc. A 5 km extension of a local logging road facilitated creation of seven trenches on the western side of the Hail Claims. Excavation of 1,500 m^3 of material and the taking of 31 channel samples along 3 m bedrock lengths resulted. A ground magnetic survey conducted the same year included 9,000 vertical component observations at 15 m intervals over 137 km.

6.2 Historical Surface Exploration – *Cont'd*

(c) Noranda / Québec Cartier Joint Venture

Noranda and QCM formed a joint venture with Noranda as the project operator in late 1970 for continued exploration on the combined properties.

A soil orientation survey on the QCM grid in fall 1970 warranted a check sampling comparison of the results for the two grid systems. In 1971, Noranda re-sampled a portion of the QCM grid. Copper and zinc analysis of all soil samples and the analysis of two lines for molybdenum took place.

In 1972, exploration expanded out from the main deposit to the southwest, south, and north. Work included detailed stream sediment sampling, reconnaissance geological mapping, soil sampling, and geophysical surveying. Internal preliminary feasibility work conducted that year evaluated open pit designs of the combined Noranda and QCM deposits.

In 1973, groundwork shifted back to the deposit area, as newly constructed logging roads opened up new areas. A total of 22 km of VLF-EM surveying took place on new or re-established grids in that year.

In 1974, geological mapping of newly cut logging roads and relogging of historical drill core was the only work undertaken. Upgrades to the internal prefeasibility studies using revised parameters took place in 1973 and 1974. Results of these studies are unknown.

6.2 Historical Surface Exploration – *Cont'd*

(d) Aurun Mines Ltd.

In April 1986, Aurun Mines Ltd. (Aurun) signed an option agreement with QCM to investigate the potential of both small higher-grade and large lower-grade copper deposits and to test for the presence of precious metals in the massive sulphide layers on the QCM claims. Assessments also considered the significance of titanium-bearing minerals and the possibility of leaching low-grade copper mineralization. Work proceeded through sampling of historical trenches and selected historical drill core. Results of Au and Ag analysis showed the potential for modest credits to be attributable to these metals.

Aurun also commissioned a pre-feasibility study by Phillips Barratt Kaiser Engineering Ltd. in April 1986 that considered both the eastern QCM and western Noranda deposits. In July, 1991 QCM officially terminated the option agreement with Aurun (insolvent and in receivership as of 2014).

6.2 Historical Surface Exploration – *Cont'd*

(e) American Comstock Exploration Ltd

American Comstock Exploration Ltd (Comstock) purchased the Noranda claims and acquired an option on the QCM claims in 1996 but conducted no surface exploration, only drilling.

6.2 Historical Surface Exploration – *Cont'd*

(f) Other Operators

Several other historical operators performed exploration within the current bounds of the property but well outside the deposit area between 1970 and 2005. Table 6-1 lists the technical assessment reports of mineral exploration and development performed by all historical workers on the property as filed in the government of British Columbia Assessment Report Indexing System (ARIS).

Table 6-1: Historical ARIS Reports Filed on the Property

ARIS	Year	Area/Claim(s)	Operator	Work Program
1035 1612	1967	Hail Hail L, M, N & O	Québec Cartier Mining Company	Geochemical & geological Drilling, geological, geochemical, geophysical
2988	1970	VH	Royal Canadian Ventures Ltd.	Geochemical & geophysical
3141	1971	PY	Supertest Investments & Petroleum Ltd.	Line cutting
3151	1970	PY		Geochemical
3195 3430 3525	1971	VM #2 Hilltop, Bob, Hissy, Fill VM & VA	Royal Canadian Ventures Ltd. Dynasty Explorations Limited Royal Canadian Ventures Ltd.	Geochemical Geological, geophysical, geochemical Geological & geochemical
3781 3941	1972	PY CAP, PAC	Supertest Investments & Petroleum Ltd. MacDonald, WE	Line cutting Line cutting
5502	1975	Bullion & Pat	H Doyle & J Arden	Prospecting
5909 5929 6161	1976	Vav Lucky Strike Toreador 1	John H Kruzick JA Fennell Torwest Resources (1962) Ltd (NPL)	Geological Drilling Geological & geophysical
6220 6252 6317 6383	1977	Green Lake Have Vav	Copper Lake Explorations Ltd NPL Miller, JT Greenwood Explorations Limited	Prospecting Prospecting Geological Geophysical & prospecting
6773 6792 6878	1978	Toreador Lucky Strike AV 1-2	Highmont Mining Corp JA Fennell Cominco	Geological, geochemical, geophysical Geological Geochemical & geological
7503 7647	1979	Crown Baker Creek Area	Union Oil Company pf Canada Ltd	Geological, geochemical, geophysical Geophysical
7990	1980	Foggy 11	Barrier Reef Resources Ltd	Geological, geochemical, geophysical
10627	1982	Crown Property	Union Oil Company pf Canada Ltd	Geophysical
11462 11475	1983	Crown Property Len	 Esso Resources Canada Limited	Geochemical Drilling
12092 12904 13560	1984	Carbide Property Foggy McCorvie	Gordon Leask Esso Resources Canada Limited Newmont Exploration of Canada Ltd	Geological Drilling Geological, geophysical, geochemical
13862 14206 14505	1985	Tia Tia Reg 2 & 3	Nu Crown Resources Inc. Newmont Exploration of Canada Ltd	Geochemical & geophysical Drilling Geophysical
15236 15738	1986	Tia 14 Hail, Harper Creek	Nu Crown Resources Inc. Aurun Mines Ltd	Geophysical Geological & physical
16226 16482 17035	1987	Hail, Harper Creek Tia Tia	Aurun Mines Ltd Nu Crown Resources Inc.	Geochemical Geophysical Drilling
17555 17650	1988	Birch Group Hail, Harper Creek	Foundation Resources Ltd. Aurun Mines Ltd	Geological & geochemical Pre-Feasibility study
18970	1989	Birch Group	Foundation Resources Ltd.	Geological, geophysical, geochemical
20218	1990	Birch Group	Gemstar Resources Ltd.	Geological & geochemical
24822	1996	Hail, Harper Creek	American Comstock Exploration Ltd	Drilling
25036	1997	Willy 1 & 2	Edward Hayes	Physical work
26926	2002	Mag	Belik, GD	Mag survey
27611 28044	2005	Avery & Jones Harper Creek	Christopher O Naas	Soil & Rock sampling Soil sampling

6.3 Historical Drilling

Historical core drilling took place on the property in 11 different years between 1967 and 1996. The total length of the 191 holes drilled on the property is 30,800 m. Of these holes, 165 targeted what is now known as the Yellowhead Copper Deposit, for a total of 28,200 m or 92% of the overall drilling. The remaining 26 holes targeted four other areas distant to the deposit. Figure 6-1 is a plan map of the historical drillholes.

6.3 Historical Drilling – *Cont'd*

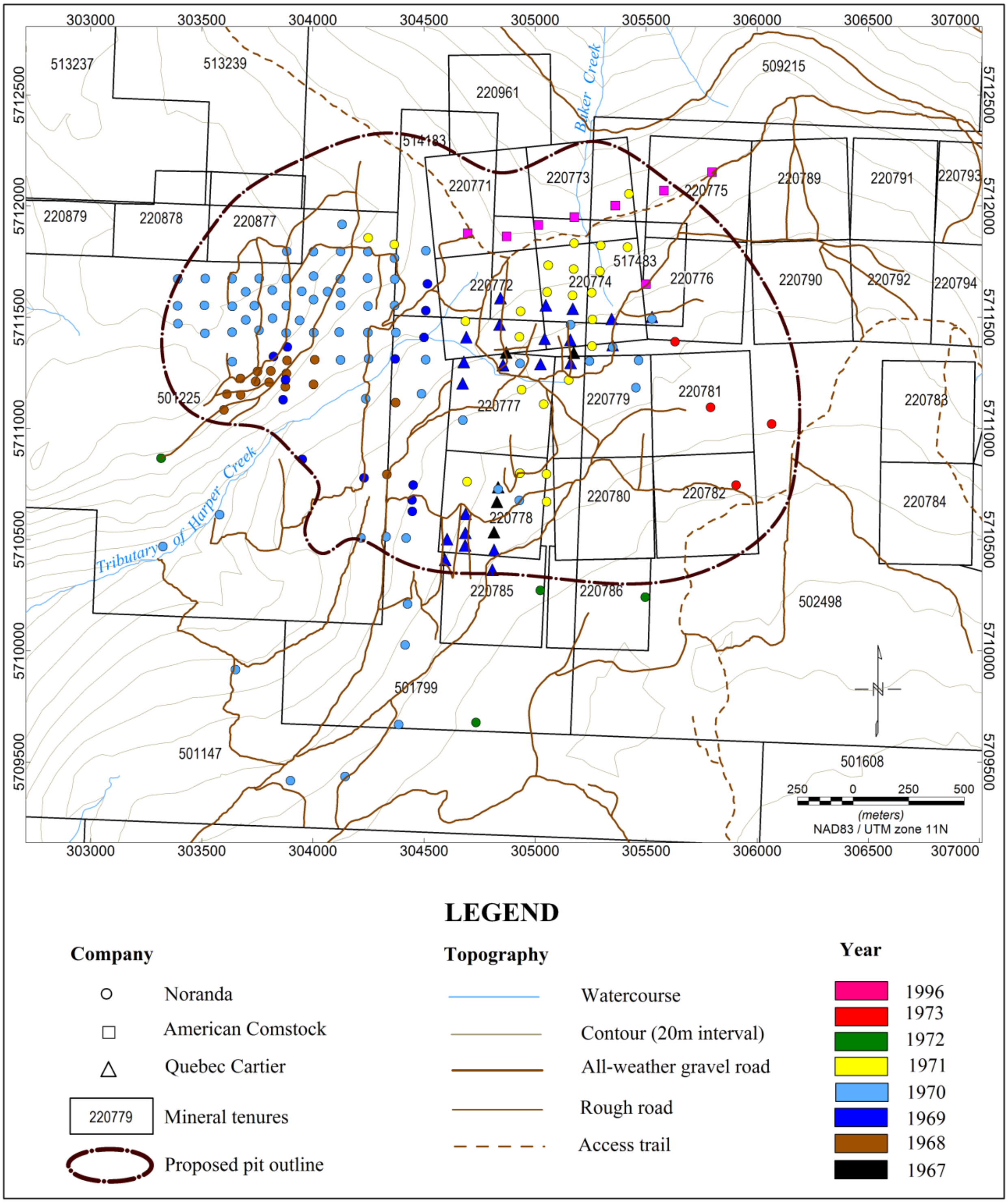

Figure 6-1: Historical Drillholes

6.3 Historical Drilling – *Cont'd*

(a) Québec Cartier

QCM collared the first diamond drillhole on the property in 1967 to target a geochemical anomaly found by surface sampling in 1966. It had an average copper grade of 0.33% over its entire cored length from 3 m to 108 m. QCM went on to complete six NQ (47.6 mm diameter) diamond drillholes in that year. The average depth of these holes was 91 m. Of the 546 m of total drilling, 40 m was overburden that was not recovered, logged or assayed. Recovery in the 526 m of cored intervals was 88%. In all, 174 samples collected at 3 m intervals were analysed for copper.

QCM resumed drilling in 1969 with 27 BQ (36.4 mm diameter) drillholes averaging 176 m in length. Drillhole orientations ranged from −45° to −85° to the south. Of the 4,739 m of total drilling, 158 m was overburden that was not recovered, logged or assayed. Recovery in the 4,581 m of cored intervals was 95%. Analysis of the 1,529 samples collected at 3 m intervals was for copper.

Due to the extremely foliated nature of the rock, QCM did not split their drill core, but instead took 1,703 whole core samples from all core drilled. Bagged samples sent to Bondar Clegg & Company laboratory in North Vancouver, BC were subject to hot acid extraction and atomic absorption spectroscopy (AAS) finish for Cu. Drill core sample preparation methods are unknown and there are no existing analytical certificates. There is no record of the insertion or analysis of any QA/QC samples.

There are no core photographs and no sample material remaining from any of the QCM core programs. Drilling on the QCM claims resumed in late 1970 under the Noranda / QCM joint venture exploration program.

6.3 Historical Drilling – *Cont'd*

(b) Noranda

In 1968 Noranda completed a 17-hole, 124 m (average length) drill program. Of the 2,106 m of total drilling, 116 m was overburden that was not recovered, logged or assayed. Recovery of the 1,990 m of cored intervals is unknown. Most drillhole orientations were vertical, with six holes drilled −60° to the south. Analysis of 710 samples collected at intervals averaging 2.8 m was for copper.

In 1969, Noranda drilled 13 holes at orientations of −60° south to an average depth of 133 m. Of the 1,734 m of total drilling, 119 m was overburden that was not recovered, logged or assayed. Recovery of the 1,615 m of cored intervals is unknown. Analysis of 528 samples collected at 3 m intervals was for copper.

Noranda resumed drilling in 1970 and completed 57 drillholes averaging 146 m in length. Drillhole orientations from this program were either −60° south or vertical. Of the 8,316 m of total drilling, 432 m was overburden that was not recovered, logged or assayed. Recovery of the 7,883 m of cored intervals is unknown. Analysis of 2,504 samples collected at 3 m intervals was for copper. Additional analysis, for zinc, lead, gold and silver took place on selected samples. Analysis for copper, gold and silver also took place for composited intervals from selected drillholes.

6.3 Historical Drilling – *Cont'd*

(c) Noranda / Québec Cartier Joint Venture

The Noranda / QCM joint venture completed a 12-hole drill program in 1970. Holes were drilled to the south at orientations from −45° to vertical and an average depth of 194 m. Of the 2,329 m of total drilling, 125 m was overburden that was not recovered, logged or assayed. Recovery of the 1,987 m of cored intervals is unknown. Analysis of 618 samples collected at 3 m intervals was for copper.

The joint venture commenced drilling again in 1971, completing 27 holes. All holes were vertical except three −60° south holes. Of the 5,594 m of total drilling, 342 m was overburden that was not recovered, logged or assayed. Recovery of the 5,468 m of cored intervals is unknown. Analysis of 1,767 samples collected at 3 m intervals was for copper.

In 1972, drilling resumed on a 4-hole program. Holes had southeast to south −60° to −70° orientations. Of the 457 m of total drilling, 12 m was overburden that was not recovered, logged or assayed. Recovery of the 445 m of cored intervals is unknown. Analysis of 50 samples collected at 3 m intervals was for copper.

In 1973, the joint venture completed a 5-hole program. The orientation of the holes was −55° south. Of the 632 m of total drilling, 27 m was overburden that was not recovered, logged or assayed. Recovery of the 605 m of cored intervals is unknown. Analysis of seven samples collected at 3 m intervals was for copper.

Noranda and the Noranda / QCM joint venture took 6,194 samples and analyzed for copper only. Sampling and assaying typically included all core recovered in the deposit area with a few minor exceptions. Outside of the main area, core sampling and assaying was much less frequent. No records of the methods of sampling, sample preparation or analysis, laboratories used, or any assay certificates exist for the Noranda and joint venture drill core analytical programs. There is no record of the insertion or analysis of any QA/QC samples.

All Noranda and joint venture core was drilled BQ size. Noranda stored boxes containing half split remaining core cross-stacked and in the open at their camp. The core remained there unsecured until moved to a storage facility in Vavenby, BC by YMI in 2008.

Availability of core photographs of historical core recovered by YMI is good for the NH series of Noranda holes. However, only limited joint venture holes are available.

Resampling of historical half core by YMI after photography consumed the remaining material. Sample assay pulps from the historical core resampling programs are well stored in a secure container at Vavenby, BC.

6.3 Historical Drilling – *Cont'd*

(d) American Comstock Exploration Ltd.

Comstock completed an 8-hole NQ2 (50.6 mm diameter) core drilling program in 1996. The holes averaged 356 m in length. They targeted deeper mineralization than previous programs. All were drilled south at −55° except one vertical hole. Of the 2,847 m of total drilling, 47 m was overburden that was not recovered, logged or assayed. Recovery of the 2,800 m of cored intervals is unknown. Analysis of 686 samples collected at 3 m intervals was for copper, molybdenum and silver. Gold, lead and zinc assays on composited 15 m intervals from one drillhole were also completed. Sampling and analysis of these holes was only for intervals with visible mineralization. This left 754 m of core unassayed.

Samples shipped to Acme Analytical Laboratories in Vancouver, BC for sample preparation and analysis for Cu, Mo and Ag were by digestion of a 1 g sample in aqua regia and analysis by inductively coupled plasma atomic emission spectroscopy (ICP-AES). Pb and Zn analyses on 15 m composites followed the same sample digestion and analytical methods as the other elements. Au analysis was by fire assay on one assay ton samples.

6.3 Historical Drilling – *Cont'd*

(e) Esso Resources Canada Limited

In 1983 Esso Resources Canada Limited drilled one NQ hole to a depth of 84 m on a geochemical and geological target 3 km northeast of the deposit on the historical Len claims. Split drill core was stored at site. Analysis for Cu, Au, Ag, Pb and Zn by Min-En Labs of North Vancouver, BC yielded no results of interest.

6.3 Historical Drilling – *Cont'd*

(f) Nu-Crown Resources Inc.

Nu-Crown Resources Inc (Nu-Crown) drilled 14 BQ holes on geophysical targets 4 km north of the deposit on the historical Tia claims. This drilling intersected anomalous to low-grade Pb-Zn-Ba mineralization. In 1985 they completed 427 m in a 4-hole program. Core sample analysis in 1985 by Acme laboratories for Au, Ag, Cu, Pb, Zn and Ba was by aqua regia digestion ICP-AES on the 81 samples (Belik, 1985). In 1987, 10 holes were completed totaling 942 m. Core sample analysis was by Eco-Tech of Kamloops, BC for Au, Ag, Cu, Pb, Zn and Ba on 107 1987 samples. All holes were drilled at −55° to the south. Drillhole collar locations and orientations are in the current YMI database but analytical results are not.

6.3 Historical Drilling – *Cont'd*

(g) Historical Surveys

Diamond drillhole collars were located in the field by transit surveys and reported in a company specific local grid. McElhanney of Vancouver, BC surveyed the QCM drillholes in 1969. Noranda contracted McWilliam, Whyte, Goble and Associates of Kamloops to undertake a legal survey of collar locations in 1971. Noranda also converted the QCM grid to the Noranda grid to integrate the geological databases of the two companies in that year. Only dip tests performed on inclined holes exist for the Noranda, QCM and joint venture data. Some inclined holes lack dip surveys and no downhole directional (azimuth) surveys exist for any of these holes. Vertical holes were not downhole surveyed.

6.4 Historical Mineral Resource & Mineral Reserve Estimates

The historical resource estimates referred to in section 6.4 are not current and most do not meet NI 43-101 Definition Standards. None of the historical estimates should be relied upon. A new resource estimate prepared in accordance with NI 43-101 Definition Standards is set out in section 14 of this report.

Aurun commissioned a pre-feasibility study by Phillips Barratt Kaiser Engineering Ltd. in 1986. The results of this historical mineral reserve and resource estimate was pre-NI 43-101.

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 project (Rennie and Scott, 2007).

Following an additional 12,656 m of diamond drilling in 34 holes the estimate was updated and reported as a NI 43-101 Resource in 2008. Table 6-2 summarizes the two estimates.

Table 6-2: 2007 and 2008 Historical Resource Estimates

Date	Cut-off Grade (% Cu)	Tonnes (thousands)	Cu Grade (%)	Contained Cu (tonnes)
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	142,200	0.33	463,900
Mar-08	0.2	65,000	0.34	221,000

Rennie & Scott 2007 / Rennie, Writ. Comm. 2008

6.4 Historical Mineral Resource & Mineral Reserve Estimates – *Cont'd*

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 drillholes completed after the 2008 resource estimate (Table 6-3).

Table 6-3: 2010 Historical Resource Estimate

Cut-off Grade (% Cu)	**Tonnes (thousands)**	**Cu Grade (%)**	**Contained Cu (million lbs)**
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

Rennie & Scott 2010

6.4 Historical Mineral Resource & Mineral Reserve Estimates – *Cont'd*

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

Table 6-4: 2011 Historical Resource Estimate

Cut-off Grade (% Cu)	**Tonnes (thousands)**	**Cu Grade (%)**	**Au Grade (gpt)**	**Ag Grade (gpt)**
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

Wardrop, 2011

6.4 Historical Mineral Resource & Mineral Reserve Estimates – *Cont'd*

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 Feasibility Study (FS) dated January 25, 2013) as shown in Table 6-5.

Table 6-5: 2012 Historical Mineral Resource Estimate

Cut-off Grade (% Cu)	Tonnes (thousands)	Cu Grade (%)	Au Grade (gpt)	Ag Grade (gpt)
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

Geosim Services Inc. Dec 2011

6.4 Historical Mineral Resource & Mineral Reserve Estimates – *Cont'd*

Geosim completed a subsequent resource estimate in 2014 for the Technical Report and Feasibility Study of the Harper Creek Copper Project issued July 31, 2014 as shown in Table 6-6.

Table 6-6: 2014 Historical Mineral Resource Estimate

Measured and Indicated Mineral Resource						**Contained Metal**		
Category	**Cut-off (Cu %)**	**Tonnes (thousands)**	**Cu (%)**	**Au (gpt)**	**Ag (gpt)**	**Cu (million lbs)**	**Au (thousand oz)**	**Ag (thousand oz)**
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						**Contained Metal**		
Inferred	0.15	119,743	0.25	0.025	1.2	660	96	4,619

Geosim Services Inc. Mar 2014

The 2014 Technical report also reported a mineral reserve 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 reserve was stated at a 0.14% Cu cut-off grade and is shown in Table 6-7.

Table 6-7: 2014 Historical Estimated Mineral Reserve

Proven and Probable Mineral Reserve					**Contained Metal**		
Category	**Tonnes (thousands)**	**Cu (%)**	**Au (gpt)**	**Ag (gpt)**	**Cu (million lbs)**	**Au (thousand oz)**	**Ag (thousand oz)**
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**

6.5 Yellowhead Mining

YMI formed as a private British Columbia company and obtained control of the project through staking, purchase and option agreements in 2005. Transactions completed in 2005 and 2006 encompassed five claim groups in the historical drilling area and contiguous parts of the Eagle Bay Assemblage that include the deposit. YMI undertook their first phase of field exploration on the project in 2006. Other sections of this report describe the work performed by YMI.

6.6 Production from the Project

There has been no production from the project to date.

SECTION 7

GEOLOGICAL SETTING AND MINERALIZATION

SECTION 7: GEOLOGICAL SETTING AND MINERALIZATION

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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). Flanking these rocks are high-grade Kootenay Terrane metamorphic rocks of the Shuswap Complex immediately to the east and rocks of the Fennell Assemblage immediately to the west. The project lies within the Cretaceous Bayonne plutonic belt represented by two large batholiths, Baldy to the south and Raft to the north.

Regional unit names (typically prefixed EB) and many of the descriptions used in sections 7.1 through 7.3 are after Schiarizza and Preto (1987) and Nass (2012a, 2012b, 2012c, 2013), except as noted.

(a) Lower Cambrian to Mississippian Eagle Bay Assemblage

The Eagle Bay Assemblage incorporates Lower Cambrian to Mississippian sedimentary and volcanic rocks subject to deformation and metamorphism during a Jurassic-Cretaceous orogeny. The Eagle Bay Assemblage divides 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).

Unit EBA of the Devonian-Mississippian succession hosts the deposit. To the south, unit EBA is over-thrusted 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.

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

7.1 Regional Geology – *Cont'd*

(b) Devonian to Permian Fennell Formation

The Fennell Formation is located northeast of the project and is comprised of Devonian to Permian oceanic rocks of the Slide Mountain Terrane. Tectonic emplacement of these units over the Mississippian rocks of the Eagle Bay Assemblage occurred in the early 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 appears to be the deep oceanic basin distal equivalent to the Eagle Bay Assemblage. There are striking similarities found in both formations and a hypothesis is that the sandstone of the Fennell Formation derived from the sandstones of the Eagle Bay Assemblage.

7.1 Regional Geology – *Cont'd*

(c) Mid-Cretaceous Bayonne Plutonic Belt

The north-south belt of mid-Cretaceous Bayonne Plutonic rocks consists of mostly peraluminous, subalkalic hornblende-biotite granodiorite and highly fractionated two-mica granites, aplites and pegmatites (Logan, 2002). The Baldy batholith to the south and the Raft batholith to the north are representative of this plutonic suite in the project area.

The west-trending multiphase Baldy batholith pluton covers approximately 650 square kilometres. It intrudes Proterozoic to middle 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.

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).

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 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, 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).

The Baldy batholith hosts a variety of mineral occurrences. 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. 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 for at least 7.5 km (Logan, 2001).

7.2 Regional Mineralization – *Cont'd*

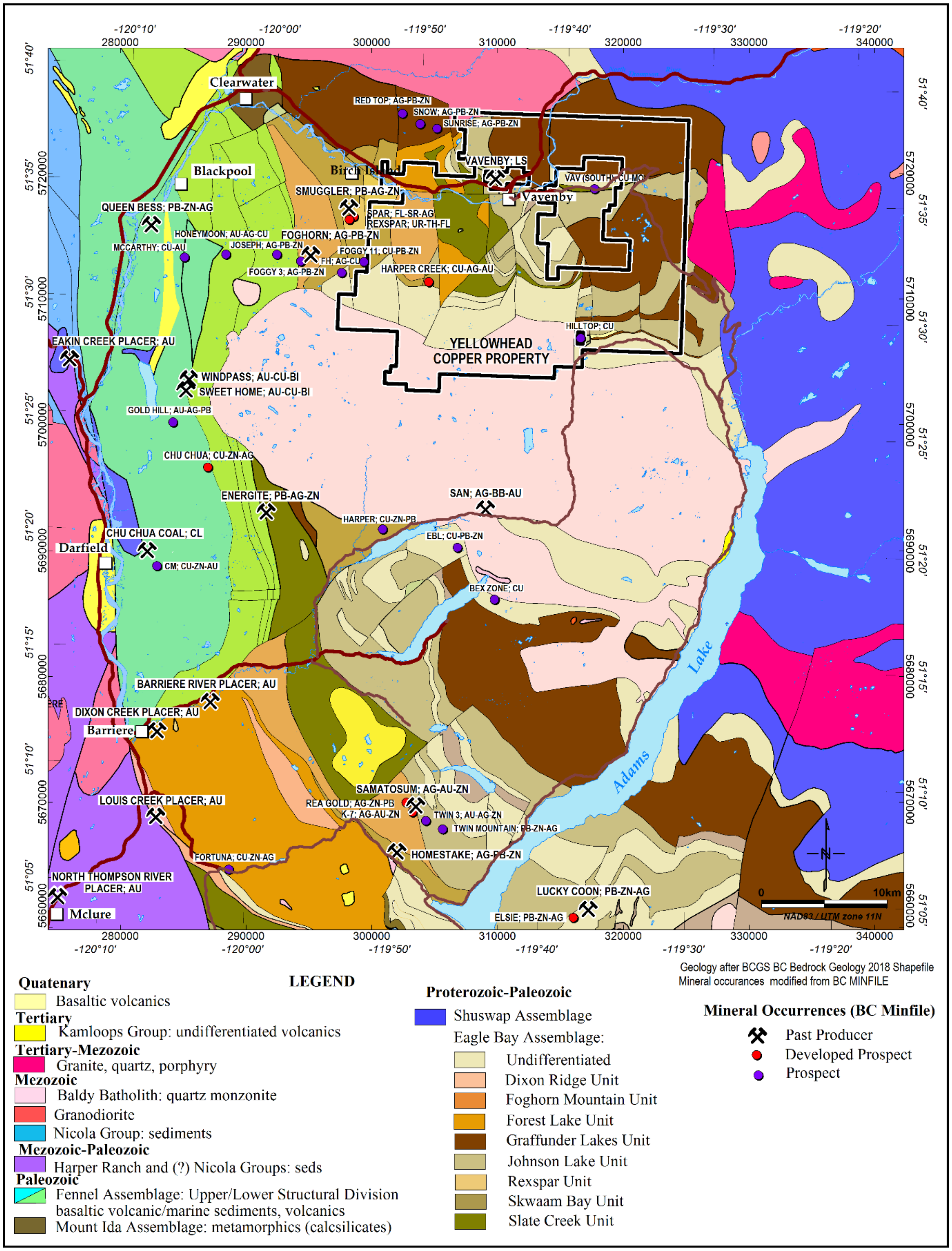

Figure 7-1: Regional Geology and Economic Setting

7.3 Project Geology

Rocks that underlie the project are primarily of the Eagle Bay Assemblage with a lithological succession interpreted as the Dgn, EBQ, EBA, EBF and EBG units of this group. This succession consists of a series of orthogneisses, metasediments, metavolcanics and metavolcanic clastics respectively, structurally overlain by the Tshinakin limestone unit belonging to unit EBG. Regional structure encompasses a complicated sequence of polyphase deformation consisting of sequences of thrust faulting, intrusion-related folding and faulting, strike-slip and normal faulting all of which imposed a complex alteration and metamorphic fabric on the rocks.

The mid-Cretaceous Baldy batholith cuts this succession at the southern end of the project and a late epidote alteration event relates to this intrusion. (Armstrong and Hawkins, 2009). Figure 7-2 is a simplified project-scale geology map modified from Paradis et al (2006).

7.3 Project Geology – *Cont'd*

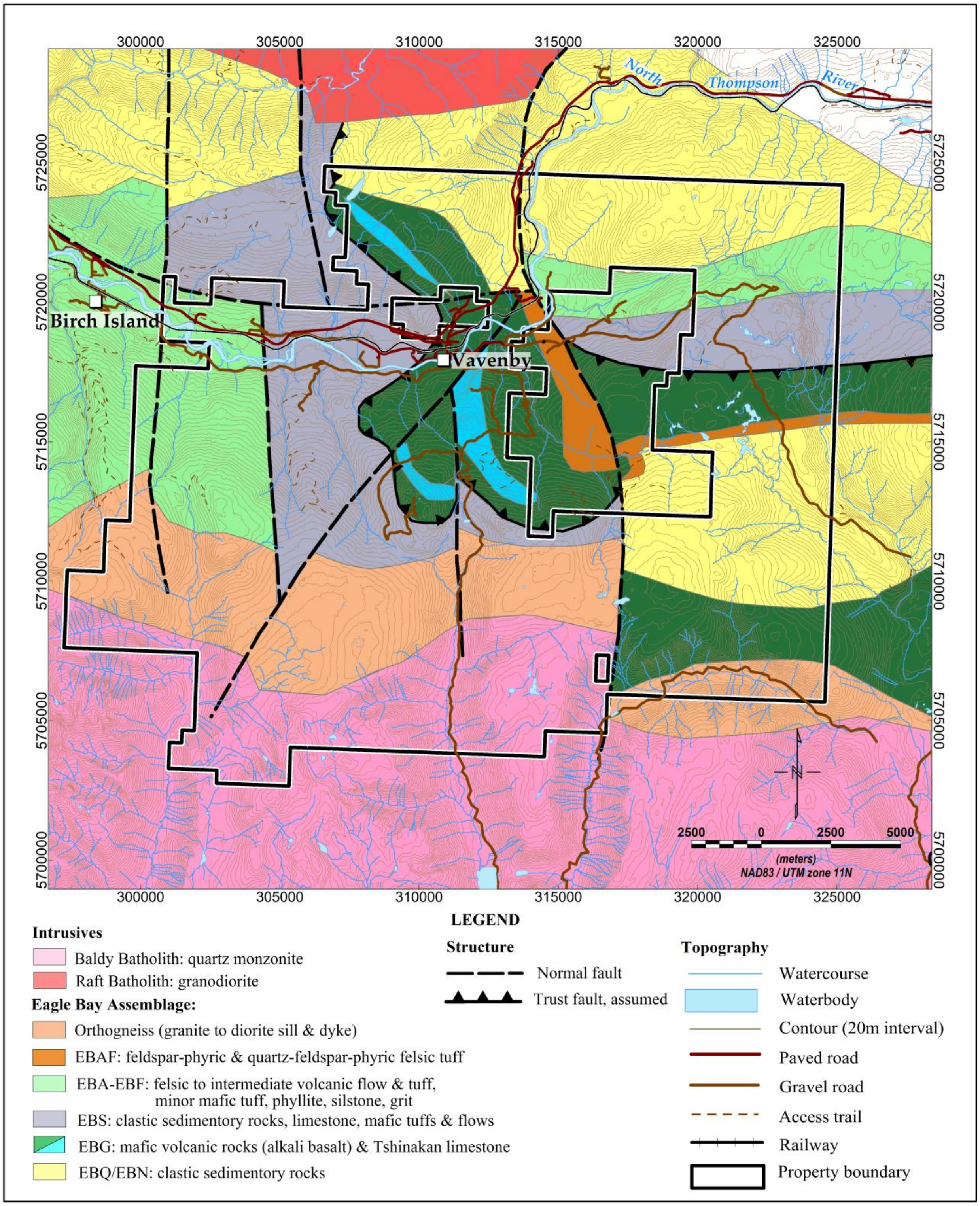

Figure 7-2: Geology Map, Yellowhead Copper Project

7.4 Project Mineralization

The principal area of mineralization on the project is the Yellowhead Copper Deposit (the Deposit). The northeast trending Harper Creek Fault separates the deposit into a west domain and an east domain (Figure 7-3). In the west domain, chalcopyrite mineralization is primarily in three copper bearing horizons. The upper horizon ranges from 60 m to 170 m in width and is continuous along an east-west strike for some 1,300 m, dipping approximately 30º north. Mineralization within this horizon occurs within felsic and mafic volcanics and volcaniclastic rock units. The middle horizon is not as well developed and is often fragmented. It primarily exists within a graphitic and variably silicified package of rocks that range from 30 m to 40 m in width at the western extent, increasing up to 90 m locally eastward, gradually appearing to blend into the upper horizon. Of the three horizons, this contains strong to intense silicification and localized tension fractures filled with mineralization. The lowest or third horizon has less definition mainly due to a lack of drill intersections. Commonly hosted within mafic to intermediate volcaniclastics and fragmental rocks, it can range from 30 m to 90 m in width although typical intersections are in the 30 m range. These horizons host within felsic and mafic metavolcanics and metavolcaniclastics and generally contain foliation-parallel wisps and bands as the dominant style of sulphide mineralization.

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

The primary focus of exploration by YMI on the property has been on the main deposit area and mineralization outside of there is not well known.

7.5 Deposit Geology and Mineralization

(a) Geological Lithologies

Metamorphic rocks of the Eagle Bay Assemblage host the deposit. Pervasive alteration and structural deformation of these host rocks has made confident identification of their protolith difficult. Four metamorphic rock types: quartz-bearing schists, non-quartz-bearing schists, phyllite, orthogneiss, comprise about 90% of lithologies drilled in the deposit and the quartz/quartz-eye schist unit comprises almost half of them. The four dominant lithologic units are coded in drill core as 7, 8, 9 and 10. Phyllites and schists are subdivided further based on their mineral or textural characteristics. Table 7-1 summarizes the geological rock type groups, subgroups, code lists and descriptions used on the project.

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.

Within the dominant schist unit 9, the sericite-chlorite-quartz schists represent the most significant component, followed by sericite-chlorite-quartz-feldspar type. 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).

Areas where pervasive alteration completely masks the geological textures assign to a unique unit number (unit 11). This unit is subdivided based on alteration product. Currently defined are silica (unit 11a) and chlorite (unit 11b).

Areas of massive sulphides, although not significant volumetrically, are assigned separately (unit 12) due to their mineralogical importance. This unit is subdivided based on the dominant sulphide.

In rare situations where the protolith is identifiable, rocks are classified accordingly as intrusives (unit 3), volcanic flows or intrusions (unit 4), volcaniclastics (unit 5) and sedimentary (unit 6). The area immediately southeast of the deposit has the most notable intersections of argillites and sandstones. Limestones, as identified in several drillholes, tend to be rare and thin. Drill core has intersected a late-stage series of andesitic dykes and sills (unit 4a) in various areas of the deposit. To date, there is only one occurrence of an intrusive (granodiorite, unit 3a) in the drilling.

7.5 Deposit Geology and Mineralization – *Cont'd*

(a) Geological Lithologies – *Cont'd*

Table 7-1: Geological Rock Type Code List and Descriptions

Code	Unit	Sub	Description
0	Overburden		Unconsolidated overburden
1	Faults		Fault zones Fault gouge, fault breccia & healed, shear zones
2	Veins		Veins Quartz, carbonate, quartz-carbonate, sulphide veins
3	Intrusives		Intrusive rock protolith Granodiorite, hornblende-biotite granodiorite, quartz monzonite
4	Volcanic flows or intrusions		Volcanic flow or intrusive rock protolith Includes late-stage andesitic dykes & sills, lamprophyre dykes
5	Volcaniclastics		Volcaniclastic rock protolith
6	Sedimentary		Sedimentary rock protolith Sandstone, argillite, limestone
7	Phyllites	7 7a 7b 7c 7d	Phyllite metamorphic rock Graphite Sericite-chlorite Calcareous chlorite-sericite Sericite-chlorite-quartz
8	Schists (<quartz)	8 8a 8b 8c	Schist metamorphic rock with minimal or no quartz content Sericite-chlorite Sericite-chlorite-fuchsite Chlorite-sericite fragmental
9	Schists (>quartz)	9 9a 9b 9c 9d 9e	Schists metamorphic rock with quartz content &/or quartz eyes Sericite hornblende-quartz-feldspar Sericite-chlorite-quartz Sericite-chlorite-quartz-feldspar Sericite-augen quartz Siliceous chlorite-sericite quartz
10	Orthogneiss		Orthogneiss metamorphic rock
11	Pervasively altered	11 11a 11b	Pervasively altered rock, protolith unknown Silica altered Chlorite altered
12	Massive sulphides	12 12a 12b 12c 12d 12e	Massive sulphides Undivided massive sulphides Magnetite dominant Pyrrhotite Pyrite Chalcopyrite

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages

Due to multiphase deformation and alteration, correlation of lithologies between drillholes is difficult. Creation of a set nine of geological packages with common characteristics and affinities maintained the lithological detail, yet simplified correlation of essentially similar geological units. The packages are coded A, B, C, D, E, Fa, Fb, G and H, where package A represents the lowest stratigraphic unit, moving up-section to package H at the top. Table 7-2 summarizes the geological packages, codes and styles of copper mineralization.

Table 7-2: Geological Package Code List and Descriptions

Code	**Description of Geologic Package Composition**	**Copper Mineralization**
H	Mafic polymictic volcaniclastics 8c, 8a, 7c +/-9a hornblende crystals, frequently calcareous & deformed 7d	No
G	Graphitic horizon, somewhat calcareous	No
Fb	Intermediate to mafic polymictic volcaniclastics 8c, 8a, 7c ± 9a hornblende crystals, somewhat calcareous	Yes
Fa	Felsic to intermediate volcaniclastics: 9c, 8c & 8a	Yes
E	Graphitic horizon: mixed 11a silicified +7a	Yes
D	Intermediate volcaniclastics & Fragmentals, somewhat calcareous: dominated by 8c/7c	Remobilized
C	Graphitic horizon	Remobilized
B	Sandy sediment dominant: 9b + 8a mafic sediments in the west. 9b graphitic of 9b + 9c felsic sediments ± 8a in the east	Remobilized
A	Orthogneiss: 10a and associated border phases 9d and others	Remobilized

Figure 7-3 is a surface plan map of the deposit area illustrating the geological packages, topographic features, drillhole collar locations and the location of the accompanying cross-sections. Figures 7-4 and 7-5 are vertical, west-looking example cross-sections at 304060E and 305420E respectively. They show geological package stratigraphy and downhole assay grade bars on drill traces and illustrate significant intersections of copper mineralization from the west and east domains of the deposit.

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

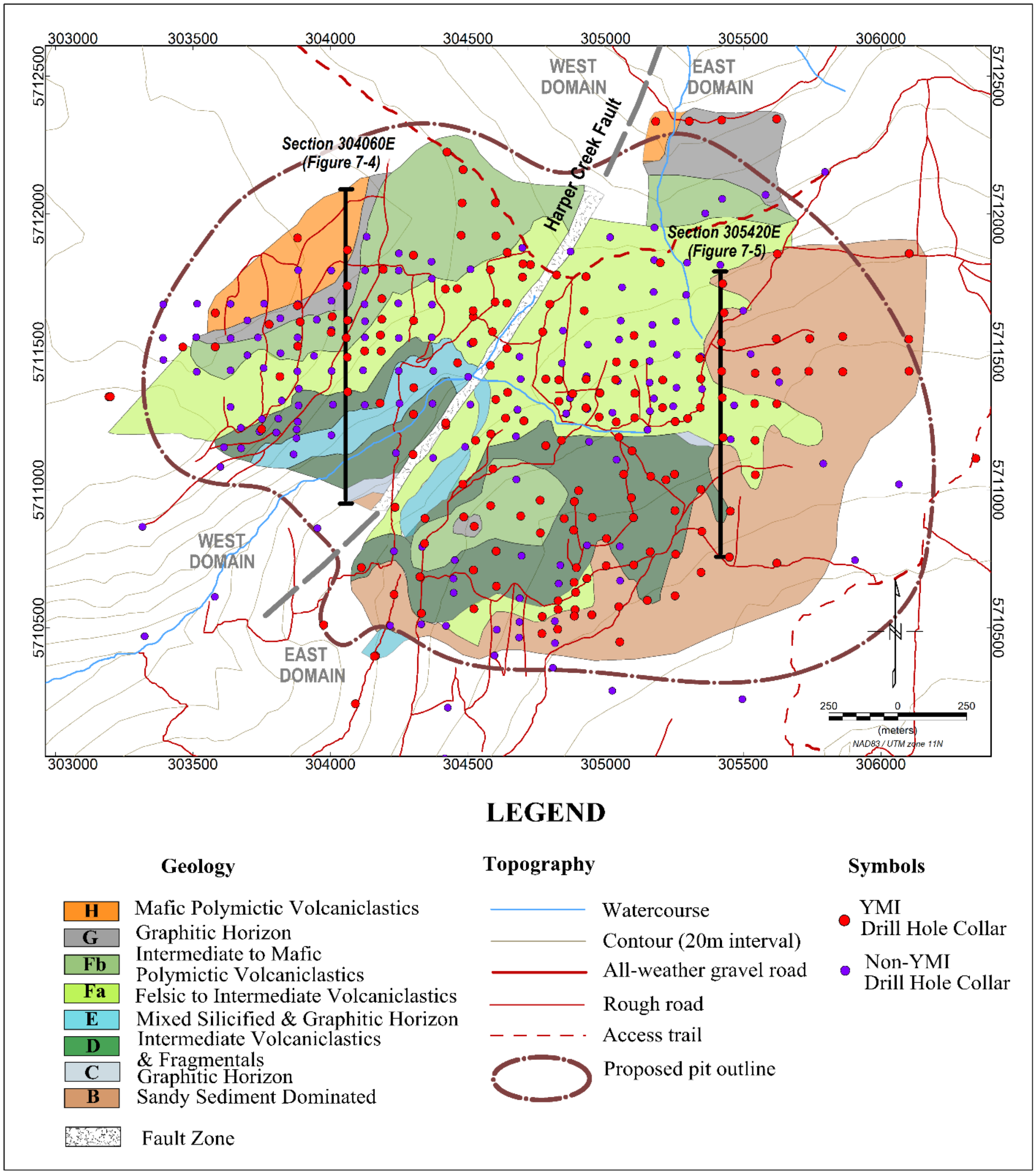

Figure 7-3: Geology & Drilling Plan

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

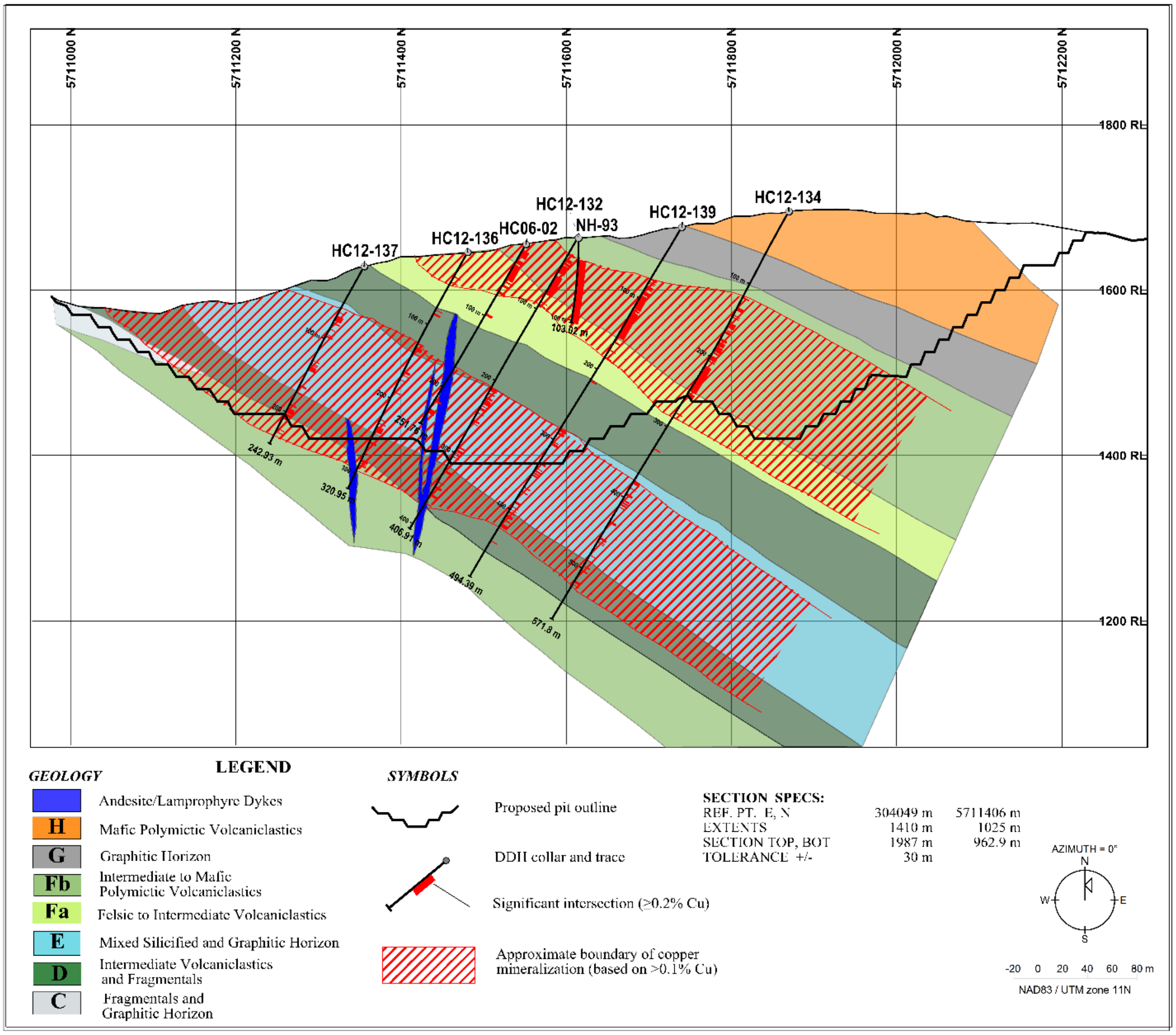

Figure 7-4: Geological Cross Section 304060E (West Domain)

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

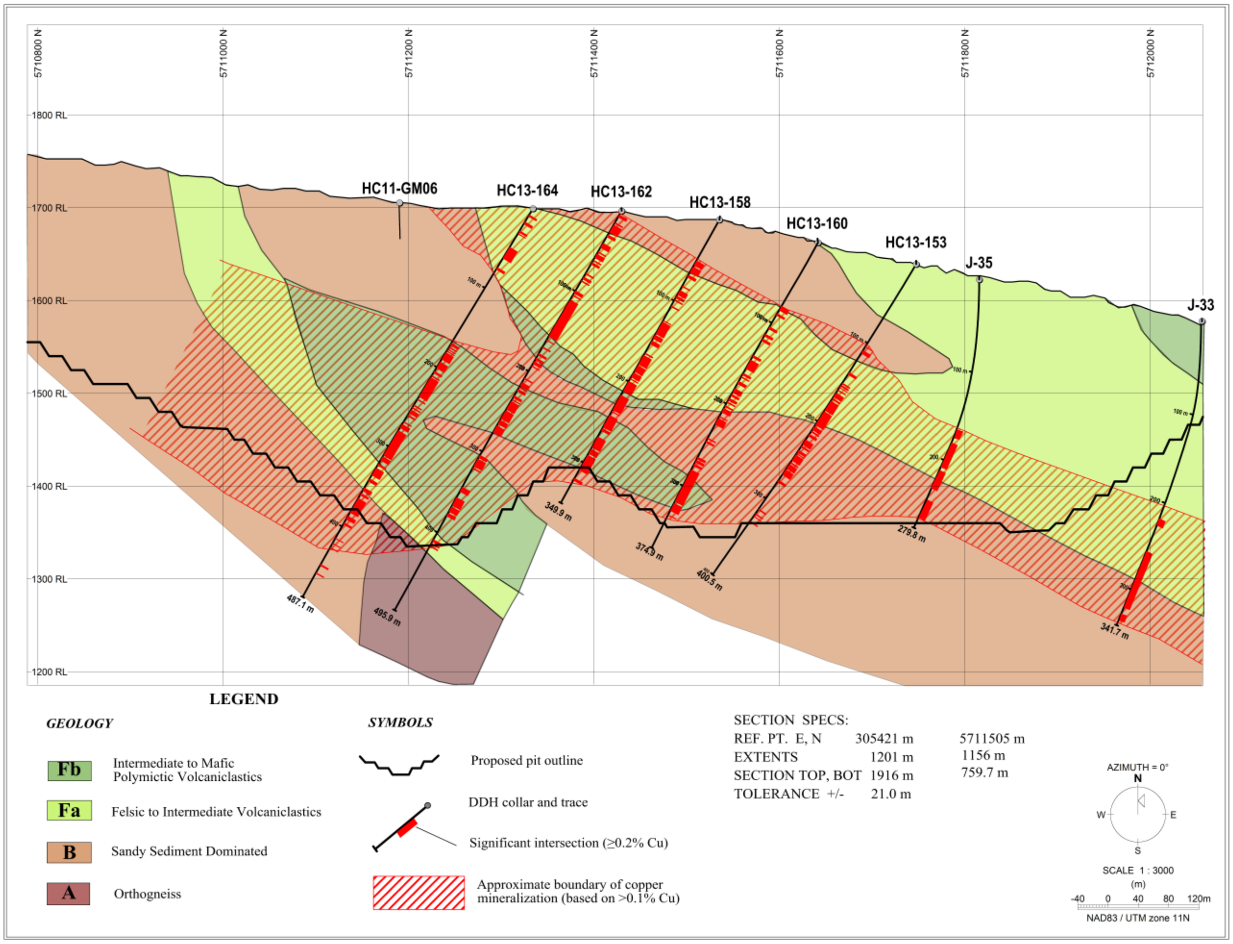

Figure 7-5: Geological Cross Section 305420E (East Domain)

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

Package A

Package A comprises the Late Devonian orthogneiss (unit 10a) and the strongly to intensely deformed marginal-phase of the orthogneiss intrusion. The latter frequently has texturally destructive deformation that classifies as sericite-chlorite-quartz phyllite (unit 7d). Interpretation is that these are strongly deformed felsic intrusives (sericite-augen quartz schists, unit 9d). This unit often cuts through the upper sections of this package.

Unit 7d occurs in the zone of intense deformation encountered immediately before the orthogneiss downhole. 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. Definition of the rocks proximal to the orthogneiss is difficult due to deformation and strong to intense biotite alteration. The colour of intensely foliated and deformed unit 7d ranges from medium green to dark green to brown, as a function of biotite content. The frequent presence of weak to moderate interstitial calcite along with the textural and compositional change is indicative of the proximity to basement rock.

Unit 7d contains foliation-parallel quartz bands and boudinage that are commonly milky, 1 cm to 15 cm wide, and internally fractured with iron carbonate and occasionally calcium carbonate infill. Cutting throughout units 10a and 7d are felsic dykes of unit 9d. These 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 1 cm 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 regionally mapped Devonian granitic orthogneiss unit Dgn. This orthogneiss, situated on the northern and southeastern portions of the Baldy batholith, overlies and intrudes metasedimentary units.

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

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 to 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 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%).

To the east, 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 5-20%.

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. Units of 9b that grade in and out of 8a horizons may indicate a siltstone version of metasediments or mafic metavolcaniclastics.

In the easternmost portion of the deposit, metasediments become the dominant lithology. Package B is 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. In the top interval, there is a graphitic component to the metasediments not seen in 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 of intervals from packages Fa, E, and/or D is intersected in the bottom half of these easterly drilled holes. This strongly intercalated zone has an increased abundance of unit 9c (up to 50%) while unit 8a decreases and becomes more rare. 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. Metavolcaniclastic rocks wane to the east and metasediments increase

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

Package B – *Cont'd*

significantly, possibly indicating that this area was previously a sedimentary basin some distance from the volcanic source.

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 through most of the deposit, unless inundated with pervasive secondary silicification. In the far eastern part of the deposit, copper mineralization occurs in greater abundance within this package. Following the unmineralized graphitic portion, mineralization is no longer generally selective to packages Fa and D, but instead occurs in large intervals throughout. This may result from 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.

Package C

This package occurs as a graphitic phyllite (unit 7a) horizon ranging from 2 to 25 m in thickness. It is common 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 is a preferred horizon for thrust faulting. Package C is therefore a marker horizon that separates packages B and D respectively in the west domain.

In the east domain, this package occurs more commonly as intercalations rather than as a distinct horizon. There the package is often absent altogether and package D overlies package B.

Sulphide mineralization within package C is low. Sulphides are mainly present as pyrite, lesser pyrrhotite and locally trace chalcopyrite. Pyrite and pyrrhotite precipitated as porphyroblasts up to 1.5 cm 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 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

Package D

This package occurs between two graphitic horizons. It is comprised predominantly of intermediate to mafic metavolcaniclastic tuffs and fragmental volcaniclastics that frequently contain secondary quartz and calcite alteration that 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.

In the west domain, package D transitions eastwardly from predominantly mafic metavolcaniclastic tuffs and silts to a package with increased intercalations of quartz-rich metasediments (unit 9b). This is observed in western drillholes 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 increase towards the east.

In the east domain, package D gradually decreases in thickness and intercalations of more felsic units (units 9b and 9c) increase in abundance 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% of the rock, 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. Package D does not occur in many of these drillholes and felsic metavolcaniclastics and metasediments are the dominant rock types. In the west domain, unit 8a is generally present with unit 9b in the D and B packages. In the far eastern part of the deposit, package D decreases and it appears that units 9b and 9c have replaced the intervals previously occupied by unit 8a.

Sulphide mineralization within package D is not consistent. There is sporadic emplacement of wide multiple sulphide lenses up to 5 m. Thick lenses more common within package Fa are not present here. Zones of sulphide mineralization are present in units 8a, 7c, and 9b and frequently transgress lithological contacts without preference to lithology. Chalcopyrite mineralization is mainly seen 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 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

Package E

Package E consists of a 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) sandstones intercalated with siltstone. Preserved within the silica-flooded host, are relict opaline-blue quartz grains, commonly observed within package B (unit 9b). Impermeable mudstone has metamorphosed to graphitic phyllite (unit 7a). However, in many places it also shows strong to intense silicification. This package may occupy a large thrust fault along weak graphitic units where silicification has resulted from increased fluid movement related to the Harper Creek normal fault. Structures including the Harper Creek Fault are abundant in the area. Package E could be related them.

Package E traces easily from west to east throughout the drillholes in the west domain and ranges from 15 to 91 m in thickness. In the east domain, the trend is discontinuous and not frequently observed. The contact between packages D and Fa does not confine silicified intervals resembling package E, as this alteration occurs randomly throughout the stratigraphy. Unsilicified graphitic intervals are also randomly present in the east domain and may represent mudstone and/or shear planes. Package E intervals in the east domain range in width from 4 to 80 m. Here, silicified and graphitic intervals are generally not associated with one another as they are in the west domain.

Sulphide mineralization within package E is strong and high-grade lenses of copper trace throughout. Chalcopyrite (<1-3%) is mainly noted as fracture-fill in tension fractures at 10° to 30° to core axis. Specularite (and locally molybdenite) are frequently present as is rare bornite. This sulphide assemblage is a useful marker within the silicified section. Interpretation is of an increased temperature gradient moving eastward within the sulphide fluid phase, as specularite appears to decrease while molybdenite and bornite increase.

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

Package Fa

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, dominate this assemblage. 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 unit 7a and 9b metasediments (without opaline-blue quartz grains) represent less than 5% of the package. Locally pervasive silica altered host intervals (unit 11a) may also be present.

Package Fa is intensely convoluted, indistinct and difficult to trace across drillholes in the east domain, similar to package D. The abundance of unit 9c decreases markedly and its occurrence ranges from 10 to 50%. Zones of texturally destructive deformation increase and unit 7d makes up to 80% of the package locally. 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 also exist in the Fa package in the east and are variable in abundance (up to 20%).

Metasediments become the dominant lithology moving eastwards and package Fa appears to decrease in size and abundance as package Fa becomes lensoidal or pinches-out.

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), that are interpreted to represent a sequence of felsic volcanics and volcaniclastics intervals. Chalcopyrite, ranging from <1 to 3%, commonly occurs as very fine-grained foliation-parallel wisps on rims of pyritic chain-of-grain bands, interstitial disseminations and locally filling tension fractures at 10° to 30° to core axis.

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

Package Fb

This package is composed primarily of polymictic fragmental chlorite schists (unit 8c) and chlorite-carbonate phyllites (unit 7c) likely derived from mafic volcanic and volcaniclastic rocks. Similar to package D, these units frequently contain secondary quartz and calcite alteration that occurs interstitially and in 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 occurs within the chlorite-carbonate phyllites. 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 preserved reasonably well, the unit shows a flow-like texture and appears similar to a welded ignimbrite. A marked increase in titanium and phosphorus, which is consistent throughout the deposit, defines this package geochemically. This package is most notable in the west domain in the northern part of the deposit and the western part of the deposit). 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%). Unit 9a 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, near surface in the south and 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 not consistently present in all successions. It is possible that these very similar looking rocks belong to two different formations.

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

7.5 Deposit Geology and Mineralization – *Cont'd*

(b) Geological Packages – *Cont'd*

Package G

Package G is a graphitic horizon ranging from 6 to 40 m in thickness 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 1 mm, to 11 cm in width. It is well foliated and appears locally fragmental in texture with lenticular to banded fragments parallel to foliation (1 mm to 6 cm). Intercalations of medium to dark grey limestone (unit 6f) occur within this package.

In the east domain, the package occurs as sporadic lenses, which do not correlate well across the deposit. It is calcite-dominant in the southwest with intercalated graphitic limestone. Centrally, dolomite is the more prominent carbonate and occurs interstitially and in foliation parallel bands.

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

Package H

This is the uppermost package of rocks within the deposit. Its known occurrence thus far is restricted to rocks observed in the far north and west of the deposit. The base of this package appears to have undergone strong to intense deformation as noted by the presence of thick intersections of sericite-chlorite-quartz phyllites (unit 7d) frequently intercalated within a succession that resembles felsic volcanic tuffs similar to those identified within the 9c unit. Local intercalations of hornblende-feldspar-quartz crystal lithic tuffs are likely representative of the regional EBF assemblage.

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

7.5 Deposit Geology and Mineralization – *Cont'd*

(c) Structure

Harper Creek Fault

The large Harper Creek Fault zone trends northeast and dips 80° to the southeast in the deposit area. This 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).

Several wide zones of pale grey to green gougy faults and localized quartz and iron carbonate-healed fault breccias commonly occur within this structure. Fault breccias within these structures include polylithic fragments, often silicified, that commonly have disseminated and fracture-fill mineralization. Quartz-iron carbonate breccias are generally barren, faulted by a later event defined by reactivated gougy sections. Common within the structure is strong to intense deformation, often seen as kink folding, in addition to abundant clay (argillic) alteration. As the structure is composed of several fault zones, thickness varies from hole to hole, however it generally ranges from 25 to 50 m in thickness. Interpretation of structural movement is oblique right lateral offset with some possible rotational movement. Drop down on the south side appears to be in the range of 60 to 100 m.

The structure also contains several mafic to andesitic dykes interpreted as late Tertiary that show no regional deformation. Many dyke intersections are gougy and brecciated, possibly due to later northerly trending faults. Several of these dykes appear to have used the Harper Creek Fault as a structural pathway.

7.5 Deposit Geology and Mineralization – *Cont'd*

(c) Structure

East Domain Structures

The east domain appears to have separated into several fault slices by a structural event. Fault structures noted throughout the domain likely caused offsets in mineralized zones as well as offsets in packages of rocks that may range from tens to possibly hundreds of metres. The actual degree of mineralized zone offset caused by these structures is unknown. Related to these fault slices is the east-west trending Larry Fault..

The orientation of these structures trends west southwest (~250°) with a northwest dip of 20° to 35°. The style is that of an imbricated thrust fault system with multiple variations in strength and orientation.

Characteristics of these structures vary with the host lithology they pass through. Feldspar-dominated units 9c, 8a, 8c, and 7c exhibit abundant foliation-parallel flaking. This is evident in core that is broken into disc shapes and with multiple foliation-parallel gouge zones where back and forth movement has occurred. More silicified and weakly foliated sericite-chlorite-quartz schist units 9b and 11a occur as broken fragments with abundant hairline fractures of no preferred orientation. Fracture surfaces within silicified areas frequently have clay and gouge. Iron carbonate and silica-healed breccias also occur within gouge zones in several areas.

7.5 Deposit Geology and Mineralization – *Cont'd*

(d) Geological Interpretation

The proposed sequence of formation for the deposit as presented in Table 7-3.

Table 7-3: 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. • Middle Cambrian to Middle 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 to Early Jurassic: first regional phase of deformation. 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. • 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. Emplacement not pinpointed with accuracy at this time. Displacement appears to be minimal. • Intrusion of quartz-feldspar porphyry, andesite, and lamprophyre dykes.
8. Erosion to current topography.

SECTION 8

DEPOSIT TYPE

SECTION 8: DEPOSIT TYPE

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8.1 Deposit Type

Interpretation of the deposit type is that of a remobilized polymetallic volcanogenic massive 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 is consistent with the volcanogenic model. Observed sulphide lenses measure many tens of metres in thickness with kilometer-scale strike and dip extents. In 2009, YMI conducted a program of field mapping, sampling, relogging, petrographic examination of existing thin sections and re-assessment of the total digestion geochemical dataset that confirmed the deposit type hypothesis for the deposit (Armstrong and Hawkins, 2009).

Support for this model is as follows:

- The generally stratabound nature of the highest grades of mineralization, which can be interpreted as deformed massive to semi-massive sulphide lenses;
- An overall metal assemblage consistent with a copper-rich VMS;
- Interpretation of widespread, lower grade mineralization as a deformed feeder or alteration zone originally located below higher-grade massive sulphide horizons; this also accounts for the overall discordance of mineralization to stratigraphy;
- Host rocks are highly altered felsic volcanic rocks within a bimodal volcanic sequence, similar to those that host many major VMS deposits globally;
- The presence in the region of numerous deposits clearly compatible with a VMS genetic model.

SECTION 9

EXPLORATION

SECTION 9: EXPLORATION

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9.1 Introduction

YMI began the company's first phase of field exploration on the project in 2005. Exploration completed between 2005 and 2013 included diamond drilling and historical core relogging (described in section 10), airborne geophysics (magnetic and electromagnetic), ground geophysics (magnetic, electromagnetic and induced polarization), soil sampling, rock sampling, geological mapping and petrographic and whole rock analysis of drill core and surface rock samples. This work largely focussed on the west-central part of the property in the deposit area. The 2014 technical report describes this work in detail and this summary derives from it. Project history included in section 6 of this report describes the exploration work carried out by previous operators on the property. There has been no exploration on the property since 2013.

Table 9-1 lists the ARIS assessment reports filed by YMI on the property since 2006, all authored by C.O. Nass, P.Geo.

Table 9-1: Yellowhead Mining ARIS Reports on the Property

ARIS Number	Work Year(s)	Work Program
28472	2006	Core logging & resampling
28812	2007	Airborne geophysics
29404		Drilling, geophysics
29732	2008	Drilling, geophysics, geochemistry
30320		Drilling
30566	2009	Geology, geochemistry, geophysics, road & reclamation
31278		Geology, geochemistry, reclamation
31986	2010	Resource modelling & estimation
32220	2011	Drilling, geochemistry
32723	2010-2011	Drilling, core relogging and geological modelling
34525	2012-2013	Drilling, geology, geochemistry

9.2 Airborne Geophysics

Aeroquest Limited helicopter-borne magnetic and electromagnetic geophysics surveys conducted in 2006 and re-processed by Insight Geophysics in 2007, included over 1000 line-kilometers at predominantly 100 m line spacing. Follow up of airborne geophysical targets identified was by ground survey.

9.3 Ground Geophysics

In 2007 and 2008, ground-based geophysical surveys included horizontal loop electromagnetic (HLEM), magnetics and induced polarization (IP). HLEM and ground magnetic survey coverage included the Harper West, Jones Creek, Northwest, and M Anomaly grids. Ground magnetic and IP survey coverage included the Harper South and southeastern area of the M Anomaly grids respectively.

A 32 line-km IP/ Resistivity survey conducted by Insight Geophysics in 2007 tested anomalous targets defined previously by ground geophysics and soil sampling. The survey identified three anomalous areas within the Harper West grid and three conductor axes within the Northwest grid. The surveys also detected conductive areas on the western edge and north-northeast of the Northwest grid. Of note on the Jones Creek grid, were three areas of coincident conductivity and anomalous soils. Results from M Anomaly grid consist of numerous profiles that may indicate the shallow depth extent of vertically trending responses.

The 40 line-km ground magnetometer survey at 25 m intervals conducted by CME Consultants in 2008 on the Harper South grid indicated a prominent boundary between higher magnetic rocks to the north and a moderate magnetic unit to the south. This corresponds with field observations of the contact between Eagle Bay orthogneiss and metavolcanic / metasedimentary units.

9.4 Soil & Rock Sampling

YMI collected 4,532 soil samples between 2006 and 2008 from eight soil sample grids and one soil line established over high priority targets identified by the airborne geophysics. Description of the sample preparation and analysis of these samples is in section 11.

Survey grid cross-lines were oriented NNW-SSE, perpendicular to the regional trend, except for the north-south oriented Northwest grid. Cross-line spacing ranged from 100 m for detailed sampling, up to 400 m for reconnaissance-scale sampling, with GPS located sample stations spaced 25 m along lines. A typical sample taken from the B-horizon at 20 cm to 30 cm depth from surface ranged from 200 to 400 g in size. Samples were of the C-horizon in areas of poor soil development. Unsampled areas include those with unsuitable material (i.e. roads, swamp). Sample bags included local grid coordinate labels along with a corresponding bar code.

The Harper South grid immediately adjacent to the proposed pit is the strongest soil anomaly identified on the property and has a number of highly anomalous Cu values over 1,000 ppm. It is 450 m long and 100 to 400 m wide and appears to be representative of the surface expression of the deposit. YMI soil sampling of the M anomaly confirmed and refined the historically identified Cu anomalies there. A coincident Zn and Cu anomaly and a moderate discontinuous Cu anomaly occur on the Northwest grid and there is a persistent Cu anomaly across the entire Avery Lake grid. The smaller Vavenby grid has a possible weak Cu anomaly and the NZ soil line has two anomalous Cu values. In terms of Cu anomalies in soils, the Summit grid is weakly anomalous and the Jones and Farmer grids are not very anomalous.

Between 2004 and 2008, 462 rock samples were collected on the property from historical trenches, sub-crop, out-crop and float. They were taken for geochemical analysis and review of lithology, alteration and mineralization and as part of a wider mapping program outside of the main deposit area. The rock sample database contains 351 samples taken by YMI in 2006 and 2008, along with results from 111 rock samples collected in 2004 and 2005 by a previous operator.

9.4 Soil & Rock Sampling – *Cont'd*

Sample size varied but was typically >100 g, large enough to incorporate a representative sample for assay. GPS located samples were marked in the field using orange or pink flagging with the sample number and described in terms of lithology and alteration with estimated mineral and sulphide abundance. Samples were marked with a sample number and placed in 20 by 30 cm poly sample bags. This program identified numerous significant copper and other base metal occurrences and several significant precious metal occurrences. Twenty percent of the 351 samples collected by YMI were greater than 0.1% Cu and seven greater than 1 % Cu. Two grab samples from outcrop in the M Anomaly grid area returned results greater than 1 gpt Au. Other samples also had appreciable Ag, Pb and Mo.

(a) Petrographic Studies

Petrographic studies completed in 2007 and 2008 included thin and polished section work and whole rock analysis on drill core and rock sample specimens. These studies led to a better understanding of lithology, alteration and mineralization characteristics of the deposit. These studies were undertaken prior to the development of the current geological model and as such, their lithological descriptions may not match the current terminology. In support of geological modeling, additional thin sections were prepared, and petrographic descriptions completed, along with whole rock analysis of these samples in 2009.

9.4 Soil & Rock Sampling – *Cont'd*

(b) Additional Studies

In 2009, a program of field mapping, sampling, relogging, petrographic examination of existing thin sections and re-assessment of the total digestion geochemical dataset was undertaken to confirm the mineralization style of the deposit (Armstrong and Hawkins, 2009). This assessment confirmed that the hypothesis that the deposit is a volcanic-hosted massive sulphide (VHMS) deposit based on key attributes:

- Mineralogy and elemental assemblage, namely chalcopyrite + pyrite + galena;
- Tabular and broadly concordant nature of the mineralization;
- 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 crosscut 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 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.

SECTION 10

DRILLING

SECTION 10: DRILLING

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10.1 Introduction

A significant amount of drilling has taken place on the Yellowhead Copper Project, totalling 95,735 m by YMI and historical operators in 408 holes. All were cored diamond drillholes. Results from these drill programs are the basis for the mineral resource estimate reported in section 14. There are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

Table 10-1 summarizes the drilling on the project by operator, year and type. Figure 10-1 is a plan illustrating the locations of YMI and historical drillholes by drilling purpose. Figure 10-2 is a drillhole plan illustrating the locations and projected traces of Yellowhead and historical holes in the deposit area with the proposed pit outline.

Table 10-1: Summary of Yellowhead & Historical Drilling on the Property

Operator	Year	Drillhole ID's	No. of Holes	Core Size	Total (m)	Avg. Depth (m)	Purpose
Québec Cartier Mining Company	1967	67-H-1 to 6	6	NQ & BQ	546	91	Exploration
	1969	69-H-1 to 27	27	BQ	4,739	176	
Noranda Exploration Co. Ltd.	1968	NH-1 to 18	17		2,106	124	
	1969	NH-18 to 30	13		1,734	133	
	1970	NH-31 to 96	57		8,316	146	
Noranda / Québec Cartier Joint Venture	1970	J-1 to 12	12		2,329	194	
	1971	J-13 to 39	27		5,594	207	
	1972	J-40 to 43	4		457	114	
	1973	J-44 to 48	5		632	126	
Esso Resources Canada Limited	1983	LBC83-1	1	NQ	84	84	
Nu-Crown Resources Inc	1985	DDH-01 to 04	4	BQ	427	107	
	1987	DDH-05 to 14	10		942	94	
American Comstock Exploration Ltd	1996	96-1 to 8	8	NQ2	2,847	356	
Yellowhead Mining Inc	2006	HC06-01 to 12	12		4,101	342	Resource
	2007	HC07-13 to 52	40		15,880	397	
	2008	HC08-53 to 75	23		7,603	331	
	2010	HC10-76 to 82	7		3,487	498	
	2011	HC11-83 to 130	48		15,571	310	
		HC11-C01 to 08	8		1,791	224	Condemnation
		HC11-GM01 to 07	8	HQ	2,433	304	Geomechanical
		HC11-GT01 to 24	24		1,267	53	Geotechnical
		HC11-M01 to 04	4	PQ	441	110	Metallurgical
	2012	HC12-131 to 142	12	NQ2	3,803	317	Resource
		HC12-GT01 to 08	8	HQ3	442	55	Geotechnical
	2013	HC13-143 to 165	23	NQ2	8,166	355	Resource
Total	**1967 to 2013**		**408**		**95,741**	**235**	

10.1 Introduction – *Cont'd*

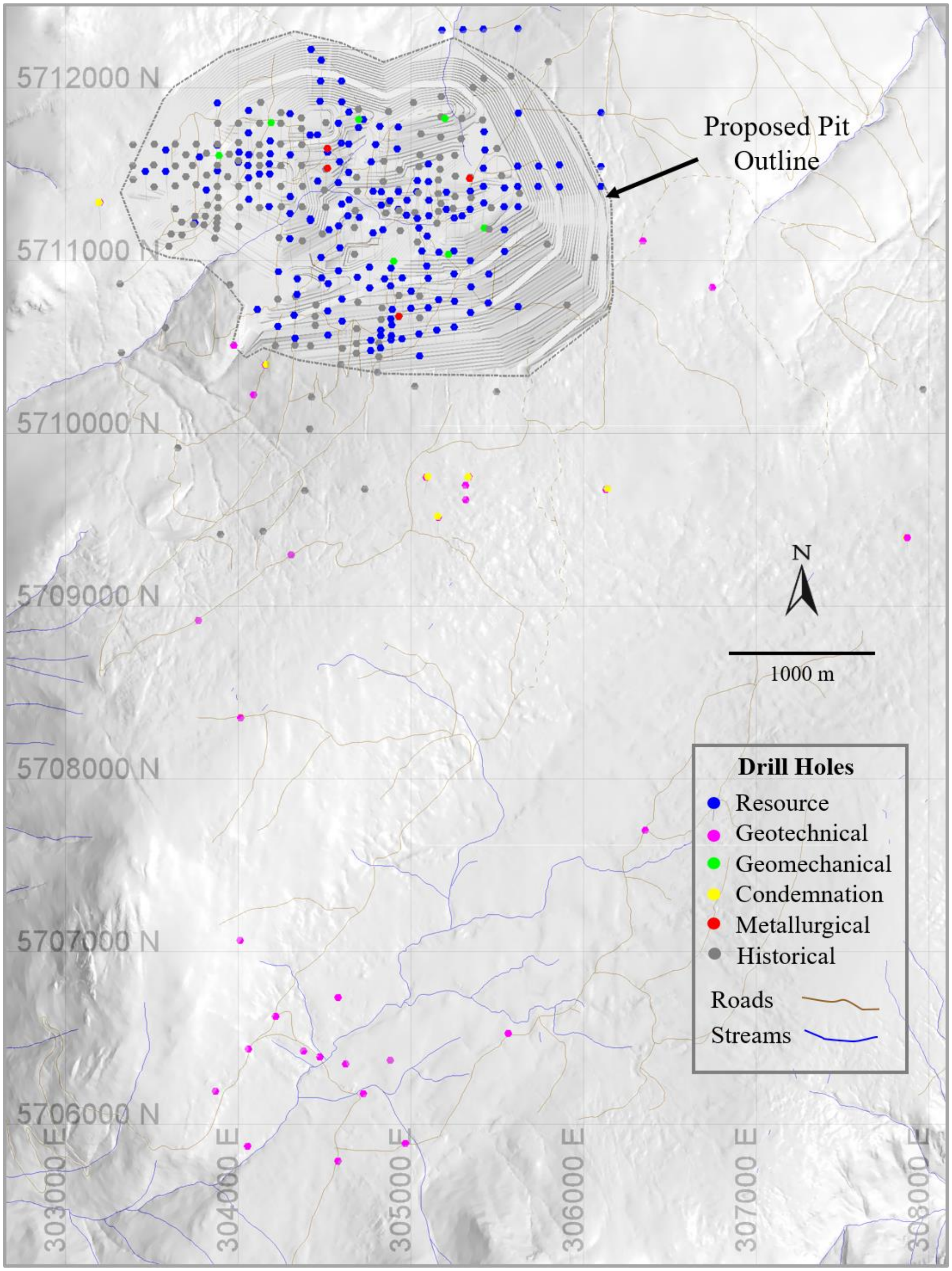

Figure 10-1: Plan of Yellowhead Drilling by Purpose

10.1 Introduction – *Cont'd*

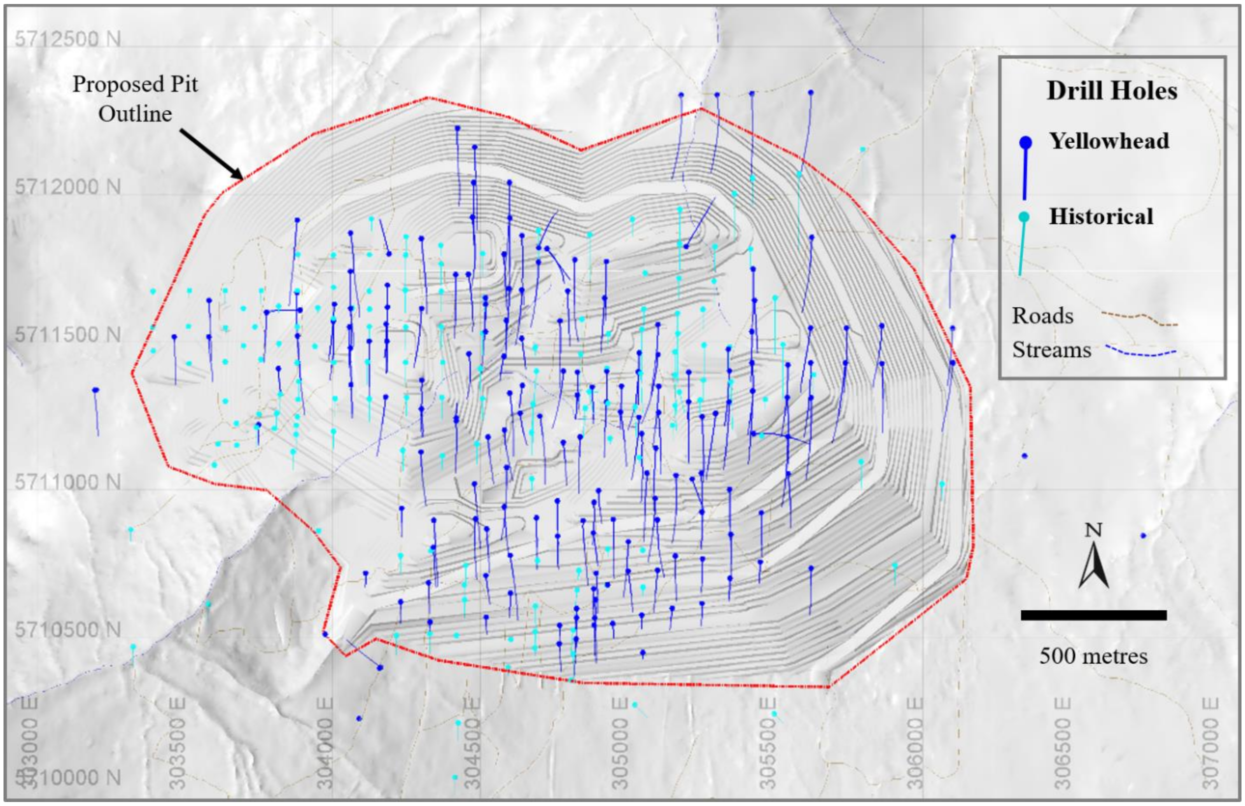

Figure 10-2: Plan of Drillholes in the Deposit Area with Proposed Pit Outline

Yellowhead relogged and resampled selected historical core in the deposit area from the Noranda 1968-1971 and American Comstock 1996 drill campaigns with the goal of verifying the historical analytical copper results. Results of this program showed good correlation of copper grades and thicknesses with the historically reported drill core intersections. Section 11.2 (a) describes the resampling and assaying work in further detail. Observations made during this program developed and improved the geological understanding of the deposit model, provided support for the hypothesis of a VMS mineralization style for the deposit and assisted in the targeting of new holes.

10.2 Drilling 2006-2013

The 64,990 m drilled in the 217 Yellowhead holes represents 68% of the total metres drilled on the property. Yellowhead and consultants geologically and geotechnically logged and photographed all core recovered from their drill programs. Over 90% of this drilling focussed on the confirmation, delineation and definition of copper resources within the main body of mineralization. Geomechanical, condemnation, geotechnical and metallurgical holes comprised the balance of the drilling.

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 stored at a secure facility in Vavenby, BC.

A typical drill run length for the Yellowhead program was 3 m with an overall average run length of 2.9 m. The average core recovery for the 20,288 drill runs cored and measured in these campaigns in the deposit area is 98% with an average RQD of 40%.

(a) Resource Drilling

Yellowhead completed 58,612 m in 165 holes of confirmation, delineation and infill drilling in support of geological modelling and resource estimation between early 2006 and mid 2013. Sampling and assaying included the entire cored length of all resource drillholes. Section 11 describes this in further detail. All holes were drilled NQ2 core size and most were oriented in a southerly direction at inclinations of −50° to −60°. Overall drill spacing in the central part of the deposit is from 50 to 70 m, increasing to over 100 m in the fringes.

In 2006, Yellowhead completed 4,101 m in 12 core holes numbered HC06-1 through HC06-12 for resource confirmation purposes. Nine drillholes targeted the western side of the deposit, while the remaining three drillholes targeted the eastern side. Drilling was oriented to the south at inclinations of −50° to −60°.

In 2007, a program to delineate and infill the northern and southern parts of the resource area included 15,880 m of drilling in 40 core holes numbered HC07-13 through HC07-52. These holes also extended below the intersections of historical holes to test the extent of mineralization at depth. Holes were oriented to the south at inclinations of −55° to −60°. A downhole orientation-marking tool used in holes HC07-39 and HC07-40 enabled orientation measurements to be made of geological features, including cleavages, foliations, veins and structures. The average of 1,933 measurements, 259° azimuth 30° N dip, confirmed the suitability of the preferred drill orientation used by Yellowhead and historical workers. All casing remained in the ground after drillhole completion for the 2006 and selected 2007 drillholes for possible re-entry purposes.

10.2 Drilling 2006-2013 – *Cont'd*

(a) Resource Drilling – *Cont'd*

The 2008 program consisted of 7,603 m in 23 core holes numbered HC08-53 to HC08-75, all oriented to the south at inclination of −60°. These infill and delineation drillholes targeted the east and southeast areas of the deposit.

There was no drilling in 2009.

The seven holes drilled in the 2010 program numbered HC10-76 through HC10-82 average 498 m in length for a total of 3,487 m. The three holes drilled in the west side and four on the east side of the deposit further extended the known depth extent of mineralization. This includes HC10-82, the longest hole at 606 m, drilled on the property. All were oriented to the south at inclination of −60°.

The extensive 2011 delineation and infill drill program totalled 15,571 m in 48 holes. The purpose was to target areas of low drilling density to increase confidence in the resource and assist in the creation of a geological model. Although most holes were drilled south at inclinations of −60°, a number of orientations deviated from this to intersect specific areas of mineralization and structure.

In the 2012 and 2013 programs, the focus on increasing the drill density in the deposit continued with 12 holes and 3,803 m and 23 holes and 8,166 m completed respectively in those years. All holes were drilled south at inclinations of −60°.

10.2 Drilling 2006-2013 – *Cont'd*

(b) Condemnation Drilling

In 2011, potential mineralization below proposed mine site infrastructure was tested in a 12-hole 1,791 m NQ2 core drilling program. These condemnation holes numbering HC11-C01 through HC11-C08 targeted proposed primary crusher, waste rock storage, mill building, truck shop, coarse ore and low-grade stockpile facilities. Holes drilled to depths of 200 m in a southerly direct at −60° except as noted. Drillhole HC11-C06 in the proposed west waste storage area was the longest. It drilled to a depth of 340 m because of its proximity to mineralization around the deposit, just 250 m to the east. HC11-C04 drilled subvertically to test a proposed low-grade storage area and proposed crusher site hole HC11-C08 drilled northwest at −47° to a depth of 246 m.

The two holes drilled closest to the deposit, HC11-C06 and HC11-C-08, had intercepts of >0.2% Cu over intervals ≥1 m, particularly the proposed crusher site hole. The other six holes did not return any significant results for copper.

10.2 Drilling 2006-2013 – *Cont'd*

(c) Metallurgical Drilling

Yellowhead completed a 4-hole, 441 m PQ core size (83 mm diameter) metallurgical drill program to collect drill core for metallurgical and crushing/grinding test-work in 2011. These drillholes twinned four historical holes NH-27, NH-29, J-4, and 69-H-22. Dawson Metallurgical Laboratories of Midvale, UT, received the crushing/grinding samples for test-work from these holes. G&T Metallurgical Services in Kamloops, BC received the remaining samples. Section 13 includes information on the metallurgical results. Sampling and geochemical analysis of 137 core samples from metallurgical drillhole HC11-M04 took place in addition to sampling in this hole specifically for metallurgical test-work.

10.2 Drilling 2006-2013 – *Cont'd*

(d) Geomechanical and Geotechnical Drilling

Knight Piésold completed a series of geomechanical and geotechnical drillholes as part of their site investigation studies. Geomechanical drilling undertaken in the proposed pit area consisted of eight HQ core size (63.5 mm diameter) drillholes totaling 2,433 m. These holes numbered HC11-GM01, HC11-GM01A, and HC11-GM02 to HC11-GM07, drilled in a variety of orientations to intersect proposed pit walls. In addition to core samples selected by Knight Piésold for the geomechanical studies, were 1,025 samples submitted for geochemical analysis from six of these holes.

Geotechnical drilling undertaken in various areas of proposed mine infrastructure consisted of a 24 HQ drillholes totaling 1,270 m in 2011. These 30 to 130 m long holes numbered HC11-GT01 to HC11-GT24 are vertical, except for proposed tailings embankment hole HC11-GT15 drilled northwest at −75°. There were 191 core samples collected and submitted for geochemical analysis from 13 of these holes.

Eight additional vertical geotechnical drillholes completed in 2012 total 442 m in length. These holes, numbered HC12-GT01 to HC12-GT08, are HQ3 core size (61.1 mm diameter). No sampling of these holes for geochemical analysis took place.

10.3 Density Measurements

The overall median bulk density value obtained from 10,739 drill core measurements in the deposit is of 2.78 t/m^3 and the average (mean) value is 2.80 t/m^3. Measurements taken by Yellowhead using the water immersion method were on dry, uncoated 10 to 12 cm long pieces of whole core. Selection was of two pieces of core from geochemical sample intervals in drillholes HC06-01 to HC07-39 (excluding HC06-08). The average of the two test values provided the density applied to each sample interval. Testing of only one piece of core took place where a lack of sufficient or appropriate sample material existed for a second test.

The Ohaus Scout Pro digital balance used for all weight determinations has 2.0 kg capacity and 0.1 g sensitivity. Calibration of the balance was with a 2 kg standard weight. Recorded measurements included water temperature, core length, dry sample weight in air and weight of the sample submerged in water. Calculation of sample specific gravity (SG) was by:

Specific Gravity = Dry weight in air ÷ (Dry weight in air – Weight in water)

Calculation of density was by the formula:

Density = Specific Gravity × Density of water

Sixty specimens re-analyzed at ALS laboratory in 2012 showed no significant differences to the Yellowhead measurements.

10.4 Survey & Topography 2005-2013

In 2005, Yellowhead converted the Noranda local grid to the NAD83 UTM Zone 11 North coordinate system, the grid currently in use on the property. As a check on the transformation, 20 historical drillholes from the Noranda, QCM, Noranda / QCM joint venture programs and all but two of the Comstock drillholes, were located in the field and resurveyed using a differential GPS. Differences were minor.

Yellowhead updated the topographic mapping based on one-metre resolution imagery in 2007. Cohesion Consulting checked the drill collars on cross section views against the 2007 topographic surface in 2019 and found no significant discrepancies.

Yellowhead staff and consultants surveyed all drillhole collar coordinates and elevations using a satellite-based Global Positioning System (GPS). The survey instrument used from 2006 to 2008 was a Trimble GeoExplorer XT Rover. Data from this unit were differentially corrected using information from the Williams Lake public domain GPS base station. Accuracy achieved by this method is sub-metre for easting and northing readings and 3 to 5 m for elevation readings. The use of drillhole collar elevations obtained from drillholes plotted on the 1 m contour interval digital terrain model provided improved accuracy.

Surveying of all drill collars from 2008 to 2013 was by a Trimble GeoExplorer XH Rover instrument utilizing a Tornado antenna. Differential correction of the collected survey points utilized data recorded by a Trimble 5700 base station and Zephyr antenna located at the Yellowhead field camp, 2.5 kilometres up the Jones Creek forest service road. Accuracy by this method is sub-metre for easting, northing and elevation readings relative to the base station. Elevations used for all drillholes during this period utilized GPS readings.

Upon completion of all resource holes, downhole surveying was by a multishot instrument utilizing a magnetic compass and inclinometer, with seven exceptions. The first two 2006 holes were by the acid-etch dip test method. Instrument failure precluded surveying in five pre-2008 holes. A single shot Sperry Sun downhole survey tool used as a backup survey system on a number of drillholes was at approximately 100 m intervals downhole as drilling proceeded.

All the condemnation and metallurgical drillholes were down hole surveyed for both azimuth and dip using digital multi-shot or single-shot instruments. Geotechnical drillholes were not down hole surveyed.

Five geomechanical holes were downhole surveyed. Downhole surveying did not take place on geomechanical holes HC11-GM03, HC11-GM07 and HC11-GM01 (abandoned and re-drilled as HC11-GM01A).

10.4 Survey & Topography 2005-2013 – *Cont'd*

Local concentrations of magnetic minerals, (i.e., magnetite and pyrrhotite), which are known to exist on the property can affect magnetic compass / inclinometer survey tool readings. Yellowhead personnel measured magnetic susceptibility of the core and reviewed downhole survey measurements for orientations that appeared suspect. Some instruments used automatically flagged measurements that appeared radically different from adjacent readings. Removal of all suspect surveys followed these assessments.

SECTION 11

SAMPLE PREPARATION, ANALYSIS AND SECURITY

SECTION 11: SAMPLE PREPARATION, ANALYSIS AND SECURITY

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11.1 Introduction

YMI and previous project operators systematically sampled and analyzed all potentially mineralized sections of drill core on the Yellowhead deposit for copper, the primary element of interest. Early operators in the 1960’s and 1970’s, typically only analyzed for copper. This expanded to include a handful of other elements in the programs of the 1980’s and 1990’s. From 2005 onwards, over 30 elements made up the standard assaying protocol for drill core, including historical core resampled and reanalysed since then. This historical core was from the Noranda, Noranda / QCM Joint Venture and Comstock drilling. Samples taken for copper assay from all historical and modern drillholes number over 55,000 with an average core length of 1.5 m. Table 11-1 lists the analytical laboratories used and the elements analyzed by the original operators for each drill program.

Table 11-1: Original Assay Laboratories & Elements Analysed – Drill Core

Years	Operator	Primary Assay Laboratory	Elements Analysed
1967, 1969	Québec Cartier	Bondar Clegg, N. Vancouver, BC & unknown lab(s)	Cu only†
1968-1970	Noranda	Eco-Tech Kamloops, BC & unknown lab(s)	
1970-1973	Noranda/ QCM JV		
1986	Aurun‡	ALS Minerals, N Vancouver, BC	Ag, Au
1983	Esso Minerals□	Min-En Laboratories, N. Vancouver, BC	Cu, Ag, Au, Pb, Zn
1985	Nu-Crown□	Acme Analytical, Vancouver, BC	Cu, Ag, Au, Ba, Pb, Zn
1987		Eco-Tech, Kamloops, BC	
1996	American Comstock	Acme Analytical, Vancouver, BC	Cu, Ag, Au, Mo, Pb, Zn
2006	YMI	Eco-Tech, Kamloops, BC	Cu, Ag, Au & 22 Elements
2007, 2008			Cu, Ag, Au & 27 Elements
2010, 2011*			Cu, Ag, Au & 32 Elements
2011*, 2012, 2013		ALS Minerals, N. Vancouver, BC	Cu, Ag, Au & 31 Elements

† Noranda assayed a small number of selected samples and composites for Au, Ag, Cu, Pb and Zn.
‡ Aurun re-assayed 38 sample intervals from seven Noranda / Québec Cartier Joint Venture drillholes for Au and Ag.
□ Esso Minerals and Nu-Crown did not drill any holes in the deposit area.
* 2011 drillholes from HC11-83 to HC11-98 assayed by Eco-Tech. Holes HC11-95, 97, 99 assayed by Eco-Tech & ALS. All other 2011 holes assayed by ALS Minerals.

11.1 Introduction – *Cont'd*

Resampling and reanalysis of historical core by YMI provided precious metal and multi-element results for 132 pre-2006 holes drilled on the deposit. The creation of two separate assay tables in the drillhole database was necessary, as it was not possible to match the original assay intervals in many instances. The primary table includes the intervals and results of copper assays only. It comprises the original Cu results from drill core intervals as sampled and assayed by the original workers. The second assay table includes Au, Ag and a number of other elements. It has different from-to intervals for many resampled historical holes, but intervals for modern holes match. Results in this second table are from sampling and analysis by YMI from 2005 onwards. Just under 55,000 assay intervals are in this table with an average length of 1.4 m. Average interval lengths for resampled historical core in the second table tend to be shorter than in the typical 3 m sample intervals of the original Cu-only samples.

11.2 Sampling by Yellowhead Mining

(a) Historical Drill Core (2005-2011)

YMI started exploring the Yellowhead Copper Deposit in 2005 by salvaging, re-logging and resampling the remaining historical diamond drill core. The objectives of this program were to confirm historically reported copper grades, perform precious metal and multi-element analyses, to obtain host rock geological information and to gain further understanding on mineralization controls.

Historical core recovered from the old Noranda camp was moved to a core processing facility in Vavenby, BC in 2005. YMI relogged and resampled historical core for assay between 2005 and 2011 using similar procedures to those described in sections 11.3 and 11.4, with important differences as noted in the following paragraph.

Resampling was of the remaining half core in its entirety due to a strong prevalent schistosity in the rock that precluded accurate quarter core sampling. Some historical core boxes were in very poor condition and sections of core were missing. These were marked as not sampled (NS) based on estimated start and endpoints. Resampling took place at nominal 3 m intervals to match the original samples wherever possible. Actual sample length varied considerably due to missing core and geological selection criteria. The average interval length of resampled sections was 2.2 m. Reanalysis of sample pulps from the 1996 American Comstock drill program also took place after retrieval from storage at Acme Laboratories.

Of the 191 drillholes completed prior to YMI’s involvement, 131 drillholes were subject to resampling and relogging. The resampled intervals totalled 18,874 m of the 30,745 total metres drilled by the historical operators, or over 66% of the historically cored intervals. A total 9,465 samples from historical core were analysed. Of the 131 reanalysed drillholes, 127 are located within, or immediately adjacent to, the deposit area.

The historical core resampling and reanalysis program was successful in validating the reported historical copper grades and providing a substantial number of additional gold and multi-element analyses.

11.2 Sampling by Yellowhead Mining – *Cont'd*

(b) YMI Drill Core (2006-2013)

YMI maintained full chain of custody control for all analytical samples from the 2006 through 2013 drill campaigns, from collection at the drill rig through to delivery at the analytical laboratory. Drill company employees, YMI employees and company consultants were in charge of the security of all drill core on the property during drilling, logging and sampling procedures. Figure 11-1 is an example flow chart of the sampling, sample preparation, security and analytical procedures for the 2013 drill program.

11.2 Sampling by Yellowhead Mining – *Cont'd*

(b) YMI Drill Core (2006-2013) – *Cont'd*

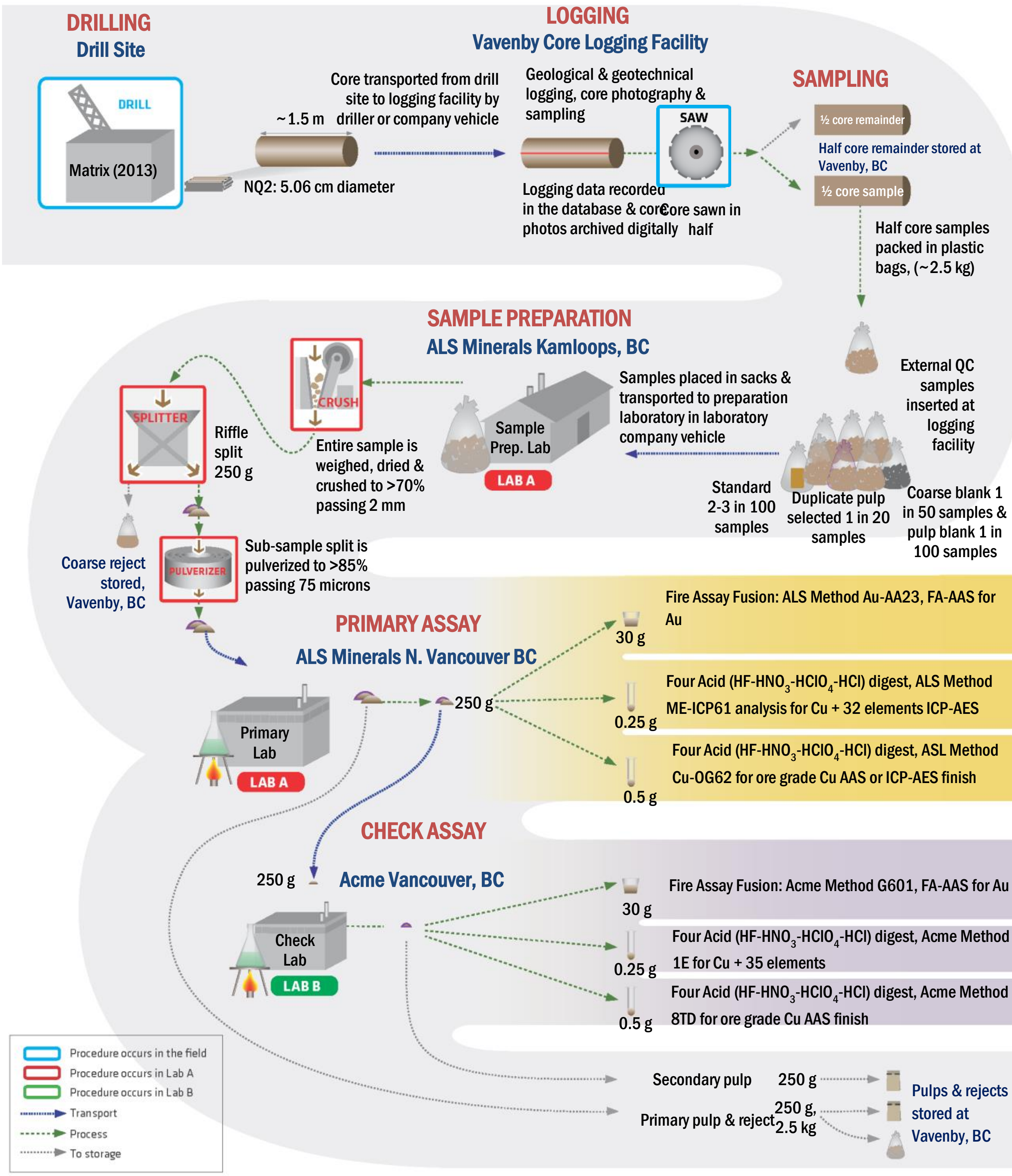

Figure 11-1: Sampling, Sample Preparation, Security & Analytical Flow Chart (2013)

11.2 Sampling by Yellowhead Mining – *Cont'd*

(b) YMI Drill Core (2006-2013) – *Cont'd*

Sample intervals are nominally 1.0 to 2.0 m in length, with breaks at changes in lithology, alteration, mineralization and core size accounting for most variations from this. Mineralization broadly tends to follow the trend of the stratigraphy and changes in mineralization intensity are often gradual and cannot be easily discriminated by inspection, consequently intervals are typically at even metre or half metre increments. The median sample interval length is 1.2 m.

Core sampling took place based at intervals marked by a geologist upon completion of logging procedures. A company technician used a diamond-bladed rock saw to cut the core in half lengthwise. Half of the core went into the appropriately numbered and tagged sample bag that was sealed and placed in a secure location prior to shipment. The remaining half went back into the core box for long-term storage. Bags containing samples were stored in a locked, secure structure pending packing and transport to the laboratory.

Sorting and scanning of bags containing drill core samples and placement into rice bags took place before transport to analytical laboratories in Kamloops, BC. Prior to 2007, delivery was by commercial courier. After that, laboratory personnel picked up the samples at the Vavenby core logging facility and took responsibility for their transport and delivery.

Eco-Tech Laboratories Ltd. (Eco-Tech) did the sample preparation and analysis for the project from 2005 to 2011 at their laboratory in Kamloops, BC. Stewart Group purchased Eco-Tech in July 2008 and continued operating the Kamloops laboratory under the Eco-Tech name until 2011. In July 2011, ALS Minerals (ALS) purchased Stewart Group and sample preparation work transferred to the ALS laboratory in Kamloops at the end of that year for balance of the program. The ALS laboratory in North Vancouver, BC completed the analytical work for the 2012 and 2013 programs.

11.3 Sample Preparation and Analysis Eco-Tech (2005-2011)

Eco-Tech, a laboratory registered under ISO9001:2008 for the provision of geochemical, assaying and environmental analytical services, performed sample preparation and analysis for the historical resampling and the 2006 through 2011 drilling and sampling programs.

(a) Sample Preparation

Dried drill core samples were subject to comminution prior to analysis. The first step was to crush the entire sample using jaw crushers and cone or rolls crushers to achieve a nominal –10 mesh (2 mm) size. Splitting of the crushed product by passing it through a Jones riffle provided a 250 g sub-sample. Preparation of the 250 g pulverized sample (assay pulp) to a >95% passing –140 mesh (0.1 mm) size was by ring and puck pulverizer. Rolling of the pulverized samples after that homogenized them further.

11.3 Sample Preparation and Analysis Eco-Tech (2005-2011) – *Cont'd*

(b) Copper Analysis

Assay-level analysis performed on all samples with elevated concentrations of copper were by aqua regia (HCl-$HN0_3$) acid digestion of a 0.5 g aliquot (analytical sub-sample) with AAS finish. Laboratory quality control procedures included repeats every nine samples and the use of certified reference materials for each batch of 35 samples or fewer.

11.3 Sample Preparation and Analysis Eco-Tech (2005-2011) – *Cont'd*

(c) Multi-Element Analysis

Multi-element analysis of all samples was by 4-acid digestion ($HF-HClO_4-HN0_3-HCl$) with ICP-AES finish. This method provided results for up to 35 elements, including Cu and Ag. There are over 44,000 results by this method in the drillhole database. Table 11-2 is a list of the elements analyzed by the 4-acid ICP method at Eco-Tech.

Table 11-2: Elements Analysed by Eco-Tech 4-Acid Digestion ICP Method

Element	Element	Element	Element	Element	Element
Ag	Ca	Hg†	Mo	S†	Ti
Al	Cd	K	Na	Sb†	U*
As*	Co	La*	Ni	Sc†	V
Ba	Cr	Li†	P	Se†	W
Be†	Cu	Mg	Pb	Sn*	Y
Bi	Fe	Mn	Rb‡	Sr	Zn

* Element not analysed in years 2005 and 2006.
† Element not analyzed in years 2005 through 2007.
‡ Only one hole analyzed (HC10-76).

Thresholds of Cu results from the 4-acid ICP method also determined which samples were re-analysed by single element Cu assay. This threshold was ≥2,900 ppm Cu from 2005 to 2008 and decreased to ≥2,000 ppm Cu between 2010 and 2011. Single element ICP values greater than the upper detection limit also triggered a small number of single element, aqua regia digestion AAS overlimit assays for silver, lead and zinc using similar methods to the Cu assays. The upper limits for these elements by the 4-acid digestion ICP-AES method is 30 ppm for Ag and 10,000 ppm for Pb and Zn.

A second multi-element ICP-AES method employed on all YMI core, surface rock and soil samples prior to May 31, 2007 was aqua regia digestion of a 0.5 g aliquot for the determination of 29 elements, including Cu and Ag. There are over 8,800 results on drill core by this method. Table 11-3 lists the elements analyzed by this method by Eco-Tech.

Table 11-3: Elements Analysed by Eco-Tech Aqua Regia Digestion ICP Method

Element	Element	Element	Element	Element	Element
Ag	Ca	Fe	Mo	Sb	V
Al	Cd	K*	Na	Sn	W
As	Co	La	Ni	Sr	Y
Ba	Cr	Mg	P	Ti	Zn
Bi	Cu	Mn	Pb	U	

* Some samples were not analysed for K.

11.3 Sample Preparation and Analysis Eco-Tech (2005-2011) – *Cont'd*

(c) Precious Metal Analysis

Gold analysis performed on all core sampled by YMI was by fire assay with an AAS finish. A 30 g aliquot mixed with litharge and appropriate fluxes was subject to fusion and cupellation at high temperatures. Analysis of the resulting doré bead after parting was by AAS with results reported in ppb. The reportable concentration range for this method is 5 to 1,000 ppb. There are almost 55,000 Au assays by this method. Values >1,000 ppb were re-analysed by the same fire assay method with a gravimetric finish and results reported in gpt (ppm).

Analysis for palladium of historical drill core samples collected in 2005 and one YMI hole in 2008 used this same analytical method, reporting units and range as the gold assays. There are Pd assays for 697 samples from 10 historical holes and 96 samples from drillhole HC08-54.

11.3 Sample Preparation and Analysis Eco-Tech (2005-2011) – *Cont'd*

(d) Whole Rock Analysis

Whole rock analysis completed by Eco-Tech on 57 core and surface rock samples selected in 2009 for petrographic analysis was on a 0.5 g sub-sample fused with lithium metaborate ($LiBO_2$) and finished by ICP-AES.

11.3 Sample Preparation and Analysis Eco-Tech (2005-2011) – *Cont'd*

(e) Surface Samples

Descriptions of the soil and surface rock sampling procedures of YMI are in section 9-4. Soil samples submitted to Eco-Tech were prepared by sieving at 80-mesh (0.18 mm) to obtain an analytical sub-sample. Samples with insufficient material for analysis at minus 80-mesh were screened at a coarser fraction and flagged accordingly. Surface rock samples were prepared in the same manner as drill core samples. Analysis of soil and rock samples was by the same aqua regia digestion ICP-AES and gold fire assay AAS methods as for drill core, with some rock samples also analysed by 4-acid digestion ICP-AES.

11.4 Sample Preparation and Analysis ALS Minerals (2011-2013)

ALS Minerals Kamloops sample preparation facility is ISO 17025:2005 certified and ALS Minerals laboratory in North Vancouver, BC is ISO 9001:2015 registered and ISO/IEC 17025:2017 certified. This accreditation also applies to mineral analysis by ALS methods for the determination of Cu, Au and multiple-elements performed on the Yellowhead samples in the 2011 through 2013 drill programs.

(a) Sample Preparation

Specifications of drill core sample preparation at ALS were drying, crushing to >70% passing 10 mesh (2 mm), riffle splitting of a 250 g sub-sample and pulverization of that sub-sample to >85% passing 200 mesh (75 micron).

11.4 Sample Preparation and Analysis ALS Minerals (2011-2013) – *Cont'd*

(b) Copper Analysis

Copper assays completed on all samples analyzed was by ALS laboratory method Cu-OG62, in which 0.5 g aliquots are subject to four acid digestion and analytical finish by either AAS or ICP-AES.

11.4 Sample Preparation and Analysis ALS Minerals (2011-2013) – *Cont'd*

(c) Multi-Element Analysis

Analysis for Cu and 32 other elements was by ALS trace level multi-element Method ME-ICP61 in which a 0.25 aliquot is subject to four acid digestion and instrumentation finish by ICP-AES. Table 11-4 lists the elements reported, units and detection limits of this method.

Table 11-4: Details of Elements Reported on ALS Method ME-ICP61

Element	Symbol	Units	Lower Limit	Upper Limit
Silver	Ag	ppm	0.5	100
Aluminum	Al	%	0.01	50
Arsenic	As	ppm	5	10000
Barium	Ba	ppm	10	10000
Beryllium	Be	ppm	0.5	1000
Bismuth	Bi	ppm	2	10000
Calcium	Ca	%	0.01	50
Cadmium	Cd	ppm	0.5	500
Cobalt	Co	ppm	1	10000
Chromium	Cr	ppm	1	10000
Copper	Cu	ppm	1	10000
Iron	Fe	%	0.01	50
Gallium	Ga	ppm	10	10000
Potassium	K	%	0.01	10
Lanthanum	La	ppm	10	10000
Magnesium	Mg	%	0.01	50
Manganese	Mn	ppm	5	100000

Element	Symbol	Units	Lower Limit	Upper Limit
Molybdenum	Mo	ppm	1	10000
Sodium	Na	%	0.01	10
Nickel	Ni	ppm	1	10000
Phosphorus	P	ppm	10	10000
Lead	Pb	ppm	2	10000
Sulphur	S	%	0.01	10
Antimony	Sb	ppm	5	10000
Scandium	Sc	ppm	1	10000
Strontium	Sr	ppm	1	10000
Thorium	Th	ppm	20	10000
Titanium	Ti	%	0.01	10
Thallium	Tl	ppm	10	10000
Uranium	U	ppm	10	10000
Vanadium	V	ppm	1	10000
Tungsten	W	ppm	10	10000
Zinc	Zn	ppm	2	10000

11.4 Sample Preparation and Analysis ALS Minerals (2011-2013) – *Cont'd*

(d) Precious Metal Analysis

Gold assays completed on all samples were by ALS Method Au-AA23 in which a 30 g aliquot mixed with litharge and borax flux was subject to fusion and cupellation at high temperatures. Analysis of the resulting doré bead after parting was by AAS with results reported in ppm to a lower limit of 0.005 and an upper limit of 10 ppm.

11.5 Analysis - Other Laboratories

(a) Geoscience Laboratories

In 2008, Geoscience Laboratories, formerly Geo Labs of Sudbury, ON, completed whole rock and trace element analyses of 27 core samples from 15 YMI and 6 historical drillholes. Sample preparation was to jaw crush, riffle split and pulverize samples in a planetary ball mill. The whole rock and trace element analytical method was X-ray fluorescence (XRF). Major oxides determined are Al_2O_3, CaO, Fe_2O_3, K_2O, MgO, MnO, Na_2O, P_2O_5, SiO_2 and TiO_2. Analysis for selected trace elements included Ba, Co, Cs, Mo, Nb, Sc, Sn, Sr, Rb, Zr and V. Other analyses included total carbon reported as CO_2, total sulphur reported as S, ferrous iron reported as FeO, moisture content, rare earth elements, high field strength elements and large-ion lithophile elements.

11.5 Analysis - Other Laboratories – *Cont'd*

(b) Check Assay Laboratories

Inter-laboratory check assays for copper done on 5% of the original assay pulps were part of the drill program Quality Assurance / Quality Control (QA/QC) protocol. Check laboratories used similar analytical methods to the primary laboratory.

Acme was the check assay laboratory for the 2006 and 2010 through 2013 drill programs. Acme analysed original assay pulps from the 2006 program for Cu by 4-acid digestion of a 0.5 g aliquot with an AAS finish. Two methods were added to the check assay protocol for the 2010 through 2013 programs, 4-acid digestion ICP-AES finish on a 0.25 g aliquot for 36 elements including copper and gold by fire assay fusion of a 30 g sample with an AAS finish.

For the 2007 and 2008 drill programs, the check assay laboratory was Assayers Canada (Assayers) of Vancouver, BC (now SGS). Assayers analysed original assay pulps for copper by nitric, hydrobromic and hydrochloric (HN03, HBr, HCl) acid digestion of a 1 g aliquot with AAS finish.

11.6 Quality Assurance and Quality Control

YMI implemented an effective external QA/QC program and applied it to the 2005 through 2013 drilling and sampling programs. Insertion of QA/QC samples was designated by the core-logging geologists at the logging facility within the regular sample stream prior to shipment of samples to the sample preparation and analytical laboratories. These QA/QC procedures were in addition to those used internally by the analytical laboratories. Table 11-5 lists the QA/QC sample types used.

Table 11-5: QA/QC Sample Types Used

Sample Type	Description	Percent of Total
Regular	Samples of actual drill core submitted for preparation and analysis at the primary laboratory.	90.8%
Duplicate	An additional split taken from the remaining assay pulp after analysis and submitted to a check laboratory. Selected over broad grade ranges.	4.6%
Standard	Control sample with mineralised material in pulverised form with a known concentration and distribution of elements of interest. Randomly inserted.	2.3%
Blank	Control sample in coarse or pulverised form with no appreciable grade used to test for contamination. Randomly inserted.	2.3%

YMI technical staff and consultants monitored the Cu results of control samples, including selected inter-laboratory duplicates, inserted standards and blanks. Failed batches resulting from control samples outside set limits, duplicated sample pairs in disagreement and high blanks were subject to review. If no field logging or coding errors were evident, laboratory reruns of affected analytical batches ensued. QA/QC review also applied the rerun results returned. Results from reruns that passed QA/QC superseded the original data in failed batches.

11.6 Quality Assurance and Quality Control – *Cont'd*

(a) Standards

Certified reference materials are assay standards used for QA/QC monitoring purposes with expected mean values and control limits. YMI inserted standards of prepackaged pulps from CDN Resource Laboratories or Ore Research that were typically 60 to 150 g in size. Table 11-6 lists the 15 different standards used in the YMI sampling programs.

YMI improved their standard insertion protocol as the drill programs progressed. The insertion rate of one standard for every 50 regular samples used from 2006 through 2008 was increased to one in 33 regular samples from 2011 onwards. This gave an effective insertion rate of about one standard for every 40 regular samples overall. Discontinuation of the practice of inserting non-blind standards, in which the analytical laboratory can readily identify the standard, occurred in late 2007. Insertion of blind standards took place from then until 2013.

Standards submitted in soil batches was at a rate of one in 100 samples and typically one per batch for surface rock samples.

Review of copper and gold results of inserted standards reported by Eco-Tech and ALS resulted in analytical reruns of a reasonably small number of batches. Reanalysis of these batches returned acceptable results for the standards and application of these revisions took place accordingly. This protocol provided good confirmation of the veracity of the copper and gold results used in the drillhole database.

Table 11-6: Assay Standards Certified Mean Values

Standard	Cu (%)	Au (gpt)	Times Used
CDN-CGS-6	0.318	0.26	20
CDN-CGS-8	0.105	*0.08**	29
CDN-CGS-9	0.473	0.34	148
CDN-CGS-12	0.265	0.29	77
CDN-CGS-13	0.329	1.01	29
CDN-CGS-15	0.451	0.57	171
CDN-CGS-22	0.725	0.64	20
CDN-CGS-24	0.486	0.487	118
CDN-CGS-27	0.379	0.432	183
CDN-CGS-29	0.585	0.228	95
CDN-CM-1	0.853	1.85	105
CDN-CM-25	0.191	0.228	35
CDN-CM-27	0.592	0.636	35
CDN-FCM-1	0.94	1.71	1
CDN-HLLC	1.49	0.83	45
OREAS 152A	0.385	0.116	169

* Provisional value only.

11.6 Quality Assurance and Quality Control – *Cont'd*

(b) Duplicates

The protocol for duplicate sample analysis was to submit the original assay pulp to a second laboratory after receipt of assay results from the primary laboratory. Sample selection was not random, but targeted representative copper grade ranges. A standard was included with each batch of duplicate pulps sent to the check laboratory. Drill core samples from the 2006 to 2008 drill programs sent the check assay laboratories numbered about 5% of the total, or one in 20 samples. This ratio decreased to 4%, or one in 25 samples, from 2010 onwards for an overall effective rate of about 4.3%.

Eco-Tech and ALS also analysed duplicate splits of assay pulps and coarse rejects and reported them on their analytical certificates.

The results of the inter-laboratory pulp duplicate analysis program on drill core samples substantiate the copper results of the original assay laboratories.

Historical core resampling programs by YMI resulted in over 2,100 half-core duplicate core assay pairs for copper. Assay results from re-assayed historical core correlate well with the historically reported copper grades from similar core sections.

11.6 Quality Assurance and Quality Control – *Cont'd*

(c) Blanks

Blanks are control samples with no appreciable grade used to test for contamination. Coarse blanks inserted for analytical QA/QC purposes consisted of visually barren crushed granite tile and decorative limestone landscape rock prior to 2012. They are not true analytical blanks, as their copper and gold content prior to insertion is unknown.

The premise for using granite tile and limestone blanks was that they contained very low levels of copper and gold. However, a number of results received on these uncertified coarse blanks during the course of the YMI drill campaigns were anomalously high for copper or gold, typically from two to 10 times the anticipated values. Possible reasons for this include mislabelling of blank and regular samples in the field, cross-contamination of samples during sample preparation and challenges of analysing high carbonate samples in a stream of generally low carbonate samples, amongst others. The overall impact of this is reasonably low, as the high blank results are still well below a reasonable threshold of what constitutes mineralized rock. However, use of these blanks for QA/QC monitoring was not ideal.

Two certified blank materials obtained from Analytical Solutions Ltd (ASL) for use in the 2012 and 2013 drill programs are designated as ASL-Blank-125 (100 g pulp blank) and ASL-Blank-10 (500 g coarse blank). Monitoring and control of the certified blanks proceeded in a similar way to the assay standards. These certified blanks provided better quality assurance and quality control. Table 11-7 lists the blanks used and average of the results received for copper and gold.

Table 11-7: Blanks Inserted

Standard	Average Cu (ppm)	Average Au (ppb)‡	Times Used
Crushed Granite Tile	11†	4.0	26
Limestone Landscape Rock	10	3.1	1,117
ASL-Blank-125 (pulp)	5.2	2.5	88
ASL-Blank-10 (coarse)	11	2.5	10

† One outlier of 457 ppm removed.
‡ Calculated based on <5 ppb value set as 2.5 ppb for calculation purposes.

11.7 Conclusion

The authors are of the opinion that the security, sampling, sample preparation and analytical methods used on the historical and modern Yellowhead Copper Project drill core is comparable to industry standard practice in mineral deposits of this type. Furthermore, the QA/QC measures and protocols used lend credence to the veracity of the drillhole database.

SECTION 12

DATA VERIFICATION

SECTION 12: DATA VERIFICATION

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12.1 Verification of Drill and Assay Data

R. Simpson, P.Geo, visited the project site on July 11 and 12, 2011 in order to review the drilling, sampling, and QA/QC procedures. During the site visit Mr. Simpson verified:

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

In 2012, R. Simpson independently audited the sample database for location accuracy, downhole survey errors, interval errors and missing sample intervals. He also reviewed the sample QA/QC results.

In 2019, the Cohesion Consulting Group (CCG) completed an audit of the Yellowhead project drillhole database. CCG reviewed the digital files comprising the drillhole database, assay certificates, geological models and supporting documents used in the mineral resource and mineral reserve estimates. The audit found no errors, omissions, QA/QC failures or differences between this drillhole database and the supporting documents of significance to the resource and reserve estimate.

12.2 Other Data Verification

Relevant sections of this report describe the verification of metallurgical, hydrological, environmental baseline and geotechnical data. The conclusion is that the data meets an acceptable standard for projects of this type.

12.3 Conclusion

Results of verification work completed on the data indicates that it is of good quality. The Qualified Person is of the opinion that the data is adequate to support geological modelling, resource and reserve estimation and economic analysis of the Yellowhead Copper Deposit.

SECTION 13

MINERAL PROCESSING AND METALLURGICAL TESTING

SECTION 13: MINERAL PROCESSING AND METALLURGICAL TESTING

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13.1 Introduction

Ore from the Yellowhead deposit is volcanogenic in origin and contains copper sulphide mineralization amenable to concentration by flotation.

Taseko acquired the project in February 2019, and since that time no additional metallurgical test work has been undertaken. The basis of process design for the project was informed from the feasibility level metallurgical test work program conducted in 2011 and early 2012 at G & T Metallurgical Services Ltd., in Kamloops, BC for YMI.

This test program consisted of a suite of open circuit batch flotation testing, lock cycle testing, ore hardness testing, a pilot plant campaign, and mineralogical characterization of both a primary master composite representing feed from the earlier phases of mine development along with a suite of composite samples representing variable lithology and discreet spatial zones within the pit. Additional laboratory comminution test work conducted in 2011 at FLSmidth of Bethlehem, Pennsylvania, was also used to inform process comminution circuit design and power requirements.

13.2 Historical Metallurgical Testing

Metallurgical testing on the Yellowhead deposit dates back to 1968. The first metallurgical test work program was undertaken at Lakefield Research of Canada Limited, where preliminary flotation tests and mineralogical examinations were conducted on composite samples of sulphide ore and oxide ore. It was concluded that chalcopyrite was the only copper mineral identified in the sulphide ore sample. The oxide ore sample contained both malachite and chalcopyrite. Fine veinlets of galena and silver were seen in both samples. The sulphide sample, with a head grade of 0.42% Cu, demonstrated amenability to flotation concentration at a primary grind of 78% passing 75 µm, yielding a 25.7% Cu concentrate grade at 83% recovery, while the oxide ore yielded poor flotation performance.

Similar performance and conclusions were drawn from a subsequent test program undertaken at the Noranda Ore Dressing Laboratory in 1971. Two composites (0.42% Cu and 0.46% Cu head grades) of diamond drill core were sent to the lab for mineralogical evaluation, and flotation test work. The mineralogical evaluation concluded the copper present in the samples was in the form of fairly coarse-grained chalcopyrite with minor amounts of bornite and covellite. Both composites were found to be quite friable and easy to grind, with a Bond rod mill work index (RWi) of 10.1 kWh/t. The flotation testing was conducted at a primary grind of 62% passing 75 µm with regrinding of rougher concentrate to produce a projected 25% Cu concentrate grade at 83% Cu recovery.

YMI conducted four exploration drilling programs on the property from 2005 – 2008, from which 0.76% Cu high grade (CME Composite) and 0.35% Cu low grade (CME LG Composite) drill core composites were sourced. These composites were sent to Process Research Associates Limited (PRA), in Richmond, BC for metallurgical testing.

Comminution testing resulted characterization of the ore as medium to soft with a metric Bond ball mill work index (BWi) of 11 kWh/t.

The effect of primary grind size on rougher flotation performance was investigated on both CME composites over a size range of 64 µm to 106 µm. Results indicated that both copper and silver recovery to rougher concentrate was essentially unaffected over the size range; however, both iron and gold recoveries were more variable with grind size. Finer grinding recovered more pyrite and increased mass pull, while the coarsest grind resulted in the highest gold grades and recoveries to the rougher concentrate. The coarsest grind size of 106um was selected for further regrind and cleaner flotation testing, where the effect of varying collector dosages, pyrite suppression reagents, and pH conditions were tested. Test results indicated that a combination of using more selective collectors such as 3418A and Sodium Isopropyl Xanthate (SIPX) combined with an elevated cleaner pH and a pyrite

13.2 Historical Metallurgical Testing – *Cont'd*

depressant would improve concentrate grade. Upon establishing best results, PRA conducted two lock cycle tests on the CME LG composite sample testing open circuit and closed-circuit cleaner configurations. The circuit with a discarded first cleaner tail returned the best result with 87.6% copper recovery to a 31.5% Cu concentrate.

Additional metallurgical testing was conducted in 2010/2011 at G&T to expand on the findings of previous testing. This work was performed on a new master composite sample with a head grade of 0.32% Cu. Mineralogical evaluation of the master composite sample yielded similar results to previous mineralogical findings with chalcopyrite being the most abundant copper bearing mineral making up 96% of the copper in the sample, with minor copper deportment associated with bornite, chalcocite, covellite, and tennantite. The primary gangue minerals found in the sample were quartz, muscovite, chlorite and carbonates.

Both BWi and RWi ore hardness testing was completed on the master composite sample. The BWI was found to be 12.9 kWh/t and the RWi was found to be 11.3 kWh/t, which was consistent with previous results, indicating the sample being medium to soft ore. Additional comminution tests to determine the SAG milling characteristics were also performed using the JK SMC testing protocol indicating that the ore was amenable to SAG milling and that the sample drop weight index is in the bottom 35% percentile with respect to ore hardness from the SMC test database.

Rougher flotation testing was completed to investigate the impact of coarser primary grind sizes in a range of 146 µm to 204 µm. The tests results indicated rougher flotation metallurgical performance was not very sensitive to primary grind size and a target grind size of 180 µm was selected for lock cycle testing. Two lock cycle tests consisting of rougher flotation, regrind stage and three stages of closed-circuit cleaning were conducted using different flotation collectors. The two tests produced consistent copper recoveries between 82% to 83% with concentrate grades ranging from 24.5% to 28.7% Cu, gold recoveries were reduced from 58% to 23% as the concentrate grade increased.

In 2011 additional testing was conducted at G&T focusing on rejection of pyrite in the flotation cleaner stage. This program used the same composite sample as the preceding G&T testing, but the majority of development work used a finer primary grind of 96 µm and a regrind size varying from 17 µm to 22 µm. A total of six lock cycle tests were conducted during this phase of work including a baseline test using conditions from the PRA program. The program concluded that suitable concentrate grade and recovery could be achieved using a combination of regrinding and lime for pyrite depression. The best performing lock cycle test produced a final copper concentrate grade of 28.7% Cu at 85% Cu recovery, and a gold recovery of 49%.

13.2 Historical Metallurgical Testing – *Cont'd*

At the conclusion of this test program a feasibility level metallurgical test program was conducted at G&T in 2011/2012 to investigate the metallurgical response from a range of ore zones within the ore deposit and further develop the process flowsheet.

13.3 Ore Characterization

(a) Sample Origin

In 2011, a drilling program sourced a sample of PQ sized core for the feasibility study metallurgical test work program conducted at G&T. Four drillhole locations were specifically selected with consideration given to obtaining a suite of sample lithologies and grades from spatially unique zones representing ore feed from the earlier pit phases of the mine life. A sample of 5,261 kg of whole diamond drill core was sent to G&T for testing and 752 kg of sample was sent to FLS for comminution test work.

13.3 Ore Characterization – *Cont'd*

(b) Sample Composite Blends

From the drillhole database, an estimate of the relative proportion of the major lithologies within the deposit was calculated. Intercepts from the four drillholes were used to construct 10 composite samples representing the major lithology and grade profiles of the deposit detailed in Table 13-1. These lithology composites were partially used in proportion to their relative abundance to construct the master composite sample designated Master Composite 2. Additionally, 6 zonal composites representing normal and low-grade samples from the south, east and west zones of the deposit were constructed. 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.

Three lithologies dominate the deposit. Approximately 72% is quartz eye schist with some slight variation in the precise breakdown of minerals. Schists (without quartz eyes) represents 13% and phyllites represents 11%. All other classifications combined represent less than 5% of the overall resource.

13.3 Ore Characterization – *Cont'd*

(b) Sample Composite Blends – *Cont'd*

Table 13-1: Metallurgical Grade Lithology Composites

Sample						Composite	
Hole	**From (m)**	**To (m)**	**Length (m)**	**Lithology**	**Est Avg Cu Grade (%)**	**Mass (kg)**	**Lithology**
HC11-M04	147	150	3	Phyllite	0.22	258	Phyllite 1
HC11-M03	22	39	17	Phyllite	0.32		
HC11-M04	98	101	4	Phyllite	0.33		
HC11-M03	96	100	4	Qz Eye Schist	0.15	488	Qz Eye Sch 1
HC11-M03	81	84	3	Qz Eye Schist	0.16		
HC11-M04	27	39	12	Qz Eye Schist	0.18		
HC11-M02	10	24	13	Qz Eye Schist	0.19		
HC11-M04	89	98	9	Qz Eye Schist	0.19		
HC11-M01	8	15	7	Qz Eye Schist	0.20	1,286	Qz Eye Sch 2
HC11-M03	8	22	13	Qz Eye Schist	0.20		
HC11-M04	119	127	7	Qz Eye Schist	0.20		
HC11-M02	71	81	10	Qz Eye Schist	0.21		
HC11-M04	39	87	48	Qz Eye Schist	0.21		
HC11-M04	140	147	7	Qz Eye Schist	0.21		
HC11-M01	30	40	10	Qz Eye Schist	0.23		
HC11-M02	96	99	3	Qz Eye Schist	0.24		
HC11-M03	39	42	3	Qz Eye Schist	0.24		
HC11-M03	5	8	3	Qz Eye Schist	0.25	944	Qz Eye Sch 3
HC11-M04	130	135	6	Qz Eye Schist	0.25		
HC11-M04	135	140	5	Qz Eye Schist	0.25		
HC11-M02	6	9	3	Qz Eye Schist	0.26		
HC11-M02	24	27	3	Qz Eye Schist	0.26		
HC11-M02	28	43	14	Qz Eye Schist	0.26		
HC11-M03	86	92	6	Qz Eye Schist	0.28		
HC11-M02	57	64	8	Qz Eye Schist	0.29		
HC11-M04	101	116	15	Qz Eye Schist	0.29		
HC11-M03	92	96	5	Qz Eye Schist	0.30		
HC11-M02	93	96	3	Qz Eye Schist	0.35	515	Qz Eye Sch 4
HC11-M03	42	71	29	Qz Eye Schist	0.44		
HC11-M03	73	74	1	Qz Eye Schist	0.46		
HC11-M02	137	140	2	Schist	0.28	112	Schist 1
HC11-M02	99	102	2	Schist	0.29		
HC11-M02	43	46	4	Schist	0.30		
HC11-M03	84	86	2	Schist	0.37	36	Schist 2
HC11-M03	74	77	3	Schist	0.46		
HC11-M04	87	89	3	Silica Altered	0.15	163	Silica Altered 1
HC11-M03	79	81	2	Silica Altered	0.19		
HC11-M02	9	10	2	Silica Altered	0.20		
HC11-M04	116	119	3	Silica Altered	0.20		
HC11-M04	127	130	3	Silica Altered	0.20		
HC11-M02	140	150	10	Silica Altered	0.33	125	Silica Altered 2
HC11-M03	71	73	2	Silica Altered	0.33		
HC11-M02	46	51	4	Vein	0.34	235	Vein 1
HC11-M01	15	30	15	Vein	0.41		

13.3 Ore Characterization – *Cont'd*

(c) Composites Elemental Content

Standard analytical techniques were used to assay each constructed composite with results summarized in Table 13-2. The copper content of the feed for the samples tested ranged from about 0.17% to 0.43%. The Master Composite 2 sample used in majority of the flowsheet development work had a copper feed grade of 0.31%, a silver feed grade of 2 gpt, a gold feed grade of 0.1 gpt and a sulphur grade of 1.95%. Very little of the copper in the feed of Master Composite 2 was either acid or cyanide soluble. The silver content was consistent across the entire sample suite with between 1 to 2 gpt silver content. Gold feed grades through the composites ranged from 0.01 to 0.1 gpt. Sulphur feed grades, in the samples analyzed, ranged from 0.95 to 3.32%. The Schist 2 sample had the lowest sulphur feed grade at 0.95%.

Table 13-2: Composite Samples Head Assay Summary

Sample Name	Element for Assay - % or gpt						
	Cu	**Fe**	**S**	**Ag**	**Au**	**Cu (ox)**	**Cu (CN)**
Master Composite 2	0.31	3.79	1.95	2	0.10	0.002	0.006
Phyllite	0.30	5.27	1.57	4	0.04	-	-
QZ Eye Sch 1	0.20	2.67	1.22	<2	0.10	-	-
QZ Eye Sch 2	0.20	2.44	1.22	<2	0.03	-	-
QZ Eye Sch 3	0.35	3.37	2.32	2	0.06	-	-
QZ Eye Sch 4	0.38	3.40	1.37	2	0.05	-	-
Schist 1	0.33	3.47	2.27	2	0.04	-	-
Schist 2	0.25	3.93	0.95	2	0.03	-	-
Silica Altered 1	0.19	2.62	1.57	2	0.04	-	-
Silica Altered 2	0.23	3.50	2.23	<2	0.07	-	-
Vein 1	0.43	3.90	3.32	2	0.05	-	-
West LG	0.27	2.88	2.04	2	0.04	0.002	0.003
West NG	0.31	3.05	1.66	2	0.03	0.008	0.007
East LG	0.17	2.26	1.08	<1	0.02	0.001	0.002
East NG	0.32	2.60	1.75	1	0.03	0.001	0.004
South LG	0.26	3.67	1.23	1	0.01	0.001	0.003
South NG	0.36	3.46	1.46	2	0.03	0.001	0.005

Note: Ag and Au are reported in gpt, all others are reported in %.

13.3 Ore Characterization – *Cont'd*

(d) Mineralogy

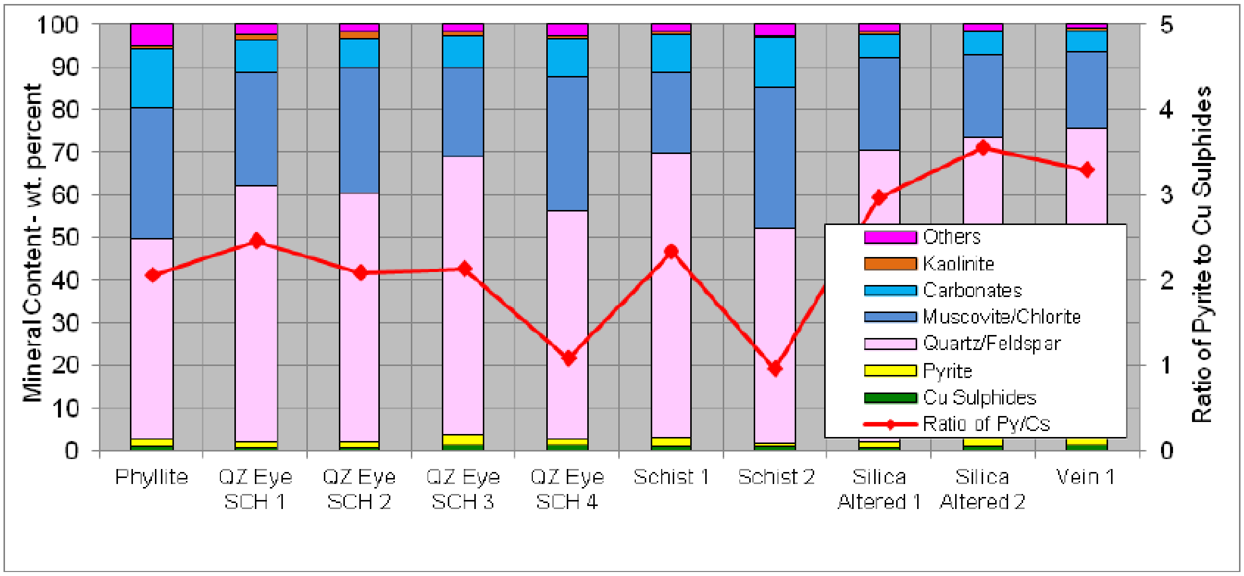
Mineralogy analysis was conducted on the master composite and the 10 lithology composite samples. Consistent with the historical mineralogy work, 97% of the copper observed in the master composite was chalcopyrite, with minor amounts of bornite and secondary copper minerals. Similarly, results from the variable lithology composite samples determined that chalcopyrite made up >98% of the copper bearing minerals, with the exception of the Silica Altered 1 lithology composite, which contained 94% chalcopyrite with 2% bornite and minor amounts of covellite and chalcocite. The mineralogical composition of each composite is shown in Figures 13-1 to 13-3.

Conclusion drawn from mineralogy results were:

- The sulphide mineral content varied from about 2 to 5% across the suite of samples tested;
- Chalcopyrite was the main copper bearing sulphide mineral observed in all the samples;
- Quartz and muscovite were the two dominant gangue mineral species;
- The pyrite to chalcopyrite ratio ranged from 1 : 1 to 3.5 : 1 with 7 out of 10 samples being below 3 : 1 ratio. A pyrite to chalcopyrite ratio less than 3 : 1 is conducive to high copper recovery by flotation;
- At a primary grind sizing of about 180 µm, the copper sulphide liberations ranged between 50 to 70%. This level of 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.

13.3 Ore Characterization – *Cont'd*

(d) Mineralogy – *Cont'd*



G&T Metallurgical Services Limited, February 2012

Figure 13-1: Variability Composites Mineral Speciation

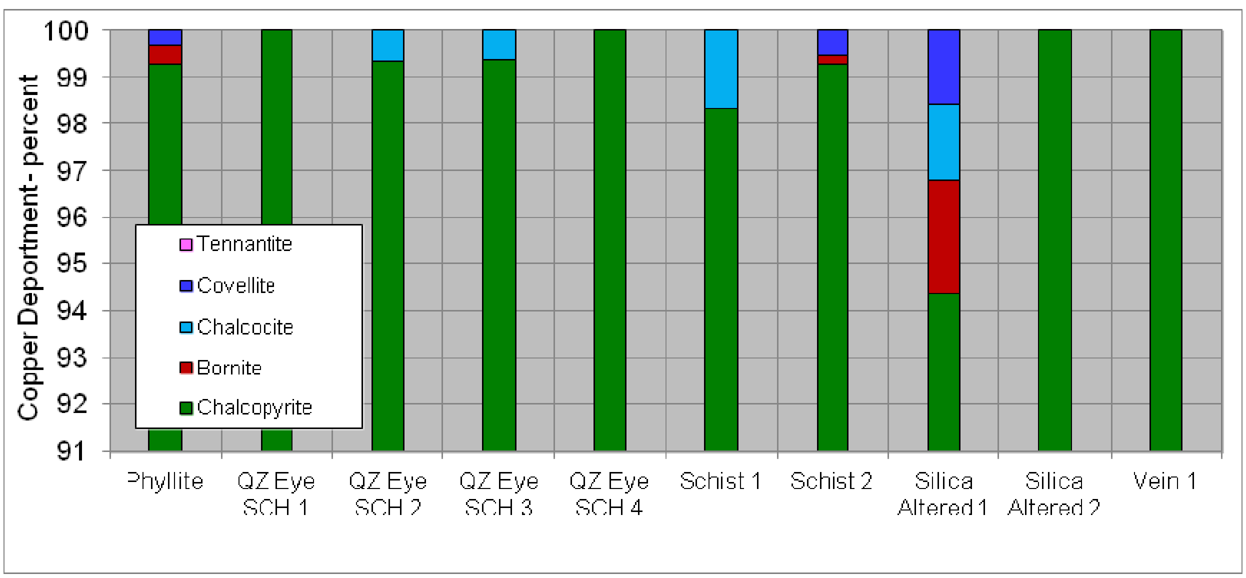


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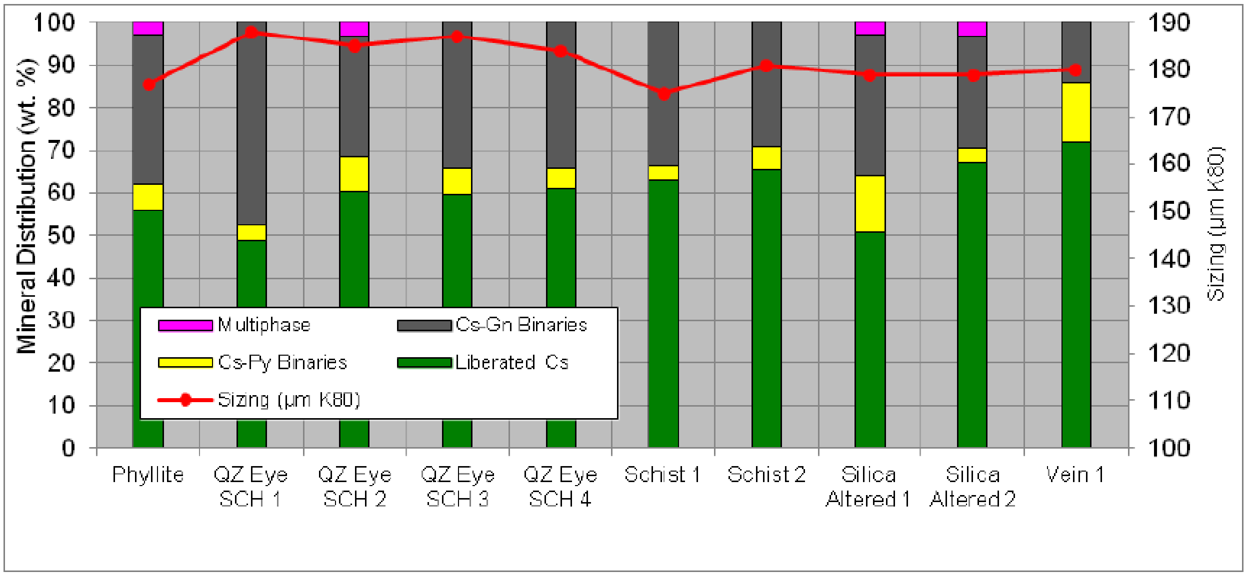
Figure 13-2: Variability Composites Copper Deportment by Mineral Species

13.3 Ore Characterization – *Cont'd*

(d) Mineralogy – *Cont'd*



G&T Metallurgical Services Limited, February 2012

Figure 13-3: Cu Sulphide Distribution by Class of Variability Composites

13.3 Ore Characterization – *Cont'd*

(e) Ore Hardness and Grindability Testing

During the Feasibility Study metallurgical testing phase, two independent ore hardness characterization test work programs were undertaken. The first program conducted at G&T on four samples designated SMC1 to SMC4. Each sample was wholly constructed from intervals from each of the four drillholes representing discreet spatial zones within the deposit. A Bond ball mill work index at a close size setting of 106 µm, Bond abrasion test, and JK SMC test was completed on each sample. The test data indicates these samples ranged from soft to moderately soft ore with respect to breakage in the ball mill. All samples tested had low abrasivity with the exception of sample SMC4. The A*b values generated in the SMC test show the samples range from soft to medium with respect to breakage in the SAG mill. The results of this testing are presented in Table 13-3.

Table 13-3: G&T Ore Hardness Testing Summary

Sample Name	Bond Ball Mill Work Index (kWh/t)	Bond Abrasion Index	A*b
SMC 1	12.2	0.084	60.5
SMC 2	12.2	0.102	42.3
SMC 3	9.5	0.042	48.9
SMC 4	10.5	0.36	51.4

The second program was executed by FLS, where 9 whole core samples with variable lithologies and identified as samples A through I were tested. Crusher work index (CWi), crusher abrasion index (Ai), unconfined compressive strength (UCS), and Bond ball mill work index (BWi) tests were performed at the FLSmidth Bethlehem Catasaqua test facility. Rod mill work index (RWi) tests were performed at Phillips Enterprises LLC in Golden, Colorado. The RWi tests were conducted by standard procedures at a closing screen of 1180 µm while the BWi test was conducted at a closed size setting of 74 um. Results are summarized in Table 13-4.

13.3 Ore Characterization – *Cont'd*

(e) Ore Hardness and Grindability Testing – *Cont'd*

Table 13-4: FLS Comminution Testing Results

Sample ID	Unconfined Compression Strength Test	Crusher Work Index	Bond Abrasion Index Test	Bond Rod Mill Work Index	Bond Ball Mill Work Index	Lithology/Hole ID
	Average	Average	Average	Average	Average	
	(PSI)	(kWh/t)	(g)	(kWh/t)	(kWh/t)	
A	2,414	5.68	0.33	10.79	19.10	Qtz Vein (M11-01)
B	NA	7.06	0.17	11.09	11.70	Phyllite – Calcareous Chlorite (M11-02)
C	4,867	8.76	0.06	11.56	10.50	Phyllite – Calcareous Chlorite (M11-03)
D	9,755	7.45	0.09	11.97	11.70	Schists (no qtz eyes) (M11-02)
E	6,955	8.07	0.43	10.95	14.00	Silica Alt. Schists (qtz eyes) (M11-02)
F	10,384	7.49	0.32	10.30	12.80	Silica Alt. Schists (qtz eyes) (M11-02)
G	8,699	6.88	0.08	11.54	10.70	Schists (qtz eyes) (M11-01)
H	NA	5.23	0.16	13.41	14.20	Schists (qtz eyes) (M11-04)
I	4,526	2.73	0.19	14.35	14.00	Schists (qtz eyes) (M11-01)
A to I	**6,800**	**6.59**	**0.20**	**11.77**	**13.19**	

The results generally conformed to historical data and the work indices were found to be generally consistent over the range of samples analyzed. Sample A which was representative of a quartz vein material was the only sample that was an outlier with a high BWi of 19.1 kWh/t. This ore type only represents <0.5% of the 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. It was elected to have a third party, KWM, review the comminution test work before finalizing any conclusions. A summary of the findings are 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.

13.3 Ore Characterization – *Cont'd*

(e) Ore Hardness and Grindability Testing – *Cont'd*

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 the independent review it was concluded that the relatively low work index strongly suggested that the ore is amenable to a conventional SAG/ball mill grinding circuit. The observed platey breakage pattern in the core boxes suggested pebble crushing was not needed. An example of "poker chip" effect on sample B is shown in Figure 13-4 below.

Figure 13-4: Platey Breakage Example on Core Sample B

13.4 Flotation Tests

During the FS test program at G&T, both open circuit and locked cycle flotation testing protocols were utilized to test a master composite and variability samples. Initial flowsheet development work was completed on a master composite. The developed flowsheet and test conditions were then utilized on the variability samples to probe metallurgical performance. Some key features from the flotation testing were:

- Potassium amyl xanthate (PAX) was used as the sulphide mineral collector;
- Methyl isobutyl carbinol (MIBC) was added to produce a stable flotation froth;
- Lime was used as a pH regulator.

(a) Master Composite Open Circuit Flotation Test

Initial rougher kinetic testing on Master Composite 2 evaluated metallurgical performance of variable grind sizes between 102 µm to 243 µm and rougher pH conditions between 8.5 and 11. The results indicated about 95% of the copper in feed was recovered in 6% of the feed mass at a primary grind size of 189 µm and pH of 11 in the rougher circuit. These conditions were considered to be the best compromise between mass pull and copper recovery to a rougher concentrate and were carried forward in the test program.

A suite of open circuit batch cleaner tests using variable regrind sizes and rougher/cleaner pH's were conducted. Regrind sizes between 16 µm to 25 µm were tested. The results from this work demonstrated that a final copper grade of 26% at a 92% total copper recovery was attainable using a regrind size of 25 µm at a pH of 11 in both the roughers and cleaner circuits. These conditions were brought forward to subsequent variability and lock cycle testing.

13.4 Flotation Tests – *Cont'd*

(b) Lock Cycle Testing

Locked cycle flotation tests were completed on Master Composite 2, zonal composites, and selected lithology composites. The results of the first locked cycle test carried out on the master composite achieved a concentrate grade of 26.3% Cu, and a recovery of 89.6% Cu, 66.8% Ag, and 57.9% Au. The test was carried out after a primary grind of 80% passing 189 µm, at a pH of 11 and using PAX as the collector. Rougher/scavenger concentrate was reground to 27 µm and cleaned at a pH of 11 using PAX as collector and MIBC as a frother. No other depressants (except lime/pH) were used. The test was repeated at a slightly finer regrind and gave similar results producing a final concentrate grade of 25.6% Cu at 90.0% Cu recovery. The results from these duplicate lock cycle test formed the basis of metallurgical performance for the process design.

13.4 Flotation Tests – *Cont'd*

(c) Variable Lithology and Zonal Composites

Following the establishment of this flowsheet and reagent schedule, the flowsheet was tested on the zonal composites and the ten grade-lithology composites used in compiling the master composite. Results generally conformed to the master composite, with the exception of two lithology samples, one of which was below the project cut off grade of 0.17% Cu and the other which consisted of a silica alteration of quartz eye schist in which copper minerals other than chalcopyrite were present in any appreciable amounts and which is atypical of the deposit.

It was observed that some of these minerals like chalcocite and covellite 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.

13.4 Flotation Tests – *Cont'd*

(d) Pilot Plant Testing

To generate sufficient concentrate for smelter acceptability tests, a 10-hour pilot plant campaign was executed at G&T using 871 kilograms of Master Composite 2 sample used as circuit feed. A feed rate of between 82 and 106 kilograms per hour was employed for the duration of the test campaign. The same developed flowsheet used in the bench scale program was employed in the pilot plant, involving a primary grind of a 180 μm followed by rougher flotation, a regrind stage, and 3 stages of dilution cleaning. The primary grinding was performed in an open circuit rod mill, operating at 55% solids by weight. Grind size was measured with grab samples and adjustments to mill rotational speed and rod charge were made to reach desired target grind size. Regrinding was completed using a 2 liter stirred mill with a regrind target sizing of P80 of 30 μm. The reagent scheme was similar to that used in the bench scale lock cycle testing; however the PAX additions were markedly lower in the pilot plant. A target pH of 11 was employed in both the roughers and the cleaner circuit, with main additions being to the rod mill and regrind mills, with supplemental pH control by two more unmetered lime additions in the rougher and first cleaner when required.

Due to the short duration of the pilot plant, some challenges with achieving good circuit stability were encountered, resulting in production of lower than target final concentrate grades at times, especially during start-up and shutdown. When circuit stability was achieved final concentrate grade and recoveries were very similar to results obtained from the lock cycle. Had the overall run time of the pilot plant been longer, it is likely this result could have been achieved more consistently. When emptying the circuit during pilot plant shutdown, a substantial mass of shutdown concentrate was also generated. The grades were lower than what would be expected for stable operation, so batch kinetic rougher tests were completed on the concentrates to upgrade them to about 26% Cu. After upgrading, just less than 8 kilograms estimated dry weight of concentrate was produced grading at about 26% Cu.

13.4 Flotation Tests – *Cont'd*

(e) Concentrate Quality

Minor element determinations were completed on samples of the final concentrate produced in locked cycle test 13 and on produced pilot plant concentrate. The minor elements were analyzed using standard analytical techniques with the results summarized in Table 13-5. It was found that the minor element data was quite similar between the concentrate from locked cycle test 13 and the pilot plant concentrate. Both concentrates were clean with deleterious minor element concentrations below typical penalty limits for the elements analyzed. Precious metal values were between 1.5 to 2.0 gpt for gold and about 122 gpt for silver.

13.4 Flotation Tests – *Cont'd*

(e) Concentrate Quality – *Cont'd*

Table 13-5: Final Concentrate Minor Elemental Composition Summary

Element	Symbol	Units	Locked Cycle Test 12	Locked Cycle Test 13	Final Conc.
Copper	Cu	%	25.6	26.3	25.5
Gold	Au	gpt	1.35	1.55	1.92
Silver	Ag	gpt	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	gpt		87	104
Bismuth	Bi	gpt		31	13
Cadmium	Cd	gpt		32	34
Calcium	Ca	%		0.36	0.67
Carbon	C	%		0.34	0.83
Cobalt	Co	gpt		0.40	110
Fluorine	F	gpt		101	151
Lead	Pb	%		0.21	0.17
Magnesium Oxide	MgO	%		0.40	1.36
Manganese Oxide	MnO	%		0.013	0.031
Mercury	Hg	gpt		<1	<1
Molybdenum	Mo	%		0.010	0.020
Nickel	Ni	gpt		206	350
Phosphorous Pentoxide	P2O5	gpt		76	418
Selenium	Se	gpt		127	2
Silicon Dioxide	SiO2	%		2.67	7.12
Zinc	Zn	%		0.49	0.35

13.4 Flotation Tests – *Cont'd*

(f) Process Recovery Projections

A metallurgical recovery model based on process tailings grade was developed by Laurion Consulting Inc in the previously published 2014 technical report. From the test work results, equations were developed to estimate the metallurgical recovery of copper, gold and silver as a function of head grade and are depicted in Figures 13-5 to 13-7. These recovery models were incorporated into the resource and reserve estimates discussed in subsequent sections.

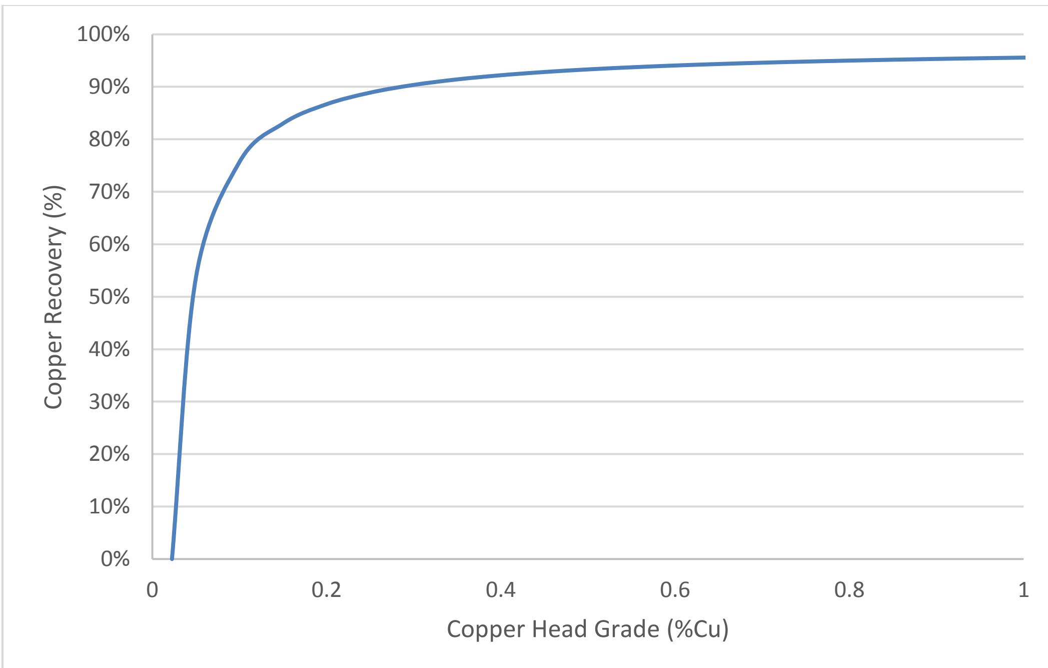


Figure 13-5: Copper Recovery vs. Copper Head Grade

13.4 Flotation Tests – *Cont'd*

(f) Process Recovery Projections – *Cont'd*

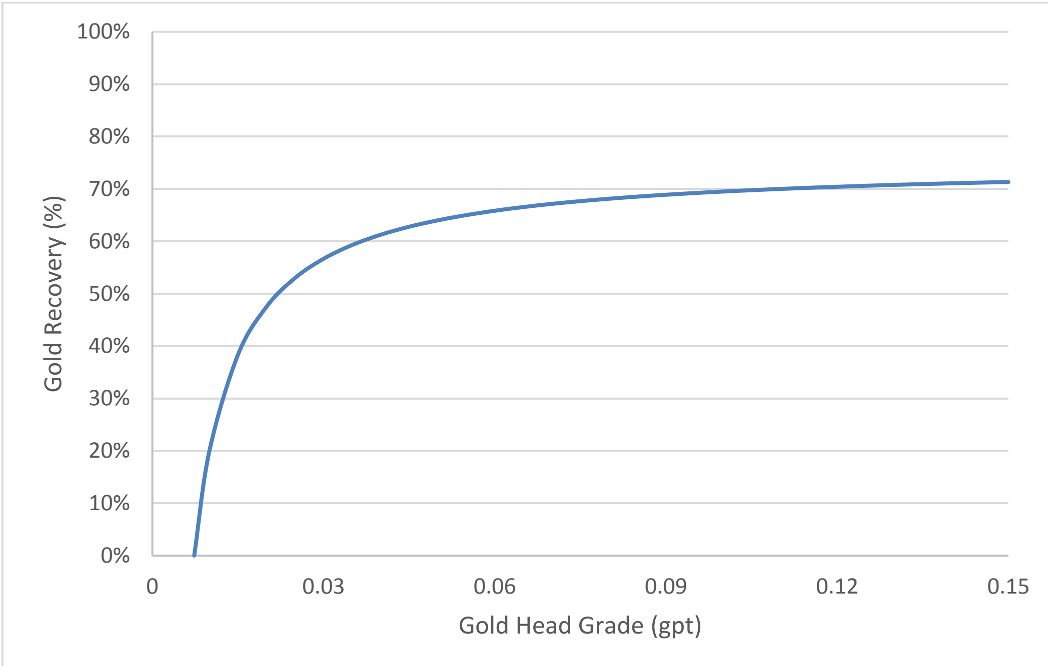


Figure 13-6: Gold Recovery vs. Gold Head Grade

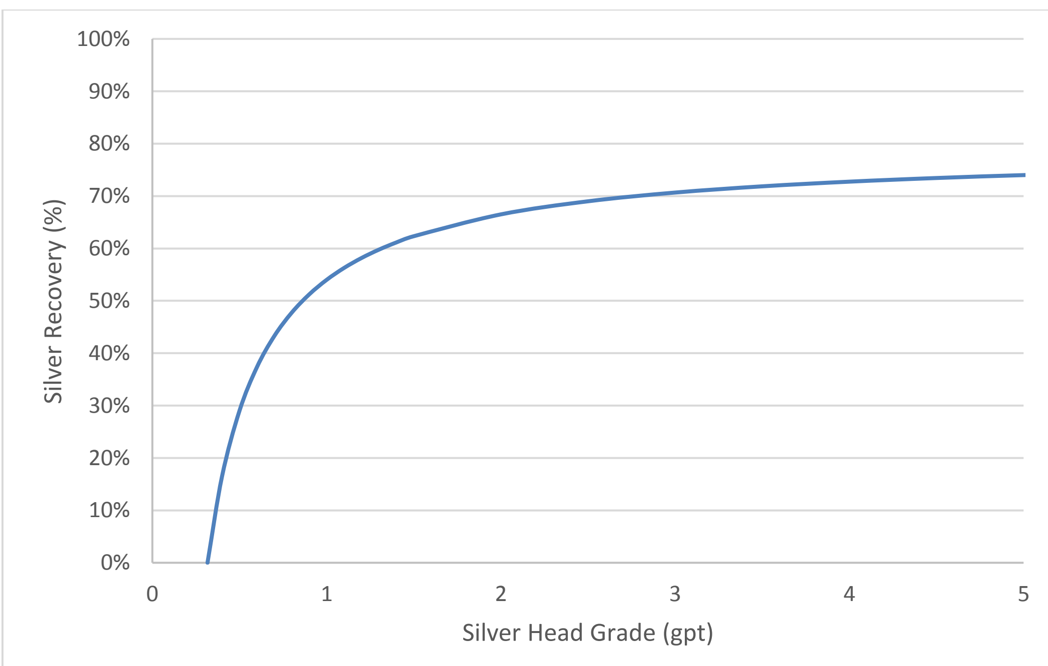


Figure 13-7: Silver Recovery vs. Silver Head Grade

13.5 Conclusion

The proposed process for the Yellowhead project consists of a conventional milling circuit to recover copper via grinding, rougher flotation, regrinding of rougher concentrate, followed by a cleaner flotation circuit. All comminution testing conducted to date suggest the ore is soft to moderately soft and very amenable to both SAG milling and ball milling.

Mineralogy characterization on ore samples from the deposit demonstrate that chalcopyrite is the dominant copper bearing mineral making up >98% of the copper species in the majority of the deposit, with the exception of the Silica Altered lithology composite which was found to contain 94% chalcopyrite with 2% bornite and minor amounts of secondary sulphides.

Lock cycle testing conducted on the master composite sample produced a final copper concentrate grade of 26% copper at about a 90% total copper recovery. The final concentrate produced from the lock cycle testing and pilot plant produced a clean concentrate with minor deleterious elements below typical penalty limits at smelters, and also containing payable gold and silver credits.

Future metallurgical test programs undertaken for the project should consider testing more copper sulphide selective collectors, reduction of pH conditions in the roughers, and further coarsening of the primary grind with minimal impact on metallurgical performance of the process. This could reduce overall process operating costs and further improve process economics for the project.

SECTION 14

MINERAL RESOURCE ESTIMATE

SECTION 14: MINERAL RESOURCE ESTIMATE

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14.1 Exploratory Data Analysis

The last exploration work on the Yellowhead project resource was documented in the technical report titled "Technical Report & Feasibility Study of the Harper Creek Copper Project", dated July 31, 2014, filed on www.sedar.com under YMI's profile. There have been no additional relevant exploration results within the resource area nor changes to the resource block model since that time.

The sample database for the project contains results from 353 core holes (90,779 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 (1,545 m) were also drilled in 2011 but were outside of the resource area. A total of 24 geotechnical holes (1,270 m) were also completed in 2011. The drilling used to develop the resource model is summarized in Table 14-1.

Table 14-1: Resource Drillhole 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	174	526
NH-1 to 17	1968	Noranda	17	BQ	2,106	709	1,988
69-H-1 to 27	1969	Quebec Cartier	27	BQ	4,739	1,528	4,579
NH-18 to 30	1969	Noranda	13	BQ	1,734	532	1,615
J-1 to 12	1970	Noranda	12	BQ	2,329	617	1,894
NH-31 to 95	1970	Noranda	57	BQ	8,316	2,503	7,654
J-13 to 43	1971	Noranda	27	BQ	5,594	1,728	5,354
J-40 to 42	1972	Noranda	4	BQ	457	39	118
J-44 to 48	1973	Noranda	5	BQ	625	13	40
96-1 to 8	1996	American Comstock	8	NQ	2,847	686	2,046
		Subtotal 1967-1996	**176**		**29,292**	**8,529**	**25,813**
HC06-01 to 12	2006	YMI	12	NQ2	4,101	2,536	4,029
HC07-13 to 52	2007	YMI	40	NQ2	15,880	12,569	15,602
HC08-53 to75	2008	YMI	23	NQ2	7,603	6,991	7,496
HC10-76 to 82	2010	YMI	7	NQ2	3,486	2,637	3,406
HC11-83 to 130	2011	YMI	48	NQ2	15,571	11,865	14,930
HC11-GM01 to GM07	2011	YMI	8	PQ	2,433	1,025	1,291
HC11-M01 to M04	2011	YMI	4	PQ	441	137	143
		Subtotal 2006-2011	**142**		**49,516**	**37,760**	**46,897**
HC12-131 to 172	2012	YMI	12		3,803	2,547	3,466
HC13-143 to 165	2013	YMI	23		8,166	5,206	7,259
		Subtotal 2012-2013	**35**		**11,969**	**7,753**	**10,726**
Total			**353**		**90,778**	**54,042**	**83,436**

14.1 Exploratory Data Analysis – *Cont'd*

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

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

Copper shows a moderate positive correlation with gold and a weaker positive correlation with silver with correlation coefficients of 0.23 and 0.13 respectively (Figure 14-4).

Gold and silver 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

14.1 Exploratory Data Analysis – *Cont'd*

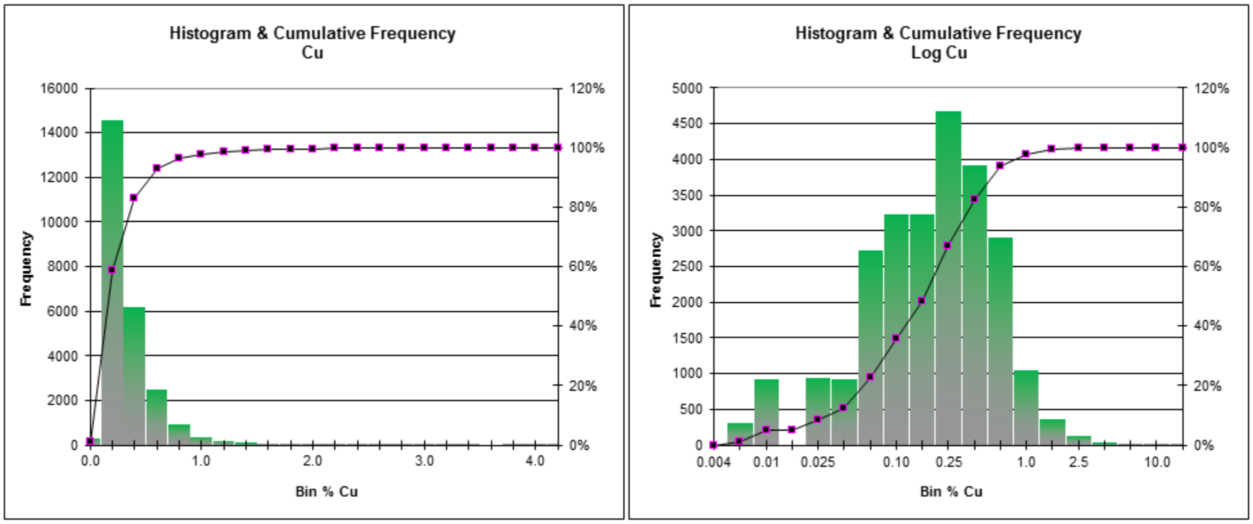

Figure 14-1: Frequency Distribution of Copper

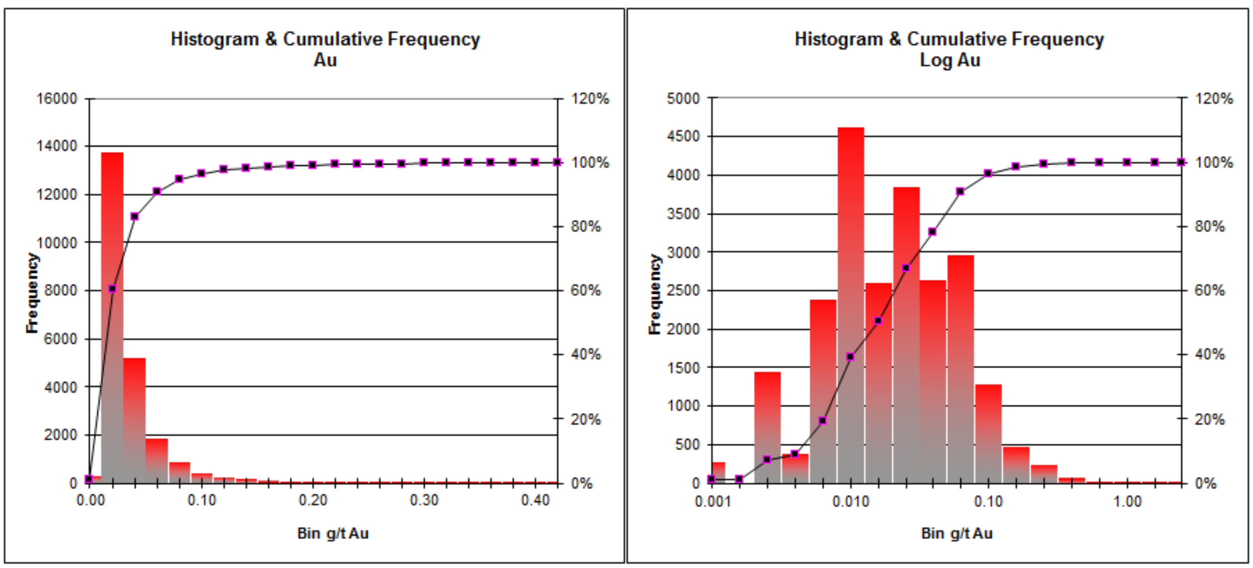

Figure 14-2: Frequency Distribution of Gold

14.1 Exploratory Data Analysis – *Cont'd*

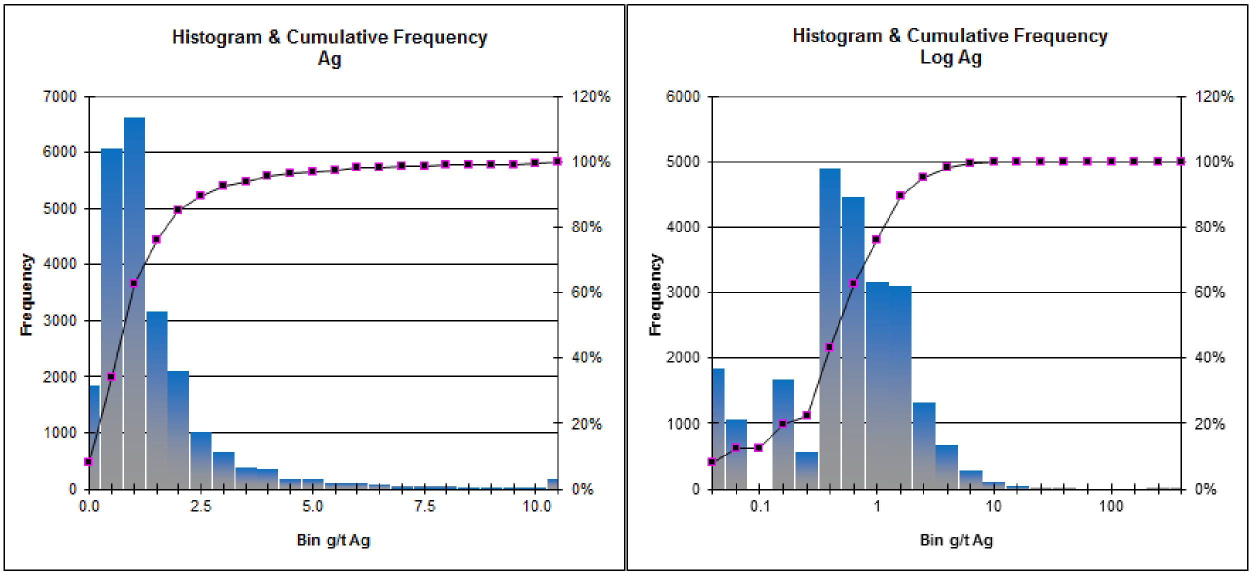

Figure 14-3: Frequency Distribution of Silver

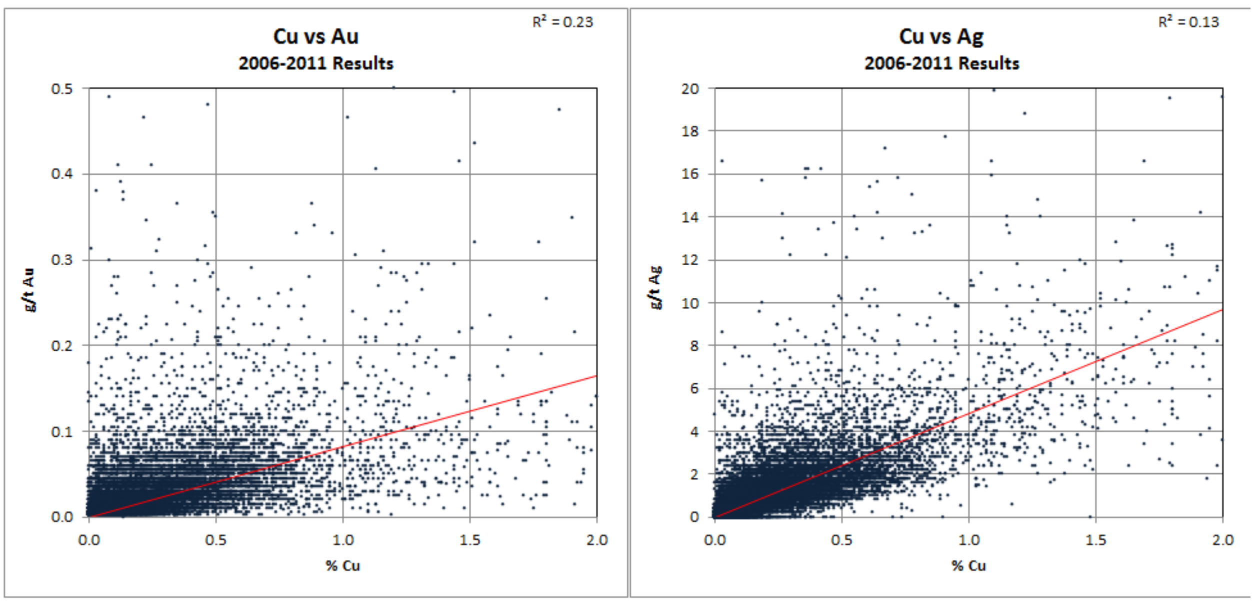

Figure 14-4: Scatterplot of Copper vs Gold and Silver Sample Data

14.1 Exploratory Data Analysis – *Cont'd*

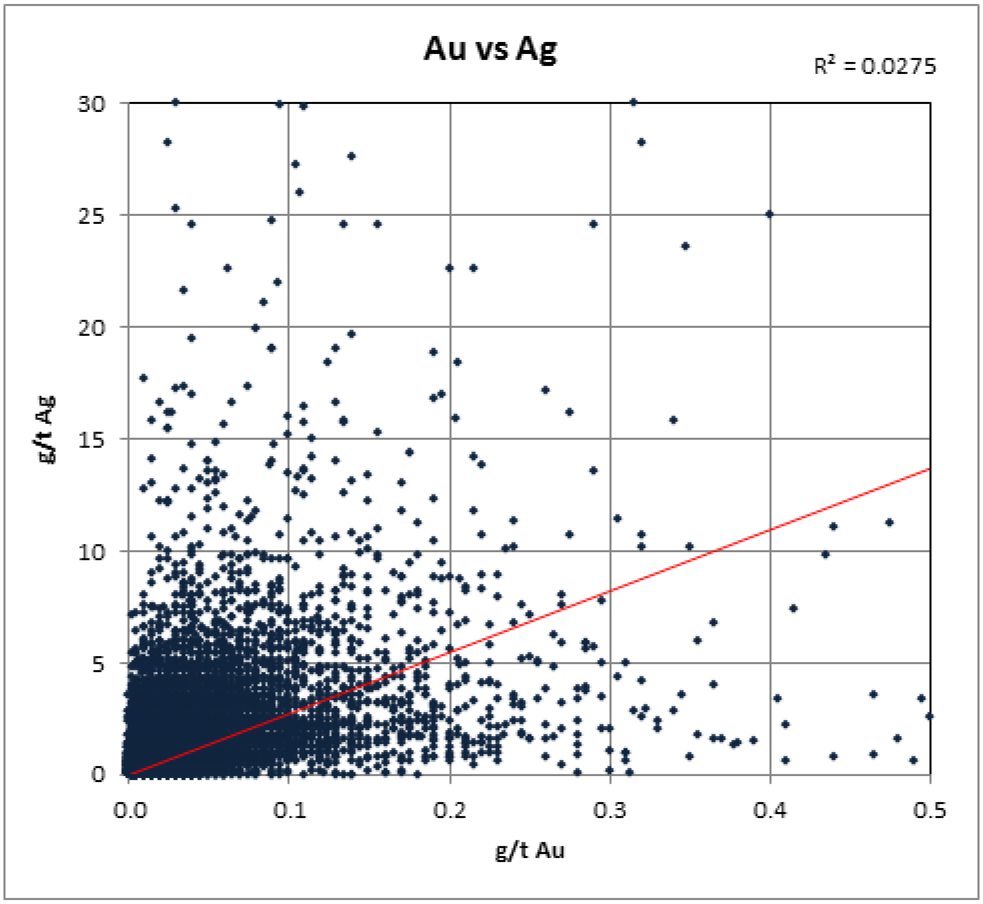

Figure 14-5: Scatterplot of Gold vs Silver Sample Data

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 (CPP) were examined for outlier populations and decile analyses were performed for copper, gold and silver 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	gpt	4
Ag	30	gpt	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 southwest-northeast direction and dips steeply to the southeast. 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.

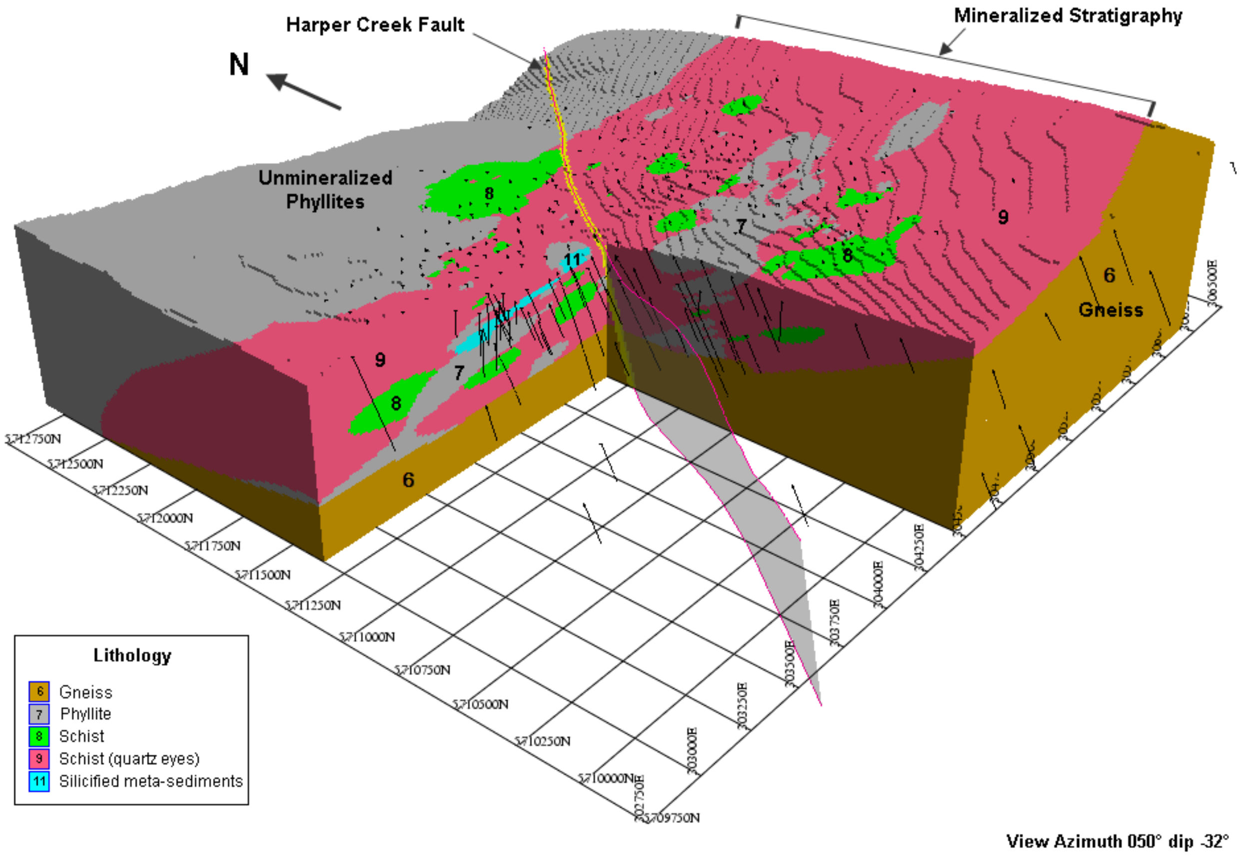

Figure 14-6: Block Model Lithology

14.3 Deposit Modeling – *Cont'd*

In order to further constrain the block model grade estimation, gradeshells based on a 700ppm copper 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 northwest and southeast zones.

A bedrock surface digital elevation model was constructed in Surpac based on drillhole data and projected to the edges of the resource model.

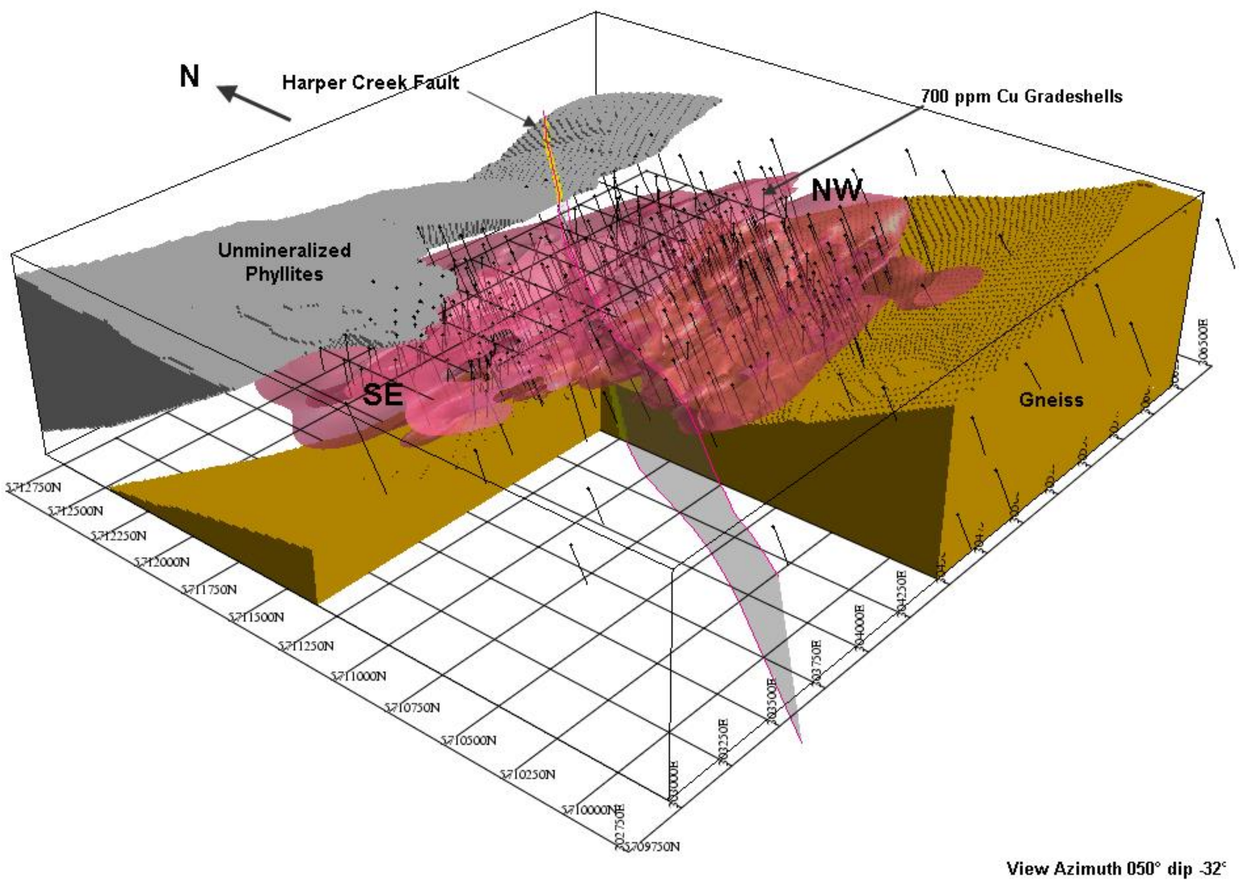

Figure 14-7: Gradeshell Constraints

14.4 Compositing

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

Table 14-4: Composite Statistics (Geosim)

	Copper in 700ppm Cu grade shells			**Gold in 700ppm Cu grade shells**			**Silver 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	12.0
Median	0.17	0.19	0.18	0.035	0.019	0.019	1.5	1.0	0.9
Mean	0.30	0.30	0.30	0.010	0.032	0.033	0.5	1.4	1.5
Wt avg	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 as shown in Table 14-5. 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 copper, gold and silver were modeled using composites falling within the domain constraint in order to determine search parameters and anisotropy. Maximum ranges for copper in both zones were 250 m while gold and silver had modeled ranges of 250 m in the southeast zone and 200 m in the northwest zone. Variogram model parameters for copper, gold and silver 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 12 m x12 m x12 m. 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.

Copper, gold and silver grades within the northwest and southeast 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.

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

14.7 Block Model and Grade Estimation Procedures – *Cont'd*

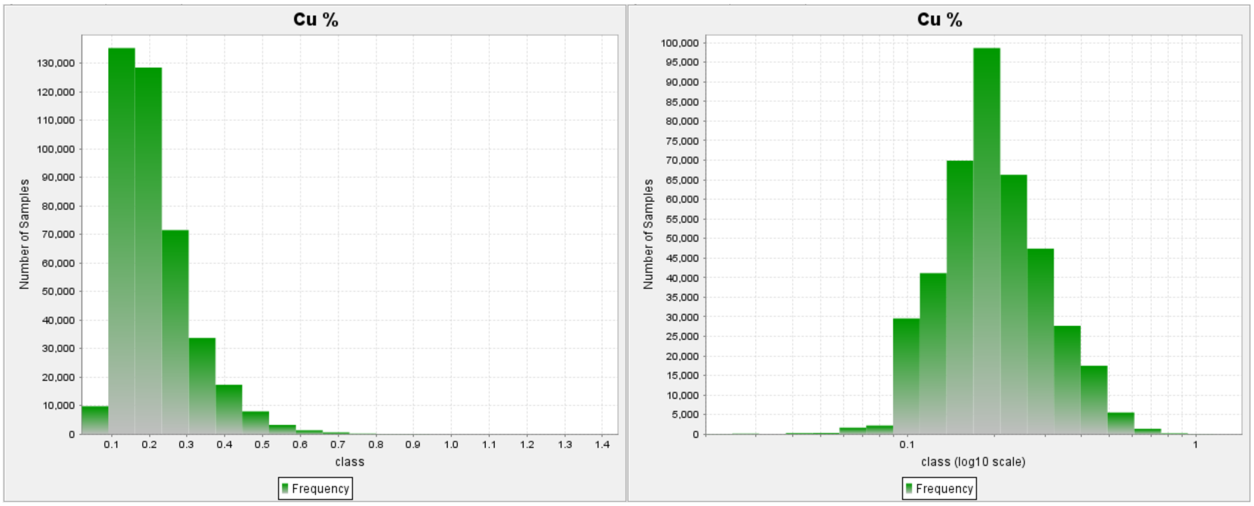

Figure 14-8: Frequency Distribution of Copper Grades in Block Model

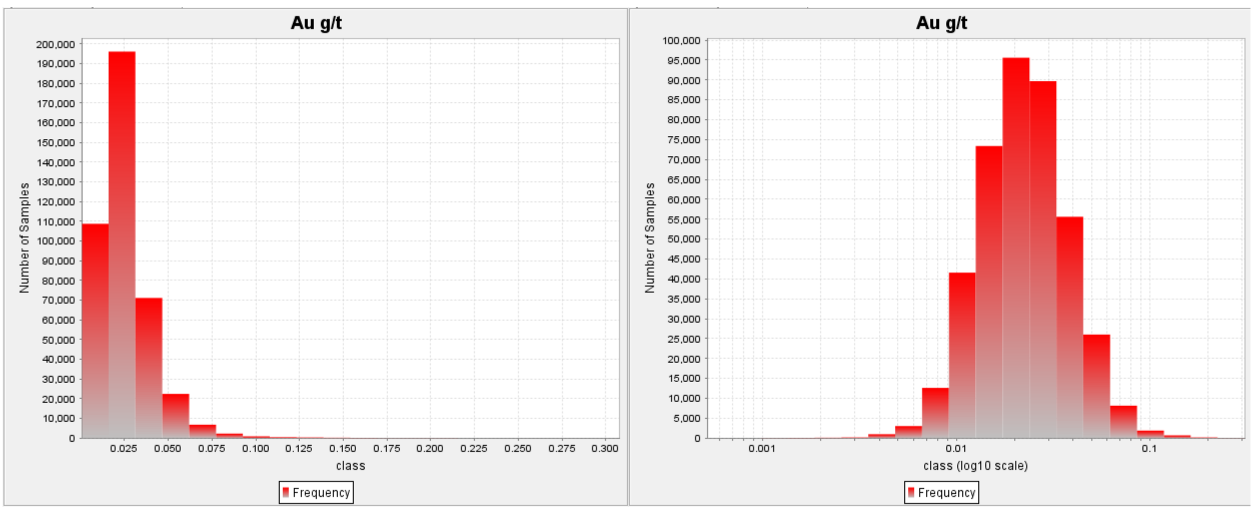

Figure 14-9: Frequency Distribution of Gold Grades in Block Model

14.7 Block Model and Grade Estimation Procedures – *Cont'd*

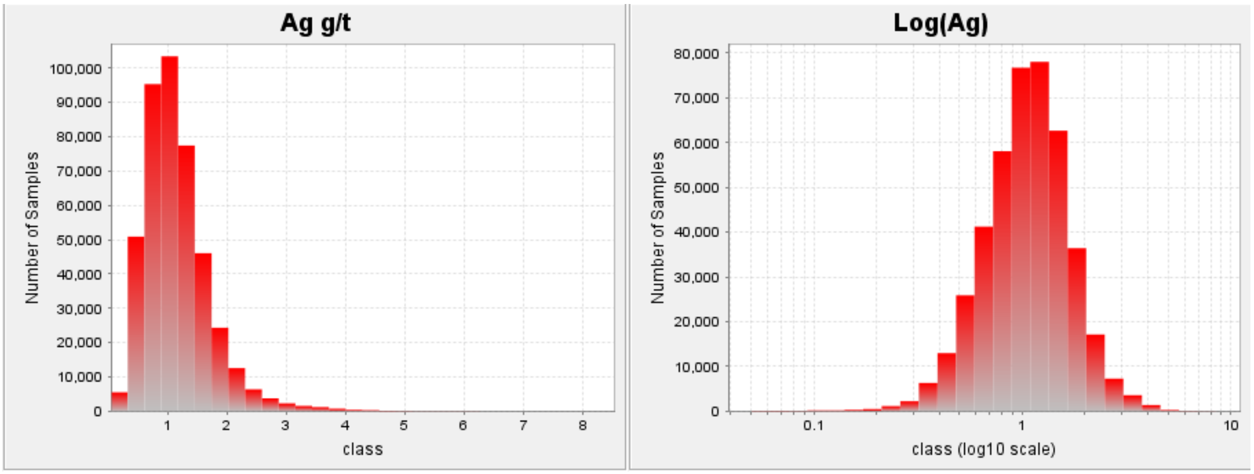

Figure 14-10: Frequency Distribution of Silver Grades in Block Model

14.8 Mineral Resource Classification

Resource classifications used in this study conform to the 2014 CIM Definition Standards:

Mineral Resource

*A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.*

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Measured Mineral Resource

*A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.*

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Indicated Mineral Resource

*An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.*

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

14.8 Mineral Resource Classification – *Cont'd*

Inferred Mineral Resource

An ***Inferred Mineral Resource*** *is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.*

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Resource Classification

Blocks were initially classified as measured if they were estimated in the 1st pass with a minimum of 4 composites from at least 2 drillholes within 82.5 m 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 they 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 125 m 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 14-11.

14.8 Mineral Resource Classification – *Cont'd*

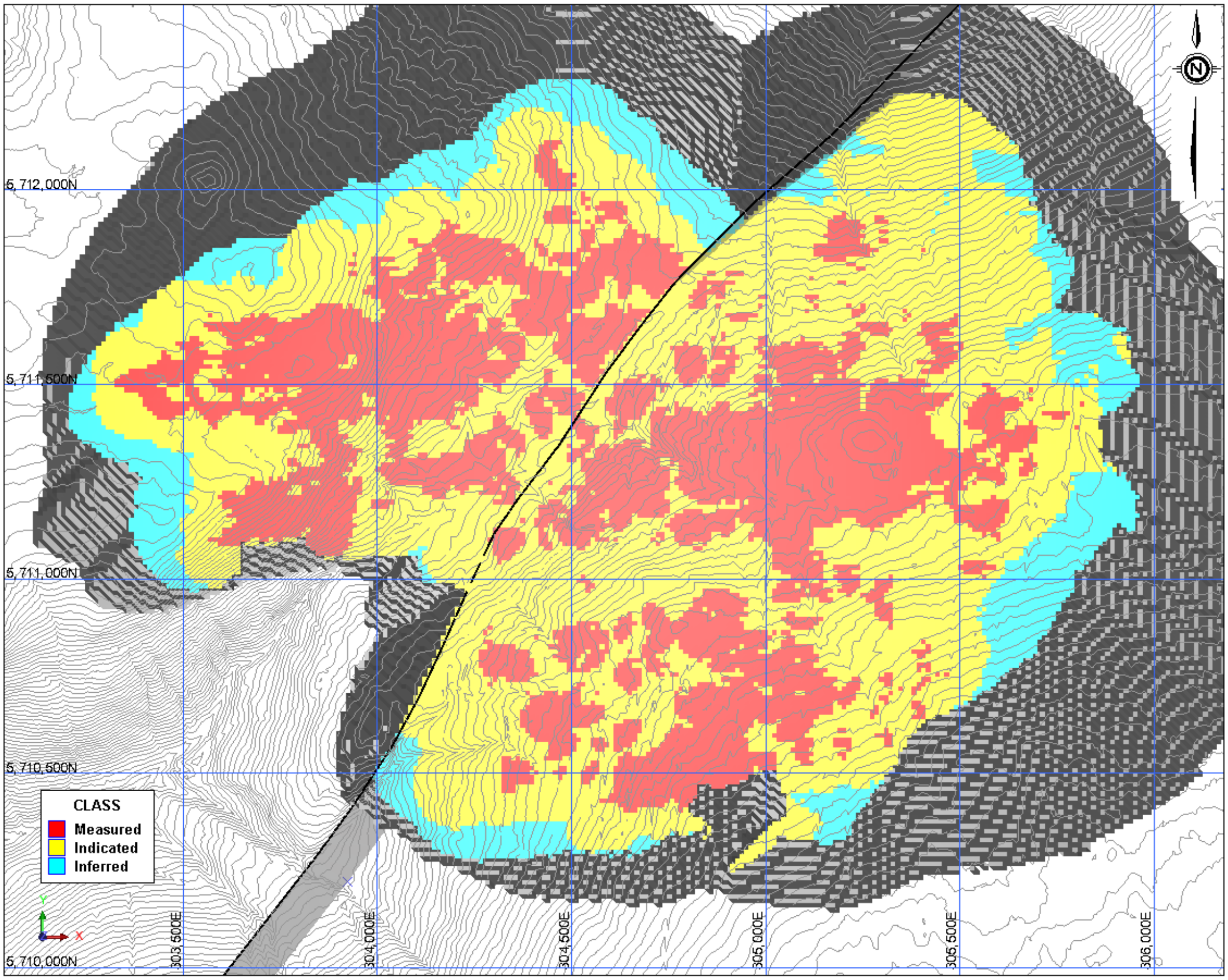

Figure 14-11: Block Classification – Plan View

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 copper. 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 (gpt)**	**Ag (gpt)**
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 beighbour	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 14-12 to Figure 14-14).

14.9 Model Validation – *Cont'd*

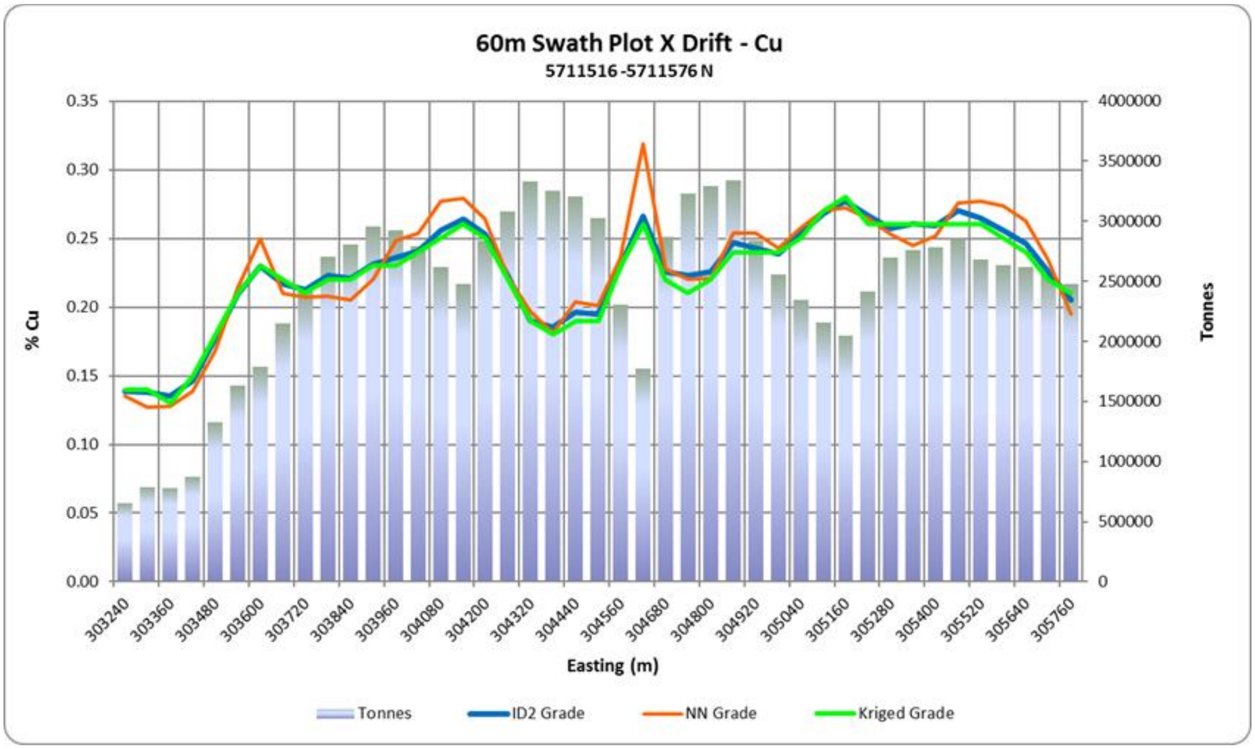

Figure 14-12: Copper Swath Plot (E-W) at 5711516-5711576 North

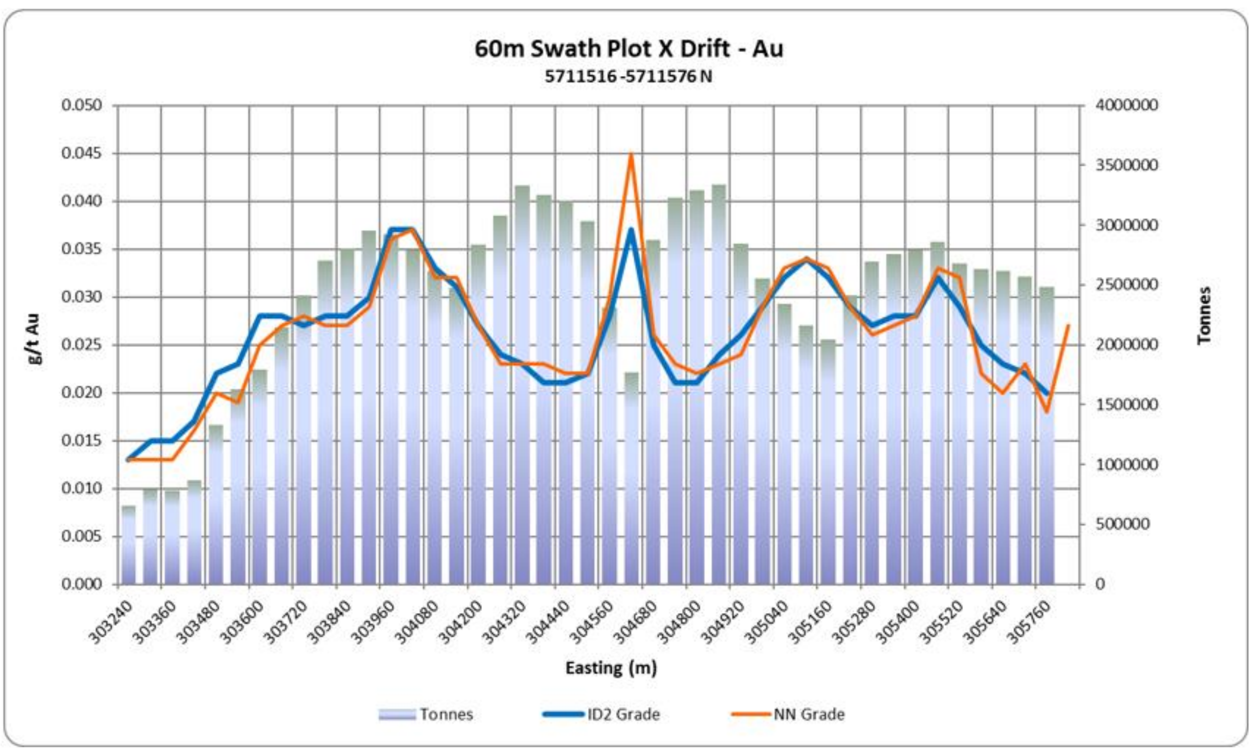

Figure 14-13: Gold Swath Plot (E-W) at 5711516-5711576 North

14.9 Model Validation – *Cont'd*

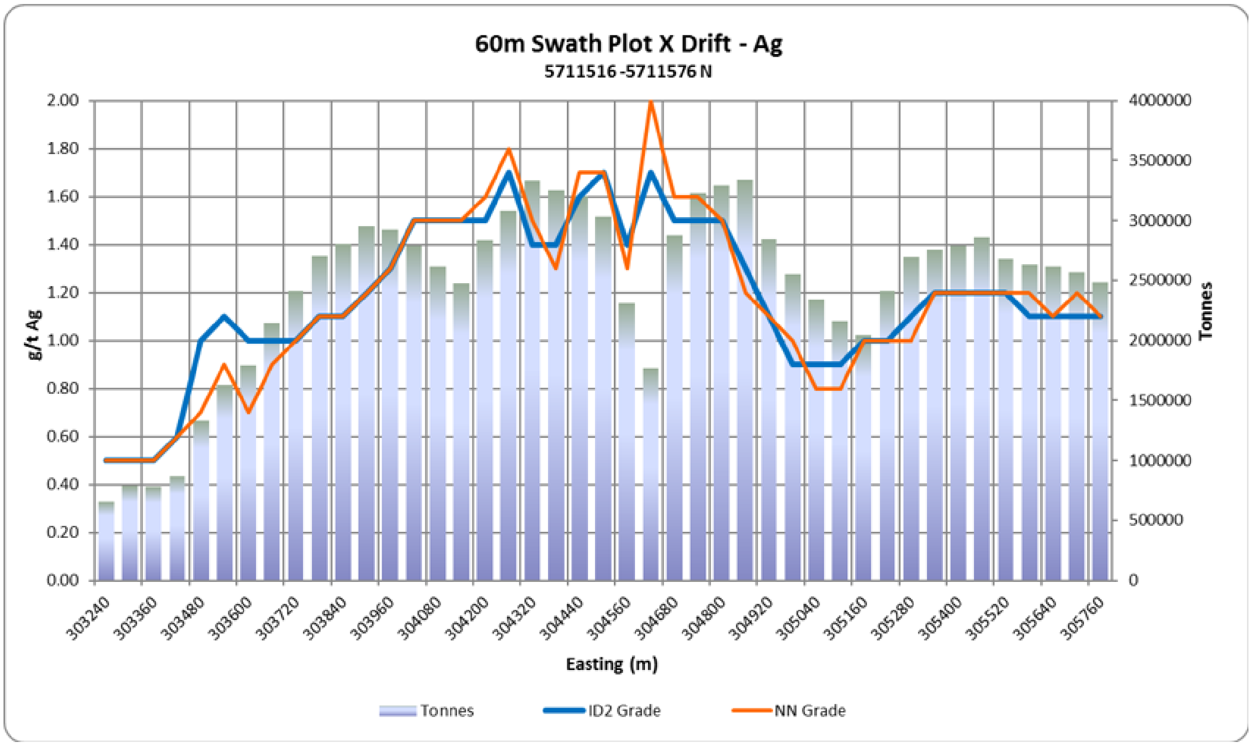

Figure 14-14: Silver Swath Plot (E-W) at 5711516-5711576 North

14.10 Mineral Resource Estimate

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 optimization was generated to constrain the resource within the block model. Metal prices used were US$3.25/lb Cu, US$1,300/oz Au and US$17.00/oz Ag at a foreign exchange rate of US$0.80 : C$1.00. Metal recoveries are based on recovery models discussed in section 13 of this report applied to block grades with average recoveries of 89% for copper, 55% for gold and 59% for silver at a 0.15% copper cut-off grade. Combined processing and G&A costs were set at C$5.25/t milled. Pit-rim mining cost for ore and waste were C$1.86/t mined with a bench increment of C$0.029/t mined. Pit slopes were set based on wall azimuth as outlined in Table 14-10. No allowances were made for mining losses and dilution.

Table 14-10: Open Pit Slopes by Azimuth

Wall Azimuth	Pit Sector	Wall Slope
0°-115°	North, West	40°
115°-230°	Southwest, South	30°
230°-360°	West, Northwest	40°

Profiles of the pit with estimated copper grade distributions are included in Figures 14-15 to 14-20.

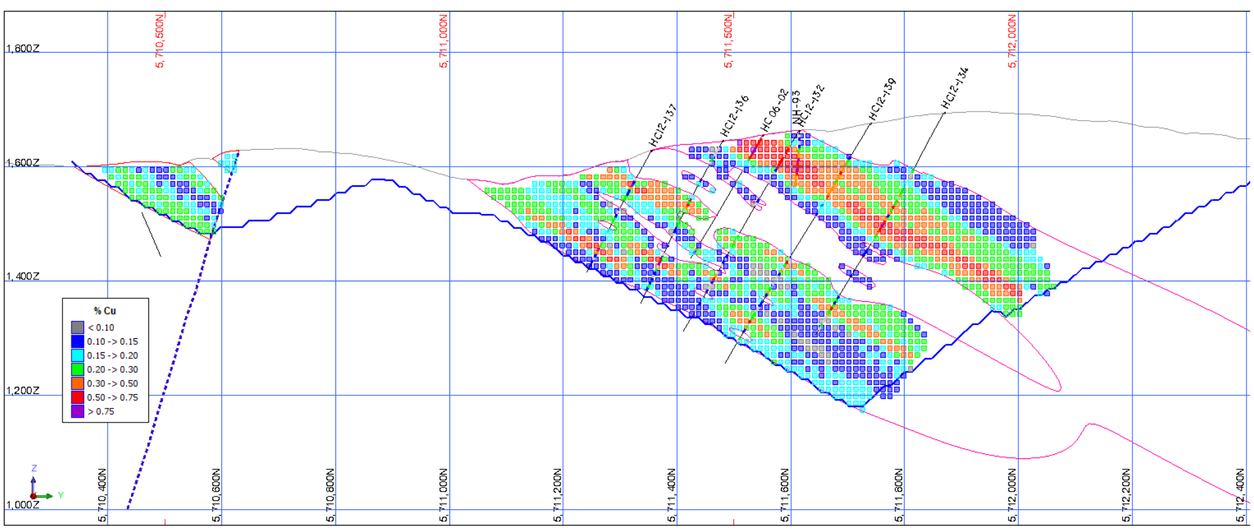

Figure 14-15: Block Grade Distribution Section 304060E

14.10 Mineral Resource Estimate – *Cont'd*

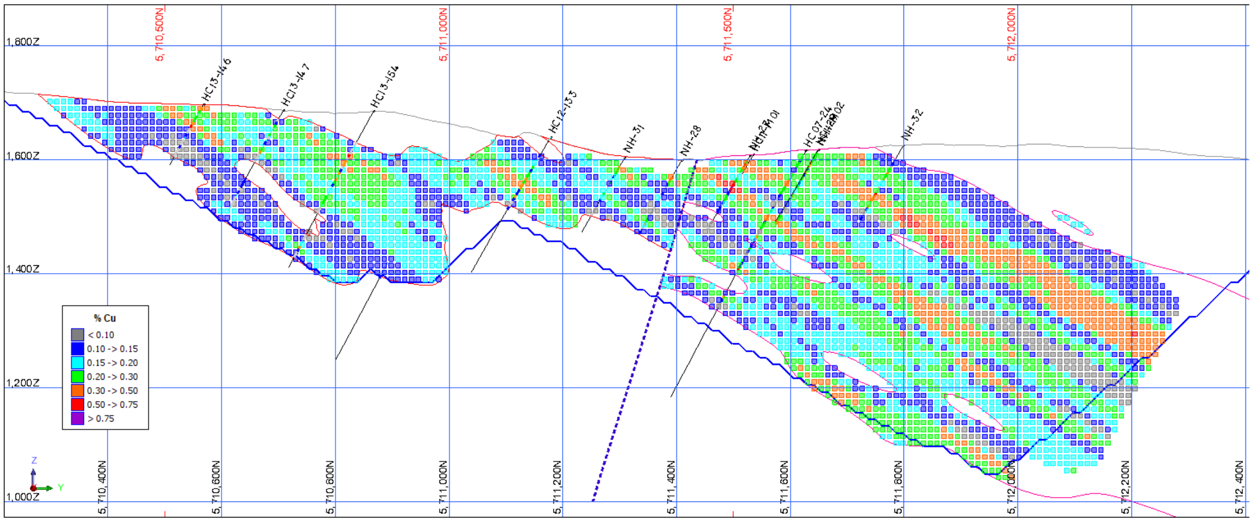

Figure 14-16: Block Grade Distribution Section 304518E

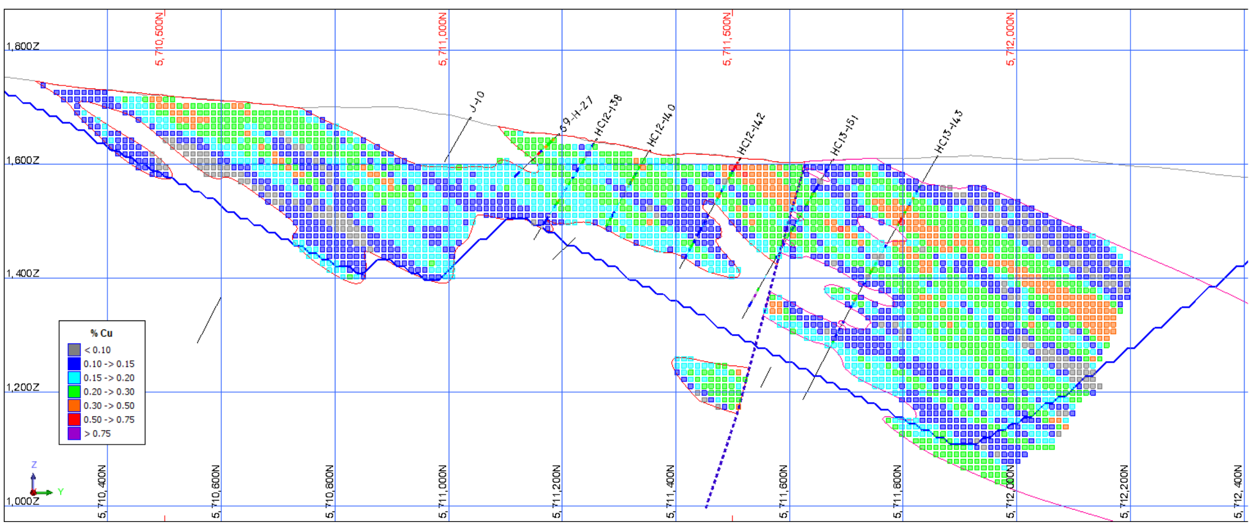

Figure 14- 17: Block Grade Distribution Section 304650E

14.10 Mineral Resource Estimate – *Cont'd*

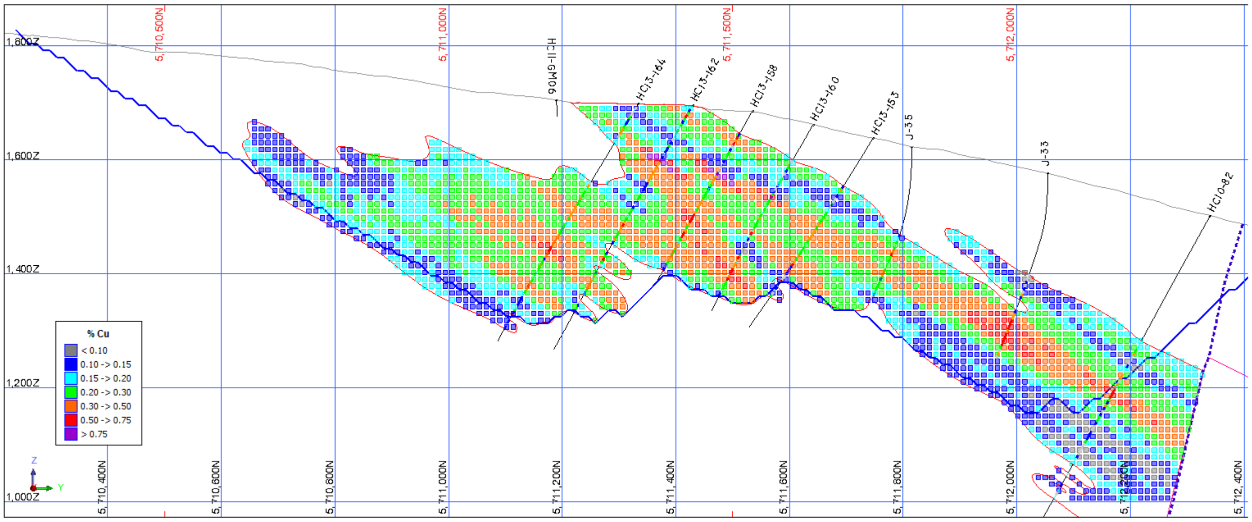

Figure 14-18 : Block Grade Distribution Section 305418E

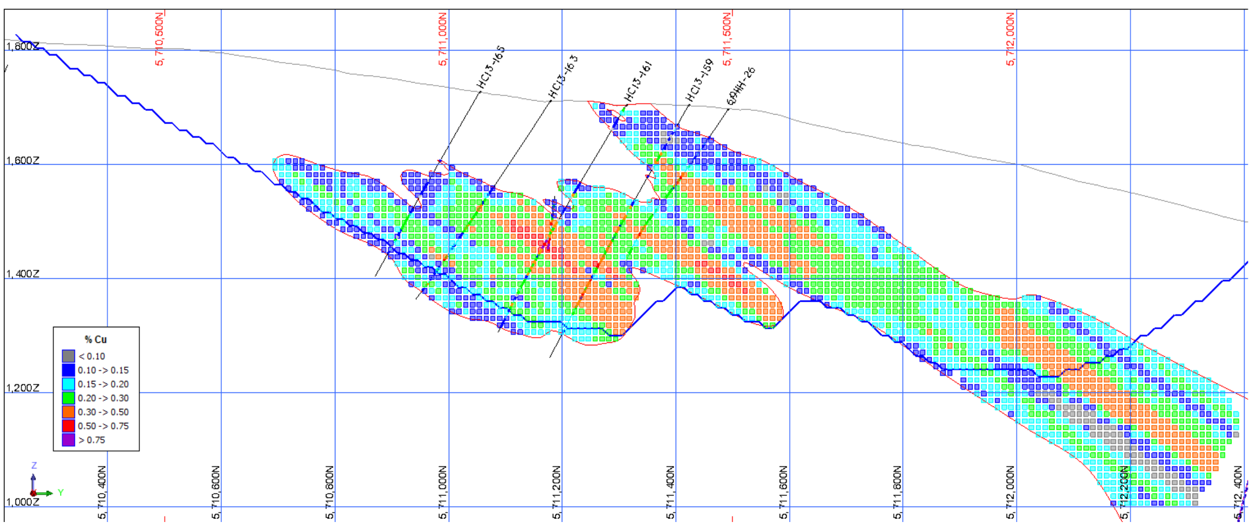

Figure 14-19: Block Grade Distribution Section 305538E

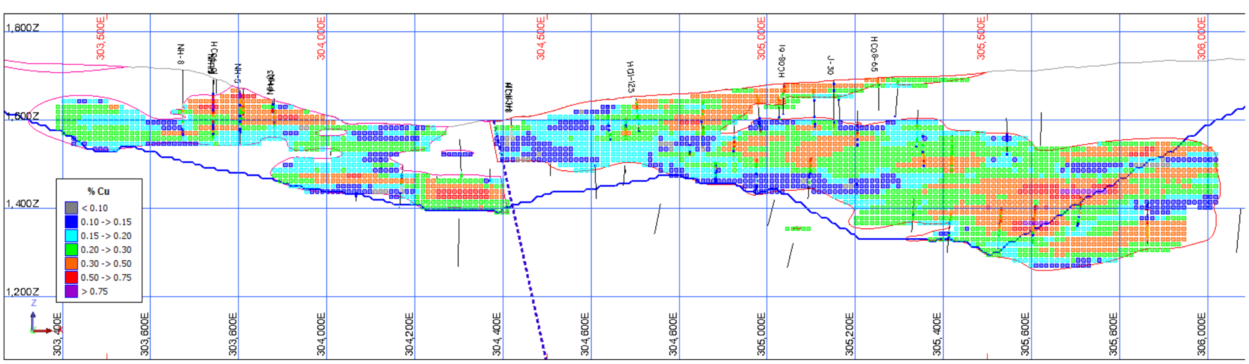

Figure 14-20: Block Grade Distribution Section 5711228N

14.10 Mineral Resource Estimate – *Cont'd*

Table 14-11 presents 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 0.15% Cu is reasonable compared to other large-scale open pit copper mines in British Columbia.

Table 14-11: Mineral Resource Estimate December 31, 2019 (Geosim)

Cut-off Grade (% Cu)	Tonnes (millions)	Cu Grade (%)	Au Grade (gpt)	Ag Grade (gpt)
Measured				
0.30	173	0.39	0.043	1.6
0.25	269	0.35	0.037	1.4
0.20	403	0.31	0.033	1.3
0.15	**561**	**0.27**	**0.029**	**1.2**
0.10	705	0.24	0.027	1.2
Indicated				
0.30	140	0.37	0.041	1.5
0.25	261	0.33	0.036	1.4
0.20	445	0.28	0.031	1.3
0.15	**730**	**0.24**	**0.027**	**1.2**
0.10	933	0.21	0.025	1.2
Measured + Indicated				
0.30	313	0.38	0.042	1.5
0.25	530	0.34	0.037	1.4
0.20	847	0.29	0.032	1.3
0.15	**1292**	**0.25**	**0.028**	**1.2**
0.10	1639	0.22	0.026	1.2
Inferred				
0.30	23	0.38	0.035	1.4
0.25	39	0.33	0.033	1.3
0.20	68	0.28	0.030	1.3
0.15	**109**	**0.24**	**0.026**	**1.2**
0.10	157	0.21	0.024	1.1

Notes:

1. Mineral resource estimate prepared by Mr. R. Simpson, P.Geo., of GeoSim Services Inc. with an effective date of 31 December 2019. Mineral Resources are classified using the 2014 CIM Definition Standards.
2. An optimized pit shell was generated using the assumptions stated in section 14.10.
3. Totals may not sum due to rounding.
4. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

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;
- Foreign exchange assumptions;
- Assumptions that all required permits will be forthcoming;
- 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, socioeconomic, 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 in the estimation of mineral reserves and mineral resources and corresponding grades being mined or assigned to future production. The estimation 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, as well as 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 changes in quantity of mineral reserves, mineral resources, grade, or strip 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.

SECTION 15

MINERAL RESERVE ESTIMATE

SECTION 15: MINERAL RESERVE ESTIMATE

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15.1 Assumptions, Parameters and Methods

(a) Pit Size Determination

The extent of potential reserve is initially determined by application of the Lerchs-Grossmann “zero profit” technique. This methodology derives a series of nested pit shells, based on a series of consistent cost and recovery calculations over a range of commodity price assumptions. The pit wall angles used are a simplified version of consultant-recommended overall pit slope angles.

By increasing commodity prices in a stepwise fashion, the methodology incrementally expands the limits of each pit shell in all directions until the point where the net value of the last increment in each shell is zero. Pits are determined using measured and indicated resources only.

A preferred pit shell is selected by evaluating the derived nested pit shells on the basis of a number of metrics including supporting commodity price, approximate cash flow, strip ratio, metal production, equipment requirements, and number of operating years. The pit shell selected is the reserve basis shell and is used as a guide to develop the detailed pit design.

The input parameters used to derive the reserve basis pit shell are detailed in Table 15-1. Metal recovery estimates were produced based on grade recovery models developed from the metallurgical testing discussed in section 13 of this report.

All costs are in Canadian dollars (C$) and units are metric unless stated otherwise.

15.1 Assumptions, Parameters and Methods – *Cont'd*

(a) Pit Size Determination – *Cont'd*

Table 15-1: Lerchs-Grossmann Inputs

Model Input	Value
Copper Price	US $2.40/lb
Gold Price	US $1000/ oz
Silver Price	US $13.50/ oz
Exchange Rate	US $0.80 = C$1.00
Pit Rim Mining Cost – Overburden	$1.62/tonne mined
Pit Rim Mining Cost – Non-PAG Waste	$1.81/tonne mined
Pit Rim Mining Cost – PAG Waste	$2.23/tonne mined
Pit Rim Mining Cost – Ore	$1.72/tonne mined
Bench Incremental Cost	$0.029/bench
Processing Cost	$4.34/tonne milled
Water Treatment Cost	$0.07/tonne milled
G&A Cost	$0.84/tonne milled
Sustaining Capital	$0.25/tonne mined
Copper cut-off grade	0.17% Cu
Average Copper Recovery*	90%
Average Silver Recovery*	56%
Average Gold Recovery*	59%
Off-Property Costs	$0.47/lb Cu
Payable Copper in Concentrate	96.1%
Payable Silver in Concentrate	90%
Payable Gold in Concentrate	90%
Overall Slopes	Range from 30 to 40 degrees

* Average metal recoveries calculated for ore contained within the pit optimization limits

15.1 Assumptions, Parameters and Methods – *Cont'd*

(b) Pit Design

The ultimate pit design is based upon the selected Lerchs-Grossmann (LG) pit shell. Access ramps, sector-specific wall angles, practical mining development considerations and scheduling factors were incorporated into developing the ultimate pit.

Overall pit slope design is based on recommendations made by geotechnical consultants shown in Table 15-2.

Table 15-2: Overall Pit Slope Design Criteria

Zone	Azimuth	Maximum Overall Slope
North and Northwest Facing Slopes	120º to 225º	30°
All Other Slopes	0° to 120° 225º to 360º	40°

The ultimate pit outline is illustrated in Figure 15-1.

Figures 15-2 and 15-3 provide selected cross sections across the ultimate pit.

15.1 Assumptions, Parameters and Methods – *Cont'd*

(b) Pit Design – *Cont'd*

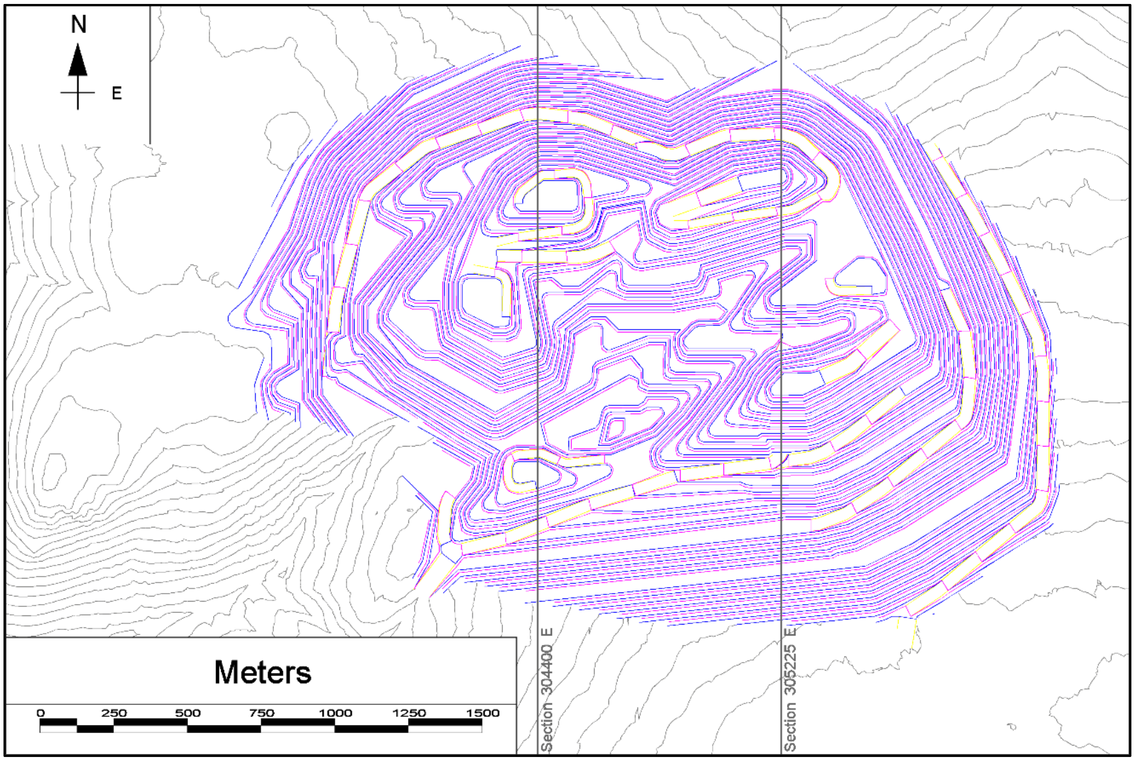

Figure 15-1: Ultimate Designed Pit – Plan View

15.1 Assumptions, Parameters and Methods – *Cont'd*

(b) Pit Design – *Cont'd*

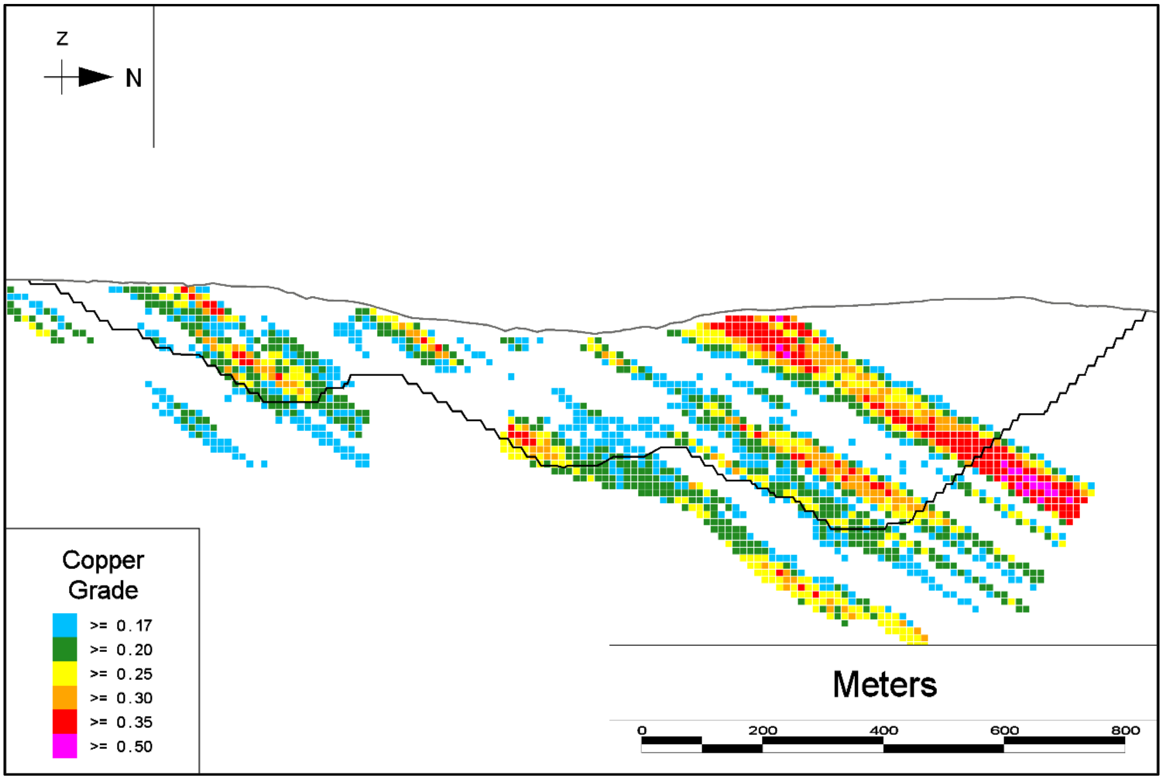

Figure 15-2: Ultimate Designed Pit – Section 304400E Looking West

15.1 Assumptions, Parameters and Methods – *Cont'd*

(b) Pit Design – *Cont'd*

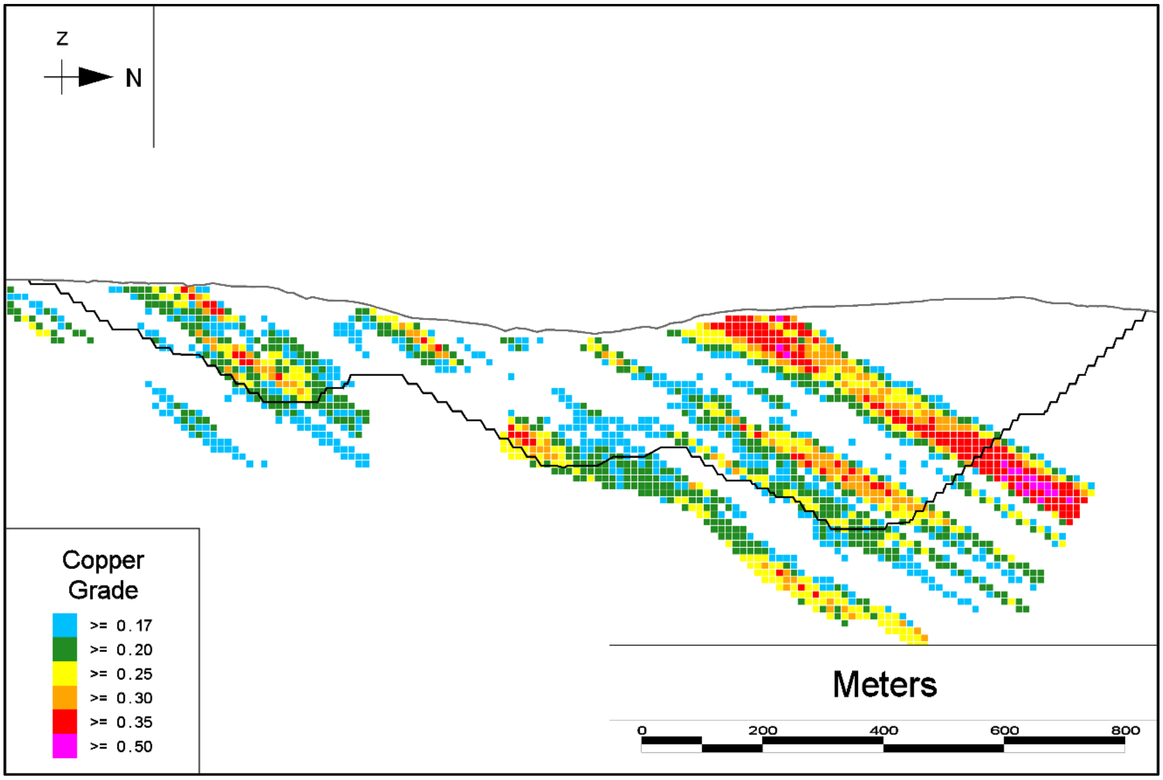

Figure 15-3: Ultimate Designed Pit – Section 305225E Looking West

15.1 Assumptions, Parameters and Methods – *Cont'd*

(c) Cut-Off Grade

An optimum cut-off grade was selected by developing a series of mine schedules and corresponding cash flows at various cut-off grades within the reserve basis pit shell. The cash flows were evaluated on the basis of annual cash flow, annual metal production, capital requirements, and NPV. The analysis resulted in selecting a copper cut-off grade of 0.17%. In the opinion of the author, the current cut-off grade is appropriate based on the grade distribution of the orebody, mill capacity, forecast long range metal prices, capital costs, and operating costs.

15.2 Mineral Reserves

Reserve classifications used in this study confirm to the 2014 CIM Definition Standards:

***Modifying Factors** are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.*

*A **Mineral Reserve** is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.*

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

*A **Probable Mineral Reserve** is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.*

*A **Proven Mineral Reserve** is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.*

In order to meet the requirements of NI43-101 with respect to determining the economically mineable part of the resource, an LG shell was determined through the process discussed in section 15.1. This shell formed the basis for the detailed pit design, scheduling of the mine and the development of a cash flow. This pre-feasibility study includes adequate information on mining, processing, metallurgical, economic, and other relevant modifying factors that demonstrate, at the time of reporting, that economic extraction is justified.

15.2 Mineral Reserves – *Cont'd*

Proven and probable reserves are derived from measured and indicated resources respectively, that are contained within the final ultimate design and are above the stated copper cut-off grade. Table 15-3 summarizes the proven and probable mineral reserves as of December 31, 2019.

Table 15-3: Yellowhead Mineral Reserves

Category	**Tonnes (millions)**	**Cu (%)**	**Au (gpt)**	**Ag (gpt)**	**Cu Eq. * (%)**
Proven	458	0.29	0.031	1.3	0.31
Probable	359	0.26	0.028	1.2	0.28
Total	817	0.28	0.030	1.3	0.29

*Copper Equivalent is based on an 90% copper recovery, US$3.10/lb copper price, 56% gold recovery, US$1350/oz gold, 59% silver recovery, and US$18.00/oz silver price.

The reference point for the reserves is the point where the ore is delivered to the processing plant. The mineral reserves presented in Table 15-3 are contained within the mineral resources stated in section 14 of this report.

It is the opinion of the author that the classification of proven and probable mineral reserves as estimated in Table 15-3 meets the definitions of proven and probable mineral reserves as stated by NI 43-101 and defined by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines.

15.3 Mineral Reserve Sensitivity to Mining, Metallurgical, Infrastructure, Permitting, and Other Relevant Factors

As with any mining operation there are a number of factors that may have a material and adverse impact on the operating performance, operating costs, and revenue estimated as the basis for resources and reserves in this report. The mineral reserve estimate was based upon economic parameters, geotechnical design criteria and metallurgical recovery estimates detailed in this report. Changes in these assumptions may impact the mineral reserve estimate.

Relative to the estimates presented in this report, increases in operating costs and/or reductions in estimated revenue, whether due to metallurgical recovery, commodity prices, or exchange rates, will negatively impact economic valuation of the project. However, the conservative commodity price assumptions relative to consensus pricing used to confine the reserve and the use of an optimized cut-off grade as opposed to a break-even cut-off grade will accommodate some variability in these factors without affecting the reserve estimate.

The project will require licenses and permits from various governmental authorities. There can be no assurances that Taseko will be able to obtain all necessary licenses and permits that may be required to carry out all proposed development and operations.

SECTION 16

MINING METHOD

SECTION 16: MINING METHOD

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16.1 Introduction

The Yellowhead Development Plan 1 (YDP-1) envisions an open pit mine utilizing conventional truck and shovel mining techniques. The equipment utilized in this operation would be typical of that found in today's large open pit operations. Open pit operations are planned to supply a conventional copper concentrator with 90,000 tpd of ore at a cut-off grade of 0.17% copper. Ore would be delivered to a primary crusher located at the southwestern rim of the ultimate pit. An ore stockpile would be built during the first five years of operation to maximize ore grade delivered to the mill during that period and provide an operating contingency. Potentially acid generating (PAG) waste rock would be stockpiled inside the tailings storage facility while non-acid generating (NAG) waste and overburden would be stockpiled in conventional waste storage locations proximal to the open pit.

16.2 Pit Design

The pit design is based on the selected Lerchs-Grossmann pit shell described in section 15 of this report. Access ramps, sector-specific wall angles, practical mining development considerations and scheduling factors were incorporated into developing the ultimate pit with intermediate phases.

Slope design for the Yellowhead pit is based on recommendations made by geotechnical consultants shown in Table 16-1. A single-bench configuration of 15 m high benches is used based on the scale of mining equipment selected. Steeper inter-ramp slopes up to 150 m high are used with enlarged berms or haul roads breaking up larger slopes to honor overall slope requirements.

Table 16-1: Pit Slope Design Criteria

Zone	Azimuth	Bench Face Angle	Inter-Ramp Angle	Maximum Overall Slope
North and Northwest Facing Slopes	120 to 225º	60º	35°	30°
All Other Slopes	0 to 120° 225 to 360º	70º	44°	40°

Haul roads are designed 40 m wide to allow for double-lane hauling including allowances for berms and ditches. Single-lane, 27.5 m wide roads are used to maximize ore extraction and mining width at pit bottoms. Road grades are limited to 10% with flat switchbacks.

For scheduling purposes, the ultimate pit has been divided into five interim phases. The mine schedule considered the following objectives in order to ensure efficient and practical mining operations:

- Target areas of higher copper grade to maximize copper production early in the mine plan;
- Maintain sufficient mining width on each bench for efficient operations in each phase;
- Limit vertical bench mining rate to no more than 6 benches per year;
- Supply enough non-acid generating (NAG) waste rock to meet material requirements for ex-pit infrastructure construction activities; and
- Provide an efficient ramp system that minimizes haul distances to ore and waste destinations.

16.3 Mine Dewatering

A dewatering system is designed to remove surface runoff and groundwater inflows from the open pit. The system is designed to initially remove water from the starter pits and would be expanded as the pit depth increases. Water would be pumped to the to the process water pond and to the TSF via the mill.

16.4 Waste Rock Storage

The total pit waste rock produced would be 1.1 billion tonnes. This includes

- 50 million tonnes of overburden type waste;
- 560 million tonnes of non-acid generating (NAG) waste rock; and
- 500 million tonnes of potentially acid generating (PAG) waste rock.

Overburden waste consists of the unconsolidated materials located above bedrock. Overburden of sufficient quality for reclamation use would be segregated from NAG waste rock and stockpiled in several locations surrounding the pit.

NAG waste rock is planned for use in constructing the initial (TSF) main embankment. The north and northwest embankments to be constructed later in mine life would also be constructed from NAG waste rock. Surplus NAG waste rock not designated for embankment construction would be stored in four locations located to the south and southwest of the open pit.

PAG waste rock would be placed within the TSF facility for ultimate subaqueous storage.

In-pit storage of waste rock is planned for later in the mine life when final pit walls have been exposed. Both NAG and PAG wastes would be stored in-pit with PAG stored only such that it is ultimately stored subaqueously as the pit fills with water in closure.

Waste rock stockpiles are designed based on the recommendation of geotechnical consultants. In-pit dumps and other temporary slopes that would ultimately not require resloping in closure are designed using slopes of 1.3 : 1.

16.5 Ore Storage

Ore is classed into the following three categories: PAG ore, high-grade NAG ore and low-grade NAG ore using a cut-over grade of 0.25% copper.

PAG ore and high-grade NAG ore mined during the pre-production period would be stockpiled within the ultimate pit footprint and processed in year 1. Excess low-grade NAG ore mined would also be stockpiled within the ultimate pit footprint and west of the ultimate pit adjacent to the primary crusher for processing in years 6 through 11.

The overburden, NAG and PAG waste rock storage areas in their ultimate configurations are shown in Figure 16-1.

16.5 Ore Storage – *Cont'd*

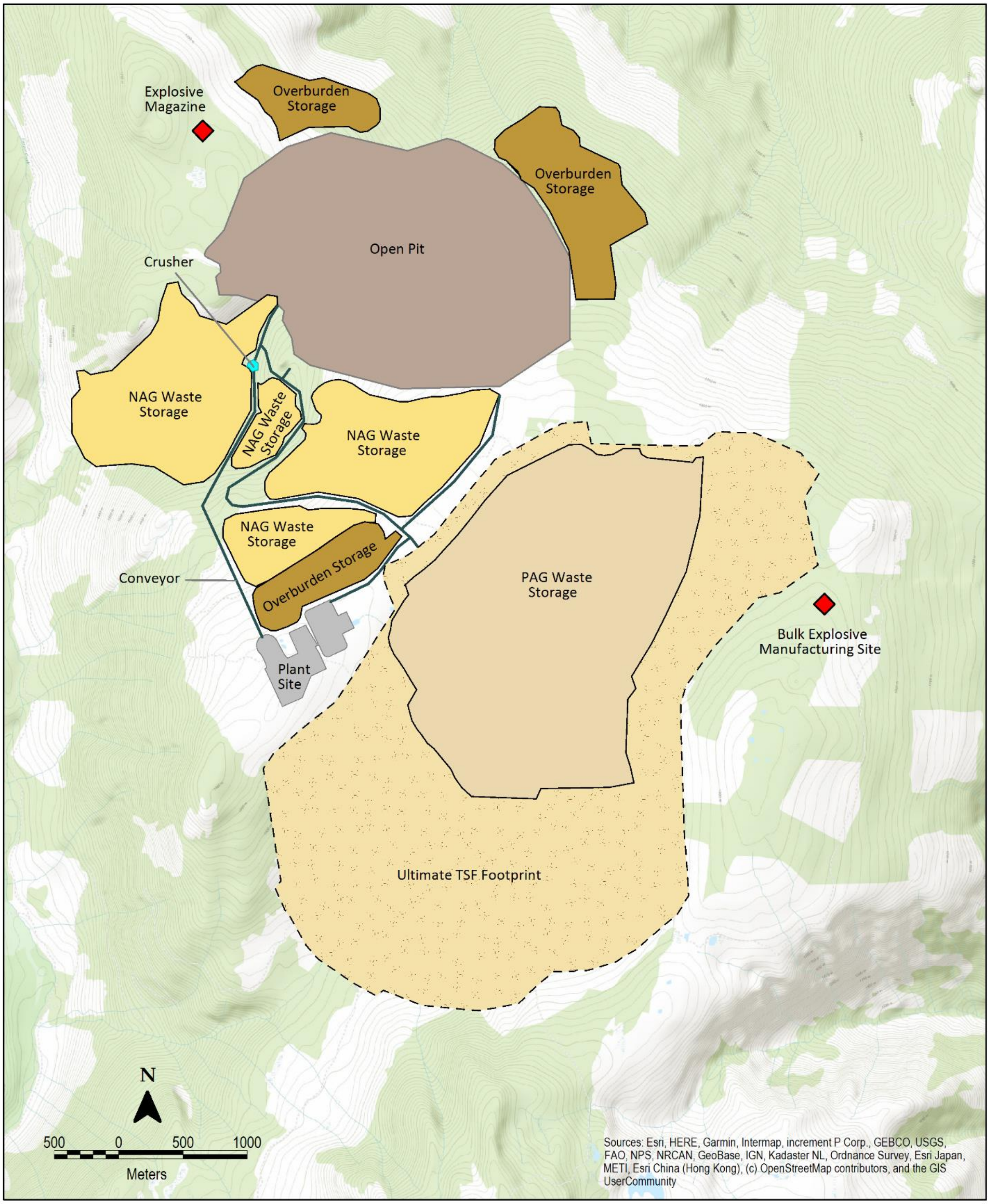

Figure 16-1: Ore and Waste Storage Areas – Year 25

16.6 Dilution and Ore Loss

For reserve and planning purposes dilution and ore loss are considered to be zero for the following reasons:

- The deposit shows good lateral and vertical continuity at the cut-off grades applied for scheduling;
- There is a broad width to the ore zones on individual benches;
- A detailed grade control program will be implemented; and
- Internal dilution is reflected through sample compositing and interpolation techniques used to generate the resource model.

16.7 Major Mine Equipment

The major mining equipment fleet is listed in Table 16-2.

Table 16-2: Major Mining Equipment

Unit	Capacity	Maximum Fleet Size
Electric Rotary Drill	311 mm hole size	5
Electric Rope Shovel	55 m^3	3
Diesel Hydraulic Shovel	36 m^3	1
Front-End Loader	30 m^3	1
Haul Truck	290 t	25

Production fleet equipment requirements have been determined using industry standard first principle-based calculations of productivities and equipment hours required to meet the annual production requirements. Truck productivities are based upon cycle times calculated between each mined bench to the various material destinations and are used in conjunction with the mine schedule to determine the annual truck fleet requirements.

A fleet of support equipment consisting of track dozers, wheel loaders, motor graders and service vehicles is also included.

16.8 Production Schedule

Pre-production mining focuses on pre-stripping of pit phases 1 and 2, establishment of an ore stockpile to support mill start-up, construction of the main haul roads to the various material destinations, construction of the starter dam for the TSF main embankment and filling the primary crusher pad. Ore mined during the period would be stockpiled inside the ultimate pit footprint.

For the first 5 years of operations, ore supply is planned from pit phases 1 and 2 which would be completed in years 4 and 5 respectively. Mining of both ore and waste from phase 3 would begin in year 4.

Mining in years 6 through 10 is planned from phases 3 and 4 with mining in phase 4 beginning in year 6. Ore would be supplied from phase 3 continuously and from phase 4 starting in year 8. The ore supply would be supplemented with low-grade ore mined and stockpiled in the previous five years.

In years 11 through 15, mining is planned in pit phases 3, 4 and 5 with phase 5 starting in year 11 and phase 3 concluding in year 14. Ore would be supplied from phases 3 and 4 throughout this period with phase 5 mining in waste only.

Mining would continue in pit phases 4 and 5 throughout years 16 to 20. Both phases supply ore continuously through this period but only minor amounts of ore are supplied from phase 5 until year 19.

From years 21 through 25 mining would predominately occur in phase 5 with mining in phase 4 completed in year 21.

Figure 16-2 shows mill feed by phase and Figures 16-3 through 16-8 show End of Period (EOP) maps for end of pre-production and years 5, 10, 15, 20, and 25 of the proposed project schedule.

The summarized annual production schedule results are provided in section 22 of this report.

16.8 Production Schedule – *Cont'd*

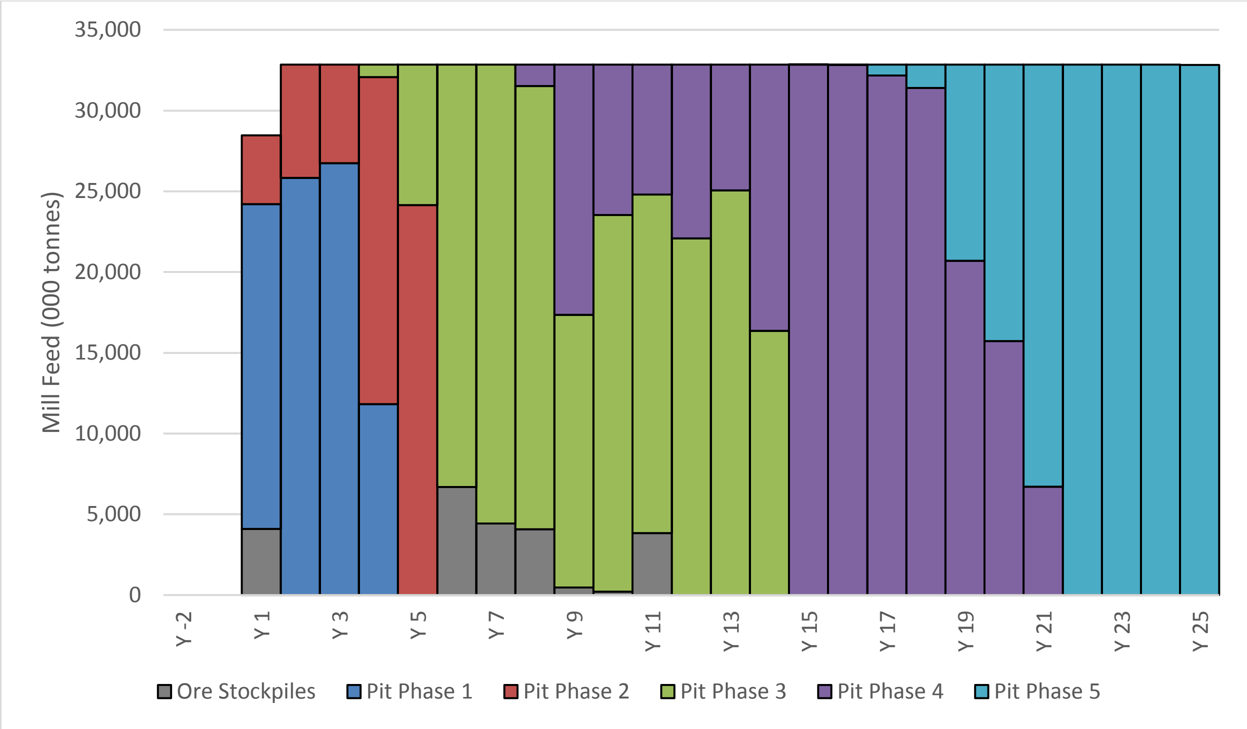

Figure 16-2: Mill Feed by Phase

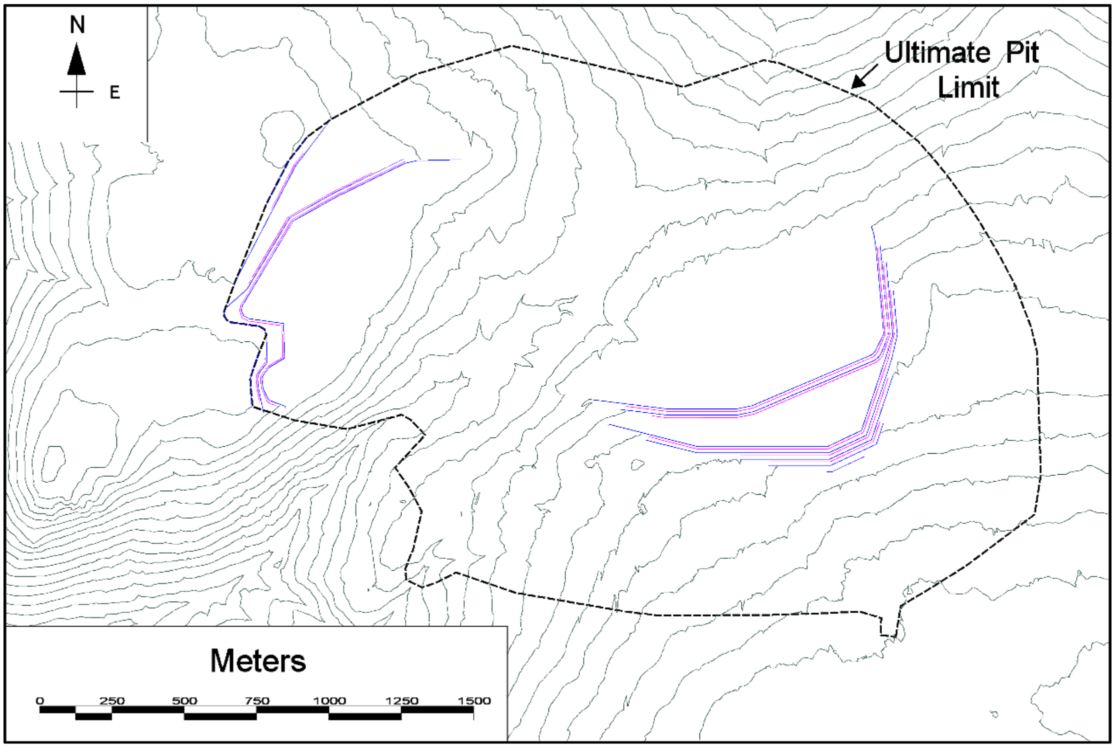

Figure 16-3: End of Pre-Production

16.8 Production Schedule – *Cont'd*

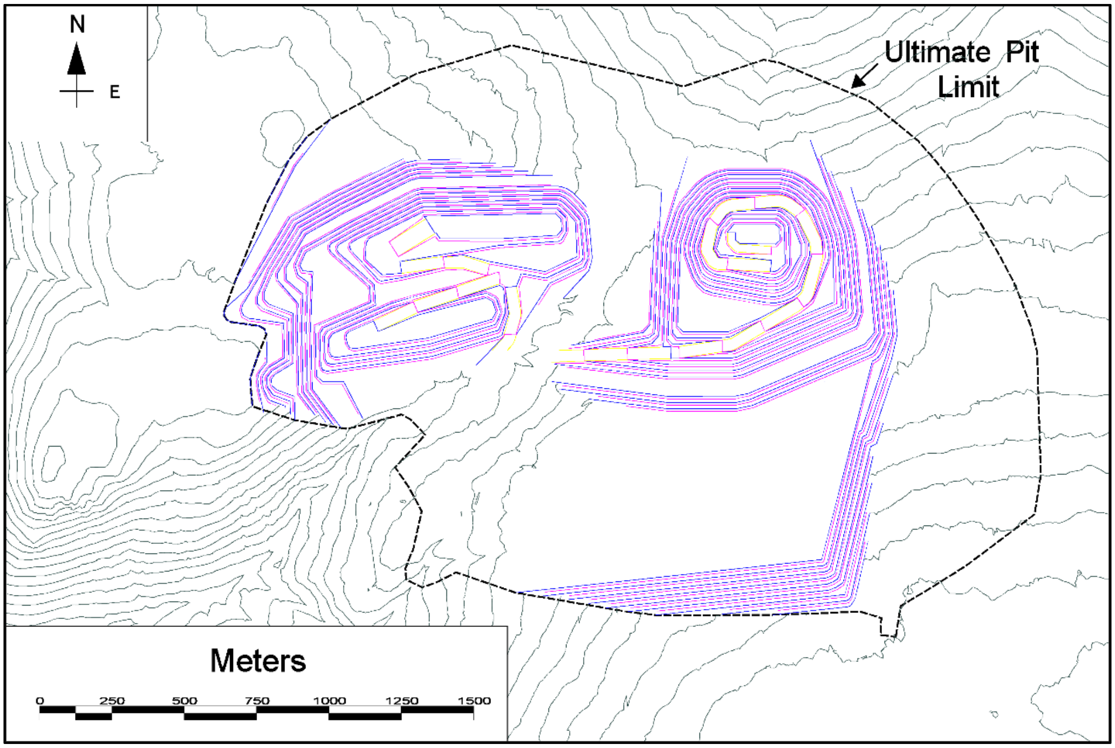

Figure 16-4: End of Year 5

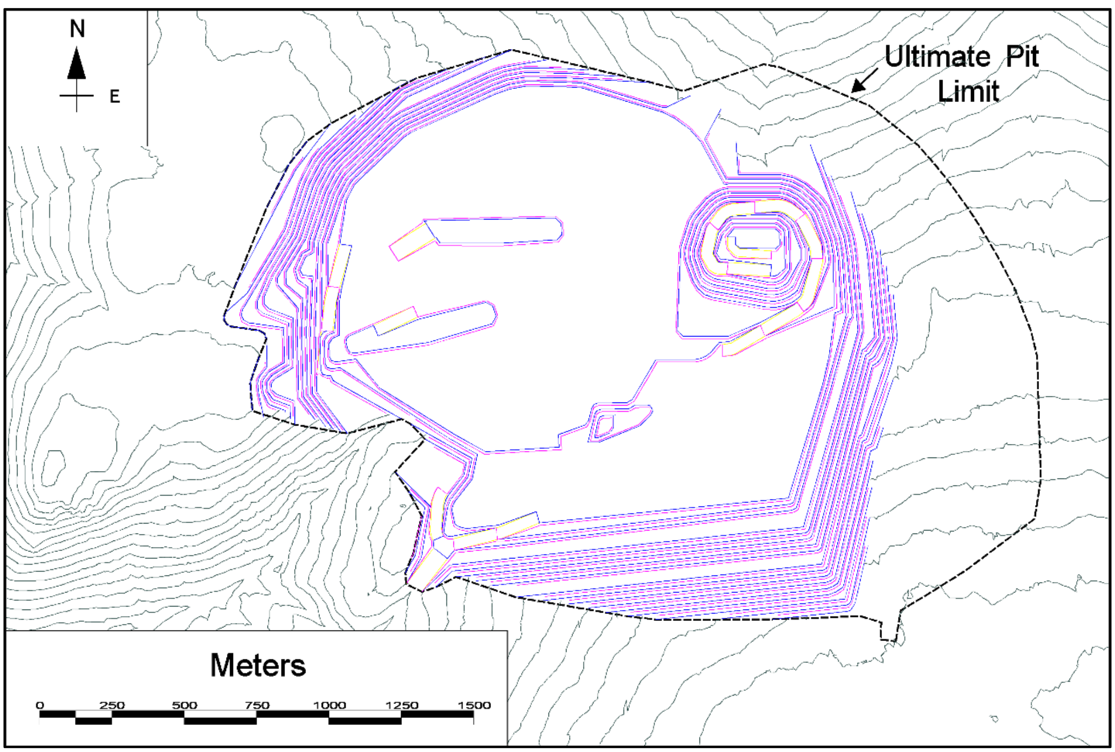

Figure 16-5: End of Year 10

16.8 Production Schedule – *Cont'd*

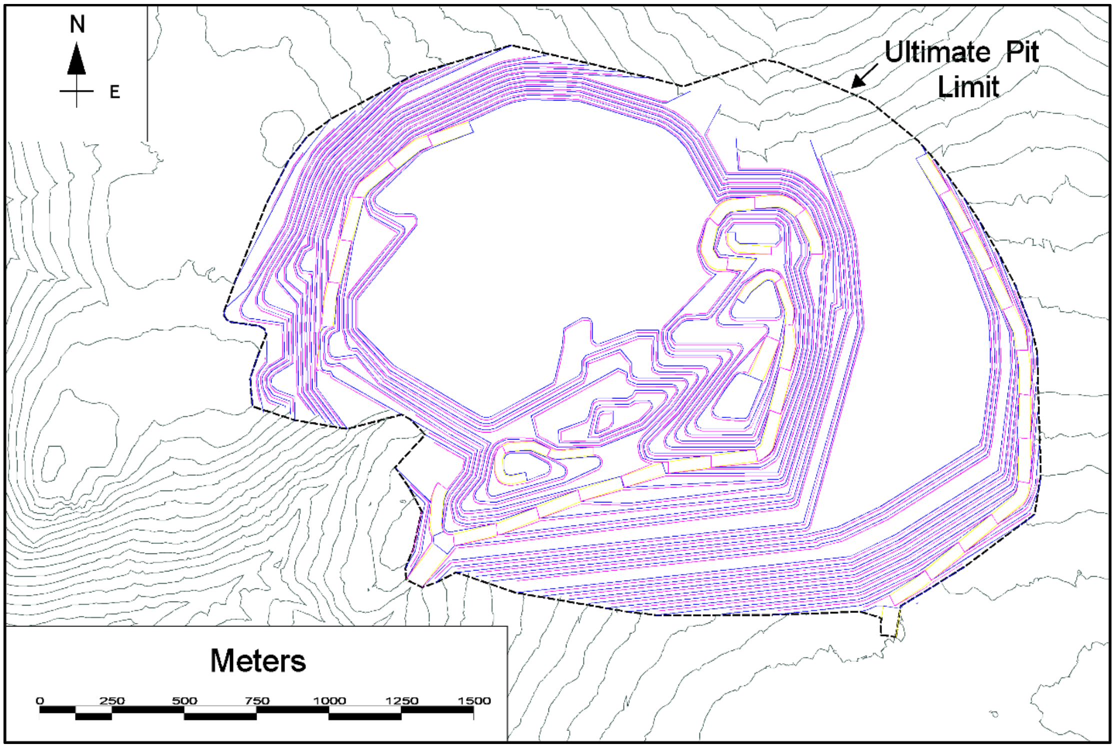

Figure 16-6: End of Year 15

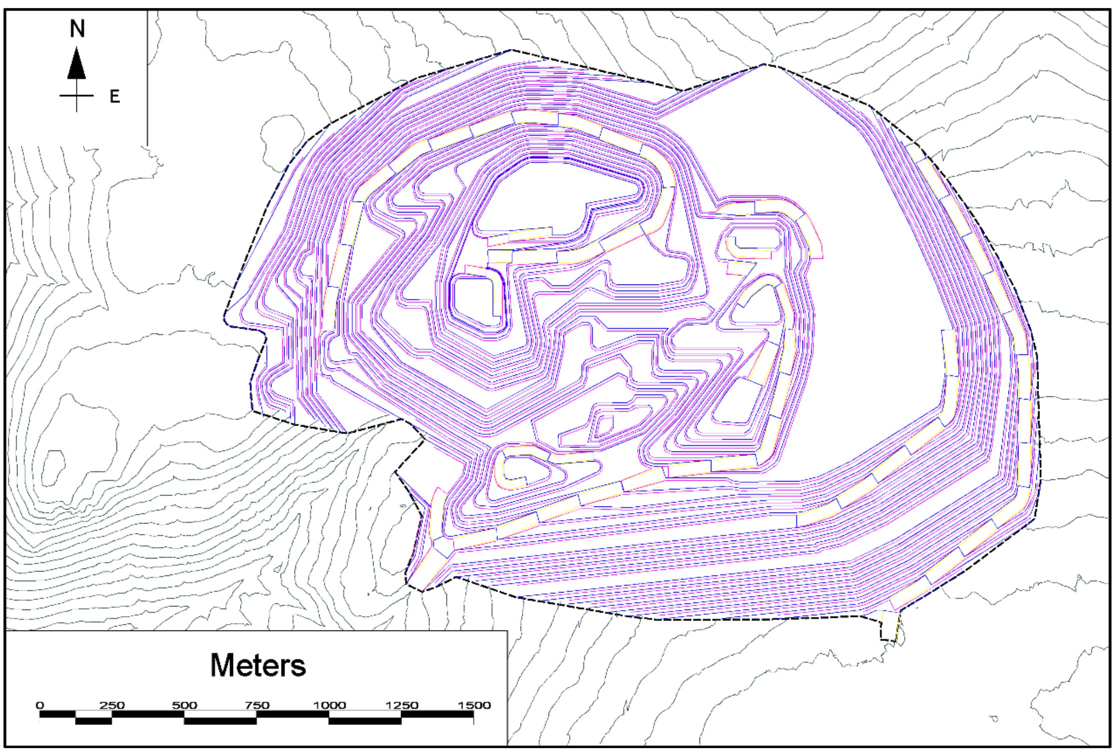

Figure 16-7: End of Year 20

16.8 Production Schedule – *Cont'd*

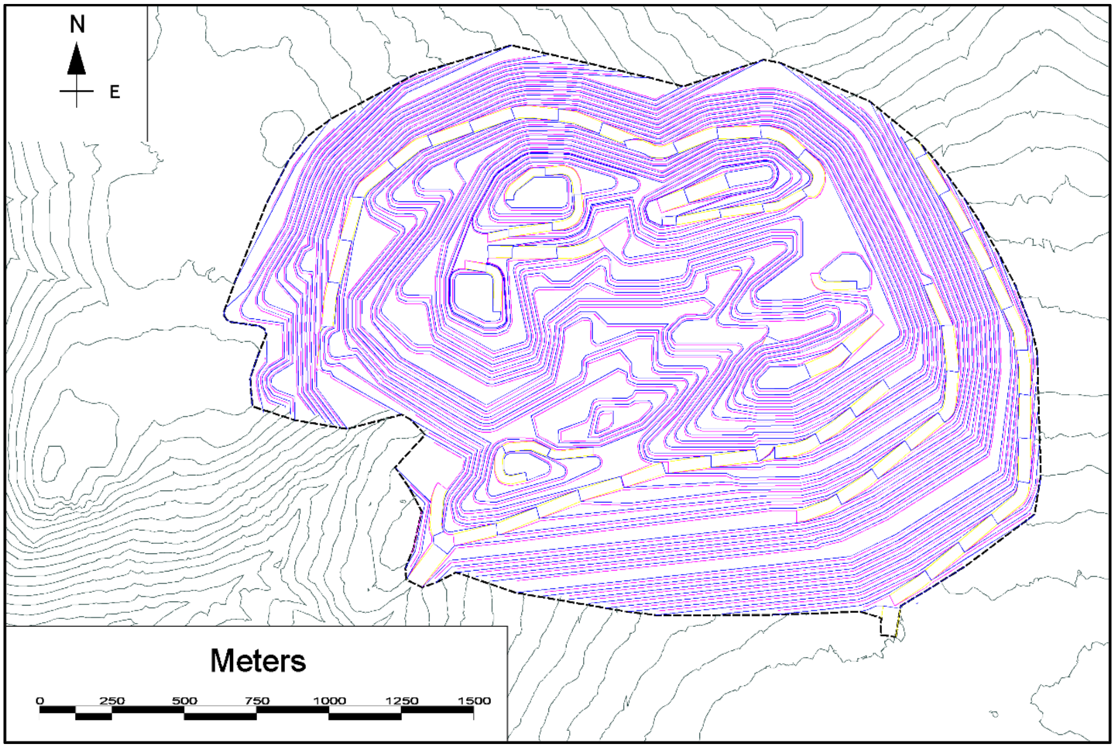

Figure 16-8: End of Operations (Year 25)

SECTION 17

RECOVERY METHOD

SECTION 17: RECOVERY METHOD

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17.1 Introduction

The proposed process plant for the Yellowhead ore is a conventional sulphide concentrator utilizing three stages of comminution, three stages of flotation and concentrate dewatering. The concentrator has been designed for simplicity of operations and maintenance and to meet the project metallurgical targets. Process design and equipment sizing undertaken were informed by results obtained from the 2011/2012 feasibility metallurgical test program conducted at G&T as discussed in section 13 of this report.

The concentrator is designed to process 90,000 tonnes per day of ore and produce a marketable copper concentrate containing payable amounts of silver and gold. The concentrator would consist of a gyratory crusher fed ROM ore from the open pit. The product from the crusher would be transported via overland conveyors to a coarse ore stockpile. Ore from the stockpile would be reclaimed and fed to two parallel SAG-ball mill circuits which produce feed for a single rougher flotation bank. The rougher flotation concentrate would be reground with two parallel vertical stirred mills prior to being reprocessed in a two stage cleaner flotation circuit which would include both tank and column flotation cells. Sulphide minerals would be collected with a conventional xanthate collector and pyrite rejected using lime.

The final concentrate would be dewatered by thickening followed by filtration to meet transportation moisture requirements prior to being conveyed to the final concentrate stockpile. The final concentrate would be trucked off-site to a proximal rail load out facility for subsequent transport to the Port of Vancouver or direct rail to other North American markets.

Both rougher and first cleaner flotation tailings would be transported separately to the tailings storage facility. Process water from the TSF would be reclaimed and recycled back to the process plant for reuse.

A simplified process flowsheet is presented in Figure 17-1.

17.1 Introduction – *Cont'd*

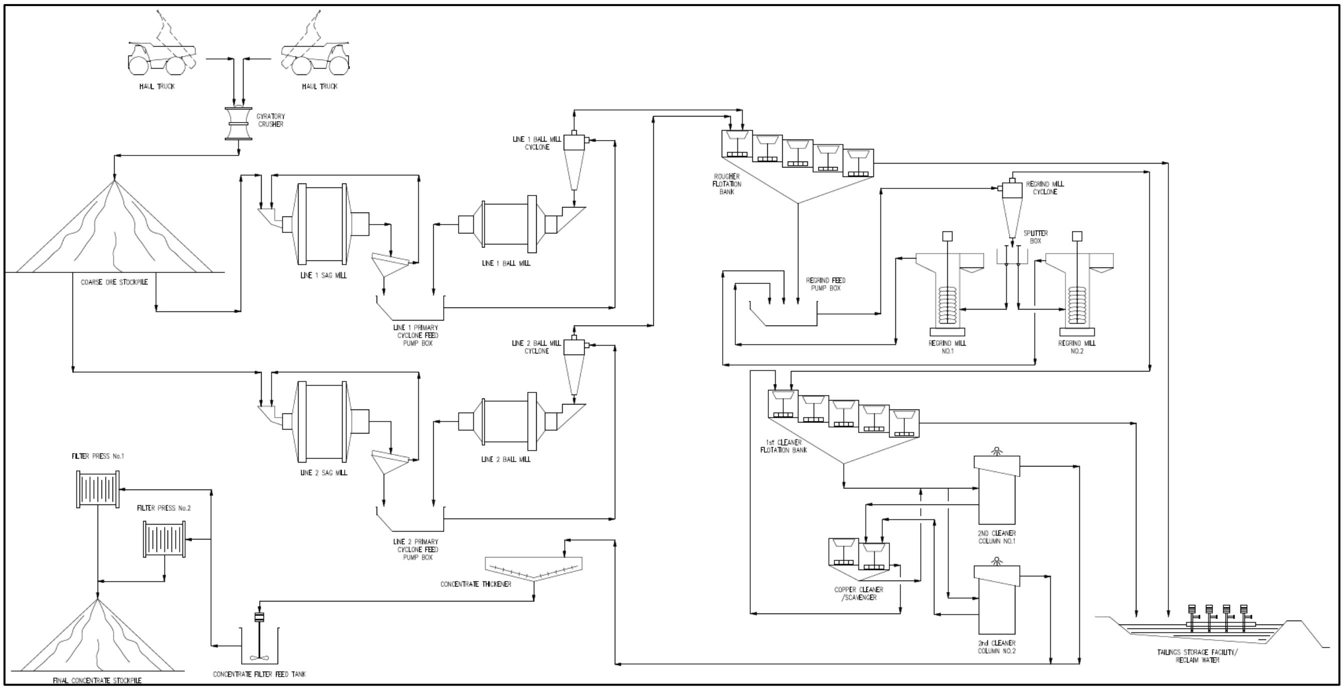

Figure 17-1: Simplified Process Flowsheet

17.2 Plant Design & Equipment

The crusher facility is planned to be located near the pit rim, with crushed ore being transported via an overland conveyor system to the concentrator, located near the topological high between the pit and TSF areas. The TSF is proposed to be located south of the concentrator facility and tailings would be transported from the concentrator via gravity pipelines.

The process plant is designed with the following unit operations:

- Crushing and overland conveying;
- Coarse ore stockpile and reclaim;
- Primary grinding;
- Rougher flotation;
- Concentrate regrind;
- Cleaner flotation;
- Concentrate dewatering;
- Concentrate storage and transportation;
- Tailings storage and water reclaim;
- Reagents handling and distribution;
- Assay and metallurgical laboratory;
- Water supply systems.

The following sections provide details on the process design criteria and each of the process unit operations.

An overall general arrangement for the concentrator facility is presented in Figure 17-2.

17.2 Plant Design & Equipment – *Cont'd*

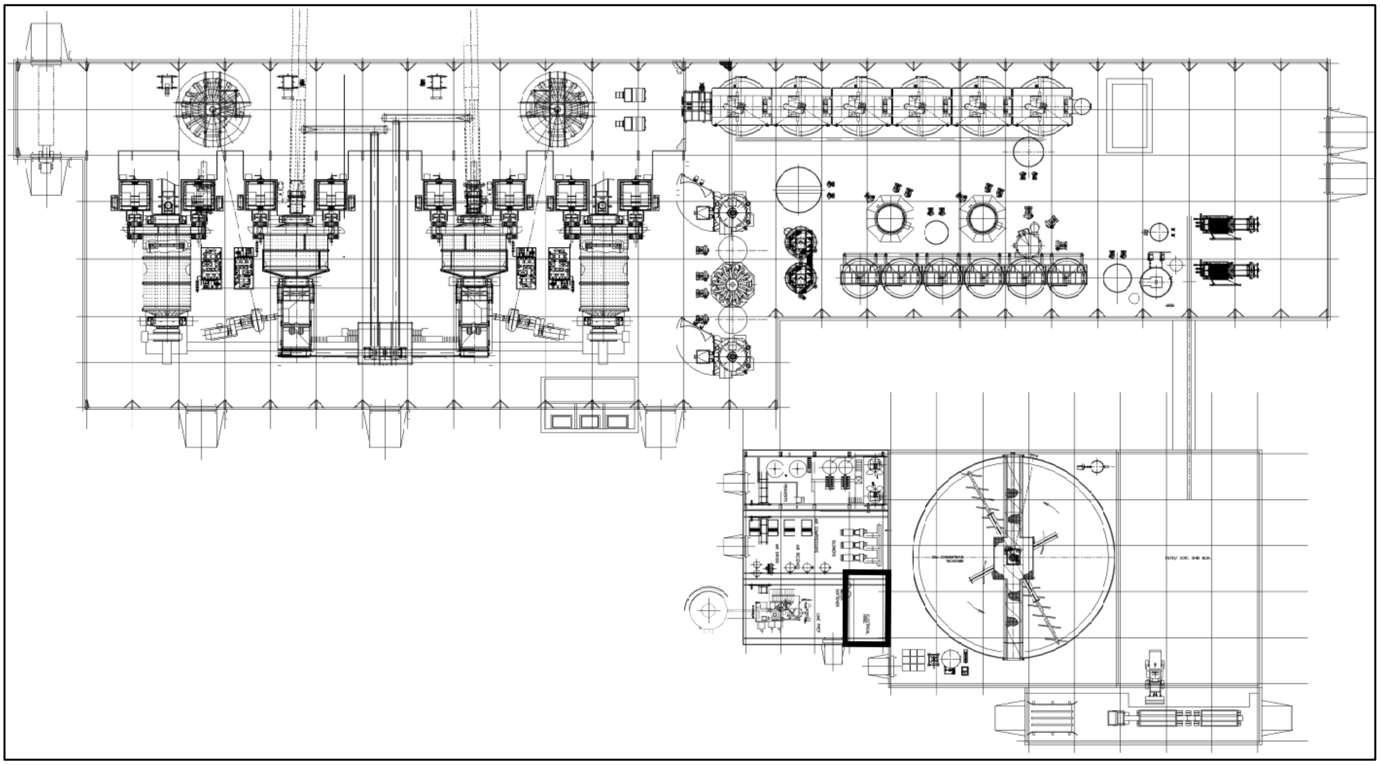

Figure 17-2: Concentrator General Arrangement Drawing

17.2 Plant Design & Equipment – *Cont'd*

(a) Major Design Criteria

The process facilities are designed to achieve a nominal throughput rate of 90,000 tpd, or 33 million tonnes per annum. Table 17-1 summarizes the major design criteria used for the facilities.

Table 17-1: Major Process Design Criteria

Design Criteria	Unit	Value
Operating Days	Days	365
Operating Time	Hours	24
Daily Throughput	tpd	90,000
Annual Throughput	tpa	32,850,000
Design Processing Rate	tph	4,076
Design Process Plant Copper Recovery	%	90
Design Final Concentrate Grade	% Cu	26
Crusher Availability	%	85
Grinding and Flotation Availability	%	92
Ore Specific Gravity	t/m3	2.8
Crusher Work Index	kWh/t	6.6
Bond Rod Mill Work Index	kWh/t	11.8
Bond Ball Mill Work Index	kWh/t	13.2
A*b		50.8
Primary Crushing Rate, dry	tph	7,500
Grinding and Flotation Process Rate, dry	tph	4,076
Ball Mill Product Size, 80% Passing	Micron	180
Regrind Product Size, 80% Passing	Micron	20

The SAG and ball mills were sized based on energy calculations using ore hardness results from SMC grindability and Bond work index testing described in section 13 of this report. The specific energy requirements for the regrind circuit were benchmarked from a pilot plant campaign conducted on an ore sample of comparable ore hardness with a similar target regrind product size.

The flotation cells were sized based on the estimated slurry flow rates and the flotation retention times as determined from lock cycle laboratory tests described in section 13. Typical scale-up factors were applied for sizing flotation cells and a minimum number of cells were applied based on experience to avoid short circuiting.

17.2 Plant Design & Equipment – *Cont'd*

(b) Crushing

The crusher facility is designed with a single gyratory crusher with a double-sided dump pocket for the mine haulage trucks. The facility would be located on south-west edge of the open pit to minimize ore haulage distances. The crusher will be serviced by a fixed hydraulic crane and a rock breaker. The crusher and conveyor system have been sized to process ROM ore at design rate of 7,500 tonnes per hour (tph), which is an excess capacity of approximately 45 percent more than process plant throughput. This excess crushing capacity provides operating and maintenance flexibility while minimizing feed disruptions to the process plant. The 80% passing product size generated at the discharge of the crusher is expected to range between 160mm to 250mm, depending on the crusher gap setting. The crusher product would discharge into a surge bin sized to hold approximately two truckloads of material. From the surge bin the crushed ore would discharge via an apron feeder which meters the crushed material onto the conveyor system that transports the ore onto the coarse ore stockpile. The crushing facility would also be equipped with a dust suppression/collection system to control any fugitive dust that is generated during crushing, material loading, and related operations.

The major equipment in this area includes:

- One 1,200 kW gyratory crusher: 1,524 mm x 2,794 mm (60 ft x 110 ft);
- One apron feeder: 2,438 mm x 10,100 mm;
- One hydraulic rock breaker;
- One fixed hydraulic crane;
- One 930 kW 1,828 mm (72 ft) x 360 m sacrificial conveyor;
- Two 1,500 kW 1,524 mm (60 ft) wide overland belt conveyor with a total length of 1.7 kilometers;
- One 1,500 kW 380 m long and 1,524 mm wide stacking conveyor to transport ROM to the coarse ore stockpile;
- Dust suppression systems.

17.2 Plant Design & Equipment – *Cont'd*

(c) Stockpile and Reclaim

The coarse ore stockpile is designed with a live storage capacity of 45,000 tonnes. The crushed ore would be reclaimed from the stockpile via two parallel conveying systems with three apron feeders installed on each conveyor line.

The apron feeders for each grinding line have been sized to allow nominal design throughput rates to be attained by operating only two out of the three feeders. The reclaimed ore from the apron feeders would discharge onto a belt conveyor, transporting the crushed ore to the SAG mills.

Each SAG mill feed conveyor has been designed with 30 percent excess capacity compared to nominal plant throughout, and would be equipped with a belt scale to measure and meter the SAG mill throughput at a controlled rate. The reclaim area would be equipped with a dust collection system to control fugitive dust generated during loading and transport of the crushed material.

The major equipment in this area includes:

- Six 22 kW 1,219 mm (48 ft wide) x 7,000 mm apron feeders;
- Two 447 kW 1,828 mm (60 ft wide) and 243 m long conveyor belts;
- Dust suppression system.

17.2 Plant Design & Equipment – *Cont'd*

(d) Primary Grinding

Primary grinding consists of two parallel SAG mill and ball mill circuits. Variable speed dual pinion driven SAG and ball mills have been incorporated in the circuit design. The grinding mills are driven by water cooled low speed induction motors, and the electrical drive systems for all of the mills are identical to keep equipment standardized with interchangeable parts.

Each grinding line is designed with a SAG mill which discharges onto a vibrating double-decked screen equipped with spray bars to wash down any entrained fines on the screen oversize. The screen oversize would be returned to the feed of the SAG mill via a pebble conveying system. Consideration has been made in the design for installation of a future pebble crusher. Screen undersize would report to a primary cyclone feed pump box where it would be combined with the ball mill discharge and pumped via a single centrifugal pump to a hydrocylone cluster. The underflow from the cyclones would be fed to the ball mill and the cyclone overflow advanced by gravity to the rougher flotation circuit. The ball mills have been designed for a circulating load of 350 percent and to produce a product size of 80% passing 180 μm. Reject steel from the SAG mills would be recovered via belt magnets installed on the pebble recycle conveying system, while reject steel from the ball mills would be collected via trommel magnets installed on the ball mill discharge. Steel media would be loaded into the mills via skips from steel media storage bins located on the south wall of the grinding circuit.

The major equipment in this area includes:

- Two dual pinion 17 MW SAG mills – 11 m x 6.2 m (36 ft x 20.25 ft) driven by variable frequency low speed induction motors;
- Two dual pinion 17 MW ball mills: 7.9 m x 13.4 m (26 ft x 44 ft) driven by variable frequency low speed induction motors;
- Two pebble recycle conveying systems, consisting of three conveyors including a high-angle conveyor;
- Two hydrocyclone clusters containing fifteen– 700 mm hydrocyclones per cluster;
- Two vibrating double-deck Screens: 3.6 m x 7.3 m with 7.5^{o} incline;
- Two Primary Cyclone Feed Pumps: 28x36 – 1,565 kW Slurry Pump.

17.2 Plant Design & Equipment – *Cont'd*

(e) Flotation and Regrinding Circuits

The ground ore from both grinding lines would be combined and processed in the flotation and regrind circuits to recover the valuable minerals. The recovery process would consist of rougher flotation, concentrate regrind, and two stages of cleaner flotation.

Copper Rougher Flotation Circuit

The rougher flotation circuit is designed with a single bank of forced air flotation tank cells fed the cyclone overflow product from both primary grinding lines. The rougher flotation circuit would produce a concentrate which would be pumped to the regrind circuit and a tailing stream which would gravity flow to the TSF. Flotation reagents added to the rougher flotation would be lime as a pH regulator, PAX as collector, and MIBC as a frother. Provisions have been made for additional reagents should they be required.

The major equipment in this area includes:

- Six 500 m^3 rougher flotation tank cells

Regrind Circuit

The regrind circuit is designed with two vertical stirred mills operating in parallel. Rougher concentrate slurry would be pumped from the regrind cyclone feed pump box to a cluster of regrind cyclones. The cyclones would classify the slurry with the underflow being split to feed the vertical stirred mills. The discharge of the vertical mills would be returned to the regrind cyclone feed pumpbox by gravity flow. The regrind cyclone overflow would transport the classified circuit product to the cleaner flotation circuit. The regrind hydrocyclone and pumping system has been designed for a circulating load of 250 percent. Lime would be added to the circuit to maintain slurry pH targets for downstream processing in the cleaner flotation circuit.

The equipment used in the regrind circuit includes:

- Two 3,355 kW stirred mills;
- One hydrocyclone cluster containing eighteen (18) 400 mm hydrocyclones (15 operating/3 standby);
- Two 12 x 10 -220 kW hydrocyclone feed pumps (one operation and one standby).

17.2 Plant Design & Equipment – *Cont'd*

(e) Flotation and Regrinding Circuits – *Cont'd*

Cleaner Flotation Circuit

The cleaner flotation circuit would consist of an open circuit first cleaner flotation stage and a closed circuit second cleaner stage. Reground rougher concentrate would be pumped to the first cleaner stage consisting of a bank of six forced air flotation tank cells. The tailings from the first cleaner cells would flow by gravity to the TSF in a dedicated pipeline. The concentrate from the first cleaner flotation cells would be pumped to the second cleaner stage consisting of two parallel flotation columns equipped with an external hydrodynamic sparging system to maximize fine particle recovery. The concentrate from both second cleaner columns would be the final copper concentrate and be pumped to the copper concentrate dewatering circuit. The tailings from both columns would be pumped to a second cleaner scavenger stage consisting of two forced air tank flotation cells. Concentrate from the scavenger cells would be reintroduced back to the second cleaner column feed, while the tailings would be returned to the feed of the first cleaners.

The equipment used in the cleaner scavenger circuit will include:

- Six 160 m^3 first cleaner flotation tank cells;
- Two 5 m x 12 m second cleaner flotation columns operated in parallel;
- Two 50 m^3 second cleaner scavenger flotation tank cells.

17.2 Plant Design & Equipment – *Cont'd*

(f) Concentrate Dewatering

The concentrate dewatering circuit consists of a high rate thickener, pressure filter, and material handling equipment to stockpile concentrate for shipment to the smelters. Final concentrate generated from the flotation columns would be pumped to the concentrate thickener where flocculent would be added to aid the settling process. The thickener underflow, estimated at 55 to 65 percent solids density, would be pumped to the concentrate stock tank and then pumped to the pressure filters for further dewatering to approximately 8 percent moisture. Filtered concentrate would be transported by conveyor to a stockpile prior to it being transported by truck to the off-site concentrate handling facility in Vavenby. The filtrate generated from the pressure filters would be returned to the concentrate thickener as dilution water. The concentrate thickener overflow would be collected and sent to the TSF and reclaimed back to the process water pond to be reused as process make-up water.

The concentrate dewatering circuit will include the following key equipment:

- One 35 m high rate thickener;
- One fully automated flocculant mixing and dosing system;
- Two parallel vertical plate pressure filters;
- Conveying system to concentrate stockpile;
- Slurry pumps, including the high head pumps for the pressure filters.

17.2 Plant Design & Equipment – *Cont'd*

(g) Tailings Storage and Water Reclaim

The TSF is proposed to be located to the south of the concentrator. The rougher and first cleaner flotation tailings from the concentrator would be transported to the TSF separately via gravity. Process water would be reclaimed back from the drained tailings and recycled back to the concentrator for re-use via a pump-back systems and the associated process water pond. Further details on the TSF can be found in section 18 and further details on the reclaim water system can be found in section 17.2 (j).

17.2 Plant Design & Equipment – *Cont'd*

(h) Reagent Handling and Distribution

The reagent facility would be located adjacent to the main concentrator and is designed to include systems for mixing, storing and distributing the various reagents required to facilitate the flotation process. Reagents used in the process would include:

- Potassium Amyl Xanthate (PAX) as a primary copper sulphide mineral collector;
- Provisions for a secondary trial collector;
- Methyl Isobutyl Carbinol (MIBC) as flotation frother;
- Lime as a pH regulator;
- Flocculant as settling aid in concentrate thickener;
- Antiscalent;
- Anti-freeze reagent.

Each reagent would have its own bulk handling, mixing, storage and distribution systems. The reagent facility would be equipped with appropriate ventilation, eye-wash stations, safety showers, fire and safety protection equipment.

Solid reagents such as PAX would be mixed with fresh water to a required strength in an agitated mix tank, then subsequently transferred to a holding tank. Frother would be delivered to site in bulk and off-loaded pneumatically directly to the holding tank. The holding tanks would be connected to a reagent distribution piping network equipped with metering pumps to deliver reagents to their respective process dosing points.

Lime would be delivered to site in bulk and off-loaded pneumatically into a lime silo. A lime slaking system utilizing stirred mill technology would be used for preparing milk of lime, which would be pumped from the holding tank, to points of addition using a closed loop distribution ring-main and control valves.

Liquid reagents such as antiscalent and antifreeze would not be diluted and would be pumped directly from bulk containers to their points of addition using metering pumps.

An automated flocculent skid located by the concentrate thickener would be used to prepare dilute flocculant solutions to be pumped to the feed well of the concentrate thickener to aid settling.

17.2 Plant Design & Equipment – *Cont'd*

(i) Assay and Metallurgical Laboratory

The assay and metallurgical laboratory, located south of the concentrator, would be equipped with the necessary analytical instruments to provide all routine assays for the mine, the process plant, and the environmental departments. Some of the major analytical instruments includes:

- Sulphur and carbon determination furnace (LECO);
- Atomic adsorption spectrophotometer (AAS);
- Fire assay equipment.

The metallurgical laboratory would be equipped with appropriate equipment to undertake routine test work to monitor and improve plant metallurgical performance.

17.2 Plant Design & Equipment – *Cont'd*

(j) Water Supply Systems

All of the process water would be distributed to the plant site from the process water pond. The bulk of the process water would be supplied from water reclaimed from the TSF via a reclaim barge and water pumping system. The reclaim barge consists of six vertical turbine pumps to feed the process water pond. Some supplementary water from pit dewatering and site collection ponds would also be pumped to the process water pond. The pond has been designed to have a storage capacity of 26,000m^3, amounting to approximately three hours of storage capacity based on nominal plant water demand. Separate fire water, and process water pumping systems have been designed to draw from the pond as required.

Major equipment in the area:

- One 14.6 m x 29.2 m x 3.7 m floating barge;
- Six 1,865 kW reclaim water barge pumps;
- Four 745 kW reclaim water booster pumps.

17.3 Energy Requirements

The annual power consumption for the concentrator (MWh per year) is based on the plant operating 24/7 with an availability of 92%. The primary grinding circuit would consume the largest proportion of energy in the concentrator at about 74 percent. The average and total consumed electrical loads in the concentrator are summarized by area in Table 17-2 below.

Table 17-2: Energy Requirements by Concentrator Area

Area	Average Load (kW)	Consumption (MWh per year)	% of Total
Primary Crushing	750	6,600	1%
Overland Conveying/ Stockpile Reclaim	5,800	51,000	7%
Primary Grinding	60,000	520,000	74%
Flotation & Regrind	9,800	86,000	12%
Reagents	100	800	0.1%
Dewatering	200	2,000	0.3%
Water Supply System	2,600	23,000	3%
Other (Lighting, Heat, Ventilation, Compressed Air)	1,500	13,000	2%
Total	**80,000**	**700,000**	**100%**

Note: Totals may not add due to rounding

17.4 Instrumentation & Control System

The process facilities would be controlled using a distributed control system (DCS). The DCS marshalling cabinets, servers, and controllers would communicate via a fully redundant ethernet network to form a plant wide control system. Motor starters, VFD's as well as field devices would be controlled by the DCS controllers via Devicenet. Process control and monitoring for the facility would be performed in two operator control rooms utilizing graphical operator stations. The operator control rooms would be located in the primary crusher area and the concentrator building.

17.5 Staffing Requirements

The concentrator is designed to operate 24 hours a day, 365 days a year. The workforce would be composed of technical, operational and maintenance personnel. The facility would be operated by four crews of area specific operators reporting to a crew supervisor. Maintenance work would be supervised by electrical and maintenance supervisors and conducted by trades consisting of millwrights, welders, pipe fitters, electricians and instrumentation personnel. The mill technical, operations and maintenance departments would each have a multi-level supervisory structure to ensure safe and efficient operations.

SECTION 18

PROJECT INFRASTRUCTURE

SECTION 18: PROJECT INFRASTRUCTURE

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18.1 Introduction

The infrastructure, services and ancillary facilities required for the project include the following:

- Site access;
- Power supply and site electrical distribution;
- Plant site area;
- Crusher and conveyor facilities;
- Concentrator building;
- Water management and treatment;
- Tailings distribution and storage facility (TSF);
- Maintenance facilities;
- Warehouse and storage facilities;
- Explosives facilities;
- Construction camp;
- Administrative and dry facilities;
- Site security and first aid;
- Fuel storage and dispensing;
- Sewage collection and treatment;
- Overburden, waste rock and ore storages;
- Rail load-out facility.

The proposed project site layout is shown in Figure 18-1.

18.1 Introduction – *Cont'd*

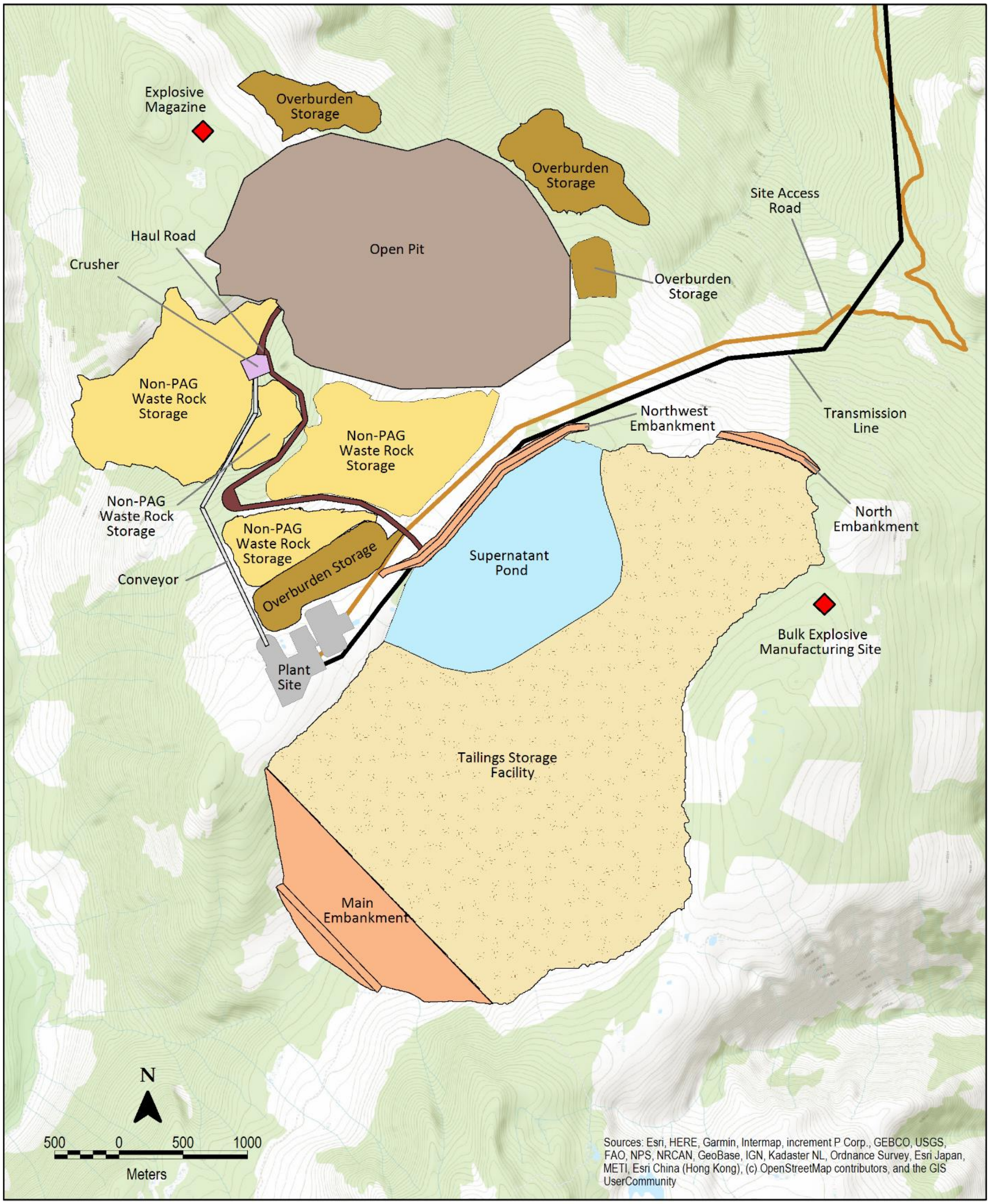

Figure 18-1: Site Layout

18.2 Site Access

Road access proposed to the project site is from Highway #5 at the town of Vavenby via 24 km of existing FSRs. These FSRs will require minor upgrading for operations traffic, such as widening, alignment and surface reparation. A 2.5 km extension from the end of the FSRs will be required to reach the plant site.

During the construction phase, oversized loads would use an existing secondary site access road. This alternate route crosses the North Thompson River at the Birch Island Lost Creek Bridge, which has been designed for heavier loads. This route follows the Birch Island Lost Creek Road after crossing the North Thompson River for 15.6 km, until it intersects with the proposed access road from Vavenby.

A rail load-out facility is designed to be constructed at an existing rail siding on a property owned by Taseko near Vavenby. Concentrate would be trucked from the plant site to the rail load-out facility where it would be loaded onto trains and transported to North American markets and/or to the port of Vancouver for overseas shipping.

18.3 Power Supply and Electrical Distribution

Electrical power for the project would be supplied by BC Hydro from the Vavenby substation. The current Vavenby substation would need upgrades from BC Hydro to be able to provide stable power to site. The company proposes to construct a 22 km overhead transmission line to bring power from the Vavenby substation to the project site.

The 138 kV overhead transmission line would terminate at the main substation, located at the plant site, and provide the site with 25 kV and 4.16 kV distribution voltages.

Emergency power would be provided by two 2 MVA stand-by generators located by the main substation. The generators would be installed during the pre-production phase to support construction efforts and afterwards the generators will be on standby for emergency power generation.

18.4 Plant Site Area

The designed plant site is located along the ridge between the TSF and the open pit and would host the concentrator, main electrical substation, mobile equipment maintenance shop, admin, warehouse and supporting facilities. The plant site is designed for adequate space between structures to allow for efficient construction of the facilities as well as long term maintenance and safe transportation of materials and personnel during operations.

18.5 Concentrator and Supporting Buildings

The primary crusher building would be located near the crest of the pit and will be fed by 290 tonne haul trucks from two dump pockets. The crushed ore would be delivered via a 2.4 km overland conveyor to a coarse ore stockpile, with a 45,000 tonne capacity, next to the concentrator.

The concentrator building is designed as a pre-engineered building consisting of both grinding and floatation circuits. Apron feeders underneath the coarse ore stockpile would feed material to the grinding circuit which would then feed the flotation circuit. The flotation circuit would produce a concentrate to be thickened and dried using filter presses and then conveyed into the concentrate shed for temporary storage until trucked to the rail load-out. Tailings would be discharged from the concentrator through pipelines for deposition in the TSF.

The assay lab is designed as a modular building located immediately to the south of the west side of the concentrator building.

The reagent building would also be a modular building located immediately south of the east side of the concentrator and house the reagents, air blowers and lime distribution systems. A lime storage silo would be located outside the reagent building for storage of lime for processing requirements.

18.6 Reclaim Water Storage and Distribution

Reclaim water would be pumped from a floating pump barge within the TSF to a process water pond next to the concentrator. The designed reclaim barge includes six pumps and a separate de-icing pump and bubbler system for winter operations.

Large diameter, parallel HDPE pipelines would connect the reclaim barge to the process water pond at the concentrator site. Water from the process water pond would be used in the concentrator and as feed water to the water treatment plant (WTP) for clean water discharge from site.

18.7 Water Treatment Plant

The WTP would be located adjacent to the process water pond, and is designed as a stand-alone plant used for processing site contact water. The water treatment plant would be fed by the process water pond. The initial water treatment plant is proposed for construction in year 2 and commissioning in year 3. It would house a metal removals circuit, an electroreduction circuit and a selenium ion exchange circuit. The water treatment plant would be constructed using a modular design to meet increasing treatment capacity requirements throughout the life of mine. It is proposed to be constructed in phases, in years 2, 9 and 20. Clean water would be discharged into Harper Creek through an HDPE pipeline. A reagent storage area would be located inside the building.

18.8 Fire Protection

Water for fire suppression would be sourced from the process water pond. A prefabricated pump station, including electrical pumps and supported by diesel backup pumps, is designed to deliver water to the all of the plant site area buildings in the event of fire emergencies.

Dry agent fire suppression would be available in all motor control centers and electrical rooms to maintain the integrity of electrical infrastructure during a fire.

18.9 Potable Water Supply, Storage and Distribution

Potable water is planned to be sourced from wells around the plant site and pumped to the potable water treatment plant, which will be a stand-alone plant only used for processing potable water. The water pumped from the wells would be treated and stored in the potable water treatment plant's storage tank and pumped to distribution points around the plant site.

18.10 Plant Maintenance Building

The fixed plant maintenance shop is designed as a pre-engineered structure located on the east side of the concentrator. The maintenance shop would service the concentrator, thickener and water treatment plant buildings.

18.11 Mobile Equipment Maintenance Shop

The mobile equipment maintenance shop would be a pre-engineered building. The designed mobile equipment maintenance shop includes a haul truck wash bay, four haul truck service bays, eight medium duty bays, four light duty bays, light duty wash bay and an adjacent welding tent sized for truck box repair and rebuilds. The mobile equipment maintenance shop offices and lunchroom would be located on the second floor of the building.

18.12 Warehouse Building and Cold Storage

The warehouse building is planned to be located to the west of the mobile equipment maintenance shop with associated cold storage laydown immediately adjacent to it. The warehouse and cold storage area would be used for the storage of parts and materials required for both mine and concentrator operations.

18.13 Explosives and Magazine

An onsite explosives plant is planned to be located near the southern end of the TSF. The explosives plant would be operated by an explosives manufacturer. The explosives magazines required to store accessories would be located on the north side of the open pit.

18.14 Construction Camp

The construction camp facility is proposed to be located near the town of Vavenby at the rail load-out facility property. The single story prefabricated modular building would support a peak construction workforce of 540 personnel. The camp would include full services including dormitories, kitchen and dining facilities. The camp would be removed after the construction phase is complete.

18.15 Administrative Building

The administration building is designed as a 2-storey prefabricated modular building sized to support engineering, operations and administrative staff. The building would initially serve as the construction team office and be repurposed for operations personnel after construction concludes.

18.16 Mine Dry Building

The mine dry would be a stick-built building hosting two separate changing areas. The dry would include offices for mine personnel. The mine dry has been sized for all site mining and milling operations workforce.

18.17 Security, Safety and First Aid

A gatehouse with first aid services, located at the entrance to the plant site, would provide access control of personnel and vehicles onto site.

Adjacent to the gatehouse would be an emergency response building, including a classroom, training area, ambulance, and mine rescue vehicles.

A small parking lot would be located outside the gatehouse for suppliers and visitors. Employees would be bused to the site from an employee parking lot at the rail load-out site.

18.18 Fuel Storage

Diesel fuel for mining and support equipment is designed to be supplied from a series of double walled, skid mounted diesel tanks located adjacent to the primary crusher platform on the crusher laydown area.

A secondary fueling station for diesel and gasoline, both contained in double walled storage tanks, would be located by the mobile equipment maintenance shop for ancillary mobile units and for fueling trucks after maintenance work.

18.19 Sewage Collection and Distribution

All plant site sewage would report to a membrane style biological treatment plant. The sewage plant building would comprise of a series of connected modular shipping container units designed for a treatment capacity suitable for all personnel on site. Treated water from the sewage digestor would be discharged into tailings, and solids wastes would be transported off site by a contractor.

18.20 Site Water Management

Site infrastructure is planned to segregate contact and non-contact water throughout the life of mine. Precipitation that falls as contact water would be diverted towards collection ponds where the water would be pumped to the process water pond. The collection ponds would manage sediment throughout site construction and operation. Any excess water pumped to the process water pond would flow through a spillway and diversion channel towards the TSF for storage. The contact water would be used in the concentrator processes. Non-contact water would be discharged into the receiving environment through ditching and piping.

18.21 Tailings Storage Facility

Tailings produced by the concentrator would be discharged into the TSF through a series of large diameter HDPE pipelines in two separate streams; PAG cleaner tailings and NAG rougher tailings. The NAG rougher tailings would be cycloned to create downstream raises of the main embankment and a tailings beach, while the PAG cleaner tailings would be separately deposited subaqueously. Tailings would be gravity discharged from the concentrator to the TSF and interrupted by drop boxes to reduce the energy in the pipeline. Starting in year 2, the NAG rougher tailings are planned to be cycloned and the coarse cyclone underflow material would be used to construct the downstream shell of the tailings embankment, while the finer overflow material would be deposited to create a tailings beach on the upstream face of the embankment.

The TSF is proposed to be located in the valley on the south side of the plant site, downstream of the concentrator. The main embankment would initially be constructed as a water retaining starter embankment, constructed in a downstream fashion beginning in year -2. The starter embankment would consist of a low permeability core zone, filter zone and a rock fill shell. Mine waste would be used as rockfill starting in year -1 and cease in year 5, while cycloned sand would be deposited concurrently from years 2 to 5. After year 5, cycloned sand would be used to construct centreline raises on top of the starter embankment to a final height of 165 m with a 3.5H : 1V downstream slope.

By year 18, two additional embankments will be required to provide storage capacity for operations. The north and northwest embankments would be constructed in years 12 through 16, to ensure completion prior to the year 18. The north embankment is designed as a water retaining downstream constructed embankment that would support the deposition of tailings along its upstream face, while the northwest embankment is designed as a water retaining centreline constructed embankment armoured with a rockfill upstream face to prevent erosion from wave run up. Both embankments would be constructed at a 2H : 1V slope.

The TSF has been designed for secure and permanent storage of 714 million tonnes of tailings and 467 million tonnes of PAG waste rock during the 25-year life of mine requiring a total storage volume of 779 million cubic meters.

The starter embankment would retain water beginning in year -1, to capture the required water volume for mill start-up. The tailings deposition strategy would develop large beaches to keep supernatant water away from the main and north embankments and cover the PAG mine waste subaqueously. NAG rougher tailings would be used to build a tailings beach upstream of the main embankment, which would develop by year 5 and ultimately approach a beach width of 1.8 km. The pond would be pushed towards the north end of the TSF until year 16 when non-PAG rougher tailings would be concurrently deposited at the main and north embankments. The resultant beach upstream of the north

18.21 Tailings Storage Facility – *Cont'd*

embankment would be 1.3 km wide pushing the pond to the northwest to cover the mined PAG waste rock subaqueously. The PAG cleaner tailings would be deposited subaqueously in the supernatant pond to prevent any potential acid leaching. PAG waste rock generated from the mine would cease after year 20 and be fully covered by deposited tailings by the end of the mine life.

Seepage to ground water throughout the basin below the TSF would be controlled to a large degree by the glacial till liner existing within the TSF area. If necessary, the till will be augmented to ensure that the hydraulic conductivity of the basin meets a design specification that limits basin seepage to an acceptable level.

Seepage through and under the main embankment will be minimized with a low permeability core in the starter embankment and with large beaches to keep the supernatant pond far from the embankment and lower the phreatic surface in proximity to the embankment.

Any seepage losses from the main embankment or abutments would be directed to the seepage pond at the toe of the main embankment. Cyclone underflow material used for construction of the main embankment would include water which would also be directed to the seepage pond at the toe of the TSF. All water reporting to the seepage pond downstream of the main embankment would be pumped back into the TSF through the main embankment seepage pump back system.

Any seepage losses from the north embankment would be similarly minimized by a low permeability core and establishing a large beach. Any seepage losses from the north embankment would be directed to and collected in the seepage pond at the toe of the north embankment and be pumped back into the TSF through the north embankment seepage pump back system.

Seepage losses from the northwest embankment would be similarly minimized by a low permeability core. The seepage losses from the northwest embankment would be directed to and collected in the site water management system and be pumped into the process water pond.

The TSF pond volume is planned to be maintained at an annual average of 13 million cubic meters roughly equivalent to 2 months storage with a maximum storage volume of 18 million cubic meters during freshet. At the end of the mine life the tailings pond water would be drained into the open pit through a spillway.

18.22 Overburden, Waste Rock and Ore Storage

Storage of overburden, waste rock and ore materials is required on site throughout the life of the operations. Storage would be constructed to easily capture and collect all contact water through ditching and drainage into lined ponds. These lined ponds would be sized to receive and pump the estimated peak daily flow during freshet to the process water pond where water would be used in processing or overflow into the TSF through a spillway and ditches.

SECTION 19

MARKET STUDIES AND CONTRACTS

SECTION 19: MARKET STUDIES AND CONTRACTS

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19.1 Market Studies and Contracts

Copper is a key commodity used extensively for all urban and industrial development and will continue to be so for the foreseeable future. Most industry experts believe the long-term fundamentals of the copper market are strong. While the price of copper has been relatively volatile over the past five years, the overall market supply and demand fundamentals have improved and there is an expectation that the market will shift from a balanced market to a deficit. Lower copper pricing over the past number of years has resulted in mining companies investing less in development projects, leaving a gap in the global project pipeline. With very few new major copper mines currently under construction, existing mines depleting and global demand for copper growing, a significant copper deficit is projected over the next three to five years.

The copper concentrate is estimated to have a 25.5% copper grade with payable amounts of gold and silver and no element approaching typical smelter penalty levels. While there are currently no contracts in place for the sale of concentrate, it is expected that the clean nature of the concentrate would make it attractive to a large array of smelters globally.

For evaluating the project, Taseko has relied on long term street consensus commodity pricing as of December 2019.

The offsite costs associated with concentrate transport, port storage, stevedoring, shipping, treatment and refining have been incorporated into the economic analysis of the project based upon Taseko's current experience at it's Gibraltar Mine.

Standard procurement contracts will be required for construction, materials delivery and some site services.

The qualified person has reviewed these costs and commodity prices and they support the assumptions in the technical report.

SECTION 20

ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

SECTION 20: ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

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20.1 Environmental Studies

(a) Baseline Studies

Baseline studies were performed between 2007 and 2014. The results of these are summarized as follows.

Climate and Air Quality

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 on site can range from summer highs of up to +26° C to winter lows down to -35° C. The mean annual precipitation is estimated to be 1,259 mm at an elevation of 1,837 masl, with about 40% falling as snow and 60% falling as rain. 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 1.6 m/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%.

Industrial activities within the regional area include forestry, the CN rail line passing through Vavenby, and traffic utilizing the Yellowhead Highway route (Highway #5) between Kamloops and Edmonton. A sawmill in Vavenby is shutdown as of the writing of this report. Right-of way clearing and installation of the Trans Mountain pipeline paralleling Highway #5 is not yet scheduled and will be a short-term activity. Overall baseline air quality in the project area is good due to the limited local emissions and the project's remoteness.

Noise

Potential noise sources in the surrounding area include the town of Vavenby, approximately 10 km northwest of the project. There is a rail line passing through Vavenby, and active logging in the area surrounding the project, along with a network of FSRs. Highway #5 runs along the North Thompson River and at its closest point is approximately 7 km to the north of the project and is a source of traffic noise in the area. Other activities contributing to noise in the area include tourism and recreation activities such as hunting, fishing and skiing.

20.1 Environmental Studies – *Cont'd*

(a) Baseline Studies – *Cont'd*

Terrain and Soils

Glacial till overlies much of the bedrock in the project area, while a surface veneer of colluvium is generally present in the areas of steeper terrain. Surficial soils locally comprise organic soils and silt-rich glacial lake deposits, which are particularly prone to erosion.

Glaciofluvial outwash deposits and the fluvial terrace of the North Thompson River valley occur in the areas of the proposed power line and rail load-out facility respectively. A probabilistic seismicity assessment for the project indicates the project is at low risk of a damaging seismic event.

Hydrology

The site is located within the Columbia Mountains hydrologic zone. The annual peak flow regimes of the watersheds in this hydrologic zone are dominated by spring snowmelt. Autumn rainfall events also can contribute significant amounts of water.

Water Quality

Baseline Study streams had near-neutral to slightly alkaline pH and waters were soft to moderately hard with turbidity highest during the freshet period (May to June) due to the greater volumes of discharge within streams. Concentrations of total and dissolved metals exhibit distinct seasonality, with the highest concentrations per site generally occurring during high-flow freshet periods.

Hydrogeology

The project is located within the Shuswap Highlands physiographic region, with bedrock typically having low permeability and low well yields and valleys containing permeable layers of glacial and post-glacial sediments capable of high-yield wells.

Baseline studies to date have included borehole drilling, monitoring well installation and development, hydraulic testing, geophysical survey, and groundwater quality sampling. Groundwater has been analyzed for a suite of variables, including physical variables, major ions, and trace metals.

20.1 Environmental Studies – *Cont'd*

(a) Baseline Studies – *Cont'd*

Aquatic Resources

Baseline studies conducted on fish, fish habitat, and aquatic ecology found the fish community downstream of the project is composed of bull trout, mountain whitefish, torrent sculpin, longnose dace, and several salmon species. The distribution of fish is affected by the presence of natural barriers preventing fish from occupying the upstream reaches of creeks. As a result, the creeks within the project footprint are non-fish bearing.

The Harper Creek Watershed supports bull trout (the species that extends furthest up the watershed) and downstream populations of chinook salmon, coho salmon, sockeye salmon, and rainbow trout.

Vegetation

The project area is comprised of low stands of ponderosa and jack pine, hemlock spruce, with alder and birch occupying the creek drainages and North Thompson River valley. A large portion of the project area has been previously logged. The following two biogeoclimate zones are present within the mine site area:

- Engelmann Spruce – Subalpine Fir
- Interior Cedar – Hemlock

Two Red- and five Blue-listed vascular plant species, three Red- and two Blue-listed mosses, and 21 listed lichen species occur within 1 km of the mine site. While 12 ecological communities at risk were identified as potentially occurring in the project area, only three blue-listed communities were found during baseline studies. Eight wetlands totalling 200 ha and approximately 3,000 ha of old-growth forest have been identified in the project area.

20.1 Environmental Studies – *Cont'd*

(a) Baseline Studies – *Cont'd*

Wildlife

Wildlife species of conservation concern identified as potentially being present in the Canfor-Vavenby Defined Forest Area are included in Table 20-1.

Table 20-1: Wildlife Species of Conservation Concern with Potential to Occur in the Project Area

Species	Conservation Status			
	BC List	**COSEWIC**	**SARA**	**MBCA**
Amphibians				
Western Toad *(Anaxyrus boreas)*	Blue	SC	1-SC	
Birds				
Barn Swallow *(Hirundo rustica)*	Blue	T	-	Y
Common Nighthawk *(Chordeiles minor)*	Yellow	T	1-T	Y
Harlequin Duck *(Histrionicus histrionicus)*	Yellow	-	-	Y
Lewis's Woodpecker *(Melanerpes lewis)*	Blue	SC	-	Y
Long-billed Curlew (*Numenius americanus*)	Blue	SC	-	Y
Olive-sided Flycatcher *(Contopus cooperi)*	Blue	T	1-T	Y
Mammals				
Grizzly Bear *(Ursus arctos)*	Blue	SC	-	
Southern Mountain Caribou *(Rangifer tarandus caribou)*	Blue	E/SC	1-SC	
Wolverine *(Gulo gulo)*	-	SC	-	
Fisher (*Pekania pennanti*)	Blue	-	-	
Badger (*Taxidea taxus jeffersonii*)	Red	E	E	
Little Brown Myotis *(M. lucifugus)*	Yellow	E	-	
Northern Myotis *(M. septentrionalis)*	Blue	E	-	
Fringed Myotis *(M. thysanodes)*	Blue	Data deficient	-	

BC List: Red = Threatened, Blue = Special Concern, Yellow – Secure
COSEWIC Codes: E = Endangered, SC = Species of Special Concern, T = Threatened
SARA (*Species at Risk Act*): T = Threatened, SC = Special Concern, 1 = Schedule 1
MBCA (*Migratory Birds Convention Act*): Y = Yes

20.1 Environmental Studies – *Cont'd*

(b) Environmental Assessment

The previous owner of the property submitted an Application for an Environmental Assessment Certificate, also known as the Environmental Impact Statement, to the BC Environmental Assessment Office and to the Canadian Environmental Assessment Agency (CEA Agency) in January 2015. This document described a previous design of the project and was required to fulfill the requirements of both the British Columbia Environmental Assessment Act (BC EAA; 2002), and the Canadian Environmental Assessment Act (CEAA; 1992).

The EAO terminated the provincial application in July, 2018 due to inactivity on the file and Taseko withdrew the federal application in May, 2019.

Although the environmental assessment was not completed, concerns typical of proposed mine development were identified during the EA process by regulatory personnel, the public, and First Nations. These were related to excess stored water and potential effects on water quality, fish and aquatic habitat, wildlife, trapping, vegetation, cattle, and First Nations values.

Taseko has addressed these concerns in the current design and will restart the application process with the provincial and federal governments once project redesign is sufficiently advanced. Taseko believes there will be no issues remaining that would materially affect the ability of Taseko to extract minerals as part of developing the project.

20.2 Waste Rock and Tailing Storage, Water Management and Site Monitoring

Plans for waste rock storage are provided in section 16 while tailings storage and water management are described in section 18.

Taseko will hold and maintain necessary permits for any work that takes place in, on, or about the mine and will comply with all provisions of provincial and federal legislation, regulations, conditions of permits issued, and the BC Mines Act "Health, Safety and Reclamation Code for Mines in British Columbia" (Code). A full list of monitoring and reporting obligations associated with the project will be developed during the permitting process. Monitoring activities associated with necessary permits, authorizations, licenses, regulations and the Code may include:

- Workplace contaminants to ensure employees are not exposed to airborne concentrations of chemical agents or noise in excess of the levels specified in section 2.1.1 of the Health, Safety and Reclamation Code;
- Surface and ground water quality monitoring downstream of the project area;
- Air quality in the vicinity of the project infrastructure and emission sources;
- Aquatic life downstream of the project area;
- Development and maintenance of an annual inventory of greenhouse gas emissions;
- Soils handling and reclamation throughout mine life to ensure that reclamation is successful and that a self-sustaining vegetation which cover meets end land use objectives is established;
- Geotechnical stability of structures, including pit walls and embankments;
- TSF performance;
- Waste rock handling including material volumes.

Post-closure activities may include water treatment and a continuation of environmental monitoring conducted during the history of the project including:

- Periodic geotechnical inspections, such as the TSF embankments;
- Continued evaluation of water quality and flow rates downstream of the project;
- Continued evaluation of aquatic life downstream of the project; and
- Soil and vegetation monitoring on reclaimed landscapes.

Taseko will be responsible for all environmental monitoring and reclamation programs until such time as all permit conditions have been fulfilled and Taseko has been released from all obligations under the BC Mines Act.

20.3 Permitting

(a) Environmental Assessment

A mining project similar to that proposed for the project typically goes through a formal environmental review process and if approved can then receive the necessary permits and approvals for construction and operation.

Taseko has engaged with both the BCEAO and the IAC regarding the Yellowhead project but it is not yet formally in the environmental assessment process.

The BC review process will be required under BC EAA's Reviewable Projects Regulations that stipulates that any new mineral mine that has a production capacity of 75,000 tonnes per year or more is reviewable under the BC EAA. The current plan estimates processing of 33 million tonnes per year of ore. BCEAO is expected to confirm that an assessment is required by issuing an EAA Section 10 Order which will state that, in order for the project to proceed, an EA certificate needs to be issued after the review of the EA application.

Federally, the Impact Assessment Act came into effect in August 2019 and applies to projects described in the Physical Activities Regulation. This regulation identifies new metal mines with ore production capacity of 5,000 tpd or more, or a metal mill with an ore input capacity of 4,000 tpd or more. As the project exceeds these capacities, submission of an Initial Project Description is required by the proponent to IAC.

Subsequent agency consultation with federal and provincial authorities, indigenous groups, and the public results in the agency preparing a Summary of Issues. The proponent then submits a Response to the Summary of Issues and a Detailed Project Description so the agency can determine whether an impact assessment is required. It is expected that the agency will confirm that an impact assessment is required.

20.3 Permitting – *Cont'd*

(b) Federal Permits, Licenses, Authorizations and Approvals

For explosives storage, approval will be required under the Explosives Act. An authorization from Transport Canada will be required for aeronautical clearance for the overhead transmission line crossing of the North Thompson River. Other federal permits, licenses or approvals that may be required for the construction, operation, or closure of the project are the following:

- Environment Canada – Metal Mining Effluent Regulations (MMER) under the Fisheries Act, as water will ultimately be discharged from the site during operations and into post closure;
- Fisheries and Oceans Canada – Fisheries Act authorizations may be required, although current field data and presence of downstream barriers suggests that the mine site area is not providing habitat to any fish species, and proposed transmission line crossings will be designed to avoid habitat disruption in riparian areas.

It is expected that during the EA process and further discussion with federal departments, the nature of any federal authorizations will be confirmed.

20.3 Permitting – *Cont'd*

(c) BC Permits, Licenses, Authorizations and Approvals

A list of provincial permits, licences and approvals that may be required for the project follows:

- BC Ministry of Energy, Mines and Petroleum Resources (BCMEMPR):
    - Mineral Tenure Act:
        - Mining Lease;
    - Mines Act Permit:
        - Approval of the Mine Plan;
        - Approval of the Reclamation Plan;
- BC Ministry of Forests, Lands and Natural Resource Operations (BCMFLNRO):
    - Land Act Authorizations:
        - Licence of Occupation;
    - Water Act:
        - Approvals for "Changes In and About a Stream" (section 9);
        - Water licences for water wells, new sediment control/detention ponds and surface water diversion, storage and use;
    - Forestry Act Licence:
        - Occupation Licence to cut;
    - Heritage Conservation Act:
        - Section 14, Inspection Permit;
        - Section 12, Site Alteration Permit;
    - Provincial Forest Use Regulation:
        - Special Use Permit for use of new and existing road access;
- BC Ministry of Environment (BCMOE):
    - Environmental Management Act permits:
        - Effluent Discharge Permit (e.g., SSMF, sewage, etc.);
        - Air Discharge Permit;
        - Discharge to Land Permit – disposal of refuse;
        - Fuel Storage Permit;
        - Sewage Registration – sewage disposal facility;
- Ministry of Transportation (MOT):
    - Transportation Act, Motor Vehicles Act:
        - Utility Permit;
        - Interior Health Authority;
    - Public Health Act:
        - Food Premises Permit;
        - Drinking water;
        - Filing of Certification Letter for sewage disposal facility;

20.3 Permitting – Cont'd

(c) BC Permits, Licenses, Authorizations and Approvals

- Drinking Water Protection Act and Regulations:
  - Construction Permit;
  - Operating Permit;

It is expected that during the EA process and the exchanges with BC regulatory authorities, more specific requirements will be refined.

There are currently no permit applications under review with provincial or federal regulatory bodies.

20.4 Social and Community Relations

(a) 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.

Taseko is committed to hiring local people. During the construction period the project is expected to employ a peak workforce of approximately 900 people. When fully operational, the project will support about 550 direct jobs and approximately 1,500 indirect jobs in the area.

20.4 Social and Community Relations – Cont'd

(b) 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, Taseko understands that Adams Lake is a member of the Lakes Division of the Secwepemc Nation 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.

20.5 Mine Closure and Costs

(a) Reclamation and Closure

In British Columbia, mining companies are required to reclaim mine disturbance when mining is complete in accordance with the Code.

The definitive closure phase will begin at the cessation of mineral processing and tailings deposition. At this time water treatment will be discontinued while all contact water is directed to the open pit. Decommissioning of site infrastructure and reclamation will be completed early in this period. The following are key activities related to the closure period:

- Allow the supernatant pond to flow through the TSF spillway to the open pit. This will continue until TSF water quality enables direct discharge to the environment;
- Direct all other contact water to the open pit until water quality at specific sources allows direct discharge to the environment;
- Stabilize and revegetate TSF embankments and beaches;
- Remove site buildings and infrastructure, recontour site and revegetate;
- Recontour and revegetate waste rock stockpiles and roads. Sufficient road access will remain to maintain closure and post-closure activities;
- Establish open pit water reclaim infrastructure to feed water treatment plant starting in post-closure;
- Environmental monitoring and follow-up including monitoring of water quality, reclamation success, stability of remaining site infrastructure, annual reporting to government.

The post-closure phase begins when the open pit has filled with water and treatment of pit water is restarted. Physical activities in this period are related to water management and treatment. Environmental monitoring will continue including monitoring of water quality, reclamation success, stability of remaining site infrastructure, and annual reporting to government. The transmission line will be removed when water treatment is no longer required or an alternate power source is provided. Further discussion of post-closure requirements will occur during the EA and subsequent permitting processes. This period will continue until all conditions of the Code and permits have been fulfilled and Taseko has been released from all regulatory obligations.

20.5 Mine Closure and Costs – *Cont'd*

(b) Mine Closure Costs

Before any work on a site is conducted, the province requires companies to provide security in accordance with the Code.

The reclamation security amount will be developed as part of the permitting phase.

SECTION 21

CAPITAL AND OPERATING COSTS

SECTION 21: CAPITAL AND OPERATING COSTS

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21.1 Pre-Production Capital Costs

(a) Pre-Production Capital Cost Summary

A summary of the pre-production capital costs estimated for the project is provided in Table 21-1. All costs shown are in Q4, 2019 Canadian dollars. Foreign currency exchange rates utilized for the capital cost estimate are listed in Table 21-2 based on Q4 2019 Canadian dollars.

Table 21-1: Summary of Pre-Production Capital Costs

Area	Total Pre-Production Capital ($ millions)
Mining Equipment*	100
Pre-Production Mining Costs	69
Processing Facilities	486
Tailings Storage Facility & Water Collection	132
Ancillary Facilities	79
On-Site Infrastructure	101
Off-Site Infrastructure	19
Subtotal Direct Costs	**986**
Indirect Costs	143
Owner’s Costs	49
Contingency	168
Subtotal For Indirect Costs	**360**
Grand Total	**1,347**

Note: totals may not add due to rounding
*Includes down payment and lease costs in pre-production years only.

Table 21-2: Foreign Currency Exchange Rates

Canadian $	Currency	Exchange
1.00	US Dollar	0.80
1.00	Euro	0.67

No allowances have been made for escalation, interest and financing, taxes or working capital in the capital cost estimate. The accuracy level for the estimate is ±20%.

Further details on the basis for these costs are included in the following sections.

21.1 Pre-Production Capital Costs – *Cont'd*

(b) Direct Costs

Mining Equipment and Pre-Production Mining Costs

The mining equipment capital cost estimates are based on budgetary quotes supplied by equipment manufacturers. All capital costs are FOB to the project site and include recommended options, assembly and commissioning. Other mine capital includes the cost of logging and grubbing the pit area in preparation of mining and installation of power distribution to the pit area. The capitalized pre-production mining costs are derived from the mine operating costs estimated for the material mined in the two years prior to mill start-up except for the cost of material delivery to tailings facility construction which is included in the tailings cost.

All of the major mining equipment will be purchased new except for production drills which will be purchased used. Major mining equipment (shovels, trucks, dozers, cleanup loaders and graders) will be leased at current mine equipment lease terms over a period of five years.

Table 21-3: Mine Pre-Production Capital Costs

Capital Item	Total Pre-Production Capital ($ millions)
Mine Equipment Lease Payments*	74
Mine Equipment Purchases	14
Light Vehicles, Maintenance Equipment & Technology Systems	10
Subtotal All Mine Equipment	**98**
Other Mine Capital	2
Capitalized Pre-Production Mining Costs	69
Subtotal Other Mine Capital	**71**
Total Mine Capital	**169**

Note: totals may not add due to rounding

*Includes down payment and lease costs in pre-production years only.

21.1 Pre-Production Capital Costs – *Cont'd*

(b) Direct Costs – *Cont'd*

Processing Facilities

This area includes all of the process equipment, structures and systems required to produce a copper concentrate from ROM ore feed. The facilities included are the primary crusher, material handling systems, coarse ore stockpile and reclaim system, grinding circuits, mineral separation circuits, concentrate dewatering, process water reclaim and distribution system and mill general facilities. The direct capital costs for the area are detailed in Table 21-4.

Table 21-4: Concentrator Direct Pre-Production Capital Costs

Area	Total Pre-Production Capital ($ millions)
Crushing & Conveying	74
Stockpile & Reclaim	34
Grinding	137
Mineral Separation	56
Dewatering	22
Process Water Reclaim & Pond	26
Mill Building & General	136
Total	**486**

Note: totals may not add due to rounding

Tailings Storage Facility and Water Collection

This area includes all of the systems, structures and equipment for the TSF and site contact water collection systems. The area includes the main embankment, NAG rockfill mining and hauling costs for embankment construction, rougher and cleaner tailings pipelines and spigoting system, main dam seepage collection and return pumping system, site contact water collection and pumping systems, pit dewatering system and the roads required to access all of this infrastructure. The direct capital costs for this area are detailed in Table 21-5.

21.1 Pre-Production Capital Costs – *Cont'd*

(b) Direct Costs – *Cont'd*

Table 21-5: TSF and Water Collection Pre-Production Capital Costs

Activity	Total Capital ($ millions)
Tailings Embankments & Earthworks	54
Rockfill Mining & Hauling Costs	25
Tailings Mechanical Systems	31
Water Collection & Management	21
Total	**132**

Note: totals may not add due to rounding

Ancillary Facilities

This area includes the ancillary systems and structures required to support the site mining and processing operations. This includes the mobile equipment shop, warehouse, fuel storage, earthworks for explosives manufacturing site, assay laboratory, reagent storage, reagent make-up, rebuild shop, main office, mine dry and various other outbuildings around the site. The direct capital costs for this area are detailed in Table 21-6.

Table 21-6: Ancillary Facilities Pre-Production Capital Costs

Area	Total Capital ($ millions)
Mine Ancillary Facilities	44
Process Ancillary Facilities	10
Site Ancillary Facilities	24
Total	**79**

Note: totals may not add due to rounding

On-site Infrastructure

This includes the infrastructure on the mine site required to support the site mining and processing operations. The area includes the plant site preparation, bulk site earthworks, plant site roads, fire protection systems, potable water system, sewage treatment system, main substation, site power distribution network, emergency power generators, site communications network and process control systems. The direct capital costs for this area are detailed in Table 21-7.

21.1 Pre-Production Capital Costs – *Cont'd*

(b) Direct Costs – *Cont'd*

Table 21-7: On-Site Infrastructure Pre-Production Capital Costs

Activity	Total Pre-Production Capital ($ millions)
Plant Site Earthworks	38
Utilities and Services	4
Power Distribution	44
Site Communications & Process Controls	14
Total	**101**

Note: totals may not add due to rounding

Off-site Infrastructure

This includes the infrastructure external to the mine site which is required to support the operation, including power supply from the Vavenby substation to site, offsite communications, site access road, and rail load-out facility. The total direct capital cost for this area is shown in Table 21-8.

Table 21-8: Off-Site Infrastructure Pre-Production Capital Costs

Activity	Total Pre-Production Capital ($ millions)
Site Power Supply	9
Roads and Communications	2
Rail Load-Out	8
Total	**19**

Note: totals may not add due to rounding

21.1 Pre-Production Capital Costs – *Cont'd*

(c) Indirect Costs

This area includes the costs for services and temporary infrastructure required on the site to support the construction and pre-development mining activities. The project indirect capital costs are detailed in Table 21-9.

Table 21-9: Indirect Pre-Production Capital Costs

Item	Total Pre-Production Capital ($ millions)
Temporary Construction Facilities & Services	20
Construction Camp, Catering	19
Vendor Representatives	1
Start-Up & Commissioning	4
Epcm	89
Capital and Maintenance Spares	9
First Fills	16
Owner's Costs	32
Contingency	168
Total	**360**

Note: totals may not add due to rounding

Temporary Construction Facilities & Services

This area includes all of the temporary infrastructure required to execute the project as well as construction support services and mobile equipment not supplied by the construction contractors. The estimate was based on the anticipated project schedule and recent project experience. The items estimated in this cost include, but are not limited to, the following:

- Temporary construction service and warehouse facilities;
- Construction and site maintenance equipment not supplied by contractors;
- Materials testing and quality assurance;
- Site survey;
- Site maintenance;
- Waste management;
- Material off-loading and construction warehouse services;
- Construction power supply;
- Scaffolding;
- Site security, safety and fire protection;
- Janitorial services;
- Owner supplied worker transportation to site.

21.1 Pre-Production Capital Costs – *Cont'd*

(c) Indirect Costs – *Cont'd*

Construction Camp

A temporary construction camp made up of modular units will be mobilized and demobilized based on the project construction schedule. Camp catering, janitorial, maintenance, waste water treatment and potable water services have been incorporated in the costs. An initial 150 person camp will be mobilized at the start of the construction phase during the initial site preparation work expanding to a 540 person camp at peak utilization. Average camp operating costs are estimated at $83/person.

Vendor Representatives

Vendor representative costs were based on historical project data and include both vendor requirements during construction and commissioning. The costs include the vendor service rates as well as the anticipated vendor travel, lodging and expenses costs.

Start-up & Commissioning

These costs include the required contract support to start-up and commission the site facilities. The owner's team costs related to start-up and commissioning are included in Owner's Costs. The items included in this area are:

- Contractor support to assist with the pre-commissioning and commissioning of the facilities;
- Electrical equipment and protective relay setting and testing;
- Contract process control system support;
- EPCM commissioning support.

EPCM

The project EPCM costs were estimated on a percentage basis from the project direct costs accounting for items which were quoted as design build and items which will be managed by the owner.

Capital & Maintenance Spares

The capital and maintenance spares were estimated based on a percentage of the purchase cost for mechanical and electrical equipment with an additional allowance for a spare grinding mill motor. The maintenance spares were estimated based on 2.5% of direct costs.

21.1 Pre-Production Capital Costs – *Cont'd*

(c) Indirect Costs – *Cont'd*

First Fills

The first fills includes the costs for purchase of the necessary consumables to commence operations at the site. Costs for the purchase of grinding media, reagents, lubricants, fuel, mine tires and miscellaneous supplies are included in the estimate based on vendor supplied quotes.

Owner's Costs

The Owner's Costs estimated for the project include the anticipated costs incurred by the owner from the time the project is authorized to proceed through to production. Costs for work preceding a project authorization are not included in the estimate. The items estimated in this cost include, but are not limited to, the following:

- Owner's project management personnel;
- Pre-production mine engineering personnel;
- Ramp up and training of permanent operations, maintenance and administration personnel;
- Field office costs and supplies;
- Environmental testing and monitoring;
- Recruiting and relocation;
- Transportation and accommodations costs for owner's personnel;
- Insurance;
- Taxes, fees and licenses;
- Off-site road maintenance.

Contingency

The capital cost estimate includes separate contingencies on the pre-production mining and the construction of the process facilities and associated infrastructure to cover costs of materials and labour within the scope of the project but not estimated. A 5% contingency has been applied on the acquisition of the mining fleet and mobile equipment. An average 20% contingency has been applied to direct costs for the construction of process facilities and associated infrastructure. An average 15% contingency has been applied to direct costs for the construction of the TSF. The contingency levels for each discipline and area were determined by the project team based what they believe is appropriate in consideration of technical risk and the level of engineering work performed for this study.

21.1 Pre-Production Capital Costs – *Cont'd*

(d) Basis of Estimate

The capital cost estimate is based on the use of all new equipment and materials for the project except for production drills which will be purchased used. The direct cost estimate includes supply and installation of the equipment and materials required to construct all of the permanent facilities associated with the project. Mine production equipment (shovels, trucks, dozers, cleanup loaders and graders) are purchased on a capital lease basis. The major permanent facilities for the project scope are:

- Pre-production mining and pit equipment;
- Infrastructure, roads and site preparation;
- Process buildings;
- Crushing, material handling and process facilities;
- Water reclaim and distribution system;
- Assay laboratory;
- Administration building;
- Warehouse;
- Cold storage;
- Mobile equipment maintenance shop;
- Fuel storage;
- Fixed plant maintenance shop;
- Mine sry;
- Power supply and distribution;
- Emergency generators;
- Plant site services and utility systems;
- Tailings storage facility and deposition systems;
- Site contact water collection and management systems;
- Plant mobile equipment.

Labour rates for each required construction trade were set based on current rates received from British Columbia contractors. A crew composite labour rate for each trade was calculated which includes:

- Base labour wage rate;
- Benefits and burdens;
- Overtime allowance;
- Small tools and consumables;
- Safety supplies;
- Contractor overhead and profit;
- Appropriate crew compositions;
- Contractor travel allowance.

21.1 Pre-Production Capital Costs – *Cont'd*

(d) Basis of Estimate – *Cont'd*

The rates are based on all installation work being done by external contractors carried out on a schedule of 21 10-hour workdays, followed by 7 days off with overtime premium included in the labour rates.

The capital cost estimate includes a total of approximately 1.6 million man hours of direct and indirect labour associated with construction activities. 1.3 million man hours of the total are associated with direct construction activities. The average labour rate in the estimate for all construction activities is approximately $127 per man hour.

Project direct costs were estimated based on the following information:

- Site topography, layout and preliminary general arrangement drawings as well as process flow diagrams, equipment lists, electrical single line diagrams and some drawings from previously constructed projects where applicable;
- Budgetary quotations for the supply and erection of the major process and ancillary buildings and the supply of major equipment;
- Secondary and ancillary equipment prices based on a combination of budgetary quotations and database prices from recently completed projects;
- Prices for bulk construction materials were based on database prices from recently completed projects;
- Labour rates sourced from contractors in the Province of British Columbia;
- Equipment installation time and labour efficiency based on recent project experience adjusted for site specific conditions and vendor guidelines where appropriate;
- Freight costs to site based on a combination of budgetary quotations and recent project experience.

Capital Cost Exclusions

The follow items are excluded from the capital cost estimate:

- Escalation;
- Financing costs and interest during construction except for leased mining equipment;
- Costs due to currency fluctuations;
- Scope changes;
- Schedule delays;
- Reclamation bonding;
- Closure costs;
- Salvage values;

21.2 Sustaining Capital

(a) Sustaining Capital Cost Summary

Sustaining capital is estimated to be $624 million for the life of the project. The sustaining capital estimate includes a water treatment plant, staged TSF embankment construction, additional water collection systems, additional mining equipment, mining equipment lease payments, and general sustaining capital through the life of the mine. Sustaining capital costs are shown in Table 21-10.

Table 21-10: LOM Sustaining Capital Costs

Area	Total Sustaining Capital ($ millions)
Water Treatment Plant	48
Staged TSF Construction	80
Water Collection Systems	12
Incremental Mine Capital	38
Mining Equipment Lease Payments	237
General Sustaining Capital	209
Total	**624**

Note: totals may not add due to rounding

21.2 Sustaining Capital – *Cont'd*

(b) Sustaining Capital Components

Water treatment will be implemented in stages based on water discharge requirements with the initial treatment plant and discharge system constructed in year 2 followed by expansions in years 9 and 20 to increase treatment and discharge rate.

On-going tailings capital costs for operating the tailings facility includes:

- Raises to the rockfill starter dam up to year 5;
- Toe drains and foundation preparations for transitioning to cyclone sand construction in years 1 through 6;
- Mechanical systems for transitioning to cyclone sand dam construction including stationary and mobile equipment;
- Construction of the north and northwest tailings embankments and seepage collection ponds in years 11 through 16;
- Piping and spigot systems for depositing tailings at the north embankment in year 14;
- Seepage collection system upgrades, tailings cyclone feed pumps and miscellaneous mechanical upgrades throughout the project life.

Water collection system upgrades are included in years 2 and 10 for incremental pit dewatering system upgrades as the pit expands and for implementation of additional contact water collection.

Additional mine capital is included as the equipment requirements for the mine plan increase and as new pit phases are developed. This includes an additional used rotary production drill in year 4, five additional haul trucks in years 13-15 and development costs for new pit areas in years 3 and 10.

Mining equipment lease payments are included for mining equipment as discussed in the section 21.1 (b). Lease payments continue from start of production through year 5 of the project.

General sustaining capital accounts for maintaining the integrity of the mining, processing and support equipment as well as the site facilities and buildings through capital replacements and major capital repairs. General sustaining capital is estimated to average $0.11 per tonne mined over the life of mine.

21.3 Operating Costs

(a) Operating Cost Summary

Onsite operating costs comprise mining, processing and general and administration. Average onsite costs for the project are summarized in Table 21-11.

Table 21-11: Summary of Onsite Operating Costs

Area	Cost per Tonne Milled ($/t)
Mining	4.53
Processing	4.65
G&A	0.79
Total onsite	**9.97**

Note: totals may not add due to rounding

Offsite costs include copper concentrate transportation costs, smelter fees and deductions, and royalty payments. Average offsite costs are US $0.39/lb.

21.3 Operating Costs – *Cont'd*

(b) Mine Operating Costs

The mine operating cost estimates are built up from first principles and include fuel, lubricants, tires, ground engaging tools, consumables, routine parts and non-routine component replacement costs, operating and maintenance labour. Table 21-12 below summarizes the mining operating costs used in this study.

Table 21-12: Unit Mining Costs

Mine Process	Cost Per Tonne Mined ($/t)
Drilling	0.10
Blasting	0.32
Loading	0.24
Hauling	0.82
Pit Support	0.26
General Mine Expense	0.19
Total Mining Cost	**1.93**

Note: totals may not add due to rounding

The equipment productivities for the primary mining equipment (drills, shovels and trucks) are calculated from the basic operating capacities of the equipment, the travel speed characteristics of the trucks, and the haul road profiles as described in section 16 of this report. Equipment operating hours determined in the mine plan are multiplied by the hourly consumables consumption rates and unit operating costs to calculate the total equipment operating costs for each year.

Support equipment operating hours are estimated based on experience and historical performance from the Gibraltar Mine which is a similar sized, mature facility operated by Taseko. As with the primary mining equipment, costs are derived from estimated operating hours in each year and consumable consumption rates.

Blasting costs are based on a vendor quote for operating a down-the-hole delivery basis assuming a bulk explosives manufacturing facility is constructed at the project site. Bulk explosives quantities are calculated based on mining rates and powder factors used in the mine plan. Blasting accessories costs are based on the calculated number of blastholes and unit costs of accessories per hole.

General mine expenses include staff and supervision costs, pit dewatering activity and non area-specific operating expenses. These costs are based on factors derived from experience at the Gibraltar Mine and labor calculations described below.

21.3 Operating Costs – *Cont'd*

(b) Mine Operating Costs – *Cont'd*

Operations labor requirements are calculated based on equipment hours estimated in the mine plan for each class of equipment. Additional labor for non-equipment support roles has been estimated based on experience from the Gibraltar Mine. Maintenance labor requirements are based on current staffing at the Gibraltar Mine for a nearly identically-sized mining fleet. Labor costs are calculated based on total manpower counts and fully burdened annual labor costs.

Technical services are allocated to G&A.

21.3 Operating Costs – *Cont'd*

(c) Process Operating Costs

Process operating costs incorporate crushing, conveying, grinding, flotation, concentrate dewatering, process water reclaim, general building services, tailings cycloning and deposition, site contact water collection and water treatment costs. Table 21-13 summarizes typical annual and unit costs by area and category.

Table 21-13: Process Operating Costs

Mill Input	Cost Per Tonne Milled ($/t)
Crushing, Conveying & Stockpile Reclaim	0.22
Primary Grinding	2.32
Flotation & Regrind	0.71
Process Water Reclaim	0.05
Maintenance	0.33
Labor	0.55
General Mill Expense	0.21
Subtotal of Direct Milling Costs	**4.39**
Tailings Deposition & Management	0.10
Water Collection & Management	0.07
Water Treatment & Discharge	0.09
Subtotal Tailings & Water	**0.26**
Total Milling Cost	**4.65**

Note: totals may not add due to rounding

Operating costs for the direct milling unit operations include reagents, consumables, mill and crusher liners, grinding media, and electricity consumption. Input costs were obtained from vendor supplied quotes and consumption rates were estimated based on the laboratory scale test work described in section 13 of this report combined with operating experience derived from the Gibraltar Mine.

Mill maintenance costs include all general parts, component rebuilds and replacement equipment for all areas of the concentrator and related systems. Costs have been estimated using a factor of the direct costs for the concentrator.

Mill labor costs include staff and hourly manpower for operating and maintenance areas. They are based on the Gibraltar Mine wage structures and are fully burdened. Manpower structures were developed based on process requirements.

General mill costs include general mill utilities, assay lab costs, contract services requirements, mill mobile equipment costs and site services. Costing is based on first principals buildups combined with experience from the Gibraltar Mine.

21.3 Operating Costs – *Cont'd*

(c) Process Operating Costs – *Cont'd*

Tailings management includes all aspects of managing the tailings sand deposition and cyclone operations required for raising the embankments. Mobile equipment costs are included using the same methodology as described for mine support equipment. An allowance for contract engineering services is included for annual safety reviews and routine engineering work by an independent engineer of record.

Water management systems will be used to collect all site contact water and deliver it to the mill via the process water pond and ultimately into the tailings storage facility. Costs are estimated based on the water volumes collected at various locations and the electrical consumption required to pump the water to the process water pond. The water management costs increase over time as the mine impacted areas expand. An allowance is made for routine maintenance based on factored direct capital costs.

Water treatment costs begin in year 3 of the project and escalate similarly to the water collection costs. Water treatment costs include all consumables and dedicated operators required to operate the WTP. Water discharge costs are minimal as clean water will be discharged using gravity to the watercourses below the mine.

21.3 Operating Costs – *Cont'd*

(d) General and Administration Costs

General and administration (G&A) costs for the project include the labour cost as well as expenses and services associated with the following:

- Mine engineering;
- Materials management;
- Human resources;
- Safety and security;
- Accounting;
- Environmental monitoring;
- Personnel transport to/from site;
- Insurance;
- Taxes, fees and licenses;
- General administrative costs.

The G&A labour costs for employees were based on the organizational structure developed for the project and salaries based on operating experience at the Gibraltar Mine. Other G&A costs, including site consulting requirements and recruiting costs, were estimated based on a combination of operating experience at the Gibraltar Mine, budgetary quotations and estimates as appropriate.

Table 21-14 summarizes the G&A costs by category.

Table 21-14: General & Administration Costs

Life of Mine G&A Costs	Cost Per Tonne Milled ($/t)
Mine Engineering	0.11
Environmental Monitoring	0.11
Materials Management	0.14
Human Resources	0.12
General Administrative	0.29
Head Office	0.02
Total G&A Cost	**0.79**

Note: totals may not add due to rounding.

21.3 Operating Costs – *Cont'd*

(e) Offsite Costs

Offsite costs include transportation costs, smelter fees and deductions, and royalty payments. Average offsite costs are summarized in Table 21-15.

Table 21-15: Summary of Offsite Costs

Area	US$/lb
Concentrate Transportation	0.18
Smelter Fees & Deductions	0.21
Royalty Payments	0.00
Total Offsite Cost	**0.39**

Note: totals may not add due to rounding

Concentrate transportation costs include:

- Trucking to Vavenby and rail load-out operations by a contractor;
- Rail transportation costs to the Port of Vancouver including rail car leasing costs;
- Port storage and handling fees;
- Sampling;
- Ocean freight to overseas smelters.

All concentrate transportation and handling costs are estimated using the Gibraltar Mine rates except for trucking and load-out operations which are based on a vendor supplied quote.

Smelter costs include treatment costs for concentrate and refining costs for all contained metals. No deleterious elements are expected in the project concentrate. Current market rates for smelter terms have been used.

Two royalties exist for the project and are considered in the offsite costs. The first is a $3.0 million lump-sum royalty payment is expected in the first year of operation. This royalty is based on an effective date from 2010 and escalated to current dollars based on the effective date of this report. The second is a 2.5% NSR royalty associated with six mineral claims and is not subject to escalation.

SECTION 22

ECONOMIC ANALYSIS

SECTION 22: ECONOMIC ANALYSIS

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22.1 Introduction

The mineral reserves are supported under the cost and performance data presented in the previous sections of this report. Metal prices are based on street consensus metal pricing as of Q4 2019 and long-term foreign exchange rates based on Taseko's expectations informed by historical exchange rates as shown in Table 22-1. A discounted net present value (NPV) cashflow model using a discount rate of 8% is used for the valuation basis with an effective date of December 31, 2019. Results of the valuation are presented on a 100% basis and assume no debt financing costs except for mining equipment leases discussed in section 21 of this report. All values are in Canadian dollars unless otherwise noted.

Table 22-1: Street Consensus Long-Term Metal Pricing and Foreign Exchange Rate

Long-Term Forecasts	Metal Price
Copper Price	US$3.10/lb
Gold Price	US$1,350/oz
Silver Price	US$18.00/oz
Foreign Exchange	US$0.80 : CAD$1.00

22.2 Pre-Tax Cashflow

Pre-tax economic indicators for the project are presented in Table 22-2.

Table 22-2: Pre-Tax Economic Valuation for the Yellowhead Project

Economic Indicator	Value
Average Annual Pre-Tax Cash Flow	$270 million
Pre-Tax NPV at 8%	$1.3 billion
Internal Rate of Return	18%
Payback Period	4.2 years

The pre-tax cashflow for the project is presented in Table 22-3.

Table 22-3: Pre-Tax Yellowhead Project Cashflow

Project Period		Pre-Production Total	Years 1-5 Total	Years 6-10 Total	Years 11-15 Total	Years 16-20 Total	Years 21-25 Total	Grand Total
Copper Production	(M lbs)		1,030	760	840	850	960	**4,440**
Gold Production	(000 oz)		120	70	80	80	90	**440**
Silver Production	(000 oz)		3,760	3,550	3,850	4,600	3,640	**19,400**
Operating Profit	(C$ B)		2.0	1.0	1.2	1.2	1.9	**7.3**
Capital Costs	(C$ B)	1.3	0.4	0.1	0.1	0.1	0.0	**2.0**
Net Cash Flow	(C$ B)	-1.3	1.6	0.9	1.1	1.1	1.9	**5.3**

Note: totals may not add due to rounding

22.3 Taxes and Royalties

The Yellowhead project is 100% owned by Taseko. Two royalty obligations exist on the project and are included with the offsite operating costs discussed in section 21 of this report.

Profits will be subject to taxation by the provincial and federal governments. At long-term metal prices, the project's estimated tax payments are summarized in Table 22-4.

Table 22-4: Estimated Project Taxes (LOM)

Item	Value
BC Mineral Taxes	$0.6 billion
Corporate Income Taxes	$1.3 billion
Total Taxes	$1.9 billion

(a) BC Mineral Tax

Currently the provincial government in British Columbia collects taxes relating to mineral production referred to as BC Mineral Tax. BC Mineral taxes are assessed under a two-part system, made up of Net Current Proceeds Tax and Net Revenue Tax.

Net Current Proceeds Tax applies at a rate of 2% to operating cash flow from production. This tax applies until the producer has recovered applicable capital investments and a reasonable rate of return, at which time the Net Revenue Tax will apply at a rate of 13%. The total tax collected under both Net Revenue Tax and Net Current Proceeds Tax will not exceed 13%.

The development of the project will be eligible for a new mine allowance under the BC Mineral Tax. The new mine allowance is calculated as 1/3 of eligible capital expenditures from the development of the new mine and is applied in determining the Net Revenue Tax. BC Mineral taxes are deductible against corporate income taxes.

22.3 Taxes and Royalties – *Cont'd*

(b) Income Taxes

Currently corporate taxpayers resident in Canada are subject to a federal income tax rate of 15% and taxpayers resident in British Columbia are subject to a further 12%, for a total combined corporate income tax rate of 27%.

Taxable losses generated in a given year may be carried forward for 20 years and applied to taxable income when it arises and carried back 3 years and applied against taxable income if applicable.

Costs associated with exploration and development are allocated to certain resource pools and deductible against taxable income. Canadian Exploration Expenses (CEE) may be carried forward indefinitely and are fully deductible against taxable income. Canadian Development Expenses (CDE) may be carried forward indefinitely and are deductible against taxable income up to a maximum of 30% per year on a declining balance basis.

A depreciation rate of 25% per year is applied to capital expenditures on mining production equipment.

22.4 After-tax Economic Indicators

The project after-tax economic indicators, assuming current federal and provincial tax laws are in force are presented in Table 22-5.

Table 22-5: After-Tax Economic Valuation for the Yellowhead Project

Economic Indicator	Value
After-Tax NPV at 8%	$0.7 billion
After-Tax IRR	14%

22.5 Sensitivity Analysis

An analysis was performed to demonstrate the sensitivity of the economic valuation for the project to fluctuations in metal prices, ore grade, operating costs and capital costs on a before-tax basis. These results are shown in Figures 22-1 and 22-2. Project economics are most sensitive to copper price and ore grade followed by operating cost. Economics are less sensitive to capital cost and only marginally sensitive to precious metal prices.

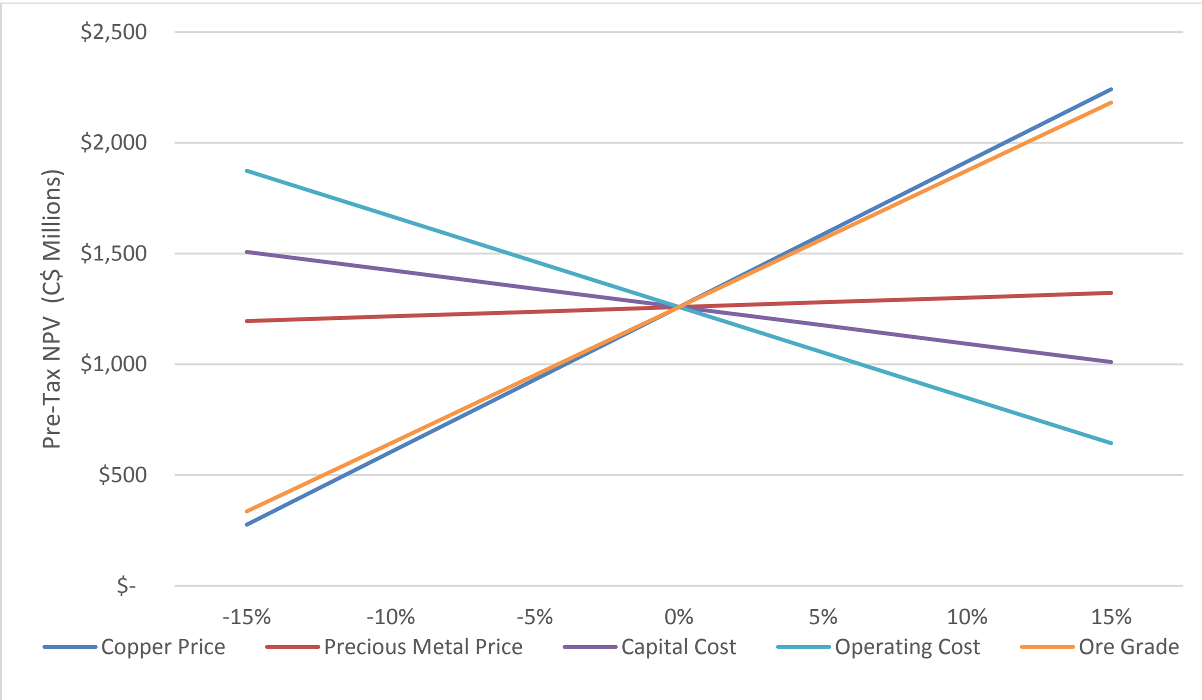

Figure 22-1: Before-Tax NPV Sensitivities

22.5 Sensitivity Analysis – *Cont'd*

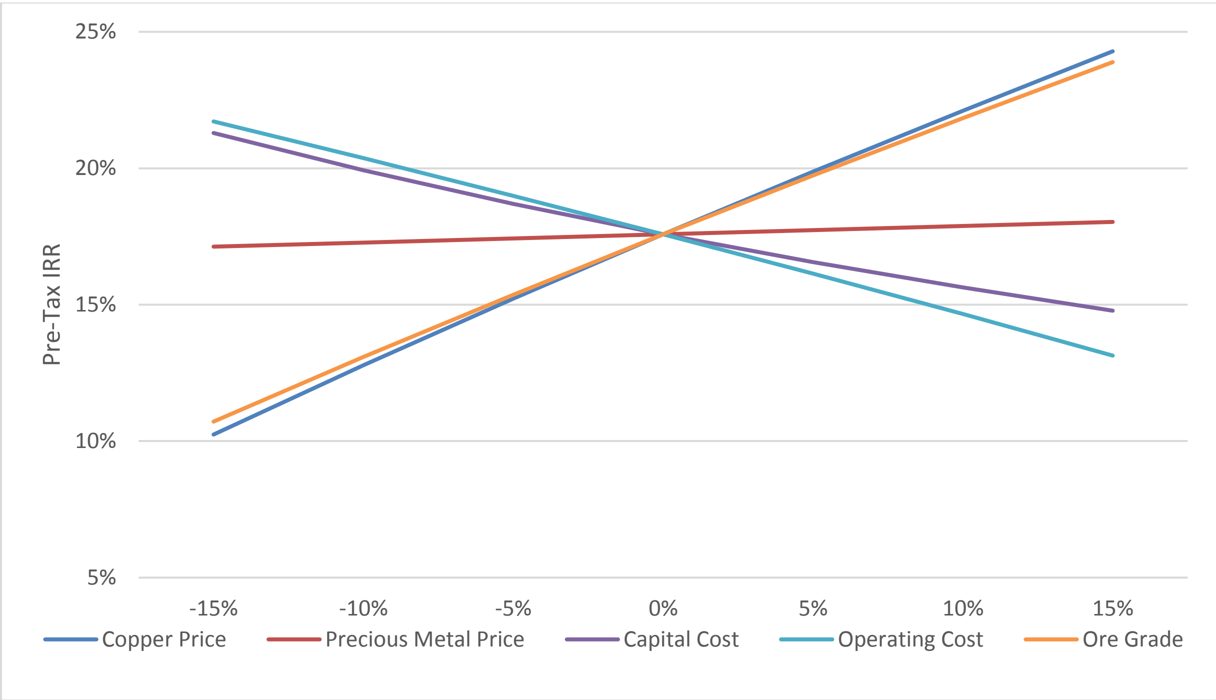

Figure 22-2: Before-Tax IRR Sensitivities

SECTION 23

ADJACENT PROPERTIES

SECTION 23: ADJACENT PROPERTIES

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23.1 Adjacent Properties

There are no adjacent properties as defined by NI 43-101.

SECTION 24

OTHER RELEVANT DATA AND INFORMATION

SECTION 24: OTHER RELEVANT DATA AND INFORMATION

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24.1 Other Relevant Data and Information

In the opinion of the author, there is no additional information necessary in order to make the technical report understandable and not misleading beyond that included in this report.

SECTION 25

INTERPRETATION AND CONCLUSIONS

SECTION 25: INTERPRETATION AND CONCLUSIONS

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25.1 Tenure and Environmental Liabilities

Taseko Mines Limited, through its wholly owned subsidiary Yellowhead Mining Inc., is the 100% owner of the Yellowhead mineral claims. All mineral claims are in good standing.

Six claims are subject to a 2.5% NSR royalty while 31 claims are subject to a 3% NSR royalty, capped at C$3.0 million, subject to inflation.

An application has been submitted to the BC Mineral Titles Office to convert 40 claims to a mining lease.

The Yellowhead property is subject to environmental liabilities related to previous exploration drilling programs. Funds to cover the expense of these reclamation activities are held in trust and are fully recoverable by Yellowhead Mining Inc. once the site has been rehabilitated.

25.2 Exploration and Geology

Evaluation of the exploration programs and results available to the effective date of this report indicates that:

- The geology is sufficiently well understood to support the mineral resource and mineral reserve estimations presented in this report;
- The drillhole database contains all drilling data collected on the project to date and has been structured for resource estimation;
- QA/QC with respect to the results received to date is acceptable and protocols have been well documented;
- As of December 31, 2019, the Yellowhead deposit is estimated to contain a measured and indicated resource of 1.3 billion tonnes grading 0.25% copper, 0.028 gpt gold, and 1.2 gpt silver using a cut-off grade of 0.15% copper. An additional 110 million tonnes averaging 0.24% copper is classified as inferred;
- As of December 31, 2019, the Yellowhead deposit is estimated to contain a proven and probable reserve of 817 million tonnes grading 0.28% copper, 0.030 gpt gold, and 1.3 gpt silver using a cut-off grade of 0.17% copper. This reserve is contained within the resource stated above.

25.3 Mining

The evaluations of the mining options available to effectively recover copper from this deposit indicate that:

- The Yellowhead property contains adequate mineral reserves to develop an open pit mine and supply a process plant with 90,000 tpd of ore for a period of at least 25 years;
- The detailed pit design is consistent with the design basis optimum pit shell and meets the recommended geotechnical design parameters. The final pit limit can be subdivided into 5 phases with adequate mining width for the selected mine fleet and an efficient ramp system provides access between the mining benches, the waste storage areas, and the primary crusher;
- The mine design provides a reasonable basis for the production schedule meeting the targeted mill feed rate of 90,000 tpd with a consistently sized mining fleet;
- Equipment and fleet sizing is based on appropriate assumptions and is adequate for the operation proposed;
- Mining losses and mining dilution are appropriately considered;
- The design and mine schedule are to a pre-feasibility level of study;
- The mine schedule uses only measured and indicated blocks within the resource estimate. Inferred resources are treated as waste.

25.4 Metallurgy and Processing

The evaluation of the metallurgy and processing options available to effectively recover copper from this deposit indicate that:

- A process that utilizes commercially available mineral processing unit operations consisting of crushing, three-stages of comminution, sulphide flotation and concentrate dewatering can be used to produce a clean copper concentrate with no penalty level deleterious elements;
- Recovery of copper can be expected to average 90%;
- Recovery of gold and silver can be expected to average 56% and 59% respectively;
- A processing facility can be constructed at a nominal throughput of 90,000 tpd of feed ore;
- Process tailings from the concentrator can be co-deposited with PAG waste rock in a tailings storage facility located in proximity to the processing facility.

25.5 Infrastructure

The infrastructure required has been adequately identified to support the project at the designed capacity. The design and cost estimation is to a pre-feasibility level and there are no known conditions that would preclude the establishment of the infrastructure as designed.

25.6 Environment

Environmental baseline studies have been advanced by a number of consultant groups to a level commensurate with initiating an environmental effects assessment.

25.7 Capital and Operating Costs

The estimation of capital and operating costs are based on a pre-feasibility level of engineering and are current to December 31, 2019.

25.8 Economics

The economics of mining and processing the stated reserves of this project are appropriate to demonstrating that, as of December 31, 2019, extraction can reasonably be justified.

25.9 Risks and Opportunities

The following project risks and opportunities have been identified:

(a) Risks

- Should the ore processed in the plant be different than the current master composite or samples representing variable lithology, then process recoveries, grades, and quantities may be different;
- Predictions on expected unit process recoveries may be different than those achievable in a pilot or industrial scale plant;
- Should the costs or availability of process reagents and steel consumables materially change, this could materially change the operating costs;
- The economics of the project are directly related and sensitive to the price of copper. While copper markets and demand are well established, copper prices are affected by numerous factors beyond the company's control, including demand growth, expectations with respect to the rate of inflation, the exchange rates of the United States dollar to other currencies, interest rates, and global or regional political, economic or financial situations;
- Site operating costs are subject to variation from one year to the next due to factors, such as changing strip ratio, ore grade, minerology, cost of supplies and services (for example, electricity and fuel) and the exchange rate of supplies and services denominated in foreign currencies. No assurance can be given that the estimates of production and unit cash costs of production will be achieved. Failure to achieve production or cost estimates or material increases in operating or capital costs could have an adverse impact on the project's profitability;
- The project is subject to currency exchange rate risk because the price of copper is denominated in United States dollars and, accordingly, the project's revenues would be received in United States dollars. The company's expenses would be almost entirely denominated in Canadian dollars. Taseko currently does not engage in foreign exchange hedging. Any strengthening of the Canadian dollar without a corresponding increase in commodity prices would negatively impact the profitability of the project;
- The project will require licenses and permits from various governmental authorities. There can be no assurances that Taseko will be able to obtain all necessary licenses and permits that may be required to carry out all proposed development and operations;
- Typical mining risks include adverse geological or ground conditions, adverse weather conditions, potential labour problems, and availability and cost of equipment procurement and repairs;
- There are uncertainties inherent in estimating mineral reserves and mineral resources.

25.9 Risks and Opportunities – *Cont'd*

(b) Opportunities

- There is a significant opportunity to reduce lime consumption and evaluate more selective flotation reagents while maintaining recovery, resulting in reduced processing cost in the concentrator and potentially increasing concentrate grade;
- Metallurgical test work to date indicates relatively low recovery sensitivity to primary grind size. There is an opportunity to increase the primary grind size and reduce both capital and operating cost;
- Opportunities exist to optimize water management and treatment infrastructure, resulting in reduced capital and operating costs while maintaining environmental protection.

SECTION 26

RECOMMENDATIONS

SECTION 26: RECOMMENDATIONS

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26.1 Recommendations

The following section identifies recommendations for two phases of work to advance the project towards a production decision. The two phases are not contingent on one another.

(a) Environmental Assessment

As per the provincial and federal regulatory requirements outlined in section 20 it is a reasonable assumption that an environmental assessment of the Yellowhead project will be required before the project can proceed to obtain permits for construction and operation.

Although a significant amount of baseline data has been collected and evaluated in support of an environmental assessment, additional site investigation data validation and preparation of an Environmental Impact Statement (EIS) will be required to proceed through an environmental assessment. It is recommended that this work be completed.

A summary of the scope and cost of this work is presented in Table 26-1.

Table 26-1: Cost Estimate for Environmental Assessment

Scope of Work	**Cost ($ millions)**
Baseline Data Validation and Collection	1
Geotechnical Site Investigation	2
Hydrogeological Drilling	1
Site Investigation Supervision and Data Analysis	1
EIS Preparation	3
Total	**7**

26.1 Recommendations – *Cont'd*

(b) Process Optimization

Metallurgical test work completed to date is consistent with the design, costing, and recovery which supports the mineral reserve that is the subject of this technical report. The author is of the opinion that there is significant opportunity to optimize the cost and recovery of the processing facilities through additional metallurgical test work. It is recommended that this work be completed before advancing to detailed design on the project.

A summary of the metallurgical bench test work and cost of this work is shown in Table 26-2.

Table 26-2: Cost Estimate for Proposed Metallurgial Test Work

Metallurgical Bench Test Work	**Cost ($ thousands)**
Evaluate Reduced Lime Consumption	30
Evaluate Reduced Grind Requirements with More Selective Flotation Reagents	60
Additional Filtration and Settling Test Work	10
Total	**100**

SECTION 27

REFERENCES

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I, Richard Weymark, P.Eng., MBA, of Vancouver, British Columbia, hereby certify that:

a) I am an employee of Taseko Mines Ltd., with a business office at 15th Floor, 1040 West Georgia Street, Vancouver, British Columbia. In my position as Chief Engineer, on behalf of Taseko Mines Limited, I authored this technical report on the mineral reserves at the Yellowhead Copper Project which was announced on January 16, 2020.

b) This certificate applies to the technical report titled "Technical Report on the Mineral Reserves at the Yellowhead Copper Project, British Columbia, Canada", dated January 16, 2020 which has an effective date of January 16, 2020.

c) I am a graduate of the University of British Columbia in Vancouver, B.C. (B.A.Sc. in Mining Engineering). I have practiced my profession for 11 years since graduation in 2008, in various roles, including supervisory positions, overseeing mine design and planning, resource and reserve estimation, open pit operations, business improvement, tailings dam construction, cost estimation, environmental assessment and project evaluation aspects. I am a member in good standing of Engineers and Geoscientists British Columbia, license number 46355. As a result of my experience and qualifications, I am a qualified person as defined in National Instrument 43 – 101 *Standards of Disclosure for Mineral Projects* ("NI 43 – 101").

d) My most recent personal inspection of the property was on September 11th, 2019.

e) I am responsible for the compilation of all sections of this report.

f) I am not independent of Taseko Mines Limited.

g) I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with this Instrument.

h) To the best of my knowledge, information, and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading as of the effective date of the report.

Signed at Vancouver, British Columbia on the 16th day of January, 2020.

"Signed and Sealed"

Richard Weymark, P.Eng., MBA